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BURRA MINES COPPER ORE
Treatment Methods

by

B. H. J. Waters

Investigated by: Industrial Chemistry Section

Officer in Charge: R. E. Wilmshurst

P. A. Young. Director

THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES
Adelaide South Australia

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SUMMARY

Background

Results of previous work showed that copper could be extracted from Burra oxidised copper ore by acid leaching, but that leaching with ammoniacal solutions or flotation gave only low recoveries. However, the samples on which this work was done were considered to be not representative of the ore now available for treatment and a further study of extraction techniques was requested.

Objective

The aim of the present programme was to investigate various methods of treating the ore and to determine which method would be the most economical.

Summary of Work Done

Concentrates could be produced by flotation or gravity methods, but the concentrate grades and the recoveries were too low to justify the adoption of either of these methods. Both agitation and percolation leaching with sulphuric acid can achieve leaching efficiencies of 90% with acceptable acid consumption. Since the capital cost of percolation leaching equipment is lower than that for agitation leaching equipment, the former system is preferred. Because of the small proportion of ore of a suitable size, the use of a photoelectric ore sorting machine cannot be justified. Leaching tests carried out with either sodium cyanide or ammoniacal solutions gave only low recoveries and neither method would therefore be of value.

Direct precipitation of copper in the leach liquor, with subsequent recovery by flotation was shown to be feasible, but the recovery of copper and the grade of product obtained were not sufficiently high to show any advantages over conventional leaching processes.

The estimated cost of a plant treating 1000 tons daily of ore containing 1.7% Cu is $1.3m when using the standard leach-CCD-cementation procedure. For a plant using the leach-precipitation-flotation route the estimated cost is $1.1m. Operating costs are estimated to be $3.3-3.6 per ton of ore, with a surplus of return over cost of approximately $4 per ton of ore by either process.

The break-even grade, based on direct costs alone, is 0.65% Cu.

Conclusions

Percolation leaching appears to be the most satisfactory way of treating the ore if production of liquor for conventional treatment is desired. However, any acid leaching process would only be economical if the ore did not contain excessive amounts of gangue carbonate minerals.

Further work is also considered to be justified on the leach-precipitation-flotation process as applied to ore ground to suitable size for agitation leaching. The operating cost of this process appears to be lower than those of other processes considered.

1. INTRODUCTION

A request was received from the South Australian Government Department of Mines to conduct an investigation into the recovery of copper from oxidised ores and dumps at Burra. Earlier work on samples of Burra ore had been carried out by a number of investigators. Gartrell and Blaskett (1941) had shown that acid leaching gave good recoveries but high acid consumption, whereas preliminary ammonia leaching suggested that only low recoveries would be obtained. Weir (1957) used flotation, heavy media and gravity methods, and roasting under reducing and chloridising conditions but with little success. Hosking and Sheridan (1960) carried out further flotation studies but the recoveries obtained were unfavourably low. However, this earlier work was carried out on samples now considered by the Department of Mines, not to be representative of the ore body and further samples were submitted.

The further work envisaged included petrological and mineralogical examination of the samples, flotation studies, leaching investigations with acid, cyanide and ammonia leachants, and a study of methods of recovering copper from the liquor.

2. MATERIAL EXAMINED

Four samples of Burra copper ore were received on 31st August, 1964 and were identified as follows:

a.	KRW 154/64	A 3317/64
b.	KRW 155/64	A 3318/64
c.	KRW 156/64	A 3319/64
d.	KRW 157/64	A 3320/64

Instructions were received from the sponsor for the preparation of a composite of the four samples for beneficiation tests. The sampling procedure was as follows:

> Each sample was air-dried and then stage crushed through a jaw crusher and rolls crusher to minus $^1/_4$-inch mesh. Petrological samples were taken from the minus $^1/_4$-inch mesh material after thoroughly mixing by coning and quartering. The petrological results are given in Appendix A.
> A composite sample (Sample H) was prepared by mixing 200 lb of each of samples KRW 154/64 and KRW 155/64, with 400 lb of each of samples KRW 156/64 and KRW 157/64. The total 1200 lb was mixed thoroughly by coning several times.
> A 100 lb sample was taken from the whole and divided into two lots of 50 lb. One 50 lb portion was sent to USA to American Cyanamid Co. for testing and the second 50 lb portion was retained as a duplicate sample. After a portion had been cut out for assay and mineralogical examination, the remainder of the composite sample (1100 lb) was stored for beneficiation work. The assay and mineralogical results are given in Appendix A.

A concentrate sample (Sample J) was prepared by combining various products of flotation tests.

Three further samples of Burra copper ore were received on 18th May, 1965 and were identified as follows:

Sample A	This sample was from the 200,000 ton dump which contained a quantity of high carbonate gangue. The sample contained 0.92% Cu and 11.2% CO_2.
Sample B	This sample was taken from half-way down the track into the open cut, co-ordinates 14029N, 10355E. It contained 1.26% Cu and 0.9% CO_2.
Sample C	This sample was from the bottom of the open cut to the NW of the water, co-ordinates 14310N, 10415E. It contained 2.24% Cu and 2.74% CO_2.

Equal portions of Samples B and C were combined to give Sample BC. The analysis and sizing of this composite sample are tabulated below:

			Sample BC %
Total:	Cu		1.75
	CO_2		1.82
Sizing:	- ¼	+ ⅛ in.	15.2
	- ⅛ in.	+ 8 mesh	10.0
	- 8	+ 12 mesh	18.5
	- 12	+ 16 mesh	5.7
	- 16	+ 22 mesh	9.0
	- 22 mesh	+100 μ	33.9
	-100	+ 10 μ	5.4
	- 10 μ		2.2

3. ANCILLARY MATERIALS

Flotation Reagents:

Potassium amyl xanthate
Myrtan
Pine oil
Fuel oil
Hydrolysed palm oil
Sodium Hydrosulphide

American Cyanamid Reagent 633
American Cyanamid Reagent S3275
American Cyanamid Reagent 610
Emigol
Aerofroth 70
Aerofloat 208

Heavy Media Separation:	Bromoform sp gr 2.85.
Leaching:	Concentrated sulphuric acid (commercial) Sodium cyanide Ammonia sp gr 0.880
Copper Recovery:	Magnesium carbonate Iron turnings.

4. EQUIPMENT

The equipment used in the tests included:

Laboratory stainless steel rod mill,	10 x $7\frac{1}{4}$ in. ID with 15 1-in. stainless steel rods.
Photometric sorter	
Scrubber,	20 in. dia x 14 in. deep
Laboratory rolls agitator	
Percolation leaching vessels,	$9\frac{1}{2}$-in. dia $3\frac{1}{2}$ ft deep rubber lined, mild steel.

5. EXPERIMENTAL PROCEDURE AND RESULTS

5.1 Flotation

Mineralogical examination of sized fractions of minus 20-mesh head sample (Sample H) indicated little composite material even in the coarsest fraction. The material was ground to minus 52 mesh for flotation of the copper minerals, because at this particle size good liberation of the few composites present was achieved.

To avoid overgrinding of the copper minerals, and because of the high clay content of the material, the minus $\frac{1}{4}$-inch head sample was screened on 52 mesh and the oversize fraction ground in the rod mill to minus 52 mesh. Both fractions were then combined and conditioned in the Fagergren cell for flotation.

Test 1. The test conditions are shown in Table 1 and the results are shown in Table 2.

The reagent Myrtan was added as a calcite depressant in Test 1 because the results of earlier tests on Burra ore had indicated that calcite was present in substantial amounts. However, the mineralogical examination (the report of which was received after Test 1 was completed) did not reveal the presence of calcite. Myrtan was therefore omitted in the next test and instead American Cyanamid Reagent 633 was used as a depressant for the micaceous gangue minerals which appeared to float readily with the copper minerals in Test 1.

Test 2. The test conditions were similar to those of Test 1 except that the Myrtan was replaced by AC633 (1.0 lb/ton).

Test 3. The micaceous gangue minerals still floated freely in Test 2 and it was decided to try an alcohol-type frother rather than the pine oil in the hope of achieving better selectivity. Test 3 was carried out under the same conditions as Test 1 except that Aerofroth 70 was used instead of pine oil.

Test 4. The sample was ground to minus 52 mesh. Other test conditions were as shown in Table 3.

The three stage concentrates were combined and cleaned twice.

Test 5. The conditions for this test were similar to those of Test 4 except that American Cyanamid Reagent 610 was added (0.5 lb/ton) as a gangue depressant. The results of Tests 1 to 5 are shown in Table 2.

Test 6. This test was similar to Test 4 except that palm oil mixture[(a)] was added to the first rougher stage at the rate of 0.5 lb per ton.

Test 7. This test was similar to Test 4 except that American Cyanamid Reagent S-3275 was added as an extra promoter for the copper minerals.

Test 8. In this test sodium hydrosulphide was used instead of sodium sulphide and American Cyanamid Reagent 633 was also added to aid gangue depression. The test conditions are shown in Table 5.

The froth appeared clean after the second stage addition and the concentrates from both stages were combined and cleaned without further reagent addition.

Test 9. In this test, which was similar to Test 8, 2.0 lb of the palm oil mixture per ton was added with the other reagents to try to improve the recovery of copper. Rougher flotation only was employed.

The results of Tests 6 to 9 are shown in Table 4.

At this stage, a typical rougher tailing was examined to determine the distribution of the copper. A portion of the tailing from Test 5 was screened on 100, 200 and 300-mesh screens and each fraction was assayed for copper. The results which are shown in Table 6, revealed that most of the copper was in the minus 300-mesh fraction.

Test 10. Because the major part of the copper in the tailings was in the fine particle size range, desliming the flotation pulp was decided on. The objective was to achieve maximum copper recovery from the deslimed material after which the slime could be combined with the flotation concentrate. The test conditions for the deslimed pulp were similar to those for Test 8 (Table 5) and the results are shown in Table 7.

Test 11. A 1100-g charge of the minus ¼-inch feed was prepared by screening on 52 mesh and grinding the oversize to pass 52 mesh. A size analysis of the minus 52 mesh combined oversize and undersize is shown in Table 8.

The minus 52-mesh charge was deslimed by settling and decantation of the minus 35-microns (nominal) quartz. The sands were conditioned and floated under the conditions shown in Table 9.

The results of this test are shown in Table 10 along with the weight and assay of the slime fraction and the cumulative weight and assay of the combined slime and rougher concentrates (Stages 1 and 2).

A mineralogical report on the rougher tailing from this test is given in Appendix B.

(a) Palm oil mixture:

	% by weight
Hydrolysed palm oil	0.45
Diesel fuel oil	0.45
Emigol	0.10
Water	99.00

5.2 Heavy Liquid Separation

Test 12 was carried out to evaluate the feasibility of gravity concentration.

The ore was ground to minus 52 mesh and deslimed as described for Test 11. The deslimed material was split into two fractions by screening on 200 mesh. Heavy liquid separations were carried out on each fraction using Bromoform of 2.85 sp gr. The results of this test are shown in Table 11.

The slime fraction was combined with the two heavy fractions from heavy liquid separation. The results are shown in Table 12.

5.3 Ore Sorting – Sample A

A sample weighing approximately 2 tons was screened, without crushing, into various size ranges. Fractions coarser than 2 inches were washed to remove adhering slime.

An attempt was made to separate brown, high carbonate gangue from rocks of a different colour, using a photometric sorter. Poor results were obtained by this method and the rocks were separated by hand sorting.

It is thought probable that a fairly sharp separation could be made with a photometric sorter using a special filter. However, tests were not continued in view of the small percentage by weight of the ore that could be treated by this method.

The results of hand sorting are shown in Table 13. In the coarse, green fraction, 17.2% of the copper was recovered.

5.4 Attrition Conditioning

Ore (Sample H) was crushed to pass a ½-inch screen. A sample weighing 100 lb was agitated for 4 hours with 7 gallons of water and the equivalent of 0.025 lb natural Myrtan per ton of solids. The scrubbed solids were screened into size fractions for assay. Results are shown in Table 14.

5.5 Leaching

5.5.1 Cyanide Leaching

Cyanide leaching was carried out at a single pulp density (40%) using various amounts of sodium cyanide. The conditions of leaching and the results are given in Table 15.

5.5.2 Acid Leaching

Agitation Leaching. Agitation leaching using sulphuric acid was carried out on a pulp at 40% solids content with acid of various concentrations. For several of the tests the acid was added at the beginning of the leaching but for the remaining tests the acid was added at a rate sufficient to maintain a constant pH. The conditions of the experiments and their results are shown in Table 16.

All the liquors prepared by leaching at a constant pH (Tests 4-7) deposited either a brown or white precipitate upon standing.

Free acid was determined in the liquor from Test 1.

The actual acid consumption was found to be 170 lb sulphuric acid per short ton of ore.

The preliminary acid leaching tests on copper ore ground to minus 200 mesh indicated that 90% of the copper could be extracted using 100 lb sulphuric acid per short ton of ore. Several tests were done to determine the leaching efficiency at other particle sizes. Copper and iron were determined in the residues and leach liquors. The conditions and the results obtained are given in Table 17.

The extraction of copper did not vary greatly with particle size. The amount of iron dissolved decreased sharply when material coarser than minus 200 mesh was leached. The effect of using smaller quantities of acid on the minus 52-mesh material was investigated. The conditions and the results obtained are given in Table 18.

A little more than the quantity of acid theoretically required to dissolve all the copper was required to leach 80% of the copper, but a large excess of acid was necessary to increase the recovery to 90%. The results of leaching at 40 and 50% solids are shown in Table 19. The leaching efficiencies were almost identical.

Leaching rates were determined for minus 52-mesh and minus ¼-inch material, using 75 lb of sulphuric acid per short ton of ore in each case. The results shown in Table 20 indicate that for a given particle size there was very little difference in extraction at 1 and 2 hours.

The minimum leaching time required was 30 minutes for minus 52-mesh material and 1 hour for minus ¼-inch material.

A single leaching test was done on a concentrate sample. The conditions of leaching and the results obtained are given in Table 21. The result of leaching the ore under similar conditions (Test 12) is included in the table for comparison. The results show that most of the copper was leached from the concentrate at an acid consumption only slightly in excess of the theoretical requirements.

Settling Tests. Settling tests were done on a diluted leach slurry. A sample of ore was leached under conditions of leach No. 17 (Table 20). The pulp was then diluted with 3-4 times its own volume of water. Synthetic and natural flocculants were added to samples of the slurry and the settling rates were observed. The results of these tests are shown in Figure 1. From the settling rate observed during each test the thickener area was calculated. The basic data used in the calculations and the results obtained are shown in Table 22.

Percolation Leaching

Test 1. A 110-lb sample of ore was leached with 10% sulphuric acid solution. The quantity of acid used was equivalent to 95 lb per ton of ore and was calculated to be 75% in excess on the basis of copper requirements or 50% in excess on the basis of total carbonate present. The main purpose of this test was to determine the general physical characteristics of percolation leaching, and a number of different flowrates were used during the test.
The maximum flowrate which could be obtained once the ore was saturated was found to be 0.6 gallon per square foot per hour.
The volume of liquor retained by the bed after draining overnight was approximately 14 litres.

Following the passage of 22.9 litres of acid, sufficient water was passed to displace the acid retained in the bed. The bed was not further washed.
The 25.2 litres of effluent collected contained 450 g of Cu which corresponded to an extraction of approximately 50%.

Test 2. A 115-lb sample of ore was leached with 23.8 litres of sulphuric acid at a nominal flowrate of 0.5 gallons per square foot per hour. The excess acid was at the same level as for Test 1. Some difficulty was experienced in maintaining a pumping speed at the low levels required, and the bed was left in contact with the final 14 litres of acid overnight, displacement with water beginning on the following morning.
The first 23.8 litres of effluent (displaced acid) contained 21.4 g Cu per litre, equivalent to a total of 509 g Cu. This corresponds to the extraction of 56% of the original Cu. The next 13.6 litres of effluent approximately corresponding to a 1 displacement volume wash had a copper concentration of 8.2 g per litre equivalent to a total of 112 g Cu. This corresponds to the extraction of 12% of the original copper.

Test 3. A split leach was carried out in which 23.8 litres of fresh 10% acid (95 lb per ton of ore) was delivered to the top of the partly depleted bed of ore from Test 2. The effluent was permitted to drain on to a fresh bed of ore weighing 115 lb.
The calculated copper content of the partly depleted ore was 294 g, including that in solution but held up in the bed.
Following the passage of the acid, wash water was pumped onto the depleted bed.
During the course of this test samples were taken at intervals of the effluent from the first bed.
The nominal first bed volume effluent from the second bed (15.2 litres) contained 0.5 g Cu per litre, which was considerably less than the copper content on entry. A measurement of the pH gave a value of 6.5, and it appeared that the acid remaining in the wash liquor had been consumed in column 2 and the copper had been precipitated.
The mean copper concentration of the acid leach liquor from column 1 was approximately 7.2 g per litre equivalent to 171 g of copper in solution or 19% of the original copper present.
The 24-litre-acid effluent from column 2, which followed the 15.2-litre wash water displacement contained 17.0 g Cu per litre, equivalent to 308 g Cu. Presumably 171g of this amount had been carried down from column 1.
Unfortunately movement of column 1 to a higher level prior to Test 3 had caused the bed to slump 1 inch, and it became impossible to maintain the desired flowrate, resulting in the test continuing overnight with resultant inadequate control of sampling.

Test 4. This test was identical with Test 2 except that the acid liquor was made up with wash water from Test 3, and had a copper concentrate of 6.8 g per litre.
The first 4 litres of effluent was barren. Presumably the copper in solution initially was precipitated when the pH rose, as suggested for Test 3.
The next 20.7 litres out had a copper concentration of 25.0 g per litre or 518 g of copper. If it is assumed that all the copper entering in the acid solution eventually passed out with the effluent, 350 g of copper was extracted from the ore by the passage of the acid. This corresponds to 38% of the available copper.
The next 14.5 litres, representing the first displacement wash had a concentration of 7.3 g per litre and contained 106 g of copper. The overall extraction in Test 4 was thus 50%.

Test 5. A 115-lb sample of ore was leached with 42 litres of 4.1% sulphuric acid solution equivalent to 66 lb of acid per ton. The quantity of acid used was calculated to be 22% in excess on the basis of copper requirements. The ore bed was flooded to a depth of approximately 3 inches with acid, and the acid was allowed to percolate at the natural drainage rate, which was approximately constant throughout the rest at 0.5 gallons per square foot per hour. Following the passage of the acid, water was passed through the bed.
The first 5.5 litres of effluent collected was barren with a pH of approximately 6.
The next 60.5 litres of effluent composed of 42 litres of acid liquor and 1 displacement wash contained 61.2% of the available copper.

Test 6. This test was similar to Test 5, except that 47.5 litres of 4.1% sulphuric acid (i.e. 75 lb acid per ton) containing 1.0 g Cu per litre was used. Vibration from the operation of adjacent heavy equipment caused the ore bed to slump and the percolation rate was only 0.3 gallons per square foot per hour throughout the test. Samples of the liquor were collected regularly and the copper concentration and pH were determined. The results are shown graphically in Figure 2. For the purpose of this test an operating (i.e. undrained) bed volume was taken as 16 litres.
The first 0.5 bed volume of effluent contained 0.2% of the available copper. The next 2.5 bed volumes, representing the remainder of the acid displaced, contained 72.4% of the copper in the ore plus the 4.6% already present in the leach liquor. The copper concentration of this liquor was 19.9 g per litre. The first bed volume of wash water displaced contained 4.6% of the copper.
The overall recovery was thus 77%, and the test corresponded to an equilibrium situation in which the amount of acid in the leach liquor equalled that which would be provided by the wash water.

5.5.3 Leach-Precipitation Flotation

Three leach-precipitation flotation tests were conducted on minus 8-mesh ore. The ore was leached with sulphuric acid and the leach residue coarser than 60 mesh was separated from the slurry. The copper was precipitated from the solution in the minus 60-mesh slurry by contact with metallic iron and recovered by flotation.

Leaching. The following leaching conditions were applied in all tests: Samples of ore weighing 280 g were agitated at 45% solids for 3 hours with 120 lb sulphuric acid per ton.

Material coarser than 60 mesh was screened from the residue and washed to remove copper bearing solution. The washings were added to the minus 60-mesh slurry and the plus 60-mesh fraction was later combined with the flotation tailing.

Precipitation. Small scale tests were carried out to determine the conditions necessary for precipitation of copper from the pulp with metallic iron. The free acid in the leached pulp was neutralized by adding dry powdered magnesium carbonate until the pH reached 3.0 (approximately). The estimated leach efficiency for minus 8-mesh ore was 85%.

Iron turnings equivalent to 1.2 times the amount theoretically needed to precipitate the copper were added to the stirred, neutralized pulp. Samples were taken at 5-minute intervals, centrifuged rapidly and the remaining soluble copper was determined in the clear liquor.

For 400 g of Sample H the estimated amount of copper leached was 5.8 g. The weight of metallic iron required to give the desired excess was 6.0 g.

Results of the tests are tabulated below:

Precipitation of Copper by Iron

Time min	Cu Remaining in Liquor
0	21.3
5	9.2
10	2.3
15	0.6
20	0.3
30	nil

Complete precipitation of the copper was achieved in 20 to 30 minutes.

The conditions determined by small scale tests were then applied to precipitation of copper in the minus 60-mesh portion of the leached slurries.

The minus 60-mesh slurries were agitated for 30 minutes with 20 g of iron turnings in Tests 1 and 2 and 10 g in Test 3.

The slurries of Tests 1 and 2 were agitated in 2-litre bottles on laboratory rolls. Test 3 slurry was agitated in a rectangular glass vessel with a glass propellor-type agitator. Excess $MgCO_3$ (120 lb/ton) was added to the slurries after precipitation of the copper; and agitation was continued for 15 minutes. The pH after this neutralization was 5.0-5.2. Residual iron turnings were removed from the slurries with a magnet, weighed and assayed for copper.

Flotation. Tests were carried out under various conditions.

Test 1. The pulp was conditioned for 1 minute with 0.04 lb Aerofroth 65 per ton and floated for 4 minutes to remove silicates which would tend to report in the copper concentrate during copper flotation. After silicate flotation the pulp was conditioned for 10 minutes with the following reagents:

	lb/ton
Potassium amyl xanthate	0.07
Aerofloat 208	0.07
Aerofroth 65	0.04

Air introduced to the pulp during conditioning appeared to increase the rate of collector coating of the copper. A copper concentrate recovered during 10 minutes of flotation contained 17.4% copper; the overall recovery was 54.8% of the copper in the original ore.

Test 2. Flotation of silicates was carried out as in Test 1, after which the pulp was conditioned for 10 minutes with the following reagents:

	lb/ton
Dow reagent Z200	0.20
Aerofroth 65	0.04

A rougher concentrate was recovered during 10 minutes of flotation.
The rougher concentrate was conditioned for 10 minutes with 0.20 lb Z200 per ton and 0.04 lb Aerofroth 65 per ton and floated for 3 minutes.
A cleaner concentrate was produced containing 68.2% copper; the overall recovery being 63.8% of the copper in the original ore.

Test 3. Silicates were not removed prior to flotation of copper. The pulp was conditioned for 8 minutes with 0.2 lb Z200 per ton and 0.04 lb Aerofroth 65 per ton and floated for 10 minutes.
The rougher concentrate was cleaned once without additional reagents.
Flotation time was 4 minutes.
The cleaner concentrate contained 30.4% Cu and the overall recovery was 55.6% of the copper in the original ore. The calculated grade of the rougher concentrate was 17.5% copper. Recovery at this grade was 83.2%.
Results of the tests are shown in Table 23.

5.5.4 Leaching with Ammoniacal Solutions

Leaching tests were done on Sample BC under various conditions.

The principal variables were:

Reagents used in addition to ammonium hydroxide

Concentration of reagents

Leaching time

Ore size

Ore condition.

Most of the tests were done on minus 8-mesh ore but for some the ore was crushed to minus 52 mesh. Some tests were done on ore which had been heated to 300°C in an attempt to decompose hydrated copper silicates.

Leaching conditions and results are shown in Tables 24 and 25.

6. DISCUSSION

6.1 Flotation

It would appear from the results of the tests that about 80% of the copper in the ore is readily floatable. The form of the copper remaining in the tailing is not known but is suspected to be as chrysocolla which is difficult to concentrate by flotation. The most promising line of investigation appears to be in desliming the ore to remove fine copper minerals which do not respond to flotation and attempting to effect better recovery of the coarser copper minerals in the deslimed sands. The flotation product can then be combined with the slime fraction to provide a low grade product for leaching which might be approximately one third of the weight of the original material.

6.2 Heavy Media Separation

The results of the heavy liquid separations indicate that gravity concentration appears to be applicable to the ore. This form of treatment may be more economical than flotation as a pre-concentrating process for ultimate leaching.

6.3 Photometric Sorting Tests

Although photometric sorting proved to be impractical it was thought that a fairly sharp separation could be made using a special filter. However, only a small percentage by weight of the ore could be treated by this method.

6.4 Attrition Conditioning

There was no apparent preferential size degradation of the high carbonate or copper minerals.

6.5 Leaching

6.5.1 Cyanide Leaching

The cyanide leaching tests showed that the extraction of copper using cyanide was unattractive because of a low leaching efficiency (approx. 40%) and a high cyanide consumption (approx. 6 lb of NaCN per lb copper extracted).

6.5.2 Acid Leaching

Agitation Leaching. The carbonate content of the composite ore Sample (H) was much lower than was expected. The amount of carbonate present was less than that equivalent to the amount of copper, if present as azurite or malachite. At least 30 to 40% of the copper in the ore would have to be present as non-carbonate minerals. The sulphur content of the ore was so low as to preclude the possibility of copper being present as sulphide, therefore the non-carbonate copper mineral would probably be chrysocolla. The higher leaching efficiency when sulphuric acid was added initially rather than at a constant pH was possibly due to the silicate copper material dissolving in the stronger acid solution.

The efficiency was affected by the leaching conditions in the following manner:

1. The amount of copper extracted using 100 lb sulphuric acid per ton of ore was independent of the particle size of the ore for sizings of minus $\frac{1}{4}$ inch or less. The iron extraction increased when the ore was ground to a particle size of minus 200 mesh.
2. Using a slightly greater amount of acid than was theoretically required to dissolve the copper, 80% of the copper was extracted.

A comparison of the results of leaching Tests 3 and 7 shows that a higher extraction was obtained when acid was added initially than when the pH was kept constant.

Under suitable conditions at least 90% of the copper was leached using 100 lb sulphuric acid per short ton of ore.

Percolation Leaching. The most favourable extraction obtained was from the ore bed used in Tests 2 and 3, from which 87% of the copper was extracted. However, this result was obtained under conditions which would require an acid consumption of 190 lb acid per short ton of ore, and produce considerable quantities of low grade liquor. Although Test 3 was incomplete, the results indicated that there may be no advantage in carrying out a split leach in preference to a single leach, and that difficulty might be experienced in extracting a significant part of the copper in the trailing column.

The results of Test 6 showed that an extraction of 77% of the copper could be achieved for an acid consumption of 75 lb of acid per short ton of ore. Since the pH of the effluent did not fall below 3.5 it seems probable that all of the acid supplied was consumed, and thus the provision of a greater excess of acid may have resulted in a higher recovery. In this test, the use of wash solutions containing copper for acid make up did not significantly reduce recovery. In Test 4 however, the use of liquor of higher copper concentration, i.e. 7 g Cu per litre resulted in a lower extraction efficiency.

The precipitation of copper carbonate from solutions as a result of the total consumption of acid appears to be of considerable value, since the first 0.5 bed volume of effluent may be rejected as barren and the copper concentration of the following liquor is thus upgraded. Separation in this manner could possibly be utilized to greater advantage if a deeper ore bed were used.

6.5.3 Leach-Precipitation Flotation

All of the dissolved copper was precipitated by slightly more than the theoretical amount of metallic iron in less than 30 minutes provided that slurry was stirred constantly. Larger amounts of metallic iron could be added in practice as long as the surplus was removed by magnetic separation before the copper flotation stage.

Neutralizing the free acid by adding magnesium carbonate served mainly to lower the consumption of metallic iron. High carbonate copper-bearing ore could be used for this purpose.

6.5.4 Leaching with Ammoniacal Solutions

The tests were not sufficiently comprehensive to allow the system to be completely evaluated. However, the leaching efficiencies were generally poor and much lower than those achieved by acid leaching. The main observations made from the limited amount of work done are tabulated below:

1. The leaching efficiency was improved by increasing the concentrations of reagents but reached a maximum value at between 1.6 and 3.2N for both ammonium hydroxide and electrolyte.

2. Leaching efficiency increased only slightly with increase of leaching time beyond 1 hour.

3. Slight improvement in leaching efficiency resulted from grinding the minus 8-mesh ore to minus 52 mesh before leaching. The improvement was probably due to an increase in the dissolution rate of copper over the short leaching period of 1 hour.

4. Roasting the ore at 300°C before leaching resulted in slightly increased copper extraction. The temperature of roasting was chosen arbitrarily and may not represent the optimum condition.

5. The addition of sodium chloride, either alone or with ammonium sulphate had no effect on the leaching efficiency.

6. Some improvement in overall leaching efficiency was achieved by releaching the residue with fresh solution or by increasing the solution to solids ratio. However, the maximum leachable amount of copper probably would not exceed approximately 60% of the total copper as illustrated by Test 20 in which ore was exhaustively leached with fresh solution.

7. CONCLUSIONS AND RECOMMENDATIONS

Most of the investigations were terminated before the optimum conditions were established, but some treatment methods were shown to have significant advantages over others.

The samples examined had only a low concentration of non-copper carbonate minerals and satisfactory extractions were obtained by leaching with sulphuric acid. Although both agitation and percolation leaching methods achieved leaching efficiencies of approximately 90%, the capital cost of percolation leaching equipment would be lower than that of agitation leaching.

Both flotation and gravity separation could be used to produce concentrates which would comprise 33 to 51% of the feed and would contain between 73 and 80% of the copper, but it is unlikely that concentration before percolation leaching could be economically justified.

The use of a photoelectric ore sorting machine, or the substitution of sodium cyanide or ammonia leachants for sulphuric acid, does not appear to have any advantages. Nor would direct precipitation of copper in the leach liquor, with subsequent recovery by flotation yield an economically preferable system, since either the recovery is low or the product is of too low a grade to be of economic value.

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Mineralogical and Petrological Section:	R. Townend
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APPENDIX A

ANALYSIS PETROLOGY AND MINERALOGY OF BURRA COPPER ORE

1. REPORT OF PETROLOGICAL EXAMINATION

KRW154/64-157-64: TS14699-14704. These four samples consist of limonitic silts and quartzose breccias. In the silts (154-155) clay minerals may form a high proportion of the rock, while chlorite and muscovite are present only occasionally. Angular quartz grains and microcline feldspar are prominent, as are opaque grains, now mostly composed of goethite, and averaging 0.01 to 0.03 mm.

The copper carbonates, malachite and azurite occur principally as late infillings in cracks, and in recrystallized quartz lenses, associated with opaque mineral.

Sample 156/64 is of uneven composition, being rich in chlorite or rich in clay minerals. Quartz and microcline feldspar grains are present sporadically while malachite is present to the exclusion of azurite. Sample 157/64 is a quartz feldspar breccia, where the matrix is either chloritic or micaceous. Talc occurs in minor quantities, associated with chlorite.

2. ANALYSIS OF COMPOSITE SAMPLE H

Component		%
Copper,	Cu	1.58
Iron,	Fe	2.04
Sulphide,	S	0.11
Sulphate,	SO_4	0.01
Carbonate,	CO_2	0.34

Spectrographic analysis detected the following elements in the corresponding concentration range:

Element	%
Si	100 -10.0
Al, Mg, Fe, Cu	10 - 1.0
Ca, Ti, K, Na	1.0 - 0.1
Ba, B, Rb, Li	0.1 - 0.01
Zr, Mn, Sr, Cr, V, Pb, Zn, Ni, Ga	0.01 - 0.001
Co, Sn, Bi, Mo, Ge	0.001 - 0.0001

3. REPORT OF MINERALOGICAL EXAMINATION

Composite Sample H. The sample was split into plus 100, plus 200 minus 100, and minus 200,fractions. It was then examined as grain mounts using refractive index oils. The sample as a whole contained very little composite material, even in the coarsest fraction. The main mineral constituents are quartz, alkali feldspar, chlorite and sericitic matrix, with minor quantities of azurite and opaques. Azurite is the main copper mineral in the plus 200- minus 100- and minus 200-mesh (BSS) fractions, whereas malachite is rather sparsely distributed. No other copper minerals were detected.

Opaque material is most common in the finest fraction, but is probably mostly goethite after pyrite. Accessory minerals are confined to the minus 200-mesh size.

APPENDIX B

MINERALOGICAL EXAMINATION OF BURRA COPPER TAILING

MA/4: 3748. Optical examination of the sample did not reveal malachite or azurite. As chrysocolla was suspected, a gravity separation was undertaken using a TBE/alcohol mixture of density 2.42. Optical examination of the resultant fractions revealed very minor traces of recognisable chrysocolla in the light fraction (rounded grains, composed of pale green radiating fibres, with a high birefringence and moderate refractive index) and none in the heavy.

X-ray analysis of the light fraction gave negative results but this is not surprising as a subsequent assay has now shown that copper was mainly concentrated (90%) in the heavy fraction. The problem is complicated by the disagreement in the literature on the nature of chrysocolla. Until a good standard sample is obtained and documented, positive identification of the mineral is impossible. In the present case either chrysocolla has a density greater than 2.42, or the copper is present in another form.

TABLES 1 TO 25

FIGURES 1 AND 2

TABLE 1: FLOTATION OF BURRA MINES COPPER ORE
Test 1 Conditions

Pulp:	% solids		30
Reagent:	Sodium sulphide	lb/ton	4.0
	Potassium amyl xanthate,	lb/ton	2.0
	Pine oil,	lb/ton	0.15
	Myrtan,	lb/ton	1.0
Conditioning time:		min	5
Rougher flotation time;		min	5
Cleaner flotation time;		min	4
Re-cleaner flotation time;		min	3

TABLE 2: FLOTATION OF BURRA MINES COPPER ORE
Results of Tests 1-5

Test No.	Product	Weight %	Assay Cu, %	Distribution Cu, %
1	Recleaner concentrate	6.5	7.95	30.8
	Recleaner tailing	5.2	2.27	7.0
	Cleaner tailing	25.6	1.53	23.3
	Rougher tailing	62.7	1.04	38.9
	Feed (calc)	100.0	1.68	100.0
2	Recleaner concentrate	2.7	16.70	26.1
	Recleaner tailing	9.3	2.47	13.4
	Cleaner tailing	28.0	1.60	26.1
	Rougher tailing	60.0	0.98	34.4
	Feed (calc)	100.0	1.72	100.0
3	Recleaner concentrate	3.8	11.45	24.8
	Recleaner tailing	3.6	2.52	5.2
	Cleaner tailing	27.0	1.54	23.6
	Rougher tailing	65.6	1.24	46.4
	Feed (calc)	100.0	1.75	100.0
4	Recleaner concentrate	12.9	5.98	44.6
	Recleaner tailing	3.1	2.11	3.8
	Cleaner tailing	15.0	1.61	14.3
	Rougher tailing	69.0	0.95	37.3
	Feed (calc)	100.0	1.73	100.0
5	Recleaner concentrate	11.2	6.32	41.4
	Recleaner tailing	2.6	1.97	3.0
	Cleaner tailing	15.2	1.73	15.3
	Rougher tailing	71.0	0.98	40.3
	Feed (calc)	100.0	1.72	100.0

TABLE 3: FLOTATION OF BURRA MINE COPPER ORE
Test 4 Conditions
Pulp: % solids 30

Stage	Reagents	lb/ton	Conditioning Time min	pH	Flotation Time min
1	Sodium sulphide Potassium amyl xanthate Pine oil Sulphuric acid	2.0 1.0 0.1 0.5	3	7.6	3
2	Sodium sulphide Potassium amyl xanthate Sulphuric acid	2.0 1.0 0.5	2	7.6	2
3	Sodium sulphide Potassium amyl xanthate Sulphuric acid Pine oil	2.0 1.0 0.5 0.1	2	7.6	2

TABLE 4: FLOTATION OF BURRA MINE COPPER ORE

Results of Tests 6-9
Pulp: % solids 30

Test No.	Product	Weight %	Assay Cu, %	Distribution Cu, %
6	Cleaner concentrate	15.8	5.02	46.0
	Cleaner tailing	14.2	1.78	14.5
	Rougher tailing	70.0	0.98	39.5
	Feed	100.0	1.73	100.0
7	Cleaner concentrate	11.8	6.43	44.0
	Cleaner tailing	5.8	2.18	7.3
	Rougher tailing	82.4	1.02	48.7
	Feed	100.0	1.73	100.0
8	Cleaner concentrate	8.1	9.00	41.7
	Cleaner tailing	13.6	1.74	13.5
	Rougher tailing	78.3	1.00	44.8
	Feed	100.0	1.75	100.0
9	Rougher concentrate	19.4	4.25	47.7
	Rougher tailing	80.6	1.12	52.3
	Feed	100.0	1.73	100.0

TABLE 5: FLOTATION OF BURRA MINE COPPER ORE

Test 8 conditions
Pulp: % solids 30

Stage	Reagents	lb/ton	Conditioning Time min	pH	Flotation Time min
1	AC633	1.0	3	7.8	3
	Sodium hydrosulphide	2.0			
	Potassium amyl xanthate	1.0			
	Sulphuric acid	0.5			
	Pine oil	0.1			
2	Sodium hydrosulphide	2.0	2	7.8	3
	Potassium amyl xanthate	0.5			
	Sulphuric acid	0.5			

TABLE 6: SIZING AND DISTRIBUTION OF COPPER
Test 5
Rougher tailing

Screen Fraction Mesh BSS	Weight %	Assay Cu, %	Distribution Cu, %
+ 100	5.2	0.44	2.5
-100 + 200	15.8	0.35	5.9
-200 + 300	14.8	0.37	5.9
-300	64.2	1.23	85.7
Feed (calc)	100.0	0.92	100.0

TABLE 7: SIZING AND DISTRIBUTION OF COPPER
Test 10 Results

Product	Weight %	Assay Cu, %	Distribution Cu, %
Slime fraction	21.6	2.35	29.0
Cleaner concentrate	4.9	12.80	35.8
Cleaner tailing	5.1	1.71	4.9
Rougher tailing	68.4	0.78	30.3
Feed (calc)	100.0	1.76	100.0

TABLE 8: SIZING OF MINUS 52-MESH FEED

Nominal Width of Aperture	Equivalent Mesh	Weight	
µ	BSS	%	Cum. %
300	+ 52	-	-
210	+ 72	3.0	3.0
150	+100	3.5	6.5
105	+150	8.9	15.4
75	+200	6.3	21.7
53	+300	16.0	37.7
	-300	62.3	100.0

TABLE 9: FLOTATION CONDITIONS
Test 11

Stage	Reagent	lb/ton	Conditioning Time min	pH	Flotation Time min
1	Sodium sulphide	2.0	5	7.6	5
	Potassium amyl xanthate	1.0			
	Pine oil	0.1			
	Sulphuric acid	0.5			
	Fuel oil	0.2			
2	Sodium sulphide	2.0	2	7.6	3
	Potassium amyl xanthate	1.0			
	Sulphuric acid	0.5			

TABLE 10: FLOTATION RESULTS
Test 11

Product	Weight		Assay		Recovery	
	%	Cum %	Cu %	Cum %	Cu %	Cum %
Slime fraction	28.8	-	2.60	-	45.1	-
Rougher concentrate (Flotation stages 1 and 2)	22.5	51.3	2.73	2.66	37.0	82.1
Rougher tailing	48.7	-	0.61	-	17.9	-
Feed (calc)	100.0	-	1.66	-	100.0	-

TABLE 11: HEAVY LIQUID SEPARATION
Results

Product Fraction	Weight %	Assay Cu %	Dist Cu %		Liquid Fraction Weight	Cu %	Dist Cu %
Slime	30.0	2.62	45.1	-	-	-	-
+ 200	22.0	1.04	13.2	Heavy	1.2	8.8	6.1
				Light	20.8	0.59	13.2
-200	48.0	1.51	41.7	Heavy	2.2	17.4	22.0
				Light	45.8	0.71	18.7
Feed (calc)	100.0	1.74	100.0				

TABLE 12: HEAVY LIQUID SEPARATION
Combined Slime and Heavy Fractions

Material	Weight %	Cu %	Distribution Cu %
Slime	30.0	2.62	45.1
+200 mesh (heavy)	1.2	8.8	6.1
-200 mesh (heavy)	2.2	17.4	22.0
Composite (calc)	33.4	3.82	73.2

TABLE 13: ORE SORTING
Sample A

Size	Product	Weight	Cu		CO_2	
		%	%	Dist. %	%	Dist. %
-6 +4	Green	2.0	5.6	12.6	2.6	0.5
-4 +3	Green	0.7	3.0	2.4	1.0	0.1
-3 +2	Green	0.8	2.4	2.2	2.5	0.2
-6 +4	Brown	0.9	1.05	1.1	29.1	2.3
-4 +3	Brown	0.8	0.24	0.2	22.5	1.6
-3 +2	Brown	1.7	0.44	0.8	23.0	3.5
-2 +1	Composite	7.4	1.15	9.6	17.8	11.6
-1 +½	Composite	8.8	1.20	11.9	15.8	12.4
-½	Composite	76.9	0.65	59.2	10.0	67.8
Calc head		100.0	0.89	100.0	11.3	100.0

TABLE 14: ATTRITION CONDITIONING
Sample A

Size	Weight	Cu		CO$_2$	
	%	%	Dist %	%	Dist %
-½ , +¼ in.	7.8	1.56	11.4	17.5	12.2
-¼ in. + 6 mesh	9.3	1.78	15.5	17.3	14.4
- 6 +12 mesh	7.8	1.64	12.0	16.8	11.7
-12 +22 mesh	4.4	1.24	5.1	16.2	6.4
-22 +44 mesh	6.7	0.74	4.6	12.5	7.5
-44 mesh (sand)	33.8	0.66	20.8	9.5	28.5
-44 mesh (slime)	30.2	1.08	30.6	7.2	19.3
Calc Head	100.0	1.07	100.0	11.2	100.0

TABLE 15: CYANIDE LEACHING
Results of Tests 1-4
Ore: -200 mesh
Pulp density: 40% solids
Weight of ore: 200 g
Temperature: Ambient

		Leach Number			
		1	2	3	4
NaCN added,	g	4.0	6.0	8.0	8.0
NaCN/Cu ratio of equivalents,		1.7	2.5	3.3	3.3
Time of leach,	hr	4	4	2	4
Cu extracted,	g/l	3.8	4.6	5.0	5.2
Cu extracted,	%	27	36	40	42
NaCN consumed, lb/lb Cu extracted		4.7	5.3	6.4	6.1

TABLE 16: SULPHURIC ACID LEACHING

Results of Tests 1-7
Ore: -200 mesh
Pulp density: 40% solids
Weight of ore: 200 g
Temperature: Ambient

		Leach Number						
		1	2	3	4	5	6	7
H_2SO_4 added,		20	10	10	3.4	4.0	6.8	9.0
Equivalent H_2SO_4 lb/short ton of ore		200	100	100	34	40	68	90
Actual H_2SO_4/theoretical H_2SO_4 required		4.30	2.10	2.10	0.73	0.82	1.50	1.40
Final pH of liquor		0.72	1.37	1.41	3.5(a)	3.5(a)	2.5(a)	2.0(a)
Time of leach,	hr	4	2	4	4	6	4	4
Cu extracted,	g/l	12.5	11.4	12.7	2.9	5.1	7.6	9.3
Cu extracted,	%	91	83	94	23	31	56	68
H_2SO_4 consumed per pound Cu extracted,	lb	7.0	3.8	3.4	4.7	4.2	4.0	4.2

(a) pH maintained throughout leach at this value.

TABLE 17: EFFECT OF PARTICLE SIZE ON LEACHING

Results of Tests 8-10
Weight of ore: 200 g
Pulp density: 40%
Leaching temperature: Ambient
H_2SO_4 added: 10 g
% theoretical for sol of Cu: 190

			Leach Number		
			8	9	10
Particle size,			-200(a)	-52(a)	-¼ in.
Time of leach,		hr	1	4	6
Pulp		pH	1.7	1.5	1.3
Cu extracted,		%	89	91	92
Fe extracted,		%	12	1.5	1.3
Pound H_2SO_4 consumed/lb Cu extracted			3.4	3.1	3.2
Liquor characteristics:	Volume,	ml	500	500	500
	Cu,	g/l	5.93	6.45	6.20
	Fe	g/l	0.94	0.141	0.13
Residue characteristics:	Weight,	g	189	189	188
	Cu,	%	0.199	0.173	0.144
	Fe,	%	1.83	2.50	2.72

(a) Mesh, BSS.

TABLE 18: EFFECT OF AMOUNT OF ACID ON LEACHING
Results of Tests 9, 11, 12

Weight of ore:	200 g
Mesh of ore:	-52
Pulp density:	40%
Leaching temperature:	Ambient

			Leach Number		
			9	11	12
Amount of acid,	g		10	7.5	6.0
Amount of acid,	lb/short ton ore		100	75	60
Amount of acid,	% theoret for sol of Cu		190	140	110
Time of leach,	hr		4	2	2
Pulp,	pH		1.5	1.6	2.2
Cu extracted,	%		91	81	80
Fe extracted,	%		1.5	0.7	0.5
H_2SO_4 added, lb/lb Cu extracted			3.1	2.7	2.4
Liquor characteristics:	Volume,	ml	500	500	500
	Cu,	g/l	6.45	5.47	4.95
	Fe,	g/l	0.141	0.064	0.046
Residue characteristics:	Weight,	g	189	190	188
	Cu,	%	0.173	0.345	0.325
	Fe,	%	2.50	nd(a)	2.27

(a) Not determined.

TABLE 19: EFFECT OF PULP DENSITY ON LEACHING

Results of Tests 13-16
Weight of ore: 200 g
Particle size: -52 mesh
Leaching temperature: Ambient

			Leach Number			
			13	14	15	16
Pulp density,	% solids		40	50	40	50
Amount of acid,	g		7.5	7.5	6.0	6.0
Amount of acid,	lb H_2SO_4/short ton ore		75	75	60	60
Amound of acid,	% theoret for sol. Cu		140	140	110	110
Time of leach,	hr		0.5	0.5	6	6
Pulp,	pH		1.5	1.4	2.3	2.6
Cu extracted,	%		78	77	82	78
Fe extracted,	%		0.7	0.4	0.6	0.5
Pound			2.8	3.0	2.2	217
Liquor characteristics:	Volume,	ml	500	500	500	500
	Cu,	g/l	5.43	5.23	5.37	4.49
	Fe,	g/l	0.061	0.039	0.049	0.040
Residue characteristics:	Weight,	g	190	190	189	174
	Cu,	%	0.40	0.41	0.305	0.364
	Fe,	%	nd[a]	2.50	2.27	2.26

(a) nd - Not determined.

TABLE 20: EFFECT OF LEACHING TIME

Results of Tests 13, 17, 11, 18, 19, 20
Weight of ore: 200 g
H_2SO_4 added: 7.5 g
H_2SO_4 added: % theoret for sol. of Cu, 140
Leaching temp: Ambient

Conditions	Leach Number					
	13	17	11	18	19	20
Leaching time,	½	1	2	½	1	2
Particle size,	-52(a)	-52	-52	-¼(b)	-¼	-¼
Pulp density, % solids	40	40	40	50	50	50
Pulp pH	1.5	1.7	1.6	1.4	1.4	1.4
Cu extracted,%	78	83	81	70	79	77
Fe extracted,%	0.7	0.7	0.7	0.3	0.3	0.3
H_2SO_4 added, lb/lb Cu extracted,	2.8	2.7	2.7	3.2	2.6	3.0
Liquor characteristics: Volume ml	500	500	500	500	600	500
Cu, g/l	5.43	5.63	5.47	4.71	4.88	4.95
Fe, g/l	0.061	0.068	0.064	0.029	0.028	0.030
Residue characteristics: Weight, g	190	191	190	190	190	191
Cu, %	0.40	0.31	0.345	0.52	0.405	0.38
Fe, %	nd(c)	nd	nd	2.54	2.50	2.46

(a) Mesh, BSS.
(b) Inch.
(c) nd - Not determined.

TABLE 21: CONCENTRATE AND ORE LEACHING

Results of Tests 12, 21
Pulp density: 40%
Leaching temp: Ambient
Particle size: -52 mesh

		Leach Number	
		12	21
Material,		ore	concentrate
Weight of material,	g	200	100
Weight of H_2SO_4,	g	6.0	10.0
Amount of H_2SO_4,	% theoret. for sol. of Cu	110	110
Amount of acid,	lb H_2SO_4/short ton of material	60	100
Time of leach,	hr	2	1
Pulp pH		2.2	1.8
Cu extracted,	%	80	90
Fe extracted,	%	0.5	1.3
H_2SO_4 added, lb/lb Cu extracted		2.4	1.8
Liquor characteristics:	Volume ml	500	250
	Cu, g/l	4.95	21.9
	Fe, g/l	0.046	0.10
Residue characteristics:	Weight g	188	89
	Cu %	0.325	0.675
	Fe %	2.27	2.15

TABLE 22: SETTLING TESTS

Results of Tests 1-3
Sp gr ore residue: 3.18
Sp gr leach liquor: 1.01

		Test Number		
		1	2	3
Flocculant		None	S3384	Jaguar MD3
Amount used,	lb/ton solids	-	0.1	0.8
Settling rate,	ft/hr	5.3	25	33
Slurry: weight,	g	547	547	537
volume,	ml	500	500	500
Pulp density: initial,	% solids	12	12	10
final,	% solids	69	71	67
Thickener area,[a]	sq ft solids/24 hr	1.6	0.35	0.35

[a] The thickener area was calculated using the equation -

$$A = 1.33\,(F-D)/RS$$

where A = thickener area in sq ft/ton solids/24 hr

F = Weight ratio of liquid at rate R.

D = Weight ratio of liquid to solid of settled slurry

R = Settling rate, ft/hr

S = Specific gravity of liquor.

TABLE 23: LEACH-PRECIPITATION FLOTATION

Test No.	Product	Weight %	Assay Cu, %	Distribution Cu, %
1	Frother concentrate	3.3	1.42	2.6
	Fe turnings	-	5.75	17.2
	Rougher concentrate	5.5	17.4	54.8
	Routher tailing plus coarse leach residue	91.2	0.49	25.4
	Head[(a)]	-	1.66	-
2	Frother concentrate	3.5	2.44	5.1
	Fe turnings	-	2.55	7.7
	Cleaner concentrate	1.6	68.2	63.8
	Cleaner tailing	2.3	3.64	4.9
	Rougher tailing plus coarse leach residue	92.6	0.34	18.5
	Head[(a)]	-	1.63	-
3	Fe turnings	-	3.17	3.0
	Cleaner concentrate	2.95	30.4	55.6
	Cleaner tailing	4.7	9.40	27.6
	Rougher tailing plus coarse leach residue	92.3	0.24	13.8
	Head[(a)]	-	1.59	-

(a) Calculated.

TABLE 24: LEACHING WITH AMMONIACAL SOLUTIONS
Results of Tests 1-10

Conditions	Test Number									
	1	2	3	4	5	6	7	8	9	10
NH_4OH Normality	1.7	1.7	1.7	1.7	1.7	1.7	1.7	3.2	0.2	1.7
NH_4Cl Normality	0.2	0.8	1.6	3.2	1.6	1.6	1.6	1.6	1.6	-
$(NH_4)_2SO_4$ Normality	-	-	-	-	-	-	-	-	-	0.8
NaCl Normality	-	-	-	-	-	-	-	-	-	-
Solids, w/w	40	40	40	40	40	40	40	40	40	40
Ore size, mesh	-8	-8	-8	-8	-52	-8	-8	-8	-8	-8
Leaching time, hr	1	1	1	1	1	4	24	1	1	1
Cu leached, %	3	15	32	30	34	30	34	25	8	12

TABLE 25: LEACHING WITH AMMONIACAL SOLUTIONS
Results of Tests 11-20

Conditions	Test Number									
	11	12	13	14	15	16	17	18[a]	19[a]	20[a]
NH_4OH Normality	1.7	1.7	1.7	0.2	1.7	1.7	1.7	1.7	1.7	1.7
NH_4Cl Normality	-	-	1.6	-	-	-	-	1.6	1.6	1.6
$(NH_4)_2SO_4$ Normality	3.2	0.8	-	0.8	0.8	-	-	-	-	-
NaCl Normality	-	-	-	0.3	0.3	0.8	1.6	-	-	-
Solids, w/w	40	40	40	40	40	40	40	40	20	nd[b]
Ore size, mesh	-8	-52	-8	-8	-8	-8	-8	-8	-8	-8
Leaching time, hr	1	1	1+1	1	1	1	1	1	1	nd[b]
Cu leached, %	30	26	39[c]	3	12	5	5	34	43	63

(a) Ore heated to 300°C for 1 hour before leaching. Loss of weight on heating = 2.5%.

(b) Extracted repleatedly in Soxhlet apparatus with fresh solution until no further copper was dissolved by prolonged contact with ore.

(c) Residue re-leached under identical conditions (compared with Test 3).

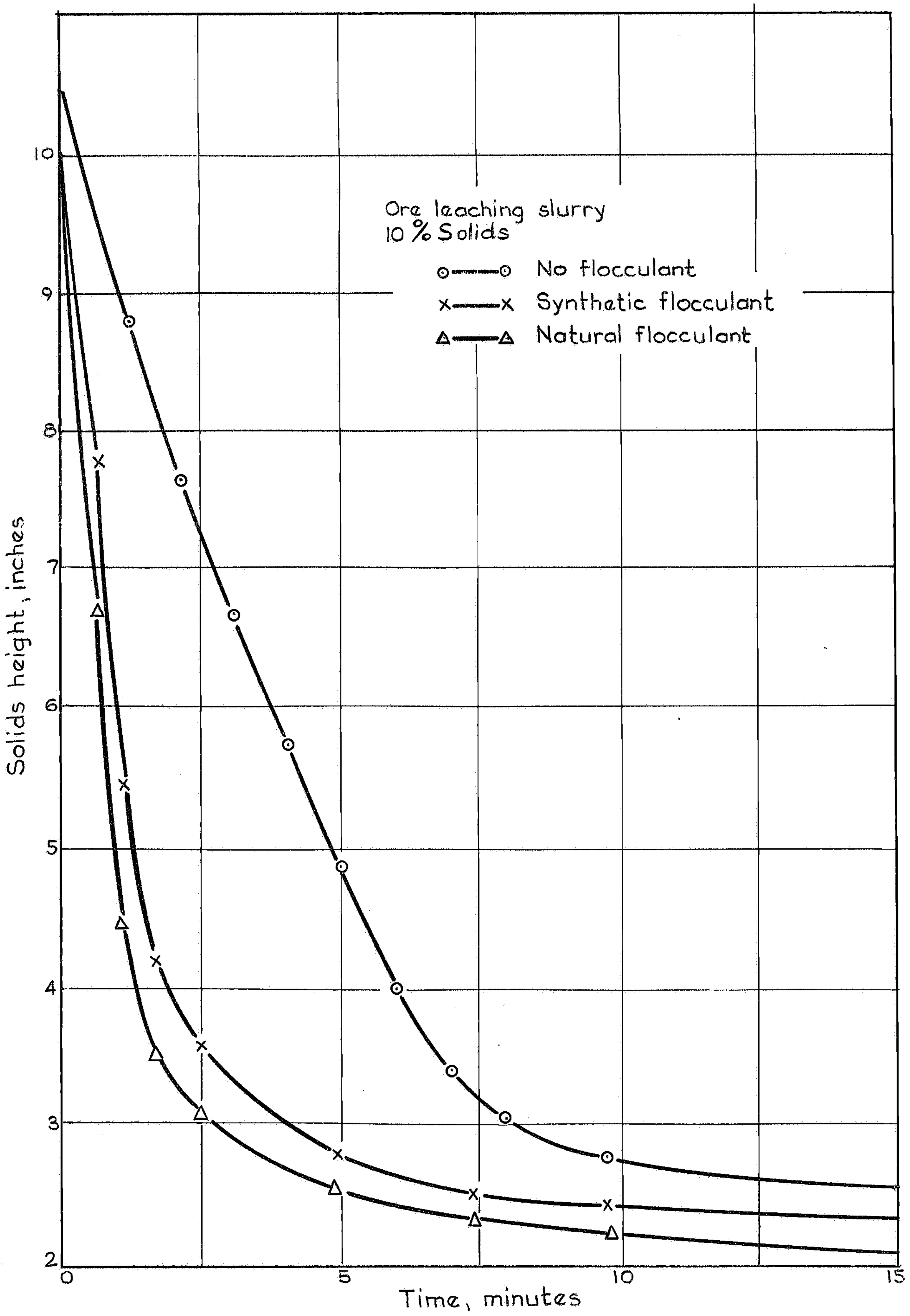

FIG. 1: SETTLING TESTS

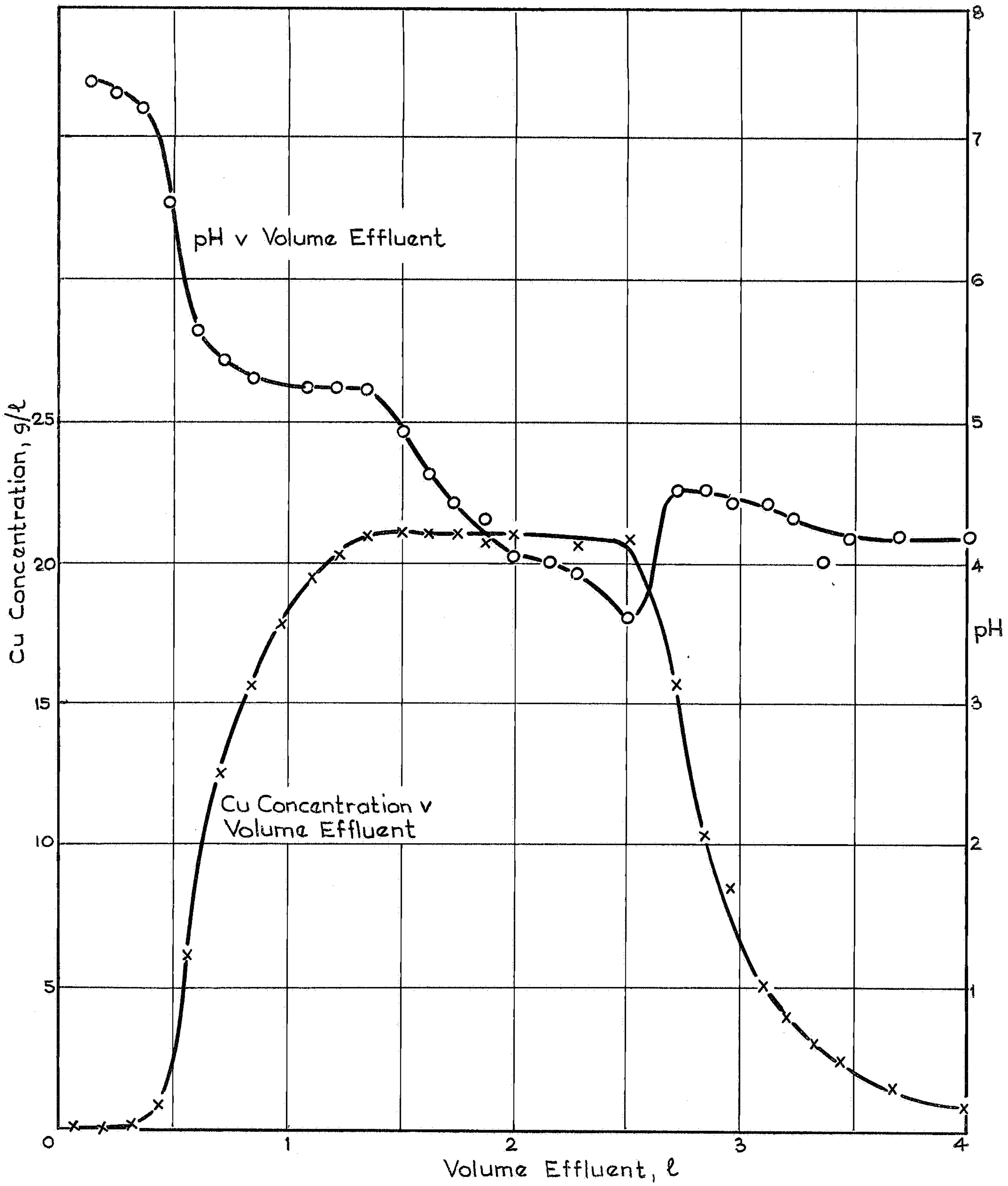

FIG. 2: RELATIONSHIP BETWEEN COPPER CONCENTRATION pH AND VOLUME OF EFFLUENT