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THE

CHLORINATION PROCESS.

BY

E. B. WILSON, E.M.

FIRST EDITION.

FIRST THOUSAND.

NEW YORK:
JOHN WILEY & SONS.
LONDON: CHAPMAN & HALL. LIMITED.
1907.

ROBERT DRUMMOND, ELECTROTYPER AND PRINTER, NEW YORK.

INTRODUCTION.

THE treatment of refractory gold-bearing ores is of interest to those who are or have been engaged in mining the precious metal.

Some wise person has stated "that all is not gold that glitters"; and if he were alive and a miner he could have added two other facts which history has established, viz.:

That all gold-bearing rocks do not contain gold in paying quantities; also, that some gold-bearing rocks contain considerable quantities of gold, but are commercially valueless.

The greater part of the money sunk in gold-mining ventures has been due to the above facts; but should the statement be discredited, inquiry of those who have lost money in the past will substantiate it, we believe.

The chlorination method of treating refractory ores has been some years before the public, and while old yet it is new.

It has added much to the world's store of gold, and is destined to add more as it becomes generally practised, as it has assumed a place in metallurgy from which it cannot be dislodged.

The process is not one of great difficulty, although it has been belittled by those who, claiming to be mining experts, were unable to practise it, and who therefore proposed either a less economical plan of treatment, or brought financial loss upon those who followed their advice. The writer takes occasion here to express his obligations to those whose data he has included in this volume, and trusts he has given them proper credit.

The author recognizes that owners of mines as well as students and those practically engaged are interested in this subject. He has endeavored for that reason to place the subject in such a form that any one who reads can understand. The chemical formulæ are of little value except to those who are in practice or intend to practise; their omission would, however, detract considerably from the value of the book for reference and completion of the subject.

The author trusts his endeavors will meet with the approval of those interested.

E. B. WILSON.

JUNE, 1897.

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CONTENTS.

THE CHLORINATION PROCESS.

CHAPTER I.

LIXIVIATION BY CHLORINE SOLUTIONS.

LIXIVIATION is the term applied to the abstraction by a liquid of the soluble part of a mineral, or aggregation of minerals. An aggregation of minerals is a rock.

Lixiviation is therefore the abstraction by a liquid of the soluble part of a rock.

The useful processes for separating gold from ores by lixiviation are limited chemically by the few known solvents for it, and commercially by the cost required to effect profit. We are therefore at present confined to the following processes:

Lixiviation by hyposulphite soda solutions, or the Russell Process, for silver first;

Cyanogen solutions;

Bromide solutions;

Bromide-cyanogen solutions;

Chlorine solutions.

They are all based upon the solubility of gold in aqueous solutions of the chemicals named. Lixiviation by chlorine solutions is what we are particularly to deal with. Rose, in his "Metallurgy of Gold," gives as the relative dissolving powers of the last three chemicals in 1-per-cent solutions, with each solution at the temperature of 60° Centigrade, the following:

Chlorine	in	1¼	hours	dissolves	4.49% of gold.
Bromine	"	1¼	"	"	6.46% " "
Cyanide	"	1¼	"	"	0.57% " "

All these wet processes require that the ore be subjected to preliminary treatment before using the solutions. With chlorine solutions preliminary treatment is elaborated more than in the bromine or cyanide solutions.

All four processes seem to occupy a distinctive field of their own, but also encroach upon each other's territory. In such cases the choice of the process should depend upon which will extract the greatest percentage of gold at the least cost.

Could we say that any one of the processes was the best, this choice would be an easy matter; but as each ore differs in character, it is not possible to say that any one of the above processes is the best under all circumstances; and then the choice of process becomes difficult, and can only be decided by the metallurgist after experiment.

Chlorination (by which we mean the leaching of gold ores by chlorine solutions) does not save the silver content of the ore, because during preliminary treatment the ore is subjected to chloridizing roasting, which forms an insoluble compound in water, known as Silver Chloride.

The reverse is the case with gold, which forms in chloridizing roasting a gold chloride soluble in water. But other metals may do the same; consequently it becomes necessary to remove the other metals by oxidation, or by oxidizing roasting.

Under certain conditions gold unites with chlorine, forming the gold chloride known as trichloride of gold, or auric chloride, where one triad atom of gold unites with three monad atoms of chlorine, forming a molecule of auric chloride whose molecular weight is 302.31, and whose chemical symbol is $AuCl_3$. The chief source of chlorine is common salt ($NaCl$), which contains about 60 per cent of this substance. As a gas it has a greenish yellow color and a very disagreeable odor, producing on inhalation a suffocating cough. (The cough may be relieved by breathing ammonia or ether.)

Cold water absorbs about twice its volume of chlorine gas, being converted gradually into hydrochloric acid (HCl) by the chlorine uniting with the hydrogen of the water.

Slaked lime, or calcic hydrate ($CaO + H_2O$), when exposed to chlorine gas forms a chloride of calcium and hypochlorite of calcium, or what is known as bleaching-powder, with formula $CaCl_2 + CaCl_2O_2$.

Commercial bleaching-powder contains from 20 to 35 per cent of available chlorine. It forms a homogeneous white powder, possessing a smell of hypochlorous acid, gradually becomes moist on exposure to the air and decomposes with absorption of water and carbonic acid. It should therefore be kept away from the atmosphere. Bleaching-powder derives its chief value from the hypochlorite of lime which it contains.

Hypochlorous acid is so weak an acid that its salts are easily decomposed. Carbonic acid gas decomposes it readily. The salts of hypochlorous acid are unstable compounds, the same as the acid; and the calcium hypochlorite gives the bleaching-powder its chief value both for bleaching and the chlorination process, from the fact it yields its chlorine readily. When either hydrochloric or sulphuric acid are added to bleaching-powder a quantity of chlorine equal to the quantity in the hypochlorite is evolved. The reaction is as follows:

$$\text{I.}\begin{cases}2HCl + CaCl_2O_2 = 2HOCl + CaCl_2;\\ 2HCl + 2HOCl = 2H_2O + 2Cl_2.\end{cases}$$

$$\text{II.}\begin{cases}CaCl_2+CaCl_2O_2+H_2SO_4=2CaSO_4+2HCl + Cl_2O_2;\\ 2HCl + Cl_2O_2 = 2H_2O + 2Cl_2.\end{cases}$$

In the first case half the chlorine is obtained from hypochlorite and half from the hydrochloric acid.

In the second case the sulphuric acid decomposes the chloride and hypochlorite, liberating all the chlorine in both compounds.

It may be possible to obtain liquid chlorine in a concentrated state, thus reducing the bulk compared with bleaching-powder. To dissolve gold by chlorine, the latter must be in a free or nascent state, that is, as a gas, or in a liquid state, uncombined with other chemicals, as chlorine water. The gold must be in a metallic state, and the chlorine will then combine with it if oxygen be present. Whenever metallic gold is dissolved in nitro-muriatic acid ($HNO_3 + HCl$) chlorine is liberated in the presence of oxygen, and forms with gold the compound $HAuCl_4 + 2H_2O$, known as chlo-auric acid, a very peculiar combination, the reaction of which is as follows:

$$4HCl + CaCl_2O_2 + Au + HNO_3 = \\ HAuCl_4 + CaCl_2 + NO_3 + 2H_2O.$$

The deep yellow solution obtained gives upon evaporation yellow crystals of the double chloride of gold and hydrogen ($HAuCl_4$), and this cautiously heated solidifies to a red crystalline mass, soluble in water, alcohol, or ether, and is auric chloride, $AuCl_3$. The dissolving of gold from ores by the above process would not be feasible on account of dissolving other impurities as well, together with the subsequent difficulties and expense in separating the gold from the base impurities, and it is merely mentioned as an example of the active principle of chlorine in its attack upon gold when it can be liberated from its compounds.

It can be liberated from its compounds as readily as in the above example. The practical results, however, depend upon its liberation in contact with gold, in aqueous solutions, free from impurities, whereby its usefulness may not be impaired, and from which the auric chloride formed can be precipitated.

Plattner in 1856 proposed what is now known as the Plattner system of chlorination, which he accomplished successfully by the following steps:

1. He subjected the ores to a roasting process for the purpose of oxidation, and driving out of the ore those substances which would be acted upon by free chlorine.

2. He leached the roasted ores with water, the ores having been previously saturated with chlorine gas.

3. He precipitated after filtration the auric chloride thus formed in the second step by means of ferrous sulphate.

The foundation for the chlorination process of to-day was thus built, and whatever improvements have been made are along the lines adopted by Plattner.

It is often necessary, in treating sulphides containing iron and other base metals, to chloridize and roast as well. This is especially necessary with such ores as contain much sulphur, arsenic, antimony, or other such volatile compounds which should be displaced, and which simple oxidizing roasting will not accomplish.

For chloridizing roasting, common salt ($NaCl$) is mixed with the ore (as explained under the heading Roasting, p. 20), converting those substances not volatilized into soluble and insoluble chlorides.

Chlorine has a remarkable tendency to act on metals and the oxides of metals, and so long as volatile substances are combined with a metal during roasting very little chlorine escapes; but after these are displaced by heat and chlorine the chlorine itself escapes—and this is peculiar, since chlorine is very volatile.

Due observations made relative to the amount of volatile matter in an ore, and the affinity between chlorine and that substance, allows us to advance one step nearer the recovery of gold by chloridizing roasting than by oxidizing roasting.

In the present state of the art we cannot treat ores by wet processes as they come directly from the earth; or could ores be subjected to treatment without roasting for chlorinating, they would be more readily treated by some cheaper process; consequently preliminary treatment is a necessity. Again, the chief object of chlorination is to recover gold from refractory ores where cheaper processes will not. We therefore find chlorination practised upon sulphides, tellurides, arsenides, and other similar ores whose combination of substances can be broken by oxygen or chlorine with the assistance of heat. As these minerals form but a small proportion of the ore, they are concentrated to reduce the bulk before treatment, and thus save waste in time, fuel, chemicals, and capital. Concentration may follow amalgamation unless the ore be too refractory, but in either case crushing of some description is preliminary to concentration and subsequent chlorination. The foregoing brief description of the object sought will allow the reader to follow intelligently the process, which should proceed as follows:

1. Preparation of the ore.
2. Roasting.

3. Roasting appliances.
4. The process.
5. Filtering.
6. Precipitating.
7. Refining.
8. The cost of treatment.
9. The plant required.

Under the above headings may be introduced details not of minor importance, for the attention to details has perfected the process to its present condition of usefulness.

Precious-metal mining has reached the stage where capital invested judiciously will as surely bring returns as any manufacturing business where equal care is exercised. The difficulties formerly encountered are to a very great extent overcome, and chlorination is but one of the improvements in that direction. We do not mean to convey the idea that all difficulties are circumvented, nor do we mean to convey the idea that lixiviation is able to do more than assist in recovery of gold, and we particularly advise parties intending to enter into gold-mining to beware of low-grade milling propositions which carry less than $10 per ton of gold where chlorination is to be practised.

While low-grade bodies of ore are more uniform and continuous than "bonanzas," there is a limit to their "low-gradeness," especially if refractory. That one low-grade mining company with a large mill is able to treat its ore at a good profit is no criterion that another large mill in a different locality can treat the ore in that locality at all. We advise in all mining enterprises the consultation of trained mining engineers, practically, scientifically, and technically educated.

CHAPTER II.

PREPARATION OF THE ORE.

WERE the whole mass of ore to be treated by chlorination as it came from the mine, it would be economy to crush the ore dry preliminary to roasting. The treatment of such large masses of ore would require an immense outlay of capital for chlorinating purposes, together with increased size of the plant in general. For instance, if but one tenth of the ore carried mineral, the mechanical arrangement would require to be ten times as large as where the mineral had been concentrated; larger supplies of chemicals and fuel would also be needed. There are rare instances where clean silicious ores could be treated in this manner, but where they occur once, the probabilities are they do not occur again in ten thousand instances.

The ore, that is, the vein rock containing mineral, is mined, cobbed, assorted, loaded into suitable conveyances and transported to the mill, as the first step in preparation. In some instances the vein rock will not show mineral, but it is there; and in such instances the rock must be treated with the mineral streak, provided it carries gold. Vein rock between the hanging and foot walls very often is totally barren, and again will carry more precious metals than the mineral streak proper. To avoid wear and tear upon the machinery in the first instance, and to avoid loss in the second, judicious assays should be made and recorded of all vein matter broken in the mine.

When the ore reaches the mill, it is unloaded over a chute which has an inclination of 40° and upwards. This chute has for its floor in some part of its length a number of iron bars, separated by spaces, to allow the finer ore as it comes from the mine to pass through to the rolls, while the coarser passes over the bars into the rock-crushers, and from the latter to the rolls. These screen-bars are termed "grizzlies," and assist the crushers very much by their separating material already fine enough for the rolls, which would otherwise interfere with the amount of ore crushed in the rock-breakers.

The most suitable rock-crushers are of the Blake and Gates type.

The former has been longer in use, so that by some it is considered to be the only crusher of moment. The largest size of the Blake we believe is No. 20, with an opening to receive rocks 13 × 30 inches and under, which it can crush to sizes of 1½ to 1¾ inches in diameter. The capacity given is 10 tons per hour, which it only reaches under most favorable circumstances. A fair average of its capacity would probably be 8 tons per hour.

The Gates crusher with the same horse-power can crush double this amount in the same time. Against the Gates crusher is its weight, and while it crushes more it wears more.

For fine crushing, two small crushers are better than one large crusher, as setting the jaws to crush fine hinders the amount of product which can pass through. However, in small machines the opening or mouth for the reception of the rock is smaller, and would require finer ore, possibly sledging, in order to feed; consequently a large crusher, assisted by two smaller ones to receive its product, would do the most economical work.

Mr. Blake puts the limit of economy in the use of multiple-jaw crushers at No. 10 screen (100 holes to the square inch). If rolls are to be used, and then stamps, there is not so much saving in crushing fine by rock-breakers as would appear, as all ore must be thoroughly dried; and if 10-mesh screened ore were admitted to the stamp battery the coarse sand would simply pack, and the agitation necessary to keep the ore stirred up in the mortar would be lacking. For the above reasons multiple-jaw crushers for fine crushing can be dispensed with in most cases where rolls and stamps are used, and the jaws of the crusher set to size of ore which will work well in the rolls. It is advisable to use crushers whenever possible to reduce the ore for the rolls, say to 1½ inches diameter, and have the rolls crush this to, say, ¼ inch diameter before admitting to the stamps. In this way the product of the stamps may be increased.

The rolls mentioned are cylinders with steel tires turning towards each other, the ore passing between them. They are set close, with heavy springs or swinging pillow-blocks so arranged as to give if necessary to allow large pieces of rock or a coupling-pin or piece of drill steel from the mines which has found its way into the ore to pass through without breaking the rolls. They are also in some cases fitted with magnets to attract any iron or steel which would dent the faces of the rolls.

It was formerly customary to gear these rolls, but that added to the wear and tear; they are now almost universally run by pulleys and belts. If the ore which passes the rolls is not sufficiently crushed, it is screened and returned to the rolls for recrushing with an additional supply from the crushers.

It is at times feasible to crush fine enough with crushers and rolls to chlorinate without stamp-milling.

For this purpose, however, we must have an easily pulverized rock, and return all the product which does not pass the desired screen. When the rock is hard this method is destructive to screens, wearing them out quickly and requiring that the ore be free from moisture to pass the screen. For fine crushing the most satisfactory arrangement is the stamp-mill, especially if amalgamation is to be practised before chlorination.

The action of these crushing-machines is mashing, and they do not round the particles of ore as do certain classes of pulverizers which pulverize by abrasion. The latter class are not suitable for gold or silver milling, as they do not crack the grains of ore in a proper manner; however, when roasting is to precede the process this matter is not of such moment as for amalgamation or cyaniding. The Huntington and Chilian mills are also excellent crushers in place of stamps.

If free silver be present in the ores, chloridizing roasting followed by amalgamation may be practised, or the ore treated first to recover the gold by chlorination and afterwards by hyposulphite of soda to recover the silver, or the reverse; but there must be in each case silver values greater than gold. Roasted gold ore does not amalgamate readily, while silver gives good results. Another consideration is that there will be a loss of gold if that be not treated first as it is generally customary to add chemicals t brighten the mercury, which becomes sickened when roasted ore is amalgamated.

We can therefore consider all silver lost when chlorination is practised or a good portion of the gol lost where silver is recovered first.

The ends in view will determine the crusher to b used. Coarse crushing may be possible, and whil objectionable for good roasts and clean concentrates has advantages over fine crushing, which is alway attended with considerable fine ore and slimes ver inconvenient in filtering or drainage to wet processes With chlorination, however, this drawback is somewhat removed by concentration of the ore, and th subsequent roasting which follows, changes the shap of the mineral particles from compact to porous thus facilitating drainage and leaching.

The crushing process is followed by concentration that is, a separating of the mineral from its gangue c vein rock not containing mineral. These concentrates are usually sulphurets of some description which make the ore refractory, and thus necessitat chlorination; in fact, the usefulness of the process hinges upon this class of ores.

To concentrate the tailings from the crushers the are conducted to jigs, where they are washed fre from slimes. The sands and lighter particles are carried from the jigs to vanners, buddles, bumping-tables, or other similar arrangements. These machines, by the aid of water, separate the lighter particles of sand from the heavier particles of mineral: the former are washed away, while the latter are collected. It may not be necessary to jig in all instances, but it is better to do so where large quantities of sand might otherwise go to the tables, and there interfere with the work. When fine crushing is practised from 40 to 60 mesh screen (i.e., 1600 or 3600 holes to the square inch respectively), from 30 to 60 per cent of the product is slimes, and unless these are separated from the sands and concentrated separately, they will be lost. To accomplish this the ore is separated into fine and coarse, each carrying values. This product is now carried to jigs, which, having more even sizes to deal with, are able to treat with a greater degree of certainty. The surplus products of the jigs then go to the tables—the coarse to one, the fine to another. This order may be somewhat varied, but it is absolutely necessary for chlorination that the concentrates should be clean, to give the best extraction results and the least loss of values in concentration. Slimes will adhere to the sands unless sufficient water is used to separate them; and as they usually are rich in precious metal, if the appliances mentioned do not answer, they should b run into settling-tanks.

The cost of the preparation of ore is against th use of chlorination at times, but the main choice o a process is, other matters being equal, the value saved.

If we take, for example, an ore carrying 2 oz. gold ($40), with 16 oz. of silver ($9.60), we may save 95 per cent of the gold and lose all the silver: in money this loss is $12.08. If by cyaniding we recover 90 per cent of both values, the loss amounts to $4.96; our choice would therefore naturally be the use of the cyanide process. If now we take a $40 gold ore with no silver value, the loss by chlorination with 95 per cent recovery would be $2 against $4 by the cyanide recovery of 90 per cent. The choice then narrows down to the question; can chlorination be practised as cheaply as cyaniding upon a given ore? No direct answer can be given, since chlorination has more preparation of ore to contend with, and if in our example the margin of $2 is consumed by the extra manipulation of the ore, cyaniding would be the best. (See Cost of Treatment.) There is a limit where the difference in favor of cyaniding upon strictly gold ore occurs, which is that point where the extra expenses of chlorination are counterbalanced by increased extraction. For instance, in an $80 gold ore with 95 per cent extraction there is a difference of $4 in favor of chlorination, which is amply sufficient to favor its use in preference to cyanide extraction in most instances.

CHAPTER III.

ROASTING THE ORE.

HAVING obtained the mineral in the ore as concentrates, by the preparation noted in the preceding chapter, our attention must be directed to freeing that mineral from the base metal compounds which it contains. To accomplish this we have recourse to oxidization, which in our case is the application of heat with oxygen present in sufficient quantities to unite with the base metals, forming oxides of metals which volatilize.

Where air only is requisite for the purpose, the roasting is termed "oxidizing roasting," but where other substances must be employed to assist decomposition, such as common salt, the operation is termed "chloridizing roasting."

The chlorine from the salt (NaCl) seems to answer a twofold purpose, since it assists not only oxidation, but forms combinations with metals which volatilize or else become insoluble.[*] To roast thoroughly, the

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[*] See p. 7.

ore should not be subjected to a high heat at the commencement of the operation: in fact at no time should it fuse or melt, but the heat may be increased towards the close of the roast.

If we allow fusion in roasting we cannot attain our object, for it is next to impossible to separate some substances, such as iron and sulphur, which have become united by heat, and no substances which have fused are as susceptible to oxidization as before, while its action or destruction of the leaching compounds are nearly as great as when they were in the raw state.

Certain substances admit of oxidization and will volatilize to a certain point, after which it becomes almost impossible to oxidize them more; at this stage chlorine assists on account of its great affinity for metals, and still further reduces the amount of the remaining impurities.

Yet with the assistance of chlorine it is not possible to free some metals from impurities such as iron sulphides. The greater quantity of sulphur is readily volatilized, but the last 2 per cent of sulphur from an ore containing above 10 per cent sulphur will require as much labor, care, and expense, if not more, than the first 8 per cent.

When sulphur has been removed to the lowest limit, which is about one quarter of one per cent, the ore is said to be "roasted dead," since it is in such a condition that we cannot further reduce it without great expense, and is practically in such a form its injury to the leaching process can be tolerated.

The affinity of sulphur for other substances than oxygen modifies the process of roasting somewhat, as is seen where chlorine in the presence of hot silver is reduced to silver chloride, or where, when arsenic is to be evaporated, carbon is added to the mixture, producing a combination more easily evaporated than the oxide.

Chloridizing roasting deals chiefly with sulphur compounds, with the object in view of freeing the ore from that compound; no doubt there will be met in practice other difficult substances to be eliminated. It must be borne in mind that roasting is not melting, and that the latter must be avoided if it be the object to accomplish a good roast, and high extraction by chlorination. The following substances are of frequent occurrence, consequently their action in roasting is worth consideration.

Iron cannot by any means be freed entirely from sulphur. Roasting will reduce it to about 8 per cent, chloridizing roasting to about 0.25 per cent, at which point elimination ceases.

Sulphide of zinc (zinc blende) is slow to oxidize, and is not purified of all its sulphur. It further volatilizes freely, and carries off the precious metals associated with it when exposed to moderate heat.

Copper sulphides can be readily freed from their sulphur by slow roasting at a moderate heat, with or without salt.

Tellurides act in a manner similar to zinc.

The sulphide of bismuth, being easily fused, is difficult to oxidize, since it melts at a low temperature.

Galena is nearly as difficult to oxidize as bismuth on account of its fusing at low temperature.

Mercury and silver are readily liberated from ore by heat. The former, however, volatilizes at a low heat, and requires great care when roasting to avoid this.

Mercury used to collect gold and silver as amalgam is retorted by volatilization and the fumes condensed in a cooled vessel without much loss of quicksilver.

Silver is readily converted into silver chloride by roasting with salt. This salt is insoluble in water, but dissolves in solutions of hyposulphite of soda. The Russell process is based upon this fact.

Sulphides of antimony are difficult to oxidize, because extremely fusible.

The sulphides of nickel are easily oxidized, forming pure oxides. The same applies to cobalt.

Arsenic sulphides are readily oxidized, both the arsenious and sulphurous acids forming volatile compounds.

Phosphorus cannot be freed from iron by roasting; neither can titanic acid.

Arsenic is also difficult to remove from iron.

Fine ore roasts quicker and better than coarse; but ore not sized, that is, mixtures of coarse and fine ore, offer uneven surfaces to the action of heat and oxygen, one getting in the other's way, so to speak. To assist in exposing the various surfaces of the ore to the reagents, they are stirred or rabbled, either by hand as in plain reverberatory furnaces, or by fixed rabbles as in moving hearth-furnaces, or by movable rabbles as in the Spence furnace, or by falling from shelves as in revolving furnaces. This rabbling also performs another office: it avoids fusing, sintering, or melting of the ore during roasting by its agitation.

The roasting ore, after moisture has been driven entirely from it by heat, becomes like so much quicksand, flowing rather than hanging together in a pasty mass.

It is not always necessary that ore should be extremely fine: in some instances such fine ore would be a hindrance rather than a help, since it would cause more labor in preparation, and greater care in roasting and also in filtering. The principle to be governed by, is to crush only to that coarseness which will roast readily, and yield the greatest per cent of gold to chlorine. The filtering is a secondary matter, for when once the auric chloride has been obtained in solution the gold may be recovered from it, even if time is long drawn out in the process. As noted under preparation of ore (p. 16), ores undergo a complete change in physical structure when roasted; they are also liable to be converted into non-refractory ores; at any rate, they are not refractory to chlorine solutions. Mr. Daggett says: "The leaching of roasted ores is quicker done for any process than the leaching of raw ores, unless the soluble salts formed in roasting have received a prior leaching, when it is about the same as for raw ores."

This is a measure true in chlorination, where sometimes difficulty in filtering occurs when fine slimes prevent the passage of the liquor. In other leaching the ore packs to such an extent that it is next to impossible to drain at all, not more than a quarter of an inch per hour being recorded with raw ore, while with ores which do not form a clayey compact mass impervious to water filtration is comparatively rapid.

Where the base metals present in the ore are in excess of the volatile substances, it may be policy to hasten oxidation by adding small quantities of iron pyrites, and allow the sulphur driven off to combine with base metals before adding salt for chloridizing roast.

Formerly it was thought necessary to have a certain percentage of sulphur in the charge for its effect in chloridizing. This, Mr. Brückner thinks, is a mistake. In his judgment the iron pyrites is added for volatilizing the arsenic and antimony, and for preventing the formation of arseniates and antimoniates of silver and gold which resist chlorination; very little sulphur, or, in its absence, quartz, is sufficient to evolve, when in contact with a very small percentage of salt, the chlorine necessary for chloridizing all the silver contained in the charge, after the arsenic and antimony have been driven off. Thus an addition of from 1 to 10 per cent of iron pyrites is in most cases required for the success of the roasting process. Ores which are apt to cake at a low temperature are mixed with 10 to 20 per cent of rich tailings. Ores containing lime and alumina have to be mixed with silicious ores so that silica shall be in excess. (Dr. R. W. Raymond, A.I.M.E., 1885.)

Ores containing considerable sulphur in them must have care to keep down the heat which the burning sulphur creates, and salt should not be added at this stage, since the heat of the burning sulphur may be sufficient to volatilize the chlorine and cause a loss of gold as well. When salt is added at the commencement of a roast in the reverberatory furnace a loss of gold occurs, but not to so great an extent, and sometimes not at all if the salt is added on the hot hearth or end of the roast with not more than two per cent of sulphur present.

At the end of the roast soluble chlorides are formed as readily as when mixed with the ore at the commencement.

Prof. Christy of the University of California ascertained that sulphides roasted without salt lost little of their precious metals, but that with 3 per cent salt added the loss in one instance was 30 per cent of the gold and 50 per cent of the silver. This very great loss he attributes to high temperature and the telluride in the ore.

Plattner from his experiments states " that loss of silver increases with the temperature in roasting, with the looseness or porosity of the ore, and the freedom with which silver combines with other substances." " The loss also increases with time of roasting." He used artificial mixtures for his roasting tests to ascertain the volatility of gold, such as arsenides and sulphides. " His conclusions were, that a loss of gold can take place only in oxidizing roasting, when the operation is carried on so rapidly that fine particles are carried off mechanically."

Prof. Christy and others coincide with him in his deductions with regard to oxidizing roasting, but add that " he does not seem to have been aware of the volatility of gold in chloridizing roasting." Kustel is said to have recorded a loss of 20 per cent of gold in oxidizing roasting of certain tellurides of gold and silver, attributing the loss to the volatilization of tellurium; and says: "If salt is present during roasting, the chloride of tellurium volatilizes, and it is possible that this volatilization causes gold to volatilize as well." Mr. C. H. Aaron, Prof. Christy says, was the first to publish anything definite on the losses of gold in chloridizing roasting. He pushed the roasting purposely with and without salt; the salted ore upon assay was found to contain but one half as much gold as the unsalted.

Mr. C. A. Stetefeldt gives an account of his investigations with chloridizing roasting, and found the losses to be from 42.8 to 93 per cent of the total gold. He also states " that volatilization of gold no doubt takes place with copper chlorides," but adds, as Mr. Butters' experiments show, " that copper chlorides are not essential to produce this loss." "Mr. C. Butters found that with gold ore free from copper no loss occurred in oxidizing roasting, but when chloridizing roasting was undertaken in a muffle with 5 per cent salt, there was a loss of from 68 to 85 per cent of the gold." " He found also that an increased quantity of salt up to 10 per cent caused no increased loss of gold."

Prof. Christy has shown by a large number of experiments, the results of which are published in his admirable paper contributed to the American Institute of Mining Engineers,* that muffle tests with salt gave a higher percentage of gold loss when the salt was added at the end of the roast rather than at the commencement. This being the reverse of practical experience in continuous roasting, he accounts for it as follows: "Where batch roasts are made and the whole lot of ore is kept at the same temperature throughout (as in a muffle), when the gold chloride has once formed and left the batch of ore, that is the last of it. At the end of a roast more gold being exposed, the addition of salt produces a greater loss of gold."

"In practice with continuous roasting, if the salt is mixed throughout the ore from the start, a continuous volatilization goes on and likewise loss, where, on the other hand, if salt be added at the finish, or hot end of the furnace, the long surface of unsalted cooler ore condenses the chloride of gold." "Part of this cooler ore is giving off sulphurous acid gas, and this with steam from the burning fuel offers excellent

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* Vol. XVII. p. 43.

means for the reduction of chloride of gold in the furnace; but the most efficient means is in the pyrites themselves."

The factors entering into "salt roasting" are salt, temperature, and time.

The amount of salt necessary for a chloridizing roast will depend upon the amount of oxidation required, and this must be governed by the ore. The less salt used the better, but enough must be used to drive off the sulphur and form silver or base metal chlorides; the leaching which follows will then give a high percentage of recovery.

Aaron says that salt should be below 4 per cent. Butters, Stedefelt, and Prof. Christy, and the deductions of others from practical work, bear out Mr. Aaron's statement.

Four per cent of salt is equivalent to 80 lbs. to the ton of ore. That amount of salt is probably more than required for driving off impurities, and what cannot unite with impurities present will unite with gold to form gold chloride; but if more than sufficient be present, chlorine, being readily volatile, will escape with loss of precious metal. A few tests with the ore will determine the percentage of salt most advisable for roasting with an ore.

Temperature has been mentioned previously as of importance in chloridizing roasting; we therefore refer to it again. It is necessary in continuous roasting that at first slow heat and low temperature should take place; as the ore moves forward towards the fire the heat increases from the burning sulphurous acid gas; when nearer to the fire, the sulphur fumes having ceased, the heat is increased. Had the temperature been as great at the commencement as at the finish, the ore would have fused and matted; but the volatile and combustible gases having been driven off at low red heat, there is not much danger of fusion on the hot hearth at a cherry-red heat. The loss of gold from volatilization with salt commences at 100° C. and increases until it reaches its maximum at 250° C.

Below red heat the loss diminishes, but increases to a maximum above melting heat.

Prof. Christy gives the following losses of gold in a stream of chlorine gas with different temperatures:

At incipient redness the standard loss was 0.05 per cent in half an hour.

At a low red heat it is 0.10 per cent.

At a cherry-red heat it is 0.25 to 0.35 per cent.

At incipient yellow it is 0.40 per cent.

At melting heat it is 0.50 + per cent.

These results show the proper regulation of temperature to be important in chloridizing roasting; also, that the proper place for salt is at the end or hot hearth of the roast; furthermore, that at that end an over supply of salt is not beneficial, since the chloride of sodium with other impurities might produce fusion by making a flux.

It has been ascertained that loss of gold may occur from volatilization when a high heat is maintained for a long time if salt be mixed with the ore. From the foregoing it may be surmised that increased temperature does not assist roasting with salt, and consequently time cannot be gained except at the expense of gold, and afterwards leaching.

Time, again, as a factor in chloridizing roasting depends upon the amount of sulphur or other volatile impurities. While roasting, as stated, can be forced up to a certain limit by mechanical contrivances and automatic furnaces, "dead roasting" cannot materially be hastened.

The amount of impurities may be so small and so readily reduced, that mere contact with heat will allow reduction, but generally time and patience are required.

In one instance we have ore oxidized from 30 to 7 per cent of sulphur by falling slowly through the furnace; in another instance we find it requires 32 hours to "roast dead" from a 30% sulphur ore, and at times it may require even more than this.

In chlorination so much depends upon the roasting that no specified time can be stated that will be applicable to every ore. By hastening the operation we run chances of ruining the roast by melting; and also loss of the metal we desire to save, when tellurides, arsenides, and zinc sulphides are volatilized. The Austin process of pyritic smelting, which is practised under certain circumstances to utilize the heat of the burning sulphur and melt the metal to a matte, fully illustrates the heat which may be evolved from pyritic ores, and which in our case must be avoided. The whole process of chlorination depends upon the care taken to obtain a uniform and complete roast.

The deductions we may make as far as regards pyritic ore roasting are:

1. Sulphur can be removed in part by heat, more fully by a plentiful supply of air with heat, and practically eliminated by salt and heat.

2. Sulphates can be decomposed by air, heat, and chlorine, combined or separate; but sulphides must be converted into sulphates by heat with air, and finally with heat, air, and chlorine gas, in our case.

3. Fusion or sintering prevents further desulphurization or formation of sulphates, therefore oxidation or chloridizing roasting.

4. Loss of gold is not apt to occur in oxidizing roasting where care is used, while with salt added to the ore at the commencement instead of the end of

the roast it is considerable, even when great care is taken.

Prof. Christy found by experiment that the loss in the latter instance was 308 times greater than in the former.

5. Salt should be added at the end of the roast after sulphur has ceased to be evolved, and in proportions which have by experiment proved to be sufficient.

Loss of weight occurs when ore is roasted by the volatilization of volatile compounds. The roasted ore is consequently richer in precious metal per ton than raw ore. To make one ton of roasted ore from 33⅓ per cent sulphur ore will require 1.3 tons of raw ore. Salt will add a trifle to the weight of the ore, but the other impurities driven off will counterbalance that.

If the raw ore assays $15 per ton in gold and no loss occurs, the roasted ore should assay one third more, or $20 per ton.

The theory of roasting is explained as follows:

1. The conversion of sulphides into sulphates.

2. By raising the temperature the sulphates are decomposed in the following order of the metal compounds:

Iron, copper, silver, nickel, zinc, lead.

Mr. A. Theis gives the following simple plan for testing the degree of roast in the furnace: "A small portion of the ore from the furnace is boiled in water and stirred with a bright iron rod. The least stain of sulphur on the rod will show that roasting is not complete."

Second Test.—Ferricyanide of potassium will indicate the absence of any ferrous salt. A portion of the roasted ore is boiled, and if any ferrous salt be present in the ore a blue coloration will be given to the liquor on the addition of ferricyanide of potassium. This only indicates a dead roast as far as iron is concerned.

CHAPTER IV.

ROASTING-FURNACES.

EFFICIENT roasting-furnaces must allow easy control of the heat, abundant access of air to the hot ore, and rapid removal of the products of combustion. These specifications require large, flat hearths, that the ore may be spread out thin, otherwise small ore charges would be necessary; large throats leading to stacks; and small, shallow fireplaces, that the fire may be easily regulated. Should the draft not be sufficient to remove the products of combustion quickly, artificial draft must be resorted to, or the stack made higher and larger in the flue. This forced draft will require the construction of dust-chambers, and if much arsenic or other volatile substances are present they will be necessary under all circumstances, as the current of hot air will carry off the fumes and dust, and with them fine gold, either mechanically or chemically.

Full details of the various furnaces invented and discarded, with their good and bad points, is beyond the province of this work; we will therefore confine our attention to those which are in common use and the causes which have led up to their adoption.

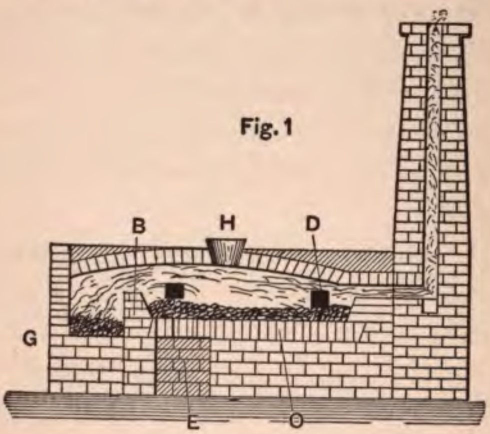
Where one furnace will give entire satisfaction in one locality it will not in another, and the chloridizing roasting for the chlorination process requires much more careful work than the same roasting for the Russell process, or for the amalgamation of silver ores after roasting. Still, at times it is within the province of chlorination to use the mechanical furnaces to great advantage where low percentages of base metals and sulphides are in the ore; but where they are above 8 per cent, "dead roasting" is a difficult matter, and we have no authentic accounts which will allow us to assert that for our purpose mechanical roasting above that limit can be accomplished.

There are two classes of furnaces to choose from—those favored by mechanical engineers, and those liked by metallurgists.

At this date inventors realize that the sale of a furnace will not lead to success in their line; consequently they wish to test the ore to be roasted, when, if their furnace proves efficient, they will guarantee it to do the work at a certain cost per ton.

Roasting sulphides commenced with pile-roasting in the open air. This method oxidized a portion of the pile, smelted another portion, and answers to-day where matte is wanted, but is not advisable for chlorination. The assortment, the grading, the sledging, handling, and carting, and, finally, the re-roasting of fully one third the pile, made inventors in Wales look for some method of oxidizing roasting which would do better and cheaper work in less time. The result was the reverberatory furnace, which has been modified, enlarged, and improved in shape as the objects to be attained became better known. They were first constructed as in our illustration, where the charge was roasted and withdrawn into a cooling pit; from this construction they were lengthened to one long hearth; and, finally, steps were added to assist in working the charges forward, as it became evident nothing could be gained by increased heat in the roast.

Fig. 1



The rabbling or stirring next attracted attention. What is known as the Spence furnace is an English invention of Peter Spence. These have rakes attached to a rigid frame, which are operated by automatically reversing the engine. The adjustment is such that ore is admitted to the furnace at the same rate it is discharged, and falls from hearth to hearth as desired. The clogging and wear of the rakes was an objection, but this is said to be overcome by cooling the rake.

The American modification of the Spence furnace is known as the O'Hara. In this furnace the rakes are moved forward by an endless chain. It is provided with two hearths, and the rakes are attached to carriers which extend through slots in the side walls and which move the chain, thus being away from the heat and sulphurous acid of the burning and roasting ore. This furnace will roast 40 tons per day down to 7 per cent of sulphur. The wear and tear upon the mechanical parts is high in spite of the precautions taken to obviate it.

The Pearce Turret furnace has hollow arms moved by simple machinery situated in the open space within the annular bed. These arms are cooled by air, which, passing through them, is forced against the rakes and on the ore.

The Spence furnace built with steps is continuous in its action, as many as five steps and hearths being used.

Mr. Peters gives an account of the furnaces which he built at Butte, Montana, as 64 × 16 feet outside, with four hearths, each 14 × 15 feet wide in the clear. The ore is charged from a hopper upon the hearth farthest from the fire in 3600-lb. charges, and in that position will become a bright red in two hours. Each furnace calcines 11 tons ore in 24 hours, from 30 to below 4 per cent sulphur, and consumes 2 cords pine wood, only the labor of two men being required. A similarly constructed furnace is used at the Treadwell mill in Alaska, where dead roasting is required.

The Brown Horseshoe furnace is a somewhat later invention, and does good work on low-sulphur ores.

To obtain a " dead roast " from high-sulphur ores we are obliged to leave the mechanical furnaces, and either use them in preliminary roasting, followed by the use of the reverberatory for finishing roast, or else discard them for the reverberatory entirely.

The disadvantages in a measure overcome by the mechanical rabbling of the Spence and other furnaces can be further overcome by the revolving furnaces of the Brückner type, or the revolving pan-furnace with fixed rabbles, but in either case the dead roasting must be done in the reverberatory.

The shaft furnaces, such as the Stetefelt, are fairly good for chlorination of silver ores, but will not answer for the gold chlorination process.

Ores carrying a high percentage of arsenic, zinc, tellurium, bismuth, antimony, and a low percentage of sulphur can be sufficiently roasted in the mechanical furnaces at a low cost, but the advantages of these furnaces for a high percentage of sulphides in the ore have yet to be demonstrated.

The old reverberatory, like the beehive coke oven, seems to hold its own against all comers as far as uniformity and product is concerned.

The reverberatory furnace is so well known among professional men, the description would be omitted were it not the object of this book to reach the student and non-professional man not so well informed.

In our illustration, Fig. 1, *H* represents the hopper through which ore is charged to the hearth *O*. On this hearth it is spread out in a thin uniform layer. The hopper is provided with a cover, which is closed as soon as the oven is charged. *G* is the fire-box provided with grate-bars, fire-doors, and ash-pit. The fuel usually burned is wood, as it is free from sulphur and ash, which might in a measure reach the ore, and, furthermore, a wood fire is more readily controlled. The flame from the burning fuel is separated from the ore on the hearth by the bridge *B*. This bridge also raises the flame against the fire-brick arch, which reflects it down upon the ore. It is this reflected or turned back heat which gives this style of furnace its name.

As the flame approaches the stack it becomes cooler, and for that reason the arch pitches towards the hearth gradually until it meets the throat, where after passing the ore it goes through a straight flue and up the chimney. The apertures *DD* are working-doors through which the ore is spread upon the hearth as well as stirred during roasting. *E* is the discharge-pit for the calcined ore.

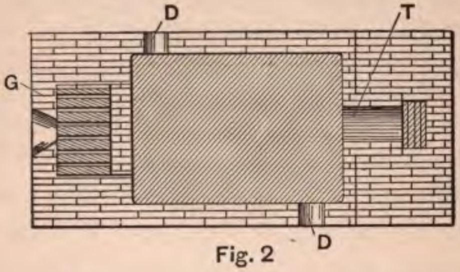


Fig. 2

By reference to the figure it may be seen that considerable heat might be saved if the furnace was longer, or longer with steps, the under side of the throat of the first furnace being raised to answer as a fire-bridge for the second in the latter instance.

With the same object in view return-flues are built underneath the hearth; again, one furnace is built above the other in such a way that the flame passes from the first furnace up and back over the hearth of the upper one and then up the stack.

The latter style of furnace is termed a "double-hearth," and is worked as follows: The top is used for drying the ore, after which it is dropped to the first hearth, upon which it is allowed to remain twelve hours. This charge is then dropped to the lower hearth and a new charge placed in the upper. After remaining a sufficient time on the lower hearth to allow sulphur fumes to cease, salt is added and well stirred into the ore. The time required for this roasting is about twenty-four hours.

The furnace has the advantage of utilizing space, of continuous roasting, of creating little flue-dust, and lessening fuel expenses. The size of such a furnace is 18 feet long, 15 feet high, and with low fire-brick domes or arches, cast-iron working-doors, and other appurtenances will cost approximately $1.50 per cubic foot.

Step-furnaces are built as in our illustration, but one furnace is raised three to four feet above the hearth of the other. The ore can be more readily kept separated and transferred from one hearth to the next in this furnace. The domes can be brought lower to the ore as well, where with one long continued reverberatory this cannot so readily be accomplished. These furnaces are about 60 feet long and 15 feet wide, will roast 3 tons per day, and have a capacity of about 9 tons of raw ore. The charges are kept separate and the roasting is continuous. One hearth reverberatory furnace of the above dimensions will cost $2 per square foot of ground-space; step reverberatories will cost $2.50 per square foot of ground-space.

The working of such furnaces is illustrated as follows:

Charged at 8 A.M. 2500 lbs. raw concentrates, well dried and free from moisture.

8.30 A.M. Sulphur commences to flame on the 4th hearth. Fire good.

10 A.M. Moved charge to 3d hearth; sulphur burning briskly; ore hot and red. Fire almost without fuel. Charged the 4th hearth.

2.30 P.M. Moved ore to 2d hearth, large quantities of sulphur fumes being given off. Fireplace dark; ore red. Moved ore from the 4th to the 3d hearth; sulphur burning freely. Charged the 4th hearth.

3.30 P.M. Combustion of sulphur on 2d hearth; decreasing ore swelling slightly. Fireplace dark. Ore on 3d hearth giving off large quantities of sulphur; ore dark red. Ore on the 4th hearth commencing to burn.

4.30 P.M. Combustion of sulphur ceased on 2d hearth; added 3 per cent salt and mixed well. Added fuel to the fire. Ore on the 3d hearth giving off large quantities of sulphur fumes; ore on 4th hearth burning briskly.

5 P.M. Charges passed from 2d to 1st hearth, 3d to 2d, 4th to 3d, and new charge placed on the 4th hearth. Fire hot, but not much chlorine fumes given off.

5.30 P.M. Chlorine fumes ceased. Fire red-hot. Ore swelling slightly on 2d hearth; sulphur burning briskly on 3d hearth; sulphur just commencing to flame on the 4th hearth.

6 P.M. Charge drawn from the 1st hearth into the pit. Ore moved from 2d to 1st hearth, from 3d to 2d, from 4th to 3d, and new charge added to the 4th.

In this continuous manner roasting is carried on day in and out.

The roasted ore is not immediately removed from the pit; it is left there some time, to allow chlorine fumes to pass off.

It is then taken to the cooling floor and allowed to become cold, when it is dampened and elevated to the chlorinator floor.

Wetting down roasted ore when red-hot is likely to cause "balling," in which condition it is not in proper shape for chlorinating, it being difficult for chlorine to penetrate the balls and extract the gold.

In this connection we may also note that raw concentrates should be kept moistened until ready for use, when they are dried quickly; otherwise they will be likely to form lumps, which are difficult to roast thoroughly, and thus again a loss in leaching from imperfect oxidation.

The revolving-hearth furnace is an iron pan which carries the ore to be roasted. It is geared to revolve horizontally, and has fixed rabbles which stir up the ore as it comes in contact with them, thus exposing new faces to the heat. These furnaces do excellent work, but, as before noted, are not able to do "dead roasting" without the aid of a small reverberatory attached. Prof. Phillips* gives the dimensions of one used by Mr. Theis, which roasted 3 tons of concentrates in 36 hours, desulphurizing pyrites from 32 to 0.25 per cent.

Diameter of the hearth or pan...........	12 ft.
Depth.................................	8 in.
Height of dome from centre of pan......	30 in.
Furnace-wall thickness.................	14 in.

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* Vol. XVII, Trans. A. I. M. E.

Fire-Lox	6 ft.
Grate-surface, 2 × 3	6 ft.
Fire-bridge height	2 ft.
Throat length, 4 ft.; height 16 in., area 5 ft. 4 in.	
Working-doors (one each side)	8 × 16 in.

Reverberatory attached:

Length	14 ft.
Width inside	6 ft.
Spring of arch	2 ft.
Working-doors (one each side)	8 × 16 in.
Dust-chamber	4 × 3 × 20 ft.
Spring of dust-chamber arch	18 in.

The cost given for roasting by this furnace one ton of concentrates:

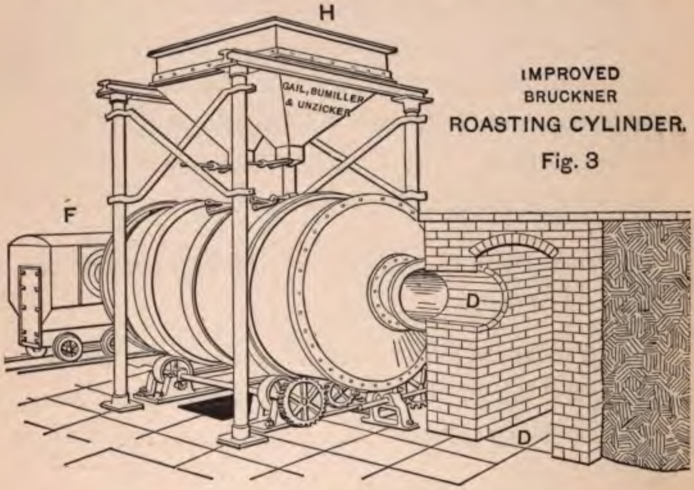
½ cord wood, at $1.40	$0.70
12 hours' labor, at 9 cts.	1.08
Motive power	0.25
Cost per ton of concentrates	$2.03

As it requires 1.3 tons of such concentrates to make one ton of roasted ore, the cost of roasting one ton of ore would be $2.64.

Of the revolving-cylinder furnaces we must say the same as for the other mechanical furnaces.

The Brückner furnace as originally designed was intermittent in discharge, and could roast but one charge at a time. It was a wrought-iron cylinder, lined with 4-in. curved fire-brick, usually 18 to 22 feet long and 6 to 8.5 feet in diameter.

The cylinder had two orifices, one at each end; a manhole for the receipt of the ore, and its discharge after roasting. The motion was around a horizontal axis, but as improved the cylinder turns on rollers geared to the motive-power shaft.



IMPROVED BRUCKNER ROASTING CYLINDER.
Fig. 3

The two contracted orifices, one at each end, are for the fire and escape of the products of combustion. In Fig. 3 an improved Brückner roasting-cylinder is shown, one end connected with a movable furnace, the other with the dust-chamber *D*. The movable furnace *F* is connected with the cylinder until the charge commences to blaze, when it is removed, and only reattached to complete the roast. *H* represents the hopper through which the ore is charged.

These furnaces are better adapted to the roasting of silver ores, in which case they will roast 9 tons concentrates inserted in one charge from 30 to 5 per cent sulphur in 24 hours.

In some cases the cylinder is made with four manholes or discharging-doors, and lined with fire-brick at the throats and red brick inside the cylinder.

The 8.5 × 18.5 feet cylinder has a capacity of 9 tons, and weighs 45,000 lbs.

The throat-linings require 100 fire-brick.

Fire-box, 500 fire-brick and 4135 red bricks.

The body of cylinder, 4300 bricks.

If movable fire-box is used, 2800 more fire-brick are required.

These furnaces when used for chloridizing roasting and chlorination, possess the advantage of allowing desulphurization of base ores before salt is added, thus obviating one difficulty of mixing the salt with the ore at the commencement of the roast. The Brückner furnace has some of the advantages of the reverberatory, as well as its disadvantages, increased. The percentage of sulphur may be decreased, also the heat readily controlled. The charge may be retained in the furnace as long as desired, and tested for the degree of roast required.

It was the intention of Mr. Brückner to make the furnace automatic and continuous by placing two furnaces in a line, but stepped one below the other and connected by a feed-pipe. The fire was to be made at the lower cylinder and circulate through both, thus effecting a saving in fuel. The results for comparison between Brückner and reverberatory furnaces are not in favor of the Brückner for dead roasting for a high-sulphur ore. Where low-sulphur ores are roasted, the advantage must be with the Brückner.

The disadvantages of the furnace are:

Care must be taken not to let the heat get " too light a yellow," as the charge will ball.

The charge has to be heated to a " dark yellow," and become sticky to avoid flue-dust, in which case it balls if the ore be fed dry. If the charge is wet the ores have to be heated to a dark red, which is the condition for flue-dust to form. The damper must be closed and a hot fire kept up until the ore is a dark yellow, when it becomes sticky and liable to ball. This balling is a great drawback to chlorinating, as the ore thus roasted must be recrushed before being placed in the leaching-barrels; but in such masses it cannot be properly " roasted dead." The makers say it is possible to roast ores to as low a percentage of sulphur in the Brückner furnace as in a hand reverberatory furnace if the operators are experienced.

For desulphurizing where dead roasting is not required these furnaces are well liked, one firm using 168 of them, roasting from 40 to 7 per cent sulphur with 167 lbs. of coal to the ton of ore roasted. There is another advantage which will apply to almost any furnace—a further decrease in the cost of roasting by the use of fuel-gas. The simplicity of modern fuel-gas producers allows almost any fuel to be converted into gas at a moderate cost, and the gas fixed or purified to avoid any deleterious mixtures with the ore. The Mond Gas Producer will produce 60,000 cubic feet of gas from one ton of inferior coal, coke, or peat; and equally good results may be obtained from our modern producers, such as Loomis or Taylor's, although they do not claim as much.

The Brückner furnace has been improved in various details, among which is Clark's oxidizing and desulphurizing apparatus, which brings the air in contact with the ore by means of a pipe running through the furnace. This pipe is water-jacketed to keep it cool, while air is forced through it and from apertures in it to the cylinder. This principle of adding more air is correct in every detail, as naturally what air is admitted to the throat rises to the top of the cylinder without coming in contact with the ore, and so out the stack. The air forced in, however, through the pipe would come in contact with the ore and drive the sulphur fumes from the burning ore to the top of the cylinder and allow quicker oxidation.

These furnaces are revolved at the rate of 100 feet per minute, and should not be above that. If the charge is working properly, it goes half-way up the sides of the furnace and slides back; if above this point it is too hot, if below this too cold.

The cost of a Brückner furnace will depend upon freight rates from some manufacturing point.

A 20 × 8 furnace will weigh 45,000 lbs. It is composed of iron work as follows: 20 × 8 ft. iron cylinder, one movable fire-box, one 60 ft. × 30 in. iron stack, one conveyor, one hopper and driving-gear, driving-shaft, countershaft, pulleys, pillow-blocks, etc., complete.

There are needed 3400 fire-brick, 10,000 red brick, 8 bbls. fire-clay, lime, and cement.

The labor of one machinist, two bricklayers, and three helpers, 10 days.

The building for the furnace will be 30 × 20 ft.

The cost in dollars and cents will approximately be:

Cylinder and appurtenances	$2,500.00
Fire, red bricks and lime	335.00
Labor	210.00
Building 30 × 20 at $1.50 per sq. ft. of ground surface	900.00
Total	$3,945.00

To which must be added the freight.

The Howell or Oxland furnace is a modification of the Brückner. The Howell-White differs but little from the Howell, and what is termed the improved White is a trifle different yet. They are all rotating cylinders, provided, as in the Howell-White, with cast-iron shelves, which raise the ore and let it drop back as it rotates. They are made 30 feet long and 5 feet in diameter, lined entirely throughout with 4½-inch curved fire-brick. The Howell-White has but one end lined with these bricks, while the metal on the smaller portion is exposed. This smaller portion has cast-iron spirally arranged shelves, which showers the ore through the flames as it turns.

The roasting capacity of a 60 in. × 27 ft. furnace is stated to be from 30 to 45 tons per day. The amount of brick required for lining furnaces and 20 feet of dust-chamber is 28,000 red brick and 2700 fire-brick; fire-clay is also required.

The Improved White roasting-furnace differs only in being a long cast-iron revolving cylinder, lined throughout with fire-brick. These fire-brick are curved to the inner circumference of the cylinder, and are 4½ inches thick. The shelves are made by inserting key-brick 9 inches thick in rows, which make a projection of 4½ inches. These shelves run nearly the entire length of the cylinder. The greater the number of shelves the more dust is made, and for this reason the shelves have been reduced from 8 to 6.

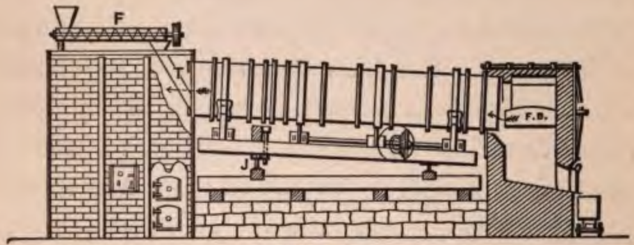


Fig. 4.

A furnace 60 inches in diameter and 30 feet long will require 28,000 red brick, 9000 fire-brick, and 3 bbls. of fire-clay. The capacity is from 30 to 45 tons per day. The weight of these cylinders is about 28,000 lbs., including bearing-wheels, chairs, sole-plates, gearing, and bolts, exclusive of brick and other paraphernalia.

These cylinders are inclined toward the fire end, and fed at the upper end by a screw feeder situated as shown in the figure. They are continuous in operation, discharging the product regularly into a pit at the lower end, from which the roasted pulp is withdrawn as required.

Working the ore towards the fire is the true theory of perfect roasting, while the expenditure of power—7 to 12 H.P.—is not unreasonable for the size of furnace.

The shelves, as mentioned, create considerable dust, and this requires mechanical drafting to remove with the products of combustion.

This drafting creates another difficulty: dust leaves the cylinder, as does fine ore, before it is thoroughly chloridized. We have noticed also that if salt is used the chances for loss of gold are greatly advanced, when the salt is mixed with the ore at the commencement of the roast.

To roast this flue-dust Mr. Hoffman added a furnace at the dust-chamber end, which necessitated two fires. This did not entirely obviate the difficulty; and in some instances with automatic charging one third of the ore went into the dust-chamber instead of through the cylinder, and not being sufficiently chloridized had to be reroasted with salt. To charge out of line of draft one or two patents were taken out, and in another instance a collar was inserted, which formed a sort of chamber which possibly stopped the draft and some of the dust.

Mr. Rumsey claimed for this latter patent that the chamber formed at the head of the cylinder held the dust until roasted, after which it would fall to the bottom of the cylinder and move with the heavier ore down the cylinder and out with the roasted pulp.

The Hoffman roasting-furnace is a revolving cylinder of the Brückner type, with a fire-place at each end, also a flue.

The flues are between the fire-places and cylinder, descending to dust-chambers which are connected with the main flue. By means of dampers the current of air can be reversed to go in either direction, and thus expose the ore charge to a uniform temperature.

The furnace is suitable for ores which require a very high or low roasting temperature.

The Brown Horseshoe Furnace is constructed in the shape of a circle, with one fifth of the circle left out. The cross-section gives an arch the same as the common reverberatory. The stirrers are mechanical, and move on carriages running on tracks in chambers on each side of the furnace. The ore is charged and discharged automatically, always working against the hot furnace. The fumes may affect the carriers and moving mechanism, with all the protection offered. The stirrers make the round of the furnace, and then stop to cool off in the one-fifth open space; they work automatically. The stirrers do not, it is claimed, by this cooling process become overheated.

The points of superiority are claimed for this furnace as follows:

1. It is simple in construction, requiring no more brick or iron work than the ordinary reverberatory furnace of the same length of hearth.

2. It is 50 per cent less in cost than any other mechanically stirred furnace of the same capacity.

3. The operating mechanism is most easy to manage, and least liable to get out of order.

4. The carriages moving the rakes, standing half the time in the open air, are kept thoroughly cooled, and are at all times perfectly accessible.

5. Less manual labor is required, one man on a shift taking care of the machinery and fires for a furnace of 40 to 60 tons daily capacity.

6. The hearth being on one plane, no dust is raised by the falling ore, and no loss of heat, as is the case with furnaces having upper and lower hearths.

7. The feed is automatic, introducing any required quantity of ore with the passage of each rake, the amount being governed by a counterpoised lever which weighs each charge.

8. The machinery is perfectly noiseless in operation.

9. All the journals that are exposed in any manner to the heat are fitted with ball- or roller-bearings requiring no lubrication.

It is claimed to give a "dead roast" on high-sulphur ores. The author has had no opportunity to thoroughly inspect the working of this furnace, and no data but the maker's to go by. The reports received, however, are such as to warrant the assertion that this furnace is a great improvement in mechanical roasting-furnaces, and worthy further investigation.

Mr. R. P. Rothwell gives an account of one of the Howell-White furnaces used by him at Deloro, Canada, in roasting ores carrying 40 per cent arsenic. He was able to obtain 93 per cent of the gold by its use. The arsenic fumes being very dense, flues as well as dust-chambers were necessary. The forced draft was obtained by a Guibal fan.

The dryer was an inclined revolving cylinder 48 inches diameter at the mouth and 36 inches diameter at the throat, with a conical addition at the throat making its total length 22 feet. Fire passed through this cylinder, from which the ore dropped in a continuous manner into a boot, and was raised by elevator buckets into No. 1 roasting cylinder.

This was a revolving cylinder 30 × 5 feet, was similar in lining and shelves to the Howell described above. From this the ore ran direct into a second roasting cylinder 20 × 4 feet, lined as above, and in which the roasting was completed. The cost of roasting these ores is given as 60 cents per ton, the furnace roasting 10 tons in 24 hours. This ore is more readily roasted than simple sulphurets would be.

The obstacles encountered were flue-dust, loss of fine gold carried away by the dust, aided by the arsenic fumes and volatilization.

CHAPTER V.

THE LEACHING PROCESS.

HAVING prepared the ore for roasting, and by roasting for leaching, we come to that stage of the process where the outlay from the foregoing is to be recovered with interest. In the first chapter we dealt somewhat at length upon this subject, but additional information of the action of chlorine and its, to us, most interesting compound of gold will not be out of place.

Auric chloride, the most important compound of gold, is a red crystalline mass when evaporated to dryness, soluble in water. When combined with other metal chlorides, it forms double salts, termed chloro-aurates, the general formula of which is $MCl, AuCl_3$, where M represents an atom of a monad metal. These compounds are mostly yellow in crystals, but red when deprived of the water of crystallization. It seems to make no difference to chlorine whether the gold is cold or warm, provided no other impurities are present with which chlorine unites.

If auric chloride ($AuCl_3$) be heated above 130° C. subchloride of gold is formed; and by warming AuCl, or gold sponge, in $AuCl_3$ a double chloride, AuCl, $AuCl_3$, is formed.

Pratt says $AuCl_3$ heated in chlorine gas forms a higher gold chloride.

Chlorine gas for chlorination is usually generated from solutions of bleaching-powder * by sulphuric acid. We may consider the bleaching-powder to have the formula $CaO + Cl_4$, and then to have had its Cl displaced by the air, carbonic acid, or oxygen, and become $CaOCl_3$. The reaction then would be:

$$Au + CaOCl_3 + H_2SO_4 = AuCl_3 + CaSO_4 + H_2O.$$

This may not be the direct action—that point seems to be in doubt; but the ultimate reaction is as given, a gold chloride being formed when nascent chlorine is liberated by sulphuric acid in the presence of gold.

The chlorination process is based upon this reaction.

The Plattner process of chlorination is practised as follows:

The ore is crushed, concentrated, roasted with salt,

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* Bleaching-powder is composed of $CaCl_2 + CaCl_2O_2$, calcium hypochlorite.

Roscoe (VI., p. 176) considers hypochlorite to be the important factor. The result for our purpose is, however, ultimately the same, viz., the production of $AuCl_3$.

and sifted into large wooden vats, which are slightly raised at one side to insure drainage.

There is a filter-bed (described in Chapter VI) in the bottom of these tanks, through which the liquor containing the auric chloride is drained, and collected for precipitation (see Chapter VII, p. 81).

The vats are of a size to suit the operator—say 9 feet in diameter and 3 feet deep, or 12 feet in diameter and 4 feet deep. The depth, however, should not exceed 4 feet, unless mechanical contrivances are employed to remove the tailings from the vats after leaching. These vats should be lead-lined, although this is not absolutely necessary, but will prevent leakages and possibly loss of gold chloride in solution.

This process requires about four days, and on that account four leaching-vats, to allow of one being cleaned and charged daily. The charge for a 9 × 3 ft. tank is 4 tons, and for a 12 × 4 ft. tank 7 tons, of roasted concentrates.

In order to sift the concentrates into the tank so as to lie loosely and avoid packing, they are slightly dampened, with enough water to form a ball in the hand when squeezed, which will crumble, however, when the hand is opened and pressure removed. This looseness allows the chlorine gas to permeate the ore, which it could not so readily do were it packed in the tank. A coarse screen over the tank is used for this sifting, and the tanks nearly filled with ore, but space enough left to assure the ore being covered with water after gassing. The gas is generated in an apparatus (usually two generators), which causes it to flow into the bottom of the tank at two opposite points and work upwards through the ore. This gassing continues until ammonia held over the ore gives off the dense fumes of ammonium chloride; the tanks are then covered with lids, and the lids luted with clay to prevent any escape of gas. This operation requires about three hours.

The tank being charged with chlorine gas is left standing with the gas in contact with ore two days. The gas is not forced into these tanks by pressure, but continues to generate and go into them for a time after the covers are placed on them. The amount of gas required is fairly well known for one class of ores, as is the amount of chemicals, so that the generators are charged accordingly to supply that quantity.

On the third day the ore is leached by filling the tanks with water, which immediately dissolves the gold chloride and holds it in suspension.

The gold chloride is now fixed, and can only be precipitated by some metal salt, as metallic gold. This solution is now filtered off by removing the plugs or opening lead spigots at the bottom of the vat. Wash-water is added until upon testing no chlorine appears to be present, when we may consider the leaching to have been completed. Water is added in sufficient quantities to keep the tank full during this process. This leaching process requires from four to five hours.

G. W. Small * gives an account of tank chlorination at the Plymouth Mining Co.'s chlorination-works in California, where the ore was passed through a 35-mesh screen, 1205 holes to the square inch, which allowed quick leaching and a recovery of 95 per cent of the gold. The liquor from the tank is run or pumped into settling-tanks, where sulphuric acid is added, to hold in suspension any impurities which the excess of water has dissolved from the ore, and also to allow mechanical impurities in suspension to settle in these tanks rather than in the precipitation-tanks. After settling, the liquor is drawn off into precipitation-tanks, where the gold is thrown down as a brownish precipitate by ferrous sulphate, which process will be explained in Chapter VII.

If lime or talc is present in the concentrates, the ore in roasting may have its lime converted into sulphide of lime. This sulphide must be entirely decomposed, or the leaching in the vat or barrel will prove unsatisfactory.

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* Trans. A. I. M. E., vol. xv. p. 307.

The chlorine will be converted into calcium chloride, and, furthermore, hydrogen sulphide will be evolved, which will precipitate the gold from the auric chloride solution as fast as formed until the excess of chlorine gas ceases. This will be lost with the tailings, unless reroasted and retreated.

"To use the Plattner process on lime ores, no calcium sulphide should be present in the material when it enters the leaching-vat."

Whenever the lime is in the shape of chloride of calcium, leaching can occur without loss; but no roasting test will determine the presence of calcium sulphide.

If carbonates be present in the ores, the hypochlorite of calcium in the bleaching-powder will be decomposed, and consequently the hypochlorous acid will remain inert.

The Mears process had for its object a saving in time over the Plattner process. It is so similar in details with the Theis process that the two can be described together. The chlorine gas was generated for the Mears process outside the chlorinating-barrel, while in the Theis process it is generated inside the barrel. The latter process has entirely excluded the former, as far as barrel chlorination is concerned, on account of its simplicity and its removal of the objectionable features of the former.

BARREL CHLORINATION.

The chlorinator, Fig. 5, was with the Mears process a cast or sheet iron cylinder capable of withstanding a pressure of 60 lbs. per square inch. This cylinder was lined with sheet lead weighing 10 lbs. to the square foot. It was fitted with cast-iron cylinder heads, which were securely bolted. It revolved upon a hollow trunnion, through which the gas was forced into the interior of the cylinder by means of a pressure-pump. This trunnion was arranged to admit at one end an iron lead-lined gas-pipe, termed a "goose-neck," one end being in the trunnion, the other outside, upon which was a pressure-gauge to record the

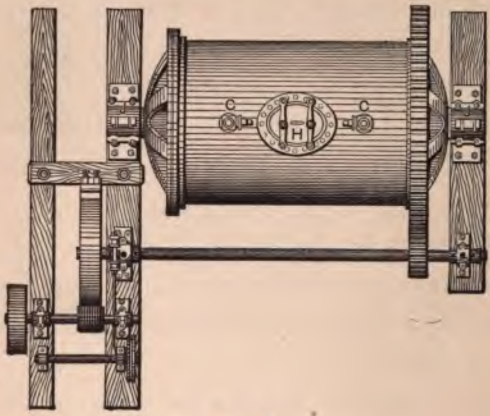

Fig. 5.

pressure inside the cylinder. The barrel was charged with roasted ore and chemicals, then rotated from 15 to 20 times per minute.

The Newberry-Vautin process used compressed air as well as the pressure generated by the chlorine gas. The air was to be compressed in wooden barrels to 100 lbs. per square inch, and that with the gas pressure was to permeate the rock. It had to permeate the rock or burst the barrel—we think the latter, as the process has come down to generating the pressure to 20 lbs. in the barrel, and this pressure is proved to be necessary on account of the preliminary treatment of the ore.

Mr. Theis ascertained that pressure was merely an accompaniment to barrel chlorination, and might be as readily obtained inside the barrel as out, the main object to be attained being the liberation of chlorine in the presence of gold, and under such conditions as would mechanically compel the gas to attack the gold in the least possible time.

He therefore discarded the generator, gas-storage tank, and pressure-pump, and employed chloride of lime for the chlorine gas and sulphuric acid to liberate it, charging these chemicals in suitable proportions directly into the cylinder.

This enabled him to do away with the hollow trunnion, and substitute solid shaft trunnions securely bolted to the heads and provided with tight and loose pulleys. The cylinder for a one-ton chlorinating charge was made 42 inches diameter and 5 feet long. The "goose-neck," which was continually leaking, was next discarded, and in its place was substituted a lead valve, by which the gas in the cylinder was tested to better advantage, thus obtaining more uniform extraction.

The generation of gas in the cylinder is tested from time to time. The pressure-gauge might show considerable pressure in the cylinder, but there was no means by which it could be determined whether that pressure was chlorine gas or some other gas. In case it was the latter, and the gas shut off from the generator, the extraction was poor; but by means of the lead cock chlorine is readily detected, and if free chlorine is not found present on testing, a time suitable having elapsed, more lime and acid are added. It can be seen that an appliance which allows the gases in the cylinder to be tested, rather than by an uncertain pressure-gauge, is of immense importance.

The illustration given, Fig. 5, is a one-ton chlorinating barrel of the Theis pattern. The cylinder may be cast or wrought iron with flanged ends. The heads are cast iron, with solid cast trunnions to rotate in suitable boxes. These heads are bolted to the cylinder, and made perfectly air-tight.

The cocks for testing are shown on each side of the manhole *H* at *C*, *C*.

The illustration gives the barrel with a geared flange at one end, and driven by a small wheel connecting with the pulley-wheel; these barrels, however, may have the trunnion extended, and upon it loose and tight pulley-wheels for motive-power attachments. They may also be turned directly by belts passing around the barrels. With barrels larger than three tons capacity it is advisable to build them of wrought iron or steel boiler-plate, capable of standing 150 lbs. pressure to the square inch. They should also rotate on tires bolted to them, as shown in the Brückner furnace and Figs. 5*a* and *b*, these tires receiving their revolving motion from the rollers upon

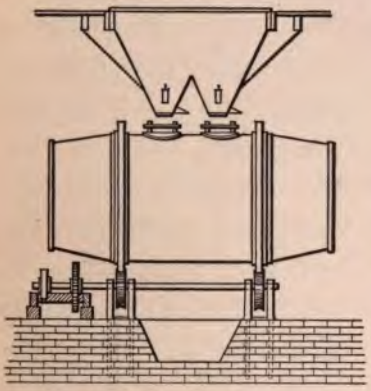

Fig. 5a. Chlorinating Barrell.

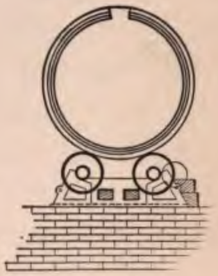

Fig. 5b.

which they rest. The rollers are keyed to shafts and connected by gears to motive-power shafts.

A five-ton barrel, 60 inches in diameter and 9 feet long, could receive six charges in 24 hours, and is therefore capable of chlorinating 30 tons of roasted ore or concentrates per day.

This size requires no more attention than a one-ton chlorinator, and is consequently less expensive to work. There is no specified limit to the length of these chlorinators: they can be made up to 20 feet, and chlorinate 20 tons at a charge. Their cost in such instances would not be as much as 4 five-ton chlorinators, while their capacity would be 120 tons per day. With such large-sized chlorinators, from five tons upwards, their ends should taper as do barrels—first, to allow the liquor to drain from the centre of the barrel or larger diameter, and, second, to allow of smaller heads which can be handled more readily, and which if subjected to high pressure need not be reinforced by braces.

Fig. 5*a* gives an outline of such a barrel, with heads made of ¾-inch boiler-plate. The bolts which fasten the heads are to be countersunk inside.

The shell is ⅜-inch boiler-plate, 10 feet long, 60 inches inside diameter at the centre, and 50 inches inside diameter at the ends. The object of bolting the cylinder-heads is to allow of quicker repairs and examination of the lead lining, also to change the filter-bed with more facility. The lead lining is bolted to the shell by rows of flat-headed bolts or lead rivets spaced about 18 or 20 inches apart. The sheet lead is dressed back against the iron shell with lead hammers or wooden mallets. There are two manholes instead of one, as in Fig. 5; size, 10 × 12 inches. The Colorado Iron Works, the E. P. Allis Co., and Fraser & Chalmers make a specialty of chlorinating plants; full particulars for these barrels can be obtained from them, together with their cost. The transverse joints should be single-riveted, the longitudinal double-riveted, since the cylinder may be called upon to withstand 60 to 80 lbs. pressure.

The speed per minute for such barrels should not be over 100 feet.

The method of charging these barrels is as follows:

The barrel is first partially filled with water; the proper amount of chloride of lime is added; on top of this the roasted ore is placed, and finally the proper amount of sulphuric acid. The manhole covers are then put on and screwed down tight, after which the barrel is set in motion. The water first put in the barrel is for the purpose of making an easy-flowing pulp. The lime is added next, that it may more intimately mix with the ore, which latter will work through it. The sulphuric acid is added last, so that little gas may be generated before the covers are on and the barrel set in motion, and also as a precaution of safety. About one fifth more acid is added by weight than chloride of lime, the object being to convert all the lime possible into calcium sulphate, to remain on the filter as well as to set free all the chlorine gas from the chloride of lime. When lime passes the filter-boxes a bulky precipitate is produced in the settling-tanks, or if it passes the latter the precipitation-boxes; an excess of acid in the chlorinator will avoid this.

Mr. Theis advocated and practised the division of the chemical charge.

He first charged one half the amount of the lime and acid into the ore and water mixture, then closed the chlorinator and rotated it three or four hours, after which the remaining half of the chemicals were added, and the chlorinator rotated two or three hours longer. By this method a possible saving of chemicals may occur, for if the ore were not properly roasted there would be free chlorine in the cylinder, but if properly roasted very little will be free. The time required for this process is from four to eight hours, between five and six hours being the average.

The presence of free chlorine is detected by the means of the lead cock mentioned, and the cylinder rotated at least one hour after its detection to insure proper absorption of the gas by the water and metal. The ore is then discharged upon a shallow filter-bed below the barrel, or the cocks opened, when filtering is to take place from pressure applied inside the barrel.

Aqua ammonia may be used as a test for free chlorine, as stated in the Plattner process.

The cause for the presence of gas being in the generator, other than chlorine, or the cause why chlorine gas is not properly absorbed, may be traced almost entirely to improper roasting. If copper be present more bleaching-powder and acid are needed, and again the recovery may be so low that the whole charge must be dried and reroasted. Suppose, for example, we have roasted ore which we know assays $80 per ton, and from which we generally recover 95 per cent or $76, but from improper roasting we are only able to extract 80 per cent or $64. It will pay to reroast this charge and rechlorinate if we can recover 95 per cent, or $15.20 of the $16 present in the tailings; but the cost of chlorination was but $3.50 in the first place, and by double work it has mounted to $7.

With proper care in roasting the percentage of recovery by barrel chlorination will be between 90 and 98 per cent as the extremes and 94 as the average extraction.

Mr. Rothwell, one of the first to employ the Mears process, states that with a good roast 40 lbs. per square inch was given in the cylinder, but with a poor roast but 25 lbs. per square inch was recorded on the pressure-gauge in a certain time. This may be explained in the first instance by the gold forming chloride and leaving an excess of gas, in the second by other chlorides being formed besides gold.

CHAPTER VI.

FILTERING.

THE chloride of gold having formed in the chlorinator, water is added to make a liquid pulp. The barrel is then revolved a few times to wash out the chloride from the ore, and the pulp dumped direct upon the filter-bed. The barrel is now washed by water and a few more revolutions given it, this wash-water in turn is added to the filter-bed, the barrel is now ready for another charge. This practice may vary somewhat: the liquor may be decanted from the barrel direct into the filtering-tanks, wash-water added, and the ore and water run onto the beds; or the filtering may take place from the barrel itself, if provided with an asbestos or sand filter, in which latter instance pressure may be applied to hasten the filtering.

Sand-filters are constructed as follows: For a one-ton charge they are lead-lined boxes 6 feet wide, 8 feet long, and 18 inches deep; for five-ton barrels they are 8 feet wide, 15 feet long, and 3 feet deep.

The bottom is covered with clay or other substance impervious to water and not acted upon by action of acid and chlorine, such as perforated glazed tile. The bottom has a fall of about one inch in 8 feet. Upon the floor clean gravel-stones ½ to 1 inch in diameter are placed in a layer about 2 inches thick. Another layer of finer gravel-stones is placed over this about 1 inch thick, and upon this finer gravel about 1 inch thick, and lastly fine clean quartz sand, making a filter about 5 or 6 inches thick. To prevent the filter from getting uneven surfaces, strips of board 1½ to 2 inches wide are laid about 10 inches apart on its surface. This filter is then flooded from below upwards until the water stands over the surface. The pulp from the chlorinator when discharged cushions on the water and slats, preventing the surface from becoming uneven, also the filter from packing, and allows the pulp to flow evenly over the face of the filter-bed. The corks or rather plugs from the lower end of the filter-bed are now removed, and the liquor allowed to flow into the settling-tank. The filtering requires three or four hours, and is followed by a wash-water of from 150 to 300 gallons per ton of ore, or until no reaction is given, when the filtered water is tested with ferrous sulphate ($FeSO_4$).

The reaction, if trichloride of gold is present, would present a reddish-brown precipitate of very fine brown gold, which reaction may be expressed as follows:*

$$2AuCl_3 + 6FeSO_4 = 2Au + Fe_2Cl_6 + 2(Fe_2O_3, 3SO_3).$$

The filtrate should be quite clear, and the filtering accomplished as speedily as possible. So long as the solution shows the presence of free chlorine, when the last wash-water is leaving the filter, the ore has been thoroughly leached.†

For each barrel there are four filter-boxes, so that at least one will be ready for use. When these filter-boxes are deep or have been used some time the rate of drainage is not satisfactory. Instead of draining at the rate of three or four inches per hour, the drainage is not more than one inch or less per hour. Mr. E. G. Spilsbury advocated decanting the first water and the barrel wash, and only dumping the last wash and ore on the filter. Prof. Phillips does not approve of this method, and considers time lost rather than gained by so doing.

Mr. R. P. Rothwell was one of the first to wash

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* See Chapter VII.

† Chlorine water decomposes iodine compounds, setting free iodine when not in excess. A dilute solution of metallic iodide mixed with starch paste acquires upon the addition of a little chlorine water a blue tint at once, which becomes colorless again upon the addition of more chlorine water.

The presence of chlorine is also indicated if a white precipitate is formed by a few drops of silver nitrate, $AgNO_3$.

the ore in the barrel and decant the liquor; the ore when thoroughly washed in the barrel was removed to the dump by a stream of water, not going into the filter. This system required much time for washing.

The next step was to dump into a filter-box and wash under pressure; but this was objectionable because of channels being formed in the filter-bed, through which the wash-water passed instead of uniformly percolating through all the ore, making the washing very irregular.

Mr. J. E. Rothwell turned his attention to washing and filtering in the barrel under pressure. For this purpose he had drainage-holes made in the barrel on one side and on the other holes for connection with pressure-pump. At first he employed an asbestos-cloth filter, which he found difficulty in obtaining suitable for the purpose, and high-priced. The fibre of the cloth was destroyed by the pressure, making the process expensive; otherwise it was a success.

To overcome these objections, he devised and constructed a sand-filter inside the barrel, which lasts about one month, and then has to be replaced, when subjected to a pressure of 20 to 30 lbs. per square inch. He gives a description of this filter in vol. LX of the *Engineering and Mining Journal*, p. 274. Mr. Rothwell advocates this filter which works satisfactorily. It is one of his inventions and we believe it is considered worthy of universal adoption, although the mechanical construction seems to be faulty on first appearance; but when the barrel has been in motion a while the swash, which must be considerable at first, will probably be overcome by the ore remaining against the inner periphery of the cylinder. The question of time *versus* economy in filtering is one of considerable moment, and yet to be fully demonstrated: with slow filtering, time is consumed, but less cost entailed; with quick filtering, less time is required and more cost added to the process. The sand-filters require replacement as well as the filters placed in the barrel, but they are more readily repaired and at much less cost. They do not require power, but power after installation of a plant does not add much for filtering, and is not a separate expense; it belongs to the barrel.

The Plattner process requires a filter in the bottom of the tank, in which the ore is leached. Upon the floor of the tank, which is slightly inclined for drainage, are laid ¾-inch strips of wood about one foot apart. On top of these and at right angles, with one-inch spaces between them, 6-inch boards are placed. Upon this false bottom loose pieces of quartz rock or clean gravel are placed. On these finer pieces in layers until the top layer is clean fine sand. Upon this sand, at right angles to the false bottom, boards are laid quite close together, as from these boards the ore must be shovelled from the tank after leaching and drainage.

The same difficulties will occur with the use of this filter as in the previous mentioned sand-filters, which can only be remedied by shovelling out the quartz and sand and laying a new filter-bed.

CHAPTER VII.

PRECIPITATION.

THE liquor from the filter-beds is conducted or pumped to storage-tanks. If the liquor be muddy after passing the filter-beds, or if it contains impurities in suspension, it is treated with sulphuric acid to precipitate them in these settling-tanks. The time for the liquor to remain in these tanks will depend upon the clearness of the solution; not less than two hours and sometimes 24 hours may be required. These tanks are of sufficient size to hold the liquor from one day's run of the filters. The accumulations of liquor in these stock tanks is drawn off as desired for the precipitation-tanks.

These latter tanks are usually not over 4 feet deep and 6 feet wide by 8 feet long, and of sufficient number to allow the precipitate to settle three days. Each tank of the above size will hold liquor from 3 tons of ore, and consequently a 60-ton daily plant would require 60 such tanks and occupy a floor-space of 4320 feet, allowing one foot between rows. This floor-space can be economized by making the tanks longer and wider, but they should not be made deeper. With vats 12 × 16 feet, or 6 × 32, or 8 × 24, the floor-space required will be, with one foot between tanks, 3600 square feet. When made deeper than 4 feet cleaning up becomes unhandy.

Precipitation-tanks are lead-lined to avoid leakage, and lead-lining, not being very stiff, requires skilled work to avoid leakage when tanks are made large. After the precipitation-tank is nearly filled with liquor from the stock-tanks it is treated with ferrous sulphate, with the following reaction:

$$2AuCl_3 + 6FeSO_4 = 2Au + Fe_2Cl_6 + 2(Fe_2O_3.3SO_3).$$

The precipitated oxide of gold is a dark reddish-brown powder, brought about by an interchange of metals in the presence of oxygen. The ferrous sulphate (copperas) should be made fresh daily for use, it being converted by the oxygen of the air into various basic sulphates if allowed to remain exposed any great length of time. Its precipitating qualities will be weakened where $Fe_2(SO_4)_3$, or ferric sulphate, is formed in particular.

To manufacture ferrous sulphate ($FeSO_4$), scrap iron is thrown into a wooden tank, and sulphuric acid added until hydrogen is evolved. After standing some time the liquor is drawn off for use, and more water, iron, and sulphuric acid added for the next day's solution.

It requires about three days to settle the gold precipitate, which is formed in a very finely divided brown powder. The fluid in which the gold is suspended has a blackish-blue color by transmitted light.

To ascertain if precipitation be complete, the solution is tested every 24 hours.

The absence of chlorine and a sweetish odor are fair indications of a complete precipitation, but for a certainty a small quantity of the liquor is stirred thoroughly in a beaker with $FeSO_4$. When the precipitate in the tanks has settled, the liquor above is either drawn or siphoned off, a fresh solution from the stock-tanks admitted, and again the copperas solution added. At stated intervals the settled precipitate is taken up from the precipitation-boxes and placed in smaller lead-lined boxes, where it is settled again, and what little liquor remains siphoned off. The filtrates are then well washed with boiling water until free from iron salts, after which they are collected on filters, dried, and melted into bullion. The bullion can be raised to 990 + degree of fineness where care is used in skimming. Borax and soda are usually used as a flux in this melting.

A more recent method introduced by Mr. Werner Langguth presents some advantages over the above method, which we believe will in time supersede the ferrous sulphate method. A description given by him of the process as carried on at the Golden Reward chlorination-works is found in vol. XXI, p. 314, of the A. I. M. E. Transactions. The disadvantages of this process are: Impure bullion, and extra work in refining on account of the gold precipitate being in the form of gold sulphide; also that other impurities are thrown down by the hydrogen sulphide used for precipitating the gold; and, finally, the intricate apparatus suggested, as shown in Fig. 6.

The advantages of this process are economy in space for precipitating-tanks; quickness in precipitating, and recovery from the auric chloride solution.

The trichloride of gold solution is pumped into a tank, *PPT*, which is raised about 25 feet above the filter-pump, *FP*, for head. We do not see the advantage of this arrangement, especially when compressed air is used. This tank has a capacity of 7000 or more gallons, say of a size 10 × 12 × 12 feet, made of strong two-inch pine plank, lined with light sheet lead. The generators, *G*, are two in number, made of boiler-plate, and capable of standing a pressure of 150 lbs. to the square inch. Their size is about 2½ feet in diameter by 4 feet high. The shape is cylindrical, with cylinder-heads securely bolted to the shell and provided with manholes. One generator is for the production of sulphurous acid (SO_2), and has an iron pan on a tripod for the reception of sulphur, which is burned in this generator to produce the gas. It is not lead-lined.

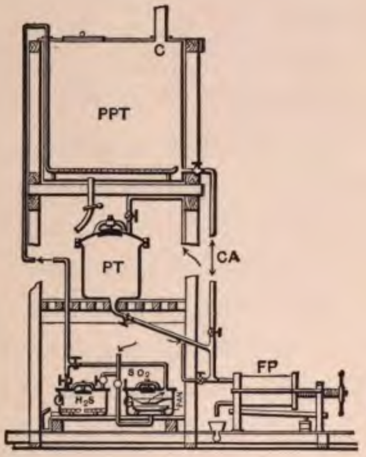

Fig. 6.

The other generator is for the production of hydrogen sulphide (H_2S), similar in size and construction, but lead-lined, and has no tripod and pan.

Both generators are connected with air-pressure, to force the gases into the precipitation-tanks, and also with a discharge-hole at the bottom for cleaning out the refuse after each run.

The pressure-tank (see Fig. 6, *PT*) is constructed of boiler-iron to withstand a heavy pressure. It is cylindrical in shape, size 4 × 4½ feet, inside measurement, fitted with manhole and proper threaded holes for receiving the pipes from the air-compressor, and also for pipes conducting the precipitants from the clean-up to the filter-press.

The liquor containing the precipitate is run into this pressure-tank, and from that into the filter-press, at stated times. The liquor freed from the precipitate or the liquor above it in precipitation, is run directly from the precipitation-tank into the filter-press as soon as the precipitate has settled sufficiently. A lead pipe leads up from the generators to this tank over its side and down to within four inches of the bottom, when it turns at right angles and runs across the tank. This pipe has holes on the under side about one-eighth inch in diameter and six inches apart. When air carrying the gases is forced through it, the gases rise up through the liquor, and by the agitation which it causes soon brings the liquor in contact with it.

The chloride of gold liquor having been forced into the precipitation-tank, roll sulphur is placed in the iron pan of the sulphurous acid generator and ignited. Compressed air for combustion and forcing the sulphurous acid gas into the tank is then admitted to the generator. The object of this gas is not to precipitate gold, but to destroy any excess of free chlorine which may be in the liquor. This generator forms sulphurous acid by oxidation of sulphur as follows:

$$S + 2O = SO_2.$$

The reduction of free chlorine then takes place rapidly, as expressed by the formula

$$SO_2 + 2Cl + 2H_2O = H_2SO_4 + 2HCl.$$

This changes the color of the liquor from that of yellow to clear blue in a short time, producing a white fog of sulphurous acid and chlorine gas, which is led up *C*, a ventilating chimney. When no trace of chlorine is detected (see tests for chlorine, pp. 63, 73) the gold is precipitated by sulphuretted hydrogen (H_2S) formed in the other generator by the action of sulphuric acid on iron matte (iron sulphide). Hydrogen sulphide gas is forced up through the solution by compressed air, as in the former instance. The reaction is expressed by the equation

$$H_2SO_4 + FeS = FeSO_4 + H_2S.$$

Before the acid is put in the generator twice its bulk of water is added.

The precipitation of gold by the hydrogen sulphide takes place according to the following reaction:

$$2AuCl_3 + H_2S + 2H_2O = 2Au + 6HCl + SO_2.$$

The solution is tested in about one hour to ascertain if all the gold is precipitated by filtering the liquor to be tested through filter-paper, and adding a few drops of $FeSO_4$. When no precipitate is formed on the test liquor the solution is allowed to stand two hours, and then drawn off to within 4 inches of the bottom of the tank, and passed through the filter-press.

The precipitation will take about one hour and the settling two, the time required to pass through the filter-press about three, or a total of six hours to precipitate and collect the gold. When this is compared with three days, the saving in time is noticeable. The tank of 7000 gallons capacity is capable of holding the filterings of 20 tons ore where 350 gallons of water is added in leaching and washing.

The clean-up occurs once or twice a month, when the collected filtrates from the bottom of the precipitating-tank are passed into the pressure-tank, and from there into the filter-press, where they are dried by compressed air. These filtrates contain impurities, which will require careful roasting and smelting to give a bullion from 800 to 930 fine gold. Mr. J. E. Rothwell has precipitated gold from solutions carrying copper by this method without precipitating the copper, which is almost invariably precipitated from such treatment in the laboratory.*

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* **Potassium nitrate precipitates metallic gold in solution. Potassa**

Precipitation becomes difficult when impurities are in the solution, such as lime and magnesium, which give bulky precipitates.* This can be obviated by the addition of acid, or by filtering through charcoal. This latter process is carried on as a method for preventing the bulky precipitates formed by ferrous sulphate from mixing with the gold. Just what the reaction is between pulverized charcoal and trichloride of gold is unintelligible, but the charcoal is certain to quickly decompose it, and collect the gold in and upon its surface. If the lime, magnesium, or arsenic be fixed or held in solution by acid, they pass through at times without decomposition, but not always. In the latter case the utility of charcoal as a precipitant over ferrous sulphate is nothing—in fact, as it adds new complications, is a hindrance, and to be avoided. There may be more or less slimes as well which pass the filter: these collect mechanically upon the charcoal, and when burned with the charcoal form with the ash a mixture which requires considerable extra work to get rid of; then, again, at times the lime and magnesium forming a precipitate as well give virtually a new gold ore, difficult to handle.

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or soda when added in excess will leave the solution clear, but on being warmed and tannic acid added, will separate the gold (Johnson's *Fresenius*, p. 191).

* *E. & M. Journal*, vol. LX. p. 323.

The liquid precipitants have all the same disadvantage, viz., the trouble necessary in collecting the precipitated metal; but gold does not, as a rule, deliver itself up from ores as bullion without trouble, and that is to be expected. Mr. J. T. Blomfield suggests the following plan for gold precipitation:

He forms subsulphide of copper by fusing sulphur and copper together (Cu_2S). This is crushed to pass a 100-mesh sieve, and is then used as a filter substance.

The liquor, filtered through copper sulphide, showed no gold present upon testing with Fe_2SO_4.

He suggests three vessels containing this precipitant: the first to catch the gold, the second to catch any gold which may have passed the first, and the third as a possibility.

The first vessel, Cu_2S, becomes fully charged with gold before any traces appear in the second. This vessel is then removed, the second moved to take its place, and the third the second's, while a new vessel is added for the third.

The residue is fused with nitre and borax. The reaction he gives as follows:

$$3Cu_2S + 4AuCl_3 = Au_2S_3 + Au + 6CuCl_2.$$

This equation should be changed somewhat, as the reaction recorded is hardly possible, since gold will not unite with sulphur in the cold, and under such conditions as given. The following is therefore advanced as the probable reaction:

$$3Cu_2S + 4AuCl_3 = 4Au + 6CuCl_2 + 3S.$$

CHAPTER VIII.

REFINING PRECIPITATES.

To obtain the metal precipitated in a pure form, that it may be handled in bulk of a known value, it is reduced by heat, which operation is termed refining.

The brown precipitate of gold produced by the action of ferrous sulphate upon the trichloride of gold is collected, washed on filter-paper, then treated with acid to remove any possible impurities, dried, placed in a graphite crucible with some flux — either soda or borax, and then placed in the furnace to melt.

Gold bullion is said to be fine when absolutely pure; it is then termed 1000 fine. To obtain such bullion requires an artist and very fine manipulation; to obtain it above 900 fine is not as difficult, especially from the precipitates of ferrous sulphate; but from the precipitate given by hydrogen sulphide the task is more difficult and the bullion less pure, ranging in fineness from 800 to 950, while in the former case it may be obtained with less trouble from 950 to 990 fine.

The reddish-brown precipitate of metallic gold obtained by this operation is washed with warm water and filtered to remove any acids or salts of iron present, or any other substances soluble. If insoluble salts of metals or substances are present, such as arsenic, zinc, copper, silver, sulphur, they can only be removed by acids to reduce them, and then evaporation and washing, or by an oxidizing roast, as practised at the commencement, but on a smaller scale, in a muffle.

The gold precipitated by $FeSO_4$ is placed in a graphite crucible, and with a little borax added is introduced into the furnace. The oxygen is driven off by the heat, the gold gradually melting, while the borax forms a liquid slag upon the surface of the molten metal.

This slag has absorbed the impurities which were not soluble in water, and as the slag is lighter than gold, by carefully skimming the surface it may be nearly all removed.

The oxide of gold is a peculiar compound, but little known or understood. Gold when heated may be deprived of oxygen, but it will not unite with its own or other oxides. Gold is but little volatile, and may be exposed to a strong heat for some time after melting; in fact it is frequently boiled. With nitre the boiling is so marked that larger crucibles are required as well as covers to keep it within bounds. The opposite is the case when volatile metals are combined with gold. They when driven off by heat carry gold with them; noticeably is this the case with the metals zinc, lead, arsenic, antimony, bismuth, and tellurium. To obviate this as much as possible the gold must be kept below a boiling-point, and be protected by slag, formed from a suitable flux. Gold with pure borax as a flux assumes a whitish color in melting; with saltpeter or common salt it retains its rich yellow color.

The heat should at first be mild, but may be gradually increased, and maintained during the melting, and for some little time afterwards. The crucible with its contents is then removed, and the gold poured into moulds to form ingots of gold. These ingots are sent to the nearest assay-office (Government assay-office wherever possible), tested for fineness, and sold.

The assay-offices do not like to accept base bullion on account of the trouble they encounter in refining it, nor will they pay as good price for impure bullion on account of the extra work and chances for loss in further refining.

Mr. Langguth having introduced the improved method of precipitating referred to in the last chapter, whereby he obtained auric sulphide, was obliged to refine these sulphides after collecting them from the filter-press.

Hydrogen sulphide precipitates gold from neutral or acid solutions. From the cold solution the precipitate is Au_2S_3; from boiling solutions, Au_2S. These precipitates are insoluble in acids, but soluble in "aqua regia"—a mixture of nitric and muriatic acids. If we should use the latter method we would virtually have the same experience to go through with as were we to use $FeSO_4$, and in the end must use the latter precipitant, although the liquor to be treated would be much less in quantity.

The aurous sulphides may be dissolved in yellow ammonium sulphide or by yellow potassium or sodium sulphides, but there is nothing to be gained by so doing.

Mr. Langguth places the precipitates and the filter-cloth in pans, which he introduces into muffles in a roasting-furnace, to drive off the sulphur, arsenic, and antimony, or whatever other volatile substances are present.

This oxidation can be accomplished in about three hours, when the mass presents a reddish or yellowish-brown appearance. If salt roasting could take place without loss of gold, purer bullion could be obtained but as it is, care has to be used in first heating to avoid loss of gold by volatilization. These roasted sulphides are now removed from the muffle, and being hard are placed in a pulverizing drum with cobble-stones to assist pulverization, after which the flux is added in suitable proportions and thoroughly mixed with the gold sulphides.

If the ore treated has been silicious or acid, the flux is borax or soda; on the other hand, if it has been basic, a silicious flux of sand or quartz is added. The gold mixture is now placed in a crucible, and this and its contents placed in the melting-furnace. The sulphur remaining from the roast is partially driven off by heat and absorbed by the flux, leaving metallic gold as follows:

$$Au_2S_2 + 4O = 2Au + 2SO_2.^*$$

After melting, and being kept at a high temperature some time, the crucible and contents are removed from the furnace and poured into conical moulds of suitable capacity. The bullion separates from the slag in these moulds as a conical button in the bottom, from which it is taken when cool, remelted, and cast into ingots for shipment.

The slags contain considerable gold, and are therefore pulverized and separated by water, the gold being

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* The author quotes Mr. Longguth's equations.

added to the next melting, while the tailings are mixed with lead and metallic iron, and melted in crucibles. The lead bullion resulting is cupelled, yielding the remainder of the gold. The slags from the second melting are too poor in gold to handle.

No serious losses occur by this method, but slight losses do occur, and from what has been said previous are due to vaporization and to mechanical agencies.

CHAPTER IX.

RÉSUMÉ OF CHLORINATION.

PLATTNER'S application of chlorine gas to cold gold is the foundation of the process. Mr. G. F. Deitken was the first to make a practical and commercial success of the process in the United States, at a mill situated in Grass Valley, California. The pioneer works of the East were under the charge of the present editor of the *Engineering and Mining Journal*, Mr. R. P. Rothwell; they were situated at Deloro, Canada, and were worked by the Mears process. The pioneer works of the South were at Haile Gold Mine, South Carolina, where the Theis method was brought out. The pioneer works of the West, as far as barrel chlorination is concerned, is the Golden Reward Chlorination Mill, Deadwood, S. D.

The tank-lixiviation seems to have made way almost entirely for the barrel-chlorination process, and the Mears for the Theis.

The barrels were originally constructed to contain one ton, but are now made to hold as high as ten tons, the latter size being several hundred dollars less in cost than two smaller barrels of the same capacity. They are also just as efficient and are more readily operated, requiring also less space than two separate barrels of the same capacity.

The improvements which Mears made over tank-chlorination were chiefly those which economized in time. He based his process upon pressure, to force the chlorine to attack the gold and, further, to mix and turn up the ore; that the chlorine gas might have more opportunity to do so, he rotated the barrel. By these improvements he reduced the time from three days to six hours, and also the attendance proportionally. Mr. Theis found the Mears process too cumbersome, and introduced the chemicals into the barrel, which not only generated the gas, but produced pressure sufficient for all purposes. This was a most important improvement, since it overcame trouble and expense engendered by the generation of gas outside the barrel. It also overcame several mechanical defects incident to such generation, as well as doing away with the pressure-gauge and its leaky goose-neck appendage.

Mr. Rothwell's addition—the filtering from the barrel under pressure—was the next improvement; and finally may be added Mr. Langguth's precipitation process.

These improvements have increased the capacity of the plants from 5 to 150 tons daily capacity, and at the same time decreased the cost of operating, and increased the extraction in a given time. This not only speaks well for the process, but certifies its usefulness. The original Theis plant is still in operation, and we are informed has paid over $500,000 in dividends from low-grade sulphur ores.

The mill site and locality will determine the arrangement of the mill, machinery, and labor-saving appliances to be adopted.

The advantages of a hillside are not so great as would warrant the building of a long tram-road to obtain a gravity fall, as in gold or silver milling, since the arrangement must be as automatic as possible to avoid elevating machinery. But in our case the ore will have to be conveyed to the dryer bin either on a level or elevated, and from the dryer to the rolls and from them to the concentrators, and finally to the roasters, cooling-floor, and leaching-barrels.

The preference would be a crusher which can handle 30 to 40 tons per hour, this being passed over $\frac{3}{4}$-inch grizzlies to two smaller crushers capable of handling the product—say 100 tons daily for the plant. Of the 100 tons passed to the large crusher 30 per cent will be of a suitable size for the dryer.

This relieves the small crushers of 30 tons of ore. The small crushers will produce 30 per cent of ore which will screen ¼ inch, and thus relieve the dryer of 21 tons daily, for the moist ore will generally be found in the first crushings and screenings. The small ore from the second crushers can go direct to the rolls; the finer ore to the dryer will also be treated quicker; and that dried ore mixed with the fine direct from the crushers will screen readily. In some cases by drying ore of small size thoroughly it has been found that ore crushed to ¼-inch mesh will pass the rolls and screen as high as 70 per cent through a 20-mesh screen. The advantage of dryers and the fine crushing thus becomes doubly apparent. The finer the ore delivered to the dryer the quicker it is deprived of moisture and the better the product of the rolls.

The ore which passes through the coarse rolls should screen at least 25 per cent 30-mesh, 25 per cent 1/16-mesh, and the balance between ⅛ and ¼. To throw ¼-mesh ore on a No. 30 screen is very destructive to screens; for this reason two screens are used, the inner being ⅛-mesh, the outer 30-mesh. The product passing the ⅛-mesh goes to the fine rolls, the balance back to the coarse rolls—possibly 25 per cent. The same proportion will probably hold true in the case of the fine rolls, 25 per cent being fine enough to pass the 30-mesh screen with one pair rolls, while 50 per cent would pass two pairs rolls with one elevating.

The use of two pairs coarse rolls and two pairs fine rolls is therefore economy in power, screens, and roll tires.

If concentrating machinery be used and ore crushed to 30-mesh separators and jigs, fine and coarse may answer; but as there is much ore above 30-mesh, it may be necessary to use separators and vanners in preference to jigs where the ore crushes easily.

In the case of clayey ores rolls will not answer, and either Chilian mills, Huntington mills, or stamps must be used for crushing, and this is followed by vanner concentration.

If the gold be coarse and can be saved in part by amalgamation, Huntington mills or stamps may be used.

If fine crushing is required, Chilian mills or stamps must be used, the limit of roll crushing being 30-mesh screen.

The power will consist of a first-class automatic cut-off engine 150 H.P. being sufficient for a 100-ton mill. This should have feed-water heater in preference to condensing apparatus, unless water is very valuable and needs to be economized. There should be an electric-light engine, and a smaller auxiliary engine to run the chlorinators and furnaces at night, in case all the power were not needed. This latter, while not necessary, will be found to be economical in fuel.

The boilers and feed-pump are a part of the plant. Flue-boilers are not as economical as tubular boilers, although the latter require more attention.

The dryer is a cast or wrought iron revolving cylinder, similar in construction to furnaces, with cast-iron shelves which pick up the ore and shower it through the flames. They are either one diameter the whole length, with jack-screws to elevate one end, thus giving a quicker or slower discharge as required; or they are made larger at one end than the other, to work the ore forward as it is dried. The dryer is rotated on rollers, similar to revolving roasting furnaces.

The ore is fed in at one end and is dropped into a pit at the other, from which it is conveyed automatically by elevators or scraper lines to the pulverizers. The storage-tanks and other appliances of the plant have already been mentioned.

The laboratory is a necessary appendage, which should be fitted up with assayer's outfit, also a roasting and smelting furnace, and in case of amalgamation an amalgam retort.

Pumps also may be required about the plant, and a storage-room for chemicals as well.

The screens are either hexagonal or round, made in sections that wire cloth may be readily replaced in frames which fit to the screen, spiders, and frame. They are of two compartments, as mentioned, the inside screen being heavy wire netting or metal plate, with holes punched in it. The outer screen is wire cloth; at least one foot should be allowed between the inner and outer screens.* A Root blower will probably be required to keep down the dust from the rolls and screens.

Salt-storage bins should be in the vicinity of the furnaces where chloridizing roasting is practised, and these bins should be of sufficient size to carry a good supply unless it can be obtained readily. Automatic feeders should be used for crushers and pulverizers; also automatic machinery, as far as details will permit.

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* The Berthelet separator is advanced by H. F. Brown as a substitute for revolving screens.

CHAPTER X.

COST OF CHLORINATION.

THE factors which enter into the cost of the process preclude giving a cost which will cover every case.

a. These are cost of mining and transportation to the mill—two items which will vary according to the character of the rock, the depth of the mine, the water encountered, and the distance of the mine from the mill. This cost will be increased or decreased by the above, and further influenced by the situation of the mine and mill with reference to the transportation of supplies and product from railway connections.

b. The cost of milling enters into the calculation, which depends upon the character of the rock, the ease with which it is crushed, and its arrangements in detail for close and automatic work.

c. The cost of concentration and the amount of concentrates saved, together with the degree of care taken to obtain clean concentrates for roasting.

d. The character of the furnace, and the degree of desulphurization dependent upon the amount of sulphur in the ore, or other substances which need to be eliminated, determine the cost in this department, which must necessarily differ in different localities or for different ores in the same locality.

e. The cost of chemicals will vary according to the facilities by which they may be obtained at the mill, and will also vary for different characters of ores, thus making their cost indefinite. The labor required in the laboratory for precipitating and refining is also an indefinite consideration.

f. The superintendent's and office expenses will vary for each mine, a good manager generally, however, saves his wages several times over during the year.

Much is said about this process for low-grade ores; and for the benefit of those contemplating the use of this process on low-grade propositions we would suggest they advise with a mining engineer before so doing, and not to attempt it on ores carrying less than $6 per ton of gold if the mine be not well developed, in which latter case with favorable circumstances the process should be remunerative.

Mines which are but partially developed do not offer enough stoping ground for steady supply of suitable ore, and development is always more expensive than mining. With mines well opened on plenty of stopes the mining expenses are reduced, and more than cover the cost of development. The average cost of mining will be about $1.45 per ton where power drilling, hoisting, and pumping is carried on, which figure also includes timbering.

The dumping and tramming to the mill, and the dumping there, will add ten cents per ton to the above cost.

The milling will average on hard rock where stamps are used $1, and the concentrating and the handling of tailings and product about 50 cents more.

After concentration the cost of roasting may be considered, and this will vary from 30 cents in the Cripple Creek district to $3.75 in others. At the Haile Mine, where wood and labor are cheap in comparison with other localities, the cost is placed at $2.62, for a ton of roasted concentrates, oxidized to a "dead roast" so that but 0.25 per cent of sulphur remains in the ore.

Then in order comes the cost of chemicals, which will vary from $2 to $3 per ton of concentrates, while the labor and other expenses will add about $5 to it.

In taking these different items up in detail, it must be borne in mind that we are dealing with concentrates, and, further, roasted concentrates.

If we have an ore running $6 per ton as it comes from the mine, and we can concentrate 90 per cent of the value, the ore should assay $5.40. If the ore runs 33⅓ per cent in sulphur, one ton of roasted ore will require 1.3 tons of concentrates—possibly more; or 13 tons of raw ore will make 1 ton of roasted concentrates. The value of the roasted ore will also be raised, by its quantity being lowered, provided no loss occurs other than mentioned, the value of 13 tons mine ore; 1⅓ tons of raw concentrates or 1 ton of roasted ore, will be $70.20.

With miners' wages at $2.50 and coal at $5, if 2.5 tons of ore are drilled, shot, cobbed, loaded, and hoisted per man employed at the mines the work is excellent.

The cost then would be about as follows:

Labor	$1.00	per ton.
Powder, fuse, and caps	.15	" "
Fuel	.15	" "
Timbering	.15	" "
Supplies	.10	" "
	$1.55	

Or $19.15 for one ton of concentrates.

Adding 10 cents per ton of mine ore for transportation, which includes wear and tear on rolling-stock and motive power, oil-waste, runners and dumpman's wages at the mill, the cost for 13 tons is $1.30.

With wood at $2.50 per cord and labor $2.00, the cost of roasting one ton of concentrates would be—

½ cord wood at $2.50	$1.25
12 hours' labor at 20 cents	2.40
Motive power	.35
Cost of roasting 1.3 tons concentrates	$4.00

The cost of milling would be, with 60 stamps, in hard rock:

Labor	$0.35
Supplies	.02
Fuel	.25
Machinery	.10
Oil and waste	.03
Illumination	.02
Lumber	.02
	$0.79

Or for 13 tons $10.27.

The cost of concentration would be about as follows:

Labor	$0.26
Repairs	0.10
Supplies	0.10
Fuel	0.15
	$0.61

per ton, or $7.93 for 13 tons ore.

The chemicals may be more or less than the quantities given; we have endeavored to take an average:

30 lbs. chloride of lime at 3 cents	.90
35 " sulphuric acid (H_2SO_4) at 3 cents	1.05
10 " H_2SO_4 for settling-tanks	.30
10 " H_2SO_4 for ferrous sulphate	.30
40 " salt at ½ cent	.20
Labor of 2 men at $2	4.00
Labor of 1 man at $3.50	3.50
Motive power	.35
Laboratory expenses	1.50
Cost of chlorinating	$12.10

We have not added the proportional part of superintendent's and office expenses, which would reach 10 cents per ton raw ore, or $1.30 per ton of concentrated roasted ore. There is still another item, such as amortization, which may be placed at 25 cents per ton of concentrates roasted. Our cost is then as follows:

Mining 13 tons ore	$19.15
Transportation to mill, etc.	1.30
Milling	10.27
Concentrating	7.93
Roasting	4.00
Carried forward	42.65

Brought forward................	$42.65
Chlorinating and recovery	12.10
Superintendent and office expenses..	1.30
Amortization.....................	3.25
Total cost of chlorinating one ton roasted ore..................	$59.30

Or per ton of mine ore $4.56.

The total value of the ore concentrated is $70.20; as we cannot extract all of this value, only about 92+ per cent, we have for a margin of profit the difference between $64.58 and $59.30, or $5.28—about 40 cents per ton of ore mined.

This margin we consider too small for mining and milling for safe work, as it does not allow 10 per cent for contingences and requires that 100 tons be mined daily for 155 consecutive days before the interest on the investment, if it be $100,000, is paid. With a free-milling gold ore this profit could be increased from $52.80 on a 100-ton daily mill to $116, and only extract 60 per cent of the gold; the remainder in the tailings could then undoubtedly be concentrated and treated at a profit sufficient to warrant the expenditure of the money on the additional plant.

The first cost of installation is high, because of the necessary crushing, concentrating, roasting, filtering, and precipitation requirements. A complete concentrating plant for 50 tons daily, which includes crushing plant, will cost, according to its elaborateness, between $30,000 and $40,000.

For chlorination must be added the furnaces and other paraphernalia equally expensive, which with buildings will bring the cost up from $60,000 to $80,000. The process is not able to recover coarse flake gold, and does not recover silver, but under certain conditions it is the best process known. It has in its favor the treatment of high-grade sulphurets with a high extraction. In this respect it is superior to cyanide, even when the ore is roasted for both processes. It covers a field distinctly its own, and has certainly come to remain as one of the metallurgical operations of the future.

In some instances tailings can be treated at a very low figure; in others the cost of roasting varies from 30 cents to $4.60 per ton, and the chlorinating from $2.18 to $4.60 per ton of concentrates. The Golden Reward has reduced the cost to $3, while the Haile Mine cost has been reduced from $4.84 to $3.50 per ton. We may expect, therefore, greater reductions when chemicals and roasting are reduced in cost.

For the benefit of those who inquire why this process has not been more generally adopted, it should be stated that those who run mines are not capable of carrying it on, and, knowing their weakness, let it alone; again, parties having mining machinery to sell advise the use of their machinery rather than chlorination; and, lastly, mine owners, thinking they will have to pay higher salaries to good men, are willing to suffer loss of gold rather than do so. Other matters, such as inability to purchase the necessary plant, must be considered as a further hindrance to its introduction. For that reason custom mills do a fair business in some localities.

Wherever chlorination has been introduced it remains, thus testifying to its worth as a means for extracting gold from refractory ores.

FINIS.

INFORMATION.

WATER FOR MILLING PURPOSES.

For stamps, 1.2 to 3 gallons per stamp per minute.
" plain-belt vanners, ¾ to 2 gallons per minute.
" corrugated-belt vanners, 1½ to 3 gallons per minute.

NOTE.—To the vanner water, which is clear, must be added the pulp from the stamps.

For boilers, 8 gallons per horse-power per hour.
" each settler, 1 gallon per minute.
" " pan, 2 gallons per minute.
" chlorinating barrel, 80 to 100 gallons per ton of ore.
Wash-water, 200 to 300 gallons per ton of ore.

Of the above water, 50 per cent of that used for engine can be condensed and used over again.

Of that water used by stamps, settlers, and pans, all but 20 per cent may be settled and used over again.

The chlorinating and wash water cannot be used again in the leaching or washing process where ferrous sulphate or hydrogen sulphide is the precipitating agent, but it may be used when charcoal is the agent.

METRIC CONVERSION TABLE.

7000 grains = avoirdupois pound.
5760 " = troy pound.
Av. lbs. × 1.21527 = troy pounds.
" ozs. × .9115 = " ounces.

One gramme = 15.433 grains.
One kilogramme = 2.2047 pounds av.
One " = 2.6778 " troy.

Meter = 39.3710 inches.
Millimeter = .03937 "

		1 inch	=	.0254 meters.
1 ft.	=	12 inches	=	.3048 "
1 yd.	=	36 "	=	.9144 "
1 fathom	=	6 ft.	=	1.8287 "
1 pole	=	$16\frac{1}{2}$ "	=	5.0291 "
1 furlong	=	660 "	=	201.16 "
1 mile	=	5280 "	=	1609.351 "

To change millimeters into decimals of an inch, multiply the number of meters by .03937.

$\frac{1}{64}$	of an inch is, in decimals,	.015625
$\frac{1}{32}$	" " " " " "	.03125
$\frac{1}{16}$	" " " " " "	.0625
$\frac{1}{8}$	" " " " " "	.125
$\frac{1}{4}$	" " " " " "	.25
$\frac{1}{2}$	" " " " " "	.50

THE THERMOMETER RULES FOR CONVERTING FAHRENHEIT INTO CENTIGRADE OR RÉAUMUR.

Water boils..	Fahr. 212°,	Cent. 100°,	Réau. 80°.
Ice melts....	" 32°,	" 0°,	" 0°.

$212^\circ - 32 = 180$ and $\frac{180}{100} = \frac{9}{5} = 1.8$ C., or $\frac{180}{80} = \frac{9}{4} = 2.25$ Réau.

Temp. Fahr. = $(\frac{9}{5} + 32)$ Cent. or $(\frac{9}{4} + 32)$ Réau.
" Cent. = $(\frac{5}{9} - 32)$ Fahr. or $\frac{5}{4}$ Réau.
" Réau. = $(\frac{4}{9} - 32)$ Fahr. or $\frac{4}{5}$ Cent.

To convert Fahr. to Cent.: Subtract 32 and divide by 1.8.

To convert Fahr. to Réaumur: Subtract 32 and divide by 2.25.

To convert Cent. to Fahr.: Multiply by 1.8 and add 32 to the product.

To convert Cent. to Réaumur: Deduct $\frac{1}{5}$.

To convert Réaumur to Fahr.: Multiply by 2.25 and add 32 to the product.

To convert Réaumur to Cent.: Add $\frac{1}{4}$.

SCREENS.

Wire screens take their numbers from the meshes in the linear inch; slotted screens, from the number of needle used in punching the plates. The slots are usually ½ inch long, running diagonally, but can be obtained either running lengthwise or crosswise of the sheet.

Wire screens pass a greater product than slot-screens, but wear faster.

Number of Needle.	Meshes per Inch.	Width of Slot, Inches.	Equivalent Birmingham Gauge.
1	12	.058	$22\frac{3}{8}$
2	14	.049	$22\frac{3}{8}$
3	16	.042	$22\frac{2}{8}$
4	18	.035	$22\frac{3}{8}$
5	20	.029	$23\frac{1}{4}$
6	25	.027	24
7	30	.024	$24\frac{1}{2}$
8	35	.022	25
9	40	.020	26
10	50	.018	27
11	55	.016	28
12	60	.015	28

Higher numbers of wire screens can be obtained of the dealers.

COLORS EXPRESSIVE OF TEMPERATURES CORRESPONDING TO DEGREES FAHRENHEIT AND CENTIGRADE.

Faint red........	960° F.	515° C.
Dull "	1260° F.	680° C.
Cherry-red.......	1650° F.	898° C.
Yellow or orange .	2010° F.	1099° C.
White..	2370° F.	1299° C.

The following metals volatilize below red heat: Arsenic, at 356° F. or 180° C.; mercury, slightly at ordinary temperature, boiling at 680° F. or 362° C.

The following metals volatilize at red heat: Potassium, sodium, tellurium, magnesium, zinc.

The following metals volatilize at white heat: Zinc, antimony, lead, bismuth.

Metals which give up their oxygen to heat: Mercury, silver, gold, platinum, palladium, rhodium, iridium, and osmium.

Metals which retain their oxygen at high temperatures, and cannot be reduced by heat alone: Potassium, sodium, calcium, magnesium.

CONCERNING METALS.

Name.	Specific Gravity.	Chemical Symbol.	Atomic Weight.	Fusibility, Fahr.
Aluminum	2.6	Al	27.4	very high h.
Antimony	6.7	Sb	122	932°
Arsenic	5.7	As	75	400°
Bismuth	9.74	Bi	210	476°
Copper	8.8	Cu	63.4	1996°
Iron	7.6	Fe	56	2500°
Lead	11.4	Pb	207	594°
Mercury	13.58	Hg	200	− 39°
Zinc	6.8	Zn	65	770°
Tellurium	6.1	Te	128	600°
Silver	10.5	Ag	108	1950°
Gold	19.4	Au	197	2000°
Manganese	5.85	Mn	55	very high h.
Potassium	.865	K	39.1	136°
Sodium	.972	Na	23	194°
Platinum	21.2	Pt	197.4	very high h.
Palladium	11.3	Pd	106.6	high heat
Nickel	8.2	Ni	58.8	" "
Chromium	5.9	Cr	52.2	" "

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