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Leaching copper tailings

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LEACHING COPPER TAILINGS.

BY

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AND

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THESIS

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There are, scattered throughout the country, enormous amounts of rock that, by the discovery of a quick, cheap, and practiable method of extracting their metallic content could be turned into valuable ores. This has already been accomplished for gold and silver by the adaption of the cyanide method to their treatment.

The next in order, both because of the value of the metal and the amount of material available in mill dumps and in the ground, is copper. There are dumps containing millions of tons of this material lying idle because of the fact that with the present methods of treatment, it can not be handled at a profit.

Now if we had an ore carrying 1% copper, or worth 3 dollars, this ore could not be treated by smelting because of the cost. Concentration would also be out of the question because of the facts that it would cost about 50 cents per ton to concentrate the ore, and the tails will carry about half of the value to the dump. This would mean that the cost of concentration, including the tailing loss, would be 2 dollars to concentrate a 3 dollar ore. This excessive cost would eliminate concentration from the treatment.

On the other hand, if we had an efficient leaching scheme, we could get nearly all of the copper from the ore, and at a cost of less than either the smelting or the concentration.

From this we see that if our copper ores could be

treated for what it costs per ton to treat many of our
gold ores, the profit would be doubled. With the present
tendency along the lines of cheap mining, a cheap
leaching scheme would put much of the material that is
at the present considered as waste rock, into the class
of valuable ores.

There is another class of material that could be
handled at a good profit, by means of a leaching
scheme. There are many dumps in the country that will
carry from .5% to .75% copper. These dumps have already
been ground to a size suitable for leaching, and hence
the cost of mining and crushing could be done away
with. This would bring the cost of leaching the copper
to simply the cost of extracting the copper from the
ore, and refining it.

The material could be shoveled from the dump by a
steam shovel, then carried to the mill and leach it.
The shoveling can be done at a cost of say 3 cents per
cubic yard. The ore could then be transported to the
mill by one of the cheap mechanical means, in vogue at
the present time. It would then be put into leaching
tanks and treated by a solution, which would of nec-
essity be cheap to make and conserve, then precipitate
the solution on scrap iron, the method that is at
the present time used on copper bearing mine waters.

There is very little information on the leaching
of copper ores, for all of the fact that it is one of

the most attractive of all concentration schemes. The
theory is very simple, consisting, essentially of the
following steps; the ore is leached by a solution of
such a constitution that the copper will go into sol-
ution as one of its many salts. This rich solution is
then passed over some cheaper metal, that is higher in
the electro-motive series than is copper, which will
replace the copper in solution and precipate it out as
cement copper. All that is now necessary to do to the
copper to prepare it for market, is to refine it by
one of the common refining schemes.

The copper often occurs in the material to be leached
in an unsoluble condition. In order to make the leach-
ing a success it is necessary to put this copper into
a soluble condition. According to M.Eissler, in his
Hydro-Metallurgy of Copper, this can be done in two
ways, (a) by changing the insoluble copper sulphides
into the soluble copper sulphate, and (b) the changing
the insoluble copper sulphides into the soluble cop-
per chlorides.

The first of these processes can be carried out by
three general schemes, which are as follows;—(1)The
formation of the sulphates by a process of weathering,
(2) By means of a slow roast with an excess of air,
(3) The roasting of the ore with ferrous sulphate or
other easily decomposed sulphates.

In order to make the process successful on the low

grade ore under consideration, it is necessary to
limit the time and cost to a very reasonable factor.
It is necessary to limit the time because of the fact
that the interest on the money that is represented by
the ore under treatment will amount to quite a large
item. The cost of the treatment must be kept down be-
cause the ores to which the leaching scheme would be
applicable are   low grade and a large treatment
cost would allow no margin of profit.

The first of these methods mentioned, the weather-
ing of the ores to form the sulphates, is sometimes
used in foreign countries. To be successfully used
in connection with leaching, under conditions that
hold in this country, would be inpractical because of
the length of time that is required and the amount
of capital tied up in the weathering ore.

For a different reason, that of cost, it would
be necessary, in our case, to also eliminate the sec-
ond and third methods, that of slow roasting with
air and the roasting with ferrous sulphate. From the
best figures that are obtainable, those of M. Eissler,
the cost of heap roasting, which is the cheapest method
of roasting, under the most favorable conditions is 50
cents per ton, in the first case. The cost of roast-
ing with ferrous sulphate would be even more expensive
because the cost of the chemicals would have to be
added to the actual cost of the roast.

It can be plainly seen that the addition of even 50 cents per ton added to the cost of treatment of an ore worth only 1.50 dollars would make the process unprofitable and hence improbable. For these reasons this step in the process of leaching must be omitted.

The second method of preparing the sulphides for leaching is to change them to the soluble chlorides, and according to M. Eissler this can be done by two methods which are as follows;—(a) The wet method or the treatment of the ore with ferric chloride, or with ferrous chloride and hydrochloric acid.

This method seems to have no objections because in all, except the unusual cases, the cost can be kept down to a reasonable amount as the ore can be leached by the same solution that was used to change the sulphides to the chlorides. As can be seen, the principal trouble with this method is the cost of the ferrous chloride and the hydrochloric acid that is used in making the solution.

The second method of M. Eissler is (b) to give it a roast with sodium chloride. This second method, because of the cost can be eliminated as was that of roasting for the sulphates.

It would seem that from the above that the condition in which the copper is contained in the ore will have a good deal to do with the success of leaching, that is, some ores will give up their values more easily

to some solutions than to other solutions. It would follow that the best way to determine the adaptibility of an ore to treatment by a leaching scheme, is simply to try the ore with different strengths of various solutions, and then from the assay of the tails after leaching, figure the extraction. This is substantially the method of testing a gold ore for treatment with a cyanide solution. This is the method that was used in this case for testing the possibility of leaching the copper tailings.

The ore used,was a low grade copper tailing. The ore is highly silicous, which fact is favorable to leaching, as the SiO_2 does not interfere with the reactions between the copper in the ore and the solution. The screen analysis of the ore is as follows;—

On 80 mesh = .25%

100 mesh= 20%

150 mesh= 21%

200 mesh= 19%

Through 200 mesh = 38%

Tylers Standard Screens used for analysis.

Owing to the fine condition of the ore we were not able to tell in what condition the copper was held. The fact that, at the present time, most of the mines have worked through the oxidized zones, we may safely assume that the copper in this ore is there as the sulphide, and most probably as chalcopyrite.

The first step in the process was to ascertain the amount of copper that was contained in the ore, in order that, after leaching, we could figure the extraction. This was carefully done by the Iodide Method. We made eight or ten determinations, taking between three and five grams of the ore to test each time, and the average of all of our results gave .62% copper. With copper at 15 cents per pound, this would make the ore worth 1.86 dollars per ton of 2,000 pounds.

The apparatus that was necessary in the carrying out of the leaching process was very simple.

We weighed out an amount of ore into a beaker. On this beaker was then placed the date, time, the amount of ore taken, the amount and kind of solution taken, and it was set away until the leach was finished. The solution was then filtered off, the time of the end of the test and the number that was given to the experiment was then placed on the beaker, and the beaker with the dried tails kept until the assay had been successfully made.

For the sake of convenience and ease of figuring results, all of the solutions that were used for the leaching were made up to a strength of 1%, and then difference amounts of this one solution were used. In all of the work , we used the assay ton system. That is,we would weigh out different numbers of assay tons of the ore and then leach it with different numbers of

assay tons of the solutions,and in this way it was
very easy to tell just how much copper we were getting
out per ton of ore taken or per ton of solution used,
as the case might be.

The solutions that were used were as follows:—
$KHSO_4$ (Potassium acid sulphate), H_2SO_4 (Sulphuric acid),
$FeSO_4$ (Ferrous sulphate), $Fe_2(SO_4)_3$ (Ferric sulphate),
HCl (Hydrochloric acid), $FeCl_2$ (Ferrous chloride), $FeCl_3$
(Ferric chloride), $FeCl_3+NaCl$, $FeCl_2+CaCl_2$, and $FeCl_2+$
$CaCl_2+NaCl$.

The first leachings that were made were with 2 as-
say tons of ore and 4 assay tons of solution, or
in that ratio. These were let stand for various lengths
of time, varing from 70 to 193 hours. They were then
filtered, washed by decantation, and the residue dried and
assayed. The assay of the tails were made by the Iodide
Method, using $Na_2S_2O_3$ to precipate the copper in-
stead of aluminum, as is more commonly done. In all
of the assays of the tails we took five grams for analysis,
because when this large amount is taken the liability
for errors in the results is lowered. The results were
unsatifactory, as in every case the extraction was
very low.

On the next thirty two experiments we used a dif
ferent method of testing the solutions. Instead of
taking 2 assay tons of the ore and 4 assay tons of the

solution, we used one assay ton of the ore and from
1 to 5 assay tons of the solution, except the 2 tons of
solution, as we had,already, the results for this ratio
of ore and solution.

The results for the various experiments with the
extraction follow;

No	Amount of Ore	Amount Solution	Kind of Solution	Assay Tails	Time of Leaching	% Ext.
1	2 A.T.	4 A.T.	1% KHSO$_4$	.583%	72 hrs	7
2	2 "	4 "	1% H$_2$SO$_4$	.601%	72 "	4
3	2 "	4 "	1% Fe$_2$(SO$_4$)$_3$	.598%	70 "	4
4	2 "	4 "	1% FeCL$_2$	.606%	77 "	3
5	2 "	4 "	1% Fe CL$_3$	.583%	77 "	7
6	2 "	4 "	1% Fe CL$_2$ + Co CL$_2$	.610%	96 "	2
7	2 "	4 "	1% Fe SO$_4$	.575%	97 "	8
8	1 "	2 "	1% FeCL$_3$ + { na CL / Ca CL$_2$ }	.588%	168 "	6
9	1 "	4 "	1% KHSO$_4$	.575%	193 "	8
10	1 "	4 "	2 A.T. FeCl$_2$ 1% / 1 " HCL, 1 A.T. Ca Cl$_2$ 1%	.454%	168 "	27
11	2 "	4 "	2 A.T FeCl$_3$ 1% / 2 " HCL 1%	.588%	190 "	6
12	2 "	4 "	2 A.T. FeCl$_3$ 1% / 2 " Fe$_2$(SO$_4$)$_3$ 1%	.596%	167 "	5
13	2 "	4 "	1% Fe CL$_2$ + No CL	.588%	187 "	6
14	2 "	4 "	1% Fe CL$_3$ + Ca CL$_2$	.578%	190 "	8
15	2 "	4 "	1% HCL	.578%	120 "	8
16	1 "	4 "	1% H$_2$SO$_4$	.580%	193 "	8
17	2 "	4 "	2 A.T. HCL 1% / 2 " H$_2$SO$_4$ 1%	.556%	169 "	11
18	1 "	4 "	1% Fe$_2$(SO$_4$)$_3$	.585%	169 "	6
19	2 "	4 "	2 A.T H$_2$SO$_4$ / 2 " KHSO$_4$	.566%	186 "	9

No	Amount of Ore	Amount Solution	Kind of Solution	Assay of Tails	Time of Leaching	% Ext.
1	1 A.T.	1 A.T	1% $KHSO_4$	.599 %	234 hrs	3.4%
2	1 "	3 "	"	.551 %	"	11.2 "
3	1 "	4 "	"	.546 %	"	12.0 "
4	1 "	5 "	"	.526 %	"	15.2 "
5	1 "	1 "	1% H_2SO_4	.567%	"	8.6 "
6	1 "	3 "	"	.546%	"	12.0 "
7	1 "	4 "	"	.599 %	"	3.4 "
8	1 "	5 "	"	.504 %	"	18.8 "
9	1 "	1 "	1% $FeCL_3$	.535 %	"	13.8 "
10	1 "	3 "	"	.515 %	"	17.0 "
11	1 "	4 "	"	.515 %	"	17.0 "
12	1 "	5 "	"	.504 %	"	18.8 "
13	1 "	1 "	1% $FeSO_4$	.588 %	"	5.2 "
14	1 "	3 "	"	.493 %	"	20.4 "
15	1 "	4 "	"	.546 %	"	12.0 "
16	1 "	5 "	"	.536 %	"	13.8 "
17	1 "	1 "	1% $Fe_2(SO_4)_3$	.567 %	"	8.6 "
18	1 "	3 "	"	.536 %	"	13.8 "
19	1 "	4 "	"	.526 %	"	15.2 "
20	1 "	5 "	"	.576 %	"	7.1 "

No	Amount of Ore	Amount Solution	Kind of Solution	Assay of Toils	Time of Leaching	% Ext.
21	1 A.T	1 A.T.	1% HCL	.599 %	2 34 hrs	3.4%
22	1 "	3 "	1° "	.536 %	"	13.8 "
23	1 "	4 "	"	.609 %	"	1.8 "
24	1 "	5 "	"	.536 %	"	13.8 "
25	1 "	1 "	1% Fe CL₂	.620 %	"	———
26	1 "	3 "	"	.546 %	"	12.0 "
27	1 "	4 "	"	.493 %	"	20.4 "
28	1 "	5 "	"	.526 %	"	15.2 "
29	1 "	1 "	1% Fe CL₃ + NaCL	.599 %	"	3.4 "
30	1 "	3 "	"	.526 %	"	15.2 "
31	1 "	4 "	"	.599 %	"	3.4 "
32	1 "	5 "	"	.536 %	"	13.8 "

As can be seen from the results, the extraction
varies from 20% to nearly nothing. The 20% is equiliv-
ant to about 2.5 pounds of copper or a value of 37.5 cents.
With sulphuric acid at 2 cents a pound, a one per cent
solution would cost 40 cents. We can, therefore see
that it costs 40 cents worth of acid to extract 37.5
cents worth of copper. In this way we can see that the
copper will not even pay for the cheapest solutions,
to say nothing of the costs for labor, putting the ore
in the mill, and the other incidentals that would be
necessary to carry on the process. As the other ex-
tractions are even lower than this, we can say that with
our methods and solutions, the leaching of this ore
could not be carried on at a profit.

From the experiments that have been carried on
by other men, notably by David Cole, of the Cananea
Copper Company, we know that the oxidies and carbonates
will go into solution easily, and they have given nearly
100% extraction with sulphuric acid or ferric sulphate.
As our results from the leaching by these solutions
give as the highest extraction only from 18 to 20%, we
can assume that the copper that did not go into solution
was in some form other than the oxide or carbonate.

Thus it would seem, from our results that 80% of
the copper is held there as the sulphide. As stated
before, in order to get the copper by leaching, the sul-
phides would have to be changed first to the sulphate

or chloride by some expensive method. From this, it would follow, that in order to get all of the copper by the solutions that we used, it would be necessary to add to the cost of leaching, the expense of changing the sulphides to the sulphate or chloride, and with as low grade an ore as this one, this would be out of the question because of the cost.

We might now go ahead and make some assumptions as to the properties that a solution that would leach this ore economically, should have. It would of necessity be cheap as the margin of profit, at the best, is small. The solution must next have the power to dissolve the sulphides of copper, as well as the oxide or the carbonate. This is true because most of the copper in this ore is there as the sulphide. Next the solution must be such that the copper can be precipated from it easily and with little waste consumption of the precipating agent. Again, the solution must be such that the cement copper that it gives will be pure or containly only impurities that can be easily removed by refining.

The authors are certain that none of their solutions fullfilled any of the above conditions. They are also sure that unless some solution is found that will fullfill these conditions better than any that have been tried so far, the possibility of of the economical leaching of this ore will be small.

The references used in this thesis are as follows:—

I. W. Eissler. The Hydro—Metallurgy of Copper.

II. W. Greenawalt. Hydro—Metallurgy of Copper.