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THESIS

for the Degree of
Metallurgical Engineer.

Metallurgical Operations Of Pyritic Smelting

as carried on

at

Ducktown, Tennessee.

Byron John Snyder.

June 1st. 1910.

Approved
D. Copeland.

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I also desire to thank Dr. G. R. Glenn, Pres., North Georgia Agricultural College for his cooperation.

I desire to extend my sincere thanks to the Director of M.S.M. and his Professors for their kind consideration.

Introduction,-

The Practice of Pyritic Smelting is a comparatively new method. It has made possible the extraction of metals from very low grade ores economically. In fact, the direct smelting of sulphide ores of any sort is a very modern metallurgy. This method is extensively used in the United States and British Columbia and will prove to be very valuable in the future developement of other regions.

Pyritic smelting may be considered as the oxidizing smelting of pyritic ores in a blast furnace with the view to producing matte which collects the values.

A short discussion of copper smelting in general, and especially pyritic smelting, will be necessary in order to make this paper of value.

There are several objects to be considered in pyritic smelting which are more or less dependent upon each other, which are as follows,

First:: It is desired to save a preliminary roasting of the ore by burning off as much of the sulphur as possible in the blast furnace and, at the same time, furnish enough ferrous oxide for the slag.

Second: To use as little carbonaceous fuel as possible for both chemical and commercial reasons and to use the heat of oxidation of the ore or material smelted to its greatest extent.

Third: Concentrated combustion at the tuyeres must be avoided, except just enough to give the necessary smelting heat. Very concentrated action will use up all the oxygen in one small zone. Gases after passing the concentrated zone of combustion have no further oxidizing power.

Fourth: To avoid all possible reducing action. In blast furnace smelting ordinarily coke and carbon monoxide are strong reducing agents therefore these are wanted present as little as possible. This end is accomplished by using the least possible amount of coke, keeping the smelting down at the lowest point so that carbon dioxide may be had instead of carbon monoxide.

Fifth: In pyritic smelting an excess of oxygen must be had in every part of the furnace. Theoretically speaking no coke added at all would promote this condition best, but this is never realized in practice. Furnaces however are run with a very small amount of fuel ; three to six percent is considered good work. The efficient use of the heat of ~~heat of~~ oxidation of pyrite is made to take the place of coke. As a matter of fact the same set of conditions that satisfy one practice of pyritic smelting will satisfy all of them, generally speaking.

Sixth: All fuel is bad that has carbonaceous matter in it.

Seventh: All cold air that is blown in the furnace must be heated to the kindling temperature and this takes fuel By blowing in preheated air this fuel would be saved. A hot blast has other advantages. It takes up more volume than a cold blast and that fact ensures a more even distribution of the oxidizing action or combustion and avoids a concentration. Hot blast makes the running of the furnace more regular and even and also tends to keep the smelting zone down at the tuyeres, where it belongs.. With difficult slags a hot blast is often a necessity. Cold air striking this sort of a slag at the tuyeres is very liable to chill it instantly. The theory and use of the hot blast is good,,especially if waste heat in some form can be used to heat the blast. However the waste gases from a copper blast furnace, especially a pyritic blast furnace,, do not carry a large amount of carbon monoxide and they cannot be burned directly as can be done in the case of an iron blast furnace in which there are a large amount of combustible gases.The waste gases from a pyritic smelting practice are mainly sulphur dioxide,which may be utilized in making crude sulphuric acid for fertilizer purposes.So if the blast in this sort of smelting is preheated the waste heat of the slag must be utilized which is very difficult, or it must be heated by means of an outside fuel which is not economical. It loses the advantage in the way of economy.For this reason it is not more widely used.

In pyritic smelting the most energetic oxidizing must be done before the actual smelting zone is reached or before the material becomes fused. The ideal material would be sulphides that are difficult to smelt and to hold their form until the actual smelting zone is reached. Then it fuses with the slag easily so as to not use too much heat.At the same time the charge should stay open and as loose as possible and should be run with a comparatively low ore column and a hot top. These conditions are hard to realize.

The fact that smelting is pyritic implies that iron would occur in the slag in a considerable percentage, so the slags in this sort of smelting are moderately fusible. The slags are usually high iron slags, hence moderately fusible. It is not uncommon in pyritic smelting to be necessary to add a barren siliceous ore or some ore that contains a large amount of silica. This makes the slag more infusible. As a matter of fact this sort of smelting can be done with a difficult slag, that is, a broad range of slags may be had in pyritic smelting.

A large amount of low pressure blast is used which requires very large tuyeres and permits of rather wide furnaces if the material does not contain too much fines. A furnace is wanted as wide as possible at the tuyeres so as to avoid concentrated combustion.

Theoretical pyritic smelting is not possible, however it takes very little extra heat to accomplish it.

A few words in regard to copper slags will not be out of place.

Copper slags can have a very wide range of the silica content. As long as there is sulphur present in the copper charge the copper will join with the sulphur perfectly and will not form a silicate. There also need be no fear of volatilizing the copper by having slags of difficult fusibility. There are no limitations in a copper slag as in a lead slag. The silica in a copper slag can run from twenty five to over fifty percent.Also it is not entirely necessary to have large amounts of iron present as ferrous oxide, as with lead. There are copper blast furnaces making slags with the ferrous oxide as low as four percent, or in other words there is an unlimited swing as most any material may be blasted in a blast furnace. The sulphur formed in this case is a "protecting agent" for the copper.

For several reasons copper ores are not roasted completely to metallic copper and not to matte. The slags would be too rich in copper therefore too expensive.They would also be very difficult to roast completely.

The first question that comes up in smelting a sulphide ore for copper is, what concentration can it be made? Or, in other words, how many tons of ore can be put in to one ton of matte. The greater the concentration the lighter will be the cost per the original ton of ore for treating further the one ton of matte. If twelve tons of ore are smelted and eleven tons of this ore can be dumped as slag and have all the values in one ton of matte, the cost of bringing that matte will be divided by twelve when referring back to the original ore.

The principal factors in determining the amount of concentration possible are the percentage of copper in the original ore and the percentage of iron and sulphur accompanying the copper.

Experience has proven in all sorts of matte smelting that it never pays to make the first matte too high in copper as this implies too great a loss of copper in the slag and some other difficulties which are mechanical. The copper loss in the slag of matte smelting is invariably in the form of shots or pellets.

The process of converting the copper matte into metallic copper is a very important manipulation. A short discussion of this process will not be amiss.

If air is blown through hot liquid copper matte contained in a vessel lined with siliceous material certain reactions take place, on which the Bessemerizing Process depends. There are two stages, the slag forming stage and the blister forming stage. The slag forming stage lasts about forty or fifty minutes.

The first reaction that takes place is that the ferrous sulphide contained in the matte loses its sulphur according to the following reaction,

$$FeS + 3O = FeO + SO_2$$

and as soon as the ferrous oxide is formed it attacks the siliceous lining of the converter according to the following reaction,

$$2FeO + SiO_2 = Fe_2SiO_4$$

During this stage this reaction continues until all the iron has been slagged, in the method just given.

The cuprous sulphide in the matte remains unchanged during this slagging. If any cuprous oxide is formed it is immediately converted into cuprous sulphide according to the reaction given below.

$$FeS + Cu_2O = Cu_2S + FeO$$

The different stages of converting may be told by the condition of the flame coming from the converter. The flame at first is usually blue tinged with a slight rose coloration. As soon as the reaction begins energetically (which is after a few minutes) and the flame has more body it has an apple green color. This green coloration becomes fainter as the end of the slag forming stage approaches. It becomes whitish as the matte gets nearly to the stage of white metal(pure cuprous sulphide) you see at intervals in the flame quick flashes of azure or purplish blue flame. When these flashes get to be of a certain frequency the converter is turned down and the slag of ferrous silicate is poured off. The matte appears asblack spots on a rabble pulled back and forth through the slag.If high grade they are active and dance back and forth ; it is a sign that the material is practically up to the white metal stage.When the slag has been skimmed off as clean as possible some scrap copper, high grade matte or white metal is sometimes thrown in and the converter starts on the blister forming stage.

When the converter is first turned up the flame issuing from it usually has a greenish-orange color and as the process continues and more metallic copper is formed the color of the flame gets darker, finally to a copper bronze color and more wavering.The above point of finishing is the most difficult to determine. It takes long experience to become an adept in determining this point correctly.

When the moment of finishing comes small particles of material strike the converter head and bound off. The flame appears as if it were filled with particles and resembles a gauze made of metallic copper. The converter is then turned down, the slag present usually is very stiff, so that the metallic copper will flow very easily from under it leaving the slag in the converter. If the slag is present in large amounts it is tapped out afterward, if there is only a small amount present it is left in the converter and fresh matte is poured in on it and the process starts all over again .

The reactions of the blister process are as follows,

$$Cu_2S + 3O = Cu_2O + SO_2$$

$$2Cu_2O + Cu_2S = 6\ Cu + SO_2$$

The above reactions continue until all the copper is in the metallic form. They are exothermic, this keeping the contents of the converter in a liquid form.

The slag formed during the first converting stage is a very fusible one. The siliceous lining of the converter is eaten away very rapidly. The slag contains high iron and is very liquid. It is very important during the latter part of this slag forming stage to not carry the blowing so far as to create any cuprous oxide that would go into the slag. The blowing should only be carried to the cuprous sulphide stage and not beyond this. If the slag takes up any cuprous oxide, which after accumulating a small amount, will react with any remaining cuprous sulphide and large volumes of sulphur dioxide will be liberated, metallic copper being at the same time precipitated, which will be so sudden and so violent that often a large part of the contents of the converter will be thrown out over the floor of the building, which is due to overblowing.

At the end of the slag forming stage, if the converter is not at the proper temperature it is regulated. If the converter is too hot or too cold during this blister forming stage it works slowly and the tuyeres are difficult to keep open. It being too cold it is natural that the reaction will lag, if too hot copper silicate forms and this is rather infusible and noses form over the tuyeres. A cold converter shows a peculiar brassy yellow coloration to the flame. This is remedied by throwing in cord wood or coal, if too hot, cold copper scrap is thrown in.

The linings of a converter must be siliceous Any siliceous material that will bind and stand tamping and that does not contain much sulphur will serve for a converter lining.

For the reason that ferrous sulphide is converted into ferrous oxide as the result of the blow it must have silica with which to combine. Were it not for its presence, an infusible mass of iron oxide would soon accumulate within the converter. It would not be possible to run with a basic lined or water jacketed shell. This has been several times undertaken, since it would save considerable expense and time lost in relining, but such experiments have always failed. The life of the lining is less as the grade of matte is lowered because with so much ferrous oxide coming from it, a

such converters are run

more basic slag is formed. The lining also falls off in lumps owing to the higher corrosive action. Thus, the necessity of a siliceous lining may be seen.

As above stated, with a low grade matte a large amount of slag is made and the lining is used up rapidly. If the material available contains gold or silver a very low grade matte may be an advantage as these metals will go in the copper and as the values from a large amount of lining goes into a very small amount of copper, hence its advantage. With a high grade matte the amount of iron is small, so that the gold or silver bearing lining loses its advantage largely, although these values are still clear gain.

There are several types of converters used to convert to metallic copper, which are, Upright, Barrel and Selecteur, the first two being the most used in this country.

The Upright converter is a very large one and is used very extensively in Bessemer steel practices. The largest ones of this type are eight feet in diameter and fourteen feet high. These require high blast pressure, fifteen pounds, which is one of its disadvantages. The first charge for this converter is about five tons of matte and each charge following is increased. The advantage of this converter is that the lining may be put in very easily.

The Barrel converter is the most extensively used. Two sizes are in general use. The chief disadvantage is the difficulty of getting in a firm lining. A much lower pressure blast can be used, about ten pounds, which is one of its chief advantages.

The Selecteur converter is in successful use in France and is of the upright type with a belly in the side. When turned down the copper collects in the belly and is drawn off thro an orifice.It is blown until a small amount of copper is formed, again brought up to white metal, skimmed, above being unconverter matte, when turned down the copper collects in the pocket, blowing is resumed,etc.

In all large plants when converting is to be done the liquid matte is tapped direct from the blast furnace fore hearths into ladles and then poured by means of traveling cranes into the converters. This is called, "direct method of converting" and is very economical.In small plants where the output is not absolutely reliable in quantity it is not uncommon for the matte to be remelted in a reverboratory furnace which keeps it hot until it is needed.

The converter slag formed is usually pretty high in copper so it is resmelted in the blast furnace recovering most of the copper.

Situation, Character of the ore, etc. of the Ducktown Dist.

The Ducktown copper district is situated in the extreme south east corner of the State of Tennesee, in Polk County.The names of the two principal copper mines of thisdistrict are"The Burra Burra"and"The Isabella" owned by the Tennesee Copper Co. and The Ducktown Sulphur and Iron Company.

The ore of this district consists of sulphides of iron and copper, mainly iron sulphide. The deposits outcrop on a small upland plateau inclosed by higher mountains and made into trenches by stream channels. The country rock is a thinly foliated mica schist with occasional intercalated layers of gneiss. The rocks are commonly regarded as part of the "Ocoee" series of cambrian age, but they differ from the rocks of this series as exposed in the Hiwassee Gorge. The foliation runs north-northeast and the beds dip fifty degrees south east. The rocks consist of biotite and quartz, indicating a sedimentary origin. The ore consists of massive pyrrhotite, containing interspersed particles and stringers of chalcopyrite and pyrite, together with small amounts of galena and zinc. Calcite, zoisite, quartz and occasional bunches of garnet occur, with actinolite in some deposits. The copper content averages a little better than 2%, in some instances nearly 3%.

The deposits consist of long and exceptionally thick masses of solid ore, rarely with a few included slivers or horses of country rock. These deposits occur on the lines of fracturing and probable faulting and rock crushing. They are classed as replacement veins. Diagonal faulting of the ore bodies has also occurred. The composition of the ore of this district ~~are~~ is very constant.

Mine Equipment and Disposition of the Ore.

The surface equipment at the Burra Burra Mine of the Tennesee Copper Company consists of electrical hoist, trackways from mine to car dump, crushers, etc. The ore is hoisted on a cage in self dumping steel cars, being then crushed to the proper size (not larger than six inch), conveyed to the ore bins which supply the rail-way cars.

The underground equipment consists of pumps for pumping seep water from the mine, haulage systems, tracks, etc. This mine is a little ever 950 ft. deep.

The ore after being hoisted from the mine, crushed to proper size, etc. is dumped into railway cars and transported by railway three miles to the smelter plant, where it is dumped into storage bins of five thousand tons capacity. There are four rows of these storage bins, with a railway track over each, with compartments,into which is dumped ore, quartz, and coke, each into separate bins.

The ore after reaching these bins is in fit condition for direct smelting. These bins are elevated so that a narrow guage electric haulage system, with cars etc. may be run under them, into which is dumped the ore charge in the correct proportions for smelting. There are three scales on each track for weighing the charge. The quartz is dumped into the car first, the ore next, and the coke last.It is hauled directly from here to the charging floor,this track being continuous or in the form of a loop.

Metallurgical Treatment of the Ore.

Introductory Remarks,-

The metallurgical treatment of the ore at this plant is strictly a pyritic smelting operation.The green ore being directly smelted in a blast furnace in the first operation to a matte of about ten percent copper.This matte is then reconcentrated in a blast furnace to a matte which is about thirty or thirty five percent copper. The first concentration is about seven into one forming the above mentioned matte.

The ore, as will be seen in the following pages is of a very constant composition, so that the charge hardly ever need be varied.It contains on an average of about three percent copper and gold and silver values to only a very slight extent, so that the value of the ore lies in its copper content.

This practice is very unique in that an ore with such low values is smelted economically.

Forms of Furnaces used,-

The forms of furnaces used are the ordinary rectangular form of copper matting blast furnace. They are of two sizes, several larger ones and one small one for concentration purposes.The blast furnace and its accessory apparatus will be dwelt upon in more detail in the following pages.

General Smelting Operations,-

An outline of the general smelting operations will be given in this section.The ore charge is brought from the storage bins and dumped into the furnace

It being then smelted under pyritic smelting conditions to matte and slag, the matte and slag running continuously from the furnace into the settler. The slag is dumped on the slag bed by means of an electric haulage system. The first matte made being in the neighborhood of ten percent copper is run into a dump car and transported by means of an electric locomotive to a slightly inclined yard in front of the furnace building and there poured into flue dust beds. There are about fifteen of these beds in the neighborhood of twenty feet wide and sixty feet long. All of the flue dust made is utilized in this manner After it has solidified and cooled, it is broken by hand by means of an ordinary pick. It is very soft and crumbly and is easily broken up in this manner. It is then loaded into railroad cars and brought to the smelter bins. Pouring the hot molten matte on this flue dust makes the cooled matte also very porous and very acceptable for further smelting. Thus may be seen the first smelting operation in this practice.

The first matte produced is then smelted together with quartz, converter slag and cleanings from the furnace, so as to produce a matte of sufficiently high grade to convert commercially successfully. This is found to work best by making a matte of about thirty to thirty five percent.

As before mentioned the conversion of this low grade matte to a high grade one is done in the small furnace. This matte concentration process is much slower than the first process..

The matte and slag is tapped from this furnace also into a fore hearth or settler. The depth of the matte in this forehearth is ascertained by dipping an iron bar into it. The slag adheres to the bar and the matte does not but leaves only a darkened place. In this manner the contents or depth of the matte is determined so that the matte may be drawn off at the proper time.

This high grade matte is then tapped into a large matte ladle, which is electrically operated, and conveyed to the converter where it is converted to metallic copper.

The conversion of high grade matte to metallic copper is discussed at length under the topic of converting.

Operation of the Blast Furnace in general,-

The feed floor of the blast furnace is on the same level with the continuous or loop track upon which the electric haulage runs, from the ore bins. The ore is brought around to the feed floor in trains of four cars each.

The quartz is loaded in the car first, then the ore and the coke on top. These cars are self dumping and dump the charge into the furnace so that as it falls to the bed of ore it is approximately spread in the most beneficial manner for smelting. The charging was originally done by hand, then the coke only, was charged by hand, finally the whole charge is dumped in from the car. This has proven to be the most satisfactory manner and dispenses with the men on the charging floor, cutting out this expense.The charging tracks spoken of above run on both sides of the furnace so that it may be charged alternately on both sides of the furnace.

A car of charge consists of about two tons of ore, one-half tons of quartz, and 180 pounds of coke. This is varied somewhat, if crusts are forming the coke is slightly increased.

Great care is used in charging the blast furnace so that crusts do not form, it being carefully watched and at the first appearance of crusts the furnace man bars it off with a long iron bar.If the crusts are of too great a thickness to be removed by hand the attendant drives a long three inch bar down thro the crust and the electric locomotive is hitched to the bar and in this manner it is loosened. This is the greatest trouble to be found in running the blast furnace at this practice.

The depth of charge in the large furnace is about twelve or thirteen feet, this being found to give a better elimination of sulphur than a bed with less depth of charge.

There are six blast furnaces of the largest size, fifty-six by one hundred and eighty inches,each one smelting on an average of four hundred tons of ore per day of twenty four hours, or about 9500 tons per month.This allows for the furnace being out of commission part of the time for repairs.

The smaller furnace, which is forty-two by one-hundred and twenty inches smelts on an average of two hundred and seventy five tons per day of twenty four hours.

The blast pressure supplied to the furnaces consists of a pressure of 42 inches and is furnished by six Nordberg Blowers, each one supplying 22000 cubic feet of air per minute..

The Blast Furnace and its Accessory Apparatus,-

The blast furnaces used are the ordinary rectangular water jacketed copper matting blast furnace, with the exception that the cast iron brick lined crucible has been replaced by water jackets of chrome steel and cast iron sole plates.The sizes of the blast furnaces used have already been mentioned in the preceding pages.

The tuyeres used are tolerably large, there being twenty six on the rear side and twent four on the front side. They are about four inches in diameter.

The total height of the furnace from tuyere to feed floor is eighteen feet, of this fourteen feet are available for depth of charge, the top four feet being added simply to get proper distribution of the charge.

The charge doors of the furnace are about four feet wide and extend the whole length of the furnace so that large space of access may be had to the furnace in case the barring of crust formations becomes necessary.

The main difference in this furnace and other furnaces of like type is that it is practically water jacketed all the way down from the feed floor.

The matte and slag run continuously from the furnace through a water cooled spout into the fore-hearth for settling the matte from the slag.This forehearth is made of circular shape and is lined with chrome brick. It is sixteen feet in diameter and four feet high. This affords ample capacity for settling a very large amount of matte.

The slag runs from the fore hearth through a trapped spout to a slag car of about five tons capacity, which is tilted by means of a worm screw. This car is handled by an electrically driven locomotive to the slag dumps.

The matte is conveyed from the matte concentrating furnace by means of a large matte ladle handled by an electrically operated crane. This ladle is of cast steel and designed to give the least radiating surface to the molten matte. The trunnions are cast on the body and are made hollow to allow the insertion of a guiding bar for manipulating the ladle in pouring. A wide and high pouring lip is provided which allows a thick stream to be poured without spilling.

The gases and flue dust from the furnaces are passed through a large sheet iron pipe about eight feet in diameter, where the flue dust has a chance to settle and the gases may pass to the sulphuric acid plant,which is about two hundred feet away. The flue dust is cleaned from this receptacle at intervals,thro openings provided for this purpose.

Gases from the Blast Furnace,-

The gases coming from the blast furnace consist mainly of sulphur dioxide and nitrogen. The sulphur dioxide is used to make commercial sulphuric acid.

The method used for the manufacture of this acid is the old lead chamber process, without any improvements. A larger sulphuric acid plant is in erection for the utilization of all the sulphur dioxide fumes so that none at all will escape into the atmosphere.The building of this large plant has been made necessary by the action of the courts of Tennesee as the escapement into the atmosphere has entirely denuded the surrounding country of all plant life.This will, however, be controlled entirely when the new plant has been completed.No doubt the smelter was of more benefit to the surrounding country than all the farms in the vicinity put together, but this made no difference, so the new acid plant is in the process of erection from this cause.

A large stack six feet in diameter on the inside and 325 feet high has just been completed so that the gases that escape into the atmosphere will be sufficiently diluted so as to not prove harmful to the surrounding vegetation.

The commercial sulphuric acid made is used in the south to make fertilizer and has proven to be a very valuable by-product.It is shipped in steel lined tanks made on flat cars. It has been found that this commercial acid does not attack the steel tanks to any great extent. The tanks were formerly lead lined but this was found to be an unnecessary expense.The sulphuric acid made is 60 deg. Baume'.

Chemical Composition of the Ore Smelted,-

A chemical analysis, of the most careful manipulation, was carried out on the ores of the Ducktown district from different parts of the ore bodies and gave, the following analyses.

	#1	#2	#3	#4
Copper	2.70%	2.69%	2.29%	3.01 %
Iron	36.00	42.16	35.11	34.09
Sulphur	25.36	30.18	29.12	23.15
Silica	19.50	9.00	16.90	23.15
Calcium oxide	6.30	7.30	6.20	6.20
Magnesium oxide	2.60	1.30	2.10	2.40
Zinc	2.62	2.00	1.70	1.10
Aluminum oxide	.90	1.56	2.91	3.85
Manganese	.80	.70	trace	trace
Carbon di-oxide	3.10	3.25	3.65	3.00
Totals	99.88%	100.14%	99.98%	99.91%

The fire assay of several samples of the ore for silver and gold gave an average of 0.2^{0} to 0.3^{0} of an ounce for the silver and a trace of gold.

Qualitative tests for tellurium and selenium were made on several samples which gave strong traces in some others giving such as were almost blanks.

The methods of analysis used on the above samples were as follows,

The copper was determined by the iodide and cyanide methods.

The iron was determined by means of Stand. Sol N/10 $KmnO_4$.

The sulphur was determined Gravimetrically by weighing as $BaSO_4$.

The silica was determined grav. by weighing as SiO_2.
The lime was determined grav. by weighing as CaO also volumetricallyby means of N/10 Stand Sol of $KmnO_4$.
The magnesium oxide was determined gravimetrically by weighing as $Mg_2P_2O_7$.
The zinc was determined volumetricallyby a Standard solution of Potassium ferrocyanide.
The Aluminum was determined gravimetrically weighing as Al_2O_3.
The manganese was determined volumetrically by Volhard's method.
The carbon dioxide was determined gravimetrically by collecting in strong KOH solution and weighing.

The writer did not consider it of sufficient importance to go further into the chemical analyses than the mere mention of the methods used.

Action of Fluxes and Influence of Foreign Material.

The only flux added in this practice at this plant is barren quartz. This is added in sufficient amounts to produce a fusible iron silicate with the large percent of iron present, which is oxidized to oxide, then combining with the silica present produces a fusible silicate. This is mostly the singulo-silicate of iron. It seems that, with a given ore and a given furnace, the combination makes its own type of slag,from which it is impossible to make any radical changes. The slag from the first operation runs about 40% silica. Here it makes no difference whether the quartz in the charge is increased or diminished the slag remains about the same,40% silica,and the only change is the increase or decrease in the grade of matte produced.

The method of running the furnaces is to vary the quartz charge so as to keep a constant grade of matte. The first grade of matte as before mentioned runs about 9 or 10% ~~matte.~~ copper.

The zinc present in the ore tends to make an infusible slag but there being such a small amount present in this ore it is not a serious factor, it going partly in the slag as ZnO and partly in the matte and partly up the chimney, forming accretions on the walls of the furnace, condensing in the dust chamber, or escaping into the atmosphere.

The only other constituent of any extent is lime and this is rather beneficial as it forms a light slag which assists or allows the matte to settle more quickly and perfectly, also producing a tolerably fusible slag of bi-silicate form.

The small amount of manganese present undoubtedly forms a singulo-silicate and is also partly carried away in the slag and a very small amount going into the matte,also.

Fuels used in the Blast Furnace.

The only fuel of foreign addition used in the blast furnace in this practice is from 3 to 4% of the charge in the form of coke.This amount is sometimes varied but not to any great extent.The heat of the reaction of the oxidation of the sulphide of iron being relied upon to furnish most of the heat.

Chemistry of the Blast Furnace.

The chemistry of the blast furnace in this case of pyritic smelting is no different from that of other practices and as this has been dwelt upon in the preceding pages it will not be repeated here.

Furnace Products.

The furnace products consist of matte, first and second grade and the slags from these two, flue dust, and sulphur dioxide and will be considered in this order.

Matte, First grade and Analysis,-

This matte is very liquid and flows freely over the flue dust beds. An evolution of gas takes place when this hot molten matte comes in contact with the flue dust and sulphur dioxide gas ~~seems to~~ escapes freely.

A smaple is taken every hour from each furnace and a composite made at the end of the day from each furnace for analysis.

A very careful analysis of this matte was made which gave the results on the following page.

Matte Analysis,-

Copper	10.62%
Iron	54.12
Sulphur	25.02
Zinc	2.69
Manganese	0.28

As may be seen from the above analysis the copper runs a little more than ten percent. This value fluctuates from 9.25% to 10.50%.

Matte, Second, and Analysis,-

This matte is also very liquid. As in the first matte a sample is taken from the concentrating furnace every hour and a composite sample made at the end of the day. This ~~slag~~ matte on analysis gave the following results,

Copper	33.15%
Iron	34.09
Sulphur	26.01
Zinc	1.98
Manganese	.22

The second matte contains about 33% of copper, sometimes it is a little lower than this and sometimes it is a little higher, it how ever usually contains about 30% of copper.

Slag, First Matte,-

This slag upon analysis is found to contain very little copper. The samples are taken hourly, chilled and a composite sample of the days run is made for analysis. Once a week composite samples are made up from the daily samples from each furnace and assayed for copper by electrolysis. The electrolytic method gives a little higher results than the ordinary cyanide method.

The slag was analyzed by ordinary slag analyses methods, the copper being determined electrolytically. The following results were obtained from the slag of the first operation of matting.

Copper	0.23%
Silica	41.30
Ferrous oxide	40.10
Aluminum tri-oxide	5.50
Calcium oxide	7.20
Magnesium oxide	2.40

Slag, Second matte,-

This slag contains a slightly larger amount of copper than the slag from the first matte. There is a slightly larger amount of ferrous oxide present and a somewhat smaller amount of silica.The following results were obtained from an analysis of the slag from the second matte.

Copper	0.37%
Silica	38.40
Ferrous oxide	45.80
Aluminum tri-oxide	3.40
Calcium oxide	7.30
Magnesium oxide	1.90

Bessemerizing.

Number and kind of Converters used,-

The Bessemerizing plant at this practice consists of five ten-ton horizontal barrel-type converters, hydraulically operated. They are so made that they may be taken apart in two halves for the purpose of relining. The top is lifted off with the electric crane.The converter top is a steel casting with a removable cast iron snout. The top is reinforced by flanges on the sides. The lugs for lifting the converter shell or top off for the purpose of relining are made part of the stiffening flanges and are of sufficient strength for the two of them to carry the full weight of the shell loaded with its lining.The converter top is fastened to the body of the converter by cotter bolts to facilitate the opening and closing of the shell.

The blast pipe is connected to the converter by a flexible joint which allows for slight discrepancies in the alignment between the shell and the fixed blast pipe, also permits of the shell rotating.

The gas from the conversion of the matte is conducted through a five foot sheet iron pipe to the main chimney which finally conducts it to the sulphuric acid plant.

Composition of the Converter Lining and Method of placing,

The converter lining consists of waste furnace cleanings, old fire brick,which is crushed to sufficient fineness, so that with the addition of a small amount of clayey material for binding it may be used as such. A force of three men is kept busy all the time putting in new converter linings. The placing of the lining consists in balling up the material and placing it by hand and afterward tamping by means of a compressed-air driven hammer suspended by chain blocks. The lining is tamped to a very hard consistency by this machine and left to air dry for a short time after which time it is ready for tempering.

Tempering the lining,-

If the lining is not thoroughly tempered its life is lessened. The shell is dried out by putting in a fire of a little wood, followed by coke, a light fan blast being admitted at the tuyere openings. After the lining has been thoroughly dried in this manner it is theh ready for a first charge. The capacity of the converter is lessened by a new lining, which increases however as the number of charges increase as the lining is eaten away more each time. When the lining becomes dangerously thin the converter shell becomes red hot. Sometimes this spot can be kept cool by a stream of water from a hose, and the charge finished, but if not, the charge is poured out and transferred to another converter.

A lining lasts from five to eight blows as it has been found at this practice to be cheaper to renew the ~~charge~~ lining than to run a great number of blows.

Character of Matte and handling of Matte charged to Converter,-

The matte charged to the converter comes from the fore hearth of the concentrating furnace. This is transported in a large cast steel ladle lined with clay or loam and poured directly into the converter. The matte as it comes from the settler is in the proper condition ~~of~~ for conversion, it being at about the right temperature ,etc.

Length of Blow,-

The length of blow for converting the high grade matte to metallic copper at this plant is from fifty minutes to one hour. This is varied somewhat at times, it being controlled by the appearance of the gases issuing from the converter as before stated.

Disposition of Metallic Copper,-

The metallic copper is poured from the converter into molds of elongated truncated form, carried on a small railroad truck made of I beams. It is then allowed to solidify. The space between the beams is of such width that by simply overturning the molds the copper bars can be dumped between them, thus avoiding the necessity of taking the molds off the truck for the purpose of removing the bars. These ingots of copper weigh 200 pounds.

Disposition of Converter slag,-

As the separation of the slag and the values is incomplete in the converter all slag is returned to the blast furnace for resmelting, thereby saving the copper, which amounts to considerable.

Chemical Analysis of the Resulting Copper.

Sampling, Method of,-

The copper bars are sampled by boring with a $\frac{3}{4}$ inch drill, The first bar is bored about an inch from one end, the next bar about an inch farther on, and so on until the whole space of the bar has been utilized, each place being bored on a different bar. In this manner the bar is sampled. This has been found to give about as fair a sample as any other method that can be used.

A very careful electrolytic analysis gave the percentage of copper as 99.48% pure.The ordinary electrolytic method for copper was used, depositing the copper on a platinum electrode and weighing it as such.

Several fire assays were made on the copper for silver and gold, scorifying off the copper etc., which gave an average of 12.00 ounces of silver per ton and about0.018 ounces of gold.

The metallic copper also gave traces of tellurium, selenium and arsenic.

The method used in the chemical laboratory for these routine determinations on copper slags, etc. is the one mentioned above.The electrolytic bench is fitted up to make ten copper depositions at the same time.The apparatus has revolving electrodes of platinum. Another form of apparatus has been perfected by these people. The only difference in this method and the ordinary one is that a greater number of determinations may be carried on at the same time without so much first outlay

Conv. Blast

for apparatus, that is the electrodes used are small. This method how ever is a very much slower one than the first one mentioned.

The chemical laboratory seemed to be neglected but a new one is to be built in the near future to be equipped with modern apparatus and conveniences.

Most of the output of copper from this district ,the Bessemer copper, is exported to Europe.

The Copper Output of this District,-

The output in copper for this district for the last three years has been pretty constant. In 1907 the copper production was nearly 18,900,000 pounds and in the year 1908 the copper production was nearly 19,500,000 pounds. The copper product for this district during the year 1909 was ~~something like~~ about 19,000,000 pounds. The price of copper decreased so much during the year 1908 that the values for the year lessened something ~~like~~ over a million dollars.

The incidental production of silver and gold amounts to about $33,000 to $42,000 per year. Of this amount $3500 to $4,000 is given by the gold content, from Tenn.

Over 98% of the copper, gold and silver produced by Tennesee is from this district, in fact this is almost the largest producer of the eastern states, only one other of any importance; this is the Milan copper mines in New Hampshire.

The above production of silver and gold from this district perhaps might be increased if the copper were all refined in this country, as part of it is shipped abroad, this is lost to the statistics of the U.S.

Tennesee furnishes about two percent of the total production of the United States and ranks sixth in the amount of copper produced in this country.

The Isabella Plant.

The practice at this plant is also a pyritic smelting practice and consists of smelting the raw ore, which has the same composition as that found on the Tennesee Copper Company's property it being the same lead of ore, to a low grade matte of 20 to 22% copper in the first operation which is then reconcentrated to a 50% copper matte.

An ordinary Herreshoff furnace is used for the smelting here, which has a depth of about nine feet and has a cross sectional area at the tuyeres of about 22 or 23 feet. The furnace has a water cooled blast trapping spout and a brick lined settler of about 5 feet by 4 feet and 18 inches high inside dimensions.

The capacity of this furnace is about 60 to 75 tons of raw ore. This may however be increased under favorable conditions.

The blast pressure used is from 16 to 18 ounces and is furnished by an ordinary Connersville blower direct driven.

Incrustations form on the inside of the furnace, but these are found to not be very detrimental and are barred off occasionally.

The coke charged in the furnace is about 3 1/2% of the charge in the first operation and about 6 or 7% in the second making a total average of about 4%, which is about the same as that used in the previously described process of pyritic smelting.

The slag losses are about 0.3% in the first operation and about .5% in the second, which is also about the same as the slag losses at their neighbors plant.

The flue dust is utilized in the same manner as at the T.C.Co. practice, i.e. by running the molten matte over it. The flue dust made amounts to about 50 lb. per ton of original ore.

The slag and matte as it comes from the furnace enters the forehearth, flowing out one side of the settler is granulated by dropping into a launder supplied with a very forceful stream of water. It is then elevated and dumped into cars for dumping on the slag dump or shipping for rail road ballast.

The low grade matte is poured on the flue dust beds and after the proper accumulation has taken place it is reconcentrated in the Herreshoff furnace, it being used alternately, to a 50% matte which is not converted at this plant but is shipped for conversion.

The point of the greatest interest at this plant is the method of charging the ore in the furnace. The ore is shoveled into a car of about six feet in width and twelve feet in length ~~together with the charge~~. This car is fitted with a movable or sliding bottom which has flexible joints every five inches. This bottom moves over rollers placed at each end of the car. When the charge is ready to be dumped this car is moved forward and as it does so it pushes the horizontal sliding top of the furnace away as it moves on and at the same time the jointed bottom of the car moves and charges the ore in a very flat perfect layer. A charge is made every seven minutes. This is a very unique and advantageous method of charging.

The concentration at this plant is about seven to one by the first operation and two and one-half to one in the second. This is not a fixed limit as a lower grade matte has been concentrated to a 50% matte successfully. This is controlled by the operation of the furnace speed, blast, etc.

The sulphur dioxide fumes from the furnace are utilized in the manufacture of crude sulphuric acid, the lead chamber process being used.

Ore Charge Used.

The Eng. and Min. Jour. 1903,5 -2 contains a good form of calculation of charge used here which can hardly be improved upon. The quality and composition of the ore being so much the same, the same ore charge is used indefinitely with the exception of the amount of coke. This is doubled for an hour or two at the instance of the head furnace-man.

From the foregoing figures the charge is very easily calculated. The slags made are found to agree very well with the calculated ones. It may be tabulated in the following manner, which is given on the following pages.

Analyses of First Operation.

Materials Smelted.

Constituents	Ore	Quartz	Slag.Chg.	Coke
Copper	2.744	.....	.73	......
Iron	36.519	1.45	39.20	2.30
Sulphur	24.848	.32	1.75	1.58
Silica	18.548	96.79	30.90	8.41
Lime	7.294	.23	8.51	T
Magnesia	2.672	T	2.71	T
Zinc	2.556	.....	2.88	none
Alumina	.911	.32	1.90	3.56
Manganese	.77	T	.85	none
O, etc.	.....	.38	11.37	1.00
Carbon	.....	......	.....	83.86
CO2 etc.	3.138	.....	.....	.....
Loss Ign.	.....	.39	.....	.....
Totals	100.00	99.88%	100.80 %	100.71%

Products.

	Matte	Flue-Dust	Slag
Copper	20.00	2.20	.37
Iron	47.15	30.80	38.84
Sulphur	24.00	16.51	1.74
Silica	.44	23.92	32.60
Lime	.10	4.45	8.24
Magnesia	T	1.38	3.44
Zinc	2.05	2.98	1.54

	Matte	Flue-Dust	Slag
Alumina	.82	1.94	1.50
Manganese	.53	.56	.80
O, etc.	4.91	15.26	10.88
Carbon	.....		
CO2 etc	.....	.....	.....
Loss Ign.	.....	.....	.....
Totals	100.00%	100.00	99.95%

Synthesis of Charge and its Products- First Operation.

	Lbs.	Cu	Fe	S	SiO2	CaO
Ore	1000	27.44	365.19	248.48	185.48	72.94
Quartz	80	.....	1.16	.26	77.43	.18
Slag	145	1.06	56.84	2.54	44.81	12.34
Coke	34	....	.78	.54	2.86	....
Totals	1259	28.50	423.97	251.82	310.58	85.46
Ded. as below	428.95	25.10	65.75	235.88	6.69	1.26
Bal. O to Fe, Zn & Mn Slag	938.24	3.40	358.22	15.94	303.89	84.20

	C	MgO	Zn	Al2O3	Mn	Dif.	
Ore	......	26.72	25.56	9.11	7.70	31.38	
Quartz	.....	.....	.....	.26	....	.71	
Slag	.....	3.93	4.18	2.76	1.23	15.31	
Coke	28.51	.....	.....	1.21	....	.10	
Totals	28.51	30.65	29.74	13.34	8.93	47.50	
Ded. as below	28.51	.35	15.63	1.50	.78	47.50	
Bal. O to Fe Zn & Mn slag	.....	30.30	14.11	11.84	8.15	.....	O 108.19

Deductions.

These willbe given on the next page as there is room for them here.

Deductions.

Products	Lbs.	Cu	Fe	S	Si02	CaO
Matte 20% Cu	122.65	24.53	57.83	29.44	.54	.12
Flue dust recovered	25.71	.57	7.92	4.24	6.15	1.14
Volatilized	280.59	....	....	202.20	....	....
Totals deduct. above	428.95	25.10	65.75	235.88	6.69	1.26

Products	C	MgO	Zn	Al2O3	Mn	Dif.
Matte 20% Cu	....	....	2.51	1.00	.64	6.04
Flue dust recovered	....	.35	.77	.50	.14	3.93
Volatilized	28.51	....	12.35	....	....	37.53
Totals deduct.	28.51	.35	15.63	1.50	.78	47.50

Analyses pertaining to Second Operation.

Materials Smelted.

Constituents	Matte	Ore	Lab. Samp.	Quartz	Coke
Copper	20.00	2.79	2.45	....	....
Iron	47.15	43.26	31.07	1.45	2.30
Sulphur	24.00	29.18	14.84	.32	1.58
Silica	.44	10.01	22.66	96.79	8.41
Lime	.10	6.32	5.71	.23	T
Magnesia	T	1.39	2.03	T	T
Zinc	2.05	2.56	2.05	...	none
Alumina	.82	1.00	1.15	.32	3.56
Manganese	.53	.69	.75	T	mone
O, etc.	4.91	....	.....	.38	1.00
Carbon	....	.....	17.29	....	83.86
Co2 etc.	....	2.80	....	....	.....
Loss Ign.	....	.....	....	.39	.....
Totals	100.00	100.00	100.00	99.88	100.71

Products.

Constituents	Matte	Flue-Dust	Slag
Copper	49.63	2.49	.60
Iron	25.24	24.79	43.99
Sulphur	23.00	8.91	1.19
Silica	.26	31.43	33.72
Lime	T	3.31	2.03
Magnesia	T	1.18	.57
Zinc	1.53	3.81	2.12
Alumina	T	3.93	2.16
Manganese	.30	.30	.50
O, etc.	.....	19.85	12.86
Carbon	.....		
CO2 etc.	.....	.....	.....
Loss Ign.	.....	.....	.....
Totals	100.05%	100.00	99.74%

Synthesis of Charge and its Products,-Second Operation.

Charge	Lbs.	Cu	Fe	S	SiO2	CaO
20% Matte	1000	200.00	471.50	240.00	4.40	1.00
Raw Ore	170	4.74	73.54	49.60	17.02	10.74
Lab. Samplings	34	.83	10.56	5.05	7.70	1.94
Slag	160	1.17	62.72	2.80	49.44	13.62
Quartz	330	......	4.78	1.06	319.41	.76
Coke	95	......	2.19	1.50	7.99	T
Totals	1789	206.74	625.29	300.01	405.96	28.06
Deduct. as below	763.36	199.61	104.33	285.86	4.81	.40
Bal. O to Fe, Zn & Mn Slag	1182.18	7.13	520.96	14.15	401.15	27.66

Continued

Charge	C	MgO	Zn	Al2O3	Mn	Dif.:	
20% Matte	.....	T	20.50	8.20	5.30	49.10	
Raw Ore	.....	2.36	4.35	1.70	1.17	4.78	
Lab. Samplings	.....	.69	.70	.39	.26	5.88	
Slag	.....	4.34	4.61	3.04	1.30	16.90	
Quartz	.....	T	.....	1.06	T	2.93	
Coke	79.67	T	.....	3.38	....	.27	
Totals	79.67	7.39	30.16	17.77	8.09	79.86	
Ded. as below	79.67	.14	6.60	.47	1.61	79.86	
Bal. O to Fe	.....						0
Zn & Mn Slag	.....	7.25	23.56	17.30	6.48	......	156.54

Deductions.

	Lbs.	Cu	Fe	S	SiO2	CaO
Matte 49.63%	401.60	199.31	101.36	92.18	1.04	T
Flue Dust Recovered	12.00	.30	2.97	1.07	3.77	.40
Volatilized	349.76	......	......	192.61	....	....
Totals	763.36	199.61	104.33	285.86	4.81	.40

Continued

	C	MgO	Zn	Al2O3	Mn	Dif.
Matte 49.63%	....	T	6.14	....	1.57	.....
Flue Dust recovered	....	.14	.46	.47	.04	2.38
Volatilized	79.67	...	.....	....	....	77.48
Totals	79.67	.14	6.60	.47	1.61	79.86

Thus from the above figures the method of figuring the charge may be seen. The slags etc . found by analysis agree with the figured ones very well.

This plant is subordinate to the Tennesee Copper Company's plant in regard to capacity, ~~oduction~~ etc., it however being constructed so that it may be enlarged to five times its present capacity without very much trouble or expense outside of first cost of machinery.

The Operations at this plant are very similar to the ones carried on at the previously described plant, the main points of difference being the method of charging and the disposition of slag, and difference in type of furnace used.

Finis.

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