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REPORT OF INVESTIGATIONS

LEACHING OXIDIZED COPPER ORES:
EFFECT OF STRENGTH OF ACID IN LEACHING SOLVENT



BY

JOHN D. SULLIVAN AND G. L. OLDRIGHT

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OF ACID IN LEACHING SOLVENT¹

By John D. Sullivan² and G. L. Oldright³

INTRODUCTION

In leaching copper ores knowledge of the rate of dissolution of the copper minerals and of the impurities is desirable. With knowledge of these two factors and data as to what minerals are present in the ore the metallurgist has an excellent start on planning a leaching cycle to fit that ore. The rate of dissolution is rather difficult to predict, and the time required to leach a given ore may differ markedly from that required by another ore. The whole study as related to mixed oxidized and sulphide ores is one of relative rates of dissolution of the different copper minerals and the impurities in sulphuric acid and in ferric sulphate. The aim of leaching is to choose a cycle and strength of solvents to extract a maximum amount of copper and a minimum of impurities. The consumption of reagents per unit of copper dissolved is a function of the strength of the reagents used and the relative rates of attack on copper minerals and impurities. An opportunity may exist of obtaining the kind of electrolyte wanted for electrolytic precipitation and yet of saving on the consumption of reagents by modifying the strength of ferric sulphate and of sulphuric acid during the cycle to fit the time needed for dissolving the sulphide and oxidized minerals present.

Former papers⁴ have considered the rate of dissolution of azurite, malachite, tenorite, chrysocolla, cuprite, chalcocite, bornite, and covellite in common leaching solvents. A paper soon to be written will consider some of the less important copper minerals. The present paper considers the rate of dissolution of copper from ores of the Southwest in sulphuric acid of varying strength and the ratio of sulphuric acid consumed to copper dissolved. In this study two ores containing the copper principally in an oxidized condition were used. A study of other ores, especially those containing high percentages of sulphide copper minerals, would be desirable, but since consumption of acid has paramount interest,

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4 Sullivan, J. D., Dissolution of Various Oxidized Copper Minerals: Rept. of Investigations 2934, Bureau of Mines, 1929, 9 pp.

Sullivan, J. D., and Oldright, G. L., The Dissolution of Cuprite in Sulphuric Acid and in Ferric Sulphate Solutions: Rept. of Investigations 2967, Bureau of Mines, 1930, 9 pp.

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Sullivan, John D., Chemistry of Leaching Bornite: Tech. Paper 486, Bureau of Mines, 1931, 20 pp.

Sullivan, John D., Chemistry of Leaching Covellite: Tech. Paper 487, Bureau of Mines, 1930, 18 pp.

especially in leaching ores of the Southwest, the first experiments were limited to oxidized ores and to sulphuric acid as the leaching solvent. Continuation of this work was interrupted owing to exigencies and lack of personnel. The data collected were thought sufficient, however, to justify this publication.

ACKNOWLEDGMENTS

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ORE USED

The Ajo ore was typical of that used in leaching operations at that place. The Silverbell Imperial ore was a coarse-grained, highly silicified and sericitized porphyry. The screen analyses are given in Table 1 and the chemical analyses in Table 2.

Table 1.- Screen analyses of ores

Size	Per cent	
	Ajo ore	Silverbell Imperial ore
- $\frac{1}{4}$ inch +10-mesh	59.4	65.8
-10 +28 mesh	19.9	19.9
-28 +100 mesh	10.5	8.3
-100 +200 mesh	3.6	2.0
-200-mesh	6.6	4.0

Table 2.- Chemical analyses of ores

Material	Per cent	
	Ajo ore	Silverbell Imperial ore
Copper		
Total	1.17	1.05
Acid-soluble ¹985	.88
Alumina		
Total	14.90	11.84
Acid-soluble ²67	1.60
Iron	2.61	4.85
Sulphur15	.17
CaO70	1.65
MgO	- -	3.33
Manganese24	- -

- 1 Copper soluble in 24 hours at room temperature in a solution containing 5 per cent, by weight, of sulphuric acid and saturated with sulphur dioxide.
- 2 Alumina soluble in 24 hours at room temperature in a solution containing 10 per cent, by weight, of sulphuric acid.

Table 4.- Leaching manipulations

Time, hours	Ajo ore										Silverbell Imperial ore				
	Test 1	Test 2	Test 3	Test 4	Test 5	Test 6	Test 7	Test 8	Test 9	Test 10	Test 11	Test 12	Test 13	Test 14	Test 15
0 (start of test)	Added 2,000 c.c. ¹ solution containing 10.32 grams H ₂ SO ₄ per liter	Added 2,000 c.c. ¹ solution containing 26.96 grams H ₂ SO ₄ per liter	Added 2,000 c.c. ¹ solution containing 50.35 grams H ₂ SO ₄ per liter	Added 2,000 c.c. ¹ solution containing 104.70 grams H ₂ SO ₄ per liter	Added 2,000 c.c. ¹ solution containing 105.80 grams H ₂ SO ₄ per liter	Added 2,000 c.c. ¹ solution containing 52.40 grams H ₂ SO ₄ per liter	Added 2,000 c.c. ¹ solution containing 10.44 grams H ₂ SO ₄ per liter	Added 2,000 c.c. ¹ solution containing 105.80 grams H ₂ SO ₄ per liter	Added 2,000 c.c. ¹ solution containing 52.40 grams H ₂ SO ₄ per liter	Added 2,000 c.c. ² solution containing 10.48 grams H ₂ SO ₄ per liter	Added 2,000 c.c. ² solution containing 25.42 grams H ₂ SO ₄ per liter	Added 2,250 c.c. ¹ solution containing 9.96 grams H ₂ SO ₄ per liter	Added 2,250 c.c. ¹ solution containing 24.40 grams H ₂ SO ₄ per liter	Added 2,250 c.c. ¹ solution containing 50.35 grams H ₂ SO ₄ per liter	Added 2,250 c.c. ¹ solution containing 104.70 grams H ₂ SO ₄ per liter
4	Drained and added 1,000 c.c. solution containing 10.32 grams H ₂ SO ₄ per liter	-----	-----	-----	-----	-----	Drained and added 1,000 c.c. solution containing 10.44 grams H ₂ SO ₄ per liter	-----	-----	-----	-----	-----	-----	-----	-----
8	Do.	Drained and added 1,000 c.c. solution containing 26.96 grams H ₂ SO ₄ per liter	-----	-----	-----	-----	do.	-----	-----	-----	-----	do.	Drained and added 1,500 c.c. solution containing 24.40 grams H ₂ SO ₄ per liter	-----	-----
12	Do.	-----	-----	-----	-----	-----	do.	-----	-----	do., start of test	-----	do.	-----	-----	-----
24	Do.	do., 8 hours	Drained and added 1,000 c.c. solution containing 50.35 grams H ₂ SO ₄ per liter	Drained and added 1,000 c.c. solution containing 104.70 grams H ₂ SO ₄ per liter	Drained and added 1,000 c.c. solution containing 105.80 grams H ₂ SO ₄ per liter	Drained and added 1,000 c.c. solution containing 52.40 grams H ₂ SO ₄ per liter	do.	-----	-----	-----	-----	do.	do., 8 hours	Drained and added 1,500 c.c. solution containing 50.35 grams H ₂ SO ₄ per liter	Drained and added 1,500 c.c. solution containing 104.70 grams H ₂ SO ₄ per liter
28	Do.	-----	-----	-----	-----	-----	do.	-----	-----	do., 12 hours	do., start of test	do.	-----	-----	-----
32	Do.	-----	-----	-----	-----	-----	do.	-----	-----	do., 12 hours	-----	do.	-----	-----	-----
36	Do.	do., 8 hours	-----	-----	-----	-----	do.	-----	-----	-----	-----	do.	-----	-----	-----
48	Do.	do.	do., 24 hours	do., 24 hours	Drained and added 1,000 c.c. solution containing 10.44 grams H ₂ SO ₄ per liter	do., 24 hours	Drained and added 1,000 c.c. solution containing 25.42 grams H ₂ SO ₄ per liter	-----	Drained and added 1,000 c.c. solution containing 52.40 grams H ₂ SO ₄ per liter	Added 2,000 c.c. same solution as above ⁴	Added 1,000 c.c. same solution as above ⁵	do.	do., 8 hours	do., 24 hours	Do., 24 hours
52	Do.	-----	-----	-----	-----	-----	-----	-----	-----	-----	-----	do.	-----	-----	-----
56	Do.	do., 8 hours	-----	-----	-----	-----	-----	-----	-----	-----	-----	-----	-----	-----	-----
60	-----	-----	-----	-----	do., 48 hours	-----	do., 48 hours	-----	-----	-----	-----	Drained and added 1,500 c.c. solution containing 10.08 grams H ₂ SO ₄ per liter	do., 8 hours	-----	-----
72	Do., 4 hours	do., 8 hours	do., 24 hours	do., 24 hours	do.	Drained and added 1,000 c.c. solution containing 10.22 grams H ₂ SO ₄ per liter	do.	Drained for 1/2 hour; returned effluent solution to ore	-----	do., 48 hours	do., 48 hours	do.	do.	do., 24 hours	-----
84	Do.	-----	-----	-----	do.	do.	-----	-----	-----	-----	-----	do.	-----	-----	-----
96	Do.	do., 8 hours	do., 24 hours	do., 24 hours	do.	do.	Drained and added 1,000 c.c. solution containing 52.40 grams H ₂ SO ₄ per liter	-----	-----	do., 48 hours	do., 48 hours	do.	do., 8 hours	Drained and added 1,500 c.c. solution containing 50.80 grams H ₂ SO ₄ per liter	Do., 24 hours
104	Do.	-----	-----	-----	-----	-----	-----	-----	-----	-----	-----	do.	-----	-----	-----
108	-----	-----	-----	-----	-----	do., 72 hours	-----	-----	-----	-----	-----	-----	-----	-----	-----
120	Do., 4 hours	-----	-----	-----	do., 48 hours	do.	-----	-----	do., 48 hours	do., 48 hours	do., 48 hours	do., 60 hours	do., 6 hours	do., 96 hours	-----
128	Drained and added 1,000 c.c. solution containing 10.58 grams H ₂ SO ₄ per liter	-----	-----	-----	-----	-----	-----	-----	-----	-----	-----	do.	-----	-----	-----
144	Do.	do., 8 hours	do., 24 hours	do., 24 hours	do., 48 hours	do., 72 hours	do., 96 hours	Drained and washed with 9 applications of tap water of 1,000 c.c. each; added in 3 days	-----	Added 2,000 c.c. solution containing 10.00 grams H ₂ SO ₄ per liter ⁴	do., 48 hours	do.	do., 8 hours	do., 96 hours	-----
168	Do.	do.	-----	-----	do.	do.	-----	-----	-----	-----	-----	do.	do.	do.	Do., 24 hours
192	Do.	do.	-----	-----	Drained and washed with 9 applications of tap water of 1,000 c.c. each; added in 3 days	do.	-----	-----	-----	Drained and washed with 9 applications of tap water of 1,000 c.c. each; added in 3 days	-----	do.	-----	-----	-----
216	Do.	-----	-----	-----	-----	Drained and washed with 9 applications of tap water of 1,000 c.c. each; added in 3 days	-----	-----	-----	do.	do., 48 hours	do.	do.	do., 96 hours	Do., 24 hours
240	Drained and washed with 9 applications of tap water of 1,000 c.c. each; added in 3 days	Drained and washed with 9 applications of tap water of 1,000 c.c. each; added in 3 days	Drained and washed with 9 applications of tap water of 1,000 c.c. each; added in 3 days	Drained and washed with 9 applications of tap water of 1,000 c.c. each; added in 3 days	-----	-----	Drained and washed with 9 applications of tap water of 1,000 c.c. each; added in 3 days	-----	-----	Commenced washing; 9 liters of tap water trickled onto ore in 2 days	Commenced washing; 10 liters of tap water trickled onto ore in 2 days	-----	-----	-----	-----
264	-----	-----	-----	-----	-----	-----	-----	-----	-----	-----	-----	do., 60 hours	do.	do., 96 hours	-----
288	-----	-----	-----	-----	-----	-----	-----	-----	-----	-----	-----	do.	do.	do.	Do., 24 hours
336	-----	-----	-----	-----	-----	-----	-----	-----	-----	-----	-----	Drained and washed with 9 applications of tap water of 1,500 c.c. each; added in 3 days	Drained and washed with 9 applications of tap water of 1,500 c.c. each; added in 3 days	Drained and washed with 9 applications of tap water of 1,500 c.c. each; added in 3 days	Drained and washed with 9 applications of tap water of 1,500 c.c. each; added in 3 days

¹ Includes 300 c.c. used to wet the ore.² Added to container from which it was trickled onto the ore.³ Trickled onto ore at rate of approximately 4 liters per 24 hours.⁴ Trickled onto ore at rate of approximately 2 liters per 24 hours.⁵ Trickled onto ore at rate of approximately 1 liter per 24 hours.⁶ Trickled onto ore at rate of approximately 1/2 liter per 24 hours.

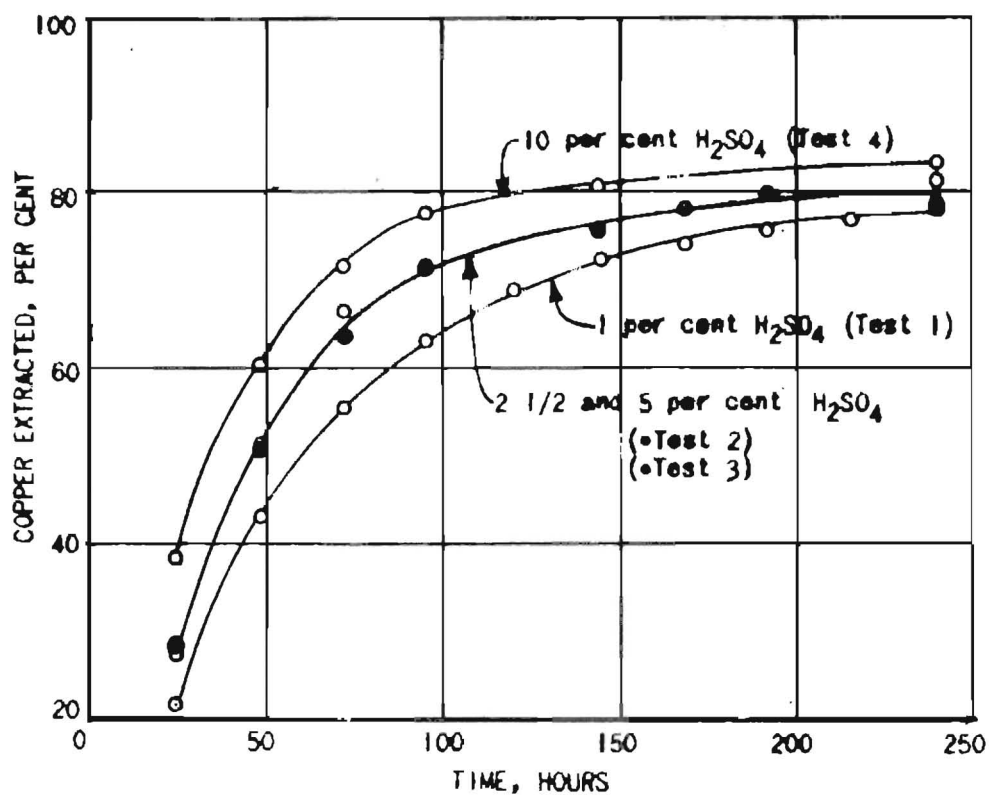


Figure 1a.—Rate of extraction of copper from Ajo ore

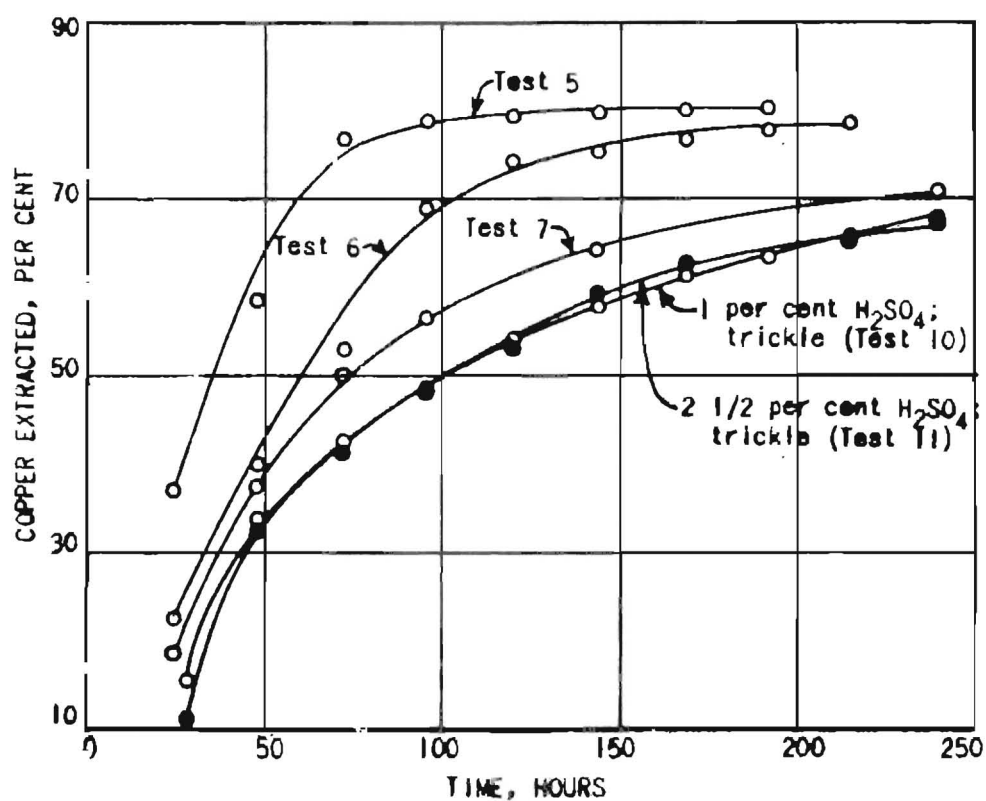


Figure 1b.—Rate of extraction of copper from Ajo ore

EXPERIMENTAL WORK

In all tests 11 pounds of ore was leached in a circular lead vat 7 inches in diameter. In flood-percolation tests the vats were 8 inches deep; in trickle open-drainage tests, 42 inches deep. A lead pipe burned into the bottom of each vat was used for draining the solutions from the ore. Cocoa-fiber matting was used as a filter bottom. The usual procedure consisted of adding 300 c.c. of leaching solution to 11 pounds of dry ore, and mixing the mass thoroughly to insure complete wetting. The wet ore was charged into the leaching vat. When flood-percolation leaching was used the procedure was as follows: Solution was added by downward percolation to flood the ore about one-half inch. After a definite time solution was drained from the ore through the exit pipe at the bottom of the vat; draining was continued one-half hour. The time the leaching solvent was allowed to remain on the ore before draining and replenishing with fresh solution varied with the strength of acid in the solvent. The volume of fresh leaching solution added during each cycle was constant and was approximately the same volume as that of the effluent solution coming from the ore. At the end of the leaching period the residue was washed with tap water. The washing was also a flooding operation.

When open-drainage leaching was employed the procedure was as follows: Solution was trickled onto the ore from an overhead container. It gravitated through the ore and was collected in a jar. After leaching the residue was washed with tap water by the trickle principle.

Eleven tests were made on Ajo ore and four on Silverbell Imperial ore. All tests were made at room temperature, but the temperature varied with different tests. Temperature data are given in Table 3.

Table 3.- Temperature data

	Ajo ore												Silverbell Imperial ore		
	Tests 1, 2, 3, 4			Tests 5, 6, 7, 8			Test 9			Tests 10, 11			Tests 12, 13, 14, 15		
	8 a.m.	12 m	4 p.m.	8 a.m.	12 m	4 p.m.	8 a.m.	12 m	4 p.m.	8 a.m.	12 m	4 p.m.	8 a.m.	12 m	4 p.m.
Mean temp. air °C	24	26	27	21	22½	23½	21½	23	24	21½	23	24½	23	25½	26
Maximum temp. do... °C	26	27	28	23	24	24	23	24	25	23	24	25	24	26	27
Minimum temp. do... °C	20½	23	25	18	21	21½	17	20	23½	17	20	23½	22	23	24
Mean temp. solution °C	24	25	26	21	21½	22	21	21½	23	20	21	22½	23½	24	25
Maximum temp. do. °C	27	27	26½	23	23½	23½	22	22	23½	21½	22½	23½	24	25	25½
Minimum temp. do...°C	20½	22	25	17	19	19½	17	19	22½	17	19	21	22½	23	23½

The leaching manipulations of all tests are given in Table 4.

Table 5 summarizes the results of the 11 tests of Ajo ore. In Figures 1A and 1B the percentage of copper extracted is plotted against time. The percentage of copper extracted is based upon the copper in the effluent solutions and does not account for the dissolved copper remaining in the ore and not removed by washing. Especially during the early

part of the test the actual percentage of total copper rendered soluble may be considerably higher than that dissolved and recovered. In Figure 2 the ratio by weight of sulphuric acid consumed per gram of copper extracted is plotted against the percentage of the total copper extracted. The calculations were made as follows: The analyses of the effluent solutions gave the amount of sulphuric acid and copper that came off the ore during each draining. The concentration of acid in the original leaching solution was known. The difference between the weight of sulphuric acid in the effluent solution and in a like volume of heads solution was taken as the acid consumed. From this difference and the weight of copper in the effluent solution the ratio of acid consumed to copper dissolved was calculated. All ratios are based upon cumulative weights. These ratios are not strictly correct, because the assumption is made that the entire solution on the ore before draining was homogeneous and that the effluent solution drained from the ore had the same concentration of copper and of sulphuric acid as the solution remaining on the ore after draining. However, all tests were calculated in the same manner, so results are comparable. Such calculations were not made in tests 5, 6, and 7, since the concentration of acid in the solution added to the ore varied at different times.

The true ratio of acid consumed to copper dissolved during the entire test should be taken from Table 5, which gives the over-all consumption, including the wash solutions.

In all figures, tables, and the discussion that follows solutions containing approximately 10, 25, 50, and 100 grams of sulphuric acid per liter have been designated as 1, 2½, 5, and 10 per cent sulphuric acid, respectively.

Data corresponding to those for Ajo ore are given for Silverbell Imperial ore in Table 6 and in Figures 3 and 4.

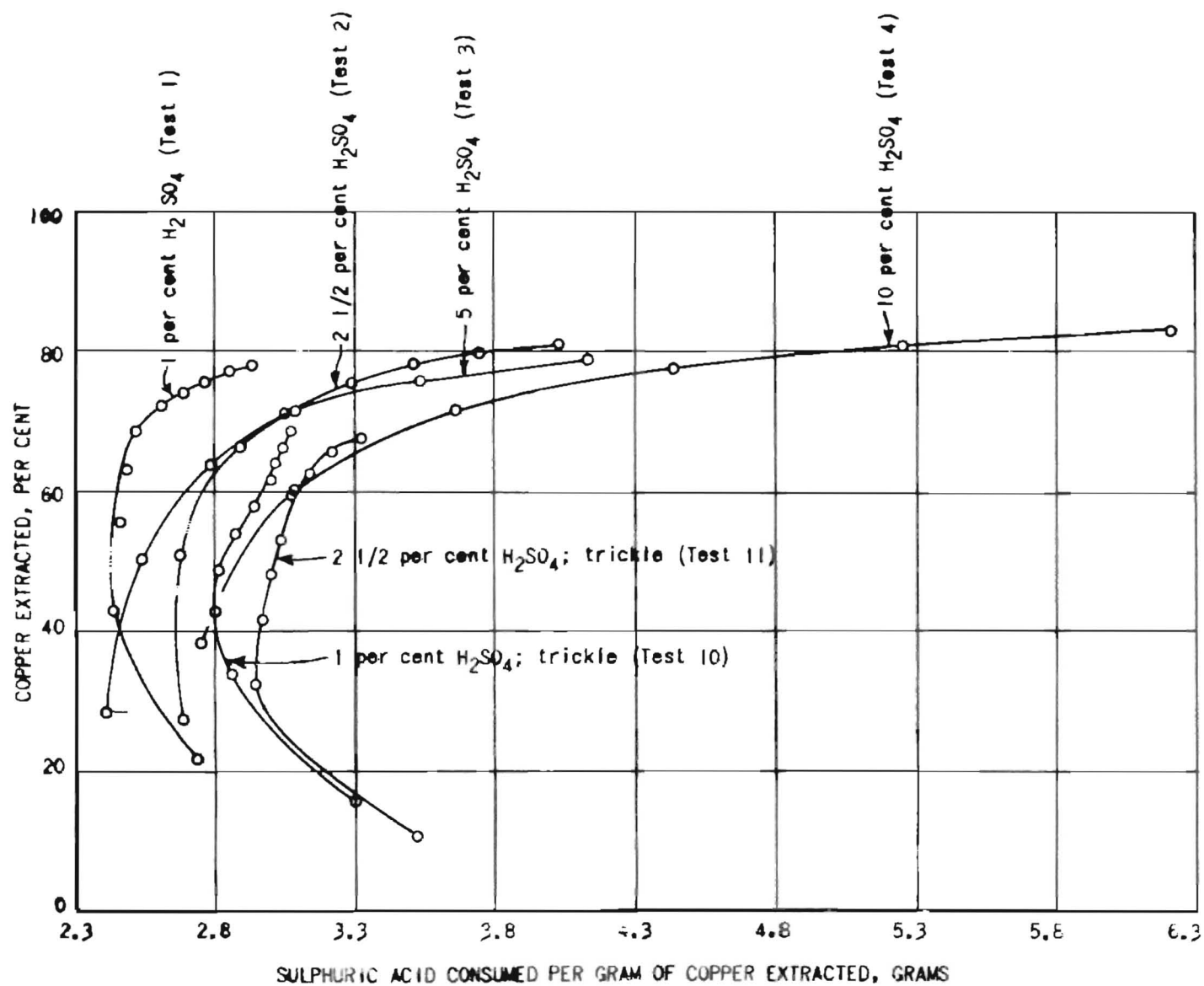


Figure 2.- Consumption of acid in leaching Ajo ore

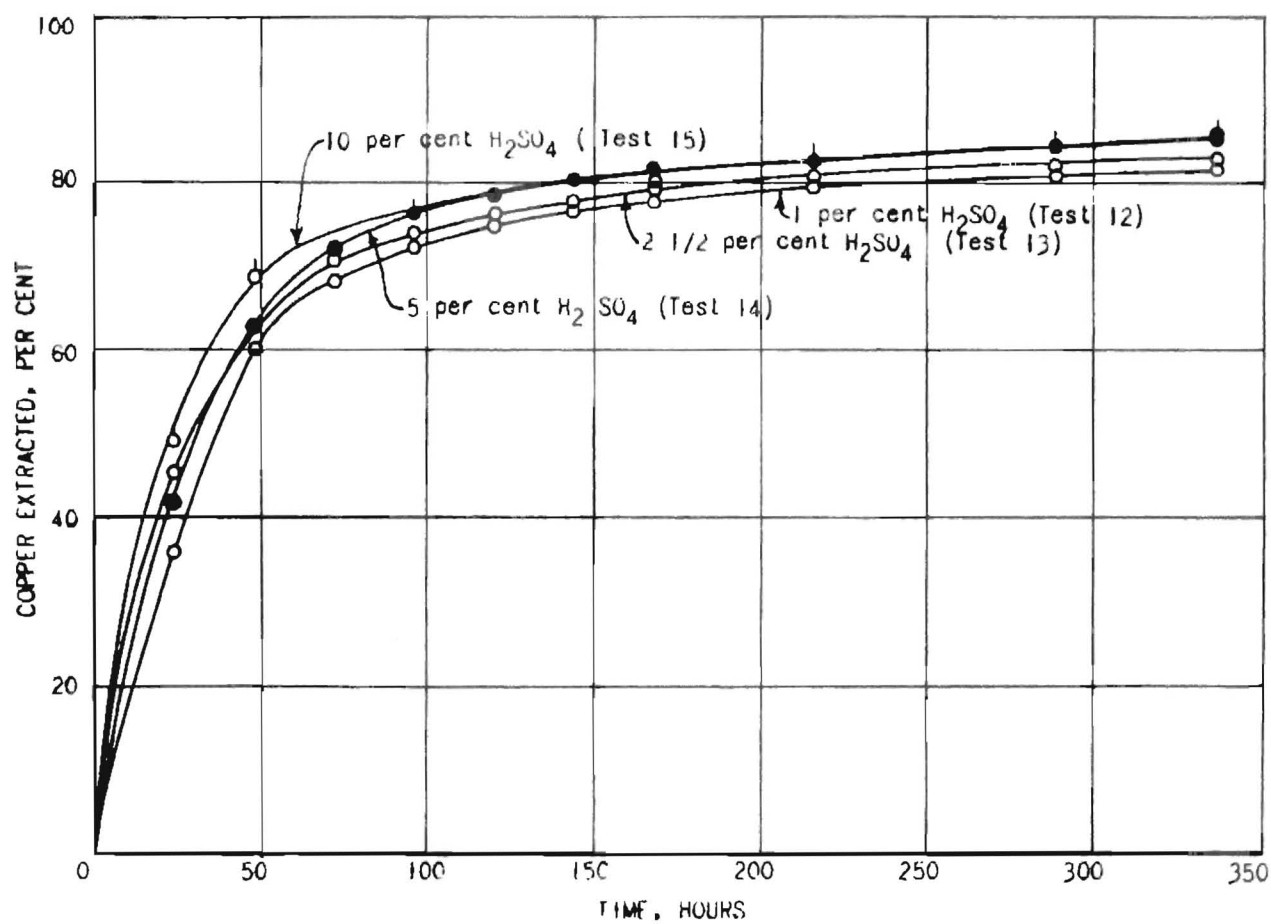


Figure 3.- Rate of extraction of copper from Silverbell Imperial ore

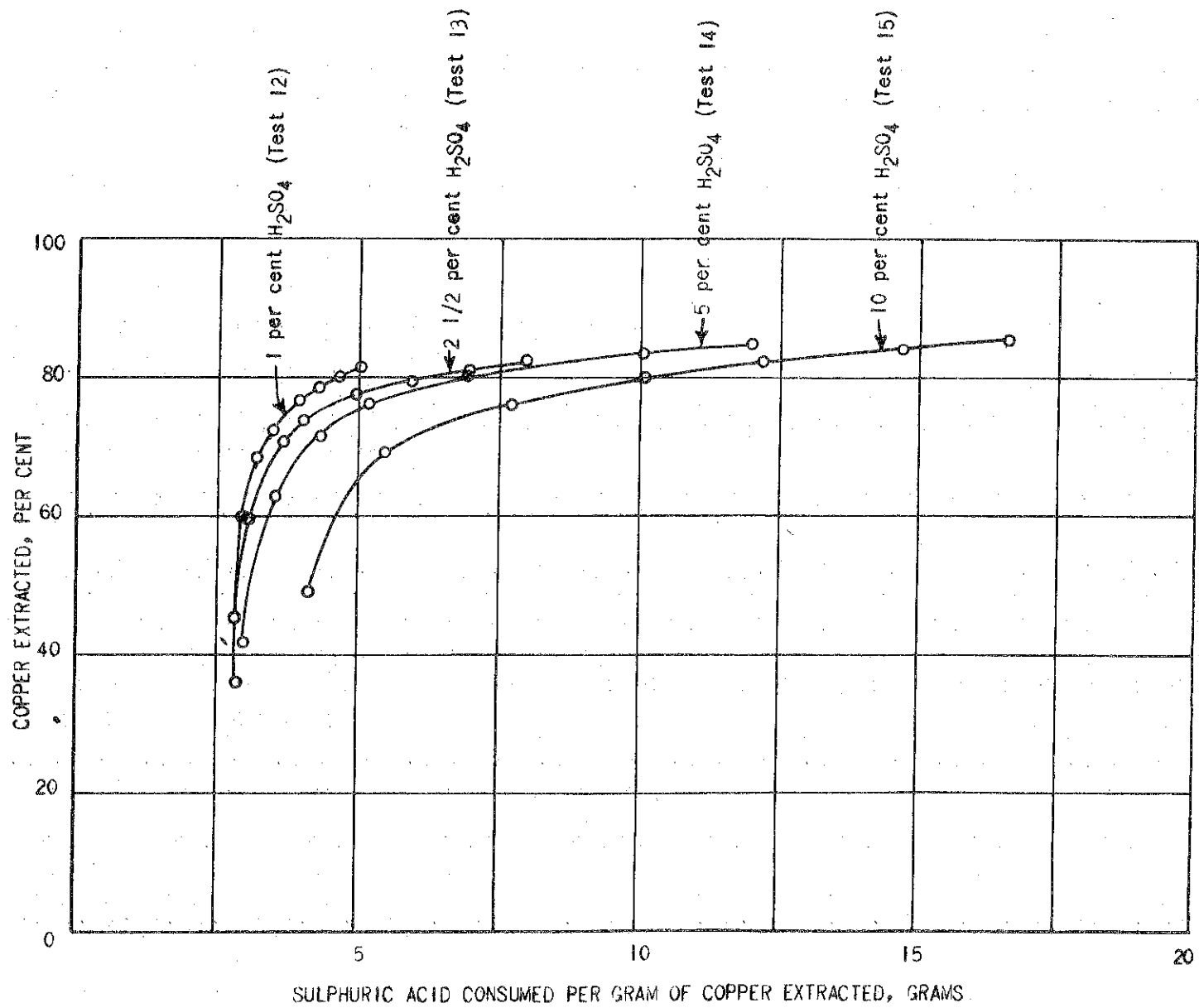


Figure 4.-- Consumption of acid in leaching Silverbell Imperial ore

Table 5.- Summary of results of tests of A10 ore

	Test 1	Test 2	Test 3	Test 4	Test 5	Test 6	Test 7	Test 8	Test 9	Test 10 ⁵	Test 11 ⁵
Sulphuric acid in leaching solution Per cent	1	2 $\frac{1}{2}$	5	10	(1)	(2)	(3)	(4)	5	1	2 $\frac{1}{2}$
Copper in heads, total, Per cent	1.17	1.17	1.17	1.17	1.17	1.17	1.17	1.17	1.17	1.17	1.17
Copper in heads, acid-soluble Per cent	.985	.985	.985	.985	.985	.985	.985	.985	.985	.985	.985
Copper in tailings, total Per cent	.20	.13	.155	.115	.195	.17	.22	.18	.40	.345	.295
Copper in tailings, acid-soluble Per cent	.08	.05	.055	.055	.055	.05	.08	.06	.275	.28	.18
Copper dissolved (from solution analyses) Grams	46.5	48.2	47.8	50.1	47.8	46.6	46.8	45.7	41.4	41.6	41.9
Copper in heads, total, Grams	58.4	58.4	58.4	58.4	58.4	58.4	58.4	58.4	58.4	58.4	58.4
Copper dissolved (on heads and solutions analyses), total Per cent	79.7	82.4	81.9	85.8	81.8	79.9	80.1	78.2	71.0	71.2	71.8
Iron dissolved Grams	12.0	20.8	28.3	52.1	21.5	15.5	18.0	16.4	9.0	10.7	15.0
Alumina (Al ₂ O ₃) dissolved Grams	17.5	25.0	29.0	48.7	25.5	18.2	23.5	18.1	12.5	16.3	18.5
Sulphuric acid consumed, Grams	131.6	211.4	232.8	383.4	191.6	160.7	175.8	168.4	125.3	138.9	165.2
Sulphuric acid consumed per gram copper dissolved, Grams	2.83	4.38	4.87	7.65	4.01	3.45	3.76	3.69	3.03	3.34	3.94
Time of acid contact Days	10	10	10	10	8	9	10	6	8	10	10
Time of washing Days	3	3	3	3	3	3	3	3	3	2	2

1 10 per cent for 2 days; 1 per cent for 6 days.

2 5 per cent for 3 days; 1 per cent for 6 days.

3 1 per cent for 2 days; 2 $\frac{1}{2}$ per cent for 2 days; 5 per cent for 6 days.

4 10 per cent. Same solution on ore throughout; no fresh solution added.

5 The ore in tests 10 and 11 was leached by the trickle open-drainage principle.

Table 6.- Summary of results of tests of Silverbell Imperial ore

		Test 5	Test 6	Test 7	Test 8
Sulphuric acid in leaching solution	Per cent	1	2½	5	10
Copper in heads, total	Per cent	1.05	1.05	1.05	1.05
Copper in heads, acid-soluble	Per cent	.88	.88	.88	.88
Copper in tailings, total	Per cent	.225	.16	.135	.12
Copper in tailings, acid-soluble	Per cent	.08	.06	.045	.045
Copper dissolved (from solution analyses)	Grams	42.8	43.4	44.6	45.0
Copper in heads, total	Grams	52.4	52.4	52.4	52.4
Copper dissolved (on heads and solutions analyses), total	Per cent	81.6	82.9	85.2	85.9
Iron dissolved	Grams	24.1	46.6	78.4	109.7
Alumina (Al ₂ O ₃) dissolved	Grams	39.7	67.4	92.7	119.8
Sulphuric acid consumed	Grams	233.5	363.5	570.0	825.1
Sulphuric acid consumed per gram copper dissolved	Grams	5.46	8.37	12.77	18.33
Time of acid contact	Days	14	14	14	14
Time of washing	Days	3	3	3	3

DISCUSSION

To make all results strictly comparable the strength of acid on the ore at all times should be constant, or the solution on the ore should be drained off and replenished with fresh solution at equal intervals. From a practical standpoint, however, one would not drain solution from the ore and replenish it with new solution unless the strength of reagents in the solution were lowered to such an extent that a change was necessary or desirable. One would not change a solution containing 5 per cent of acid so frequently as one containing only 1 per cent. In the trickle-leaching experiments where open drainage was used the condition of constant strength of acid on the ore was most nearly fulfilled. In the flood-percolation tests the stronger the acidity the less frequently the solutions were drained from the ore and replenished with fresh solution. Since the rate of dissolution of minerals varies so much with changes in temperature the temperature differences between various tests should be small to make results comparable.

When Ajo ore was leached with solutions containing 1, 2½, 5, and 10 per cent of sulphuric acid, respectively, the rate of dissolution increased with increasing strength of acid, although the percentage extractions resulting from the 2½ and 5 per cent solutions were approximately the same. The probable explanation of the rates being nearly the same for the two intermediate strengths of acid is that the solution was changed more frequently for the 2½ per cent than for the 5 per cent solution. Actual analyses showed that in some instances the effluent solutions from the test employing 2½ per cent acid were higher in acidity than those from the test employing 5 per cent acid.

In test 3 the leaching solution added contained 5 per cent sulphuric acid throughout the entire test; and in test 4, 10 per cent sulphuric acid. In test 5 the solution contained 10 per cent sulphuric acid for the first 48 hours; in test 6, 5 per cent sulphuric acid for the first 72 hours. All other conditions being equal, one might expect the percentage extractions in tests 4 and 5 to be identical for the first 48 hours and in tests 3 and 6 for the first 72 hours. Figures 1A and 1B show that this not true. At the end of 24 hours

the percentages of copper extracted and recovered in the effluent solutions were 28.4, 22.7, 38.2, and 37.1 for tests 3, 6, 4, and 5, respectively. In 48 hours the percentage extractions were 50.2, 40.3, 60.2, and 58.7 respectively. The differences were greater for the 5 per cent than for the 10 per cent solutions. The probable explanation for the variations of parallel tests is found in the fact that in tests 1 to 4, inclusive, the mean solution temperature during the first day was about 27°C. and during the second day about 25°C. In tests 5 to 8, inclusive, the temperature was approximately 19°C. during the first 48 hours.

In the trickle-leaching experiments with Ajo ore the rate of dissolution in 1 and in 2½ per cent sulphuric acid was virtually the same. However, 3.94 grams of sulphuric acid was consumed per gram copper extracted when 2½ per cent acid was used, whereas only 3.34 grams was consumed when 1 per cent acid was employed.

The data in Figure 2 show the over-all consumption of acid per unit of copper extracted. In the flood-percolation tests after about 60 per cent of the copper was extracted the consumption of acid increased markedly. This can be illustrated best if one takes the ratio of acid consumed per unit copper extracted from one sampling to the next. The ratio from the period of the previous sampling to the 72-hour sampling was 2.51, 3.95, 3.73, and 6.64 in tests 1, 2, 3, and 4, respectively. The time from the previous sampling was 16, 16, 24, and 24 hours, respectively. From the previous sampling to the 240-hour sampling the ratios were 10.2, 21.7, 19.5, and 39.2, respectively. When 10 per cent sulphuric acid was used only 1.38 grams copper was dissolved and 54.15 grams sulphuric acid was consumed from the 144 to the 240 hour period.

In the trickle-leaching tests the consumption of acid per gram copper extracted during the entire test was slightly higher than in the flood-percolation test, when 1 per cent acid was used, but the ratio was lower for the 2½ per cent acid. However, during the later stages of the extraction the ratios from one sampling to the next were lower when trickle leaching was followed. In the 216 to 240 hour period the ratios were 3.97 and 6.35 in the trickle-leaching tests employing 1 and 2½ per cent acid, respectively. The percentage extractions, however, were lower in the trickle-leaching than in the flood tests. The temperature was also lower. Since the consumption of acid did not vary greatly with acidity up to an extraction of 50 or 60 per cent of the total copper, since stronger acidity increased the rate of dissolution of the copper minerals, and since in trickling the solutions through the ore the consumption of acid did not increase so rapidly near the end of the test as it did in flood leaching, in leaching an ore such as that from Ajo one might use flood-percolation methods and strong acid to dissolve rapidly a large percentage of the copper and then finish the extraction with weaker solutions added by the trickle principle. This would increase the complexity of the leaching operation, but savings in cost of acid might justify a more complicated leaching cycle, especially in places where acid is expensive.

The rate of extraction of copper from Silverbell Imperial ore did not vary so much with strength of acid, although it did increase to a certain extent with increases in strength of acid. The consumption of acid was very high, and it increased markedly with increase in strength of acidity. As shown in Table 6, the consumption of sulphuric acid per gram copper extracted during the entire test was 5.46, 8.37, 12.77, and 18.33 grams, for solutions containing 1, 2½, 5, and 10 per cent H_2SO_4 , respectively. The consumption of acid increased enormously as the percentage of copper extracted increased. When 1 per cent acid was used 5.13 grams of copper was extracted and 13.17 grams of sulphuric acid was consumed from the 12 to the 24 hour period. The consumption of acid was 2.56 grams per gram copper

extracted. In the 288 to 316 hour period 0.32 gram copper was extracted and 10.07 grams sulphuric acid was consumed, giving a ratio of 31.4. When 10 per cent acid was used 10.31 grams copper was extracted in the 24 to 48 hour period, and 90.43 grams acid was consumed; the ratio was 8.77. From the 288 to 336 hour period 0.61 gram copper was dissolved and 95.26 grams sulphuric acid was consumed, giving a ratio of 156.3. This last ratio is so high that one could not economically leach the last of the copper from such an ore by the use of strongly acidic solutions. Silverbell Imperial ore is not suitable for leaching with acidic solvents. The high acid consumption is due to the large amount of iron and alumina dissolved from the ore and to the large amount of acid-soluble magnesia present. It is possible that a preliminary heat treatment might render the iron and alumina less acid soluble. This was not studied experimentally.

The authors believe that this work has shown quantitatively that using a leachable ore such as the Ajo ore employed in these experiments a large percentage of the copper can be extracted rapidly from the ore by a strongly acidic solution without undue consumption of acid. After about 60 per cent of the copper is dissolved (about 84 per cent of the copper is nonsulphide and 16 per cent sulphide) the rate of dissolution does not change markedly with changes in acidity, whereas the consumption of acid does. Weaker solutions should be used in the later part of the leaching cycle. Trickling solution over the ore near the end of the cycle would also decrease the consumption of acid.

The rate of dissolution of the various oxidized and sulphide minerals of copper and the requirements of solutions to give high current efficiency in the electrodeposition of copper from solutions obtained in leaching lead to the following speculation as to leaching mixed oxidized and sulphide ores. In electrolytic precipitation to obtain high efficiency of current one needs clean solutions low in ferric salts. The efficiency of current is so low when solutions are high in ferric iron that from the economical standpoint one has little to choose between electrolytic and scrap-iron precipitation. Strongly acidic solutions attack the gangue more vigorously than weak ones, but a large percentage of the oxidized copper can be extracted from the ore before the attack on the gangue becomes appreciable. Earlier work has shown that the rate of dissolution of chalcocite and of bornite is virtually independent of the strength of ferric iron in the leaching solution over a wide range of concentration. An ore therefore might be leached for a short time with a relatively strongly acidic solution containing little or no ferric iron. A large percentage of the oxidized and some of the sulphide copper would be dissolved. The pregnant solution containing very little impurities then could be brought to the tankhouse, the copper deposited electrically, and a high current efficiency realized. The residue remaining in the vats then could be treated with a solution containing enough ferric iron to dissolve the sulphide copper minerals and enough free acid to dissolve the remaining oxidized copper, but low enough in acidity to prevent much attack on the gangue. The solutions from this operation could be brought to iron-precipitation launders and the copper recovered as cement copper. If it was necessary to regenerate a certain amount of ferric sulphate, it might be done by some suitable method such as the bureau's ferric sulphate-sulphuric acid process, or electrolytically in a separate section of the tankhouse. Such a scheme might add to the number of leaching vats necessary, but the added cost of construction might easily be offset by decrease in cost of acid and precipitation, the two major costs in the hydrometallurgy of copper.

It would also be possible to dissolve the oxidized copper minerals with sulphuric acid and after washing to regrind the residue containing the sulphide copper and recover the latter by flotation. This procedure would yield solutions highly desirable for electrodeposition of the copper and a concentrator feed containing the copper largely as sulphide.

Since the water of the Southwest usually is slightly alkaline it would not be particularly difficult to reduce the acidity of the residue in the leaching vats. Unless the operator wishes to drop pyrite in the flotation process, slightly acidio solutions are not undesirable. The double process of leaching followed by flotation was tried on a mixed oxide-chalcocite ore some years ago and not used because another process was slightly cheaper, although it saved less copper. The great improvements in flotation reagents during the last 10 years make it seem likely that the combined process would now be successful economically as well as metallurgically.