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Analysis and metallurgical treatment of copper ores from Ajo Basin, Arizona

Harry Carleton Chamberlain

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FOR THE

SUBJECT:

"Analysis and Metallurgical Treatment of Copper Ores from Ajo Basin, Arizona."

H. C. CHAMBERLAIN.

JUNE 9, 1905.

THESIS
FOR THE

Degree of Bachelor of Science
IN
MINE ENGINEERING.

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ANALYSIS AND METALLURGICAL TREATMENT OF
COPPER ORES FROM ÁJO BASIN, ARIZONA.

The purpose in mind, in writing this thesis, is to select the most economical method of treating the ores found in the Ájo Basin. The ore on the surface, and to a depth of thirty to forty feet, is almost exclusively carbonates in a silicious gangue.

Immediately underlying the carbonate ore comes the sulphide ore in its natural state, lying below any action of the elements, and is in various forms. The sulphide has been found in every shaft to its lowest depth.

We will divide this work up into two main propositions:

First: A Blast Furnace proposition, with a converter, and the advisability of installing an electrolytic refining plant.

Second: A Leaching proposition of the carbonate ores.
We will now take up these two propositions, and finally make a summary of the data.

**Original analysis of Ore.**

<table>
<thead>
<tr>
<th></th>
<th>Cu.</th>
<th>Ag.</th>
<th>Au.</th>
<th>Fe. Al₂O₃</th>
<th>SiO₂</th>
<th>S.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sulphide Ore</td>
<td>60.38%</td>
<td>.23 oz</td>
<td>.024 oz</td>
<td>1.23</td>
<td>1.13</td>
<td>13.35</td>
</tr>
<tr>
<td>Carbonate Ore</td>
<td>13.44%</td>
<td>.00 oz</td>
<td>.00 oz</td>
<td>1.16</td>
<td>1.61</td>
<td>73.4</td>
</tr>
</tbody>
</table>

We will first consider the sulphide.

As the results of this table are not conclusive that the ore will run as high in copper as is shown in the table, we will recalculate to the basis of a 15% ore.

So then the following table is the result of figuring on a 15% ore, with the constituents in about the same ratio as the average Arizona ores of this nature.

<table>
<thead>
<tr>
<th></th>
<th>Cu.</th>
<th>Fe. Al₂O₃</th>
<th>SiO₂</th>
<th>S.</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ore</td>
<td>15.00</td>
<td>40.1</td>
<td>1.4</td>
<td>16.8</td>
</tr>
</tbody>
</table>

The results shown are obtained from the following computations:

Assuming that ore will run in Cu, Fe and S about the same as average ores of Arizona, working on a basis of 7.5% Cu., making a 40% matte, losing 60% in roasting and smelting, we formulate analysis as follows for ore:-

1 ton ore contains 70 lbs Cu.

70 Cu requires 17.7 lbs S.

87.7 lbs Cu₂S make \[\text{xxx}\] 1 ton of ore.
87.7 lbs Cu₂S make 175 lbs 40% matte, or 175 - 87.7 = 87.3 lbs FeS in 175 lbs matte.

87.3 lbs FeS = 32 lbs S and 55.3 lbs Fe

Total S in 175 lbs matte -------------- 49.7 lbs.

Assuming this as 40% of original S in one ton ore, we have 124.25 lbs S in one ton of ore.

Then a 15% Cu ore would have 4.3 times as much S = 534.27 lbs sulphur, or runs 26.7% S, (See Comp. above)

15 lbs Cu would require 3.3 lbs S., so 26.7 lbs - 3.3 = 23.4 lbs, which must be in form of FeS.

56 : 32 : X : 22.9

X = 40.1

Ore must run 40.1% Fe.

Then 15 + 40.1 + 26.7 = 81.8

100 - 81.8 = 18.2% to be divided among SiO₂ and Al₂O₃ in ratio of original analysis on 60% ore, which was

Al₂O₃ : SiO₂ :: 1 : 11.72.

18.2 = X + y

X = 1.4 = % Al₂O₃ present.

X : y :: 1 : 11.72 y = 16.8 = % SiO₂

But 40% matte from 15% ore would use up 11.9 lbs Fe for every 100 lbs ore, leaving (40.1 - 11.9) or 28.2 lbs Fe to go into slag (which in FeO is equal to 36.25 lbs FeO)

Slag.

As for a source of lime for our slag, assume a limestone containing 51% available CaC to be on hand.
4.

We will now try and see how a slag in the form of a singulo-silicate works out.

This singulo-silicate = 2 RO: SiO₂, or must have two atoms of O in base (49)

For the two atoms of O in the SiO₂, so the factor for FeO is 2FeO : SiO₂

or 144 : 60 = .416

Then 36.25 FeO X .416 = 15.06 lbs SiO₂ reg.

1.4 Al₂O₃ X .873 = 1.22 " " 

16.28 " total "

But we have in our ore 16.50 lbs SiO₂

So we have .50 lbs excess, over what is required to form singulo-silicate. Then we want to add enough limestone to flux this excess of SiO₂.

Assuming that the bases in the limestone are figured outside, and that we have our limestone with 51% CaO for fluxing purposes, so then to flux this .5 lbs of SiO₂ (excess) it will require

.5 X 1.36 = 93 lbs CaO

(The factor for SiO₂ in terms of CaO is 1.36)

This .93 lbs CaO = X lbs limestone.

100 : 51 = X : .93

X = 1.82 lbs limestone reg.

So we see that, as we only need 1.82 lbs of limestone to complete the singulo-silicate, this is a very small factor when we consider how many pounds of extra fluxes might have been necessary,
so we will say then that the singulo-silicate is satisfactory.

Calculations of matte.

100 lbs of 40% copper matte contains 50% CuS, and 50% FeS in the ratio of Fe = 31.6% and 10.4% S of analysis of matte Cu - Fe - S

40 31.6 28.4

We now have to calculate the proper amounts of Cu, S and Fe it will take to form a matte.

Our ore contains for matting

Cu - 15
S - 26.7
Fe - 11.9 lbs reg for a 40% matte from 15% ore.

Assuming that 60% of the total sulphur in the ore is burnt off in smelting

26.7 X .60 = 16.020, leaving 10.68 lbs sulphur to go into the matte with the Cu and Fe.

\[
\text{Cu}_2 : S = 126 : = 15: X
\]

\[
X = 3.8 \text{ lbs S to form Cu}_2S. \text{ This leaves 7.88 lbs S for the FeS.}
\]

\[
\text{Fe} : S = 56 : 32 = 11.9 : X
\]

\[
X = 6.8 \text{ lbs of S needed to form FeS.}
\]

The excess S then would be 1.08 lbs.

We assumed that 60% of the total sulphur would be burnt away, but as this is not constant, this extra 1.08 lbs would be taken care of, or would enter the slag.
Blast Furnace Charge.

For 100# Ore

<table>
<thead>
<tr>
<th>Material</th>
<th>Amount</th>
</tr>
</thead>
<tbody>
<tr>
<td>15 lbs Cu</td>
<td></td>
</tr>
<tr>
<td>40.1 lbs Fe</td>
<td></td>
</tr>
<tr>
<td>1.6 lbs Al₂O₃</td>
<td></td>
</tr>
<tr>
<td>16.8 lbs SiO₂</td>
<td></td>
</tr>
<tr>
<td>26.7 lbs S</td>
<td></td>
</tr>
<tr>
<td>1.87 lbs Limestone</td>
<td></td>
</tr>
</tbody>
</table>

Total: 101.87 lbs based on a 100 lb charge.

In a copper blast furnace the coke-ash of the coke is always self fluxing, so it does not need to be considered in the computations of the charge.

Blast Furnace.

In the following table there will be found dimensions of a furnace to handle 200 tons per 24 hours.

<table>
<thead>
<tr>
<th>Dimension</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Horizontal section at throat</td>
<td>66' X 100</td>
</tr>
<tr>
<td>&quot;    &quot; tuyeres</td>
<td>42' X 100</td>
</tr>
<tr>
<td>Area at tuyeres</td>
<td>52.35 sq. ft.</td>
</tr>
<tr>
<td>Height tuyeres to throat</td>
<td>7' 3-7/8&quot;</td>
</tr>
<tr>
<td>&quot;    &quot; top of crucible</td>
<td>10&quot;</td>
</tr>
<tr>
<td>Height water jacket</td>
<td>8&quot;</td>
</tr>
<tr>
<td>Bosh, inches in feet</td>
<td>12' in 6' 4&quot;</td>
</tr>
<tr>
<td>Crucible, depth</td>
<td>8&quot;</td>
</tr>
<tr>
<td>Forehearth, fixed or movable</td>
<td>Fixed</td>
</tr>
<tr>
<td>&quot;    &quot; shell dimensions</td>
<td>3' X 5-1/2' X 3'</td>
</tr>
<tr>
<td>Tuyers, number</td>
<td>16</td>
</tr>
<tr>
<td>&quot;    &quot; diameter</td>
<td>3&quot;</td>
</tr>
<tr>
<td>Description</td>
<td>Value</td>
</tr>
<tr>
<td>-------------------------------------</td>
<td>-------</td>
</tr>
<tr>
<td>Tuyere-ratio</td>
<td>5.5</td>
</tr>
<tr>
<td>Blast, pressure, oz</td>
<td>16</td>
</tr>
<tr>
<td>Blast, cu. ft. per ton of ore</td>
<td>57,000</td>
</tr>
<tr>
<td>Blast, temperature in °C</td>
<td>175</td>
</tr>
<tr>
<td>Charge (ore and flux), weight, lbs</td>
<td>2027</td>
</tr>
<tr>
<td>Charge, tons in 24 hours</td>
<td>200</td>
</tr>
<tr>
<td>Charge, tons per sq. ft. hearth area</td>
<td>7.88</td>
</tr>
<tr>
<td>Coke per cent, ash</td>
<td>20</td>
</tr>
<tr>
<td>Coke per cent on charge</td>
<td>13</td>
</tr>
<tr>
<td>Men, number in eight hour shift</td>
<td>5.2</td>
</tr>
<tr>
<td>Matte, % on charge</td>
<td>5</td>
</tr>
<tr>
<td>Matte, % cu.</td>
<td>40</td>
</tr>
<tr>
<td>Slag, SiO₂</td>
<td>16.5</td>
</tr>
<tr>
<td>FeO</td>
<td>36.2</td>
</tr>
<tr>
<td>CaO</td>
<td>.03</td>
</tr>
<tr>
<td>Al₂O₃</td>
<td></td>
</tr>
<tr>
<td>Cu.</td>
<td></td>
</tr>
<tr>
<td>Ag.</td>
<td></td>
</tr>
<tr>
<td>Sp. gr.</td>
<td></td>
</tr>
</tbody>
</table>
Blast Furnace.

We aim now to treat 400 tons per 24 hours, but instead of using one large furnace, we will consider two furnaces each with a capacity of 200 tons per day.

Much available data on such work was found in Prof. H. C. Hofman's "Notes on the Metallurgy of Copper of Montana," from which manual the following data has been derived:

In the plant outlined roughly in these notes, we will assume that the ore is to all be treated in the blast furnace without previous roasting in a reverberatory furnace. In a case of this kind, it has been found best to use a blast furnace with a narrow horizontal section, when the furnace has to smelt all of the material, both "fines" and coarse ore.

So then we will take a furnace 48 inches wide, the length depending on charge per square foot of grate area, and the capacity required, which is 200 tons per 24 hours for each furnace.

This figures out to be 180 inches for the length, making the charge in tons per square foot of grate area = 50.35.

These figures and other dimensions and important data are found in the table given.

The best blast furnaces are of the same general type; they are oblong and water jacketed, and have a crucible partly external, and partly internal, which discharges the slag-matte mixture continuously over a raised spout, whereby the blast is trapped. The matte is settled in a large cone-fourth, while the slag overflows, to be granulated and removed by water, or to be collected in waste slag cars and
conveyed by some electric means to the dump and poured.

The general tendency is to increase the size of the furnace. The distance between tuyeres varies from 42 inches to 56 inches. For large output, is generally 56 inches.

The greatest length of furnace at tuyeres is 180 inches, and this seems to be the limit with the present mode of construction of jackets. The height of the furnaces increases with the distance between tuyeres from 7 to 8 feet. But the charge in an 18 foot furnace is kept about 4 feet below the throat, so that the actual working height is reduced to 14 feet.

All furnaces are run with a top more or less hot. The water jackets are all made of low carbon steel. They are 2' high, reaching to a short distance below the feed door, the remaining space being of brick.

There are usually two jackets on a side, and one on the ends; the lower tiers of side jackets only have a bosh, the amount of which is very small, while the upper tiers are vertical, as are generally the end jackets. The gases are always drawn off above the feed floor.

The charge in most works is introduced by hand, but in the case of a plant of 400 ton capacity per 24 hours, mechanical feeding has taken the place of hand feeding.

The charge is brought in by electric or compressed air traction to the side of the furnace on trucks with tilting carriage, the doors are raised by compressed air, and the charges slid in alternately from one side to the other.
11.

This method of mechanical feeding has been carried on satisfactorily in many places, even though some engineers maintain that the successful working of a blast furnace is much governed by the care given to feeding, and that this method is not wholly satisfactory.

The hearth is built as solid from the foundation in most cases. It rises some distance above the furnace floor, in order to furnish the height necessary for the forehearth or settler, and the disposal of the waste slag.

The hearth is lined to a depth of 24 to 30 inches with fire brick. The crucibles have a depth ranging or varying from 8 to 28 inches. The deep crucibles, while their life is limited to three months, a much shorter life than the shallow crucibles, are, nevertheless, more commonly used.

The slag-matte is usually discharged from the center of one of the sides of the furnace. The water cooled tymp is made of cast iron, or preferably copper.

The tuyere-stock in common use, consists of a cast iron tuyere-box, firmly attached to the jacket, and connected by a sheet-iron tuyere-pipe, having a gate, with the bustle pipe.

As far as the settling of matte is concerned, oval and oblong hearths are sufficient, as the freedom of corner in waste slag depends to a large extent upon the length of the path the slag can travel before it overflows.
The circular forehearth, 14' in diameter and 56" high, with a 9" brick lining and a 9" brasure backing, holds about 60 tons of matte when new. In order to prevent breaking out, the shell is sprayed with water.

All forehearths have, of course, a tapping slot for the matte. It is closed in some cases by a cast iron plate, but more commonly by a solid copper plate with a tap hole. The matte is tapped from the forehearth into a ladle, which discharges its contents either into flat cast iron molds, holding about 3000 lbs, or into the converter. The practice of bringing the converter to the forehearth of the blast furnace has become obsolete.

The waste slag is caught in waste-slag pots of 5 to 6 tons capacity, and hauled to the dump, or it is granulated.

Near the bottom of the launder there is often found a mechanical sampler, driven by the granulating water, which takes out the whole of the stream of granules at stated intervals, and furnishes thus a true sample of all waste-slag made.

In blast furnace work, especially in Arizona, even the freedom from all concentration and calcination processes, does not leave the smelters from the occurrence of "fines."

The clayey and friable nature of many of the ferruginous and calcareous carbonates favors the formation of fines, which aside from the heavy loss entailed by their escape through the stack, clog the furnace, obstruct the blast, and being sifted down between the coarser bumps of ore and fuel, reach the smelting in a cold and unprepared condition causing the chilling of the crucible and the
growth of long noses from each tuyere, which may meet in the middle, forming a central core of semi-fused material that may necessitate the termination of the campaign.

Some Arizona smelters remedy the loss in flue dust by the construction of flues and dust chambers. The clayey nature of most of the fines, and the hot and dry climate, assist the process of bricking these fines, although sometimes an addition of milk of lime is sometimes found necessary to bind the particles together with sufficient firmness.

The slag from the converter always carries from one to two per cent of copper, and quite enough to cover the cost of its smelting where fuel is cheap; while the large percentage of iron that it carries neutralizes the 510% of the "green fines," which finds no base in its own composition, all the iron that it contains being combined with sulphur and consequently unavailable for slag formation.

The slag resulting from this practice is distinguished by its freedom from copper, owing to the overwhelming amount of low-grade matte present, which cleanses the light silicious slag to an unprecedented degree. This slag is very acid, owing to the large amount of silica present, and is very thick and sticky, but free from value.

So then, we could re-run all our slag that we have from our sulphide ore, with part of the raw sulphide in the charge, and get a good matte and a clean slag.
Convertor Plant.

At the present time, the Parsons-Klepethko method for treating matte is generally employed in the best plants.

A converter charge is tapped from the blast furnace forehearth into a steel ladle, managed by an electric traveling crane, and poured into the converter. Ladles varying in size: a common form has a diameter of five feet, and a height of three feet and ten inches. The lining, ordinary loam, is plastered on by hand, dried by an air current and then by a fire (wood or coke); converter slag, to the thickness of three inches is then poured in. Matte ladles are used, also, as slag ladles. One lining is good for nine and one-half tons of copper.

Generally the matte, after being blown to white metal and skimmed, is blown to coarse copper, without any further addition of fresh matte. In case of a matte running less than 50%, which applies to our case of 40% matte, doubling charges is a common procedure.

After skimming the slag, the white metal is brought forward to coarse copper. By these double charges considerable amount of copper is produced in one blow, to be poured into the form of shipping-ingots, 200 to 250 lbs, or anodes, or into a refining-furnace, to be refined before casting into anodes.

We will select our converter of the David-Yankees type, or barrel converter swinging in a horizontal plane, which is the most common in the new plants.

The most common size is 7' in diameter, and 10'6" long. It
receives an initial double charge of five tons, and a final double charge of twelve tons of 40% matte.

The main reason for replacing the upright by the horizontal converter is the low blast pressure required by the latter; 10 lbs to the 15 lbs required by the upright.

This advantage, however, suffers greatly by the difficulty in making the lining firm, so there is not so much advantage for either type.

In all plants the converters are placed in a single row, forming a straight line. Back of them is the main flue, with projecting hoods, into which the converters discharge their gases, vapors and finely divided particles of flue dust. The main flue ends in a dust-chamber connected with a stack.

Linings.

The lining of converters, both of body and of caps, is still causing trouble in plants, and in most favorable cases the body has to be removed after twenty-eight tons of material have been treated.

The following table gives composition of several converter linings:

<table>
<thead>
<tr>
<th>Greenist, granular sandstone, Great Falls</th>
<th>SiO₂</th>
<th>Fe</th>
<th>Al₂O₃</th>
<th>CaO</th>
<th>MgO</th>
<th>Loss on ignition</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>64.4</td>
<td>5.9</td>
<td>16.5</td>
<td>1.1</td>
<td>1.8</td>
<td>4.7</td>
</tr>
</tbody>
</table>

| Crushed quartzite with 5% slag           | 88.6 | 2.2| 4.0   | 1.6 |     |                 |
| Quartzite                                | 90.9 | 2.7| 3.2   |     |     |                 |
| Slimes                                   | 60   |    | 4.5   | 15  |     |                 |
The material for the cap has to be more plastic than is necessary for that of the body.

The lining usually consists of body-lining material, and 5 to 10% clay, and is mixed with more water than the body-material.

The chemical action in some instances plays even a subordinate part to the mechanical wear, by the swash of the liquid charge which causes part of the lining to break off. This is especially observable in the barrel-converter.

The average time to complete a five ton charge is two hours and fifty-five minutes. The blowing of a single charge takes forty-seven minutes, and a double charge takes eighty-two minutes.

The labor required per shift for a single converter is made up of one skimmer, one puncher, one helper, and one sampler.

The general yield of metal in roasting, smelting and converting, in treating a 15% ore and about 4 ozs silver per ton, is Cu - 92 4 % and Ag - 95%.

Data on a Barrel Converter.

Grade of Matte - - - - - - - - - - - - - - - - - - 40% Cu.
Horizontal section of tuyures - - - - - - - - - - - - 6.66 sq. ft.
Lining material per converter - - - - - - - - - - - - 8.40 tons
Lining per ton copper resulting - - - - - - - - - - - .24 tons
Copper per converter - - - - - - - - - - - - - - - - .20 tons
Tuyures, number - - - - - - - - - - - - - - - - - - 14
" diameter - - - - - - - - - - - - - - - - - - 3/4
Maximum pounds possible per charge - - - - - - - - - - 9
Number of Converters required.

Have a charge of 2037 lbs per ton of ore, making a total of 814,800 lbs charge per day for 400 tons capacity.

Percent of matte if charge = 5%, so there would be 40,740 lbs or about 20 tons of matte per day. Each converter will treat ten tons of matte on a double charge, which makes a final charge of about four or five tons of copper depending on the grade of matte. Each converter will handle about five charges per day, or 50 tons of matte.

Then one converter at this rate would handle all of the matte that two furnaces will turn out in one day, but allowing for repairs on them, and relining, there should be \textit{at least three} converters in the plant, so that there should be no delay.

Casting Machine.

There are several different types, but this matter need not be discussed fully. The copper is cast into anodes of various shapes and sizes.

Two men are needed for each shift to look after the machine. The improved form of the Walker Casting Machine is about the best casting machine.

Electrolytic Plant.

We will now consider a plant for refining the blister copper from the converter.

A few figures from the Anaconda workings will furnish material for a successful treatment.
An ordinary converter anode weighs 500 lbs, and is sufficiently corroded in about eighteen days to be withdrawn and regarded as scrap, a new anode being put in its place.

The Anaconda anodes are 24 to 24-1/2" wide, 32-7/8" long, and 1-1/4" thick at the top and 1" at the bottom. They have two top lugs, 6" wide with holes, and are suspended in pairs from iron copper covered cross bars (1/2 X 1-3/4"), 19 to a tank, by means of copper hooks 3/8" thick. An anode weighs 230 lbs, and is corroded in thirty-seven days.

The cathodes are made in the same way, in tanks set apart for the purpose. They are slightly longer and wider than the anodes. At the Anaconda they are 11" wide by 33" long, and weigh 3/4 to 1 lb.

The upper rim of the starting sheet is bent and clamped over the ends of a piece of sheet copper, 4-1/2" wide, by 11" long; four cathodes go to a cross bar; the electrode distance ranges from 1-1/2" to 2".

The tanks are made of a 3" blank and are lined with 8 lb lead, and the bottoms have a board cover to protect the lead lining from pieces of anode copper that may become disturbed. The tanks are arranged in single rows.

The electrolyte contains with the heavier current 170 grams concentrated sulphuric acid, and 42 grams copper per liter; with the lighter current, 150 grams acid and 40 grams copper.

The resistance appears to increase when the copper rises
above 42 grams. The temperature of the electrolyte at the head tank is 64 degrees C and 50 degrees C; and the circulation is 6 and 5 gallons per minute. Solutions are admitted and drawn off, either at the ends or the sides, precautions being taken by means of perforated lead baffle plates to have an even distribution, and this avoid stirring up any mud.

At the Anaconda cathodes are removed only when the anodes are completely corroded. After two days of depositing, they are, however, taken out and straightened. A tank receives its anodes all at once; in the same way all the cathodes are taken out by one operation.

The anode mud is discharged through an opening in the bottom of the electrolyzing vat, after the solution has been drawn off from the side, into a V shaped trough leading into a tank; from this it is drawn into an acid egg and forced into a tank in the refining-department, drawn onto filter-cloths, washed, dried, sampled, sacked and sold. Until recently, the mud was treated at the works. It was melted down in a reverberatory furnace, refined by the addition of nitre, cast into small ingots and parted with sulphuric acid, the silver being precipitated by means of the copper.

The anode scraps are re-smelted in the converters.

The refining works at Anaconda are driven by steam power; have 1400 tanks 8' long by 4' wide and 4' deep, in sets of 200 each.

A tank is charged with 76 anodes, and 30 cathodes; the current is of 10 amperes per sq. ft of cathode area. There are 7 dynamos, each generating 4000 amperes at 60 volts for 200 vats.
Summary of the Above Figures.

From the figures above we could design a refining plant for our Arizona ores, but the conditions are such that to install an electrolytic plant in a region generally barren of coal, and with a none too abundant supply of wood, and no natural source of water power, one can see that the expense of operating a plant of this kind would be very great. So with these points in mind it seems best to follow the example of nearly all of the Arizona copper companies and sell the blister copper from the converter direct to the refineries in the East.

A representative of Clark's Mines in Arizona told the writer that electrolytic refining was too expensive a method to employ, and it was much more economical to sell the blister copper direct to the refineries.

Thus for economical treatment, this point we will consider as settled.

Leaching of Ores.

As there are such a variety of ways to leach copper ores, some of them fairly successful under certain conditions, and others not at all, so we will not take up any special process, but will discuss it in general.

Nearly all of the various processes aim to dissolve the copper out of the ore in its natural state, or after a desulphatizing roast, and get the copper in the form of sulphate or chloride. Then it is precipitated from the solution by zinc or iron: scrap iron,
gathered up from about the country, is generally employed, as it is cheaper.

The great disadvantage in the leaching of ores is the great amount of time taken up in comparison with the output, also to make a paying proposition out of it a great deal of capital must be sunk in leaching vats. This requires a large area of ground surface, which would be quite a factory where the claim or property was small.

Then too, the consumption of sulphuric acid would be considerable. The idea of manufacturing acid from the chimney gases at the plant, calls for a special outlay of money, and is not generally employed, the expense being greater than if the acid was bought outright.

There is one scheme, however, suggested that might be worth while to experiment on and see how it would work on the surface carbonate. After you have your copper dissolved in the form of copper sulphate, filter the solution, and precipitate the copper, electrolytically.

Now comes the main point: Blow $SO_2$ into the solution from which the copper has been precipitated, thus regenerating the sulphuric acid of solution, and using the same solution a number of times. The $SO_2$ can be kept on hand in storage tanks.

However there is one point that must not be overlooked. If the gangue contains much iron and aluminum, the acid will take these up into solution, and after a few regenerations with the $SO_2$ the solution will become so charged with precipitate that it will not filter, and hence the operations must stop.
But in the surface carbonates, that are in abundance, analysis shows about 15% each of iron and aluminum, so that this ore seems to be just about suited for such a process.

Prof. W. W. Garrett, of Missouri School of Mines, conducted a series of experiments on this method under certain conditions, and found that it worked admirably for ores running low in iron and aluminum.

Summary.

We will now gather up the points and see what conclusions can be drawn.

The mine itself, in a barren, hot country, located 50 miles from the railroad, would, of course, necessitate the hauling of everything by wagon, both mill supplies, products or the mine, etc., until a railroad could be built.

A railroad could be built at comparatively low cost, as the country is low and level, and very little grading would be required.

The object is, then, to select the method that would bring the largest returns in the shortest time possible, so as to allow for increase in capacity.

The computations in the first part of this paper show the sulphide ore to be a first-class blast furnace proposition, only a few pounds of limestone being required to complete the charge. At the same time a first-class blast furnace proposition is the most direct and quickest method of converting copper ore into metallic copper. As for the gold and silver values in the ore, unless very high, they would still throw the balance in favor of the blast furnace.
The analysis does not show very high value in gold and silver, but that one analysis could not be taken as a fair average for the whole field, as gold especially has such a wide variance.

Also, from the experience of many mines of this type, in all probability the sulphide ore on attaining a great depth becomes higher in iron and sulphur, thus requiring a large amount of silica to neutralize them. Then having a great amount of the surface carbonate on hand, rich in silica, this would be invaluable in smelting the sulphide ore; and every bit of the silicious carbonate could be used along with the smelting of the sulphide.

This last point alone would seem to settle all doubt as to the advisability of putting in a leaching plant. For if the carbonate was all used up in a leaching plant, when it came to treating the sulphide a great amount of fluxing material would be required to neutralize the excess of iron and sulphur.

Conclusion:

Having a first-class blast furnace proposition, quick and most economical compared to a leaching proposition, which at some future date would be all used up, and necessitating finally a smelting plant, it would seem best, by all means, to install a good blast furnace plant on the grounds.

H. C. Chamberlain.