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# A Preliminary Study of the Separation of the Copper Sulfides from Sphalerite and the Effect of Certain Reagents on Some of the Pure Copper Minerals in Synthetic Mixtures

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A Preliminary Study of the Separation of the Copper  
Sulfides from Sphalerite and the Effect of Certain  
Reagents on Some of the Pure Copper Minerals in  
Synthetic Mixtures.

By

Gailen T. Vandel

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1931 MS. METALLURGY

REPORT  
VANDEL

Part 1.

Experiments on the selective flotation of the  
copper sulfides from sphalerite .

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## Introduction

The problem of separating the copper sulfide minerals from sphalerite, in copper - zinc ores, has been a difficult one. This is largely due to the lack of adequate research and the small amount of data obtainable on the behavior of copper and zinc sulfide minerals in flotation circuits.

The fact that each of the copper sulfides react differently with different reagents has made the scope of this problem much larger than it was first thought to be.

The original line of procedure in these experiments was to selectively separate the copper minerals from the sphalerite by depressing the zinc sulfide. Further work showed this to be quite impossible with the present knowledge of flotation if bornite or chalcocite is present. In order to gain a more thorough knowledge of the behavior of the copper sulfide minerals it was finally found necessary to treat each copper mineral independently in synthetic mixtures of the mineral and granite.

It is thought that the successful separation of the copper sulfides from sphalerite might be effected by data obtained from the synthetic mixtures.

The copper - zinc ore for this research problem was obtained from the Badger State Mine, Butte, Montana. The mixture was complex; besides the ordinary siliceous gangue the ore contained pyrite, chalcopyrite, chalcocite, bornite, covellite, tetrahedrite, sphalerite and possibly tennantite. A study of polished surfaces under the microscope showed the ore to contain various intimate associations of the copper minerals and sphalerite. Treatment of the ore involves a problem in grinding as well as one of separation of the copper sulfides from

from sphalerite.

The following tests and experiments were of such an individual nature that it was thought best to make conclusions at the end of each test or series of tests. In view of this fact these tests are presented in the order in which they were conducted in the research laboratory.

## Experiment 1

The decision was made to first try a bulk flotation on the ore and determine its behavior in the flotation machine. A charge of ore 500 grams, and water 300 grams was placed in the pebble mill and ground for 30 minutes.

The flotation test was made in a mechanically-agitated Ruth machine. Terpeneol was added as a frother, starting with 0.06 lb. / ton and later adding 0.18 lb. / ton more. Sodium carbonate 8 lb / ton and heptyl mercaptan 0.10 lb. / ton were added to the flotation machine. The resulting concentrate was heavy in pyrite, copper sulfides and sphalerite. The process was truly one of bulk flotation.

Another test was run following this and the same charge placed in the ball mill with the exception of adding the sodium carbonate 8 lb. / ton to the ball mill.

The same flotation machine was used and terpeneol 0.06 lb. / ton and amyl xanthate 0.10 lb. / ton added to the machine. Unlike the mercaptan this amount of xanthate caused over collecting and additional amounts of frother would not remove the effects.

## Experiment 2

In the light of the nonselectivity of these two tests it was decided to use potassium cyanide to depress the sphalerite. The flotation run consisted of making four concentrates using 0.01, 0.02, 0.07, and 0.02 lb. / ton respectively of ethyl xanthate in increasing amounts as given. Each concentrate showed increasing amounts of pyrite and the last concentrate was largely pyrite. As before there was little evidence of any selectivity in the operation.

### Experiment 3

Another test was now run using calcium oxide 1.0 lb./ton, sodium carbonate 4.0 lb. / ton, and 0.2 lb. / ton of potassium cyanide. The reagents were added to the ball mill with the charge of ore, as before. The concentrates were made with increasing amounts of xanthate as before. No apparent difference from that of the previous test was noticed. Observations were made with a binocular microscope.

Tests run similar to the preceding ones substantiated the results already obtained. From these results it is concluded that potassium cyanide does not depress sphalerite in the presence of some of the copper minerals. Coarse sphalerite when pure is one of the least floatable of the sulfide minerals; but in these tests is shown to be the most readily floatable. It is assumed from this that the sphalerite has been coated with copper ions and is really acting as covellite, a very readily floatable mineral.

### Experiment 4

Continued experiments for the purpose of finding a reagent to depress the sphalerite were now run. Several tests similar to the one that follows were run but the results presented in this one are the most graphic.

The charge to the ball mill was as follows; ore 500 grams, water 300 grams, calcium oxide 4.0 lb. / ton, potassium cyanide 0.4 lb. / ton, zinc nitrate 1.0 lb. / ton and the charge was ground for 30 minutes.

The following reagents were added to the Ruth machine; ethyl xanthate 0.05 lb. / ton, terpeneol 0.06 lb. / ton. A rougher concentrate and a rougher tailing were made and the pH



of the pulp maintained at 10.6. The rougher concentrate was put back in the cell and terpineol 0.06 lb. / ton and potassium dichromate 8.0 lb. / ton added. The decrease in pH value to 9.4 is due to the acidity of the dichromate. The cleaner concentrate was again recleaned and a recleaner concentrate and tailing made. The flow sheet is shown in fig. 1.

Table 1 indicates that the sphalerite is more readily floatable than the copper minerals with chalcocite coming next, if an oxidizing agent is used in cleaning. The amount of bornite present is shown to be very small. The estimated copper and zinc content is shown to be very close to that of the assay value presented in table 3. This difference can be accounted for by the fact that a deslimed product was used in making the briquetted specimens.

Table 2 shows the constituents of the cleaner tailing and it is quite evident that the bornite is depressed much more than the other copper minerals, with chalcocite next in order. It was thought that the addition of zinc nitrate would facilitate the depression of the sphalerite in a cyanide circuit. The tests proved this to be just the opposite and it was concluded that the oxidation of the copper was too rapid to allow the zinc ion to have any effect. The continued floating of the sphalerite ahead of the copper minerals again substantiates the theory that the sphalerite is acting as covellite.

It seems reasonable to assume that sphalerite cannot be depressed in the presence of the copper minerals where cyanide is being used as the depressant and in particular in the presence of bornite.

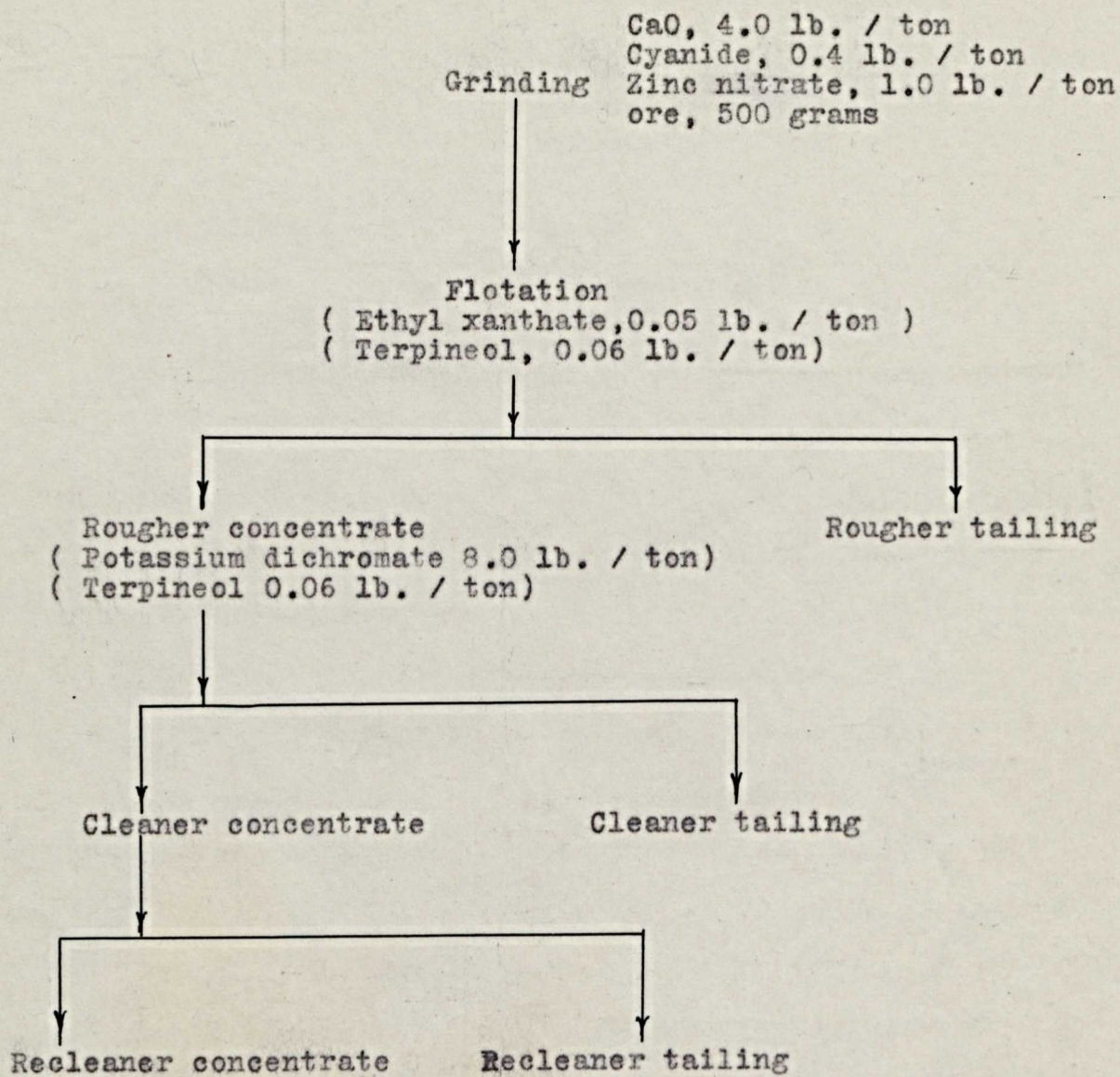


Fig. 1. Flow sheet of flotation operation, experiment 4.

Table 1

Microscopic study of recleaner concentrate, from  
a briquetted specimen.

## Free copper minerals:

Chalcocite -----	18
Bornite -----	2
Total	20

## Free zinc minerals:

Sphalerite -----	151
------------------	-----

## Binary copper minerals:

Chalcopyrite - bornite -----	7
Bornite - chalcocite -----	1
Total	8

## Ternary copper minerals:

None

## Binary copper zinc minerals:

Sphalerite - chalcocite -----	4
Sphalerite - chalcopyrite -----	1
Sphalerite - bornite -----	2
Total	7

## Ternary copper zinc minerals:

Chalcopyrite - sphalerite- bornite ----	2
Chalcopyrite - sphalerite - chalcocite -	1
Total	3

Pyrite ----- 1

Silicate ----- 3

Total 4

Grand Total 193

Estimated zinc content ----- 50 %

Estimated copper content ----- 10 %

Table 2

Microscopic study of the cleaner tailing from a  
briquetted specimen.

Free copper minerals:

Chalcocite -----	27
Bornite-----	47
Chalcopyrite -----	2
total	76

Free zinc minerals:

Sphalerite -----	17
------------------	----

Bianary copper minerals:

Chalcocite - bornite -----	20
Bornite - covellite -----	1
Bornite - chalcopyrite -----	25
Chalcocite - chalcopyrite -----	4
total	50

Ternary copper minerals:

Chalcocite - bornite - chalcopyrite -----	3
Bornite - covellite - chalcocite -----	1
total	4

Bianary copper - zinc minerals:

Sphalerite - chalcopyrite -----	1
Pyrite -----	4
Silicates -----	5

Grand total 157

Estimated zinc content ----- 7 %

Estimated copper content -----52 %

Table 3

Assay values on the products from experiment 4 using potassium cyanide and dichromate as zinc depressants.

Product	per cent		weight, grams
	copper	zinc	
Rougher tailing	0.11	3.0	428.2
Cleaner tailing	37.2	9.1	12.0
Recleaner tailing	21.8	5.0	8.7
Recleaner concentrate	11.4	43.0	27.0

## Experiment 5

Table 4 and 5 shows the results obtained in four tests, the first two of which were run for the express purpose of determining the desirable reagent to use as a collector and to find the smallest amount of cyanide necessary to depress the pyrite. The desire was to also make some comparisons between amyl xanthate and amyl mercaptans as collectors.

The calcium oxide and the potassium cyanide were added to the ball mill and the other reagents to the 2000 gram Denver Sub 'A' machine as shown in the tables. In the last two tests a rougher concentrate was made, returned to the cell, additional reagents added and the cleaner concentrates floated at intervals after the reagents had been added.

In test 1 the recovery is much better for both copper and zinc than indicated in test 2. This shows that the amyl mercaptan is less desirable than amyl xanthate as a collector for the copper and zinc sulfides. The amount of iron floated with amyl xanthate is much in excess of that floated when the mercaptan is used as a collector. This shows that the amyl mercaptan is much more selective in so far as the floating of

pyrite is concerned.

The zinc recovery in the first three is much better than for copper. Microscopic examination of the tailings show them to contain increasing amounts of copper minerals with increase of pH from 11 to 12 or over.

The separation of the copper zinc minerals is none too good in any of the tests but a step in the right direction seems to have been made when calcium oxy chloride was chosen as a depressant for the copper minerals. Test 4 shows cleaner concentrate 1 to contain 14.05 per cent copper and 45.5 per cent zinc which is better than results obtained heretofore. Perhaps if enough calcium oxy chloride were added to the circuit and the products cleaned enough times a fairly good separation could be effected. This is evidence that the sphalerite is floated ahead of the copper minerals and especially bornite. This of course substantiates the theory of oxidation of the copper minerals which in turn produce copper ions to activate the sphalerite.

If any of the copper minerals present in the pulp of the flotation circuit were to react with the calcium oxy chloride to produce a water soluble copper oxide at the surface of the copper sulfide mineral then the mineral would be less readily floatable and in reality depressed. If this coating were to dissolve off, as it is certain that it would, copper ions would be produced in the flotation circuit. These copper ions would replace a zinc element at the surface of the sphalerite and make the surface of that mineral into covellite. This would be the same condition that exists when copper sulfate is used to activate sphalerite that has previously been depressed with cyanide ions. This is likely what happens and would seem a reasonable explanation for the behavior of sphalerite in flotation circuits containing copper minerals.

Table 4

Reagents added to the ball mill and the rougher flotation circuit in tests of experiment 5.

Test	CaO	KCN	Amyl Xanthate	Ethyl Xanthate	Amyl Mercaptan	Terpineol	pH*
1	12	0.25	0.05			0.06	12.2
2	12	0.25			0.05	"	12.0
3	10	0.20			0.05	"	11.8
4	10	0.20		0.05		"	11.9

\* These values are for the rougher circuit only.

The charge to the ball mill was 2000 grams of ore.

Table 5

Metallurgical results obtained from tests of experiment

Test	1			2			3			4				Copper Recovery				Zinc Recovery			
	Cu	Zn	Fe	Cu	Zn	Fe	Cu	Zn	Fe	Cu	Zn	Fe	1	2	3	4	1	2	3	4	
Rougher Concentrate	46.7	21.4	19.5	17.9	24.6	13.4							92.5	71.0			96.0	80.0			
Rougher Tailing	0.85	0.18	5.4	0.88	0.69	7.2	0.34	0.21	9.1	0.13	0.27	5.4									
Cleaner Concentrate 1							17.1	32.5	8.4	14.1	45.5	6.4					36.7	17.3		45.1	
Cleaner Concentrate 2							22.2	33.4	14.3	19.3	17.4	23.1					51.7	76.0		50.1	
Cleaner Tailing							1.61	1.29	14.6	2.51	1.72	14.5					1.5	2.5		0.79	

5 lb./ ton of calcium oxychloride added to the cell before taking off the first concentrate in the roughing operation of test 4.

0.2 lb./ ton of copper sulfate and 10 lb. / ton of calcium oxide added to the cleaning operation before taking off the cleaner concentrate 1 in test 3. In test four ethyl xanthate 0.05 lb. / ton and terpineol 0.12 lb. / ton were added before taking off cleaner concentrate 1.

Amyl mercaptan 0.1 lb. / ton was added in the cleaning operation of test 3 before pulling the cleaner concentrate 2.



## Experiment 6

In line with the knowledge gained from the tests in experiment 4, several tests were run on synthetic mixtures of bornite, sphalerite and granite. The small pebble mill was used for grinding and the small 200 gram Utah type laboratory flotation machine used in floating the pulp.

In removing the first concentrate the only reagents added were terpineol and calcium oxy chloride. The resultant product was a clean zinc concentrate as shown by examination under the microscope. Upon adding amyl xanthate as a collector a clean copper concentrate was obtained. In all cases the bornite was depressed and the sphalerite floated readily.

With such good results as these preliminary experiments a test was run using a 500 gram charge and floated in the Karaut machine. The ball mill charge was granite 420 grams, sphalerite 40 grams, bornite 40 grams, calcium oxide 2 lb. / ton and calcium oxy chloride 2 lb. / ton. The grinding time was 1 hour as it was believed necessary to have a finer product than in past experiments.

The flow sheet of this operation is shown in fig. 1 and the metallurgical results in table 5. It is interesting to note that 82.4 per cent of the zinc is recovered without the addition of any collector whatsoever and that only 12.4 per cent of the copper is recovered. The regrind of the rougher concentrate 2 was principally for the purpose of bringing the copper minerals in contact with the calcium oxy chloride for a longer length of time and not to make a finer product.

These results are perhaps the best obtained on the separation of copper from zinc so far in these experiments but it must be remembered that the conditions are some what more

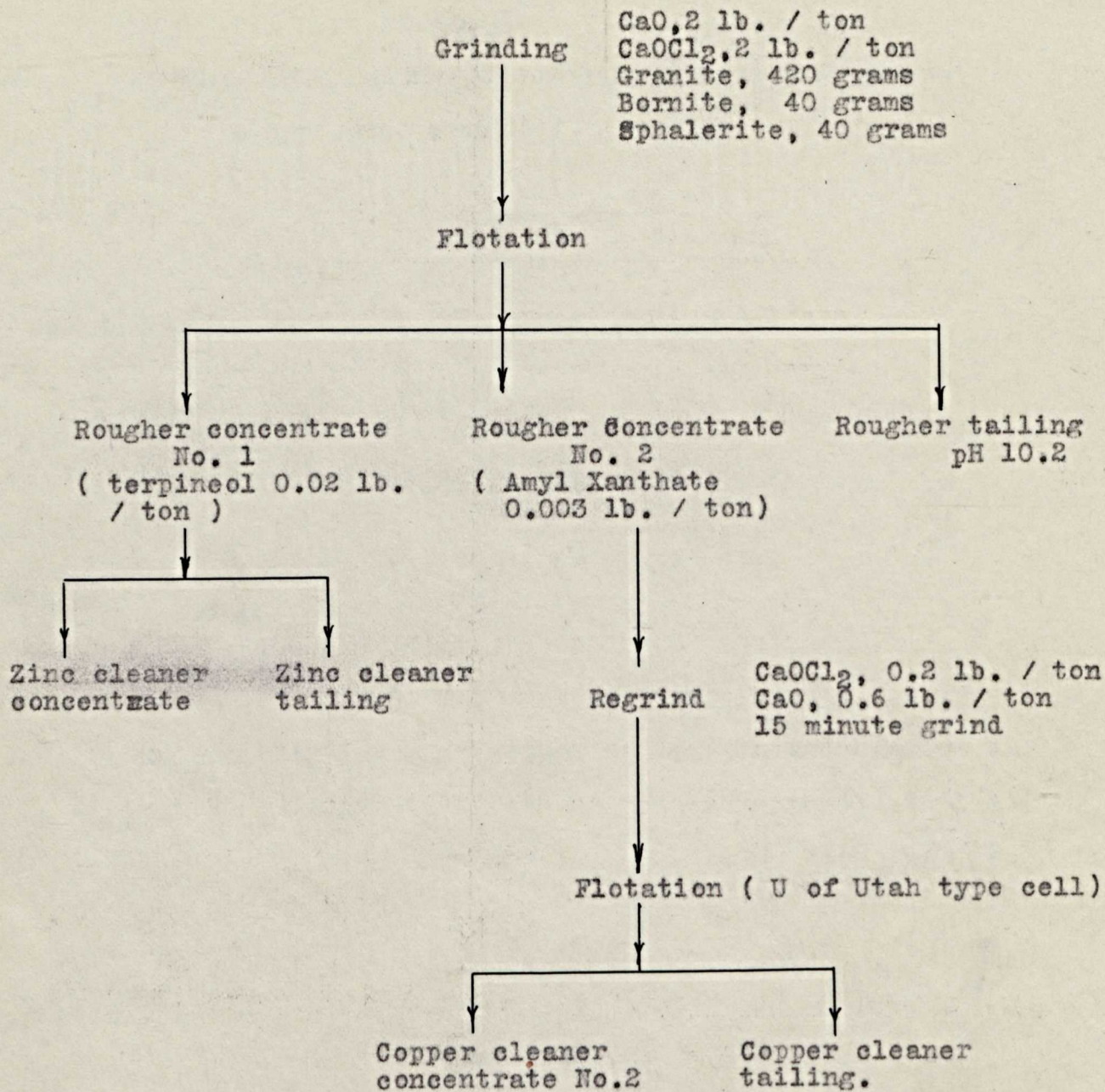


Fig. 2.- Flow sheet of operation in tests of experiment 6.

Table 6

Results from separation of copper and zinc sulfides in  
a synthetic mixture.

Product	Weight	Per cent		Recovery	
		Cu	Zn	Cu	Zn
Zinc cleaner concentrate	42.0	6.55	55.30	12.4	82.4
Zinc cleaner tailing	14.0	5.69	4.40	3.6	2.1
Copper cleaner concentrate	12.5	46.80	19.50	26.4	8.7
Copper cleaner tailing	25.0	49.20	4.50	55.5	4.0
Rougher tailing	400.0	0.12	0.19	2.1	2.7

Selectivity index of copper to zinc in roughing -- 5.5  
 " " " " in zinc cleaning-3.3  
 " " " " in Cu cleaning --2.1

Part 2.

Experiments of pure copper minerals in synthetic  
mixtures.

The problem of getting pure minerals in sufficient quantities has been almost impossible in so far as the Butte camp is concerned. Supposedly pure bornite was found, upon examination under the microscope, to contain either chalcopyrite, chalcocite or covellite disseminated through it. A magnification of 500 diameters was sometimes necessary to discern it. The same was found to be true of chalcopyrite, chalcocite and covellite; the contamination usually being bornite or covellite in the case of the chalcocite and the contamination of the chalcopyrite with bornite. The enargite obtained from the Anselmo mine appeared quite pure but upon further examination was found to contain veinlets of tetrahedrite. The enargite from the Badger State mine and the Pilot Butte mine was more usually contaminated with chalcocite.

The bornite used in the following tests was obtained from Idaho and was comparatively pure. The only contamination besides quartz was small veinlets of covellite which constituted about 0.5 per cent of the total bulk of sulfides.

The chalcopyrite used in the tests was obtained from the United Verde Copper Co., Jerome, Arizona. The impurity in this mineral was largely pyrite which amounted to about 3.5 per cent of the total sulfides.

The enargite used in the tests was obtained from the 2300 ft. level of the Anselmo mine. This mineral was found to contain quartz and pyrite upon preliminary examination. Upon a more thorough study of the mineral under the microscope it was found to also contain another white mineral which did not etch with cyanide as did the enargite. This mineral was in the form of veinlets and amounted to approximately 15 per cent of the total sulfides. From the very close resemblance of tetrahedrite and enargite and the difficulty with which tetrahedrite

etches the veinlets were assumed to be tetrahedrite.

The granite used in all the tests came from Butte and was free from the copper minerals. The magnetite contained in the granite was eliminated by tabling before using in the flotation tests.

### Chalcopyrite

The mineral assayed approximately 29 per cent copper and was ground to 35 / 100 mesh before charging to the ball mill. The granite was ground to 28 / 100 mesh. The ball mill charge consisted of granite, 450 grams, chalcopyrite 50 grams, water 200 grams and calcium oxide and potassium cyanide to suit. The grinding time in the pebble mill was 45 minutes.

In floating the above charges the Kraut laboratory machine was used and the following reagents were added to the machine; potassium amyl xanthate 0.02 lb. / ton, terpineol 0.10 lb. / ton. The time of flotation was 6 minutes in all tests.

#### Experiment 1.

These tests were with varying amounts of calcium oxide which were run for the purpose of determining the effect of calcium oxide or increased alkalinity upon the floatability of chalcopyrite.

In these tests the increase in effectiveness of the frother was quite noticeable with increase in pH. The grade of concentrate increases as the calcium oxide is increased up to 6 lb. / ton but decreases if the calcium oxide is increased further than 6 lb. / ton and up to 16 lb. / ton.

The time of floating was 6 minutes, as mentioned before, but this was entirely unnecessary except in the case of test 1 and 2. In test 4 three minutes suffices to float all the chalcopyrite. The tendency of the coarse material to float

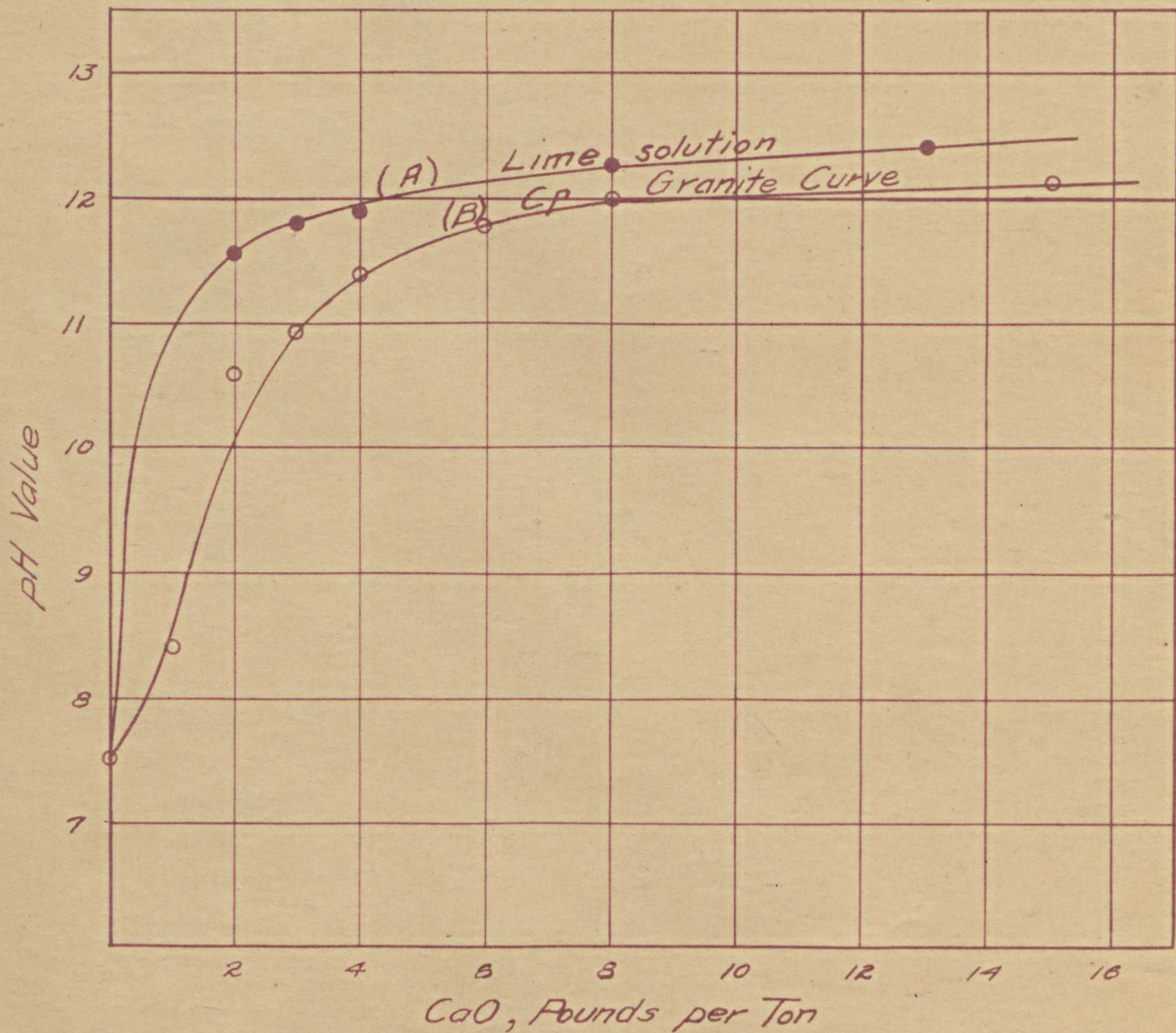


Fig. 3.- Curve B shows the increase in pH with additions of calcium oxide to chalcopyrite - granite mixtures. Curve A was obtained from the dilution of a saturated lime solution in amounts equal to the pulp mixtures of the synthetic ore of curve B.

first was noticeable.

The data presented in table 7 shows that the addition of calcium oxide has no apparent effect on the recovery of chalcopyrite but does effect the grade of concentrate which is perhaps due to the varying degree of effectiveness of the frother with changes in pH value. Table I also shows a recovery of 97.8 per cent copper in a circuit with no calcium oxide added and a recovery of 99.3 per cent copper with the use of 8 lb. / ton of calcium oxide. This is a difference of only 1.5 per cent in the recovery and might be easily accounted for in loss of weight of the samples or the error that is encountered in assaying a tailing having such a small amount of copper as 0.03 to 0.07 per cent.

Table 7

Metallurgical results showing the effect of calcium oxide on the recovery of chalcopyrite.

Test	lb. / ton			wt., grams		per cent		Recovery	pH
	Amyl Xanthate	Terpineol	CaO	Con	Tailing	Cone	Tailing		
1	0.02	0.10	0.00	55.0	445	27.63	0.076	97.8	7.5
2	"	"	1.00	54.0	447	28.83	0.025	99.2	8.4
3	"	"	2.00	53.5	445	28.64	0.051	98.4	10.6
4	"	"	3.00	53.5	447	29.04	0.038	98.7	10.9
5	"	"	4.00	52.5	450	29.14	0.063	98.0	11.4
6	"	"	6.00	52.5	449	29.55	0.036	99.0	11.8
7	"	"	8.00	54.0	443	28.94	0.025	99.3	12.0
8	"	"	16.00	59.5	443	26.22	0.063	98.2	12.1

Experiment 2

These tests were run to determine the effect of potassium cyanide upon the floatability of chalcopyrite. The ball mill



charge was the same as in experiment 1 with the exception of the addition of 6 lb. / ton of calcium oxide to each charge with varying amounts of cyanide as desired.

The Kraut machine was again used in the flotation tests and the conditions maintained the same as in the previous experiment. The same amount of frother and collector were also used.

Fig.3 shows the effect of potassium cyanide on the recovery of chalcopyrite in a high alkaline circuit. The recovery is shown to be decreasing in almost a straight line with additions of potassium cyanide. The mineral that was recovered with the use of 0.60 lb. / ton was very fine and it seems to be true in most all the tests that the finer sizes are the most difficultly depressed.

Table 8

Metallurgical results showing the effect of cyanide on the recovery of chalcopyrite.

Test	KCN	wt., grams		copper, per cent			Selec Indices	pH
		Conc	Tailing	Conc	Tailing	Recovery		
1	C.O	52.5	449	29.55	0.04	99.0	75.3	11.8
2	0.1	55.0	465	26.40	0.17	94.8	33.8	11.9
3	0.4	36.5	447	24.90	1.60	55.8	7.4	11.8
4	0.6	31.0	471	14.80	2.32	29.6	3.4	11.8
5	0.8	27.5	473	9.80	2.78	17.0	2.2	11.8

Fig. 4 shows that the selectivity of the operation is comparable to the recovery. The selective indices as calculated for chalcopyrite to gangue show a decrease from 75.3 to 2.14.

It is possible that the high alkalinity of the circuit has a noticeable effect upon the depression of the chalcopyrite in the presence of cyanide ion. At pH 9 it would perhaps take

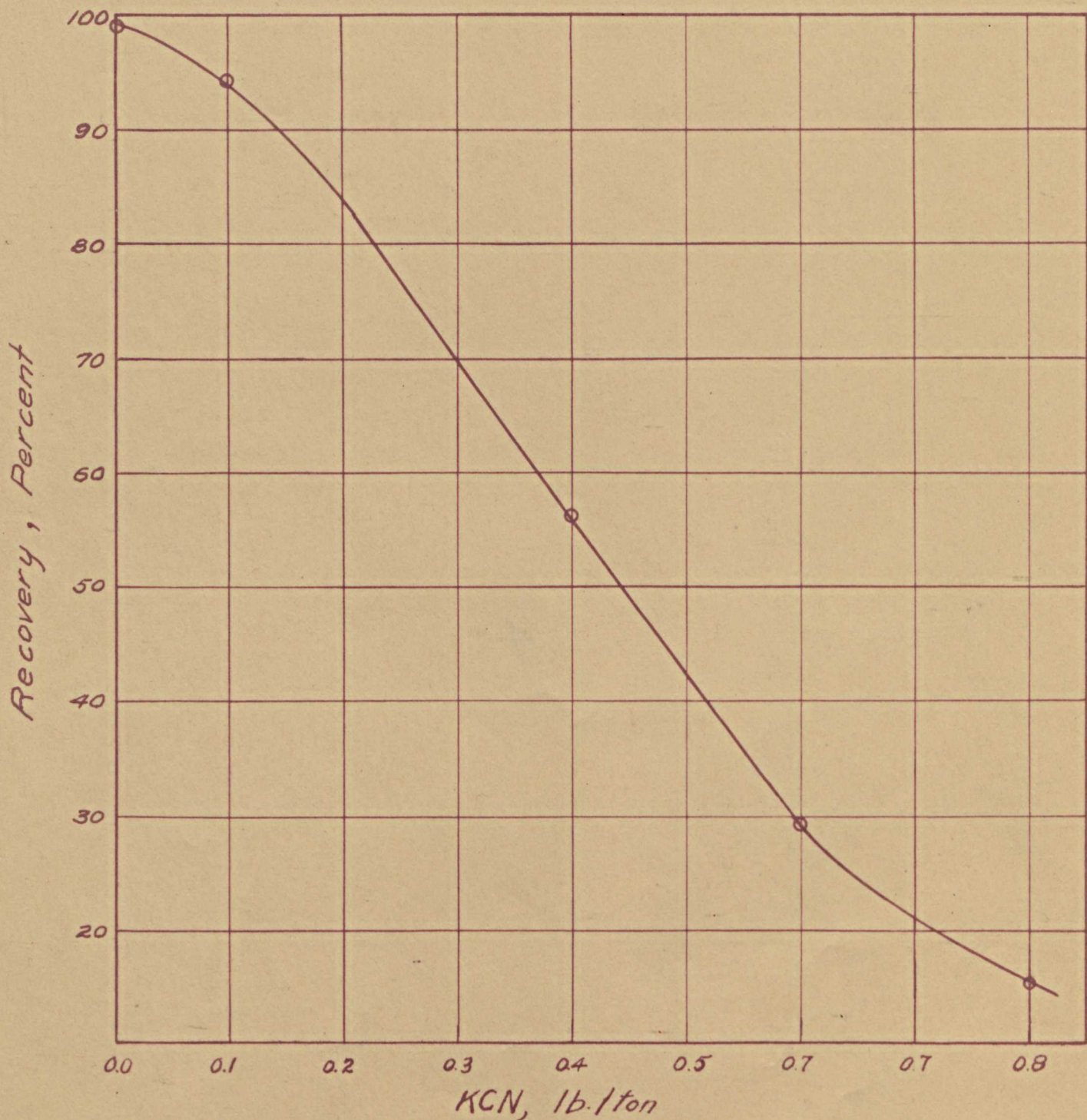


Fig. 4.- This shows the decrease in recovery of chalcopryrite in a high alkaline circuit of pH 11.8. Terpeneol 0.10 lb. / ton and amyl xanthate 0.02 lb. / ton were the reagents added to the cell.

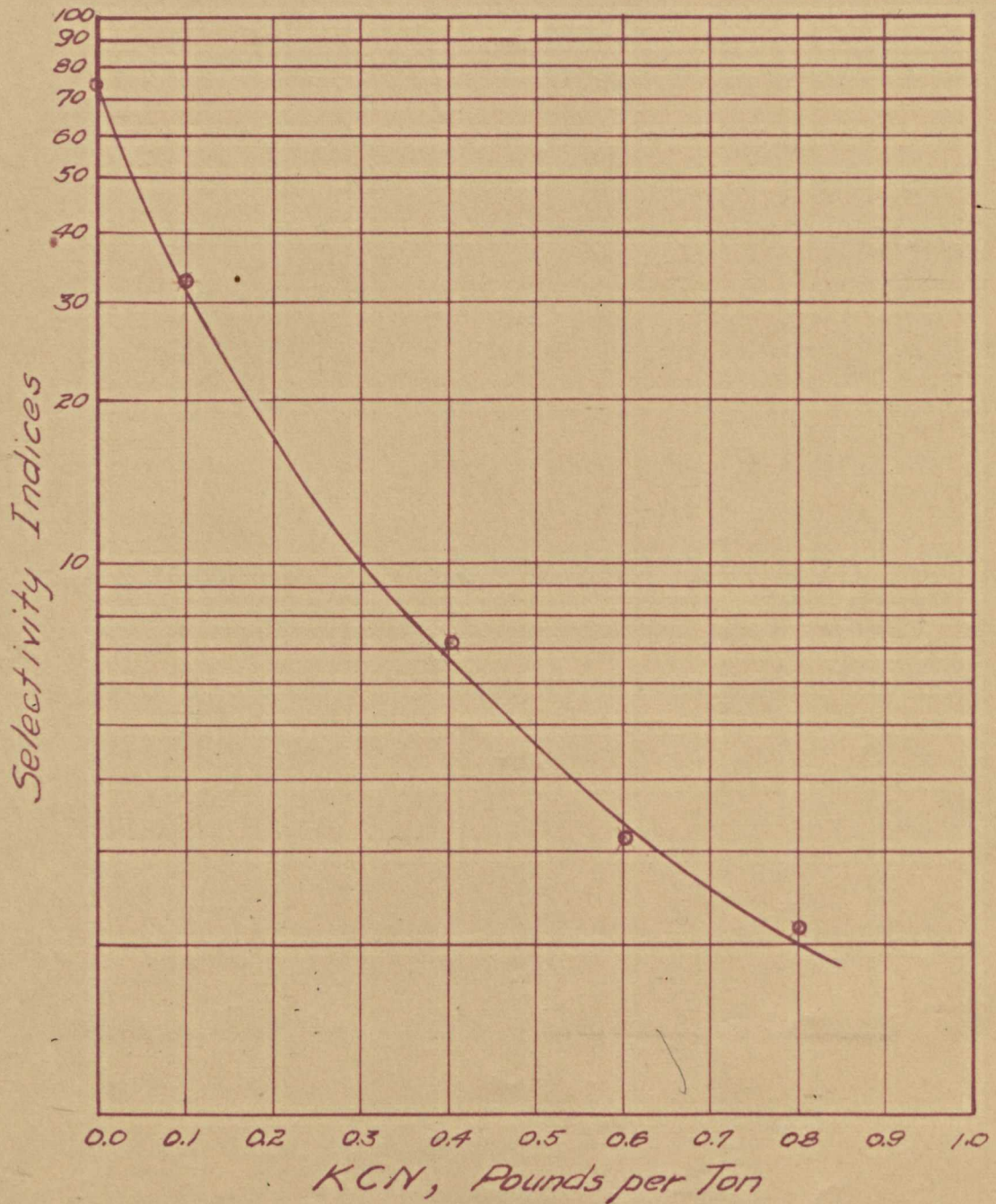


Fig. 5 .- Selectivity of flotation of chalcopyrite with increasing amounts of potassium cyanide.

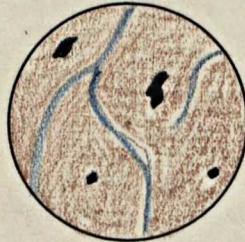
2 or 3 times as much potassium cyanide as at pH 11.8 to depress the chalcopyrite. This is an important factor in the flotation of chalcopyrite where it is necessary to use cyanide to control the flotation of the pyrite present. Close control of both calcium oxide and cyanide appears indicated.

### Bornite

#### Experiment 3.

These tests were run for the purpose of determining the effect of calcium oxide upon the recovery of bornite in a flotation circuit. Due to a limited amount of bornite and the impossibility of obtaining more at the present time it was quite impossible to run similar tests to determine the effect of potassium cyanide.

The bornite used in these tests was quite pure. The following sketch is a microscopic study of a polished surface. The blue indicates the veinlets of covellite, the brown the bornite and the black blebs quartz.



The bornite was ground to 28 / 35 mesh and treated with a 10 per cent potassium cyanide solution, washed thoroughly, and then treated with a 5 per cent hydrochloric acid solution. The mineral was allowed to dry without heating so as to keep it from oxidizing as little as possible.

The ball mill charge consisted of granite 180 grams, bornite 20 grams, water 150 grams, and calcium oxide in the amounts desired. The grinding time for each charge was 30 minutes.

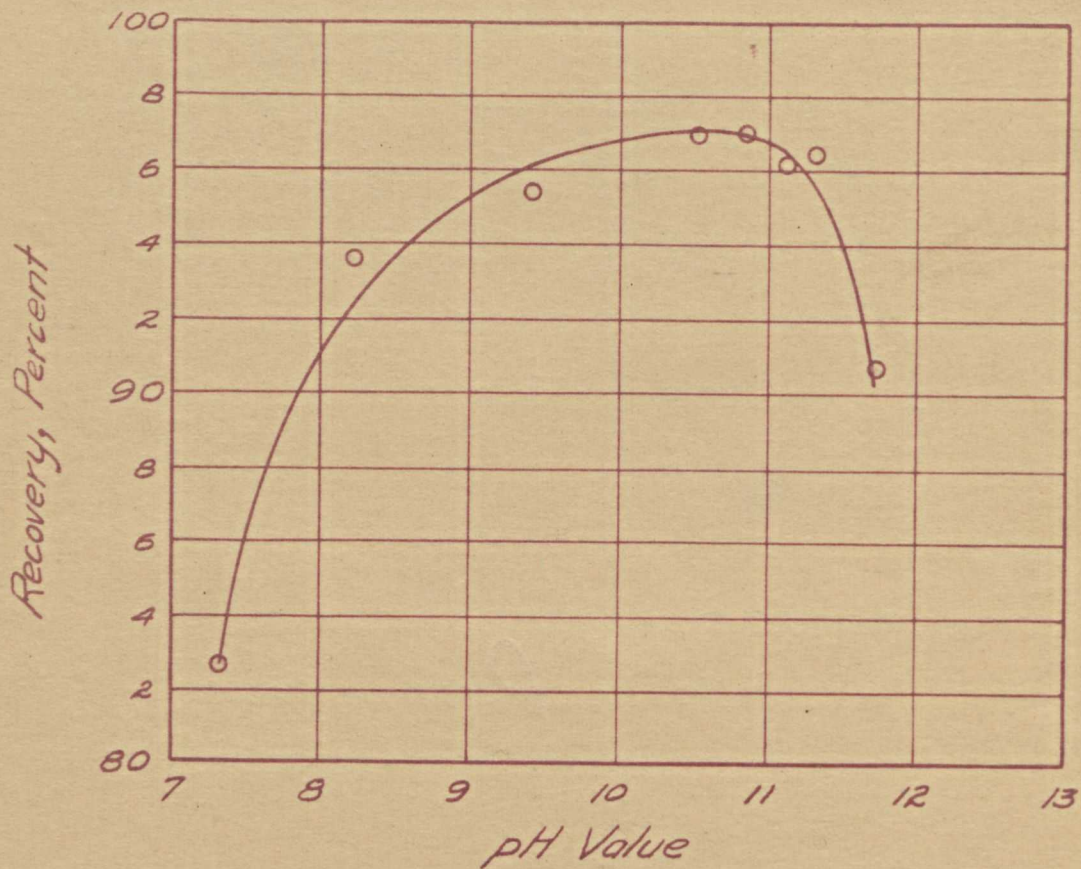


Fig. 6.- Recovery of bornite with varying pH. Amyl xanthate 0.02 lb. / ton, terpineol 0.15 lb. / ton and calcium oxide to give the desired pH were the only reagents added.

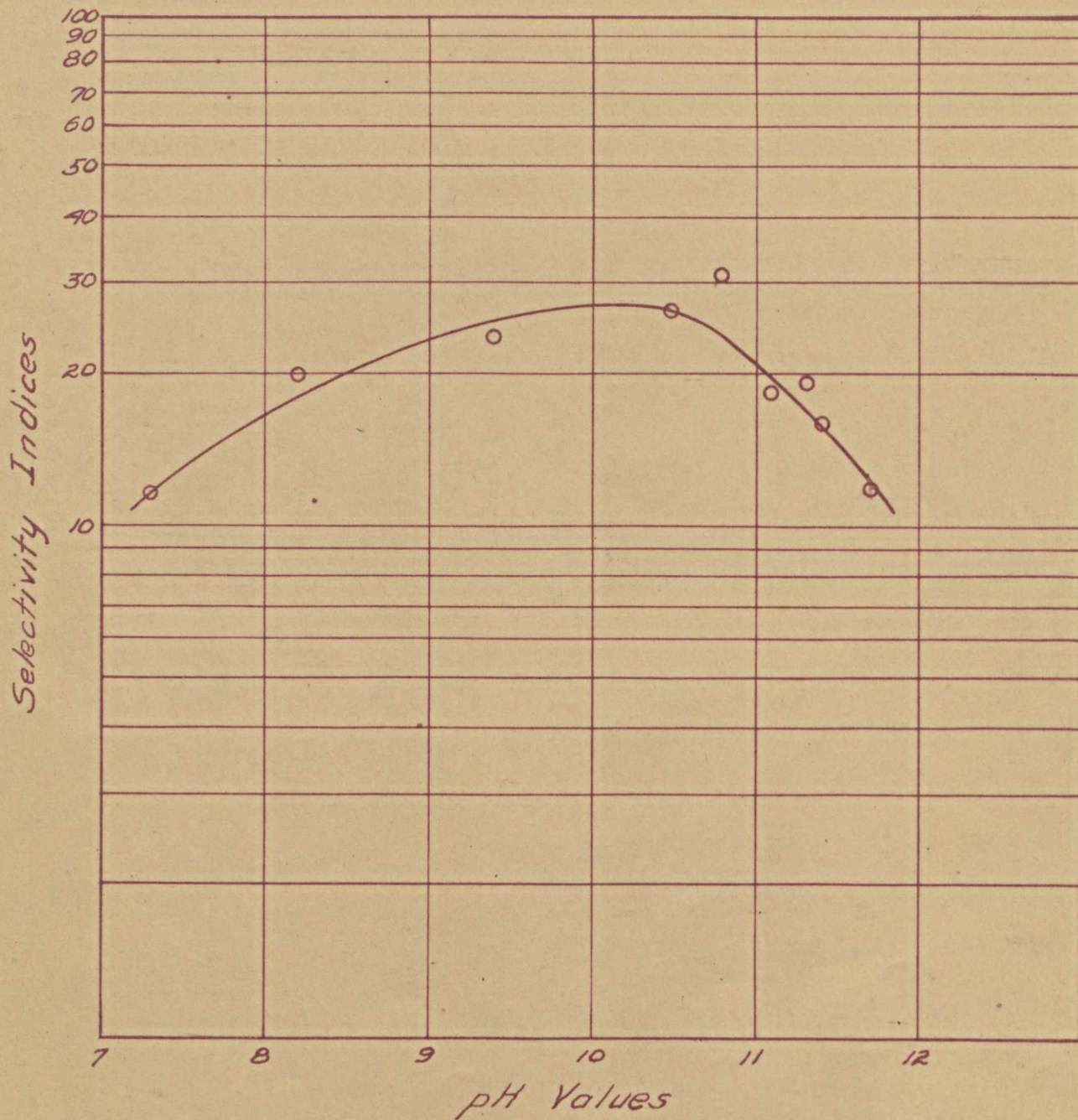


Fig. 7.- Selectivity of bornite to gangue in flotation of synthetic mixtures.

The flotation was conducted in a 500 gram Denver Sub 'A' machine and the time of floating was 6 minutes in all tests. The following reagents were added to the flotation machine; amyl xanthate 0.02 lb. / ton, and terpeneol 0.15 lb. / ton.

Table 9

Metallurgical results obtained showing the effects of calcium oxide on the recovery of bornite.

Test	lb. / ton		Wt., grams		copper, per cent			pH	Selec Indices
	CaO	Conc	Tailing	Conc	Tailing	Recovery			
1	0.0	19.7	181.0	44.2	1.04	82.7	7.3	11.80	
2	1.5	17.8	175.0	40.8	0.28	93.6	8.2	19.95	
3	2.0*	23.0	181.0	44.6	0.27	95.6	9.4	23.50	
4	3.0*	23.5	174.0	43.5	0.19	97.0	10.5	26.90	
5	4.5	22.0	177.0	47.8	0.20	97.0	10.8	31.10	
6	6.0	27.5	173.5	31.7	0.24	96.2	11.1	18.65	
7	4.0*	28.5	172.0	35.8	0.22	96.4	11.3	19.30	
8	8.0	27.5	173.5	34.6	0.28	95.0	11.4	16.35	
9	12.0	27.5	173.5	33.0	0.46	91.8	11.7	12.18	

\* The calcium oxide used in these tests had not been left exposed to the air and consequently were stronger in their effect per unit of weight. The air forms a coating of calcium carbonate on the surface of the calcium oxide which is less soluble in water and of course the effect of the calcium oxide is less per unit of weight.

These tests show that the optimum pH for the flotation of bornite is around 10.5 to 11.0. The recovery drops off very rapidly after this range is reached. This indicates a close control of calcium oxide if the best results are expected. The highest selective index coincides with the optimum recovery.

Enargite  
Experiment 4.

These tests were run with the intention of keeping the conditions as near the same as possible to those in experiment 3 so that a comparison could be made between the two tests.

In these tests the granite was ground to 28 / 48 mesh and the enargite to the same size before charging to the ball mill. The following charge was added to the ball mill; granite 180 grams, enargite 20 grams, water 150 grams and calcium oxide as shown in table 10. The grinding time was 30 minutes.

The reagents were added to the flotation machine as before and the flotation operation conducted in the 500 gram Denver Sub 'A' machine. The time of flotation was again 6 minutes.

It was impossible to keep the frother and collector the same as in the test on bornite. In test 1 shown in table 10, 0.02 lb. / ton of amyl xanthate and 0.15 lb. / ton of terpineol were added. The froth showed conditions of over collecting and 0.10 lb. / ton of terpineol added. This partially removed the effects but was not sufficient to completely remove the over collecting. A thin film of glossy mineral had to be raked away before the froth would rise to any extent.

In tests 2 to 9 inclusive the xanthate was cut to 0.10 lb. / ton and the frother maintained at 0.15 lb. / ton. In tests 2 and 3 the pulp still showed the effects of over collecting and it was not until the flotation had proceeded for over 2 minutes that this condition was removed. The froth in both these tests was rather brittle. In test 4 the froth showed greater stability and the effects of over collecting was apparently removed. In tests 7, 8 and 9 a good froth was maintained throughout but it was a bit voluminous due to the high pH in the last tests.



Table 10

Metallurgical results showing the effect of calcium oxide on the recovery of enargite.

Test	CaO	wt., grams		Copper, per cent			pH	Selec Indices
		Conc	Tailing	Conc	Tailing	Recovery		
1	0.0	18.50	178.0	33.5	0.75	82.05	7.8	11.9
2	0.0	21.00	182.0	27.6	0.87	78.50	7.6	8.5
3	1.0	19.25	180.0	30.3	0.98	77.30	8.4	9.0
4	1.5	18.75	182.5	29.7	1.22	71.40	8.9	7.8
5	2.5	16.00	184.0	33.3	1.27	69.40	10.1	9.2
6	3.0	13.75	189.0	34.7	1.46	63.30	10.3	9.2
7	6.0	14.25	188.0	27.1	1.96	51.20	11.1	5.5
8	9.0	10.50	190.5	22.1	2.59	32.10	11.6	3.9
9	12.0	10.50	193.0	13.8	3.06	19.65	11.7	2.5

Table 10 shows the recoveries to decrease with increasing amounts of calcium oxide and increasing pH. Fig. 8 shows the recovery to decrease very rapidly between the range of pH 11 and 12 and reaches a minimum of 19.65 per cent in the last test.

The recovery of 78.5 per cent in test 2 is not good and unlike bornite the recovery does not reach a maximum for a given pH and then decrease, but drops off from the beginning. The poor maximum recovery is likely due to the excessive sliming of the enargite mineral. This sliming is quite noticeable in the settling of the tailings. The water in all cases is black from the finely divided enargite and it was necessary, even with 12 lb. / ton of calcium oxide, to add still more flocculating agent to facilitate settling.

The selectivity of the operation does not materially decrease until a pH of 10.3 is reached. It appears that the optimum pH for floating enargite lies somewhere below 7.6.

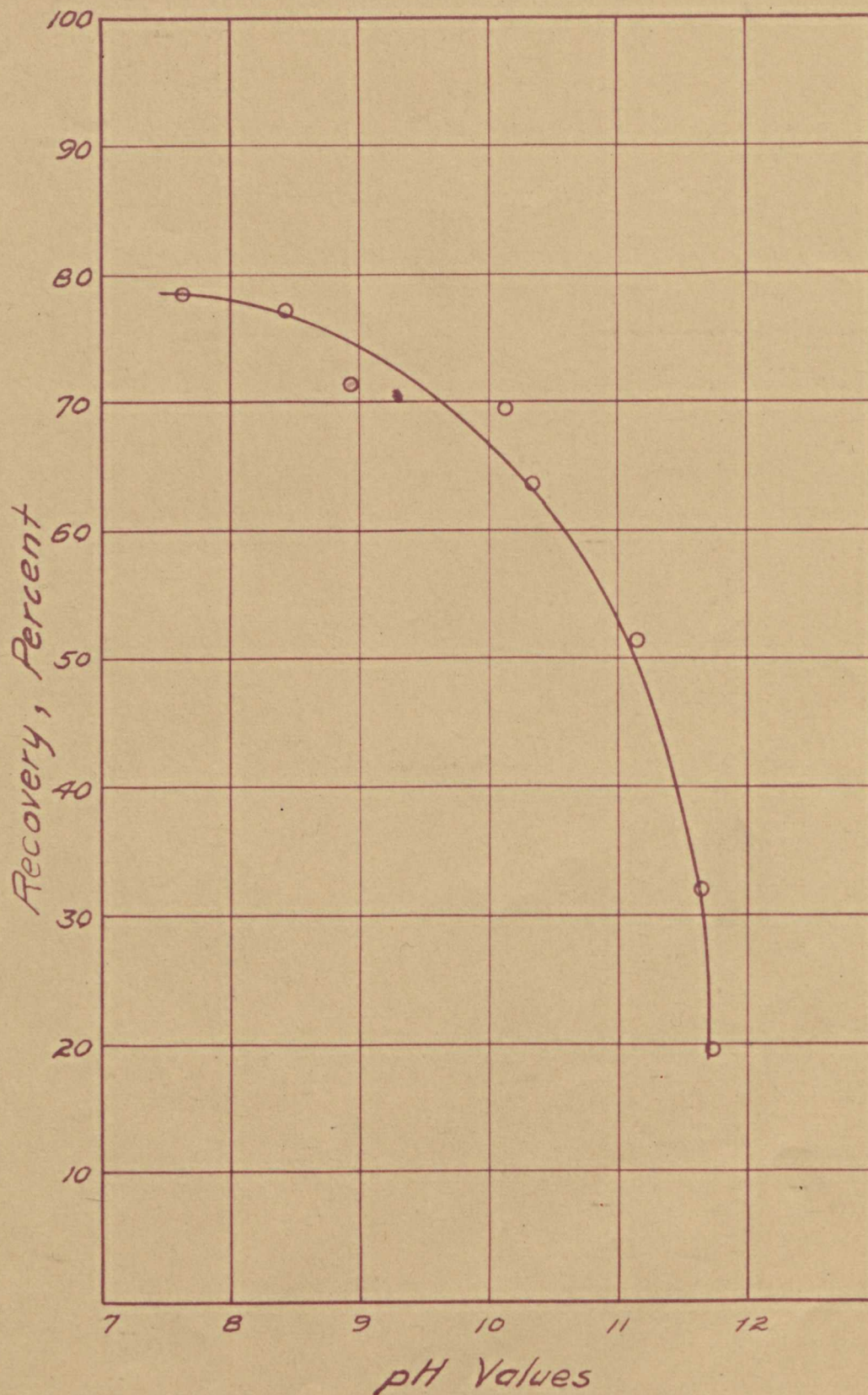


Fig. 8-- The recovery of enargite with varying pH. Amyl xanthate 0.01 lb. / ton, terpineol 0.15 lb. / ton and calcium oxide to suit were the reagents used.

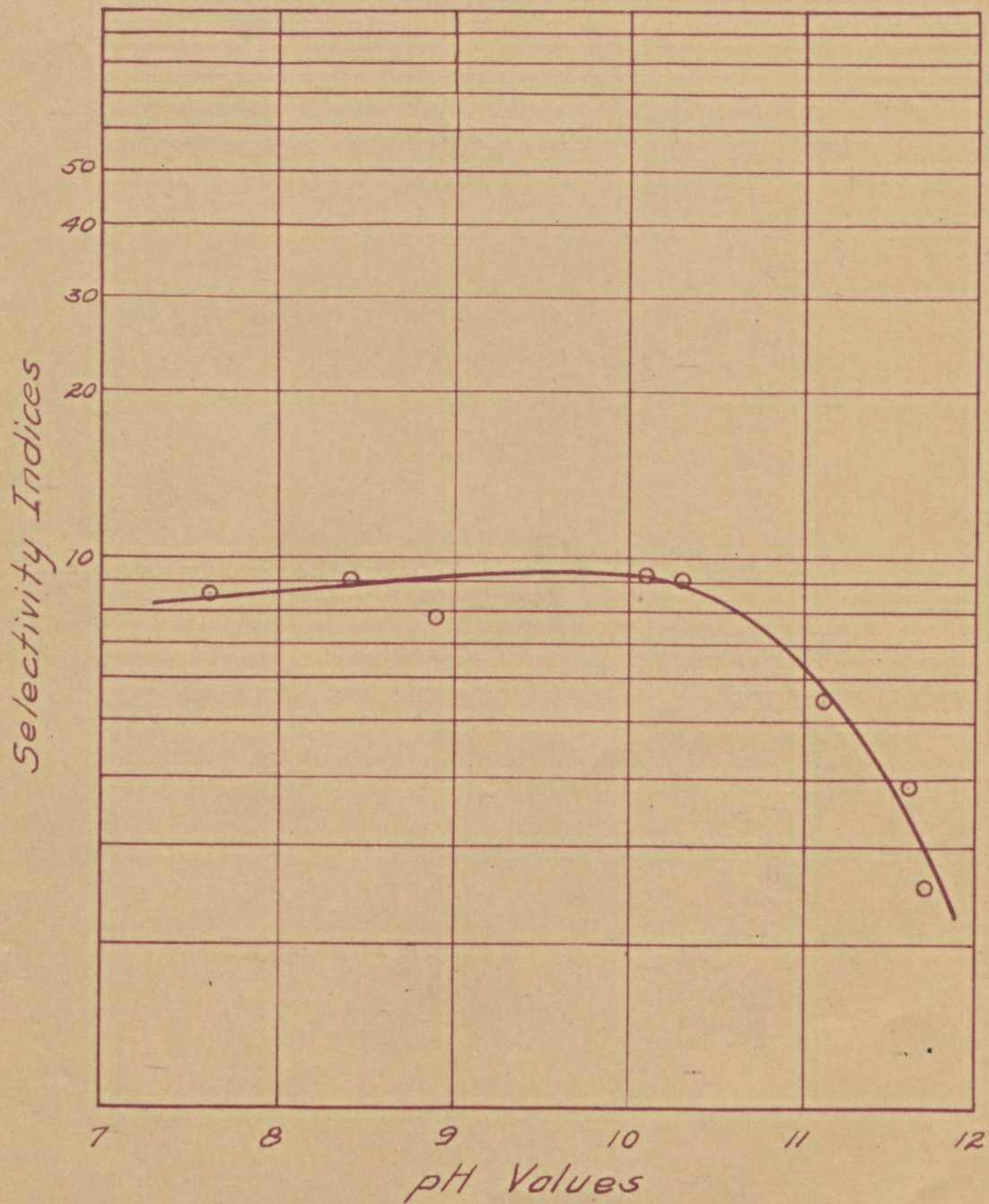


Fig. 9.- Selectivity of enargite to gangue in flotation of synthetic mixtures.

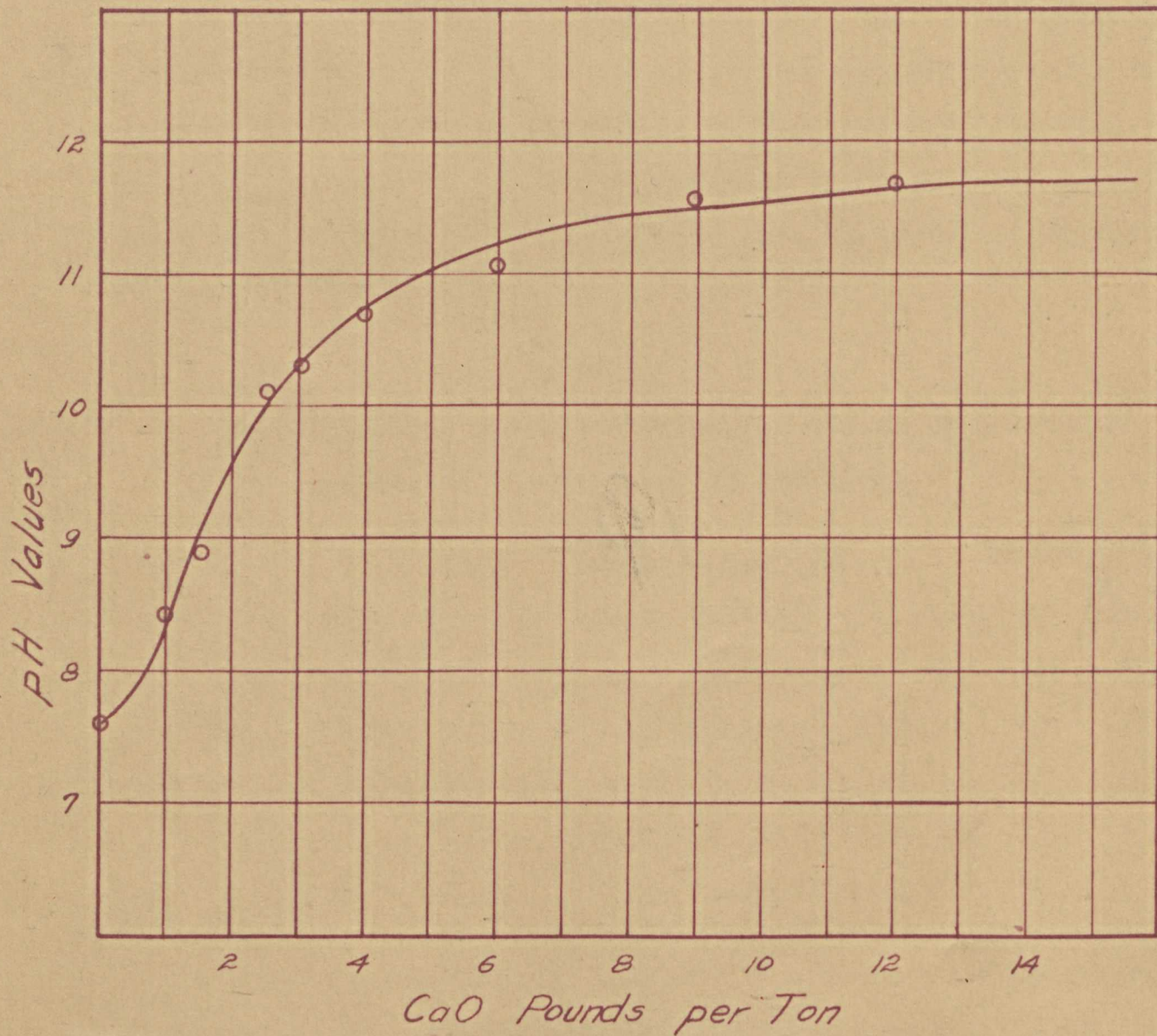


Fig. 10.- The change in pH was obtained with varying amounts of calcium oxide added to the synthetic mixtures of enargite and granite.

This would not seem to be such good practice as the conditions for maintaining a good froth at that pH are very poor.

#### Experiment 5

The following tests were conducted to determine the effect of potassium cyanide on the recovery of enargite. Again the conditions during the flotation operation were maintained the same as in the tests of experiment 4. The charge to the ball mill was the same as the above mentioned tests with the exception of adding 1.5 lb. / ton of calcium oxide and amounts of potassium cyanide as shown in table 11.

Table 11

Metallurgical results showing the effect of potassium cyanide on the recovery of enargite.

Test	KCN	Wt., grams		Copper, per cent			pH	Selec Indices
		Cone	Tailing	Cone	Tailing	Recovery		
1	0.0	18.75	182.5	29.7	1.22	71.4	8.9	7.8
2	0.1	16.50	181.0	32.1	1.31	71.8	8.8	9.0
3	0.3	13.50	189.5	27.8	1.78	53.5	8.7	5.9
4	0.5	10.00	190.0	28.4	2.34	38.9	10.1	5.3
5	0.6	10.50	190.0	28.4	2.27	40.8	10.0	5.4

The same machine for floating was used as in the previous test and the reagents added to the machine were as follows; amyl xanthate 0.01 lb. / ton, and terpeneol 0.225 lb. / ton.

It was impossible to keep a froth with 0.15 lb. / ton of terpeneol due partially to the low alkalinity of the pulp. Even with this additional amount of frother a good froth was not obtained until enough cyanide was added to give a pH value of 10.0. The case of over collecting is even more noticeable than in experiment 4.

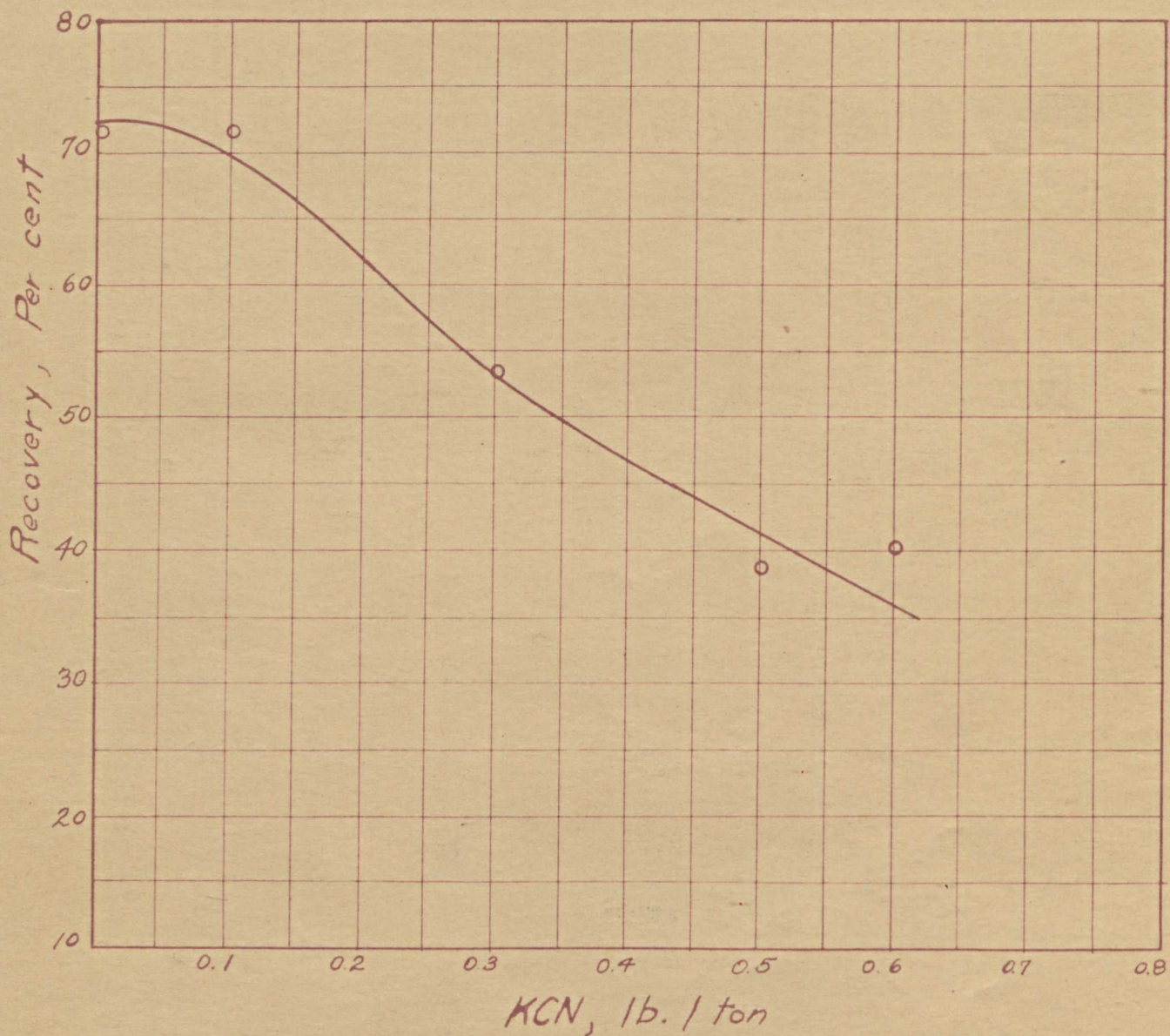


Fig. 11.- The recovery of enargite with varying amounts of potassium cyanide. Amyl xanthate 0.01 lb. / ton and terpineol 0.225 lb. / ton were the other reagents added.

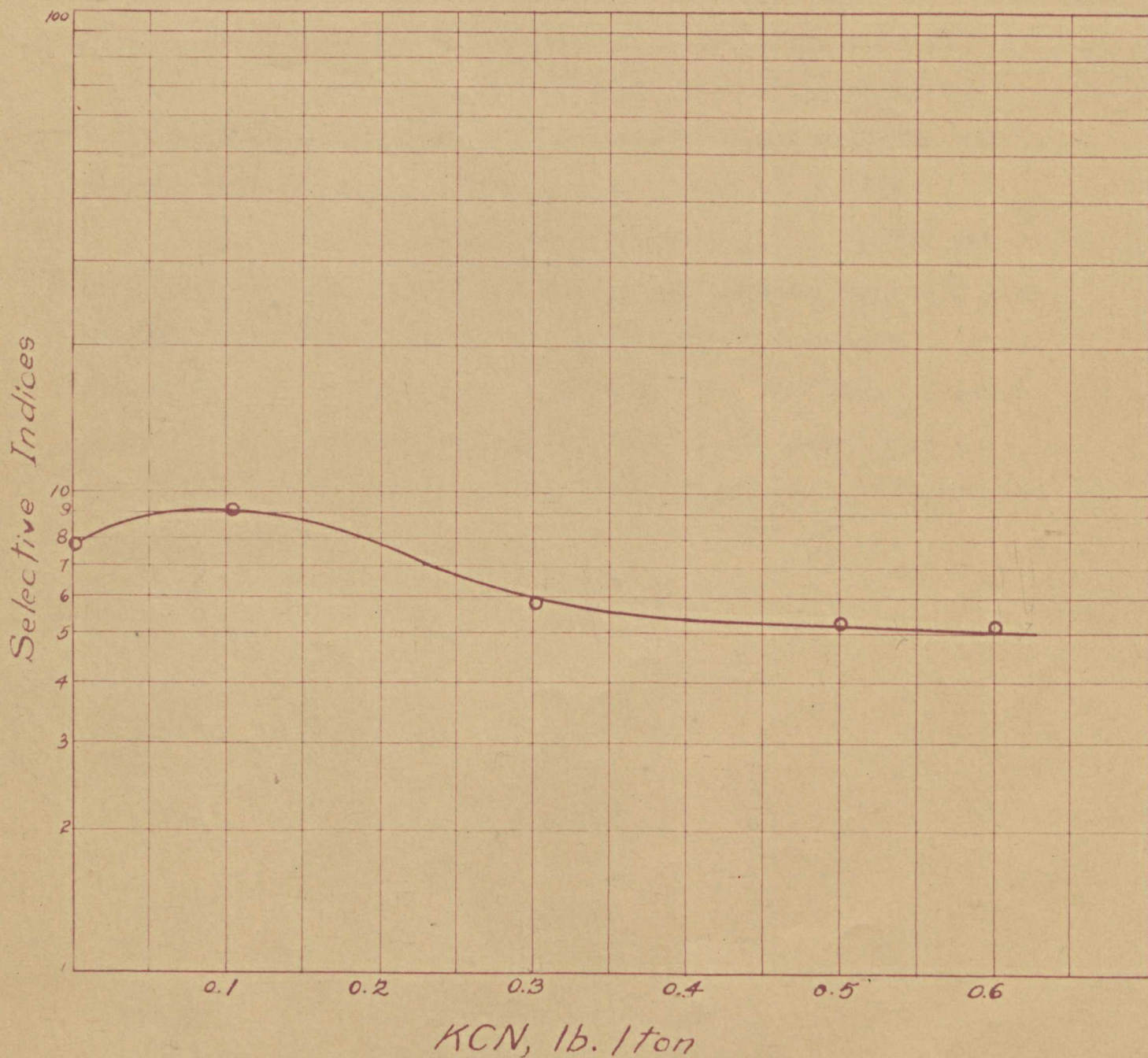


Fig. 12.- Selectivity of enargite to gangue in flotation of synthetic mixtures with varying amounts of potassium cyanide.

A possible explanation of the cause of over collecting with the enargite is offered. The surface of the enargite is no doubt oxidized. Upon the addition of small amounts of cyanide only the oxide coating is attacked, but upon adding increasing amounts of cyanide the oxide coating is dissolved off and the sulfide surface of the mineral is free to be attacked by the cyanide. This then allows the xanthate to react with the sulfide surface rather than the oxide surface of the enargite.

The problem of keeping conditions uniform in this set of tests was quite difficult and was perhaps not successfully done. The froth was very poor in the tests using low amounts of cyanide which made it difficult to rake off the thin film of mineral uniformly without recovering some of the gangue. As the cyanide amount was increased the froth became better making the recovery better.