UNIVERSITY OF NEVADA

THE MOST ECONOMICAL METHOD OF TREATING THE SULPHIDE ORES OF GOLDFIELD.

A Thesis

Submitted to the Faculty of the College of Engineering

In Candidacy for the Degree of

Mining Engineer.

By

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Mines Thesis
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THE MOST ECONOMICAL METHOD OF TREATING THE SULPHIDE ORES OF GOLDFIELD.

Since the advent of local treatment the problem of recovering the greatest amount of the contained values in the various sulphide ores of the Goldfield District has been one upon which no small amount of experimenting has been done, and notwithstanding this fact, one upon which there is still considerable difference of opinion even among those most intimately concerned with the treatment of the Goldfield ores. The fact that the three leading milling plants of the District employ different methods of handling their sulphides will lend support to this statement.

In a series of tests and calculations, I have attempted to determine the method of handling which would yield the greatest profit on characteristic Goldfield ore.

The ores of the Goldfield District consist essentially of sulphide minerals, with a quartz gangue. The principal sulphide is that of iron in the form of both pyrite and marcasite, the former predominating, the latter being incidental in amount; also occurring in small and variable quantities are sulphides of copper, antimony, arsenic, bismuth and the sulphide, chloride, sulph-antimonides and sulph-arsenides of silver, but the silver found in the Goldfield ores is so small that it is of very little commercial importance. The mineral, Goldfieldite, peculiar to this district alone, is composed of copper sulphide, bismuth, arsenic, antimony and tellurium and is of quite frequent occurrence.

Copper sulphide is found associated with sulphide of arsenic, forming the mineral enargite.

Sulphides of arsenic and antimony are found associated with copper sulphide, forming the mineral famatinite. Copper is also found associated with the sulphide of iron, forming chalcopyrite.

Owing to the frequent occurrence of copper minerals in the Goldfield ores, I have, in addition to the experiments on the most characteristic sulphide ores of the District, extended the same methods of treatment to samples containing noticeable amounts of the copper mineral enargite. The first lot of samples is from the Combination Fraction mine and are typical of the greatest number of the ores of this district. Silicified dacite and blue quartz form the gangue and pyrite, marcasite and chalcopyrite the noticeable minerals, although in quantity the latter is exceedingly small. The gold is in a free state and readily yields between 30 and 45% to amalgamation by grinding to 16 mesh.

The second lot of samples is from the Jumbo Extension mine and these samples are similar in every respect to those from the Fraction mine, but they contain copper in quantities varying from 0.27% to 0.54%. The copper bearing mineral is enargite which is a compound sulphide of copper and arsenic ($3\,Cu_2S$-As_2S_3). The treatment of ores coming from the Combination Fraction mine will first be considered in detail; after which I will present the results of cyanide tests on the samples from the Jumbo Extension mine, as the quantity of this character of ore is too small to justify a detailed consideration. However, the effect of a small amount of copper mineral in practically the same character of ore, may be of interest, hence their addition to this thesis.

The samples from the Combination Fraction assayed $14.40 per ton. The concentrates from this ore averaged $103.40 per ton. Wilfey concentrators were used after the product had been ground to 16 mesh, in the batteries.

The first method of treatment to be considered will be that of concentration before sliming and the shipment of the concentrates to the smelter. Sliming the tailings from the concentrators and their treatment with potassium cyanide then follows. This method has been followed at the Nevada Goldfield Reduction Mill on almost the entire production of the Combination Fraction mine, except at such times as the sands have been separated from the slimes and leached separately. For the concentration of the coarse product (12 to 30 mesh) necessarily obtained where regrinding is employed, the Wilfey concentrator is most suitable, and has been and is being used at the Nevada Goldfield Reduction Mill. By this method of concentration approximately 61% of the sulphides are saved. The values of the concentrates range from $40.00 to about $200.00 per ton, but for convenience and better comparisons with future tests, I will base all calculations on the grade of concentrates above mentioned, viz. $103.40 per ton. The freight rate on this grade of concentrates would be $9.25 per ton, while the smelter charges would be $8.50 per ton. The product would contain approximately 10% moisture, which would bring the freight to $10.12, making a combined freight and treatment cost of $18.62 per ton. The smelters pay $19.50 per ounce for gold in concentrates assaying between five and fifteen ounces per ton, which would deduct $2.59 from the assay value of each ton of concentrates, as an ore assaying $103.40 per ton would represent a gold content of 5.17 ounces. As all calculations in this thesis are based on a valuation of

$20.00 per ounce for gold, the above settlement would favor the smelters to the extent of $.50 per ounce, which would be a deduction of $2.59 per ton of concentrates. In addition to the above costs and deductions, the following would necessarily enter against concentration:

	Per Ton Ore Milled.	Per Ton Concentrates
Power	$ .031	$1.39
Labor	.04	1.80
TOTAL	$ .071	$3.19

This would give a total cost of $24.40 per ton of concentrates, leaving $79.00 or 76.41% net profit. The power costs are based on 6 Wilfey tables handling 75 tons ore in 24 hours, power costing $.0135 per kilowatt hour. From shipments made, forty-five tons of Fraction ore are necessary to produce one ton of concentrates, which justifies the use of this ratio in the above table.

From the Wilfey Concentrators the tails are lifted by Frenier sand pump, first to a 4 foot cone classifier with 3/4" discharge, where a portion of the slimes are taken off, thence the remaining product drops directly into an Akins classifier, which permits of a clean product of sands being delivered to the tube mill. The product from the cone and Akins Classifiers are directed to a common launder which leads to the Frenier Sand pumps and these lift the slime to the settling tank where the solution is drawn down as closely as possible, leaving the pulp at a consistency of about 150 and 170 specific gravity. The latter density is attained and easily handled by the agitators and pumps on a product giving the following screen test:

Remaining on Mesh	Per Cent
80	2.5
100	11.5
120	6.5

Finer grinding than this screen test is required on the raw ores of the district but this degree of fineness has been found sufficient and most economical on some old tailings that are being worked at the mill.

The object of this close decantation is obviously to get the pulp as free as possible from water and replace it with stock KCN solution, thereby preventing an over-accumulation of the latter. Sufficient stock solution is added to make the pulp of about 1.33 specific gravity, at which density agitation is usually carried on. During the addition of the stock solution, sufficient potassium cyanide is added to bring the strength up to at least two pounds to the ton of solution.

In test 8B given below in which the sulphides were panned out of a sample coming from the mortar, but not amalgamated, I have followed the general mill practice prior to the time that concentration was abandoned in favor of sliming and cyaniding the concentrates with the remainder of the pulp. This test and all others which follow were made in closed bottles strapped to a revolving wheel. Samples were taken from these bottles at periods stated in each of the tables which follow, by first shaking the bottle well, then pouring a small portion therefrom. While this is the only satisfactory method of taking samples from bottles, still it is not wholly reliable unless the pulp is of a very uniform grade of fineness and specific gravity. Samples thus taken are necessarily small and it is plain that an un-uniform mixture of the heavier and lighter particles will give erroneous results. In this work, notwithstanding great care on this particular point, I have experienced errors which check assays traced directly to this cause.

However, the final residues would not be affected from this cause and their results will be given preference over intermediate samples even should the latter show better extractions.

TEST 8B

Hours agitated	KCN per ton	Protective Alkalinity per ton	Oz.Gold per ton	Value per ton		Per Cent Ext.
0	3.7	1.3	.44	$8.80	Orig.	----
12	2.2	.1	.03	.60)		93.1
15	2.4	1.5	---	----)		----
23	2.1	1.3	.03	.60)	Tails)	93.1
32	.6	.2	.01	.20)		97.7
45	1.8	1.0	.02	.40)		95.4

KCN and Protective alkalinity in the above table as also in all those which follow are given in pounds per ton of solution. The specific gravity of the above sample was 1.332 or a ratio of 2.1 part solution to 1 of pulp. The fineness is shown by the following screen test:

	Mesh	Per Cent.
Passed	120	99.6
"	150	98.9
"	200	96.9

Here 93.1% of the total values in the slimes was recovered in 12 hours at a cost shown in the following table:

	Tube Milling	Cyaniding	Filtering
Power	$.198	$.036	$.066
Labor	.02	.06	.107
Potassium Cyanide		.693	-
Lime		.106	-
Lead Acetate		.028	-
Materials and up-keep	.063	.014	.047
	$.281	$.937	$.220

Not included in these costs is a double lift with Frenier sand pumps which would add $.038 per ton or a total of $1.476. The above figures for tube milling are based on a 5 x 22 foot mill making 24 revolutions per minute; of cyaniding with a 75 ton charge of 1.33 specific gravity, agitated with arms and one 6" Butters Centrifugal Pump. Of filtering, upon an installation of 54 leaves operated sixteen hours per day, the pulp being circulated with one 6" Butters Centrifugal Pump.

From Test 8B, 23 hours agitation showed no greater extraction, while 32 hours shows an apparent increase of 4.6%, while the final results of 45 hours agitation shows only an increase of 2.3%. This apparent increase in extraction after 32 hours agitation is accounted for by the uncertainty of samples poured from bottles, as previously explained, and as the extraction appears higher at this point than at the end of 45 hours agitation, the results of 32 hours will not be given consideration here. The total remaining residue is more reliable and will be considered instead. 45 hours agitatio in Test 8B shows an increase in extraction of value of 2.3% of $8.80 or $.202. The potassium cyanide consumed amounts to .6 pounds per ton of solution or 1.26 pouhds per ton of ore, which at $.22 per pound amounts to $.277. Figuring on the above cost of $.11 per ton for 12 hours agitation, 33 hours agitation together with the cyanide consumed during this time, would total $.577 for a recovery of $.202 which marke this as uneconomical. As previously shown, the concentrates when shipped to the smelter netted 76.41% of the assay value.

The ore from which the sample used in test 8 B was taken, assayed $14.40. The tails, after concentration, assayed $8.80, $5.60 going into the concentrates. The net profits on the concentrates would be 76.41% of $5.60 or $4.278. Test 8 B shows an extraction of 93.1% of $8.80 or $8.19 net, a cost of $1.476 which gives a net profit of $6.714 per ton on the tails, or a total net profit of $10.504 per ton, which is a saving of 76.33%

METHOD NO. 2.

This method consists of sliming and cyaniding direct, without concentration or amalgamation. The pulp used in test 8 B was taken from the sample from which test 8B was made.

TEST 8 A.

Hours agitated	KCN per ton	P.A. per ton	Oz.Gold per ton	Value per ton	Extracted Per Cent
0	4.7	2.4	.72	$14.40 Orig.	----
12	3.6	011	.08	1.60 T	88.8
15	3.6	0.8	---	---- a	----
23	3.5	.5	.10	2.00 i	86.1
32	3.3	.5	.05	1.00 i	93.
45	2.9	.5	.05	1.00 s	93.

Specific gravity of Test 8 A was 1.184 or a ratio of solution to pulp of 3 to 1. Fineness same as in test 8 B. In test 8A 12 hours agitation shows a consumption of 3.3 pounds cyanide per ton or 72.6 cents, while 32 hours shows 4.2 pounds or 92.4 cents per ton. The extra hours of agitation cost 18.2 cents or a total cost of 38 cents.

The increased extraction of 4.2% amounts to 60.5 cents or a profit of 22.5 cents per ton. Since longer agitation gives greater cyanide consumption without greater extraction, it is apparent that 32 hours is the most economical time of agitation. 32 hours agitation would be conducted at a cost shown by the following table:

Power	$.096
Labor	.16
KCN	.924
Lime	.17
Lead Acetate	.028
Materials and up-keep	.037
	$1.415

Tube milling and filtering as previously given would bring this to $1.916 per ton. Test 8 A shows an extraction of 93%

or $13.392. Deducting the above cost from this would give us a net profit of $11.476 or 79.69%.

That this ore will yield its value to a much weaker solution without any extension of time, is shown by Test 11A. In fact Test 11A shows a slightly higher extraction in a shorter period of time than Test 8A, but this is due to the fact that the pulp used in Test 11A was ground finer than that of 8A, all having passed 150 mesh and 97.98% passed 200 mesh in 11A. For the sake of comparison, I will here quote a screen test made on mill charge No. 94, which is representative of the fineness of grinding on raw ore at the Nevada Goldfield Reduction Mill.

	Mesh	Per Cent
Passed	150	96.8
"	200	90.4

TEST IIA

Hours Agitated	KCN per ton	Protective Alkalinity per ton	Oz.Gold per ton	Value per ton		Per Cent Extracted
0	2.0	1.2	.72	$14.40	Orig.	0.
12	.4	.0	.06	1.20)	T	91.4
16	2.6	1.4	---	----	a	----
22	2.2	1.2	.04	.80)	i	94.5
32	1.7	1.0	.04	.80)	l	94.5
45	1.6	.8	.04	.80)	s	94.5

Specific Gravity 1.184 or a ratio of 3 of solution to 1 of pulp.

While Test IIA shows a higher extraction in shorter time than Test 8A, the final net returns of the latter are slightly in excess, due to a higher consumption of cyanide. The higher consumption of cyanide in a weaker solution is clearly due to the latter also being weaker in alkalinity.

METHOD NO. 3.

This method consists in sliming, cyaniding and concentration after cyaniding. For use in the determination of the practicability of this method, a sample of charge No.94 was taken at the Butters filters from which the sulphides were panned and assayed. The sample assayed $8.00 per ton; samples of previous charges agree very closely with this test but the samples taken for this work are from the same lot of ore as that treated in Charge No.94, hence its use as a basis of calculation at this point. Freight from Goldfield and treatment at smelter on concentrates averaging under one ounce per ton would total $8.00 per ton, thus consuming the entire values in the concentrates for these two items alone. The smelters pay $19.00 per ounce for gold in concentrates averaging under five ounces per ton. This in addition to the concentration costs would mark method No.3 as being conducted at a loss.

METHOD NO. 4.

Concentrating, sliming and cyaniding of concentrates and slimes separately. From a number of tests run on the same concentrates previously mentioned, the best results were obtained with a solution of 7.8 lbs. per ton at the start and .7 lbs. at the end of 45 hours agitation, the solution not having been strengthened during the period of agitation; these results are shown in Test 6 B.

TEST 6 B.

Hours agitated	KCN per ton	Protective Alkalinity per ton	Oz.Gold per ton	Value per ton		Per Cent Extracted
0	7.8	1.4	5.17	$103.40	Orig.	----
12	3.4	0.15	1.64	32.80	Tails	68.2
12	3.4	2.7	----	-----		----
22	2.2	2.7	1.08	21.60		79.1
33	1.1	2.6	.80	16.00		84.5
45	.7	2.5	.66	13.20		87.16

Ratio of solution to pulp 2-1/2 to 1, all passed 150 mesh screen, 99.2% passed 200 mesh.

Longer agitation might possibly have given a higher extraction but when it is recalled that those sulphides assayed $8.00 per ton or 92.2% extraction in mill Charge No.94 after 36 hours agitation, with the remainder of the pulp, against $13.20 or 87.16% when treated separately for 45 hours, it is apparent that better results at a lower cost can be obtained by treating the sulphides with the remainder of the pulp. This would mark Method No.4 as being uneconomical.

Test 6A with a 2 pound solution,- Ratio of solution to pulp 2.2 to 1,-gave an extractinn of 59.6% on these sulphides; Test No.4 with a 4 pound solution,- Ratio of solution to pulp, 2 to 1,-82.3%; Test 6C with 7.8 pound solution, same as 6B, Ratio of solution to pulp,-1-1/2 to 1, 78.6%.

The conclusion to be drawn from the foregoing experiments is that Method No.2 should be adopted. Methods Nos. 3 and 4 were shown to be uneconomical in favor of No.2, while the latter method shows a net recovery of 79.69% against 76.33% in No.1. In simplicity, Method No.2 also excels. In the adoption or arrangement of a plant, Method No.2 would have the advantage in that the installation of concentrators or additional agitators would be avoided, thus effecting a considerable saving in machinery and in building space.

In this work I have purposely omitted salaries, precipitation, assaying, refining and other costs which did not have a direct bearing on a comparison of the different methods. The cost items given, particularly those of power, are influenced by the fact that the mill in consideration, is built partly on the flat plan, requiring the use of Frenier sand pumps to deliver the pulp from batteries to the tube mill and a second lift by these pumps to deliver the tube mill product to the agitators. From the agitators the pulp is delivered to the Butters filters by 6 inch Butters Centrifugal pumps. This item is included in agitation costs.

TESTS ON JUMBO EXTENSION COPPER-BEARING SAMPLES

Test No. 1

Hours Agitated	KCN Per Ton	Protective Alkalinity Per Ton	Oz.Gold Per Ton	Value Per Ton	Per Cent Extracted
0	2.4	2.7	1.72	$34.40 Orig.	0
4	2.0	.5	----	-----	----
5	2.0	2.1	----	-----	----
16	1.7	2.1	.52	10.40)	69.8
16	2.05	2.0	---	-----)T	----
25	1.4	1.6	.38	7.60)a	78.0
25	2.2	1.6	---	----)i	----
41	1.7	1.4	.32	6.40)l	81.5
41	2.2	1.4	---	----)s	----
49	1.6	.9	.32	6.40	81.5

This sample contained .54% copper, specific gravity 1.184, or a ratio of 3 parts solution to 1 of pulp. Fineness all passed 150 mesh, 97.6% passed 200 mesh.

TEST No. 2.

Hours Agitated	KCN Per Ton	Protective Alkalinity Per Ton	Oz.Gold Per Ton	Value Per Ton	Per Cent Extracted
0	2.4	2.7	2.22	$44.40 Orig.	0.
4	2.1	1.1	----	-----	----
16	2.0	.8	1.26	25.30) T	43.3
25	2.0	.8	.88	17.60) a	60.4
41	1.9	.7	.84	16.80) il	62.1
49	1.9	.6	.72	14.40) s	67.6

The samples from which Test No.2 were taken assayed .27% copper. Specific gravity of Test No.2, 1.184, or a ratio of 3 of solution to 1 of pulp. Fineness 87.9% passed 150 mesh; 80.7% passed 200 mesh.

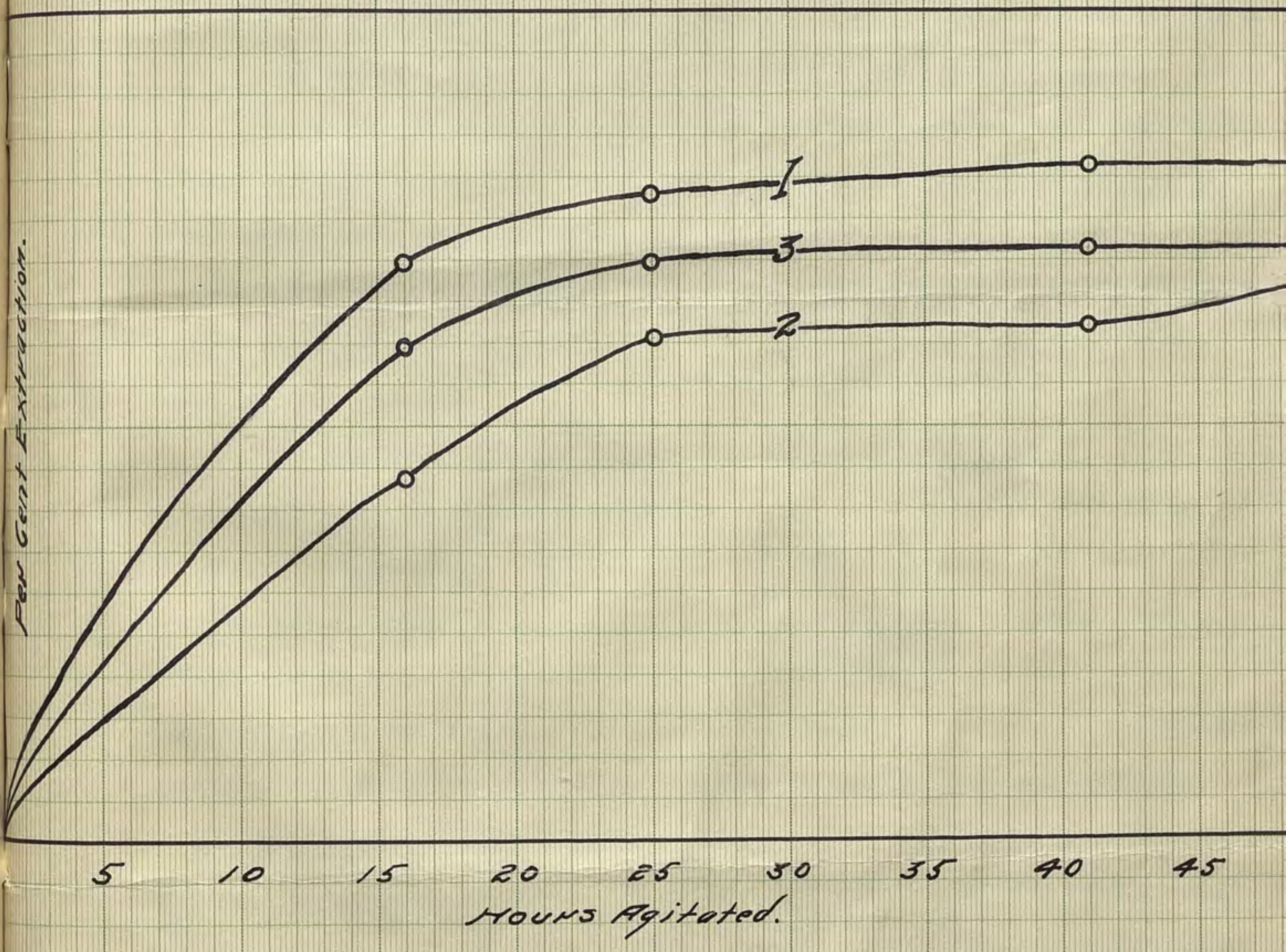
Test No. 3.

Hours Agitated	KCN Per Ton	Protective Alkalinity Per Ton	Oz. Gold Per Ton	Value Per Ton		Per Cent Extracted
0	2.4	2.7	1.52	$30.40 Orig.		0
4	2.2	.8	----	----		----
5	2.1	2.6	----	----		----
16	2.0	2.5	.62	12.40)	T	59.2
25	1.7	2.1	.46	9.20)	a	69.8
25	2.1	2.1	----	----)	i	----
41	1.7	2.0	.44	8.80)	l	71.1
41	2.1	2.0	----	----)	s	----
49	1.7	1.6	.44	8.80)		71.1

Copper content .34%. Specific gravity 1.184. Fineness 96.7% passed 150 mesh; 85.3% passed 200.

The effect of fineness is noteworthy in the three foregoing tests. Regardless of copper content, the extractions appear almost proportional to the size of the product, notwithstanding the fact that all are ground to what is generally considered a good slime. That the copper had its effect, however, may be seen by comparing the results of test No. IIA previously given and Test No.1. IIA contained no copper. The screen analyses on these two tests were almost identical, 97.98% having passed 200 mesh in IIA, while 97.6% was finer than 200 mesh in Test No.1.

As the difference in extraction in these three tests, due to difference in size, is somewhat pronounced at a stage of fineness where such would hardly be expected, especially as conditions under which all were run, were the same, I will plat the curves.


Per Cent Extraction.
1
3
2
5
10
15
20
25
30
35
40
45
Hours Agitated.

Test IIIA.

Hours Agitated	KCN Per Ton	Protective Alkalinity Per Ton	Oz. Gold Per Ton	Value Per Ton		Per Cent Extracted
0	2.7	2.5	.84	$16.80	Orig.	0.
11	2.5	2.5	.38	7.60	Tails	54.8
$20\frac{1}{4}$	2.4	2.3	.26	5.20		69.0
$35\frac{3}{4}$	2.2	2.3	.20	4.00		76.8
$43\frac{3}{4}$	2.0	2.0	.18	3.60		78.6

Specific gravity 1.193. Fineness same as Test No.3

Test No. 3A was made from a portion of the same samples from which Test No.3 was made, but in 3A, the sulphides were panned out. This would indicate that a recovery of 88.1% could be made on sample No.3 by concentration, shipping concentrates and cyaniding tails, against 71.1% by sliming and cyaniding concentrates with remainder of pulp.

It would be impossible to determine in any manner, aside from a regular mill run, on this class of ores, what effect the high amount of sulphides of copper and arsenic would have on the precipitation in the zinc boxes, on the bullion produced and on fouling of solutions, but it is obvious from a comparison of the above tests, that the gold in the sulphides of this class of ore does not yield readily to cyanide. Since the specific gravity of enargite is 4.44, considerable of that mineral would also go into the concentrates, thus giving a slime much more suitable for cyaniding, and by close concentration it is possible that the rebellious minerals would be reduced to a point wherein little trouble would be given in precipitation or fouling of solutions, and wherein a marketable bullion would be obtained. This in connection with the fact that concentration and sliming tails showed a recovery of 88.1% against 71.1% by sliming and cyaniding concentrates with remainder of pulp, would indicate that close concentration, shipping of concentrates, and cyaniding tails would be the most profitable method of local treatment of these ores, and one which would compete favorably with direct shipment to the smelter, which is the present practice.

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