

RESEARCH ON WISCONSIN ORE

- by -

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in the Department

of

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For reference in flotation, "Flotation" by A.M. Gaudin was used freely; for the Cyanidation, "Manual of Cyanidation" by Hamilton was used, while for the roasting tests, advantage was taken of Bulletins of the Australian Institute of Mining and Metallurgy.

PROBLEM

To devise an economically profitable method of treatment for the ore of the Wisconsin Mine.

CONCLUSION

The results of the examination were generally unsatisfactory. The results of the tests and the microscopic examination established the fact that it is improbable that the ordinary methods of milling can give a high recovery of gold.

Four principal processes were tried. They were flotation, flotation followed by cyanidation, roasting and cyanidation and straight cyanidation.

Flotation, under the present circumstances, is not practical due to the small size of the gold particles and the difficulty of separating arsenopyrite and pyrite. A recovery of 96% of the gold can be obtained in 62% of the feed.

Flotation followed by cyanidation was not satisfactory.

Low temperature roasting before cyanidation did not improve the recovery.

Straight Cyanidation has greater possibilities than any of the other methods used in these tests. While the preliminary treatment has no effect upon the recovery of the gold it is evident that fine grinding warrants consideration.

We understand that mill heads can be maintained at 0.35 oz. or \$12.20 per ton, with gold at \$35 per ounce. With a 59% recovery, this gives a gross production of \$7.25 per ton. If the tonnage is sufficiently large, the property might be operated at a profit.

If the recovery rose to 66% (the recovery for fine grinding),

the gross production would be \$8.05 per ton. Whether the extra cost of installation, operation and depreciation of the additional equipment necessary would be more than \$0.80 per ton should be ascertained.

Smelting is the most logical course, but as this is impossible under the existing circumstances, it is apparent that a satisfactory method of treatment has not yet been established.

LOCATION OF PROPERTY

The property is in the Nelson Mining Division, two or three miles south-southwest from the forks of Midge Creek, which empties into the east side of Kootenay Lake; about 20 miles southerly from Proctor, B.C.

The mine is reached from the railway at Midge Creek station. A narrow road is practically completed for about $2\frac{1}{2}$ miles up the creek; from there to the mine there is a pack trail which can be improved into a road. The total distance from the railway to the mine is fourteen miles. The elevation at the mine is about 6300 feet, which is about 4500 feet above the railway at Midge Creek.

ANALYSIS OF ORE

Au - 0.44 oz./ton	Pb - 0.33 percent
Ag - 1.88 oz./ton	Sb - Tr.
Fe - 29.7 percent	S - 30.5 "
Cu - 0.43 "	Insol. - 29.5 "
As - 8.50 "	CaO - --
Zn - 0.93 "	MgO - --
Arsenopyrite	18.5 percent
Pyrite	48.9 "
Galena	0.35 "
Chalcopyrite	1.2 "
Sphalerite	1.5 "

CHARACTER OF ORE

The bulk of the ore was crystalline masses of pyrite and arsenopyrite which were fractured and brecciated, with chalcopyrite forming veinlets along the fractures. In these veinlets were also sphalerite, galena, telluride mineral, tetrahedrite (?) and small particles of an unknown mineral.

Part of the gold occurs in finely disseminated particles fairly evenly distributed. The presence of a gold mineral is indicated by superpanning and infrasizing results. Microchemical tests show the presence of a telluride, and fusion on a pyrex glass shows a gold telluride. In the chalcopyrite can be seen minute particles, too small to be identified, of a yellow mineral which may be gold.

The principal gangue mineral is quartz.

SCREEN ANALYSIS

The - 200 mesh products were sized by the Haultain Infra-sizer and microscope.

20 Minute grind,

1000 grs. ore and 1000 grs of water in Mill.

INFRASIZING RESULTS FOR - 200 MESH

MESH	% Wt
‡-65	0.0
65/100	0.1
100/150	0.2
150/200	2.4
-200	<u>97.3</u>
	100

MESH	% Wt.	GOLD % Dist.
‡ 250	19.8	25.1
250-400	21.7	14.4
400-560	18.9	17.3
560-850	12.3	19.5
850-1100	8.7	7.0
1100-1700	7.9	8.4
- 1700	<u>10.5</u>	<u>8.3</u>
	100.	100.

20 Minute grind

500 grs ore and 500 grs water in mill.

MESH	% Wt
‡ 250	0.36
250-400	14.00
400-560	28.40
560-850	22.85
850-1100	15.04
1100-1700	7.08
- 1700	12.20
	100.

PREVIOUS METALLURGICAL WORK

Previous work on the Wisconsin ore has been carried out by the Ore Dressing and Metallurgical Laboratories of the Department of Mines at Ottawa, the Consolidated Mining & Smelting Company of Canada, Limited at Trail, B.C., and the Ore Dressing Laboratories at the University of B.C.

The results at Ottawa showed that Flotation was not practical and that by Cyanidation only slightly over 50 percent of the gold could be recovered. Smelting was recommended as the only profitable means of treating the ore, although the feasibility was not thoroughly investigated.

The C.M. & S. Co., tests were principally conducted along the lines:

1. Roasting & Cyanidation. Recovery 20%
2. Roasting & Chlorination. Recovery 80%
3. Roasting with 5% Soad Ash (McKay method) followed by Cyanidation. Recovery 82%

4. Amalgamation. Recovery 3%
5. Tabling - Recovery depended on bulk present.
6. Direct Cyanidation. Recovery 42%
7. Flotation - Unsuccessful

The roasting methods were considered by the Company to be unsatisfactory. As the temperatures varied between 780° and 1450°C this process cannot be used due to the proximity of the Wisconsin property to the American border and the large amount of sulphur and arsenic fumes that would be evolved.

Flotation Tests only were made at the University. The results were not satisfactory. The series of tests indicated that (a) the gold recovery in a concentrate depended almost entirely upon the bulk of that concentrate and (b) that it was not possible to separate Arsenopyrite and pyrite by selective flotation.

Two tests (6,13) were selected from this series as having the best possibilities for satisfactory results, & served as a basis for the earlier experiments in the present work.

As the duplicate #6 was abandoned without assaying it will not be recorded, but #13 is as follows:

TEST #13

Charge to Ball Mill

Ore - 1000 grs

Water - 1000 grs

Lime - 2#/ton

KCN - 0.1#/ton

Time of Grinding - 20 minutes.

CONCENTRATE 1

Added	Aerofloat #15	- 0.079#/ton
	Cresylic Acid	- 0.068#/ton

Conditioned - 2 mins

Skimmed - 10 mins

pH - 10.1

CONCENTRATE 2

Added Cu SO₄ - 1.0# /ton

Skimmed - 10 mins

CONCENTRATE 3

Added Conc. H₂ SO₄ - 5 cc.

#301 - 0.02 #/ton

Pine Oil #5 - 0.08 #/ton

Conditioned - 2 mins

Skimmed - 15 "

pH. - 3.9

CONCENTRATE 4

Skimmed - 10 Mins

RESULTS

PROD.	Wt Grs.	% Feed	Au oz/ton	Au % Rec.	Ag oz/ton	Ag % Rec.	Cu %	Cu % Rec	As %	As % Rec
C1	30.0	3.0	2.40	21.2	16.48	23.6	11.2	86.5	12.2	2.5
C2	25.0	2.5	0.60	5.6	4.12	6.1	1.2	10.2	16.6	3.1
C3	330.0	33.0	0.43	42.7	2.60	45.3	Tr	--	30.9	62.4
C4	30.0	3.0	0.40	4.8	4.00	5.2	Tr	--	16.0	4.3
T	<u>585.0</u>	58.5								
	1000.0			74.3		80.2		96.7		72.3

It must be noted before comparing this test with similar ones described later, that the head sample for the ore used in above

test ran 17.55% Arsenic and 35.7% Iron.

PREPARATION OF ORE PRIOR TO TESTING

Approximately 25 pounds of ore that had been ground to pass through a 10 mesh screen remained from the previous year. It was thought that there would be insufficient for the proposed tests, so a new batch weighing about 50 pounds was secured. This was stage-ground to 100% - 10 mesh and mixed with the first batch, thus ensuring a supply of ore that would be consistent throughout and fairly representative. Both samples had been exposed to possible oxidation for over a year, but it was unavoidable.

By constant riffing about 400 grs. of ore were obtained from the above mixture. This amount was disc-pulverized until it in entirety passed through a 100 mesh screen. It was split once more and 200 grs obtained which provided a good representative sample from which the head assays were determined.

F L O T A T I O N

CONCLUSIONS

RECOMMENDATIONS

THEORY

&

TESTS

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P R E A M B L E

The objective in the flotation tests has been the recovery of gold in some suitable form which will render it amenable to one or more subsequent processes.

During these tests, two possibilities have been kept in mind, first to concentrate the gold in a sufficiently small bulk and high content to warrant shipping to a smelter for further treatment. It was necessary to reduce the amount of metals such as arsenic to a quantity insufficient to incur penalties. The second possibility was to obtain a concentrate containing over 20% Arsenic so that it could be shipped as an arsenic concentrate, containing gold. Both courses were found to be impossible.

The reason for the impossibility of the concentration of the gold in a small bulk was probably due to the very fine dissemination of the gold throughout, both as native gold and a gold telluride. The cause of failure for the second case was due to the intergrowth of the pyrite and arsenopyrite and the necessity of extremely fine grinding to unlock one from the other. Another reason is the close similarity between pyrite and arsenopyrite in their reactions to flotation reagents. This has been confirmed by¹ Wark & Cox in their research work with contact angles. In it they show that the two minerals react in the same manner, there being only a very slight difference in sensitivity to depressants when the pH is high, around 11. At this range the arsenopyrite is slightly more reactive and can be depressed, but extreme care must be taken in the regulation of the reagents.

1. I.W. Wark & A.B. Cox - "Principles of Flotation, VI"

CONCLUSIONS

1. Arsenopyrite & pyrite cannot be separated by selective flotation, with the present reagents. This is discussed under the previous heading.
2. A higher percentage of arsenic in a concentrate can be obtained when the pulp is slightly acid.
3. Practically without exception the percentage recovery of the gold depends, in direct proportion, on the bulk of the concentrate collected.
4. The use of sodium sulphite to replace potassium cyanide does not appreciably affect the recovery and has no effect in the selective flotation of arsenopyrite and pyrite.
5. Collectors #208 and #301 added to the ball mill and aero-float 15 to the cell, gave the best gold recoveries.
6. The collector 2-6 (Pentacol Xanthate) apparently was no different from the other collectors in its action.
7. No advantage is to be gained by cleaning a bulk concentrate.

RECOMMENDATIONS

1. Should any further work be done in the flotation of an arsenopyrite concentrate, it would be advisable to keep the pH between 5.0 and 7.0.
2. Any attempt to separate the two should be made at a pH of 11.0.

THEORY OF FLOTATION

While this paper is not concerned with flotation,^{only} it is thought advisable to present a short review showing the relationship and importance of the various factors.

HISTORY

Flotation has passed through many stages in the evolution of the modern process.

The first attempt in modern times was the separation of sulphides from earthy matter by means of oily agents. Later results were improved by the use of modifying agents such as acids and acid salts². A further stage was the introduction of gas as the buoyant medium, resulting in a decided reduction of the amount of oils used. It was not until 1924 that Perkins³ patented the use of Xanthates, thus marking the beginning of the modern trend of flotation by chemical collectors.

Many discoveries in the use of inorganic salts have had far-reaching results in the flotation of and concentration of minerals that could not be treated previously, and they have advanced the process of selective flotation considerably.

The accurate classification of the reagents is difficult, but for ordinary purposes it is desirable to discuss them largely in terms of their functions. They will be discussed under the headings of;

- | | |
|-----------------------------|-------------------------|
| (1) Frothers | (4) Depressants |
| (2) Collectors or promoters | (5) Sulphidizing Agents |
| (3) Activators | (6) Regulators |

2. Rickard, T.A.- "History of Flotation". Mining Sc. Press, 114. Pgs 365 - 369; 401 - 406.

3. U.S. Patent #348,157.

FROTHERS

A frothing agent must create a froth capable of bearing non-wetted particles denser than water to the surface, but may or may not possess the property of making the minerals less wettable by water. Pure liquids do not froth, as only substances which have different surface tensions are capable of foaming. Organic compounds are used almost invariably, as small amounts dissolved in water lower the surface tension very appreciably. This may be understood from a consideration of Gibbs adsorption equation -

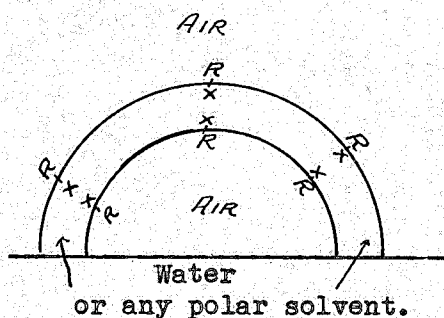
$$a = \frac{C}{RT} \frac{dr}{dc} \frac{dr}{dc}$$

in which "a" is the amount of material adsorbed on the interface, T is the absolute temp., R is a constant, C the concentration in the bulk of the liquid and $\frac{dr}{dc}$ the rate of change of the surface tension with concentration. In organic compounds, where surface tension is low and capillary active substances are easily adsorbed, $\frac{dr}{dc}$ is large, therefore small concentrations cause a considerable drop in surface tension. In organic substances however, which tend to raise the surface tension and be poorly adsorbed, $\frac{dr}{dc}$ will be small and the concentration would have to be large to produce much change in the tension.

The extent to which the organic substances affect the surface tension of water is closely related to their structure. The length of the hydrocarbon chain has a definite effect, as pointed out by Traube. In a homologous series each higher homologue has a solubility 1/3 that of the preceeding one, while the surface tension is depressed three times, with the result that the best results are obtained with a homologue midway in the series.

Frothing compounds have structural formulae that are composed of two constituents having opposite properties; one part of the molecule is polar, which is water avid, the other non polar, which repels it. The latter part consists of a hydrocarbon chain, while the polar section may consist of oxygen in the (CO), (COOH), or (OH) forms, or nitrogen in the amino (NH₂), or nitrile (CN) form. The last two are avoided where frothing only is desired, as they ionize and therefore have collecting properties. Taggart, Taylor and Ince⁴ have come to the conclusion that all good frothers contain one polar group only.

The mechanism of the bubble may be represented by a cross section of a portion of a bubble⁵. To understand it one must remember the character of the molecules with one water repellent and one water avid portion. The non-polar part is represented by "R" and the polar part by "X"



As a result of these affinities the molecules arrange themselves as indicated, with the polar group dissolving in water and the non-polar part sticking out into the air. Consequently the bubbles are lined with a monomolecular sheath of water repellent and chemically inactive groups

4. Taggart, Taylor & Ince, "Experiments with Flotation Reagents"
A.I.M.M.E. 68 (1923)

5. "The Story of the Bubble", C.I.M.M. Bulletin, July 1935. Pg. 349.

oriented outwards, thus preventing coalescence of the bubbles and producing a more permanent froth. The effect of the dissolved substances on frothing may be briefly explained by a consideration of the film of the solution. As the film is stretched the liquid comes from the bulk of the pulp and dilutes the film, thereby increasing the surface tension and counteracting the stretching force. This gives the film a greater elasticity than that obtained from pure liquids whose surface tension cannot be increased by a stretching of the surface. Therefore by constant adsorption and changing of the surface tension an equilibrium can be quickly reached in which the stretching force balances the surface tension and ensures the stability of the froth.

The principal requirements of a froth are;

1. It must be an organic substance.
2. It's molecules should be heteropolar and consist of one or more hydrocarbon radicals attached to one polar group.
3. There should be only one polar group and it should contain oxygen in the form of the hydroxal (OH), carboxyl (COOH), or carbonyl (CO); or nitrogen in the amine (NH₂) or nitrile (CN) form.
4. It must not ionize materially.
5. Must be readily available at reasonable cost.

In practice, the most widely used frothers are cresylic acid and pine oil, with the former gradually replacing the pine oil.

Locally used frothers, such as camphor oil in Japan and eucalyptus oil in Australia, are excellent frothers and are usually less expensive than the two more important ones.

COLLECTORS

The term collecting agent goes back to the days of oil flot-

ation, at which time it was used to designate certain oils capable of bringing sulphide minerals in the froth to greater abundance.

There are two types, (a) collecting oils, which are not used in modern plants, and (b) chemical collectors which act by adhering to the surface of certain mineral particles either in their natural state or in some altered chemical form, producing a non-polar coating that repels water but attaches itself to gas bubbles. They may be divided into two varieties, those that form definite compounds by metathesis with the surface of the minerals, and those that do not.

Xanthates, which play an important part in modern practice, appear to be an example of the first variety. They act on base-metal sulphides by double decomposition between an oxidized coating and the reagent, followed by later decomposition so that the final coating need not be a base-metal Xanthate.

CHARACTERISTIC STRUCTURE OF COLLECTORS

The promoter molecules, like those of the frothers are composed of two parts, polar and non-polar. Unlike the frothers, however, the collectors ionize and the polar part of the molecule reacts with the mineral to form an insoluble compound or cause adsorption on the surface.

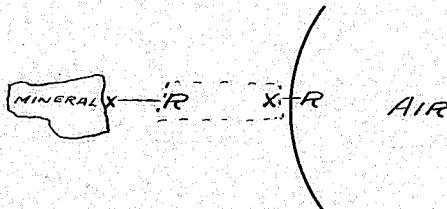
As in the case of frothers increased length of the hydrocarbon chain results in an increased effectiveness of the collector. The optimum for the number of carbon atoms has not yet been established, although at present five is thought to be the most satisfactory.

ACTION OF COLLECTORS

Many theories regarding the attachment of a particle to a bubble have been advanced, but not one has been generally accepted.

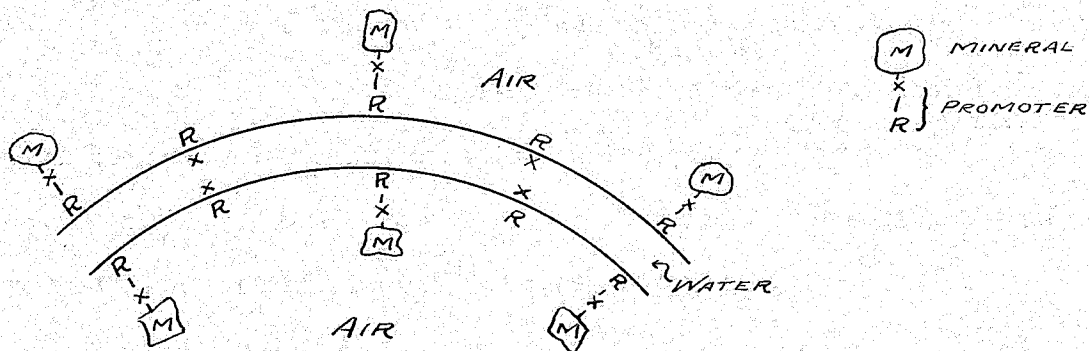
It is possible that, when the bubble is in the pulp, the

polar part of the collector unites with the mineral, providing a monomolecular surface with the hydrocarbon chain oriented outwards. This unites with the polar part of the bubble, causing a union of the particle and bubble.



Some other additional support must also be necessary. This may be due to the sinking of the particle into the bubble wall until a definite contact angle is reached.

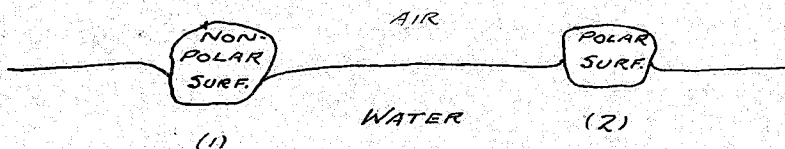
For union in the froth, a different explanation is required. A theory advanced by Dr. Christmann⁶ is that of dissolution of the hydrocarbon of the froth film, so that a mineral froth would be constituted as shown -



It is not likely that the two hydrocarbons mutually dissolve, but rather that there is adsorption between the two, resulting in the adherence of the particle to the bubble.

SELECTION IN GAS - SOLID ATTACHMENT

The gas solid attachment pictured as resulting from the encounter of bubbles and particles must be selective between minerals having different surfaces, consequently the polarity of the surfaces is important.

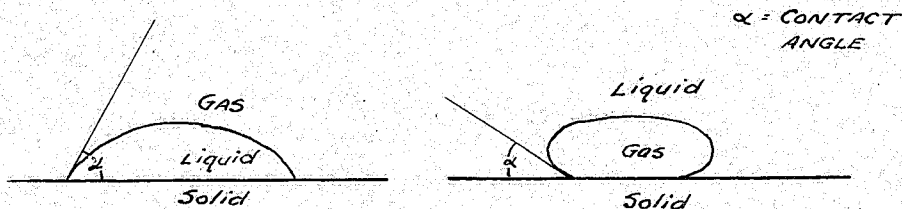


If a particle having a non-polar surface (1) encounters an air bubble the contact angles at the solid surface indicate a tendency for the gas to displace the water at the solids surface.

Similarly if a particle having a polar surface (2) encounters an air bubble the contact angles indicate a tendency for the water to displace the gas at the solids surface⁷.

CONTACT ANGLES

Much work has yet to be done on the contact angles between gasses, liquids and solid surfaces. Recent research⁸ has indicated that a great deal of valuable information, which can be used in the interpretation of flotation phenomena may be obtained by measuring the angles formed between various mineral surfaces and bubbles.



7. Gaudin, A.M. "Flotation", page 99.

8. I.W. Wark, A.B. Cox - "Transactions of A.I.M.M.E. Vol. 112

FROTH FORMATION

As the bubbles move upward there is a downward draining of the water and the suspended substances in the water. As the draining proceeds the walls of adjacent bubbles come in contact at one point and gradually assume a definite polygonal shape as the pulp drains. The particles suitably prepared adhere to the bubble surfaces and as they are close together have a filtering action which traps some of the gangue.

COLLECTORS USED IN PRACTICE

- (a) Fatty Acids and Soaps for non-metallic minerals.
- (b) Xanthates and their oxidation products.
- (c) Organic hydrosulphides (mercaptans) and their oxidation products.
- (d) Substituted thioureas (thiocarbonilid), dithiophosphates, amines, azo compounds.

FLOTATION:

ACTIVATING AGENTS

Agents which through their effect upon the surface of otherwise non floatable or poorly floatable minerals make them amenable to flotation with the usual collecting agents.

Generally speaking they are compounds which are capable of producing less soluble salts with the collecting agents than any of the metal compounds at the surface of the minerals to be floated. They must also be capable of first reacting with the mineral surface to form compounds whose solubility is less than that of the metal compounds at the surface of the minerals.

The metals whose organic salts and oxides are the least soluble are Cu, ^{Pb}~~Pb~~, Hg. Consequently they are, in salt form, the most effective activating agents.

The best example is copper sulphate in connection with sphalerite

Copper sulphate changes the surface of the blende to covellite (CuS) which is readily floated by the regular promoters.

DEPRESSANTS

The function of depressors is to prevent, either temporarily or permanently as desired, the flotation of certain mineral constituents of complex ores subjected to selective flotation without preventing the primarily desired mineral or minerals from being readily floated.

Among the more common depressants are:

LIME - has a depressing action on gold and all sulphides, particularly pyrite.

KCN - has a depressing action on pyrite, arsenopyrite and sphalerite

DICHROMATES - specially for galena

SULPHIDIZING AGENTS

These compounds are capable of changing the surfaces of oxidized minerals by coating with a sulphide film. They are used rarely and evidence is increasing which shows that they are of little use in flotation, even with oxidized ores.

The most common agent is sodium sulphide. It was found⁹ that sodium sulphide is detrimental in the flotation of precious minerals.

REGULATORS

The function of the regulator is to modify the alkalinity or acidity (pH) of a solution.

The action of the regulator may be said to be three-fold.

(1) It precipitates soluble salts from the solution

(11) Depresses certain sulphides by affecting the mineral surface

9. U.S. Bureau of Mines "Report of Investigations" Metallurgical Division
June, 1935.

CONCLUSION

The tremendous strides made in the last few years can be recognized when it is noted that, apart from the metallic sulphides that were originally floated, the more difficult ores such as; native metals, phosphates, some carbonates and oxides, can now be concentrated.

METHODS

The charges of ore were ground in a cylindrical rod mill with a watertight cover.

The mill was rotated for a specified time at 41 R.P.M. The pulp was then transferred to the flotation cell and diluted to the required density. The reagents were added in the desired quantities and the pulp agitated. The cell used was a Denver Sub-aeration with belt drive and a capacity of 1000 grs of ore. Skimming was done by hand with a metal scraper. The concentrates and tails were dried and assayed at the end of the run.

The reagents used, excepting soda ash, lime, cyanide, zinc sulphate, and frothers were made up in aqueous solutions so that 1 cc. equalled 1#/ton of ore when 1000 grs of ore were used.

The pH was determined by the Leeds & Northrup indicator.

The reagents used, weight of ore, pulp density, time of grinding and general conditions pertaining to the individual tests are given with the assays and recoveries of each. All results are transferred to the appendix in table form and are also shown on graphs, both individual and composite.

Recovery calculations are based on the equation -

$$\frac{\text{Wt of element in concentrate}}{\text{sum of weights of element in concentrates + tails}} \times 100 = \% \text{ Recovery}$$

TEST I.To Duplicate Test 6 of last year.

Charge to Ball Mill	Ore	1000 grs.
	Water	1000 grs.
	KCN	0.5 #/ton.
	Na ₂ CO ₃	2.5 #/ton.
	ZnSO ₄	0.5 #/ton.

Time of grinding - - 20 minutes.

Cell.

Concentrate I.

Added	Pine Oil #5	0.24 # / ton.
	Cresylic Acid	0.54 # / ton.
	Aerofloat #25	0.06 # / ton.

Conditioned 2 minutes

Skimmed 20 minutes

pH. - 6.85.

At first the bubbles were tough and heavily mineralized. After 5 minutes they were smaller and very copious. After 15 minutes the bubbles became clean. Then the froth turned darker and concentrate 1B started.

Concentrate I B.

Added	(K-Et. Xanthate	0.197 # / ton.
	(CuSO ₄ (5% sol.)	2.00 # / ton.

Conditioned 2 minutes

Skimmed 48 "

pH. - 7.0.

The bubbles were small and copious and the froth was good.

Concentrate. 2.

Added	{ K-Et. Xanthate	0.197 # / ton.
	{ CuSO ₄ (5% sol.)	2.00 # / ton.
Conditioned	-	2 minutes
Skimmed	-	12 "
pH	-	7.02

The froth was composed of small and copious bubbles carrying few sulphides.

Conclusion.

Note. This test was not assayed, as a new batch of ore arrived and it was thought that better results could be obtained with the fresh ore.

TEST 2.To Duplicate Test 13 of Previous Year.

Charge to Ball Mill	Ore	1000 grs.
	Water	1000 grs.
	KCN	0.10 # / ton.
	Lime	2.0 # / ton.

Time of grinding - 20 minutes.

Cell.

Concentrate 1.

Added	Cresylic Acid	0.108 # / ton.
	Aerofloat #15	0.438 # / ton.
Conditioned	-	2 minutes.
Skimmed	-	10 minutes
pH	-	7.2

The bubbles at first were large, tough and well-armored, but after three minutes became small and copious. The small bubbles were dirty, but soon cleared.

Concentrate 2.

Added	CuSO ₄	1.0 # / ton.
Conditioned	-	2 minutes
Skimmed	-	10 "
pH	-	7.2

The persistent froth was composed of small, lightly mineralized bubbles.

Concentrate 3A.

Added	Pine Oil	0.8 # / ton.
	#301	0.02 # / ton

Conditioned	2 minutes
Skimmed	15 "
pH	7.2

The froth was fairly persistent, the bubbles were small and more mineralized than those of previous test. Skimming was continued 15 minutes to duplicate last year's results, but as mineral was still coming up, skimming was continues into another pan.

Concentrate 3 B.

Skimmed	9 minutes
pH	7.4

This is a continuation of 3A. Skimming was continued until the bubbles were clean. The froth was the same as 3 A.

Concentrate 4.

Skimmed	13 minutes
pH	6.8 "

This concentrate was taken to duplicate last year's results. The froth was the same as in 3B, and little mineral was carried over.

Conclusion.

Note. The concentrate of this test were not assayed as a new batch of ore arrived and it was thought that better results could be obtained with the fresh ore.

TEST 3.

To duplicate Test 13 with the New Mixture of Ore.

Charge to Ball Mill

Ore	-	1000 grs.
Water	-	1000 "
KCN	-	0.1#/ton
Lime	-	2.0#/ton

Grinding Time -- 20 minutes.

Cell

Concentrate 1.

Added	Cresylic Acid	0.108 # / ton
	Aerofloat #15	0.438 # / ton
Conditioned	-	2 minutes
Skimmed	-	10 "
pH	-	7.0

Small and poorly mineralized bubbles. They were dirty for first 5 minutes, then clean for remainder of time.

Concentrate 2.

Added	Copper Sulphate	1.0 # / ton
Conditioned	-	2 minutes
Skimmed	-	10 "
pH	-	7.0

Froth was very similar to previous concentrate but less mineralized.

Concentrate 3.

Added	Pine Oil	0.03# / ton
	#301	0.02 # / ton

H ₂ SO ₄ (1:1)		5 cc.
Conditioned	--	2 minutes
Skimmed	--	15 "
pH	--	4.4

At first, a few large bubbles that were very heavily mineralized, came up. Almost entire surface of cell was covered with a silvery scum that was very persistent. After a six minute period the bubbles became smaller and more copious. Mineralization was heavy but less than at start. Skimmed 15 minutes to duplicate last year's test, but as mineral was still being brought up, continued skimming into another pan.

Concentrate 3 B.

Skimmed	-	12 minutes
pH	-	4.4

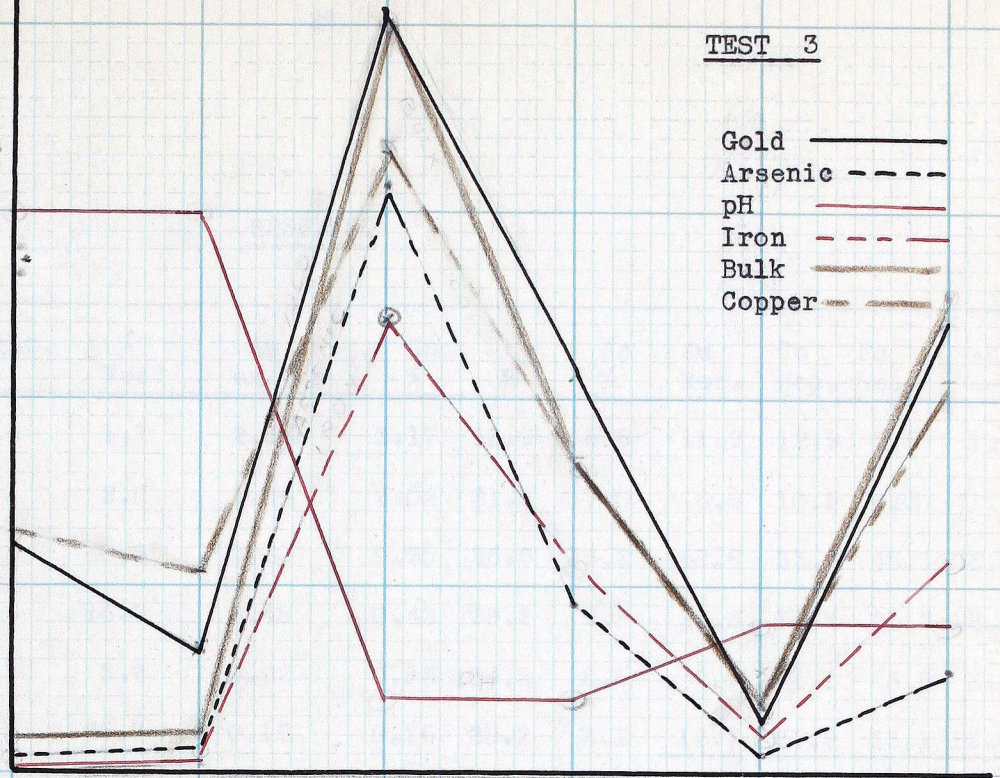
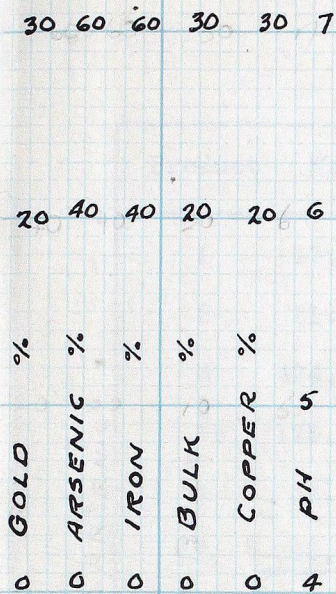
Continuation of #3. Froth was same as 3, except that the bubbles were less mineralized.

Concentrate 4.

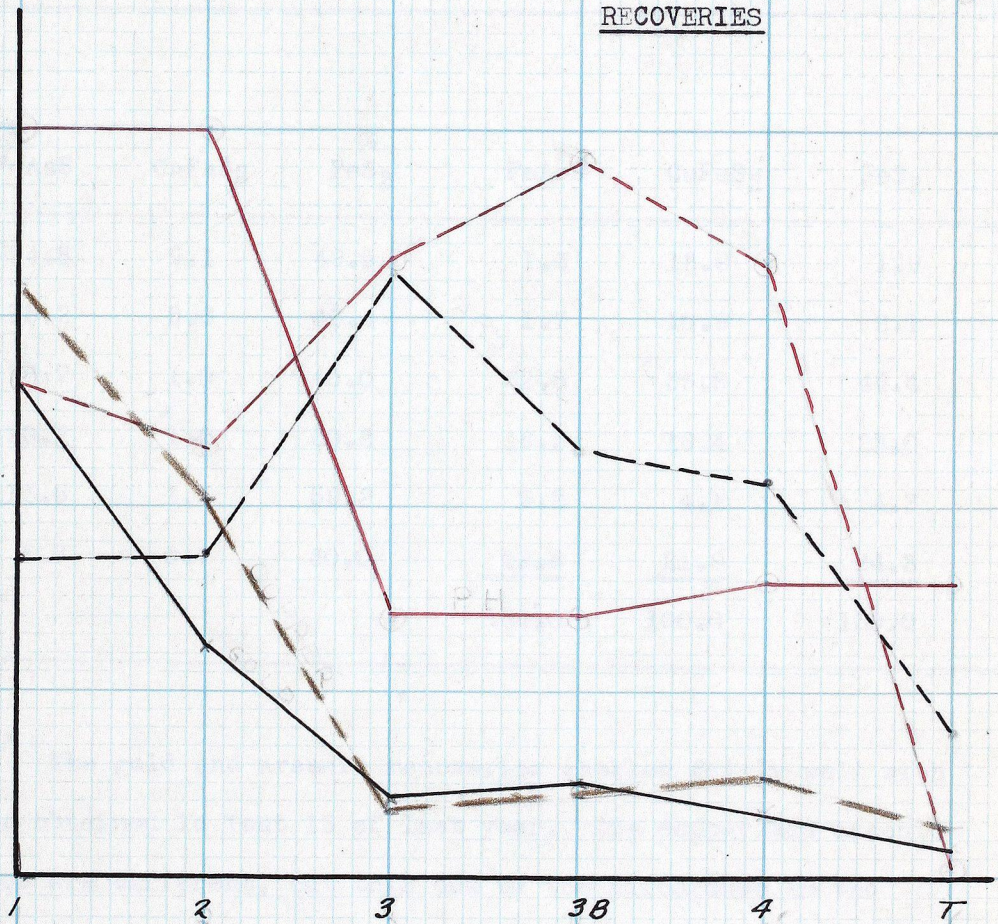
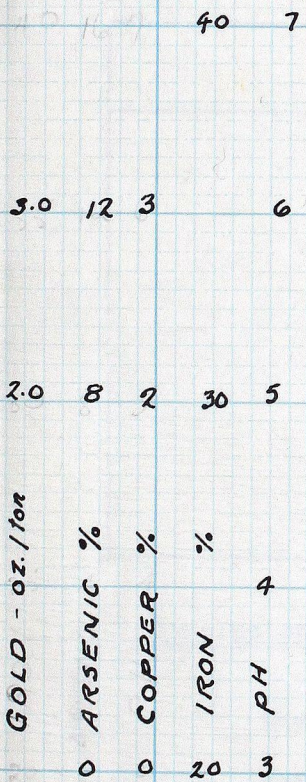
Skimmed	-	13 minutes
pH	-	4.6

A fourth concentrate was taken to duplicate last year's test. The froth was weak and bubbles were dirty and poorly mineralized.

Gold _____
 Arsenic - - - -
 pH _____
 Iron - - - -
 Bulk _____
 Copper - - - -



RECOVERIES



GRADES

RESULTS.

Product	Wght.	% Feed	Au oz/ton	Cu %	Fe %	As %	Au Rec.	Cu Rec.	Fe Rec.	As Rec.
L	17.6	1.7	2.68	3.17	33.9	6.8	11.9	12.9	1.9	1.3
2	21.1	2.0	1.24	2.02	31.5	7.0	6.6	10.9	2.0	1.7
3	424.3	40.6	0.40	0.35	36.6	13.2	42.6	33.8	48.1	62.8
3b	175.3	16.8	0.48	0.43	39.2	9.3	21.3	16.4	21.2	18.3
4	33.7	3.2	0.32	0.53	36.5	8.5	2.9	4.1	3.8	3.2
T	<u>368.5</u>	<u>35.7</u>	0.16	0.26	20.2	3.1	<u>14.7</u>	<u>21.9</u>	<u>23.1</u>	<u>12.8</u>
	1040.5	100.0					100.0	100.0	100.0	100.0

% FeAsS	% CuFeS ₂	% FeS ₂	FeAsS	CuFeS ₂	FeS ₂
14.8	9.1	49.5	1.3	12.9	1.9
15.2	5.8	46.4	1.7	10.9	2.1
28.7	1.0	48.0	62.8	33.8	44.4
20.2	1.2	60.2	18.3	16.4	23.0
13.5	1.5	59.2	3.2	4.1	4.3
6.7	0.7	30.0	<u>12.8</u>	<u>21.9</u>	<u>24.3</u>
			100.0	100.0	100.0

Conclusions.

The gold and arsenic recoveries checked fairly well with those obtained in Test 13 of last year. The copper and arsenic grades are different, but this due to the difference in the heads of the two batches.

TEST 4.

To duplicate Test 3 using less sulphuric Acid.

Ball Mill

Ore	-	1000 grs.
Water	-	1000 grs.
KCN	-	0.1 # / ton
Lime	-	2.0 # / ton

Grinding Time - 20 minutes.

Cell

Concentrate 1.

Added		Cresylic Acid	0.108 # / ton
		Aerofloat #15	0.438 # / ton
Conditioned	-	2 minutes	
Skimmed	-	10 "	
pH	-	7.2	

The bubbles were small and not well mineralized. They were dirty for the first five minutes and then fairly clean to end of skim. The persistence of froth was good.

Concentrate 2.

Added		CuSO ₄	1.0 # / ton
Conditioned	-	2 minutes	
Skimmed	-	10 "	
pH	-	7.2	

The froth was very similar to #1, but the bubbles were poorly mineralized.

Concentrate 3.

Added		Pine Oil	.08 # / ton
		#208	.02 # / ton

1:1 H₂SO₄ 3 c.c.

Conditioned - 2 minutes

Skimmed - 15 "

pH - 5.2

Obtained a good froth of tough and heavily armored bubbles, though not as heavily armored as in concentrate 3, test 3.

Concentrate 3b.

pH - 5.2 Skimmed - 12 minutes

The froth was same as concentrate 3, but bubbles became gradually cleaner.

Concentrate 4.

pH - 5.4 Skimmed - 13 minutes

Obtained poor froth of weak, dirty bubbles, carrying little mineral.

RESULTS.

Conc- entrate	Weight	Percent of feed	AU	CU	FE	AS	AU	CU	FE	AS
			Au Assay oz/ton	Cu %	Fe %	As %	Au Rec	Cu Rec	Fe Rec	As Rec
1	13.4	1.3	2.96	4.45	32.3	5.5	10.4	13.5	1.4	0.8
2	20.2	2.0	1.16	1.94	28.3	7.1	6.2	8.9	1.9	1.6
3	452.2	45.0	0.48	0.43	36.3	13.2	56.8	43.6	55.1	70.3
3b	146.9	14.6	0.36	0.31	39.1	8.0	13.8	10.3	19.4	13.7
4	19.8	1.9	0.32	0.67	30.0	6.4	1.7	2.9	1.9	1.5
T	354.0	35.2	0.12	0.26	17.1	2.9	11.1	20.8	20.3	12.2
	1006.5	100.0					100.0	100.0	100.0	100.0

0.4

TEST 4

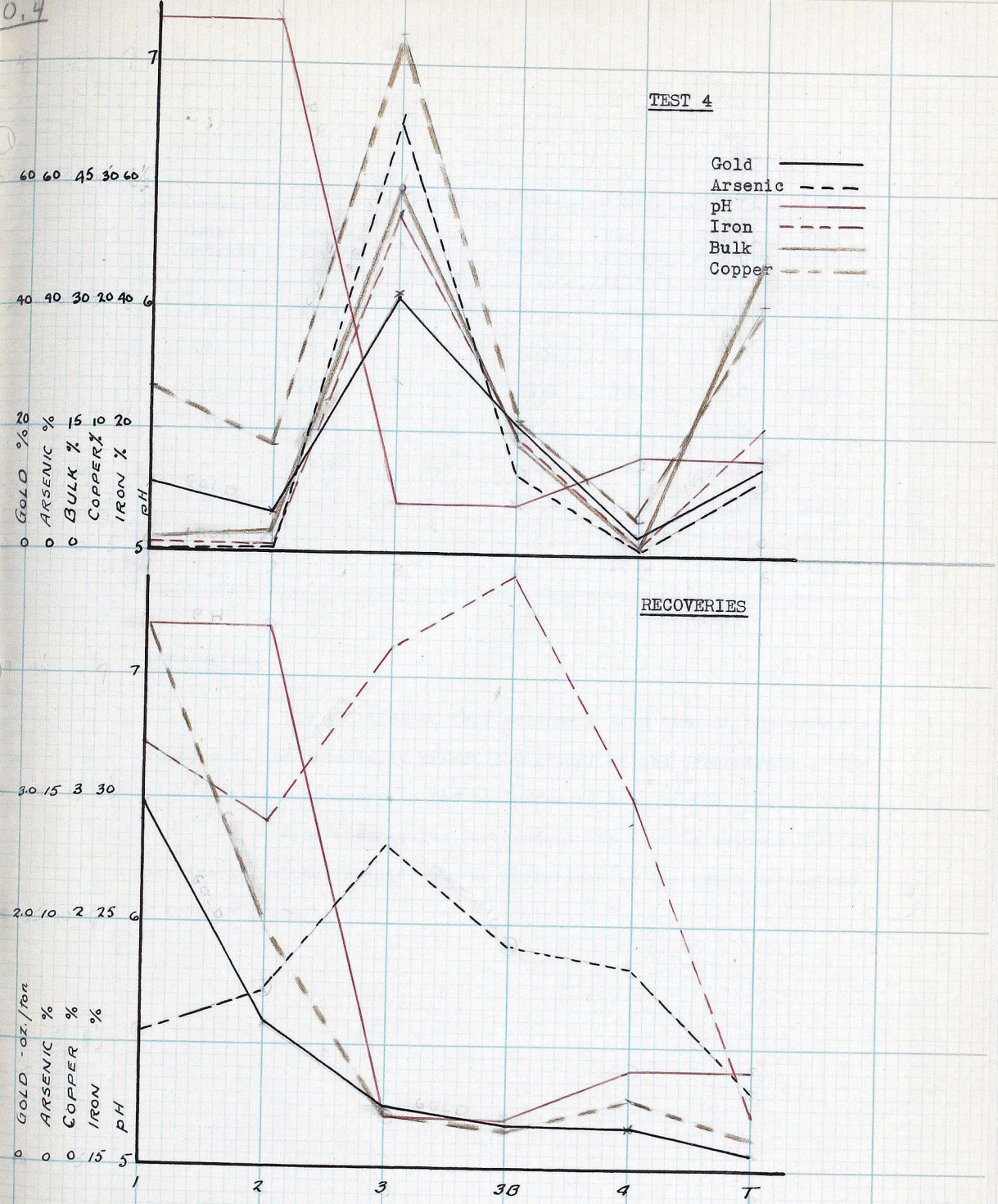
60 60 45 30 60
40 40 30 20 40 6
20 %
%
%
%
%
pH

Gold ———
Arsenic - - -
pH ———
Iron - - -
Bulk ———
Copper - - -

3.0 15 3 30
2.0 10 2 25 6
GOLD - oz./ton
ARSENIC %
COPPER %
IRON %
pH

RECOVERIES

GRADES



	FeAsS	CuFeS ₂	FeS ₂	FeAsS	CuFeS ₂	FeS ₂
Conc- entrate	Grade Arseno- pyrite	Grade Chalco- pyrite	Grade Pyrite	Rec. Arseno- pyrite	Rec. Chalco- pyrite	Rec. Pyrite
1	11.9	12.7	46.0	0.8	13.5	1.5
2	15.4	5.5	40.5	1.6	8.9	2.0
3	28.7	1.2	43.0	70.3	43.6	47.6
3b	17.4	0.9	62.0	13.7	10.3	22.3
4	13.9	1.9	46.6	1.5	2.9	2.3
T	6.3	0.7	27.9	12.2	20.8	24.3
				100.0	100.0	100.0

Conclusion.

The addition of less sulphuric Acid than in the previous test, does not materially affect the grades of the concentrates. The slightly higher recovery is due to increased bulk of the concentrates.

From the results for concentrate 3 it is evident that the decrease in sulphuric acid made no difference in the grade percentage of arsenic present.

TEST 5.

To Duplicate Tests 3 and 4, but using No Sulphuric Acid.

Charge to Ball Mill.

Ore	-	1000 gms.
Water	-	1000 "
KCN	-	0.1 # / ton
Lime	-	2.0 # / ton

Grinding time - 20 minutes

Cell

Concentrate 1.

Added	-	Cresylic acid	0.108 # / ton
		Aerofloat	0.438 # / ton
Conditioned	-	2 minutes	
Skimmed	-	10 "	
pH	-	8.65	

Obtained fair froth, which could have been stronger.

Bubbles were weak and dirty.

Concentrate 2.

Added	-	CuSO ₄	1.0 # / ton
Conditioned	-	2 minutes	
Skimmed	-	10 "	
pH	-	8.37	

Froth was weak and dirty. Bubbles were larger.

Concentrate 3.

Added	-	Pine Oil	.08 # / ton
		#301	.02 # / ton
Conditioned	-	2 minutes	
Skimmed	-	15 "	
pH	-	8.20	

Froth was composed of small, weak, dirty bubbles, which were not nearly as heavily armored as those in the tests where H_2SO_4 was added.

Concentrate 3 B.

Skimmed - 12 mins. pH - 7.98

Froth was same as 3, composed of small, weak, dirty bubbles, which were not heavily armored.

Concentrate 4.

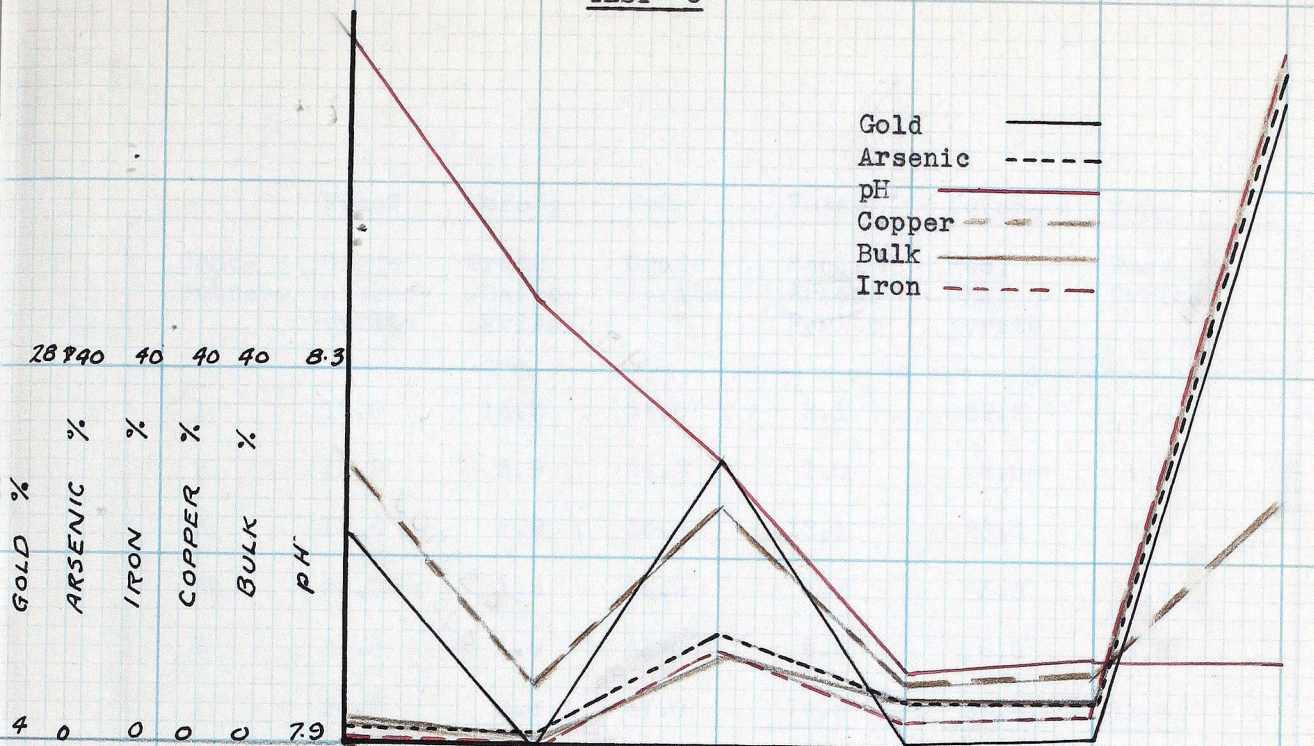
Skimmed - 13 mins. pH - 8.00

Froth was again weak and dirty, and was composed of small bubbles carrying little mineral.

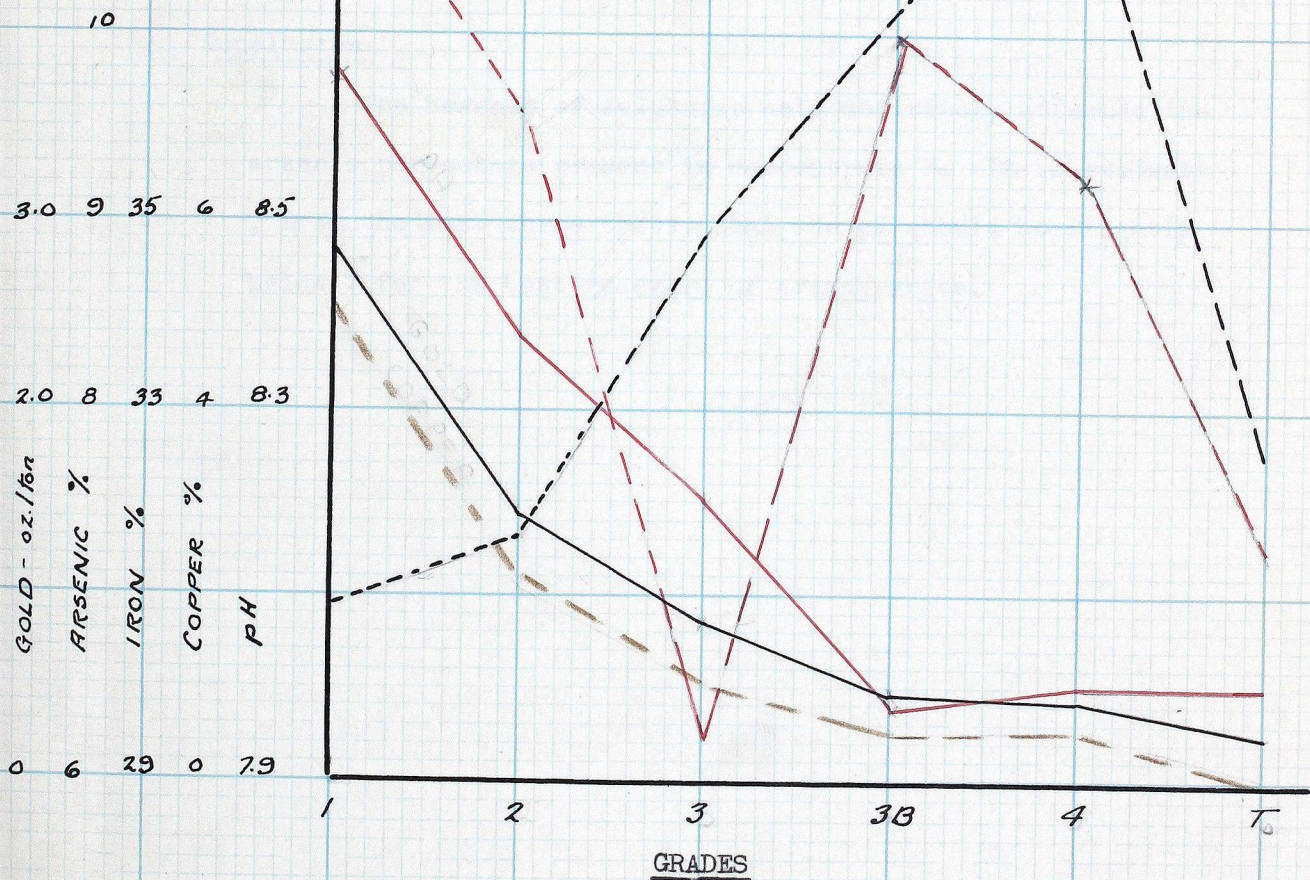
RESULTS.

Conc- entrate	Weight	Percent of Feed	Gold oz/ton	Cu %	Fe %	As %	Au Rec	Cu Rec	Fe Rec	As Rec
1	25.3	2.5	2.84	5.0	39.0	6.9	17.9	29.3	3.1	2.1
2	11.1	1.1	1.44	2.06	36.0	7.3	4.0	5.5	1.3	1.0
3	109.1	10.8	0.84	1.01	29.4	9.0	22.8	25.4	10.0	12.1
3b	37.5	3.7	0.44	0.54	37.0	10.2	4.1	5.6	4.4	4.7
4	41.7	4.1	0.42	0.66	35.5	10.8	4.4	6.8	4.6	5.6
T	783.3	77.8	0.24	0.15	31.3	7.7	46.8	22.4	76.6	74.5
	1008.0	100					100	100	100	100

TEST 5



RECOVERIES



	FeAsS	CuFeS ₂	FeS ₂	FeAsS	CuFeS ₂	FeS ₂
Conc- entrate	Grade Arseno- pyrite	Grade Chalco- pyrite	Grade Pyrite	Rec. Arseno- Pyrite	Rec. Chalco- pyrite	Rec. Pyrite
1	15.0	14.3	54.0	2.1	29.3	3.0
2	15.9	5.9	54.3	1.0	5.5	1.3
3	19.6	2.9	26.4	12.1	25.4	6.4
3b	22.2	1.5	46.9	4.7	5.6	3.9
4	23.5	1.9	41.0	5.6	6.8	3.8
T	16.7	0.4	47.0	<u>74.5</u>	<u>27.4</u>	<u>81.6</u>
				100.0	100.0	100.0

Conclusion.

The absence of sulphuric acid definitely affected the arsenic percentage present in concentrate 3. It is evident, from the last three tests, that the pH should be slightly below 7 for the best recovery of arsenopyrite.

Test 6.To Duplicate Test 3, But Using More Sulphuric Acid.

Charge to Ball Mill

Ore	-	1000 gms.
Water	-	1000 gms.
KCN	-	0.1 # / ton
Lime	-	2.0 # / ton

Grinding Time - 20 minutes.

Cell

Concentrate 1.

Added	-	Cresylic Acid	0.108 # / ton
		Aerofloat #15	0.438 # / ton
Conditioned	-	2 minutes	
pH	-	8.68	
Skimmed	-	10 minutes	

The froth was weak; bubbles were dirty and not strong. The froth was similar to test #5, C 1, except that bubbles were cleaner at the end.

Concentrate 2.

Added	-	CuSO ₄	1.0 # / ton
Conditioned	-	2 minutes	
pH	-	8.18	
Skimmed	-	10 minutes	

The froth was weak and dirty, similar to Test #5, C 2.

Concentrate 3.

Added	-	Pine Oil	0.08 # / ton
		#208	0.02 # / ton
		1:1 H ₂ SO ₄	10. c.c.
Conditioned	-	2 minutes	
Skimmed	-	15 "	pH - 3.67

At first, obtained froth of big, heavily armored bubbles, but after two minutes, the bubbles became smaller, even, well-armored and persistent. This was the most persistent froth so far obtained in the tests.

Concentrate 3 B.

Skimmed - 12 minutes

pH - 3.67

Got a tough, persistent froth, of small, heavily armored bubbles at first, but it gradually and rapidly weakened, and at the end was very weak.

Concentrate 4.

Skimmed - 13 minutes

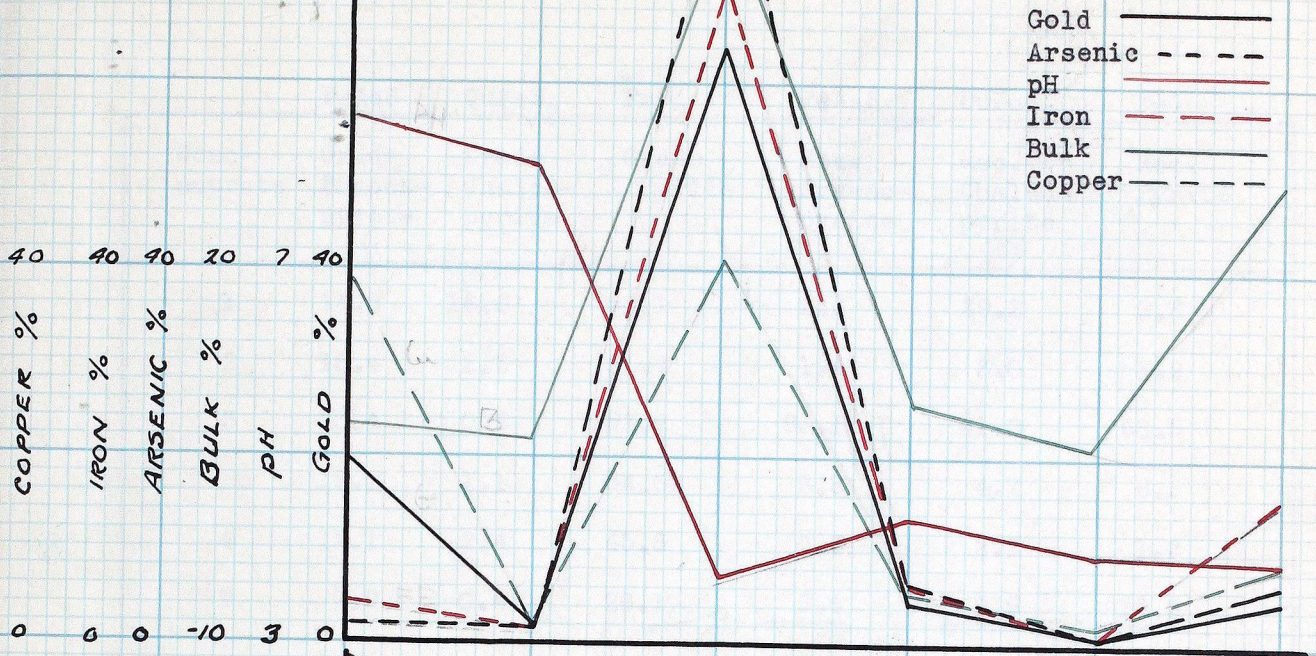
pH - 4

The froth was poor. The bubbles were small, weak and dirty, and carried little mineral.

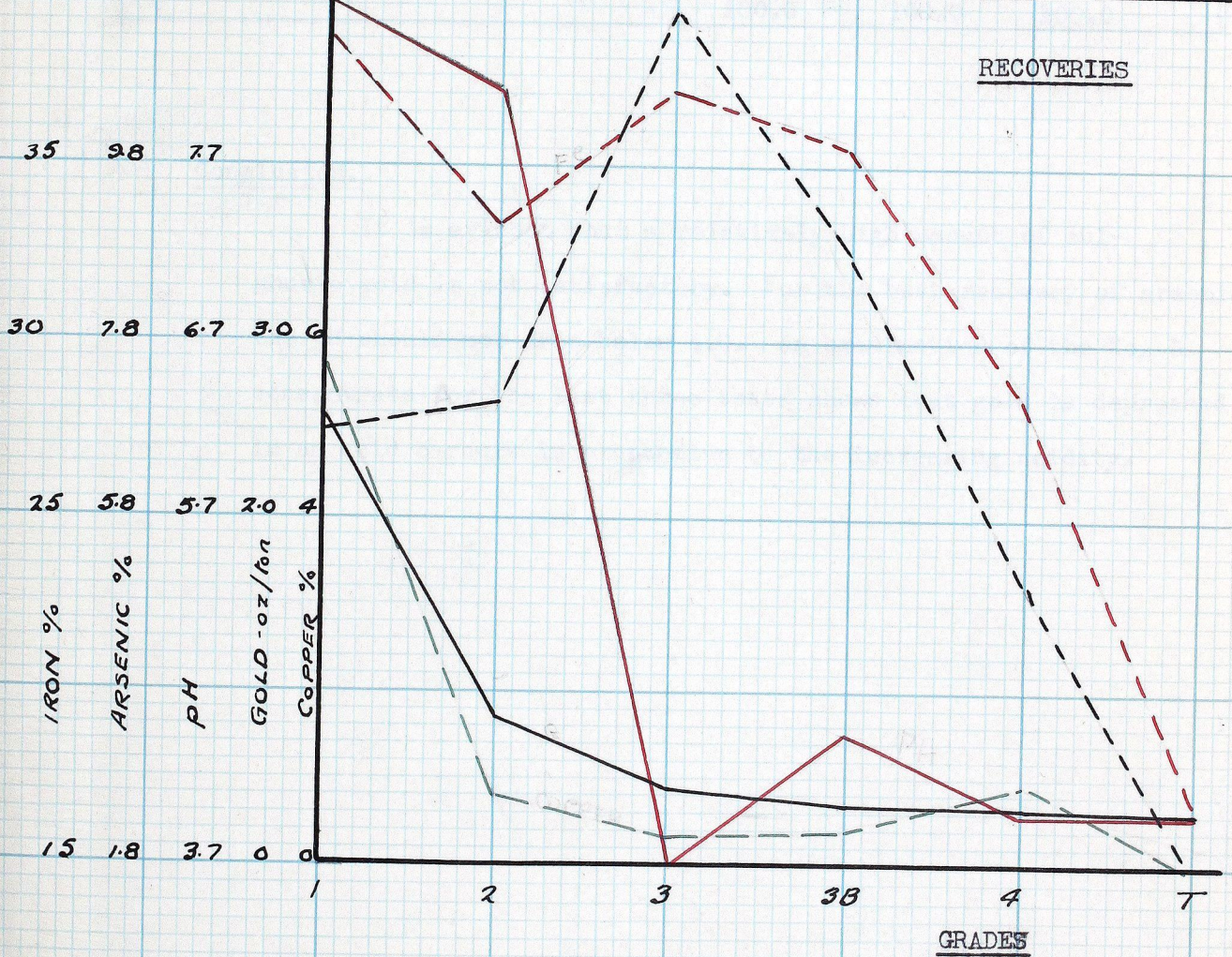
RESULTS.

Conc- entrate	Wght.	% of Feed	Au	Cu	Fe	As	Au	Cu	Fe	As
			Au oz/tón	Cu %	Fe %	As Rec	Au Rec	Cu Rec	Fe Rec	As Rec.
1	31.6	31.1	2.64	5.8	38.7	6.8	20.2	39.2	4.0	2.6
2	17.5	1.7	0.80	0.8	33.3	7.1	3.4	3.0	1.8	1.4
3	602.2	58.9	0.44	0.32	37.1	11.7	64.8	41.0	71.1	83.1
3b	55.2	5.4	0.32	0.40	35.8	8.8	4.4	4.7	6.3	5.8
4	11.9	1.1	0.32	0.83	28.8	5.1	1.0	2.2	1.1	0.7
T	<u>304.1</u>	<u>29.8</u>	0.08	0.15	16.3	1.8	6.0	<u>9.9</u>	<u>15.7</u>	<u>6.4</u>
	1022.5	100.0						100.0	100.0	100.0

TEST 6



RECOVERIES



	FeAsS	CuFeS ₂	FeS ₂	FeAsS	CuFeS ₂	FeS ₂
Conc- entrate	Grade Arseno- pyrite	Grade Chalco- pyrite	Grade Pyrite	Rec. Arseno- pyrite	Rec. Chalco- pyrite	Rec. Pyrite
1	14.8	16.6	54.1	2.6	39.2	3.6
2	15.4	2.3	51.7	1.4	3.0	1.9
3	25.4	0.9	53.2	83.1	41.0	68.5
3b	19.1	1.1	54.5	5.8	4.7	6.5
4	11.1	2.4	45.8	0.7	2.2	1.2
T	3.9	0.4	28.1	<u>6.4</u>	<u>9.9</u>	<u>18.3</u>
				100.0	100.0	100.0

Conclusion.

It is evident that a relatively high amount of sulphuric acid is not satisfactory. For the best recovery of arsenic the pH should not drop below 4.0. An examination of the No. 3 concentrate for the last three tests shows that gold is depressed in an acid circuit in proportion to the increasing acidity.

TEST 7.

To Test Collectors # 208 and # 301. and Frother Pine Oil.

Charge to Ball Mill

Ore	1000 gms.
Water	1000 gms.
Na ₂ CO ₃	4 # / ton
KCN	0.1 # / ton
#208	0.05 # / ton.

Grinding Time - 20 minutes.

Cell

Concentrate 1.

Added	-	Pine Oil	0.68 # / ton
Conditioned	-	2 minutes	
Skimmed	-	30 "	
pH	-	7.82	

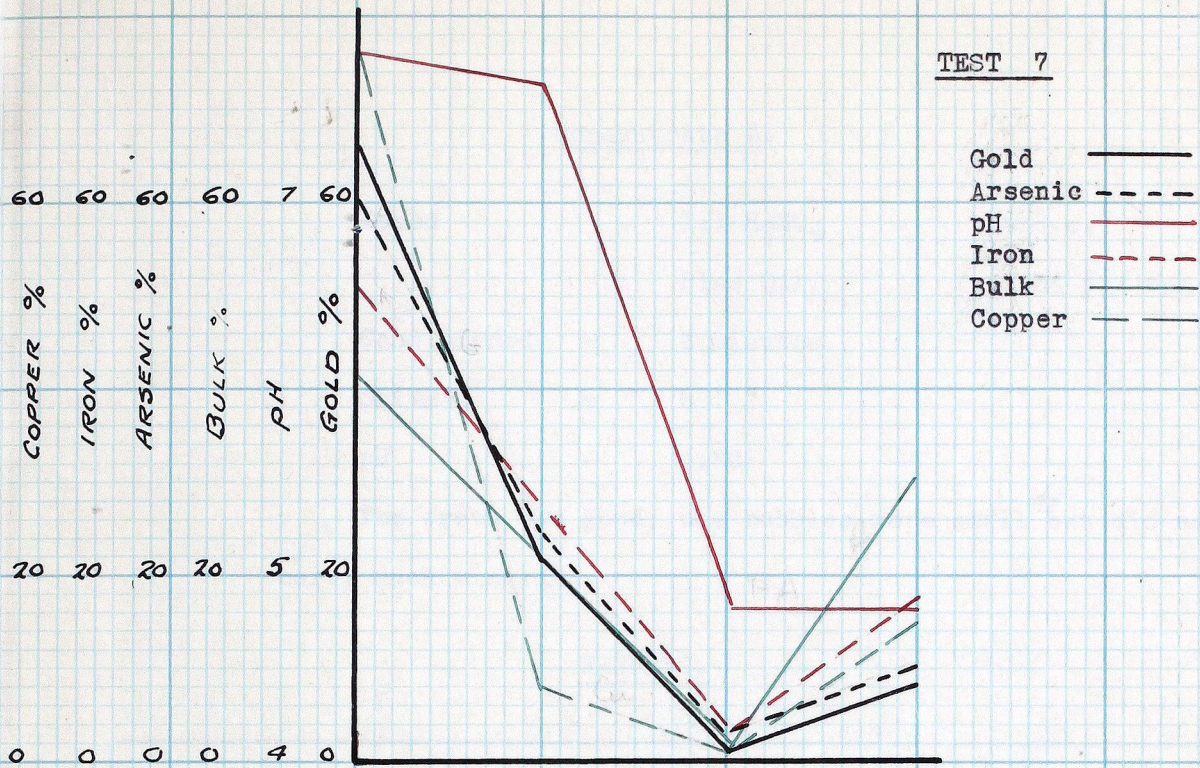
For the first minute, the bubbles were small, and the froth weak and dirty. Then got a froth of large and small bubbles, which were over-mineralized. After fifteen minutes, the froth was much better; it had just the right persistence, the bubbles were of medium size and not too heavily mineralized.

Concentrate 2.

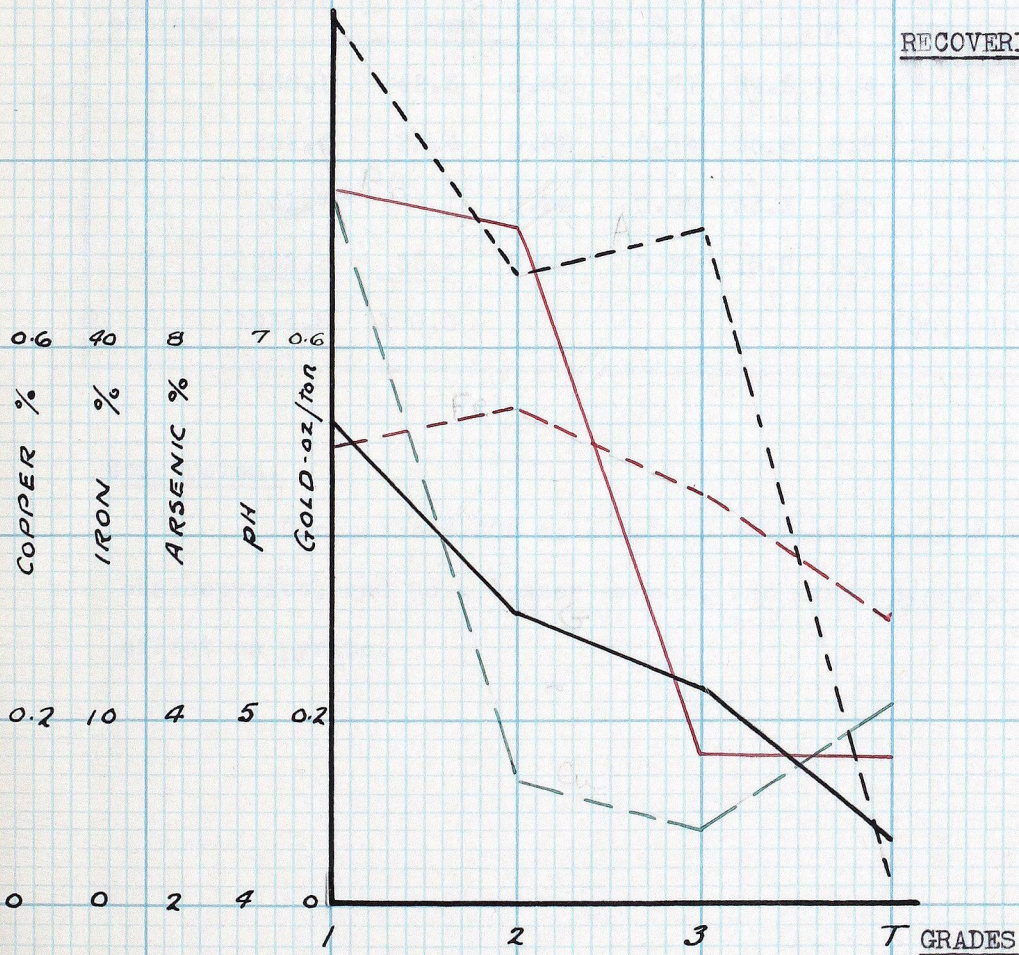
Added	-	#301	0.07 # / ton
Conditioned	-	2 minutes	
Skimmed	-	15 "	
pH	-	7.68	

The bubbles were not as large as in Concentrate #1, but were well mineralized. The froth had the right consistency.

TEST 7



RECOVERIES



Concentrate 3.

Added 1:1 H₂SO₄ 3 c.c.
 Pine Oil 0.34 # / ton
 Conditioned - 2 minutes
 Skimmed - 6 "
 pH - 4.82

The froth was good; the bubbles were small but did not carry much mineral.

RESULTS.

Conc- entrate	Wght.	% of Feed	Au oz/ton	Cu %	Fe %	As %	Au Rec	Cu Rec	Fe Rec	As Rec
1	438.1	42.3	0.52	0.77	34.9	11.4	67.8	76.0	50.9	60.8
2	232.4	22.4	0.32	0.14	36.6	8.8	22.0	7.6	28.3	24.9
3	31.7	3.1	0.24	0.09	33.3	9.3	2.3	0.7	3.5	3.5
T	<u>332.3</u>	<u>32.2</u>	0.08	0.21	15.8	2.7	<u>7.9</u>	<u>15.7</u>	<u>17.3</u>	<u>10.8</u>
	1034.5	100.0					100.0	100.0	100.0	100.0

Conclusion.

Cloocetor # 208 is a better collector than # 301 for gold, chalcopyrite, and to a lesser extent, for arsenopyrite. It has no effect on pyrite.

TEST 8.

To Duplicate Test 7, Using Cresylic Acid as a Frother.

<u>Charge to Ball Mill.</u>	Ore	-	1000 gms.
	Water	-	1000 gms.
	Na ₂ CO ₃	-	4 # / ton
	KCN	-	0.1 # / ton
	#208	-	0.05 # / ton
<u>Grinding Time</u>	-		20 minutes.

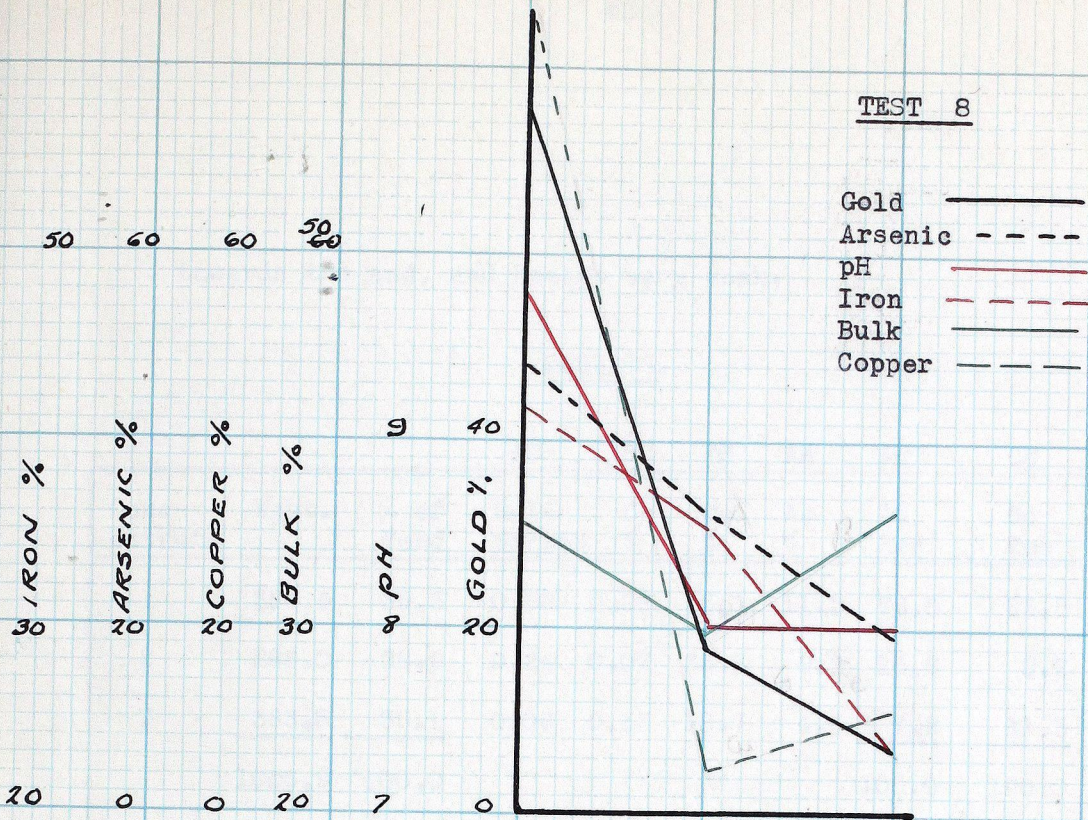
Cell	<u>Concentrate 1.</u>	
Added	-	Cresylic Acid 0.11 # / ton
Conditioned	-	2 minutes
Skimmed	-	30 "
pH	-	9.74

For the first minute, the bubbles were dirty; then for the next ten minutes, the froth was excellent. The bubbles were of medium size, and not too well mineralized, while the froth had just the right persistence. After ten minutes, the froth gradually weakened and the bubbles became less mineralized.

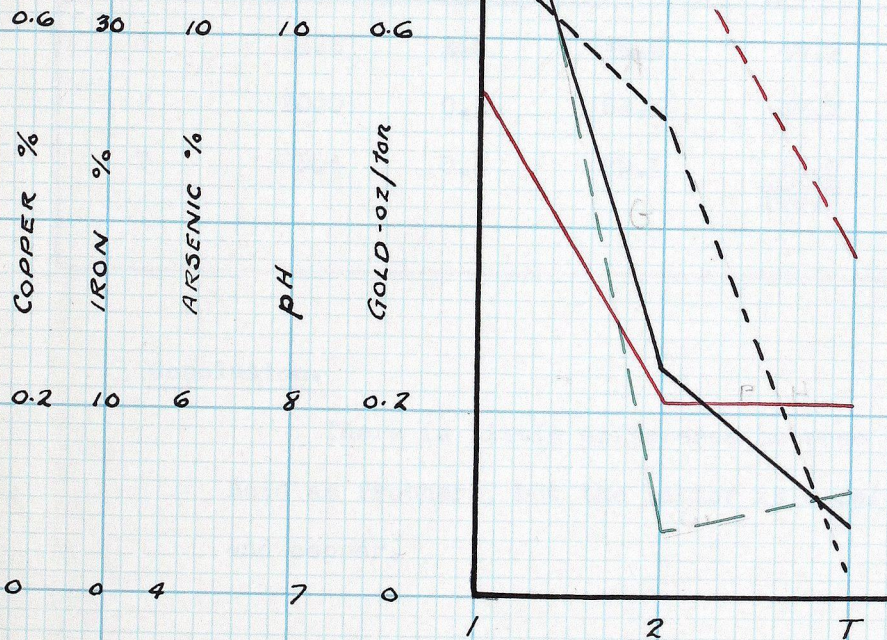
	<u>Concentrate 2.</u>	
Added	-	#301 0.07 # / ton
Conditioned	-	2 minutes
Skimmed	-	15 "
pH	-	8.03

For the first minute, a thick silvery scum was formed. Then got a froth composed of large, mineralized bubbles, which gradually became less mineralized. The bubbles at first, were large, but gradually became smaller, then increased in size

TEST 8



RECOVERIES



GRADES

towards the end, and became very weak.

RESULTS.

			AU	CU	FE	AS	AU	CU	FE	AS
Conc- entrate	Wght.	% of Feed	Gold Oz/ton	Cu %	Fe %	As %	Au Rec	Cu Rec	Fe Rec	As Rec
1	355.4	35.3	0.88	0.98	35.1	11.3	76.0	83.5	41.5	48.9
2	288.0	28.6	0.24	0.07	35.8	9.2	16.8	5.2	35.2	32.2
T	<u>363.5</u>	<u>36.1</u>	0.08	0.13	18.7	4.3	<u>7.2</u>	<u>11.3</u>	<u>23.3</u>	<u>18.9</u>
	1006.9	100.0					100.0	100.0	100.0	100.0

	FeAsS	CuFeS ₂	FeS ₂	FeAsS	CuFeS ₂	FeS ₂
Conc- entrate	Percent Arseno- pyrite	Percent Chalco- pyrite	Percent Pyrite	Recovery Arseno- pyrite	Recovery Chalco- pyrite	Recovery Pyrite
1	24.6	24.6	48.6	48.9	83.5	39.7
2	20.0	0.2	54.5	32.2	5.2	36.0
T	<u>29.54</u>	0.4	29.1	<u>18.9</u>	<u>11.3</u>	<u>24.3</u>
	100.0			100.0	100.0	100.0

Conclusion.

There is little difference between Pine Oil and Cresylic Acid as frothers, but the latter is a collecting power for gold and copper.

TEST 9.

To Duplicate Test 7, Using Aerofloat # 15 as a Frother.

Charge to Ball Mill

Ore	1000 gms.
Water	1000 gms.
Na ₂ CO ₃	4 # / ton
KCN	0.1 # / ton
# 208	0.05 # / ton

Grinding Time - 20 minutes.

Cell

Concentrate 1.

Added	-	Aerofloat # 15	0.088 # / ton
Conditioned	-	2 minutes	
Skimmed	-	30 minutes	
pH	-	8.83	

For the first minute the bubbles were small and weak; then a tough, over-persistent froth formed. The bubbles were over-mineralized, and gradually increased in size until, after fifteen minutes, they were the right size. At this point, both the froth and bubbles were good, and stayed that way until the end of skimming. However, the bubbles gradually became less mineralized.

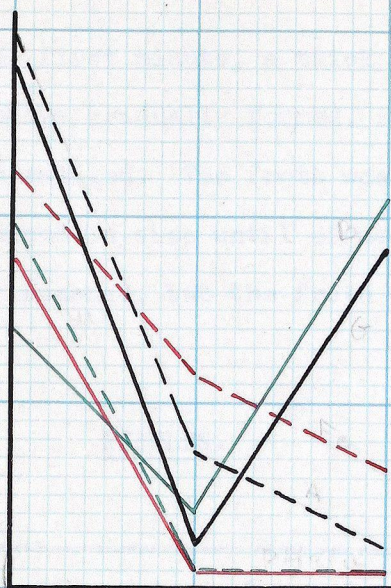
Concentrate 2.

Added	# 301	.07 # / ton
Conditioned	2 minutes	
Skimmed	15 minutes	
pH	8.05	

TEST 9

100 40 40 40 9 40
 COPPER %
 IRON %
 ARSENIC %
 BULK %
 PH
 GOLD %

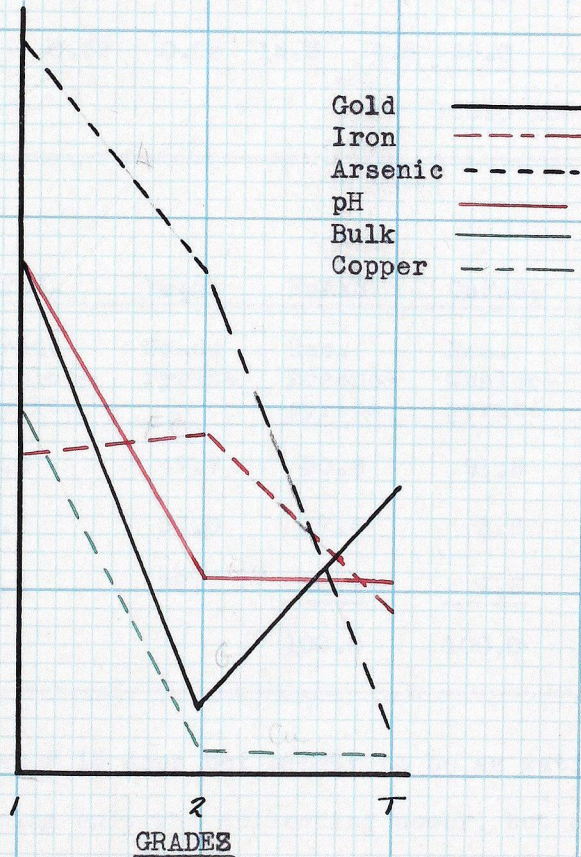
0 20 20 20 8 0



RECOVERIES

10 9 0.6
 1.0 40 8 8.5 0.4
 COPPER %
 IRON %
 ARSENIC %
 PH
 GOLD - oz/ton

0 0 4 7.5 0



In the first minute, a thick scum of mineral formed, the bubbles gradually formed, some of them large and heavily mineralized. The froth was over persistent. The bubbles reduced in size until of medium size, and mineralization lessened, but the froth was too tough the whole time.

RESULTS.

Conc- entrate	Wght.	% of Feed	Au oz/ton	Cu %	Fe %	As %	Au Rec	Cu Rec	Fe Rec	As Rec
1	358.4	34.8	0.56	1.05	35.0	11.9	56.5	91.8	42.3	50.2
2	248.1	24.1	0.08	0.07	37.4	9.5	5.5	3.6	31.4	27.6
T	<u>422.0</u>	<u>41.1</u>	0.32	0.05	18.6	4.5	<u>38.0</u>	<u>4.6</u>	<u>26.3</u>	<u>22.2</u>
	1028.5	100.0					100.0	100.0	100.0	100.0

	FeAsS	CuFeS ₂	Fe ₃ S ₂	FeAsS	CuFeS ₂	FeS ₂
Conc- entrate	Grade Arseno- pyrite	Grade Chalco- pyrite	Grade Pyrite	Rec. Arseno- pyrite	Rec. Chalco- pyrite	Rec. Pyrite
1	25.9	3.0	47.7	50.2	91.8	39.5
2	20.7	0.2	57.0	27.6	3.6	32.6
T	9.8	0.2	28.7	<u>22.2</u>	<u>4.6</u>	<u>27.9</u>
				100.0	100.0	100.0

Conclusion. Aerofloat # 15 does not appear to be as satisfactory a frother concerning only the collecting power, but as a frother, only, it was considered better than either Pine Oil or Cresylic Acid, due to toughness of bubbles and general behaviour.

TEST 10.To Add # 301 to the Ball Mill Instead of # 208

Charge to Ball Mill

Ore	1000 gms.
Water	1000 gms.
Na ₂ CO ₃	4 # / ton
KCN	0.1 # / ton
# 301	0.05 # / ton

Grinding Time - 20 minutes.

Cell

Concentrate 1.

Added	-	Aerofloat # 15	-	0.088 # / ton
Conditioned	-	2 minutes		
Skimmed	-	30 minutes		
pH	-	8.82		

For the first minute, the bubbles were small, weak and dirty. For the rest of the time the froth and bubbles were, on the whole, good, although the bubbles were slightly over-mineralized.

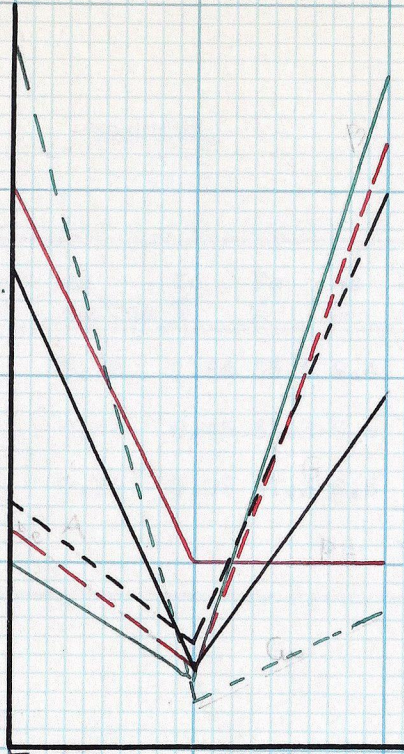
Concentrate 2.

Added	-	Aerofloat # 15	0.44 # / ton
		# 208 #	0.07 # / ton
Conditioned	-	2 minutes	
Skimmed	-	15 "	
pH	-	8.4	

The forth had the right persistence, the bubbles were of medium size and lightly mineralized.

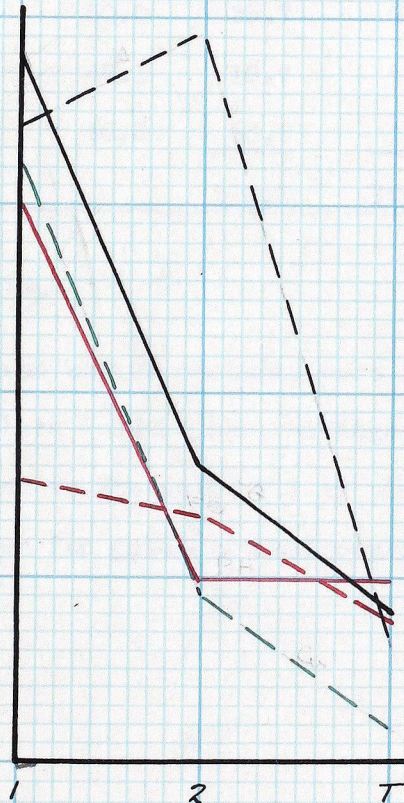
TEST 10

IRON %	ARSENIC %	COPPER %	BULK %	pH	GOLD %
60	60	60	60	8.8	60
20	20	20	20	8.4	20
0	0	0	0	8.2	0



RECOVERIES

COPPER	IRON	ARSENIC	pH	GOLD - oz./ton
1.5	50	9	8.8	0.6
0.5	30	7	8.4	0.2
0	20	6	8.2	0



Gold —————
 Arsenic - - - - -
 Iron - - - - -
 pH —————
 Bulk —————
 Copper - - - - -

GRADES

RESULTS.

Conc- entrate	Wght.	Percent of Feed	Au oz/ton	Cu %	Fe %	As %	Rec Au	Rec Cu	Rec Fe	Rec As
1	206.2	20.1	0.78	1.64	35.2	9.4	52.7	77.0	24.1	25.7
2	82.8	8.1	0.32	0.43	33.3	9.9	8.7	8.0	9.2	10.7
T	<u>737.0</u>	<u>71.8</u>	0.16	0.09	27.3	6.6	<u>38.6</u>	<u>15.0</u>	<u>66.7</u>	<u>63.6</u>
	1026.0	100.0					100.0	100.0	100.0	100.0

	FeAsS	CuFeS ₂	FeS ₂	FeAsS	CuFeS ₂	FeS ₂
Conc- entrate	Grade Arseno- pyrite	Grade Chalco- pyrite	Grade Pyrite	Rec. Arseno- pyrite	Rec. Chalco- pyrite	Rec. Pyrite
1	20.5	4.7	50.0	25.7	77.0	23.3
2	21.5	1.2	48.3	10.7	8.0	9.0
T	14.4	0.3	40.5	<u>63.6</u>	<u>15.0</u>	<u>67.7</u>
				100.0	100.0	100.0

Conclusion.

The gold and copper contents were increased in this test and there was less gold in the tailings. This indicated that # 301, when used in the Ball Mill in place of # 208, is a better collector for gold and chalcopryrite, but not for arsenopyrite.

TEST 11.To Attempt To Get a Bulk Concentrate.Charge to Ball Mill.

Ore	1000 gms.
Water	1000 Gms.
Na ₂ CO ₃	4 # / ton
K ₂ CO ₃	0.1 # / ton
# 208	0.05 # / ton
# 301	0.05 # / ton

Grinding Time - 20 minutes.

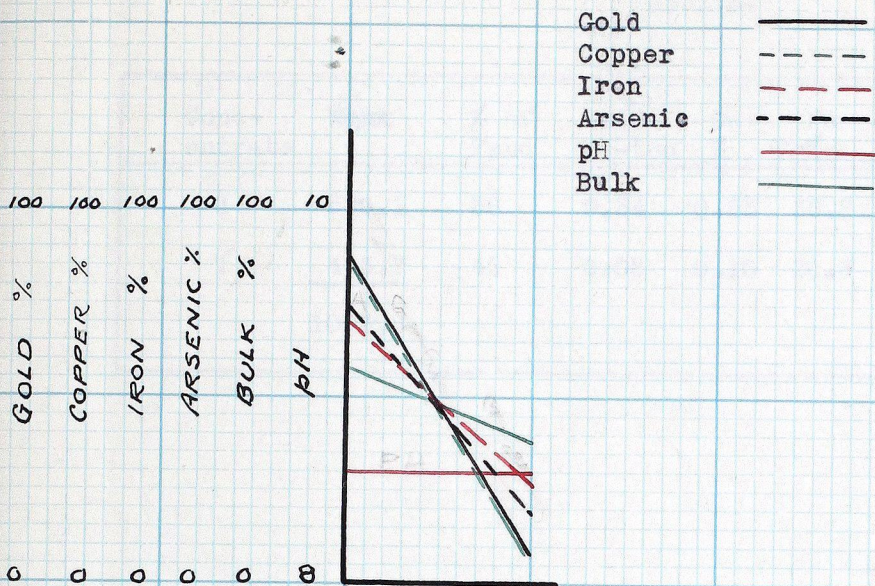
Cell

Concentrate 1.

Added	-	Aerofloat # 15	.088 # / ton
Conditioned	-	2 minutes	
Skimmed	-	30 "	
pH	-	8.63	

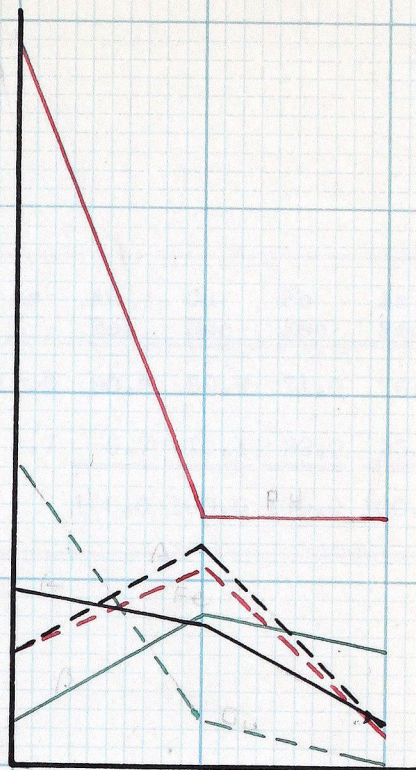
At first the bubbles and froth were good, then the bubbles became too large and overmineralized, and the froth too tough. The froth gradually weakened so after fifteen minutes, one drop of Aerofloat # 15 was added. The froth and bubbles were good until the end of skimming.

TESTS 11 & 12

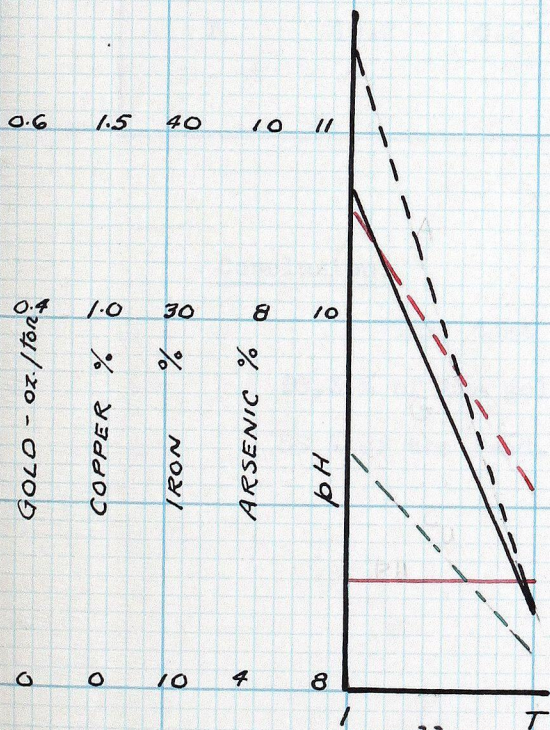


11

RECOVERIES

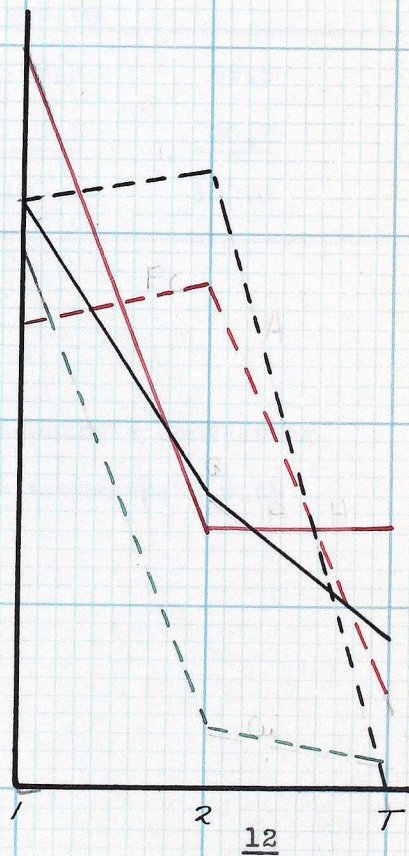


12



11

GRADES



12

RESULTS.

Conc- entrate	Wght	% of Feed	Gold oz-ton	Cu %	Fe %	As %	Au Rec	Cu Rec	Fe Rec	As Rec
1	580.1	58	0.54	0.65	36.8	11.0	90.3	89.9	71.0	76.3
T	<u>423.7</u>	42	0.08	0.10	20.3	4.7	<u>9.7</u>	<u>10.1</u>	<u>29.0</u>	<u>23.7</u>
	1003.8						100.0	100.0	100.0	100.0

Conc- entrate	Grade Arseno- pyrite	Grade Chalco- pyrite	Grade Pyrite	Rec. Arseno- pyrite	Rec. Chalco- pyrite	Rec. Pyrite
1	23.9	1.5	53.2	76.3	89.9	69.8
T	10.2	0.2	31.5	<u>23.7</u>	<u>10.1</u>	<u>30.2</u>
				100.0	100.0	100.0

Conclusion.

This test showed that a bulk concentrate containing 90.3 % of the gold can be obtained in a concentrate with 58 % of the feed.

TEST 12.To Test The Effect Of High Alkalinity.

Charge to Ball Mill.

Ore	1000 gms.
Water	1000 gms.
NaOH	6 # / ton
# 208	0.05 # / ton

Grinding Time - 20 minutes .

Cell

Concentrate 1.

Added	-	Aerofloat # 15	0.044 # / ton
Conditioned	-	2 minutes	
Skimmed	-	30 "	
pH	-	12.0	

At first a very tough dirty froth of tiny bubbles formed. The bubbles slowly became larger and more mineralized, until they were of medium size. The froth at first, was over persistent; later it improved, but was not satisfactory at any time.

Concentrate 2.

Added	-	#301 1:1 H ₂ SO ₄	0.07 # / ton -sufficient to bring down the pH.
Conditioned	-	2 minutes	
Skimmed	-	25 "	
pH	-	9.42	

A silver scum coated the surface of the pulp. Liver bubbles completely coated with mineral formed, and the froth was over persistent. Bubbles gradually decreased in size, and became less mineralized until after eight minutes the mineralization was satisfactory.

RESULTS.

Conc- entrate	Wght.	% of Feed	Gold oz/ton	Cu %	Fe %	As %	Au Rec	Cu Rec	Fe Rec	As Rec
1	254.4	24.8	0.64	1.45	35.3	10.4	45.6	82.3	29.8	30.8
2	447.4	43.7	0.32	0.14	37.1	10.7	40.0	14.1	55.0	55.5
T	<u>321.9</u>	<u>31.5</u>	0.16	0.05	14.2	3.7	<u>14.4</u>	<u>3.6</u>	<u>15.2</u>	<u>13.7</u>
	1023.7	100.0					100.0	100.0	100.0	100.0

Conc- entrate	FeAsS Grade Arseno- pyrite	CuFeS ₂ Grade Chalco- pyrite	FeS ₂ Grade Pyrite	FeAsS Rec. Arseno- pyrite	CuFeS ₂ Rec. Chalco- pyrite	FeS ₂ Rec. Pyrite
1	22.6	1.8	51.0	30.8	82.3	29.4
2	23.3	0.9	54.5	55.5	14.1	55.2
T	8.1	0.4	21.1	<u>13.7</u>	<u>3.6</u>	<u>15.4</u>
				100.0	100.0	100.0

Conclusion.

This indicates that a fairly high alkalinity does not affect the selective separation of pyrite and arsenopyrite.

TEST 13.To Clean A Bulk Concentrate.

Charge to Ball Mill.

Ore	1000 gms.
Water	1000 gms.
Na ₂ CO ₃	4# / ton
# 208	0.05 # / ton
# 301	0.05 # / ton

Grinding Time - 20 minutes.

Cell

Concentrate 1.

Added	-	Aerofloat # 15	0.088 # / ton (total)
Conditioned	-	2 minutes	
Skimmed	-	30 "	
pH	-	8.63	

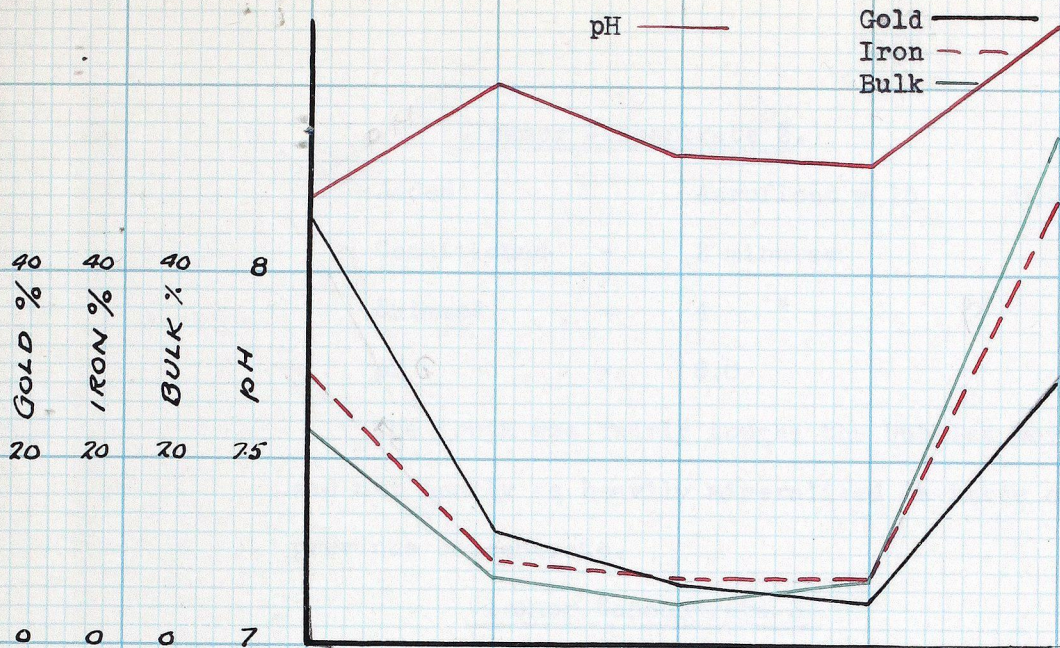
The froth was weak and dirty for the first minute; then large, over mineralized bubbles and an over persistent froth formed. The froth gradually weakened and the bubbles became less mineralized. After fifteen minutes, one drop of Aero-float # 15 was added, and a good froth of lightly armored bubbles of medium-size was obtained.

Cleaner Concentrate 1.

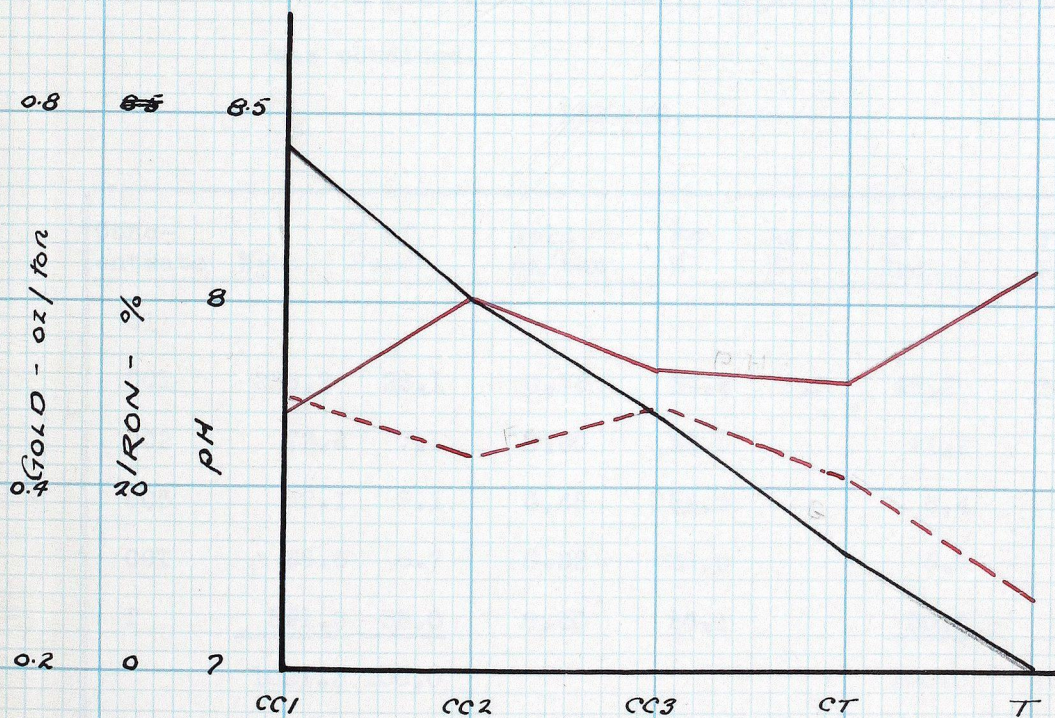
Added	
Pulp dilution	4:1 (roughly)
Skimmed	6 minutes
pH	7.7

Large, tough, heavily-mineralized bubbles formed, but they gradually decreased in size and load of mineral.

TEST 13



RECOVERIES



GRADES

Cleaner Concentrate 2.

Added		Aerofloat # 15	0.044 # / ton
Conditioned	-	2 minutes	
Skimmed	-	4 "	
pH	-	8.0	

The froth was fairly tough; the bubbles were small, and not nearly as heavily mineralized as those of the previous concentrate.

Cleaner Concentrate 3.

Added		Aerofloat # 15	0.044 # / ton
Conditioned	-	2 minutes	
Skimmed	-	6 minutes	
pH	-	7.80	

A good froth of small, lightly mineralized bubbles was obtained.

RESULTS.

Conc- entrate	Wght.	% of Feed	Gold oz/ton	Fe %	As %	Au Rec.	Fe Rec.
C1							
CC1	236.2	23.1	0.76	34.8	8.0	46.2	28.6
CC2	72.8	7.1	0.60	31.5		11.2	8.0
CC3	52.1	5.1	0.48	33.9		6.4	6.2
CCT	68.6	6.7	0.32	30.8		5.7	7.4
T	<u>591.6</u>	<u>58.0</u>	0.20	24.1		<u>30.5</u>	<u>49.8</u>
	1021.3	100.0				100.0	100.0

Conclusion.

Results obtained in this attempt to separate the pyrite and arsenopyrite by cleaning a bulk concentrate, were not satisfactory. These results bear out the final conclusion reached that pyrite and arsenopyrite react similarly to all reagents and cannot be separated satisfactorily.

TEST 14.To Get A Bulk Concentrate Using Z-6 (Pentasol Xanthate)
as a Collector.

Charge to Ball Mill

Ore	1000 gms.
Water	1000 gms.
Na ₂ CO ₃	4 # / ton

Cell

Concentrate 1.

Added	Z-6	0.12 # / ton
	Aerofloat # 15	0.098 # / ton (total)
Conditioned	- 2 minutes	
Skimmed	- 28 "	
pH	- 8.98	

A good froth of well mineralized bubbles was obtained.

RESULTS.

Conc- entrate	Wght.	% of Feed	Gold oz/ton	Fe %	As %	Rec Au	Rec Fe
1	456.5	45.2	0.62	34.3	8.9	76.4	55.8
T	<u>554.3</u>	<u>54.8</u>	0.16	22.4		<u>23.6</u>	<u>44.2</u>
	1010.8	100.0				100.0	100.0

Conclusion.

This collector was not satisfactory as it produced a concentrate carrying only 76.4 % of the gold in 45.2% of the bulk.

TEST 15.To Get A Blanket Concentrate.

Charge to Ball Mill.

Ore	1000 gms.
Water	1000 gms.

Time of Grinding - 20 minutes.

The charge from the ball mill was washed over a
corduroy blanket.

Conc.	Wht.	% Feed	Au. oz/ ton	Au. Recov.
1	179.8	18	0.44	18.0

Conclusion.

This test showed that blanket concentration of this ore is not satisfactory. From the results obtained in this test and the following one, it is evident that there is no concentration, the recovery depending entirely on the bulk remaining on the blanket.

TEST 16.To Get a Blanket Concentrate and Then Float the Blanket Tailings.

Charge to Ball Mill

Ore 1000 gms.
 Water 1000 gms.
 Na_2CO_3 4 # / ton
 KCN 0.10 # / ton

Time of grinding - 20 minutes.Concentrate 1.

The charge from the ball mill was washed over a corduroy blanket. The tailings were filtered, put into the cell and diluted 4:1.

Cell

Concentrate 2.

Added	-	# 301	0.10 # / ton
		Aerofloat # 15	0.088 # / ton
		1:1 H_2SO_4	0.7 c.v. (to lower the pH)
Conditioned	-	2 minutes	
Skimmed	-	15 "	
pH	-	6.9	

One drop of Aerofloat # 15 produced a weak froth, so one more drop was added. This produced large, heavily mineralized bubbles at first, but they lessened in size and mineral burden until after three minutes, the froth was good.

Concentrate 3.

Added	-	CuSO_4	1 # / ton
		# 301	0.05 # / ton
		Aerofloat #15	0.044 # / ton
Conditioned	-	2 minutes.	
Skimmed	-	12 "	
pH	-	6.78	

TESTS 14 & 16

60 40 30 40 8
 GOLD %
 IRON %
 ARSENIC %
 BULK %
 pH

20 0 10 0 6

14

RECOVERIES

16

60
 GOLD - oz./ton
 IRON %
 ARSENIC %
 pH

40 11 9
 0 20 9 7

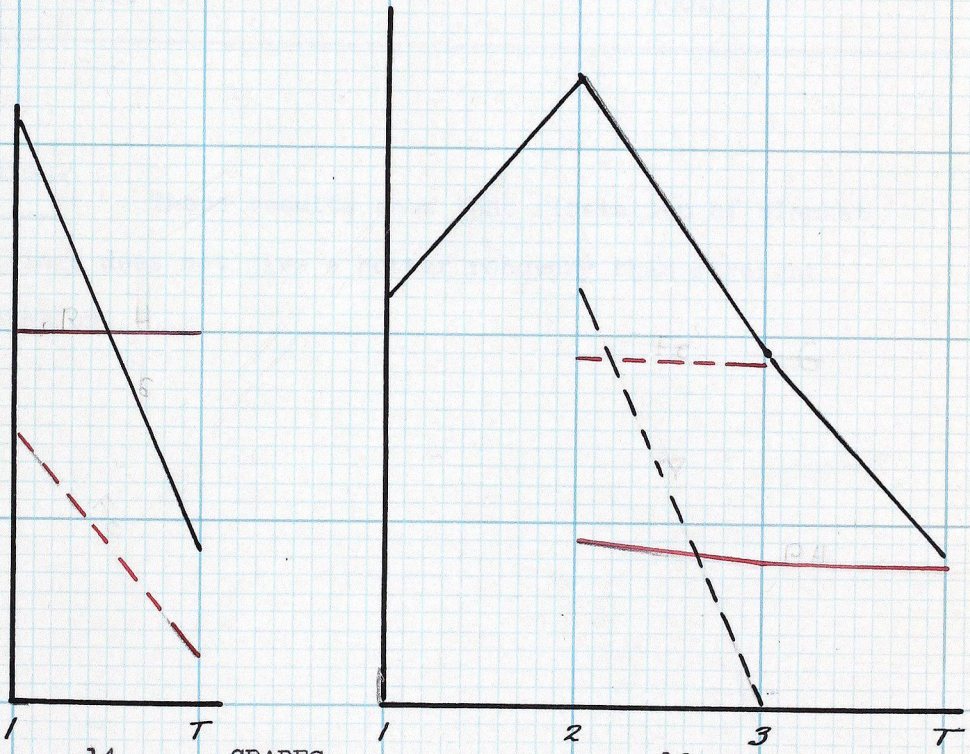
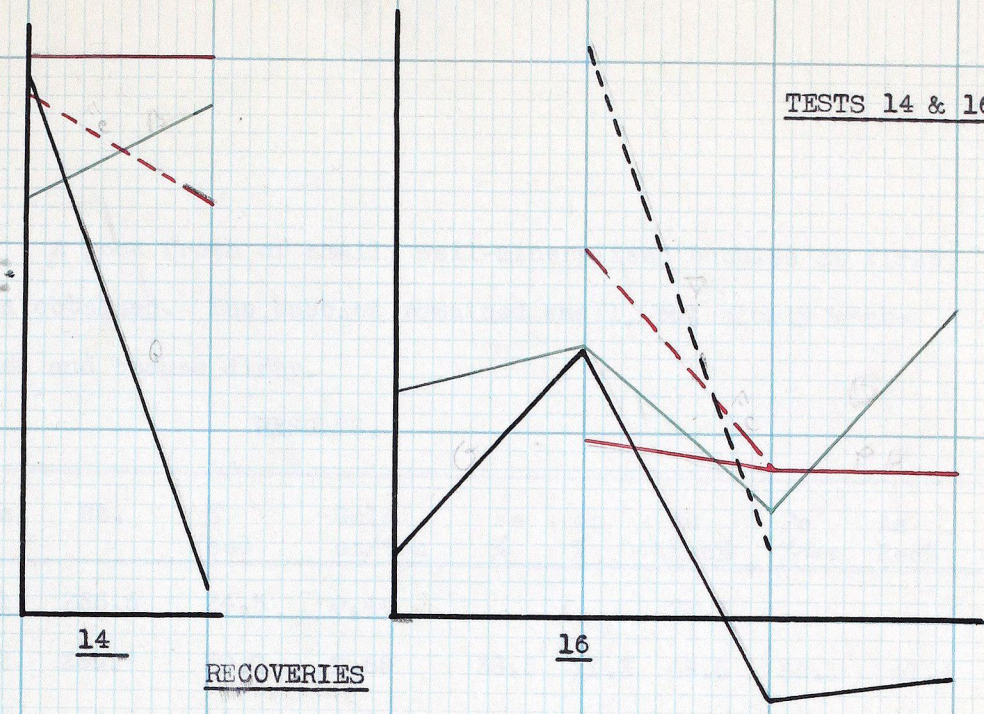
14

GRADES

16

Gold —————
 Iron - - - - -
 Bulk —————

Arsenic - - - - -
 pH —————



A good froth of small, well-mineralized bubbles at first, was obtained. The bubbles remained small, but slowly weakened and became dirty.

RESULTS.

Conc.	Wt.	% Feed	Gold oz/ton	Fe %	As %	Au Rec%	Fe %Rec.	As Rec%
C-1	236.5	24.5	0.44			27.3		
C-2	285.7	30.0	0.68	38.8	11.3	48.5	39.1	41.4
C-3	114.9	12.1	0.38	38.7	9.0	10.7	15.7	13.2
T	<u>317.3</u>	33.4	0.16			<u>13.5</u>		
	954.4					100.0		

Conclusion.

These results show that floatation of blanket tailings does not give a better recovery than straight flotation.

TEST 17.

To Use Sodium Sulphite (Na_2SO_3) Instead of KCN in the Ball Mill.

Charge to Ball Mill

Ore 1000 grs.
 Water 1000 grs.
 Na_2CO_3 4 # / ton
 # 208 0.05 # / ton
 # 301 0.05 # / ton
 Na_2SO_3 0.10 # / ton

Time of Grinding - 20 minutes.

Cell

Concentrate 1.

Added - Aerofloat # 15 0.32 # / ton
 Conditioned - 2 minutes
 Skimmed - 30 "
 pH - 7.98

The froth was good, as were the bubbles, Aerofloat #15 being added whenever the froth got weak.

RESULTS.

Conc.	Wt.	% Feed	Au oz/ton	Fe %	As %	Au Rec%	Fe Rec.	As Rec.
1	510.7	51.0	0.68	39.2	10.8	81.1	66.2	70.8
T	<u>507.0</u>	<u>49.0</u>	0.16			<u>18.9</u>		
	1007.7	100.0				100.0		
6								

Conclusion.

There is no appreciable difference between these results and those obtainable for test #11, showing that sodium sulphite has little effect.

TEST 18.

To Duplicate Test # 17, But Using More Sodium Sulphite.

Charge to Ball Mill.

Ore	1000 gms.
Water	1000 gms.
Na ₂ CO ₃	4 # / ton
Na ₂ SO ₃	1.0 # / ton
# 208	0.05 # / ton
# 301	0.05 # / ton

Cell

Concentrate 1.

Added	-	Aerofloat # 15	0.132 # / ton (total)
Conditioned	-	2 minutes	
Skimmed	-	30 "	
pH	-	8.42	

The bubbles were small and dirty at first, then became well mineralized. They increased in size until quite large. Whenever the froth became weak, Aerofloat # 15 was added.

RESULTS.

Conc.	Wght.	% Feed	Au oz/ton	Fe %	As %	Au Rec	Fe Rec	As Rec
1	589.4	58.0	0.64	38.3	10.6	86.5	75.1	79.8
	<u>425.7</u>		0.14			<u>13.5</u>		
	1015.1					100.0		

Conclusion.

Other than an increased bulk, the addition of more solution of sodium sulphite had no effect.

TEST 19.To Separate Pyrite and Arsenopyrite at a Low pH.

Charge to Ball Mill.

Ore	1000 gms.
Water	1000 gms.
Na ₂ CO ₃	4 # / ton
Na ₂ SO ₃	1 # / ton

Time of Grinding - 20 minutes,

Cell

Concentrate 1.

Added	-	Aerofloat # 15	0.088 # / ton
		#301	0.10 #n / ton
		H ₂ SO ₄ (1:1)	3.2 c.c.
Conditioned	-	2 minutes	
Skimmed	-	7 "	
pH	-	5.6	

A good, peristent froth of large, well-mineralized bubbles was formed. The bubbles became smaller, at end of six minutes, more Aerofloat # 15 was added. In an attempt to get a high grade concentrate the time of skimming was cut down.

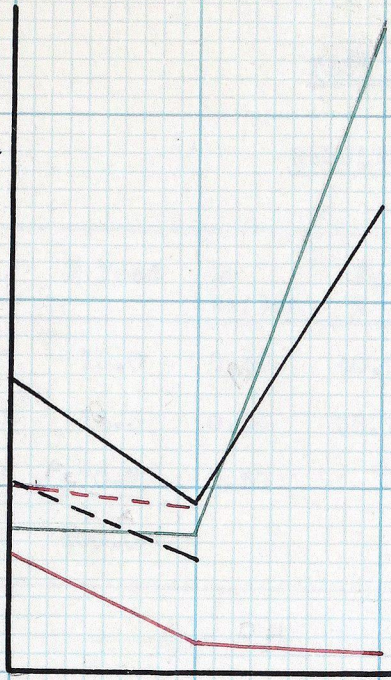
Concentrate 2.

Added	-	Aerofloat # 15	0.044 # / ton
		# 208	0.05 # / ton
		CuSO ₄	1.0 # / ton
Conditioned	-	2 minutes	
Skimmed	-	5 minutes	
pH	-	5.12	

The froth was tough and composed of small, fairly well mineralized bubbles, which soon became clean.

TESTS 19 & 20

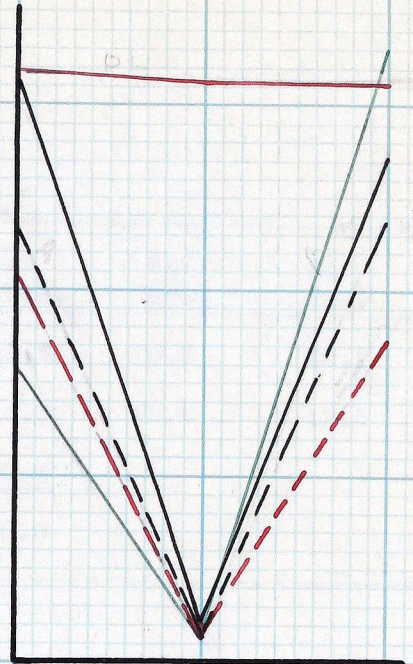
40 40 40 40 7
 GOLD %
 IRON %
 ARSENIC %
 BULK %
 PH
 0 0 0 0 5



19

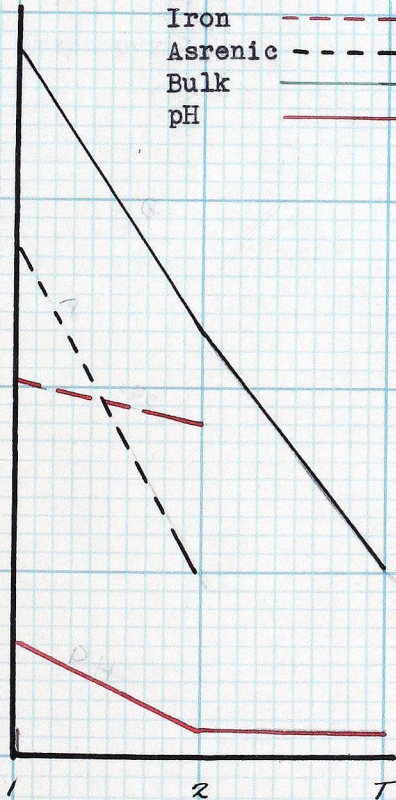
RECOVERIES

20



Gold ———
 Iron - - - -
 Arsenic - - -
 Bulk ———
 pH ———

0.7 50 11
 GOLD - oz/ton
 IRON %
 ARSENIC %
 PH
 0.3 30 7 6
 0.1 20 5 5



GRADES

TEST 19.RESULTS.

Conc.	Wght.	% Feed	Au oz/ton	Fe %	As %	Au Rec.	Fe Rec.	As Rec.
1	150.7	15.0	0.86	40.2	10.5	30.6	20.2	20.2
2	144.3	14.3	0.56	38.1	6.8	19.0	18.4	12.5
T	<u>711.9</u>	<u>70.7</u>	0.30			<u>50.4</u>		
	1006.9	100.0				100.0		

Conclusion.

Test unsatisfactory as there is no separation of pyrite and arsenopyrite.

TEST 20.

To try selective flotation by Taking a Pyrite concentrate and then an arsenopyrite concentrate.

Procedure - To float the pyrite with a small amount of Potassium Ethyl Xanthate followed by a more powerful collector for the arsenopyrite.

Charge to Ball Mill	Ore	1000 gms.
	Water	1000 gms.
	Na ₂ CO ₃	4 # / ton

Time of Grinding - 20 minutes.

Concentrate 1.

Added	-	Pine Oil #5 -	0.069 # / ton
		Pot. EthylXanthate	0.01 # / ton
Conditioned	-	2 minutes	
Skimmed	-	35 "	
pH	-	8.17	

Got a good froth of well-mineralized bubbles.

Concentrate 2.

Added	-	Pine Oil # 5 -	0.069 # / ton
		Z-6	0.40 # / ton
		CuSO ₄	1.5 # / ton
Conditioned	-	2 minutes.	
Skimmed	-	10 "	
pH	-	8.15	

A persistent, dirty froth with small, fairly mineralized bubbles was obtained.

RESULTS.

Conc.	Wght.	% Feed	Au oz/ton	Fe %	As %	Au Rec	Fe Rec	As Rec
1	320.0	32.0	0.62	40.4	10.5	63.2	40.1	47.3
2	20.9	2.1	0.44	36.1	9.7	3.0	2.4	3.6
T	660.4	65.9	0.16	28.0	5.3	<u>33.8</u>	<u>57.5</u>	<u>49.1</u>
						100.0	100.0	100.0

Conclusion.

The Test was a failure as there was no separation of the pyrite and arsenopyrite.

TEST 21.To Get A Bulk Concentrate And Clean It in a High Alkaline Circuit.

Charge to Ball Mill

Ore	1000 gms.
Cyanide	0.1 # / ton
Na ₂ CO ₃	4.0 # / ton

Time of Grinding - 20 minutes.Bulk Concentrate

Added	-	Aerofloat # 15	-	0.088 # / ton
		# 208	-	0.05 # / Ton
		# 301	-	0.05 # / ton

Conditioned - 2 minutes

Skimmed - 19 #

pH - 8.63

At first the bubbles were too large, but soon improved, producing a good froth of over-mineralized bubbles.

Cleaning of Bulk Concentrate.

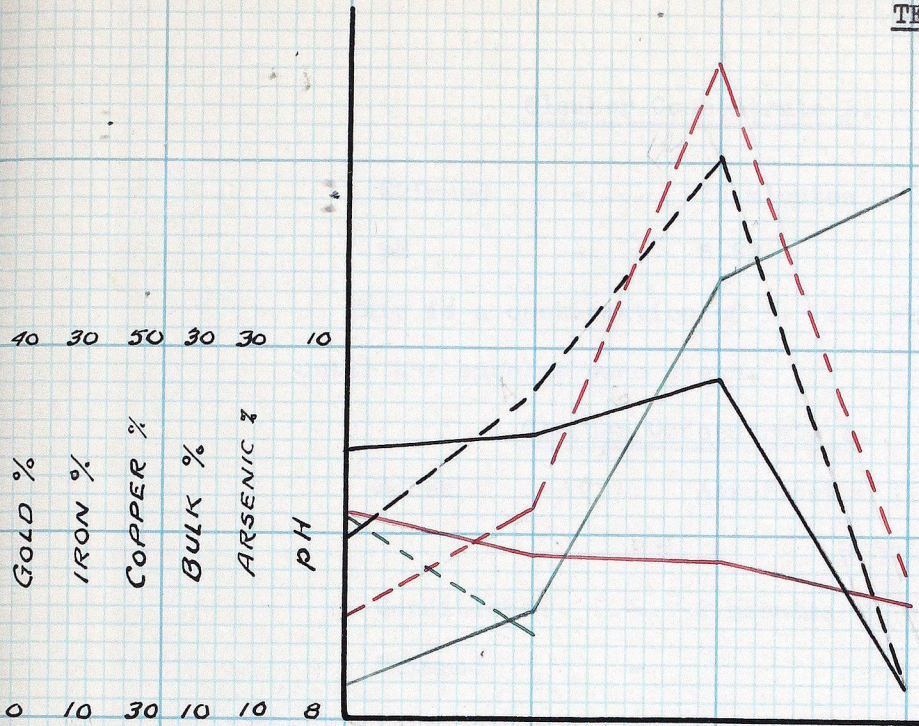
The concentrate of 582.0 gms. was transferred to the 500 gm. cell and diluted to 4:1 pulp density. Sodium hydroxide was added to raise the pH to approximately 11.0. It was found that the first addition brought the pH to 12.5+. As this was too high, a few drops of # sulphuric acid were added. This was agitated for one minute and the pH was found to be 11.5. Evidently the agitation was insufficient as at the conclusion of the run the pH was 9.10.

Concentrate 1 - Cleaner.Added - NaOH & H₂SO₄ (1:1) pH - 9.10

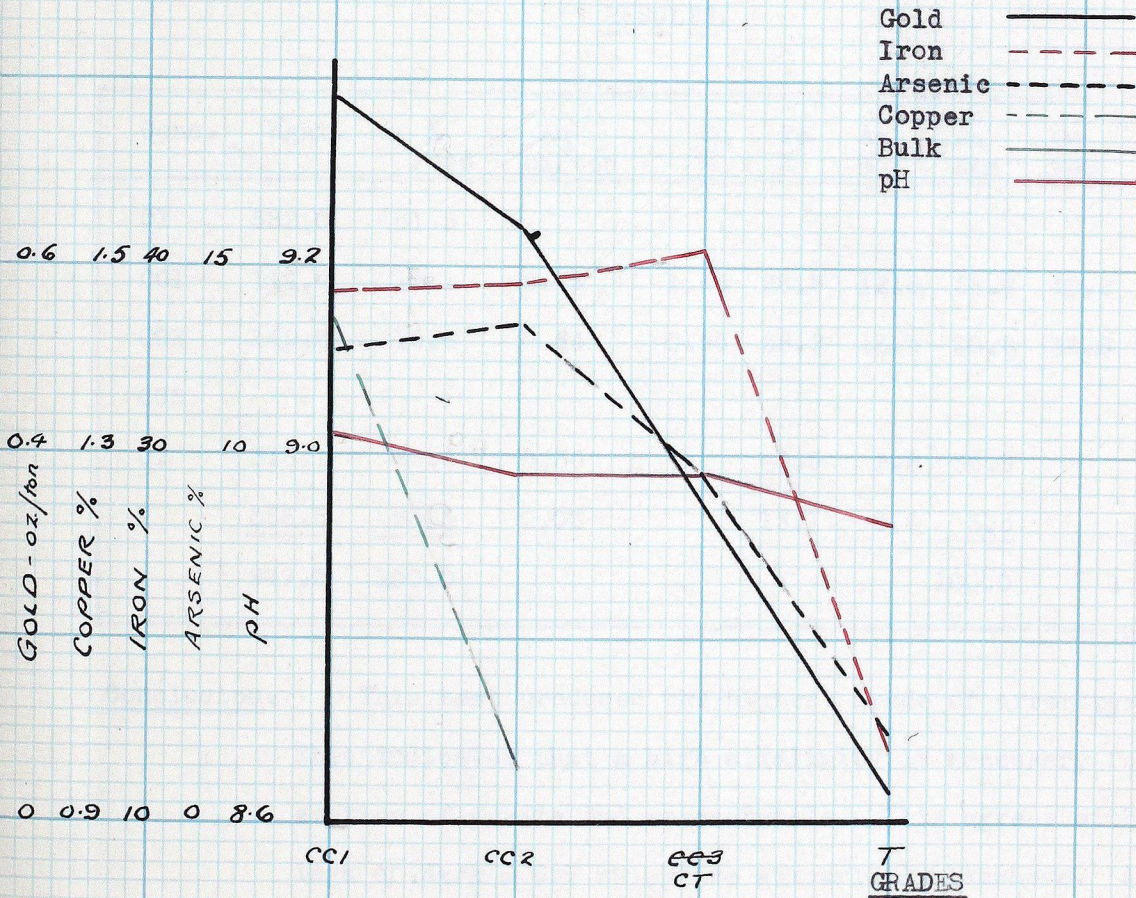
Conditioned - 1 minute Skimmed - 8 minutes

The froth was good and the bubbles were well mineralized.

TEST 21



RECOVERIES



Cleaner Concentrate 2.

Skimmed - 35 minutes

pH - 8.92

This is a continuation of 1. The skimming was continued until the bubbles were clean.

Cleaner Concentrate 3.

Added - Aerofloat # 15 0.044 # / ton
301 0.01 # / ton

Conditioned - 2 minutes

Skimmed -

This test was abandoned due to mechanical trouble, but the froth was very similar to 1 and 2.

RESULTS.

Conc.	Wght.	% Feed	Au oz/ton	Cu %	Fe %	As %	Au Rec	Cu Rec	Fe Rec	Au Rec
C1	582.0	61.5								
CC1	114.4	12.1	0.78	1.44	38.5	12.8	28.8	40.2	15.7	20.0
CC2	149.5	15.8	0.64	0.95	38.9	13.5	30.8	34.8	20.8	27.7
CC3										
CT	318.1	33.6	0.36		40.5	9.2	36.9		46.1	40.2
T	<u>365.5</u>	<u>38.5</u>	0.03		13.1	2.4	<u>3.5</u>		<u>17.4</u>	<u>12.1</u>
	947.5	100.0					100.0	100.0	100.0	100.0

Conclusion. This test produced the highest grade of arsenopyrite obtained. This indicates that a high alkalinity is necessary for the separation of pyrite and arsenopyrite. This agrees with the findings of Wark & Cox which are summarized elsewhere. Although the arsenic content was 13.59 it was not a commercial arsenic concentrate.

GAS FLOTATION

GAS FLOTATION

These tests were not successful so the remarks in the conclusion for each test will be sufficient. There was not enough time to make a thorough investigation so the matter was dropped when the second test was finished.

The hydrogen sulphide gas was generated in the usual manner and introduced into the cell through the ordinary air passage.

Carbon dioxide was made by the action of 1:1 H_2SO_4 on lumps of Soda Ash. The gas entered the pulp through the same air passage.

For the results, see tests #22 and 23.

TEST 22.

To Duplicate Test # 4, Using H₂S as the Gas Phase.

Charge to Ball Mill.

Ore	1000 gms.
Water	1000 gms.
Lime	2 # / ton
KCN	0.00 # / ton

Time of Grinding - 20 minutes.

Concentrate

Cell	Cresylic Acid	0.109 # / ton
	Aerofloat # 15	0.044 # / ton
	# 301	0.05 # / ton

Conditioned - 2 minutes.

As a non-mineralized froth was obtained, 1.0 # / ton, CuSO₄ was added. This had no effect so an extra 0.15 # / ton of # 301 was added. The froth was still non-mineralized, so the test was abandoned.

Conclusion. The H₂S fouds the pulp by precipitating sulphur which ~~coats~~^{coag} the mineral particles.

This is explained by a consideration of the law of mass-action which is shown by the equation:-

$$\frac{(H)^2 (S)}{H_2S} = K$$

As the ionization constant K is low, an increase of the hydrogen ion concentration due to an acid circuit necessitates a proportional decrease in the sulphur ion concentration. This forces the sulphur to precipitate forming a sulphur coating on the mineral which inhibits the flotation.

TEST 23.To Duplicate Test # 4, Using CO₂ as the Gas Phase.

Charge to Ball Mill

Ore	1000 gms.
Water	1000 gms.
Lime	2.0 # / ton
KCN	0.10 # / ton

Time of Grinding - 20 minutes.Concentrate 1.

Cell	Cresylic Acid	0.11 # / ton
	Aerofloat # 15	0.044 # / ton
	# 301	0.05 # / ton
Conditioned	- 2 minutes	
Skimmed	- 10 "	
pH	- 6.67	

The froth was weak and dirty while the bubbles were small and lightly mineralized.

Concentrate 2.

Added	- CuSO ₄	1.0 # / ton
Conditioned	- 2 minutes	
Skimmed	- 10 "	
pH	- 6.60	

The bubbles were small and lightly mineralized; the froth was weak and dirty.

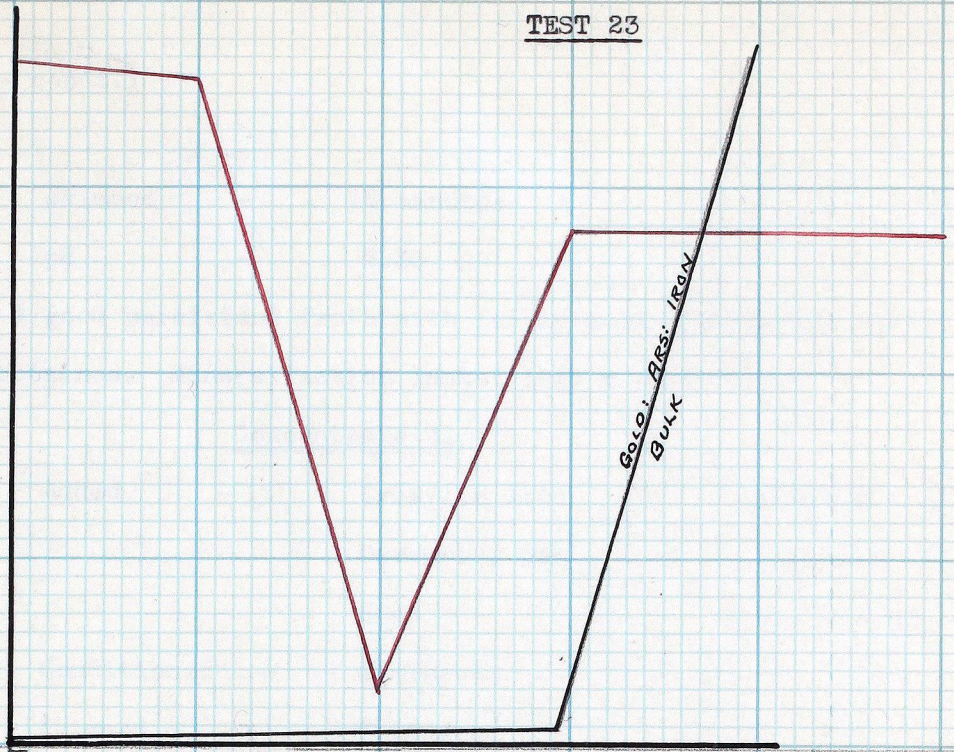
Concentrate 3.

Added	- Pine Oil	0.08 # / ton
	#301	0.02 # / ton
Conditioned	- H ₂ SO ₄	3 c.c. of 1:1
	2 minutes.	

TEST 23

GOLD %
ARSENIC %
IRON %
BULK %
pH

50 50 50 50 5
0 0 0 0 3



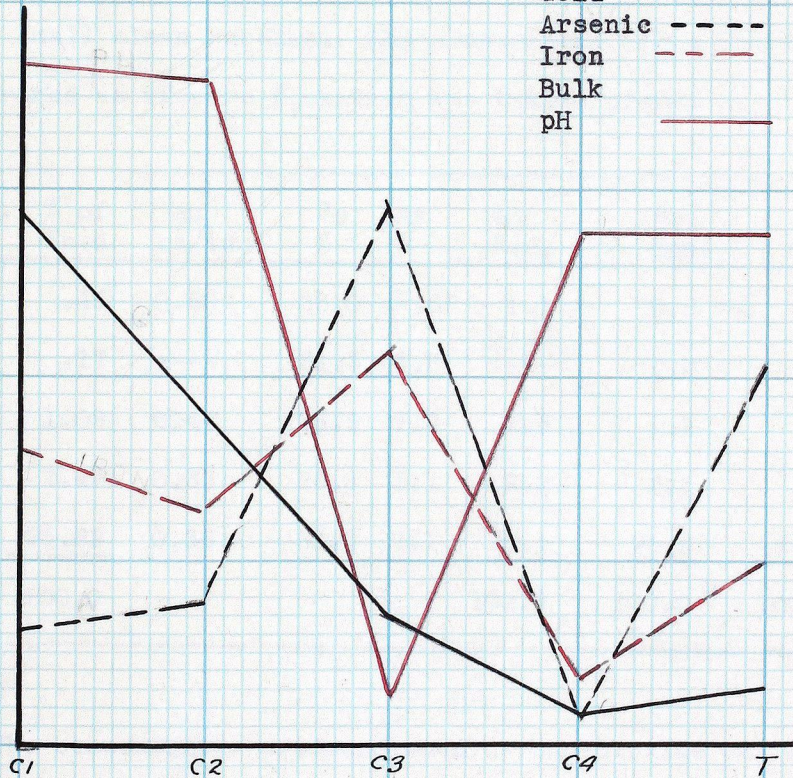
RECOVERIES

Gold ———
Arsenic - - -
Iron - - -
Bulk ———
pH ———

0.9 40 8 6

Gold - oz/ton
IRON %
ARSENIC %
pH

0.3 25 5 3



GRADES

(Test 23 - con't.)

Skimmed - 15 minutes

pH - 3.30

A good persistent froth with fairly well-mineralized bubbles was obtained. Mineralization lasted only a short time.

Concentrate 4.

Added - Cresylic Acid - 0.218 # / ton

Barrett # 634 - \pm 0.25 # / tonSoap - \pm 4.0 # / ton

Conditioned - 2 minutes

Skimmed - 7 minutes

pH - 5.78

At first the bubbles were large and heavily mineralized. After 5 minutes they changed rapidly producing a good froth of small clean bubbles. The concentrate had a peculiar appearance as if some flotation was taking place.

Conc.	Wght.	% of Feed	Au oz/ton	Fe %	As %	Au Rec.	Fe Rec.	As Rec.
1	4.6	.45	0.88	33.0	5.65	1.1	0.5	0.5
2	1.8	.17	----	31.3	5.8	---	0.3	0.2
3	11.0	1.09	0.44	35.9	7.9	1.3	1.3	1.5
4	24.8	2.47	0.33	26.8	5.1	2.2	2.1	2.1
T	<u>967.0</u>	<u>95.82</u>	0.36	30.0	7.1	<u>95.4</u>	<u>95.8</u>	<u>96.7</u>
	1009.2	100.0				100.0	100.0	100.0

Conclusion.

The concentrates produced when carbon-dioxide was used as the gas phase were lower in grade than these of test 4. This is due to the formation of carbonic acid in the solution and the probable resultant coating of carbonate on the mineral surfaces, thereby reducing the floatability.

CYANIDATION

CONCLUSIONS

RECOMMENDATIONS

THEORY

&

TESTS

-oOo-oOo-oOo-

SUMMARY OF RESULTS OF CYANIDATION

The results are summarized in the following items. Further particulars can be found in the descriptions for each test. It is concluded-

- (1) That no matter what type of preliminary treatment the ore receives, excepting fine grinding, the percent recovery of the gold by cyanidation is practically the same.
- (2) That the consumption of lime and cyanide can be lowered considerably by the use of litharge. This appears to be an example of a chemical oxidizing agent proving beneficial.
- (3) That bromocyanide is not necessary.
- (4) That fine grinding will increase the gold recovery and at the same time increase the consumption of the lime and cyanide.
- (5) That 48 hours are sufficient for agitation and that a longer period is unnecessary and increases the reagent consumption.
- (6) The following results were obtained from the tails of test 37, which was a direct cyanidation experiment.

Size Mesh	Size u	Wt. Grs	% Wt.	Gold oz/ton	Gold Dist'n
‡ 250	‡ 56	0.5	0.36	--	--
250-400	56-40	19.4	14.00	0.22	12.80
400-560	40-28	39.3	28.40	0.19	22.45
560-850	28-20	31.6	22.85	0.20	18.95
850-1100	20-14	20.8	15.04	0.26	16.22
1100-1700	14-10	9.8	7.08	0.28	8.23
-1700	-10	16.9	12.20	0.42	21.30
		138.3	100.0		100.0

A recommendation for further work in connection with cyanide tails can be found under heading "Recommendations for Further Work". See No. 4.

- (7) The following results were obtained from the infrasizing of test 34, which was straight cyanidation after fine grinding.

Size Mesh	Size u	Wt Grs.	% Wt.	Gold oz/ton	Gold Dist'n
‡ 250	‡ 56	0.1	0.07	--	--
250-400	56-40	0.2	0.13	--	--
400-560	40-28	0.5	0.33	--	--
560-850	28-20	3.3	2.20	0.10	1.44
850-1100	20-14	21.8	14.54	0.18	16.92
1100-1700	14-10	40.1	26.75	0.18	31.02
-1700	-10	83.9	<u>56.00</u>	0.14	<u>50.62</u>
			100.0		100

The results show that the gold remaining in the very finest grind is the chief source of loss.

RECOMMENDATIONS FOR FUTURE WORK

If at any time further tests are made on the Wisconsin Ore, the following suggestions might be of some help.

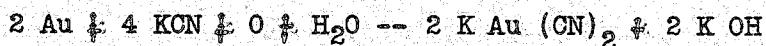
- (1) The possibilities of very fine grinding should be examined. Although the gold recovery in test 38 was 72.7%, the grinding was beyond the present practical range.
- (2) The use of bromocyanogen should be studied. Although in these tests the addition did not materially increase the recovery, it should be noted that there was no excess at the end of the test, although there had been midway thru' the run.
- (3) Fresh ore should be used as much as possible, as it was found that oxidation lowered the recovery.
- (4) An analysis of the tails from a standard cyanidation test on the ore was made and reported elsewhere. It is suggested that a similar

test be made by infrasizing the residue and separating each tube product into pyrite and arsenopyrite and assaying each concentrate. This would show the extent of the action of the cyanide upon each mineral.

THEORY OF CYANIDATION

CHEMISTRY OF PROCESS:

The usual reaction given for the dissolution of gold and metallic silver in cyanide solutions is known as Elsner's equation



The action of cyanide on silver sulphide is not as well known, but it might be written as

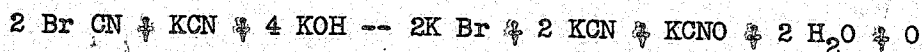


FUNCTION OF OXYGEN

Oxygen appears to be an indispensable factor, either directly or indirectly, in the dissolution of gold & silver by cyanide solutions. The most generally useful agent for this purpose is atmospheric oxygen and usually a sufficient amount is absorbed by the solutions in their circulation.

Chemical oxidizing agents are generally not satisfactory as they oxidize the cyanide to cyanate, which is useless in that form. Two agents that were used in the tests were, however, successful, litharge or lead oxide, and, to a lesser extent, bromocyanogen.

The litharge was very useful for reducing the lime and cyanide consumption, and bromocyanogen for raising the recovery slightly. Julian and Smart¹⁰ say that the activity of the latter is not due to the liberty of the cyanogen, though that probably occurs, but to a liberation of oxygen.



Should an ore contain a reducing agent sufficient extra oxygen and cyanide must be added to make possible the proper dissolution of the precious metals.

An increase in temperature increases the rate of dissolution of the metals.

STRENGTH OF THE SOLUTION IN CYANIDE

There is no advantage gained by increasing the strength of the solution beyond that which is actually needed for the dissolution of the precious metals. A further increase leads to chemical and mechanical losses of the cyanide which are unnecessary.

METHOD OF CYANIDATION

The charge was placed in the rod mill with an equal amount of water, and ground for 20 minutes. The pulp was filtered to reduce the water content and put into an inverted bottomless acid bottle. Air at a low pressure was passed in in such a manner that it entered the pulp from the bottom and kept the contents in constant motion. The reagents were added at this point.

The lime and cyanide contents were checked several times during each run to assure an excess of each.

At the end of the run a final analysis was made of the reagents and the solution filtered off. The residue was dried and assayed for gold.

The recovery was calculated from the relation

$$100 \times \frac{\text{Assay of Residue}}{\text{Head Assay}} = \text{percent loss of gold.}$$

TEST 24.

To Cyanide The Ore.

Charge to Ball Mill

Ore	500 gms.
Water	500 gms.
Lime	10 # / ton of ore
Cyanide	5 # / ton of solution

Time of Grinding - 20 minutes.Cyanidation

Pulp Density 3:1

Time of Cyanidation 48 hours

Lime Consumption.Lime addedLime Recovered

2.5 gms.

Titrated, 25 c.c. at 0.035 % -- 0.00875 gms.

" , 25 c.c. at 0.028 % -- 0.007 gms.

Left at finish , 1450 c.c. at 0.029% - 0.402 gms.

1500 c.c. 0.417 gms.

Lime lost -- 2.5 - 0.42 -- 2.08 gms.

Lime consumption / ton of ore -- $\frac{(2000)(2.08)}{500}$ -- 8.32#

Cyanide Consumption.KCN addedKCN recovered

5 gms

Titrated, 25 c.c. at .005 % -- .00125 gms.

5 "

" , 25 c.c. at .285 % -- .071 gms.

10 gms.

Left at finish, 1450 c.c. at .19 % -- 2.755 gms.

1500 c.c. 2.837 gms.

KCN lost -- 10 - 2.84 -- 7.16 gms.

∴ KCN consumption -- 7.16 # / ton of solution.

Au recovery -- 59.1 %

Conclusion.

This indicates that straight cyanidation of the ore will give a 59 % recovery of the gold with a moderately low reagent consumption.

TEST 25.To Roast the Ore at a Low Temperature and Then Cyanide.

<u>Roasting.</u> Temperature of Roasting	475° C.
Time of Roasting	2 hours
Weight of charge before roasting	500 grs.
" " after "	437.8 grs.
Percent loss in weight	12.4%

The roasted charge was thoroughly mixed with lime (16#/ton), moistened, and allowed to stand for forty-eight hours. Then it was cyanided.

<u>Cyanidation.</u> Pulp Density	3:1
Time of Cyanidation	48 hours

Lime Consumption.

<u>Lime Added</u>		<u>Lime Recovered</u>
4.0 grs.	Titrated, 25 c.c. at 0.004%	0
2.2 grs.	" 25 c.c. at 0.041%	0.0102 grs.
<u>2.2 grs.</u>	Left at finish, <u>1263</u> c.c. at 0.040%	<u>0.5052</u> grs.
8.8 grs.	1313 c.c.	0.5154 grs.

Lime consumption $8.4 - 0.52 = 7.88$ grs.

Lime consumption / ton ore $\frac{(2000)(7.88)}{437.8} = 36.0 \text{ \#}$

Cyanide Consumption

<u>KCN Added</u>		<u>KCN Recovered</u>
4.4 grs.	Titrated, 25 c.c. at 0.0%	0
<u>8.8 grs.</u>	" , 25 c.c. at 0.40%	0.100 grs.
13.2 grs.	Left at finish, 1263 cc. at 0.18%	<u>2.318</u> grs.
		2.418 grs.

KCN consumed $13.2 - 2.42 = 10.78$ grs.

KCN consumption / ton of ore $\frac{(2000)(10.78)}{1751} = 12.32 \text{ \#}$

Gold Recovery 61.4 %

Conclusion

The slight increase in recovery does not warrant roasting before cyanidation, nor the increased cost of lime and cyanide.

TEST 26.

To Make A Bulk Concentrate and Then Cyanide It.

Charge to Ball Mill

Ore	-	1000 gms.
Water	-	1000 gms.
Na ₂ CO ₃	-	4 # / ton
KCN	-	0.10 # / ton
# 203	-	0.05 # / ton
# 301	-	0.05 # / ton

Time of grind - 20minutes

Flotation

Cell

Concentrate

Added	-	Aerofloat # 15	-	0.132 # / ton
Conditioned	-	2 minutes		
Skimmed	-	30 minutes		
pH	-	8.63		

A good, persistent froth of well-mineralized bubbles was obtained.

Gold recovery = 88.3 %

Cyanidation

Time of Cyanidation	-	48 hours
Pulp Density	-	3:1
Consumption	-	Lime - 8.1 # /ton
		KCN - 3.05 # /ton
Gold Recovery for Cyanidation	-	61.3 %
Overall gold recovery	-	54.0 %

Conclusion. Results obtained indicate that the decreased gold recovery would not warrant concentration by flotation before cyanidation.

TEST 27To Make A Bulk Concentrate for a Future Cyanide Test.

Charge to Ball Mill

Ore	-	1000 gms.
Water	-	1000 gms.
Na ₂ CO ₃	-	4 # / ton
KCN	-	0.10 # / ton
# 208	-	0.05 # / ton
# 301	-	0.05 # / ton

Time of Grind - 20 minutes

Cell

Concentrate

Added - Aerofloat # 15 0.132

Conditioned - 2 minutes

Skimmed - 30 "

pH - 8.63

Concentrate - 670.0 grs.

Tailings - 355.2 grs.

1025.2 grs.

Gold Recovery - 89.0 %

TEST 28.To Test the Effect of Litharge on Cyanidation.

Charge in Ball Mill

Ore	-	500 gms.
Water	-	500 gms.
KCN	-	5 # /ton of sol.
Lime	-	15 # /ton of ore
Litharge	-	1.5# /ton of ore

<u>Time of Grinding</u>	-	20 minutes.
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<u>Time of Cyanidation</u>	-	48 hours
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<u>Pulp Density</u>	-	3:1
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Consumption

Lime	-	9.64 # / ton
------	---	--------------

KCN	-	1.72 # / ton
-----	---	--------------

Gold Recovery	-	56.8 %
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Conclusion.

It is evident that litharge reduces considerably the lime and cyanide consumption without increasing the gold recovery. The reagent consumption decrease is probably due to the weak oxidation power of the litharge, Should a cyanidation plant be erected, the use of litharge to reduce the consumption of the lime and cyanide, from an economical standpoint, should be considered.

TEST 29

To Test The Effect of Litharge on Cyanidation using
Less Lime than Test 24.

Charge to Ball Mill

Ore	-	500 gms.
Water	-	500 gms.
Lime	p	5 # / ton of ore
KCN	-	5 # / ton of sol.
Litharge		1.5 # / ton of ore

Time of Grinding - 20 minutes.

Time of Cyanidation - 48 hours

Pulp Density - 3:1

Consumption

Lime	-	3.12 # / ton of ore
KCN	-	0.54 # / ton of solution

Gold Recovery - 54.6 %

Conclusion.

The consumption of lime and cyanide was again lowered through the use of litharge. The difference in lime content made little difference.

TEST 30.To Roast the Ore in Presence of Lime and then Cyanide.

<u>Roasting.</u>	Temperature	-	550° C.
	Time of roast	-	2 hours
	Charge to furnace	-	500 gms.
	Wght. after roasting	-	<u>452.8 gms.</u>
	Loss in weight	-	47.2 gms.
	Per cent loss	-	9.4 %
	Lime added	-	5 # /ton ore

The ore was disc-pulverized to -100 mesh and the lime added before roasting in a reducing atmosphere. Before cyanidation the lime was washed out and the ore reground for 10 minutes in a ball mill with 500 gms. of water.

<u>Cyanidation</u>	<u>Time of Cyanidation</u>	-	48hours.
	<u>Pulp density</u>	-	3:1
<u>Consumption</u>	Lime	-	- 10.5 # / ton ore
	KCN	-	- 3.7 # / ton solution
	Gold Recovery	-	- 56.9 %

Conclusion.

Results obtained again show that roasting before cyanidation does not improve recovery. It is interesting to note that roasting in the presence of lime decreases the consumption of both lime and cyanide. For comparative results consult Test 25.

TEST 31.

To Roast the Ore in Presence of More Lime than Test 26 and
Then Cyanide.

Roasting.

Temperature	-	550° C.
Time of Roast	-	2 hours
Charge to furnace	-	500 gms.
Weight after roast	-	<u>460 gms.</u>
Loss in weight	-	40 gms.
Per cent loss	-	8.0 %
Lime added	-	25.0 # / ton ore

The ore was disc pulverized to -100 mesh and the lime added before roasting in a reducing atmosphere.

Before cyanidation the lime was washed out and the ore reground for 10 minutes in a ball mill with 500 gms. water.

<u>Time of Cyanidation</u>	-	45 hours
<u>Pulp density</u>	-	3:1

NOTE. Due to an accident, the bottle was broken and the test was not finished.

TEST 32.

To Test the Effect of Bromocyanide on the Ore.

Charge to Ball Mill	Ore	-	500 gms.
	Water	-	500 gms.

Grinding Time - 20 minutes

Cyanidation

Time of Cyaniding - 48 hours

Pulp
Pulp Density - 3:1

The ore was cyanided for 24 hours with KCN and then for 24 hours more with BrCN and KCN. The latter was added after the solution was reduced with H_2SO_4 until an excess alkalinity of 0.008 % CaO was reached.

Consumption.

Lime	-	5.88 # / ton ore
BrCN	-	1.26 # / ton of solution
KCN	-	5.0 # / ton " "

Gold Recovery - 63.6 %

NOTE. When end of test was reached, there was no BrCN left. This test was rerun as Test 35.

Conclusion.

The gold recovery was slightly increased. This is probably due to the increased liberation of free cyanide from the bromocyanide.

TEST 33.To Test the Effect of Oxidation.Charge to Ball Mill.

Ore	-	500 gms.
Water	-	500 gms.

<u>Time of Grinding</u>	-	20 minutes.
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After grinding, the water was filtered off and the ore left standing in a moist condition for 5 days.

Cyanidation.

<u>Time of Cyanidation</u>	-	48 hours
----------------------------	---	----------

<u>Pulp Density</u>	-	3:1
---------------------	---	-----

Consumption.

Lime	-	7.64 # / ton ore
------	---	------------------

KCN	-	4.07 # / ton of solution
-----	---	--------------------------

Gold Recovery	-	45.5 %
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Conclusion.

The low recovery in this test is probably due to oxidation of one of the sulphides, and the oxidized material coating the gold.

TEST 34.To Test The Effect Of Fine Grinding.

Charge to Ball Mill

Ore	-	500 gms.
Water	-	500 gms.

<u>Time of Grinding</u>	-	1 hour 30 minutes.
-------------------------	---	--------------------

Cyanidation

<u>Time of Cyanidation</u>	-	48 hours
----------------------------	---	----------

<u>Pulp Density</u>	-	3:1
---------------------	---	-----

Consumption.

Lime	-	20 # / ton of ore
------	---	-------------------

KCN	-	9 # / ton of solution
-----	---	-----------------------

Gold Recovery	-	65.9 %
---------------	---	--------

Conclusion.

Fine grinding increased the gold recovery slightly, and the consumption of lime and cyanide considerably. While the consumption might be lowered by the use of litharge, it is doubtful that the increased recovery would warrant the extra cost of the finer grinding.

TEST 35.To Test the Effect of Bromocyanide on the Ore.

NOTE. This is a repeat of Test 32 with more careful control of excess lime and BrCN.

Charge to Ball Mill	Ore	-	500 gms.
	Water	-	500 gms.

<u>Time of grinding</u>	-	20 minutes.
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Cyanidation

<u>Time of Cyanidation with KCN</u>	-	44 hours
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" " " " " BrCN	-	4 hours
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<u>Pulp Density</u>	-	3:1
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First charge	-	Lime	-	6 # / ton of ore
		KCN	-	5 # / ton of solution

Second Charge	-	KCN	-	5 # / ton of solution
		BrCN	-	0.23 # / ton of solution

Consumption		Lime	-	3.84 # / ton of ore
		KCN	-	9.17 # / ton # solution.
		BrCN	-	0.23 # / ton " " " .

Gold Recovery	-	56.7 %
---------------	---	--------

Conclusion. The use of bromocyanide would not raise the recovery sufficiently to pay for the increased cost of reagents.

TEST 36

To test the effect of Cyanide (KCN) on a concentrate without using any lime to give protective alkalinity.

NOTE - The concentrate from Test 23 was taken and 500 grams weighed out.

No further grinding was done.

CYANIDATION

Time of Cyanidation - 48 hrs.

Pulp Density - 3:1

CHARGE

KCN - 10#/ton of ore

NOTE

During the run, when samples of the solution were taken for determination of the protective alkalinity and cyanide content, it was found that colloidal particles turned the solution a light brown color. This prevented accurate determination by titration due to the need for a colorless solution in the methods previously described. Consequently it was ascertained only that there was an excess of cyanide and that the solution was alkaline.

At the end of the run the solution was still colored, so after filtering, lime was added to the filtrate to flocculate the particles and give a clear solution, suitable for titration. This gave two residues, one containing the main bulk of the tailings and no lime; the other the finer particles, or slimes.

CONSUMPTION

KCN - 8.61 #/ton of solution

PH at end of run = 11.0

Gold Recovery - 61.4%

Overall gold recovery 54.6%

TEST 36CONCLUSION

Results indicate that lime is not essential for the recovery of gold. From tests made during the run, it was ascertained that the pulp was alk^kaline during the entire period. The gold recovery was the same as the corresponding test with lime while the cyanide consumption was a trifle more.

TEST 37

To Cyanide the ore and float the tailings.

Charge to Ball Mill ore - 500 grs

 water - 500 grs

Time of Grinding - 20 minutes

CYANIDATION

Time of Cyanidation - 48 hours

Pulp Density - 3:1

Additions - Lime 10# /ton of ore

 KCN 10# /ton of ore

Consumption - Lime - 9.32# / ton of ore

 KCN - 7.91# / ton of solution

Gold Recovery - 59.0%

FLOTATION

No concentrate was taken. When the tailings were transferred to a 500 gram machine and the reagents added, it was found that a dirty froth of small bubbles was formed.

TEST 38

To Grind Extra Fine and Cyanide for forty-eight (48) Hours.

Charge to Ball Mill Ore - 200 grs

Water - 200 grs

Time of grinding - 3 hours.

Time of Cyanidation - 48 hours

Pulp Density 5:1

Added - Lime - 20# /ton ore

KCN - 10# /ton ore

This test was agitated for 12 hours before any reagents were added, and the PH was found to be 10.03.

Consumption Lime - 19.85# /ton ore

KCN - 9.75# /ton solution

Gold Recovery - 72.7%

CONCLUSION

The gold recovery was increased approximately 15% by the finer grinding. Should the cyanidation method of treatment be used, it must be decided whether the increased cost of grinding would be warranted by the increase in the recovery of gold.

TEST 39

To Grind Extra Fine and Cyanide for Ninety Six (96) hours.

Charge to Ball Mill - Ore 200 grs

water 200 grs

Time of Grinding - 3 hours.

CYANIDATION

Time of Cyanidation - 96 hours

Pulp Density 5:1

Added Lime - 30# /ton of ore

KCN - 15# /ton of solution

Consumption - Lime - 29.70# /ton ore

KCN - 14.70# /ton of solution

Gold Recovery - 72.7%

CONCLUSION

The results indicated that an increase in time of contact between the cyanide and gold is not necessary, as the gold recovery was the same for both periods. The lime and cyanide consumption is also greatly increased.

ROASTING

R O A S T I N G

Low temperature roasting tests were made in an attempt to drive off a part of the sulphur and expose the ore more completely to the action of cyanide. The conclusions reached, however, were that roasting had no effect upon the final gold recovery and that the consumption of lime and cyanide was increased considerably.

METHODS:

500 grs. of ore were taken and ground in a disc-pulverizer to pass a 100 mesh screen. They were put into a small oven with a piece of charcoal to form a reducing atmosphere, and the front plugged with fire clay. Finally the oven was placed in an electric furnace which had been heated to the required temperature and left in it for a specified time. At the end of run the charge was weighed to determine the loss. See Tests 25 & 30.

BLANKET CONCENTRATION

BLANKET CONCENTRATION

The results of this test confirmed the conclusions previously reached that a satisfactory blanket concentrate could not be made. The grade of the product was the same as that of the mill feed - 0.44 oz /ton, so it was obvious that the gold recovery depended entirely upon the bulk.

The blanket covered the bottom of a rectangular trough set at an angle of 15-20° from the horizontal. The ore, previously diluted to a pulp density of 8:1, was added at the top. A continuous spray of water washed the ore down the slope.

For results, see tests #15 and 16.

ASSAY METHODS

ASSAY METHODSGOLD AND SILVER:

The gold and silver content of the ore was determined by the niter method. The majority of the gold assays for the various products was run by the standard nails method, which was checked and found to be sufficiently accurate for the purpose. Silver determinations were not made for any of the tests.

COPPER:

Several methods of analysis were tried at first, but a satisfactory end point in titration could not be obtained. This was probably due to the arsenic which had not been entirely eliminated. Finally the method of precipitation of the minerals as sulphides by hydrogen sulphide gas was tried. Although a long analysis it was accurate, and consequently was adopted for all future copper assays.

C O P P E R

1.0 grs of ore in copper flask

6 grs sodium sulphate

10 cc. H_2SO_4

Fuse thoroughly until mass is a light yellow color.

Cool sufficiently to add 30-40 cc. water.

5 drops HCl.

Heat at near boiling or boiling temp. for half hour

Filter - Ppt. - Pb.

Filt - Fe, Cu, Sb, As.

Dilute to 300 cc. and warm. Pass in H_2S gas until all sulphides are ppted. and solution clear.

Filter Ppt. - Cu, As, Sb, Pb, Sn.

Filt - Fe.

Return ppt. to same beaker washing paper with Conc. sodium polysulphide solution.

Digest at temp. below boiling until soln. clear and dark ppt. coagulates.

Filter Ppt. - Cu, Pb.

Filt.- As, Sb, Sn.

Return ppt. to same beaker washing with water, then hot 1:1 HNO_3 , then Bromine water.

Add 5 cc. conc. H_2SO_4 to ppt. lead.

Bake.

Add water and filter out lead.

5 cc. HNO_3 and evaporate nearly to dryness.

Make just alkaline with 1:1 NH_4OH . Boil to expell excess.

COPPER (Continued)

Just acidify with Acetic Acid.

Cool thoroughly.

2 grs. KI and let stand 5 minutes.

Titrate with sodium thiosulphate using soluble starch soln. as indicator.

IRON - Dichromate method for Sulphides.

Take 0.5 grs. into 250 cc. beaker.

10 cc. water and 15 cc. HNO_3

Warm till red fumes all driven off.

15 cc. HCl and warm.

10 cc. Chlorate mixture and warm

10 cc. 1:1 H_2SO_4

Fume nearly to dryness.

Cool and add 25 cc. water

5 cc. HCl - boil

While still hot, add Sn Cl_2 drop by drop until yellow color disappears and add 1 drop in excess.

Cool quickly and add about 15 cc. Hg Cl_2

Titrate with Potassium dichromate using potassium Ferricyanide as indicator. Use white plate. End point reached when blue in drops disappears.

Reagents;

Stannous chloride - 60 cc. in 600 cc. HCl .

Mercuric chloride - saturated solution

Potassium ferricyanide - small crystal in 50 cc. water

Potassium dichromate - To standardize - weigh up 0.7 grs

ferrous ammonium sulphate, acidify with HCl ,

1 drop of Sn Cl_2 , excess Hg Cl_2 and titrate.

SULPHUR:

Take 0.5 grs of ore

40 cc. water and 10 cc. nitric chlorate.

Heat to dryness overnight

5-6 grs Na_2CO_3 and 25 cc. water.

Boil 10 minutes, dilute to 100 cc. and boil.

Filter through: No. 1 Whatman paper. Wash twice with hot water.

Ppt. - Fe and other impurities.

Filt.- Na_2SO_4

20 cc. HCl to ppt.

Add excess Na_2CO_3 until get further ppt. Then 25 cc. water.

Boil 10 minutes, dilute to 100 cc. and boil,

Filter. Add this filtrate to first one.

Neutralize combined filtrates with HCl, add 2 or 3 cc. excess, boil.

Add hot BaCl_2 and boil until BaSO_4 ppts.

Filter, ignite and weigh as BaSO_4

$$\frac{\text{Wt. of Ba SO}_4 \times 0.1375}{\text{Wt of sample}} \times 100 = \% \text{ Sulphur.}$$

High sulphide ore and concentrates are usually taken down slowly or overnight with the chlorate mixture.

ARSENIC - Distillation Method

This method was found to be more accurate than that of fusion before distillation. The error is probably due to the volatilization of some of the arsenic.

Take 0.5 grs. ore in 100 cc. beaker

25 cc. water, 10 cc. chlorate mixture, digest.

10 cc. of 1:1 H_2SO_4 - to fumes overnight

Put in distillation flask, add some brick grape nuts, 3 grs. Ferrous Chloride or Cuprous Chloride, 2 grs. Ferric Chloride, 90 cc. HCl.

Receive in 400 cc. beaker containing 100 cc. water

Distill until bumping. (at 120° for half hour)

Neutralize with NH_4OH .

Just acid with HCl. Cool.

Make alkaline with $NaHCO_3$

Add starch and titrate with Iodine Solv.

REAGENTS:

Iodine - 50 grs. KI in 75 cc. water. Add 25.5 grs. of I_2 crystals and leave in warm place until all in solution. Dilute to 2 litres.

To Standardize - Put 0.99 grs. As_2O_3 in a 400 cc. beaker with NaOH and some water. Just acidify with HCl. Dilute to 300 cc. and cool. Make alkaline with $NaHCO_3$. Add starch and titrate.

Cyanide in Pulp.

Take 25 c.c. of pulp

Filter to clear

Add several drops of 5% KI solution

Titrate with Silver Nitrate solution

As soon as yellowish ppt. is just permanent when viewed against black background the end point is reached.

Lime in Pulp.

Take above solution and add a few drops of Phenolphthalein

Titrate against Oxalic acid until red color disappears.

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GRAPH SHOWING
PERCENTAGE RECOVERIES
FOR
EACH TEST



FLOTATION (CONT'D)

TEST NUMBER	CONCENTRATE NUMBER	WEIGHT	PERCENTAGE WEIGHT	GRINDING	CONDITIONING	SKIMMING	PH	REAGENTS (LBS PER TON)																ASSAY				RECOVERY PERCENT					
								TO BALL MILL								TO FLOTATION CELL								GOLD OZ./TON	COPPER PERCENT	IRON PERCENT	ARSENIC PERCENT	GOLD	IRON	ARSENIC	COPPER	PULP DENSITY	
								KCN	Na ₂ CO ₃	NaOH	LIME	# 208	# 301	Na ₂ SO ₃		PINE OIL #5	CRESYLIC ACID	AEROFLOAT # 15	AEROFLOAT #20	# 301	# 208	K E.T. XANTHATE	Z-6										CuSO ₄ 5%
15		179.8	18	20																				0.44					18.0				
16				20				0.10	4.0															0.44					27.3				
	C-1	236.5	24.8																					0.68	38.8	11.3			48.5	39.1	41.4		4.1
	C-2	285.7	30.0		2	15	6.9											0.088	0.10				0.7					10.7	15.7	13.2			
	C-3	114.9	12.0		2	12	6.78											0.044	0.05				1.0										
	T	317.3	33.2																					0.16					13.5				
		95.4	100.0																										100.0				
17				20																													
	C-1	510.7	51.0		2	30	7.98		4.0		0.05	0.05	0.10																				
	T	507.0	49.0															0.132															
		101.77	100.0																					0.16					18.9				
18				20																													
	C-1	589.4	58.0		2	30	8.42		4.0		0.05	0.05	1.0																				
	T	425.7	42.0															0.132															
		101.51	100.0																					0.14					86.5	75.1	78.8		4.1
19				20																													
	C-1	150.7	15.0		2	7	5.60		4.0					1.0																			
	C-2	144.3	14.0		2	5	5.12											0.088	0.10					3.2									
	T	711.9	71.0															0.044		0.05			1.0										
		1006.9	100.0																														
20				20																													
	C-1	320.0	32.0		2	35	8.17											0.067				0.01											
	C-2	20.4	2.1		2	10	8.15											0.069						0.40	1.5								
	T	660.4	65.9																														
		1001.3	100.1																					0.16									
21				20				0.10	4.0																								
	C-1	582.0	61.5		2	19	8.63																										
	CC-1	114.4	12.1		1	8	9.10											0.088		0.05	0.05												
	CC-2	149.5	15.8			35	8.92																										
	CC-3				2																												
	CT	318.1	33.6															0.044		0.01													
	T	365.5	38.5																														
		347.5	100.0																														

GAS FLOTATION

TEST NUMBER	CONCENTRATE NUMBER	WEIGHT	PERCENT WEIGHT	GRINDING MINS.	CONDITIONING MINS.	SKIMMING MINS.	pH	REAGENTS (LBS PER TON)										ASSAY			RECOVERY %			PULP DENSITY	GAS				
								BALL MILL		FLOTATION CELL								GOLD OZ./TON	IRON PERCENT	ARSENIC PERCENT	GOLD	IRON	ARSENIC						
								LIME	KCN	CRESYLIC ACID	AEROFLOAT #15	# 301	CuSO ₄ (5%)	H ₂ SO ₄ (1:1) (cc)	BARRETT #634	SOAP													
22	C1			20				2.0	0.10																				
					3					0.109	0.044	0.05				No	MINERALS	FLOATED											H ₂ S
23				20				2.0	0.10																				
	C1	4.6	0.45		3	10	6.67			0.109	0.044	0.05						0.88	33.0	5.65		1.1	0.5	0.5		4.1		CO ₂	
	C2	1.8	0.17			10	6.60						1.0						31.3	5.8			0.3	0.2					
	C-3	11.0	1.09			15	3.23					0.02	3.0					0.44	35.9	7.9		1.3	1.3	1.5					
	C-4	24.8	2.47			7	5.78			0.218				10.25	14.0			0.33	26.8	5.1		2.2	2.1	2.1					
	T	967.0	95.82															0.36	30.0	7.1		95.4	95.8	96.7					
		1009.2	100.0																			100.0	100.0	100.0					

CYANIDATION

ROASTING										CYANIDATION				FLOTATION																				
TEST NUMBER	WEIGHT GRS	GRINDING MINS	CYANIDATION TIME MINES	ROASTING TIME MIN 3	TEMPERATURE ROAST °C	WEIGHT AFTER ROAST GRS	LOSS OF WEIGHT GRS	PERCENT LOSS	LIME CONSUMPTION	KCN CONSUMPTION	BY CN CONSUMPTION	LITHARGE CONSUMPTION	PULP DENSITY		GOLD oz /TON TAILS	GOLD RECOVERY		ORE	CONCENTRATE WEIGHT	GRINDING MINS	CONDITIONING MINS	SKIMMING MINS		BALL MILL				CELL		GOLD oz / Ton	GOLD RECOVERY	TOTAL GOLD RECOVERY	REMARKS	
																								Na ₂ CO ₃	KCN	# 208	# 301	AEROFLOAT #15						
24	500	20	48						8.36	7.16			3.1		0.18	59.1																	59.1	
25	500		48	90	475	437.8	62.2	2.4	36.0	12.32			3.1		0.17	61.4																61.4		
26	607.6		48						8.1	3.05			3.1		0.25	61.3		1000	607.6	20	2	30			0.05	0.05		0.132	0.63	88.3	54.1	CYANIDATION OF BULK CONCENTRATE		
27			SEE	TEST	36																													
28	500	20	48						9.64	1.72		1.5	3.1		0.18	56.8																56.8		
29	500	20	48						3.12	0.54		1.5	3.1		0.19	54.6																54.6		
30	500	20	48	90	550	452.8	47.2	9.4	10.5	3.73			3.1		0.19	56.9																56.9		
31			BOTTLE	BROKEN																														
32	500	20	40						5.88	5.00	1.26		3.1		0.16	63.6																63.6		
33	500	20	48						7.64	4.07			3.1		0.24	45.5																45.5		
34	500	90	48						20.0	9.0			3.1		0.15	65.9																65.9		
35	500	20	48						3.84	9.17			3.1		0.19	56.7																56.7		
36	500		48						8.61				3.1		0.17	61.4		1000	670	20	2	30	4.0	0.10	0.05	0.05		0.132		89.0	54.6	CYANIDATION OF BULK CONCENTRATE		
37	500	20	48						9.32	7.91			3.1		0.18	59.0																59.0		
38	200	180	48						19.85	9.75			5.1		0.12	72.7																72.7		
39	200	180	96						29.1	14.7			5.1		0.12	72.7																72.7		