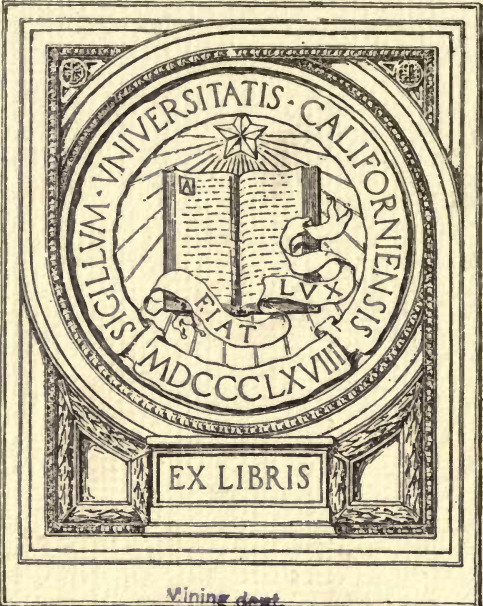


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HYDROMETALLURGY OF SILVER

WITH SPECIAL REFERENCE TO

CHLORIDIZING ROASTING OF SILVER ORES AND THE
EXTRACTION OF SILVER BY HYPOSULPHITE
AND CYANIDE SOLUTIONS

By

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PREFACE

THE silver ores which are treated by a hydrometallurgical process are mostly complex sulphide ores consisting of quite a number of different metal sulphides. In order to render soluble the silver contained therein the ore is roasted with an addition of salt (chloridizing roasting), by which process the silver is converted into silver chloride. In this chemical reaction, however, all or nearly all the constituent minerals of the ore participate, which makes the process rather complicated and we may even say delicate, because the formed metal chlorides are volatile and induce the silver chloride to volatilize too, and in order to keep this loss at the minimum great care and skill are required.

The solvent, whether this be sodium hyposulphite or concentrated brine, will extract all the silver which was converted into chloride, and the final result of extraction depends, therefore, entirely on the quality of the roasting. This being the case, it is obvious that chloridizing roasting is the most important part of the process, and that a metallurgist can not expect to be successful in the hydrometallurgy of silver unless he has a thorough knowledge of chloridizing roasting and the ability to apply skilfully this knowledge in actual practice. For this reason a large part of this treatise is devoted to the art of chloridizing roasting, which I consider to be especially necessary, as there is no new literature on this subject, though great advance has been made in it.

Chloridizing roasting was originally studied and practised in relation to amalgamation. In amalgamation not only the silver amalgamates, but the base-metal chlorides also amalgamate, which greatly interferes with the process, causing a poor extraction, a great loss of silver and mercury, and the production of very base bullion. To avoid these difficulties and to make the process applicable to a greater variety of ores, these objectionable metal chlorides are partly expelled and partly converted into

oxides by increasing the temperature of roasting. The expulsion of the volatile metal chlorides, however, induces quite a percentage of the otherwise not volatile silver chloride to volatilize, thus causing a considerable loss of silver.

I was the first to introduce the process of lixiviation with sodium hyposulphite in Mexico, in 1868, and made a special study in actual practice of chloridizing roasting, and in course of time, and as experience was gained with different ores, became convinced that chloridizing roasting as practised for amalgamation was not the proper way to roast for lixiviation. A large percentage of base-metal chlorides in the roasted ore is detrimental to a successful extraction of the silver by amalgamation, while in lixiviation they do not interfere with the extraction; why then should we expel and destroy these chlorides by increased heat at a sacrifice of silver? The expulsion of the volatile compounds by increased heat is the sole cause of the silver loss by volatilization, and if we avoid this we consequently will avoid this loss, or at least reduce it to the minimum. I therefore modified the mode of chloridizing roasting, inasmuch as, instead of expelling the volatile compounds by increased heat, I endeavored to retain them as much as possible in the roasted ore by using the lowest permissible temperature — a temperature sufficiently high to produce the chemical reaction but not high enough to expel the metal chlorides. Of course such a condition cannot be maintained with theoretical exactness, but I succeeded in greatly reducing the loss of silver by volatilization even with ores rich in arsenic. By this modification in chloridizing roasting a marked step forward was made in the hydrometallurgy of silver.

The second part of this treatise is devoted to the extraction of the silver from the roasted ore by different solvents. The last chapter deals with the cyaniding of silver ores. This process, so extensively and successfully used for the extraction of gold, is still more or less in its experimental stage with regard to silver ores. However, such very promising results have been obtained with certain ores, that further experiments and a thorough investigation of this subject are to be recommended. For complex auriferous silver ores a combination of the sodium hyposulphite and the cyanide processes is most promising.

By far the larger part of this treatise is a record of my long years of experience, studies, and experiments on a commercial

scale rather than the product of compilation, and it will be found that attention is paid to many things apparently unimportant but which, in actual practice, I have found to be of great importance, and on the correct execution of which the success of the enterprise may often depend.

I hope that this treatise will prove a friend and adviser to the student of hydrometallurgy as well as to the metallurgist in active service in the field.

OTTOKAR HOFMANN.

KANSAS CITY, Mo., January, 1907.

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PART I

CHLORIDIZING ROASTING OF SILVER ORES

I

THEORY OF CHLORIDIZING ROASTING

THE object of chloridizing roasting is to convert the silver in the ore into silver chloride, in which state, while not soluble in water, it becomes soluble in sodium hyposulphite and other solutions like hot concentrated brine, potassium cyanide, etc., by means of which it can be extracted from the ores. It is one of the most complicated, and in the hydrometallurgy of silver the most important, of metallurgical operations. The results of the subsequent extraction of the silver by the solvent depend entirely on the quality of the roasting. Silver chloride dissolves easily, and even a very dilute solution of sodium hyposulphite will extract all the silver which was converted into chloride during roasting, so that it is of the greatest importance that this part of the process be executed intelligently, and with great care and skill.

The ores which are subjected to chloridizing roasting are usually complex sulphide ores, though in some instances ores almost free of sulphides are roasted successfully, but these are exceptional cases. To effect chloridizing roasting chlorine has to be generated in the ore while being subjected to heat. This is done by an addition of salt (sodium chloride) to the ore. But not only the silver is converted into a chloride; all the constituent parts of the ore also undergo a change, quite frequently even the gangue. During the first part of the roasting the sodium chloride remains indifferent, while the metal sulphides oxidize, forming metal sulphates and sulphurous acid; then by the action of these sulphates on the salt rather complicated reactions take place, by which metal chlorides, chlorine, hydrochloric acid, and sulphurous chloride are formed.

The decomposition of the sodium chloride and the chlorination of the silver and other metals is effected in the furnace in different ways:

- (1) In oxidizing the metal sulphides there is always, besides

sulphurous acid gas, some sulphuric acid gas formed. Not so much in the beginning as later, when part of the metal sulphides, especially the iron, have changed into oxides, which then act as a contact substance on the sulphurous acid, converting it into anhydrous sulphuric acid, which then decomposes the sodium chloride. The formation of sulphuric acid increases much if a liberal amount of air is permitted to enter the furnace, and as the sulphuric acid plays an important part in chloridizing roasting, provision should be made, in the construction of the furnaces, that they may receive as much air as required.

(2) By the reaction between metal sulphates and the sodium chloride, by which metal chlorides and sodium sulphate are formed. This is the principal reaction for the formation of chlorides. The metal sulphates which act most energetically in this respect are those of iron and copper, for which reason ores containing an ample amount of iron pyrites and some copper sulphides will be found to chloridize the best.

(3) Besides chlorine, there is also hydrochloric acid formed, owing to the moisture in the air and fuel. Hydrochloric acid acts very energetically, and sometimes it is of advantage to produce larger quantities of it, in which case steam is admitted into the furnace to supply an extra amount of moisture.

(4) The fumes of volatilized salt (sodium chloride) act also in chloridizing the ore. Quartz decomposes the salt, forming silicate of soda and chlorine, but it takes a rather high heat for this reaction, and only in exceptional cases does it come into play.

(5) Volatile metal chlorides act also, chloridizing the silver, whereby they are reduced to subchlorides or changed into oxides. Cupric chloride acts very energetically in this respect.

If salt and ore are charged together, the salt is not decomposed until the formation of sulphates begins, and the first stage in roasting is, therefore, a mere oxidizing process. Whatever sulphuric acid is formed during this period by the oxidation of the sulphides acts more readily on the base metals, forming sulphates, than on the salt. Likewise it acts more readily on the lime and other earthy matters of the gangue. The principal part of the chlorination takes place by the reaction between the metal sulphates and the salt. The oxidizing and chloridizing periods are quite distinct, and can be easily observed by the appearance of the ore in the furnace and by the smell of the fumes of a sample

taken from the charge. During the oxidizing period the glow of the surface of the ore is much brighter than the inside, and the particles brought to the surface by stirring brighten instantly to a lighter red. The fumes of a sample have a strong, choking smell of sulphurous acid. During the chloridizing period, if an excessive fire is not kept up, the surface of the ore assumes a very dull red, while the deeper layers are of a brighter glow, which, however, becomes dull shortly after the particles are brought to the surface. The fumes of a sample have a mild but distinct odor of chlorine. The charge swells and becomes loose and woolly.

It was mentioned above that the ores, which are subjected to chloridizing roasting, are mostly complex argentiferous ores, and as the roasting is much influenced by the behavior of the constituent parts of the ore and has to be modified according to the requirements of one or the other of the constituents, a knowledge of the behavior of the different minerals and the changes they undergo during chloridizing roasting is therefore indispensable in order to conduct the process intelligently.

BEHAVIOR OF DIFFERENT MINERALS IN CHLORIDIZING ROASTING

Iron Pyrites. — During the oxidizing period, sulphurous and sulphuric acids are formed, of which the former escapes entirely, while part of the latter combines with lime and other earthy matters of the gangue, and part combines with the iron, forming sulphates. The iron changes into ferrous and ferric sulphates and into ferric oxide. The ferrous and ferric sulphates act on the salt, forming ferrous and ferric chlorides and sodium sulphate, while some of the chlorine combines with sulphur to form sulphurous chloride, which escapes as gas. In the course of the process both these iron chlorides give off their chlorine, chloridizing the silver and changing into ferric oxide. In practice the reaction is not quite so complete, and in the finished charge we find, besides the ferric oxide, some ferric sulphate and some ferrous and ferric chloride.

The iron chlorides decompose easily and act as the principal chloridizers, for which reason it is very desirable, in fact often necessary, to have iron sulphides in the ore.

Copper pyrites consists of the sulphides of copper and iron. During oxidizing, cupric sulphate, cuprous and cupric oxides are formed. The cuprous oxide, however, soon changes into cupric. During chloridizing, sulphurous chloride (which volatilizes), cupric and cuprous chlorides are formed. Both these copper salts melt below red heat, and are absorbed by the ore, thus becoming finely divided through the ore and coming in intimate contact with the silver. Both are volatile. At a higher heat the cupric chloride gives off part of its chlorine, chloridizing the silver and changing into cuprous chloride. In presence of steam, hydrochloric acid, cuprous and cupric oxides are formed.

The iron sulphide of the copper pyrites undergoes the same chemical changes as the iron pyrites. For this reason, and for the fact that cupric chloride gives off part of its chlorine, copper pyrites is a very good producer of chlorine during roasting.

In the roasted charge we find cupric oxide, cupric sulphate, ferric oxide, ferric sulphate, ferrous and ferric chlorides, and cuprous and cupric chlorides. If such a charge is subjected to a prolonged roasting at a high heat (dead roast), all the iron as well as the copper will be changed into oxide.

Other copper ores, like gray copper ore, fahlerz, etc., undergo the same changes.

Galena (lead sulphide) undergoes the changes much slower. It cakes easily, for which reason the temperature in the beginning has to be kept low until most of its sulphur has been oxidized. During oxidizing, sulphurous acid, lead oxide, and lead sulphate are formed. The lead sulphate does not decompose the salt at a roasting heat and, therefore, does not take an active part in the generation of chlorine. When air has free access, most of the lead is converted into sulphate and but little into chloride, while, if the supply of air is limited, much more lead chloride is formed, and thus becomes a consumer of chlorine. It is volatile, and volatilizes without giving off any chlorine. Lead oxide is volatile too, while the sulphate remains more indifferent. In the roasted ore we find lead sulphate and lead chloride, but much less of the latter.

By the above it can be seen that lead sulphide is not a desirable constituent part of a roasting charge.

Zinc Blende. — During the oxidizing periods zinc sulphide

changes into zinc oxide and zinc sulphate, but the sulphate does not act decomposingly on the salt. By the action of the chlorine and hydrochloric acid zinc chloride is formed, which is very volatile and goes off in heavy fumes, which increase when the temperature is raised. These escaping fumes induce the silver to volatilize, for which reason ores rich in zinc blende have to be roasted at a low heat to avoid an excessive loss of silver.

In the roasted ore we find principally zinc oxide, then zinc sulphate and chloride.

Zinc blende, as a rule, contains more or less iron sulphide, some of its varieties as much as 22 and even 28 per cent. The iron sulphide takes, of course, an active part in the generation of chlorine; still it takes much skill to chloridize satisfactorily the silver contained in zinc blende. This subject will be treated exhaustively in another chapter.

Arsenical Pyrites. — This consists of arsenic sulphide and iron sulphide. Arsenic is very volatile and begins to come off from the ore in dense fumes right at the beginning and before other sulphides are ignited. During this part of the process much arsenate of silver is formed, up to 50 and 54 per cent. of the total silver contained in the ore. This silver compound is soluble in a solution of sodium hyposulphite. During the chloridizing period, however, most of it is decomposed without volatilizing, if the temperature is kept low, but it volatilizes very readily at a high heat, causing a heavy loss in silver. Such ores have to be roasted at a very low heat. This subject is exhaustively treated in another chapter.

In roasting arsenical pyrites, arsenious oxide, sulphurous chloride, arsenic chloride, and ferric chloride are formed and volatilized. In the roasted charge we find ferric oxide, ferric sulphate, ferrous and ferric chlorides, and some ferric arsenate.

Antimony Sulphide. — This mineral we find quite frequently in complex silver ores, and if it occurs in large quantities the roasting has to be conducted very carefully and at a very low heat on account of its great volatility, which can cause a heavy loss of silver. During oxidizing it changes to oxide of antimony, of which a large portion is volatilized as such. During chloridizing antimony trichloride and sulphurous chloride are volatilized. In the roasted ore we find the antimony as antimony antimonate.

Quartz. — We find quartz quite frequently as gangue of the ore. At a proper roasting temperature quartz remains indifferent, but at a very bright heat it decomposes the salt, forming sodium silicate and chlorine. There are works in operation in which, for want of sulphur in the ore, the chlorination of the silver is produced partly by this reaction and partly by the chloridizing action of volatilized salt. It requires a high heat and a large percentage of salt. Quartz is the most desirable gangue in chloridizing roasting.

Carbonate of Lime (Lime Rock). — This mineral, which occurs quite frequently as gangue, or part of the gangue, acts as a rule unfavorably in chloridizing roasting. It takes an active part in the process. It combines with the sulphuric acid which is produced by the combustion of the metal sulphides, and it decomposes also the metal sulphates, forming sulphate of lime and metal oxides, thus preventing them from acting on the salt. It decomposes also metal chlorides, forming calcium chloride and metal oxides. Calcium sulphate is indifferent and does not act on the salt. If there is more carbonate of lime in the ore than can be converted into sulphate and chloride, part of it will be found in the roasted ore as caustic lime, which acts decomposingly on the silver chloride, especially so in the subsequent treatment for extraction, causing a poor result. If, however, there are more sulphides in the ore than necessary to convert the lime into sulphate and chloride, usually a good chlorination of the silver can be obtained, with the further advantage that the final silver precipitate will be very rich in silver, almost free from base-metal sulphides, and easily convertible into metallic silver of great fineness. Therefore, if lime is present in the ore in moderate quantities it is beneficial to chloridizing roasting. The loss of silver by volatilization will be found moderate, as most of the volatile chlorides are converted by the lime into oxides, which then are not volatile and will not induce silver chloride to volatilize.

Porphyry, Clay, Slate, and Other Gangue Minerals Containing Alumina. — F. Sustersic made the very interesting observation that under certain conditions a great loss of silver may be caused by the presence of alumina. The chlorine acts on the alumina, forming aluminum chloride, which is extremely volatile and induces the silver to volatilize. The conditions under which

this unfavorable reaction takes place were not ascertained. As a rule the gangue minerals named in this paragraph are more or less indifferent, and do not exercise a bad influence in chloridizing roasting.

CLASSIFICATION OF ORES IN RELATION TO CHLORIDIZING ROASTING

By the above-described behavior of the different minerals in chloridizing roasting it is apparent that chloridizing roasting of complex silver ores is undoubtedly one of the most delicate of metallurgical operations. The treatment has to be modified in accordance with the character of the ore, and the character of an ore in relation to chloridizing roasting depends on the nature of the different sulphide minerals and the gangue accompanying them. The sulphide minerals can be classified as:

(1) Those, like iron and copper pyrites, gray copper ore, silver copper glance, and argentite, which form in roasting sulphates which act on the sodium chloride and liberate the chlorine.

(2) Those, like galena and zinc blende, which form sulphates remaining indifferent to sodium chloride.

(3) Antimonial and arsenical silver minerals, which form antimonates and arsenates of silver.

The gangue either remains indifferent, like quartz and porphyry, or it takes an active part, like limestone, and minerals containing magnesia.

If an ore consists of minerals of the first class, together with an indifferent gangue, chloridizing roasting offers no difficulties nor does it require much skill, and a high chlorination can be obtained without much loss of silver by volatilization; nor does it matter whether the salt is added to the charge before entering the furnace or after it has been subjected to a partial oxidizing roasting.

The process of chloridizing roasting becomes more difficult if one or both minerals of the second class are present in large quantities, even if associated with an indifferent gangue. The roasting of this class of ore is elaborately treated in Chapter XI. With such ores the time the salt is added becomes very important. If added before the charge enters the furnace a very inferior chlorination is obtained, as is also the case if the salt is added before the oxidizing period has sufficiently advanced, or if

it is added when the period has too far advanced. The temperature and air supply require much attention.

The roasting is not less difficult if all three classes are represented, especially in connection with a gangue like limestone, which takes an active and often injurious part in the operation. This class of ore is treated elaborately in Chapter XII.

II

CRUSHING OF THE ORE

THE fineness to which an ore has to be reduced in order to give the best roasting result depends on the chemical and physical character of the material. As a rule, finely pulverized ore roasts quickly and gives a better result than a coarser material. Ores which decrepitate when charged in the furnace, or ores which during the combustion of the sulphur swell and disintegrate like iron pyrites, can be crushed rather coarse and still will give good chloridizing results. The ore of Sombrerete, Zacatecas, Mexico, gave good roasting results if crushed through a screen with 10, even with 8, meshes to the linear inch, though the ore contained much zinc blende and galena. The zinc blende, however, was of that kind which decrepitates, and besides, the ore was crushed in a stamp battery. In crushing in a battery the larger portion of the material is much finer than the size of the meshes calls for. This is particularly the case with heavy ores. It is doubtful if the same good result could have been obtained if the same ore had been crushed through rolls, because rolls produce a pulp much more uniform in size, with a much larger percentage corresponding in size with the size of the meshes of the screen.

I made some experiments in this direction with ore of the San Francisco del Oro mine, near Santa Barbara and Parral, Chihuahua, Mexico. The ore consists principally of a very dense zinc blende and finely divided galena. The zinc blende did not decrepitate. The zinc blende and the galena were the principal silver-bearing minerals of the ore.

A series of roasting experiments was made with ore crushed through 20- and through 40-mesh screens. The ore was crushed in a stamp battery. It was found that the ore crushed through 20-mesh required a much longer time and was 27 per cent. less chloridized than the ore crushed through the 40-mesh screen. The material which passes through a battery screen of certain

size is much finer than the size of the meshes. Heavy ore makes a much finer pulp through the same screen than lighter ore. The pulp of the Del Oro ore, obtained by crushing through battery screens No. 20 and No. 40, was sifted through sieves of different fineness, and the following figures obtained:

| BATTERY PULP WHEN SIFTED THROUGH SIEVE | CRUSHED THROUGH SCREEN No. 20 | CRUSHED THROUGH SCREEN No. 40 | CRUSHED THROUGH SCREEN No. 20 | CRUSHED THROUGH SCREEN No. 40 |
|----------------------------------------------|-------------------------------------------------------------|-------------------------------------------------------------|--------------------------------------------------------|--------------------------------------------------------|
| | PERCENTAGE OF MATERIAL PASS- ING THROUGH THE SIEVE | PERCENTAGE OF MATERIAL PASS- ING THROUGH THE SIEVE | PERCENTAGE OF MATERIAL REMAINING ON THE SIEVE | PERCENTAGE OF MATERIAL REMAINING ON THE SIEVE |
| No. 30 | 93.8 | 100 | 6.2 | 0.0 |
| No. 40 | 87.3 | 100 | 12.7 | 0.0 |
| No. 60 | 78.8 | 98.95 | 20.2 | 1.05 |
| No. 80 | 71.2 | 93.80 | 28.7 | 6.20 |
| No. 90 | 67.1 | 90.50 | 32.9 | 9.50 |

These figures show how exceedingly fine a heavy ore is crushed in a battery, even through a screen with comparatively coarse meshes. Though 67.1 per cent. of the material which was crushed through screen No. 20 was finer than sieve No. 90, the average chlorination of quite a number of compared roastings was 27 per cent. less than that of ore crushed through battery screen No. 40. This indicates how essential it is to crush such ores fine.

It is frequently argued in favor of coarse crushing that coarser crushed ore permits in the subsequent lixiviation a free percolation of the solution.

While to a certain extent coarsely crushed ore permits a somewhat quicker filtration, the increase (if extremes are avoided) is slight and of not much practical value. If a finely crushed ore filters too slow for an extraction by filtration it will filter too slow if it is crushed coarser, because in crushing always a certain amount of very fine powder (slime) is formed, no matter what kind of a pulverizing machine is used, and if the nature of the ore is such as not to undergo much of a physical change in roasting, the pulp in either case will contain sufficient slimes to interfere with a free percolation. A free percolation does not depend on the coarseness of the pulp nor on the nature of the gangue; it depends almost entirely on the nature of the sulphides and on the proportion of metal sulphides and the gangue. Besides the chemical changes which an ore undergoes during chloridizing

roasting, a change of its physical condition also takes place. Lead sulphate, which is formed in roasting, melts easily at a roasting temperature and is absorbed by the gangue and metal oxides. The same is the case with cuprous and cupric and with ferrous and ferric chlorides. They melt even below red heat and also penetrate the ore. By doing so, these metal salts collect all the dusty particles or slimes of the gangue and metal oxides into small porous globules and flakes, in which changed condition the ore permits a free percolation. This is the cause why a chloridized ore filters so much better than a raw ore, and if the ore contains a sufficient amount of metal sulphides it will filter well whether crushed very fine or whether it is crushed coarser. If finely pulverized the conditions for the chemical reactions, however, are much more favorable.

The melting of the metal chlorides and lead sulphate and their absorption by the ore causes the loosening and swelling of the charge, making it what is called "woolly," during the chloridizing period. It assumes a moist appearance and can be stirred without dusting, and does not evade the hoe as during the oxidizing period, but can be banked and collected into a pile. To maintain this condition the charge has to be agitated from time to time, otherwise a crust will be formed on the surface.

If for want of a sufficient amount of sulphides in the ore the formed chlorides and sulphates are insufficient to cause this physical change, the ore will remain dusty, run like water on the cooling floor, and will filter very slowly. On the other hand, if the amount of sulphides, especially lead sulphide, is too great in proportion to the gangue and metal oxides, the latter will get so saturated that they cannot maintain their loose condition, and form lumps. If the temperature is kept moderate these molten chlorides and lead sulphates will act like a cement, but will not go into chemical combination with the silica, and in most cases the lumps will be found to be porous and soft and as well roasted as the finer part. But if the heat is kept too strong silicates will be formed and the lumps will become dense and hard, and the chlorine will be unable to penetrate them and act on the silver. Particles of undecomposed sulphides will be enclosed in them and cause a poor extraction. The silver can be extracted from such lumps only if they are ground and re-roasted with steam, by which hydrochloric acid is formed, which acts on

the silicates. Without steam only a small percentage of the silver can be extracted.

Iron sulphides do not participate so much in changing the physical condition of the ore as lead or copper sulphides do, because the chlorides of iron easily give off their chlorine and change into oxide, which then acts like gangue. The most effective agent is the lead sulphide, the main part of which is changed into sulphate, which is but very little volatile at a roasting heat and does not undergo any further changes, thus much improving the filtering quality of an ore.

In working the refuse dump of the Cusihiuriachic mine, Chihuahua, Mexico, containing from 25 to 30 oz. silver per ton, I found that, while a satisfactory chlorination of the silver could be obtained, the silver could not profitably be extracted on account of the exceedingly slow filtration caused by too great an excess of porphyry gangue. It occurred to me to add a small percentage of galena, and the effect was very gratifying — the ore filtered well. Later the slow filtration trouble was overcome by applying trough lixiviation.

III

PERCENTAGE OF SALT REQUIRED

If all the chlorine of the salt could be transferred to the silver only an insignificant amount of salt would be required, but as other metals, which usually are present in much larger quantities than silver, are also chloridized, a correspondingly large percentage of salt has to be added to the ore. The amount to be added depends on the nature of the ore and has to be ascertained empirically in each individual case. It is best to commence with a high percentage, say 10 per cent., of salt, and to reduce the salt 1 per cent. in each succeeding roasting charge until 3 per cent. is reached. Ores which can be chloridized with less than 3 per cent. are very rare. The roasted charges are tested in the laboratory for silver chloride. It will be found in most cases that 10 per cent. of salt does not produce a higher chlorination than 6 or 5 per cent., and the experimenter will decide on the least amount of salt which produces as good a chlorination as the next larger amount, and will adopt that percentage. There are instances, however, where it will be found of advantage not to produce the highest possible chlorination, but to be contented with a somewhat inferior extraction. This is the case when the cost of the extra amount of salt exceeds the value of the additional amount of silver gained. This occurs usually in treating the lower grade ores in remote localities, where the price of salt is high.

Ores containing a large percentage of lead and zinc require less salt than ores rich in iron and copper sulphides, because the main part of the lead is converted into lead sulphate, which remains indifferent during the chloridizing period and does not consume any chlorine. This is also the case with zinc, which is mostly converted into zinc sulphate and oxide, which remain indifferent. Most of the iron and copper, however, is converted first into chlorides before they change into oxides, and of course

these are heavy consumers of chlorine, and the ore therefore requires more salt. For instance, the ore of the San Francisco del Oro mine in Mexico, which is very heavily mineralized, containing zinc 24.08 per cent., lead 11.92 per cent., iron 7 per cent., copper 0.5 per cent., and sulphur 21.35 per cent., required only $3\frac{1}{2}$ to 4 per cent. of salt.

An excess of salt does not improve chlorination; on the contrary, in many instances I have observed that the chlorination already gained was reduced by adding more salt. For this and for economical reasons an excess, therefore, should be avoided, especially as undecomposed salt in the roasted ore is not advantageous in the subsequent extraction.

THE PROPER TIME TO ADD THE SALT

The generation of chlorine in the furnace does not commence until, by the oxidation of the sulphur, metal sulphates have formed, which then act on the salt. The first part of the process, therefore, is an oxidizing process, whether the ore contains salt or not, and in this respect it would be immaterial at what time the salt were added. That the ore sustains a heavier loss of silver by volatilization if the salt is added before the oxidizing period is not conclusively proved, and actually there is no reason for it. Chlorides are not formed until the sulphates are formed, and therefore the presence of salt cannot cause a greater volatilization of the silver. An ore which is apt to lose silver on account of its arsenic and antimony sustains the larger part of its loss during oxidizing roasting.

There are ores, however, which cannot be chloridized successfully if the salt is added to the ore in the beginning. This is the case with ores which contain a large percentage of a dense argentiferous zinc blende, or argentiferous galena, or both, as the principal silver-bearing minerals of the ore. The reason why such ores have to be first subjected to an oxidizing roasting before the salt is added is the following:

Zinc blende, if subjected to oxidizing roasting, changes into zinc oxide and zinc sulphate, while sulphurous acid escapes. The process of oxidizing the zinc blende progresses but slowly, especially if the mineral is very dense. Iron and copper sulphides, on the other hand, oxidize easily and are converted into sulphates

long before this is the case with the zinc and lead sulphides. Zinc and lead sulphates do not act decomposingly on the salt, while iron sulphate does so energetically. Now, if the ore and salt are charged together we will find that the iron sulphate, as soon as it is formed, will act on the salt, producing chlorine and transforming itself into chloride and oxide. The chlorides of iron are volatile, and also give off the chlorine, changing into oxide. While this process is going on the zinc and lead sulphides are only partly oxidized, and as the chlorine in roasting has but very little effect on the raw zinc blende and galena, the silver contained therein will not be chloridized by the time the generation of chlorine and the action of iron chloride has ceased. The consequence is a very inferior roasting result. If, however, the ore is charged into the furnace without salt and subjected to an oxidizing roasting until the zinc and lead sulphides are oxidized, or to a certain extent oxidized, and then the salt is added, the generated chlorine and the iron chlorides will find the silver in a state in which it will combine with the chlorine. Iron sulphate requires considerable heat to be decomposed directly into oxide and sulphuric acid, and if the heat during oxidizing roasting is kept low, there will be sufficient iron sulphate in the charge to decompose the salt, and a quite satisfactory chlorination of the silver will be effected. To this class of ores belong those of the San Francisco del Oro mine, Chihuahua, Mexico, and of Sombrerete, Zacatecas, Mexico.

Another instance of great difference in the behavior of the ore, whether the salt was added to the ore in the battery or during the oxidizing period in the furnace, I experienced during my investigation of the chloridizing roasting of the calcareous arsenical silver ore at Yedras, Sinaloa, Mexico. The gangue of this ore consisted of silicious limestone and calcspar, while the ore proper consisted of argentiferous arsenical pyrites, a moderate amount of fine-grained black zinc blende, arsenical fahlerz, and some iron pyrites. When the roasting was done in the Brückner furnace there was a marked difference in the behavior of the ore. When the salt was added in the battery, the ore swelled, became woolly, kept on one side of the revolving furnace, and when discharged did not dust and remained in a pile on the cooling floor. When the salt was added toward the end of the oxidizing period, the ore did not assume the moist appearance so charac-

teristic in chloridizing roasting, but remained very loose and level in the revolving cylinder, and when discharged made much dust and ran on the cooling floor like water. The percentage of silver chlorination was in both cases about the same, but the ore which contained the salt at the beginning formed a very large amount of hard balls, which increased in size as the roasting progressed. They consisted of concentric layers and were smooth and hard. They were well chloridized, but the silver could not be extracted unless they were first pulverized, as they were too dense to permit the solution to percolate through them.

The sulphureted part of the ore had no tendency to form lumps, as numerous experiments with concentrates of the same ore showed. In this case we have an instance in which the time of adding the salt was conditioned by the nature of the gangue (see Chapter XII).

If the salt is added later, it is not necessary to dry and pulverize it; in fact it is better not to do it. It saves expense, and, besides, it is difficult to spread it uniformly over the charge, and in places where more salt drops it is apt to form lumps. The action of finely pulverized salt commences immediately on touching the ore, and in doing so it becomes sticky, which makes it difficult to divide and to mix it evenly. This is still more the case in a Brückner furnace. If coarse salt is added, the crystals, which usually are of the size of beans and have more or less moisture, coming in contact with the hot ore decrepitate quite rapidly. The particles fly in all directions, striking the roof and sides and falling back to the ore. When decrepitation ceases the salt will be found much more evenly scattered over the charge than this can be done by a shovel, and the disintegrated particles are small enough for the purpose. The chemical action does not commence quite as soon as with pulverized salt, and a much better mixing can be secured. Of course, larger lumps of crystals cemented together, or pieces of salt crust, have to be mashed first. Salt fuses and is absorbed by the ore, thus coming in intimate contact with the sulphates. A Brückner furnace should not be set to revolve until decrepitation ceases.

To extend the oxidizing roasting to such a degree as to produce a "dead roast," that is, to convert all the convertible sulphates into oxides and then to produce the chlorination by an addition of a mixture of calcined copperas (ferrous sulphate) and salt, is

by far too slow and expensive a method to be adopted in practice.

To add the salt during crushing produces a very uniform mixture of ore and salt and simplifies operations in roasting, for which reason it is preferable to do so if the nature of the ore permits it; still it is not frequently done in practice, unless the construction of the furnace demands it, because by crushing ore and salt together the crushing capacity of the machinery is reduced by the amount of salt added, and even more if the latter is not previously very well dried.

IV

LOSS OF SILVER BY VOLATILIZATION

SILVER chloride as such is not volatile, but if influenced by the volatilization of other chlorides it becomes volatile. A high heat, therefore, indirectly causes a larger loss of silver by the expulsion of larger quantities of volatile chlorides. Other conditions being equal we shall always find the loss of silver to be in direct proportion to the chemical loss in weight an ore sustains. In other words, the charge of the same ore that during roasting sustains the least chemical loss in weight sustains also the least loss of silver by volatilization. The term "chemical loss in weight" is used in distinction to the loss an ore sustains during roasting by dusting, which is a mechanical loss.

The logical consequence of the above facts is that the operator, while he endeavors to obtain a high silver chlorination, should be at the same time careful to expel as little as possible of the volatile chlorides. He will be greatly assisted in this endeavor by keeping the ore in a thick layer, and by using low heat and plenty of air. If a small charge is thinly spread over a large hearth more volatile chlorides will be expelled, and the ore will lose more in weight and in silver than when a larger charge is roasted in the same furnace. This is the reason why, as a rule, the loss in weight and in silver in a large Brückner furnace, in which the ore lies two feet thick, is less than in a reverberatory, and why small samples roasted on a roasting dish in the muffle show so much greater loss of silver than the same ore does when roasted on a large scale in the furnace.

It will be found that ore roasted at a low heat with sufficient air will lose less in weight, because a large part of the volatile chlorides, which at a higher heat would be expelled, will then remain in the ore. For amalgamation it is desirable, in fact necessary, to expel the volatile chlorides as much as possible, because they take an active part in amalgamation and make the

quicksilver smeary and inactive, causing a poor silver extraction and a very low-grade bullion. These chlorides, however, do not seriously interfere in the lixiviation process; in fact, it is one of the principal advantages of lixiviation over amalgamation that in it the volatile chlorides do not need to be expelled, and therefore the roasting of most ores, even those rich in arsenic and antimony, can be conducted with a very small loss of silver by volatilization. In metallurgical books we always find the great loss of silver pointed out as an objection to all processes which require chloridizing roasting of the ore. Formerly chloridizing roasting was principally used and studied in relation to amalgamation; little or no attention was paid to roasting for lixiviation, or to the fact that this process allowed a modification of roasting by which its objectionable features could be obviated.

I made chloridizing roasting the subject of special study, and found that it could be conducted with just as little loss of silver as oxidizing roasting, if care was taken to expel as little as possible of the volatile chlorides. The chemical reaction between salt and the sulphates takes place at a very low heat, in fact at a lower heat than is generally believed, while on the other hand it takes quite a high heat to expel thoroughly the volatile chlorides; therefore, in roasting for lixiviation the temperature can be kept as low as the nature of the ore permits during oxidizing, and lower still during chloridizing, and yet have the ore well prepared for the subsequent extraction of the silver. During chloridizing the ore ought to be kept in a thick layer and stirred only at intervals to diminish the volatilization of the chlorides.

In the old method of chloridizing roasting the aim was to free the ore by heat from metal chlorides that are objectionable for the subsequent extraction of the silver, while in the new method the aim is to retain in the ore as much of the chlorides as possible and to remove them by leaching with water previous to the extraction of the silver. If we take into consideration the fact that the otherwise not volatile silver chloride becomes volatile by the volatilization of other metal chlorides, it is quite logical that the volatilization of the silver will be greatly reduced by the modified method.

In roasting the calcareous arsenical silver ore at Yedras, Sinaloa, Mexico, by the modified method, with plenty of air, the loss in weight was only 3.5 per cent. and the loss in silver by

volatilization only 1.8 per cent., while if roasted by the old method the loss in weight was found to be from 7 to 13 per cent., while the loss of silver was 15 to 25 per cent. and more.

METHOD OF ASCERTAINING THE LOSS OF SILVER BY VOLATILIZATION

In order to roast skilfully it is of great importance to ascertain frequently the loss of silver by volatilization, but to do this it is necessary to know the loss in weight the ore sustains. This, however, is accompanied with great difficulty if it is done in the old way by actual weighing of the charge before and after roasting, necessitating the careful cleaning of the furnace and the dust-chambers before and after the process. In many cases this is not possible without seriously interfering with the regular work, and at all events it is accompanied with so much trouble and expense that if the shrinkage in weight is once ascertained, this figure is used in all subsequent calculations, though the conditions under which the roasting is performed, such as heat and draft or the character of the ore, may have changed. That such figures are not very reliable will be readily understood, but still more incorrect is the method some adopt of roasting 10 or 20 grams in the muffle and then taking the difference in weight before and after roasting as the loss in weight the ore sustains in roasting; by this means the loss of that particular sample in the muffle is ascertained, but not the loss the ore would lose in the furnace. Just as incorrect is the practice of roasting 10 grams in the muffle, of using the roasted 10 grams for an assay, and of comparing the assay value per ton with the assay value per ton of the raw ore. This gives us only the amount of silver this particular sample lost by volatilization, but it gives no information as to how much the ore loses if roasted in the furnace, because the conditions under which the roasting in the two cases takes place are very different with regard to temperature, draft, time, and thickness of the layer.

To conduct the roasting properly it is not of great importance to know how much the ore loses by dusting, for this is merely a mechanical loss, and the fine ore particles carried away by the draft are easily collected in dust-chambers. The loss due to the volatilization of the chlorides is the serious one. These fumes

are often richer in silver than the ore, are difficult to collect, and easily escape. We have, therefore, to find how much the ore in the furnace loses in weight by volatilization in order to obtain a correct basis for a calculation of the loss of silver in roasting.

I adopted the following method, which gives sufficiently correct results for practical purposes, can be performed in the assay office in a few hours, and is at all events more correct than if the loss in weight of the ore is ascertained by actual weighing of the charge and flue-dust.

Ten grams of the raw pulp, containing the same percentage of salt as the ore in the furnace, are placed in a roasting dish and roasted in the muffle for half an hour or an hour; then the sample is removed from the muffle, allowed to cool, weighed, returned to the muffle, roasted again for half an hour, and then weighed again. This is repeated until two weighings are alike, or until in the last half-hour the ore does not lose more than 2 or 3 mg.; then the difference between the original weight and that of the last weighing, expressed in percentage, gives the highest possible loss the raw ore can suffer.

Ten grams of a sample of roasted ore, corresponding with the sample of raw pulp, are placed in a roasting dish, and also roasted in the muffle until two weighings agree, or the difference between two consecutive weighings is not more than 2 or 3 mg. The difference between the first weighing (10 grams) and the last, expressed in percentage, gives the weight which the roasted ore is still capable of losing if subjected to prolonged roasting. If we deduct, therefore, the capable loss from the highest possible loss, we obtain in percentage the loss in weight the ore has suffered during roasting in the furnace by volatilization.

In the following, the weighings are given of one of the tests which I made with ore roasted in Brückner cylinders at Yedras, Mexico:

RAW ORE, CONTAINING 7 PER CENT. SALT

| | |
|---------------------------------------------------------------|-----------|
| Original weight. | 10 grams. |
| After 1 hour roasting in the muffle. | 9.35 " |
| After $\frac{1}{2}$ hour more roasting in the muffle. | 9.23 " |
| After $\frac{1}{2}$ hour more roasting in the muffle. | 9.21 " |

Ten grams - 9.21 grams = 0.79 grams = 7.9 per cent. highest possible loss in weight.

ROASTED ORE

| | |
|-----------------------------------------------------------|-----------|
| Original weight..... | 10 grams. |
| After 1 hour roasting in the muffle..... | 9.65 " |
| After $\frac{1}{2}$ hour more roasting in the muffle..... | 9.51 " |
| After $\frac{1}{2}$ hour more roasting in the muffle..... | 9.50 " |

Ten grams - 9.5 grams = 0.5 grams = 5 per cent. loss, which the ore, roasted in the furnace, was still capable of sustaining by dead roasting.

| | |
|-------------------------------------------|---------------|
| Highest possible loss of raw ore..... | 7.9 per cent. |
| Capable loss of roasted ore..... | 5.0 " |
| Actual loss in weight in the furnace..... | 2.9 per cent. |

The gangue of the Yedras ore is limestone. Agreeing weights, however, are more quickly obtained if ores have quartz gangue, while ores containing considerable manganese take a longer time, and require more patience. It is advisable to pulverize the ore carefully once or twice in a porcelain mortar during the test, in order to break up small lumps which have formed.

As this test is so quickly and easily done, it gives the metallurgist the means of ascertaining the most favorable temperature and proper time, and of controlling the work of the man in charge of the furnace. The mere difference of assay value between raw and roasted ore is no guide, as can be seen in the following table, in which the results of a few tests are given, which I made with the ore of the Cusihuriachic Silver Mining Company, Chihuahua, Mexico:

| ASSAY VALUE OF RAW ORE PER TON OUNCES | ASSAY VALUE OF ROASTED ORE PER TON OUNCES | HIGHEST POSSIBLE LOSS OF RAW ORE PER CENT. | CAPABLE LOSS OF ROASTED ORE PER CENT. | ACTUAL LOSS IN WEIGHT SUSTAINED IN ROASTING PER CENT. | LOSS OF SILVER BY VOLATILIZATION PER CENT. | REMARKS |
|---------------------------------------|-------------------------------------------|--------------------------------------------|---------------------------------------|-------------------------------------------------------|--------------------------------------------|-------------------------|
| 46.0 | 43.6 | 7.0 | 6.0 | 1.0 | 6.2 | |
| 41.8 | 41.6 | 7.3 | 5.5 | 1.8 | 2.2 | |
| 43.2 | 41.0 | 7.5 | 5.3 | 2.2 | 7.2 | Roasted in "C.'s" shift |
| 43.2 | 41.6 | 7.5 | 6.2 | 1.3 | 4.9 | Roasted in "L.'s" shift |
| 51.0 | 50.0 | 7.8 | 6.6 | 1.2 | 3.1 | |

The roasting was done in Howell furnaces. Each of the above tests was made with average samples of a whole day's roasting. The third and fourth sample, however, represent the ore roasted in one day, one roasted in C.'s shift (night) and the other in L.'s shift (day). Corresponding samples were taken of the raw ore. The salt was added to the ore in the battery. The

assay showed that the value of the raw ore was in both shifts the same, while the assay value of the roasted ore of both men was nearly the same, L.'s assaying only 0.6 oz. more. Judging by the assays, we are apt to think that the work of both men was nearly alike, but by referring to the column showing the percentage of silver volatilization, we find C. lost 7.2 per cent. while L. only 4.9 per cent. silver. These figures also tell the cause why C. lost more silver. The ore roasted by him showed a capable loss in weight of 5.3 per cent. while that roasted by L. showed a capable loss in weight of 6.2 per cent. C. therefore roasted at too high a heat, expelling unnecessarily more volatile chlorides, and by doing so increased the loss of silver.

V

METHODS OF ROASTING

CHLORIDIZING SELF-ROASTING

THIS mode of roasting can only be successfully performed with highly sulphureted ore and in a furnace the construction of which permits the roasting of large charges, like the Brückner type of furnaces. We have seen above that in roasting for the process of lixiviation with sodium hyposulphite it is not necessary to expel the metal chlorides by increasing the heat to bright red toward the end of the chloridizing period, but that, on the contrary, the roasting should be conducted at a low heat to the very end, to retain in the roasted ore as much of the metal chlorides as possible in order to reduce the loss of silver by volatilization. Reflecting on this principle, it occurred to me, while roasting heavily sulphureted ore in a Brückner furnace, that the charge if once ignited may, by the oxidation of the sulphides, produce and keep in store sufficient heat to finish the chloridizing part of the process without applying any additional heat. Experiments showed that this could be successfully done, and that not only was 50 per cent. of the fuel saved, but that, while the chlorination of the silver was 5 per cent. higher, the loss of silver by volatilization was materially less than by applying a second fire. It was possible in this way to roast 10.6 tons of ore with one cord of wood.

The charges should not be smaller than $4\frac{1}{2}$ to 5 tons, otherwise the heat stored in the ore will die out before the roasting is finished. When the furnace is charged, a strong fire is kept up until the ore has fairly started to roast; then the fire is allowed to go out, or if necessary pulled out, and the fire-door left open to allow a sufficient supply of air to pass through the furnace. The heat gradually increases though the fire is out. The charge maintains nearly a horizontal position. In due time the ore loses its brightness, increases in volume, and begins to assume a more erect

position, leaning against that side which moves upward. The chloridizing period has commenced. While during oxidizing the ore looks bright and the furnace lining dark, just the reverse can be observed during chloridizing: the surface of the ore looks dark while the lining, emerging from the ore, looks red. Of course, that part of the ore which is brought up by the motion of the furnace is also red, but it quickly darkens.

It will be found that the chlorination of the silver is finished before the red heat of the charge has entirely died out, and this is the proper time to discharge the furnace, in the first place to avoid loss of time, and secondly to avoid dusting. A chloridized ore when still red does not dust much in discharging, while when it gets completely dark, but is still hot, it dusts considerably more than if handled when quite cool.

If the ore is rich in sulphides, the salt can be added, if required, during the oxidizing period, but this ought to be done quickly in order not to cool the furnace too much.

I adopted the term "chloridizing self-roasting" for this mode of roasting because, after the ore is ignited and the fire is removed, it passes through the oxidizing and through the chloridizing period without requiring any further attention. One man can attend to quite a number of furnaces.

The ore thus roasted is roasted at the lowest possible temperature.

CHLORIDIZING HEAP-ROASTING

If silver ore which has been subjected to chloridizing roasting is left in a pile when discharged from the furnace, it will retain a dark-red heat for many hours, during which time the process of chlorination continues. I found that, if the chlorination of the silver is accomplished in the furnace up to 85 or 90 per cent., the increase in chlorination amounts to respectively 2 and 1 per cent., and that this increase takes place principally during the first two or three hours. By extending the time only an insignificant increase takes place. This, however, is different if the chlorination in the furnace be less advanced at the time of discharge. In such a case a large increase in chlorination takes place on the cooling floor. C. A. Stetefeldt made the interesting and valuable observation that even in a very poorly roasted ore the chlorination of the silver can be brought up to a high

percentage if the ore is left in a pile on the cooling floor. In roasting the ore of the Lexington mine he found it to be of such a nature that a silver chlorination of only about 47 per cent. could be obtained in the shaft of the Stetefeldt furnace. This partially roasted ore was piled on the cooling floor while his roasting experiments were going on. An examination of the roasted ore after twelve hours, however, showed an increase in chlorination of from 47 to 90 per cent. The ore was too heavily charged with sulphide to be suitable for a complete roasting in this furnace, and only a partial oxidation took place, but when piled in a heap the oxidation continued, forming sulphates which, acting on the salt, produced the chlorination. The temperature produced by the slow oxidation was sufficiently high for the chemical reaction. This observation may lead to the adoption in practice of a new method of chloridizing roasting, which we properly may call "chloridizing heap-roasting."

It is apparent that, if a chloridizing roasting could be performed just by exposing the ore to a short roasting in the furnace, and then leaving it to itself in a pile outside the furnace until cool, the advantages gained would be great, metallurgically as well as financially. This method, however, is only applicable to ores not too heavily charged with zinc blende and galena, as I once had the opportunity to convince myself. When experimenting with the heavy zinc-lead ore of the San Francisco del Oro mine, Chihuahua, Mexico, containing 24 to 25 per cent. zinc, 11.9 per cent. lead, 7 iron, and 21 sulphur, I also tried the Stetefeldt furnace. This was done more to obtain positive figures and a complete record of my investigation than in expectation of obtaining satisfactory results. The ore when removed from the shaft of the furnace emitted large volumes of sulphurous acid gas. No chlorine could be detected and the chlorination obtained did not exceed 15 to 16 per cent. The ore was piled on the cooling floor. There it continued to roast, emitting sulphurous acid fumes for several days, until it finally cooled without showing a perceptible increase in chlorination.

Heap-roasting was tried again by me, with the ore of Sombrerete, Zacatecas, Mexico, which contained 8.9 per cent. zinc, 9.5 per cent. lead, 16.8 per cent. iron, and 26.4 per cent. sulphur. Though this ore is considerably lighter in zinc and lead than the ore of the San Francisco del Oro mine, it was still too heavy to

be tried with the means of a Stetefeldt furnace. In order to promise success the oxidation of the metal sulphides had to be brought to a more advanced state than can be done in a Stetefeldt furnace, especially as the Sombrerete ore also required an oxidizing roasting to a certain state before adding the salt. The experiment was made with the aid of reverberatory furnaces. In three adjoining reverberatory furnaces three charges of one ton each were oxidized until the color of the ore commenced to change to brown, but still contained many black particles, and still smelled quite strongly of sulphur. Then 6 per cent. of salt was scattered over the surface of the ore. Immediately after adding the salt the three furnaces were discharged simultaneously, and the hot ore of the three charges was piled into one heap in the yard outside the building and left there to chloridize. After lying for fourteen hours, it was found, by inserting the sampling rod, that the ore inside the pile was still red hot, and that the fumes of the sample still smelled strongly of sulphurous acid. The color had, to a great extent, changed from brown to red. A test of the sample showed that only 12.6 per cent. of the silver was chloridized. After twenty-three hours it was found that the temperature inside the heap was considerably lower, but still high enough for the generation of chlorine. A distinct odor of chlorine was emitted from the sample, but none of sulphur. The chlorination of the silver had increased to 74.2 per cent. After thirty-eight hours the ore had cooled down to an extent that no more chemical reaction could take place. The heap was spread out and sampled. The color of the ore was as red as that of charges finished in the furnace. The chlorination of the silver was found to have increased to 85 per cent. There is no reason why the chlorination could not have been raised to 90 or 95 per cent. and higher, if the proper temperature could have been maintained longer, but the heap being so small, containing only three tons, it lost its heat before the chlorination was finished.

Only during the time of dumping the hot ore on a pile could the fumes be seen. As soon as the pile was completed visible fumes ceased to emanate. A strong odor of sulphurous acid could be observed for quite a number of hours, indicating that oxidation was still continuing, but no fumes could be seen. When the chloridizing period commenced the odor of sulphurous acid ceased, but no odor of chlorine could be noticed in its place,

nor did any visible fumes emanate from the pile. But from a sample taken from the inside of the pile light fumes could be observed, accompanied by an odor of chlorine, indicating that no volatile chlorides were emitted from the pile, and that the generated chlorine went into combination with the metals of the ore. The ore being undisturbed and in a thick layer, an excellent opportunity existed for this chemical reaction. In roasting in a reverberatory furnace it can plainly be observed that the ore on the hearths, even on the chloridizing hearth, will not emit much visible fume, but as soon as the ore is disturbed by the movements of the rake heavy fumes will be emitted. Now, these emanating volatile fumes are the very cause of the volatilization of the silver. It is therefore apparent that, if the creation of such volatile metal chlorides can be avoided, the loss of silver will be reduced to the minimum — that is, to the loss which will occur during oxidizing roasting, and which, in most cases, is very small.

On examination of the ore roasted in that experimental heap, it was found that much more metal subchlorides than chlorides were formed as compared with roasting in the furnace. As a much better utilization of the chlorine takes place if the ore is in a heap and left undisturbed than when spread over a hearth in a comparatively thin layer, it is to be expected that roasting in heaps will require less salt. This agrees with observations I have made by roasting the same ore in a Brückner and in a reverberatory furnace. In the Brückner furnace less salt was required while a higher chlorination was obtained, together with a smaller loss of silver by volatilization. In the Brückner furnace the ore is in a much thicker layer than in the reverberatory, which causes the better results.

The experiment at Sombrerete was made under very unfavorable conditions. The heap was too small, containing only three tons, and was exposed from all sides to the cooling action of the air, so that the chemical reaction ceased before the chlorination was completed. Notwithstanding this, the results obtained showed that 85 per cent. of the silver was chloridized, and if we take into consideration the increased furnace capacity, the reduction in the consumption of fuel and salt, this method surely offers sufficient advantages to justify further investigations and experiments. To maintain favorable conditions the hot ore

should be dumped into bins made of bricks or stone masonry, holding 30 to 40 tons. The number of these bins will depend on the roasting capacity of the works and on the time a heap will require to complete the roasting. The bins which have to be placed on the cooling floor should be open on the side toward the cooling floor, or provided with a good sized iron door, to permit free access, because chloridized ore as a rule does not run and has to be poked down. The top of the bins should be provided with hoods, to take off the sulphur gas which will emanate from the ore for some time.

Chloridizing heap-roasting may prove to be the most rational mode of chloridizing roasting. A higher percentage of silver will be chloridized with less loss of silver and at a smaller cost than if the roasting is finished in the furnace, no matter what type of roasting furnace may be used. Of course, the ore has to contain sufficient sulphur — not less than 8 to 10 per cent.

The reverberatory furnaces at Sombrerete roasted from 60 to 80 tons of ore per day, and the space available as a cooling floor was inconveniently small for the regular work, and a repetition of the experiment in a proper kiln was, therefore, not practicable.

CHLORIDIZING ROASTING WITH STEAM

If steam is admitted into the furnace during the chloridizing period, it forms hydrochloric acid, which decomposes the sulphides, expels arsenic and antimony, and chloridizes the silver with great energy, even metallic silver, on which chlorine acts but slowly. It acts also on metal silicates and chloridizes the silver contained therein, which otherwise would remain entirely indifferent to the action of the chlorine. The heavy zinc-lead ore of the San Francisco del Oro mine, Chihuahua, Mexico, when in course of experiments it was passed through the Stetefeldt furnace, showed a chlorination of only 15 to 16 per cent. It still contained 8 per cent. sulphur, and in order to bring the ore in better condition for the extraction of the silver it was re-roasted. The chlorination, however, could only be increased to about 44.2 per cent. On examination of the ore as it came from the Stetefeldt furnace it was found that by dropping through the shaft the main portion of the ore was transformed into minute globules, which showed that the ore was partially slagged, and

the silver contained in these globules resisted the action of the chlorine. After reroasting, these globules felt between the fingers just as sharp and glassy as before, but when the reroasting was done in presence of steam the result was different. The chlorination increased from 15 to 66.6 per cent. and the globules became soft and could be powdered between the fingers. Of course, a chlorination of 66.6 per cent. is very inferior, but this fact does not interest us just now. The ore had been spoiled in the Stetefeldt furnace, which made it impossible to produce a satisfactory chlorination. The present interest is the fact that these experiments demonstrated the beneficial effect of steam in roasting. Without steam the chlorination was only 44.2 per cent. while with steam it was 66.2 per cent. These figures represent the average of a considerable number of charges. The globules which remained unchanged when roasted without steam became soft and assumed the color of roasted ore.

The same conditions were maintained in both cases, with regard to the percentage of salt to the temperature applied, etc. The improved results can therefore be credited solely to the action of the steam.

If an ore is rich in lead it is hardly possible to avoid the formation of lead silicate during the oxidizing period, and the silver contained therein will not be chloridized during the subsequent chloridizing period, and consequently will enrich the residues and be lost. Roasting with steam is, therefore, much to be recommended for ores containing galena, especially if the galena is rich in silver, which is very often the case in complex ores.

Objections have been frequently made against roasting with steam, based on the assumption that it much increases the consumption of fuel, but in actual practice it will be found that the increased consumption is not serious at all. Waste steam from the engine can be used, but even if live steam is applied it is not necessary to use it in such volumes as to cause a marked drop in the temperature of the furnace. A moderate application answers the purpose. Sometimes the mere keeping of water in the ash-pit has a decidedly beneficial effect. The steam has to enter the furnace at the fire end and under the flame, so that it comes well in contact with the ore. The steam becomes superheated mostly at the expense of the ore next to the fire-bridge, thus preventing an overheating of that part of the charge.

The use of steam may also greatly reduce the loss of silver by volatilization. This is mostly noticeable with ore containing rich antimonial fahlerz and zinc blende besides antimonial galena. In working the ores of the Silver King mine of Arizona, I had the opportunity to make very interesting observations with regard to the effect of steam in reducing the loss of silver by volatilization.

As steam has not on all kinds of ore such a striking effect as in this case, it will be instructive to give a short description of the Silver King ore. This was a complex ore, and consisted of the following silver-bearing minerals:

(1) Native silver in close contact with fahlerz, silver copper glance, zinc blende, and in some instances with galena. It was brittle enough, so that a large part of it was pulverized in the battery. It occurred in the shape of wire, flakes, solid grains, and in large chunks, and in such quantities that it had to be removed from the mortars of the battery twice a week by means of shovels. This silver was of a bright white color, 0.975 fine, and did not contain any gold.

(2) Silver copper glance with 70.3 per cent. silver, 9.8 per cent. copper, 17.4 per cent. sulphur.

(3) Antimonious fahlerz, containing over 3000 oz. of silver per ton. This mineral was the most important constituent part of the ore.

(4) Zinc blende, of which there were three varieties:

(a) Zinc blende found in large and quite transparent crystals of a lustrous green color. This was the poorest of the silver-bearing minerals of the ore, but it was highly interesting from its beauty as a specimen. It contained only 10.2 oz. silver per ton.

(b) Brown zinc blende occurred in solid masses and in large quantities, frequently permeated with wire silver, and contained 97.7 oz. silver per ton.

(c) Black zinc blende was more scarce, and contained 40.8 oz. silver per ton.

(5) Galena occurred in two varieties: the fine-grained antimonious with 185 oz., and the coarsely crystallized with only 29 oz. silver per ton.

(6) Peacock copper ore, with 450.6 oz. silver per ton.

(7) Copper pyrites.

(8) Iron pyrites.

The gangue consisted of quartz, heavyspar and some por-

phyry. The average value of the ore as furnished to the mill was 161.4 oz. per ton.

This ore was roasted with 10 per cent. of salt, in a large size revolving furnace of the Brückner type, but with a modification specially designed by myself for this ore. On account of the antimonious fahlerz, the antimonious galena and the heavyspar, the ore caked very easily. For this reason, and to avoid excessive loss of silver by the antimony, the ore had to be roasted at a very moderate heat. The furnaces were 16 ft. long. It was found that the roasting could not be done properly with a furnace of common construction, with a fireplace only at one end of the cylinder, as the ore either did not receive enough heat at the farther end, or, if it did, it was overheated and caked at the end nearest to the fire. To overcome this difficulty the cylinder was provided at each end with a fireplace and flue arrangement (see Figs. 15, 16, and 17). These two fireplaces were worked alternately. After the ore was charged, the furnace was set in slow revolving motion, and fire kept up in one of the fireplaces. The flame traversed the furnace, and smoke and gases escaped through the flue, in front of the opposite fireplace. After a lapse of one hour, fire was made up in the other fireplace, the damper reversed, and flame and gases allowed to pass through the furnace in the opposite direction. The changing of the fire was kept up during the whole time the charge was in the furnace, only the intervals were not quite as frequent as in the beginning. This system of double fireplace and flues proved to be of great advantage in securing a very uniform roasting; the ore from both ends was chloridized up to the same percentage, while, when the ore was roasted in a furnace with a fireplace at one end only, the farther end showed a less chlorination of from 5 to 10 per cent. Besides, it enabled the operator to roast at a low and uniform heat.

Notwithstanding the capacity of the furnaces to roast at a moderate and uniform heat, the loss of silver by volatilization proved to be exorbitant, being not less than 38 per cent., while at the same time the chlorination was low on account of the large percentage of metallic silver in the ore, which was but imperfectly converted into silver chloride by the chlorine. A jet of steam was then tried, which was applied right under the flame and directed toward the side where the ore was. There was a steam

jet at each end, but only the one was operated which corresponded with the end at which was the fire. The roasting results thus obtained were very satisfactory. An average of many charges showed that the loss by volatilization was reduced from 38 per cent. to 2 per cent. while the average chlorination of 67 furnace charges proved to be 94.4 per cent. and in some cases as high as 96.8 per cent.

This roasting example illustrates that with certain ores the application of steam is of vital importance. The ores of the Silver King mines could not have been worked by a hydro-metallurgical method if the steam had not so greatly reduced the loss of silver, and increased the percentage of chlorination. There are some ores which do not need steam, but in most cases a larger or smaller jet of steam, according to the nature of the ore, does beneficially assist the chemical reactions.

CHLORIDIZING ROASTING OF SILVER ORES CONTAINING GOLD

There are two combinations of gold and chlorine: the aurous and the auric chloride. The latter is soluble in water and is formed when finely divided gold is brought in contact with chlorine gas at a common or moderately warm temperature. At a temperature of 230 deg. C. it changes into aurous chloride, which, however, on further heating, decomposes into metallic gold and chlorine. Owing to this property of the chlorine compounds of the gold, neither of them will be formed in the furnace during chloridizing roasting. The temperature in the furnace is too high for them to exist, and the gold on discharge of the furnace will be found in the metallic state. The aurous chloride, which is not soluble in water but is soluble in a solution of sodium hyposulphite, resists a much higher temperature than does the auric chloride, and it will form at a temperature much higher than the decomposing point of the auric, which temperature, however, has to be kept below red heat.

Based on this property of the gold chlorides I adopted a *modus operandi* by which I was able to extract 75 to 80 and even 90 per cent. of the gold contained in the silver ore simultaneously with the silver by sodium hyposulphite.

If the ore leaving the furnace is not allowed to cool quickly,

but, on the contrary, is made to cool slowly by dumping it to a large pile and leaving it undisturbed until it is cool, which takes several days, it will be observed that the generation of chlorine still continues for a considerable time. The cooling of the heap begins from the outside and progresses toward the inside, and the chlorine, which is generated at the inside, in escaping will meet a layer of ore sufficiently cooled for combination with the gold contained therein. It will form the aurous chloride, because the temperature is still too high for the auric chloride to exist. The formation of aurous chloride will progress toward the inside in proportion to the cooling of the pile. The cooling does not need to be continued beyond the time when a sample taken from the inside does not emit any chlorine.

If no precaution is taken to cool the ore slowly, only a small percentage of the gold will be converted into aurous chloride, and the gold extraction, therefore, will be very small. The beneficial effect of slow cooling on the chlorination of the gold contained in auriferous silver ore can also be observed in experimenting on a small scale, which will be illustrated by some results which I recently obtained in conducting some laboratory investigations respecting the ore of the Lucky Tiger mine, Sonora, Mexico. An analysis of the sample showed the ore to consist of:

| | | |
|---------------|--------|-----------------|
| Iron..... | 2.92 | per cent. |
| Zinc..... | 3.36 | " |
| Lead..... | 1.15 | " |
| Copper..... | trace. | |
| Antimony..... | trace. | |
| Sulphur..... | 2.54 | per cent. |
| Silica..... | 89.54 | " |
| Silver..... | 108.16 | ounces per ton. |
| Gold..... | 0.36 | " " |

Two lots of 100 grams each were roasted with salt on roasting dishes in the muffle at a dark-red heat. The amount of salt as well as the temperature and roasting time were for both lots exactly the same. When roasting was completed one lot was withdrawn and allowed to cool at a place away from the muffle, while the other was placed in a hot roasting-dish and covered with another hot roasting-dish, then removed from the muffle, but placed right in front of it. Thus the one lot was allowed to cool quickly, while the other was made to cool slowly. The quickly cooled ore showed a gold chlorination of 20.8 per cent. that is, 20.8 per cent. of it could be extracted with sodium

hyposulphite, while the slowly cooled ore showed a gold chlorination of 74.7 per cent. The result of this experiment clearly demonstrates that the chlorination of the gold takes place outside the furnace and is caused by slow and gradual cooling. The conditions in this experiment were not as favorable as they would have been on a large scale, because the generation of chlorine in so small a lot as 100 grams ceases soon, while in a large pile it continues for many hours.

While this mode of roasting gives very satisfactory results with silver ores containing 1 oz. of gold per ton or less, it will not be quite satisfactory for ores richer in gold, in which case a cyanide solution should be applied after the extraction of the silver. In the above experiment the residues of the quickly cooled ore were treated with a cyanide solution, by which the gold extraction was raised from 20.8 per cent to 86.07 per cent.

CHLORIDIZING ROASTING FOR AMALGAMATION

We have seen that the aim in chloridizing roasting for the extraction of the silver by lixiviation with sodium hyposulphite is to convert as much as possible of the silver into silver chloride and at the same time to expel as little as possible of the volatile base-metal chlorides, in order to reduce the loss of silver by volatilization to the minimum. This modification in chloridizing roasting was an important step forward in the hydrometallurgy of silver because thereby its weakest and most objectionable feature, the volatilization of the silver, was reduced to the minimum.

Chloridizing roasting was first devised, studied, and executed to meet the requirements of barrel and later of pan amalgamation. For this process not only as much as possible of the silver has to be converted into silver chloride, but it is of the greatest importance to expel or to decompose the base-metal chlorides, too, because if this is not done these chlorides will amalgamate with the mercury as well as the silver and produce an impracticably large amount of amalgam and a very low-grade bullion. Besides this, the mercury containing much of such amalgam has no tendency to unite when the pulp is diluted, but remains in minute globules, which partly float on the surface of the pulp as a dark scum and partly are carried off with the residue, thus causing a very large loss of mercury and, of course, of the silver it contains.

The mercury loses much of its decomposing energy on silver chloride; it becomes "foul," which results in very rich residues. In short, it is absolutely necessary in working complex silver ores by amalgamation to expel and decompose the base-metal chlorides. This is done by heat.

The salt is added either together with the ore or after the oxidation of the sulphides has pretty well advanced. During the oxidizing period the temperature is kept moderate, partly to prevent caking but mostly to form as much metal sulphates as possible, because these sulphates will act on the salt and change into chlorides and also produce chlorine, and, in presence of steam, hydrochloric acid. If the ore is rich in sulphurets the heat created by their oxidation can so increase as to make it necessary to let the fire go out entirely, but close attention has to be paid to start the fire again as soon as a pronounced decrease of the temperature can be observed. If an ore charge is allowed to cool too much it takes considerable time and fuel to bring the temperature up again to the desired degree. During the oxidizing period the ore ought to be frequently stirred — continually, if possible. The beginning of the chloridizing period is indicated by the swelling of the ore; it becomes "woolly" and emits heavy white fumes. The heat is then gradually increased until toward the end the charge assumes a bright cherry red. During roasting the charge is several times turned, that is, the half toward the fire-bridge is removed toward the flue, and that from the flue side is brought toward the fire-bridge. This is done in the following way: The whole charge is raked together into a long heap extending from the bridge to the flue end. This ridge of ore is made nearer to the working doors, to lay bare as much of the hearth as possible. Now, by means of a shovel the ore from the fire-bridge is moved and dumped back of the ridge, commencing close at the flue and proceeding toward the fire-bridge until the middle is reached, then the other part of the ridge is moved toward the bridge. Then the whole charge is spread again.

During roasting, samples are taken and tested with regard to the behavior of the ore toward mercury. If it is found that by adding some water and some mercury to the ore the bright surface of the latter becomes immediately covered with a black skin, it is an indication that the roasted ore still contains an

injurious amount of base-metal chlorides, and roasting has to be continued at an increased temperature in order to decompose them.

The practice of decomposing the base-metal chlorides by heat and increased roasting time is naturally connected with much loss of silver by volatilization. The late G. Küstel proposed a much more rational means than heat to destroy the base-metal chloride. In his treatise, "Roasting of Gold and Silver Ores," he says:

"It is very difficult to get rid of all the base chlorides; they are formed under the action of chlorine and hydrochloric acid. The most of the metal chlorides are volatile, and a part is carried off through the chimney. Another part of the chlorides gives off some of its chlorine, whereby sulphates, undecomposed sulphurets, antimonates, and arsenates are chloridized. Chlorides which are disposed to transfer chlorine to other metals in combination with sulphur or arsenic are: the proto chloride of iron and of copper, the chlorides of zinc, lead, and cobalt. When in this way the most of the metals are chloridized, the base metals, principally iron and copper, are losing their chlorine gradually, being first converted into sub-chlorides, and then into oxides. The roasting for this purpose must continue with increased heat, even when the chlorination of the silver is finished. At an increased heat, the base metal chlorides lose their chlorine, while the chloride of silver remains undecomposed, unless a very high temperature should be applied. This process requires a long time, consequently also more fuel. The decomposition of these chlorides is greatly assisted by the use of 5 to 6 per cent. of carbonate of lime in a pulverized condition. Lime does not attack the chloride of silver, but it is not advisable to take too much of it, as it would interfere to some degree with the amalgamation. The pulverized lime-rock must be charged toward the end of the roasting. First, two per cent. is introduced by means of a scoop, the whole well mixed, and then examined either with sulphide of sodium or in the following way:

"A small portion of the roasted ore is taken in a porcelain cup or glass, and mixed with some water by means of a piece of iron with a clean metallic surface. If the iron appears coated red with copper, some more lime must be added. In place of iron—especially if no copper, but some other base metal is present—some quicksilver is mixed with the sample. In the

presence of base-metal chlorides, the quicksilver is coated immediately with a black skin.

“When endeavoring to expel the base metals by heat, the loss of silver, in presence of much antimony, lead, and copper, should be investigated very carefully. Under certain circumstances it is not uncommon to find a loss of even 50 per cent. of the silver, if the chloridizing roasting is carried on at a high heat for a great length of time. The loss increases with the duration of roasting and with the degree of temperature. When such ore is under treatment, it is necessary to take samples during the roasting, and to examine the same for the amount of chloride of silver, and also for its loss, and to stop roasting when the highest percentage of chloride of silver is obtained without reference to the condition of base metals.”

Mr. Küstel proposed a still more rational method for removing from the roasted ore the base-metal chlorides which are so obnoxious to amalgamation. Instead of destroying them by heat and extended roasting time, he removed all soluble chlorides and sulphate of zinc by leaching the ore with hot water previous to amalgamation. The ores at Flint, Idaho, turned out such base amalgam that further working proved to be ruinous; but after Mr. Küstel applied his method the ore became most suitable for amalgamation, and very satisfactory results were obtained.

It is apparent that this method is quite an advance in amalgamation; not only is the amalgamating energy between silver and mercury much increased, thus resulting in a better extraction of the silver, but the volatilization of the silver in the furnace, and the loss of mercury and silver in the process of amalgamation, are much reduced.

An important point in this process should not be overlooked, namely, that chloride of silver, while not soluble in water, is soluble in a concentrated solution of metal chlorides. The dissolving energy of such a solution on silver chloride increases with its concentration and its temperature. Hot water has to be used in order to remove the lead chloride, and therefore the first part of the outflowing solution will be as a rule rather concentrated, and at the same time, being warm, will dissolve quite a noticeable amount of silver chloride. This, however, can be regained by collecting the solution in large tanks and by diluting

the same. If sufficient water is added, all the silver chloride will precipitate, and if enough time can be given will settle on the bottom together with lead chloride. If there is copper in the ore the solution should be made to flow through tanks filled with scrap iron, by which the copper and silver are saved.

Lead chloride amalgamates; lead sulphate does not; and in roasting for amalgamation it is therefore of importance to convert as much of the lead as possible into sulphate and as little of it as possible into chloride.

Lead sulphate, if once formed, remains indifferent and unchanged during the balance of the process. Mr. Küstel, in roasting the plumbiferous silver ores for amalgamation at Plomosas, Mexico, made the observation that if the ore was roasted in a reverberatory with a roof of only 20 or less inches above the roasting floor the bullion obtained contained 500 to 600 parts of lead in 1000, while if the same ore was roasted in a reverberatory with high roof, 27 or 30 inches, the bullion obtained was almost free of lead. In the first instance much lead chloride and oxichloride was formed, which amalgamated, while in the second instance nearly all the lead was transformed into lead sulphate. The reason of the different results is apparent. In the furnace with the higher roof there was sufficient space left between the flame, which travels next to the roof, and the surface of the ore to permit a free access of air, and in presence of ample live air more sulphuric acid is formed, which acts on lead oxide, oxichloride, and chloride, converting them into sulphate. If the supply of air is limited by the low arch, insufficient sulphuric acid is formed to convert all the lead into sulphate. In support of this theory speaks the fact that C. A. Stetefeldt, by roasting the ore from Ontario, Utah, in a Stetefeldt furnace, found all the lead contained in the ore to be in the state of sulphate.

With regard to the extraction of the silver by sodium hyposulphite it is immaterial whether the lead in the roasted ore is in the state of sulphate or chloride. If cold water is used for removing the base metal chlorides, which is generally the case, but a small portion of the lead chloride is removed, and the same is still contained in the ore when the latter is subjected to leaching with sodium hyposulphite, in which solution it is soluble, as well as the sulphate, which is not soluble in water.

VI

CONSUMPTION OF FUEL

THE fuel mostly used in roasting is wood, not so much so because it is the fuel with which it is the easiest to regulate the temperature, but because the mines and works are usually situated in more or less remote mining districts, where wood is easier and cheaper to be procured than other fuel. Bituminous coal and gas from gas producers answer the purpose perfectly well.

The quantity of fuel required depends mostly on the nature of the ore, but also on the construction of the furnace. Ores rich in sulphur, especially if a large part of it is combined with iron, as in iron pyrites, require the least amount, because by the combustion of the sulphur much heat is developed in the ore itself, so that the process needs only to be slightly assisted by fire. In fact, ores with 20 or 22 per cent. sulphur, if roasted in a Brückner furnace, need only to be heated until it is ignited. The balance of the heat required for the oxidation and chlorination is then furnished by the ore, the same as in chloridizing self-roasting, by which method 10 or more tons of ore can be roasted with one cord of wood. A highly sulphureted ore which does not contain too much zinc blende or galena but contains considerable pyrite could be roasted chloridizingly without the use of any fuel except what is needed to start the furnace. We have seen above, in heap-roasting, that only a partial oxidation of the ore in the furnace is required; and if for this purpose a continually discharging furnace of suitable construction is used, like those of the McDougal type as employed in sulphuric acid factories, or of the Howell type (if the latter is lined with bricks the whole length and only a very slight inclination given, and provided with a ring-flange at both ends so as to retain a somewhat larger amount of ore in the furnace in order to produce more heat and to preserve it better) a continuous chloridizing roasting could be successfully effected without the use of fuel.

These continually discharging furnaces will have to discharge

into closed pits, which in starting will have to be first well heated like the furnaces. From these pits or vaults the ore is then moved and charged into the roasting bins. The vaults should be sufficiently large to prevent cooling.

Ores, very poor in sulphur, containing only about 2 per cent., require much fuel, and if the same is expensive should not be roasted in a reverberatory furnace, because the conditions prevailing in a reverberatory furnace as regards utilization of heat are very unfavorable. Heat penetrates but very slowly into any pulverized material, especially if the same is left undisturbed. If a fresh charge is placed on the hearth of a reverberatory and spread over it, it will cool the bottom, and if it is not a muffle furnace, the supply of heat from that source will soon end, and the heating will be effected only on the surface by the radiating heat of the flame which travels next to the roof of the furnace; but heat, as stated, penetrates very slowly into pulverized material, and therefore only a small percentage of the heat produced by the fuel will be effective. To better utilize the heat it is necessary that continually new particles of the ore be brought to the surface to be exposed to the radiating heat. This, however, for obvious reasons, can be done but very imperfectly in a hand-stirred furnace. As soon as the charge becomes hotter than the bottom of the furnace the bottom will draw heat from the charge, thus exerting a cooling action on the latter. To make the furnace long helps, but not very much; and if the roasting of such dry ore has to be done in a reverberatory furnace it is more rational to make the hearth short and to build the furnace in two stories, in which case the roof of the lower hearth will heat the bottom of the upper one.

The most effective and at the same time the most fuel-saving roasting furnace for this class of ores is undoubtedly the Stetefeldt furnace, in which the flame is made to ascend through a high shaft, while the ore in a fine shower falls down the shaft and against the flame. There is no other roasting furnace in which the heat is utilized to such advantage as in the Stetefeldt; ten to twelve tons of such dry ore can be roasted with one cord of wood — a result which cannot be obtained in a reverberatory furnace. This peculiarity of the Stetefeldt furnace, which is so advantageous for dry ore, makes it also unsuitable for ores heavily charged with sulphurets.

If circumstances do not allow the erection of a Stetefeldt furnace, the next best fuel-economizing roasting furnace for such ores is the Howell-White.

If the ore permits a closer sorting without causing too much waste of silver, it will be well to do so in order to raise the sulphur contents of the ore, because this will not only reduce the consumption of fuel, but also improve the extraction and shorten the roasting time. In case the ore is not very rich, and concentrates well, it may be advisable to concentrate part of it and to add the concentrates to the balance of the ore, or to sort close and to concentrate the second-class ore, just as, according to circumstances, is found to be the most rational.

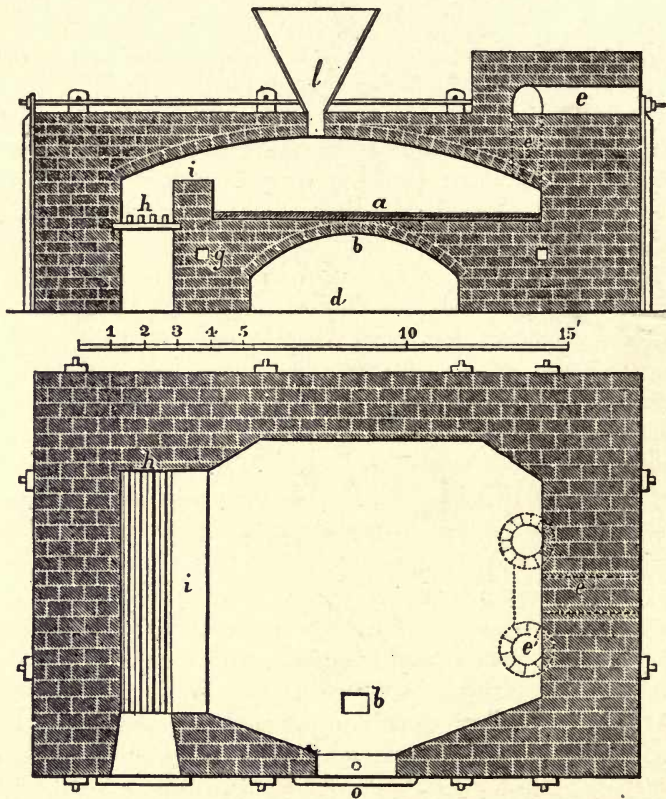
VII

REVERBERATORY FURNACES WORKED BY HAND

THE reverberatory furnace is a horizontal hearth furnace provided with a fireplace and grate at one end and a flue at the opposite end, and with working doors on one or on both of the two long sides. The hearth is separated from the fireplace by the fire-bridge. It is the oldest and the most primitive type of roasting furnace, but notwithstanding this it is the furnace which can be applied to any kind of ore, except those the nature of which prohibits chloridizing roasting altogether, like ores containing too much lead. Its construction gives the operator full control over the process. It offers facilities to maintain any condition the nature of an ore requires; and it is, when substantially built, very durable, requiring little repair, and makes but little flue-dust. The reverberatory is, in fact, the ideal furnace for chloridizing roasting, and would be exclusively used for this purpose if it were not for the fact that it has to be operated by hand, which makes the cost of roasting too high in localities where labor is costly. This was the cause which gave the impulse in the silver districts of the Pacific coast of the United States to the invention and construction of quite a variety of mechanical furnaces, all of which are labor-saving, and if applied to the proper ore do excellent service. They cannot be used, however, for so many kinds of ore as the reverberatory. Each of them has its peculiarities, with which the character of the ore has to comply, and it is therefore of the greatest importance, if mechanical furnaces are to be erected, that a thorough study of the nature of the ore should precede the selection of the furnace.

The Single-Hearth Reverberatory. — This is the oldest style of a roasting furnace. Figs. 1 and 2 represent the vertical and the horizontal section respectively: *a*, hearth; *s*, roof; *h*, fireplace; *i*, fire-bridge; *e* and *e'*, flue; *b*, bottom discharge opening; *d*, vault for placing the wheelbarrow to receive the roasted ore; *p*, charge

hopper in the roof of the furnace. (This hopper is provided with a slide which, when drawn, permits the charge to drop into the furnace. It has to be large enough to hold a full charge, and ought to hang on a proper framework, so that its weight does not rest on the roof of the furnace); *w*, buck stays and anchors.



FIGS. 1 and 2. — SINGLE-HEARTH REVERBERATORY FURNACE.

The single-hearth furnaces are not in use any more for roasting ore on a large scale. They are too wasteful with regard to fuel; the heat of the gases is not utilized. We find them, however, very useful for experimental work and for burning the precipitate in lixiviation works.

The Two-Story Single-Hearth Furnace. — This is a considerable improvement on the single-hearth furnace. It is shown in

Fig. 3, which represents in a longitudinal section the general arrangement: *a, a* are the lower and upper hearth; *r*, lower fireplace; *b*, flue connecting the lower with the upper hearth; *b'*, flue in the arch of the upper hearth, whence the gases are led to the dust-chambers; *f*, working doors of the upper hearth (the working doors of the lower hearth are on the opposite side); *r'*, an auxiliary fireplace for the upper hearth, which is smaller and is used only when a fresh charge enters the furnace, to assist in heating and to ignite the same more quickly.

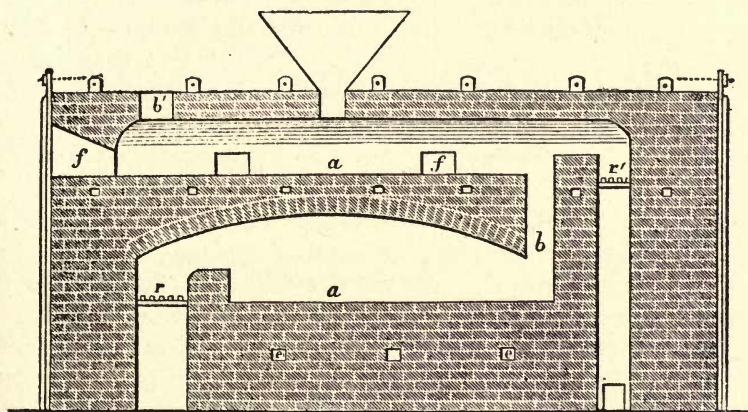


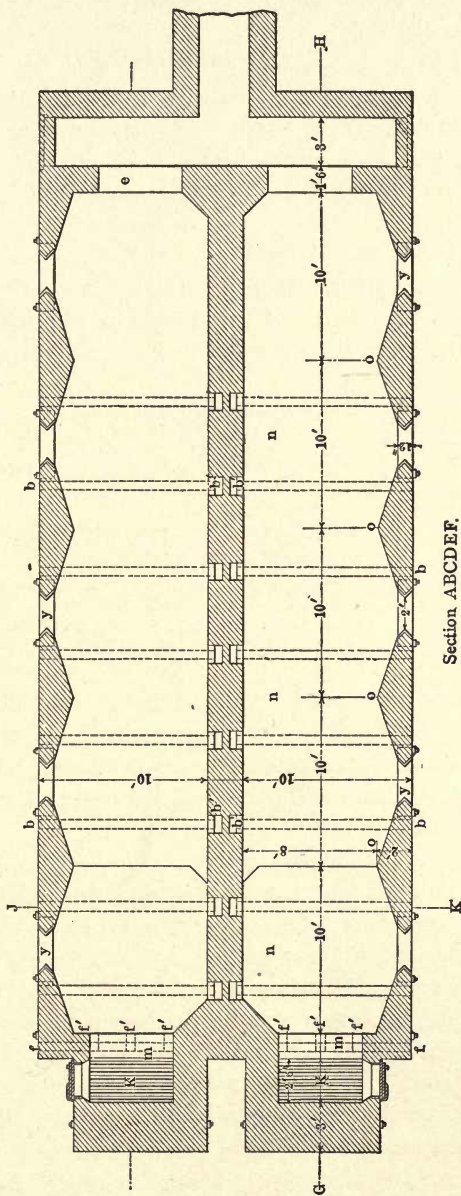
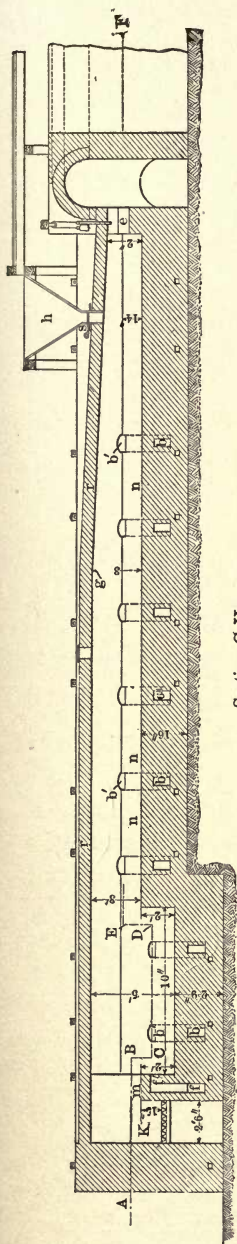
FIG. 3. — TWO-STORY, SINGLE-HEARTH REVERBERATORY FURNACE.

As soon as the charge of the lower hearth is finished and removed the upper charge is dumped down through the flue *b*. To facilitate this operation, and to permit the use of a hoe instead of the slower-working shovel, there is at the end side of the furnace the door *f*, through which is inserted the hoe with which the charge is pushed to drop through *b*.

The Long Reverberatory Furnace. — A further step in the development of the reverberatory furnace was the construction of long furnaces with three to five hearths on the same level, or in flat steps of three to five inches rise. The length depends on the nature of the ore. For highly sulphureted ore, especially if it contains much iron pyrites, the length may be extended to 50 ft. without the aid of an auxiliary fireplace. As a rule each hearth is made 10 ft. long and 10 ft. wide. The arch, which can be made rather high at the fire end, ought to slope down toward the flue end to

throw the heat of the moving gases more toward the bottom at the part of the furnace remote from the fire. If the arch is made straight, each succeeding hearth should be a few inches above the preceding, by which the same object is attained. These steps serve at the same time as a mark for each hearth, and assist in preventing the charges from getting mixed.

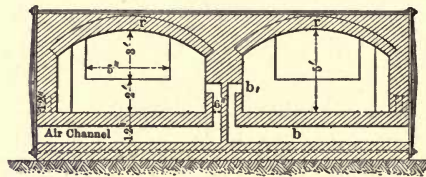
These long furnaces are either built singly or in pairs back to back, as shown by Figs. 4, 5 and 6. Single furnaces are to be preferred, as they offer the opportunity of providing working doors on both sides, which not only facilitates the working of the charge but also permits a free access of air from both sides, which is of great advantage. The construction of these single furnaces, however, is much more costly, and at the same time requires a good deal more space and consequently much larger buildings. A reverberatory furnace with working doors on one side only has two dead places, that is, places which are not reached by live air, for which reason the process of roasting on these places is not only retarded, but also such places become, as a rule, overheated and often cause there caking of the ore. These places are: the part of the hearth next to the fire-bridge extending almost to the first working door, and the part along the back wall of the furnace extending from the fire-bridge to the flue, so that, if no provision is made for air to enter at these points, which we very often find to be the case, we have to consider the furnace to be of faulty construction. The oxygen of the air is the very life of the process, and if the same is withheld, or its access obstructed, the bad effect will invariably manifest itself by an inferior roasting result and a higher loss of silver by volatilization. These dead spaces are avoided by constructing air channels leading from the front under the hearth and entering the furnace through openings in the back wall as shown by *b* and *b'*, Figs. 4 and 6. The fire-bridge is also provided with an air channel, *f*, which is in communication with three openings, *f'*, through which the air enters the furnace. The effect of this additional air supply is very noticeable. The temperature through the whole width of each hearth is quite uniform and no overheating next to the fire-bridge takes place. At the same time the fire-bridge is much protected by the cooling effect of the air. If steam is to be used in roasting, the channel *f* serves for this purpose. The steam pipe, which is provided with three holes while the end is closed



FIGS. 4 and 5. — LONG REVERBERATORY FURNACE.

by a cap, is inserted through *f*. The holes in the pipe are so divided that each one is located right under each upraise of the channel.

The long furnace, as represented by Figs. 4, 5 and 6, was designed by me for chloridizing the calcareous and arsenical ores of the Anglo-Mexican Mining Company at Yedras, Sinaloa, Mexico. This ore was of a very peculiar character, and apt to sustain an unusually heavy loss of silver if certain conditions in the treatment were not scrupulously maintained. The dimensions of different parts of this furnace were designed to conform with the peculiarities of this ore, but the general arrangement was not altered; and the diagrams will well serve to illustrate the construction of the long reverberatory furnace.



Section J.K.

FIG. 6. — LONG REVERBERATORY FURNACE.

It is always advisable to make the fireplace (see K, Fig. 5) sufficiently wide, because it gives the means to regulate the temperature, and if too narrow may make it impossible to supply sufficient heat. The heat of the first and second hearths is mostly supplied by the combustion of the sulphides on the preceding hearths. Wood requires a wider fireplace than coal; 2 ft. 6 in. will be sufficient for wood fire. The sides and roof of the fire-box should be lined with fire-bricks in order to resist better the heat and the wear caused by wood and tools. The brickwork encasing the fire-box should be substantial and well braced. The depth should not be made much longer than the length of the wood, which is usually cut 4 ft., so that a depth of 5 ft. is sufficient. It is well to keep water in the ash pit, not only to lengthen the life of the grate bars but also to make a limited amount of steam, which benefits the roasting very much.

The top of the fire-bridge (see *m*, Fig. 5) should not be too high above the grate, in order that the furnace may receive

as much as possible of the radiating heat of the fire; the width differs according to circumstances, and is made from 12 to 18 in. A 12-in. bridge should be made entirely of fire-bricks, but if wider can be lined on both sides with them. The space above the fire-bridge is to be made large enough so that the hot gases can enter the furnace freely without receiving any back pressure, which manifests itself by flames and smoke coming out between the frame and door of the fire-door after each new addition of wood. In case the space is too small the flame recoils, becomes short and overheats the fire-box without furnishing sufficient heat to the parts of the furnace farther off from the fire. If this space is large enough the flame rolls slowly, touching the roof, and after a fresh addition of wood extends 25 to 30 ft. into the furnace. The fire-bridge ought never to be built without air channels, as described above.

The length of the hearth (see *n*, Fig. 5) depends, as above mentioned, entirely on the nature of the ore. If an ore is rich in sulphur the hearth can be made 50 ft. long, but this is the limit. It is the heat created by the combustion of the sulphides which makes the working of such long furnaces possible. An ore poor in sulphur never could be heated sufficiently to commence roasting if 40 or 50 ft. away from the fire, and, therefore, a large part of the furnace would be inactive, causing only unnecessary extra labor to move the ore. To insert additional fireplaces does not offer any advantages; on the contrary, it hinders the execution of a delicate chloridizing roasting. A very uneven heating of the charge is caused by them. Near the inserted fireplace the charge gets hot, and often hotter than it ought at that stage of roasting, and when moved to the next hearth gets cooler again, which is not advantageous to chloridizing roasting. The insertion of additional fireplaces is only justified in mechanical continually discharging hearth furnaces like the O'Harra and the Ropp furnaces, which are made 100 ft. long and even longer, and are very effective in labor saving and under proper conditions do very good work; but their applicability is confined, like that of other continually discharging mechanical furnaces, to certain kinds of ore, and they do not permit a really delicate roasting. On the first, or charge hearth, the ore ought to become well heated, so that, shortly after transferring it to the second hearth, blue flames can be observed when the ore is stirred. Before it is

removed from here the oxidation of the sulphides ought to be well started.

Of course, it takes experience and skill to judge the proper length to be given to the furnace for a certain ore. Ores containing 20 to 22 per cent. sulphur, with considerable iron pyrites, will roast well in a furnace 40 to 50 ft. long, provided the ore does not contain an excess of zinc blende. For ores with about 8 per cent. sulphur a furnace 30 ft. long is sufficient, and ores containing only 2 to 3 per cent. sulphur should not be roasted in a long reverberatory, but either in a Stetefeldt or a Howell furnace.

Each hearth of the long reverberatory furnace is made 10 ft. long. If a step is given to each hearth, it serves as a mark, but if the whole hearth is level, the points, *o*, of the pillars of the front wall mark the lines. It is not advisable to make the hearth too wide, thus trying to increase the capacity of a furnace. It should be borne in mind that the charges have to be worked and moved by hand labor, and that extra long tools are very hard to handle; they tire out the man, in consequence of which the part of the charge next to the back wall will be worked less than the part from the middle toward the front. For the same reason the handle of the hoe and the shovel should be made of gas pipe, in order to make them as light as possible. To the end of the pipe is forged a solid rod 24 to 30 in. long, to which the hoe or shovel is attached. Nothing smaller than a $1\frac{1}{2}$ -in. pipe should be taken, in order to make the handle sufficiently stiff and at the same time easy for the workman. The hand has an easier hold on a $1\frac{1}{2}$ -in. handle than on an inch handle.

To make the width of the hearth, measured from the front of the working door to the back wall, 10 to 11 ft. will be found convenient, but it should never exceed that. The height of the hearth above the working floor should be 2 ft. 9 in. to 3 ft., so that the workman can throw his weight on the handle of the tool when required, which assists him much.

Of great importance is it to prepare the bottom well so that it remains level and does not sink in different places. It would be too expensive to build the whole part below the hearth solid with bricks, for which reason only the sides and ends are built solid, while the inside is filled with stones, gravel and sand. The filling has to be done carefully, so that no hollow spaces are left. The filling is commenced with coarse rock of 5 to 6 in.,

then, after about 8 in. in depth are filled, finer material is used and washed down between the spaces with water. Some stamping is to be recommended. Then another layer of 3 to 4 in. of rock is added, the spaces filled with sand with the help of water, and stamped again. Then gravel, and as final layer sand, is used. The top of the sand layer should come within an inch of the width of a brick if the upper edge of the brick is to be flush with the bottom of the working-door frame. This done, an inch layer of clay is spread with the trowel. A straight-edge and level ought to be used. Then time has to be given for the clay to dry, while work is being done on other parts of the furnace. If the furnace is very long several cross walls should be made, well connected with the side walls.

When the clay has sufficiently dried, the brick pavement is laid. The hardest bricks are selected for it. They are set on the narrow edge and with their long side parallel with the fire-bridge, which makes the hoe slide easily over them. They can either be set dry and the spaces filled with sifted sand, or they can be set in clay, but always as close together as possible. The hearth next to the fire should always be set dry in sand to permit the bottom to expand without bulging. For the binding rods which pass under the hearth, channels should be made of bricks, so that these rods will be kept cool by air and can be easily inserted or withdrawn.

As regards the roof *r*, the arch over the hearth should be made pretty flat in order to spread the flame, but care should be taken not to go to the extreme. A rise of the arch of 14 to 17 in. over a 10 ft. wide hearth is about proper; to give such an arch only a rise of 5 to 6 in. is a great mistake if any durability of the furnace is expected. It has to be considered that in the heat the hearth will expand, and if the joints between the bricks are not very close and carefully made, the arch will soon cease to be an arch, but become a flat plane which in course of time even becomes an inverted arch, and soon will cave. The arch is usually made 9 in. thick. If square bricks are used, which is generally the case in remoter mining districts, it is much better to throw two $4\frac{1}{2}$ -in. arches one on top of the other than to make only one arch by setting the bricks on their narrow edge, because then the joints are so much wider on the outside than on the inside that they have to be filled with clay and brick chips. This filling offers

very little resistance and makes the arch weak. The foot of the arch should rest on the long sides of the furnace and extend also over the fireplace. It is not necessary to cover the latter with a separate cross arch.

With regard to the height of the roof above the hearth bottom it can be considered, as a rule, that the height should be the greatest on the two hearths next to the fire, and then gradually diminish to the flue end. In ascertaining this distance one has to be guided by the character of the ore. For certain ores the roof of the hearth next to the fire has to be made much higher than on the following hearth, as is shown in Fig. 4, where the roof of this part of the furnace is made 5 ft. above the hearth bottom, or 2 ft. higher than that of the adjoining hearth. This is accomplished by a step of 2 ft., which, however, is an exceptional case. Usually a long furnace will do good work if the highest part of the arch of the finishing hearth is made 30 to 36 in. above the bottom and then slopes down to 24 in. at the flue end. This will allow live air to enter between the ore and the fire gases. The spring of the arch, as stated, should not be made less than 14 to 17 in. These dimensions are given for highly sulphureted ore, and have to be varied according to the character of the ore.

In the endeavor to reduce the consumption of fuel to a minimum we often find that other more important points have been sacrificed to this one object. One of them is to make the arch and sides too low. The flame, being pressed down by the low arch, comes in contact with the ore, and exercises a reducing action, which is adverse to the principle of chloridizing roasting. The space between the ore and the roof is almost completely filled with gases from the combustion of the fuel, and the live air has no opportunity to enter deeper into the furnace. The little that enters through the working doors is forced to the side, doing some good work on its way to the flue, but the main portion of the ore depends for its oxidation on the small volume of undecomposed air mingled with the fire gases. Thus a furnace may have plenty of draft but not enough air. This defect in the construction is felt still more if in a long furnace an ore is to be roasted which cannot stand a high heat without caking or suffering a great loss of silver by volatilization, because it is not possible to keep the heat on the finishing hearth as low as required

and at the same time insure the working on the more remote hearths to the best advantage.

Instead of using a wooden center to build the arch, quite frequently damp sand is used for this purpose. The sides are finished first, including the skew back, then the center line is marked on the bottom of the furnace and wooden sticks set up on that line at distances of 3 to 4 ft. and kept in position by some moist sand. The length of these sticks must correspond with the height of the roof at their respective places. All the working doors and other openings are closed with boards and then the furnace is filled with moist sand, or tailings. This done the shape is given to the arch with a straight-edge and trowel. The finishing is done with a thin layer of lime mortar, on top of which the arch is built. Clay should be used as mortar for the arch. After several days the sand is removed, but not until buck stays and binding rods are placed in position; otherwise, when by removing the sand the weight and side pressure of the flat arch are thrown on the sides, they may give way and cause the arch to cave.

The charge hopper *h*, is made of stout sheet iron and should be large enough to hold a full furnace charge, which consists of 1500 lb. to one ton of ore. Not to burden the arch with this weight, the hopper is flanged with angle iron around the rim and hangs on a framework which rests on two sides of the furnace, as shown in Fig. 4. These hoppers are usually made square, and the narrow end, which is provided with the slide, *S*, corresponds with a cast-iron extension which passes through the arch. This piece of casting should be shaped to correspond with the circle of the arch, so it will act as a keystone. The hopper is filled by means of dumping cars running on an elevated track.

The flue-hole should always be made in the end wall of the furnace, and not in the roof, because the flue (see *e*, Fig. 4) on top of the furnace is much in the way. It is of the greatest importance to have ample draft in the furnace. The draft in the furnace depends on the suction power of the stack and on the size of the flue-hole. If the latter is made too small the gases will be throttled and the furnace will smoke. No good roasting can be performed in such a furnace; and as the draft is such an important factor in roasting, the flue should be made full large and be provided with a damper to regulate the draft. Dampers are

indispensable, especially if a number of furnaces are worked by the same chimney. The furnaces next to the flue leading to the chimney will receive an excess, while those situated farther away may not receive sufficient draft.

I found that through insufficient supply of air not only the chlorination suffers, but the loss of silver also increases. If the roof of a furnace is of proper height to permit air to enter between the ore and the gases, and the fire-bridge and the back of the furnace are provided with air-ducts, the interior of the furnace can be observed, which makes it easier to regulate the draft. The draft must be so regulated that the fumes evolved are kept in motion in all parts of the furnace. If they stagnate around the ore, or if the furnace assumes a nearly uniform heat throughout the entire length, it is always a sign of insufficient draft, and if the draft is not increased the result will invariably be a high loss of silver and a low chlorination. On the other hand, if the flame coming from the fireplace becomes short and pointed and travels very swiftly, it is a sign of too much draft, which causes an unnecessary consumption of fuel and may cool the furnace too much for it to do good work.

If the flue opening in the end wall of the furnace is made 4 to 5 ft. wide, the sides 9 in., and the spring of the arch 12 in., it will answer for all kinds of ores.

For the working door *f*, (Fig. 5) the best design for the reverberatory furnaces is represented by Figs. 7 and 8. It is 18 in. wide and 10 in. high. Its design is ingenious and simple. Each side is in the form of an angle, which enables the laborer to reach through one door with his hoe all points of a 10-ft. hearth, while if the two sides were straight it would require two doors to each hearth to reach all the points. The top and bottom plate of the frames extending beyond the sides, and the angular shape of the latter, make it possible to cement solidly the door frame into the wall, without the use of any anchor-bolts, which are not of much use anyway, as they invariably work loose in a short time and then loosen the bricks around the door. A movable iron plate serves as door.

Each door is provided in front with a 2½-in. roller, the object of which is to facilitate the working of the charge. The long handle of the tool rests on it and in stirring it revolves, and thus, while taking the weight of the tool, lessens the friction. The

full advantage, however, of this arrangement is only obtained when the movement of the hoe is at right angles with the roller. As soon as the angle is changed the hoe will partly slide, or even will slide altogether, while the roller stops. A far better contrivance is that designed by G. Küstel, and shown in Figs. 9 A, 9 B and 9 C. Instead of the long roller he provided the door frame

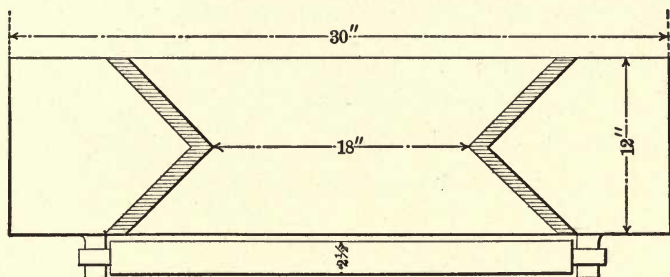


FIG. 7. — PLAN OF WORKING DOOR.

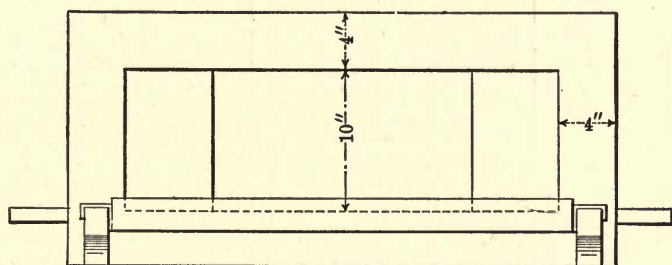


FIG. 8. — ELEVATION OF WORKING DOOR.

with a grooved wheel, which rests in a fork movable on a pivot. The groove is large enough to take the handle of the hoe. In working, the hoe moves easily to and fro on the wheel, which, on account of the pivot in the frame, will turn to any direction in obedience to the hoe and revolve with the same ease.

When the furnace is built, it has to be dried carefully with a slow fire for several days. It is well to keep a small fire on each hearth, so that the whole furnace will get about the same heat. If fire is kept on the fireplace only it will take a very long time to dry such a long furnace.

In starting the furnace it is best to charge each hearth through the working door with crude ore. Each charge will have to remain on the same hearth until the charge on the finishing hearth

is chloridized, which will be quite a number of hours if the ore is highly sulphureted, but after the first charge is drawn and the others are moved forward one hearth each, the time required on the finishing hearth will be less, and soon the whole furnace will be in good working condition. The discharging is done through the working door of the finishing hearth, into wheelbarrows. Shortly before discharging commences the charge of each succeeding hearth is raked into a pile on the forward half. As soon as the finishing hearth is discharged the forward movement of the charges begins, and when done a new charge is dropped on the first hearth. On those hearths on which the oxidizing takes

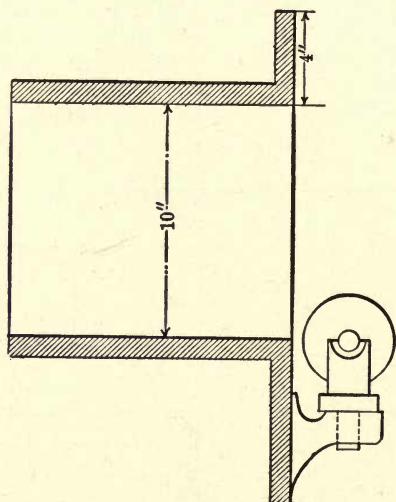
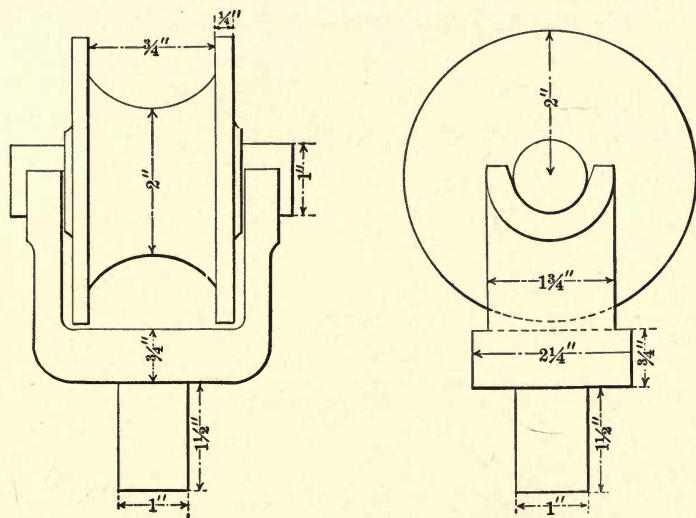


FIG. 9A. — THE KÜSTEL WORKING DOOR.

place the charges should be worked diligently to expose the ore as much as possible to the oxidizing action of the air. This not only shortens the roasting time but more sulphuric acid is formed, producing a better sulphating of the ore. When the ore becomes woolly, that is, when the chloridizing period sets in, much less stirring is required. On the finishing hearth the ore should be raked to a thick layer in the center of the hearth, the space next to the fire-bridge and the back wall being kept clear. From near the working door the ore should also be pushed farther into the furnace, to prevent the cooling of that portion of the charge.

A thick layer diminishes the loss of silver by volatilization. On this hearth the ore should be stirred only a few times. If the salt is to be added in the furnace, it is soon found by observation and assays on which hearth this has to be done to obtain the best results. Always consider both chlorination and volatilization.

With regard to the fire the operator has to be entirely guided by the temperature required on the finishing hearth. If the ore is such that it loses much silver if exposed to too high a heat, which can be ascertained before the furnace is built, then, in order not to reduce the working capacity of the furnace by keep-



FIGS. 9 B and 9 C. — DEVICE FOR WORKING DOOR.

ing so low a heat as the ore requires, the roof of the finishing hearth has to be made much higher above the hearth. This is best done by dropping that hearth a step lower than the other hearths. The level of the fireplace, however, should not be dropped too, but should be put in proper position for the other hearths, which, of course, will make a higher fire-bridge for the finishing hearth.

Three men are sufficient to attend a furnace 50 ft. long, with one helper for each two furnaces. When a charge is drawn, two of the men and the helper wheel the roasted ore to the cooling floor, while the third man pulls the charge. The fire is cared for by the man who attends to the two hearths next to the fire

on which the ore does not need to be worked so frequently as on the other hearths. The removal of the ashes is done by the helper. The wood is supplied by one yard man to all the furnaces; likewise the hopper is filled with ore by one man for all the furnaces. The roasting capacity of a 50-ft. furnace is about 8 to 9 tons in twenty-four hours.

THE TWO-STORY LONG FURNACE

As a further improvement in the hand-worked reverberatory furnace we have to consider the two-story furnace. Instead of building the furnace in one direction, say 40 ft. long, it is built

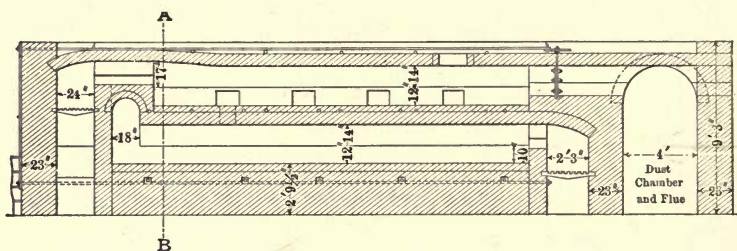


FIG. 10. — LONG REVERBERATORY FURNACE, TWO-STORY.

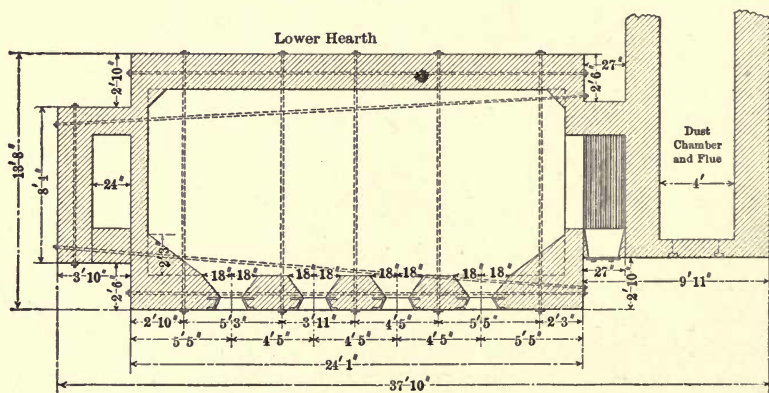


FIG. 11. — LONG REVERBERATORY FURNACE, TWO-STORY.

in two stories of 20 ft. hearth each. There is by this method not so much loss of heat by radiation. The hot arch of the lower story warms the bottom of the upper hearths, and a new charge is more quickly ignited than if the hearths are all on one level. Figs. 10, 11 and 12 illustrate the construction of such a furnace.

The lower two hearths are 11 ft. long each, while the upper two are 10 ft. each. A number of these furnaces were erected by me for the Mexican Santa Barbara Mining Company, to roast the heavy zinc-lead ores of the San Francisco del Oro mine. In this furnace 10 tons of ore were roasted in twenty-four hours. The oxidizing was done on the upper two hearths. Through an opening in the bottom of the second upper hearth the charge was

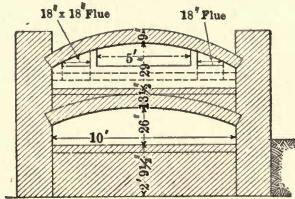


FIG. 12. — SECTION THROUGH A B
(FIG. 10).

dumped into the first hearth of the lower furnace. The salt was added in the upper hearth just before dropping the charge into the lower hearth. In this way the salt was well mixed with the ore. The construction of these two-story furnaces permits the insertion of an auxiliary fireplace across the width of the hearth, which can be used to advantage, because the heat is spread over the whole width of the hearth, which is not the case in a long furnace, where, if an auxiliary is used, the fire has to enter the side of the furnace. In roasting the San Francisco del Oro ore the auxiliary fire was used only for a short time after a new ore charge entered the furnace, for the purpose of reducing the cooling effect of the cold charge on the other charge, which was in the state of roasting, and to ignite the fresh charge more quickly. The fire was stopped as soon as the fresh charge showed the sulphur flame. The gases from the lower furnace entered the upper furnace through two flues, one on each side of the firebridge proper, so the rising gases did not interfere with the flame of the auxiliary fire.

At Sombrerete, Mexico, when it became necessary to rebuild two of the 40-ft. furnaces, I changed them into 20-ft., two-story furnaces, and increased thereby the working capacity of each by nearly two tons per day.

VIII

MECHANICAL ROASTING FURNACES

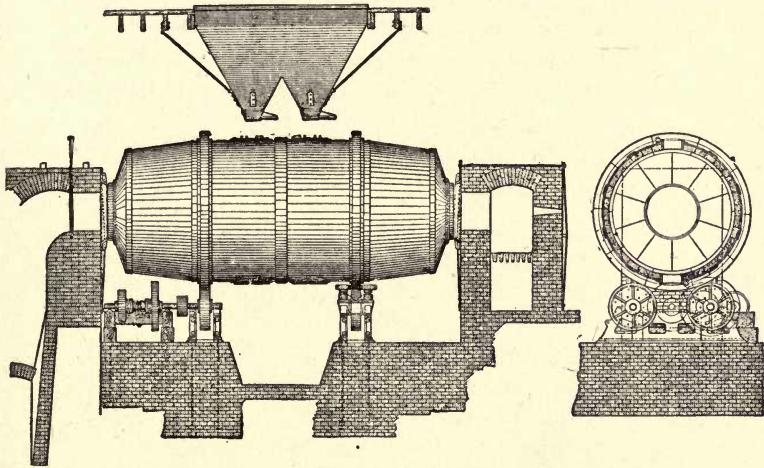
In this chapter will be included only such mechanical roasting furnaces as are specially adapted for chloridizing roasting. There are two classes of such furnaces: one in which the ore is roasted in charges; and the second in which the roasting is continuous, that is, furnaces in which at one end a continuous stream of raw ore enters, while at the other a continuous stream of roasted ore leaves the furnace.

As stated above, in no mechanical furnace can the process of roasting in all its stages be so well controlled as in a reverberatory furnace worked by hand, and, therefore, their applicability is much more limited to certain classes of ore. However, if applied to suitable ores, they do very good work, and, where labor is expensive, are more economical. In Mexico, where labor is cheap, the mechanical furnaces proved successful only in exceptional cases. All mechanical furnaces are connected with more or less machinery and require frequent replacement of the wearing parts. These parts of machinery and castings are sent from the United States, and are rather costly by the time they land in some remote mining place in the mountains. This, however, would not be so important a factor if it were not for other inconveniences connected with it, as the long time it takes to get these parts, and, perhaps the sudden breakdown of parts of which duplicates may not be on hand, requiring quite a long shutdown of one of the furnaces, which always represents a large percentage of the roasting capacity and with it of the producing capacity of the works.

The erection of mechanical furnaces for ores for which they were not suitable has caused many serious failures.

(1) MECHANICAL FURNACES FED BY CHARGES

(a) *The Brückner Revolving Furnace.* — This ingenious device of a chloridizing furnace was successfully introduced by its inventor, Mr. Brückner, in Colorado, in 1867. The furnace consists, as illustrated by Figs. 13 and 14, of a cylinder of boiler iron, the ends of which are closed save a circular central opening on each end. Two circular tracks are fastened around the cylinder placed at even distances from the ends. With these tracks the cylinder rests on four strong wheels. A revolving motion is imparted to the cylinder either by friction, in which case one of



FIGS. 13 and 14. — BRÜCKNER ROASTER.

the four wheels is made to revolve, or the motion is imparted by pinion and cogs, in which case a cast-iron ring with cogs is fastened to the cylinder. At one end of the cylinder is placed a fire-box, the throat of which corresponds with the central end opening of the cylinder, while the opposite opening corresponds with a circular hole in the flue. Four doors are placed diametrically opposite, two on each side. These doors serve for charging and discharging the furnace. Above the furnace is placed a hopper large enough to hold a charge of ore. The hopper has two outlet spouts, each provided with a slide, which correspond with the furnace doors. The shell as well as the ends are lined

with bricks. Provision is made in the driving mechanism to regulate the speed from one revolution in one minute to one revolution in three minutes. Some of the furnaces are so constructed that the two ends of the cylinder are slightly contracted in order to facilitate the discharging of the ore, but this complicates the lining and reduces the capacity, and is actually not necessary, as a straight cylinder discharges very nicely, and if a little of the charge does remain in the furnace, it helps to heat the new charge.

The manipulations of this furnace are as follows: The two doors of one side are opened and the furnace revolved until the doors come right under the two spouts of the hopper, when the two slides are withdrawn and the charge allowed to drop into the furnace. Then a quarter turn is given to bring the doors in proper position for the man to close them. They are fastened as tight as possible by means of a wedge. Though the lid and the flange of the cast-iron door frame are faced it is not possible, especially if the furnace has been in use for some time, to close the door perfectly tight, and when the furnace revolves, some ore will leak out through each door. This takes place only as long as the charge is crude and stops when actual roasting is in progress. This leakage is the more annoying as the ore is crude and mixes with roasted ore underneath the furnace. To prevent this leakage the joint of lid and flange should be plastered from the outside with clay, or better with paste of sifted wood ashes and salt. Before charging the furnace should be well heated, and during charging the draft checked so that not too much dust is carried into the flue.

In the beginning a strong fire is kept, but as soon as the sulphides are well ignited the fire is allowed to go out. The heat developed by the oxidation is considerable, and if further increased by the fire a caking and balling of the ore would take place. These furnaces are usually 6 ft. in diameter inside the lining and 16 ft. long, and take a charge of 5 to 5½ tons of ore. Usually two charges can be roasted in twenty-four hours. About three to four hours of each charge the furnace can run without fire, then the salt is added and the roasting continued with a moderate fire. A hole back of the flue permits the observation of the temperature and the taking of samples with a long-handled scoop. During chloridizing the charge in the furnace assumes an inclined

position up to 45 deg., the weight of which acts against the direction of the motion, and if the clutch is thrown out in order to stop the furnace, this weight will pull the furnace back nearly a quarter of a turn. To open the doors to add the salt or to discharge the furnace, it is necessary that the furnace should be stopped at a position convenient to the roaster man. In order to accomplish this the end of an iron bar is pressed between the track and the wheel at the side at which the movement of both is outward. When the doors are in the right position the clutch is thrown out, but the furnace is prevented from revolving back by the iron bar, which becomes clamped in very tightly. By means of a short-handle shovel the salt is introduced through the two doors and well scattered over the whole surface of the ore. The salt decrepitates violently and in this way becomes very evenly divided. After the salt is added and the furnace revolves again, the ore becomes woolly, as in the reverberatory, and assumes a still more upright position.

If the ore is sufficiently sulphureted and the salt is added it will not be necessary to start the fire again. There is enough heat stored in the charge to finish the process of chlorination (see remarks on chloridizing self-roasting), in which case the consumption of fuel is very small. This mode of roasting is only admissible if the ore is to be roasted for lixiviation. For amalgamation a second fire is indispensable, because the base-metal chlorides have to be decomposed or volatilized.

When the charge is finished two cars are pushed under the furnace, one for each door. They are large enough to receive the full charge of the furnace. All four doors are opened, the lids kept in position by a proper contrivance, and the furnace is revolved again. While the furnace is prepared for discharging, a good strong fire should be started again, to heat it for the next charge. The receiving cars are made narrow and long, so that no ore is dropped beyond the rim of the car, because, especially in the beginning, the ore will pass through the door over a large arc.

The cooling floor is situated 5 or 6 ft. lower than the track under the furnace, which track extends some distance on iron trestle over the cooling floor. The body of the cars is shaped like a hopper with bottom discharge, and closed by a slide which is worked by a lever. As each car holds about $2\frac{1}{2}$ tons and is

hot, they are pulled out from under the furnace by means of a windlass and chain.

The revolving motion of the Brückner furnace should be slow. There is not the least advantage in whirling the ore around and around in the furnace. When new ore particles are brought to the surface it takes some time to undergo oxidation; why not, then, give them the required time to be in contact with the air before again immersing them under the surface? I found in some works large Brückner furnaces set to make two and even two and a half revolutions per minute, whereas a speed of one revolution in three minutes is ample. A Brückner furnace, when charged, weighs about 16 tons; this divided on four wheels makes 4 tons to the wheel, a rather heavy weight, especially if we consider that the whole of this weight is pressing against the space of contact between the ring-track of the furnace and the surface of the wheel, which is very small. The effect of this high pressure is shown by the way the wheels and the ring-tracks wear. When the furnace is revolving it can be observed that continually thin scales of iron up to the size of a finger nail are dropping from the face of the wheels and tracks. Now if two revolutions are made in one minute, instead of one revolution in three minutes, the wear will be six times as great, to say nothing about the greater power which is required for the faster motion. If the capacity of the furnace should be increased by it six times it would be different, but this is not the case; a charge roasted at a high furnace speed takes just as long to be finished as a charge does when rotated at a moderately slow speed. Besides much increasing the wear, a high speed can cause serious trouble if the ore has a tendency to ball, in which case it may happen that the whole charge is transformed into balls from the size of a pea to that of a cocoonut, without leaving any fine stuff at all.

One of the main advantages of the Brückner furnace is the fact that the ore in this furnace is kept in a thick layer and still permits a thorough oxidation and chlorination. As explained above, the loss of silver by volatilization is much less if the ore during roasting can be kept in a thick layer, because a large portion of the evolving volatile chlorides is kept back in the ore as by a filter, and consequently much less silver will be carried away.

Under equal conditions, it will be found that an ore loses less

silver if chloridized in a Brückner than in any other furnace in which the ore is roasted in a thin layer.

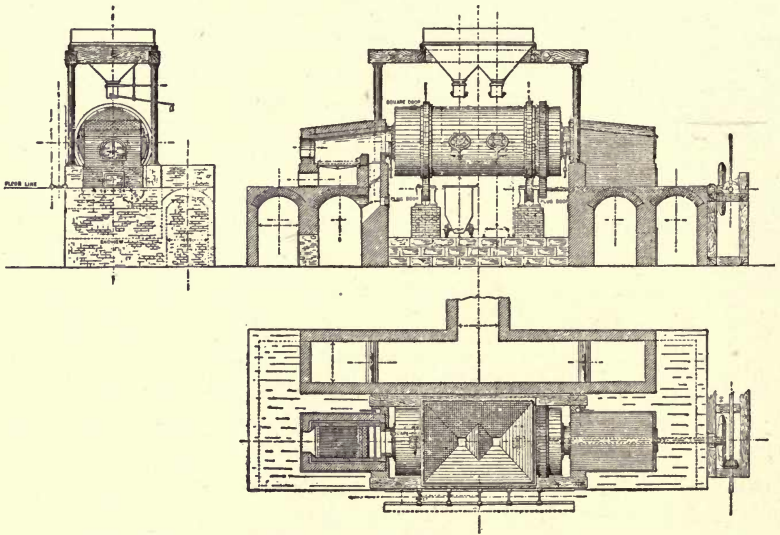
Ores not suitable for the Brückner furnace are those which, on account of a large percentage of lead, antimony, etc., cake easily, because by the continuous rolling of the ore any lumps which may have formed cannot be mashed shortly after they are formed, but increase much in size by the continual rolling, which makes them hard and dense.

(b) *The O. Hofmann Improved Brückner Furnace.* — In the Brückner furnace the ore is exposed to a rather uneven heat. The part next to the fire receives always the highest heat, while that at the other end, 16 ft. away, will receive much less, in some cases not even enough without overheating the fire end. This is a defect of the furnace which is of no consequence if a highly sulphureted ore is treated, which can be subjected to self-roasting, because it creates ample heat by itself to become uniformly hot through the entire length of the furnace; but it is a defect much felt if ore poor in sulphur is to be roasted, which has to receive nearly all the required heat from the fire, or an ore which has to be roasted at a low heat, because it is apt to lose much silver by volatilization, or lumps easily at a higher temperature (see chloridizing roasting with steam, page 34). Mr. Brückner was aware of this defect, and he tried to remedy it by inserting into the furnace a diaphragm made of cast-iron pipes. This diaphragm was set at an angle of about 15 deg. to the axis. It had a diagonal position extending through the whole length of the furnace and was intended to move the ore from one end to the other and back, in order to produce a uniform heating of the charge. This device, however, did not give the satisfaction expected and was soon abandoned, especially on account of the inconvenience the diaphragm caused in cleaning the furnace from the crust, which has to be done from time to time, and of the short life of the pipes, though they projected through the shell of the furnace to permit air to pass through.

Confronted with the necessity of obviating this defect of the Brückner furnace, because the rich ores of the Silver King mine, Arizona, could not be roasted successfully in the Brückner owing to this defect, I changed the arrangement of the furnaces, inasmuch as I attached a fire-box and flue arrangement to each end of the furnace, which enabled me to heat either

end of the furnace by changing at intervals the course of the flame.

Figures 15, 16, and 17 represent the arrangement. Between the fireplace proper and the furnace is situated the flue, extending downward to the dust-chambers, Fig. 15. This flue is provided with a damper. The other end of the furnace is provided with exactly the same arrangement. The dust-chambers from both ends are connected with the main flue leading to the stack. Before they connect there is an additional damper, Fig. 16, on each side, to make it sure that, if the dampers of one side are closed, no draft passes through on that side.



FIGS. 15-17. — HOFMANN IMPROVED BRÜCKNER FURNACE.

If the fire is kept at one side for some time the dampers of that same side are opened and those of the other are closed, and the fire started there. The flame now traverses the furnace in the other direction, and the ore at that end will be exposed to the same heat as the other was before. This changing can be done at intervals to suit the character of the ore. The changing of the fire does not cause any trouble, as the opposite fireplace is still warm enough to ignite the wood when the change is made.

The results obtained with this furnace have been very satis-

factory. I obtained with it good results in roasting the ores of the North Mexican Silver Mining Company, Mexico, which were very poor in sulphur and could not be heated sufficiently at the opposite end with a fire at one end only. With the double arrangement quite satisfactory results were obtained.

If a number of these furnaces are built in a row the fires of all of them have to be changed at the same time, which is necessary on account of the dampers in the two wings of dust-chambers and on account of the man attending the furnaces. If the fires are all on one side he will and can attend better and to more furnaces than if the fires are partly on one and partly on the other side.

There is quite a serious omission in the construction of the regular Brückner furnace, and that is that there is no provision made for the admittance of live air into the furnace. Air being such an important factor in roasting it is absolutely necessary to have some means to admit air if the roasting is to be conducted intelligently.

To keep the fire-door open for this purpose is not to be recommended, as this shortens the flame. In the Hofmann furnace provision is made for an air inlet by making the circular cast-iron throat of the fire-box longer than usual and by leaving in the lower half of the same a sufficiently large opening for the air. The size of the opening can be regulated by a hinged door and a lever.

On the periphery of the furnace and near each end is a small door for taking samples. These small doors are easy to handle, and they can be opened and closed and a sample taken without stopping the furnace, which is quite convenient.

The spouts of the hopper have to be made to stand pretty high above the furnace in order to permit the passage underneath of the door and the eye to which the door is keyed when open for discharging. This causes considerable spilling of ore during charging. To prevent this, the spouts are each provided with a sliding sleeve kept in position by a lever and weight, Fig. 17. When the furnace is stopped with the doors open for charging, the weight of the lever is removed and the sleeve lowered until it projects into the door. Then the hopper slide is pulled. This arrangement permits very clean work.

The lining of the furnace has to be done very carefully. The

door frames projecting inside should conform with the circle of the lining. Specially made bricks to fit the circle of the arch should be used only; if not, the lining will soon come out. To facilitate the work of lining it is well to rivet to the steel shell of the furnace six angle-iron ribs the whole length of the cylinder, which will divide it into six equal sections. The projecting part of the rib should be $3\frac{1}{2}$ in., so that when the bricks are laid the latter project half an inch above the rib. The groove which is formed by it is filled with clay. Thus each section forms an arch for itself, the angles serving as skew-backs. Each should be well keyed. In lining, the bricklayer can then bring the cylinder always in the most convenient position for his work, as there is no danger of caving even if that part of the lining which was made first comes to stand right above him. If, in course of time, from some reason or other, part of the lining should become defective, the bricks of the bad section can be taken out and replaced without disturbing the other parts of the lining. The ends should be lined first, so that the lining of the cylinder will abut against it.

As the furnaces of this type when charged are very heavy, the tracks with which they rest on the wheels should be made very solid, allowing considerable iron for wear. At the places at which the two tracks are to come a wrought-iron band an inch thick and as wide as the track, including its two-flange projection, is to be strongly riveted to the shell. To this band the track is fastened by means of tap-screws. Bolts will not answer because, if the track is worn and has to be replaced by a new one, it will be found that many of the nuts are so tightly roasted in that the bolts will be turned off in trying to unscrew the nuts, and to renew these bolts means the taking out of a part of the lining, which ought not to be. If a tap-bolt breaks, the piece in the wrought-iron band can be bored out and a new tap-bolt put in without difficulty. It is best to have the track cast in about four segments. The joints of the segments should not be square with the track, but slanting, because the furnace will revolve more smoothly and the chipping of the edges of the joints will be much less.

As in any other roasting furnace, in course of time a crust is formed, which has to be removed from time to time. This is usually done by cooling down the furnace and having men remove

the crust by picks, shovels and other tools. When cold the crust is quite hard, and the removal of it endangers and weakens the lining. While hot this crust, however, is rather soft, which suggested to me a method by which the furnace can be quickly cleaned without cooling or shutting down the furnace. When the furnace needs cleaning, after it has been discharged, a charge is made up of bricks, fire-bricks if possible, and the furnace revolved while a very strong fire is kept. The heat softens the crust, and the bricks in moving in the furnace gradually shave off the crust without injuring the lining. This is done in a few hours, and when the bricks and crust are discharged the furnace is not only in shape but also heated to receive a new charge. Besides this, the men are relieved from an unhealthful job.

There are other roasting furnaces which properly belong to the class of mechanical furnaces fed by charge, like the revolving-hearth furnaces; but they never were used much in actual practice, they are not very convenient, and are of small capacity, so that I can without hesitation leave them undescribed.

(2) MECHANICAL ROASTING FURNACES WITH CONTINUOUS FEEDING

To this class of furnaces belong the O'Harra, the Ropp the Howell-White and the Stetefeldt. The Brown and Pearce furnaces, while they are excellent for oxidizing roasting, are not quite suitable for chloridizing roasting.

(a) *The O'Harra Furnace.*—This furnace was the first mechanical furnace with continuous feeding devised on the Pacific slope. It is very ingeniously arranged and is, in improved form, still in use. In the following I give a description of this furnace by the late G. Küstel, who had much experience with it:

“This furnace was first tried in 1862 or 1863 in Dayton, Nevada, and later three of them were built in Flint, Idaho. The main feature of this furnace is the endless chain to which two oval rings are attached, the wings being as wide as the cross-section of the hearth. To these rings are fastened the plows or shoes by which the ore is gradually pushed forward. The hearth of the furnaces built in Flint were 104 ft. long and nearly 5 ft. wide. Eighty feet of this hearth were covered by an arch 12 in. high; attached to it were three fireplaces — two on one side, and

one between the two on the other side. At one end was the feeding hearth, which was not covered by the arch, and on which the ore was continually delivered from the stamp battery by mechanical contrivances. The motion of the ore was effected by an endless chain, passing over two chain wheels, one at each end. To this chain two oblong flat rings were attached, each provided with eight shovels or plows, so arranged that while one of the rings shoveled the ore toward the center line, the other pushed it back again toward the sides every three or four minutes (or in shorter intervals if more ore is charged). The ore not only changed its place to the right and left, but it also moved forward by degrees, so that in course of six hours from the beginning it commenced to be discharged, passing 18 ft. over a cooling hearth. On both ends of the furnace were iron doors hung on hinges which were opened by the ring every time it passed.

“The whole plant at Flint was arranged to work automatically. The five batteries, of five stamps each, had on both long sides endless screws, by which the crushed ore was forwarded in proportion as it discharged from the battery, and dropped into an elevator. Having been lifted about 15 ft., it was conveyed again by endless screws along the feeding hearths of all three furnaces. The discharge of this conveyor was so regulated that each feeding hearth received an even part of the ore. The ore mixed with 5 per cent. of salt was spread on iron plates behind the batteries (heated by the hot gases from the furnaces, conveyed through the flue and under the plates). When charged into the battery, the ore was not further handled till it came out of the furnace perfectly roasted.

“There is only one obstacle connected with this mechanical furnace. The shoes or shovels, touching the sides of the furnace, wear off by degrees, leaving a space which is taken up by the ore. This part of the ore along the wall hardens and increases in amount in the furnace till new shoes are put in. By these the crust of one-half to three-quarters of an inch thick is broken off and carried out. From the Rising Star ore these crusts contain nearly as much silver chloride as the well-roasted ore; they are, nevertheless, disagreeable, but some means might be devised by which this inconvenience could be avoided.

“The ore from the Rising Star mine at Flint contained argentiferous fahlerz, miargyrite, ruby silver, zinc blende, galena,

iron pyrites, and sulphide of antimony. At an average the ore contained between 69 to 77 ounces silver per ton and some gold. The gangue was quartz. It was crushed through sieves with forty holes to the inch, together with 5 per cent. of salt. The ore in the furnace, when reaching the first fireplace, commenced to roast oxidizingly. Between this fireplace and the second, which was on the other side, the chlorination began at an increased heat. Between the second and third fireplace the chlorination was finished at a high red heat. Although not more than 5 per cent. of salt was added, the roasted ore contained about 90 per cent. of the silver converted into a chloride. The gases, containing free chlorine and chloride combinations emitting chlorine, coming in contact with the surface of the ore while passing over it for a

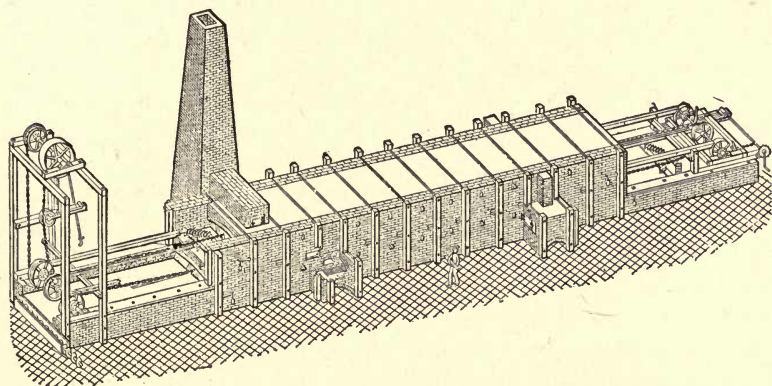


FIG. 18.—O'HARRA FURNACE.

space of eighty feet, have a chloridizing influence on it, replacing thus a certain amount of salt.

"The capacity of the three furnaces was more than twenty tons. Each one could easily treat ten tons of the Rising Star ore in twenty-four hours. The roasted ore was treated by amalgamation in pans."

Mr. Küstel continues:

"O'Harra's furnace is now greatly improved (Fig. 18). It is built in two stories, so that when the chain comes out of the lower hearth it turns into that of the upper story. The chain is heavy and there is no trouble whatever. Although the chain in its course through the red-hot furnace is exposed to red heat,

it nevertheless does not become so hot as to suffer any injury from it; there is a wooden framework on either end of the furnace over which the chain and plows move in the open air, which prevents them from getting too hot. The furnace, of the latest construction, is eight feet wide and from forty to a hundred feet long, with four fires, two on each side, directly opposite each other. The first two fires, where the ore comes in contact therewith, are divided, so that one-half of the flame goes direct to the lower, and the other half to the upper hearth, through an opening in the arch of the fire-chamber. There is a fire-clay damper to regulate the flame. The other two fires are opposite each other, so that the heat is uniform over the whole hearth. These last two fires are regulated by the ash-pit dampers.

“The endless chain has two triangular frames, with plowshoes on each side. The cooling space is built in proportion to the length of the hearth. A furnace of this kind, fifty feet long and eight feet wide, can roast from thirty to forty tons in twenty-four hours, with the help of only two men, consuming about two cords and a half of wood. The chlorination runs up to 90 and 95 per cent.

“The working of this furnace is not expensive, as two men, one at daytime the other at night, can attend the roasting of forty tons.

“A remarkable feature of O’Harra’s furnace is the very small amount of dust that is carried off by the draft. Another peculiarity of the furnace is for drying ore in pieces the size of a man’s fist. One furnace near Shasta, California, dries 40 tons of ore in twenty-four hours, at a small expense.”

The construction of this furnace does not offer many facilities for regulating the process, and therefore it can be used for chloridizing roasting only for a particular class of ore. To roast in this furnace ores which are apt to lose much silver by volatilizing would be rather risky. Moreover this furnace is not suitable for roasting ores which require the addition of salt after the sulphating period, at least not in its present construction. However, this furnace is well adapted for quite a variety of ores, especially those which do not cake easily and contain a considerable quantity of iron pyrites.

(b) *The Ropp Furnace.* — It is apparent that by dragging the plows on the bottom of the furnace, as in the O’Harra furnace,

the bottom and the rabblers or plows will suffer much wear, and not less the chain, especially when highly sulphureted ore is roasted.

Alfred von der Ropp has obviated these obstacles by a very ingenious construction of his furnace. Figs. 19 and 21 B represent a horizontal section, and Fig. 21 A an elevation of his furnace, while Fig. 20 is a cross-section through fireplace and hearth.

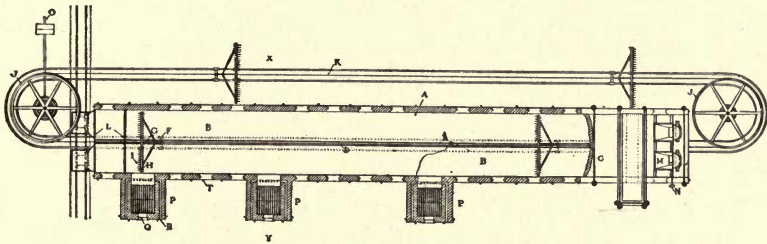


FIG. 19. — HORIZONTAL SECTION OF ROPP FURNACE.

It is a one-story straight hearth furnace, 105 ft. long and 11 ft. wide in the clear. The hearth is longitudinally divided into two even parts by a slot, which extends the whole length. The two sides of the slot project above the hearth surface to prevent the ore from falling into it. The slot communicates with a tunnel, *E*, underneath the furnace. This tunnel is provided with a track

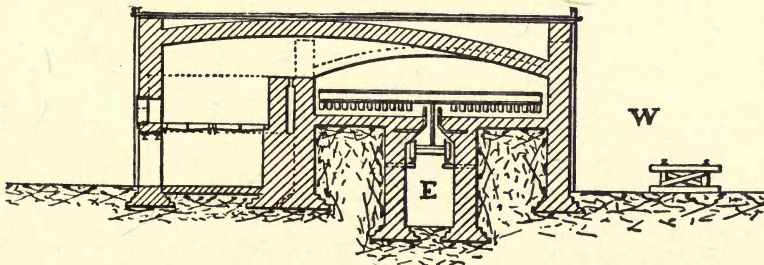


FIG. 20. — CROSS-SECTION OF ROPP FURNACE.

on which travels a four-wheeled truck or carriage. To this truck is fastened an iron arm which extends through the slot into the furnace, and to the end of which is attached a cross arm extending over the whole width of the hearth. This cross arm is provided with adjustable rabblers, which are designed not only to stir the ore, but also to move it gradually forward. An endless wire

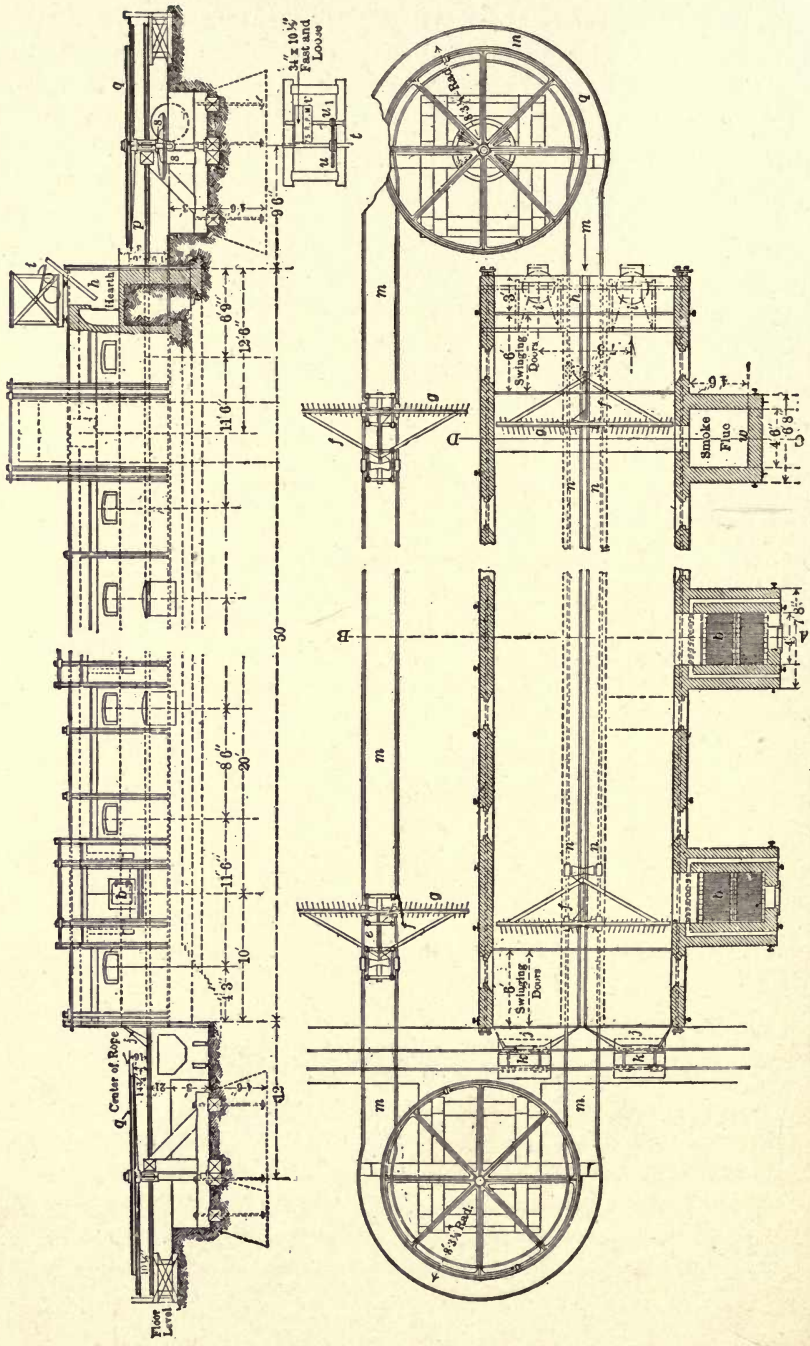


FIG. 21, A and B. — ROPP FURNACE, LONGITUDINAL ELEVATION AND PLAN.

rope moves the truck and with it the rabble arrangement. The truck, after passing underneath the furnace, turns a curve and returns on the outside to the other end of the furnace, where it enters the furnace again. On this track run four carriages or trucks, which are placed at even distances from each other. They are fastened to the wire rope. The rabbles are so arranged that if the preceding rake turns the ore toward the right the following one will turn it to the left. The rabbles can be set more or less slanting as circumstances require, and also can be lowered or raised. The tunnel underneath the furnace is high enough for a man to enter and to pass underneath the truck in case any repairs should be required. Three fireplaces provide the necessary heat. They are all on one side of the furnace and are constructed as shown in Fig. 20. The feeding of the ore is done automatically by Challenge feeders, at one end of the furnace. At the opposite end the roasted ore is brought out, each rake pushing forward a certain amount and dumping it into two iron cars. The two ends of the furnace are closed, like the O'Harra furnace, by swinging doors, which are opened by fenders attached to the rake and which drop back in position after the rake has passed. Numerous doors on both sides of the furnace serve for regulating the admission of air and give access for cleaning the hearth.

The driving mechanism is of simple construction. Two bevel gears transmit the power to a horizontal sheave around which the steel wire rope travels. A similar sheave is arranged on the opposite end of the furnace. The rope is entirely outside the furnace and is therefore perfectly protected from the heat, and so are the carriages. The rakes returning on the outside of the furnace for so long a distance have sufficiently cooled at the time when they enter the furnace again.

It is claimed that the furnace has a capacity of 36 tons in twenty-four hours roasting an ore which contains 20 per cent. sulphur, 8 per cent. lead and $17\frac{1}{2}$ per cent. zinc. This furnace is also excellently adapted for oxidizing roasting of iron pyrites and copper matte.

With respect to the roasting conditions of this furnace they are the same as prevail in the O'Harra, but the mechanical construction is much superior to that furnace.

(c) *The Howell-White Furnace.* — About the simplest and

cheapest continuous mechanical furnace, and at the same time perhaps the most effective, is the Howell-White. It is very simple in construction, requires little repair and very little power to run it, while its capacity is quite large.

This furnace is a revolving cylinder, open on both ends. One of them is connected with the fireplace while the other enters into the flue. The cylinder is made of cast iron in flanged sections, which in setting up are bolted together. The two sections nearest to the fire are lined with bricks to protect the iron, and in order to have the inside diameter, after lining, the same through the whole length of the furnace, the diameter of these two sections is 10 inches larger than the balance. The furnace rests on the top of five wheels all in one line and properly divided, as can be seen in Fig. 22. It is kept in place by four rollers which

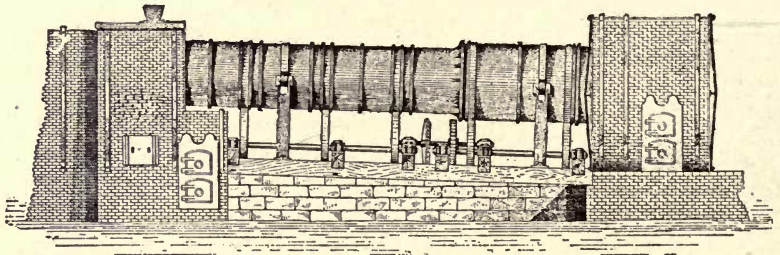


FIG. 22. — HOWELL-WHITE FURNACE.

touch the furnace half-way up its diameter, which are kept there by strong, stiff iron frames. The furnace is made to revolve on top of the wheels to diminish friction and consequently to make it turn more easily. It has a slight inclination toward the fire. The flue end of the cylinder is provided with a flange to prevent the ore from falling into the dust-chamber. The feeding is done through an inclined cast-iron spout, which passes through the arch of the dust-chamber and extends into the furnace to within five or six inches of the periphery. The feeding is done from a hopper by means of a worm, the speed of which is adjustable. To the inside of the unlined part of the cylinder are riveted several iron ribs in the shape of a spiral. The object of these ribs is to lift the ore, when the furnace is revolving, and to shower it through the flame. As this causes much dust, which is carried away by the draft, an auxiliary fireplace is arranged at that end,

the flame entering the dust-chamber right under the end of the furnace.

The roasted ore leaves the furnace gradually at the fire end and drops into a vault underground but conveniently accessible from the cooling floor. For this purpose a space about 12 in. wide is left between the fire-bridge and the furnace end. The

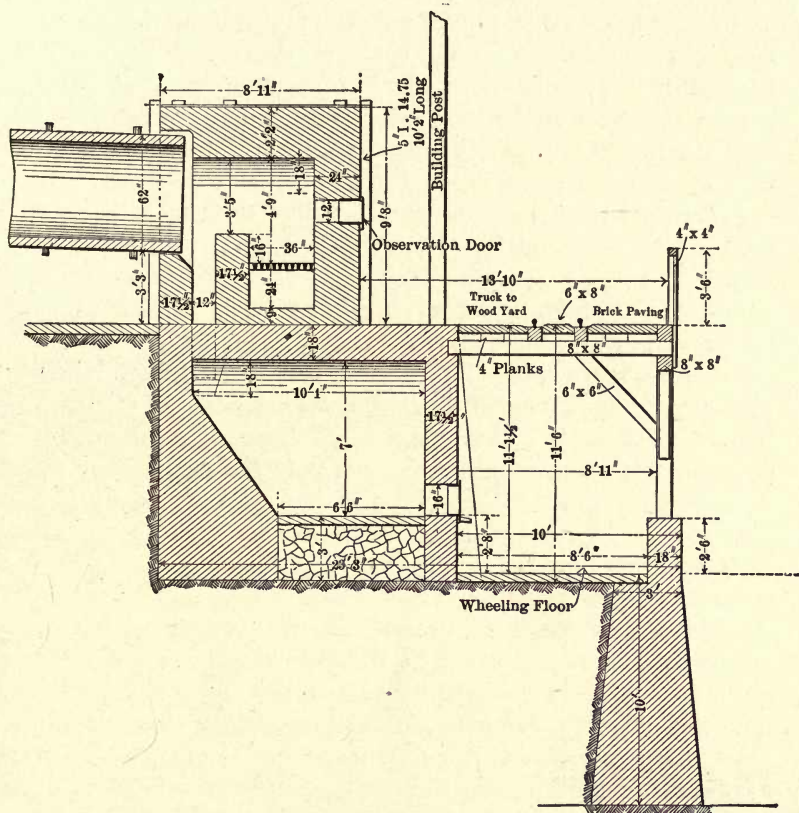


FIG. 23.—HOWELL FURNACE, DISCHARGE END AND ORE-VAULT.

revolving speed of the furnace is adjustable; in most cases $2\frac{1}{2}$ to 3 revolutions per minute is sufficient. The forward movement of the ore in the cylinder is caused by the revolving motion, and the speed, therefore, regulates the amount of ore which passes through and the amount which is retained in the cylinder while in operation. Fig. 23 represents a longitudinal section showing

the relative position of fireplace and furnace, the ore-drop and the ore-vault.

The constant showering of the ore, brought up by the ribs, through the draft causes a separation of dust and sand, the dust being carried into the dust-chamber. The amount of dust deposited in the chambers as compared with that of the roasted ore dropping into the vault is from 30 to 50 per cent. The dust is well roasted, provided proper attention is paid to the auxiliary fire and that it is kept strong enough. This, however, can only be done if the fire is intrusted to an extra man. If it is made part of the duty of the man attending the fire at the discharge end, the auxiliary will always be more or less neglected. One man at each end can attend to three furnaces.

There are ores, however, which will not stand entering the sudden heat of the auxiliary without caking. Even if the dust is roasted well by the auxiliary fire, the formation of so much of it is very annoying and inconvenient. Actually there is no need to make so much dust with this furnace.

This excessive amount of dust is caused by the ribs, which produce a shower of ore through the swiftly moving gases. I found that these ribs are not necessary. There is no accumulation of ore in this furnace; in fact there is only a comparatively small stream of ore passing through, even if roasting is conducted at the rate of 30 tons in twenty-four hours. By the revolving motion of the furnace, this thin layer of ore is made to expose continually new particles to the action of heat and air, and the ore at the discharge end will be found just as well roasted without as with ribs; in fact better, because it will contain a larger percentage of fine material. Besides, the furnace after being incrustated offers a rough surface to the ore, which prevents its sliding swiftly and spreads it over quite a large surface. By removing the spiral ribs from the Howell furnaces of the Cusihiuriachic Mining Company, Mexico, I very much diminished the formation of dust; in fact so much so that the maintenance of the auxiliary fire was not justified, and was abandoned altogether. The dust from right behind the furnace was removed twice a day and elevated to the feed-bin, thus mixing it with the crude ore and feeding it into the furnace again.

To have an elevator between each two furnaces, which can be made to discharge into either of the two feed-hoppers, is very

convenient, not only for elevating the dust from the chambers, but also to elevate the ore-sweepings, which always accumulate around a roasting furnace.

This furnace radiates considerable heat, which affects the driving-belt and shortens its life; it is, therefore, much better to drive the furnace by means of sprockets and link chain.

It ought to be mentioned that there are two vaults beneath each furnace which can be filled alternately by turning a wing which is placed right under the roasted ore drop. On each side of this drop there should be a small door to permit the entrance of air into the furnace and to give access for tools in case it is necessary.

If one of the vaults is filled with ore, it is advisable not to discharge it until the other is nearly filled. This gives the ore an opportunity to improve in chlorination.

When the crust in the furnace becomes too thick it can be easily removed by inserting a number of bricks at the flue end. In revolving the bricks will shave off the soft crust and bring it out. When the bricks are charged the feed has to be stopped for a while; but as soon as they have moved away 4 or 5 ft. from the end, charging can be commenced again.

(d) *O. Hofmann's Modified Howell Furnace.* — The Howell furnace is a very efficient furnace; its first cost is small as compared to its roasting capacity; it does not need many repairs, and requires but very little manual labor. The furnace does not carry much ore at a time, and is therefore not excessively heavy; the way it is arranged, revolving on top of wheels, the friction is much reduced, so that the furnace revolves very easily and not more than two, perhaps two and a half, horse-power is required. All these favorable qualities make the Howell furnace a very desirable one; but it has the drawback, in common with all the continuous furnaces, that the salt has to be mixed with the ore before it enters the furnace. We have seen above that there are ores which cannot be chloridized unless the salt is added later during roasting, and as the construction of the Howell furnace does not permit of this, it makes it unfit for such ores, which is to be regretted considering the numerous advantageous features of this furnace.

In my metallurgical investigations of the heavy argentiferous zinc-lead ores of the San Francisco del Oro mine, Chihuahua,

Mexico, I was confronted, besides other roasting questions, with the problem as to the best method of chloridizing these ores in a Howell furnace. By mixing the salt with the ore in the stamp battery the ore became sticky, incrusting the furnace rapidly, and when it left the furnace consisted mostly of lumps without being much chloridized. If the ore was charged without salt, it remained loose and sandy, but it dusted so much that by the draft an almost perfect separation of coarse and fine took place, the latter being carried into the dust-chamber, while the coarse sand dropped in the pit, but, on account of its large percentage of lead and zinc blende, insufficiently oxidized, so that the salt, which was added from time to time to the ore in the pit and stirred, had but very little chloridizing effect. The best chlorination obtained was only 29 per cent. To diminish this separation a small percentage of salt, from 1 to 2 per cent. was added in the battery. The effect was remarkable; without balling or incrusting the furnace the dusting was practically stopped. When the balance of the salt was added in the pit a chlorination of 67 per cent. was obtained.

Based on this observation a modification of the arrangement in front of the furnace was made. Between the fireplace and the discharge end of the furnace a shallow pit was inserted, which was in communication with a reverberatory furnace 6 by 8 ft. The bottoms of both were on the same level. The fireplace of the reverberatory was only 24 in. wide. The gases from the reverberatory passed through the pit and the furnace. When a charge of about 1400 lb. had accumulated in the pit the same was pushed into the reverberatory, salt added and well mixed. There the ore was kept until another charge had accumulated in the pit. By this modification very satisfactory chlorination was obtained. A very low fire was kept on both the fireplaces.

(e) *The Stetefeldt Furnace.* — This furnace consists of an upright shaft 30 to 40 ft. high, which is connected near the top with a descending flue. On top of the shaft is a feeding machine, which showers the ore into the shaft. In the lower part are two fireplaces opposite each other, so that the descending ore meets the hot ascending gases. The principle on which the construction of this furnace is based is at variance with that of any other chloridizing furnace. The ore falls very finely divided through a glowing atmosphere of chlorine, sulphurous acid,

oxygen and fire gases, metal chlorides and volatilized salt, and the whole complicated reactions which take place in chloridizing roasting are completed in the incredibly short time of a very few seconds. Mr. Stetefeldt derived his idea from the Gerstenhöfer pyrites roaster, in which the ore is fed into a shaft, wherein numerous shelves are so arranged that the ore drops from one shelf to the other, resting on each for some time. The experimental furnace Mr. Stetefeldt built was of such a construction arranged so that fire gases were permitted to enter the shaft. The shelves, however, caused much inconvenience by incrustation, etc., and by observations he made during these experiments he thought it justifiable to repeat the experiment without the use of any shelves. The results were so gratifying that he, adhering to this new principle, gradually developed his furnace to the present much improved construction.

Fig. 24 represents a vertical section of this furnace. *A*, shaft; *G*, returning flue; *K*, receiver or hopper forming the bottom of the shaft; *C, C*, the two fireplaces; *T*, slit for the fire gases; *U*, air ducts to produce a perfect combustion, and at the same time to cool the fire arch; *E*, ash-pit, the iron door of which is provided with a slide to regulate the air inlet; *O, O*, doors for the insertion of tools for cleaning the walls. The returning flue, *G*, is provided with doors, *R*, which serve to clean it. *D* is an auxiliary fireplace, which is constructed like the fireplaces of the shaft and which serves to roast the large amount of flue-dust which this furnace makes. Passing the chamber *H* the dust enters *V* and the larger part of it settles in the bottom hoppers, *I, I*, from which it is drawn into iron cars by moving the dampers *S, S*. The rest of the dust is collected in a number of dust-chambers, *Q*, which are connected with the chimney by means of a long flue. *P, P* are doors for observation and cleaning. Below the shaft-hopper, *K*, is the slide door *L*. By pulling this slide the accumulated roasted ore drops into a large iron car. *B* is a cast-iron frame with water-jacket on top of the shaft on which the ore-feeding machine is placed. Above it is an ore bin (not shown in the diagram) from which the ore is fed into the machine by means of a worm.

The feeding machine is shown in Fig. 25. *A* is a cast-iron, water-cooled frame placed on top of the shaft and provided with the damper, *B*, which is withdrawn when the furnace is in opera-

tion, but which is inserted if the feeding machine stops for any repair or for exchange of screens. *C* is a cast-iron grate to which on the upper side is fastened the punched screen, *D*, which is made of a steel plate with holes of an eighth of an inch. Above the punched screen is placed a frame, *E*, to the bottom of which is fastened a coarse wire screen, *F*, with about three meshes to

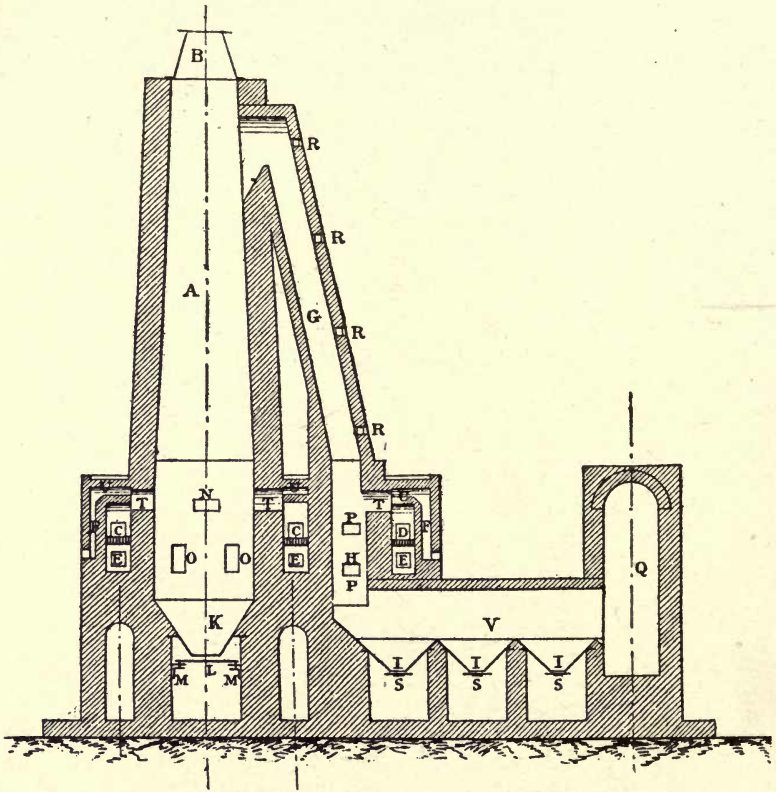


FIG. 24. — STETEFELDT FURNACE.

the inch, made of heavy wire. The frame, *E*, rests upon friction rollers, *G*. The brackets, *H*, which hold the friction rollers, can be raised or lowered by set-screws so that the wire screen can be brought closer or less close to the punched screen. The brackets *K* carry an eccentric shaft, by which an oscillating motion is given to the frame *E*. To the brackets *N* are fastened transverse stationary blades, *O*, which extend close to the wire screen and

which can be adjusted by the nuts *P*. These blades keep the ore uniformly spread over the screen when the machine is in motion. The ore is usually crushed through a 40-mesh screen.

Before starting the feed the furnace has to be well heated, which takes thirty-six to forty hours. After the speed of the feeder is set and the proper temperature is ascertained by observation and numerous assays, the roasting itself requires but very little attention beyond the maintenance of a uniform temperature.

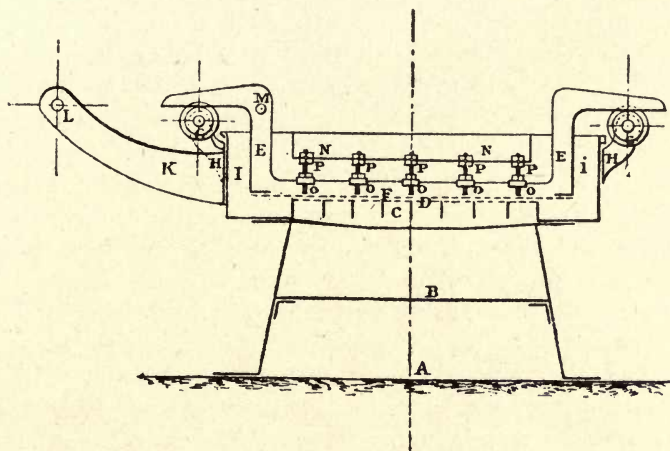


FIG. 25. — FEEDING MACHINE, STETEFELDT FURNACE.

The capacity of the furnace is very large, roasting, according to the size of it and the character of the ore, from 20 to 50 tons in twenty-four hours. The consumption of fuel is rather small. With one cord of wood about 8 tons of ore can be roasted. The loss of silver by volatilization is less than if the same ore was roasted at the same heat in another furnace, because, according to Plattner, the loss of silver does not depend merely on the temperature the ore is roasted at and the character of the ore, but also on the length of time the ore is subjected to the roasting temperature. In the Stetefeldt furnace the ore is roasted almost instantaneously and does not suffer that part of the loss which is caused by long heating.

This furnace is well adapted to roast ores not heavily charged with sulphides, containing 5 to 8 per cent. sulphur, or even 10 per cent., if no, or only a small percentage of, galena and zinc

blende is present. Ores heavily charged with sulphides are not suitable, especially if they contain a large percentage of galena and zinc blende. These minerals need a low, gradually increasing temperature, which conditions cannot be maintained in the Stetefeldt furnace. If highly sulphureted ore is fed into the shaft the temperature in the upper part of the shaft extending closely to the feeding machine becomes intense, and the conditions required for roasting such ores become reversed. Instead of being exposed to a low and then gradually increasing temperature, the ore encounters the hottest zone first, and combustion is so rapid that when it drops to the bottom it will be found to consist mostly of minute globules, formed by partial melting of the ore particles. Besides, much crust is formed at the bottom of the shaft as well as on the sides, which keeps falling down in large chunks. Even the iron grate on which the punched screen of the feeding machine rests becomes clogged with incrustation, which occurs so frequently as to make a regular feeding impossible. This was the reason that while a large number of this type of furnace were in successful operation on the Pacific slope, in Nevada, California, Utah, etc., all the numerous attempts to work the more mineralized ores of Mexico were failures.

For the proper ore, this furnace is undoubtedly the cheapest to roast ores, especially for those which, on account of their small percentage of sulphur, cause a large consumption of fuel. The conditions necessary to heat such ores quickly and effectively are excellent, because every particle is exposed to the heat when it passes in a shower through the flame.

G. Küstel had once to roast in a Stetefeldt furnace ores which were too poor in sulphur to produce sufficient chlorine. Sulphureted ore to mix with the dry ore could not be procured, either. Mr. Küstel solved the problem by burning brimstone in a cast-iron pan and conveying the gas into the furnace near the bottom of the shaft. This could not have been done successfully with any other furnace.

IX

COLLECTING THE FLUE-DUST

THE formation of dust in roasting is unavoidable. In some of the furnaces the amount of dust carried off by the draft is comparatively small, as in the reverberatory and Brückner, while in others, like the White-Howell and the Stetefeldt, it is excessive. The dust consists of ore particles, more or less roasted, carried out of the furnace by the draft, and of condensed fumes of volatilized metal chlorides and oxides. The former are much easier collected than the latter.

The problem of collecting this dust is quite a difficult one, but is very interesting and of great importance in metallurgy. The idea of collecting the dust by conveying the gases into large chambers in order to reduce the speed of their movement, thus giving time for the dust to settle, is not a correct one. Close observations have shown that in such a chamber a much larger accumulation of dust will be found close to the walls than on other parts of the floor. Besides, the walls will be found to be covered with scale-like formations of the dust, the pointed part of these scales turned against the current of the gases. When these scales become too heavy they peel off and drop to the floor, hence the larger accumulation of dust on the floor near the foot of the walls. The cause of this greater precipitation can be no other than the friction between the dust-charged gases and the walls. It can also be observed that wherever the flue makes a sharp turn the precipitation of dust is greater, because the friction between wall and gases is much greater if the latter are forced to make a sudden change in their course than if they are permitted to follow a straight-line course. Furthermore, it can be observed that, if an obstacle is placed in the flue, against which the moving gases have to strike, it will cause a precipitation of dust which increases with the swiftness of the gas current.

These observations demonstrate that a slow movement of the gases, as attained with large dust-chambers, is by far not so effective as a swift movement with increased friction, and therefore much better results will be obtained if the dust-chambers are so constructed as to offer a large wall surface as compared with the area of the cross-section of the chambers, and if the same are arranged in a zigzag fashion.

Based on the above observations I have devised and constructed a dust-collecting arrangement which is very effective and gives much satisfaction.

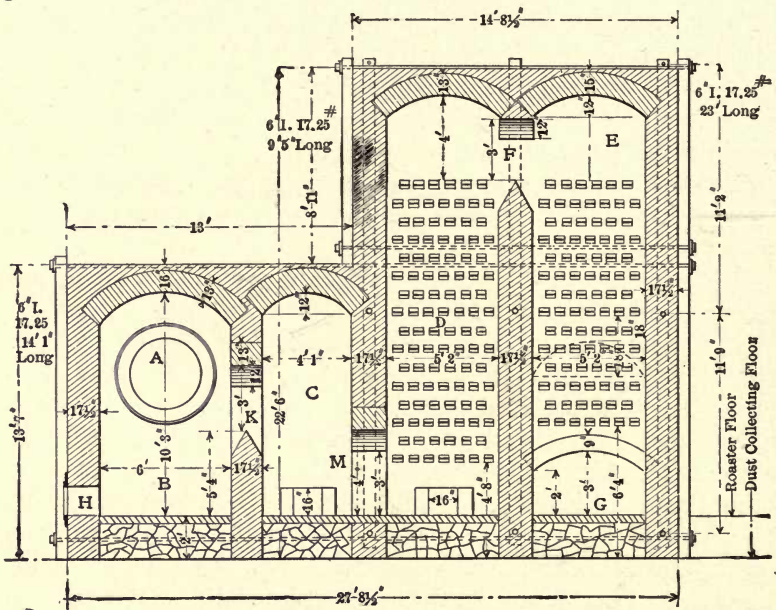


FIG. 26.—VERTICAL SECTION OF HOFMANN DUST COLLECTOR.

O. Hofmann's Flue-Dust Collector.—Fig. 26 represents in a vertical section the dust-collecting arrangement in connection with a White-Howell roasting furnace. *A*, feed end of the furnace; *B*, first chamber, the end of the furnace projecting a few inches into it. Here the coarsest part of the dust accumulates and is removed from time to time through the door *H*. From here the gases pass through the opening, *K*, into the second chamber, *C*, and from there through the arch, *M*, into the collecting shaft, *D*, which they ascend, leaving it through the flue, *F*, and entering

the second collecting shaft, *E*, in which they descend, leaving the same through *G*, which makes connection with the general flue leading to the chimney.

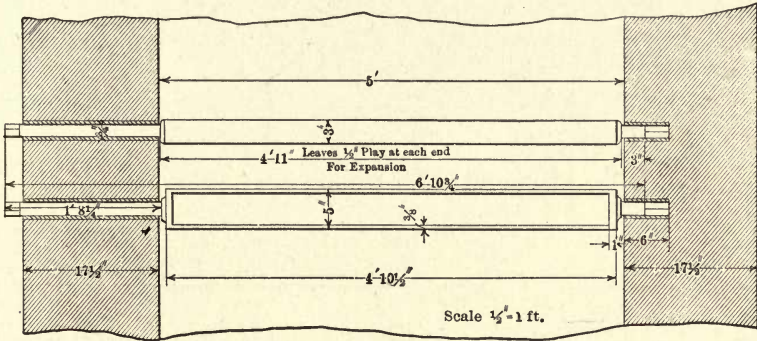


FIG. 27. — DETAILS OF BARS AND BEARINGS, HOFMANN DUST COLLECTOR.

In the collecting shafts, cast-iron double channel irons are arranged in rows, leaving a space 3 in. wide between each two. These channels are closed at each end (Fig. 27) and extend as round bars $1\frac{3}{4}$ in. in diameter and 1 ft. $8\frac{1}{2}$ in. long at the front and 3 in. long at the opposite side. These two cylindrical exten-

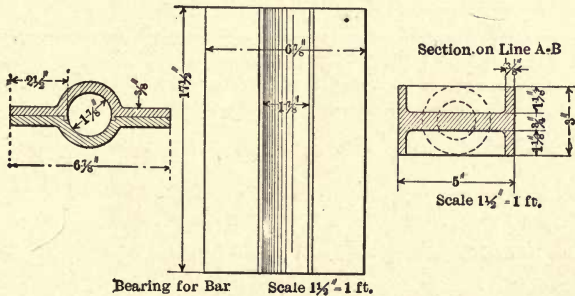


FIG. 28. — DETAILS OF BARS AND BEARINGS, HOFMANN DUST COLLECTOR.

sions rest in cast-iron bearings, as shown in Fig. 28. The longer bar extends through the front wall of the shaft, and is cast square at the very end, which part projects out of the wall, while the shorter rests in a bearing inserted in the back wall of the shaft. These channels can be turned by a socket wrench slipped over

the square end. Each row is 10 in. above the other, and the position of the channels is such that the channels of one row are placed right above the open spaces of the row below, as shown in Fig. 29. The bearings are $1\frac{7}{8}$ in. in diameter while the cylindrical parts of the bars are only $1\frac{3}{4}$ in., in order to allow room for expansion and to permit an easy turning of the bars when hot.

The gases from the furnace entering the shaft *D*, through *M* (Fig. 26), in ascending partly strike the channels of the first row and partly pass through the open spaces. That part, however, which strikes the channel rebounds and is also forced to pass through the open spaces. Passing through, they strike against

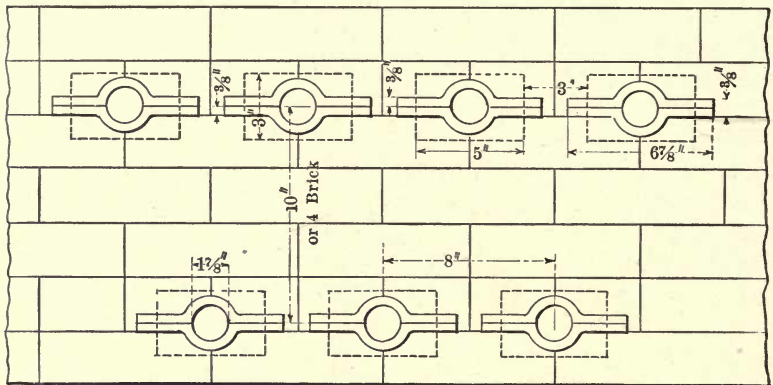


FIG. 29.—POSITION OF BARS, HOFMANN DUST COLLECTOR.

Bearing should be made $1\frac{7}{8}$ in. diameter, or $\frac{1}{8}$ in. larger than the bar, so that the latter can be easily turned when hot.

the channels of the next row above, rebound, and force their way again through the open spaces. This is repeated until the gases leave the top row and enter the second shaft, *E*. Here the same play of the gases takes place, only that they have to descend through the shaft.

It is apparent that the numerous objects placed in the path of the gases and against which they have to strike, and the large surface which they offer for friction will produce a very effective precipitation of the fumes and dust. This is actually the case. I erected at the works of the United Zinc and Chemical Company, Argentine, Kansas, a system of three such collecting shafts. Blende-pyrite ore was roasted in mechanical furnaces to

produce sulphur dioxide gas for the manufacture of sulphuric acid, and it was of importance that the gases should enter the Gay-Lussac tower as free of dust as possible. After the system was in operation for some time an investigation was made as to its efficiency. The fumes passed first through a down flue and a piece of straight flue, in which the coarse dust accumulated before it entered the first shaft. It was found that the dust on the bottom of the first shaft had the color of the roasted ore, showing that the main part of it consisted principally of mechanical ore-dust. The dust on the bottom of the second shaft was of a light pink color, showing that the main part of it consisted of precipitated white fumes of lead, iron and zinc. On the bottom of the third shaft the dust was perfectly white, showing that all the ore-dust was precipitated before it reached the third shaft, which was proved by actual analysis of this dust, showing it to consist of the sulphates of lead, zinc and principally iron, while almost no insolubles were present. In the first shaft were found 6.65 cu. ft. of dust; in the second, 4.15 cu. ft., and in the third only 1.55 cu. ft. This very rapidly decreasing volume of dust found in each succeeding shaft illustrates the great efficiency of this system. I may mention as a further illustration that, at the bottom of the down flue, where the coarse settled, the layer of dust was 7 in. deep. In the piece of straight flue next to the first shaft the layer was only $1\frac{1}{2}$ in. thick, while in the first shaft it was 8 in.

The channels have to be shaken and turned from time to time, depending on the dusting qualities of the ore and furnace. Once or twice a week will be found sufficient. The shaking and turning should, of course, be commenced at the top row. Fig. 30 shows the dust-collecting and flue arrangement in connection with two White-Howell furnaces.

This dust-collecting arrangement is compact, and very effective, and ought to be inserted in all the works where ores are roasted, either in order to regain the valuable dust and the volatilized silver, copper, lead, etc., or to prevent the entering of the dust and volatilized substances into the subsequent chemical process.

A very effective method of collecting the dust is the bag system, in which the gases are forced by fans through long bags made of burlap, flannel or muslin, which act as filters. The gases,

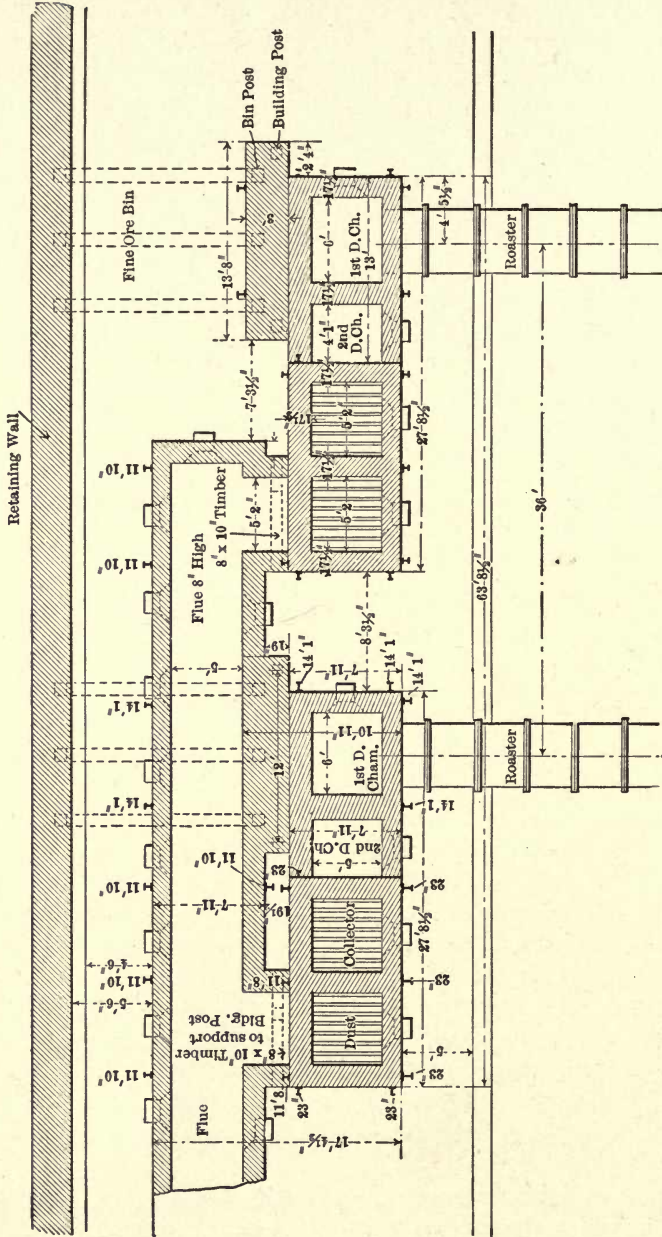


FIG. 30. — HORIZONTAL SECTION, HOFMANN DUST COLLECTOR.
Arrangement for two White-Howell Furnaces 52 x 62 in. x 34 ft. 1 in.

however, have to be first cooled sufficiently so as not to ignite the bags, which cannot always be easily and cheaply accomplished. Besides, if, as in a sulphuric acid factory, the furnace gases have to enter the process hot, the bag system cannot be applied. This system is often used in smelting works.

SULPHATING ROASTING

THIS mode of roasting, which has the object of converting the silver into silver sulphate, in which state it is soluble in water, is only used if silver is to be extracted with hot water by Ziervogel's method.

The material to be suitable for this roasting has to consist principally of copper and iron sulphides, of which the former has to predominate, and has to be free of, or to contain only in small quantities, the sulphides of lead, zinc, arsenic, and antimony. For this reason it is exclusively used for argentiferous copper matte. In this roasting the copper and iron have to be converted into oxides, while the silver has to be changed into a sulphate. The transformation of the silver into sulphate is done almost exclusively by the sulphuric acid fumes which result from the decomposition of cupric sulphate at a higher heat. Cupric sulphate and ferrous sulphate are formed in the first stage of roasting. Ferrous sulphate is decomposed at a much lower temperature than cupric sulphate, in fact at a temperature not high enough for the formation of silver sulphate, so that the sulphuric acid generated by the decomposition of the ferrous sulphate is of very little avail for the formation of silver sulphate; it is, however, of great effect in the formation of cupric sulphate, which then, at a higher heat, sulphatizes the silver.

A certain percentage of iron sulphide is therefore advantageous for this roasting process, but if the iron sulphide is in excess the formation of silver sulphate, and with it the extraction, suffers.

At Mansfeld, Germany, where this mode of roasting and the subsequent extraction of the silver with hot water was originated by Mr. Ziervogel, the roasting charge consisted of sulphur 19.32 per cent., copper 58, iron 9.18, lead 2.48, zinc 4.31, manganese

0.15, nickel 0.43, cobalt 0.83, silver 0.286, insoluble 1.08 per cent., and permitted an extraction of 91 per cent. of the silver, while at Schemnitz, Hungary, a matte containing 88 per cent. iron sulphide and only 1.5 per cent., of copper sulphide, which was tried by this method, permitted only an extraction of 73 to 75 per cent. of the silver.

This roasting is a very delicate process and has to be conducted with great care and skill, otherwise inferior results will be obtained.

At Mansfeld the roasting is done in a two-story reverberatory furnace. The operations are as follows:

Six hundred pounds of pulverized copper matte are charged on the upper hearth, spread, and about 5 lb. of slacked bituminous coal scattered over the charge, and stirred. This addition of coal is made merely to help heat the charge in order to hasten the operation. A matte richer in iron sulphide does not need the addition of coal, because iron sulphide ignites quicker than copper sulphide. The charge is stirred continually, and the lumps which form have to be mashed with the furnace shovel. They are soft and easily mashed, and are more numerous if the material contains more iron than if it is poorer in iron. They are caused by the conversion of the ferrous sulphate into basic ferric sulphate, which melts easily. In this period the iron oxidizes before the copper, and by the action of the sulphuric acid changes into ferrous sulphate, which later at an increased heat gives off sulphuric acid fumes and changes into ferric oxide and basic ferric sulphate. The copper sulphide is converted into cupric sulphate, but more by the acid fumes of the ferrous salt than by the action of the air.

The time of roasting on the upper hearth is governed by the time required to finish the charge on the lower hearth. During this period, which lasts from five and one-half to six hours, the charge has to be turned twice so that all parts of it are exposed to the same heat.

When the lower hearth is clear, 25 lb. of slack bituminous coal is spread over the charge, which then is drawn to the drop-hole in the bottom of the hearth, through which it falls to the lower hearth. When the coal is mixed with the charge, burning gases are emanating from the ore.

At the end of the first half-hour the ore on the lower hearth

commences to glow brighter than it did on the upper hearth, caused by the higher temperature kept here and the further oxidation of the sulphur. The thickness of the charge, which is about $2\frac{1}{2}$ in., swells, on account of the burning sulphur, to $3\frac{1}{2}$ or 4 in. In order not to burn the coal in the charge too quickly by the action of the air, and to give it better opportunity to act on the salts in the ore, the draft is very much checked while the charge is raked continually and very briskly, in order to avoid as much as possible the formation of lumps. This is done for an hour, after which time all the coal is consumed. Then the charge is turned, the part from the hotter place to the cooler, and that from the cooler to the hotter place. After this the draft is increased to its full capacity in order to produce a strong oxidation by the inflowing air. This cools the charge after a while until it becomes almost dark. To judge the end of this period, a sample is taken from the middle of the hearth, cooled, the fine separated from the lumps, and by means of a spatula a ridge is made of the fine in a porcelain saucer. The saucer is held slightly inclined, and some water, drop by drop, is poured behind the ridge. The water is first absorbed by the sample, but after being saturated a clear liquor slowly flows out from the other side of the ridge. By the color of this liquor and its behavior toward salt the progress of the roasting is judged. If the roasting was conducted right, by this time the liquor should have a clear blue color and by the addition of some salt should give a light precipitate of silver chloride, which is a sign that the silver sulphating has commenced. If the liquor has a dirty greenish color it shows that some ferrous sulphate is still undecomposed and a continuation of the oxidizing period is required.

The coal, which is added to the charge and vigorously raked and mixed with the latter while the draft is much checked, acts on the neutral sulphates, which are converted into basic sulphates while sulphurous acid escapes. After the dampers are opened and the full draft is given to the furnace, the sulphides which may still exist will be completely roasted and all ferrous sulphate will disappear, which is necessary to be accomplished before the sulphating of the silver takes place. The cuprous oxide oxidizes to cupric, and at the end of this period the material should consist of the free oxides of iron and copper, basic salts of iron, copper and zinc, and neutral sulphates of copper, zinc, man-

ganese, some silver oxide and nearly all the balance of the silver as sulphide.

When it has been ascertained by the above test that the roasting has advanced to the proper stage, the ore is ready for the sulphating of the silver. It will be remembered that at the end of the previous period the material had cooled down almost to darkness by the increased draft. The temperature has to be increased again, but care is to be taken that this is done with a clean oxidizing flame, and that the latter does not touch the ore, so that none of the cupric oxide is reduced to cuprous oxide or to metallic copper, as both of them would precipitate metallic silver during the subsequent lixiviation, which silver would remain in the residues. For this purpose very dry, thin limb-wood should be used only. Pine is not to be recommended on account of its pitch, which causes a smoky flame. The charge has to be continually raked. After an hour it becomes dark red hot, and later increases to bright red. The strong fire has to be kept up uninterruptedly. If after two and a half hours the material near the fire-bridge is completely roasted, the charge is turned and roasting continued for one-half to three-quarters of an hour. The material is properly roasted when the solution emerging from the ridge of a sample on the saucer is only of a faint but clear blue color, showing that but little cupric sulphate is present, while by an addition of salt a heavy white precipitate of silver chloride is formed.

If the heat is too high, some of the silver sulphate will be reduced to metallic silver and all the cupric sulphate will be decomposed, and therefore the liquor of the test will be colorless.

By increasing the heat to bright red during this period, the fuming sulphuric acid liberated from the neutral cupric sulphate does not act so energetically on the silver sulphide as the sulphuric acid fumes resulting from the decomposition of the basic cupric sulphate; hence the addition of fine coal at the beginning of this period.

The time required for roasting a charge is eleven to twelve hours, of which five and a half to six are consumed on the lower, and, naturally, just as many hours on the upper hearth. All in all, about four charges or 2400 lb. of material will be roasted by each furnace during twenty-four hours.

The loss of silver in Mansfeld was found to be 7.06 per cent.

which was caused partly mechanically by flue-dust, partly by volatilization of silver oxide, which, however, in the cooler regions of the dust-chambers decomposed into silver and oxygen. In the roasted matte 91.74 per cent. of the silver was converted into silver sulphate and was extractable, while 1.20 per cent. silver remained in the residues.

XI

CHLORIDIZING OF ARGENTIFEROUS ZINC- LEAD ORE

IN this chapter the detail records are given of investigations of chloridizing roasting of argentiferous zinc blende and galena ore, which I had the opportunity to make on a large working scale.

The chloridizing roasting of this class of ore had not previously been made the subject of a thorough investigation on a large scale, and the record of such experiments and investigations may be of interest and practical value.

The San Francisco del Oro mine is situated near Santa Barbara and Parral, Chihuahua, Mexico. The ore contains on an average 26 to 30 oz. silver per ton, besides a trace of gold. The principal silver-bearing minerals are zinc blende, of which the ore carries 37 per cent. and more, and galena, of which it contains from 13 to 19½ per cent. The heavy-solid occurrence of the ore and the great width of the vein permits very cheap mining. The cost per ton does not exceed \$1.50 Mexican currency, including hoisting and a slight hand-assorting.

(a) *Zinc Blende*. — The fine-grained black zinc blende predominates, containing about 25 oz. silver per ton; but there occurs also a brown blende of a peculiar luster, somewhat resembling bronze-colored mica. It is richer in silver than the black blende, assaying from 55 to 70 oz. silver. The blende contains considerable cadmium.

(b) *Galena*, with 40 to 50 oz. silver per ton, is finely impregnated in the zinc blende, and can scarcely be detected with the eye, and only a very small portion occurs as defined galena, which makes it impracticable to lessen the percentage of lead in the ore by hand-sorting. An attempt was made to sort out the lead ore for shipping, but it was soon abandoned because the

amount of pure lead ore thus obtained was too small to pay for such close work. Besides, the percentage of lead in the ore was reduced only 0.5 per cent.

(c) *Iron Pyrites*. — Either intermixed with the zinc blende or intersecting the same in narrow streaks. It contains about \$12 gold per ton, but constitutes only a comparatively small percentage of the ore.

(d) *Copper Pyrites*. — Occurs seldom and then only in very small quantities.

(e) *Native Silver*. — Now and then specimens with metallic silver in flakes or wire are found.

(f) *Gangue*. — The minerals forming the gangue are quartz and calcespar.

The ore looks like solid zinc blende; it is heavy and solid, showing but very little gangue. In the following table two analyses are given. Each one represents the average of large lots of ore. The one is unassorted, just as it was extracted from the mine, while from the other the galena and gangue were sorted.

ANALYSIS OF SAN FRANCISCO DEL ORO ORE

| | UNASSORTED ORE | ASSORTED ORE |
|------------------------|-------------------|-----------------|
| Zinc..... | 24.08 | 25.50 |
| Lead..... | 11.92 | 11.56 |
| Iron..... | 7.00 | 6.50 |
| Manganese..... | 0.70 | 0.53 |
| Cadmium..... | 0.16 | 0.30 |
| Antimony..... | 0.50 | 0.52 |
| Copper..... | 0.72 | 1.02 |
| Alumina..... | 1.30 | 3.65 |
| Calcium carbonate..... | 9.82 | 8.00 |
| Sulphur..... | 21.35 | 21.01 |
| Nickel..... | 0.20 | |
| Silver..... | 0.10 | 0.12 |
| Gold..... | trace | trace |
| Soluble silica..... | 0.92 | |
| Insoluble gangue..... | 21.32 | 19.41 |
| Total..... | 100.09 | 98.12 |

Near the surface the ore contained more free galena and less zinc blende; and it is said that the first owners made quite a financial success by smelting the ore in Mexican furnaces. However, when the character of the ore changed, the mine changed

hands, and all subsequent attempts to work the ore proved a failure. Though the ore was offered much cheaper than other ore to the custom mills of Parral, only small quantities were bought, being merely used as flux for the oxidized ores of the Veta Grande and other mines of the district to facilitate chloridizing roasting. Many attempts, however, were made to work the ore by itself, but without success. Even smelting, which ought to have been out of the question, was tried. The main difficulty was found to be in roasting. The ore caked very readily and the silver could not be chloridized, at least not above 17 or 20 per cent. After many unsuccessful attempts, further trials were abandoned until an English company purchased the property.

The exceedingly refractory character of the ore and the very discouraging experience of others induced the managing director of the English company to have elaborate experiments made before erecting a mill near the mine, the new mill to be constructed in conformity with the observations and experience derived from the experiments. For this purpose the Bosque mill at Parral, an old 25-stamp lixiviating mill, was purchased, and I was commissioned to conduct the experiments.

The Mill. — It consisted of 25 stamps, rock-breaker and self-feeders, one large-size Stetefeldt furnace, claimed to roast 60 tons per day, one brick-lined revolving cylinder furnace of the White-Howell type, 24 ft. long and 4 ft. in diameter, and two leaching plants, one of 11 the other of 10 leaching vats, of various sizes, averaging about 10 ft. diameter by 3 ft. 6 in. depth, which, for want of grade, were almost buried in the ground. The whole arrangement of the mill was ridiculously inconvenient, causing a never-ending handling of the ore, supplies and products. For experimental purposes, however, the mill was good enough, especially as I had the privilege of erecting any other roasting furnace which I should consider advisable to experiment with.

ROASTING EXPERIMENTS

Theory. — As shown by the analysis, the ore consisted principally of the sulphides of zinc, lead and iron. The other sulphureted minerals occurred in such small quantities that it was not necessary to pay any attention to the chemical part they took in the process of chloridizing roasting.

Zinc blende, if subjected to the oxidizing action of the air, is converted into zinc oxide and zinc sulphate, while sulphurous acid escapes. In presence of salt, zinc sulphate remains indifferent, and does not decompose the salt, at least not at the temperature used in chloridizing roasting. If pure zinc blende, finely pulverized, and mixed with salt, is placed on a roasting dish, and roasted in the muffle, no chlorine gas can be detected, even if exposed to a bright red heat. Even freshly prepared zinc sulphate mixed with salt and exposed to the heat of the muffle does not produce any chlorine gas. Zinc blende, therefore, does not take an active part in producing chlorine during chloridizing roasting, at least not enough to be of practical value. In the roasted ore we find, therefore, the zinc mostly as oxide and sulphate and so also in the flue-dust.

If galena is subjected to a chloridizing roasting, especially in presence of sufficient air, most of the lead is converted into a sulphate, which, as such, like the zinc sulphate, does not react on the salt, and therefore does not generate chlorine.

Iron pyrites is converted into ferric oxide and into ferrous and ferric sulphates, both of which react very energetically on salt, and generate chlorine.

We have, therefore, two non-generators and only one generator of chlorine in the ore. The non-generators of chlorine, galena and zinc blende, however, contain all the silver, while the iron pyrites carries only some gold, but no silver. This is a very important point to take into consideration. The next important point is the fact that on account of the great density of the zinc blende it requires a long time to oxidize. Likewise the galena requires long roasting at a low heat, while iron pyrites decomposes quickly, and in presence of salt generates chlorine at a period of the roasting when neither the zinc blende nor the galena are sufficiently oxidized to yield their silver to the action of the chlorine. If, therefore, the salt is mixed with the ore in the stamp-battery, the chlorine produced by the reaction of ferric sulphate and salt is lost, and a very imperfect chlorination of the silver takes place, no matter how long roasting may be continued and how much salt may be used. For instance, when the ore was roasted with 12 per cent. of salt in the Stetefeldt furnace, the roasted ore contained but 1.38 per cent. of chlorine, of which 0.8 per cent. was combined with sodium, which represents 1.3 per

cent. undecomposed sodium chloride. The salt used in roasting contained but 78.25 per cent. of sodium chloride, and the 12 per cent. of salt represents, therefore, 9.39 per cent. Not taking into consideration the loss in weight which the ore sustained in roasting, we can assume that 8.09 per cent. sodium chloride was decomposed or volatilized, while not more than 15 per cent. of the silver was chloridized, showing that the silver was not yet in a proper condition to be acted upon by the chlorine at the time when the reaction took place between the iron sulphate and the salt. Only a very small percentage of chlorine was found to have combined with other bases. The chlorine was practically an entire loss. Similar observations were made by roasting in other furnaces.

In roasting an ore like the San Francisco del Oro ore, it is therefore of the greatest importance to add the salt afterward and not to mix it with the ore in the stamp-battery. But this is not the only condition to be observed. We have to take into consideration that in this case we are relying on the sulphates of iron to generate chlorine, and that these sulphates easily decompose, forming oxides. If, therefore, the oxidizing period is continued until the zinc blende and galena are well oxidized, which takes a long time, we will have no iron sulphate left to decompose the salt, and in consequence will have a very badly chloridized ore. At a high temperature these iron salts decompose more quickly, giving off their sulphuric acid, and in order to retain them as long as possible the ore has to be roasted at a low temperature. To know the proper time when the salt is to be added is of the greatest importance; this knowledge, however, can be obtained only by repeated tests and very close observation. The most suitable time to add the salt for the San Francisco del Oro ore was found to be when the black color of the ore turns brown but still shows black particles. If at that time the salt is added, a distinct odor of chlorine can be observed, which lasts during the whole of the finishing period; while, if the salt is added too soon or too late, no chlorine evolves from the ore during this period.

The best results with this ore could undoubtedly be obtained by subjecting it first to a dead oxidizing roasting, then adding a mixture of green vitriol and salt. But the ore is not rich enough to permit such an expense.

While the incapacity of the zinc and lead sulphates to de-

compose the salt makes the process of roasting difficult and complicated, it offers, on the other hand, the advantage that it reduces materially the consumption of salt. An addition of four or five per cent. of salt gives the same result as eight or ten per cent. If the roasting is very carefully conducted even three per cent. gives good results.

Remarks. — The most difficult and at the same time the most important process in the treatment of ores by wet methods is undoubtedly the chloridizing roasting. It is always the safest plan for the operator to roast as thoroughly as possible. If the silver is well chloridized, the sodium hyposulphite will extract all the silver chloride and frequently will leave the tailings even poorer than indicated by the chlorination test, without the use of additional solutions or chemicals, thus saving time and expense, and not complicating the process. A high chlorination does not necessarily involve a high loss by volatilization. I have recorded instances in which the volatilization of silver was greater in imperfectly chloridized charges than in well-chloridized ores (see Chapter IV, "Loss of Silver by Volatilization"). Solutions, by which we can correct a badly roasted charge, like chloride of copper applied during base-metal leaching, or by which part of the unchloridized silver can be extracted, like potassium cyanide, or, as in some instances, Russell's extra solution, are very useful and acceptable; but to neglect roasting and to rely for closer extraction on these solutions is a rather dangerous practice.

Being convinced that the successful working of the San Francisco del Oro ore depended on a successful roasting, and knowing the great difficulty which the nature of this ore offered to chloridizing roasting, particular attention was paid to this process, and careful studies were made of the peculiarities of the ore. The principal points to ascertain were: first, the mode of treatment which the ore required with regard to temperature, roasting time, draft, and the proper time for adding the salt; and, second, which of the furnaces would comply best with the requirements and at the same time perform the work the cheapest.

ROASTING IN THE STETEFELDT FURNACE

The furnace was a large-size Stetefeldt furnace, claimed to roast 60 tons per day, and was built according to the best improved design.

Though quite elaborate experiments were made, it was not possible to obtain good results; in fact, they were far from being satisfactory. Nevertheless they are interesting and, at the same time, useful, inasmuch as they establish the fact that an ore like the Del Oro, containing 25.5 per cent. zinc, 11.56 per cent. lead, and 21 per cent. sulphur, is by far too refractory for the Stetefeldt furnace. Such an ore requires to be submitted to a long and gradually increasing temperature before the salt is added. The principle of the Stetefeldt furnace, however, is just the reverse of this important condition, and the results, as a matter of course, could not be satisfactory.

Notwithstanding that there was a very powerful draft through the furnace, and that the ore was crushed through 40-mesh screen, only about three-eighths of the ore (by volume) was carried by the draft into the descending flue and deposited at the bottom of the chambers, while about five-eighths of it came down the shaft. Actual dust was not carried beyond the second dust-chamber, and even there only small quantities deposited, owing to the great specific gravity of the ore.

After heating the furnace gradually for three days, charging was begun. There were 20 stamps running, which crushed through 40-mesh screen from 20 to 22 tons per twenty-four hours, according to the amount of salt used. The experiments were begun with an addition of 8 per cent. salt, increasing the amount during the time of the experiments to 12 and finally to 16 per cent. It was also tried to roast oxidizingly and to add the salt at intervals to the ore at the bottom of the shaft, but without success. The furnace was kept running for seven days and was then stopped, as no signs of improvement in the work could be noticed and the cooling floor was filled with badly roasted ore.

Before making any comments, the average results obtained under different conditions are given:

ROASTING WITH 8 PER CENT. SALT

| | SHAFT OZ. PER TON | DESCENDING FLUE OZ. PER TON |
|-------------------------------------------|----------------------|--------------------------------|
| Average of raw ore including salt | 32.37 | 32.37 |
| Average of roasted ore | 34.41 | 29.67 |
| Average of leach tailings | 28.62 | 23.72 |
| Average of chlorination | 16.90 per cent. | 20.20 per cent. |

ROASTING WITH 12 PER CENT. SALT

| | SHAFT OZ. PER TON | DESCENDING FLUE OZ. PER TON |
|---------------------------------------|----------------------|--------------------------------|
| Average of raw ore including salt.... | 31.00 | 31.00 |
| Average of roasted ore..... | 31.33 | 27.50 |
| Average of leach tailings..... | 26.60 | 10.78 |
| Average of chlorination..... | 15.20 per cent. | 60.80 per cent. |

ROASTING WITH 16 PER CENT. SALT

| | SHAFT OZ. PER TON | DESCENDING FLUE OZ. PER TON |
|---------------------------------------|----------------------|--------------------------------|
| Average of raw ore including salt.... | 30.32 | 30.32 |
| Average of roasted ore..... | 33.39 | 29.81 |
| Average of leach tailings..... | 28.35 | 22.30 |
| Average of chlorination..... | 15.20 per cent. | 25.20 per cent. |

An experiment was also made merely to oxidize the ore without adding any salt:

OXIDIZING ROASTING

| | SHAFT OZ. PER TON | DESCENDING FLUE OZ. PER TON |
|------------------------------------|----------------------|--------------------------------|
| Average of roasted ore..... | 34.91 | 28.57 |
| Average of leach tailings..... | 32.65 | 17.41 |
| Average of extractable silver..... | 6.50 per cent. | 39.20 per cent. |

The oxidized ore from the descending flue was charged into a tank, treated at first with a diluted solution of cupric chloride, then leached with water and afterward with sodium hyposulphite, by which tailings were obtained containing 14.43 oz. per ton, showing an extraction of 49.5 per cent. silver.

REROASTING THE ORE FROM THE SHAFT

The ore which had passed through the Stetefeldt furnace once was sifted, to free it from lumps, and charged a second time.

| | SHAFT OZ. PER TON | DESCENDING FLUE OZ. PER TON |
|-------------------------------------|----------------------|--------------------------------|
| Average of roasted ore | 31.49 | 31.49 |
| Average of leach tailings | 29.60 | 24.79 |
| Average of chlorination | 9.40 per cent. | 22.30 per cent. |

Observations and Comments.—If we compare the above results we find that those obtained in the shaft were pretty nearly equally bad, whether more or less salt was used. In the descending flue 12 per cent. of salt gave the best result (60.8 per cent. chlorination). An excess of salt lowered the chlorination. The same observation was made afterward in the Howell and reverberatory furnaces.

The ore from the shaft has a very dark, almost black, color, and emits volumes of sulphurous acid gas when discharged, but no chlorine. By leaving the ore in a pile on the cooling floor it continues to emit sulphurous acid gas for a couple of days, without a marked increase in the percentage of chloridized silver taking place. In dropping through the shaft the main portion of the ore is transformed into minute globules, which show that, while the ore falls, it is partially slagged. Trying to avoid this, the fire was lowered so much that the lower part of the shaft was quite dark, while on the auxiliary grate the fire was allowed to go out entirely. This, however, did not produce any change; the ore came down now as before in globules, while in the descending flue the temperature continued to be very high. It is apparent that the ore when sifted into the shaft creates by the sudden combustion of the sulphurets an extremely high temperature in the upper regions of the shaft, which causes the suspended ore particles to melt and slag to globules. Also that under such circumstances silicates are formed, which, incrusting the ore particles, prevent their further oxidation and chlorination.

An analysis showed that the roasted ore still contained 8 per cent. of unoxidized sulphur. Ores containing not more than 8 per cent. sulphur usually roast well in a Stetefeldt furnace, and it was expected that by charging the ore a second time good results might be obtained. The ore was sifted, to free it from lumps, and charged again. The results, however, proved to be worse than those obtained in the first roasting. The chlorination

of 15 and 16 per cent. was reduced to 9.4 per cent. The ore maintained its dark color and continued to emit heavy fumes of sulphurous acid. It is not probable that part of the silver chloride of the first roasting was decomposed by passing a second time through the furnace; it is more likely that the dropping of the chlorination in the shaft was caused by mechanical separation. Part of the lighter ore particles which contained less lead and were better chloridized in the first roasting were carried over into the flue during the second roasting, thus seemingly reducing the original percentage of chlorination. Besides part of the silver chloride may have slagged during the second roasting.

There are, however, other circumstances which also act disadvantageously. For instance, it was found after the first roasting that the ore in the shaft contained 8.48 per cent. unoxidized sulphur and 11.19 per cent. lead, while that from the descending flue contained only 0.51 per cent. unoxidized sulphur and 3.11 per cent. lead. This shows that a separation takes place, the shaft receiving the main portion of the lead, while the lighter minerals, among them the iron pyrites, are carried over into the descending flue. The better oxidation and chlorination, as well as the higher temperature in the flue, are principally due to this separation. But as the main bulk of the ore drops through the shaft such a separation is disadvantageous.

Another interesting fact has to be recorded. Using so much salt, and obtaining such an imperfect oxidation and chlorination, we should naturally expect to find most of the salt undecomposed in the ore. This, however, is not the case. In the shaft the ore contained only 1.38 per cent. chlorine, of which 0.8 per cent. was combined with sodium, representing only 1.30 per cent. undecomposed sodium chloride. The material from the first dust-chamber contained 0.38 per cent. of chlorine, of which 0.16 per cent. was combined with sodium representing only 0.27 per cent. sodium chloride, while the material from the descending flue contained but 0.2 per cent. chlorine. The fine white dust from the last dust-chamber contained 0.42 per cent. chlorine and a great deal of sulphuric acid. The salt, therefore, was decomposed, and the chlorine, either as hydrochloric acid or as chlorine, escaped as gas without effect. If it were volatilized we ought to have found more of it in the last dust-chamber.

Another bad feature is the formation of lumps. In the upper

region of the shaft, the ore-dust, wherever it comes in contact with the hot walls, sticks to and incrusts them. This crust peels off and drops down in pieces. It is almost raw, and some of the larger pieces, when broken, show the texture of matte. They form in large quantities. When the ore was sifted for reroasting, there was not less than 25 per cent. of the whole ore in the shape of hard lumps. These lumps contained only 21.87 oz. silver per ton, and 0.82 per cent. chlorine, of which 0.28 per cent. was combined with sodium, equal to 0.46 per cent. sodium chloride.

Besides the chemical difficulties, a very annoying mechanical difficulty was encountered. The ore-dust in the same way as it incrusts the walls of the shaft also incrusts the lower side of the screen of the feeding machine, and thus stopped up the holes, which necessitated a frequent changing of the screen, as often as twice a day.

Heavily sulphureted ores, especially if they carry zinc, require more draft in roasting than less sulphureted ores. This is especially the case with the Stetefeldt furnace, where the ore is exposed for such a short time to the action of the air and heat. In compliance with this theory all the air-doors with which the furnace is provided were opened, but the result did not improve.

REROASTING THE ORE OF THE STETEFELDT FURNACE IN THE MODIFIED HOWELL FURNACE

The partly roasted ore from the Stetefeldt furnace, after sifting, was fed into the Howell furnace. Having previously ascertained that the ore contained only 1.3 per cent. of salt, 6 per cent. and sometimes 8 per cent. more salt were added. The rate of feeding was changed from time to time in order to test the working capacity of the cylinder for this ore; thus the rate varied from 5 to 9 tons per twenty-four hours. The following results are the averages of 33 charges:

| | |
|--------------------------------|---------------------------|
| Average of roasted ore..... | 31.42 oz. silver per ton. |
| Average of leach tailings..... | 17.55 oz. silver per ton. |
| Average of chlorination..... | 44.20 per cent. |

The consumption of wood in reroasting proved to be much greater than in roasting the raw ore. The main portion of the sulphur, especially that of the pyrites, which easily ignites, hav-

ing been burnt off in the Stetefeldt furnace, the ore did not create any heat by itself, and all the required heat had to be furnished. To reroast 8.3 tons it took 26 cargas of wood (12 cargas = 1 cord) = 0.27 cord per ton, while it took only 16 cargas to roast 10 or 11 tons of raw ore, which is equivalent to 0.13 cord per ton of ore. The roasting capacity of the furnace was not increased by roasting this material, which contained only about 8 per cent. sulphur, but on the contrary was diminished as compared with the raw ore containing over 21 per cent. sulphur.

The reroasted ore was of a red-brown color, smelled of chlorine, and did not emit any sulphurous acid gas, but it still consisted principally of little globules. Quite a large portion of the globules remained black, no matter how long the ore was kept in the furnace. Some of them were magnetic, but the great majority were not. Between the fingers the reroasted pulp felt sharp, like pulverized glass. The temperature was kept at a proper degree, and, the dust-chambers and furnace having been previously cleaned, there was an abundant draft, still it was not possible to obtain more than 44.2 per cent. chlorination. The cause of this failure was undoubtedly the silicates which were formed during the roasting in the Stetefeldt furnace.

APPLICATION OF STEAM

A jet of steam was introduced into the reverberatory of the modified Howell. The hydrochloric acid which was formed by the action of the steam was of decidedly beneficial influence, and considerably improved the result; still the result did not give entire satisfaction. The same percentage of salt was used and the same temperature maintained as in the foregoing experiments, and the improved results are therefore exclusively due to the action of the hydrochloric acid on the silicates. The following figures are the average of thirteen charges:

| | |
|---------------------------------|--------------------|
| Average of reroasted ore | 31.00 oz. per ton. |
| Average of leach tailings | 10.37 oz. per ton. |
| Average of chlorination..... | 66.60 per cent. |

The ore still contained a considerable amount of these little globules, but they had changed their color to red brown, and between the fingers the ore felt soft and not so sharp and glassy as when roasted without steam.

CONCLUSIONS

The experiments in roasting the argentiferous zinc blende and galena ore of the San Francisco del Oro mine in a Stetefeldt furnace have shown:

(1) An incomplete oxidation of the sulphureted minerals, the main portion of the ore still containing 8.48 per cent. unoxidized sulphur when roasted with salt, and 7.6 per cent. when roasted without salt.

(2) An insufficient chlorination of the silver. The highest chlorination in the shaft was only 16.9 per cent., and as 62.5 per cent. of the whole volume of the ore dropped into the shaft, the somewhat higher chlorination obtained in the descending flue could not much improve the average chlorination.

(3) That the principle of the Stetefeldt furnace is contrary to the conditions, of which the maintenance is so essential to roasting ores containing much zinc blende and galena. Instead of permitting the ore to be subjected for a longer time at a low but gradually increasing temperature, the ore, entering the furnace, encounters immediately the highest temperature, which is detrimental to the roasting of such ores.

(4) That on account of the sudden exposure of the raw ore particles to such a high temperature they melt to minute globules, which makes the ore unfit for further treatment.

(5) That a concentration of the lead minerals takes place in the shaft, which is disadvantageous.

(6) That about 25 per cent. of the ore when passing through the furnace is changed into hard lumps of almost raw ore, and that the construction of the furnace does not offer any means to prevent it.

(7) That the lower side of the feeding screen becomes rapidly incrustated and the holes obstructed, requiring a too frequent exchange of screens.

These observations taken together prove beyond doubt that the Stetefeldt furnace is not suitable for the San Francisco del Oro ore, and consequently for no ores heavily charged with zinc blende and galena.

ROASTING IN THE WHITE-HOWELL FURNACE

The furnace which was used was not a regular Howell. It consisted of a revolving cylinder of uniform diameter, the shell being made of boiler iron and lined with bricks the whole length. The principle on which it works, however, is identical with that of the Howell, and this name is used here merely to indicate the type of furnace.

The Howell, like the Stetefeldt furnace, requires the salt to be added to the ore before entering the furnace. With some ores this is immaterial, but it is a point of the greatest importance for the Del Oro ore. If the salt is previously added the ore becomes sticky, incrusts the furnace rapidly, and when it leaves the furnace consists mostly of lumps, and what is still worse, without being chloridized. If the ore is charged without salt it remains dry and sandy, but a very annoying separation takes place. The fine particles are carried by the draft into the dust-chambers, and only the coarse sand passes through the furnace, without being sufficiently desulphurized. If, then, salt is added in the drop-pit, only a small percentage of the silver becomes chloridized; the best results gave only 29 per cent. chlorination.

In order to diminish the separation, two per cent. of salt was added to the ore in the battery, while the balance of the salt was added in the drop-pit. This small percentage of salt made the ore sticky enough to diminish considerably the dusting, without causing the formation of lumps or too heavy an incrustation of the furnace.

By this mode of roasting the chlorination improved considerably, the average of three days' run being 67 per cent. It was soon evident, however, that the Howell furnace, as such, could not roast the Del Oro ore. The results were not uniform and reliable, being sometimes high, sometimes low; and notwithstanding the greatest care the average could not be brought above 67 per cent. But the roasted ore was in a good condition; it was unfinished but not spoiled, as in the Stetefeldt furnace, and there was reason to expect that, by an alteration which would give the ore more roasting time and allow a better regulation of the temperature, good results would be obtainable.

ROASTING IN THE MODIFIED HOWELL FURNACE

In front of the furnace I constructed a shallow drop-pit and a fireplace, the long side of the fireplace being opposite the discharge of the furnace, so that the flame before entering the furnace had to traverse the drop-pit. To one side of the drop-pit, and communicating with it, there was attached a small reverberatory furnace 6 x 8 ft., the bottom of both being on the same level. The reverberatory contained one working door and a 24-in. fireplace. When enough ore had accumulated in the pit to make a charge for the reverberatory, it was pushed by means of a hoe into the reverberatory. Each charge consisted of about 1400 lb. While starting the furnace a strong fire was kept in both fireplaces, but after the process was in operation the fire in front of the cylinder was much lowered; in fact, so much so that half the grate-bars remained bare of wood. Only now and then a thin stick of wood was added, just enough to prevent the drop-pit from getting chilled. Two per cent. of salt was added to the ore in the battery.

If the roasting is properly conducted, the blue flame of the ignited pyrites can be observed in the back part of the cylinder. Next to it and reaching beyond the middle of the cylinder the ore assumes a higher temperature, forming a belt of bright-red heat. In this region the principal oxidation takes place, and the increase in temperature is caused by the oxidation and not by its position nearer to the fire. The part of the furnace next to the fire and nearly one-third of the whole length ought to look dark. The furnace is mostly heated by the combustion of the sulphides, and receives but little supply from the fireplace and from the reverberatory. In fact, the ore while in the cylinder should be left as much as possible to roast in its own heat. This is a very important condition to maintain. The object is to convert as much as possible of the galena and zinc blende into sulphates and oxides before generating chlorine, and to avoid until then as much as possible the decomposition of the iron salts. This can only be done by maintaining a low heat after the combustion of the pyrites. An excess of heat is invariably connected with an excessive loss of silver by volatilization and by a low chlorination. Galena and zinc blende roast quicker and better in a low than in

a high heat. When the ore leaves the cylinder and drops into the pit it should be of a very dull red heat, while the color after cooling should be dark yellow-brown.

If the temperature is so kept, neither the odor of chlorine nor much of sulphurous acid can be detected. At an increased heat, sulphurous acid emits again strongly, showing that the oxidation is not yet completed. As the temperature in the cylinder is mostly produced by the combustion of the sulphureted material, the main means of regulating the same is the feed. If too much ore enters the furnace the belt of bright-red heat increases, advancing more and more toward the front, and finally the whole furnace assumes this temperature. The ore dropping into the pit is very hot, emits heavy fumes, and overheats the pit. If then removed into the reverberatory, it takes a very long time to be finished, necessitating an interruption in the feed of the cylinder. On the other hand, if insufficient ore is charged, the belt of bright-red heat gets smaller and moves toward the back end of the furnace.

When the properly prepared ore enters the reverberatory furnace, the salt is added and the temperature is somewhat increased. It commences to fume, and swells, without forming more lumps than an ordinary ore. In the beginning strong fumes of sulphurous acid emit, but soon cease, and chlorine appears. The charge is finished if the fumes assume a mild and sweetish smell of chlorine; as long as they smell strong, roasting has to be continued.

A series of experiments was made to ascertain the smallest amount of salt practicable, and it was found that 4, 6, 8, and 10 per cent. give about equal results. Twelve per cent. begins to make the ore too sticky and produces less chlorination; 3 per cent. is sufficient if the roasting is very carefully conducted, but then only 1 per cent. has to be added in the battery and 2 per cent. in the furnace. Four per cent., however, is safer, as then the result does not depend so much on the skill and good-will of the laborers.

The roasting capacity of the furnace proved to be much less for this ore than for ordinary ore. Not more than 8½ tons could be roasted in twenty-four hours. It is true the cylinder was only 24 ft. long, but even with a 32-ft. cylinder it cannot be expected to roast more than 12 tons per day. Each charge had to remain two hours on the reverberatory hearth. Though the ore was

roasted in the reverberatory at a somewhat increased heat, yet the temperature could not be increased beyond dull red without losing too much silver by volatilization.

ADDITIONAL CHLORINATION AFTER THE ORE HAS LEFT THE FURNACE

Some ores gain much in chlorination if left hot in a pile for some time. This is mostly the case when an ore is insufficiently roasted, or when the nature of the ore is such as to require a long roasting time at a low heat. Another additional chlorination can be produced by moistening the ore and leaving it for several hours in a pile. This is usually the case if the ore contains copper. Roasted ore containing caustic lime should not be left moist on the cooling floor. The most important additional chlorination, however, takes place, according to numerous observations of mine, during base-metal leaching. The roasted Del Oro ore either contained much caustic lime — which, however, is hardly possible, as the raw ore is so rich in sulphur — or some other chemical substance which acted decomposingly on the silver chloride, because it could not be moistened on the cooling floor without sustaining quite a loss in chlorination. To prevent this, the vats were charged about one-third with water and the hot ore dumped into it, thus producing a hot base-metal solution. The observation was made that by this practice not only was the decomposition of the silver chloride avoided, but that a considerable increase in the silver chlorination took place, in some instances as much as 12.9 per cent. (see following table, charge No. 11).

If the original chlorination, however, was 75 per cent. or more, this additional chlorination amounted to much less. By adding some cupric chloride to the water in the vat before dumping the ore, I found that badly roasted charges gained in chlorination as much as 34 to 38 per cent. (see table, charges Nos. 9, 15 and 16).

These are very important observations, and give the operator the means of correcting badly roasted charges.

RESULTS

The table on page 116 is a record of the results obtained in roasting the San Francisco del Oro ore in the modified Howell furnace. It represents a two weeks' run. As each tank charge contained

WORKING RESULTS OBTAINED IN ROASTING EXPERIMENTS WITH THE MODIFIED HOWELL FURNACE

| DATE | PER CENT. OF SALT USED | VALUE OF RAW ORE | | VALUE OF RAW ORE CONTAINING SALT | NO. OF TANK CHARGE | VALUE OF ROASTED ORE | | VALUE AFTER LEACHING IN THE LABORATORY | | PER CENT. OF CHLORINATION WHEN THE ORE LEFT THE FURNACE | VALUE OF VAT TAILINGS | | PER CENT. OF ACTUAL CHLORINATION CALCULATED FROM THE VAT TAILINGS | LOSS IN WEIGHT THE ORE SUSTAINED DURING ROASTING | LOSS OF SILVER BY VOLATILIZATION | LOSS OF SILVER BY TAILINGS | PER CENT. OF CHLORINATION GAINED IN THE VAT | PER CENT. OF ACTUAL SILVER EXTRACTION | NUMBER OF OUNCES SILVER EXTRACTED | REMARKS | |
|-------|------------------------|------------------|---------|----------------------------------|--------------------|----------------------|---------|----------------------------------------|---------|---------------------------------------------------------|-----------------------|---------|-------------------------------------------------------------------|--------------------------------------------------|----------------------------------|----------------------------|---------------------------------------------|---------------------------------------|-----------------------------------|---------|---------------------------------------------------------|
| | | OZ. PER TON | PER TON | | | OZ. PER TON | PER TON | OZ. PER TON | PER TON | | OZ. PER TON | PER TON | | | | | | | | | PER CENT. |
| APRIL | | | | | | | | | | | | | | | | | | | | | |
| 20 | 5½ | 34.21 | 32.33 | 32.33 | 9 | 31.34 | 11.66 | 37.2 | 7.75 | 75.3 | 2½ | 5.2 | 24.5 | 38.1 | 70.3 | 24.04 | | | | * | Assorted ore. |
| 21 | 6 | 34.50 | 32.43 | 32.43 | 10 | 30.91 | 9.04 | 71.8 | 7.26 | 76.6 | 2½ | 7.1 | 21.8 | 4.8 | 71.1 | 24.52 | | | | | |
| 23 | 5 | 35.09 | 33.34 | 33.34 | 11 | 31.49 | 8.89 | 71.8 | 4.33 | 84.7 | 2½ | 7.8 | 14.1 | 12.9 | 78.1 | 27.40 | | | | | |
| 24 | 5 | 33.76 | 32.07 | 32.07 | 12 | 29.74 | 8.16 | 72.6 | 3.29 | 89.0 | 2½ | 9.5 | 10.0 | 16.4 | 80.5 | 27.17 | | | | | |
| 25 | 5 | 31.67 | 30.09 | 30.09 | 13 | 29.59 | 6.99 | 76.4 | 3.57 | 88.0 | 2½ | 4.1 | 11.5 | 11.6 | 84.4 | 26.72 | | | | | |
| 26 | 5 | 29.09 | 28.64 | 28.64 | 14 | 29.16 | 7.58 | 74.0 | 4.25 | 85.5 | 3½ | 1.7 | 14.3 | 11.5 | 84.4 | 24.55 | | | | | |
| 27 | 4½ | 27.35 | 26.02 | 26.02 | 15 | 25.95 | 13.27 | 51.9 | 4.25 | 84.7 | 3½ | 3.7 | 15.7 | 32.8 | 80.6 | 22.04 | | | | | |
| 28 | 4 | 28.32 | 27.19 | 27.19 | 16 | 24.49 | 12.89 | 47.4 | 4.48 | 81.8 | 3½ | 13.0 | 15.8 | 34.4 | 71.2 | 20.16 | | | | * | Ore slightly assorted. |
| 29 | 4 | 26.02 | 24.98 | 24.98 | 17 | 22.16 | 4.95 | 77.7 | 4.76 | 78.7 | 3½ | 14.2 | 19.0 | 1.0 | 66.8 | 17.38 | | | | | |
| 30 | 5 | 25.43 | 24.42 | 24.42 | 18 | 22.74 | 5.24 | 77.0 | 4.48 | 80.3 | 3½ | 10.1 | 18.3 | 3.3 | 71.6 | 18.29 | | | | | |
| MAY | | | | | | | | | | | | | | | | | | | | | |
| 1 | 5 | 24.37 | 23.15 | 23.15 | 19 | 21.57 | 4.66 | 78.4 | 4.84 | 77.6 | 3½ | 10.0 | 20.0 | 0.8 | 70.0 | 17.05 | | | | | Unassorted ore, just as it was broken down in the mine. |
| 2 | 5 | 23.94 | 21.97 | 21.97 | 20 | 22.45 | 5.39 | 76.0 | 3.87 | 82.8 | 3½ | 1.3 | 17.0 | 6.8 | 81.3 | 19.46 | | | | | |
| 3 | 4 | 24.28 | 23.31 | 23.31 | 21 | 22.01 | 5.83 | 73.6 | 4.84 | 78.1 | 3½ | 8.7 | 20.0 | 4.5 | 71.3 | 17.31 | | | | | |
| 4 | 4 | 26.02 | 24.98 | 24.98 | 22 | 21.87 | 6.12 | 72.1 | 4.30 | 80.4 | .. | 15.5 | 16.6 | 8.3 | 67.9 | 17.66 | | | | | |
| | 4.7 | 28.85 | 27.44 | 27.44 | | 26.10 | 7.90 | 68.4 | 4.76 | 81.6 | 3 | 7.9 | 17.0 | 13.3 | 74.9 | 21.69 | | | | | Averages. |

*Charges thus marked were treated with cupric chloride during base metal leaching.

the whole ore of twenty-four hours' roasting, it offered a good opportunity to follow each charge from the raw ore down to the tailings, and to ascertain for each charge the loss by volatilization, gain by additional chlorination, etc.

Taking the averages of the results, we find the silver chlorination when the ore left the furnace, 68.4 per cent.; additional chlorination, 13.3 per cent. or a total chlorination of 81.7 per cent. The low average chlorination of the ore when leaving the furnace was caused by the three badly roasted charges — 9, 15 and 16. The other eleven charges gave an average of about 75 per cent.

The total or actual chlorination of 81.7 per cent. may seem to be low, but if we consider that the ore is of low grade, averaging only 28.8 oz. per ton, and that 1 per cent. represents only 0.28 of an ounce silver; and also consider that the ore contains about 37 per cent. zinc blende and 13 to 19½ per cent. galena, which carry all the silver, and that the ore was pronounced as being too refractory for chloridizing roasting, we have to count the work done by the modified Howell furnace as very satisfactory, especially as such a chlorination secured the success of the enterprise owing to the cheapness of mining and reduction.

LOSS OF SILVER BY VOLATILIZATION

The loss of silver by volatilization in these experiments was ascertained by the method described in a previous chapter.

Owing to the fact that a great portion of the lead and zinc sulphides is converted into sulphates, the San Francisco del Oro ore loses but a small percentage of its weight during roasting. The tests showed a loss of 2½ and 3½ per cent.

With these figures, and the assay value of the raw and roasted ore, the loss of silver by volatilization was calculated. The extremes were 1.3 per cent. and 15.5 per cent. while the average gave 7.9 per cent. The figures contained in the corresponding column of the table illustrate how variable this loss is, and what a severe loss of silver can be caused by even slight oversights. I found the Del Oro ore to be more sensitively disposed for such loss than many others, even antimonial ores which I had treated before. The least increase of the temperature above dull red causes a heavy loss, even if this increase of the temperature lasts

only a very short time. Thus two or three thin sticks of wood, if thrown on the fire before needed, may materially increase the loss.

The loss by volatilization is not in direct proportion to the per cent. of chloridized silver. Frequently a well chloridized ore suffers much less loss than a badly chloridized one. Referring to the table we find, for instance, in charge No. 16 the silver was chloridized only to 47.2 per cent. while the loss by volatilization was as high as 13 per cent. Again, in charge No. 20, 76 per cent. of the silver was chloridized, while the loss by volatilization amounted to only 1.3 per cent. We find the same in charges Nos. 12, 14, etc.

THE ROASTED ORE

The roasted ore contains only a small percentage of lumps. These are not hard, but porous, and fall to powder if kept in contact with water for some time. If the ore is left dry in a pile it hardens. If left undisturbed for a week or two it becomes so hard that it requires the use of a pick to loosen it. In water, however, it softens easily again. The color is usually red-brown, but occasionally, if there is less iron pyrites in the ore, it is yellow-brown.

The analysis of the unsorted ore after roasting (see above analysis of the raw ore) is here given. The ore was roasted with 5 per cent. of salt.

ANALYSIS OF UNASSORTED ROASTED ORE

| | |
|-----------------------------------|--------|
| Gold..... | trace. |
| Silver..... | 0.09 |
| Lead..... | 9.00 |
| Iron..... | 6.00 |
| Zinc..... | 22.45 |
| Lime (calculated as caustic)..... | 5.65 |
| Antimony..... | 0.75 |
| Copper..... | 0.60 |
| Cadmium..... | 0.10 |
| Alumina..... | 3.09 |
| Soda (calculated as caustic)..... | 3.79 |
| Sulphuric acid..... | 13.16 |
| Chlorine..... | 0.88 |
| Soluble silica..... | 8.00 |
| Insoluble gangue..... | 18.61 |
| Oxygen of the oxides..... | ? |

This analysis shows that the heavy metals were principally converted into sulphates, and that only a small portion of them,

if any, can be present as chlorides. The 0.88 per cent. of chlorine may be due to undecomposed salt.

If the furnace crust is not removed from the furnace for some time, it changes its color. In some parts it is greenish white; in others, flesh-colored. It gets very hard, and when moistened with water generates heat and slacks like lime. I had never made this observation before, not even with the very calcareous ore of Las Yedras. It is to be regretted that it was overlooked to make an analysis of this crust. It is difficult to believe that this phenomenon could be caused by caustic lime, because the latter could not well exist in an atmosphere of sulphuric and sulphurous acid and of chlorine.

CONSUMPTION OF WOOD

Owing to the large quantity of sulphureted minerals in the ore, and the very low temperature at which the Del Oro ore has to be roasted, the consumption of wood is very small. After the furnace is heated and the cylinder incrustated, it takes hardly any fire in front of the cylinder to maintain the proper temperature. In the reverberatory addition a little more fire is needed, but much less than ordinary ores require.

During two weeks the wood was weighed (it was bought by weight) and the total consumption during this time was found to be 220 cargas of 300 lb. With this amount of wood 115.8 tons of ore were roasted, which gives 1.8 cargas per ton of ore. Twelve cargas of the Parral wood are equal to one cord, and if we express the consumption in cords we find that with one cord of wood the furnace roasted 6.3 tons of ore.

COST OF ROASTING IN THE MODIFIED HOWELL FURNACE

The cost of roasting 8½ tons per twenty-four hours was as follows:

| | |
|--------------------------------------------------|-------------------------------|
| Labor..... | \$6.60 |
| 4 per cent. salt, 680 lb. at 1.27¢..... | 8.63 |
| 15.7 cargas wood at 75¢..... | 11.77 |
| Steam power, 10 cargas wood at 75¢..... | 7.50 |
| Oil, light, tools, etc..... | 2.00 |
| Management, office, mechanic's assay office..... | 1.78 |
| | <u>\$38.28</u> ÷ 8.5 = \$4.50 |
| Cost per ton..... | \$4.50 Mexican currency. |

To ascertain the cost of steam power a separate boiler was used for the furnace. It is apparent that by using a boiler for only one small furnace the expense per ton of ore will be much greater than if with the same boiler and engine several large furnaces are operated. But this was the only way of getting an estimate. The steam for working the pumps and preparing the calcium sulphide was supplied by the same boiler, and had to be charged to roasting.

As the statement of cost is made only $8\frac{1}{2}$ tons per day, it would be misleading if the whole expenses for management, mechanic's assay office, etc., should be charged to the $8\frac{1}{2}$ tons, as those expenses will be about the same for 100 tons per day, the intended capacity of the new mill. The expenses for management, etc., were, therefore, calculated for 100 tons per day and the $8\frac{1}{2}$ tons charged in proportion. But there are three departments in the mill, viz., stamping, roasting, and leaching, and each department has to be charged with one-third of this expense. The above item of \$1.78 represents, therefore, one-third. In the statement of cost 4 per cent. salt was put down because subsequent experiments proved this amount to be sufficient.

SUMMARY

It must be borne in mind that the figures contained in the above table are the results of experiments. In other words, these figures were obtained under different treatments with regard to salt, temperature, time, etc., and the averages, therefore, do not represent the best obtainable results. This is especially the case with the loss by volatilization, which will be less after the men acquire more skill in maintaining the proper temperature. But as the roasting results obtained with the modified Howell furnace under above conditions are good enough to secure a profitable reduction of the ore, we may just as well accept these averages as a basis for calculations and estimates.

RECAPITULATIONS

| | |
|----------------------------------------------------|--------------------|
| Average value of raw ore without salt | 28.85 oz. per ton. |
| Average value of raw ore containing salt | 27.49 oz. per ton. |
| Average value of roasted ore | 26.10 oz. per ton. |
| Average value of vat tailings | 4.76 oz. per ton. |
| Average per cent. of chlorination | 81.6 per cent. |
| Average per cent. of actual extraction | 74.9 per cent. |

| | |
|------------------------------------------------------------------|--------------------------|
| Average number of ounces silver extracted per ton..... | 21.9 ounces. |
| Average percental loss by tailings..... | 17.0 per cent. |
| Average percental loss by volatilization..... | 7.9 per cent. |
| Average per cent. of salt used..... | 4.7 per cent. |
| Average number of tons roasted per day with one furnace..... | 8.5 tons. |
| Average cost of roasting one ton of ore..... | \$4.50 Mexican currency. |
| Average consumption of wood per ton, including steam power | 3 cargas. |

ROASTING IN THE REVERBERATORY FURNACE

The appliances for roasting consisted of a large size Stetefeldt furnace and one 24-ft. revolving cylinder furnace, as described above. After the Stetefeldt furnace proved to be a failure with the Del Oro ore, the roasting capacity became reduced to that of the revolving cylinder, or 8½ tons. In order to increase the roasting capacity to the stamping capacity, and for the sake of further experiments, four two-story reverberatory furnaces were erected, the lower hearth of 220 sq. ft. surface and the upper of 210 sq. ft. surface (Figs. 10, 11, and 12).

Each two-story furnace took four charges of one ton each. When one charge was finished all the others were moved forward, and on the first hearth a new charge dropped through an opening in the arch. From the second hearth the charge was dropped on the lower hearth, through an opening in the bottom near the working door. The upper hearth was used exclusively for oxidizing roasting, and the 4 per cent. of salt for better mixing was added while the charge was dropped on the lower hearth. As the proper time when the salt is to be added had proved to be a factor of great importance in roasting this ore, the process was conducted on the appearance of the ore at the second hearth. When the charge on that hearth showed it to be in proper condition to receive the salt, the ore from the finishing hearth was discharged, the charge from the third hearth moved on the finishing hearth and the charge on the second hearth dropped on the third, while the salt was added. Every two and one-half to three hours a charge was done, and therefore each two-story furnace roasted from 8 to nearly 10 tons per twenty-four hours, according to the quantity of lead contained in the ore. Zinc blende roasts quicker than galena. Each charge was ten to twelve hours in the furnace.

The chlorination results were so near those obtained in the

modified Howell furnace that no details need to be given here, but details and observations will be given of a charge which was subjected to a prolonged oxidizing roasting, because they are rather interesting. A charge of the Del Oro ore was placed directly on the hearth nearest to the fireplace (finishing hearth) and kept there until finished.

OXIDIZING ROASTING

First hour. — Assay of raw ore, 35.57 oz. per ton.

During this hour the ore had just fairly started to ignite. A sample when cold had not changed its color, and looked like raw ore. In this and the following hours a sample was leached first with water, then with sodium hyposulphite, and to each filtrate calcium sulphide was added.

Wash-water: no precipitate.

Hypo-solution (1 per cent.): no precipitate, light coloration.

Concentrated hypo: a heavy precipitate, consisting mostly of iron and lead.

Assay of ore, 33.54 oz. per ton; after leaching with hypo, 33.54 oz. per ton; no soluble silver.

Second hour. — Toward the end of this hour the ore commenced to lose its own heat, caused by the combustion of the pyrites. During this hour no fire was kept up. The color of the sample when cold was a dark greenish-brown.

Wash-water: no precipitate or discoloration, therefore no salts soluble in water.

Hypo-solution (1 per cent.): a slight precipitate.

Hypo-solution (3 per cent.): a heavy precipitate.

Assay of ore, 34.41 oz. per ton; after leaching, 34.12 oz. per ton; no soluble silver.

Third hour. — During this hour the ore had lost its own heat, and fire was started again, but the temperature was kept very low. The color of the sample when cold was brown-yellow, more brown than yellow.

Wash-water: no precipitate, no coloration, no soluble salts.

Hypo-solution (1 per cent.): considerable precipitate.

Assay of ore, 36.74 oz. per ton; after leaching, 35.28 oz. per ton; soluble silver, 1.46 oz. per ton, or 3.9 per cent.

Fourth hour. — The temperature was somewhat increased

during this hour, but still kept rather low. The color of the sample when cold was of a much lighter yellowish brown.

Wash-water: the first indication of precipitate, showing that until the end of the fourth hour no zinc sulphate had been formed.

Hypo-solution (1 per cent.): considerable precipitate.

Assay of ore, 35.57 oz. per ton; after leaching, 31.78 oz. per ton; soluble silver, 3.79 oz. or 10.6 per cent.

Fifth hour. — The temperature still kept at dull red. Color of sample when cold brown-red, showing that at this time some oxide of iron had formed.

Wash-water: considerable precipitate of a yellowish-white color, mostly zinc and cadmium.

Assay of ore, 35.13 oz. per ton; after leaching, 22.89 oz. per ton; soluble silver, 12.24 oz. per ton, or 34.8 per cent.

Sixth hour. — The same low heat; color of sample when cold darker red-brown.

Assay of ore, 33.83 oz. per ton; after leaching, 20.41 oz. per ton; soluble silver, 13.42 oz. per ton, or 39.6 per cent.

Seventh hour. — The same temperature; sample when cold still darker red-brown.

Assay of ore, 30.74 oz. per ton; after leaching, 13.41 oz. per ton; soluble silver, 16.33 oz. per ton, or 54.9 per cent.

Eighth hour. — The same moderate roasting temperature. No more sulphurous acid gas could be noticed. The ore commenced to look dead. The color of the sample when cold did not change in this nor in the following hours.

Assay of ore, 30.62 oz. per ton; after leaching, 13.55 oz. per ton; soluble silver, 17.05 oz. per ton, or 55.7 per cent.

Ninth hour. — The temperature was slightly increased, but the ore did not fume, nor could sulphurous acid be noticed.

Wash-water: a very heavy, yellowish-white precipitate.

Hypo-solution (1 per cent.): a heavy precipitate, consisting principally of lead, zinc and silver, no copper.

Assay of ore, 29.30 oz. per ton; after leaching, 12.09 oz. per ton; soluble silver, 17.21 oz. per ton, or 58.8 per cent.

Tenth hour. — Temperature again slightly increased; no fumes, no sulphurous acid noticeable. The ore remains fine and loose; no lumps.

Assay of ore, 29.16 oz. per ton; after leaching, 12.24 oz. per ton; soluble silver, 16.92 oz. per ton, or 58.0 per cent.

Eleventh hour. — During this hour the temperature was considerably increased, bright red; the ore remained fine and loose; no lumps; no sulphurous acid noticeable.

Wash-water; a yellowish-white precipitate of very clear color.

Assay of ore, 30.03 oz. per ton; after leaching, 13.26 oz. per ton; soluble silver, 16.77 oz. per ton, or 55.8 per cent.

As the ore during this hour presented the appearance of a dead roasted ore, oxidation was not carried any further, but 6 per cent. of salt was added at the end of the eleventh hour, to make some further observations.

At the end of the eleventh hour and just after adding 6 per cent. salt. — The salt was added, quickly stirred and a sample taken.

Wash-water: dark-colored precipitate.

Assay of ore, $28.74 + 6$ per cent. = 30.46 oz. per ton; after leaching, $11.66 + 6$ per cent. = 12.36 oz. per ton; soluble silver, 18.80 oz., or 59.4 per cent.

Twelfth hour. — The temperature lowered for chloridizing. The ore commenced to fume after the salt was added, but the fumes were very thin and light.

Wash-water: black precipitate.

Assay of ore, $28.13 + 6$ per cent. = 29.81 oz. per ton; after leaching, $13.12 + 6$ per cent. = 13.90 oz. per ton; soluble silver, 15.90 oz. per ton, or 53.3 per cent.

Thirteenth hour. — Same temperature; a mild, faint odor of chlorine perceptible; no lumps, and when the ore was left in a heap to cool it did not harden like other charges roasted with salt. At the end of this hour the charge was removed from the furnace.

Wash-water: dark precipitate, almost black.

Assay of ore, $28.42 + 6$ per cent. = 30.12 oz. per ton; after leaching, $11.07 + 6$ per cent. = 11.73 oz. per ton; soluble silver, 18.39 oz., or 61 per cent.

I found that the assay value of the ore during oxidizing roasting dropped from 35.57 oz. per ton to 29.74 oz. This was caused principally by an increase in weight which the ore sustained by oxidation, especially the lead and zinc of which a large percentage is converted into sulphate if roasted at such a low heat. This took place principally during the seventh hour, when the ore assumed a dark red-brown color. I further found that during oxidizing roasting considerable of the silver is con-

verted into a state in which it is soluble in sodium hyposulphite. It shows first at the end of the third hour (3.9 per cent.) and gradually increases until it reaches its maximum (58.8 per cent.) at the end of the ninth hour, and then diminishes during the next two hours to 55.8 per cent.

By adding 6 per cent. of salt and continuing to roast for two hours, the percentage of soluble silver increased only 2.2 per cent. above the maximum obtained in oxidizing roasting, notwithstanding that at the time the salt was added the ore was not dead roasted, but still contained sulphates soluble in water. The salt soluble in water, however, was zinc sulphate, which does not act on the salt; the iron sulphate was decomposed by that time. This experiment illustrates the great importance of adding the salt at a certain time during oxidizing roasting — that is, at a time before the iron sulphate is decomposed and the oxidation of the lead and zinc sulphides has fairly advanced.

Only a part of the soluble silver in the oxidized ore was present as a sulphate. From a sample of nine hours' oxidizing, when leached with water, only 7.39 oz. silver per ton could be extracted, while the sodium hyposulphite extracted 17.21 oz. per ton. In other words, of the 58.8 per cent. of soluble silver only 25.2 per cent. was sulphate of silver, while the balance of 33.6 per cent. was some other silver salt, not soluble in water but soluble in sodium hyposulphite, probably silver antimonate, though the raw ore contained only one-half per cent. of antimony.

TREATING THE OXIDIZED ORE WITH CUPRIC CHLORIDE

A sample of the nine hours' oxidizing roasting was moistened with a dilute solution of cupric chloride and left in that condition for three hours, then leached with water and hypo solution: Assay of ore, 29.30 oz. per ton; after leaching 7.29 oz. per ton; soluble silver, 22.01 oz. or 75.2 per cent. Increased extraction by using cupric chloride 4.8 oz. per ton, or 19.4 per cent.

CONSUMPTION OF WOOD IN THE REVERBERATORY FURNACE

The furnaces were built in pairs, two being connected with one flue. During four weeks the wood consumed by one pair of these furnaces was weighed. During this time 507 tons were roasted at a consumption of 672 cargas of wood, or 1.3 cargas per

ton. If expressed in cords, we find that one cord of wood roasted nine tons of ore, which is an exceedingly small consumption.

COST OF ROASTING IN THE REVERBERATORY FURNACE

Statement of cost for two two-story furnaces roasting 18 tons per day:

| | |
|------------------------------------------|--------------------------|
| Labor..... | \$22.60 |
| 23.4 cargas of wood at 75¢..... | 17.55 |
| 4 per cent. salt, 1440 lb. at 1.27¢..... | 18.28 |
| Tools, etc..... | 4.00 |
| Management, office, mechanics, etc..... | 3.77 |
| | \$66.20 ÷ 18 = \$3.67 |
| Cost per ton | \$3.67 Mexican currency. |

CONCLUSIONS

To form an opinion as to which roasting furnace is the most suitable for the San Francisco del Oro ore, we have to take into consideration only the modified Howell and the reverberatory. The Stetefeldt did not roast the ore, and the Brückner was not tried, because I knew by experience that a large Brückner furnace could not roast more than five or six tons per day of such a heavy ore, and therefore refrained from incurring the expense of erecting a Brückner furnace just for experimental purposes.

Both furnaces, the modified Howell and the reverberatory, gave about the same results, and the loss by volatilization was also nearly the same. The cost of roasting, however, is different. The cost per ton of ore in the modified Howell is found to be \$4.50, and in the reverberatory \$3.67. Difference in favor of reverberatory, 83c. per ton. Besides this, the reverberatory furnace creates much less flue-dust than the Howell. The latter forms a great deal of flue-dust which is far from being roasted, even if provided with an auxiliary fireplace. The labor question has also to be taken into consideration. The reverberatory requires more manual labor, and in a locality where labor is scarce, as is often the case in Mexico, it may be more advantageous, under certain conditions, to adopt the Howell notwithstanding the greater cheapness of the reverberatory.

XII

CHLORIDIZING OF CALCAREOUS ORES

I WAS engaged by the Anglo-Mexican Mining Company to investigate and improve the roasting of their ores at Yedras, Sinaloa, Mexico.

The ore is treated by the lixiviation process with sodium hyposulphite, and consists of argentiferous arsenical pyrites, fine-grained black zinc blende, arsenical fahlerz, some iron pyrites, and occasionally ruby silver, while the gangue consists of silicious limestone and calcspar.

The chloridizing roasting of highly calcareous silver ores in combination with argentiferous arsenical pyrites has, like the ore treated in the preceding chapter, not often, if ever, been the subject of a thorough investigation on a large scale, and in the following pages the results and observation of such an investigation are given in detail.

The only analysis of the ore which I could obtain is the following, made in San Francisco several years before the experiments were made:

| | Per cent. |
|------------------------|-----------|
| Silica..... | 15.13 |
| Sulphur..... | 13.31 |
| Arsenic..... | 9.82 |
| Iron..... | 17.33 |
| Alumina..... | 1.35 |
| Zinc..... | 4.92 |
| Lead..... | 1.78 |
| Carbonate of lime..... | 33.78 |
| Magnesia..... | 2.58 |

This analysis, however, does not represent the average of the ore which was delivered from the mine to the mill. Frequent concentration tests showed that the mill ore contained much more gangue matters than the analysis shows. However, it may serve to give a general idea of the ore. While experimenting I

felt very much the want of a chemical laboratory and a chemist at the works.

The roasting facilities at the Yedras mill consisted of four revolving Brückner cylinder furnaces, each 16 ft. long, and eight long reverberatory furnaces. The Brückners had been abandoned for several years, because former operators, so I was told, could not get satisfactory results with them. The chlorination was exceedingly low (40 to 50 per cent.), and the loss of silver by volatilization extremely high, while the main part of the roasted ore was rolled up into balls ranging from the size of an orange up to 15 in. and upward in diameter, the inside of which was not roasted. The subsequently erected reverberatory furnaces also gave very poor results — 65 to 70 per cent. chlorination, with a loss of silver by volatilization of 20 to 35 per cent. though fewer lumps were formed.

ROASTING IN THE BRÜCKNER FURNACES

I knew by experience that as a rule ores can be roasted with less loss of silver by volatilization in the Brückner than in the reverberatory furnace, and as the enormous loss of silver which the ore had sustained in the reverberatory was the most important question, I started the long-abandoned Brückner furnaces again. The previous failures with them I ascribed to the application of too high a temperature and an insufficient supply of air. The formation of balls I expected to diminish by judiciously regulating the heat and the revolving speed of the furnace, and if I succeeded in chloridizing well these lumps, their formation would not be a serious matter, for they could be pulverized in a ball-mill before charging the ore into the leaching vats.

My improvements on the Brückner cylinder, which consist of a fireplace and flue arrangement attached to each end of the furnace, enable me to apply the flame alternately through either end, and thus to use much longer cylinders, and provide fully for the free access of air between the fireplace and the throat of the furnace, which, strange to say, is not the case with the common Brückner furnaces. In a reverberatory furnace air can enter through the working doors; Stetefeldt provided his furnace with air-channels through which the required supply can be regulated; the Howell furnace is provided with an air-door; but the common

Brückner furnace has no proper means of regulating the supply of air, and either the fire-door must remain open or only a very limited amount of air can enter the furnace. If the whole fire-box is built of brick it is easy enough to make the proper change, but if it is made of boiler-iron lined with brick, it is not easy to make the necessary alterations in a remote mining camp. In this case, the fire-boxes being of brick, I added 12 in. to their length, moved the grate bars toward the front, and inserted an air-channel behind the new fire-bridge. The fresh air entering through this chamber not only assisted in oxidizing the ore, but also aided the combustion of the fuel. The speed of the furnace, which had been $2\frac{1}{2}$ revolutions per minute, was reduced to one revolution in $1\frac{3}{4}$ minutes, which materially reduced or altogether avoided the formation of large balls and diminished the quantity of dust, while on the other hand the speed was sufficient to expose every particle of the ore to the action of air and heat; in fact, a still slower speed, possibly even one revolution in three or four minutes, would have been preferable had it been obtainable.

The ore was crushed dry in the stamp battery with 7 per cent. of salt, and passed through a 24-mesh screen. The furnace charge was 4 to $4\frac{1}{2}$ tons, and a strong fire was kept in order to quickly ignite the ore. In about an hour, and before the sulphur commenced to burn, heavy arsenic fumes were given off, entirely obscuring the interior of the furnace. After one and one-half or two hours a sulphur flame could be observed to enter the flue, which was a sign that the sulphurets were sufficiently ignited to continue combustion without the aid of fire. The fire was then allowed to go out and the fire and air-doors were kept wide open. The temperature gradually increased by the combustion of the sulphur until it reached a certain maximum, at which it remained for several hours. After two or three hours the arsenic period was over, the heavy fumes disappeared, the interior of the furnace became clear, and the glow of the ore presented a beautiful pink color. No chemical loss of silver took place during the arsenic period. The oxidizing period continued for three to four hours more, the ore remaining at about the same temperature throughout, then fumes commenced to rise from the ore, and gradually increased, but never became so dense as during the arsenic period, though still dense enough to make the interior of the furnace invisible. The chloridizing period had commenced, and the odor

of sulphurous acid and chlorine could be observed in the samples taken during the first period; but later the odor of sulphurous acid disappeared and only chlorine could be detected. In many instances, however, no smell of chlorine could be noticed during the whole time of roasting, as explained further on. Looking through the open fireplace, which by this time had cooled down, the whole interior seemed to be glowing, though the ore itself was not visible on account of the zinc fumes, which were illuminated by the glow of the red-hot ore and made a beautiful sight, and at the same time afforded a good opportunity to observe the temperature in the furnace. So soon as a decrease in temperature was noticed, the fire was started again and kept up for three to four hours, when the charge was considered finished. It took altogether from twelve to fourteen hours to roast a charge.

During the oxidizing period the ore maintained an almost level position in the cylinder and had a liquid-like appearance. The ore particles on the surface, however, could be seen constantly moving; on the side where the furnace moves up fresh ore came to the surface as if emerging from a liquid, moved slowly across, and sank as soon as it touched the down-moving side of the cylinder. The ore increased considerably in volume during the first part of the chloridizing period; but after it had reached the maximum it commenced to shrink again, assuming a heavy sandy condition, where before it was loose and woolly, and finally occupied no more space in the furnace than the raw ore did. A short time before the second fumes commenced to rise the charge assumed a more inclined position, attaining nearly 45 deg.

There was a great and puzzling irregularity in the results. Sometimes a number of successive charges gave satisfactory results; then at once the chlorination dropped without any apparent cause, the ore, the amount of salt, the temperature and the treatment being unchanged. The only noticeable difference was the amount of silver contained in the ore; and as a rule the richer ore gave the better results, though it was evident the quantity of silver as such could not influence the result materially. Close investigation finally showed the true cause of the trouble to be the more or less favorable proportion between the gangue (carbonate of lime) and the sulphureted matters in the ore; and this explanation corresponded with the fact noticed that richer ores usually gave better results than poorer ores. No marked

difference between the two could be noticed in an inspection of the ore before it went into the battery.

The following tables of the results obtained with ores containing less and ores containing more carbonate of lime illustrate these differences:

ORE CONTAINING LESS CARBONATE OF LIME

(7 per cent. salt mixed in the battery.)

| NO. OF CHARGE | VALUE OF RAW ORE PER TON | VALUE OF ROASTED ORE PER TON | VALUE OF LEACH TAILINGS PER TON | CHLORINATION |
|---------------|--------------------------|------------------------------|---------------------------------|---------------|
| | Oz. Silver | Oz. Silver | Oz. Silver | Per Cent. |
| 65 | 72.12 | 76.12 | 16.20 | 79.6 |
| 66 | (a) | 64.86 | 12.60 | 80.5 |
| 67 | 65.64 | 64.80 | 9.00 | 86.2 |
| 68 | 68.04 | 68.46 | 10.56 | 84.6 |
| 69 | 66.30 | 72.16 | 13.68 | 81.1 |
| 70 | 66.78 | 73.62 | 12.36 | 86.0 |
| 71 | 68.16 | 71.16 | 11.88 | 83.3 |
| 72 | 66.24 | 68.64 | 12.90 | 81.3 |
| 73 | 68.52 | 73.82 | 14.70 | 80.0 |
| 74 | 66.00 | 75.24 | 11.16 | 85.4 |
| 75 | 64.38 | 66.42 | 12.06 | 81.9 |
| 76 | 63.78 | 66.30 | 9.86 | 85.9 |
| 77 | 68.28 | 68.10 | 11.42 | 83.3 |
| 78 | 58.50 | 61.02 | 10.80 | 82.4 |
| | <u>862.74</u> | <u>970.72</u> | <u>169.18</u> | <u>1161.5</u> |

(a) Sample of raw ore lost.

Average of raw ore..... 66.36 oz. per ton.
 Average of roasted ore..... 69.33 oz. per ton.
 Average of leach tailings..... 12.08 oz. per ton.
 Average of chlorination..... 82.96 per cent.

The roasted ore contained 2.97 oz. more silver per ton than the raw ore.

ORE CONTAINING MORE CARBONATE OF LIME

(7 per cent. salt mixed in the battery.)

| NO. OF CHARGE | VALUE OF RAW ORE PER TON | VALUE OF ROASTED ORE PER TON | VALUE OF LEACH TAILINGS PER TON | CHLORINATION |
|---------------|--------------------------|------------------------------|---------------------------------|---------------|
| | Oz. Silver | Oz. Silver | Oz. Silver | Per Cent. |
| 79 | 61.06 | 60.36 | 15.90 | 73.7 |
| 80 | 62.46 | 64.20 | 19.14 | 71.2 |
| 81 | 60.18 | 64.08 | 16.14 | 74.9 |
| 82 | 56.70 | 56.52 | 11.76 | 79.2 |
| 83 | 55.08 | 55.44 | 12.72 | 77.1 |
| 84 | 57.12 | 53.88 | 20.80 | 61.4 |
| 85 | 60.34 | 58.98 | 14.52 | 75.4 |
| 86 | 59.64 | 57.60 | 20.04 | 65.3 |
| 87 | 58.50 | 56.10 | 18.18 | 67.6 |
| 88 | 49.92 | 52.86 | 15.18 | 71.3 |
| 89 | 55.38 | 54.90 | 12.84 | 76.7 |
| 90 | 55.80 | 52.56 | 12.90 | 76.2 |
| 91 | 57.60 | 51.36 | 9.12 | 81.3 |
| 92 | 57.00 | 55.80 | 13.20 | 76.4 |
| 93 | 55.02 | 55.74 | 15.00 | 73.1 |
| 94 | 58.32 | 56.64 | 14.64 | 74.2 |
| 95 | 57.00 | 57.36 | 7.62 | 86.8 |
| | <u>977.12</u> | <u>964.38</u> | <u>249.70</u> | <u>1261.8</u> |

Average of raw ore.....57.47 oz. per ton.
 Average of roasted ore.....56.72 oz. per ton.
 Average of leach tailings.....14.68 oz. per ton.
 Average of chlorination.....74.22 per cent.

The raw ore contained 0.75 oz. more silver per ton than the roasted ore.

Comparing the average results, they are found to be decidedly in favor of the higher sulphureted ore. To obtain further information I concentrated some of the ore, and made mixtures of certain proportions of concentrates and barren gangue, mostly carbonate of lime, and roasted these different mixtures as well as the concentrates in the muffle, treating all samples alike, using 7 per cent. of salt, and roasting each half an hour. The concentrates used were obtained from ore which did not represent the average richness, assaying only 35.70 oz. per ton, and therefore the different proportions I made were much poorer in silver than I afterward found corresponding ones in the bulk of the run. Only the concentrates were assayed, the values of the mixtures being calculated.

These tests showed that the silver-bearing minerals of this ore do not offer any difficulties to a good chloridizing roasting;

on the contrary, they are easy to roast to a high percentage without showing any tendency to ball. The difficulties the ore offered were therefore caused by the gangue; and the great variation in the results was due to a greater or less favorable proportion of sulphureted matters to lime.

| NO. OF SAMPLE | CONCENTRATES | BARREN GANGUE | VALUE OF MIXTURE PER TON | VALUE OF LEACH TAILS PER TON | CHLORINA- TION |
|------------------|--------------|------------------|--------------------------------|------------------------------------|-------------------|
| | Per Cent. | Per Cent. | Oz. Silver | Oz. Silver | Per Cent. |
| 1 (a) | 100.0 | | 96.0 | 2.91 | 97.0 |
| 2 (b) | 75.0 | 25.0 | 72.0 | 5.38 | 92.6 |
| 3 (c) | 62.5 | 37.5 | 60.0 | 4.72 | 92.2 |
| 4 (d) | 50.0 | 50.0 | 48.0 | 5.38 | 88.8 |
| 5 (e) | 25.0 | 75.0 | 24.0 | 5.47 | 77.2 |

(a) Fuming profusely during chloridizing period; strong and pure smell of chlorine; a great deal of base-metal chlorides soluble in water was formed; no tendency to form lumps shown. (b) During chloridizing heavy fumes; distinct but moderately strong smell of chlorine; no base-metal chlorides formed; no tendency to form lumps shown. (c) Much less fumes; very little chlorine observable; none in water-soluble base-metal chlorides; no tendency to form lumps. (d) Very little fumes; no chlorine; no base-metal chlorides; showed tendency to form lumps. (e) No fumes except at a high heat, and then but little; no chlorine; none in water-soluble chlorides; showed much tendency to form lumps.

Carbonate of lime in presence of heated sulphureted minerals will change partly into calcium sulphate, which does not act on sodium chloride, and partly into caustic lime, which decomposes the metal sulphates and chlorides, and also, though less rapidly, the silver chloride. If, however, carbonate of lime is greatly in excess, only a very small amount, if any, of base-metal sulphates is formed to decompose sodium chloride, the greater part being changed directly into oxides. If no salt is present these sulphates are, of course, quickly changed into oxides. For this reason no chlorine can be detected if the salt is added after the ore has been subjected to a partial oxidizing roasting.

Some of the more calcareous charges were followed closely, but at no stage of the roasting could any soluble salts or chlorine be detected if the salt was added after the ore had been oxidizing for some time. If the salt was pulverized with the ore in the battery traces of such salts were found, but never in large quantities.

As there were neither sulphates, which at a high temperature

decompose salt, nor chlorides, which at a high heat give off chlorine, an increased temperature during the last part of the roasting could not be of any benefit. In fact, it had even a very bad effect. The caustic lime, which at a low temperature seemed to be comparatively indifferent to silver chloride, decomposed it energetically at a high temperature. The roasting of such ores had therefore to be finished at as low a temperature as possible, contrary to the usual practice. This fact must be taken into consideration in constructing long reverberatory furnaces. The arch of the first and second hearths nearest to the fire should be very high. (Figs. 4, 5 and 6.)

Charge No. 86 of the more calcareous ore was roasted for eight hours without fire; then, when the ore commenced to lose heat, a second fire was applied for three hours. Before starting the second fire a sample was taken through the entire length of the furnace, and the chlorination was found to be 73.4 per cent., and after three hours' second fire it was only 65.3 per cent.

The moderately increased temperature at that stage of the roasting reduced the chlorination 8 per cent. A number of such observations were made and finally the mode of roasting was changed, and the ore was allowed to roast entirely by the oxidation of the sulphureted matters without the application of a second fire.

Highly calcareous charges, containing only about 20 to 30 per cent. sulphurets, had a dead and sandy appearance, no chlorine could be noticed, and it required a high heat to make the ore fume. These fumes were light and were probably zinc oxide. Not even at a very high heat could any chlorine be detected. If, however, a very strong draft of air was allowed to pass through the furnace from the beginning and during the whole time of roasting, some chlorine was generated and could be detected. The ore fumed moderately at a low temperature and 70 to 75 per cent. of the silver was rendered soluble in sodium hyposulphite. This indicated that by a rapid supply of air more sulphuric and less sulphurous acid was generated, and that some of the sodium chloride was decomposed by the former.

The beneficial effect of a great surplus of air was still more noticeable with highly sulphureted ore in the reverberatory furnace. Differences as high as 25 to 30 per cent. in chlorination were observed between two charges of the same ore, which were treated

alike, except that one furnace had excessive draft while the other had insufficient air. Silicious ores can be well chloridized with a moderate supply of air, while calcareous ores need an excess. This fact is of great importance, and due attention should be paid to it in constructing furnaces.

An ore which by hand concentration showed 40 to 45 per cent. sulphurets generated considerable chlorine; it fumed more freely and at a lower temperature; and though no base-metal chlorides were formed, 80 to 86 per cent. of the silver was chloridized when temperature, time, and supply of air were properly regulated. Ore showing by hand concentration 50 to 55 per cent. of sulphurets permitted a chlorination of 90 per cent. and over.

In the Brückner furnace the ore behaved very differently, according as the salt was added in the battery or in the furnace, and it was therefore necessary to examine each case separately.

ADDING THE SALT IN THE FURNACE

If the ore was allowed to oxidize without salt a certain percentage of the silver would be rendered soluble in sodium hyposulphite, but not in water, owing to the arsenic contained in the ore. This change takes place principally at the very beginning of roasting and at a very low temperature. A sample taken from charge No. 128 an hour after the combustion of the sulphurets had commenced, while the arsenic was still fuming strongly and oxidization had but just commenced, the appearance of the sample being still that of raw ore, yielded no less than 44.58 per cent. of the silver with sodium hyposulphite. Continuing the oxidation, the amount of soluble silver increased, but not so rapidly as in the first hour, as was shown by the following tests:

Charge No. 128, after 1 hour oxidizing roasting, yielded 44.58 per cent. of its silver with sodium hyposulphite.

Charge No. 128, after 1½ hours oxidizing roasting, yielded 43.44 per cent. of its silver with sodium hyposulphite.

Charge No. 128, after 2½ hours oxidizing roasting, yielded 45.32 per cent. of its silver with sodium hyposulphite.

Charge No. 128, after 3½ hours oxidizing roasting, yielded 57.66 per cent. of its silver with sodium hyposulphite.

No marked increase in soluble silver could be noticed by continuing the oxidation still further.

A certain percentage of silver can be extracted from some

ores with sodium hyposulphite without roasting, and in order to determine whether this extractable form of silver was originally contained in the ore, or was formed during even such a slight oxidizing roasting as charge No. 128 showed, where in one hour 44.58 per cent. became extractable, a sample of raw ore was lixiviated, but not even a fraction of 1 per cent. of silver could be extracted. As this silver combination soluble in sodium hyposulphite was principally formed in the first stage of oxidizing roasting, during the time the arsenic was being expelled, it was undoubtedly silver arsenate.

When the salt was added at any time during the oxidizing period and well stirred in with a hoe, the percentage of soluble silver either increased suddenly a few per cent. or decreased, but in neither case could this percentage be maintained or further increased. Decomposition commenced immediately and the percentage of extractable silver diminished rapidly, when, after two to four hours, according to the amount of salt added, the maximum was reached.

In the following tables are given the detailed history of three charges, roasted with 10, 7, and 4 per cent. of salt respectively. The maximum of decomposition was reached in two, three, and four hours:

CHARGE NO. 116

(10 per cent. salt added during roasting.)

| DESCRIPTION (a) | VALUE OF ROASTED ORE CONTAINING SALT PER TON | VALUE OF LEACH TAIL- INGS PER TON | SOLUBLE SILVER |
|--------------------------------------------------------------------------------------------------------|----------------------------------------------------------|--------------------------------------------|-------------------|
| | Oz. Silver | Oz. Silver | Per Cent. |
| Raw ore Oz. per ton, 62.52 | | | |
| After 5 hours and 20 minutes roast- ing and just before adding salt Oz. per ton, 61.32 | | | |
| The same calculated to contain 10 per cent salt. Oz. per ton, 55.19 | | 28.80 | 53.1 |
| Directly after adding salt and stirred | 57.24 | 26.04 | 54.5 |
| Twenty minutes after adding salt . . . | 62.04 | 31.92 | 48.6 |
| One hour after adding salt | 61.98 | 39.48 | 36.3 |
| Two hours after adding salt | 63.24 | 46.68 | 26.2 |
| Three hours after adding salt | 62.10 | 42.90 | 31.0 |
| Four hours after adding salt | 64.32 | 41.64 | 35.3 |
| Five hours after adding salt | 64.20 | 40.02 | 37.7 |

(a) After 5 hours and 20 minutes oxidizing roasting 10 per cent. salt was added and the charge stirred with hoes. The maximum of decomposition was reached in 2 hours after the salt was added, at which time 51.92 per cent. of the soluble silver was rendered insoluble. A decomposition of 10.82 per cent. took place in the first 20 minutes; in the next 40 minutes, 22.57 per cent.; in the next 60 minutes, 18.53 per cent.

CHARGE NO. 123

(7 per cent. salt added during roasting.)

| DESCRIPTION (a) | VALUE OF ROASTED ORE CONTAINING SALT PER TON | VALUE OF LEACH TAIL- INGS PER TON | SOLUBLE SILVER |
|------------------------------------------------------------------------------|----------------------------------------------------------|--------------------------------------------|-------------------|
| | Oz. Silver | Oz. Silver | Per Cent. |
| Raw ore..... Oz. per ton, 58.62 | | | |
| After 4 hours roasting and just be- fore adding salt.. Oz. per ton, 59.70 | | 21.36 | 64.27 |
| One hour after adding salt..... | 59.84 | 27.00 | 54.75 |
| Two hours after adding salt..... | 56.04 | 35.40 | 34.69 |
| Three hours after adding salt..... | 55.92 | 41.16 | 26.40 |
| Four hours after adding salt..... | 54.60 | 34.08 | 37.59 |
| Five hours after adding salt..... | 54.00 | 30.96 | 42.65 |
| Six hours after adding salt..... | 57.06 | 27.12 | 52.48 |
| Seven hours after adding salt..... | 50.52 | 23.10 | 54.28 |
| Eight hours after adding salt..... | 55.20 | 20.04 | 63.70 |

(a) The charge, subjected for 4 hours to oxidizing roasting before 7 per cent. salt was added, yielded 64.27 per cent. of the silver soluble in sodium hyposulphite. The maximum decomposition was reached in 3 hours after the salt was added, at which time 58.92 per cent. of the soluble silver was rendered insoluble. The decomposition during the first hour was 14.81 per cent.; the second hour, 31.21 per cent.; the third hour, 12.90. During the next 5 hours we find the amount of soluble silver gradually increasing until at the end of the fifth hour nearly the same amount of silver was rendered soluble as the ore contained before the salt was added. This would indicate that by a continuation good results may finally be obtained, but the heat of the charge is exhausted before that time, and a second fire causes decomposition again.

CHARGE NO. 125

(4 per cent. salt added during roasting.)

| DESCRIPTION (a) | VALUE OF ROASTED ORE CONTAINING SALT PER TON | VALUE OF LEACH TAIL- INGS PER TON | SOLUBLE SILVER |
|------------------------------------------------------------------------|----------------------------------------------------------|--------------------------------------------|-------------------|
| | Oz. Silver | Oz. Silver | Per Cent. |
| Raw ore.....Oz. per ton, 59.70 | | | |
| After 3 hours roasting and just before adding salt..Oz. per ton, 58.32 | | 26.58 | 54.42 |
| Just after adding salt and stirred ... | 52.44 | 20.84 | 61.79 |
| One hour after adding salt..... | 56.16 | 23.94 | 57.38 |
| Two hours after adding salt..... | 56.10 | 29.40 | 47.60 |
| Three hours after adding salt..... | 54.90 | 42.24 | 23.07 |
| Four hours after adding salt..... | 58.56 | 46.80 | 20.09 |
| Five hours after adding salt..... | 56.50 | 39.66 | 29.25 |

(a) After 3 hours oxidizing roasting 54.42 per cent. of the silver in the ore was soluble in sodium hyposulphite, then 4 per cent. salt was added and mixed with the ore by hoes. A sample then taken through the entire length of the furnace showed 61.79 per cent. soluble silver, or a gain of 7.39 per cent. in these few minutes. Decomposition soon set in, however, and the maximum was reached in four hours after the salt was added, at which time 67.48 per cent. of the soluble silver was rendered insoluble. The decomposition during the first hour was 7.13 per cent; the second hour, 15.83 per cent.; the third hour, 39.70 per cent.; the fourth hour, 4.82 per cent.

Summing up the results of these experiments it was clear that (1) at a very early stage of oxidizing roasting quite a high percentage of the silver was converted into a combination (undoubtedly silver arsenate) which was soluble in sodium hyposulphite, and which seemed to resist well the decomposing action of the lime. (2) A part of this soluble silver was decomposed by the action of the salt, the decomposition commencing almost immediately after the addition of salt and continuing until a maximum was reached, when a reaction took place by which soluble silver was again formed — most likely silver chloride. (3) A larger percentage of salt produced the decomposition quicker, but not so thoroughly as a smaller percentage, viz.:

Charge No. 116, with 10 per cent. salt, reached the maximum in 2 hours, with 51.92 per cent. of soluble silver decomposed.

Charge No. 123, with 7 per cent. salt, reached the maximum in 3 hours, with 64.27 per cent. of soluble silver decomposed.

Charge No. 125, with 4 per cent. salt, reached the maximum in 4 hours, with 67.48 per cent. of soluble silver decomposed.

The singular fact was developed that when salt was added in the reverberatory furnace during the oxidizing period no such decomposition took place. Lixiviating tests made hourly and half-hourly after the salt was added showed a gradually increasing chlorination. This strange behavior of the ore in the Brückner seems to be a reaction of silver arsenate and sodium chloride, but why such a reaction does not take place in the reverberatory I cannot explain. No lumps or balls were formed in this mode of roasting, but the peculiar reaction prevented a higher chlorination and rendered this method of roasting impracticable.

ADDING THE SALT IN THE BATTERY. — SELF-ROASTING

When the salt is added in the battery it becomes thoroughly mixed with the ore, and both enter the furnace together. Though there is also a considerable part of the silver converted into a salt soluble in sodium hyposulphite at a very early stage of roasting, due to the arsenic in the ore, yet no decomposition of the arsenate of silver seems to take place. On the contrary, a gradual increase of soluble silver is observed up to a very advanced stage of roasting, when all the sulphureted minerals are converted into oxides. If the roasting is continued beyond this point, or if the temperature is raised before or after this point is reached, then the caustic lime acts on the silver chloride and decomposes it.

When beginning the experiments I conducted the roasting in the Brückner the same as with certain highly sulphureted silicious ores, viz.: (1) by raising a strong heat to start combustion; (2) oxidizing without fire until the temperature commenced to decrease; (3) chloridizing and finishing at a higher temperature with a second fire. In the course of the experiments, however, the employment of the second fire caused the decomposition of a considerable part of the soluble silver already formed. A charge was therefore roasted without using a second fire, leaving the ore to complete the roasting in the heat created by the combustion of the sulphurets, and much better results were attained. For the sake of comparison I roasted and finished a number of charges without the additional fire and then a number with the second fire, using ore of the same grade and maintaining similarity in all other conditions, and found that with self-roasting the average chlorination was 5 per cent. higher, while the consump-

tion of wood was reduced more than one-half. With a second fire the average value of the roasted ore was 0.46 oz. silver less than the value of the corresponding raw ore, while in self-roasting the average value of the roasted ore was 1.91 oz. silver higher than that of the raw ore, indicating smaller loss by volatilization. I call this mode of roasting "self-roasting," because the ore, once ignited, requires no further attention, and is left entirely to itself until the heat has nearly died out. Self-roasting simplifies the manipulation. It is only necessary to maintain fire for about two hours, when the charge can be left to itself until the time of discharging, which is about an hour before the red heat dies out. Thus one man can attend to quite a number of furnaces. Care has to be taken, however, to make large charges, as small charges do not maintain the heat long enough, and they "freeze" before the roasting is completed.

Having obtained by this mode of roasting an average of 82.9 per cent. chlorination, with occasional results of 85 and 86 per cent. (see Table) with calcareous ore which offered so many difficulties to chloridizing roasting, it seems that it may be possible to chloridize properly nearly all kinds of highly sulphureted ores in this way; but, as large charges work better than small ones, it is advisable to have the revolving cylinders large enough to hold 5 or 6 tons of ore.

By roasting in this manner more subchlorides are formed, and less volatile chlorides expelled; this, however, does not interfere much with the subsequent lixiviation. The advantages gained will more than overbalance the slight extra expense caused by the increased consumption of sulphur, and the refining of a somewhat baser precipitate, especially if the diminished loss of silver by volatilization, which loss is principally caused by the expulsion of the volatile base-metal chlorides, is taken into consideration. If the ore contains copper, an increased formation of cuprous chloride will even be beneficial for the subsequent extraction by lixiviation.

The consumption of wood in self-roasting was found to be only one cord for each 10.6 tons of ore, while in roasting with a second fire only 4 to 4.5 tons could be roasted with a cord of wood.

BALLING OF THE ORE

Another noticeable difference in the behavior of the ore, due to the addition of the salt in the battery or in the furnace, was the formation of balls or lumps in the first case, while in the latter instance the ore remained loose without forming balls, and when discharged ran on the cooling floor like water. When the salt was added in the battery it assumed a more solid form, did not spread over the cooling floor, and did not dust, but contained a great many balls. These balls originated during the early part of the chloridizing period and at first were of the size of a pin's head. Gradually they assumed larger dimensions, and when the charge was finished the majority of them were from the size of a pea to that of a walnut. They were smooth, hard, and heavy, consisting of concentric shells, and were formed even if a second fire was not used. By reducing the speed of the furnace from $2\frac{1}{2}$ revolutions per minute to one revolution in $1\frac{3}{4}$ minutes, and by adopting self-roasting, which was equivalent to roasting at the lowest possible heat, the number and size of these hard balls were greatly reduced, but were not altogether prevented.

Roasting during the chloridizing period was tried with an intermittent motion. The furnace was allowed to make one revolution and was then stopped for fifteen minutes, when another revolution was made, and so on until the charge was finished. This reduced but did not prevent the formation of balls. Then the furnace was stopped entirely during the chloridizing period, intending to allow the completion of the roasting process without any further movement, though it extended the time required for roasting. This method would very likely have had the desired effect had not another difficulty made it impracticable. The surface of the ore commenced to harden, forming a crust, which increased in thickness, was spongy and porous, and would not have interfered with the lixiviation, but threatened that by the time the charge was finished the whole mass might have hardened, which would have caused great difficulty in discharging, and would also have endangered the furnace through the whole mass clinging to one side and then dropping suddenly as the furnace revolved.

Though annoying, this formation of hard balls would not be a serious obstacle, as by repeated tests it was found that they

were well roasted; in fact, always a slight percentage better chloridized than the fine stuff. By separating the coarse from the fine, and crushing the former in a ball-mill or through rolls, the ore would be well prepared for lixiviation. This extra handling does not signify much when the great difference in expense between roasting in a Brückner and roasting in a reverberatory furnace is taken into consideration, to say nothing of other advantages. Other lumps were formed toward the end of the operation which were larger, but soft and porous, had a rough surface, fell apart when brought in contact with water, and did not interfere with lixiviation.

The sulphureted minerals of this ore had no tendency to cake and form lumps. When the concentrates, free from gangue, were roasted, the pulp remained perfectly loose and sandy, even if only occasionally stirred during roasting; the formation of these small hard balls must therefore have been caused by the gangue. They were soft while hot and were readily crushed, but when cold became hard and brittle, and when broken showed concentric layers. They were not caused by excessive heat, because they formed even when the heat was kept as low as the combustion of the sulphureted minerals permitted; neither was their appearance that of overheated ore. They were probably caused by the formation of a double salt of calcium sulphate and sodium sulphate (glauberite), $\text{Na}_2\text{SO}_4 + \text{CaSO}_4$, a salt which fuses easily. This would explain the singular fact that no lumps were formed if the salt was added in the furnace after the charge had been oxidizing for some time; in which case the salt could not be thoroughly mixed with the ore, and therefore did not come in such close contact with the sulphureted matters as the lime did, on which sulphuric acid acts so much more energetically than on sodium chloride that very little, if any, of the sodium chloride was converted into sulphate. For the same reason free chlorine could be detected only in exceptional cases if the salt was added in the furnace. Neither was it possible to produce a chlorination of the silver. The 40 to 60 per cent. soluble silver was arsenate of silver.

In my muffle experiments with different mixtures of concentrates and gangue (limestone) the tendency to form lumps commenced with the proportion of 50 concentrates to 50 gangue, and this tendency increased with the percentage of gangue; it did

not exist if the mixture contained more sulphurets than gangue, and if the ore had been assorted at the mine to meet this requirement the formation of balls would doubtless have been entirely avoided, the chlorination would have been much better, and the Brückner furnaces could have been used instead of the more expensive reverberatories. It may be mentioned that the ore from the mine had to pass successively through three large ore-bins before reaching the mill. These were always kept full, in order to have ore in reserve, and it took 10 to 14 days from the time the ore was dumped into the bin at the mine before it reached the battery; quick changes for experimental purposes were therefore impossible.

ROASTING IN THE REVERBERATORY FURNACES

There were in all eight reverberatory furnaces in operation, of the following dimensions:

Four furnaces, 50 ft. long by 10 ft. wide, containing 5 hearths each.

Two furnaces, 40 ft. long by 10 ft. wide, containing 4 hearths each.

Two furnaces, 30 ft. long by 10 ft. wide, containing 3 hearths each.

The arch at the highest place near the fire was 27 in. above the hearth, further away only 20 in., and at the last hearth 18 in. The sides were 10 in. high. The furnaces were built in pairs, placed back to back, and each hearth had but one small working door 8 x 12 in.

While tolerably good chlorination may be obtained with silicious silver ore in furnaces of the above-described construction, they surely were not of suitable design for the roasting of calcareous silver ores, as in the first place insufficient provision was made for a free and well-located air inlet; the small working door furnished air only for the ore near it, which fumed, while further in and toward the back of the furnace it presented the appearance of an inactive glowing mass of high temperature, and did not emit any visible fumes. In the second place, the arch, especially the one over the hearth nearest to the fire, was not high enough to keep the temperature low enough for calcareous ores during the chloridizing period. There was no remedy for these defects except reconstruction. By keeping the fire-door continually wide open much better results were obtained, but they were not satisfactory until the furnaces were reconstructed.

At Yedras I was astonished to find that the ore in the reverberatory furnaces on the finishing hearth was subjected to almost a white heat, while care was taken to avoid cooling the furnace by excluding the air as much as possible. The fire-door was kept closed, and the very small working doors, of which there was one on each hearth, were closed as soon as stirring was completed. One of the charges roasted in this way consisted of one ton of ore; four men were attending the furnace, which was 50 ft. long, and 5 per cent. salt was added on the third hearth after the ore had been nearly five hours in the furnace. The following is a detailed record of same:

1. Raw ore without salt. Sample of ore taken from the hopper before charging gave 67.08 oz. silver per ton.

2. Same charge. Sample taken from the first hearth after the ore had been two hours in the furnace, and just before the charge was moved to the second hearth, the hearth being dark, gave off a pretty strong smell of sulphurous acid and contained 65.40 oz. silver per ton of ore.

3. Sample taken from the second hearth, after having been there for two hours and twenty minutes, and just before the charge was to be moved to the third hearth. Charge dark-red, strong fume (arsenic period), with strong smell of sulphurous acid, gave 66.06 oz. silver per ton of ore.

4. Sample taken from the third hearth just before the salt was added. Charge red-hot, strong smell of sulphurous acid, gave:

| | | |
|----------------|-----------|----------------------------------------------------------|
| Ore..... | 67.92 oz. | } 53.08 per cent. silver soluble in sodium hyposulphite. |
| Leach tailings | 31.44 oz. | |

5. Sample taken from third hearth after having roasted just one hour with salt. Strong fumes and smell of sulphurous acid; color, after cooling, light-brown; gave:

| | | |
|----------------|-----------|---------------------------------------------------------|
| Ore..... | 63.60 oz. | } 54.4 per cent. silver soluble in sodium hyposulphite. |
| Leach tailings | 29.04 oz. | |

6. Sample taken from fourth hearth after the charge had been there one hour, and had roasted two hours with salt. The ore woolly, and sample exposed to air fumed strongly; color, after cooling, red-brown; temperature red; gave:

| | | |
|----------------|-----------|-------------------------------------------------------|
| Ore..... | 60.72 oz. | } 53 per cent. silver soluble in sodium hyposulphite. |
| Leach tailings | 28.44 oz. | |

7. Sample taken while charge was being moved to fifth hearth and after roasting three hours and twenty minutes with salt. Temperature light red; sample fumed strongly when exposed to air; inside of the furnace clear, fumed only near the working door; slight smell of chlorine and sulphurous acid; ore commenced to assume a sandy consistency, and had the appearance of an over-heated ore; color, after cooling, red-brown; gave:

Ore 70.98 oz. }
Leach tailings 27.12 oz. } 61.8 per cent. silver soluble in sodium hyposulphite.

8. Sample taken from fifth hearth after roasting five hours with salt. No smell of sulphurous acid, very little of chlorine; temperature very light red, almost white; ore sandy; color, after cooling, greenish brown, gave:

Ore 68.94 oz. }
Leach tailings 23.94 oz. } 65.2 per cent. silver soluble in sodium hyposulphite.

9. Sample taken when the ore was discharged, after having been eleven hours in the furnace and six hours roasted with salt. Very little smell of chlorine, none of sulphurous acid; ore very sandy, mixed with lumps; color, after cooling, greenish brown; gave:

Ore 65.40 oz. }
Leach tailings 21.06 oz. } 67.2 per cent. silver soluble in sodium hyposulphite.

The ore sustained during roasting a loss in weight by volatilization of 13.8 per cent., and the loss of silver by volatilization amounted to 16 per cent.

By comparing the different samples it was found that at the time the salt was added there was 53.8 per cent. of the silver soluble in sodium hyposulphite. This soluble silver was an arsenate and was formed in the same manner as that obtained in the Brückner furnace during the arsenic period. During the next two hours of roasting with salt there was no change in the soluble silver. This is unlike the behavior of the ore in the Brückner, in which during this period a marked decomposition of the soluble silver took place. Although in the reverberatory the salt was also added after the ores had been oxidizing in the furnace for nearly five hours, and the arsenic period was over, no soluble silver was decomposed by the salt. During the next two hours it remained the same, and commenced to increase only after the third hour with salt, and after the ore had reached the

region of light-red heat on the fourth hearth. The increase, however, amounted to only 8 per cent. and on the finishing hearth, where the ore was exposed to an almost white heat for two hours and forty minutes, a further increase of 5.4 per cent. occurred — in all 13.4 per cent. — while the loss of silver by volatilization, which took place principally during this period, amounted to 16 per cent. Of the 67.2 per cent. of silver rendered soluble in sodium hyposulphite, 53.8 per cent. was due to the action of the arsenic, and only 13.4 per cent. to the action of the salt. To gain 13.4 per cent. chlorination, 16 per cent. of silver was sacrificed, a very thoughtless operation; and this was by no means one of the worst results. With most of the charges 25 to 30 per cent., and even more, of the silver was lost, to gain a small percentage in chlorination, as is shown further on.

The low chlorination was due to the insufficient supply of air, which prevented the formation of sulphuric acid to satisfy the lime and to act on the salt, while the great loss of silver was caused by excessive heat and insufficient air. The different behavior of the ore in the two furnaces as regards the decomposition of the soluble silver after the salt was added is remarkable, and is thus far unexplainable.

While experimenting with the Brückner furnaces, the roasting in the reverberatory furnaces was conducted in the manner just described, and the experiments with these commenced after those with the Brückners were completed. One preliminary experiment, however, was made, to demonstrate to the metallurgist in charge the effect of the air; for he insisted that the Yedras ore could only be chloridized at a very light-red heat, and he excluded the air as far as possible, so as not to cool the furnace.

As mentioned above, the furnaces were not of proper construction, and the only means of getting more air into them was by leaving the fire-door wide open. The beneficial effect of the air was quite striking, though no change was made in the temperature, the roasting proceeding at the same high heat as before. The interesting record is given in the following tables:

ROASTING WITH 5 PER CENT. SALT AT A VERY HIGH HEAT
WITH CLOSED FIRE-DOOR

Reverberatory Furnace, No. 3

| DATE | VALUE OF RAW ORE PER TON (a) | VALUE OF ROASTED ORE PER TON | SOLUBLE SILVER | LOSS OF SILVER BY VOLATILIZA- TION |
|------------------|------------------------------------|------------------------------------|-------------------|------------------------------------------|
| | Oz. Silver | Oz. Silver | Per Cent. | Per Cent. |
| January 13 | 63.03 | 56.52 | 67.7 | 17.8 |
| 14 | 62.04 | 50.04 | 69.6 | 25.5 |
| 15 | 63.78 | 49.44 | 57.1 | 27.2 |
| 16 | 63.78 | 52.80 | 49.8 | 25.4 |
| 17 | 64.08 | 60.00 | 43.0 | 13.9 |
| 18 | 61.20 | 49.08 | 43.2 | 26.3 |
| 19 | 60.60 | 45.00 | 63.2 | 31.7 |
| 20 | 61.56 | 49.50 | 68.8 | 26.1 |
| 21 | 60.00 | 48.48 | 68.1 | 25.7 |
| 22 | 58.20 | 45.48 | 76.8 | 27.8 |
| Average | 61.82 | 50.63 | 60.7 | 24.7 |

(a) Difference of value between raw and roasted ore, 11.9 oz. per ton.

ROASTING WITH 5 PER CENT. SALT AT A VERY HIGH HEAT
WITH OPEN FIRE-DOOR

Reverberatory Furnace, No. 3

| DATE | VALUE OF RAW ORE PER TON. (a) | VALUE OF ROASTED ORE PER TON | SOLUBLE SILVER | LOSS OF SILVER BY VOLATILIZA- TION |
|------------------|-------------------------------------|------------------------------------|-------------------|------------------------------------------|
| | Oz. Silver | Oz. Silver | Per Cent. | Per Cent. |
| January 23 | 57.66 | 49.56 | 85.3 | 20.6 |
| 24 | 55.50 | 47.04 | 79.7 | 21.6 |
| 25 | 54.36 | 50.40 | 79.6 | 14.3 |
| 26 | 63.30 | 55.20 | 83.9 | 18.0 |
| 27 | 63.84 | 59.22 | 77.7 | 13.9 |
| 28 | 64.86 | 61.08 | 73.5 | 12.6 |
| 29 | 65.52 | 57.00 | 65.5 | 19.2 |
| 30 | 66.78 | 59.40 | 77.2 | 17.4 |
| 31 | 66.00 | 58.80 | 74.5 | 17.3 |
| February 1 | 62.10 | 55.68 | 79.6 | 16.7 |
| 2 | 57.30 | 53.40 | 75.7 | 13.7 |
| 3 | 54.54 | 48.96 | 76.2 | 16.6 |
| 4 | 55.20 | 52.80 | 77.5 | 11.2 |
| 5 | 57.60 | 51.96 | 75.8 | 12.0 |
| 6 | 60.66 | 55.14 | 80.5 | 15.6 |
| 7 | 61.68 | 53.70 | 72.1 | 19.2 |
| 8 | 64.50 | 65.64 | 68.9 | 5.5 |
| 9 | 55.80 | 55.56 | 75.5 | 7.6 |
| 10 | 57.96 | 48.48 | 80.7 | 22.3 |
| 11 | 62.46 | 56.52 | 76.8 | 16.0 |
| 12 | 67.02 | 59.94 | 69.5 | 10.7 |
| Average | 60.70 | 55.02 | 76.4 | 15.3 |

(a) Difference of value between raw and roasted ore, 5.68 oz. per ton.

By comparing the two tables we find that from the very first day the fire-door was kept open a marked improvement in the results took place. The averages show this plainly:

| | |
|--------------------------------------------------|-------------|
| | Per Cent. |
| Average loss of silver—fire-door closed..... | 24.7 |
| Average loss of silver—fire-door open..... | <u>15.3</u> |
| Difference in favor of open fire-door..... | 9.4 |
| | Per Cent. |
| Average of soluble silver—fire-door open | 76.4 |
| Average of soluble silver—fire-door closed | <u>60.7</u> |
| Difference in favor of open fire-door..... | 15.7 |

Thus a reduction of 9.4 per cent. in the loss of silver and an increase of 15.7 per cent. in soluble silver, or a total saving of 25.1 per cent. of the silver in the ore, was effected by simply admitting more air into the furnace.

Preparatory to my experiments with the reverberatory furnaces I reconstructed some of them, to adapt them to the requirements of the ore. The arch was raised to 3 ft. with the exception of the hearth furthest from the fire on which the ore was charged and which was much lower. The working doors and the flue were enlarged, and air-channels constructed to permit air to enter along the fire-bridge and along the back wall of the furnace.

Roasting was then commenced at a much lower temperature, and with a more liberal supply of air, much better results being attained. The average chlorination of one month rose to 81.7 per cent. while the loss of silver was reduced to 1.7 per cent. In the new furnaces the ore fumed over the whole hearth instead of only near the working doors as before, and the temperature was much more uniform in all parts of each hearth. To roast at a low heat requires much greater attention and skill than at a high heat, and it took some time before the men became accustomed to it.

After numerous experiments to determine the proper temperature and draft, the following rules were adopted and will apply to all calcareous sulphureted silver ores:

1. The fumes evolved must be kept in motion in all parts of the furnace. If they stagnate around the ore, or if the furnace assumes nearly a uniform heat throughout its entire length, it is always a sign of insufficient draft, and if the draft is not increased the result will invariably be a high loss of silver and a low chlorination.

2. In no part of the furnace should the ore attain a light-red heat. The fire should be regulated entirely according to the temperature required at the finishing hearth nearest the fire, where the temperature should be kept so that the ore has rather a dark surface if not stirred, but a dark-red heat when the hoe enters it. The ore, however, should fume, and the temperature never go below the point required to evolve fumes. The presence of fumes always indicates that the temperature is not too low, while the dark surface of the ore shows that the temperature is not too high.

An arch still higher than 3 ft. facilitates the maintenance of the conditions prescribed in the second rule, and at the same time permits a stronger fire for the more remote hearths without injuring the charge on the finishing hearth. I rebuilt furnaces Nos. 3 and 4 and made the arch of the finishing hearth 5 ft. high, the sides 3 ft. 4 in., and the fire-bridge 2 ft. high; the next two hearths were raised so that the arch was only 3 ft. high, the sides 16 in.; the next arch 27 in. high, sides 16 in.; and the charging-hearth arch 24 in. high and sides 14 in. (Figs. 4, 5 and 6). These gave the best and most uniform results of any of the furnaces; the proper temperature on the finishing hearth being maintained much easier, because the flame following the roof was so far above the ore that it would have taken an excessive fire to overheat the charge. The chlorination obtained was very satisfactory, and the loss of silver by volatilization was reduced to a minimum. The average results of six weeks' working were: chlorination, 83.8 per cent; loss of silver, 0.8 per cent.

The consumption of wood in these furnaces was larger than in the others: in the reverberatories with 3-ft. arch it amounted to 0.098 cord per ton of ore roasted; and in the furnaces with a 5-ft. high arch 0.32 of a cord more per ton of ore was used; but the better results, and the ease with which the temperature could be controlled, overbalanced the extra consumption of wood.

When the roasting was properly conducted the percentage of chlorination invariably depended, as in the Brückner furnaces, upon the proportion of sulphureted minerals and calcareous gangue of which the ore was composed. The more sulphureted matters the ore contained the better was the result. But little difference was noticed in the formation of lumps whether the salt was added in the battery or in the furnace; in both cases

lumps were formed, but as a rule they were more porous and softer than those of the Brückner, and could easily be leached.

CONCLUSIONS

My experiments in roasting the calcareous and arsenical ore of Yedras have shown:

1. That the main difficulty in chloridizing the ore to a high percentage was caused by the excess of lime in the ore. The ore at Yedras was assorted, as in all silver mines, with the view to obtaining a certain grade in silver, and no attention was paid to the relative proportion of lime and sulphureted minerals, which was of so great importance. In order to obtain regular chlorinations of over 90 per cent. it was necessary that the ore should contain not less than 50 per cent. of sulphurets; but to accomplish this it was necessary to break the ore smaller, so that more of the barren limestone and calcespar could be thrown out, and enable the sorters to pick out a poor grade of second-class ore. Pieces containing much iron pyrites, even if they did not contain silver, should be thrown with the first-class ore.

The accumulation of a second-class ore was not very desirable at Yedras, because it could only be treated by concentration, and the water supply was very limited, except during the rainy season. However, I consider it more rational to accumulate a second-class ore dump at the mine for future treatment than a rich tailings pile at the mill, because it is very difficult to extract the silver from tailings, while the chances are that after further development the mine may produce more water; but even if it should not, the accumulation of second-class ore would not be more than could be concentrated during the rainy season. This question is a very important one and due attention should be paid to it. The outlay would be comparatively small compared with the large amount of silver lost in the tailings.

By improvements in the mode of roasting I succeeded in increasing the chlorination from 65 to 81 per cent. and above, and in reducing the loss of silver from 25 to 1.7 per cent. and less, thereby nearly doubling the production. If the ore had been properly sorted the increase would have been at least 10 per cent. more.

2. I proved that such ores could be roasted with a trifling

loss of silver by volatilization if the roasting was conducted as described above, and that the enormous loss which the ore formerly sustained was caused by too high a temperature and an insufficient supply of air.

3. I demonstrated that such ores, properly assorted, could be successfully and very cheaply roasted in the Brückner furnace by the self-roasting process, if the salt is added to the ore in the battery, provided suitable provisions be made to separate and pulverize the hard but well-chloridized balls which form in these furnaces.

PART II
EXTRACTION OF THE SILVER

XIII

LIXIVIATION WITH SODIUM HYPOSULPHITE

PERCY and Hauch were the first who proposed to convert the silver in ores into chloride by roasting with salt and to extract the silver with a solution of sodium hyposulphite, but Von Patera was the first to apply this method in actual practice by treating with it the rich and complex silver ores of Joachimsthal, Bohemia. These ores were formerly treated by amalgamation and also by the Augustin method, which both required chloridizing roasting; but, on account of the great loss of silver which these ores sustained during chloridizing, only the poorer grades were worked by these methods. Patera, however, changed the mode of roasting, inasmuch as he applied steam, whereby the loss of silver by volatilization was much reduced, and made it possible that even the richest varieties of the Joachimsthal ore could be successfully chloridized. By using sodium hyposulphite as solvent he introduced a process much cleaner and superior to either the amalgamation or Augustin method. It was, however, executed on a rather small scale; the roasting was done in charges of 500 to 600 lb., and the lixiviation was performed in tubs holding about 200 lb. of roasted ore.

This process was introduced at Joachimsthal in 1858. Ten years later, in the fall of 1868, I introduced it successfully at La Dura, Sonora, Mexico, on a large scale, and in the following years at Trinidad, Las Bronzas, San Marcial, in Sonora, Mexico; then at Silver King, Arizona; Monitor, California; Cusiuhiriachic, Chihuahua, Mexico, and at other places.

In course of time, as experience with different ores was acquired, the process was much improved chemically, as well as with regard to appliances.

The principle on which this process is based is the property of silver chloride of being insoluble in water, while it dissolves

readily in a solution of sodium hyposulphite, even if the solution is very dilute. To convert the silver, which in nature mostly occurs as sulphide, into chloride, the sulphide ore is roasted with salt (sodium chloride), as we have seen in the first part of this treatise. The solution dissolves only that part of the silver which is converted into chloride, and the extraction depends, therefore, entirely upon the quality of the roasting. The chloridizing roasting is undoubtedly the most important part of the process; and it cannot be too strongly impressed upon the minds of operators to pay their principal attention to this operation, and to make a thorough study of the behavior and characteristics of their respective ores.

Before going into the details of the different operations it will be more instructive to give a short description of the process. In the first part of this treatise we have seen that, if a complex ore be subjected to chloridizing roasting, not only the silver sulphide but also the other metal sulphides are converted into chlorides, subchlorides, sulphates and oxides. A number of these salts are soluble in water and have to be removed from the roasted ore by leaching with water before silver extraction, in order to prevent them from entering the silver solution and afterward the silver precipitate. This operation is termed "base-metal leaching." The leaching with water, however, dissolves also undecomposed salt, if such should be present, and the sodium sulphate which was formed in roasting.

If by test it is ascertained that the soluble chlorides and sulphates have been removed from the ore, a diluted solution of sodium hyposulphite is applied on top of the ore. This solution by descending through the ore dissolves the silver chloride. This operation is termed "silver leaching." The outflowing solution is collected in special tanks (precipitation tanks). To this silver solution a solution of calcium or sodium sulphide is added, by which the silver and some other metals present are precipitated as sulphide. By agitating the solution the precipitate collects in dark-brown flakes, which settle quickly to the bottom of the tank. When settled the clear solution is decanted and returned to the process, and is used over and over again indefinitely. The precipitate is filtered, washed, dried and charged into a small roasting furnace to burn off the excess of sulphur which it contains. This is done at a very low heat, and only as

long as the blue sulphur flame lasts. This free sulphur in the precipitate is derived from the precipitant. The calcium and sodium sulphides are both used as polysulphides. In precipitation only one atom of sulphur combines with the metal, while the balance drops down as such.

The calcined silver precipitate is finally melted on a lead-bath in a cupeling furnace and refined, or it is shipped and sold to smelting works.

After this synopsis of the process, I will proceed to describe and discuss the different operations and appliances.

BASE-METAL LEACHING

The roasted ore is charged into wooden tanks, of which each is provided with a filter bottom. The size of these tanks depends on the intended working capacity of the works, the filtering quality and the chemical nature of the roasted ore. Deep tanks should never be used; in fact, the rule should be observed not to make them deeper than 5 ft., and the diameter not to exceed 16 to 20 ft. Tanks with too large a diameter are difficult to discharge. The reason why tanks should not be made deeper than 5 ft. will be given below.

Construction of the Leaching Tank. — Fig. 31 represents a vertical and Fig. 32 a horizontal section of a leaching tank 16 ft. in diameter and 5 ft. deep, made of 3-in. clear lumber. To use lighter lumber than 3 in. is not an economical policy. The lumber should be red wood or cedar, preferably the former, because it is more indifferent to the action of chemicals than any other soft wood. The bottom can be made of 12-in. plank, but the staves ought not to be made over 6 in. wide, in order to facilitate binding by the hoops. It is very important that the tank should be tight in order not to lose by leakage valuable silver solution, and therefore it should be made very carefully. From the factory the parts are furnished in crated bundles and the edges suffer considerably in transportation; besides, these edges have usually only a rough saw-finish, and in putting the tank together they ought to be planed by a skilled carpenter. For this purpose several extra staves are always sent with the shipment. The bottom planks should be kept in position by wooden pins. Before erecting the tank the timber on which it is going to set should be prepared, then the bottom planks placed

on top of it and pinned together, then the staves with their groove slipped into the edge of the bottom and driven tight. The groove in the staves should be $\frac{3}{4}$ in. deep. The tank with its weight has to rest on a circular bed of timber specially prepared, which must be thick enough to leave the staves clear and free from weight. Timbers 6 x 6 in. will answer; but they ought not to be further apart than 12 in., as shown in Fig. 31. The best hoops are those made in two or three sections of $\frac{7}{8}$ or 1-in. round iron, and

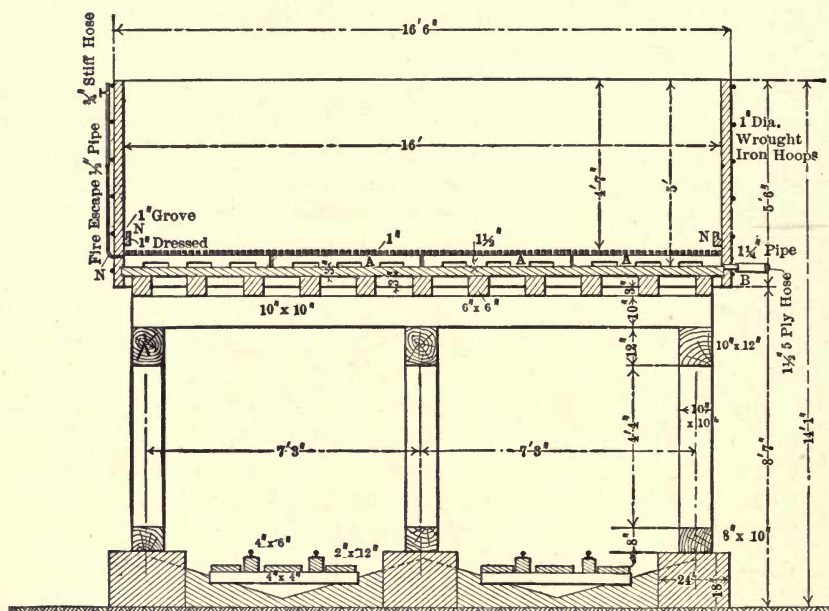


FIG. 31.—LEACHING TANK, VERTICAL SECTION.

provided with cast-iron lugs for tightening. They resist chemical action for a long time, while flat hoops are quickly destroyed. It is well to paint them with two coats of asphaltum varnish before placing them. One of them should be placed at a level with the bottom, or only a very little below, so as not to interfere with the outlet pipe of the tank. The hoops should be 12 in. apart. The bed on which the tank rests should be made so that the tank inclines toward the front 1 to $1\frac{1}{2}$ in.

If a tank has been in operation for some time it will be found that some material has accumulated under the filter bottom,

which material has to be removed from time to time, so as to permit a full flow of the filtrate from all parts toward the outlet. To facilitate this cleaning the filter bottom should be made in sections, small enough to be handled by two men. To the pieces *A*, (Fig. 31), which are 4 in. high and $2\frac{1}{2}$ in. wide, 1×1 in. laths are nailed 1 in. apart. Square notches cut into the pieces

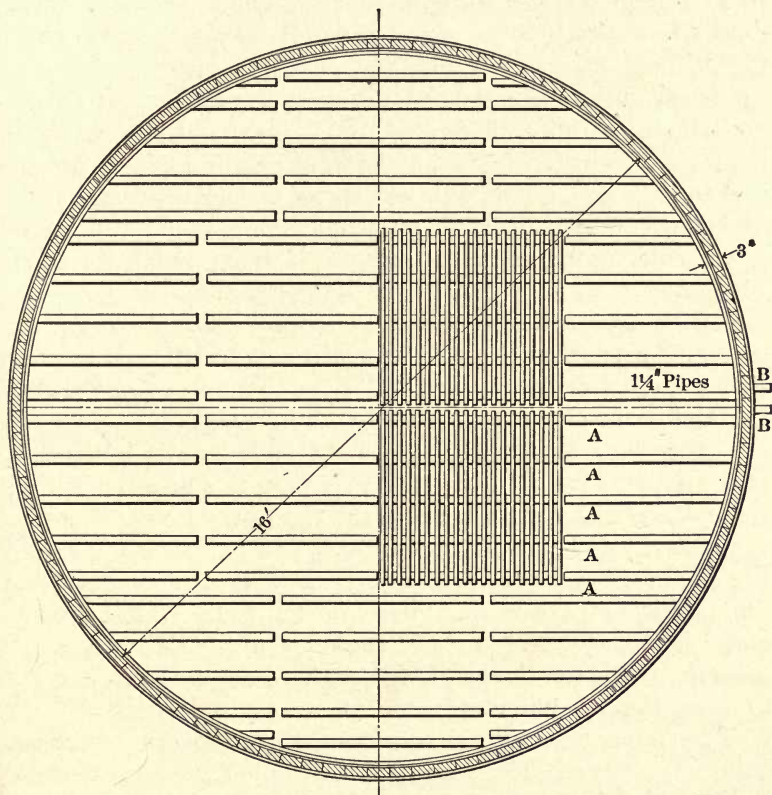


FIG. 32.—LEACHING TANK, PLAN.

This shows filter bottom arranged in fourteen sections. Two sections are complete.

A permit the flow of the solution between the sections. Fig. 32 illustrates the arrangement of these sections, showing two of them completed. All the lumber of which the filter bottoms are made should be planed and painted with asphaltum varnish, and the heads of the nails well driven into the laths, and covered with white lead and asphaltum. The upper edges of the laths

should be beveled to prevent them from cutting the filter cloth. For economical reasons it is well to order the laths together with the tanks, and not attempt to make them at the mine.

About one inch above the filter bottom the tank is provided with a groove. This groove is made by nailing first a wooden strip 1 x 2 in. around the inner side of the tank, then by nailing to this strip another 1 x 4 in. This gives a groove 1 x 2 in. deep. It serves to tuck in the ends of the filter cloth. *B, B*, Figs. 31 and 32, are the lead outlet pipes. It is not advisable to use larger outlet pipes than 1½ in. because they have to be connected with a rubber hose about 5 ft. long, and if they are of too large a diameter, flatten, and do not permit a flow in proportion to their size, as the solution has no pressure. Over a 1½-in. lead pipe a 1½-in. hose can be pushed, and if it is 5-ply will remain round and discharge a full stream. If the solution flows out with force it is a sign that the outlet is not large enough, and a second or even a third one has to be provided. To check the stream is a poor practice, because a free percolation is of great advantage, as well with regard to the extraction as to the time required for it.

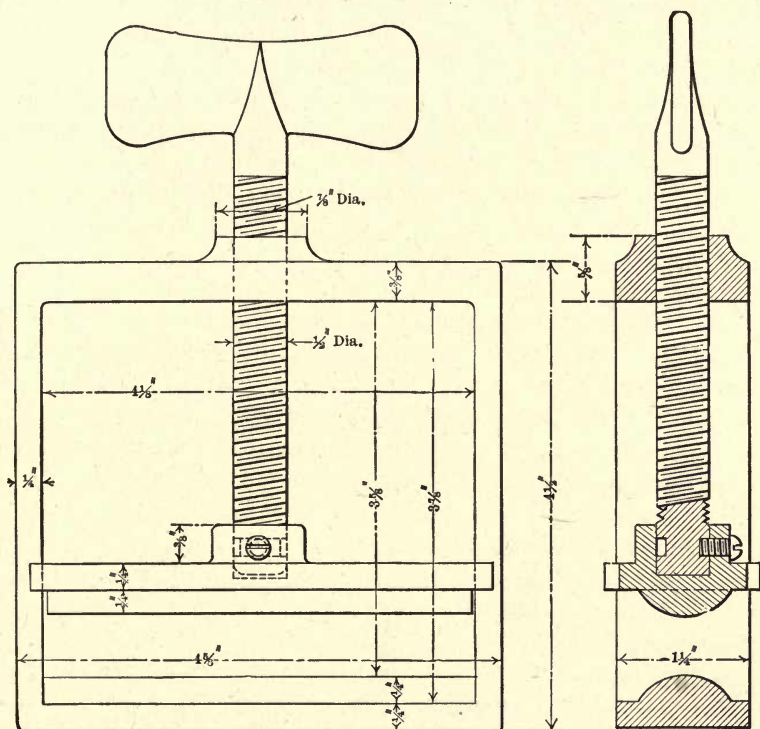
Under but close to the top of the filter bottom is the ½-in. or ¾-in. lead pipe *N*, Fig. 31. This pipe is connected with a rubber hose which extends up to the rim of the tank. The hose can be closed tight by a clamp-screw. This pipe serves to permit the air from under the filter to escape. This is of importance if the working of a tank has to be interrupted. If the solution outlet is stopped, the air under the filter will be brought under pressure by the solution, and, if there is no escape for it, it is apt to break through the filter.

I have found the use of hose clamps in lixiviation works not only very convenient but also economical. Valves are more or less corroded by the solutions and are of short life, while the clamp, doing the service of a valve, does not come in contact with the solutions. These clamps are best when made of brass. They are not in the market and have to be made to order. Figs. 33 and 34 give the construction and size of a clamp for a 1½ to 2-in. hose. Smaller sizes for smaller hose have to be kept on hand also.

The tanks ought not to be placed on a solid terrace, as we often find done; it is much more rational to place them on

trestles, because a leak in the tank is easily discovered, and, besides, it gives room underneath to run cars for the removal of the residues.

Solubility of Silver Chloride in Base-Metal Chlorides and Salt. — The base-metal chlorides as well as sodium chloride have the property of dissolving silver chloride, their dissolving energy increasing with the temperature and concentration. It is there-



FIGS. 33 and 34.—BRASS CLAMPS FOR 1½- AND 2-IN. HOSE.

Full size.

fore not advisable to leach with hot water or to use too deep leaching vats, even if the filtering quality of the ore permits it, because the water by passing through a thick layer of ore becomes highly charged with these salts and considerable silver chloride will be dissolved. A very dilute chloride solution does not dissolve silver chloride, and in order to prevent the base-metal chlorides from dissolving silver, the washing of the ore should be

so conducted that a sufficiently dilute solution will be produced.

Prevention of Loss of Silver in Base-Metal Solutions. — In tank lixiviation this can only be accomplished to a certain degree (not perfect) by two methods which were introduced by myself: (1) The water is allowed to enter the vat below the filter until it gradually rises above the ore. Thus the most concentrated portion of the solution will appear above the ore, and if then diluted with a stream of water and the course of filtration reversed a large part of the silver chloride will be precipitated and remain in the ore, while the outflowing solution will be dilute, although not sufficiently so as to be entirely free of silver. (2) A quicker but not quite so effective a method is to fill the tank partly with water and then to dump the ore into it, either dry or moist. Enough water should be taken to rise about 4 in. above the charge when complete. By this method the outflowing solution will not be so dilute as by the first, but much more so than it would be if the ore were charged in an empty tank and then leached from above. However, the only method by which this problem is actually solved is trough lixiviation, also introduced by me. By this operation the ore can be brought at once in contact with such quantities of water as to produce a solution so dilute that not a trace of silver will be dissolved. More about this method of lixiviation will be said further on.

The amount of silver dissolved during base-metal leaching varies greatly, and depends on the nature of the ore, the thickness of the layer, the amount of salt used in roasting, the mode of treatment in the vat, and the temperature of the outflowing solution. In some works, usually in large ones, the ore is charged steaming hot from the cooling floor, or if for special reason it be charged dry it will enter the tank even hotter, and as a matter of course the first part of the outflowing solution will be very hot and contain considerable silver. To illustrate how the amount of silver dissolved by the base-metal solution varies under different conditions, a few examples may be given:

San Francisco del Oro, Parral, Mexico. — Heavy zinc-lead ore with iron pyrites, roasted with 4 per cent. salt; the tanks partly filled with water and the ore dumped dry and cool into it; average value of roasted ore 26.1 oz. per ton; charge, 8 tons; leaching time, 8 hours; silver dissolved, 1 per cent.

Sombrerete, Zacatecas, Mexico. — Heavy ore containing zinc blende, galena, iron pyrites and some sulphureted copper minerals, roasted with 6 per cent. salt; the tank partly filled with water and the ore dumped dry and warm into it; average value of roasted ore, 42.6 oz. silver per ton; charge, 52.5 tons; leaching time, 12 hours; silver dissolved, 1.23 per cent.

Cusihuiriachic, Chihuahua, Mexico. — Not so heavy an ore, containing galena, zinc blende, some iron and copper pyrites, some silver-copper glance and also some ruby silver; roasted with 8 per cent. salt; the ore wetted on the cooling floor and charged slightly warm into an empty tank and leached from above; average value of roasted ore, 47.9 oz. silver per ton; charge, 8 tons; leaching time, 53 hours; silver dissolved, 2.5 per cent.

Determination of Amount of Silver Dissolved. — Mr. Russell reports the amount of silver dissolved from the Cusihuiriachic ore as 11.6 per cent. if the ore is charged cold and dry and leached from above. Though Mr. Russell roasted with 10 per cent. of salt I believe his figure to be much too high. His method of ascertaining the amount of silver dissolved is not correct. He determines on a sample in the laboratory the amount of salts soluble in water contained by the roasted ore, and from this figure he calculates the amount of silver which ought to be contained in the ore after leaching with water, compares it with the actual amount found in it by assay, and then estimates the amount of silver dissolved. This is not correct, because a large charge in the works cannot be so thoroughly washed as a small sample in the laboratory, and therefore the calculated value of the washed ore will be much too high, and consequently the calculated amount of silver dissolved.

The correct way to ascertain the amount of silver dissolved is to collect separately in vats the whole base-metal solution of one charge and measure its volume. If it be then determined how much silver is contained in 1000 c.c., it is an easy matter to calculate the total amount of silver dissolved. To take the sample out of the vats would not be correct, because the solution being collected from different vats will not be uniform in concentration and therefore will not be uniform as to tenor of silver. Besides, a large portion of the silver will be precipitated by the gradual dilution of the first concentrated brine, which, to a certain extent, will escape the sample, even if the contents of the

vat be agitated. However, there is no difficulty in obtaining a correct sample. A small rubber tube, terminating with a glass tube drawn to a fine point, is inserted in the outlet of the vat and left there during the whole time of base-metal leaching. Thus a very fine stream of the outflowing solution is obtained, and if collected in a glass vessel of proper size, a representative sample of about two or three gallons of the whole solution is obtained. The volume of this sample is correctly measured and all the heavy metal salts contained therein precipitated with calcium or sodium sulphide. The precipitate is collected on a filter, dried, weighed and assayed, and the amount of silver contained in it is calculated. With this figure and the total volume of solution collected in the vats the total amount of silver dissolved during base-metal leaching may be estimated, and by knowing the weight of the charge the amount dissolved per ton can be calculated and expressed in percentage. The figures of silver dissolved under different conditions quoted above were obtained by this method and therefore may be considered correct.

Effect of an Excess of Salt. — If the proper amount of salt be used in roasting, so that all, or nearly all, be decomposed, and the leaching be done by one of the improved methods, it may be assumed that the amount of silver dissolved will never exceed 3 per cent. An excess of salt will cause a large percentage of silver to be dissolved, reaching 60 to 70 per cent. in my own experience. In this case the ore was of such a nature that it required a large excess of salt in roasting to produce a satisfactory chlorination of the silver. This very interesting occurrence will be discussed below.

Precipitation of Silver from Base-Metal Solutions. — There are several ways of precipitating the silver contained in the base-metal solution, including precipitation by dilution with water, to be described further on, and precipitation by one of the following reagents: (1) Milk of lime. This produces a very voluminous precipitate which is rather difficult to handle, and from which the silver cannot be extracted unless there is a smelting furnace connected with the leaching works. In such a case lime may be used to advantage, especially since it is so cheap a reagent, but care must be taken not to use the base-metal solution too concentrated, in which case it is coagulated and much trouble in further handling is caused. (2) Calcium or sodium sulphide.

These precipitants are used in most of the lixiviating works, but they are rather expensive, because they precipitate base metals also. If sodium sulphide be added gradually to a tank charge of base-metal solution, it will be observed that the precipitate formed first is the richest in silver, and that it becomes poorer as precipitation goes on. A complete precipitation, however, can be produced only when all the base metals are thrown down. Since the principal portion of the silver is precipitated in the beginning, and only the smaller portion remains in solution with the main bulk of the base-metal salts, the point will be reached in precipitating when the cost of the precipitant will exceed the value of the precipitated silver, and therefore a complete precipitation is not advisable. In most lixiviation works only a partial precipitation is performed, after which the solution with the remainder of the silver, which it did not pay to precipitate, is allowed to run to waste.

Recovery of Silver from Waste Liquor by Precipitation with Copper. — Although the waste liquor will be found very low in silver, still if its large volume be taken in consideration it will appear that the loss occurring in this way during a year is important. This loss, however, can be diminished greatly by precipitation with copper, which acts most energetically if finely divided like cement copper, and if the solution be warm. To effect a perfect desilverization it is necessary to treat the heated solution in charges and to use mechanical appliances to produce an intimate contact with the cement copper, but the base-metal solution, after a partial precipitation with calcium or sodium sulphide, is not rich enough to warrant an expensive and complicated treatment, and it will be found to be more profitable to employ a more primitive method, even if not all of the silver is recovered. The true economy in metallurgy, as in any other industry, is to save the most at the least expense. As soon as its expense exceeds the value of the recovered metal, a method ceases to be practicable, no matter how interesting it may be. That method should be adopted which is best suited to local conditions.

The largest part of the dissolved silver can be saved by conveying the solution through a series of flat tanks in which cement copper is so divided that it offers a large surface of contact to the solution. The exhaust steam of the engine can be used to increase the temperature of the stream, the steam being made

to enter the solution through pipes about 12 to 18 in. below the surface in different tanks of the system. If the base-metal solution contains copper, the pipe projecting into the solution ought to be of lead. The steam being condensed a vacuum is formed, and very little if any back pressure to the engine will be noticed. Wherever the climate allows it these tanks can be built in the yard outside the works, without roof or shelter. If the ore contains copper, sufficient cement copper will be formed by placing scrap iron in the tanks, but if copper is wanting, the cement copper has to be made from a dilute solution of blue vitriol. Ores which are treated by lixiviation are seldom entirely free from copper, and for this reason it is well to place some scrap iron with the cement copper.

Base-Metal Leaching at Sombrerete. — At Sombrerete, Mexico, I conducted the base-metal leaching in the following manner: The ore is very permeable, and Stetefeldt and Russell, who built the works, erected leaching vats 15 ft. 6 in. in diameter and 7 ft. 6 in. deep, capable of receiving 55 to 58 tons of roasted ore. These tanks were decidedly too deep. The ore filtered freely enough to permit a deep charge, and in silver leaching this would not interfere, but in base-metal leaching it caused much trouble. The ore was not wetted on the cooling floor to produce additional chlorination, but was charged dry. The water passing through 7 ft. of roasted ore became so saturated with salts that they crystallized and blocked the filter, the space below the filter, and the outlet pipe, which interfered greatly with the work and was very annoying. I overcame this difficulty by filling the vats to the depth of about 3 ft. with water and dumping the ore into it; frequently the ore had to be charged while quite hot in order to make room on the cooling floor, and consequently the outflowing base-metal solution was warm and dissolved more silver chloride than it would have done if more favorable conditions could have been maintained. But notwithstanding this, the amount of silver dissolved was much less than when the ore was leached from above.

If a base-metal solution containing silver chloride be diluted with water the silver chloride will precipitate as such. In the Sombrerete works there were two base-metal precipitation vats 9 ft. 9 in. in diameter and 9 ft. deep; and in order to take advantage of the above reaction I allowed the base-metal solution to

run into both simultaneously, by which method I got the concentrated portion of the solution, which contained the most silver, divided evenly into the two vats. This left considerable room in them for diluting, and by allowing the less concentrated and, finally, the very weak solution to run evenly into both vats, I obtained a uniform solution dilute enough to cause the main portion of the dissolved silver chloride to be precipitated. Silver chloride thus precipitated being very finely divided requires a long time to settle. For the purpose of effecting quick settling and at the same time precipitating more silver, 5 to 10 gal. of sodium sulphide solution were added to each vat and agitated. In this way a part of the silver chloride was converted into sulphide, while the undecomposed part was readily collected by the flocculent precipitate of copper, lead and other base metals, and after agitation was interrupted the whole precipitate would settle quickly.

After the precipitate has settled, the clear solution is decanted by a stiff rubber hose, which enters the vat close to the bottom and projects above the surface of the liquor. By lowering this a few inches below the surface the liquor flows out, and if the hose is lowered gradually as the liquor runs out it is always the clearest part of the latter which is drawn off. This method of decantation is better than a series of tubes inserted at different levels in the side of the vat, because during precipitation these tubes are filled with precipitate, and in decanting the first solution from each tube contains precipitate and consequently has to be conveyed to filters.

Outside the building were constructed a series of flat square tanks $2\frac{1}{2}$ ft. deep, built of stone and mortar, well plastered and coated with asphaltum varnish, which were provided with a number of movable wooden double benches, or shelves, loaded down with scrap iron. In this way the scrap iron was well distributed throughout the tanks and offered a large surface to the solution. They were so arranged that the flowing solution would move in its whole depth and had to take a zigzag course.

The ore of Sombrerete contains about 2 per cent. copper, and the outflowing solution for a considerable time is colored green by cupric chloride, and a good deal of copper is precipitated by the scrap iron. The precipitated cement copper incrusting the iron is loose and spongy and offers a very large surface for pre-

cipitating the silver. It ought not to be disturbed by stirring the solution, since thereby the copper falls to the bottom and does much less service. However, it is difficult to precipitate all the silver, and it requires a rather large number of tanks. Silver dissolved in a cupric chloride solution cannot be completely precipitated until all the cupric chloride is decomposed; therefore as long as the solution leaving the last tank still gives a reaction for copper it can be assumed to contain some silver also. This indicates the necessity of erecting one or more additional tanks. In Sombrerete these tanks were cleaned once a month, and the cement copper obtained from them contained 500 to 600 oz. silver per ton, and 60 to 70 per cent. copper.

Cupric chloride in solution in contact with cement copper, especially at an elevated temperature, is converted into cuprous chloride, which settles as a heavy white crystalline precipitate. Cuprous chloride again in contact with metallic iron is converted into metallic copper, and the iron into ferrous chloride. It is clear that in these tanks, in which the cement copper surrounds the iron, an intimate contact between the iron and the cuprous chloride is not possible, and that consequently the cement copper will contain a large percentage of cuprous chloride. This is not a desirable associate for the cement copper, because in the subsequent treatment it always causes a large loss of copper and silver, and therefore the cuprous chloride ought to be decomposed. This should be done shortly after the cement copper is taken out of the tanks, because if left exposed to the air the cuprous chloride will change into oxichloride. The quickest and most rational way is to charge the cement copper in a revolving barrel with an addition of water, salt and some light scrap iron. The barrel is made of 3-in. staves and heads, and is bound with copper hoops. It is constructed like an amalgamation barrel. The inside is lined with hardwood lath about $1\frac{1}{2}$ in. thick, so that the body of the barrel is protected from the wear and tear. Four longitudinal ribs, projecting about $2\frac{1}{2}$ in., are inserted diametrically in the lining, which produces a lively mixing of the charge. The axles, or trunnions, which are made of brass, or better of bronze, are provided with a strong flange by which they are bolted to the heads of the barrel. Each is bored and provided with a stuffing-box through which a copper pipe enters. One of the pipes is turned downward close to where it enters, but must

not project so far into the lower half of the barrel that the scrap iron and the cement copper will strike it. Through this pipe steam is introduced to heat the pulp. The pipe that passes through the other trunnion is turned upward high enough to just clear the ribs of the barrel. This pipe serves as outlet for the gases and steam and leads outside the building. A manhole on the side of the barrel, provided with a copper frame and lid, serves for charging and discharging, while another in the head gives entrance in case the lining is to be renewed or other repairs are to be made.

When the barrel is charged with cement copper, water and iron, salt is added, steam turned on, and the barrel is put in revolution. A warm solution of salt (sodium chloride) dissolves readily cuprous chloride, and from this solution the iron precipitates the copper with great energy. Water should be used moderately, just enough to produce a lively movement of the pulp, say three to four times the volume of the cement copper, and to produce a strong brine without using too much salt. The reaction creates heat and the escape pipe has to be watched. When the steam commences to come out freely the steam inlet ought to be closed. The side of the barrel is provided with a plug-hole through which the operator can take a sample by means of a flask fastened to a copper wire. If the filtrate does not show a reaction for copper the process is finished. This takes about 45 minutes. Below the barrel is placed a square flat filter tank, above which is a large, strong iron screen with $2\frac{1}{2}$ -in. holes, the frame of which rests on four car-wheels on rails, so that it can be moved easily or withdrawn entirely from the tank. The contents of the barrel are discharged on this screen, and the cement copper washed into the filter tank by a stream of water, while the scrap iron remains on the screen. Then some sulphuric acid is added to remove the basic iron. This acid solution is allowed to pass through the cement copper, after which the latter is well washed. By this method I obtained cement copper containing 90 to 95 per cent. copper.

If the cement copper has to be prepared from blue vitriol it may be desirable to use it for a longer time, in which case it will become much richer in silver, but it has to be taken out occasionally and treated with dilute sulphuric acid to remove the basic salts. Thus purified the cement copper acts again with energy.

In order not to interfere with the regular work, the tanks have to be so arranged that during cleaning half of them remain in operation while the others are disconnected and cleaned. As soon as half are cleaned, the scrap iron or cement copper is put in place and the solution allowed to enter again while the other tanks are cleaned.

Precipitating the Dissolved Silver Chloride by Dilution with Water. — This method was first recommended and introduced by me. All alkaline and metal chlorides have the property, when concentrated, of dissolving silver chloride, and dropping it again as such when diluted with water. The precipitation takes place in proportion to the dilution, and if sufficient water be added all the silver will be precipitated. This method of desilverizing the base-metal solution is undoubtedly the most effective and cheapest, all that is required being a sufficient supply of water and a few more vats for base-metal precipitation than are usually found in lixiviation works. I used this method first at the Silver King mill in Arizona in 1880 to 1882 with very good results, then in various other localities where the supply of water permitted it, and also at the mill of the Hidalgo Mining Company, at Parral, Mexico, in 1894.

Experience at Parral, Mexico. — The observations I made at Parral were very interesting, and I shall give them in detail. The ore which was treated consisted principally of galena, lead carbonate, and blende, and was almost free from iron pyrites. Neither galena nor blende produces in roasting with salt much chlorine, especially if the blende belongs to that variety which contains little or no iron sulphide. In this case the chlorination of the silver depended principally on the chlorine produced by the action of the quartz on the salt and by the direct action of volatilized sodium chloride. Such roasting requires a large excess of salt and a high heat. The roasting was conducted in a White-Howell furnace with 9 to 10 per cent. of salt. Experiments were made with the aim of reducing the proportion of salt, but as soon as the amount fell below 9 per cent. the extraction suffered so much that it was more rational to use a higher percentage of salt. If the ore had contained sufficient iron pyrites 3 to 4 per cent. of salt would have been sufficient, because lead-zinc ores in presence of iron pyrites require less salt for a successful roasting than any other class of complex ores; but since the iron pyrites

was wanting and an excess of salt had to be used, and since zinc blende and galena act but slightly on the salt, the roasted ore contained a large amount of sodium chloride.

The lixiviation mill of the Hidalgo Mining Company is well constructed and arranged, and reflects credit on J. T. Long, who erected it. This mill has a very large cooling floor, on which the roasted ore is allowed to cool dry for three days. Notwithstanding this long time, the ore, when charged, is still hot enough to impart to the outflowing base-metal solution for a rather long time a temperature of 140 to 200 deg. F. The excess of salt contained in the roasted ore dissolved readily in the water of this temperature, forming a highly concentrated brine in which 60 to 70 per cent. of the silver was dissolved, and therefore the principal extraction was done during base-metal leaching. This, of course, required a careful treatment of the base-metal solution.

Sodium sulphide was used as precipitant, but a large quantity was required on account of the large amount of lead contained in the solution. If a zinc blende ore containing iron pyrites is roasted with the proper amount of salt and charged cool into the vat the base-metal solution will contain but little lead, if any, provided cold water is used; but if a concentrated hot solution of sodium chloride is formed in leaching, the result is entirely different, since this dissolves not only lead chloride, but also lead sulphate, and the solution will contain large quantities of these lead salts. If the solution be very concentrated, and allowed to cool, large crystals will be formed, while the mother liquor remains clear. If the solution be not very concentrated there will be only a crystalline precipitate which turns the solution milky. Neither the crystals nor the crystalline precipitate contain much silver. The main part of the silver chloride, lead sulphate and lead chloride remains in the solution, but if this be diluted with water it turns milky, forming a heavy white precipitate; and if sufficient water is used the white precipitate will contain all the silver and lead dissolved in the solution before diluting.

For illustration I shall record my observations made on a certain charge. The ore when charged was rather hot. The leaching was done from above. The solution flowing out first had a temperature of 200 deg. F. and was unusually concentrated. Some solution was collected in a large beaker and left to cool. A large amount of transparent crystals was formed. After pouring

off the clear mother liquor a quantity of water was added to the latter, which caused a heavy white precipitate. This was separated from the solution by filtration, and the three substances were assayed for silver with the following results: (1) The crystals contained 7.6 oz. silver per ton; (2) the white precipitate had 3386 oz. silver per ton; (3) the filtrate had none. Of this charge of ore, which contained 19.6 oz. silver per ton, 62.6 per cent. of the silver was extracted during base-metal leaching.

Working by this method I found it more convenient to do the precipitation in the trough leading from the lixiviating vats to precipitation vats, by allowing a stream of water to enter the trough. In flowing some distance the solution and water become thoroughly mixed, and the solution is perfectly desilverized before it enters the vats. The stream carries along the white precipitate, which settles in the precipitation vats. To ascertain whether the solution receives the proper quantity of water or not, a sample of the diluted solution just before it enters the vat is taken and filtered. Of the filtrate 50 c.c. are poured into a beaker and 150 c.c. of cool, clean water are added. If it remains clear after a few minutes it shows that no more water is needed; if it turns milky the stream of water has to be increased. Thus by decreasing or increasing the stream the proper dilution can be maintained. Of course the base-metal solution can also be diluted in the precipitation vats, but by this method, if the two streams simultaneously enter the vat, it is difficult to ascertain the proper proportion, and if part of the vat is filled with solution and then water is added there will be a considerable loss of time. Where mechanical devices are used for agitation it is advisable to have the solution agitated while the vat is filling; where agitation is performed by hand, it ought to be done from time to time, to make the precipitate settle more quickly. The tanks are filled to about 12 to 18 in. below the rim, to leave room for more water in case the proper proportion was not maintained in the trough and a correction is necessary. When this is done a short time is allowed for the heavy part of the precipitate to settle, then one quart of sodium sulphide is added and the solution agitated again. This is done to convert the very fine particles of the precipitate, which otherwise would remain suspended for a long time, into sulphide, in which state they assume a flaky condition and settle much more quickly.

The lixiviation tanks were charged with 30 tons of roasted ore, and the base-metal leaching required about twenty-four hours. If the precipitation was not done by water but by sodium sulphide, the base-metal solution filled $7\frac{1}{4}$ tanks of 3500 gal. each, and 53 gal. of the precipitant were used. The solution running to waste still contained 0.19 oz. silver per 1000 gal., or 0.15 oz. per ton of ore leached. If the precipitation was carried so far that all the base-metals were precipitated, the solution running to waste did not contain any silver, but then the cost of the precipitant exceeded the value of the silver saved. If the precipitation was effected by dilution with water in the trough, 22 precipitation charges of 3500 gal. each were filled. The consumption of sodium sulphide amounted to 2.75 gal. only, and the clear solution running to waste did not contain any silver at all. This result shows that it requires on an average about two parts of water to one part of solution. Of course, in the beginning, when the out-flowing solution is very concentrated, considerably more water has to be added than in the above proportion, but the volume of water to be added decreases as leaching progresses, and toward the end it is less than the volume of solution.

On account of the large quantity of lead which the base-metal solution carried, and which is precipitated by water as well as by sodium sulphide, there was not much difference in the silver contents of the precipitate whether obtained by water or by sodium sulphide. In this particular case, however, a great difference in the silver contents of the precipitate was observed if the water for leaching was first applied below the filter or from the top. In the former instance it contained 1200 oz. silver per ton; in the latter only 800 oz.

The precipitation with water is undoubtedly the cheapest and most effective mode of treating the base-metal solution, and it ought to be used in all works where there is an ample supply.

The Use of Cupric Chloride During Base-metal Leaching.—The chlorination of the silver, and correspondingly the extraction, can be increased greatly in ores which do not contain any or only a little copper, by use of a solution of cupric chloride while base-metal leaching is in operation. The cupric chloride can be prepared in the works by boiling one part of blue vitriol with two parts of salt in a wooden vat by direct application of steam. The cupric chloride is made in large quantities and used as re-

quired. It takes from 3 to 4 lb. of blue vitriol and 6 to 8 lb. of salt per ton of ore. To avoid the formation of too concentrated a solution the cupric chloride is not added until the leaching has proceeded one or two hours. It is better not to add the whole of it at once, but to divide it so that one-fourth is added to about 8 in. of water above the ore; and when this charge is disappearing below the surface another charge of 8 in. water and the second fourth of the cupric chloride is made, and so on until all is used. If the outflowing solution is green, it ought to be collected in a separate vat, and can be used over again either for the same charge or for the next one following, according to circumstances. It is best to lift this solution by means of a lead steam siphon or injector. If an ore contains so much copper that the base-metal solution is colored green, this reaction, which produces an additional chlorination of the silver, takes place in every charge in the regular operations of the process, and this is one of the reasons why cupriferous ores are the easiest to treat and permit the closest extraction. However, even with such ores, better results are obtained if the green base-metal solution is collected separately and allowed to pass through the charge a second time. Any defect in the roasting will be much lessened, and uniformly good results will be obtained. The more inferior the roasting the higher will be the percentage of gain in chlorination. At Parral, where I treated the San Francisco del Oro ore containing 25.5 per cent. zinc, 11.56 per cent. lead and 1.02 per cent. copper, on imperfectly roasted charges by the use of cupric chloride 32 to 38 per cent. in chlorination were gained. At the Silver King, Arizona, where I treated an ore containing a large percentage of copper, I increased the extraction a little by using part of the green solution over again. With ore from the Veta Grande of Parral, which contained mostly zinc blende, galena, and lead carbonate, and no copper, I experimented on a large scale with cupric chloride, and obtained much better and more uniform results.

SILVER LEACHING

After the ore is freed from base-metal salts by leaching with water, a weak solution of sodium hyposulphite is introduced on top of the ore. It is of great importance to work with a weak solution. If the solution is strong, 1, 2, or 3 per cent., and an attempt is made to maintain this as standard strength, it can be

done only by frequent additions of sodium hyposulphite; else the volume of the stock solution will decrease rapidly and the preparation of a new stock solution will soon be required, because the ore absorbs a large volume of solution, and when, after the silver extraction is concluded, this solution is displaced by water, it is impossible to regain the same volume of the original strength. It will be found that only a comparatively small portion of the outflowing solution will have the standard strength, and if the displacement of the solution is done so as to maintain the same volume of stock solution, much water will enter the latter, while considerable hyposulphite salt will remain in the residues. Such a dilution of the stock solution takes place also in the beginning of the silver leaching. The "hypo" solution passing through the washed ore has to displace the water absorbed by the ore, and this cannot be accomplished without the first portion of the solution becoming mixed with the water; and since even a very dilute solution of sodium hyposulphite dissolves silver chloride, the outflowing solution has to be conveyed to the silver precipitation vats a long time before it reaches its standard strength.

Therefore it will clearly be seen that the standard strength can only be maintained by a continual addition of sodium hyposulphite to the stock solution. A continuous supply is furnished by the precipitant, whether that be sodium or calcium sulphide, because in preparing it some sodium or calcium hyposulphite is formed. As much as 5 to 7 per cent. of this salt may be found in the freshly prepared precipitant, without the latter being exposed to the action of the air for any considerable length of time. The constant supply from this source, however, does not suffice to replace the loss of hyposulphite salt which the stock solution suffers if the latter has to be kept at a strength of 1 or 2 per cent., but it does suffice, and in fact in most cases exceeds the loss, if a weak solution is used. The consequence is that in the first case sodium hyposulphite has to be bought and added, which increases the treatment expense, while in the second case not a pound of this salt has to be added, even during years of operation, and, on the contrary, water has often to be added to reduce the strength to the standard.

The following table is an interesting daily record of the sodium hyposulphite contained in the stock solution with which I worked the San Francisco del Oro ore. With the exception of June 16,

on which day, for a certain experimental purpose, 175 lb. of sodium hyposulphite were added, there was not a pound of this salt added to the stock solution:

| DATE | PER CENT. OF HYPO IN SOLUTION | DATE | PER CENT. OF HYPO IN SOLUTION | DATE | PER CENT. OF HYPO IN SOLUTION | DATE | PER CENT. OF HYPO IN SOLUTION |
|----------|----------------------------------|--------|----------------------------------|--------|----------------------------------|--------|----------------------------------|
| April 13 | 0.34 | May 11 | 0.54 | June 7 | 0.39 | July 4 | 0.76 |
| " 14 | 0.52 | " 12 | 0.56 | " 8 | 0.39 | " 5 | 0.70 |
| " 15 | 0.50 | " 13 | 0.56 | " 9 | 0.39 | " 6 | 0.69 |
| " 16 | 0.55 | " 14 | 0.56 | " 10 | 0.40 | " 7 | 0.73 |
| " 17 | 0.60 | " 15 | 0.52 | " 11 | 0.40 | " 8 | 0.78 |
| " 18 | 0.58 | " 16 | 0.49 | " 12 | 0.38 | " 9 | 0.78 |
| " 19 | 0.56 | " 17 | 0.53 | " 13 | 0.38 | " 10 | 0.79 |
| " 20 | 0.61 | " 18 | 0.53 | " 14 | 0.38 | " 11 | 0.81 |
| " 21 | 0.60 | " 19 | 0.52 | " 15 | 0.38 | " 12 | 0.63 |
| " 22 | 0.58 | " 20 | 0.57 | " 16 | 0.50 | " 13 | 0.61 |
| " 23 | 0.58 | " 21 | 0.57 | " 17 | 0.50 | " 14 | 0.58 |
| " 24 | 0.58 | " 22 | 0.59 | " 18 | 0.49 | " 15 | 0.55 |
| " 25 | 0.57 | " 23 | 0.58 | " 19 | 0.48 | " 16 | 0.56 |
| " 26 | 0.56 | " 24 | 0.54 | " 20 | 0.53 | " 17 | 0.59 |
| " 28 | 0.53 | " 25 | 0.56 | " 21 | 0.55 | " 18 | 0.60 |
| " 29 | 0.54 | " 26 | 0.58 | " 22 | 0.55 | " 19 | 0.62 |
| " 30 | 0.51 | " 27 | 0.60 | " 23 | 0.56 | " 20 | 0.61 |
| May 1 | 0.51 | " 28 | 0.60 | " 24 | 0.58 | " 21 | 0.63 |
| " 2 | 0.49 | " 29 | 0.63 | " 25 | 0.65 | " 22 | 0.64 |
| " 3 | 0.51 | " 30 | 0.63 | " 26 | 0.67 | " 23 | 0.66 |
| " 4 | 0.49 | " 31 | 0.59 | " 27 | 0.70 | " 24 | 0.66 |
| " 5 | 0.50 | June 1 | 0.49 | " 28 | 0.75 | " 25 | 0.65 |
| " 6 | 0.46 | " 2 | 0.53 | " 29 | 0.75 | " 26 | 0.50 |
| " 7 | 0.47 | " 3 | 0.53 | " 30 | 0.73 | " 27 | 0.63 |
| " 8 | 0.47 | " 4 | 0.54 | July 1 | 0.80 | " 28 | (?) |
| " 9 | 0.50 | " 5 | 0.48 | " 2 | 0.81 | " 29 | 0.59 |
| " 10 | 0.49 | " 6 | 0.43 | " 3 | 0.79 | | |

It will be noticed that the solution had a tendency to increase in strength. In order to keep it as much as possible at the standard strength of 0.50 per cent, water had to be added frequently, which accounts for most of the sudden drops. A sample of the stock solution taken after precipitation contained: hyposulphite salts, 0.52 per cent.; lime, 0.165 per cent.; sulphuric acid, 0.140 per cent.; chlorine, 0.098 per cent. The solution evaporated and heated in the muffle gave 0.775 per cent. solids. A month after, it gave 0.785 per cent. solid residues. This shows how remarkably clean the solution remained. The 0.14 per cent. of sul-

phuric acid corresponds exactly with a saturated solution of gypsum (1 part in 400 parts of water). The 0.098 per cent. chlorine is equal to 0.16 per cent. sodium chloride.

Best Strength of Solution. — In my experience I have found the best strength of the solution to be 0.25 to 0.50 per cent. Such a solution offers not only the above-mentioned economical advantage, but it produces also a precipitate much cleaner and richer in silver. Lead sulphate and cuprous chloride dissolve much more easily in a strong than in a weak solution, and therefore with a strong solution a very low-grade precipitate will result and a much larger amount of the precipitant will be required. It is a wrong supposition that a strong solution shortens the leaching time and extracts the silver better. The strong solution, dissolving readily lead sulphate and cuprous chloride, becomes heavily charged with these salts and thereby loses much of its dissolving power for silver chloride, and therefore neither the leaching time will be shortened nor will the extraction be better.

Method of Leaching. — When base-metal leaching is finished and the sodium hyposulphite solution is to be applied, it is well to do this before all the water has disappeared from the surface of the ore. The solution, then, in its course downward, follows closely the water and replaces the latter perfectly in all parts of the charge, and has less opportunity to become mixed with the water absorbed by the ore than if the water is allowed to drain from the ore before the solution is applied, and consequently the reaction, when the silver appears at the outlet, will be more precise. There is another advantage connected with it, inasmuch that if the charge be allowed to drain, the space before occupied by water will be filled by air, and when the hyposulphite solution is run on, it is very difficult, in fact almost impossible, to drive this air out again. The layer next to the surface will give up its air, which can be observed by the bubbles at the beginning, but further down in the charge the friction becomes too great, and the air, not being able to escape, will compress to let the solution pass, and by doing so will reduce the speed of filtration and prevent a quick and intimate contact between the ore particles and solution, thus increasing the time of leaching and decreasing the percentage of extraction. Therefore attention should be paid that during no time of lixiviation the liquid be allowed to sink below the

surface of the ore. In a fresh charge the ore is loose, and when water is applied on top it will, in descending, easily force downward the air, which will escape through the outlet.

Testing the Solution for Silver. — It requires close watching to ascertain the time when the silver appears at the outlet. If the liquid is tested with sodium or calcium sulphide the reaction is very uncertain, because the base-metal leaching is seldom carried to such an extent that no light clouds are formed by an addition of these reagents, and therefore the reaction cannot show the first traces of silver in the outflowing stream. Even a very dilute solution of sodium hyposulphite dissolves silver chloride, and we can safely assume that, as soon as sodium hyposulphite can be detected at the outlet, the stream contains silver; and since furthermore the liquid naturally will contain more hyposulphite salt than silver, it is safer to adopt a method by which this salt can be detected in the outflowing stream. I use the following test, which is reliable and convenient:

A small strip of starch paper is dipped into iodine solution and then held in the stream. If the blue color disappears it is a sign that the liquid contains a hyposulphite salt and consequently silver, and the stream has to be turned at once into the trough leading to the precipitation tanks. The base metals have to be leached with cold water to make this test applicable, because hot water also discolors the blue paper. I advise all who practise the lixiviation process to use this test and see that when base-metal leaching is changed to silver leaching the outflowing stream is very closely watched. By doing so they will find that shortages in silver for which they could not account before will be avoided.

Effect of Lead and Copper. — If the ore contains lead and copper we shall find both metals in the hyposulphite solution, because lead sulphate and cuprous chloride which are present in the roasted ore are not soluble in water but dissolve in sodium hyposulphite. Both these metals are precipitated together with the silver, and we find them in large quantities in the precipitate, reducing materially the fineness of the latter. To remove the lead from the solution Mr. Russell precipitates it as carbonate by adding sodium carbonate to the solution previous to precipitation with sodium sulphide. By doing so he obtains a sulphide precipitate free from lead and lead carbonate as a by-product, but

this complicates the manipulations without offering much practical advantage. Only where the sulphides are refined by melting them with iron and borax in crucibles is the use of this method justified, because if the precipitate is free from lead a silver bullion over 900 fine will result. But this treatment is too expensive and inconvenient, and is not used except in small works, and then only in exceptional cases. In large works the refining is done on a lead bath in the cupeling furnace, and lead in the precipitate is then of great advantage.

For the sake of information the Russell method was tried in a modified way. Instead of precipitating the lead in the solution after it had left the lixiviation vat, it was precipitated inside the ore charge by adding sodium carbonate to the solution before leaching. The outflowing solution was entirely free from lead. The precipitated lead carbonate remained in the ore. No injurious effect on the extraction was noticed. The residues contained about as much silver as when leached without an addition of sodium carbonate, but the precipitate was much richer in silver.

Calcium Sulphide as Precipitant, and Action of Calcium Hypo-sulphite. — If calcium sulphide is used as precipitant the solution will contain after precipitation calcium hyposulphite, and it used to be generally assumed that within a short time the original sodium hyposulphite solution was replaced by calcium hyposulphite. I discovered and demonstrated that this is not the case; in fact, that even if the original lixiviating solution were calcium hyposulphite, that compound could not exist in the process for any length of time, but would be converted into sodium hyposulphite. We read in standard works that Patera leached with sodium hyposulphite and precipitated with sodium sulphide, while Kiss used calcium hyposulphite as solvent and calcium sulphide as precipitant. It is mentioned that calcium hyposulphite possesses a greater dissolving energy for gold than the corresponding sodium salt, and that for this reason the Kiss process is more suitable for gold-bearing silver ores. However, Kiss was not leaching, as he thought, with calcium hyposulphite, but with sodium hyposulphite, and the better extraction of gold was most likely due to cooling the roasted ore slowly.

In roasting sulphureted ore with salt, sodium sulphate is formed in large quantities. This salt is not easily removed from

the ore by leaching with water. If leaching with water be prolonged it will be found that the outflowing liquid will react for sodium sulphate long after all the heavy metallic salts are removed. Therefore, when the hyposulphite solution for the extraction of silver is applied the ore still contains sodium sulphate in considerable quantity. Calcium hyposulphite reacts with sodium sulphate, forming sodium hyposulphite and insoluble calcium sulphate. If, therefore, calcium hyposulphite is brought into the stock solution by the precipitant, or when calcium hyposulphite is used as solvent, it will be converted into sodium hyposulphite by the regular operation of the process.

Time Required for Lixiviation. — The time of lixiviation varies according to the nature of the ore, its permeability, and the size of the charge. I observed that if the main part of the silver in the ore is contained in galena, the silver extraction will be slow, even if the ore filters well, while if all or the principal part of the silver is contained in copper, zinc, arsenical or antimonial minerals the extraction is quick. At the Silver King mine, Arizona, a charge of 8 tons required only nine hours' silver leaching, though it contained on an average 161.4 oz. silver per ton; while at Cusi-huiriachic, Mexico, the time required for a charge of 8 tons was fifty-three hours, the ore containing but 47 oz. silver per ton. The filtering capacity in both cases was very nearly the same. At the Silver King the silver was mostly contained in gray copper ore, antimonial fahlerz and silver-copper glance, while at Cusi-huiriachic it was principally contained in galena. I have made the same observation with ores of many other localities.

A free filtration is very important for a quick and thorough extraction. The ore particles should be brought rapidly in contact with fresh solution, which cannot be done if the filtration is slow. After the solution, descending through the ore, has dissolved a certain quantity of the salts present it loses much of its dissolving energy, and therefore the small stream of a bad filtering ore will not contain much more silver per liter than the large stream of a quick-filtering ore. The same observation can be made if the outflowing stream of a quick-filtering ore be checked and reduced to a small stream. Likewise, if lixiviation is interrupted and the solution allowed to remain in contact with the ore for some time, say over-night, the solution will not contain much more silver after leaching is resumed than the stream did

before the interruption. Ores which after roasting run on the cooling floor like water always filter badly and are not suitable for tank lixiviation. Mixing such ores with sand or chopped straw does not improve their permeability. By mixing in a small percentage of galena before roasting, however, the ore loses somewhat of its dusty condition and permits a little better percolation. The only rational way, however, of treating such ores is by trough lixiviation.

Regeneration of a Foul Hypo Solution. — The same sodium hyposulphite solution was used at the Silver King works, Arizona, for over a year and a half without requiring any addition whatsoever of sodium hyposulphite, and acted still as energetically as it did the first day. The ore permitted a very quick extraction of the silver; the silver leaching did not require more than nine to ten hours. Suddenly a change took place. The solution, which formerly gained in strength and had to be diluted from time to time, became weaker; the time required for silver leaching increased to fifty, sixty, and seventy hours. The main portion of the silver was extracted in twenty-four hours, but the balance took a very long time. The value of the precipitate dropped from \$12 per pound to \$6, and the leaching tanks, which always were ahead of the furnaces, could not keep up with them. An addition of sodium hyposulphite salt to the stock solution improved it somewhat, but only for a short time. Seeking for the cause of this strange change, I examined closely the present ore and compared it with that which was worked before, as the cause could be found only there, because the roasting as well as the leaching was done in exactly the same way as during the previous year and a half. I found that the present ore, though its general appearance was the same as before, did not carry any copper. This occurred when the works at the mine reached the 700-ft. level. Not being able to find any other difference, I suspected the absence of copper in the ore to be the cause of the fouling of the solution, and after an experiment on a small scale had proved the supposition to be correct, cupric chloride was prepared by boiling blue vitriol with salt. One precipitation tank was cleaned and filled with stock solution. Then 12 gallons of the prepared cupric chloride solution was added, stirred and precipitated with calcium sulphide. After precipitation the solution of this tank was capable of dissolving nearly 50 per cent. more silver chloride

than before. This was tried in the laboratory with freshly prepared silver chloride. The clear solution was decanted from the copper precipitate and conveyed to the pump tank to be mixed with the balance of the stock solution. During the first two days the copper precipitations were made daily. A change in the action of the solution could be observed at once. In all nine copper precipitations were made, with a consumption of 127 lb. of bluestone and 181 lb. of salt. At the end of the ninth precipitation the stock solution had almost regained its former dissolving energy. The silver leaching time dropped to fourteen and sixteen hours, and the value of the precipitate improved again to \$10 and \$11 per pound. The solution kept in this excellent condition for over a month, when signs of degeneration could be observed; a fresh precipitation, however, brought it right again. While not being prepared to offer a chemical explanation why the want of copper in the ore should cause a degeneration of the sodium hyposulphite solution, and why an addition of cupric chloride to this solution, which is precipitated before it is sent in circulation, should restore to the solution its former quality and dissolving energy, I place my observation on record with the belief that it will be of great service to many lixiviation works.

In Cusihiuriachic, Mexico, it took seventy-two hours to leach the silver from a tank charge of 8 tons, while after a treatment of the stock solution with cupric chloride it was accomplished in twelve to eighteen hours.

Filters for Leaching Tanks. — A free filtration being of great importance for the success of the process, the selection of a proper filter is, therefore, also of great importance. Some use gravel and sand without a wooden filter bottom, as at La Baranca, Sonora, Mexico. The material of such a filter is cheap enough, but its preparation is troublesome, and therefore a filter is usually kept in use until the filtration becomes so bad that it has to be replaced by a new one. At Sombrerete straw filters were once used. Short straw was spread on a wooden filter bottom about a foot thick and then the ore was dumped on top of it. This rather coarse filter did not produce a very clear filtrate when new, but after the first charge it improved in this respect and produced a clear solution; however, the outflowing stream decreased with every charge, and when the filter had to be renewed it was found that the straw had rotted and packed tightly.

Filters like those described above, which cannot be cleaned, and are therefore kept in use as long as possible, invariably cause a decrease of the working capacity of the leaching vats and with it a decrease in production and percentage of extraction. Burlap, the material used for making grain sacks, makes the best and most lasting filter. It is cut in pieces and spread over the wooden filter bottom so that one strip laps over the other about 3 in. The ends are rolled up and packed tightly around the circumference, or better into a groove as shown at *N*, Fig. 31. After each charge the strips of burlap are carefully rolled up, so that no residue which adheres to them may drop below the filter bottom, are well washed and again spread on the filter bottom. If this is done regularly the filter is kept in the best possible condition. If the washing is not done after every charge a thin layer of residue will form on the filter cloth, gradually growing thicker and harder, partly by the repeated pressure of the feet of the laborers and partly by the deposition of calcium sulphate. The gypsum also incrusts the fibers of the cloth, makes it stiff and hard, and finally stops filtration entirely. Where burlap cannot be obtained coarse sheeting is a good substitute.

End of the Silver Leaching. — The extraction of the silver from a charge may be finished while the outflowing solution still gives a considerable precipitate upon addition of calcium or sodium sulphide. It is impossible to judge by the color of this precipitate whether it contains silver or not. It is, however, important to know when all soluble silver is extracted, in order to avoid unnecessary consumption of the precipitant and loss of time. To ascertain when the end of the silver leaching has been reached a large beaker is filled with the solution and some calcium sulphide is added. The precipitate is allowed to settle, the clear solution decanted, and the precipitate poured on a paper filter and washed well. It is then removed from the filter and dissolved in nitric acid, filtered to remove the sulphur, and a drop or two of hydrochloric acid is added. If a white precipitate or only a cloudiness is produced, which by dilution and boiling does not disappear, there is still silver in the solution and the leaching has to be continued. If the solution remains clear the extraction is concluded.

The influx of sodium hyposulphite is then stopped and the solution is allowed to drain until it commences to disappear

under the surface of the ore, when a stream of water is turned on to displace the sodium hyposulphite solution absorbed by the ore. It is only necessary to continue this second application of water for a comparatively short time; just long enough to keep the same volume of stock solution on hand. Then the charge is allowed to drain as dry as circumstances permit, after which the residues are discharged. This is done by shoveling them into chutes, of which one is placed between each two tanks and which discharge into cars running underneath the tanks (Fig. 31), or into a large triangular trough beneath the vats, in which a current of water flows. The trough ought not to have an inclination less than 1 in. to the foot. In some works the residues are sluiced out. To make this method successful the vats must not be too deep nor of too large diameter. The stream of water has to be applied under pressure by means of a pump, or under good head from a storage tank.

XIV

PRECIPITATION OF SILVER

IN all modern lixiviation works the precipitation vats are provided with a mechanical contrivance to agitate the solution during and after precipitation, and only in antiquated works is this operation done by hand with a paddle. A horizontal beam about 12 in. shorter than the inside diameter of the vat is fastened to a vertical iron shaft (Figs. 57 and 58). This horizontal beam, which moves above the surface of the solution, is provided with hard-wood staves about 2 in. square and reaching down to about $1\frac{1}{2}$ in. above the bottom. These staves are so arranged that when the agitator is in motion they cut the liquid with the edge. The agitator is set in motion by a friction clutch, and it should be started gradually to avoid breaking the staves. The inside of the tank is provided with four vertical wooden wings, projecting 3 in. toward the center and reaching nearly to the bottom. They break the violent current around the periphery and throw the solution toward the center, thus causing a strong whirling motion. The agitator should make 30 r.p.m. if the diameter of the vat is not more than 8 or 9 ft. Agitators of this construction are only durable when used in vats not deeper than 6 ft. In deeper tanks the staves will break, and an agitator of stronger construction has to be used.

Another method of agitation which is very convenient is by the use of compressed air, furnished by a small compressor. From the receiver a pipe-line leads along the whole row of precipitation vats. At each vat is inserted a T with a $\frac{3}{4}$ -in. branch. These branch tubes are provided with valves and connected with a rubber hose 6 to 8 ft. long, the other end of which terminates with a $\frac{3}{4}$ -in. gas pipe. This pipe must be long enough to reach any point of the bottom and still project about 2 ft. above the rim of the tank. If the vat is full and the air is turned on the solution is put into violent motion, as if boiling. It is well to

change the position of the pipe from time to time, especially if the vat has a large diameter. This method of agitating the solution is now extensively used.

Preparing Calcium Polysulphide. — This precipitant is prepared by boiling milk of lime with pulverized sulphur. The proportion has to be ascertained by the operator, as it depends on the quality of the burnt lime rock. No free sulphur should be visible in the white residues. It is soon found out how much of the local lime has to be slacked for a certain weight of sulphur.

The boiling was formerly done in upright boilers, made of boiler iron 4 ft. in diameter and 6 to 7 ft. deep, by direct application of a steam jet. Near the bottom was a pipe outlet closed with a wooden plug. This outlet discharged into an iron tank in which a sand filter was prepared. Below the level of the bottom of this filter tank was placed the iron storage tank to receive the calcium sulphide solution. From this storage tank a pipe-line conveyed the chemical to the precipitation tanks. The boiling tank is first charged with the milk of lime, but enough margin should be left for the water condensed from the steam, which in the beginning is quite considerable. When the milk is boiling the sulphur is charged gradually, by means of a sieve, in order to scatter the same as finely as possible over the surface. If charged by a shovel the sulphur sinks quickly to the bottom, forming chunks which require a long time to combine with the lime. When boiling is concluded some cold water is splashed over the surface, which makes the foam disappear and also causes a quicker settling. When settled the clear yellow-brown solution is decanted, by means of a stiff 1-in. rubber hose, into the storage tank. This done, the plug near the bottom of the boiler is removed, and the sediment allowed to flow into the filter tank. To make it flow more freely, some water is added before removing the plug. The filtrate from the residues also flows into the storage tank. These residues are very slow in filtering and retain much calcium sulphide solution, which has to be displaced by applying water on top of the residues, which, however, is rather difficult to do properly, as the residues are very pasty and very slow filtering, so that, when they are removed by shovels, some of the calcium sulphide will be wasted.

The calcium sulphide department was much improved lately by myself. Fig. 35 represents a vertical section of the arrange-

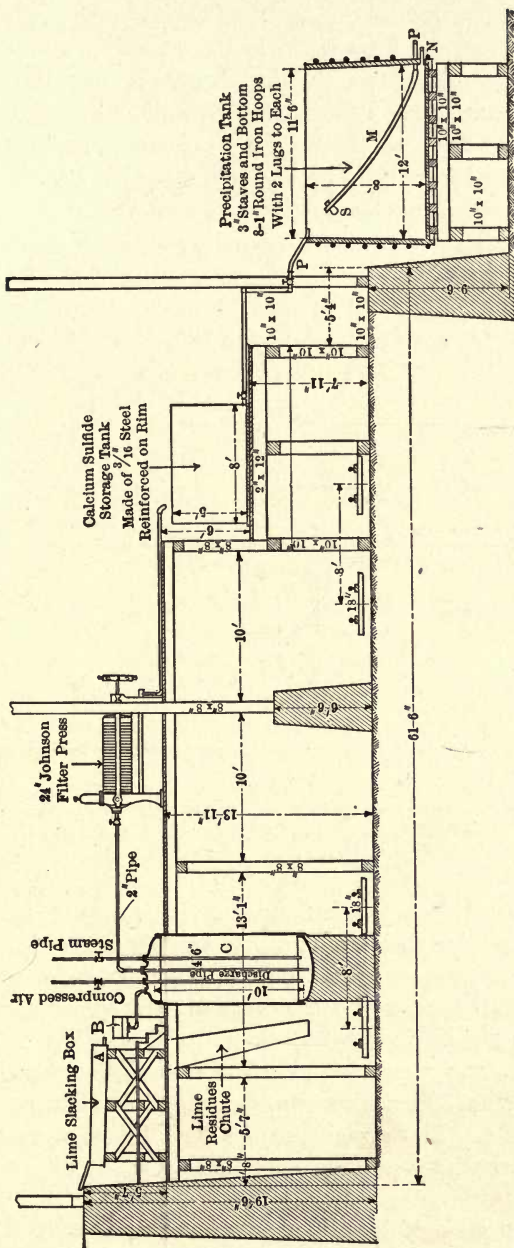


FIG. 35.—CALCIUM SULPHIDE PLANT.

ment. *A* is the lime-slacking box, nearly level with the cooling floor. When the lime is slacked the milk is allowed to flow into the distributing trough *B*, whence it fills the boiler *C*. There are two of these boilers, side by side, which can be alternately charged through the distributing trough. This trough is made of $\frac{1}{4}$ -in. steel and is represented by Fig. 36. The outlets in the bottom can be closed by wooden plugs.

The boiler *C* is represented in detail by Fig. 37. The sides are made of $\frac{3}{8}$ -in. while the bottom and top are of $\frac{7}{8}$ -in. steel. It is a boiler and pressure tank combined. Fig. 38 gives a top view. There are six openings in the top, of which one is a manhole, while the other five are smaller, and are connected with

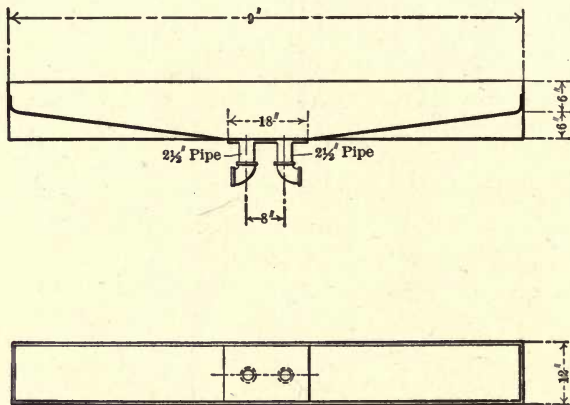


FIG. 36.—DISTRIBUTING TROUGH FOR MILK OF LIME.
To be made of steel.

pipes. Three of them reach nearly to the bottom (Fig. 35), viz.: the discharge pipe passing through the center, the steampipe, and the compressed-air pipe. The filling and air-escape pipes do not extend into the boiler. The reason why the compressed-air pipe reaches nearly to the bottom is to agitate the pulp during discharging. The manhole cover has a 4-in. flanged opening in the center (Fig. 39), which can easily be closed by a cast-iron plate and bolts. This opening serves for introducing the sulphur; it is left open during filling and boiling, and is closed during discharging.

When the milk of lime is charged steam is turned on. The opening in the manhole cover is kept open for the escape of the

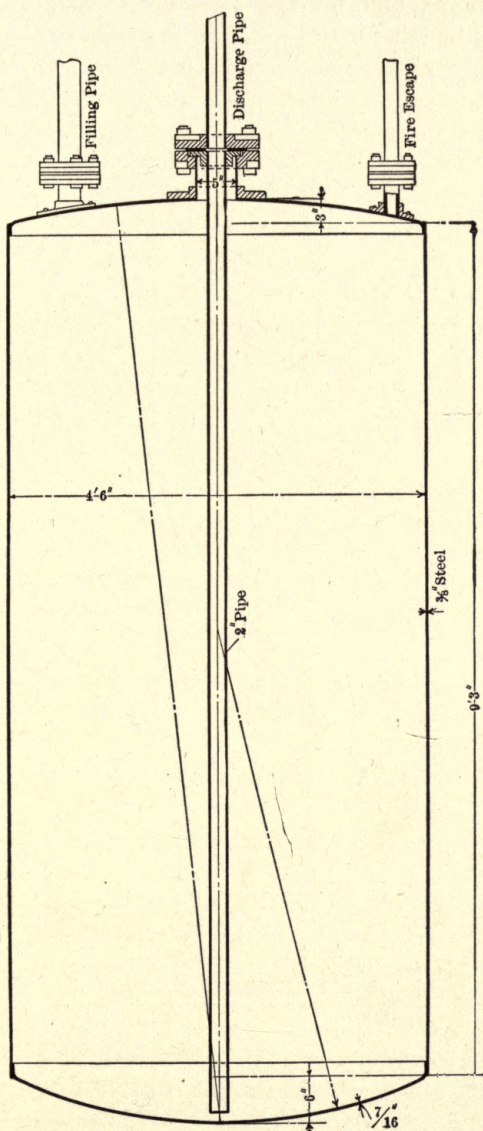


FIG. 37. — BOILER AND PRESSURE TANK FOR CALCIUM SULPHIDE.
Vertical section.

air during filling and for the escape of vapor during boiling. Room has to be left for the water from condensed steam, and also for foaming, which sometimes occurs when too much steam

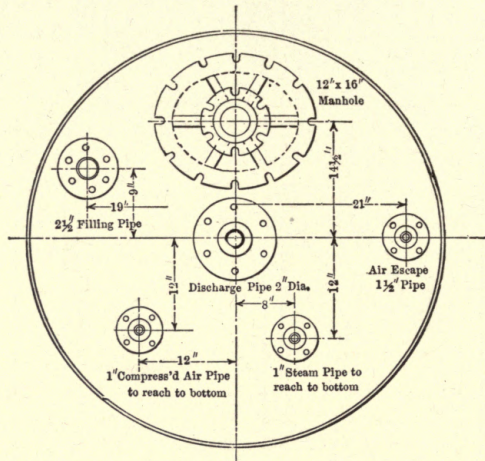


FIG. 38.—BOILER AND PRESSURE TANK FOR CALCIUM SULPHIDE.

Plan of head.

is admitted. By means of a wooden staff in which notches are cut, and which is inserted through the opening in the manhole cover, the filling is regulated.

It is not advisable, for two reasons, to make too strong a

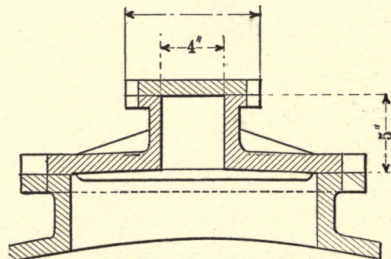


FIG. 39.—BOILER AND PRESSURE TANK FOR CALCIUM SULPHIDE.

Saddle to be riveted to head of tank.

solution. In a very strong solution crystals of bisulphide of calcium are formed, which will be found on the sides and bottom of the storage tank, and will also be formed in the pipe-line leading

from that tank to the precipitation tank, and before long will clog the pipe, necessitating the taking down of the whole line for cleaning. The second inconvenience caused by too strong a solution is the difficulty to see the end reaction in precipitating, in consequence of which the precipitator is apt to add too much of the precipitant.

I found that a solution answers well if, in boiling, to each cubic foot of water about $2\frac{1}{4}$ lb. of sulphur are taken. Of good and freshly burned lime about $2\frac{1}{2}$ lb. to the pound of sulphur is an average proportion, but, as stated above, it depends entirely on the quality of the lime.

If it is intended to fill the boiler three-quarters full, the cubic content is calculated and by it the amount of required sulphur and lime ascertained. The lime is then slacked and the milk charged into the boiler, then water is added to fill the boiler three-quarters full. Then steam is admitted. The sulphur is charged gradually when the water commences to boil. Boiling has to be continued for four or five hours, according to circumstances. If in a sample, taken with a long-handled iron ladle from the bottom after filtering, some free sulphur should be found, boiling should be continued. If this has no effect it shows that lime is wanting, which has to be added. In the next charge more lime has to be taken in the beginning.

When boiling is concluded the steam is turned off and the volume of the solution measured with the wooden staff; if the volume is short, it has to be filled up with water to the mark in the staff. Then the opening in the manhole cover is closed tight and compressed air is applied. The discharge pipe is connected with a 24-in. filter press, as shown in Fig. 35. The filtrate flows into the calcium sulphide storage tank, and from there is conveyed along and above the precipitation tanks, which are also shown in this figure. When the filter press is filled with residues, water under pressure is forced into the press to wash the residues. The wash-water is allowed to run into the storage tank and mix with the strong solution. In the floor near the lime-box is a chute, discharging into iron cars. The residues from the lime-box as well as from the filter press are thrown into this chute.

Adding the Precipitant. — The precipitant, whether sodium or calcium sulphide, is kept in a reservoir made of boiler iron, from which it is conveyed through an iron pipe to the precipitation

vats. At each vat there is attached to the pipe a hose, which is closed by a pinch-cock. In commencing to precipitate it is well to open the hose a little and to throw, by swinging it, some of the precipitant over the surface. By the appearance of the clouds which are formed, whether heavy or light, the precipitator can see at once if he has to precipitate a concentrated or dilute charge. Then he sets the solution in agitation and allows the precipitant to flow in. An experienced precipitator can judge by the color which is created when he splashes some of the precipitant over the surface, while the solution is agitated, the progress of precipitation, and knows when it is nearing the end.

While precipitation is going on the clouds which are formed become gradually lighter in color, and toward the end almost yellow. When nearly finished the influx of the precipitant is stopped, and after a few minutes the agitation also. Then the flaky precipitate is allowed to sink somewhat below the surface, and some of the precipitant is splashed over the surface. According to the appearance of the clouds more or less precipitant is added and the solution is agitated again. This operation is repeated until precipitation is complete. If by an addition of the precipitant no reaction takes place, it is well to throw some strong silver solution over the surface after the precipitate has partially settled. If the places where the silver solution fell turn reddish brown, the precipitant is in excess and more silver solution has to be added. One who is not experienced would best make this test in a beaker.

The precipitation tank as illustrated in Fig. 35 is 12 ft. in diameter and 8 ft. deep. The solution is agitated by compressed air. The two outlet pipes, *P* and *N*, are of lead. To the upper pipe, *P*, is attached on the inside of the tank the decanting hose, *M*, with the float, *S*. This should be a very stiff hose. It is kept above the solution, while the tank is filling or precipitation is going on, by a thin rope fastened to a hook on the outside of the tank. When the precipitate has settled, the rope is unhooked and the end of the hose allowed to float. The float is so arranged that it keeps the end of the hose immersed. When the float comes in near approach to the precipitate close attention has to be paid, so that no precipitate is carried out. As the decanted solution is to be used again for extracting silver, it is conveyed by means of troughs, running along in front of the precipitation

tanks or through a pipe-line to the lower storage tanks, in which it is collected and elevated to the upper storage tanks, which are placed on a higher level than the rim of the leaching tanks. The outer end of the pipe P is connected with a short piece of hose, which lies in the trough, if troughs are used, to avoid splashing. The lower outlet pipe, N , is provided with a valve and connected with an iron pipe-line, common to all precipitation tanks, which leads to a pressure tank, by means of which the solution is forced into a filter press. The upper hose P_1 serves for admitting the precipitant.

XV

TREATMENT OF THE PRECIPITATE

The Precipitate. — In precipitating the base-metal solution we have seen that not all the metals present are equally affected by the sulphur of the precipitant, and that the silver especially is more readily precipitated than the other metals. This is also the case if these metals are dissolved in the sodium hyposulphite solution, and therefore the precipitate which is obtained in the earliest stage of precipitation contains far more silver than that obtained later. Thus the operator has it in his power to make different grades of precipitate. This, however, does not offer such advantages as in base-metal leaching, and is of no direct practical value, because in order to maintain the dissolving energy of the sodium hyposulphite solution it is absolutely necessary to precipitate as perfectly as possible all the metals dissolved in it; but it explains why the black layer of sulphides, which we frequently find deposited on the surface of the ore charge, is so much poorer in silver than the precipitate itself. If the precipitation was done well, and ample time was given to the precipitate to settle, and the decantation of the solution was always performed properly and without mishap, we should not find any black deposit on the top of the ore charge; but such exact work is not always done, especially in large works, which seldom have a sufficient number of vats to give the precipitate ample time to settle.

The different sulphides settle according to their respective specific gravities. Lead, silver, and copper go down first, while antimony, zinc, iron, and free sulphur follow. While this separation is not theoretically perfect, it takes place to such a degree that the particles which settle last may contain but 30 to 100 oz. silver per ton, while the total precipitate may contain 5000 to 15,000 oz. per ton.

This black layer of sulphides deposited on top of the ore charge

after an extended lixiviation, being so much poorer than the precipitate, does not involve any notable loss of silver if it is carefully scraped off before the residues are discharged and the scrapings are mixed with the ore and roasted.

F. Sustersic's Method of Preparing the Precipitate for Refining by Cupellation. — If a precipitate contains a large percentage of copper, the refining of the same by cupellation with lead requires a large amount of lead, causes the formation of a large amount of rich by-products, and increases the cost of refining.

For the treatment of such a precipitate F. Sustersic devised a method by which he extracts the copper first, leaving the precipitate in an excellent condition for cupellation. Reporting on very interesting experiments made with such precipitate, he says:

“The crude precipitate produced by lixiviating the ores of Avino, Durango, Mexico, with sodium hyposulphite was analyzed and shown to contain:

| | | |
|-----------------------|--------|-----------|
| Silver | 4.206 | per cent. |
| Gold | 0.2064 | per cent. |
| Copper | 14.80 | per cent. |
| Lead | 16.40 | per cent. |
| Iron | 0.70 | per cent. |
| Zinc | 0.50 | per cent. |
| Chlorine | 5.10 | per cent. |
| Lime | 3.87 | per cent. |
| Sulphur | 39.40 | per cent. |
| Insoluble | 4.20 | per cent. |
| Not ascertained | 10.80 | per cent. |

“This precipitate is rather base, which was caused by the fact that the ore is very susceptible to heat and readily loses considerable silver by volatilization at only a moderate roasting temperature, and therefore had to be roasted at an extremely low heat, at which practically all base-metal salts remained in the roasted ore. To refine such a precipitate with sulphuric acid would be too expensive in Avino, and the refining with lead by cupellation would offer considerable difficulties on account of the large percentage of copper, so it became necessary to find a proper method by which the copper could be removed from the precipitate before the latter is subjected to cupellation.

“In roasting a material containing copper sulphide and silver sulphide the copper is converted into sulphate at a very low heat, while it takes a bright-red heat to convert the silver into

sulphate.¹ In accordance with this difference in the property of these two metals I made several tests and obtained very gratifying results.

“A six-inch roasting dish containing 100 grams of the precipitate was placed into a muffle the temperature of which was kept below red heat. As soon as the free sulphur commenced to burn the roasting dish was removed from the muffle and the charge continually stirred until the sulphur flame ceased. Then the charge was again placed into the muffle and roasted at so low a heat that the chemically combined sulphur oxidized without ignition. This second operation required only fifteen minutes. The color of the precipitate changed from black to greenish gray.

| | |
|--------------------------------|-----------------|
| Weight of raw charge..... | 100.00 grams. |
| Weight of roasted charge | 85.40 grams. |
| Loss in weight | 14.60 per cent. |

The roasted precipitate ought to assay 4.925 per cent. silver.
 By actual assay it was found to contain 4.9286 per cent. silver.
 The loss by volatilization, therefore, was.....*Nil*.

“An analysis of the roasted precipitate gave the following result:

| | |
|----------------------|------------------|
| Silver | 4.9286 per cent. |
| Copper | 18.50 per cent. |
| Lead | 19.30 per cent. |
| Ferric oxide | 1.75 per cent. |
| Zinc | 0.30 per cent. |
| Lime | 8.60 per cent. |
| Sulphuric acid | 37.70 per cent. |
| Insolubles | 6.80 per cent. |

“Ten grams of the roasted precipitate were leached with water until the filtrate became colorless. No trace of silver could be detected in the filtrate. The copper in solution was precipitated with zinc.

The 10 grams of roasted precipitate contained 1.85 grams copper.
 The filtrate was found to contain in solution..... 1.66 grams copper.
 Copper extracted as sulphate 89.73 per cent.

“The residues on the filter weighed 5.8 grams, and the roasted precipitate therefore lost 42 per cent. of its weight by removing the soluble substances with water. These residues were analyzed and found to contain:

¹ See Chapters on “Sulphating Roasting,” and “Ziervogel’s process.”

| | |
|----------------------|-----------------|
| Silver | 8.50 per cent. |
| Copper | 1.60 per cent. |
| Lead | 33.50 per cent. |
| Iron | 1.65 per cent. |
| Lime | 6.00 per cent. |
| Sulphuric acid | 28.50 per cent. |

Taking the loss in weight into calculation which the roasted precipitate sustained by leaching with water, if there was no silver dissolved the residues should contain 8.4965 per cent. silver.
 By actual assay it was found that they did contain... 8.5000 per cent. silver.
 Therefore, silver dissolved by leaching with water was. . *Nil*.

“These results show that by this method nearly 90 per cent. of the copper in the precipitate can be removed and recovered as metallic copper if the roasting is properly executed, while all the silver and the lead remains concentrated in the residues, which are then in a state well suited for cupellation.

“By several check-tests it was proved that no loss of silver is sustained, neither in roasting the precipitate nor in leaching the same with water.”

This method of F. Sustersic is undoubtedly a very rational way to prepare the precipitate for cupellation. Not only does it remove and recover nearly 90 per cent. of objectionable copper, but also it removes other salts, thus reducing the weight of the roasted precipitate by 42 per cent., which is connected with a corresponding enrichment in silver and lead. It permits the production of a clean litharge and greatly reduces the formation of rich slag or froth. There is nothing to diminish the usefulness of this method if the free sulphur of the crude precipitate is removed by boiling with caustic soda instead of by burning. On a commercial scale the precipitation of the copper from the solution is, of course, done with scrap iron.

Accumulation of Sodium Sulphate in the Solution. — I stated above that when the leaching with water is stopped and the charge is ready for silver leaching the ore still contains some sodium sulphate, which during silver leaching will enter the stock solution. In course of time the sodium sulphate will therefore accumulate to such a degree in the stock solution that the latter will greatly lose in dissolving energy, a much longer leaching time will be required, and the percentage of extraction will suffer much. When such trouble arises operators usually try to “freshen up” the solution by adding more sodium hyposulphite to it. This, however, will benefit it only for a very short time, and the same

trouble will appear again. Some add more hyposulphite every day, thus increasing greatly the cost of extraction without getting the solution back to its original energetic condition.

Calcium Sulphide as Precipitant. — The best method is to conduct the process so that no sodium sulphate will accumulate in the stock solution. This can be done by using calcium sulphide as precipitant. Sodium sulphate reacts with calcium sulphide, forming sodium sulphide and calcium sulphate, which precipitates, while the sodium sulphide acts as precipitant for the metal chlorides dissolved in the solution. On account of this reaction, which is of so great an advantage to the lixiviation process, I always advocate the use of calcium sulphide as precipitant.

The valuable effect of this reaction is especially felt if ore is treated which requires a large percentage of salt in roasting, whereby larger quantities of sodium sulphate are produced. Should so much sodium sulphate enter the stock solution that by the regular process of precipitation all the sodium sulphate is not decomposed, this salt will then gradually increase, notwithstanding the use of calcium sulphide, and will spoil the solution. It is therefore advisable to add at certain intervals an excess of calcium sulphide, and finish precipitation with silver solution. This will free the solution entirely from sodium sulphate.

If sodium sulphide is used this reaction does not take place, and not only will all the sodium sulphate dissolved during silver leaching accumulate and remain in the stock solution, but the amount will be increased from the precipitant, which always contains more or less sodium sulphate. In such works it is well to make provision for the manufacture of calcium sulphide, solely for the purpose of purifying the stock solution from time to time.

Treatment of the Precipitate. — The precipitate should be discharged from the precipitation tanks every other day. If it is allowed to accumulate for a longer time it changes from a flaky condition to a very fine powder, which settles slowly and will cause the solution in circulation to contain in suspension a considerable amount of exceedingly fine precipitate. The precipitate is discharged into a tank with a slowly moving agitator, varying in size according to the size of the works, from which the precipitate is drawn into a pressure tank in portions as required, and thence, by means of compressed air, it is forced into a

Johnson filter press. Pressure tanks made of cast-iron will resist longer the corroding action of the solution than those made of boiler iron. They should be made so that the top or cover can be removed when required. The relief pipe should return to the sulphide tank connected with the agitator, because the air when relieved escapes with great force, and is apt to carry along some precipitate. Instead of compressed air, steam may be used; but the pressure will be limited by the pressure in the boiler, and often may not be sufficient. The feed pipe of the press should be in connection with water and air pipes, so that after the filter press is filled the precipitate can be washed by pumping warm water through it, and partially dried by forcing compressed air through it. When so treated the precipitate will come out of the press in hard cakes, permitting a clean handling of them. The press should be placed in a separate room with cement floor. Such a floor is easily kept clean, and prevents loss of silver.

Instead of a pressure tank, a steam pump can also be used; only attention should be paid that the steam cylinder of the pump is larger than the pump cylinder, so that a pressure of 150 lb. or more can be produced with it, if it should be necessary. The pump cylinder should be provided with a bronze piston rod and liner, because iron is more or less affected by the solution, and in course of time the inside of an iron cylinder becomes very rough. Much cleaner work can be done with a pressure tank, because there is always more or less leakage around the piston rod of a pump. The filtrate is conveyed to one of the lower solution reservoirs.

It was found more convenient, cleaner, and labor-saving not to use an agitating tank for the precipitate, but to connect the pressure tank direct with the pipe-line which runs in front of the precipitation tanks and with which they are in communication. The filling of the pressure tank is done direct from the precipitation tanks, but in order to release them as soon as possible for their regular work, the pressure tank is made large enough to receive the whole of the precipitate of one tank. It is made of steel and used in an upright position.

When the charge has been forced into the filter press and the pressure tank is empty, the compressed air contained therein is allowed to escape by opening the valve of the air-escape pipe. The air rushes out with great force, carrying out moisture and

some of the precipitate hanging on the side near the air outlet, which, if left to escape free, would not only cause some loss of silver, but make the surroundings very unclean. To avoid this I designed and introduced a drum into which the air is discharged, and which gives very good satisfaction. Fig. 40 is a vertical section. The drum is 24 in. in diameter and 3 ft. 6 in. high. In alternating distances of 6 in., 4 in., 6 in., 4 in., etc., angle irons *A*, *A*, are riveted to the inside, forming circular shelves. On these shelves rest conical trays, one with the cone turned down,

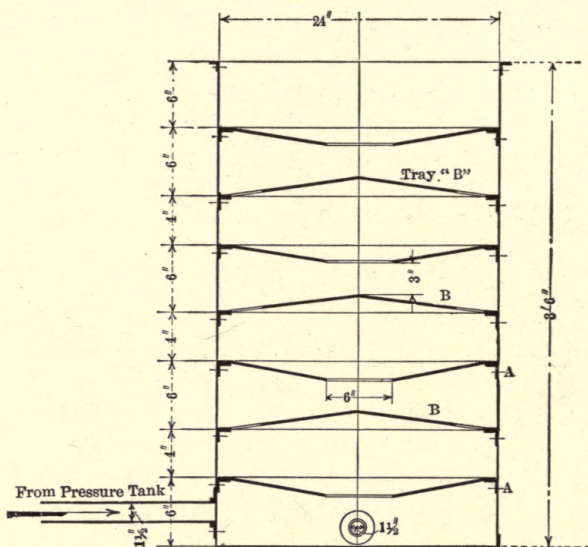


FIG. 40. — AIR BLOW-OFF DRUM.
Vertical section.

the other with the cone turned up. Those with the cone down have a 6-in. circular opening in the center, while the others have openings (*C*, *C*, Fig. 41) near the periphery. These trays are kept in place by four small bolts. Underneath the bottom tray is an outlet pipe leading to one of the lower storage tanks, while at an angle of 90 deg., enters the air-escape pipe from the pressure tank. The working of this air blow-off drum is clearly explained by the drawing.

The Pressure Tanks. — If a pressure tank is destined to lift liquors containing residues or a precipitate it ought to be used in an upright position, because it facilitates the discharge of these

solids, but if such a tank is used only for clear solutions, then it is of more advantage to have it in a horizontal position. Fig. 42 represents a horizontal pressure tank designed to lift the sodium

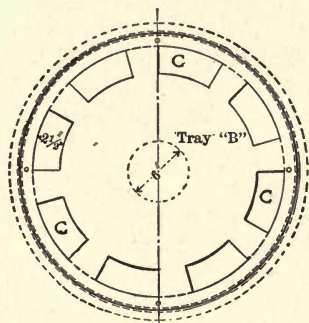


FIG. 41. — AIR BLOW-OFF DRUM.
Plan of tray.

hyposulphite solution. It is made of steel 12 ft. long and 4 ft. 6 in. in diameter. The filling pipe *P* enters at the head of the tank in order to save grade. This tank has to be placed in a pit below the bottom level of the lower storage tank; by inserting the filling pipe at the center of the head a more shallow pit will

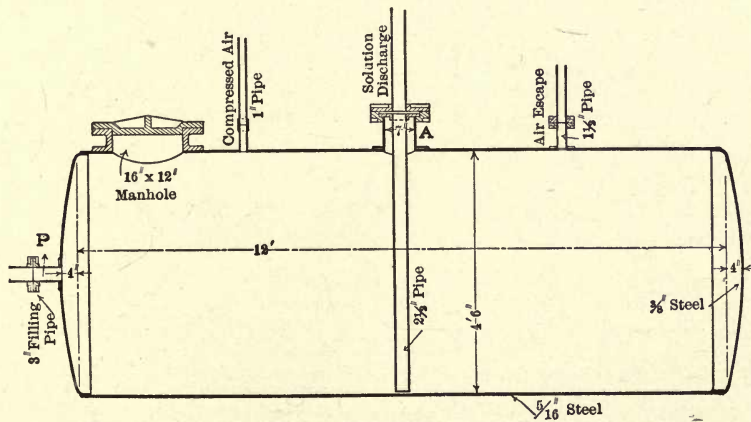


FIG. 42. — HORIZONTAL PRESSURE TANK, FOR SOLUTION.

answer, and is more convenient for work. The manhole is on top, but near one end of the tank. At *A* the solution discharge-pipe is inserted, which extends nearly to the bottom. The part

of the pipe which extends inside the tank is subject to wear and should be so arranged as to permit of quick replacement. In Fig. 43 its construction is shown in detail: *A*, a short tube, one

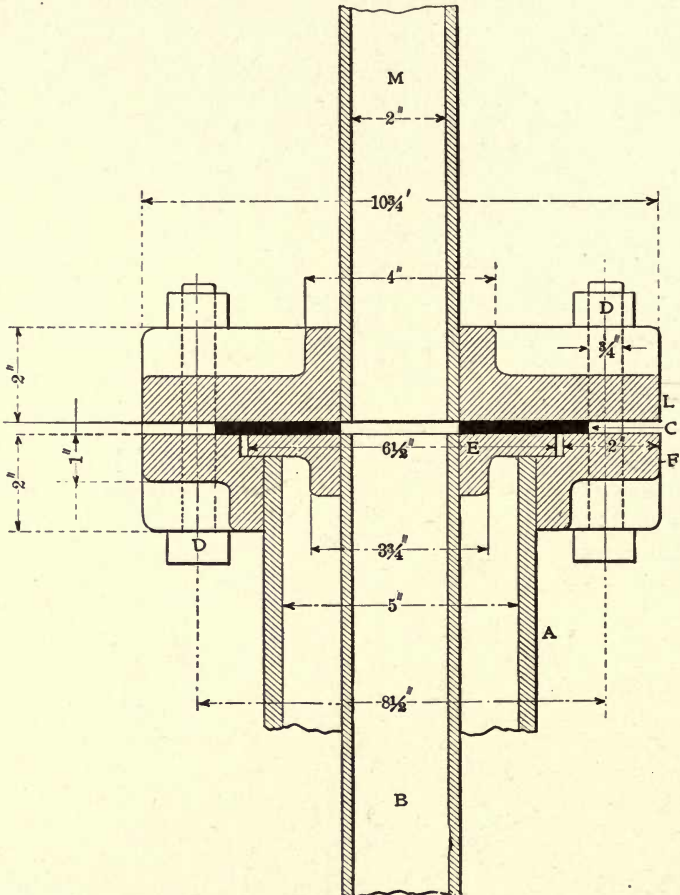


FIG. 43.—CAST-IRON FLANGE UNION FOR DISCHARGE PIPE OF PRESSURE TANK.

The dimensions here given are for 2-in. discharge pipe as used in the vertical pressure tanks. They have to be changed for other pressure tanks according to size of their respective discharge pipe. The part that extends into the tank can be easily removed and replaced.

end of which is fastened to the tank, while the other is provided with a flange *F*, of the shape shown. *B*, the pipe which extends into the tank. The upper end of it is provided with the flange

E, which is faced on both sides. This flange fits loosely into the recess of the flange *F*. When the tube is inserted a $\frac{1}{4}$ -in. rubber gasket, *C*, is placed on top and covered with the flange *L*, which is attached to the discharge-pipe *M*. This done, the flanges *L* and *F* are tightly drawn together by the bolts *D*, *D*. The rubber gasket is strongly pressed by the bolts and makes a perfectly tight joint. By the construction it can be seen that the pipe *B* can be withdrawn and replaced in a few minutes. All pipes extending into a pressure tank ought to be arranged in the way just described.

Handling of the Sodium Hyposulphite Solution. — It was stated above that the solution, by precipitating the silver and the other metals dissolved in it, is regenerated, and can be used over and over again indefinitely. As the solution works from the upper level down to the lowest level of the leaching plant by gravity, it has to be elevated again in order to keep it in circulation. The solution coming from the precipitation tanks is collected in a number of large but not too deep tanks (storage tanks). These tanks, usually four in number, communicate by means of iron pipes near the bottom. The solution coming from the precipitation tanks flows only into the first tank and into no other, but through the communicating pipes all four tanks are filled simultaneously. The fourth or last tank only is connected with the filling-pipe of the pressure tank. This arrangement of the four communicating tanks offers an excellent opportunity for settling any precipitate that may have been drawn out in decanting by carelessness. Above the leaching tanks there is another set of four storage tanks of the same dimensions, arranged exactly as are the lower tanks. Into the first of these tanks the pressure tanks lift and discharge the solution, while from the fourth one a pipe-line leads over all the leaching tanks. These upper storage tanks give another opportunity for settling. They are cleaned once or twice a year.

In large works the circulating stream of solution is quite voluminous, and it is advisable to have two solution pressure tanks, so that a constant stream can be maintained.

Removal of Sulphur. — The precipitant being a polysulphide, the precipitate will contain a large percentage of free sulphur, whether calcium or sodium sulphide is used. It is desirable to remove this free sulphur from the precipitate before the latter

is subjected to a final treatment. The best method is to boil it with caustic soda in an iron tank, the caustic soda combining with the free sulphur and forming sodium sulphide, which serves as precipitant for the silver; but care has to be taken that no excess of caustic soda enters the stock solution, because it will exercise a decomposing action on the silver chloride in the ore. To avoid this the sodium sulphide solution thus obtained is conveyed to the boiling vessels in which the precipitant is manufactured. After boiling with caustic soda the precipitate shrinks

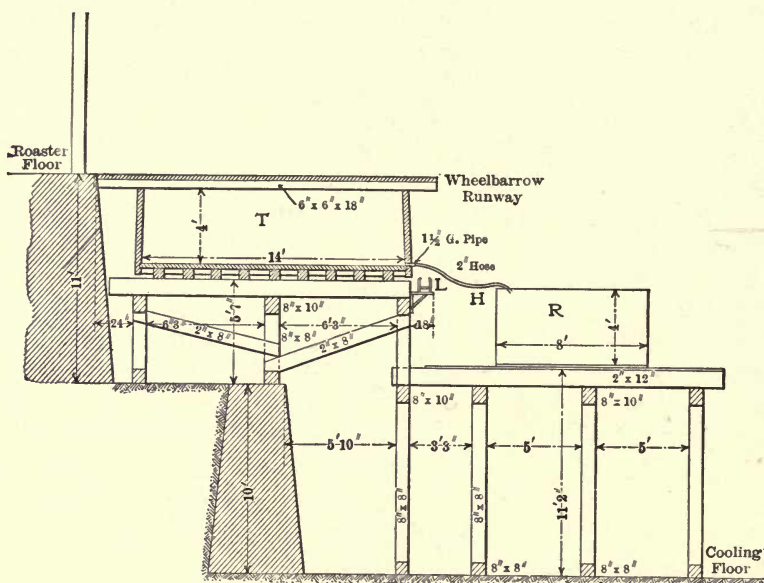


FIG. 44. — APPARATUS FOR THE MANUFACTURE OF LYE.

To be placed on cooling floor next to roaster floor.

much in volume and becomes very heavy. To separate it from the sodium sulphide solution after the main part has been decanted is a very slow process if done by common filters, and therefore it is much better to use a small filter press for this purpose. In works where no filter press is used, and a common filter has to be employed, these badly filtering sulphides can be made quick filtering by treating them with a strong silver solution to decompose all the sodium sulphide, which is the cause of the bad filtration. By this method 60 per cent. of the sulphur contained in the precipitate can be regained and brought into

a state in which it can be used again as precipitant, thus greatly reducing the actual consumption of sulphur.

F. Sustersic proposed to leach wood ashes, convert the lye by boiling with caustic lime into caustic potash and boil the precipitate with it, thus producing potassium sulphide, which is used as precipitant. I adopted and carried out this method.

In most works wood is used as fuel for the roasters and boilers, and the ashes are thrown away. By adopting this method a large part of these ashes can be utilized. Fig. 44 illustrates an arrangement for the manufacture of lye. The tank *T* is placed near the roaster floor, with its rim just a little below that floor, so that the wheelbarrow runway resting on the rim of the tank is on a level with the roaster floor. The tank is provided with a filter bottom (not shown in the figure). The ashes from the roasters and also from the boilers, if the latter are conveniently situated, are wheeled and dumped into the tank, which is filled one-third full with water. This is done to protect the tank and filter from the hot ashes, which contain many small pieces of burning coal. When the tank is filled, water is allowed to flow on top of the ashes. The outflowing lye is collected in the iron tank *R*. When the lye begins to get weak the hose *H* is placed in the trough *L*, which leads outside the building, and the charge is allowed to drain and then replaced by fresh ashes. From the iron tank *R* a pipe-line leads to the calcium sulphide boilers, which can also be used for the manufacture of caustic potash. One of the boilers is charged with lye, to which milk of lime is added. The mixture is then boiled. Samples are taken from time to time, filtered, and to the filtrate a few drops of hydrochloric acid added. If this causes effervescence boiling is continued, but if after a while effervescence is still caused by the acid, then more milk of lime has to be added. When finished the charge is pressed through the same filter press which is used for calcium sulphide. The filtrate is collected in an iron storage tank placed alongside of the calcium sulphide tank.

Fig. 45 represents a system of three pressure tanks; they serve for the treatment of the precipitate, and are placed in the refinery of the works. The pressure tank *A* is charged with precipitate from the precipitation tanks through the pipe *D*. From there it is forced through the pipe *E* into a filter press. When pressed, the precipitate is charged into pressure tank *C*,

through the opening in the manhole. Caustic potash solution is added, which is done by opening the valve of pipe *G*, which latter is connected with the caustic potash storage tank. The mixture is boiled, and when finished forced, through pipe *H*, into a second filter press. The filtrate from this press, which is potassium sulphide, flows into pressure tank *B*, through pipe *K*, whence it is lifted up through the pipe *M* and charged into the calcium sulphide boiling tank. The filtrate from this press, which is potassium sulphide, flows into pressure tank *B*, through pipe *K*, whence it is lifted up through the pipe *M* and charged into the calcium sulphide boiling tank.

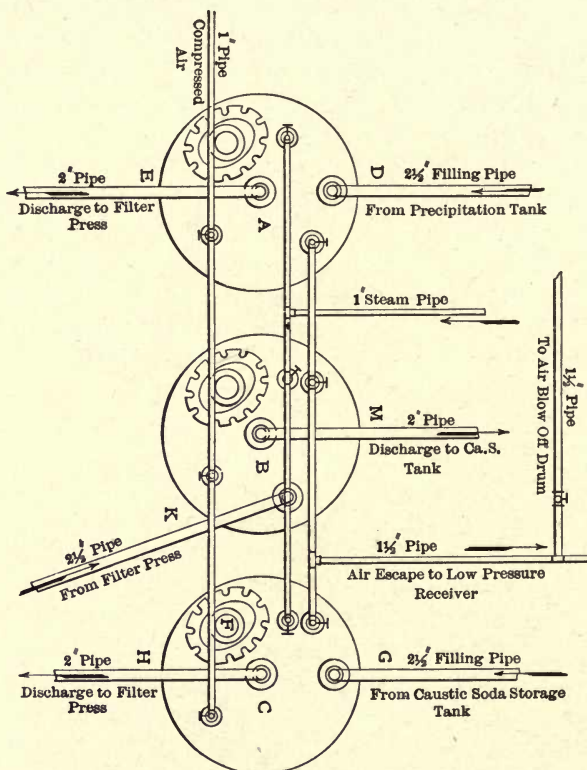


FIG. 45. — PRESSURE TANKS FOR TREATMENT OF PRECIPITATE.

As by this operation about 60 per cent. of the sulphur is regained in a shape in which it can be directly applied as precipitant, and at the same time the precipitate is freed from its surplus sulphur, and as, besides, the caustic potash does not cost more than the labor and the lime, this is surely a very economical operation.

Another way of expelling and regaining the free sulphur from the precipitate is by distillation. The moist precipitate is charged into retorts and heated, the sulphur vapors being conducted to brick chambers and condensed as flowers of sulphur. I used this method years ago on a large scale with satisfactory results as to the amount of sulphur regained, but the cast-iron retorts did not last long enough, principally on account of carelessness on the part of the men in charge, who overheated them; and as transportation of such heavy castings into the mountains was very difficult and expensive, this method was discarded. However, with careful firing, and in localities where transportation facilities are better, it can be applied to great advantage. While the extraction of the free sulphur will not be so complete as by boiling the precipitate with caustic soda or potash, the operations are fewer. The product is dry and ready immediately for further treatment, while in the former method the product has to undergo the processes of filtering, washing and drying.

A third method, by which, however, the sulphur is lost, consists in burning it off in a small reverberatory furnace. An actual roasting is not required; in fact it ought to be avoided, to prevent loss by volatilization. The sulphides ought to be charged dry, but if they are charged moist they should remain undisturbed in the moderately heated furnace until dry, to avoid the generation of rapidly evolving steam, which is apt to carry away fine particles of the already dry part of the precipitate, thus causing a loss. Even if the precipitate has been previously dried in special ovens, the heat in the beginning has to be kept very low for some time, to avoid mechanical loss by steam, because only seldom will the precipitate be perfectly free from moisture after leaving the drying ovens. Later the temperature is increased to ignite the sulphides. They commence to burn with a blue flame near the fire-bridge, and the flame spreads gradually over the whole charge. When this takes place the fire has to be lowered to avoid overheating. No stirring should be done until the flame ceases; then a gentle stirring is given. This brings up new flames, which, however, do not last long, but reappear if the charge is stirred again. Stirring is repeated until the flame ceases entirely. The temperature has to be kept so that when the flame ceases the charge is perfectly dark, which indicates that no actual roasting of the material took place and that only the free sulphur was

burned off. If the precipitate is treated in this way no loss by volatilization will take place, and that is all which is required, because for the further treatment of the precipitate an actual roasting, a changing of the sulphides to oxides and sulphates, is not necessary; in fact, is hurtful. Mr. Stetefeldt, in his book on lixiviation, stated the loss to be as high as 6 and 12 per cent., but this is not so.¹ However, there is a chance of loss by this method if it is not properly executed, especially if the precipitate contains antimony or is charged wet into the furnace.

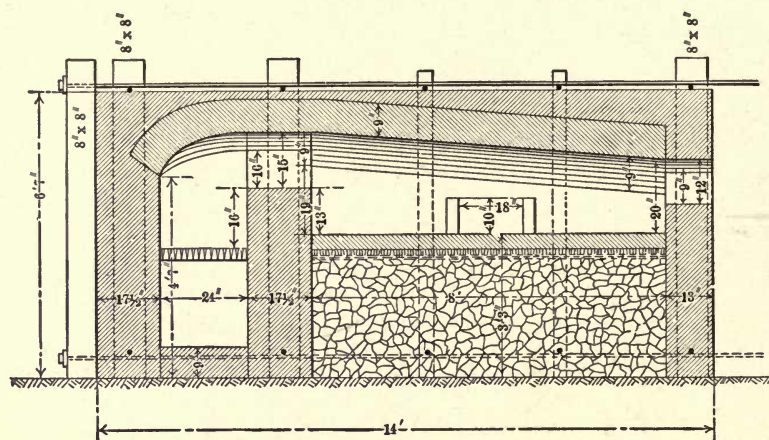


FIG. 46.—DRYING AND ROASTING FURNACE FOR SILVER PRECIPITATE.

Fig. 46 represents a vertical and Fig. 47 a horizontal section of a small reverberatory furnace for burning the silver precipitate. Where the boiling of the precipitate with caustic potash or soda is adopted this furnace can be used for drying the treated precipitate.

The first method, i.e., boiling with caustic soda or potash, is the most rational and entirely excludes any loss of silver, and offers the additional advantage of regaining about 60 per cent. of the sulphur.

Refining the Precipitate. — The refining is done with litharge

¹ Stetefeldt based his statements on reports received from Cusihiuriachic, while the works were under the management of Mr. Dagget. Later it developed that the great loss attributed to the burning of the sulphides was caused by the dishonesty of the man in charge of that part of the process, who stole systematically part of each charge.

left of it is the fireplace and to the right the flue, which is provided with a damper, while the back of the furnace is arranged to admit the blast. The furnace proper is covered with a dome made of boiler iron and lined with clay. This dome is lifted and can be swung to one side by a crane.

A cheap and excellent material for the cupel is a mixture of one part of clay (by volume) and three parts of lime rock, pulverized and well mixed. Both ingredients must be free from quartz and ore particles, for which reason, if the crushing has to be done by machinery that is also used for pulverizing ore, such machinery must first be cleaned very carefully.

After the material is well mixed part of it is spread on a clean floor, sprinkled with water, and quickly worked with shovels so that the mixture becomes uniformly moist. The mixture should not be made too moist, as otherwise in refining the bottom is apt to come up. The material is in proper condition if a handful of it, squeezed hard, forms a ball, which may be handled gently, but should crumble into its former condition by a slight pressure with the fingers. In preparing the test the perforated pan is filled about 6 in. with the prepared material, then stamped down with iron bars.

These bars are made of a piece of round iron $1\frac{1}{2}$ in. in diameter and 8 to 10 in. long, one end of which has the shape of an egg while the other is welded to a $1\frac{1}{2}$ -in. gas-pipe 5 ft. long. The material in the pan is leveled and beaten in by two men standing on boards laid across the brickwork. They commence to beat in the center, pursuing a spiral course, stamping with the egg point in a perpendicular direction, by a lift of about eight inches, striking with the rod close to each preceding stroke. When by a screw-like advance the stamping has reached the side of the pan, it has to be carried on back to the center in the same way, then again to the side and so on, till about two inches of loose material remains.

If a hole can be scratched easily with the finger in the stamped mass, the bar must be used with more force. Care must be taken always to have still two inches of loose material above the stamped mass when a new charge is put in, because if the whole is beaten hard, the next charge will not unite perfectly with the under layer. The beating on the second charge is done in the same way, and so on until the hard-beaten mass

reaches about two inches below the rim of the pan. Then a 6-in. high iron ring fitting the rim of the pan is placed on the rim, a new charge added, and stamping continued until the hard stamped mass reaches about three inches above the rim of the pan. Then the loose material is removed, the ring pulled up, and the hard mass is cut down roughly by means of a hatchet or an adze to about an inch above the rim of the pan, and then leveled with a sharp short-handled scraper. This done, a circle is scratched in, leaving a margin of about four inches around the periphery of the pan. Inside this circle the hard stamped mass is cut out spherically with a short-handled adze about 7 to 8 in. deep; then made smooth by a scraper with a curved blade. When the bottom is finished the hood is put over it by means of the crane and a gentle fire started for drying. It is well to scatter a thin layer of ashes over the surface. The fire should be kept about twelve hours, then the ashes removed and the lead charged. When the lead is melted and becomes red hot some coarsely pulverized litharge is charged, enough to cover the whole surface of the bath. A strong fire is kept and the precipitate, mixed with litharge, is put on the bath in small charges. When this is going on the draft has to be checked by the damper so that the flames will come out through the working door. This precaution is taken to reduce the amount of such valuable dust to be carried out by the draft into the dust-chambers. After the charge is introduced the damper is opened again, and when the charge is melted the upper layer is worked gently with a hoe, and a new charge is introduced. This is repeated until all the precipitate is charged. Then some more litharge is added, the working door closed and the strong fire continued. It will be noticed that, when everything is well melted, around the periphery of the bath many small bubbles will appear. This is caused by the carbonic acid which the lime rock gives off, and does not injure the bottom.

When the last charge is well melted the top layer is drawn off over the litharge bridge by means of a hoe until the surface becomes bright. Then the blast is turned on, playing on the surface, which makes the bath fume profusely. This is caused by the oxidation of the lead matte which had formed during melting and which lays on top of the lead. These fumes smell strongly of sulphurous acid. After two or three hours the surface loses

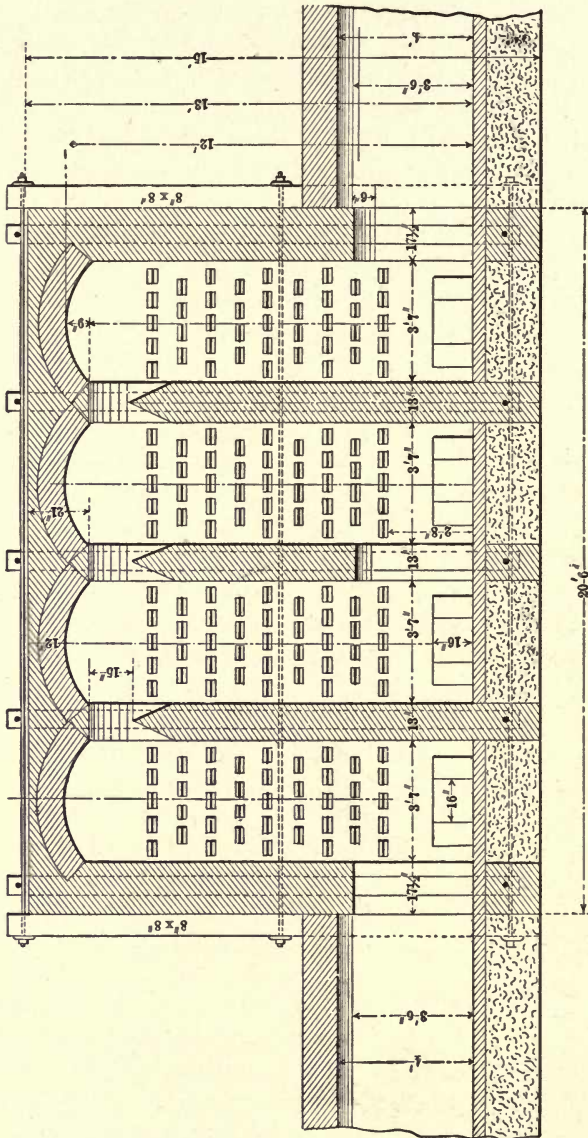


FIG. 48. — DUST-COLLECTING ARRANGEMENT FOR CUPELING FURNACE.

its brightness, a yellow-red ring of litharge forms around the periphery, and little islands of the same color are formed where the blast strikes. They float about until they reach the ring of litharge around the periphery and unite with it. The width of this ring increases until nearly the whole surface of the bath is covered. When this has happened a flat channel is cut into the litharge bridge, through which the litharge flows off. The flow is to be regulated so that the ring is kept about 8 in. wide. Care is to be taken that no metal flows over the bridge. The temperature has to be kept so that the litharge runs freely over the bridge but cuts the channel very little. If the heat is too high the litharge will cut the channel rapidly deeper and metal will flow out. On the other hand, if the temperature is too low the litharge will flow sluggishly over the bridge and form a soft crust on the inside, near the channel. Too low a temperature has to be carefully avoided, as the whole bath may freeze, especially toward the end of the process. Likewise should a too high temperature be avoided, because then the litharge cuts around the periphery of the bottom, thus ruining it, and eats out the channel too quickly, besides which more silver and lead are volatilized.

At the end of the operation, when less litharge is formed, the bath becomes covered with a net-like coat, moving on the convex surface and consisting of litharge, between which the silver glances through in spots. These spots grow larger, till at last the net breaks and the litharge slides to the sides, producing a display of colors. Before the end, but at the time when the formation of litharge becomes scanty, the iron molds are arranged on an iron bench set in front of the furnace. A long-handled iron ladle hooked to a chain reaching down from above is heated by placing it to one side inside the furnace. As soon as the surface of the silver turns bright it is ladled into the molds. During ladling the temperature has to be kept high.

Should the litharge become stiff and sluggish toward the end, even if the temperature is increased, it is a sign that there is not enough lead in the bath to separate the copper, and without loss of time some lead has to be added, but on the side of the test and not into the bath, because this would cool it too much and might freeze it.

All the products which are formed in cupeling contain silver. The first slag drawn, as well as the last litharge near the end of

the operation, is the richest, while the poorest and purest is obtained during the middle of the operation. The latter goes back to the process at the next cupellation. The products that are too rich it is best to sell to the smelting works, as it would be too expensive to treat them by themselves.

As it cannot be avoided that some dust of the precipitate, which is very valuable, is carried away by the draft during charging, and as the lead fumes from the bath also contain silver, it is very necessary to provide for an effective dust collector. Fig. 48 illustrates a four-shaft system of O. Hofmann's dust collector. It is inserted in the main flue of the refinery, so that all the fumes and dust from the different furnaces have to pass through it.

XVI

CONSTRUCTION OF TROUGHS

In lixiviating works, troughs are extensively used for conveying the base metal and the silver solution, and it is of great importance to have and keep them perfectly tight. The solutions will corrode metals, including lead, and the troughs therefore have to be made of wood, but it is a very difficult task to make and keep troughs perfectly tight, especially if they have only a slight inclination and the solutions do not move swiftly. Leakage occurs principally in places where the boards are spliced. White lead, tar, pitch, putty, etc., are of but little avail. White lead is the worst, and ought never to be used by a carpenter to make a joint waterproof. As long as the paint is wet it will be tight, but when the wood absorbs the oil and the paint dries, it contracts into numerous threads and wrinkles and hardens, forming minute channels through which the solution will find its way. The places where the white lead was applied will not swell, and the trough will be much less tight than if no cement had been used at all. In order to convey the solution to the different tanks, branch troughs have to be used. One end of these troughs is placed under the main trough, and right above it one or more holes are bored into the bottom of the main trough. These holes are closed by long wooden plugs. The loosening and tightening of these plugs is done by mallets, and therefore the main trough is subjected to considerable rough usage, and the pitch or putty will crack and start leakage.

In running the large blue vitriol plant of the Consolidated Kansas City Smelting and Refining Company of Argentine, Kansas, which I designed and erected, I was very much annoyed by trough leakage. There were over 2000 ft. of troughs in use. To guard against leakage all the troughs were lined with 6-lb.

sheet lead properly put in by experienced lead burners, but before a year passed the troughs began to leak in different places. Searching for the reason it was found that this leakage was caused by a peculiar property of the lead. The copper solution passing through the troughs was hot. It did not flow continually, but in charges, so that the lead was exposed alternately to the hot liquor and to the cooling effect of the air. The lead lining, which originally was perfectly smooth, was found to be full of wrinkles. These wrinkles could have been originated only by the expansion of the lead. By special experiments I convinced myself that lead is a metal which when heated expands, but when cooled does not contract as much as it had expanded; and if a sheet of lead is alternately heated and cooled it continues to grow larger and larger. If there is no room for free expansion, as in a trough or tank, the sheet has to fold up in wrinkles. If this is continued, the sheet gets so thin in certain places that it finally breaks.

For some time the leakage was stopped by burning a piece of sheet lead over the leaking places, but before long the leaks became so frequent that the item of trough repairing became seriously high, and it was concluded to renew all the troughs. Lead lining was of course discarded. Previous experiments with different kinds of soft wood showed that California redwood resisted best and longest the action of a hot copper solution. Other experiments were made to prepare a cement which softened but did not melt at a temperature of 90 to 96 deg. C., and which in cooling did not become hard and brittle but remained pliable. Such a cement was found by boiling lard oil with rosin and rubber and red oxide of iron. Waste pieces of sheet rubber and old pieces of rubber belting were used to supply the rubber. The fibers of the belting were removed with iron hooks and forks. The oxide of iron was added last. The boiling was done in an iron kettle with a fireplace below. The cement or paste was applied hot. Fig. 49 shows the construction of the trough in cross-section. The joints of sides and bottom are made step-shaped, as shown in the drawing, and are first coated with the hot cement, then screwed tight with brass screws. The corners are filled with triangular wooden moldings, which are first well coated with cement. They are kept in place and tightly drawn up by brass screws. If the corners are first well cemented, covered

by a strip of thin sheet rubber, and then the molding screwed on, additional security for tight joints is obtained. The troughs are made in sections of the length of the boards, and both ends cut

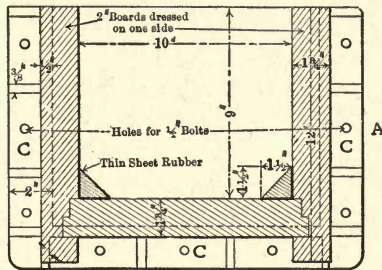


FIG. 49. — CROSS-SECTION OF TROUGH.

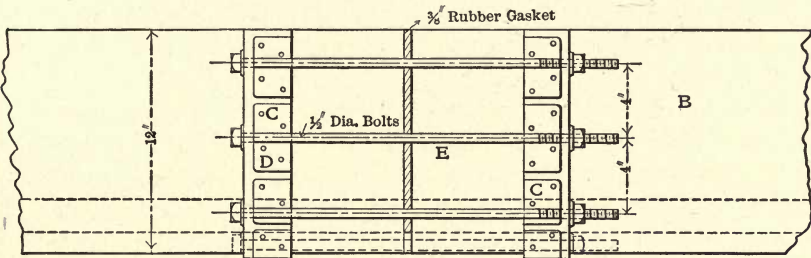


FIG. 50. — CONNECTION OF TROUGHS.

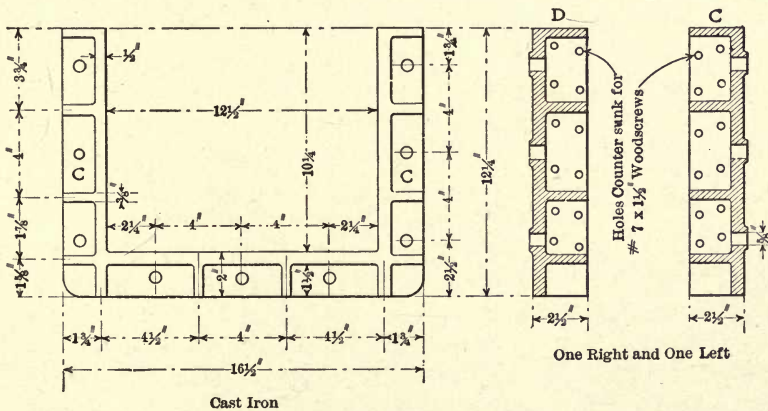


FIG. 51. — TROUGH UNION.

square. The ends of two sections are brought together, a pure rubber gasket, made to order in one piece, 1/4 or 3/8 in. thick, is

placed between them and drawn tight by bolts. Fig. 50, a side view, illustrates the manner in which the two ends of the trough are drawn together. *C* and *D* are flanged iron castings of the shape of the cross-section of the trough, but smaller, so that they will fit into a groove $2\frac{1}{2}$ in. wide and $\frac{1}{2}$ in. deep cut into the three sides of the trough and about 6 in. from the end. These castings are made in pairs—one right, one left, as shown in Fig. 51. When placed, they are fastened to the trough with wood screws, and by means of the bolts *E* the ends of two troughs are pressed together. Thus perfectly tight joints of the different sections are obtained. At distances of four feet wooden pieces are nailed across the trough to prevent the sides from spreading.

Troughs made in this way are perfectly tight and will stay so for years.

XVII

TROUGH LIXIVIATION

IN tank lixiviation, the extraction of the silver from chloridized ore by solutions of hyposulphite salts is performed by filtration. The ore particles are kept stationary, while the solvent moves down through the mass of ore. The quickness of extraction, other conditions alike, is in direct proportion to the rapidity of the movement of the solvent through the ore. The solution, if left in contact with the ore without moving, displays but very little dissolving energy. If the filtration is interrupted for ten or twelve hours, and thus solution and ore are left in complete contact for that length of time, it will be found that, when filtration is started again, the outflowing solution is but very little more saturated with silver than it was at the time of interruption, and that the ten or twelve hours were almost a complete loss in the total time of extraction. Notwithstanding the long contact, the solution had not become saturated with metal chlorides to its full dissolving capacity. A rapid movement of the solvent through the ore is essential to a quick extraction. This fact is well known; and the endeavor of leachers has been to hasten extraction by increasing the rate of filtration. Siphons, vacuum pumps and other devices have been used with more or less success, but none of them has given full satisfaction.

I have found that, if chloridized ore, after the base-metal chlorides are removed, is brought into rapid contact with a proper volume of moving sodium hyposulphite solution, the silver chloride contained in the ore dissolves almost instantly, and that it is rather the volume of the solvent than its concentrated state which produces this effect. Such favorable conditions cannot be attained in tanks. The rapidity with which a certain volume of the solvent can be brought into contact with the ore particles is limited by the speed with which the

solution descends through the ore; and thus the leaching time in tanks cannot be shortened beyond the limit set by the filtering capacity of the ore. In a trough, however, these favorable conditions can be attained by gradually introducing the ore into the moving stream of the solvent. The ore can thus be brought into rapid contact with any desired quantity of the solvent, and moves in and with the stream. The effect is astonishing. Ore charged at the upper end of a trough not longer than 12 to 15 ft. will leave the trough at the lower end as tailings, having yielded all its silver chloride to the solvent. This is accomplished during the very short time of 4.7 seconds which it takes the pulp to rush through the trough. But in order to obtain satisfactory results, that is, to extract all the silver chloride contained in the ore as shown by the chlorination test assay, it is necessary, and of great importance, to maintain a certain proportion of solvent and ore, which proportion depends on the nature of the ore. By numerous experiments I have found that all kinds of silver ores, no matter how differently they behave in tank lixiviation with regard to the length of time required for the extraction of the silver, will yield their silver chloride in the same short time. They behave all alike in this respect, but only if the proper proportion between solvent and ore which each respective ore requires is maintained. It is interesting to observe that, when two ores of different chemical character but of equal filtering property are treated in the trough, the one which in tank lixiviation requires the longer time to yield its silver chloride to the solvent will need in the trough a larger volume of solvent than the ore which requires in the tank less time for the extraction of the silver.

Lead-bearing ores require a long leaching time, for the reason that lead sulphate reduces greatly the dissolving energy of sodium hyposulphite for silver. In the ordinary lixiviation the solution becomes more saturated with lead sulphate as it descends through the ore and loses proportionally its dissolving energy. As the solubility of the lead sulphate increases with the concentration of the sodium hyposulphite solution, a stronger solution does not hasten the process; but if we bring the ore rapidly in contact with a large volume of hyposulphite solution, the latter retains enough of its dissolving energy to produce a quick silver extraction. The presence of lead sulphate, therefore, does not retard

trough lixiviation; it merely entails the use of larger quantities of solvent.

Results of much interest, obtained by me in experimenting with ore from the Cusihiuriacic Mining Company, Chihuahua, Mexico, illustrate the importance of maintaining a certain proportion of solution and ore to obtain satisfactory results. The ore contains considerable lead, and the extraction by common tank lixiviation required on an average nine hours for base-metal and 53.8 hours for silver leaching, in all 62.8 hours. It was roasted in Howell furnaces. For this experiment a 1.6 per cent. solution was used. The roasted ore was first leached with water to remove the base-metal chlorides before treating it in the trough.

The roasted ore contained27.12 oz. silver per ton.
 The chlorination test tailings called for . 3.94 oz. silver per ton.
 The ore and solution passed through 43 feet of trough.

PROPORTION OF SOLUTION AND ORE IN WEIGHTS

| | |
|-------------------------------------------------------------|--------------------------------|
| 6 weights solution to 1 ore, the trough tailings contained |14.58 oz. silver per ton. |
| 8 weights solution to 1 ore, the trough tailings contained | 6.56 oz. silver per ton. |
| 12 weights solution to 1 ore, the trough tailings contained | 5.25 oz. silver per ton. |
| 18 weights solution to 1 ore, the trough tailings contained | 4.37 oz. silver per ton. |
| 24 weights solution to 1 ore, the trough tailings contained | 4.37 oz. silver per ton. |

As the length of the troughs was in all cases alike, the time of contact of solution and ore was the same, and as the strength of the solution was also similar, the difference in the extraction was caused only by the volume of solution. The results show that the extraction is in direct proportion to the volume of solution until the maximum is reached, when an increase of solution does not improve the extraction any more. The results furthermore show that for the Cusihiuriacic ore the proportion is 18 weights of solution to one of ore, probably less, but more than 12 to 1. In this experiment the trough tailings contained 0.43 oz. per ton more silver than the chlorination test tailings called for. This shortage in extraction was caused by the fact that the 1.6 per cent. solution was too strong, as numerous subsequent experiments demonstrated. A solution containing but 0.5 per cent. sodium hyposulphite gives, as in tank lixiviation, the best results.

The idea suggested itself to investigate the behavior of the base-metal chlorides contained in the roasted ore in a moving stream of water, and it was found that they dissolved just as rapidly as the silver chloride in the sodium hyposulphite solution, and that passing through the same short length of trough the solution was accomplished, with the exception of a certain percentage of sodium sulphate, which, however, the ore is always found to contain in tank lixiviation at the time when base-metal leaching is stopped and silver leaching commenced.

We have learned in tank lixiviation that, by leaching the ore charge with water in order to remove the soluble metal chlorides, the water becomes so charged with these chlorides that it dissolves silver chloride like sodium chloride does, and that the solubility of the silver chloride increases with the concentration of the solution. To investigate this important feature of the lixiviation process, I made the following laboratory test: Freshly prepared silver chloride was introduced into solutions of sodium chloride which had a temperature of 120 deg. F., and was left in contact for five minutes, during which time the solution was vigorously agitated, and then filtered. The filtrate was tested for silver with sodium polysulphide, and it was found:

| | |
|-------------------------------------------------------------------------------------|--------------|
| Solution of 2 per cent. sodium chloride at 120 deg. F. dissolved in 5 minutes | no silver. |
| Solution of 3 per cent. sodium chloride at 120 deg. F. dissolved in 5 minutes | no silver. |
| Solution of 4 per cent. sodium chloride at 120 deg. F. dissolved in 5 minutes | no silver. |
| Solution of 5 per cent. sodium chloride at 120 deg. F. dissolved in 5 minutes | no silver. |
| Solution of 6 per cent. sodium chloride at 120 deg. F. dissolved in 5 minutes | no silver. |
| Solution of 7 per cent. sodium chloride at 120 deg. F. dissolved in 5 minutes | some silver. |

The filtrate of the 7 per cent. solution showed a faint coloring by sodium sulphide. When using an 8 per cent. solution the coloring was still very faint. Molten silver chloride poured into a 5 per cent. sodium chloride solution at 100 deg. F. decrepitated into fine powder; but the filtrate did not show any reaction for silver.

The result of this experiment shows that, if it were possible to regulate the base-metal leaching in tanks so that the outflowing solution at no time contains more than 6 per cent. of metal chlorides, the same would not dissolve any silver and could be

allowed to run to waste, instead of being subjected to a special treatment to regain the dissolved silver.

It is different in trough lixiviation. There we have it in our power to regulate the grade of concentration of the resulting base-metal solution at will, and we can therefore produce at once a solution which is sufficiently dilute not to dissolve silver chloride. This is one of the great advantages of trough lixiviation. Should the ore contain sufficient cupric chloride to make the saving of the copper an object, the whole resulting solution may be passed through a series of tanks or deep troughs filled with scrap iron. Its diluted state will not prevent the chemical reaction.

I have made a very interesting observation in respect to the solubility of the base-metal chlorides with regard to the trough principle. Two samples of complex ores from different mining districts were roasted with 8 per cent. of salt. Twenty grams of each were placed on a paper filter and leached with water. When all the soluble base-metal chlorides had been extracted, the filtrate was weighed in both instances, and it showed that the amount of water required for sample No. 1 was three times the weight of the ore, while for sample No. 2 it was seventeen times the weight of the ore. By treating the same quantity of ore one and a half minutes on the trough principle, no perfect extraction of the base metals could be obtained, until the same proportion of water and ore was used as that required in common leaching, viz.: 3 to 1 for ore No. 1, and 17 to 1 for ore No. 2. Both these ores were rather base. This behavior of the soluble chlorides toward the water as solvent is undoubtedly remarkable. If for both ores we assume the filtering property to be alike, it would follow, according to the quantity of water required, that if leached in tanks ore No. 2 will have to be leached $5\frac{2}{3}$ times as long as ore No. 1 to extract the soluble salts. The experiment, however, showed that this extraction can be accomplished from both ores in the same short time by bringing them at once in contact with their respectively required quantity of solvent.

Care has to be exercised in taking samples in trough lixiviation. The final residue sample can be taken from the tank with sampling irons in the usual way before the residues are discharged, but samples required for the observation of the process during operation have to be taken in a different way. The whole stream has to be caught in a vessel of proper size, say in an enam-

eled iron kettle of two or three gallons capacity. The proper place to take the sample is where the stream leaves the trough and drops into the settling-tank. The kettle is quickly pushed under the stream so that it receives the whole stream, and quickly withdrawn as soon as it is filled. With the proper mark attached to it the kettle is left undisturbed until the liquid becomes perfectly clear, then it is carefully decanted and filled up again with water and stirred well. This washing is done twice, when, finally, the settled pulp is evaporated to dryness, well mixed and quartered down.

In the laboratory, experiments can be made on the principle of trough lixiviation by introducing 20 grams of roasted ore, which, if the test is for silver, has previously been washed, into a graduated cylinder of 1000 c.c., in which is contained 100 c.c. or 200 c.c. of sodium hyposulphite solution, according to the proportion which is intended to be used. The top of the cylinder has to be tightly closed with the palm of the hand, and the cylinder has to be brought into a horizontal position, and then oscillated in order to make the ore and solution pass quickly from one end to the other, to imitate the current in a trough. This is done for about three-quarters of a minute or for one minute, then the contents of the cylinder are poured into a filter, washed with water to displace the silver solution from the sand and paper, dried and assayed. The same operations are required in experimenting with the base-metal chlorides, except that water is used instead of sodium hyposulphite solution.

Having by experiment found the required proportion of water and ore and solution and ore, the size of pumps, pipes, outlets, etc., can be calculated.

THE TROUGHS

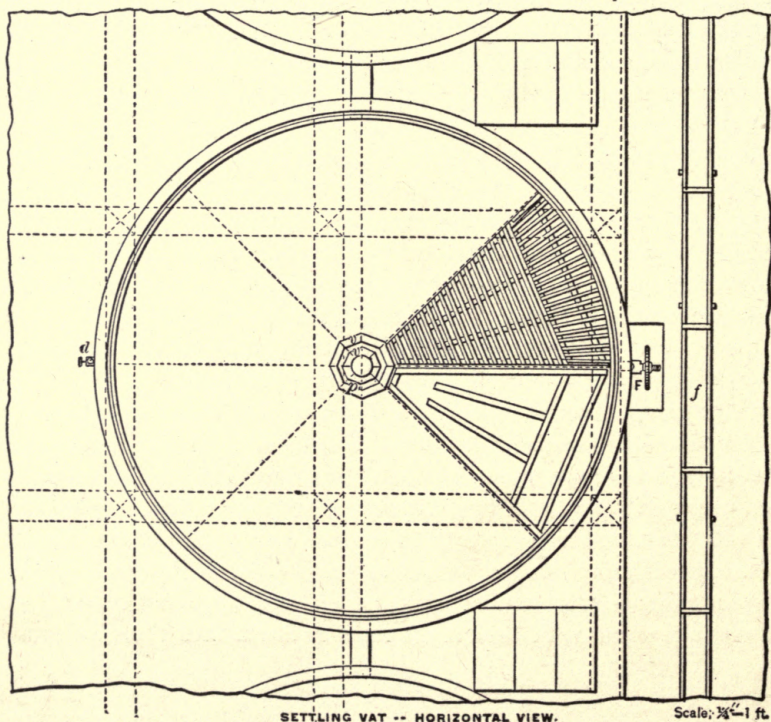
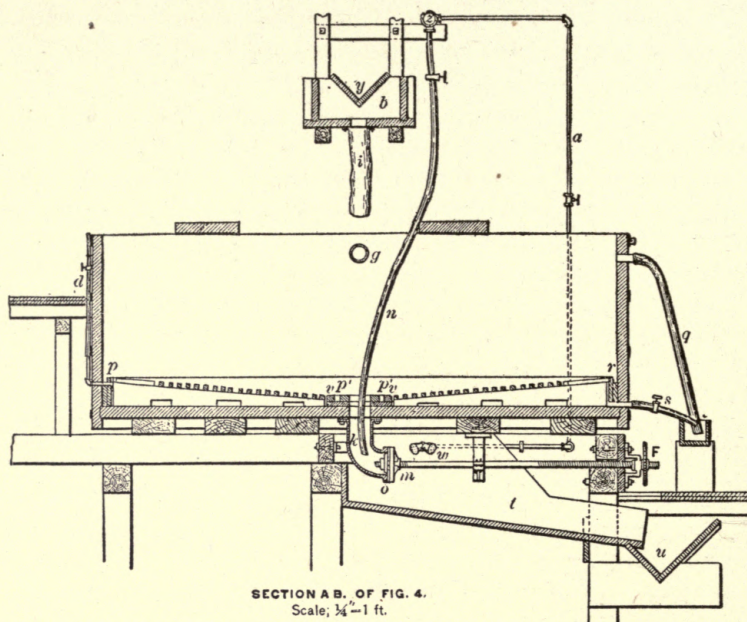
A triangular shape of the troughs is preferable, because the same quantity of water will display more energy for moving the sand than on a flat bottom. An inclination of three-fourths of an inch per foot is sufficient. We have seen that when the pulp passes through a trough 12 to 15 ft. long the extraction is completed, therefore no particular attention has to be paid to the length of the troughs. The necessary trough connection from feeder to sluice-tank and from there to the settling-tank give more than enough length. The stream in the trough moves

swiftly, and therefore very little bottom pressure will be exerted, so that it is very easy to construct these troughs so that they will be tight.

SLUICE-TANKS AND SLUICING

As the ore has to be moved from the tanks by a stream of water, it is not advantageous to give the tanks a large diameter. Twelve or 14 ft. is sufficient, though if circumstances demand it larger tanks can be used.

Figure 52 represents the vertical section, Fig. 53 the horizontal view, and Fig. 54 the front view of a settling-tank arranged for sluicing. In the center of the bottom is the discharge opening, 6 in. in diameter. The cast-iron discharge-tube, *k*, of the same inside diameter, tightly fastened to the outside of the tank bottom, corresponds with the discharge-hole. The lower end of the tube is at right angles to the upper end, and provided with flange *o*. The valve *m*, which is provided with a rubber gasket, can be pressed tightly against flange *o* by turning the wheel *F*. Flange *o* and valve *m* are made of brass. Fig. 55 shows in detail the construction of wheel *F*. Part of the valve-stem is square and rests at *m* in a square box, so that by turning the wheel *F* the valve *m* does not turn too, but moves forward or backward. By this arrangement the life of the rubber gasket is much lengthened, as no turning force is exercised against the flange *o*, but only a quiet pressure. Around the discharge opening, and fastened to the bottom of the tank, is the wooden polygon *v*, in which is cut the groove *p*₁. Around the inner periphery of the tank, and high enough to give the filter bottom an inclination of at least three-quarters of an inch to the foot, is the groove *p*. Fig. 53 illustrates the construction of the filter bottom, which is made in sections. The filter cloth is well fastened, and kept in place by driving tightly a rope into the grooves *p*₁ and *p*. The air-escape pipe *d*, which reaches to the rim of the tank, enters the latter close under the filter bottom. A piece of hose is fastened to the upper end and can be closed by a hose clamp. In Fig. 54 *q*₁, *q*, are solution outlets; *s*, filter outlet. Connecting pipes *g* and *h* (Figs. 52 and 54) have, like the discharge-tube *k*, to be well coated with asphaltum varnish. In the same figures *z* is the water-pipe; *n*, the central hose, which ought to be very stiff to resist the pressure of the ore; it reaches down into the discharge-



FIGS. 52 and 53. —SETTLING-TANK ARRANGED FOR SLUICING.

tube *k*, where it has to remain during the process of charging. Before charging the tank the discharge-tube is filled with water through the central hose, in order to keep the latter filled with water, which will prevent the inside of the hose from being obstructed by ore. In the base-metal department the central hose has to be connected with both the water- and solution-pipe. The connection with the solution-pipe serves for sluicing the ore,

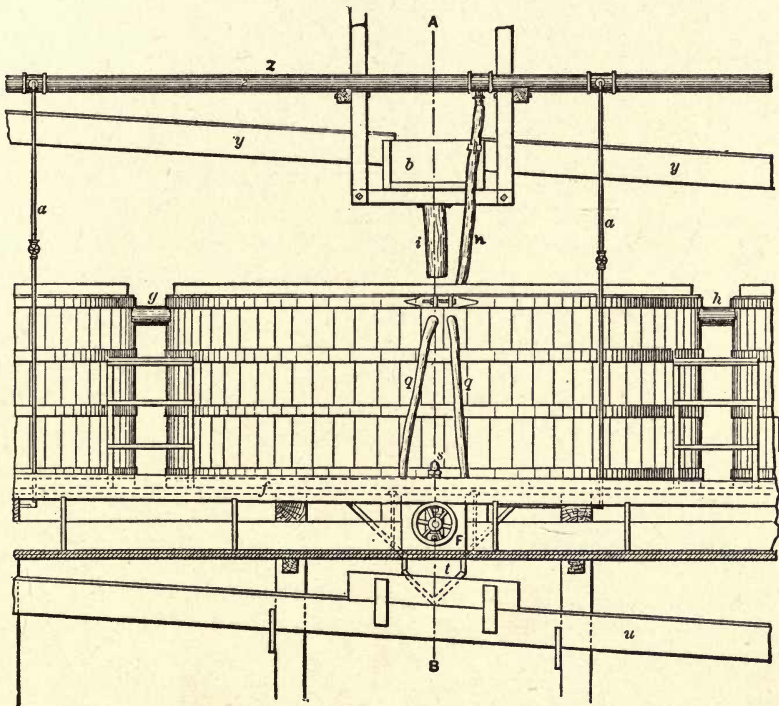


FIG. 54.—SETTLING-TANK, FRONT VIEW.

Scale, $\frac{1}{2}$ in. = 1 ft.

while the water connection is used after the tank is empty to free the tank and filter cloth from all adhering sodium hyposulphite solution by rinsing, otherwise it would get into the base-metal solution and dissolve some silver. In the silver department only the connection with the water-pipe is required, as the hose is used only for sluicing out the residues.

When a tank is ready to be discharged, the wheel *F* is turned, and thus the valve *m* pulled back. The water is injected through

the central hose, while the latter is gently moved up and down. The stream undermines the tightly packed sand, causes a continual caving in, until a funnel-shaped opening is made through its depth to the surface. Then several streams are made to play on the top, while the central hose, with checked stream, is left in position to avoid obstruction of the discharge-tube by a too sudden rush of sand.

The central position of the discharge opening and the funnel shape of the filter permit a quick and clean sluicing. The pulp leaving the discharge-tube enters the sluice-trough *t* underneath the tank, which leads to the tailings-trough *u* in the silver depart-

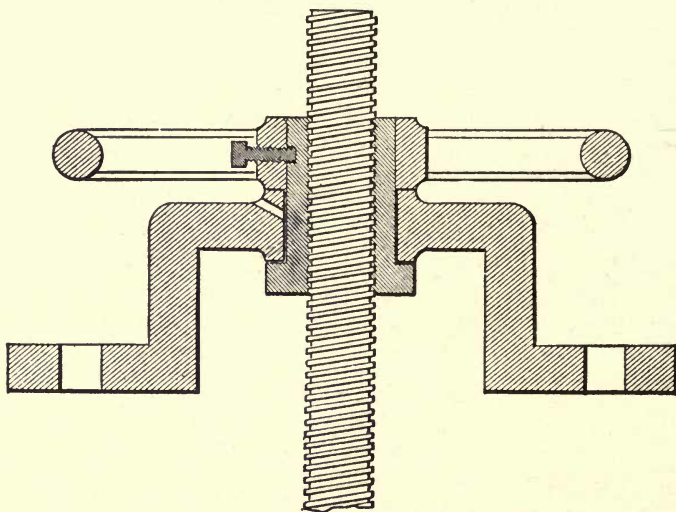


FIG. 55. — WHEEL FOR CLOSING DISCHARGE GATE.

Scale, $\frac{1}{4}$ in. = 1 in.

ment, or to the silver-leach trough in the base-metal department. The charge being sluiced out, tank and filter have to be well rinsed; the valve of branch pipe *a* is opened, and the adhering sand washed off from flange *o* and valve *m* by the double sprinkler *w*. During discharge the valve *m* has to be pulled back far enough to prevent the outflowing pulp from striking it, otherwise the rubber gasket would soon wear out. Then the discharge valve is closed again, and the tank is ready to be connected with the other tanks. In Figs. 52 and 54 *y* represents the silver-

leach trough; *b*, the intersecting box above the tank, and *i* a hose made of duck.

ARRANGEMENT AND OPERATIONS

The construction and manipulation of the sluicing tank was treated previously to the description of the general arrangement and operations of a trough lixiviation plant in order to make it better understood.

Solution is performed outside the tanks in troughs, while the ore is moving in and with the stream of solvent, and the tanks are used only to separate the solids from the liquid. The system is a continuous one; but as the lixiviation process requires two solvents, first, water for the removal of the base-metal chlorides, and then a solution of sodium hyposulphite for the extraction of silver, it has to be divided into two departments, the base-metal and the silver departments. Fig. 56 shows a complete arrangement. The upper series of tanks represents the base-metal, the lower series the silver department. The tanks in each department are placed on the same level and close together. They are connected by pipes *a*, *b*, *c*, *d*, *e*, and *f*, in such a way as to form a perfect circuit. These connecting pipes are placed a few inches below the rim of the tanks and also on a level. The diameter of these pipes depends on the daily capacity of the works and the proportion of solvent and ore to be used. Each tank has two outlet pipes *q*₁, *q*, Fig. 54, on a level with the communicating pipes, and one *S*, from under the filter. They all discharge into the base-metal solution trough which leads outside the building, either to waste or to scrap-iron tanks for the recovery of copper. Each of these tanks is arranged and constructed for sluicing, as described above.

The silver-leaching department consists of the same number of tanks of the same size and construction as those of the base-metal department, but they are placed on a lower level. They are also connected with communicating pipes forming a circuit.

On a still lower level are placed the precipitation tanks, which in this case are agitated by mechanical stirrers, and on the next platform we find the filters for the precipitate. Below this are the solution-pump vats.

The triangular trough extends above, and in the center line of each row, and wherever the trough branches off, there is

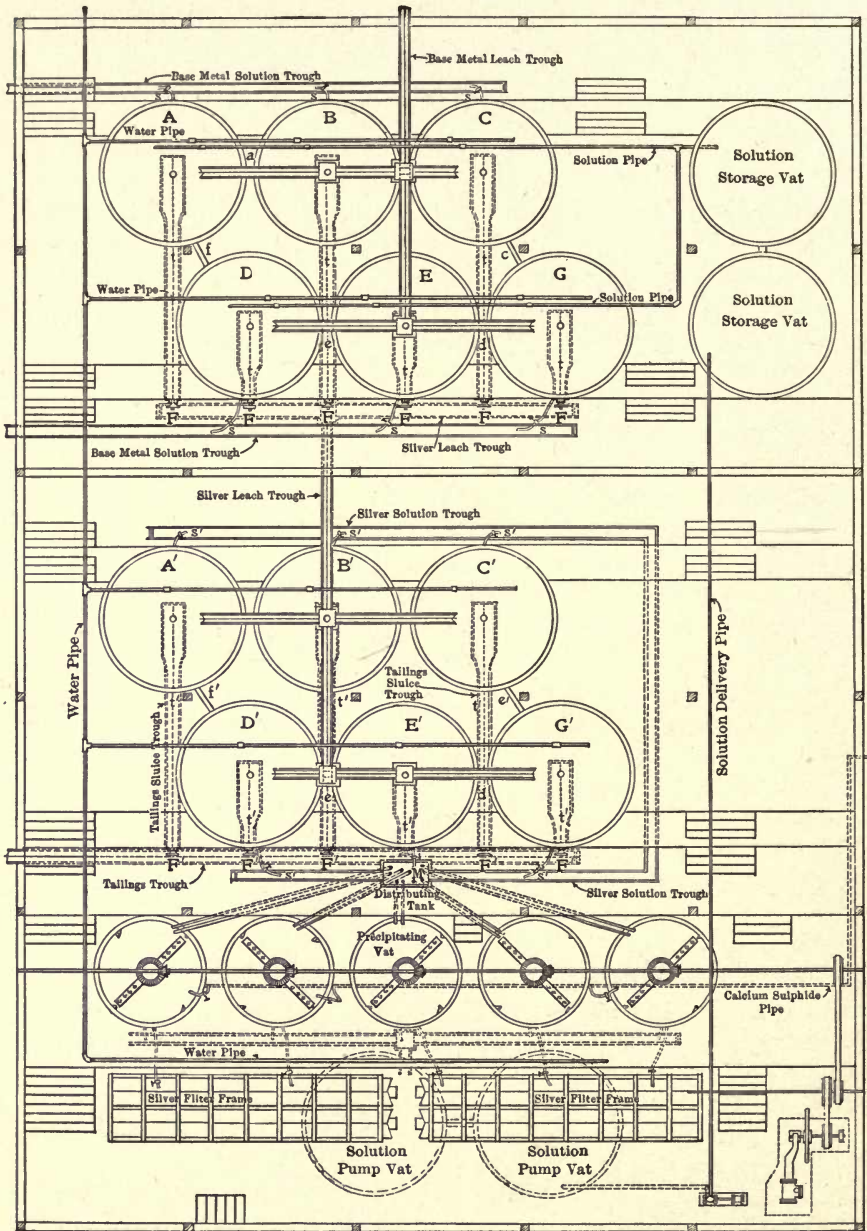


FIG. 56. — SYSTEM FOR CONTINUOUS TROUGH LIXIVIATION.
For a working capacity of 50 tons per day.

inserted an intersecting box which, if located above a tank, is provided with a plug-hole in the bottom, through which the pulp flows if that tank is to be filled. The troughs branching off from the intersecting boxes are each provided with a sliding gate, which is kept closed except in that trough into which the stream is to be directed.

The operations are as follows: The roasted ore from the cooling-floor is charged into an iron hopper which at its bottom has an adjustable worm discharge. The worm discharges into a stream of water flowing into a triangular trough which feeds the pulp into a grinding machine, in which the lumps are mashed, and which will be explained further on. Passing this lump-mashing machine, the pulp enters the triangular base-metal leach-trough, and is thus conveyed automatically to the base-metal tanks. The first tank to be filled is tank *C*, Fig. 56; but before commencing operation, the following preparations have to be made. The slide-gate of the trough leading from the intersecting box to tank *C* is opened, while those of the other two troughs are closed. The connecting pipe *c*, between tanks *C* and *G*, is closed by a plug from the inside of *C*. The outlets of all the tanks are closed except the outlets q_1 q , near the rim of tank *G*, Fig. 54. Likewise all outlets from under the filters are closed. Then the central hose *n*, Fig. 52, is inserted into the discharge-tube, the valve of which is also closed. This done, the pulp is permitted to flow into tank *C*, Fig. 56. When the solution reaches the level of the connecting pipes, it will flow through connecting pipe *b* into tank *B*, and when *B* is filled into tank *A*, and so on until the last tank *G* is filled, when the solution will leave the tank through the two outlets near the rim. As soon as tank *C* is filled with ore, the pulp is made to flow into tank *B*. The connecting pipe *b* is closed, and thus tank *C* is disconnected from the circuit. The outlet *s*, below the filter of tank *C*, is opened, and the solution is allowed to drain into the base-metal solution trough. When the solution begins to disappear below the surface of the ore, water is admitted, to press out the solution absorbed by the ore. When this is done, sodium hyposulphite solution is applied to press out the water. As soon as the liquid flowing out through filter outlet *s* shows indications of silver, the outlet *s* is closed, and the charge is ready to be sluiced for silver leaching. While tank-charge *C* is under the described treatment,

which does not take much time, tank *B* is gradually filling with ore. When filled, the pulp is made to enter tank *A*, and tank *B* is disconnected from the circuit, and its charge is treated in the same way as that of *C*.

When the charge *C* has been prepared as described above, it is sluiced out with sodium hyposulphite solution to extract the silver. Underneath the tank the pulp enters the silver leach-trough and flows down to tank *C*, of the silver department. The arrangement and construction of the tanks are the same as in the base-metal department, and when operations have been started the connections have to be set the same as described for base-metal leaching.

Charge *C*, being sluiced out with solution, the filter-outlet *s* is turned into the silver leach-trough, the hose clamp is opened, and the solution contained under the filter is allowed to flow out, and then the inside of the tank and the filter is rinsed with water, which is also allowed to flow into the silver leach-trough. This accomplished, outlet hose *s* is closed and returned to the base-metal solution trough. Then the plug of connecting pipe *c* is removed, and tank *C* is restored again to the circuit. *C* being empty, the flow in the base-metal solution trough will cease until the tank is filled again with base-metal solution.

By using the proper proportion of solvent and ore, the latter will drop as tailings into the tank, while all the silver chloride which can be extracted by the common chlorination test will be dissolved by the solution. It is well to have a special solution pipe by which a stream can be made to flow directly into the silver leach-trough close to the tank, so that if the volume of solution used in sluicing should not be sufficient, the proportion can be maintained by the use of this extra pipe.

The clear solution leaving the last tank *G'* flows through the silver solution trough into the distributing-box, and from there to any desired precipitating tank. The bottom of the distributing-box has twice as many holes as there are precipitating tanks. In these holes bent lead pipes are inserted from below and are fastened by flanges. Stiff two-inch hose-pipes, of which two lead to each precipitation tank, are attached to these pipes. The holes can be closed by long plugs. This arrangement I found quite convenient. The operator can direct the stream from the main working floor, without being obliged to creep over

all the precipitating tanks, as is customary in many lixiviating works.

When one tank is filled with tailings, it is disconnected from the circuit and the pulp is admitted into the next tank. Outlet s_1 , under the filter, is opened; the solution still contained in the tank is allowed to drain off, and the part retained by absorption is displaced by water. The tailings are then sluiced with water. Where water is scarce, the wash-water can be collected and used for sluicing out the final tailings.

It is of advantage to connect the solution-pipe with the solution-pump. In the first place a higher pressure can be obtained than from the storage tanks, and on the other hand by the number of pump-strokes per minute the volume of solution can be calculated. The extra stream into the trough outside the tank, if one should be used, it is better to have from the storage tank, as the high pressure from the pump would splash in the trough.

PRECIPITATING VATS

Figs. 57 to 61 illustrate the construction of precipitating vats and a convenient arrangement of filters of the precipitate which I devised and have operated. This construction of tank and filters is particularly adaptable for works which are not equipped with air-compressor and filter-presses. The tanks are provided with machine-stirrers, of the construction indicated in the drawings. The stirrer s (Figs. 57 and 58) has to make about 30 r.p.m. if the diameter of the tank is not more than 8 or 9 ft. It is set in motion or stopped by working the friction-clutch f (Fig. 57). The wings, g (Fig. 58), which reach near to the bottom, are about 3 in. wide and are kept in position by triangular pieces of boards. They break the violent current around the periphery and throw the solution toward the center, thus causing a strong whirling motion. In Figs. 58 and 59, a is the discharge-pipe, or decanter, for the clear solution, and b for the precipitate. Pipe d , Fig. 59, passes in front of all the precipitation tanks, and conveys the calcium or sodium sulphide solution from the reservoir to the tanks. The branch pipe c reaches above the rim of the tank and ends in a rubber hose, which is provided with a clamp. In precipitating, the stream can be conveniently regulated by the use of the clamp, and the operator, by observing the color produced by the precipitant in the moving

solution, can finish this operation in a very short time, and much more easily than by using buckets.

One man can precipitate three tanks at a time without assistance. The solution is so thoroughly agitated that a very perfect separation of the silver sulphide takes place. The separation is so perfect that the bottom of the tank can distinctly be

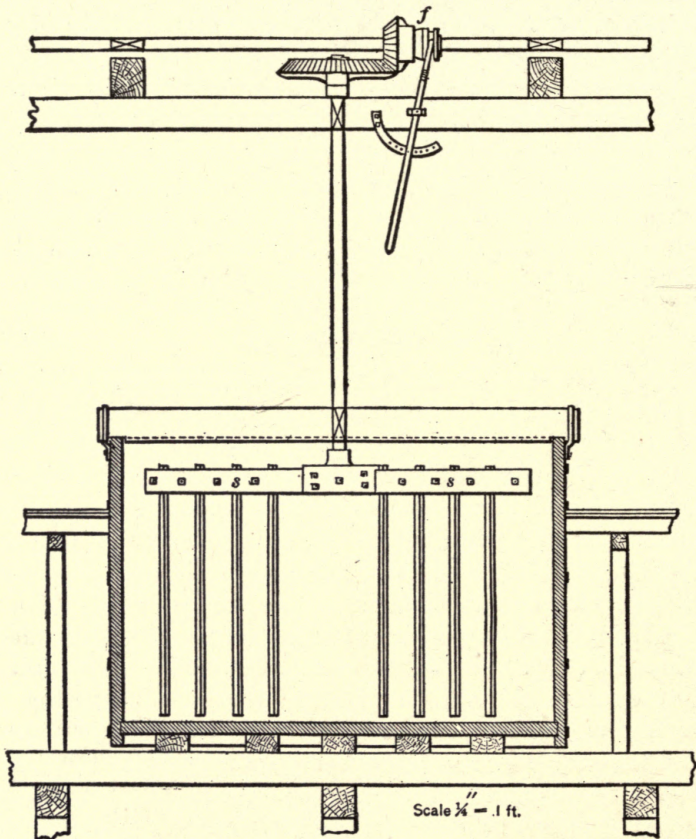


FIG. 57.—PRECIPITATION TANK, VERTICAL SECTION.

seen through 5 ft. of solution. To produce a quick and perfect separation of the precipitated silver sulphide, the solution has to be vigorously agitated. This cannot be well done in tanks of 14 or 16 ft. in diameter. It is much preferable to have smaller tanks and a larger number of them. A good size is 8 to 9 ft. in

diameter and 6 ft. deep. In some leaching works where there are large precipitation tanks in use, we find the bad practice of discharging the precipitate only once a week, in some even only once a month. Fresh precipitate forms large flakes, which settle easily and cleanly. After two or three days it assumes a dry and sandy condition, and if stirred up divides into very fine particles, which are kept suspended in the solution for a long time. The result of such a practice is that the decanted solution will not be free from precipitate when used again for extraction, and a black

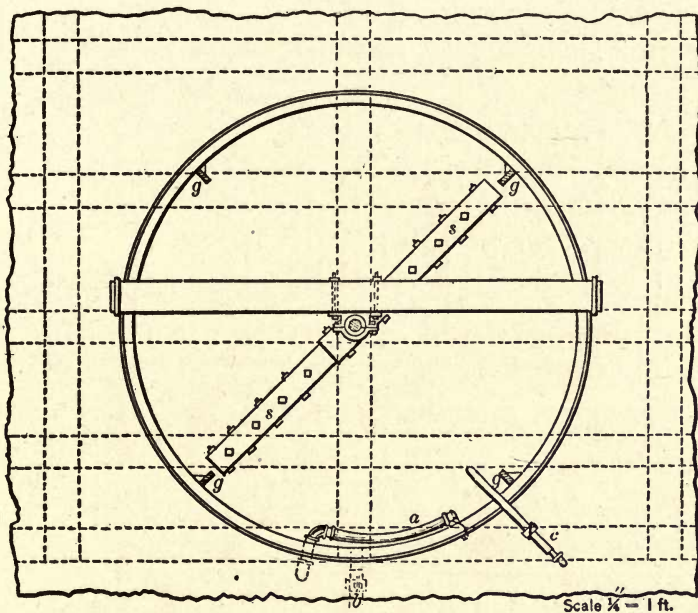


FIG. 58.—PRECIPITATION TANK, PLAN.

coating of precipitate will cover the surface of the ore in the leaching tanks. The precipitate ought therefore to be removed from the precipitating tanks every day, and should never be allowed to remain longer than two days. By using smaller tanks with the machine-stirrer, and by discharging the precipitate every day, or every other day, the circulating solution can be kept so clear that, even after a prolonged leaching of five or six days, no black coating can be observed on the top of the ore.

The machine-stirrer does good work also in discharging the

precipitate. After the clear solution has been decanted within a few inches of the precipitate, the stirrer is set in motion, and the discharge-pipe *b* opened. The stirrer agitates the precipitate

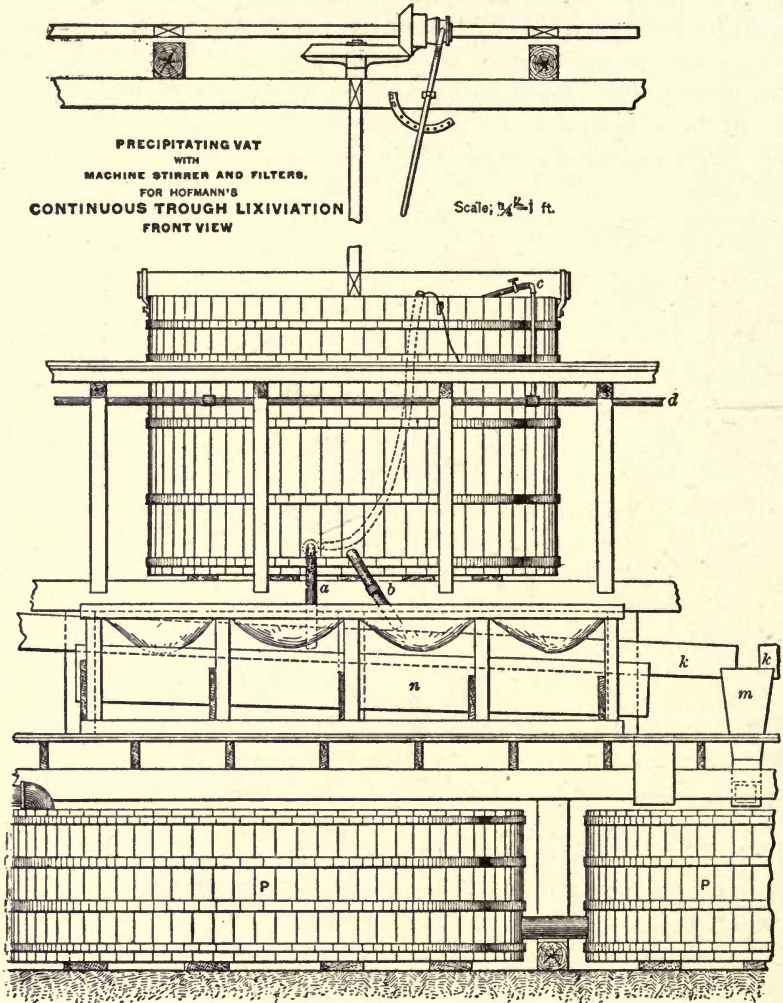
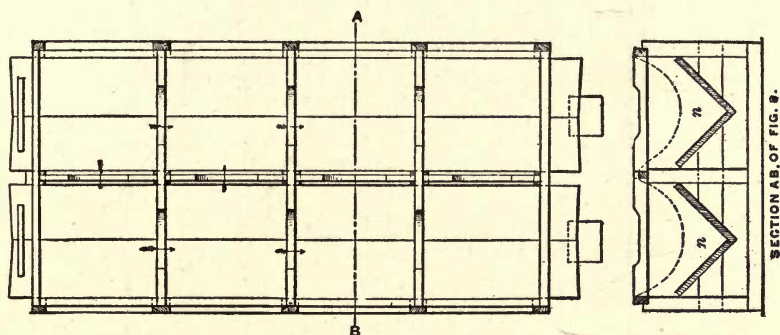


FIG. 59. — PRECIPITATING VAT.

until nearly all is discharged into the filters, or into the pressure tank by which the filter-press is filled. Thus the cleaning of a tank can be done in a very short time.

Where the filter-press is not used, a filter arrangement as shown in Figs. 59, 60 and 61 will be found very convenient. Two rows of filters are so arranged that they are in communication with each other by depressions cut into the divides of the frame. By allowing the precipitate to flow into one filter, all the adjoining filters will gradually be filled, one after the other, without requiring the attention of the operator. The filters, which are made of common cotton sheeting, are shallow, and the precipitate will thus be spread in a comparatively thin layer over a large filtering surface. Under each row of filters is placed a trough, *n*, which receives the filtrate and conveys it to the pump-tank *P*, below the floor. These filter frames are placed in front of the precipitating tanks and can be made to contain quite a number



FIGS. 60 and 61.—FILTER FRAME.

Fig. 60 is a horizontal view. Fig. 61 is a section on line A-B.

of filters, as shown in Fig. 56, where two sections serve for five precipitating tanks. Owing to the shallowness of the filters and the large filtering surface, the solution will drain off fast, and in five or six hours the precipitate will be stiff enough to be charged with wooden hand-paddles into the drying furnace.

The extraction in trough lixiviation is not produced by filtration; and the silver as well as the base-metal chlorides contained in the lumps of the roasted ore will not be extracted if the lumps are allowed to enter the trough together with the finer portion. Actual grinding is not necessary, but in order to obtain good results it is necessary first to mash the lumps. An agitator will not perform the work rapidly enough, and some quick-grinding machine has to be employed for this purpose.

Figs. 62, 63, 64 and 65 represent a grinding machine specially designed by myself for use in trough lixiviation. The mantle *m* is concave-shaped and is stationary. The muller *f*, a flatter cone than the concave of the mantle, is inserted from below, and can be lowered or raised by the screw *g*, for the purpose of regulating the fineness to which the lumps are intended to be mashed.

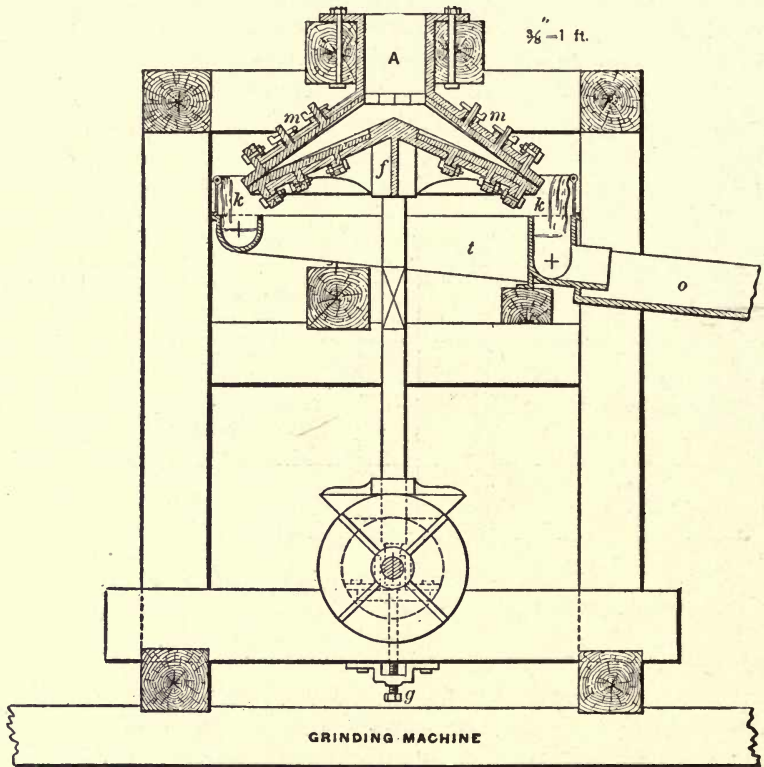


FIG. 62.—LUMP-GRINDING MACHINE, ELEVATION.

Mantle and muller are provided with exchangeable shoes and dies. The centrifugal force developed by the rotation of the muller greatly assists the discharge of the pulp. Toward the center the shoes and dies are provided with teeth, while toward the periphery their surface is smooth (Figs. 64 and 65). The teeth cut the larger lumps, the smaller ones are mashed by the smooth part of the cones. Water and ore are charged through *A*, while

the circular cast-iron trough *t* receives the pulp and conveys it to the base-metal leach-trough *o*. The canvas strip *k* prevents the pulp from flying over the rim of the trough. By far the main portion of the roasted ore is fine enough to pass through without

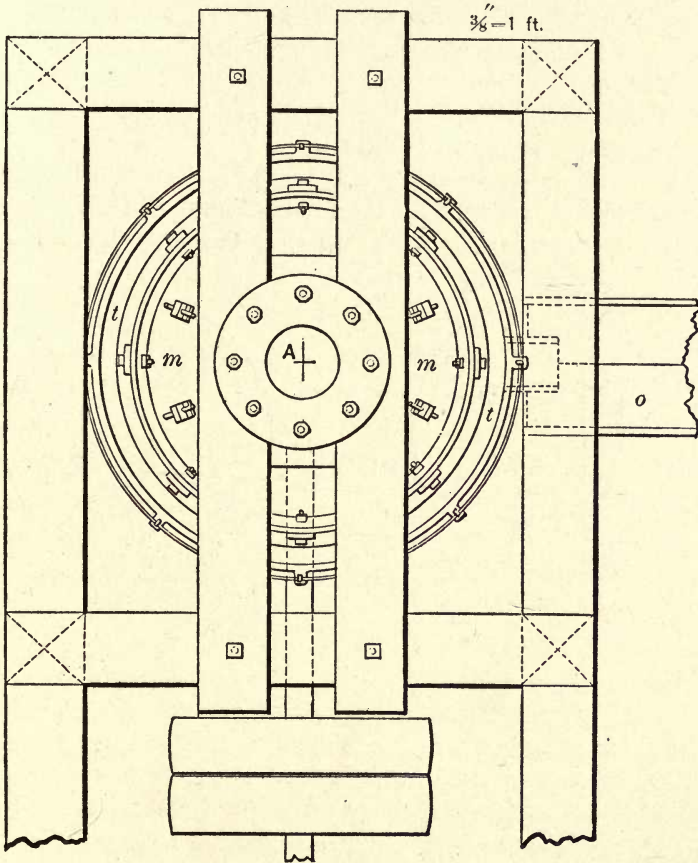


FIG. 63.—LUMP-GRINDING MACHINE, PLAN.

being affected, and only a small part will have to be mashed. For this reason, and on account of the softness of the wet material, large quantities of ore can be put through in twenty-four hours with but very little wear of the machine, which in itself is of simple and cheap construction.

PRACTICE OF TROUGH LIXIVIATION AT CUSIHUIRIACHIC.

The Don Enrique Mining Company at Cusiuhiriachic had accumulated a large dump of second-class ore, which contained about 25 oz. silver per ton, but filtered so badly on account of the large amount of porphyry in the gangue that it was not profitable to work it by tank lixiviation. After the large lixiviation works of this company were destroyed by fire, I undertook to work this second-class ore by trough lixiviation, and treated 12,000 tons by this method, at a good profit.

The work was done in the old North Mexican mill buildings, which were formerly used for tank lixiviation until the ore in the company's mine gave out. There were two sluicing-vats with

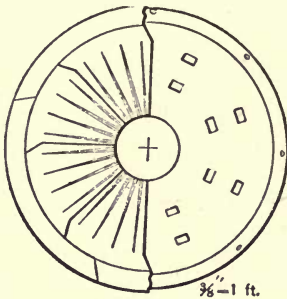


FIG. 64. — Mantle.

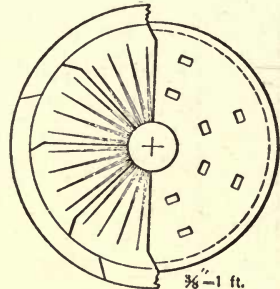


FIG. 65. — Muller.

LUMP-GRINDING MACHINE.

central discharge and flat, funnel-shaped filter bottoms, which were placed about 10 ft. above the rim of the old leaching-vats, of which there were eight, arranged in two rows. Each vat measured 14 ft. in diameter and $3\frac{1}{2}$ ft. in depth. These tanks were used as settling-tanks, and were connected each with the other by a 4-in. pipe inserted near the rim, thus forming a circuit of all the vats, so that if one tank was filled the solution could flow into the other, and from there into the next, and so on. The filter bottoms of these vats were covered with sheeting and a 4-in. layer of washed river sand. On the cooling-floor a small hopper with funnel-shaped bottom was erected and covered with an inclined screen of $\frac{1}{8}$ -in. mesh, to prevent the lumps from entering. The hopper discharged to a short screw conveyor, and the conveyor into the cups of a belt elevator. The elevator lifted the

roasted ore and discharged it into a short triangular trough, in which a stream of water was running. The trough was so arranged that the pulp could be conveyed to either of the two sluicing-tanks.

The speed of the screw conveyor being always the same, the feed of roasted ore into the trough was uniform, and the desired proportion of ore and water was easily regulated by the stream of water. When the first vat was filled the pulp was made to enter the second sluicing-vat. While the second vat filled, the liquid in the first vat became clear and was drawn off by siphons, so that when vat No. 2 was full vat No. 1 was ready to receive again the stream of pulp. This was repeated until both vats were fairly filled with washed ore. After the first charge of pulp the outlet under the filter of each vat was opened, discharging a clear stream while filling was in progress, the ore, which had by this time settled on the bottom, acting as filter. This increased the filling capacity of the vats. When both tanks were charged the solution above the ore was allowed to drain, and then water was applied to displace the solution absorbed by the ore.

The silver-leaching troughs were not longer than was necessary to reach to each of the eight settling-vats. The intersecting boxes over each settling-vat were 18 in. square and 12 in. deep, with a 3-in. round opening in the bottom, which could be closed with a wooden plug. To prevent splashing during charging a 4-in. canvas hose was fastened around the opening in the bottom, reaching down to within a few inches below the rim of the settling-vat.

Shortly before sluicing began the plug in the box above the first tank was removed and a piece of filter cloth, kept in place by several bricks, was spread directly under the canvas hose, to protect the sand filter. The pulp, dropping always on the same place in the vat, the ore, or rather the residues (because the silver is already extracted when the pulp drops into the vat), will form a cone, which, however, will never project much above the surface of the solution, because the material in the solution, being loose and lighter than the material above the solution, will slide down. Thus a tank of 14 to 16 ft. diameter will be charged pretty evenly. To fill the lower space around the periphery a short trough was placed under the drop, and by changing the position of the trough gradually the stream was directed to all

points. Long before the vat was filled with residues the solution reached the level of the communicating pipe and flowed into the next tank (No. 2), and then into No. 3 and No. 4. When tank No. 2 was filled with solution the outlet under the sand filter was opened and a clear stream of solution was discharged. The same was done when No. 3 was filled with solution, and so on. The filtration was very free, and the volume of solution entering vat No. 3 was much reduced, and still more so the overflow into No. 4, so that the solution seldom occupied more than three vats besides the one which was undergoing charging. This gave ample time to treat and discharge the residues, and to renew the filter, if necessary, before the vat was again required in the circuit. By providing each tank with a sand filter only clear filtered solution entered the precipitating vats, and no overflowing solution at all.

When tank No. 1 was filled sufficiently with residues the stream of pulp was changed to flow into vat No. 2 by opening the corresponding plug-hole and closing the one above No. 1. The communicating pipe between vat No. 1 and No. 2 was closed, and the solution allowed to drain and then displaced by water. The leaching with water was continued until the outflowing stream did not show any reaction for silver. When a vat was partly filled with residues the outlet under the filter could be opened, giving a clear stream, though the vat was still undergoing the process of filling. When vat No. 1 was disconnected from the circuit and No. 2 subjected to the operation of charging, the overflowing solution moved one tank farther and commenced to fill No. 5; when, in the course of the operation, the overflowing solution reached vat No. 8, vat No. 1 was empty and prepared to receive the overflowing solution. To prevent the overflowing stream, when entering an empty tank, from washing away the filter sand, a piece of filter cloth was spread over that part of the filter and tacked to the inside of the vat, so that the stream which flowed down on the side of the vat did not do any harm to the filter.

Samples of the pulp taken at the drop showed that all the silver chloride was extracted while the pulp was flowing through the trough, and that the ore actually dropped as spent residues into the vat. A similar experience was realized with regard to the base-metal salts. After the pulp (ore and water) passed

through the short trough from the elevator to the sluicing-vat, it was found by samples taken at the drop that all the heavy metal salts soluble in water had been dissolved, and that only some sodium sulphate still remained. Regular samples of the base-metal solution were taken, but never found to contain any silver.

TROUGH LIXIVIATION EXPERIMENTS ON A LARGE SCALE

While investigating the metallurgical problem of the lead-zinc ores of the San Francisco del Oro ore near Parral, Chihuahua, Mexico, experiments were also made to treat the roasted ore by the trough-lixiviation method. These experiments were made previous to the working of the second-class ore dumps at Cusi-huiriachic, described above. Experiences gained by this experiment were used to advantage in working the "Cusi" ore.

The Bosque mill, near Parral, was very inconveniently arranged. The main inconvenience was the want of grade; therefore, the locality did not permit the erection of a complete system for trough lixiviation, and the experiments had to be made with only one circuit of six tanks, and I was obliged to use the same troughs and tanks for base-metal and afterwards for silver leaching. The washed ore had to be removed from the tanks and brought to the head of the trough for silver leaching. Notwithstanding this inconvenience, the experiments gave very interesting results and information.

By a triangular trough, 138 ft. in length, $\frac{3}{4}$ -in. fall per foot, with a feed-box at the upper end, and intersected by five square boxes, the pulp could be conveyed to any of the six tanks of the circuit. The tanks were connected by pipes inserted near the rim.

The ore used in this experiment was roasted in the modified Howell furnace. It was charged into a running stream of water at the rate of 64 tons per twenty-four hours. The pulp passed through the whole length of trough in fifty-five seconds.

In order to find out how much of the base-metal salts were dissolved during this short time, and to ascertain the required length of trough, samples were taken at different places, dried and then subjected to a thorough washing in the laboratory, with the following results:

Roasted ore before troughing contained 12 per cent. in salts soluble in water.

(1) The sample taken after the pulp passed the entire length of 138 ft. still contained in salts soluble in water 4.9 per cent.

(2) The sample taken after the pulp passed through 58 ft. of trough still contained in soluble salts 4.5 per cent.

(3) The sample taken after the pulp passed through 12 ft. of trough still contained in soluble salts 3.6 per cent.

The above results are just in reverse order from what would be expected; but it was not possible to take the sample from the same portion of moving pulp, which may account for this irregularity. Though this assumption is rather arbitrary, we may accept it for want of a better explanation. If we take the average of the three results, we find that the pulp after troughing still contained 4.7 per cent. of salts soluble in water, or, as the roasted ore before washing contained 12 per cent. of such salts, that 60.8 per cent. can be extracted while the pulp passes through 12 ft. of trough, or in 4.7 seconds. Long troughs are, therefore, not essential for base-metal leaching. In order to ascertain if, in tank lixiviation in the usual routine, a larger percentage of the soluble salts is extracted, a sample was taken from a tub charge after it had been washed for eight hours and was ready for silver leaching. The outflowing water gave with calcium sulphide only faint white clouds, the usual indication that base-metal leaching is completed. The sample, after drying and weighing, was subjected to a second washing in the assay office, and the result showed that of the original percentage of soluble salts 61.7 per cent. were extracted by leaching in the tanks, which is only 0.9 per cent. more than was extracted in the trough in 4.7 seconds.

In both cases about the same percentages of soluble salts are retained by the ore, which only by a prolonged leaching can be removed. They are not heavy metal salts, but principally sodium sulphate and sodium chloride. In the present case mostly sodium sulphate; for an analysis of the stock solution, after three months' use, showed it to contain only 0.098 per cent. chlorine, while the white clouds produced by an addition of calcium sulphide proved to be gypsum.

TIME REQUIRED FOR BASE-METAL LEACHING

Though the dissolving of the base metals is almost instantaneous, considerable time is consumed in preparing the charge

for silver leaching, caused principally by the time required to press out the base-metal solution by water; this time was found to be 3 hours and 25 minutes for a charge of 8.39 tons. However, the total time is still 3 hours and 35 minutes less than in tank lixiviation.

The time is divided as follows:

IN TROUGH LIXIVIATION

| | hrs. | min. |
|------------------------------------------------------|------|-------|
| Leaching and filling the tanks | 3 | 6 |
| To drain remaining solution from top of the ore..... | - | 34 |
| To press out base-metal solution by water | 3 | 25 |
| To press out water by hypo solution..... | 1 | 20 |
| Total time | 8 h. | 25 m. |

IN TANK LIXIVIATION

| | hrs. |
|-----------------------------------------------|---------|
| Charging | 3 |
| Base-metal leaching | 8 |
| Pressing out with hyposulphite solution | 1 |
| Total time | 12 hrs. |

QUANTITY OF WATER REQUIRED

Sufficient water had to be used to make the pulp move freely through the trough, and to produce a sufficiently diluted base-metal solution in order not to dissolve any silver chloride. The results were obtained with 702 gallons per ton of ore, which is equivalent to about one weight of ore to three of water. When the tank was charged, and clean water turned on to press out the solution, the speed of filtration was 12 inches per hour in a tank of 10 ft 2 in. diameter, which is equivalent to 2065 gallons in 3 hours 25 minutes for a charge of 8.39 tons, or 246 gallons per ton. After silver leaching, it took 2 hours 30 minutes to press out the hyposulphite solution. Summing up, we find the total consumption of water as follows:

| | |
|-------------------------------------------------|----------------------|
| In troughing | gal. |
| Pressing out the base-metal solution by water . | 702 |
| Pressing out the hyposulphite solution by water | 246 |
| Total consumption per ton | 1129 = 150.5 cu. ft. |

In tank lixiviation the consumption of water was found to be 703 gallons per ton of ore, or 93.7 cu. ft., which shows an increased consumption in trough lixiviation of 56.8 cu. ft. per ton.

QUANTITY OF SILVER DISSOLVED BY THE BASE-METAL SOLUTION

One liter of the 702 gallons of base-metal solution was precipitated with calcium sulphide. The precipitate, after fluxing and treating like a common ore assay, returned not more than 0.0002 grams fine silver. If one liter contains 0.0002 grams silver, 702 gallons will contain 0.532 grams, which is the total amount of silver dissolved from the whole charge of 8.39 tons of ore, or 0.06 grams, equal to 0.002 oz. silver per ton. This is practically nothing, and the wash-water can therefore be allowed to run to waste, without causing any perceptible loss in silver.

SILVER LEACHING

After three tanks were filled, and the base-metal solution pressed out with water, and the water with a 0.38 per cent. sodium hyposulphite solution, the ore was allowed to drain. Then the ore was shoveled out and removed for silver leaching to the head of the trough. Being at a time rather late in the evening, the ore, saturated with hyposulphite solution, was left in a pile over night. The next morning, however, it was found that some of the silver chloride was decomposed during the night, and that the chlorination test tailings had increased from 5.24 oz. to 9.03 oz. per ton. Taking the tailings value of 9.03 oz. per ton as basis, the experiment was continued. The measured stream of solution was kept uniform, while the rapidity of charging the ore was changed according to the desired proportion. In order to ascertain the proper length of trough, samples were taken at different places, with the following results:

Assay office chlorination tailings of the pulp, 9.03 oz. per ton; strength of solution, 0.38 per cent.; proportion, one weight of ore to five of solution; rate of working, 38 tons of ore per day.

(1) Sample taken from spout of feed-box when entering the trough: Tailings, 9.8 oz. per ton.

(2) Sample taken after passing 12 ft. of trough: Tailings, 8.13 oz. per ton.

(3) Sample taken after passing 70 ft. of trough: Tailings, 8.85 oz. per ton.

(4) Sample taken after passing 100 ft. of trough: Tailings, 8.13 oz. per ton.

(5) Sample taken after passing 120 ft. of trough: Tailings, 8.60 oz. per ton.

(6) Sample taken after passing 138 ft. of trough, while dropping in tank: Tailings, 8.60 oz. per ton.

This experiment gave the very surprising information that actually only a few feet of trough are required to produce a perfect dissolving of the silver chloride. It shows that the passing through 12 ft. of trough, or in 4.7 seconds, the extraction is complete, and that longer troughs are, therefore, not necessary. This is of importance, as it simplifies the construction of trough-lixiviating works and reduces the required grade.

OTHER PROPORTIONS, WORKING THE SAME LOT OF ORE

(7) Proportion, 1 ore to 3.4 solution; working rate, 55.8 tons per twenty-four hours; tailings, 7.89 oz. per ton.

(8) Proportion, 1 ore to $2\frac{1}{4}$ solution; working rate, 84.5 tons per twenty-four hours; tailings, 9.56 oz. per ton.

(9) Proportion, 1 ore to 10 solution; working rate, 19.05 tons per twenty-four hours; tailings, 9.09 oz. per ton.

These results show that the proportion of 1 ore to 3.4 solution gave the best results, the tailings being 1.14 oz. per ton poorer than the chlorination assay called for.

A second series of experiments was made, and particular attention was paid to avoid the decomposition of silver chloride. The charges were subjected to silver leaching soon after being saturated with hyposulphite solution.

Chlorination test tailings, 5.25 oz. per ton.

Strength of solution, 0.50 per cent.

(1) Proportion, 1 ore to 3.4 solution; tailings, 3.59 oz. per ton.

(2) Proportion, 1 ore to 6 solution; tailings, 3.8 oz. per ton.

These are very satisfactory results; the tailings are as poor as, in fact poorer than, those obtained in tank lixiviation after four days' silver leaching. The proportion 1 to 3.4 proved again to be sufficient, producing tailings 1.66 oz. poorer than the chlorination test called for; the quantity of solution required for this ore is, therefore, very moderate, much less than that required in tank lixiviation.

QUANTITY OF SOLUTION REQUIRED

By using the proportion 1 : 3.4 we need 100.8 cu. ft., or 816 gal. of solution to circulate for each ton of ore. In tank lixiviation the required quantity for this ore was found to be 658 cu. ft., or 4935 gal. for each ton of ore, or about six times as much as in trough lixiviation.

TIME REQUIRED FOR SILVER LEACHING

| | hrs. | min. |
|-----------------------------------------------------|------|------|
| Troughing and filling the tank | 3 | 36 |
| Draining the solution from the top of the ore | — | 34 |
| Pressing out the solution with water | 2 | 30 |
| Total time | 6 h. | 4 m. |

In the following we will compare the total time required by the two methods from the time the ore enters the leaching works until it is ready for discharge.

TIME REQUIRED IN TANK LIXIVIATION

| | hrs. |
|------------------------------------------|-------------|
| Charging | 2 |
| Base-metal leaching | 8 |
| Pressing out the water by solution | 1 |
| Silver leaching | 96 |
| Pressing out the solution by water | 1½ |
| Total time | 108½ hours. |

TIME REQUIRED IN TROUGH LIXIVIATION

| | hrs. | min. |
|-----------------------------------------------------|-------|------|
| Base-metal leaching and filling the tank | 3 | 6 |
| To drain the wash-water from top of ore | — | 34 |
| To press out the base-metal solution by water | 3 | 25 |
| To press out the water by hyposulphite solution ... | 1 | 20 |
| Silver leaching (sluicing with solution) | 3 | 36 |
| Draining solution from top of ore | — | 34 |
| Pressing out the solution by water | 2 | 30 |
| Total time | 15 h. | 5 m. |

To work a charge of Del Oro ore by the trough system takes 15 hours 5 minutes, while by tank lixiviation it takes 108 hours 30 minutes, or about seven times as long.

FINENESS OF THE PRECIPITATE

The burned silver precipitate contained 20.9 per cent. of fine silver, while that obtained in tank lixiviation during the same week and from the same lot of ore contained only 17 per cent. fine silver.

ADVANTAGES OF TROUGH LIXIVIATION

It can clearly be seen that trough lixiviation offers many advantages over tank lixiviation, especially in large works, or if badly filtering or slowly extracting ore has to be treated. In large works, where twenty or more leaching-vats are in operation, each one in a different stage of the process, much attention is required to avoid mistakes, while in trough lixiviation care has to be given to only a few tanks. If the ore filters badly it will take a very long time to extract the silver by leaching in tanks, while in troughs the silver as well as the base-metal salts dissolve almost instantaneously, and the effect of the bad filtering will be felt only while the charge is draining and the solution is being displaced by water. It makes the treatment of very badly filtering ore possible, which otherwise could not be treated by lixiviation.

The most important advantage of trough lixiviation is the fact that this method enables the operator to bring the ore into sudden contact with any desired quantity of the solvent. This is a very important fact, as it offers the means to do away with the special treatment of the base-metal solution, because a solution of this can be made sufficiently diluted not to dissolve any silver chloride. Furthermore, the extraction of silver from lead-bearing ores is slow, and requires an extensive plant. It is well known to leachers of such ores that, while the main portion of the silver is extracted in a short time, the remaining few ounces will be tenaciously retained by the ore. Thus it happens that, while the main portion of the silver can be extracted in the first six or eight hours, the remaining eight or ten ounces of extractable silver will require three or four days, sometimes more. I have seen cases in which only one ounce per ton in every twenty-four hours of prolonged lixiviation could be extracted. This very singular phenomenon is difficult to explain. It seems that only that part of the silver is so difficult to extract which originally was contained in the lead-mineral of the ore. I have observed that whenever the galena of the ore became richer in silver, or had increased in quantity, the extraction became slow and dragging. This, together with the fact that the main portion of the silver can be quickly extracted, indicates that the slowness of the extraction is not principally due to the disadvantageous in-

fluence of lead sulphate on the dissolving energy of the solution for silver chloride, but that it must be due to some other cause, which prevents the silver contained in the lead ore from dissolving quickly except in large volumes of sodium hyposulphite solution. In troughs, when the ore is brought at once into contact with the required volume of solution, the silver dissolves almost instantaneously, and the presence of lead ore does not retard trough lixiviation: it merely entails the use of larger quantities of solution, which is undoubtedly a very advantageous feature of this method.

The cost of a plant is much less if arranged for troughs, as large quantities of ore can be treated in a comparatively small plant.

XVIII

THE RUSSELL AND KISS PROCESSES

(1) THE RUSSELL PROCESS

THE Russell process, which was so elaborately and well written up, and about which so many statements of excellent results were published, and which was in consequence thereof introduced at several places with a large expenditure of capital, has not proved a success. As this process attracted much attention and has found its way into all metallurgical text- and hand-books, it is interesting and instructive to investigate the cause of the failure.

It was claimed that the "extra solution" — a solution of a double salt of cuprous hyposulphite and sodium hyposulphite, manufactured by adding a solution of copper sulphate to a solution of sodium hyposulphite — exerted a highly energetic dissolving and decomposing action upon metallic silver, silver sulphide, silver minerals belonging to the group of antimonial and arsenical sulphides, and other silver combinations. Based on this property of the "extra solution," it was claimed that silver ores treated by this process required a less careful chloridizing roasting, or only an oxidizing roasting, and that even raw sulphureted ores could be successfully desilverized.

These claims attracted general attention, and if they had been true in the sense in which they were given out this process would have marked a decided step forward in the hydrometallurgy of silver; but these claims were based on results obtained on a very small scale, and under conditions which are not practicable to create and maintain on a large scale, while the results, even under such conditions, especially with regard to raw ores, proved to be not good enough to justify the application of the process in practical metallurgy. To protect himself against failure it is of the greatest importance that the experimenter should execute

his experiments under conditions which can be practically maintained on a large scale. As soon as he has, by working on a large scale, to change the conditions which he maintained in his laboratory tests, he will obtain very different results, and only too often will experience a complete failure.

If a half ounce of raw sulphureted ore is treated in a beaker with a large excess of a 32 per cent. "extra solution" for twelve hours with frequent stirring, as was done by Russell in the laboratory, it is experimenting under conditions which cannot be profitably maintained on a large scale, and the results thus obtained, if published, should be given as a matter of scientific interest but not as actual results of a new metallurgical process, especially if the condition under which these results were obtained are withheld. More or less silver will be dissolved by such a strong "extra solution"; a 32 per cent. sodium hyposulphite solution applied in the same manner will also dissolve some silver; but it is not practicable to work with such concentrated solutions, since they cannot be maintained without a very large consumption of sodium hyposulphite and copper sulphate. Besides, raw sulphureted ores do not filter well, and the ore would have to be treated in agitating tanks for twelve hours, and the separation of the solution from the residues would have to be effected by means of large filter-presses. These manipulations, together with the very large consumption of chemicals, would cost more than roasting with salt. The weakest part of this method as applied to raw ore, however, is that the extraction at the best is so inferior and incomplete that its application is entirely out of the question.

More favorable results can be obtained by treating raw oxidized ores, or sulphureted ores which were first subjected to a thorough oxidizing roasting, wherein there are not so serious difficulties preventing its application on a large scale, though only in exceptional cases will it be rational to employ it. A dilute solution can be used, and often 50 to 70 per cent. of the silver will be extracted. But similar results can be obtained by using a straight solution of sodium hyposulphite. Some of the San Francisco del Oro ore, a highly sulphureted lead-zinc ore, was roasted oxidizingly in a reverberatory furnace by me, and part of it treated with sodium hyposulphite and part with Russell's "extra solution." The roasted ore contained 29.3 oz. silver per ton. By leaching with sodium hyposulphite 17.2 oz. silver were

extracted, while with Russell's "extra solution" the extraction gave 17.64 oz. or 0.43 oz. silver per ton more. But an increased extraction of less than half an ounce of silver per ton does not justify the extra consumption of 7 lb. of copper sulphate and 5 lb. of sodium hyposulphite, which amount is stated to be the consumption of chemicals in the Russell process per ton of ore treated. It would be folly to oxidize an ore for lixiviation and obtain an extraction of only 50 to 70 per cent. instead of chloridizing it and obtaining an extraction of 90 per cent. and more, especially since by the modern method of chloridizing roasting the loss of silver by volatilization is greatly reduced, and does not exceed the loss occurring in oxidizing roasting. The claim that ores require only an oxidizing roasting, if treated by the Russell process, is therefore a mistake, and cannot be verified by actual and satisfactory working results.

It was mentioned above that only in exceptional cases can lixiviation of raw ores be executed successfully. One of the cases is when in oxidized ore the silver occurs as chloride. However, it is more advantageous to subject such ore first to a short red heat to melt the silver chloride. Chloride of silver, as it occurs in nature, is very dense and dissolves very slowly, but if the ore is heated the silver chloride will melt and impregnate the surrounding ore particles, in which condition it offers a large surface to the solvent and permits a much quicker extraction. Oxidized ores in which the silver occurs as antimonate can also be successfully leached. Low-grade oxidized ores, which will yield 60 to 70 per cent. by leaching raw, can also be successfully treated if, on account of the small tenor of silver in the ore, the increased amount extracted by chloridizing does not exceed the cost of roasting. However, the use of sodium hyposulphite for such ores will be found more economical than the use of the "extra solution," the additional cost of chemicals not being covered by the slight gain in extraction.

After the process was tried on a large scale without success, Russell modified his process. He adopted chloridizing roasting and leaching with straight sodium hyposulphite, and after this was done applied his "extra solution," by which he claimed to obtain a much better extraction. But this assertion was again based on laboratory experiments, which were made under different conditions than could be maintained in the works. Based

on these results it was next claimed that silver ores require a less careful chloridizing roasting, because the "extra solution" dissolves the unchloridized part of the silver. But this claim was also found to be an illusion, as has been demonstrated by many failures on a large scale. The following case is an illustration:

In Sombrerete, Zacatecas, Mexico, the lixiviation process with sodium hyposulphite was in successful operation for years, until a new company was organized to work the property on a larger scale. A new and large mill was erected to suit the requirements of the Russell process. The success in the old mill was based on a good chloridizing roasting in reverberatory furnaces. The results, however, were entirely different when the new mill was set in operation. In this the ore was roasted in a Stetefeldt furnace, which is not suitable for such heavy sulphureted ore, and the chlorination was far from being satisfactory. This gave an opportunity to demonstrate the claim that the "extra solution" exerts an energetic dissolving and decomposing action on the unchloridized part of the silver. However, the "extra solution" failed to react, and after eleven months of unsuccessful trials the company failed and the property changed hands.

The new company abandoned the Stetefeldt furnace and the Russell process, built a suitable number of reverberatory furnaces, and adopted the common lixiviation process with sodium hyposulphite with great success. The mill is still in operation and treats regularly 60 to 80 tons of ore per day.

The Cusi company at Cusihiuriachic, Chihuahua, Mexico, had a similar experience. The old lixiviation process was for years in successful operation on a scale of 50 to 60 tons per day. Induced by the glowing representations of the advantages of the Russell process, the company adopted it, but after 1½ years' trial, with heavy financial loss, the process was discarded and lixiviation with sodium hyposulphite was resumed.

(2) THE KISS PROCESS

This process was recommended for the mutual extraction of silver and gold from sulphureted auriferous silver ores. Kiss subjected the ore to a chloridizing roasting, leached with water to remove the base-metal chlorides, and extracted the silver and

gold with a solution of calcium hyposulphite. As precipitant he used calcium polysulphide. According to Kiss, in chloridizing roasting subchloride of gold is formed, which is neither soluble in water nor in a solution of sodium chloride, but is soluble in a solution of calcium hyposulphite. This method was in actual operation in several places in Hungary, and while the extraction of the silver was satisfactory (90 per cent.) the extraction of the gold varied greatly at different places; from 90 to 20 per cent. This was undoubtedly caused by the way the roasting was done at the different places. Subchloride of gold cannot exist at a temperature prevailing in the furnace, especially not if the temperature toward the end is increased, as was formerly generally done; it is then decomposed into metallic gold and chlorine. The subchloride of gold, however, is formed if the ore after being discharged from the furnace is allowed to cool slowly. If the hot ore on leaving the furnace is dumped into a pile, or better in a bin, the generation of chlorine continues down to a temperature which is low enough not to decompose the subchloride of gold, and therefore opportunity is given for the formation of this gold combination. (This has been discussed in another chapter of this treatise.) The variation of the gold extraction at the different places was therefore most likely due more to whether the chloridized ore was cooled slowly or quickly rather than to the different character of the ores.

We have seen, too, that calcium hyposulphite in contact with sodium sulphate forms sodium hyposulphite and calcium sulphate (gypsum), which precipitates, so that Kiss actually did not work with calcium hyposulphite after the solution was used for some time, but with sodium hyposulphite, as the lime salt was decomposed by coming in contact with the remaining sodium sulphate of the roasted ore. It is questionable whether calcium hyposulphite is a more energetic solvent for subchloride of gold than sodium hyposulphite; at least, it was not proved by Kiss' method, as he actually leached with the soda and not with the lime salt.

XIX

THE AUGUSTIN PROCESS

IN this process the material is roasted with salt. The resulting silver chloride is dissolved by a hot concentrated brine. From the solution the silver is precipitated by copper, and the copper by iron. The remaining solution is freed from iron and sodium sulphate and brought into circulation again.

The ore is roasted, first oxidizingly to transform the metal sulphides into oxides, and the silver sulphide as much as possible into silver sulphate, so that during the chloridizing period but little volatile or soluble metal chlorides will be formed. The material which conforms best with these requirements is copper matte. The oxidizing roasting is continued until the iron has changed into red oxide and the copper into cupric oxide. If no, or not enough, sulphates are left at the end of the oxidizing period to decompose the salt, ferrous sulphate has to be added with the salt to generate the chlorine. The roasted material is not first leached with water, but the hot concentrated salt solution is applied at once to the ore. If not all the copper was changed into oxide, and some chloride or subchloride is still present, this will be dissolved by the salt solution, and the subchloride oxidizes during the precipitation of the silver to basic insoluble chloride of copper and makes the precipitated silver impure. If the matte contains lead the hot brine will dissolve lead chloride, which in cooling precipitates as white flakes and also makes the silver impure. Such material has to be leached first with hot water.

The stream of hot brine coming from the leaching-vats is divided in many small streams, each of which is made to strike metallic copper, which is placed in small wooden tubs provided with a filter and an outlet below the filter. Under these tubs is placed another row of similar tubs, and below this still another,

so that the brine passes through quite a number of such tubs filled with copper. They are arranged on benches. Leaving the last row, the desilverized solution is conveyed to a series of tanks for the precipitation of the copper by scrap iron.

An advantageous feature of this process is that the silver is precipitated in the metallic state; from a solution of sodium hyposulphite, the silver could not be precipitated with copper without a decomposition of the sodium hyposulphite. This method, however, has many disadvantages which make it unfit for work on a really large scale. The dissolving energy for silver chloride of a concentrated hot salt solution is much inferior to that of sodium hyposulphite; it takes 68 parts of sodium chloride to dissolve one part of silver chloride, while it takes only two parts of sodium hyposulphite. The handling of large quantities of hot concentrated brine is rather unclean and troublesome. All parts of the vats and tubs above the solutions and the whole of the outside become covered with incrustations of salt; besides, it is exceedingly difficult to keep them tight, as such a concentrated and hot solution finds its way even through the fibers of the wood.

Cement copper acts more energetically in the precipitation of the silver than scrap copper, but the resulting cement silver is not quite as clean, as it usually contains more copper particles, which have to be removed by treating with hydrochloric acid.

XX

EXTRACTION WITH SULPHURIC ACID

THIS process, like the Augustin method, is exclusively used for the extraction of the silver from products of smelting, principally black copper and copper matte.

(1) EXTRACTION OF SILVER FROM COPPER MATTE

The products of this process are cupric sulphate (blue vitriol, blue stone), which goes in solution, and residues in which the silver and gold remain, from which they are finally extracted by smelting with lead ores.

(a) *The Old Method*

Besides the extraction of the precious metals, it is of great importance that the resulting cupric sulphate should be as free from iron as possible, because the blue vitriol is bought in the market as such, and if it contains ferrous sulphate its value is much reduced. To accomplish this, the common copper matte has to be first concentrated by repeated roasting and smelting, not only to enrich it in copper but to free it as much as possible from iron. Though red oxide of iron dissolves much more slowly in diluted sulphuric acid than cupric oxide, the resulting copper solution will contain by far too much ferrous sulphate to produce a marketable blue vitriol if the copper matte contains a large percentage of iron.

At Freiberg, Saxony, where this process is in operation, they concentrate the original copper matte of 38 to 44 per cent. copper by repeated roasting and smelting until it contains 70 or more per cent. copper and only 0.3 per cent. iron. This matte is crushed coarse ($\frac{1}{4}$ -in. mesh) and roasted twelve hours in a reverberatory furnace. The charge of 1000 lb. is continuously stirred, then

withdrawn from the furnace, finely pulverized and subjected during four hours to a "dead roasting." The roasted matte is sifted, the coarse pulverized and mixed with the fine.

For the treatment of the roasted matte with diluted sulphuric acid there are cylindrical upright drums 2 ft. 8 in. in diameter and $3\frac{1}{2}$ ft. high, made of $\frac{1}{2}$ -in. lead supported by an iron framework, which is covered with thin sheet lead. The drum has two outlets, a lower one for discharging the residues and one higher up for the discharge of the copper solution. Extending to near the bottom there is a lead pipe for the injection of superheated steam. This pipe is so arranged that it can be lowered and raised. The drum is charged with 3 cu. ft. of mother liquor, and 3 cu. ft. of common sulphuric acid of 45 to 47 deg. B., which gives a solution 34 to 36 deg. Beaumé. Superheated steam is now applied, and when the solution is boiling, 200 lb. of roasted matte is gradually charged with a hand shovel through a movable funnel. The pulp has to be charged continuously. After a quarter of an hour, 9 cu. ft. of mother liquor is added to the contents of the drum, which fills it to about 7 in. below the rim. The boiling with steam is continued for 4 to 5 hours, during which time stirring has to be done at intervals. The steam is turned off, and half an hour is given for the residues to settle. Then the solution is drawn off through the upper outlet and conveyed to settling tubs, while the residue, containing all the silver and about 5 per cent. copper, is transferred into a special basin, where it is washed with water. The still hot copper solution is left about an hour in the settling-tanks, during which time some basic iron salts settle. Then it is drawn off into the crystallizing tanks. The residues are dried and transferred to the lead smelting.

The blue vitriol crystals after five to seven days are removed from the crystallizing tanks. They are not of a clear blue color, but have a greenish appearance from a certain percentage of iron which they contain, and are again dissolved and recrystallized. All the solutions are very acid.

The yearly production of blue vitriol resulting from this process is about 800 tons.

(b) *O. Hofmann's Method*

We have seen that, to treat copper matte with sulphuric acid by the old method, it is required to first eliminate nearly

all the iron from it by repeated roasting and smelting, which at the same time produces a matte very rich in copper. This careful elimination of the iron is done to permit the production of a purer blue vitriol. We have also seen that in order to avoid a concentration of the resulting cupric sulphate solution for crystallization the proportion of acid and roasted matte is so regulated as to produce at once a solution strong enough for that purpose. If this is done for economical reasons, to avoid the cost of evaporation, the object is not attained, as it necessitates recrystallization because the crystals contain too much iron, notwithstanding the fact that the matte by previous treatment contained but very little iron. The solubility of red oxide of iron in sulphuric acid increases with the strength of the acid, therefore it is not advisable to bring the matte in sudden contact with too strong an acid. Cupric oxide dissolves so much easier than the iron oxide that a weak acid can be used which will readily dissolve the cupric oxide, but very little of the iron oxide, and the resulting cupric sulphate solution will be much purer. The strong first solution makes this method not suitable to be worked on a really large scale, because concentrated solutions require but little cooling to form fine crystals, which are very annoying in the separating of the residues from the solution, because these crystals will form in the filter-press, clog the filter cloth and prevent a free filtration. They will form in the mass of the residues, whence they can be removed only by a prolonged washing with hot water, causing the formation and accumulation of too large quantities of a very weak solution.

I discovered a method of purifying a cupric sulphate solution from iron, antimony, arsenic, etc., which, together with a modification of the manner in which the process of dissolving the cupric oxide is executed, made the sulphuric acid process not only much cheaper, but also applicable for working common copper matte without previous concentration, at the same time producing a very pure product of blue vitriol.

The method is based on the reaction that, if through a hot neutral solution of cupric sulphate which contains ferrous sulphate a stream of air is forced, and at the same time cupric oxide is added, ferric oxide is precipitated and cupric sulphate will go in solution. Instead of specially prepared cupric oxide, roasted copper matte is used, which contains sufficient cupric oxide for the reaction.

The manner in which the dissolving of the roasted matte is performed is modified, inasmuch as the roasted matte is not added to the bulk of acid of sufficient strength to produce at once a cupric sulphate solution of the required concentration, but the solution is gradually enriched in copper until the desired degree is reached. This is done by preparing first a 3 per cent. acid solution. Such a weak acid dissolves but very little iron oxide. Then, when hot, the copper is added gradually as a stream of roasted matte, while at the same time a small stream of 60 deg. B. acid flows into the tank. The solution in the tank is agitated vigorously by a machine-stirrer. The proportion of matte and acid has to be so regulated that the solution always maintains its strength of 3 per cent. in acid, while the content of copper increases. This is controlled by frequent volumetric tests. When the desired strength in copper is nearly attained, the influx of acid is stopped and only so much matte added as will neutralize the 3 per cent. acid. Thus the copper matte does not come in contact with a stronger acid, and the resulting solution does not contain more than 0.7 to 1 per cent. iron, notwithstanding that the treated material is very rich in iron oxide.

This process was introduced by me in the large smelting works of the Consolidated Kansas City Smelting and Refining Company, at Argentine, Kansas, for the treatment of the argentiferous leady copper matte, first on a medium large scale, but on account of its successful working it underwent an enlargement every year, until a daily producing capacity of 60 tons of blue vitriol was reached. In the following a short description of the process and the apparatus will be given.¹

The material treated is a leady copper matte containing 34 to 40 per cent. copper and 12 to 14 per cent. lead. It is crushed first in a rock breaker and then pulverized in a Krupp ball-mill of 100 tons daily capacity through a screen of 50 meshes to the lineal inch. The roasting is done in three two-story Pearce turret furnaces, each provided with three fireplaces at the lower and two at the upper hearth. For the benefit of the subsequent operation the roasting has to be conducted with great care, and with attention

¹ A more elaborate description can be found in "The Mineral Industry," Vol. VIII and Vol. X. The plant is not now in operation, the smelting works with which it was connected having been closed down and dismantled in 1902.

equally divided between the oxidation of the copper and of the iron. The copper is to be converted into cupric oxide and sulphate, and the iron into red oxide, in which state it dissolves only slowly in hot diluted sulphuric acid. The formation of cupric sulphate is very desirable, as it saves acid in the subsequent treatment, but still it is not advisable to conduct the roasting so that as much as possible sulphate is formed, because the roasted material will then contain too much soluble iron, which would make the resulting cupric sulphate solution too impure. At a comparatively early stage of the roasting nearly all the copper is in a state in which it can be extracted by diluted sulphuric acid; about 75 per cent. of it is present as oxide and 25 per cent. as sulphate. The roasting, however, cannot be considered complete at this stage, because the roasted material contains yet too much soluble iron.

At the beginning of the operation the temperature should not be raised above that produced by the combustion of the sulphur of the matte. During this period the charge assumes a rather bright red appearance, an effect due more to light than to heat, and if excess of air is admitted to the furnace but few lumps will form, even with very leady copper matte. Gradually, as the oxidation advances, the surface of the charge becomes darker, especially near the air-doors, and when the entire surface of the charge begins to darken, the fire is slightly increased to prevent cooling, as from this time on the supply of heat furnished by the oxidation decreases rapidly. If this condition is overlooked and the charge cools too much, it will be difficult to raise the temperature again to the proper degree.

The roasting is continued at a moderate temperature until no more heat is evolved by the oxidation of the material, after which the temperature must be raised to a cherry red. A skilled roaster can readily determine this point by stirring the charge; if the particles thrown to the surface brighten, the roasting has not advanced far enough, but if the entire charge presents a dead and uniform red color, it shows that this part of the roasting has been completed, and that it is time for an increase of temperature. This can now be done without danger of lumping the charge, because by this time the greater part of the sulphides has been oxidized. The increase of the temperature is necessary for two reasons: first, to hasten the oxidation of the remaining sulphides,

which would require a very long time at a low temperature, and second, in order to decompose the iron salts and to convert them into the red oxide. The task for the roaster is to convert as much iron as possible into the red oxide without decomposing the cupric sulphate present. Cupric sulphate resists considerable heat, more so than the ferrous salts, and it is possible to conduct the roasting in this way; but the increase of temperature requires judicious care, because if the heat is too high the cupric sulphate will be reduced to cuprous oxide, in which condition but half of the copper is soluble in diluted sulphuric acid. If crystals of cupric sulphate are exposed to heat and air, it will be noticed that after the acid is expelled the mass assumes a red color showing the formation of cuprous oxide. If heating is continued, it turns black by being oxidized to cupric oxide. Should cuprous oxide be formed, the amount of extractable copper will be greatly reduced. If, for instance, the extractable copper before raising the temperature was 97 per cent., after an excessive heat it will be reduced to 92 or even 90 per cent. When the roasting is done in a common reverberatory furnace, a mistake of this kind can be corrected by keeping the charge longer in the furnace and thus oxidizing the cuprous to cupric oxide. In a mechanical continuously discharging furnace, like the Pearce, however, this cannot be done, but with experience and care the decomposition of the cupric sulphate can be avoided. It is of great importance that, during the whole time of roasting, air has free access into the furnace.

It is not possible to avoid the formation of some lumps, especially in roasting leady matte, but if the roasting is conducted properly, these will be small, soft, porous, and consist of well roasted material. Roasted matte is always of coarser grain than the raw pulp, and for this reason as well as on account of the lumps it is necessary to pulverize the roasted material before treatment with sulphuric acid. This is best done in a Krupp ball-mill through a screen with 50 meshes to the lineal inch.

The dissolving at Argentine, Kansas, is done in eight agitating or stir tanks, which are arranged in two parallel rows with a track between for the delivery of the matte. The cars are scoop-shaped and are partly covered with sheet iron, and so made that the cover and scoop end come close together, leaving only a narrow slit open, so that when the car is tilted the roasted matte

runs gradually into the dissolving stir tank. Fig. 66 represents the vertical section of a stir tank, 12 ft. in diameter and 6 ft. deep. The bottom and sides are lined with boards for protection against wear from the friction of the pulp. In the center is suspended a

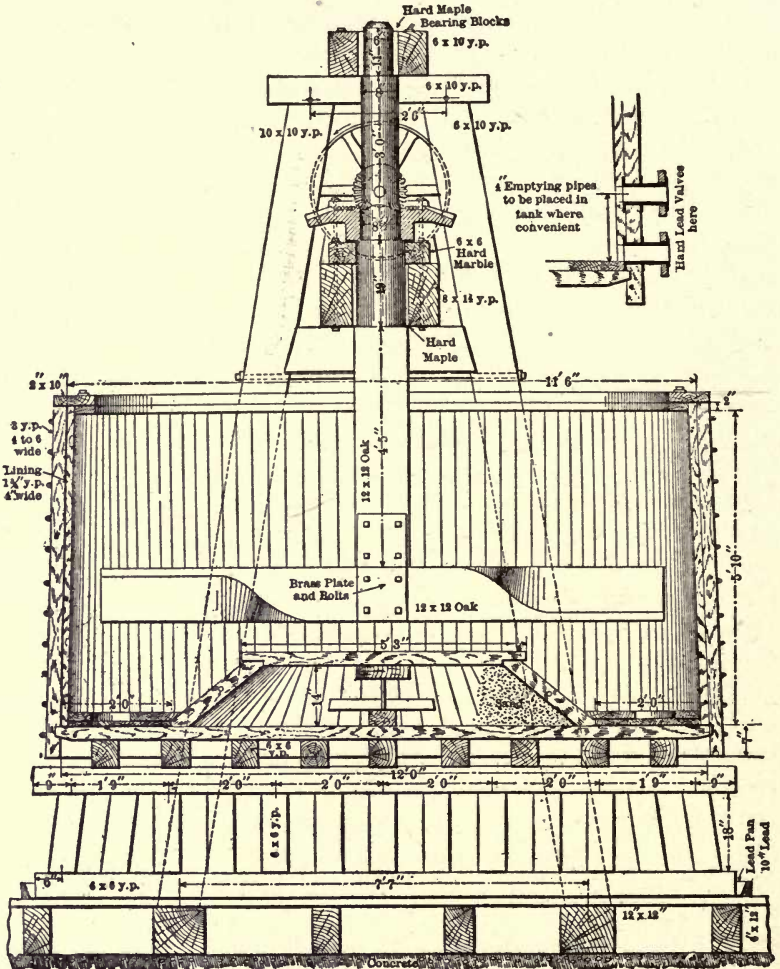


FIG. 66.—STIR TANK, VERTICAL SECTION.

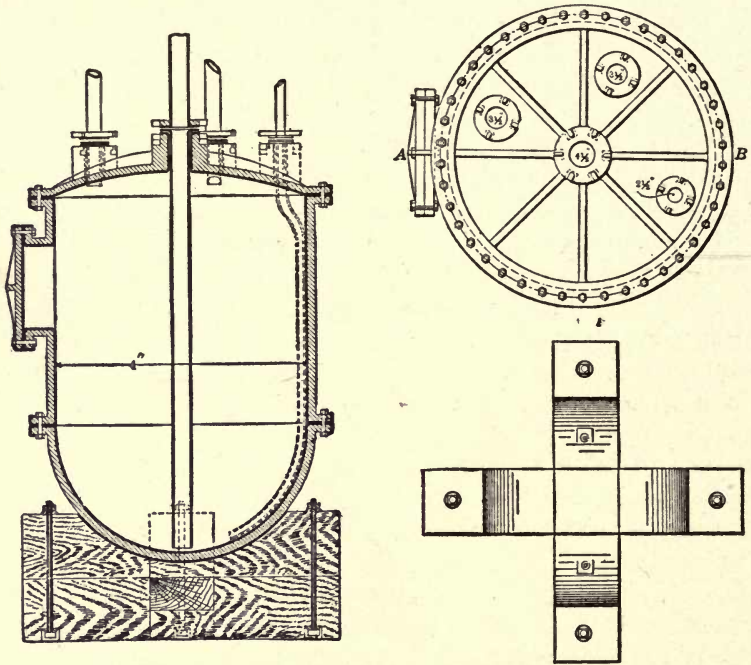
heavy wooden shaft, having fastened to the lower end, by heavy brass plates, a cross-beam cut at both ends like a propeller-blade. In the center, below the propeller and fastened to the bottom of

the tank, is a cone-shaped projection constructed of strong wooden staves, the interior of which is filled with sand. This cone forces the matte, when the propeller is in operation, toward the periphery, where it is subjected to the swift rotating motion of the liquor, thus preventing its accumulation in the central part, where the motion is much less. Two outlet pipes provided with hard-lead valves make connection with the pressure tank. These are placed one above the other to permit the withdrawal of a portion of the clear liquor if desired, in which case the paddle is stopped and the residues allowed to settle. In the usual working of the tank, however, the lower outlet pipe only is used. The wooden ring attached to the rim of the tank prevents the splashing of the pulp. The tank and trestle support are placed in a flat lead pan to collect any leakage. The upper part of the shaft is provided with a gear-wheel to receive the power.

The stir tanks are filled about two-thirds full with water, the agitator set in motion, and sulphuric acid added until the liquid shows about 3 per cent. acid. The matte is then charged gradually, and at the same time a stream of acid is allowed to flow in so as to maintain the same acid strength during charging. In this way the dissolving is accomplished with an acid strength of 3 per cent. or less, and still yields a strong solution of cupric sulphate. As stated above, it is preferable to work with weak acid, because much less iron and other impurities will be dissolved than with stronger acid. When the solution has attained a strength of about 20 to 22 deg. B., the flow of sulphuric acid is stopped and matte only charged until the solution is neutral. Toward the end it is advisable to charge the matte at intervals, and to make frequent acid tests to avoid an excess of matte. The addition of sulphuric acid to the water, and the matte to the diluted acid, produces heat, which aids the solution of the cupric oxide. This heat is not sufficient, and the temperature is further raised by a jet of steam. It is well to interrupt the charging of matte while neutralizing as soon as the solution shows 1 per cent. free acid. If the agitation is then continued this 1 per cent. of acid will be diminished, but if in half an hour after the last acid test the percentage of free acid remains the same, more matte is added until the solution is neutral; in this way a mistake of adding a large excess of matte is avoided.

Below the stir tanks are the pressure tanks, into which the

finished charge is drawn while the paddle is in motion. On account of the residues the pressure tanks are in an upright position, and are constructed as follows: The body consists of two cylindrical sections 4 ft. long each and 4 ft. 6 in. in diameter, the bottom being of a spherical shape of 2 ft. 3 in. radius. The top is rounding upward to a height of 6 in. above the rim: The sections are tightly flanged, with a rubber gasket between. Four pipe connections are made through the top; the discharge-pipe entering through the center passes nearly to the bottom, and the filling-



FIGS. 67, 68, 69.—CAST-IRON PRESSURE TANK.
67 (top right), top view. 68, section on line *AB*. 69, plan of supporting frame.

pipe, air-inlet and air-outlet pipes are conveniently arranged around it. By proper connection with the filling-pipe and by the use of valves, one pressure tank is made to serve four stir tanks. The upper cylindrical section is provided with a manhole, and the tanks are made of cast iron and lined first with lead and then with wood to protect the lead from wear by abrasion. The air pressure required is from 40 to 50 lb. Fig. 67 shows the top view and Fig. 68 a vertical section of an upright pressure tank

while Fig. 69 illustrates the supporting frame. This tank as represented by the figures is smaller than those used at Argentine, and has only one cylindrical section instead of two. The general arrangement, however, is the same. The air-inlet extends downward along the side, ending near the discharge-pipe, in order to keep the residues near the outlet in a loose condition and prevent the pipe from clogging. For this reason it is advisable to have a small stream of air enter during filling.

The pulp is forced by the pressure tank into a large filter-press, of which there are five in use. The press is 25 ft. long and has hardwood frames and plates, and holds, when filled, 5 tons of residues. The clarified solution flows from the press to lead-lined storage tanks, from which it is elevated, by means of a pressure tank, to the purifying towers. These towers were originally made of 20-lb. sheet lead incased in a cast-iron framework, which, however, were replaced by larger ones made of 4-in. California redwood. Fig. 70 represents a vertical section of such a tower. They are made of staves 16 ft. long, 9 ft. in diameter, and are well hooped with round iron. The top and bottom are flat. The towers are firmly fastened to a strong wooden trestle, which in turn is anchored to a concrete foundation. This construction is called for to guard against the oscillating movement of the tank from the action of the contained solution, which is set in violent motion by the ascending air. Inside the tower, about 19 in. above the bottom, the 4-in. lead air pipe enters, and is connected with a perforated 6-in. lead pipe, which extends diametrically to the opposite side and is closed at the further end. Outside the tower the pipe extends upward to above the top of the tower, and thence downward to the discharge-pipe of an air-compressor or to an air-receiver, connected with the compressor. This arrangement is necessary to prevent the solution from flowing to the air-pumps if the latter be not in operation. At the same level with the air-inlet, the 4-in. discharge-pipe enters, which is provided with a hard-lead valve placed close to the outside of the tower. At about the same level enters a 1-in. lead steam-pipe for heating the charge. The direct application of steam does not dilute the solution, because the evaporation, which is favored by the ascending air, fully equalizes the dilution by condensed steam. The top of the tower is provided with an 8-in. pipe extending through the roof for the escape of steam and air, which,

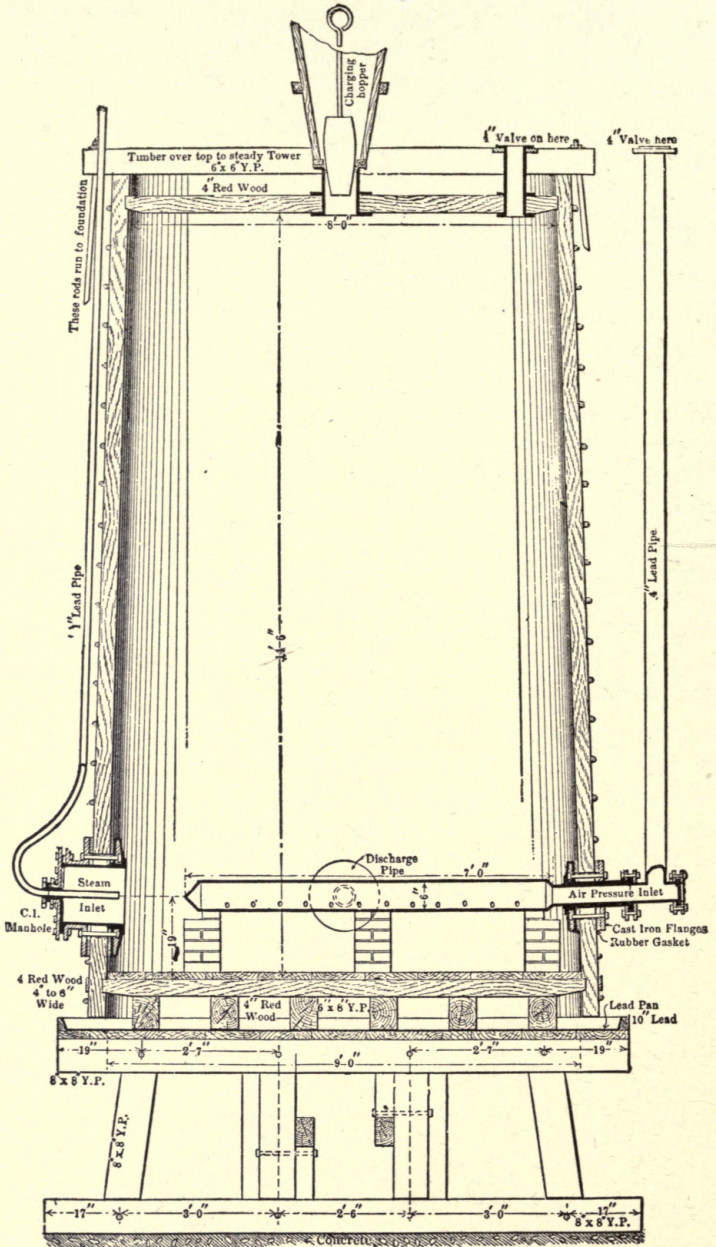


FIG. 70.—TOWER FOR REFINING CUPRIC SULPHATE SOLUTIONS.

however, first are made to pass through a lead-lined box provided with shelves arranged in a zigzag manner to precipitate all particles of liquor which may be carried out by the current of air. A 4-in. pipe also enters the top, which serves for filling the tower with crude copper solution. An opening in the center is connected with a small hopper which is filled with roasted matte. The upper third part of the tower is provided with glass tube gages to control the filling. The perforations of the air-pipe are only on the bottom side, to prevent the inflow of matte and precipitate. The steam pipe enters through the lid of the manhole, as the heat of the pipe affects the wood at its immediate surrounding.

The tower is filled with crude solution, leaving, however, sufficient room for the increase in volume of the solution, which immediately takes place as soon as the compressed air is supplied. A tower can be charged with about 5000 gallons of solution, and as three such charges can be refined in twenty-four hours the working capacity of one tower is 15,000 gallons per day. At Argentine eight such towers are in use. When the tower is filled, steam is allowed to enter, and also some air, to produce a more uniform heating of the solution. The air causes the precipitation of some basic iron salts, but I never succeeded in precipitating more than about half of the iron the solution contained, although the treatment was extended many hours. When the solution is hot (75 to 80 deg. C.) more air is admitted, and some roasted matte from the hopper is made to drop into the tower. The violent boiling motion of the solution keeps the matte in suspension, and after three to four hours the solution will be entirely free from iron, arsenic, antimony, etc. To observe and to regulate the progress of the operation the solution is tested from time to time for iron by taking samples through a small cock inserted in the side of the tower. If between two tests the content of iron is not diminished some more matte is added. It is not necessary to test for other impurities, because the iron predominates, and by the time all of it has been precipitated, no trace of any other will be found.

The cupric oxide in presence of air combines with the sulphuric acid of the ferrous sulphate, forming cupric sulphate, while the iron precipitates as oxide — a decided advantage, as the precipitant is converted into cupric sulphate, and thus enriches the solution in copper.

The refined solution leaving the tower should not be stronger than 24 to 26 deg. B. when hot, as otherwise it will cause trouble in the filter-press, for reasons above explained. Should a concentration have taken place in the tower beyond this, the solution should be diluted. The tower is discharged into a special stir tank and from there, by means of a pressure tank, forced into a filter-press. The clear and purified liquor is conveyed to the evaporating department, while the residues are subjected to an additional treatment. These tower residues are of a grayish yellow color and consist principally of precipitated iron, arsenic, antimony and some undecomposed matte, with also some basic copper sulphate. To remove the last named substance the residues are treated in a stir tank with 2.5 to 3 per cent. cold acid solution, which dissolves the basic copper salt, leaving the impurities unaffected, with the exception of iron, which is acted on very slightly.

The refined solution does not contain even a trace of silver. Whatever silver is converted into sulphate during the roasting is precipitated in the stir tanks by the ferrous sulphate present in the solution. The residues are washed well in the filter-press to remove all copper solution. The wash-water is collected separately, and used instead of fresh water for the preparation of a new charge in the stir tanks. The residues contain all the silver, gold and lead of the roasted matte; they are of a dark-red color and consist mainly of iron oxide. They are sent to the lead-smelting department for the extraction of the precious metals and the lead. On account of the large percentage of iron oxide and lead the residues are an excellent material for lead smelting. A large portion of the lead of the matte is converted into sulphate by the roasting, and therefore does not act much as a consumer of acid. As the roasted matte contains 20 to 25 per cent. of its copper as sulphate, the total consumption of acid is much less than the equivalent amount contained in the blue vitriol produced.

The evaporating department furnishes 90,000 gallons of concentrated solution daily. To supply this amount an improvement over the old pan evaporator with under-fire or steam coils was requisite, and as vacuum evaporators could not be adopted I finally devised and introduced an economical and effective evaporator, which is illustrated in Figs. 71 and 72. The

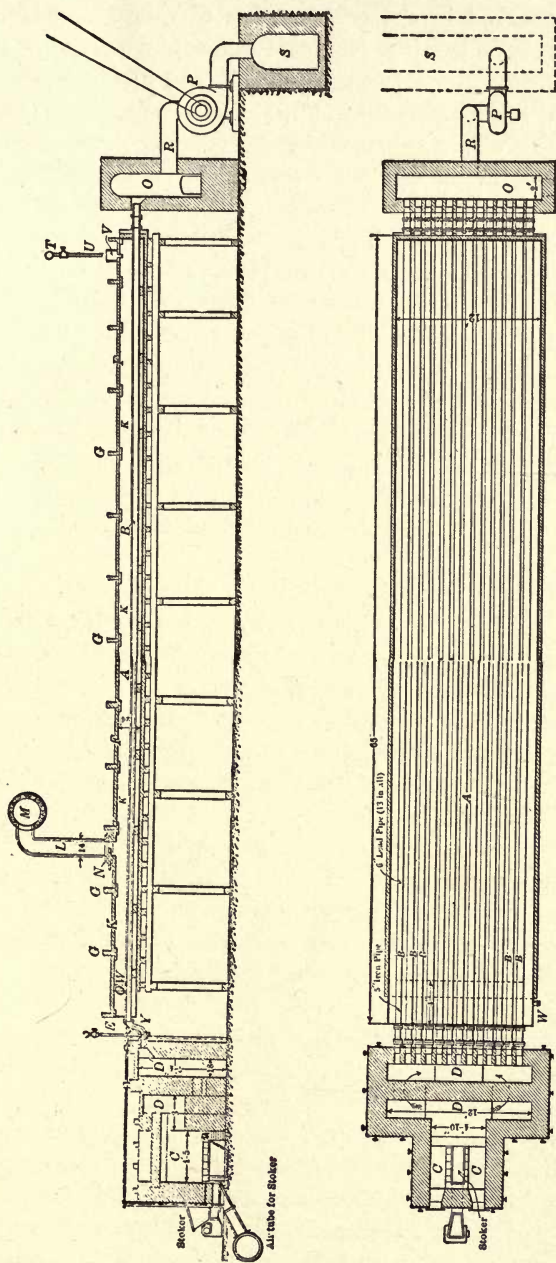


FIG. 71 and 72.—PAN EVAPORATOR, LONGITUDINAL VERTICAL SECTION AND PLAN.

principle observed in the construction of this evaporator is the application of the hot furnace gases in a manner by which almost a complete utilization of the heat contained in them takes place. The apparatus consists of a flat tank, wooden with the exception of a 2-ft. space at the front end, which is made of steel, so that the wood will not be in too close proximity to the furnace. The tank is 65 ft. long, 12 ft. wide and 2 ft. deep, and is lead lined, the two ends having much heavier lead lining than the sides and bottom. It is traversed longitudinally by 13 6-in. heavy lead pipes. These pipes rest on bricks which are properly placed on the bottom of the tank. The tank rests on wooden trestle-work of a proper height to correspond with the height of the furnace. At the furnace end in each lead pipe is inserted a 5-in. pipe 4 ft. long, provided at the outer end with a flange. The iron pipe serves to protect the lead pipe from immediate contact with the red-hot gases from the furnace. They also make the connections between the lead pipes of the tank and the iron pipes of the furnace.

The furnace is comprised of the fireplace *C*, the dust-chamber *D* and the distributing chamber *D*¹. At a proper height in the wall of the distributing chamber nearest the tank are 13 openings, in each of which is inserted a short cast-iron pipe, 5 in. in diameter, with a flange at the outer end. Each pipe is connected with its corresponding piece of iron pipe inserted in the lead pipe of the tank by a cast-iron S-shaped elbow, *Y*, which allows the introduction of the compressed-air pipe *E* for removing any accumulation of ashes in the lead pipes of the tank.

An American "underfed stoker" is used for the slack-coal fuel and affords practically perfect combustion, which is of great importance, as otherwise the lead pipes would soon become coated with soot and lose much of their efficiency to transmit the heat to the solution.

The opposite ends of the lead pipes in the tank are connected with the brick suction chamber *O*, which in turn is connected by a galvanized iron pipe, *R*, with a suction fan, *P*, the gases being discharged into an underground flue, *S*. This flue serves in common to collect the waste gases from 11 evaporators, and terminates outside the building in a brick chimney 40 ft. in height.

The top of the pan is closed with a wooden cover, and wooden joists, *G*, are placed across the pan about 5 ft. apart, having

cleats fastened to the lower side, as shown in Fig. 71. The spaces between the joists are covered with boards resting on the cleats and pushed closely together, but not nailed, so that the whole or part of the cover can be easily removed. About 14 ft. from the front end of the tank is a 14-in. suction pipe, *L*, connected with the main suction pipe, *M*, which crosses all of the evaporators to remove the water vapors. The main suction pipe, *M*, as well as the branch pipes, *L*, are made of wooden staves kept tight by hoops. *M* is connected with a large suction fan having the housing and wings of sheet copper and the shaft and arms of brass. This fan rapidly removes the vapor from each evaporating tank, and by its use the building, even in cold winter weather, and notwithstanding that 11 such evaporators are in operation, is entirely free from steam. A wooden stack outside the building serves for the discharge of the fan, and the exhaust at each individual evaporator is regulated by a wooden slide, *N*, inserted below the suction pipe *L*.

Fig. 71 gives the construction of the first or experimental evaporator. During the experiments it was found that the 6-in. lead pipes passing through both ends of the tank, and being burnt with lead tight to both the ends, did not keep their straight position, but on account of the expansion became wavy. Profiting by this experience the back end of the other evaporators were made sloping, and the pipes, instead of passing through, passed over the edge of the back end. This allowed the pipes to expand freely, and they retained their straight position.

Close to the end of the evaporating tank and resting on the cover is the lead-lined feed-box, *V*, from the bottom of which is a short pipe or nipple extending into the tank. The solution supply pipe *T*, which crosses all 11 evaporators, is connected with the large supply tanks, and serves to convey the solution to each evaporator by down-takes, *U*. The outlet of the evaporating tank is in the side near the furnace end, about four inches above the hot-air pipes, *B* (Fig. 71).

The operation is conducted in the following manner: The pan is first filled to the level of the outlet with the copper sulphate solution to be concentrated, the fire is then started and the stoker and suction fans set in motion. The big copper fan is not started until the solution becomes hot enough to generate steam. Some solution is added through *V* to keep the surface of the solution at

the same level. When it is found that the solution near the outlet has attained the desired concentration, a continuous stream of refined solution is allowed to flow into the tank from the feed-box, which starts a continuous outflow of concentrated solution through the outlet. The amount of influx is regulated by frequent hydrometer tests of the solution at the outlet. The supply of fuel by the automatic stoker being regular, the heat of the evaporator is very uniform, and once having adjusted the proper influx of the weak solution the outflowing stream will be found of quite constant concentration.

The glowing hot gases entering the tubes give off the main part of their heat to the solution within a comparatively short distance from the point of entrance, and cause this portion of the solution to boil. In the passage of the gases through the tubes they gradually come into cooler regions, and are offered an excellent opportunity to give off more of their heat to the surrounding solution, so that, when they finally leave the tubes, their temperature is much below the boiling-point of the solution; in fact, so low that the pipes at that end can be comfortably touched with the hand. In a tank 100 ft. or 125 ft. long the gases would leave at a temperature about that of the surrounding air, thus completely utilizing the heat of the gases.

The greater economy and efficiency of this type of evaporator, as compared with one having a steam coil or bottom fire, is apparent. The production of steam involves a considerable waste of heat, and in using it to evaporate liquids, its circulation through coils produces a large amount of condensed water at a temperature very nearly 100 deg. C., the heat of which is generally lost. To evaporate by direct fire under the bottom of a pan is very inefficient and wasteful; and in the present case, in which no other metal but lead can be used for the pan, it requires great care and watchfulness to avoid melting the metal.

For concentrating chemical solutions which do not affect iron the evaporator described can be constructed entirely of iron or steel, and at much less cost. The pan itself, however, should always be placed in a wooden tank to prevent loss of heat by radiation.

Very fine particles of ashes settle in the longitudinal tubes, but these ashes are very light, and by turning the valve of the compressed-air pipe *E*, one of which is attached to each tube, they are easily removed and blown into the chamber *O*.

The continually outflowing concentrated solution passes into a lead pipe common to all the 11 evaporators and is conveyed into a collecting tank, from which the liquor is elevated by means of a horizontal pressure tank into a system of troughs which pass over all the crystallizing tanks, of which there are 112, each of 720 cu. ft. capacity. The troughs are covered and so arranged that any individual tank can be filled. Wooden tanks, lead-lined, of the same size, did not answer. By the frequent changes of the temperature to which they were exposed, the lead lining continued to expand without contracting again, which caused in course of time so many leakages that it became intolerable. The present tanks are made of 20-in. thick concrete walls, which, however, do not come up to expectation either, though they are far superior to the wooden lead-lined tanks. By the sudden change in the temperature when the tank is filled with such a large volume of hot liquor, fine cracks in the walls are caused, through which, if not attended to, leakage will take place, but leakage can be prevented by plastering a little cement on the outside of the tank. An experimental tank built of bricks, however, answered the requirements of such large crystallizing tanks. The brick walls have in the center a 2-in. space filled with a mixture of asphaltum and sand, which combines with the bottom layer of the tank, thus practically forming a tank by itself, embedded in the brick work. When heated by the sudden filling of the tank with hot solution, the asphaltum softens, and when gradually cooled contracts without cracking.

On the top of each tank are movable wooden frames supporting numerous strips of lead 5 ft. long, on which the crystals form, as well as on the sides and bottom. The solution remains seven days in the tank. In discharging, the mother liquor is drawn off through a brass tube near the bottom, then the frames with the strips are lifted up by means of block and tackle attached to an overhead crawl. The crystals are knocked off from the strips with a wooden paddle and fall into the tank. The frames are then moved to one side by means of the crawl. The crystals from the sides are broken down also. The tanks are arranged in long rows intersected by several cross passages. Between each two rows is a track for transporting the blue vitriol crystals, on either side of which is a cement channel to receive and to convey the mother liquor to pressure tanks for further

handling. Rails are laid in recesses near the rim of the tanks, so that the rails of the two opposite rows of tanks form a track for a hopper mounted on wheels. The crystals are shoveled with copper shovels into this hopper, which fills the push-car underneath through a spout with slide in the center. As the tanks are 6 ft. deep the crystals have to be thrown at least 7 ft., and in doing so some of the crystals unavoidably fall back into the tanks, frequently striking the shoveler. It was found that this was very injurious to the men, especially in the summer. Their bodies became covered with deep sores, which, while not dangerous, were very painful.

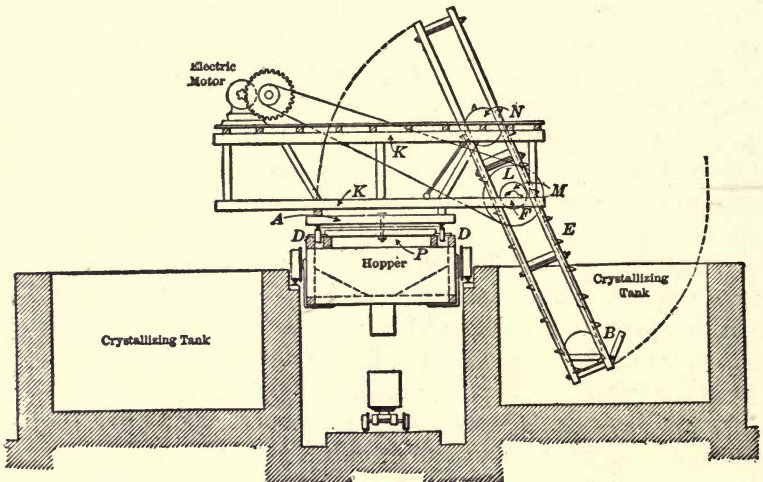


FIG. 73.—DEVICE FOR DISCHARGING BLUE VITRIOL.

To protect the men, I constructed and set in operation the device illustrated in Fig. 73. On top of the movable hopper is mounted the frame *K*, *K*, with a turntable and circular track underneath, which rests on a number of stationary wheels and is kept in place by the pin *P*. The object of this turntable is to make the apparatus available for the opposite row of tanks by rotation through 180 deg. The frame *K* is provided with a belt elevator *E*, with copper cups. On the shaft *F* are the pulleys *L* and *M*, which drive the elevator pulley *N*. The elevator can be brought to a horizontal position by means of the shaft *F*. The lower end of the elevator is provided with the boot *B*, which can be brought down to within a short distance from the bottom of

the tank and into which the crystals are shoveled. The power is imparted by an electric motor on the platform of the frame, which receives the electric current by an overhead wire and trolley. From the time this device came into operation the men were protected, as they had to shovel the crystals only a short distance above the floor.

The crystals are washed in a stream of mother liquor in a trough, and conveyed by this stream into a hexagonal revolving screen, having shaft and arms of brass. The screen itself is of maple wood, perforated. There are two such screens, to make two sizes of crystals. The mother liquor with the smallest crystals, dirt and sediment, after leaving the second screen, is conveyed to an agitating tank and heated by a steam jet to dissolve the very fine crystals. The resultant solution is sent through a filter-press, the clean liquor flowing to the storage tanks. The crystals are dried in ten brass centrifugal machines. The yearly production is about 18,000 tons of blue vitriol.

The crystals having been obtained from such a pure neutral solution are of a very deep blue permanent color, which is not affected by light, except in the direct rays of the sun. They do not change into a bluish white powder, which is the case with crystals made from an acid solution.

The trade in blue vitriol demands large crystals. In crystallizing a salt solution in large tanks it will be found that, while on the strips and sides large crystals are formed, the bottom will contain mostly small crystals. As this necessitated the dissolving and recrystallization of a large portion of the bottom crystals, and consequently of quite a percentage of the total production, this peculiarity was rather annoying, and, searching for the cause, I observed that on the surface of the cooling liquor numerous very small crystals were formed. They do not remain on the surface, but sink as soon as they are formed. These crystals are so small that they cannot be seen as such, but if a good light strikes the surface, the liquor right under the surface sparkles from light reflected on these minute crystals. It can easily be observed that they sink and that new ones are continually formed, thus producing a very shower of fine crystals from the surface to the bottom. This phenomenon is caused by the evaporation of the water on the very surface where the liquor is in contact with the air. By losing part of its water a very thin sheet of the

solution on the very surface will become so concentrated that it has to form and drop these very fine crystals, aided by the cooling effect of evaporation and contact with the air. Following up these observations I caused a stream of water to enter the tank under light pressure and level with the surface through a flat muzzle, so that the water did not mix with the solution, but covered as such the whole surface about an inch thick. This stopped the sparkling of these minute crystals entirely, and when crystallization was finished it was found that the bottom crystals were just as good as the crystals on the sides, so that they could be mixed and treated together with the other crystals, and the tedious and expensive operation of dissolving and recrystallizing of an already finished product was entirely avoided. After this successful trial the whole crystallizing plant was equipped with a proper arrangement for this purpose, so that the operator had only to fill each tank to the given mark and then to turn on the water for a short time. The mother liquor is somewhat diluted by this method, but the advantage gained greatly outweighs this disadvantage.

(2) EXTRACTION OF SILVER FROM BLACK COPPER

This process is based on the reaction which takes place if copper is moistened with warm diluted sulphuric acid in presence of air; the latter will oxidize the copper, and the acid will combine with the oxide, forming cupric sulphate.

This method originated and is still in operation at Oker, Germany, for which reason it is also called the Oker process.

The material subjected to this process is argentiferous and auriferous black copper. This is smelted in a reverberatory furnace, being partly refined and then granulated.

The granulated copper is charged into wooden lead-lined tubs 5 ft. high, with a bottom diameter of $3\frac{1}{2}$ ft. and a rim diameter of $2\frac{1}{2}$ ft. About 5 in. above the bottom there is a movable wooden filter bottom. Below this bottom an opening is cut out 4 in. high and 8 in. wide, to which is attached a lead trough. This opening is cut so large because it serves not only as outlet for the solution, but acts also as an inlet for air. The boards forming the filter bottom are perforated with inch holes. On top of the filter bottom large pieces of copper are placed first, then about one ton of granulated copper is charged on top of

the coarse pieces. This done, a spray of hot diluted sulphuric acid of 28 deg. B. (heated to 70 deg. C.) is made to play over the copper granules; this has to be done over the whole surface. The first acid charge will leave the dissolving tank colorless, without containing much, if any, cupric sulphate. The spray of acid is applied only for a short time. By the evaporation of the warm solution and condensation of the steam a gentle draft is produced through the opening under the filter bottom to the top of the tub, and the oxidation of the copper and the formation of cupric sulphate takes place. After a quarter of an hour or so some more hot acid is sprayed over the granules. The stream of acid dissolves the cupric sulphate that is formed, leaving the surface of the granules clean. The entering air oxidizes again the surface of the copper, which is again washed out by acid. This is repeated over and over again, and in fact constitutes the process. Instead of pure acid, crude copper solution, or acid mother liquor, is mixed with acid instead of water, which has a much more energetic dissolving action on the copper. The strength of such a solution is regulated at 32 to 34 deg. B.

This is undoubtedly a very slow and tedious process. The oxidizing part of the process would be much hastened if, instead of depending on the natural draft of a tub 5 ft. high, an artificial gentle stream of warm air could be forced through the tub, which instead of 5 ft. in height might be made 15 or 16 ft. high and charged with three to four times the amount of granulated copper. From time to time the tower could be flushed with cupric sulphate solution to prevent the granules from being clogged by the residue slimes.

Silver, gold, antimony, arsenic (the last coming mostly from the crude acid) and lead remain as slimy residues which are brought out from the dissolving vat with every charge of acid, and settle in a horizontal trough 160 ft. long, 30 in. wide and 14 in. deep. The solution in this horizontal trough moves but slowly, and when it reaches the outlet has cooled to a temperature only about 2 deg. above the temperature of the surrounding air. The crystals of blue vitriol which form in the trough are shoveled out on an inclined bench which lets the adhering solution drop back into the trough. This is done every three days. The mother liquor flows into a pressure tank made of 3-in. wooden staves and lined with 1-in. lead, which is well hooped. Instead of

compressed air steam is used, and the liquor is lifted to a reservoir which is placed on a floor above the dissolving tanks. This liquor contains much acid, and after mixing it with still more acid it is heated and used for dissolving the granulated copper.

To the crude vitriol which was shoveled out from the trough some silver slimes are adhering. They are first washed with a spray of water, then dissolved in a lead pan by boiling with water or with a mixture of water and mother liquor from the second crystallization. Crude vitriol is added until the solution measures 29 deg. B. Then the fire is stopped and the sediment allowed to settle. When the temperature has cooled somewhat and the solution become clear, it is very carefully decanted, so as not to draw off any of the silver slimes, and conveyed to wooden lead-lined crystallizing tanks of 147 cu. ft. capacity. Ten days is allowed for crystallization.

The silver slimes consist of:

| | |
|----------------------------|------------------|
| Silver | 3.068 per cent. |
| Metallic copper | 7.400 per cent. |
| Lead | 23.100 per cent. |
| Lime | 8.300 per cent. |
| Sulphuric acid | 16.200 per cent. |
| Antimony and arsenic | 27.000 per cent. |

The large percentage of antimony and arsenic comes from the impure chamber acid which is used for dissolving the copper. These slimes are well washed and mixed with litharge while still wet, then dried and melted; the resulting lead contains $1\frac{1}{2}$ per cent. silver.

XXI

THE ZIERVOGEL PROCESS

IN this process hot water is used as solvent for the silver, and the latter has to be first converted into sulphate, while the other metal sulphides, copper and iron, have to be converted into oxides. The only suitable material is copper matte containing a certain percentage of iron. If the matte contains too much or too little iron the extraction of silver becomes inferior. Really good results were only obtained from the copper matte of Mansfeld, which contains 80 per cent. copper sulphide, 11 per cent. iron sulphide and 0.4 per cent. silver.

The great sensitiveness of this process, and the complicated roasting it requires, limit its application. The roasting was described in Chapter X, on sulphating roasting, and we have, therefore, only to consider the extraction of the silver from the roasted matte. The roasted material is charged in lots of 500 lb. in small tubs 25 in. in diameter and 24 in. high. These tubs are provided with a filter bottom, and a number of them are placed in one row. The leaching is done with hot water of about 85 deg. C. The water passing through the roasted matte dissolves the silver sulphate, and leaves the tub through an outlet pipe below the filter bottom. It flows into a square settling-tank, which extends the whole length of the row and which receives the silver solution from all the leaching-tubs. This tank is 30 ft. long, 2 ft. wide and 1½ ft. high, and has longitudinally a partition which is lower than the rim of the tank, and over which the silver solution flows into the second half of the tank when the first one is filled. This tank serves to catch any material that may be carried out by the stream from the leaching-tubs. Below this settling-tank is placed a row of tubs 21 in. in diameter and 20 in. deep, corresponding in number to that of the leaching-tubs. Each of them is provided with a filter bottom. They serve for precipi-

tating the silver from the solution, and are filled with copper bars 1 in. thick, 5 in. wide and 14 in. long. In these tubs nearly all the silver is precipitated as cement silver. Flowing out from under the filter bottom the solution enters a lead-lined box 15 in. wide and 6 in. deep, which extends in front of all the precipitation filters. The bottom of this box, which has several outlets, is covered with small pieces of copper. Under each outlet is placed a tub with filter bottom on which bar copper and granulated copper are placed, and through which the solution from the outlets passes. In these tubs but very little silver is precipitated, and the solution when leaving them is free from silver. If the water used for dissolving the silver from the roasted matte is slightly acidified it hastens the dissolving and causes better extraction; it prevents the separation of basic salts.

The cement silver contains some metallic copper and some gypsum. For purification, it is placed in tubs and rubbed with wooden pestles to free the copper from the adhering silver, then washed in hand pans to separate the coarser copper, next placed in tubs and digested with diluted sulphuric acid during six to seven days to remove the copper and as much as possible of the gypsum, and finally it is washed with hot water. The washed cement silver contains in 1000 parts 860 to 870 parts of fine silver. It is dried and smelted. The desilverized solution is made to pass over scrap iron to precipitate any copper that may have dissolved from the matte by the acidified water. Usually, however, it is mixed with water and used over again a number of times to dissolve silver instead of using pure water for the latter purpose.

XXII

TREATMENT OF SILVER ORES RICH IN GOLD

To me was given the task of working the rich silver- and gold-bearing concentrates from the old Tarshish mine, Alpine county, California. These concentrated sulphides contained 258 oz. silver and over 10 oz. in gold per ton, which represents the average assay of five months. All the gold was contained in the sulphurets; no metallic gold could be detected. The plan of operation was self-suggesting, viz.: to subject the material to a chloridizing roasting, then to impregnate it by Plattner's method with chlorine, and then leach, first with water for the extraction of the gold, and afterward with sodium hyposulphite for the extraction of the silver. When this scheme was executed it was found, however, that while a high percentage of silver could be extracted, only about 50 per cent. of the gold was thus extractable. The reason is not easily explained. After operating for a while in this way and paying particular attention to the roasting without obtaining any better result, the operations of the process were reversed. The ore, after being roasted with salt, was leached first with water and with sodium hyposulphite for the extraction of the silver, and then treated with chlorine gas by Plattner's method. This reversing of the operations had a most beneficial influence on the result, effecting an extraction of 95 per cent. of the gold.

The operations were as follows:

(1) *Roasting.* — The roasting was done with 10 per cent. of salt, the salt being added after the oxidation had progressed for some time. Soon after the salt was added free gold could be detected by concentrating a sample of the ore in a horn spoon. Roasting was continued until the concentration test did not show any undecomposed sulphurets and but very little magnetic iron, when the concentrated part of the sample was tested with a magnet, while a large amount of very bright yellow gold was visible.

The roasted charge, after cooling, was sifted through a screen of 10 meshes to the running inch. The fine was charged into the filter tanks while the coarse was accumulated in a larger lot, then crushed dry in a battery and slightly roasted.

(2) *Base-Metal Leaching.* — In order to prevent, or rather to greatly diminish, the dissolving of silver chloride by the base-metal solution, I devised and practised the following mode of operation: Instead of applying the stream of water on top of the ore, as is usually done, the water was made, under very slight pressure, to enter the vat under the filter bottom and to ascend gradually through the ore. In this way the concentrated part of the solution which dissolves the silver chloride accumulated on top of the ore. If, then, this concentrated solution was diluted by a stream of water applied on top and the solution was permitted to flow out from under the filter bottom, the silver chloride was precipitated on and through the ore, and was dissolved again by the subsequent leaching with sodium hyposulphite. Sufficient room in the vat was left above the ore for this operation.

(3) *Leaching the Silver.* — This was done in the usual way by applying a diluted solution of sodium hyposulphite. The resulting silver bullion was 957 fine.

(4) *Second Leaching with Water.* — After the silver was extracted the solution of sodium hyposulphite was pressed out by water, and washing was continued until the outflowing liquid was perfectly free from sodium hyposulphite. Such a careful washing is necessary, because sodium hyposulphite added to a solution of gold chloride prevents the precipitation of the gold by ferrous sulphate.

The desilverized and washed ore was removed from the vat to a drying kiln, where it was left for a time till the surplus water had evaporated. After this it was charged back into the vat, still moist. This second handling and drying cannot be avoided, because the ore after leaching is too wet and tightly packed to permit a free passage of the chlorine gas.

(5) *Extraction of the Gold.* — After the extraction of the base-metal chlorides and the silver the gold is left in a metallic state, and bright and clean, permitting a very close extraction. On the inside periphery of the vat a groove was cut into the staves from the rim down, forming a shoulder or recess into which a tight wooden cover fitted. The shoulder was $2\frac{1}{2}$ in. below the rim, so

that when the 1-in. cover was put on the staves projected $1\frac{1}{2}$ in. above the cover. Around the periphery the cover was tightly luted with clay, and then water was poured on it to about the depth of one inch. This sheet of water kept the cover perfectly tight. The water, however, was not poured on the cover until the gas appeared on the surface of the ore. The cover was provided with two $1\frac{1}{4}$ -in. pieces of pipe projecting about 6 in. above the cover, and a square opening 6 x 6 in. When the chlorine gas appeared above the ore this opening was closed with a cover luted tight with clay and the water poured on top of the cover. When the gas commenced to escape through the pipes in the cover they both were closed with balls of clay. The ore was left in contact with the chlorine gas for twelve hours, and as soon as it was ready for the extraction these clay balls were removed and one of the pipes was connected with the water-pipe by a hose, while the other was connected, by means of a hose, either with another vat already prepared for chlorination, or with the ash-pit of the roasting furnace. This was done to utilize the surplus of chlorine gas, and to protect the workmen from its very injurious effect. Care was taken to place a sack, kept in place by bricks, on top of the ore right under the water inlet, in order to prevent the stream from working into the ore.

Chlorine was generated in a leaden gas generator heated by steam.

The gold solution was collected in precipitation tanks and precipitated with a solution of ferrous sulphate. Separate tanks were used for the precipitation of the silver. There was also a separate line of troughs for each metal, to guard against the entering of any sodium sulphite solution into the gold solution, because, as stated above, sodium hyposulphite prevents the precipitation of the gold by ferrous sulphate.

In working ordinary gold-bearing sulphurets by the Plattner method, the gold solution turns jet black when the iron solution is added, which is caused by the precipitation of the gold in metallic state, but in such an extremely finely divided condition that it assumes this color. The concentrated sulphurets from the Tarshish mine were very variable in their silver and gold contents, and sometimes lots were treated containing as much as \$700 to \$800 per ton in gold. When such rich gold ore is chloridized, the solution carrying out the gold is of a very lustrous yellow

color, and if ferrous sulphate solution is added red-brown clouds are formed, which rapidly sink to the bottom. There the gold accumulates in spongy lumps of great specific gravity, and some of them show scales of bright gold, which under the microscope might prove to be crystallized gold. There is but very little more time used in leaching rich gold ore than poor, on account of the great solubility of gold chloride in water.

The gold was well washed, dried and melted with borax, while the silver precipitate was melted with iron and borax in graphite crucibles.

Results. — To ascertain the working results of this method the concentrated raw sulphurets delivered to the reduction works each day were carefully weighed and assayed during a period of five months. The average value of these concentrates, as mentioned above, was 258 oz. silver and a little over 10 oz. gold per ton. The total value of the bullion shipped during this period, compared with the value of the raw sulphurets worked during this time, showed an actual extraction of silver 96 per cent. and of gold 95 per cent.

The gold obtained was of high fineness, varying from 970/1000 to 987/1000.

XXIII

CYANIDATION OF AURIFEROUS SILVER ORES

THE cyanide process is based on the fact that gold and silver in presence of oxygen dissolve in an alkaline solution of potassium or sodium cyanide. The cyanide solution, however, dissolves also other metals and their sulphides, like iron, copper, zinc, lead, antimony, arsenic, etc., but it dissolves them much slower, which condition makes the process possible, because, in dissolving, these substances decompose the potassium cyanide, and as they as a rule offer a very much larger surface than the gold and silver, they would at equal dissolving ratio consume so much potassium cyanide as to render the process financially impracticable.

Some of the above-named metals and their compounds act more energetically on the potassium cyanide than do others, but as they all act deterioratingly it is apparent that ores heavily charged with sulphureted minerals are not suitable for this process unless they are first subjected to a chloridizing roasting. This, of course, reduces greatly the applicability of the process for silver ores, because much the larger part of them are complex sulphureted ores. If the ore has to be roasted with salt, the process enters into competition with the lixiviation process with sodium hyposulphite, and then it is doubtful whether it will prove to be superior, except when the ore contains a sufficient amount of gold.

TREATMENT OF RAW ORE

To be suitable for this treatment the ore has to be but slightly mineralized and the silver ore to occur in its purer varieties, as sulphide, chloride or chlorobromide. But even from such clean ores the extraction percentage of the silver varies greatly. Some of them yield 90 per cent. and more of their silver to the solvent,

while others of similar character will yield only 50 per cent. Much more reliable and uniform results are obtained with regard to the extraction of the gold, for which reason the process is more suitable for treating clean auriferous silver ores than silver ores which do not contain any gold. Large quantities of ores can be cheaply treated by this process in simple appliances requiring but little repair, which give the means of working low-grade ores which by no other known process could be worked profitably. A number of mining properties are worked now with profit which formerly were not productive because the ore was too poor in silver and gold to be treated by other more costly processes.

Raw ores when finely pulverized permit only a very slow percolation of solution, owing to exceedingly fine slimes which are formed in pulverizing, and which, in some cases, pack so as to be practically impenetrable. To overcome this difficulty, the ore is crushed wet, and the slimes are separated from the sand either by cones and other sand-separating appliances or by conveying the pulp from the stamp battery direct, or, where concentration is practised, from the discharge of the concentrating tables to large masonry tanks to retain the sand, and the overflow of these tanks to another series of tanks to collect the slimes. The latter is a rather crude method, permitting only a very imperfect separation, and necessitates considerable handling of the material, while by the former method the filling of the tanks is done automatically. However, there may be circumstances which make the adoption of the second method necessary or even more practicable.

In the following I give an abstract of a very careful, intelligent and exhaustive record of the cyaniding of auriferous silver ores of Palmarejo, Chihuahua, Mexico, written by T. H. Oxnam, mining engineer, Palmarejo & Mexican Gold Fields, Ltd., Chinipas, and read at the Washington meeting, May, 1905, of the American Institute of Mining Engineers:

CYANIDING AURIFEROUS SILVER ORES AT PALMAREJO, MEXICO

The predominating value of the ore now treated by the Palmarejo and Mexican Gold Fields, Ltd., is silver. The method consists of wet-crushing and concentrating, followed by cyanidation of the unroasted sand and slime.

The Palmarejo mines are located in the southwestern part of

Chihuahua, on the foothills of the Sierra Madre, and at an elevation of 3200 ft. The mills, 12 miles distant, are situated on the Chinipas river, near the town of Chinipas, which is 150 miles northeast of Agiabampo, on the Gulf of California. Supplies are shipped via this port.

Mill and Cyanide Plant. — The 50-stamp mill and cyanide plant are situated on the Chinipas river, 1.5 miles east of Chinipas, at a place known locally as "El Zapote." Water-power, furnished by the river, is used to run the mill, slime plant and machine-shop. A masonry conduit 11 miles long conducts the water to a penstock a short distance above the mill, thence through a steel pipe, 1100 ft. long, tapering from 48 in. in diameter at the penstock to 22 in. at the wheel-pits; here there are four 6-ft. Pelton wheels under a 97.5-ft. head.

The ore consists essentially of a silicious matrix in which is disseminated a small percentage of pyrite. Black manganese oxide, and calcite are present in varying proportions, and small quantities of antimony and arsenic, together with traces of bismuth, also occur. Occasional traces of copper and zinc are found. The major portion of the silver occurs in the form of argentite, though a certain amount of stephanite is present, and occasionally small patches of chlorobromide and native silver.

System of Milling. — The ore, averaging 6 per cent. moisture, is brought to the mill in trains of from 9 to 14 cars, and is dumped into the main upper storage-bin, which has a capacity of 1100 tons. From this the ore is drawn over 3.5 x 10-ft. iron grizzlies having 1.5-in. openings to the 7 x 10-in. Blake rock-crushers, which run at 250 r. p. m. and crush to 2-in. size. Of the dump-ore, which is coarse and extremely hard, approximately 90 per cent. goes to the crushers; of the mine-ore, which is finer and softer, approximately 50 per cent. goes to the crushers, the other 10 and 50 per cent., respectively, falling through the grizzlies.

A secondary storage-bin (of 1100-ton) receives the ore from both grizzlies and crushers. The ore is then trammed to three small intermediate bins, each of 50 tons capacity; from here it is conveyed, by means of half-ton cars, to the hoppers of the Challenge ore-feeders. This double handling of the ore is inconvenient, but is rendered necessary because of the construction of the mill, which was originally erected for different requirements.

The stamps, when equipped with new shoes, weigh 850 lb. They drop 6 to 7 in., 100 times per min., the order of drop being 1—3—5—2—4; 20-mesh brass-wire screen, No. 26 wire, is used; the height of discharge is kept at 2 in. The stamp-duty is from 2.75 to 3.25 tons per twenty-four hours. The average stamp-duty would doubtless be somewhat increased by the installation of narrow mortars of the Homestake pattern. For some time past, forged-steel shoes have been used in preference to the cast-iron shoes of our own make. The steel shoes cost 15c. per ton of ore crushed, as compared with 18c. for the cast-iron shoes. We cast all our own dies, for which purpose the worn-out shoes and iron and steel scrap are employed. The average life of a forged-steel shoe is 95 days, while that of a cast-iron die is 33 days.

From the batteries the pulp passes directly over 10 Wilfley concentrators, running with a $\frac{7}{8}$ -in. stroke at 215 strokes per min. During the year ending July 1, 1904, the concentrator saved 0.76 per cent. by weight of the ore as a product which contained 18.28 per cent. of the gold and 17.98 per cent. of the silver of the ore.

A wooden launder conveys the pulp from the tables to the tailing elevator-wheel. The latter is 14 ft. in diameter and is of the outside-bucket type, having 22 steel buckets (each 18 in. long, 8.5 in. wide and 8.5 in. deep, with a capacity of 1025 cu. in.). The wheel is driven by a $\frac{5}{8}$ -in. plow-steel wire cable at a speed of 18 r. p. m. The discharge efficiency, as in all wheels of this type, is not high, the tailing leaving the wheel in a launder 5.5 ft. above the level of the mill launder supplying the pulp.

A large masonry sand-retaining tank (divided into four compartments, each compartment measuring 25 x 80 x 4 ft. in depth) receives the product from the wheel. Distribution is effected by a central launder in each compartment, provided with a number of 4-in. side-discharge pipes. Each compartment is provided with a removable end-discharge gate, 4 ft. wide, composed of pieces of 2-in. plank, planed smooth on the edges and sliding in guides secured to the side-posts. As the compartment fills up with sand, the discharge of these gates is raised. The discharge overflow empties into the main slime launder. Each compartment also communicates with its immediate neighbors by small side-discharge doors. The purpose of this arrangement is that the mill product may be emptying into one compartment,

from which a portion of the finer material escapes through one of the side-gates to an adjoining compartment, while the finest material is passing off in the discharge over the lowered end gate of this second compartment.

It is found, however, that a considerable quantity of the finest material will always tend to collect at the lower end of the first compartment, receiving the discharge of the elevator-wheel, to lessen which an overflow from the end gate of this compartment is also necessary. The first five or six tons of material removed from the compartments is always slimy, and is trammed a short distance to an open drying-patio, where it is spread out, sun-dried and broken up; after this it is mixed in with the coarser sand and treated in the leaching-vats. A third compartment of the sand-retaining tank is kept full of sand, which is being drained while the dry sand is trammed from the fourth compartment. Each of these compartments holds the sand of forty-eight to sixty hours' crushing in the mill. The retained sand is usually subjected to two days' draining before charging it into the leaching-vats. The fine material escaping in the overflow from the masonry retaining-tank is carried by a wooden launder to three so-called "slime pits," having an aggregate capacity of 15,000 tons. Every precaution is exercised that no slime escapes at the overflow gates of these pits; but at no time is such overflow perfectly free from suspended matter. During the 18 months (ending December 31, 1904) of the total net tonnage crushed in the mill, 19.16 per cent. went to the slime pits. Sizing tests (using the ordinary brass-wire assay screens) have shown that about 6 per cent. of this material is retained on the 100-mesh, while 85 per cent. passes a 200-mesh screen.

Although this material is chiefly slime (which on long drying cracks up into layers almost impervious to leaching), yet it is found that considerable fine but leachable sand is deposited at the head of the slime pits and near the discharge from the slime launder. About two months after ceasing to discharge into any one slime pit, this fine sand at the head of the pits dries sufficiently (during ordinary weather) to permit of being walked on; it is then conveyed, by contract labor, to the open drying-floor, or patio, together with a certain percentage of more slimy material which unavoidably becomes mixed with it. Here it is spread out, sun-dried and thoroughly broken up, after which it is

mixed in with the ordinary sand and treated by leaching. During the past year (1904) 2400 tons of very fine material from the slime pits has been so treated. By far the greater portion of the material collected in the slime pits, however, is so extremely fine and has such a clayey nature that it is almost impervious to leaching. This portion, slime, is allowed to dry as much as practicable, and is then treated by agitation in a separate plant, as will be described further on in this paper. Figs. 74 and 75 show the arrangement of the cyanide leaching plant.

Cyanidation of Sand. — The sand caught in the large masonry sand-retaining tank (after being allowed to drain as long as possible, usually from thirty-six to forty-eight hours) is trammed in half-ton cars to the cyanide leaching-vats; these are 12 in number, 30 ft. in diameter and 4.5 ft. deep. The filter bottom (which reduces the available depth to 4 ft. 2 in.) consists of a wooden, lattice framework, covered by a layer of cocoa matting, over which is stretched a filter cloth of 8-oz. duck. Two heavier grades of duck have been tried, but they reduced the rate of leaching and gave less satisfactory service than the 8-oz. cloth; 10 of the leaching-vats are constructed of No. 9 sheet steel; the other two were built on the premises of 3-in. native pine. Two additional vats, of the same dimensions and capacity, made of 3-in. redwood throughout, are in course of erection.

The sand, as charged into the leaching-vats, carries 14 to 16 per cent. of moisture; each vat holds 100 tons of dry sand. While being trammed to the leaching-vats, slaked lime is added to each car, and in the proportion of 4 to 5 lb. of lime per ton. The vats are filled and discharged by contract, for \$19 a vat, equivalent to \$0.19 per ton.

Two stock solutions are employed: the weak, of 0.25 to 0.30 per cent. KCN; the strong, of 0.75 to 0.80 per cent. KCN. The working strength of the solutions is always taken as that indicated by titration with silver nitrate (in presence of a few drops of a 10 per cent. solution of potassium iodide, as an indicator), 10 c.c. of the cyanide solution being taken for titration. For convenience, we still express the strength of our working solutions in terms of potassium cyanide, although for over a year past we have been employing sodium cyanide exclusively. Titration with silver nitrate shows that the sodium cyanide used is equivalent to about 125 per cent. of potassium cyanide. Our experience with sodium

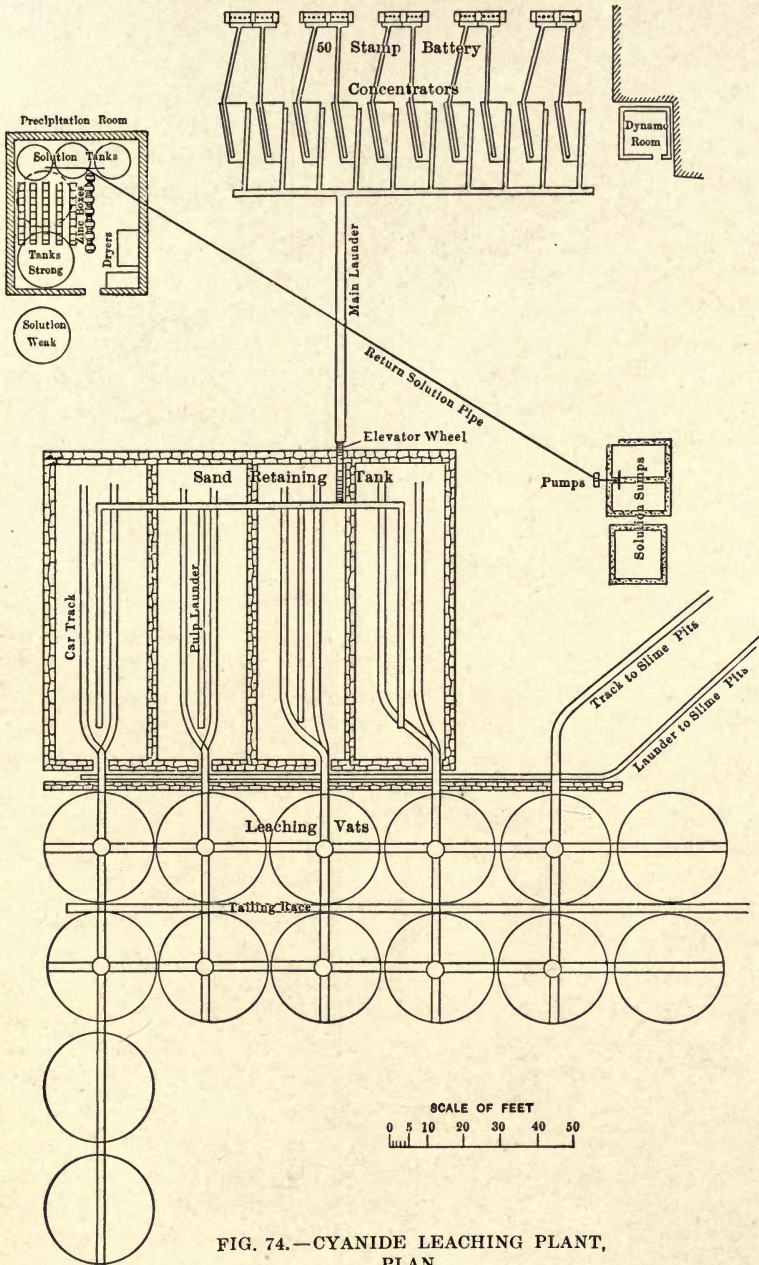


FIG. 74.—CYANIDE LEACHING PLANT, PLAN.

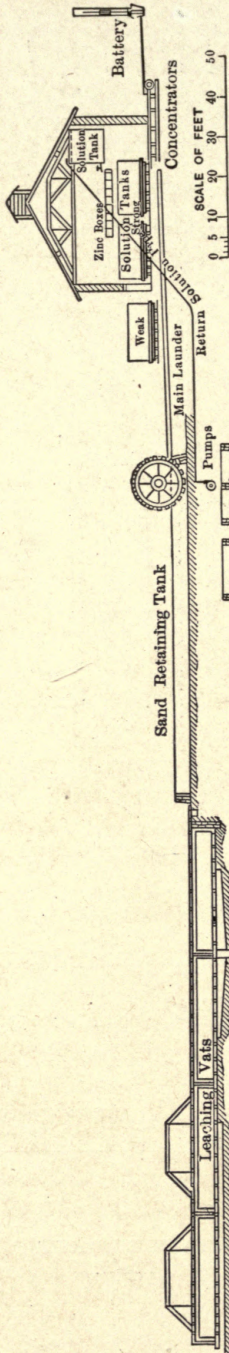


FIG. 75.—CYANIDE LEACHING PLANT, ELEVATION.

cyanide leads us to believe that it is fully as efficient as potassium cyanide. It also appears that, since commencing the exclusive use of sodium cyanide, our solutions become less fouled than previously. By the adoption of sodium cyanide a saving of 20 per cent. of the freighting expense on this article has been effected. Besides the saving in transportation expenses, the sodium cyanide appears to possess other advantages. Other things being equal, it would seem preferable to use a salt as pure as can be obtained. Absolutely pure sodium cyanide is equivalent to about 132 per cent. of potassium cyanide; a product, testing from 125 to 130 per cent. of potassium cyanide, is nearly pure. It by no means follows, however, that the ordinary commercial cyanide, rated as 98 to 99 per cent. pure, contains but 1 to 2 per cent. of impurities. That this commercial cyanide frequently carries a varying percentage of sodium cyanide is a well-known fact; and it of course naturally follows that the greater the percentage of sodium cyanide contained in the ordinary 98 to 99 per cent. potassium cyanide, the greater the percentage of impurities.

As soon as a vat is charged, from 20 to 25 tons of weak solution (carrying, as just stated, from 0.25 to 0.30 per cent. of KCN) is introduced, from the bottom, by means of a 2-in. drop-pipe, terminating in a T underneath the filter. This solution is introduced slowly in order to avoid channeling of the charge; it usually makes its appearance on top of the sand about six or seven hours after being turned on. When the solution stands 2 or 3 in. above the top of the charge, it is turned off, and the material is allowed to soak for six hours, during which time the sand will usually have settled from 3 to 4 in. The weak-solution discharge-valve at the bottom of the vat is now opened and leaching is commenced. During the next two or three days weak solution is added from the top, as rapidly as permitted by the leaching rate of the charge, until a total of from 100 to 130 tons has been applied. From 60 to 70 tons of strong solution, averaging between 0.75 and 0.80 per cent. of KCN, is now run through the charge at a somewhat slower rate, the usual time consumed by this operation being forty-eight hours. Weak solution is next run through the charge as rapidly as possible, until twenty-four hours before the time it is to be discharged; then wash-water, to the amount of 15 to 20 tons, is added in lots of 5 tons each. The residue is then ready for sluicing, which is accom-

plished by two men in about six hours, each using a 2-in. hose, equipped with a 0.5-in. nozzle and operating under a head of 72 ft. After finishing the sluicing, the canvas filter is usually swept clean with a broom; if this is not done, it is found that the filter cloth clogs with fine slime and the rate of filtration is lowered. Each vat is equipped with two 10 x 10-in. square bottom-discharge doors.

The quantity of wash-water used is regulated by the balance of the solutions on hand. Although a separate zinc-box is provided for waste solution, it is seldom used except during the rainy season; during other parts of the year but little solution is run to waste. Only two of the leaching-vats are under cover, and during the rainy season it becomes necessary to run a certain percentage of the solution to waste; each heavy rain gives the exposed vats a very appreciable quantity of water.

It has been my experience that thorough oxygenation of the material is a very desirable feature in the cyanidation of gold ores; in the case of the Palmarejo ores, this is essential to obtain the best results. Due to the fact that the major portion of the value is in silver, the actual weight of fine metal to be acted upon is much greater than is ordinarily the case with gold ores.

In order to permit as much air as possible to be supplied to the sand during treatment, the solution is frequently allowed to drain down several inches beneath the surface of the charge; air is thus allowed to penetrate the material to this depth. It is our custom to assay each charge every twenty-four hours, after the first five days of treatment. Before each sampling, the solution is allowed to drain down several inches below the surface sand, thus allowing additional opportunity for the entrance of air into the upper layer of the charge.

Under the most favorable conditions, however, the air drawn into the top layer can have but little effect on the lower half. It is doubtless due to this difference in aeration of the upper and lower portions that the lower half of the tailing will run from 1 to 2 oz. of silver higher than the upper half. Frequently this difference is even more marked, a variation of 3 or 4 oz. being obtained between the upper foot and the bottom foot of the residue.

To overcome this, and after many experiments, some time ago the practice was adopted of transferring as many charges as pos-

sible from one vat to another during the treatment. To transfer a vat means the loss of practically twenty-four hours of its available leaching time, because it is necessary to drain the charge for twelve hours before commencing to transfer it. Also, it is necessary that one of the adjoining vats be empty at the proper time to receive the transferred charge. By careful manipulation, at present about one-third of the total number of charges treated are transferred. When the two additional vats, now in course of erection, shall be completed, a greater number of charges can be transferred and the additional capacity afforded will also permit a longer treatment to offset the time lost in transferral. The transferring is done by contract for \$16 a vat, which is equal to 16c. per ton. While being transferred, the material is of course given a thorough exposure to the air; any existing lumps are broken up by the shoveling; and, roughly speaking, the bottom layer of the original charge becomes the top layer of the transferred charge. Operations are usually so timed that the transfer takes place while the strong solution is in contact with the material. During the transfer, 100 lb. of slaked lime is evenly distributed near the bottom of the vat receiving the transferred charge.

The charge is sampled just before and just after transferral, the latter sample being 1 oz. higher in silver than the former, a result doubtless due to the fact that the tendency is to obtain a larger percentage of the top half of the charge than of the lower half; and, as heretofore mentioned, the lower half of the original charge, after the transferral, becomes practically the upper half of the transferred charge.

The first solution added after the transfer is introduced slowly from the bottom, after which the regular routine treatment is continued. The value of the effluent solution from a charge is found to increase immediately after the charge has been transferred, such increase being usually from 2 to 3 oz. of silver per ton of solution.

In general, all head- and tailing-samples of the sand are taken with a 1.5-in. auger at 12 to 18 different points.

The record of a single charge, which is representative of what is regularly obtained in ordinary operations, is as follows:

Extraction, 96.27 per cent. of gold and 53.21 per cent. of silver.

Total time of treatment, including charging and discharging, 11 days.

Solution added: weak, 261; strong, 64; wash-water, 15; total, 340 tons.

All tailing-samples, with the exception of the discharged tailing, were washed before assaying.

During transferral a sample taken from upper 18 in. of charge assayed: \$0.50 of gold, 9.20 oz. of silver; and one from the lower 18 in. of charge assayed \$0.82 of gold and 11.92 oz. of silver.

The effluent solution from the leaching-vats is carried to the sump-tanks by two separate lines, one for the weak, the other for the strong solution. These tanks are of masonry and are three in number. Two of them (having a combined capacity of 65 tons) are connected and serve as a weak-solution sump; the other, having a capacity of 25 tons, is used for the strong solution. All solution draining from the leaching-vats is passed through the zinc-boxes before being returned to the vats.

The proper tonnage of strong solution is maintained by determining the strength of the effluent solution from the leaching-vats; when this strength reaches 0.35 per cent. of KCN, the solution is turned into the strong-solution sump. As a working guide for maintaining the proper alkalinity of stock solutions, they are titrated every day with the addition of about 5 c.c. of strong lime-water; 10 c.c. of cyanide solution is used in all titrations. If the addition of lime-water causes a difference of more than 0.5 lb. in the indicated strength of the solution, the quantity of lime added to the sand charged into the leaching-vats is increased.

From the sump the solution is elevated by a 3-in. centrifugal pump (900 r.p.m.), to three storage-tanks at the head of the zinc-boxes, a vertical distance of 29 ft. and a horizontal distance of 150 ft. These tanks are each 10 ft. in diameter, 8 ft. deep, and have a capacity of 19 tons. Two of these tanks are used for the weak, and one for the strong solution. The solution from the vats now passes through the zinc-boxes, from which it is led to three storage-solution tanks beneath the boxes. These storage-tanks are made of No. 9 sheet steel, each being 15 ft. in diameter, 6 ft. deep, and with a capacity of 33 tons. Two of them are used as strong-solution storage-tanks; the other, as a weak-solution storage-tank. The strong solution is brought to

the required strength by adding cyanide to the last compartment of the strong-solution zinc-box, which is reserved for this purpose. No cyanide is added directly to the weak solution.

Precipitation of Silver and Gold. — There are six zinc-boxes, five for the weak and one for the strong solution. The five weak-solution boxes are constructed of No. 10 sheet-steel, and are 2 ft. wide and 18 ft. long over all. Each box contains eight compartments, each compartment having an available zinc capacity of 24 x 24 x 18 in., equivalent to 6 cu. ft. Six compartments only are filled with zinc shavings; therefore, each box, when freshly dressed, contains 36 cu. ft. of zinc shavings, making a total of 180 cu. ft. of zinc shavings in the five weak-solution boxes.

The strong-solution zinc-box consists of seven individual round boxes or compartments, placed in series, each compartment being 28 in. in diameter and 24 in. in depth, and having an available zinc capacity of 5 cu. ft. Only six of the compartments are filled with shavings, the last compartment being reserved for the addition of the quantity of cyanide required to bring up the strong solution to standard strength. The strong-solution zinc-box has, therefore, a total of 30 cu. ft. of zinc shavings.

Records are kept of the quantities of weak and strong solution daily passing through the boxes, together with their assay values before and after precipitation. These records for the year (1904) show that 91,793 tons of weak, and 22,251 tons of strong, solution passed through the boxes; this is equivalent to an average of 251 tons of weak, and 61 tons of strong solution every twenty-four hours. During this period the flow of solution through the boxes was interrupted on various occasions for a short time, due to the ordinary clean-ups, dressing of the boxes and unavoidable delays. Without taking such stoppages into account, the average rate of flow through the boxes equaled 1.4 tons of weak solution per twenty-four hours per cubic foot of shavings, and 2.03 tons of strong solution per twenty-four hours per cubic foot of shavings.

The actual rate of flow exceeds these figures, as it is assumed that the boxes were at all times kept dressed with the maximum amount of shavings, which was seldom the case.

The shavings are cut on an ordinary zinc lathe, from No. 9 sheet zinc, the size of the sheets being 18 x 84 in. Ordinarily,

six sheets are wound on the mandrel of the lathe for one cutting. One boy, working twelve hours, cuts sufficient shavings to supply both the leaching and agitation plants, which together require an average of 120 lb. per twenty-four hours. It is found best to keep only a few days' shavings on hand; freshly cut shavings give better results than those which have been cut for some time. The customary practice of moving the zinc from the lower to the upper compartments, when dressing the boxes, is not followed, fresh zinc being added as required to the top of each compartment.

The strength of the solution running through the weak boxes will average between 0.25 and 0.30 per cent. of KCN; while that of the solution going to the strong zinc-box will average between 0.35 and 0.45 per cent. of KCN.

The average assay values per ton of the solutions entering the zinc-boxes are approximately as follows:

Weak solution, \$1 of gold and 2.25 oz. of silver; strong solution, \$1.24 of gold and 3.5 oz. of silver.

It is seldom that any trouble is experienced with the precipitation of the contained values. As a rule, the precipitation of the gold is practically perfect; that of the silver averages 95 per cent. When precipitation falls off, it is usually due to the presence of an accumulated excess of lime in the solution.

Clean-up of Zinc-boxes. — On account of structural difficulties, it is necessary to handle the precipitates more than is desirable. The boxes are cleaned twice a month. Before commencing on any box, clear water is passed through it a sufficient length of time to displace most of the cyanide solution; this requires 10 or 15 minutes. The shavings in the first compartment are thoroughly washed, after which they are removed and the water bailed out into the next compartment. The precipitates are now conveyed by buckets to the clean-up box, where they are passed through a 20-mesh screen. A small percentage of "short" zinc passes through this screen, but the greater part of such product is here separated from the finer precipitate and is returned to the boxes. The first compartment is now filled with water; the zinc contained in the other compartments is gradually transferred to it and thoroughly washed, the precipitates from each compartment being carried to the clean-up box as before mentioned. To minimize the oxidizing effect resulting from exposure of the

wet zinc to the atmosphere, the washed shavings are at once placed in the highest vacant compartment of the zinc-box and covered with solution.

The precipitate accumulating in the first compartment from the washing of the shavings, after settling for a short time, is also removed to the clean-up box. This latter is provided with three smaller settling-boxes, placed in series, which take the overflow from it. The bottom of the clean-up box is tapped by a 4-in. drop-pipe, which discharges directly into two large drying-pans beneath.

The product is now dried as much as is practicable, and then mixed, carefully sampled, assayed and sold on the premises to one of the large ore-buying companies. The moisture in the dried precipitate has averaged 0.27 per cent. during the past year.

The clean-ups are bulky; the net dry-weight of precipitate in each clean-up averaged between 1100 and 1200 lb. avoirdupois during the past year.

Considering the fact that the precipitates receive no treatment whatever beyond being passed through a 20-mesh screen, and the simple drying, as above mentioned, it is rather surprising that they carry such a high percentage of fine metal. During 1904 the assay returns, on which the sale of the precipitates is based, have averaged slightly over 20,000 oz. silver, and approximately \$8000 of gold, per short ton. By actual weight, therefore, the percentage of fine metal contained in the dried product recovered throughout this period was approximately 68.57 per cent. of silver and 1.33 per cent. of gold, making a total of 69.90 per cent. of both metals.

Two clean-ups during 1904, of a combined net weight slightly exceeding 2300 lb., gave an average assay value of 22,200 oz. of silver per ton, making the fine-silver content equal to 76.12 per cent. by weight of the precipitates.

A record of the labor employed in clean-ups shows that four men (one American and three native helpers) would readily remove 1200 lb. (net dry-weight) of precipitate from the boxes, and have the product in the drying-pans in eight hours. Based on the average assays of the precipitate for the year, this means that in thirty-two man-hours approximately 12,200 oz. of fine metal would be handled, being equivalent to a duty of 371 oz. per man-hour. This rather high duty is of course due entirely

to the fact that the precipitate contains high percentages of precious metals.

Tonnage and Extraction. — During 1904, 34,900 tons of sand were treated in the leaching-plant. This tonnage would have been considerably greater had it not been that during this period, aside from the stoppages due to general repairs, the mill was closed down for intervals aggregating 57 days for the entire 50 stamps.

The extraction for 1904 (as indicated by the assay differences between the sand charged into the leaching-vats and that being discharged) was 95.5 per cent. of the gold and 52.5 per cent. of the silver value. The combined total during the year checks closely with that called for by the sand assays. The assay value of the sand treated during this period averaged \$2.85 of gold, and slightly more than 16 oz. of silver per ton. During 1904, the returns from the precipitates were practically 1 per cent. less in gold and 0.5 per cent. more in silver than those called for by the precipitation records.

Consumption of Cyanide, Zinc and Lime. — The office records show that for 1904 the consumption per ton of sand cyanided was as follows: cyanide, 2.95 lb.; zinc, 0.96 lb.; and lime, 4.33 lb. Expressed in terms of potassium cyanide, this consumption would equal 3.69 lb. of potassium cyanide per ton of ore treated.

General Remarks. — The total quantity of solution passing through the zinc-boxes during 1904, divided by the quantity of sand for the same period, shows that, for each ton treated, 3.27 ton of solution left the leaching-vats, of which 2.63 tons was weak and 0.64 ton strong solution. It is found that a large quantity of weaker solution gives more satisfactory results than a small quantity of the strong, and it is always made an important point to pass as much weak solution through a charge as possible. Experience has demonstrated that, in a given length of treatment, a rapid leaching rate and a large quantity of solution are more efficient than a slower leaching rate and a consequently lesser quantity of solution. The solution pipe-lines and launders occasionally become quite choked in places with scale deposited from the solution. This scale, taken from lines carrying precipitated solution, contained from a trace to \$1 of gold and from 1 to 7 or 8 oz. of silver per ton. The scale deposited from the unprecipitated solution usually runs higher, several assays taken having averaged about \$5 of gold and 18 oz. of silver per ton.

Ordinarily the solutions do not become excessively fouled. They contain small quantities of iron and manganese in addition to the zinc compounds present. Alkaline sulphides are very rarely or never noticed in solution; however, sulphocyanates and ferrocyanides appear to be constantly present in fair quantities, about 0.41 per cent. of ferrocyanide and 0.048 per cent. of sulphocyanate.

The sand charged averaged about 0.09 per cent. of latent acidity; and, as a rule, it contained no free acid.

The concentrate produced is sold to the same company that buys the cyanide precipitate. An attempt had been made to treat the concentrate by cyanide, but without success. Experiments on both raw and dead-roasted concentrate (reduced to various degrees of fineness) by leaching and agitation, for varying periods of time up to 34 days, and using solutions varying from 0.2 to 2 per cent. of KCN, proved unsatisfactory.

Table I, given herewith, shows the working costs for milling and cyaniding during 1904. The cost of all supplies is increased by the heavy freight transportation expenses, as well as by the duties placed by the Mexican Government on most of the supplies used.

TABLE I—WORKING COSTS PER TON

Milling:

| | |
|----------------------------------------------|---------|
| Supplies | \$0.640 |
| Labor | 0.357 |
| Lubricating | 0.023 |
| Assay office (labor and supplies) | 0.035 |
| Concentrating | 0.092 |
| Power (ditch maintenance and supplies) | 0.234 |
| Salaries | 0.264 |
| Miscellaneous (lighting, etc.) | 0.018 |
| Management and general expenses | 0.336 |
| Total ¹ | \$1.999 |

Cyaniding:

| | |
|----------------------------------------------|---------|
| Cyanide (2.95 lb. @ \$0.63) | \$1.859 |
| Zinc (0.96 lb. @ \$0.30) | 0.288 |
| Lime (4.33 lb. @ \$0.0118) | 0.051 |
| Other supplies | 0.050 |
| Labor | 0.329 |
| Salaries | 0.371 |
| Assay office (labor and supplies) | 0.036 |
| Power (ditch maintenance and supplies) | 0.017 |
| Miscellaneous (lighting, etc.) | 0.004 |
| Management and general expenses | 0.186 |
| Total ² | \$3.191 |

¹ \$1.999 Mexican currency during this period was equivalent to \$0.95 gold.

² \$3.191 Mexican currency during this period was equivalent to \$1.22 gold.

The cost of realization on cyanide precipitate has not been included in the cyanide working costs given herewith. Transportation expenses on the precipitates are also very heavy. In addition to these comes the heavy item of government bullion taxes. The average cost of realization on cyanide precipitate, per ton of ore cyanided, is as follows: Government taxes, \$0.84; treatment charges (including transportation expenses), \$1.06; total, \$1.90. The cost of realization on the concentrate produced is also unusually high on account of the heavy transportation expenses and government bullion taxes. The average cost of realization per ton of ore crushed is as follows: Government taxes, \$0.35; treatment charges (including transportation expenses), \$1.08; total, \$1.43.

Treatment of Slime. — As already mentioned, the accumulated and currently produced slime is now being treated in a separate plant by a system of agitation and decantation, centrifugal pumps being used as the means of agitation. The slime plant consists essentially of the following parts and accessories:

Four agitation and four decantation vats, each provided with conical bottoms and connected with its own separate centrifugal pump; two solution tanks placed at the head of the zinc-boxes, which receive the solution from the decantation vats; four sets of zinc-boxes and three solution sumps, which receive the solution leaving the zinc-boxes; one special solution tank, placed at a higher level than the rest of the plant, and used principally to supply solution to the pump bearings under pressure; two ordinary 3-in. centrifugal pumps, used only for pumping solution from the sumps to any desired vat or to the upper solution tank just mentioned, they being so connected up that either pump can be used should the other get out of order. Each pump is run by a friction-clutch pulley, which enables it to be started or stopped in a moment, independently of the other pumps. A small 14 x 15-in. friction-gear hoist is used to convey the slime from the slime pits to the agitation vats. The entire plant is run by a 5-ft. Pelton wheel, making about 115 r.p.m., and operating under a head of 81 ft., using a 4-in. nozzle. Water-power is obtained by means of a 14-in. riveted steel pipe, tapping the main pipe line supplying power to the mill. This 14-in. pipe-line was brought in by mule back, riveted in 10-ft. lengths, although some difficulty was experienced in its

transportation. (Figs. 76 and 77 give plan and section of the slime plant.)

The method of treating the slime is similar to that ordinarily practised by agitation and decantation; it consists in giving a two days' agitation in the agitation vats, with from two to three times the weight of cyanide solution, followed by another two (or sometimes three) days' treatment in the decantation vats; during the latter treatment, the charge, after having been sufficiently agitated with the addition of slaked lime, is allowed to settle as much as practicable, and the clear supernatant liquor is decanted and passed through the zinc-boxes. This operation of agitation, settling and decantation of clear solution is repeated as many times as permissible within the time limit of the treatment, being ordinarily three or four decantations.

The material treated, when dried to 20 per cent. or 25 per cent. moisture, is tough and of the consistency of soft putty. It contains, however, a certain percentage of fine sand and, when viewed in vertical section, presents a somewhat stratified appearance. On long drying, it cracks into layers which are almost absolutely impervious to leaching.

The results of the sizing-test (which are given in Table II) represent an average obtained from the material treated.

TABLE II—SIZING-TEST ON SLIME

Assay value of material was \$4.13 of gold and 20.30 oz. of silver per ton.

| SIZE OF MATERIAL | WEIGHT | ASSAY VALUE | | PERCENTAGE OF TOTAL VALUES CONTAINED | |
|-----------------------------|---------|-------------|--------|--------------------------------------|---------|
| | | GOLD | SILVER | GOLD | SILVER |
| | Per Ct. | | Oz. | Per Ct. | Per Ct. |
| Retained on 80 mesh | 1.1 | \$2.38 | 14.22 | 0.63 | 0.77 |
| “ 100 “ | 2.7 | 2.06 | 13.60 | 1.35 | 1.81 |
| “ 120 “ | 5.6 | 1.96 | 13.02 | 2.66 | 3.59 |
| “ 150 “ | 3.1 | 2.27 | 14.14 | 1.70 | 2.16 |
| “ 200 “ | 2.7 | 2.16 | 13.10 | 1.41 | 1.74 |
| Passed 200 “ | 84.8 | 4.54 | 21.68 | 93.22 | 90.52 |
| Totals | 100.0 | | | 100.92 | 100.59 |

Description of the Plant. — The four agitation vats, constructed of 3-in. redwood throughout, are provided with conical bottoms, slanting at 45 deg. As shown in Fig. 78, each vat has an inside diameter of 15 ft. 7 in.; the vertical depth, from top of side staves

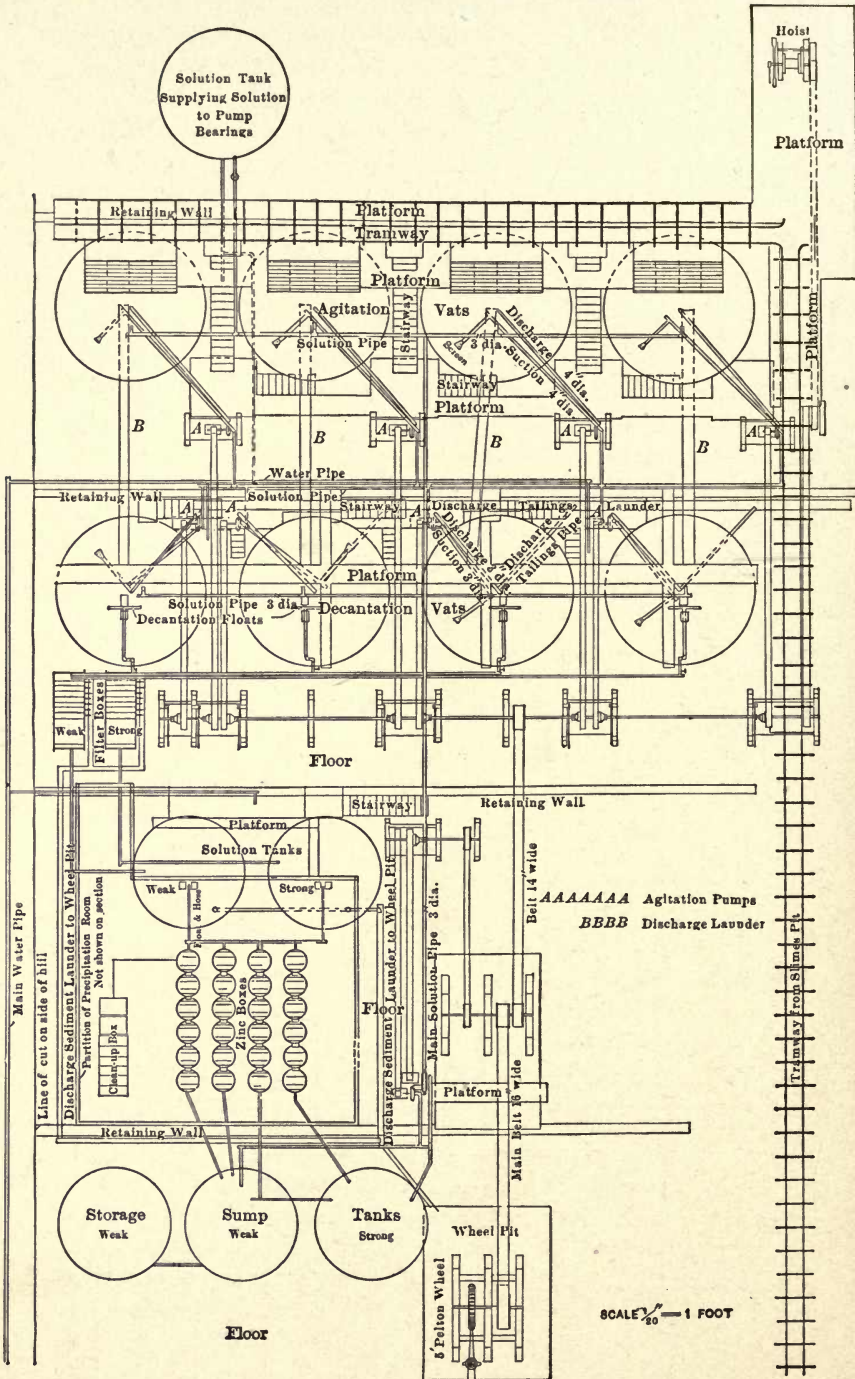
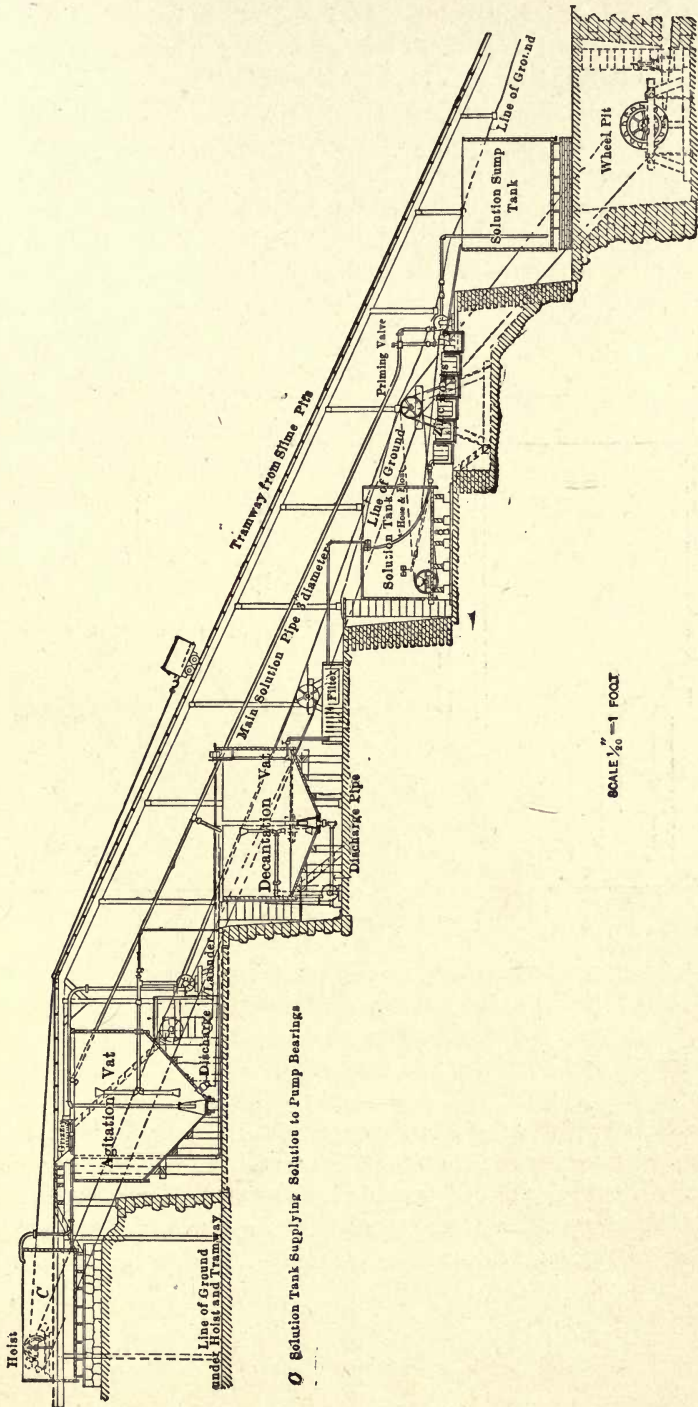


FIG. 76.—PLAN OF SLIME PLANT.



0 Solution Tank Supplying Solution to Pump Bearings

FIG. 77. — SECTION OF SLIME PLANT.

to the iron casting at point of conical bottom, is 15 ft., the inside depth of vertical side staves being 7 ft. 3 in. Each agitation vat is connected with a special manganese-steel lined 4-in. centrifugal pump, which runs at a speed of 900 r.p.m. The pump is connected with the vat by the 4-in. suction pipe *a*, which enters the vat through the side staves about 6 in. above their juncture with the bottom staves and extends nearly to the center of the vat, where it is connected by means of a movable elbow, *b*, with a short piece of 4-in. pipe, *c*, provided at the free end with a good-sized screen or strainer, *d*, made of $\frac{1}{8}$ -in. sheet iron, punched with a number of 1-in. holes; this short piece of pipe, together with

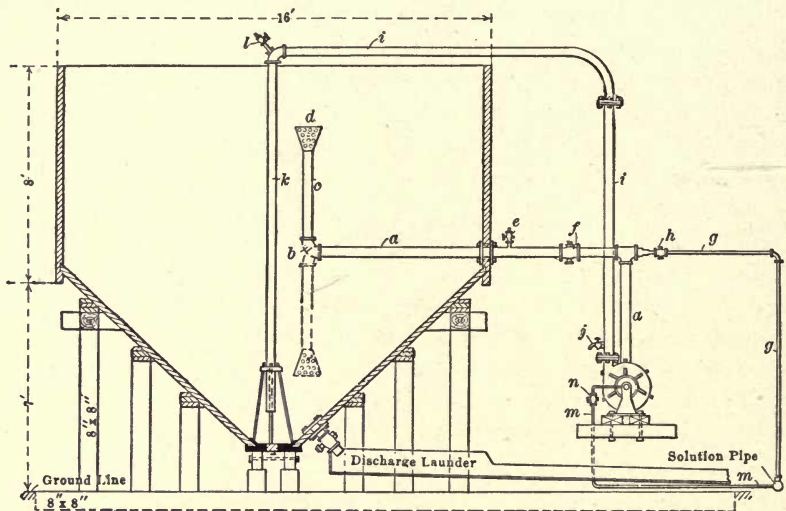


FIG. 78.—AGITATION VAT.

the screen, is of such a length that when being lowered the screen will just clear the bottom staves. The screen is provided with a small iron ring, to which is fastened a piece of rope, by means of which it can be raised and lowered.

Just outside the vat, the suction-pipe is provided with an air-cock, *e*, which admits air to the material going through the pump. This air-cock, however, is rarely used at the present time. The service-cock *f* permits the shutting off of the material from the pump at any time it may become necessary, as, for instance, to repack the stuffing-box or to examine the interior of

the pump. The 2-in pipe-line *g*, provided with the valve *h*, connects with the upper solution tank.

When it becomes necessary to shut the pump down for any length of time (either at the conclusion of the agitation of the charge or at any time during the treatment), the 2-in. valve *h* is opened and the clear solution only passes through the pump. The friction-clutch pulley running the pump is now thrown out of clutch, and after the pump has stopped the valve *h* is closed. By this means is avoided the accumulation in the pump interior of solid matter that would naturally be deposited (when the pump is stopped for any length of time) from the slimy material ordinarily passing through it.

The 4-in. discharge-pipe *i* of the pump is provided with a small bib-nosed petcock, *j*, a few inches from the body of the pump, by means of which samples can readily be taken of the material passing through the pump. The discharge-pipe passes over the top of the vat, and at a point vertically over the center of the bottom casting is provided with an elbow and drop-pipe, *k*, which reaches to within about 15 in. of the bottom casting. This pipe is held firmly in position by means of an iron clamp and four legs made of $\frac{3}{4}$ -in. bolts fastened to the bottom casting and which serve as a support. The distance of the lower end of this discharge-pipe from the bottom of vat is a matter of some importance in the agitation, and a number of experiments made along this line have indicated that the best satisfaction is obtained at a distance of 15 in. from the bottom casting.

Different shapes of nozzles have been tried at the lower end of the drop-pipe, but experience has shown that the plain 4-in. pipe-end gives satisfactory results. The discharge-pipe of the pump tends to act as a siphon when the pump is stopped at any time during the agitation, and would therefore cause inconvenience when repacking the stuffing-box or making any necessary repairs. To prevent this, air is admitted to the pipe by opening the small air-cock *l*, tapped into the elbow at the upper end of the drop-pipe. This air-cock *l* is also frequently used to allow the air entering into the charge to be agitated, it being found for this purpose preferable to the air-cock on the suction-pipe. It might be supposed that when this air-cock is open during the agitation a steady stream of the material passing through the discharge-pipe would be ejected through it; and with regard to the air-cocks

similarly situated on the pump connections of the decantation vats, such is the case. As regards the pumps connected with the agitation vats, however, the effect is found to be quite the reverse, and a rather strong air suction usually occurs when this air-cock is open.

The pump-bearing nearest the pump shell is tapped with a small pipe-line, *m*, provided with the valve *n*, which connects with the upper solution tank. By this means the bearing is supplied with clear solution under pressure, and the wear on the shaft and bearing is greatly reduced. At the commencement of operations clear water was supplied to the pump bearings and was also used for cleaning out the pumps and for priming, when necessary. It was soon found, however, that the quantity of water added in this way increased the volume of stock solution very appreciably, and, of course, an equal quantity of weak cyanide solution had ultimately to run to waste. Not only did this cause an unnecessary mechanical consumption of cyanide, but the quantity of water added through the pump bearings naturally reduced the strength of the working solution in the vat under operation, with a deleterious effect on the percentage of extraction. The quantity of solution that will be added to a vat during the usual period of agitation (from forty to forty-four hours), when the shaft and bearing are a little worn, is surprising, amounting in some cases to 15 tons, even when the greatest care is exercised. The amount of solution added in this way is naturally the least just after the pump has been equipped with a new shaft and new liners, and the bearing rebabbitted. On an average, however, the quantity of solution added to each charge through the pump bearings is from 5 to 6 tons. The agitation pumps in use, while in most respects proving very satisfactory, have nevertheless certain defects in their design, which contribute largely to the rapid wearing of the shaft and the bearing next to the pump shell, and also to the wearing of the interior, renewable manganese-steel wearing parts. The life of these parts naturally varies; but ordinarily it is necessary to equip a pump with a new shaft and with portions of the manganese-steel wearing parts, and to rebabbit the bearing every six weeks. The pumps are equipped with a pulley having a 6-in. face; but it is found preferable to use a 4-in. belt, since this reduces the weight on the pump-shaft with a consequent decrease in its wear, while a 4-in. belt runs the

pump equally as well as a 6-in. one. Wire lacing is used on all the belts.

Each agitation vat was originally provided with a 6-in. discharge opening at the center of the bottom casting. This opening was bushed down to 4 in. and was provided with a nipple and a straightway valve. The first few vats were discharged from the bottom by this means, but a deal of trouble was experienced, due to the fact that, though all the slime entering the agitation vats passed through a grizzly having 1.25-in. openings, yet the bottom valve would frequently become choked with small rocks and other material which seemed to be mixed with the first slime. This bottom discharge was therefore discontinued,

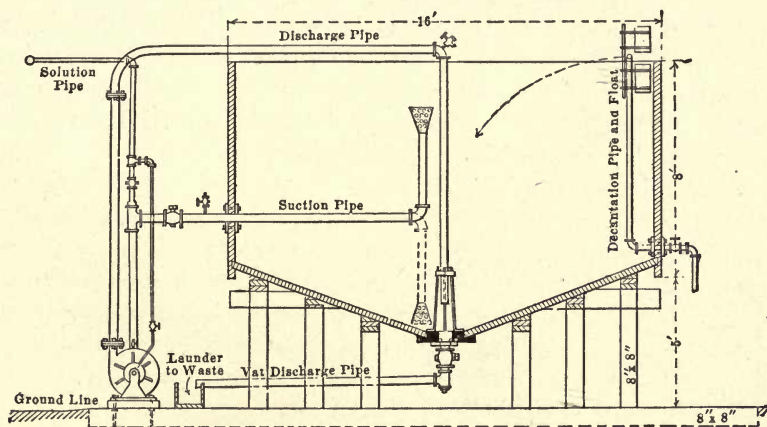


FIG. 79.—DECANTATION VAT

a hole was bored in the bottom staves 10 in. from the bottom casting, and a 3.5-in. iron service-cock was secured to the vat by means of a short nipple and iron flanges. The vats are discharged through this valve into a wooden launder which conveys the material to the corresponding decantation vat. This launder is provided with rows of 6-in. wire nails, which serve to catch any foreign matter.

The four decantation vats, made of 3-in. redwood throughout, are of the same dimensions as the agitation vats, with the exception that they are provided with conical bottoms, slanting at 20 deg. Each one is connected with an ordinary 3-in. centrifugal pump. Fig. 79 shows in detail the connection of the pump

with the vat. The vat is discharged through a 3.5-in. bottom-discharge valve and pipe, into the residue launder, from which the discharged material flows to the river. Removal of the clear solution is effected by means of a 2-in. decantation pipe and float. This pipe enters the side of the vat about 6 in. above the bottom staves and is provided with two loosely threaded elbows, which permit of the free raising and lowering of the portion within the vat. The float proper is made of two ordinary 5-gal. oil cans, soldered water-tight and painted with paraffin paint. The rate of decantation is controlled by means of the 2-in. valve just outside of the vat.

It frequently happens that the solution drawn from the decantation vats is not perfectly clear, and two filter-boxes are provided (see Figs. 76 and 77) for the partial clarification of the solution before it enters the solution tanks at the head of the zinc-boxes. Each compartment of these filter-boxes is provided with a discharge valve, by means of which the sediment deposited from the solution can be washed into a waste launder.

The solution tanks at the head of the zinc-boxes are two in number, one being used for the weak and the other for the strong solution. They are made of 2-in. redwood throughout, and are each 11 ft. 8 in. in diameter and 7 ft. 7 in. deep, inside measurements, having a capacity of 25 tors. Each solution tank is provided with a 2-in. floating hose, by means of which the clearest solution in the tanks is always supplied to the zinc-boxes. A 3-in. valved opening in the bottom of each of these solution tanks permits of the discharge of the accumulated slime into a waste launder.

Fig. 80 shows the timber foundations supporting the decantation vats, the conical bottoms resting on three beveled rings. The supports for the agitation vats are built in the same manner, the supporting rings, however, being placed to line at 45 deg. instead of at 20 deg. Fig. 81 shows the decantation vats in course of erection.

There are four sets of zinc-boxes, each set being composed of six round individual boxes or compartments, each compartment being 28 in. in diameter and 2 ft. deep, and having an available zinc capacity of approximately 5 cu. ft. One of the boxes is used solely for strong solution, and two for weak solution; the fourth being so connected up that either weak or strong solution

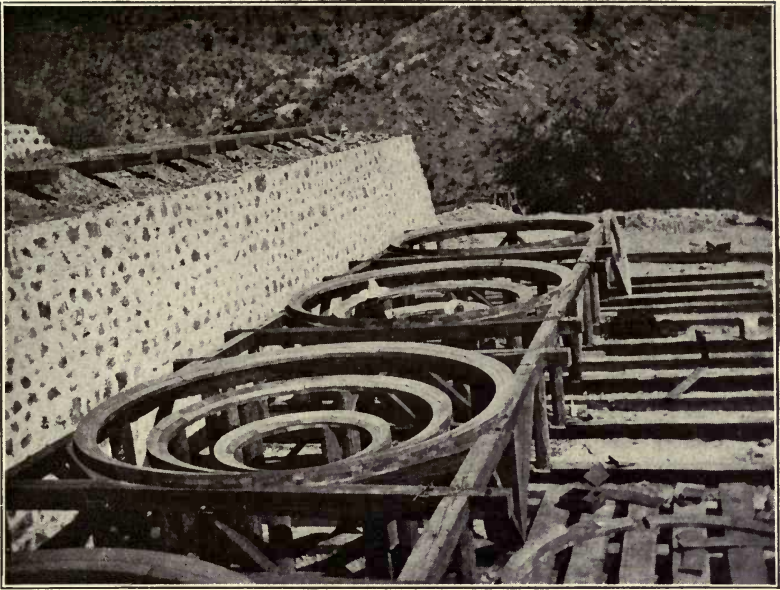


FIG. 80. — Timber Foundations supporting Decantation Vats of Slime Plant.

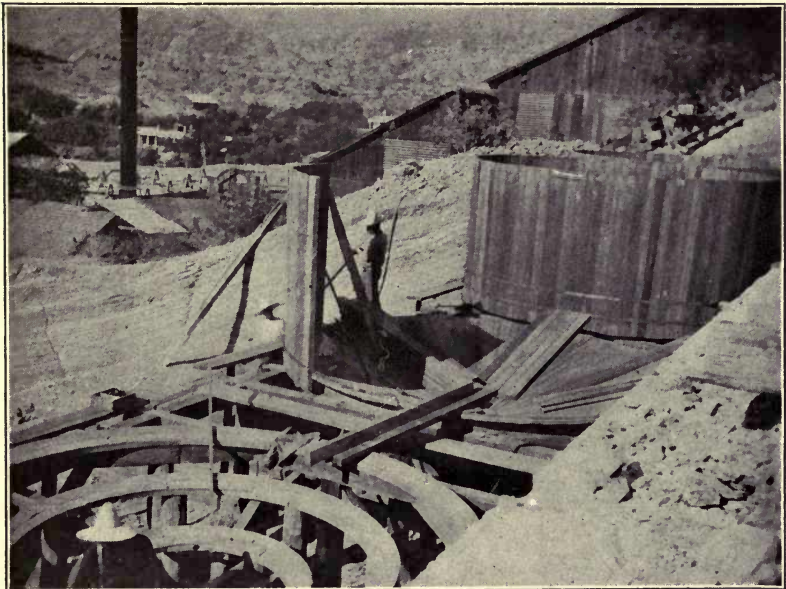


FIG. 81 — Decantation Vats in Course of Construction.

may be run through it. The solution leaving the zinc-boxes passes to three sump-tanks, made of 2-in. redwood throughout, each 11 ft. 8 in. in diameter and 9 ft. 7 in. deep, inside measurements, and of a capacity of 32 tons of solution. Two of these tanks are connected together and serve as a weak-solution sump, the other being used for the strong solution.

Fig. 82 gives a good view of the plant shortly before its completion and shows its general arrangement. Fig. 83 gives a nearer view of three of the agitation vats and shows the tops of two of the decantation vats. The 4-in. centrifugal pump (leading to No. 1 agitation vat) is seen partially connected up. Protruding from the top of the decantation vat, a little below the center of the picture, is seen the end of one of the 2-in. decantation pipes.

Method of Treatment. — The accumulated slime, after having been dried in the slime pits as much as practicable, is conveyed to the agitation vats in ordinary half-ton ore cars by means of a small, friction-gear hoist. Each agitation vat is provided with an iron grizzly (measuring 3 ft. 3 in. x 9 ft., and having 1.25-in. openings), which is suspended over to one side of the center. The content of the car is dumped on to this grizzly and the portion that does not pass of its own weight is trampled, or otherwise forced through, by boys. For some time the material being treated averaged from 20 to 25 per cent. of moisture and in this condition was lumpy and cohesive. During this period the agitation was unsatisfactory and the percentage of extraction was low. Difficulty was experienced in discharging the vats; the unagitated portion of the charge would remain in the pointed bottom of the vat as a tough, putty-like mass, after all the liquid portion had been discharged, and could only be washed out by means of a stream of solution or water under pressure. Experience demonstrates that the best condition of the material is such that, when dumped on the grizzly, it will run through of its own weight. In this state the slime carries from 30 to 35 per cent. moisture. It is desirable that the percentage of moisture contained in the slime when charged shall be as low as possible, compatible with satisfactory agitation; the greater the percentage of moisture contained in the slime, the greater will be the mechanical consumption of cyanide. The complete drying of the slime by some cheap process, followed by powdering before charging into

the agitation vats, should be productive of improved results. A charge equivalent to about 15 tons of dry slime gives more satisfactory results than does a heavier one.

TABLE III.—SETTLING RATE OF SLIME PER HOUR, WITH ADDITION OF LIME

| SETTLEMENT (IN INCHES) OF SLIME | | | | | | | |
|--------------------------------------|------------|------------|------------|------------|----------------|----------------|----------------|
| | Test No. 1 | Test No. 2 | Test No. 3 | Test No. 4 | Test No. 5 (c) | Test No. 6 (c) | Test No. 7 (c) |
| Proportion of solution to slime..... | 2.5:1 | 2.5:1 | 2.5:1 | 2.5:1 | 2.5:1 | 2.5:1 | 3.3:1 |
| Lime added per ton of slime (a)..... | 2 lb. | 3 lb. | 3 lb. | 3 lb. | 3 lb. | None (b) | 4 lb. |
| 1 hour..... | 11.0 | 10.5 | 10.0 | 16.0 | 14.0 | 15.0 | 22.0 |
| 2 hours..... | 21.0 | 19.0 | 16.5 | 25.0 | 21.0 | 24.5 | 36.5 |
| 3 "..... | 27.5 | 26.0 | 23.5 | 33.0 | 30.0 | 33.5 | 51.5 |
| 4 "..... | 33.0 | 32.0 | 30.0 | 40.0 | 39.0 | 40.0 | 54.0 |
| 5 "..... | 36.0 | 35.5 | 36.0 | 42.0 | 43.0 | 42.0 | 57.0 |
| 6 "..... | 38.0 | 38.5 | 40.0 | 43.0 | 47.0 | 43.0 | 58.0 |
| 7 "..... | 39.5 | 40.0 | 41.5 | 44.0 | 48.5 | 44.0 | 59.0 |
| 8 "..... | 40.5 | 41.0 | 42.5 | 44.5 | 48.5 | 44.5 | 59.5 |
| 9 "..... | 41.0 | 41.5 | 43.0 | 45.0 | 49.0 | 44.5 | 59.5 |
| 10 "..... | 41.0 | 42.5 | 43.5 | 45.0 | 49.0 | 45.0 | 59.5 |
| 11 "..... | 41.5 | 42.5 | 43.5 | | | 45.0 | |
| 12 "..... | 41.5 | 43.0 | | | | | |

(a) This quantity of lime was added in addition to the lime already contained in the solution; sufficient lime usually being present in solution that the addition of 5 c.c. of strong lime water to a titration (with silver nitrate), for strength of solution, would make no difference in the titration.

(b) See note (a).

(c) Tests No. 5 and No. 6 were on material from near the head of slime pits, and which therefore contained a larger percentage than usual of fine sand.

(Each 2 in. of solution equals one ton.)

Before commencing to charge the slime, about 35 tons of solution from the strong solution sump (usually of a strength between 0.12 and 0.15 of KCN) is pumped into the vat and the attached centrifugal pump started. From 75 to 100 lb. of slaked lime is added and the charging of the slime is commenced. After the required quantity of slime has been added, a sample of the material passing through the pump is taken, filtered and the clear solution titrated. The necessary quantity of cyanide to bring the solution up to strength is then added. Experiments have been made with various strengths of solution in the agitation vats;

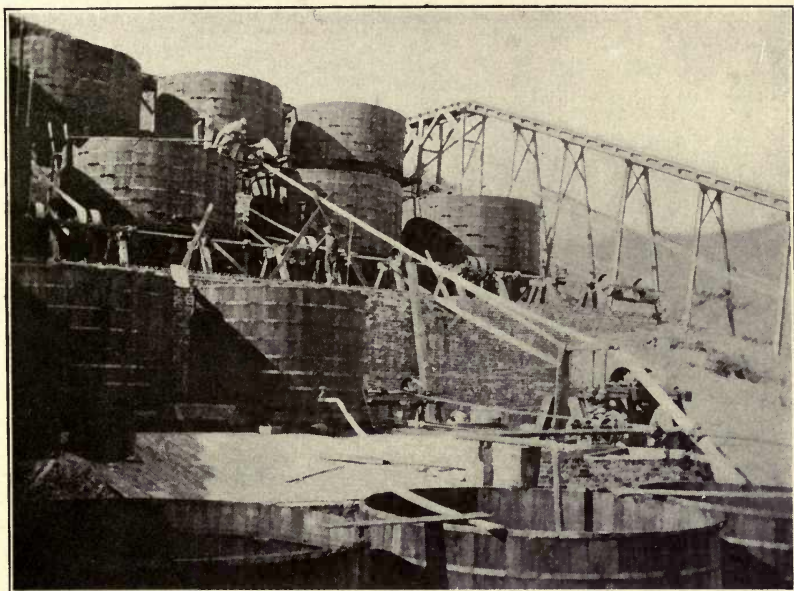


FIG. 82 — General Arrangement of Slime Plant.

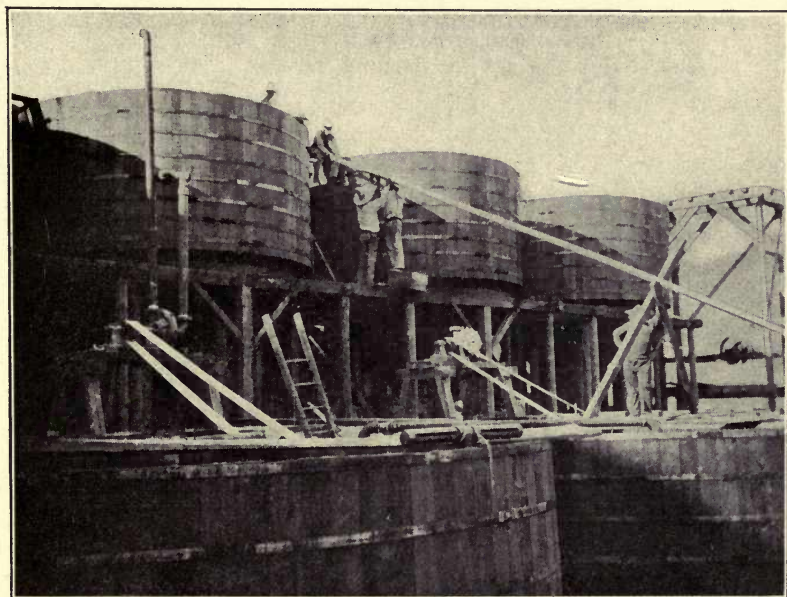


FIG. 83 — Three of the Agitation Vats and Tops of two of the Decantation Vats.

the results thus far show the 0.2 per cent. solution to give more satisfactory results than the use of a weaker solution. The solid cyanide is placed in perforated buckets or cans and suspended in the charge. It is found that unless the receptacles containing the cyanide be frequently agitated about in the charge, the cyanide dissolves exceedingly slowly. The less the proportion of solution to solid matter present, the more noticeable is this tendency of the cyanide to dissolve slowly. It is also noticed that, the thicker the charge, the slower is the action of the cyanide on the silver and gold contained in the slime. During agitation it is best to keep the screen at the end of the suction pipe just as near the surface of the charge as possible, without allowing the entrance of air. By so doing, the material passing through the pump always contains a minimum quantity of solids, and the wear on the pump is consequently lessened. In addition to this, the movement or circulation within the charge is then greatest, since the suction and discharge points are then most separated. It is quite probable that a considerable portion of the heaviest and coarsest part of the material treated does not pass through the pump at all; as, owing to its greater weight, it may never be raised to the height of the suction screen. The agitation of the mass seems to depend chiefly on the fact that the discharge issuing from the drop-pipe tends to keep the point of the conical bottom free from any settled deposit of slime, and the thickened material, constituting the lower portion of the charge, keeps constantly sliding down the inclined sides toward the bottom point. The product issuing from the discharge-pipe, being drawn from the surface of the charge, must pass upward through the entire mass above, before it can again pass through the pump.

The percentage of solid matter contained in the material passing through the agitation pumps is determined from samples taken through the bib-nosed petcock tapping the discharge-pipe a few inches above the pump shell. The pulp passing through the pumps will carry 25 per cent., by weight, of solids.

A thorough oxygenation of the mass is found to be an essential feature; it becomes more necessary as the proportion of solid matter to solution present increases. At the commencement of the operations, the small air-cock *e* (Fig. 78) was used to permit the continuous admittance of air to the suction pipe of the pump. This practice, however, was soon abandoned, because the agita-

TABLE IV. — DETAILED RECORD OF SLIME TREATMENT

Charge No. 29. Vat No. 1. Net weight of slime charged, 18 tons. Moisture in slime as charged, 31.5 per cent. Proportion of solution to slime present, 2 : 1. 100 lb. lime added at commencement of charging. Assay value of slime per ton as charged was \$4.34 of gold and 19.22 oz. of silver.

TREATMENT IN AGITATION VAT

| DATE AND TIME | AGITATION | ASSAY OF SOLUTION | | VALUES EXTRACTED BY SOLUTION | | ASSAY OF TAILING | | EXTRACTED AS PER SOLUTION ASSAYS | | EXTRACTED AS PER TAILING ASSAYS | | STRENGTH OF SOLUTION IN KCN SAMPLE TAKEN AT | | REMARKS |
|---------------|-----------|-------------------|--------|------------------------------|--------|------------------|--------|----------------------------------|-----------|---------------------------------|-----------|---------------------------------------------|-----------|-----------------------|
| | | GOLD | SILVER | GOLD | SILVER | GOLD | SILVER | GOLD | SILVER | GOLD | SILVER | VAT (a) | PUMP (a) | |
| Sept. 29. | Hours | | Oz. | | Oz. | | Oz. | | Per Cent. | | Per Cent. | | Per Cent. | |
| 4 p. m. | | \$0.95 | 0.85 | | | | | | | | | 0.11 | | Commenced charging, |
| 4 p. m. | | 0.90 | 0.96 | | | | | | | | | 0.40 | | 7 a. m. Finished 4 |
| 6 p. m. | 2 | 1.05 | 1.01 | \$2.10 | 2.02 | | | 48.4 | 10.5 | | | 0.40 | | p. m. Added 90 lb. |
| 6 p. m. | 2 | 1.15 | 1.45 | 2.30 | 2.90 | \$2.06 | 16.10 | 53.0 | 15.1 | 52.5 | 16.2 | 0.30 | | of cyanide (NaCN = |
| 10 p. m. | 6 | 1.20 | 2.13 | 2.40 | 4.26 | | | 55.3 | 22.2 | | | 0.30 | | 125 per cent. of |
| 10 p. m. | 6 | 1.30 | 2.42 | 2.60 | 4.84 | 1.44 | 14.66 | 59.9 | 25.2 | 66.8 | 23.7 | 0.22 | | KCN) at 4 p. m. |
| Sept. 30. | | | | | | | | | | | | | | |
| 2 a. m. | 10 | 1.50 | 3.35 | 3.00 | 6.70 | 1.24 | 13.20 | 69.1 | 34.8 | 71.4 | 31.3 | 0.23 | | |
| 2 a. m. | 10 | 1.55 | 3.73 | 3.10 | 7.46 | 1.14 | 11.78 | 71.4 | 38.8 | 73.7 | 38.7 | 0.23 | | |
| 8 a. m. | 16 | 1.60 | 3.99 | 3.20 | 7.98 | 1.03 | 10.98 | 73.7 | 41.4 | 76.3 | 42.9 | 0.19 | | |
| 8 a. m. | 16 | | 4.28 | | 8.56 | 1.03 | 10.68 | | 44.5 | 76.3 | 44.4 | | | |
| 2 p. m. | 22 | 1.65 | 4.65 | 3.30 | 9.30 | | | 76.0 | 48.4 | | | 0.22 | | Added 30 lb. of lime. |
| 2 p. m. | 22 | 1.60 | 4.75 | 3.20 | 9.50 | 0.92 | 10.14 | 73.7 | 49.4 | 78.8 | 47.2 | 0.22 | | |
| 8 p. m. | 28 | 1.70 | 4.76 | 3.40 | 9.52 | 0.82 | 10.34 | 78.3 | 49.5 | 81.1 | 46.2 | 0.19 | | |
| 8 p. m. | 28 | 1.65 | 4.67 | 3.30 | 9.34 | 0.82 | 10.22 | 76.0 | 48.6 | 81.1 | 46.8 | 0.18 | | |
| 12 p. m. | 32 | 1.70 | 4.83 | 3.40 | 9.66 | 0.82 | 10.26 | 78.3 | 50.3 | 81.1 | 46.6 | 0.18 | | |
| 12 p. m. | 32 | 1.80 | 4.80 | 3.60 | 9.60 | 0.92 | 10.04 | 82.9 | 49.9 | 78.8 | 47.7 | 0.18 | | |
| Oct. 1. | | | | | | | | | | | | | | |
| 4 a. m. | 33 | 1.70 | 4.94 | 3.40 | 9.83 | 0.82 | 9.85 | 78.3 | 51.4 | 81.1 | 48.7 | 0.17 | | Commenced to dis- |
| 4 a. m. | 36 | 1.75 | 4.81 | 3.50 | 9.62 | 0.82 | 9.70 | 80.7 | 50.1 | 81.1 | 49.5 | 0.17 | | charge into decan- |

(a) Vat sample taken from surface of charge, in vat. Pump sample taken from discharge-pipe, near pump.

tion was seriously affected by it. The entrance of air into the suction pipe had a detrimental influence on the capacity of the pump, and the effect was found to be injurious to the best agitation. Perhaps the chief trouble was due to the rapid rising to the surface of the imprisoned air immediately on being expelled from the discharge-pipe. The air bubbles breaking, on reaching the surface of the charge, caused a splendid surface movement that might be easily mistaken for the thorough agitation of the entire mass without effecting a proper scouring of the bottom point of the vat. The present practice is to allow the entrance of a smaller quantity of air into the mass, through the small air-cock *l* (Fig. 78).

Ordinarily a charge is agitated in the vats from forty to forty-four hours, after which it is discharged into the corresponding decantation vat, where it is usually given a two days' treatment. Should the charge from the agitation vat not fill the decantation vat, enough precipitated solution is pumped up from the strong solution sump to fill it; after agitation for half an hour, the charge is allowed to settle. Should the addition of this extra solution be unnecessary, the charge is not agitated, but allowed to settle as long as practicable, the clear supernatant solution being meanwhile decanted off. After the first settling and decantation, the vat is pumped full of weak, precipitated solution, which is usually of a strength approximating 0.1 per cent. of KCN per ton, and the charge is agitated for an hour or two by means of the 3-in. centrifugal pump connected with the vat, about 25 lb. of slaked lime being added during the agitation. The pump is then stopped and an additional quantity of slaked lime, usually about 10 lb., is sprinkled evenly over the top of the charge. After settling a few hours, the decantation pipe is lowered and the settling and decanting of clear solution continued as long as practicable. As many washes and decantations as possible within the time limit of the treatment are given in this manner. When permissible, the last wash given is of clear water, though a few of the charges have to be washed entirely with weak solution.

When treating charges containing the equivalent of 15 tons of dry slime, usually four settlings and four decantations can be effected with the forty-eight hours of treatment, each decantation averaging about 22 tons of solution; hence about 90 tons of solution is decanted in treating a 15 ton charge, and each decantation removes approximately 58 per cent. of the total solution

TABLE V. — TREATMENT IN DECANTATION VAT

| DATE | TIME | QUANTITY | KCN | SOLUTION DECANTED | | | | REMARKS |
|--------------|------------|----------|---------|-------------------|----------|-------------------|--------|---------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------------|
| | | | | ASSAY-VALUES | | TOTAL ASSAY-VALUE | | |
| | | | | GOLD | SILVER | GOLD | SILVER | |
| Oct. 1 . . . | 5 a. m. | Tons | Per Ct. | | Oz. | | Oz. | Finished receiving charge from agitation vat. 20 lb. lime added while being charged. Let settle for 3 hours. Commenced decanting. Finished decanting. Added 18 tons 0.10 per cent. KCN solution and 40 lb. lime. Agi- tated for 2 hours. Let settle for 6 hours. Commenced decanting. Finished decanting. Added 18.5 tons of 0.11 per cent. KCN solution and 40 lb. lime. Agitated for 2 hours. Let settle for 4 hours. Commenced decanting. Finished decanting. Added 19 tons of 0.09 per cent. KCN solution and 40 lb. lime. Agitated for 2 hours. Let settle for 5.5 hrs. Commenced decanting. Finished decanting. Discharged vat. |
| Oct. 1 . . . | 8 a. m. | } | 0.18 | \$1.70 (a) | 4.84 (a) | \$30.60 | 87.12 | |
| Oct. 1 . . . | 3 p. m. | | | | | | | |
| Oct. 1 . . . | 11 p. m. | } | 0.14 | 0.91 | 2.56 | 16.83 | 47.36 | |
| Oct. 2 . . . | 4 a. m. | | | | | | | |
| Oct. 2 . . . | 10 a. m. | } | 0.11 | 0.48 | 1.44 | 9.12 | 27.36 | |
| Oct. 2 . . . | 3.30 p. m. | | | | | | | |
| Oct. 2 . . . | 11 p. m. | } | 0.09 | 0.09 | 0.88 | 5.40 | 15.84 | |
| Oct. 3 . . . | 3 a. m. | | | | | | | |
| Totals . . . | | 73.5 | | | | \$61.95 | 177.68 | |

(a) In all these decanted solution samples, the value of the solution, as added to charge, has been deducted.

NOTE. — Assay of discharged residue (unwashed): \$1.03 of gold, 9.52 oz. of silver. Extraction, 76.27 per cent. of gold and 50.47 per cent. of silver. Assay of discharged residue (washed): \$0.62 of gold, 8.38 oz. of silver. Moisture contained in discharged residue, 51.4 per cent. Assay value of solution was \$0.40 of gold and 1.24 oz. of silver. Strength, 0.09 per cent. of KCN. Extraction indicated by value in decanted solutions: 79.30 per cent. of gold and 51.36 per cent. of silver. Total time of treatment: In agitation vats, 45 hours; in decantation vat, forty-eight hours.

present. Assuming the wash-agitation to be perfect, the four decantations should then theoretically contain about 97 per cent. of the total value dissolved at the time the washing was commenced. The settled pulp is discharged through the bottom valve and the 4-in. discharge-pipe into the waste launder.

Table III shows the rate of settling per hour, determined at various times on several different charges.

The pulp, ready for discharging, carries 50 per cent. of moisture, the contained solution averaging 0.07 per cent. of KCN, and having an average value of approximately \$0.40 of gold and 1.50 oz. of silver per ton. These values are higher than would be expected to remain in the solution after the several decantations and dilutions effected; yet (as has already been recorded by several different parties operating similar slime plants) the solution of value from the slime does not cease at the completion of the agitation proper, but continues throughout the washing; the value of the wash-solution is thus being constantly augmented. This feature, however, is more noticeable with the silver than with the gold, and the maximum extraction of the gold is obtained earlier. For these reasons, the solution contained in the discharged pulp will always carry more values than it should according to calculation based solely upon successive dilution and assuming the agitation to be perfect.

A portion of the sample of the pulp ready for discharging (together with its proper proportion of contained solution) is dried, the assay results being taken to represent the value of the discharged slime. Another portion of the pulp is washed and then assayed. On an average the washed sample will run about \$0.40 of gold and from 1 to 2 oz. of silver per ton lower than the unwashed sample.

The 3-in centrifugal pumps connected with the decantation vats are the ordinary pumps commonly used for pumping solutions; the only alterations being that the bearing nearest the pump shell is tapped with a $\frac{1}{4}$ -in. pipe, which supplies the bearing with solution under pressure. These pumps run about four hours in each twenty-four, and have given excellent satisfaction, the only repair work being an occasional repacking of some of the stuffing-boxes.

Tables IV and V, giving a somewhat detailed record of the treatment of one charge, may be taken to represent the usual

practice. The usual charge is now but 15 tons of slime (dry weight), while the proportion, by weight, of solution to slime has been increased to 2.5 : 1.

Precipitation. — All solution leaving the decantation vats is passed through the zinc-boxes before being reused. The zinc-boxes have to be watched closely; owing to the excess of lime present in the solution, difficulty is experienced in obtaining good precipitation. Records are kept of the quantity of solution daily passing through the boxes, together with the assay values of the solution before and after precipitation. These records show that, during the last three months, an average of practically 48 tons of strong and 117 tons of weak solution, or a total of 165 tons, was passed through the boxes daily; the average assays of the solution were as given herewith:

TABLE VI

| | STRONG SOLUTION | | WEAK SOLUTION | |
|--------------------------|-----------------|--------|---------------|--------|
| | GOLD | SILVER | GOLD | SILVER |
| | | Oz. | | Oz. |
| Entering zinc-boxes..... | \$1.05 | 2.90 | \$0.60 | 1.70 |
| Leaving zinc-boxes..... | 0.10 | 0.40 | 0.10 | 0.35 |

The zinc-boxes have a combined total shavings capacity of approximately 120 cu. ft.; the rate of flow of the solution through the boxes during 1904 averaged 1.37 ton per cu. ft. of shavings per twenty-four hours.

The highest-grade precipitate yet recovered from the slime plant assayed approximately \$6800 gold and 17,300 oz. of silver per ton.

Tonnage, Percentages, etc. — The normal capacity of the plant, while treating 15-ton charges, and allowing a two days' treatment in both agitation and decantation vats, is 30 tons per day. During the last quarter of 1904 approximately 2550 tons of slime (net dry weight) was treated, and the extraction during this period (shown by the differences between assays of the charge and of the residue) was 74.9 per cent. of the gold and 49.2 per cent. of the silver. During this period the assay value of the slime averaged \$4.35 of gold and 19.25 oz. of silver. During the last two months (March and April, 1905) 3.56 lb. of sodium cyanide (equivalent to

4.40 lb. of potassium cyanide) was used per ton of slime treated. The average extraction of silver for the last three months has been 51 per cent. The consumption of cyanide, zinc and slime per ton of dried slime treated during this time was: Sodium cyanide, 4.42 lb.; zinc, 0.957 lb., and lime, 13.95 lb. The sodium-cyanide consumption is equivalent to 5.52 lb. of potassium cyanide.

Table VII gives the operating costs per ton of slime treated.

TABLE VII. — SLIME COSTS PER TON

| | |
|----------------------------------------------|---------|
| Cyanide (4.42 lb. @ \$0.63) | \$2.785 |
| Zinc (0.957 lb. @ \$0.30) | 0.287 |
| Lime (13.95 lb. @ \$0.0118) | 0.165 |
| Other supplies | 0.238 |
| Lubricating | 0.033 |
| Labor | 0.491 |
| Salaries | 0.748 |
| Assay office (labor and supplies) | 0.066 |
| Power (ditch maintenance and supplies) | 0.621 |
| Miscellaneous (lighting, etc.) | 0.002 |
| Management and general expenses | 0.179 |
| Total ¹ | \$5.615 |

CYANIDING AURIFEROUS SILVER ORES AT SAN SALVADOR, C. A.

The following interesting notes on the working of this process in San Salvador were communicated to the *Engineering and Mining Journal*, June 1, 1905, by Alfred Chiddey:

The process was first applied to tailings from the pan amalgamation in which the ore was first roasted with salt in the ordinary way. After all these tailings had been worked, the process was applied to raw ore, with results so successful that roasting and amalgamation are now discontinued, and much poorer ore can be treated. The silver occurs presumably as sulphide and the ore contains a little copper, which apparently helps the extraction; at any rate, it seems to do no harm.

The mode of working is as follows: The ground ore, after leaving the arrastras (which are fitted with 30-mesh screens), is passed over amalgamated plates and allowed to settle in masonry

¹ \$5.615 Mexican currency during this period was equivalent to \$2.66 gold. The cost of realization on the precipitate is not included in the working cost: these expenses are high. The average cost of realization on the precipitate produced in the slime plant, per ton of dry slime, is: Government taxes, \$0.856; treatment charges (including transportation expenses), \$1.202; total, \$2.058.

tanks, where a rough classification is effected into sand and slimes. The sand is charged over gratings with 25 lb. lime per ton into the percolation vats. A cyanide solution of 0.40 per cent. strength is introduced, equivalent to nearly one-half the weight of the sand. After standing twelve hours, the cock is opened and the vat allowed to drain, the solution passing to the zinc-boxes. On the following day the surface is leveled off and raked over. The charge is then allowed to stand for six days exposed to the air, without adding more solution. On the expiration of six days, a 0.20 per cent. solution is added, and the leaching is continued rapidly without intermission for four days, at the end of which time a water-wash is employed.

The first wash that comes off, after the vat has been standing dry for six days, often contains from 12 to 20 oz. silver per ton, which will run down the second day to 6 oz., on the third day to 3 oz., on the fourth day to $1\frac{1}{2}$ oz.; the water-wash will generally be from 0.5 to 1 oz. per ton. The sand contains from 13 to 15 oz. silver and from \$3 to \$5 gold per ton before treatment. The extraction for the past 18 months has averaged from 85 to 90 per cent. of the silver and 90 to 92 per cent. of the gold.

The consumption of cyanide is a little under two lb. per ton. Lately sodium cyanide has been used with good results. It is a little cheaper, but much more deliquescent than potassium cyanide, and during the rainy season a box has to be used up immediately it is opened.

The ore contains argentite, chalcopyrite, sometimes specks of fahlerz (tetrahedrite), but the amount of copper is small, generally under one-tenth per cent. The sump solutions always contain copper, but presumably as sulphocyanate. The precipitation of gold and silver in the zinc-boxes is practically perfect.

The slime is air-dried and mixed with the sand in proportion of 1 : 2, but the time of treatment is prolonged to 18 days instead of 12, the time required for the sand. The air-dried slime is previously put through a disintegrator. This method of treating the slime is only temporary. Although the extraction is satisfactory the time of treatment is too prolonged, and the method is available only during the dry season.

The following are results on a few charges of sand:

| No. OF CHARGE | BEFORE LEACHING | | TAILINGS AFTER LEACHING | |
|---------------|-----------------|----------|-------------------------|----------|
| | SILVER Oz. | GOLD Oz. | SILVER Oz. | GOLD Oz. |
| 44 | 20.17 | 0.37 | 2.52 | 0.06 |
| 45 | 15.50 | 0.30 | 0.95 | 0.05 |
| 46 | 14.16 | 0.30 | 1.57 | 0.03 |
| 47 | 16.50 | 0.36 | 1.87 | 0.35(?) |
| 48 | 13.10 | 0.30 | 1.44 | 0.03 |
| 49 | 16.00 | 0.30 | 1.37 | 0.03 |
| 50 | 14.70 | 0.30 | 1.37 | 0.03 |
| 55 | 15.50 | 0.26 | 1.50 | 0.02 |
| 56 | 13.70 | 0.23 | 1.00 | 0.02 |
| 57 | 13.90 | 0.21 | 1.00 | 0.02 |

TREATMENT OF ROASTED ORE

As stated above, complex silver ores heavily charged with metal sulphides are not suitable to be treated raw with a cyanide solution on account of the decomposing action of these sulphides on potassium cyanide. To make this process possible, such ores have first to be roasted with salt to reduce the consumption of cyanide. The roasting has to be done as carefully as for the sodium hyposulphite process, in order to secure a minimum loss of silver by volatilization. This is accomplished by roasting at a very low heat without even raising the temperature toward the end, so that as few of the metal chlorides are expelled as possible. These metal chlorides, if expelled by heat, are the sole cause of the loss of silver by volatilization (silver chloride as such not being volatile), and it would be folly to employ this means to remove them from the ore. It is much more rational to remove them by leaching with water, though the resulting base-metal solution will have to undergo a treatment similar to that practised in the lixiviation with sodium hyposulphite, to recover the silver dissolved therein. Having the ore prepared for extraction as far as that, it seems to be more rational to extract the silver chloride and the gold subchloride first with sodium hyposulphite, and then, after the latter has been replaced in the ore by water, to apply the solution of potassium or sodium cyanide to extract the remainder of the gold and that part of the silver which was not chloridized. This procedure is advisable and much improved extraction can be thus expected, for the following reasons:

It will make the application of the cyanide solution possible

to a much larger variety of ores than if the same is applied directly after roasting and washing of the ore. Not all the metal salts formed in chloridizing roasting are soluble in water; some will remain in the ore. like cuprous chloride, lead sulphate and others, and will act decomposingly on the cyanide solution. Most of such salts, however, dissolve in a solution of sodium hyposulphite, and are removed from the ore to a great extent during silver leaching, so that the cyanide solution will thus be applied to a much cleaner ore, which may make possible the treatment of ores which contain a larger percentage of copper, lead and other metals. The action of the cyanide solution will be more energetic on those particles which escaped chlorination, and suffer less by decomposition, if the main bulk of the silver and other metals has been removed. This is also the case with regard to the gold. It will be found that, in the case of a silver ore rich enough in gold to show after roasting with salt a perceptible amount of gold when concentrated in a horn spoon, the gold assumes a much brighter color if this is done with a hyposulphite solution. In treating rich silver-gold concentrates, as related in Chapter XXII, I experienced the fact that by first roasting with salt, then extracting the gold by Plattner's method, and finally the silver with sodium hyposulphite, only 50 per cent. of the gold could be extracted, while by leaching the silver first and then applying the Plattner method, the gold extraction was as high as 95 per cent. There is no reason to doubt that, from an auriferous silver ore which was first chloridized, the cyanide solution will extract the gold more quickly and more completely if the ore be first leached with sodium hyposulphite.

The increase in cost will be but very slight, as it will not involve any handling of the charge, but will require only a continuation of the leaching with another solvent. The total time may even prove to be shorter than if only a cyanide solution was used.

It seems that very high extractions of silver and gold may be obtained by a combination of the two processes, and thorough experimenting on that line will in all probability be well rewarded. This refers, of course, only to higher grade ores that can stand the cost of roasting, and the nature of which resists a high chlorination.

John F. Allan, City of Mexico, in a very interesting paper on "Cyanide Treatment of Silver Ores in Mexico," read before the

Atlantic City meeting, February, 1904, of the American Institute of Mining Engineers, gives two examples of cyaniding silver ores containing gold which were roasted with salt before leaching:

“Leaching: Example No. 1. — The plant in question has a capacity of 700 tons per month. The chloridized pulp is charged into filters 18 ft. in diameter and 4 ft. deep, and receives as preliminary treatment four hours' soaking with water and seven hours' percolation. This water-wash is passed through special zinc-boxes, where an impure precipitate containing gold and silver is formed, and a consumption of about 0.4 lb. of zinc per ton takes place. When the salt and impurities have been removed by the wash-water, the charge receives fifteen hours' soaking and ninety hours' percolation with cyanide solution of 0.3 per cent. or 6 lb. to the ton of water, the ore receiving an equivalent of 1.5 of solution to 1 of ore, or 0.45 per cent. To displace the strong solution, twelve hours of percolation with a 0.05 per cent. weak solution is given, in the proportion of 1 of water to 2 of ore, and a final thirty hours' percolation with water-wash. The consumption of cyanide is 2.15 lb. per ton, and zinc 1.08 lb.

“The extractions are: Gold, 76; silver, 85.25; total value, 81.77 per cent. It will be observed that the gold extraction is not as good as the silver, a fact often noted in chloridized ores. The following are the costs, which can be considered as typical in small plants, although in some instances they have been reduced: Superintendence, \$0.62; labor, \$0.96; cyanide, \$1.71; zinc, \$0.29; laboratory, \$0.10; various stores, \$0.50; total, \$4.18 Mex., or, at 50c. U. S. per \$1 Mex., \$2.09 U. S. per ton.”

“Leaching: Example No. 2. — The plant is capable of treating 1800 tons per month. The treatment, with small variations, is practically the same as in Example No. 1. The consumption of cyanide is 1.73 lb. and of zinc 1.27 lb. per ton, the high consumption of zinc being due to the ore averaging over 30 oz. of silver per ton. The extractions are: Gold, 82.30; silver, 77.32; total value, 79 per cent. The costs are: Superintendence, \$0.33; labor, \$0.31; cyanide, \$0.99; zinc, \$0.33; laboratory, \$0.10; various stores, \$0.22; total, \$2.28 Mex., or, at 50c. U. S. per \$1 Mex., \$1.14 U. S. per ton.”

It is to be regretted that no analysis or description of the ore is given. The fact that in Example No. 1 the gold extraction is only 76, and the silver 85.25 per cent., and Mr. Allan's remark

that the fact of a lower gold extraction is often noted in chloridized ores, verify my experience in treating auriferous silver ores. If to the roasted ore of Example No. 1 a solution of sodium hyposulphite had been applied before the cyanide solution, the extraction of both metals would have been higher, and very likely that of gold better than the extraction of the silver.

TESTING THE CYANIDE SOLUTION FOR GOLD AND SILVER

In order to conduct the cyanide process intelligently frequent tests of the solutions for gold and silver are necessary.

*Arents' Test.*¹ — This test is based upon the fact that metallic copper will precipitate gold and silver upon its surface from acid solution. Of course the fact is not new, but its application is probably so. Arents has used the method with success; it recommends itself by the rapidity and ease with which it may be carried on.

An auriferous cyanide solution, if made acid with sulphuric acid and boiled with finely divided, pulverulent, metallic copper, will, within a short time, deposit its gold content on the copper. Any silver in the solution is also precipitated. If this mixture is now filtered, the filter and contents may at once be subjected to a crucible assay treatment, and its lead button cupeled and determinated.

If, instead of taking cement copper, or any metallic copper powder, a solution of bluestone is used after acidification, and a few small pieces of sheet aluminum are added, and the solution boiled until all the copper has come down, the result as to the precipitation of gold and silver is the same. This modification takes more time and attention in boiling. If aluminum has been used, it should go into the crucible with the filter and its contents. Commercial cement copper is particularly fitted for this test, because the acid, in taking up any basic iron or copper salts of the cement copper, renders the copper as finely divided as it is customary to obtain in the sluice-boxes of copper leachers. The finer and the more pulverulent the copper is, the greater is its surface and the more energetic the precipitation, thus permitting a minimum amount of copper to be used.

In applying the method, Arents uses, as a rule, 250 c.c. of the

¹ From a paper by Albert Arents read before the Albany meeting, Feb. 1903, of the American Institute of Mining Engineers.

solution to be tested and a few c.c. of sulphuric acid, agitates for several seconds, and then adds not less (although not much more) than one gram of cement copper. Now follows heating to boiling. This is kept up for about 10 minutes, so that the rising steam-bubbles keep the mixture well agitated. The mixture is then filtered through a 7-in. diameter gray filter-paper. No washing is done. As soon as the filtering is finished, one-third of a crucible charge of flux is added to the filter containing all the sediment of the mixture. Some of the moisture is rapidly absorbed by the flux, which permits the folding of the filter's rim upon the charge and its subsequent removal without loss or tearing. One-third of a crucible charge of flux having previously been placed upon the bottom of the crucible which is to be used for melting, the filter is transferred to the crucible, well tucked down, and the last third of the crucible charge is placed on top of the filter in the crucible. It is then ready for the furnace. The filter itself furnishes the reducing agent for the assay. Arents uses 30 grams litharge and the usual amount of borax and soda, employing a No. F crucible for melting. About 20 grams of lead are obtained. The lead button comes out bright and clean, and upon cupeling furnishes a bead free from copper.

Possibly this method of testing for gold and silver may be used upon other solutions than cyanide; also, for solutions from testing metallic copper for precious metals, when the solutions do not contain nitric acid in any form.

Alfred Chiddey's Test.—Four assay tons of solution are taken in a porcelain dish, and 20 c.c. of a 10 per cent. solution of lead acetate added; then 4 grams of zinc shaving and afterward 20 c.c. of hydrochloric acid. When the action has nearly ceased, the contents of the dish are boiled for a minute or two and filtered. The precipitate is well washed with water, moistened with alcohol, dried, wrapped with the filter in lead foil, and cupeled.

The testing of the strength of a solution in potassium cyanide is usually done by titrating the same with a standard solution of silver nitrate. This standard solution is prepared by dissolving 13.04 grams of c.p. silver nitrate in one liter of distilled water. Every 0.1 c.c. of such a solution added to 10 c.c. cyanide solution is equivalent to 0.01 per cent. of potassium cyanide. The operations are as follows:

A Mohr's burette, graduated to 0.1 c.c., is filled with the

standard silver solution. Ten c.c. of the cyanide solution which is to be tested is taken up with a pipette and emptied into a small flask which is brought under the burette. Then drop by drop the silver solution is added. Each drop produces a whitish precipitate, which, however, disappears again when the flask is well shaken. This is continued until the last drop produces a cloudiness or turbidity, which remains even after a vigorous shaking of the flask.

The scale on the burette is now read. Each tenth of a c.c. silver solution used indicates 0.01 per cent. of potassium cyanide contained in the solution.

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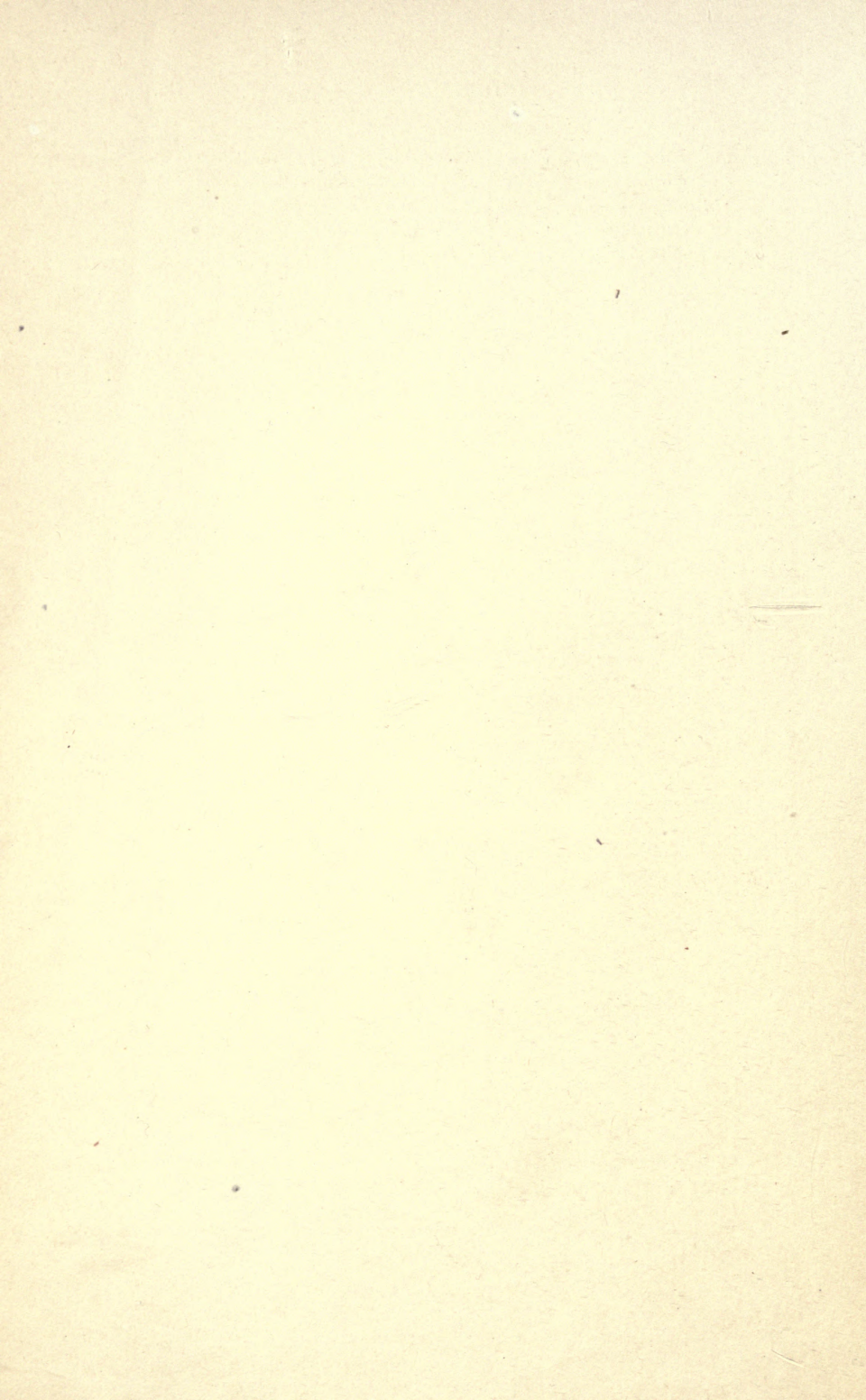
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