# CONTENTS ENVELOPE 6606

TENEMENT: Not related.

TENEMENT HOLDER: Mt Gunson Mines Pty. Ltd.

# ORE DRESSING INVESTIGATIONS.

## JOINT INVESTIGATIONS BY THE COUNCIL FOR SCIENTIFIC AND INDUSTRIAL RESEARCH AND THE SOUTH AUSTRALIAN SCHOOL OF MINES AND INDUSTRIES.

## REPORTS FROM THE BONYTHON LABORATORY.

## NEW SERIES NO. 53. C.S.I.R. NO. 47.

# PERCOLATION LEACHING OF MOUNT GUNSON COPPER ORE.

Following the report from this laboratory, New Series No. 45, C.S.I.R. No. 40, on the treatment of Mount Gunson ore, a second sample of the ore was supplied with a request for the following tests:-

(1) 24 hours' leaching on minus one-eighth inch ore using 3% and 5% sulphuric acid solutions.
(2) Tests to determine whether this ore is amenable to percolation after crushing through one-eighth inch.
(3) Tests to determine what, if any, fouling of circuit liquors may be expected in plant operations.

## NATURE OF ORE:

The second sample of ore supplied for testing appeared to be similar to the previous sample, the copper occurring chiefly as malachite finely disseminated through sandstone.

Assay results on the two samples follow:-

| | No. 1 Sample. | No. 2 Sample. |
|---|---|---|
| Gold | Nil | -- |
| Oxide Copper | 5.1% | 3.5% |
| Sulphide Copper | 0.1% | 0.2% |
| Arsenic | -- | Nil |
| Bismuth | -- | Nil |

For the purpose of leaching tests at minus one-eighth inch sizing these ores were crushed by small size reductions followed by screening over 1/8", the oversize being crushed again and so on until the final products gave the following screen fractions:-

| Mesh | Aperture (inches) | Percentages. | |
|---|---|---|---|
| | | No. 1 Sample. | No. 2 Sample. |
| Plus 5 | .1320 | -- | 0.4 |
| 5/7 | .0949 | -- | 13.5 |
| 7/10 | .0660 | 11.9 | 11.4 |
| 10/14 | .0474 | 6.0 | 8.7 |
| 14/18 | .0336 | 6.8 | 7.6 |
| 18/25 | .0236 | 8.6 | 8.3 |
| 25/36 | .0166 | 10.0 | 10.3 |
| 36/52 | .0116 | 9.9 | 9.2 |
| 52/72 | .0083 | 10.0 | 7.8 |
| 72/100 | .0060 | 10.4 | 7.8 |
| 100/150 | .0041 | 7.8 | 4.4 |
| 150/200 | .0030 | 5.1 | 2.4 |
| Minus 200 | -- | 13.5 | 8.2 |

## GENERAL:

This laboratory accepts no responsibility for results except in so far as they apply to the samples supplied for testing.

The long ton of 2,240 pounds is used throughout this report.

British Standard Sieves were used for screen analyses.

## AGITATION LEACHING:

As requested the minus one-eighth inch ore was leached by agitation for 24 hours with 3% and 5% sulphuric acid, Pernatty Lake water being used for making up the acid solutions.

Results are given in the following tabulation:-

| Leaching Time Hours. | Acid Strength % $H_2SO_4$ | % Copper in tailing. | $H_2SO_4$ used lb/ton. | Weight of tailing as % age of feed. | Copper Recovery %. |
|---|---|---|---|---|---|
| No. 1 Sample:- | | | | | |
| 24 | 3 | 0.13 | 210 | 88 | 99 |
| 24 | 5 | 0.05 | 226 | 88 | 99 |
| No. 2 Sample:- | | | | | |
| 24 | 3 | 0.06 | 125 | 91.7 | 98 |
| 24 | 5 | 0.06 | 127 | 91.7 | 98 |

In each case the ore was agitated by rolling in glass jars with four parts of dilute acid at room temperature of about 75°F.

The acid strengths given are the percentages of 100% sulphuric acid in the starting solution; no more acid was added during agitation so that the strength of the solution would fall progressively as the acid reacted with the constituents of the ore.

Acid consumption figures all relate to pounds of 100% sulphuric per long ton of ore. Theoretically the dissolution of 5.1% copper would require 180 lb. sulphuric acid per ton so that the figures for No. 1 sample show an additional consumption of 30 to 46 pounds acid per ton. Similarly, the amount of acid required to dissolve 3.64% copper is theoretically 127 pounds per ton, which checks the observed figures for No. 2 Sample, showing that Mount Gunson ore similar to the second sample contains no acid-consuming constituents except the copper minerals.

## PERCOLATION LEACHING:

30½ pounds of the minus one-eighth inch Mount Gunson No. 2 Sample was placed in a 4 inch tube, filling it to a depth of 4 feet. This was leached with a 5% solution of sulphuric acid in Pernatty Lake water, the first solution being added from below.

The test was kept going for four days, during which time the average rate of percolation was less than 100 c.c. per hour, at which rate six days would be required to pass the amount of acid theoretically required to dissolve the copper from this sample. This rate being too slow the test was discontinued.

All the No. 2 Sample of Mount Gunson ore having been used, further testing was confined to the No. 1 Sample.

No. 1 Sample as received (minus ½ inch) was treated by percolation, 29 pounds of the ore giving a depth of 3 feet 9 inches in a four-inch tube.

The percolation rate was first tested by means of tap water which was added from below, then the water drained off at the rate of 1500 c.c. per hour.

Following this reading a 5% solution of sulphuric acid in Pernatty Lake water was added, the initial drainage rate with acid, was 310 c.c. per hour, dropping to 160 c.c. per hour on the second day, then increasing steadily to 2300 c.c. per hour on the third day, at which point the discharge was throttled to give the acid longer contact with the ore.

The tabulation below gives a complete record of the test:-

| Time from start. Days. | 5% acid added c.c. | Solution drained c.c. | Copper Solution Assays. Amount sampled c.c. | % $H_2SO_4$ | Copper grams per liter. |
|---|---|---|---|---|---|
| -- | 4000 | -- | | | |
| 1 | 6500 | 5610) | | | |
| 2 | 4500 | 4570) | 13150 | 0.05 | 21.5 |
| 3 | 10500 | 10150 ) | 2400 | 0.6 | 23.5 |
| | | | 2540 | 2.0 | 14.3 |
| | | | 1430 | 2.5 | 12.0 |
| 4 | 1500 | 1520 | | | |
| 6 | 2400 | 2260 | 3540 | 0.2 | 22.4 |
| 7 | 2500 | 2620 | 2620 | 1.7 | 16.0 |
| 8 | 2400 | 2060 | 2060 | 2.55 | 14.5 |
| 9 | 2500 | 2410 | 2410 | 3.4 | 8.6 |
| 10 | 5000 | 2560 | 2560 | 3.5 | 5.7 |
| | | 1870 | 1870 | 4.1 | 2.4 |
| | Wash water | 5450 | 5450 | 4.3 | 1.6 |
| 10 | 8000 | 7840 | 7840 | 0.6 | 3.4 |

The quantities of solution added and drained do not check perfectly due to a small evaporation loss.

From the foregoing figures it is seen that 41,800 c.c. of 5% acid were added, plus 8,000 c.c. wash water, equivalent to 2090 grams sulphuric acid, and 48,920 c.c. of solution were drained containing 742 grams acid, showing an acid consumption of 1348 grams by 13082 grams ore, which is equivalent to 231 pounds sulphuric acid per long ton of ore leached.

The percolation tailing assayed 0.4% copper and weighed 91% of the feed to percolation, showing an extraction of 93% of the copper by percolation of minus one-half inch Mount Gunson ore for 10 days with 5% sulphuric acid in Pernatty Lake water.

Examination of the percolation tailing showed the presence of compact patches of slime apparently high in copper; assays of the plus and minus 100 mesh fractions of the tailing both gave 0.4% copper which indicates the presence of larger particles of copper mineral contained in the compact slime.

The first solutions drained from the percolation test were alkaline to methyl orange, so that in practice the acidity of solutions could be reduced to practically nil for scrap iron precipitation of the copper.

Screen analyses of the feed and tailing of the percolation test follow:-

| Mesh | Aperture (inches). | Percentages. Percolation Feed | Percolation Tailing |
|---|---|---|---|
| Plus 3/8" | .3750 | 1.8 | 1.9 |
| 3/8/1/4 | .2500 | 14.7 | 9.2 |
| 1/4/3/16 | .1875 | 14.3 | 10.9 |
| 3/16/5 | .1320 | 10.4 | 5.2 |
| 5/7 | .0949 | 7.1 | 5.2 |
| 7/10 | .0660 | 6.0 | 4.5 |
| 10/14 | .0474 | 5.3 | 4.7 |
| 14/18 | .0336 | 5.4 | 5.8 |
| 18/25 | .0236 | 6.4 | 8.4 |
| 25/36 | .0166 | 6.4 | 8.9 |
| 36/52 | .0116 | 5.3 | 8.0 |
| 52/72 | .0083 | 4.6 | 7.2 |
| 72/100 | .0060 | 3.7 | 5.3 |
| 100/150 | .0041 | 2.6 | 3.3 |
| 150/200 | .0030 | 1.5 | 1.7 |
| Minus 200 | -- | 4.5 | 9.8 |

These screen analyses show the production of more fine material during the percolation leaching of Mount Gunson ore. The amount of fine material in the percolation feed is about the least which could be expected from crushing of the ore in closed circuit with a screen and maintaining a high circulating load.

On minus one-half inch ore agitation treatment of the fine material and percolation of the coarse fraction may be desirable, while, if the ore is to be treated after finer crushing, removal of the fine portion for treating separately would be necessary. In this connection the minus [illegible]0 mesh fraction of minus $\frac{1}{8}$" Mount Gunson Ore was found to assay 7.4% copper compared with 5.2% copper in the total sample.

FOULING OF SOLUTION:

The 12% loss in weight on leaching the copper from No. 1 Sample and the 8.3% loss in weight on No. 2 Sample do not necessarily indicate fouling of the solution. Apparently most of the copper in the samples of ore tested existed as malachite, the carbon dioxide and water from which would also represent a loss in weight during leaching. A 12% loss in weight in dissolving 5.1% copper from the ore would be equivalent to a copper mineral containing 42% copper. Similarly an 8.3% loss in weight in dissolving 3.64% copper from the ore would be equivalent to a copper mineral containing 44% copper. The text-book figure for the copper content of malachite is 40.3%, with which the foregoing figures agree reasonably well.

A small amount of iron was noticed in the copper solutions in addition to the salts present in the saline water used.

Electrolytic precipitation of copper from these solutions would be out of the question but they should be suitable for precipitation by scrap iron.

## SUMMARY.

Two samples of minus one-eighth inch Mount Gunson Copper Ore were treated by agitation for 24 hours with 3% and with 5% sulphuric acid in Pernatty Lake water. The first sample, assaying 5.2% copper, gave recoveries of 99% of the copper with respective consumptions of 210 and 226 pounds 100% sulphuric acid per long ton of ore treated. The second sample, assaying 3.7% copper, gave recoveries of 98% ofthe copper with consumptions of 125 and 127 pounds 100% sulphuric acid per long ton of ore treated. This second sample consumed only the amount of acid theoretically required to dissolve the copper.

Percolation leaching of minus one-eighth inch ore is too slow unless the fine material is removed for separate treatment.

Leaching minus one-half inch Mount Gunson ore by percolation with 5% sulphuric acid gave a 93% recovery of the copper in ten days, the acid consumption being 231 pounds per long ton of ore leached.

The copper-bearing solutions should be suitable for scrap-iron precipitation.

(Signed) L. M. ABELL,

ORE DRESSING INVESTIGATOR.

" H. W. GARTRELL,

OFFICER-IN-CHARGE.

12th November, 1937.

**CSIRO**

ORE DRESSING INVESTIGATIONS, MELBOURNE

MELBOURNE UNIVERSITY ORE DRESSING LABORATORY, MINING DEPT., UNIVERSITY, PARKVILLE, N.2, VIC. TELEPHONE 341338, 342274

6502 16th February, 1965

Mr. H. B. Evans,
Managing Director,
Chapman, Wood, Griswold and Evans Pty. Ltd.,
Suite 20,
541 St. Kilda Road,
MELBOURNE.

Dear Mr. Evans,

We have now completed a percolation test on the sample of copper ore you supplied from Mount Gunson. I have noted the major features of the work below and will forward more complete details later.

## Head assay

In view of the small weight of sample supplied, and the need to conserve most of it at a coarse size for the percolation test, sampling could not be done accurately. However, after modifying our usual procedure in several ways the following figure was obtained.

Head assay 8.4% Cu

This agreed well with the calculated figure below. There was a fair amount of gypsum in the sample but only small amounts of calcite were noted.

## Test procedure

The sample was crushed minus 1/2 in. and placed in a 4 in. diameter tube giving a bed depth of 8½ in. Batches (2000 ml) of 5 per cent sulphuric acid were run on to the charge and allowed to stand for 1-2 hr in the early stages of the test and 24-72 hr in the later stages. The total treatment time was about 8 days. The percolation rate was relatively fast when running solution on, and when draining the bed. Copper and acid determinations were conducted on each effluent solution.

## Copper extraction

| | |
|---|---|
| Residue | 82.6% of original weight assaying 0.44% Cu |
| Calculated | feed assay (based on copper in effluent solutions and residue) 8.40% Cu |

Cu Extraction %, 95.7

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**Acid consumption**

Total acid consumption for the test was as follows.

347 lb per ton of ore

1.93 lb per lb copper extracted

The theoretical acid consumption for 8.4% Cu as $CuCO_3$ is 288 lb/ton.

**Conclusion**

The results of the test are favourable in that a high copper extraction was obtained for a reasonable acid consumption. Further tests are of course needed to investigate the effect of variation in the more important variables. These should be done on a more representative sample of ore.

Yours sincerely,

J. Woodcock.

J. T. Woodcock,
SENIOR RESEARCH SCIENTIST

# ORE DRESSING INVESTIGATIONS.

## JOINT INVESTIGATIONS BY THE COUNCIL FOR SCIENTIFIC AND INDUSTRIAL RESEARCH AND THE SOUTH AUSTRALIAN SCHOOL OF MINES AND INDUSTRIES.

## REPORTS FROM THE BONYTHON LABORATORY.

## NEW SERIES NO. 45. C.S.I.R. NO. 40.

## TREATMENT OF OPEN CUT COPPER ORE FROM MOUNT GUNSON, SOUTH AUSTRALIA.

### SOURCE OF SAMPLE:

A 115 lb. sample was received from Austral Development Proprietary Limited and was said to be representative of ore from the Mount Gunson-Pernatty Copper Deposit.

Testing was required to determine the most suitable and economical method of treatment of ore similar to the sample supplied.

### NATURE OF ORE:

A description of the ore was forwarded by Mr. A. J. Keast, and follows:-

"The ore consists of chalcocite and green oxidised copper minerals, chiefly chrysocolla, atacamite and malachite in veins and disseminations through sandstones, grits and quartzites.

A little cobalt is present in the ore but no other deleterious elements such as arsenic or bismuth are present."

The sample as received was all minus ½ inch and assayed as follows:-

| | |
|---|---|
| Gold | Nil |
| Oxide Copper | 5.1% |
| Sulphide Copper | 0.1% |

### GENERAL:

This laboratory accepts no responsibility for the accuracy of the sample supplied.

The long ton of 2240 lbs. is used throughout.

British Standard Sieves were used for all screen analyses.

The flotation cell used was a Ruth type having a capacity of 1800 c.c.

### FLOTATION:

Preliminary examination of the sample indicated that flotation was unlikely to give a satisfactory concentration of the copper, as much of the oxidised copper occurs in a very finely disseminated form throughout the silica and a considerable part of the valuable minerals would not be liberated by a reasonable degree of grinding. As flotation tests had been requested, floats under varying conditions were tried.

Oleic acid and sodium oleate with pine oil as frother and at pH values from 7 to 10 gave very poor flotation of the copper from this sample.

The use of quantities of amyl xanthate or Reagent 425 in the order of 2 to 3 pounds per ton resulted in the production of a heavy copper froth which appeared to be similar whether or not the ore were previously sulphidized.

Results from a typical test under these conditions are given below:-

| | % Weight. | Assays.<br>% Cu. | Recoveries.<br>% Cu. |
|---|---|---|---|
| Concentrate | 11.5 | 20.7 | 46 |
| Middling | 5.5 | 7.7 | 8 |
| Tailing | 83.0 | 2.9 | 46 |
| | 100.0 | 5.2 | 100 |

A screen analysis of the tailing follows:-

| | Mesh | Aperture (inches). | Percentages. |
|---|---|---|---|
| Plus | 150 | .0041 | 2.4 |
| | 150/200 | .0030 | 10.5 |
| Minus | 200 | - | 87.1 |

Conditions of test were as follows:-

**Grind:-** 60 minutes.
**Float:-** 30 minutes.
0.4 lb/ton Aerofloat 25
2.4 " Reagent 425
**Conditioning:-** 10 minutes.
0.5 lb/ton Sodium Sulphide.
**Middling:-** 15 minutes.
0.1 lb/ton Aerofloat 25
0.2 " Reagent 425
pH of tailing 7.5.

This tailing was again conditioned with sodium sulphide, Aerofloat 25 added to give a good froth, and 1.5 lb/ton Reagent 425 did not float any more copper.

Evidently flotation is unsuitable on Mount Gunson Ore similar to the sample tested.

The same reasons which prevent satisfactory flotation would also make any method of concentration by classification of doubtful value.

**LEACHING:**

Following general practice on ores of this type its suitability for sulphuric acid leaching was determined by testing.

In all cases pulps consisting of four parts of the dilute acid to one part of ore were agitated by rolling in glass jars for varying periods of time at room temperature of about 70°F.

| Leaching time hours. | Acid Strength % $H_2SO_4$ | Salt in solution % | - 1/2" ore. | | -1/8" ore. | | - 1/12" ore. | |
|---|---|---|---|---|---|---|---|---|
| | | | % Cu in tailing. | $H_2SO_4$ used lb/ton. | % Cu in tailing. | $H_2SO_4$ used lb/ton. | % Cu in tailing. | $H_2SO_4$ used lb/ton. |
| 2 | 5 | 10 | | | | | 0.2 | 237 |
| 4 | 5 | 10 | | | | | 0.1 | 244 |
| 6 | 5 | 10 | | | | | 0.1 | 244 |
| 6 | 3 | 10 | | | | | 0.2 | 212 |
| 6 | 5 | -- | | | 0.2 | 233 | | |
| 18 | 3 | 10 | 0.5 | 216 | | | | |
| 18 | 3 | -- | 0.6 | 180 | | | | |
| 24 | 5 | -- | | | 0.1 | 240 | | |
| 24 | 5 | 10 | 0.3 | 249 | | | | |
| 24 | 3 | 10 | 0.4 | 216 | | | | |
| 48 | 5 | -- | 0.2 | 240 | | | | |
| 96 | 5 | -- | | | 0.1 | 228 | 0.1 | 228 |
| 96 | 5 | 5 | | | | | 0.1 | 234 |
| 96 | 3 | -- | | | | | 0.1 | 220 |

Screen analyses of the ore after crushing to the varying sizings were as follows:-

| Mesh. | Aperture (inches) | - 1/2" ore. | - 1/8" ore. | - 1/12" ore. |
|---|---|---|---|---|
| Plus 3/8" | .3750 | 1.8% | | |
| 3/8 / 1/4" | .2500 | 14.7 | | |
| 1/4 / 3/16" | .1875 | 14.3 | | |
| 3/16 / 5 | .1320 | 10.4 | | |
| 5/7 | .0949 | 7.1 | | |
| 7/10 | .0660 | 6.0 | 11.9% | |
| 10/14 | .0474 | 5.3 | 6.0 | 5.8% |
| 14/18 | .0336 | 5.4 | 6.8 | 7.2 |

| Mesh. | Aperture (inches) | -1/2" ore. | -1/8" ore. | -1/12" ore. |
|---|---|---|---|---|
| 18/25 | .0226 | 6.4 | 8.6 | 9.0 |
| 25/36 | .0166 | 6.4 | 10.0 | 11.2 |
| 36/52 | .0116 | 5.3 | 9.9 | 12.0 |
| 52/72 | .0083 | 4.6 | 10.0 | 12.5 |
| 72/100 | .0060 | 3.7 | 10.4 | 12.4 |
| 100/150 | .0041 | 2.6 | 7.8 | 9.3 |
| 150/200 | .0030 | 1.5 | 5.1 | 5.3 |
| Minus 200 | --- | 4.5 | 13.5 | 15.3 |

The acid strengths given are the percentages of 100% sulphuric acid in the starting solution, no more acid was added during agitation so that the strength of the solution would fall progressively as the acid reacted with the constituents of the ore.

Acid consumption figures all relate to pounds of 100% sulphuric per long ton of ore. Theoretically the dissolution of 5.1% copper would require 178 pounds sulphuric acid per ton so that the figures given in tabulation show an additional consumption of 50 or 60 pounds acid per ton.

In every case the leach tailing weighed 88% of the feed to leaching.

This ore appears to be very suitable for sulphuric acid leaching, giving a leach tailing assaying 0.1% to 0.2% copper with an acid consumption of 240 pounds per ton. Time of contact of acid with ore which is required varies with fineness of crushing of the ore and initial strength of the acid. A starting strength of 3% in place of 5% sulphuric acid results in a small reduction in acid consumption and an increase in time required for the dissolution of the copper.

In a number of tests common salt was added to the tap water used for diluting the acid, as it was understood that the water available for treatment of the ore is highly saline. Salt appears to have no definite effect on either acid consumption or on time of leaching required.

Since these tests were completed a sample of Lake Pernatty water has been received and was used for laboratory tests. Results of one test on minus one-eighth inch Mount Gunson Ore using Pernatty Lake water are given below, together with a comparative test using Adelaide tap water to which no salt had been added:-

| Leaching time. hours. | Acid Strength % $H_2SO_4$. | Tailing assay % Cu. | Sulphuric acid consumed lb/ton. |
|---|---|---|---|
| Adelaide tap water: | | | |
| 6 | 5 | 0.2 | 233 |
| Pernatty Lake water: | | | |
| 6 | 5 | 0.2 | 212 |

These tests show similar results whether fresh water or saline water is used.

An analysis of the water supplied follows:-

| | | |
|---|---|---|
| Chlorine, Cl | 9561 | grains per gallon |
| Sulphuric acid (radicle) $SO_4$ | 219 | " |
| Carbonic acid (radicle) $CO_3$ | 8 | " |
| Nitric acid (radicle) $NO_3$ | Nil | " |
| Sodium, Na | 5028 | " |
| Calcium, Ca | 56 | " |
| Magnesium, Mg | 645 | " |
| Total saline matter | 15,517 | grains per gallon |
| " " " | 35.5 | ounces per gallon. |

Assumed composition of salts:-

| | | |
|---|---|---|
| Calcium carbonate | 13 | grains per gallon. |
| Calcium sulphate | 172 | " " " |
| Magnesium sulphate | 123 | " " " |
| Magnesium chloride | 2429 | " " " |
| Sodium chloride | 12780 | " " " |

pH 8.3.

## SUMMARY.

This laboratory accepts no responsibility for the accuracy of the sample of Mount Gunson Ore received for testing.

Flotation of this sample does not give a satisfactory recovery of the copper.

The ore is suitable for dilute sulphuric acid leaching, resulting in the recovery of up to 99% of the copper with an acid consumption of 240 pounds per ton of ore leached, this quantity being about four-thirds of the acid theoretically required to dissolve the copper.

Time of leaching necessary varies with fineness of crushing of the ore, so that with a starting strength of 5% sulphuric acid a 98% copper extraction is given in over 24 hours on minus one-half inch ore, in 6 hours on minus one-eighth inch ore, and 2 hours on minus one-twelfth inch ore, acid consumptions being respectively 240, 233 and 237 pounds per ton.

Leaching time is longer and acid consumption lower when starting strength is 3% than with 5% sulphuric acid.

Common salt in the leaching solution in quantities up to saturation point have no definite effect on copper recovery, leaching time or acid consumption. Water from Pernatty Lake would be suitable for leaching this ore.

The rate of leaching increasing considerably with small increase in fineness of crushing of the ore the most economical size for leaching would probably be in the order of one-eighth inch to one-twelfth inch, at which size it could readily be treated by percolation and this size is easily produced by roll-crushing.

Final decision between the possible conditions of treatment must depend on a number of factors, details of which were not supplied.

(Signed) L. M. ABELL,

ORE DRESSING INVESTIGATOR.

" H. W. GARTRELL,

OFFICER-IN-CHARGE.

23rd August, 1937.

COMMONWEALTH OF AUSTRALIA

COMMONWEALTH SCIENTIFIC AND INDUSTRIAL RESEARCH ORGANIZATION

AND THE

MINING DEPARTMENT, UNIVERSITY OF MELBOURNE

# ORE-DRESSING INVESTIGATIONS

REPORT No 663

# ACID LEACHING OF OXIDIZED COPPER ORE FROM MOUNT GUNSON, SOUTH AUSTRALIA

by

J.T.WOODCOCK

Melbourne
July,1966

# REPORT No 663

# ACID LEACHING OF OXIDIZED COPPER ORE FROM MOUNT GUNSON, SOUTH AUSTRALIA

## Abstract

Three samples of an oxidized copper ore were investigated. These contained malachite, chrysocolla, and atacamite in a siliceous gangue, with some calcite.

Sample A, which was an unrepresentative sample of high grade ore (8 per cent copper) was treated by percolation leaching of minus ½ in. ore. About 96 per cent of the copper was extracted in 7 days treatment with a sulphuric acid consumption of 363 lb/ton (2.0 lb/lb Cu dissolved).

Sample B assayed 1.6 per cent copper and was derived from percussion drill cuttings. Only 80 per cent of the copper was dissolved in 1 hr leaching with sulphuric acid of minus 10 mesh material, and longer leach times or finer grinding did not improve this.

Sample C assayed 1.3 per cent copper, and was a representative sample of percussion drill cuttings from a number of drill holes. Acid leaching of minus 14 mesh ore extracted 90 per cent of the copper in 2 hr, and longer leaching times or finer grinding gave no significant benefit. Acid consumption was about 60 lb/ton (2.3 lb/lb Cu dissolved).

A leach-precipitation-flotation test on Sample C recovered 70 per cent of the copper in a concentrate assaying 21 per cent copper, and this result could probably be improved.

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## INTRODUCTION

Three samples of an oxidized copper ore from Mount Gunson, South Australia, were submitted by Chapman, Wood, Griswold & Evans Pty. Ltd. with the request that we conduct a preliminary examination of the samples to determine the copper extraction and acid consumption during sulphuric acid leaching. One leach-precipitation-flotation test was also conducted.

Sample A, the first sample submitted, was known to be high grade, but was not representative of the orebody. The main purpose of working on this sample was to check the results of work conducted in 1937 by the Bonython laboratory, Adelaide (New Series Reports No 40 and 47).

Sample B had been prepared from percussion drill cuttings taken from the first few holes drilled.

Sample C was also composed of percussion drill cuttings, but had been prepared from drill holes throughout the volume of mineralization and was the most representative sample available in regard to grade and location of drill holes.

Note: No responsibility is taken for the results in this report except insofar as they apply to the samples investigated. Reagent consumptions are based on long tons (2240 lb). The screens mentioned are from the British Standard sieve series.

## HEAD SAMPLES

### HEAD ASSAYS

Analyses conducted on the samples submitted are set out below.

| | Sample A | Sample B | Sample C |
|---|---|---|---|
| Cu (total), % | 8.4 | 1.6* | 1.31 |
| Cu (oxide), % | n.d. | n.d. | 1.23 |
| S (total), % | n.d. | n.d. | 0.10 |
| S (sulphate), % | n.d. | n.d. | 0.08 |

n.d. Not determined
* Calculated from test products.

### NATURE OF SAMPLES

Generally speaking the samples consisted of green oxidized copper minerals disseminated in sandstones and quartzite. In Sample A the main copper mineral appeared to be malachite, but some atacamite and chrysocolla may have been present. The nature of the copper minerals in the other samples was not determined, but green particles picked out of the ore dissolved in dilute sulphuric acid and most of them effervesced while dissolving, so that they were probably carbonates.

Sample A contained numerous gypsum crystals but little calcite was observed.

### SAMPLE PREPARATION

Sample A was received in the form of lump ore (3700 g).

The whole of the sample was crushed minus ½ in., and about one-eighth riffled out and pulverized to provide a sample for head assay. The remainder of the minus ½ in. ore was used in a percolation test.

Sample B was received as 887 g of percussion drill cuttings. The sample was minus 10 mesh and was riffled into 250g and 100g lots for investigation. No direct head assay was conducted on this sample.

Sample C was received as 49 lb of percussion drill cuttings. The maximum particle size present was about ⅜ in., and about two-thirds of the sample was plus 14 mesh. The sample was crushed minus 14 mesh and then riffled into 250g lots for investigation. One 250g portion was pulverized to provide a sample for head assay.

## PERCOLATION LEACHING

## OF SAMPLE A

As noted previously a percolation test only was conducted on Sample A. The whole of the crushed ore remaining after removing the assay sample (3069g of minus ½ in. ore) was carefully placed in a 4 in. diameter tube, so as to minimize size segregation, giving a bed depth of 8½ in.

Details of percolation are shown in Table 1. The first step was to run 2000 ml. of 5 per cent sulphuric acid onto the charge by upward percolation, and to allow it to stand for 1 hour. Thereafter the standard procedure was to displace the solution in the bed by a fresh batch of 5 per cent acid, using downward percolation, and to then allow the acid to remain in contact with the bed for a convenient time that resulted in a relatively blue solution. In the early stages of the run the contact time varied from 1 to 2 hours, but in the later stages it was 24-64 hours.

Copper and free acid determinations were made on effluent solutions as shown in Table 1. Copper determinations were made by a variation of the iodide-thiosulphate method. Free acid determinations were made by titrating an aliquot of the solution with N/1 caustic soda to the first permanent precipitate. When the solution was low in copper, pH paper which changed colour at pH 4-4.5 was used as an indicator.

When the amount of copper dissolved per cycle had fallen to a relatively low proportion of the total, the bed was given four 2000 ml. water washes. The residue was then dried, weighed, sampled, and assayed. Some undissolved particles of green copper mineral were observed in the bed, and these probably occurred in parts where there had been some segregation of slimes which were not penetrated properly by the leach solution.

The total time involved in the test was about 7 days. Details of copper extraction and acid consumption are shown in Table 2.

## DISCUSSION

These results show that after 173 hours treatment (7 days) nearly 96 per cent of the copper was extracted. Most of the copper (i.e. about 90 per cent) was extracted in the first 3 days of treatment. On the remaining 4 days the copper extraction was equivalent to a little over 1 per cent per day. A higher extraction could have been obtained with longer treatment since there were some undissolved, but soluble copper minerals in the residue.

Table 1. Details of percolation leach on Sample A

| Solution added |  | Effluent solution |  |  |  | Contact time |
|---|---|---|---|---|---|---|
| Vol. ml. | $H_2SO_4$ g/l. | Prod. No. | Vol. ml. | $H_2SO_4$ g/l. | Cu g/l. | hr |
| 2000 | 50.0 |  |  |  |  |  |
| 2000 | 50.0 | 1 | 1910 | 18.7 | 19.4 | 1 |
| 2000 | 50.0 | 2 | 1980 | 20.4 | 19.4 | 2 |
| 1970 | 50.0 | 3 | 1960 | 24.8 | 16.1 | 2 |
| 2000 | 50.0 | 4 | 2000 | 14.2 | 24.7 | 16 |
| 2000 | 50.0 | 5 | 1980 | 29.3 | 13.9 | 8 |
| 2000 | 50.0 | 6 | 1980 | 30.4 | 12.9 | 16 |
| 1960 | 50.0 | 7 | 1940 | 41.0 | 5.8 | 8 |
| 2000 | 50.0 | 8 | 1990 | 41.0 | 6.3 | 24 |
| 2000 | 50.0 | 9 | 1950 | 41.9 | 4.9 | 64 |
| 2000 | Nil | 10 | 1980 | 43.0 | 1.6 | 24 |
| 2000 | Nil | 11 | 2000 | ) 2.9 | ) 0.1 | 2 |
| 2000 | Nil | 12 | 2000 | ) | ) | 2 |
| 2000 | Nil | 13 | 2020 | ) | ) | 2 |
| - |  | 14 | 1620 | ) | ) | 2 |

Feed weight 3069 g — Bed diameter 4 in.

Residue weight 2535 g — Bed depth $8\frac{1}{2}$ in.

Table 2. Metallurgical results of percolation on Sample A

| Product No. | Contact time hr |  | $H_2SO_4$ consumption |  |  |  | Cu extraction % |  |
|---|---|---|---|---|---|---|---|---|
|  |  |  | lb/ton |  | lb/lb Cu* |  |  |  |
|  | Stage | Prog. | Stage | Prog. | Stage | Prog. | Stage | Prog. |
| 1 | 1 | 1 | 120 | 120 | 4.4 | 4.4 | 14.4 | 14.4 |
| 2 | 2 | 3 | 44 | 164 | 1.6 | 3.0 | 14.9 | 29.3 |
| 3 | 2 | 5 | 36 | 200 | 1.6 | 2.6 | 12.2 | 41.5 |
| 4 | 16 | 21 | 52 | 252 | 1.4 | 2.2 | 19.1 | 60.6 |
| 5 | 8 | 29 | 31 | 283 | 1.5 | 2.1 | 10.7 | 71.3 |
| 6 | 16 | 45 | 29 | 312 | 1.6 | 2.0 | 9.9 | 81.2 |
| 7 | 8 | 53 | 13 | 325 | 1.6 | 2.0 | 4.4 | 85.6 |
| 8 | 24 | 77 | 13 | 339 | 1.5 | 2.0 | 4.9 | 90.5 |
| 9 | 64 | 141 | 13 | 352 | 1.9 | 2.0 | 3.7 | 94.2 |
| 10 | 24 | 165 | 11 | 363 | 4.8 | 2.0 | 1.2 | 95.4 |
| 11-14 | 8 | 173 | - | 363 | - | 2.0 | 0.3 | 95.7 |

* lb $H_2SO_4$ per lb copper dissolved

Residue assay 0.44% Cu

Residue weight 82.6% of original feed

Calculated feed assay 8.4% Cu

The sulphuric acid consumption was 363 lb/ton for the duration of the test and this was equivalent to 2.0 lb/lb Cu dissolved. The theoretical acid consumption for dissolving $CuCO_3$ is about 1.5 lb/lb Cu dissolved. In the first hour of treatment the acid consumption was much higher than the theoretical figure, indicating acid attack on gangue minerals (probably other carbonates). Thereafter the acid consumed was only a little over the theoretical figure.

For dissolution of 95.7 per cent of the copper present (assuming it present as $CuCO_3$) the theoretical consumption is 276 lb/ton, indicating that about 87 lb/ton was consumed in dissolving gangue minerals. This was a little higher than the consumption obtained by the Bonython laboratory, but generally speaking the results are similar.

The percolation rate was relatively high during both upward and downward percolation, but could be slower in a deeper bed. Nevertheless the results as a whole indicate that ore of this nature can be satisfactorily treated by percolation leaching with sulphuric acid.

## AGITATION LEACHING

## TESTS ON SAMPLE B

Three agitation leaching tests were conducted on Sample B. Each test was conducted at a liquid/solid ratio of 3/1 by agitation with a variable speed stirrer in a glass beaker. Test 352 was conducted by agitation of 250 g of minus 10 mesh material, Test 353 with 100 g of minus 10 mesh material, and Test 354 with 100 g of material that had been dry ground minus 100 mesh with a mortar and pestle. Because of differences in sizing and total volume, stirring conditions were not identical in each test.

Table 3. Agitation leach details for Sample B

Liquid/solid ratio 3/1

Theoretical $H_2SO_4$ for dissolution of 1.5% Cu as $CuCO_3$ = 50 lb/ton

| Time min | Test 352 -10 mesh ore 1 hr leach | | | Test 353 -10 mesh ore 3 hr leach | | | Test 354 -100 mesh ore 3 hr leach | | |
|---|---|---|---|---|---|---|---|---|---|
| | Cu g/l. | pH* | Acid addn. lb/ton | Cu g/l. | pH* | Acid addn. lb/ton | Cu g/l | pH* | Acid addn. lb/ton |
| 0 | 0 | 7.6 | 50 | 0 | 7.8 | 55 | 0 | 8.0 | 55 |
| 5 | | 1.75 | 16 | | 1.8 | 7 | | 2.1 | |
| 10 | | 1.65 | | | 1.95 | 7 | | 2.5 | 14 |
| 15 | | 1.7 | | | | | | | |
| 20 | 3.3 | 1.8 | 8 | | 1.8 | 14 | | 2.0 | 7 |
| 30 | | 1.6 | | 3.7 | 1.7 | | 3.9 | 1.85 | 7 |
| 40 | 3.95 | 1.75 | 16 | | 1.7 | | | | |
| 50 | | 1.55 | | | 1.7 | | | 1.8 | 7 |
| 60 | 4.05 | 1.6 | | 4.1 | 1.7 | 7 | 4.1 | | 7 |
| 90 | | | | 4.2 | 1.7 | 7 | 4.2 | 1.5 | 14 |
| 110 | | | | | 1.6 | | | 1.5 | |
| 120 | | | | 4.2 | 1.5 | 28 | 4.2 | 1.4 | 14 |
| 180 | | | | 4.2 | 1.45 | | 4.3 | 1.5 | |
| | Total acid | | 90 | Total acid | | 125 | Total acid | | 125 |

* pH measured before addition of acid.

Table 4. Metallurgical results for Sample B

| | Time (min) | Test 352<br>% Cu extr. | Test 353<br>% Cu extr. | Test 354<br>% Cu extr. |
|---|---|---|---|---|
| | 0 | 0 | 0 | 0 |
| | 20 | 62.7 | | |
| | 30 | | 70.7 | 73.6 |
| | 40 | 75.3 | | |
| | 60 | 77.3 | 78.3 | 77.4 |
| | 90 | | 80.3 | 79.2 |
| | 120 | | 80.3 | 79.2 |
| | 180 | | 80.3 | 81.1 |
| Total acid addition, lb/ton | | 90 | 125 | 125 |
| Final acid conc. g/l. | | 5.1 | 9.4 | 10.0 |
| Actual acid consumpt. lb/ton | | 65 | 62 | 58 |
| lb $H_2SO_4$/lb Cu dissolved | | 2.37 | 2.18 | 2.02 |
| Final residue wt., % | | 96.0 | 95.5 | 95.5 |
| Cu assay of residue, % | | 0.38 | 0.32 | 0.31 |
| % loss of Cu in residue | | 22.8 | 19.7 | 18.9 |
| Calc. Cu assay of feed, % | | 1.58 | 1.57 | 1.59 |
| Sizing (% - 200 mesh) | | 22 | 22 | 53 |

At the start of Test 352, sulphuric acid equivalent to the theoretical amount required to dissolve the copper was added, whereas in Test 353 and 354 the theoretical acid plus an excess of 10 per cent was added. During the test further acid was added as required to keep the pH below 1.8 as shown in Table 3.

The copper concentration was measured at appropriate intervals and the free acid concentration was determined at the end of the run (Tables 3 and 4).

## RESULTS AND DISCUSSION

A summary of the metallurgical results is given in Table 4. In Test 352, which was a 1 hour leach of minus 10 mesh material about 63 per cent of the copper had been dissolved after 20 min leaching but this had risen to only 77 per cent after 60 min agitation, for an acid consumption of 65 lb/ton or 2.37 lb/lb Cu dissolved. This was a relatively low extraction (leaving a residue assaying 0.38 per cent copper) and two possible reasons for this were insufficient agitation time, or too coarse a feed, with the possibility of there being too low an acid addition.

In Test 353 a 3 hour leach period was used on minus 10 mesh material and a greater amount of acid was added in the early stages of the run. This longer leaching time and higher acid strength gave only a small increase in copper extraction (to 80 per cent), and the acid consumption was about the same.

In Test 354 the ore was ground minus 100 mesh before leaching and was then leached for 3 hours with a relatively high sulphuric acid addition. This resulted in a small increase in extraction (to 81 per cent), but left a residue assaying 0.31 per cent copper.

It seems likely that the copper in the residue is present as either metallic copper, which would dissolve relatively slowly under the conditions used, or as a sulphide, which would dissolve very slowly. This point was not investigated further in view of the work to be done on the more representative sample (Sample C) and because very little sample was available, but it does warrant additional study. Possibly an addition of ferric ion would aid dissolution.

## AGITATION LEACHING

### TESTS ON SAMPLE C

Four agitation leaching tests were conducted on Sample C - two with a 'low' acid addition (one on minus 14 mesh ore and one on minus 48 mesh ore) and two with a 'high' acid addition (one on minus 14 mesh ore and one on minus 48 mesh ore). The 'low' acid addition was 46 lb/ton and was the theoretical acid required to dissolve 1.22 per cent copper as $CuCO_3$ plus an excess of 10 per cent. The 'high' acid addition was 79 lb/ton and was sufficient to maintain a pH of about 1.6 at the end of the run. In each test 46 lb/ton sulphuric acid was added at the start and then the additional acid added as shown in Table 5.

All tests were conducted with 250 g ore at a liquid/solid ratio of 3/1, and were stirred as noted for Sample B. The minus 48 mesh ore was prepared by dry grinding in a mortar and pestle. Copper determinations were made at appropriate intervals and the free acid concentration was determined at the end of the run. The final residue was filtered, washed, and assayed after each test.

### RESULTS AND DISCUSSION

Metallurgical results are shown in Table 6. In Tests 355 and 358, which were conducted with a 'low' acid addition (theoretical plus 10 per cent) about 66 per cent of the copper was extracted after 1 hour of leaching and this increased to only 68 per cent after 3 hours leaching. The pH rose to over 3 after 1 hour, and it was clear that reaction with gangue had consumed more than the 10 per cent excess acid added and reaction with copper minerals ceased. It is worth noting that the results for minus 14 and minus 48 mesh ore were very similar.

In Tests 356 and 357, in which a greater acid addition was made, the pH at the end of the tests was about 1.6. Under these conditions 91 per cent of the copper was dissolved leaving a residue assaying about 0.1 per cent copper. There was very little difference between the extraction after 2 hours and after 3 hours. The extraction was not significantly different for minus 14 mesh ore or for minus 48 mesh ore.

The acid consumption under these conditions for minus 14 mesh ore and for minus 48 mesh ore was about 60 lb/ton (or 2.4 lb/lb Cu dissolved) and was about 18 lb/ton higher than the theoretical figure of 42 lb/ton.

To sum up then, these preliminary tests show that the ore can be readily treated by crushing minus 14 mesh and then acid leaching. It might be necessary to grind finer than 14 mesh for satisfactory agitation, and this would introduce additional iron into the pulp which would consume additional acid. This should be investigated.

Table 5. Agitation leach details for Sample C

Liquid/solid ratio 3/1
Theoretical $H_2SO_4$ for dissolution
of 1.22% Cu as Cu $CO_3$ = 42 lb/ton

| Test 355<br>minus 14 mesh ore | | | | Test 358<br>minus 48 mesh ore | | | |
|---|---|---|---|---|---|---|---|
| Time<br>min | Cu<br>g/l. | pH* | $H_2SO_4$<br>lb/ton | Time<br>min | Cu<br>g/l. | pH* | $H_2SO_4$<br>lb/ton |
| 0 | 0 | 7.0 | 46 | 0 | 0 | 7.2 | 46 |
| 15 | | 2.65 | | 15 | | 2.0 | |
| 45 | | 3.0 | | 45 | | 3.3 | |
| 60 | 2.78 | 3.2 | | 60 | 2.73 | | |
| 90 | | 3.4 | | | | | |
| 105 | 2.83 | | | 120 | 2.78 | 3.6 | |
| 165 | | 3.65 | | | | | |
| 180 | 2.86 | 3.6 | | 180 | 2.86 | 3.5 | |

| Test 356<br>minus 14 mesh ore | | | | Test 357<br>minus 48 mesh ore | | | |
|---|---|---|---|---|---|---|---|
| Time<br>min | Cu<br>g/l. | pH* | $H_2SO_4$<br>lb/ton | Time<br>min | Cu<br>g/l. | pH* | $H_2SO_4$<br>lb/ton |
| 0 | 0 | 8.0 | 46 | 0 | 0 | 7.3 | 46 |
| 5 | | 2.2 | | 5 | | 1.9 | |
| 30 | | 2.4 | | 30 | | 3.1 | |
| 60 | 2.75 | 3.0 | 16½ | 60 | 2.74 | 3.3 | 33 |
| 90 | 3.22 | 2.0 | 16½ | 75 | | 1.4 | |
| 150 | 3.72 | 1.6 | | 120 | 3.58 | 1.5 | |
| 180 | 3.68 | 1.6 | | 180 | 3.79 | 1.6 | |

pH measured before addition of acid.

## LEACH-PRECIPITATION-FLOTATION TEST ON SAMPLE C

One preliminary leach-precipitation-flotation test was conducted on Sample C to obtain some data on this method of treatment.

A 250 g portion of minus 14 mesh ore was wet ground in a rubber lined mill, using steel balls, for 20 min; 165 lb/ton sulphuric acid was then added and grinding continued for a further 20 min. The pH was then 0.8 and the balls were covered with metallic copper. About 30 lb/ton of iron filings (prepared by filing mild steel) were then added (this was about 20 per cent more than the theoretical amount required to precipitate all the copper dissolved) and grinding continued for a further 10 min. The resultant sizing was 98 per cent minus 48 mesh and 58 per cent minus 200 mesh. The pulp was transferred to a flotation cell, and the mill and balls were cleaned by grinding with sand and ammonia solution (these products were assayed as noted later).

Table 6. Metallurgical results for Sample C

| | Test 355 | Test 358 | Test 356 | Test 357 |
|---|---|---|---|---|
| Time min | % Cu extr. | % Cu extr. | % Cu extr. | % Cu extr. |
| 0 | 0 | 0 | 0 | 0 |
| 60 | 66.0 | 65.2 | 68.2 | 66.7 |
| 90 | | | 79.8 | |
| 105 | 67.5 | | | |
| 120 | | 66.3 | 92.2 | 87.1 |
| 180 | 67.9 | 68.3 | 91.2 | 92.2 |
| Total acid addition, lb/ton | 46 | 46 | 79 | 79 |
| Final acid conc., g/l. | Nil | 0.2 | 2.94 | 3.0 |
| Actual acid consumpt., lb/ton | 46 | 45 | 60 | 59 |
| lb $H_2SO_4$/lb Cu dissolved | 2.40 | 2.33 | 2.43 | 2.32 |
| Final residue wt., % | 94.4 | 96.2 | 96.0 | 96.0 |
| Cu assay of residue, % | 0.42 | 0.415 | 0.12 | 0.10 |
| % loss of Cu in residue | 32.1 | 31.7 | 9.5 | 7.8 |
| Calc. Cu assay of feed | 1.26 | 1.26 | 1.21 | 1.23 |
| Sizing (%-200 mesh) | 23 | 30 | 23 | 30 |

The pulp was treated in a modified Ruth laboratory flotation cell under the following conditions.

| | | |
|---|---|---|
| Flotation time | 20 min | |
| pH | 1.8 | |
| Secondary butyl xanthate | 0.3 lb/ton ) | Added in |
| Minerec B | 0.6 lb/ton ) | three stages |

No frother was added as the froth was quite satisfactory. Metallic copper (or copper-coated iron) floated readily, as well as some gangue minerals. Results are shown in Table 7.

These results show that 70 per cent of the copper was recovered in a flotation concentrate assaying about 21 per cent copper. About 14 per cent of the copper was left on the grinding balls, and it is likely that by a change of technique (i.e. by grinding in neutral solution and then leaching and precipitating in an external vessel) that most of this copper could be recovered in the concentrate.

The total loss of copper in the flotation tailing was about 16 per cent which was made up of 9 per cent in solid tailing assaying 0.13 per cent copper, and 7 per cent in solution. It may be possible to reduce the loss in solution, but probably not that in solid tailing because the assay is similar to that in direct leach residue (Test 356 and 357 in Table 6).

These results are encouraging and further investigation is probably worth while.

Table 7. Results of leach-precipitation-flotation

| Product | % wt.* | Assay | | | % Cu |
|---|---|---|---|---|---|
| | | Cu % | Fe % | Insol % | distrib. |
| Flot. conc. | 4.4 | 21.2 | 23.5 | 22.3 | 70.3 |
| Flot. tail | 92.4 | 0.13 | | | 9.0 ) 16.1 |
| Cell soln. | 1340 | 0.007 | | | 7.1 ) |
| $NH_3$ soln. | 100 | 0.095 | | | 7.2 ) 13.6 |
| Cleaning sand | 100 | 0.085 | | | 6.4 ) |
| Calc. feed | 100 | 1.33 | | | 100.0 |

* Based on original ore.

## CONCLUSION

Results presented in this report show that ore from Mount Gunson is amenable to sulphuric acid leaching to dissolve copper. Referring particularly to Sample C (the most representative sample submitted) about 90 per cent of the copper was extracted from minus 14 mesh ore with an acid consumption of 60 lb/ton (about 18 lb/ton more than theoretical). Commercial grinding may introduce more iron into the system than laboratory work and so the acid consumption would be higher. In addition, some discard of solution containing unused acid is probably inevitable and this would also result in a higher acid consumption.

A good copper extraction was obtained on the high grade sample (Sample A) by percolation leaching. On Sample B a poor extraction was obtained during agitation leaching for reasons not evaluated.

As the results reported on Samples B and C were obtained on percussion drill cuttings, it is desirable that the results be checked on a sample of bulk ore. This work should include

1. percolation leaching of average grade material,
2. agitation leaching of ground ore as a function of grinding in an iron mill, acid addition, and leach time, and
3. leach-precipitation-flotation tests.

In addition consideration needs to be given to a source of iron for precipitation.

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19th July, 1966
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RESEARCH FOR INDUSTRY

**THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES**

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Please quote this reference in your reply:
IC 3/262/1

17th January, 1967

Your reference:

Mr. J.B. Evans,
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Suite 20,
541 St. Kilda Road,
MELBOURNE. Vic.

# MT. GUNSON ORE

## PROGRESS REPORT NO.1

1st November to 31st December, 1966

Investigation and Report by: J.D. Hayton and L.J. Weir.

Officer in Charge, Industrial Chemistry Section: R.E. Wilmshurst

R.E. Wilmshurst
for F.A. Young
Director.

| | |
|---|---|
| PROJECT NUMBER: | 3/262/1 |
| PROJECT PROPOSED: | 21/10/66 |
| PROJECT AUTHORISED: | 28/10/66 |
| AMOUNT APPROVED: | $6,000 |
| AMOUNT EXPENDED TO 21/12/66: | $3,182 |
| ANTICIPATED DATE OF COMPLETION: | February, 1967 |

# MOUNT GUNSON ORE

## 1. REVIEW OF PROGRESS

The Mt. Gunson area was inspected by Messrs. Hartley and Weir of Amdel on 18th October, 1966. Samples of ore from two sites were suggested as feed for copper extraction tests, by physical beneficiation, and by leaching.

Froth flotation of ground ore sample B4, using a sulphidising agent for oxidised copper minerals, and xanthate or di-thiophosphate type collectors, has given copper recovery into rougher concentrates in the order of 60% with rejection of 90 to 95% by weight of feed. Rougher conoentrates have been upgraded from approximately 15% Cu to approximately 40% Cu on re-flotation.

Fatty acid collectors have given similar copper recovery into rougher concentrates, but at low ratio of concentration.

Flotation using a collector for sulphide copper minerals, followed by flotation of oxidised minerals after activation with a sulphidising agent, has not increased copper recovery.

Flotation of ground ore sample H11 has given rougher recoveries of approximately 75%, and cleaner concentrate grades of approximately 40% Cu.

Flotation of a composite of ore samples B4 and H11 gave a rougher recovery of 68.1%, and an oxide cleaner concentrate grade of 43.6% Cu.

Leaching of a composite ore sample with an addition of 50 lb of $H_2SO_4$ per ton followed by precipitation of the dissolved copper on metallic iron and froth flotation of the metallic copper, gave a copper loss of 15.9% in flotation tailing. A froth concentrate assaying 80.0% Cu was obtained containing 20% of the copper in feed, the balance being associated with flotation middling and metallic iron precipitant, both of which would re-cycle in a continuous process. The nature of copper loss has not been investigated.

Leaching of a composite ore sample, ground to nominally passing 30 mesh was done by adding sulphuric acid equivalent to 60 lb per short ton of dry ore to a slurry of the ore at 60% solids in water. The slurry was agitated by a standard 6-blade turbine stirrer having a diameter of 12 inches and blade depth 2.5 inches, in an 80 gallon rubber lined tank fitted with 4 vertical stainless steel baffles. Power requirements were measured at various stirrer speeds and samples of pulp were taken at 30-minute intervals. The maximum leaching efficiency of 98% was reached within one hour of adding the acid. Settling tests were done on leached pulp diluted to 15% and 20% solids using Cyanamid flocculating agent S3384. A dosage rate of 0.15 lb per ton of dry solids appeared to be adequate in both instances.

The calculated settling area required in a C C D washing circuit was between 2.5 and 3 square feet per ton per day.

Some preliminary extraction tests with LIX 64 - kerosene were done on liquor representing a CCD washing circuit at 15% solids feed. For feed liquor containing 2.85 g Cu per litre at initial pH 2.6 at least 6 extraction stages would be needed with LIX 64, 10% v/v in kerosene at an organic/aqueous ratio of 1. Under similar conditions, at organic/aqueous ratios of 2 and 1.5 the number of stages required would be 3 and 4 respectively.

Stripping tests were done on small quantities of loaded organic with aqueous sulphuric acid solution containing 200 g $H_2SO_4$ per litre. The copper appeared to be stripped readily but insufficient data was obtained to permit construction of a stripping curve.

Several flow-sheets have been established for leach-CCD circuits based on treatment of 500 tons of ore per day at an acid addition of 60 lb per ton of dry ore. The economics of CCD circuits at feed concentrations of 15%, 20% and 25% solids are being considered. The main factor involved in the assessment is the amount of water available daily for use in the leaching and washing stages. At initial feed densities of 25 and 20% solids the number of CCD stages required for 99% washing efficiency would be 6 and 4 respectively.

## 2. WORK IN HAND

Tailing from flotation of sample B4 has been size classified, and analysis and mineralogical examination of size fractions is in hand to determine the nature of the copper losses.

Filtration tests are being done by Federated Engineers Ltd. at Artarmon, N.S.W., using the composite sample leached at 60% solids with sulphuric acid equal to 60 lb per short ton. When results are available the relative merits of filtration and CCD washing will be considered.

Economic evaluation of various proposed leach-CCD flow sheets is being continued. Suitable conditions for application of solvent extraction with LIX 64 - kerosene solution are still being investigated with the ultimate objective of establishing a continuous system for copper recovery by electrolysis.

## 3. MATERIAL EXAMINED

Four 44 gallon drums of ore crushed to 1-inch were received. The contents of the drums represented 2 samples of ore. Details are tabulated below:

| Drum No. | Sample | Footage | Locality |
|---|---|---|---|
| 427 | B4 | 5 - 10 ft | Main cut |
| 428 | B4 | 10 - 15 ft | " " |
| 429 | H11 | 5 - 10 ft | West Main Cut |
| 430 | H11 | 10 - 15 ft | " " " |

Samples of ore representing the total footage from each of the two localities were used for the various tests.

## 4. EQUIPMENT

British Standard Screens
Rolls crusher 10-inch diameter x 6-inch
Denver continuous rod-mill, 1 ft dia x 2 ft
Batch rod-mill, 7½-inch dia x 10-inch
    Stainless steel, with 15 x 1-inch dia. rods
Fagergren flotation machine, 3 litre and 5 litre
    capacity cells
Laboratory agitator
Warman cyclosizer
Leaching Tank: A rubber-lined cylindrical steel tank of 80 gal capacity. Agitated by a stainless steel 6-bladed turbine, 12 inches diameter. 2 HP 3-phase motor with revolutions counter.
Four 2¼-inch wide stainless steel baffles attached vertically at equal intervals around the inside of the tank.

## 5. ANCILLARY MATERIALS

Sulphuric acid, commercial grade
Potassium Amyl Xanthate (KAX), Cyanamid Australia Pty. Ltd.
Aerofloat 25, Cyanamid Australia Pty. Ltd.
Aerofloat 404, Cyanamid Australia Pty. Ltd.
Cresylic acid T.D., Union Carbide Australia, Pty. Ltd.
Z-200, Dow Chemical Company.
Dowfroth 250, Dow Chemical Company.
Pamak 4 (Semi-refined tall oil), A.C. Hatrick Pty. Ltd.
Naphthenic acid SP130, Shell Chemical Co.
Triton X-100 (non-ionic emulsifier), Robert Bryce & Co. Ltd.
Fuel oil
Soda ash, commercial grade.
Sodium silicate, commercial grade.
Sodium sulphide, commercial caustic-flake grade (60% $Na_2S$)

Tall oil, where used as a flotation collector, was added as a water-emulsifiable mixture of the following composition:

| | Weight % |
|---|---|
| Pamak 4 | 52 |
| Fuel Oil | 43 |
| Naphthenic Acid | 2.5 |
| Triton X-100 | 2.5 |
| | 100.0 |

## 6. EXPERIMENTAL PROCEDURE AND RESULTS

### 6.1 Sample Preparation and Head Assay

The four drums of ore were sampled by riffling, and a quarter of each was reduced in a rolls crusher to 12-mesh. The two footage samples from each locality were combined to form samples B4 and H11. The copper contents of the samples were:-

| | Cu% |
|---|---|
| B4 | 1.70 |
| H11 | 0.67 |

### 6.2 Flotation

#### 6.2.1 Grinding Tests

Portions of 500 g of sample B4 were ground in a batch rod mill at 60% solids for various times, and the ground products were sized by screening. The results are shown in Table 1.

A grinding time of 5 minutes produced material of a suitable size distribution for flotation tests.

A sample of H11 material was ground for 5 minutes. Size analyses of the ground ores are shown in Table 2.

#### 6.2.2 Froth Flotation of Copper Minerals

Tests Using Activator for Oxidised Minerals. Charges of 500 g of sample B4 were ground for 5 minutes and flotation tests were carried out using three types of collector, after activation with 1 lb of sodium sulphide per ton of ore. The pulp was conditioned with sodium sulphide in the flotation cell for 5 minutes, collector was then added, conditioned for 2 minutes, and a froth concentrate removed by flotation for approximately 5 minutes. Further additions of sodium sulphide and collector were made, and a separate froth concentrate was removed. The process was continued till no further concentrate was recovered. Pulp pH during the tests was 8.1 to 8.9. A second test using Aerofloat 25 collector was done on ore ground for 10 minutes. The stage rougher concentrates were samples for analysis separately. Results of the tests are shown in Table 3.

A test was done on ore ground for 5 minutes, using Aerofloat 25 collector under similar conditions to test 2. The combined rougher concentrates were cleaned once by re-flotation. A further test was done on ore ground for 10 minutes, in which the sodium sulphide and Aerofloat 25 additions in each of two roughing flotation stages were increased to 5.0 and 0.2 lbs/ton of ore respectively. Results of the tests are shown in Table 4.

Tests Using Collector for Sulphide Mineral, and Activation of Oxidised Minerals. Because of the relatively high losses in flotation tailing in previous tests, ground flotation feed pulp was conditioned with collector (Aerofloat 25) and a froth concentrate was removed. Sodium sulphide was then added for activation of oxidised copper minerals, and a froth concentrate was removed in 1 or 2 stages. Collector addition was 0.4 lb/ton of ore to both sulphide and oxide flotation stages, and sodium sulphide addition was 5.0 lb/ton. The oxidised rougher concentrate only was cleaned by re-flotation. Results of tests on samples B4, H11 and a composite of the two, are shown in Table 5.

Tests Using Fatty Acid Collector. Two tests were done using a tall-oil collector mixture. The ground ore pulp was conditioned in the flotation cell with 20lb of soda ash and 1.0 lb of sodium silicate per ton of ore to give a dispersed pulp at pH 9.3. For the first test, collector was added in stages of 2 lb per ton of ore, and after conditioning for 10 minutes, separate froth concentrates were removed. For the second test, the addition of collector was increased to obtain maximum recovery, and the combined rougher concentrates were cleaned. Results of the tests are shown in Table 6.

### 6.2.3 Leaching/Precipitation/Flotation.

Tests were done on a composite of ore samples B4 and H11, ground for 5 minutes.

The ground pulp was agitated with acid for 30 minutes, then iron turnings were added at the rate of 100 lb per ton of ore, and the pulp further agitated for 30 minutes. Iron turnings, with some adhering metallic copper, were removed from the pulp with a hand-magnet before flotation. The pulp was diluted to flotation feed density of approximately 20% solids, conditioned in the flotation cell with collector, and metallic copper was removed in a froth concentrate.

Conditions for the tests are shown in Table 7, and results in Table 8.

## 6.3 Leaching and Solid-Liquid Separation

### 6.3.1 Preparation of Material

A composite of samples B4 and H11, crushed to minus 12 mesh, was fed to a continuous rod-mill initially at the rate of approximately 100 pounds per hour. Rod charge in the mill was approximately 75 pounds.

The discharge pulp was held at 60% solids, and the mill feed rate was adjusted to give a product nominally passing minus 30 mesh. A feed rate of 120 pounds per hour was attained.

### 6.3.2 Leaching

Approximately 330 lb of the composite sample as a 60% solids slurry were transferred to the 80-gallon leaching tank. With the four vertical baffles in position, the solids could be kept well suspended at a stirrer speed of 40 rpm but during the leaching, speeds of up to 62 rpm were applied to allow power input to be determined. The tank was able to revolve about its central axis and would do so under the torque effect of the rotary stirrer unless a suitable retaining force was applied. This force was measured by attaching a spring balance to a point on the outside periphery of the tank and anchoring the balance to a fixed support. Measurements made at various stirrer speeds are tabulated below:-

| Stirrer Speed rpm | Force lb | Calculated shaft horse-power* |
|---|---|---|
| 40 | 3.37 | 0.03 |
| 50 | 4.87 | 0.053 |
| 60 | 6.63 | 0.091 |
| 62 | 7.0 | 0.10 |

* Calculated from relation:-

$$H\ P = \frac{2\pi r F R}{33,000}$$

where r = radius of mixing vessel (ft)
F = applied circumferential force (lb)
R = stirrer speed rpm.

Sulphuric acid equivalent to 60 lb per short ton of dry ore was added slowly over a period of 30 minutes. Samples of slurry were taken at 30 minute intervals thereafter, filtered under suction and washed. The filtrates were made to known volumes and the residues were dried at 105°C and weighed. The copper content of liquors and residues was determined and the copper distribution was calculated.

### 6.3.3 Solid-liquid spearation

Filtration: A filtration test was attempted with a 0.1 sq ft area "Eimco" test filter leaf covered in coarse duck but the cloth was blinded immediately. A portion of the unleached 60% solids slurry was sent to Federated Engineers Ltd. at Artarmon N.S.W. for filtration tests after leaching as described in Section 6.3.2. Results of these tests are awaited.

C.C.D. washing: Flocculation and settling tests were done on leached pulp diluted to 20% and 15% solids. Cyanamid flocculant S3384 was used at dosage rates between 0.05 and 0.2 lb per short ton. The settling tests were done in 500 ml capacity stoppered measuring cylinders graduated in inches and tenths to enable the height of the settling interface to be determined in relation to time after adding the flocculant. Thickener area required at each flocculant dosage was calculated by application of a modified Kinch method. The results obtained are tabulated below:

| | | | | | |
|---|---|---|---|---|---|
| Flocculant lb/short ton | 0.05 | 0.10 | 0.15 | 0.20 | 0.15 |
| Pulp solids % | 20 | 20 | 20 | 20 | 15 |
| Supernatant liquor appearance | Cloudy | Cloudy | Clear | Clear | Clear |
| Thickener area, $ft^2$/ton/day | 5.5 | 5.0 | 2.4 | 2.0 | 3.0 |

### 6.3.4 Solvent Extraction

Preliminary extraction tests were done on liquor produced during the bulk leaching test. Simulated filtration and CCD liquors were prepared as follows:-

Filtration: The pulp was diluted to 50% solids and allowed to settle. The clear liquor was assumed to be similar to that which could be produced by direct filtration of 60% solids pulp followed by minimum washing. The "filtration" liquor contained 11.4 g of Cu per litre and 0.90 g chloride per litre.

C.C.D.: The "filtration" liquor was diluted with water to reduce the copper concentration to the level expected to be found in liquors from CCD washing plants operating at various initial solids contents.

A solution of LIX 64 reagent in kerosene was used for the extraction tests. A 10% v/v solution of LIX 64 was conditioned by shaking it first with 4N sulphuric acid and then with water to remove excess acid.

Two series of tests were done:

(a) on "filtration" liquor at an organic to aqueous ratio of 4
(b) on CCD liquor at an organic to aqueous ratio of 1.

Actual operating conditions were simulated by arranging sets of 5 separating funnels containing both aqueous copper solution and LIX 64 solution and conducting "shake-out" tests so that fresh organic and aqueous solutions were introduced after each series of contacts. By repeating the shake-out series a sufficient number of times a condition of equilibrium was reached which closely resembled that attained in a continuous counter current system. The organic and aqueous phases from the final equilibrium series were assayed for copper. Equilibrium distribution curves were drawn from the data obtained.

The data obtained from the tests with liquor containing 11.4 g Cu per litre were inadequate bacause of pH effects and a suitable distribution curve could not be drawn.

Shake-out tests were done on liquor containing 2.85 g Cu per litre and at an initial pH of 2.6. Contact time was 2 minutes.

The distribution of Cu between the organic and aqueous phases and the pH values of the aqueous phases at equilibrium are shown below:

| Stage No. | Aqueous |  | Organic |
|---|---|---|---|
|  | Cu g/1 | pH | Cu g/1 |
| Initial | 2.85 | 2.62 | - |
| 1 | 0.57 | 1.6 | 0.91 |
| 2 | 1.23 | 1.7 | 1.88 |
| 3 | 1.72 | 1.9 | 2.80 |
| 4 | 2.50 | 2.2 | 3.10 |

Under these conditions practically all of the copper could be extracted in 4 counter current stages at an organic to aqueous ratio of 1:5 (Fig. 1).

## 7. DISCUSSION

### 7.1 Direct Flotation

Satisfactory concentrate grades were obtained with one stage of re-flotation of concentrates produced by activation of copper minerals with a sulphidising agent and flotation with a xanthate or di-fluophosphate type collector. However, loss of copper in tailing was approximately 40% on sample B4 (1.70% Cu) and the loss was not reduced by:

(a) Increase in reagent additions
(b) Increase in grind size from 29 to 48% passing 240-mesh.
(c) Flotation of any sulphide values present before activation of oxidised minerals.

Flotation of sample H11 (0.69% Cu) gave lower tailing losses of approximately 25% of the copper in feed. Flotation of a composite of samples B4 and H11 gave recovery between that for the individual samples as expected.

### 7.2 Leach/Precipitation/Flotation

Leaching with 50 lbs $H_2SO_4$ per ton gave satisfactory conditions in subsequent flotation, and produced high-grade concentrate.

An attempt was made to raise recovery by increasing the amount of acid added but poor flotation conditions resulted. No attempt was made to raise the pH of pulp before precipitation of copper on iron, and it is probable that poor flotation resulted from a high soluble-iron content of feed pulp.

The nature of copper loss in tailing from ore leached with 50 lbs $H_2SO_4$ per ton was not investigated. The loss is possibly due to re-solution of copper during flotation of pH approximately 4 because of the absence of free metallic iron in this stage. A light plating of metallic copper was obtained on a cast-iron filter frame on de-watering of flotation tailing.

It therefore seems likely that recoveries by leach/precipitation/flotation would closely approach those of direct leaching under optimum acid consumption, provided that free metallic iron were present in the flotation feed pulp. However, any fine iron in flotation feed would largely report to froth products, causing dilution of the final concentrate.

### 7.3 Leaching and Solid-Liquid Separation

Leaching with sulphuric acid was highly efficient. At 60% solids the amount of acid required to achieve 98% dissolution of the copper was not much above theoretical. Maximum leaching efficiency was attained in approximately one hour after adding the acid.

Solid-liquid separation in a CCD washing circuit would be straight-forward for leached pulp diluted to 15% or 20% solids if the appropriate amount of flocculant was added. At a flocculant dosage of 0.15 lb per short ton the thickener area required would be 2.5 to 3 square feet per ton per day.

Filtration tests are being done by Federated Engineers Ltd. at Artarmon, N.S.W. When the results of these tests are available, the relative merits and costs of the two systems will be evaluated.

### 7.4 Solvent Extraction

The results of preliminary laboratory tests with LIX 64 have confirmed previously held opinions that this reagent would be suitable for extracting copper from the leach liquors. Concentrated liquor such as would be derived by filtration could, undoubtedly be used, but further experimental data are required to enable the operating conditions to be determined. Extraction of more dilute liquor resulting from CCD operation appears to be feasible. The equilibrium extraction isotherm shown in Fig. 1 is representative of liquor at an initial pH of 2.6 and containing 2.85 g Cu per litre. Liquor produced from an operating plant may contain somewhat more copper and have a pH value of less than 2.0. To achieve efficient extraction in 4 or 5 stages with this liquor, some neutralization may be required, since the extraction efficiency for copper is very dependent on the hydrogen ion concentration in the liquor. Further tests are being done to determine the conditions required for efficient extraction.

## TABLE 1: EFFECT OF GRINDING TIME ON PARTICLE SIZE, SAMPLE B4

| Nominal Screen Aperture Microns | Equivalent Mesh | Weight % Grind A(2 min) | Weight % Grind B(5 min) | Weight % Grind C(10 min) |
|---|---|---|---|---|
| + 500 | + 30 | 0.2 | - | - |
| -500 + 355 | -30 + 44 | 3.3 | Trace | - |
| -355 + 250 | -44 + 60 | 15.6 | 1.2 | Trace |
| -250 + 180 | -60 + 85 | 22.2 | 7.7 | 0.4 |
| -180 + 125 | -85 + 120 | 18.5 | 26.5 | 4.4 |
| -125 + 90 | -120 + 170 | 13.2 | 22.7 | 26.6 |
| -90 + 63 | -170 + 240 | 7.2 | 12.6 | 21.1 |
| -63 | - 240 | 19.8 | 29.3 | 47.5 |
| | | 100.0 | 100.0 | 100.0 |

## TABLE 2: SIZE ANALYSIS OF GROUND ORE

| Nominal Screen Aperture Microns | Sample B4 Wt % | Sample B4 Assay Cu % | Sample B4 Distribution Cu % | Sample H11 Wt % | Sample H11 Assay Cu % | Sample H11 Distribution Cu % |
|---|---|---|---|---|---|---|
| + 250 | 1.2 | 0.86 | 0.6 | 1.3 | 0.22 | 0.4 |
| -250 + 180 | 7.7 | 1.12 | 5.0 | 8.9 | 0.28 | 3.2 |
| -180 + 125 | 26.5 | 1.18 | 18.0 | 24.4 | 0.38 | 12.0 |
| -125 + 90 | 22.7 | 1.25 | 16.5 | 22.6 | 0.50 | 14.6 |
| -90 + 63 | 12.6 | 1.65 | 12.0 | 13.2 | 0.70 | 11.9 |
| -63 + 45 | 3.3 | 3.35 | 6.4 | 5.2 | 1.40 | 9.4 |
| -45 + 33.5 | 5.4 | 1.90 | 5.9 | 5.9 | 0.94 | 7.1 |
| -33.5 +23.5 | 5.4 | 2.17 | 6.8 | 5.2 | 1.02 | 6.8 |
| -23.5 +15.5 | 3.6 | 2.23 | 4.7 | 3.6 | 1.28 | 6.0 |
| -15.5 +12.0 | 1.6 | 3.14 | 2.9 | 1.6 | 1.54 | 3.2 |
| -12.0 | 10.0 | 3.66 | 21.2 | 8.1 | 2.42 | 25.4 |
| | 100.0 | (a)<br>1.73 | 100.0 | 100.0 | (a)<br>0.77 | 100.0 |

(a) Calculated grades.

## TABLE 3: PRELIMINARY ROUGHER FLOTATION TESTS, SAMPLE B4

| Test No. | Grind | Collector (Total) lb/ton | Product | Wt % | Assay Cu % | Distribution Cu % |
|---|---|---|---|---|---|---|
| 1 | B | K.A.X. | Ro.Conc.1 | 2.2 | 14.0 | 18.4 |
| | | | " " 2 | 2.8 | 24.4 | 40.8 |
| | | 0.2 | Tailing | 95.0 | 0.72 | 40.8 |
| | | | | 100.0 | (a)<br>1.68 | 100.0 |
| 2 | B | Aerofloat 25 | Ro.Conc.1 | 1.3 | 20.8 | 15.6 |
| | | | " " 2 | 3.2 | 23.2 | 42.9 |
| | | 0.3 | " " 3 | 1.6 | 4.8 | 4.5 |
| | | | Tailing | 93.9 | 0.68 | 37.0 |
| | | | | 100.0 | (a)<br>1.73 | 100.0 |
| 3 | B | Aerofloat 404 | Ro.Conc.1 | 0.6 | 26.8 | 11.7 |
| | | | " " 2 | 1.6 | 29.6 | 28.9 |
| | | 0.5 | " " 3 | 1.1 | 11.0 | 7.4 |
| | | | Tailing | 96.7 | 0.88 | 52.0 |
| | | | | 100.0 | (a)<br>1.64 | 100.0 |
| 5 | C | Aerofloat 25 | Ro.Conc.1 | 3.8 | 6.20 | 13.7 |
| | | | " " 2 | 2.5 | 30.0 | 43.7 |
| | | 0.3 | " " 3 | 1.8 | 3.4 | 3.6 |
| | | | Tailing | 91.9 | 0.73 | 39.0 |
| | | | | 100.0 | (a)<br>1.72 | 100.0 |

(a) Calculated Grade

TABLE 4: RECOVERY OF OXIDISED COPPER, CLEANING TESTS, SAMPLE B4

| Test No. | Grind | Collector (Total) lb/ton | Product | Wt % | Assay Cu % | Distribution Cu % |
|---|---|---|---|---|---|---|
| 6 | B | Aerofloat 25 | Cl.Conc. | 1.8 | 35.2 | 37.0 |
| | | | Cl.Tail. | 5.4 | 8.0 | 25.0 |
| | | 0.3 | Ro.Tail. | 92.8 | 0.70 | 38.0 |
| | | | | 100.0 | (a) 1.72 | 100.0 |
| 8 | C | Aerofloat 25 | Cl.Conc. | 1.5 | 38.0 | 31.9 |
| | | | Cl.Tail. | 5.8 | 8.5 | 27.6 |
| | | 0.4 | Ro.Tail. | 92.7 | 0.78 | 40.5 |
| | | | | 100.0 | (a) 1.79 | 100.0 |

(a) Calculated values.

TABLE 5: 2 STAGE FLOTATION OF SULPHIDE AND OXIDISED COPPER

| Test No. | Sample | Grind | Product | Wt % | Assay Cu % | Distribution Cu % |
|---|---|---|---|---|---|---|
| 14 | B4 | C | Sulphide Ro.Conc. | 1.9 | 5.6 | 6.2 |
| | | | Oxide Cl.Conc. | 1.0 | 42.8 | 25.2 |
| | | | " Cl.Tail. | 6.8 | 7.3 | 29.2 |
| | | | " Ro.Tail. | 90.3 | 0.74 | 39.4 |
| | | | | 100.0 | 1.70 (a) | 100.0 |
| 15 | B4 | B | Sulphide Ro.Conc. | 4.2 | 4.40 | 10.6 |
| | | | Oxide Cl.Conc. | 1.5 | 39.2 | 33.7 |
| | | | " Cl.Tail. | 5.0 | 6.4 | 18.3 |
| | | | " Ro.Tail. | 89.3 | 0.73 | 37.4 |
| | | | | 100.0 | 1.75 (a) | 100.0 |
| 16 | H11 | B | Sulphide Ro.Conc. | 2.9 | 1.70 | 5.3 |
| | | | Oxide Cl.Conc. | 1.2 | 40.8 | 53.5 |
| | | | " Cl.Tail. | 3.9 | 3.8 | 16.2 |
| | | | " Ro.Tail. | 92.0 | 0.25 | 25.0 |
| | | | | 100.0 | 0.92 (a) | 100.0 |
| 17 | H11 | C | Sulphide Ro.Conc. | 7.4 | 1.60 | 13.7 |
| | | | Oxide Cl.Conc. | 1.0 | 38.6 | 44.7 |
| | | | " Cl.Tail. | 6.2 | 2.5 | 17.9 |
| | | | " Ro.Tail. | 85.4 | 0.24 | 23.7 |
| | | | | 100.0 | 0.86 (a) | 100.0 |
| 18 | B4 + H11 | B | Sulphide Ro.Conc. | 4.4 | 2.50 | 9.8 |
| | | | Oxide Cl.Conc. | 0.9 | 38.0 | 30.5 |
| | | | " Cl.Tail. | 4.7 | 5.80 | 24.4 |
| | | | " Ro.Tail. | 90.0 | 0.44 | 35.3 |
| | | | | 100.0 | 1.12 (a) | 100.0 |
| 19 | B4 + H11 | C | Sulphide Ro.Conc. | 4.3 | 2.7 | 10.7 |
| | | | Oxide Cl.Conc. | 1.0 | 43.6 | 40.4 |
| | | | " Cl.Tail. | 3.9 | 4.7 | 17.0 |
| | | | " Ro.Tail. | 90.8 | 0.38 | 31.9 |
| | | | | 100.0 | 1.08 (a) | 100.0 |

(a) Calculated Grades

## TABLE 6: FLOTATION WITH FATTY-ACID COLLECTORS

| Test No. | Grind | Collector (Total) lb/ton | Product | Wt % | Assay Cu % | Distribution Cu % |
|---|---|---|---|---|---|---|
| 4 | B | Tall-oil Mixture 4.5 | Ro.conc.1 | 1.1 | 11.8 | 7.7 |
| | | | " " 2 | 6.3 | 11.4 | 42.7 |
| | | | Tailing | 92.6 | 0.90 | 49.6 |
| | | | | 100.0 | (a)<br>1.68 | 100.0 |
| 9 | C | Tall-oil Mixture 8.0 | Re-cl.conc. | 9.1 | 7.6 | 39.8 |
| | | | " " tail. | 9.4 | 2.02 | 11.0 |
| | | | Cl.Tail. | 13.3 | 1.82 | 13.9 |
| | | | Ro.Tail. | 68.2 | 0.90 | 35.3 |
| | | | | 100.0 | (a)<br>1.74 | 100.0 |

## TABLE 7: LEACH/PRECIPITATION/FLOTATION - CONDITIONS

| Test No. | $H_2SO_4$ Consumption lb/ton | pH |  | Collector | Addition (Total) lb/ton |
|---|---|---|---|---|---|
|  |  | Leaching | Flotation |  |  |
| 7 | 50 | n.d. | 4.2 | Z200 | 0.2 |
| 11 | 75 | 1.4 | 1.9(a)<br>4.5(b) | - | - |
| 12 | 60 | 2.0 | 4.0 | Z200 | 0.2 |

(a) No flotation at pH 1.9

(b) pH raised to 4.5 using 3.0 lbs soda ash/ton - poor flotation resulted.

## TABLE 8: LEACH/PRECIPITATION/FLOTATION - RESULTS

| Test No. | Product | Wt % | Assay % | Distribution Cu % |
|---|---|---|---|---|
| 7 | Cl.conc. | 0.3 | 80.0 | 20.0 |
|  | Cl.Tail. | 4.4 | 9.0 | 33.0 |
|  | Ro.Tail. | 95.3 | 0.2 | 15.9 |
|  |  | 100.0 |  |  |
|  | Iron |  | n.d. | 31.1 |
|  |  |  | 1.20 | 100.0 |
| 11 |  | Not assayed |  |  |
| 12 |  | Not assayed |  |  |

Fig 1: EXTRACTION EQUILIBRIUM CURVE FOR LIX 64-Cu SYSTEM

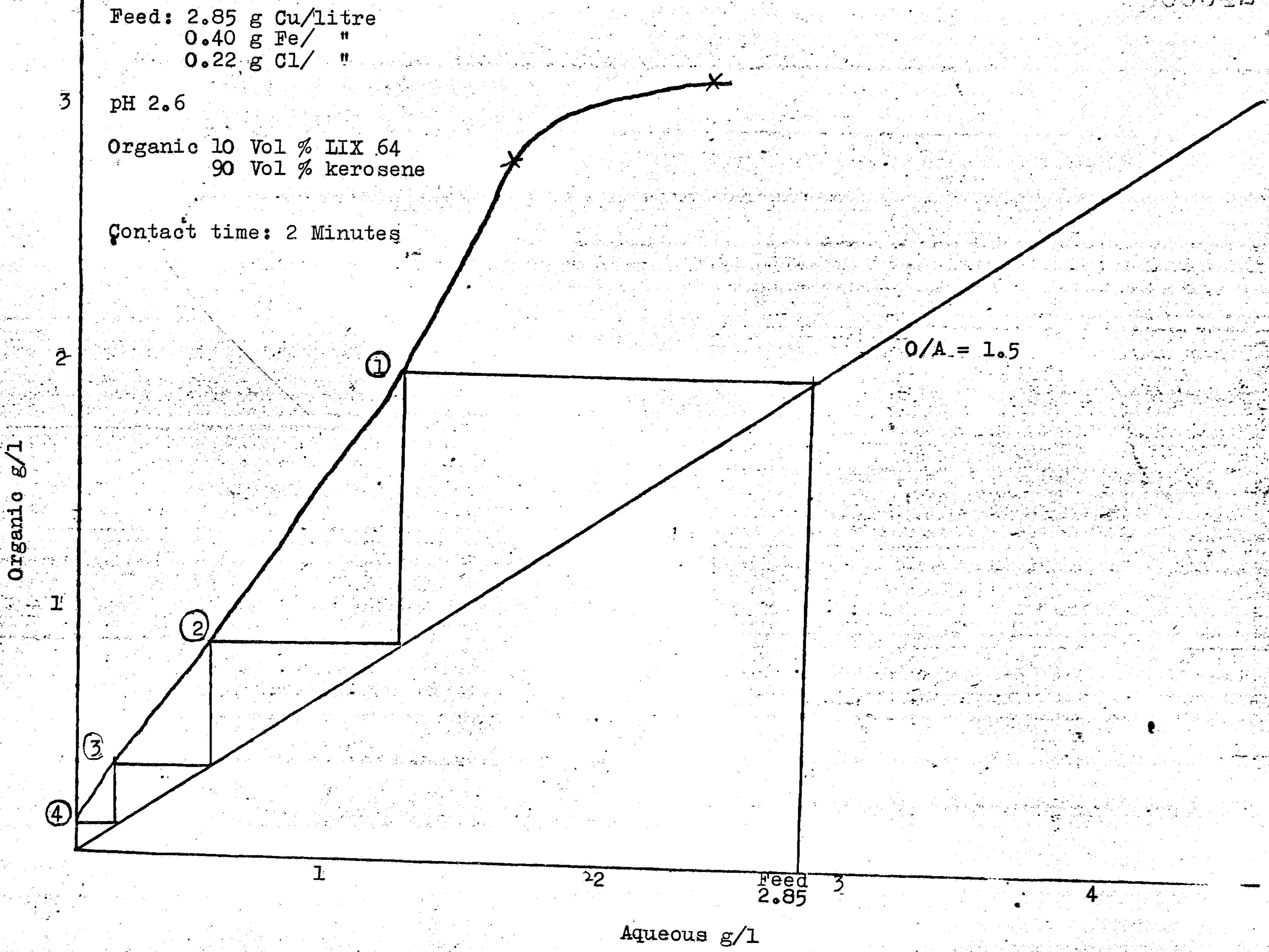


EXHIBIT NO. 8n

RESEARCH FOR INDUSTRY

**THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES**

CONYNGHAM STREET - FREWVILLE - SOUTH AUSTRALIA
TELEPHONE 79 1662 · TELEGRAMS 'AMDEL' ADELAIDE

*Please quote this reference in your reply:*

IC 3/262/1

18th January, 1967

Your reference:

Mr. J.B. Evans,
Austminex Pty. Ltd.,
Suite 20,
541 St. Kilda Road,
MELBOURNE. Vic.

MT. GUNSON ORE

PROGRESS REPORT NO. 2

To 17th January, 1967

Investigation by: J.M. Clayton, J.D. Hayton, P.K. Hosking, and L.J. Weir.

Officer in Charge, Industrial Chemistry Section: R.E. Wilmshurst

R.E. Wilmshurst

for P.A. Young
Director.

| | |
|---|---|
| PROJECT NUMBER: | 3/262/1 |
| PROJECT PROPOSED: | 21/10/66 |
| PROJECT AUTHORISED: | 38/10/66 |
| AMOUNT APPROVED: | $6,000 |
| AMOUNT EXPENDED TO 8/1/67: | $4,179 |
| ANTICIPATED DATE OF COMPLETION: | February, 1967 |

# MT. GUNSON ORE

## 1. REVIEW OF PROGRESS

Estimates of capital and operating cost have been prepared for plants to treat oxidised copper ore similar to the composite sample used in the experimental work described in Progress Report No. 1. Oxide flotation plants to treat either 500 or 1000 tons of ore daily were considered, and a more complex plant to treat 500 tons daily by acid leaching, with solvent purification and electrolytic recovery, was also costed.

Basic assumptions used throughout were:-

| | |
|---|---|
| Ore Grade: | 1.2% Cu |
| Operating year: | 330 days |
| Power Cost: | 2 cents per kWhr. |
| Water Cost: | 30 cents per 1000 gallons. |
| Operating Labour Rates: | $50 per week of 40 hours. |

No allowance has been made for access roads or railways, or for mining costs. Capital costs include complete installations, including buildings and all services, instrumentation etc and design charges. Operating costs include no amortization or return on capital, but are otherwise complete including service charges such as insurance as well as labour, reagents, maintenance, etc.

### 1.1 Oxide Flotation Plant: 500 tons per day

In this plant crushing is carried out on a one shift 5 day week basis, while milling and all subsequent operations run on a three shift 7 day roster. Feed to the primary crushers is minus 10 inch material. Principal items of equipment are:

| | |
|---|---|
| Coarse ore bin | 20 ton capacity |
| Primary crusher | 14 x 24 inch |
| Secondary crusher | 48 inch diameter shorthead cone |
| Screen | vibrating 3 ft x 6 ft |
| Fine ore bin | 1000 ton capacity |
| Feeder-conveyor | |
| Ball mill | 8 ft diameter x 10 ft |
| Pump | 4/3 inch centrifugal |
| Cyclone | 24 inch diameter |
| Flotation cells | 14-48 inch square units |
| Concentrate thickener | 3 ft diameter x 8 ft |
| Filter | 6 ft diameter, 2 discs |
| Tailing thickener | 40 ft x 10 ft |
| Pump | 4/3 inch centrifugal |

The estimated cost of this plant, installed and housed is $570,000. Suggested working capital is $200,000.

Operating costs are estimated to be:

| | |
|---|---|
| Labour: 19 operators, 4 charge hands | costing 45 cents/ton feed |
| Stores, including reagents, spares | 60 |
| Power, at 2 cents | 49 |
| Water, at 80 cents/1000 galls | 12 |
| Total | 166 cents/ton feed |

Based on 70% recovery of copper, this is equivalent to 10 cents per pound of contained copper. Mining costs, and capital charges, are additional.

### 1.2 Oxide Flotation Plant: 1000 tons/day

Here the plant is similar to the 500 ton plant. Crushing is now made a 2 shift 5 day operation, with milling and flotation continuous as before. The estimated cost, installed and housed, is $904,000. Suggested working capital is $338,000.

Operating costs are estimated to be:

| | |
|---|---|
| Labour: 20 operators, 4 charge hands | 26 cents/ton feed |
| Stores, including reagents, spares | 60 |
| Power | 46 |
| Water | 12 |
| Total | 144 cents/ton feed |

Based on 70% recovery of copper, this is equivalent to 8.5 cents per pound of contained copper. Mining costs, and capital charges, are additional.

### 1.3 Grind, acid leach, SX, Electrolysis Plant: 500 tons/day

In this plant crushing is carried out on a one shift 5 day week, while all other operations are carried out on a 3 shift 7 day basis. The crushing and grinding circuit is identical with that in the 500 ton flotation plant described earlier except that the ball mill is replaced by a rod mill 6 ft diameter x 10 ft.

Leaching is carried out at 60% solids in a 2 stage cascade circuit, using mechanical agitation. Acid consumption is taken as 60 pounds per short ton, and 98% extraction is assumed.

Separation of liquor by filtration is not recommended: counter-current decantation is preferred. Feeding 40 foot thickeners at 20% solids, and operating at 50% underflow, washing efficiency of 99% is achieved in 4 stages. Use of a 5th stage is marginal on economic grounds. Synthetic flocculant (0.15 lb/ton) is used to assist in settling.

Liquor from No. 1 thickener is clarified in a sand filter or on a precoated drum filter before going to solvent extraction.

Five extraction stages are used in which the liquor is contacted with a 10% solution of LIX 64 in kerosene. Raffinate contains 0.05 gpl of copper. The loaded solvent is stripped in 3 stages with recycle electrolyte - reduction of 10 gpl in the copper content in passing through the tank house is assumed. The tank house is standard, using copper starting sheets prepared in a small auxiliary circuit.

Schematic flow sheets are attached, showing the principal items of equipment and the mass balance over the whole plant. These flow sheets are tentative only and may differ in detail from the final ones.

The total installed cost of the plant, including crushing and grinding (but not mining) is estimated to be $2.13m. Suggested working capital is $600,000.

Operating costs are estimated to be:

| | $p.a. | $/ton Cu produced | ¢/lb Cu produced |
|---|---|---|---|
| Labour | 115,700 | 64.2 | 3.2 |
| Power | 118,000 | 65.6 | 3.3 |
| Water | 34,320 | 19.1 | 1.0 |
| LIX-64 | 20,400 | 11.3 | 0.6 |
| Kerosene | 1,860 | 1.0 | 0.05 |
| Acid* | 217,800 | 121.0 | 6.05 |
| Flocculant supplies | 25,000 | 13.9 | 0.7 |
| Maintenance | 158,300 | 93.4 | 4.7 |
| Sundry o/heads | 75,000 | 41.6 | 2.1 |
| Totals | 797,000 | 436.0 | 22.7 |

Mining and capital charges are additional

* Based on $25/ton ex Pt. Pirie plus $18.50 transport, and no effective recycle of acid. These costs may be reduced substantially by revision of transport charges. Acid consumption may also be reduced by use of raffinate from the SX section for leaching. This would require some modification to the leaching circuit and possibly use of an additional thickener. Experimental verification of the practicability of this recycle would be required.

The whole flowsheet is somewhat tentative as it is based on data which requires experimental verification in a number of areas. Effect of variations on the overall cost are likely to be relatively small except in the case of acid cost.

The flowsheet is based on use of 100 gpm of water and operation should be possible without excessive build-up of impurities in the circuit. Operation on a large scale, or with less water, would be increasingly difficult although 50 gpm at 500 tons of ore per day should not be impossible.

### 1.4 Precipitation process

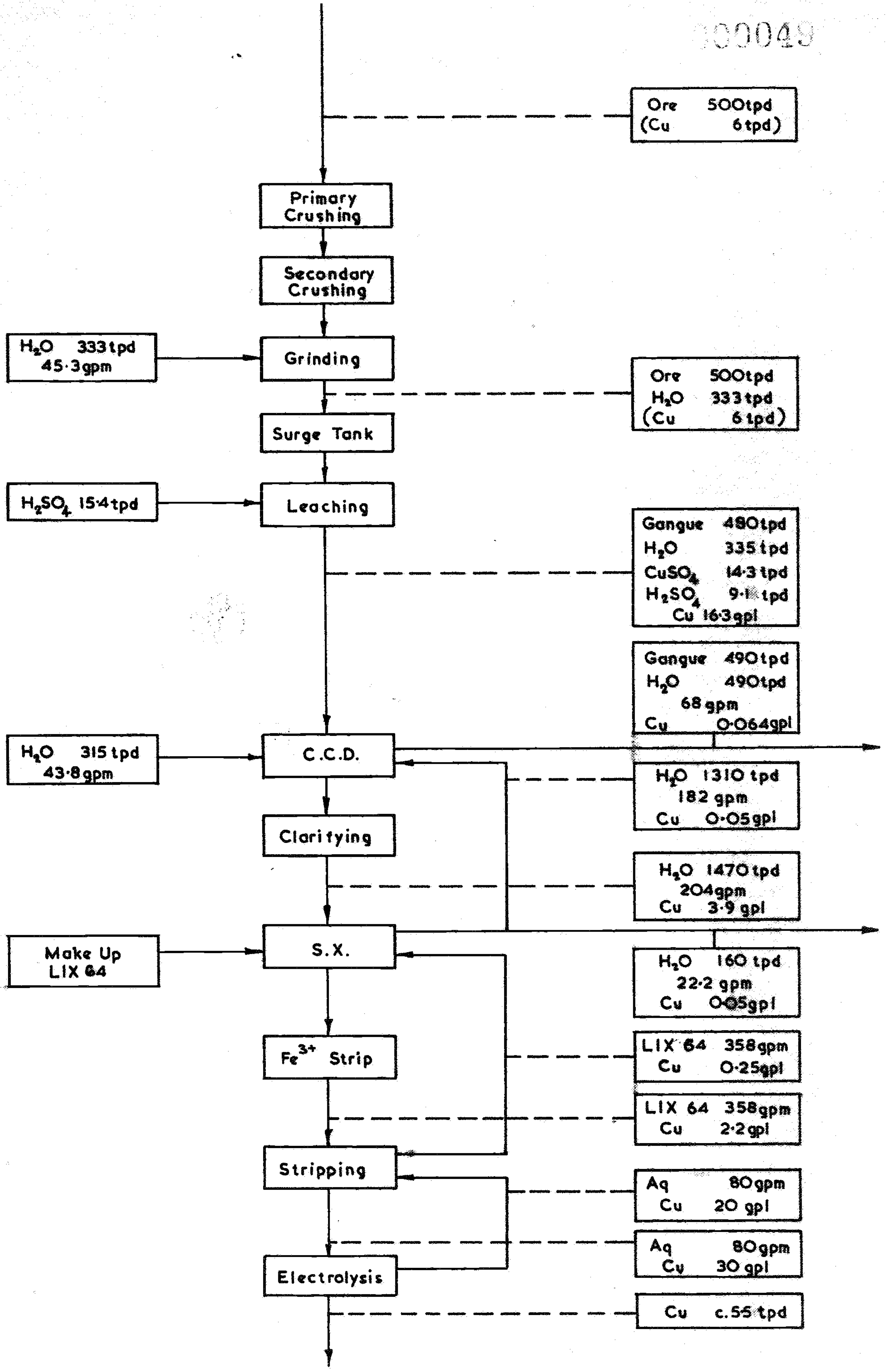
An alternative process employing precipitation with lime to remove first iron and then copper from the diluted leach liquor has been costed. The precipitate, after drying, should contain 30-40% copper as copper hydroxide mixed with gypsum. This could be leached with recycle electrolyte, thus eliminating the need for an SX plant. Significant capital economics result but the operating cost depends on cost of hydrated lime at the site. The process presents also the possibility of carrying out the electrolysis at another site, e.g. Pt. Augusta or Pt. Pirie.

## 2. WORK IN HAND

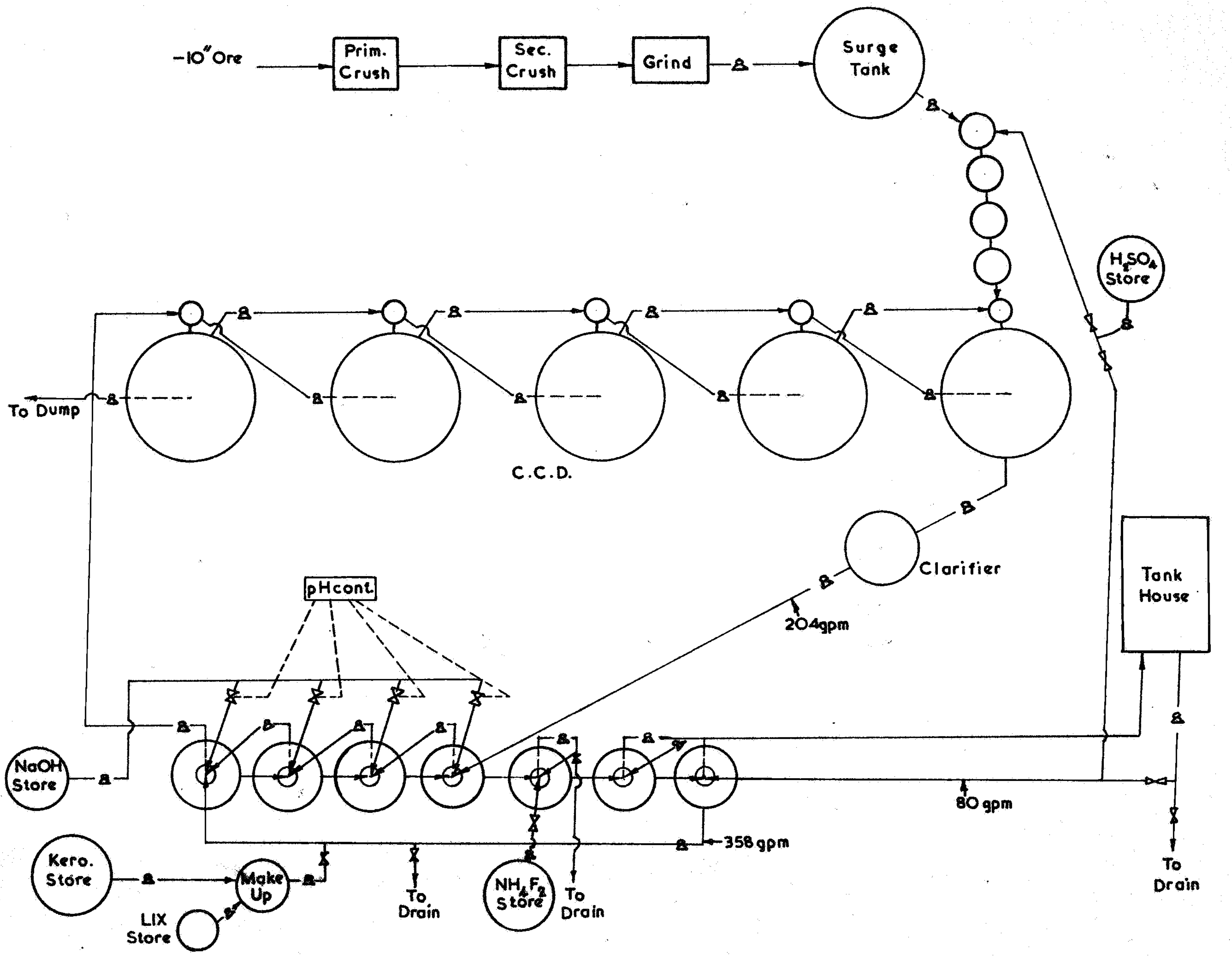
The project as originally conceived is essentially complete except for preparation of a final report.

Extensions of the project recommended for consideration are:

(a) Further work on oxide flotation
(b) Leaching experiments aimed at reducing acid consumption, and use of recycle liquors
(c) Further cost studies on alternative processes and possibly alternative plant sites.

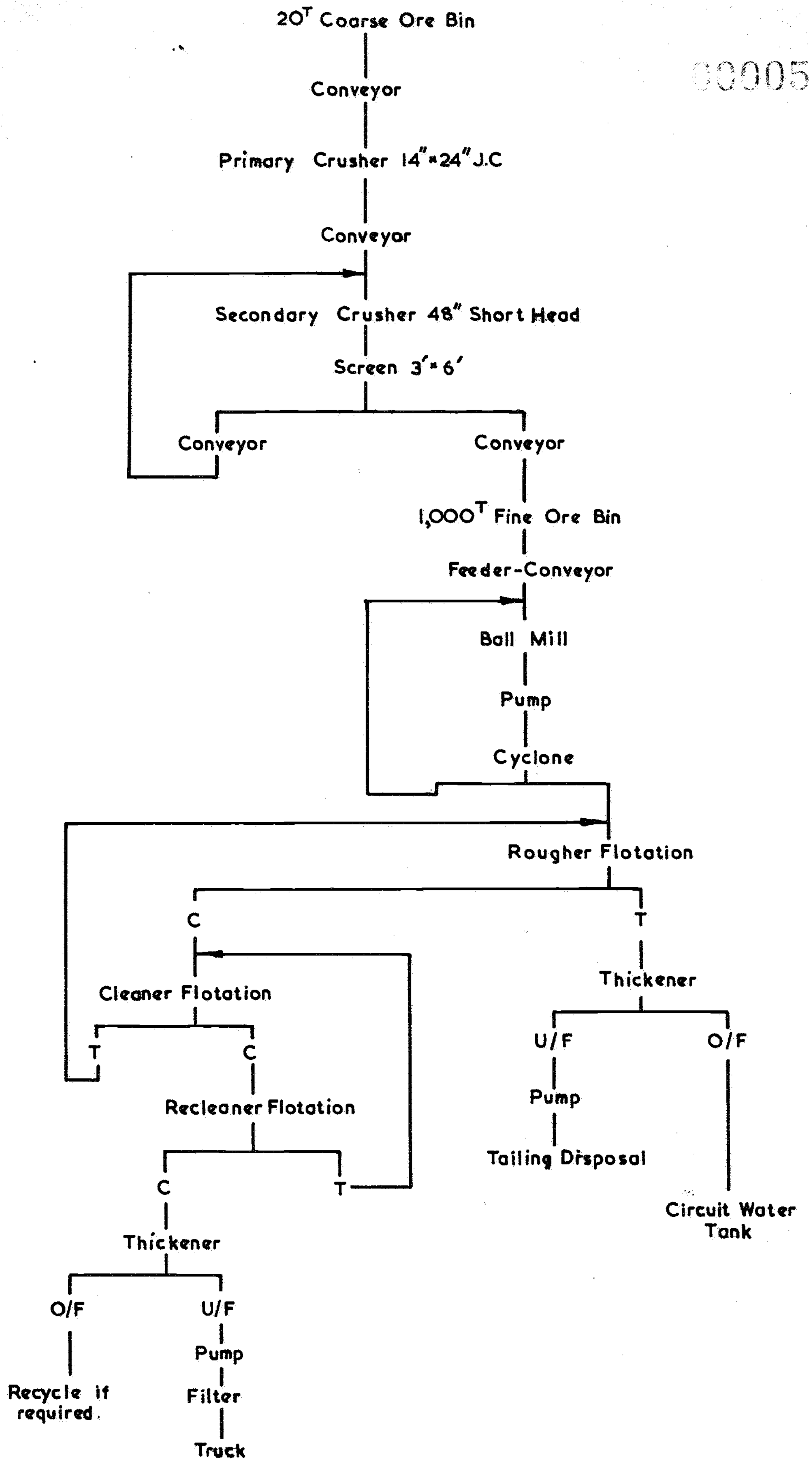


FLOWSHEET FOR EXTRACTION OF COPPER BY SX.



PRELIMINARY PLANT LAYOUT — EXTRACTION OF COPPER BY SX.

R.G. 1901



FLOWSHEET FOR FLOTATION OF OXIDE COPPER

EXHIBIT NO. 8q

GENERAL MILLS, INC. • CHEMICAL DIVISION • SO. KENSINGTON RD. • KANKAKEE, ILL. 60901 • PHONE 932-6751 • AREA 81

Mr. J.B. Evans
Austminex Pty. Ltd.
541 St. Kilda Road
Suite 20
Melbourne, Victoria.

January 25th 1967

Dear Blue,

The following is a brief summary of results obtained to date on the LIX-64 circuit.

The circuit was set up to consist of six extraction stages and 3 stripping stages. The organic consisted of 5% LIX-64 in 95% kerosine (v/v). 10% LIX-64 was preferred, but as the solvent available was limited, we assumed that chemically a 5 volume percent LIX-64, at a loading of 1.0 g/liter copper, would compare directly with 10 volume percent LIX-64, at a loading of 2.0 g/liter copper. Physically, the 5% and 10% systems will behave quite differently; therefore no attempt was made to define disengagement rates of the solvent from the aqueous. Feed to the circuit was 4.40 g/liter copper at a pH of 1.85, and the organic:aqueous ratio of extraction was maintained at about 4.4 to 1, to load the organic to 1.0 g/liter copper.

The strip section was operated at an organic/aqueous ratio of 50/1 to give a pregnant solution containing 50 g/litre copper. Strip acid entered the circuit at 150 g/liter $H_2SO_4$.

Recycles were installed on all strip stages (to recycle aqueous from the settler, back to the corresponding mixer) and were operated at a flow to give an effective organic/aqueous ratio of 5/1 in the strip mixers.

The circuit was operated for a period of 28 hours, (on an 8 hour day basis) until the feed supplied was exhausted.

Table 1 following shows average results obtained during the run.

SALES OFFICES:
ATLANTA
3390 Peachtree Rd. N. E.
Georgia 30326
BOSTON
20 Kent Street
Brookline, Mass. 02146
CHICAGO
460 So. Northwest Hy.
Park Ridge, Ill. 60068
CLEVELAND
20575 Center Ridge Roa
Ohio 44116
DETROIT
18315 West 7 Mile Road
Michigan 48219
HOUSTON
[illegible]
LOS ANGELES
[illegible]
NEW YORK
[illegible]
PHILADELPHIA
[illegible]
SAN FRANCISCO
[illegible]
ST. LOUIS
[illegible]

TABLE: 1

| Date | Operation hrs | Average flow-rates | | | Analyses g/l Cu | | | |
|---|---|---|---|---|---|---|---|---|
| | | Organic | Feed | Strip | Raffinate | Pregnant | Loaded Organic | Stripped Organic |
| 18 Jan | 8 | 104 | 26.1 | 2.6 | 0.61 | 19.7 | 1.08 | 0.01 |
| 19 Jan | 8 | 101 | 25.3 | 2.3 | 0.089 | 25.8 | 1.09 | 0.01 |
| 20 Jan | 8 | 111 | 25.1 | 2.0 | 0.056 | 32.1 | 1.00 | 0.01 |
| 23 Jan | 4 | 112 | 25.0 | 2.7 | 0.062 | 44.6 | 0.99 | 0.01 |

At the end of the run, the circuit appeared to be near equilibrium in the extraction as evidenced by the raffinate average analysis for the last 2 days, so a complete profile analysis of the extraction circuit was taken. Data are shown following as Table 2.

TABLE: 2

| Stage No: | g/l Cu Organic | g/l Cu Aqueous | Recovery of Feed % |
|---|---|---|---|
| 1 | 0.99 | 2.24 | 49.2 |
| 2 | 0.49 | 1.19 | 72.9 |
| 3 | 0.25 | 0.54 | 87.6 |
| 4 | 0.11 | 0.11 | 97.4 |
| 5 | 0.07 | 0.08 | 98.2 |
| 6 | 0.01 | 0.06 | 99.0 |

As may be seen above, the fourth stage produced a raffinate of 0.11 g/liter copper, which represents 97.4% recovery of the contained copper. Analysis of the above data (by plotting a McCabe-Thiele diagram) shows that the circuit was close to equilibrium, and it may be assumed, therefore, that this raffinate could be reasonably expected to be the result of an operating plant.

The strip circuit had not reached the 50 gram per liter pregnant solution desired at the end of the run. This was not the result of poor stripping, but simply because this portion of the circuit had not had adequate time to reach equilibrium at exhaustion of feed.

Because of this, no profile analyses were taken in this section of the circuit.

Since the leach circuit has not yet been clearly defined by AMDEL; another run was begun on the 24th January utilizing a feed to contain about 4.4 g/liter copper at a pH of 1.5, as one of the alternate leach circuits would provide a feed of this nature. This run is in progress in the same circuit as previously described, and will be completed on or about the 27th of January. Data from this circuit will be reported at the time of completion.

Once the leach is defined, the small circuit should be operated for a period of not less than 200 hours on a representative feed solution, and with 10 volume percent LIX-64, in order to define physical characteristics of the system for design of the commercial plant, and to be certain that it will behave chemically as well as the 5 volume percent LIX-64. Any possible degradation of the solvent due to $Cl^-$ content, (anticipated at about 2/g/liter) would also be checked during this run.

In summary, the data to date looks very promising, and I'll pass any further developments along as I get them.

Sincerely yours

General Mills, Inc
Chemical Division
Eldon
Eldon DeMent
Technical Representative
LIX Reagents & Systems

cc: J.E. House
D.W. Agers
C.R. Merigold
F.R. Hartley, Amdel.

EXHIBIT NO. 8r

000056

ENERAL MILLS, INC. • CHEMICAL DIVISION • SO. KENSINGTON RD. • KANKAKEE, ILL. 60901 • PHONE 932-6751 • AREA 815

Mr. J.B. Evans
Austminex Pty. Ltd,
541 At Kilda Road
Suite 20
MELBOURNE, Victoria.

27th January 1967

Dear Blue,

As noted in my previous letter of 25th January, a run of your leach liquor at pH 1.5 has been completed.

All variables, (i.e: flow-rates, staging, organic/aqueous ratios, re-cycles) were held exactly as in the previous run so that comparative data could be obtained. The circuit was again operated for a period of 28 hours, as before, on an 8 hour shift basis. Feed to the circuit was 4.45 g/liter copper at pH 1.5.

Average daily data obtained is shown in Table 1.

At the end of the run (denoted run 2) the circuit appeared to be in equilibrium, so a complete profine analysis of the circuit was taken. Data are shown in Table 2.

Examination of the data in Table 2 by means of the McCabe-Thiele diagram show that the extraction section was in equilibrium, but that the stripping section was not. It may be assumed that the raffinate produced in the fourth stage above would be close to what would be expected from an operating circuit.

Table 3 shows a comparison of recoveries obtained in the extraction sections at different pH levels, as produced by the 2 runs.

As may be seen from Table 3, pH in the feed has a very marked effect on the recovery that may be attained, and therefore should be very closely controlled.

SALES OFFICES:
ATLANTA
[illegible] Peachtree [illegible]
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BOSTON
[illegible] Street
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CHICAGO
460 So. Northwest Hy.
Park Ridge, Ill. 60068

CLEVELAND
20575 Center Ridge Roa
Ohio 44116

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18315 West 7 Mile Road
Michigan 48219

HOUSTON
[illegible]

LOS ANGELES
[illegible]
Whittier, Calif. [illegible]

NEW YORK
[illegible]
New York [illegible]

PHILADELPHIA
[illegible]
Wynnewood, Pa. [illegible]

SAN FRANCISCO
[illegible]
Palo Alto, Calif. 94303

ST. LOUIS
[illegible]
Missouri [illegible]

closely controlled.

The personnel at AMDEL have completed their investigations into the leach circuit, and are preparing 150 gallons of leach liquor. As soon as the liquor is ready, the circuit will be put into operation for a period of 200 hours of continuous operation. Data to be obtained from this run are: mixer retention time required, phase disengagement rate, and possible degradation of the reagent caused by chloride content in the feed. With such data in hand, an operating commercial circuit can be defined and designed.

The proposed leach circuit flow sheet is attached, and should produce a feed to liquid ion exchange of about 3.5 to 4.0 g/liter copper at pH of 2.0 or above. The liquor is to be produced by leaching fresh ground ore (minus 30 mesh) with a synthetic raffinate and fresh acid, and should be fairly representative of the operating plant leach.

A final report will be issued at the completion of the 200 hours run, along with recommendations for any future work.

Sincerely yours

General Mills, Inc
Chemical Division

E.R. DeMent
Technical Representative
LIX Reagents & Systems.

## TABLE: 1

| Date | Operation hrs | Average flow-rates ml/min | | | Analyses g/l Cu | | | |
|---|---|---|---|---|---|---|---|---|
| | | Organic | Feed | Strip | Raffinate | Pregnant | Loaded Organic | Stripped Organic |
| 23 Jan | 4 | 114 | 25.0 | 2.0 | 0.58 | 52.0 | 0.92 | 0.01 |
| 24 Jan | 8 | 110 | 25.0 | 2.0 | 0.61 | 54.5 | 0.88 | 0.01 |
| 25 Jan | 8 | 112 | 25.2 | 2.0 | 0.52 | 55.0 | 0.85 | 0.01 |
| 26 Jan | 8 | 112 | 25.0 | 2.0 | 0.46 | 56.0 | 0.85 | 0.01 |

## TABLE 2

| Stage No: | g/l Cu Organic | g/l Cu Aqueous | Recovery of Feed % |
|---|---|---|---|
| 1 Extractn | 0.85 | 3.35 | 24.7 |
| 2 " | 0.54 | 1.91 | 56.2 |
| 3 " | 0.27 | 1.21 | 72.7 |
| 4 " | 0.18 | 0.58 | 86.9 |
| 5 " | 0.077 | 0.46 | 89.8 |
| 6 " | 0.013 | 0.46 | 89.8 |
| 1 Strip | 0.75 | 56.5 | - |
| 2 " | 0.62 | 51.0 | - |
| 3 " | 0.013 | 21.6 | - |

## TABLE 3

| Stage No: | % Recovery Run 1, 4.40 g/l Cu pH 1.85 | % Recovery Run 2, 4.45 g/l Cu pH 1.50 |
|---|---|---|
| 1 | 49.2 | 24.7 |
| 2 | 72.9 | 56.2 |
| 3 | 87.6 | 72.7 |
| 4 | 97.4 | 86.9 |
| 5 | 98.2 | 89.8 |
| 6 | 99.0 | 89.8 |

## PROPOSED LEACH AND LIX-64 CIRCUIT

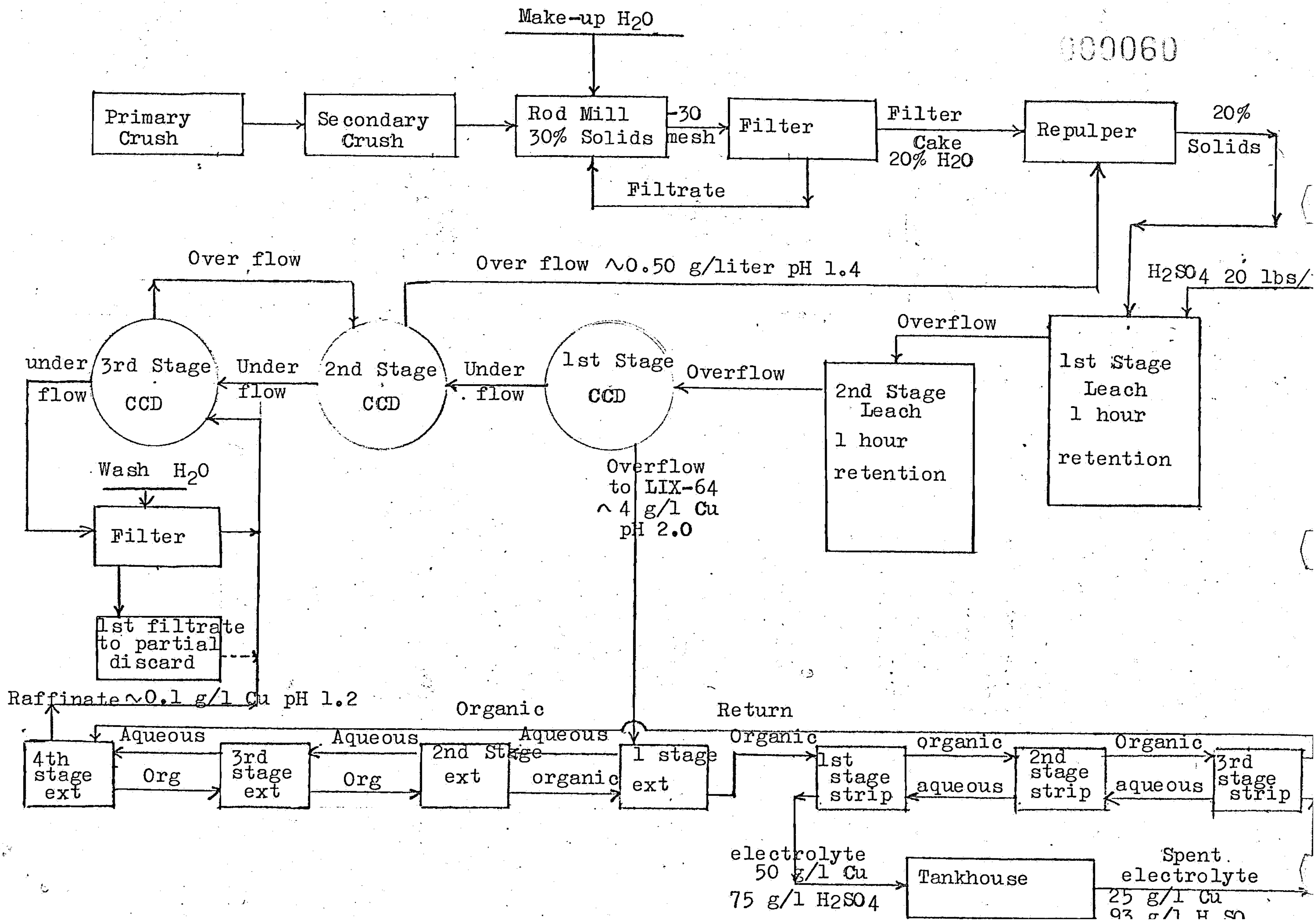

EXHIBIT NO. 8s

GENERAL MILLS, INC. • CHEMICAL DIVISION • SO. KENSINGTON RD. • KANKAKEE, ILLINOIS 60901 • PHONE 932-6751 • AREA 815

CABLE: Genmills

4/271/0
2116-67

Mr. J.B. Evans,
Austminex Pty. Ltd.,
Suite 20,
541 St. Kilda Road,
MELBOURNE, Vic.

Dear Sir:

The 200-hour run has been completed on your leach liquors as provided by AMDEL. AMDEL provided the initial leach liquor, and raffinate from the LIX-64 circuit was used for further leaching. Four complete leaching cycles were obtained during the course of the run and it would appear that the third and fourth leaches were stabilized with regard to $S04^{-2}$ and $Cl^{-}$ content, and thus were assumed to be representative of the leach liquor which would be obtained in the operating plant.

## 1. GENERAL OPERATION OF CIRCUIT

The circuit was operated with the following parameters:

1. Flow rates, average - Organic 109.8 ± 2.0 ml/min
   Feed 46.9 ± 2.0 ml/min
   Stripflow 8.6 ± 0.1 ml/min

2. Analyses, average - Feed - 3.78 ± 0.5 g/l Cu
   Raffinate - 0.37 ± 0.05 g/l Cu
   Loaded Organic - 1.74 ± 0.25 g/l Cu, 0.008 g/l Fe
   Stripped Organic - 0.28 ± 0.08 g/l Cu, 0.006 g/l Fe
   Pregnant Aqueous - 40.60 ± 2.0 g/l Cu, 0.018 g/l Fe
   Strip Feed - 22.1 ± 0.5 g/l Cu

3. LIX-64 content - 10 volume %, in lighting kerosine.

The circuit consisted of four extraction stages and three stripping stages.

Effective Organic to aqueous ratios - extraction - 2.34/1
strip - 12.77/1

It was discovered early in the run that at previously utilized flow rates (see letter of 25th January) a raffinate analysing 0.1 g Cu per litre was not being produced. On the theory that enough stages had not been provided for proper extraction, the raffinate produced from the circuit was contacted 2 minutes with fresh organic, and the aqueous resulting from this contact was analysed. Comparison with the circuit raffinate showed that no further extraction was obtained, indicating that addition of further stages would be of no benefit.

Further studies resulted in the belief that this sample of LIX-64 had a sharp cutoff point with respect to copper loading, which occurred at a pH of about 1.2. Accordingly, fresh organic was used to contact circuit raffinate (0.35 g/l Cu) which had been adjusted in pH, to determine if this was indeed the case. Results are shown in Table 1.

TABLE 1

| Raffinate pH | Cu Extracted g/l |
|---|---|
| 1.0 | 0.01 |
| 1.1 | 0.01 |
| 1.2 | 0.02 |
| 1.3 | 0.11 |
| 1.4 | 0.15 |

The results shown in this table tended to demonstrate the theorized effect.

Since LIX-64 is a hydrogen ion exchange medium, the pH of the raffinate is largely determined by the copper content of the feed, since any free acid remains as a constant. While the run was in progress, the feed copper content was varied in order to further test the above theory. Results are shown as Table 2, following.

TABLE 2

| Feed, Cu g/l | Raffinate pH | Raffinate g/l Cu |
|---|---|---|
| 3.00 | 1.3 | 0.20 |
| 3.40 | 1.2 | 0.25 |
| 3.80 | 1.2 | 0.30 |
| 4.00 | 1.2 | 0.36 |
| 4.25 | 1.2 | 0.42 |

As may be seen from the above table, extraction of copper below pH 1.2 became quite poor.

Further testing into this phenomenon was accomplished by holding the feed copper content at 4.0 g Cu/litre, and varying the input pH to the circuit. Results of this testing are shown as Table 3, following.

TABLE 3

| Feed | | Raffinate | |
|---|---|---|---|
| g/l Cu | pH | pH | g/l Cu |
| 4.0 | 2.0 | 1.2 | 0.42 |
| 4.0 | 2.2 | 1.2 | 0.40 |
| 4.0 | 2.5 | 1.3 | 0.31 |
| 4.0 | 2.8 | 1.4 | 0.15 |

Again, if the raffinate pH fell to 1.2, a decrease in the extraction efficiency was encountered.

In light of the above demonstrated effect it was assumed that pH control of the third or fourth stage would be required. However, further checking into the leaching circuit revealed that there would be a circulating load in the leaching section, and discard of tailings would yield a 99.1% recovery of copper without need for pH control in the Lix circuit. Accordingly, copper content of the feed was raised by adding $CuSO_4$ $5H_2O$ to the feed to simulate the circulating load, and the raffinate produced showed 90+% recovery, and again, losses to tailings would be only 0.6 to 0.8% of the total recovered copper. Therefore, the remainder of the run was completed without further regard to pH control, with the exception of control of pH to pH 2.0 in the leach section.

Results of leaching are not reported here, but will be reported separately by the AMDEL staff, who conducted the leaching. In brief, the last two leaches appeared to be stabilized with respect to extraneous ion build-up, and the liquors from these leaches handled identically to the first two leaches, as far as the lix circuit was concerned.

## 2. MATERIALS BALANCE

Hours of operation = 260

Feed flow = 46.9 ml/min x 60 minutes = 2.814 litres/hour
2.814 litres/hour x 260 hours = 731.6 litres

Strip flow = 8.6 ml/min x 60 minutes = 0.516 litres/hour
0.516 litres/hour x 260 hours = 134.2 litres

Copper fed to circuit

731.6 litres x 3.78 g/l Cu = 2765 g Cu contained in its feed
731.6 litres x 0.37 g/l Cu = 271 g Cu lost to raffinate

Cu recovered = 2494 g
recovery = 90.2%

Copper recovered in strip

134.2 litres x 40.6 g/l = 5448 g Cu contained in the pregnant
134.2 litres x 22.1 g/l = 2966 g Cu contained in the strip acid

Cu recovered= 2482 g

$$\text{Materials Balance} = \frac{2482}{2494} = 99.5\%$$

## 3. DESIGN DATA

When designing an lix plant, two points of information must be developed. These are: 1. Retention time required in the mixer, both for effective extraction and stripping and 2. Settler area required for the system, both in extraction and stripping.

### 1. Mixing time requirements

Mixing time requirements for extraction were determined by shaking the organic with the aqueous at an O/A ratio of 2.5 for various time intervals. The aqueous was then analysed for Cu content. Data are shown as following in Table 4.

TABLE 4

| Contact Time min. | Raffinate g/l Cu |
|---|---|
| 0.25 | 0.68 |
| 0.50 | 0.42 |
| 0.75 | 0.40 |
| 1.0 | 0.41 |

As can be seen from the table, retention times of 0.5 minutes or more result in 99+% extraction of the extractable copper. For safe design, one minute of retention time should be allowed in the extraction mixers.

For the stripping time requirements, testing was performed as above, but the stripped organic was analysed for Cu content rather than the aqueous and the contact ratio was O/A = 12/1. Results are shown as Table 5.

TABLE 5

| Contact Time Min. | Organic g/l Cu |
|---|---|
| 0.5 | 0.51 |
| 1.0 | 0.20 |
| 2.0 | 0.20 |
| 4.0 | 0.19 |

Necessary contact time appears to be 1 minute, minimum and the circuit should be designed to allow 2 minutes retention time in the stripping mixers.

2. <u>Settler area requirements</u>

These requirements were determined during the course of the 200-hour run, by fixing the flow rates to the circuit and gradually decreasing the settler area in both strip and extraction until such time as the settlers "flooded" or became filled with emulsion.

Data resulting from this test are shown as Table 6 following and figure 1 attached.

TABLE 6

| Total Flow gal/min/sq ft | Dispersion Band Width inches | |
|---|---|---|
| | extraction | strip |
| 0.48 | 0.0 | - |
| 0.67 | 0.0 | 0.25 |
| 0.89 | 0.0 | 0.25 |
| 1.07 | 0.0 | - |
| 1.23 | - | 0.50 |
| 1.35 | 0.13 | - |
| 1.49 | - | 0.75 |
| 1.85 | 0.20 | 1.50 |
| 2.52 | - | 3.00 |
| 2.81 | 0.25 | - |
| 3.20 | 0.50 | - |
| 3.88 | 4.00 | Flooded, 7+ in. |

As may be seen from the attached curves, flow rates increased beyond a critical point resulted in flooding of the circuit. General Mills has found that a 2 inch dispersion band is relatively safe, and the circuit may therefore be designed for phase disengagement rates of 3.0 gal/min/sq ft in the extraction, and 2.0 gal/min/sq ft in the strip.

## 4. DEGRADATION OF SOLVENT

No apparent degradation of the solvent as a result of $Cl^-$ attack was encountered. As a matter of fact the maximum loading of the organic slowly increased from 2.22 to 2.94 g Cu per litre during the progress of the run. This was probably a result of losses of kerosine to selective solubility in the raffinate and volatility of the kerosine employed. Such action has been assumed to be so for purposes of calculation of solvent losses shown below.

## 5. SOLVENT LOSSES

Organic was introduced to the circuit at a total volume of 8.00 litres as 10 volume % LIX-64 and 90 volume % lighting kerosine. Loading capacity of the copper at the beginning of the run was 2.22 g Cu per litre and at the end of the run was 2.94 g Cu per litre. Volume of solvent remaining at the end of the run was 5.293 litres. Addition of 2.0 litres of kerosine reduced the loading capacity from 2.94 g Cu per litre to 2.24 g Cu per litre, or the loading originally obtained. This addition brought the solvent volume to 7.293 litres, which, when subtracted from the initial inventory of 8.0 litres, represents a mixed solvent loss of 0.707 litres. Since 731.6 litres of aqueous were put through the circuit the above losses become:

$$\text{kerosine} = \frac{2.0}{731.6} = 0.26 \text{ litres/1000 litres or gal/1000 gal}$$

$$\text{mixed LIX-64} = \frac{0.707}{731.6} = 0.096 \text{ litres/1000 litres or gal/1000 gal}$$

Loss costs

assume -4.0 g/l feed = 33.36 lbs Cu/1000 gallons
kerosine = 20¢ /gallon
10% LIX-64 = 218¢ /gallon
0.26 gal/1000 gallons x 20¢ = 5.20¢
0.096 gal/1000 gallons x 218¢ = 20.93¢

Total cost 26.13¢

$$\text{cost/lb Cu} = \frac{26.13¢}{33.36 \text{ lbs}} = 0.78¢$$

## 6. IRON LOADING

Fe loading onto the organic will not be a particular problem, as a very low Fe loading was encountered (0.0080 g/l Fe) and stripping of the Fe from the loaded reagent was not very effective (0.006 g Fe/l) with the result that for every 2.0 g of Cu loaded onto the organic, there will be about 0.002 g of Fe or a ratio of 1000/1 Cu/Fe. A bleed stream will of course be required to control the Fe, but if it is allowed to build to 5 g per litre prior to bleeding, a 0.4% bleed stream to the circuit feed will suffice to control it at this level.

## 7. CIRCUIT DESIGN

Assumptions:

LIX-64 - 10 volume % in 90 volume % kerosine
Feed - 4.0 g Cu per litre at pH 2.0
Raffinate - 0.40 g Cu per litre at pH 1.2
Loaded organic - 2.0 g Cu per litre
Stripped organic - 0.20 g Cu per litre
Spent electrolyte - 25.0 g Cu per litre 100 g per litre free $H_2SO_4$
Electrolyte - 50.0 g Cu per litre 62.5 g per litre free $H_2SO_4$
Feed flow - 260 gallons/minute organic flow - 578 gal/min
O/A ratio extraction - 2.22 strip flow = 41 gal/min
O/A ratio strip - 14.0

Calculations:

Extraction

organic flow = O/A 2.22 x 260 gallons/minute = 578 gal/min
578 gallons/minute + 260 gallons/minute = total flow 838 gallons/minute or 840 gallons/minute, total flow

Mixer required = 840 gallons/minute x 1 minute retention = 840 gallon capacity or 114 cubic feet 36 inches deep + 6 inch free board = 42 inches.
42 inches deep x 74 inches wide x 74 inches long

Settler required = 840 gallons/minute ÷ 3 gal/min/ sq ft = 280 sq ft or 42 inches deep, 10 ft x 28 ft
4 such stages required for the extraction section

The strip circuit will be operated at a through-put ratio of O/A = 14.0, but we find that in order to maintain effective stripping the ratio must be held at not greater than O/A = 5. This may be accomplished by recycling aqueous from the settler of the same stage, thus the calculations above are based on O/A = 5 for stages 1 and 2. The third stage must be operated aqueous continuous in order to eliminate entrainment of strong acid and high copper-bearing aqueous in the barren organic, which would report to the last stage of extraction and cause a higher than normal raffinate. In order to maintain aqueous continuous conditions, aqueous must be recycled from the settler to maintain an O/A ratio of 1 in this mixer, thus the calculations above are based on this ratio for the third stage.

Strip Section O/A ratio = 5/1

Stage 1 and 2 Mixer Required 578 gallons/minute ÷ 5 = 116 gallons/minute aqueous..
578 + 116 = 694 gallons/minute x 2 minutes retention time = 1388 gallons x 0.134 = 192 cu ft
42 inches deep x 96 inches x 96 inches
Settler Required 694 gals/min ÷ 2.0 gal/min/sq ft = 347 sq ft
settler = 42 inches deep x 12 feet x 30 feet
2 such stages required

Stage 3 O/A ratio = 1/1

578 gallons/minute x O/A = 1 = 578 gallons/minute aqueous
total flow = 578 x 2 = 1156 gallons/minute.
1156 gallons/minute x 2 minutes retention = 2312 gallons capacity required.
2312 gallons x 0.134 = 309 $ft^3$.
42 inches deep x 122 inches x 122 inches
Settler Required 1156 gallons/minute ÷ 2 gal/min/sq ft = 578 sq ft.
42 inches deep x 12 feet x 48 feet

## 8. SOLVENT INVENTORY CONSIDERATIONS

Extraction section - 4 stages of 280 sq ft each = 1120 sq ft
1120 sq ft x 8 inches deep = 747 cu ft
747 cu ft x 7.48 gal/cu ft = 5588 gallons in settlers
Mixers contain 578 gals
x 4 = 2312 gallons in mixers
sub total 7900 gallons in extraction

Strip section
Stage 1 and 2 = 360 sq ft x 8 inches deep = 240 cu ft
240 cu ft x 7.48 gallons/cu ft = 1795 gal x 2 stages
= 2590 gal 2590 gal
Stage 3 = 578 sq ft x 8 inches deep = 385 cu ft
385 cu ft x 7.48 gallons/cu ft = 2880 gal
mixers - stage 1 and 2 = 2310 gal 2310

stage 3 = 560 gal 560 gal.
sub total 8340 gallons.

Gallons contained in circuit

extraction = 7900 gals
strip = 8340 gals

16240 gals
+ 20% surge = 3248 gals

inventory total 19488 gals
rounded = 20000 gals
at 10% LIX-64 = 2000 gals

= 40 drums at $1000/drum = $40000
and 18000 gallons kerosine at $0.20 = $ 3600

$43600

<u>Yearly anticipated losses</u>

| | |
|---|---|
| Kerosine | 39,000 gallons |
| LIX-64 | 1,100 gallons |

If any of the foregoing is unclear, please do not hesitate to get in touch with me.

Sincerely yours,
General Mills Inc.
LIX Reagents & Systems

E.R. DeMent

E.R. DeMent
Technical Representative.

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RESEARCH FOR INDUSTRY

**THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES**

CONYNGHAM STREET - FREWVILLE - SOUTH AUSTRALIA
TELEPHONE 791662 · TELEGRAMS 'AMDEL' ADELAIDE

Please quote this reference in your reply:
IC 3/262/1
Your reference:

4th April, 1967

Mr. J.B. Evans,
Austminex Pty. Ltd.,
Suite 20,
541 St. Kilda Road,
MELBOURNE. Vic.

# MT. GUNSON ORE

## PROGRESS REPORT NO. 3
(amended)

To 31st March, 1967

Investigation and Report by: J.M. Clayton, L.J. Weir.

Officer in Charge, Industrial Chemistry Section: R.E. Wilmshurst.

R.E. Wilmshurst
for P.A. Young
Director.

| | |
|---|---|
| PROJECT NUMBER: | 3/262/1 |
| PROJECT PROPOSED: | 21/10/66 |
| PROJECT AUTHORISED: | 28/10/66 |
| AMOUNT APPROVED: | $6,000 28/10/66<br>$7,500 31/1/67 |
| AMOUNT EXPENDED TO 19/3/67: | $14,216 |
| ANTICIPATED DATE OF COMPLETION: | April, 1967 |

# MT. GUNSON ORE

## 1. REVIEW OF PROGRESS

Further estimates of capital and operating costs have been prepared, using a rather different basis from those reported earlier.

Basic assumptions used in the present estimates are:

| | |
|---|---|
| Ore grade: | 1.2% Cu |
| Operating year: | 330 days |
| Power cost: | 2 cents per kWh |
| Water cost: | 80 cents per 1000 gallons |
| Operating Labour rates: | $50 per week of 40 hours |

Ore containing copper essentially as sulphide is treated by flotation only. Oxidised ore, containing copper chiefly as carbonates, is treated by acid leaching and solvent extraction to yield crystalline copper sulphate. Provision is made for leaching of flotation tailing should this be required. Estimates are presented for plants to handle either 500 or 1000 tons of minus 10 inch ore daily.

No allowance has been made for access roads or railways, or for mining costs. Capital costs include complete installations, including buildings and all services, instrumentation etc. and design charges. Operating costs include no amortization or return on capital, but are otherwise complete including service charges such as insurance as well as labour, reagents, maintenance, etc.

### 1.1 Sulphide Flotation Plant

Costs for plant to float sulphide, with tail to reject or leach.

Capital Costs

| | Equipment | Size | H.P. (installed) | Capital Cost $ |
|---|---|---|---|---|
| Crushing | Coarse Ore Bin | 20 ton | - | 2,000 |
| | Primary Crusher | 14 x 24-in | 60 | 14,000 |
| | Conveyor | | 2 | 1,000 |
| | Secondary Crusher | 48-in dia. Shorthead Cone | 150 | 38,000 |
| | Screen | 3 x 6 ft | 5 | 1,000 |
| | Conveyors | 200 ft enclosed | 5 | 30,000 |
| Ground Storage + reclaim tunnel x | Fine Ore Bin | 1,000 ton, steel | 0 | 30,000 |
| | Feeder Conveyor | 50 ft enclosed | 2 | 15,000 |
| | | | 224 | 131,000 |

Note: Crushing equipment is common for 500 or 1,000 ton/day plants, operating on 2 shifts or 3 shifts as required.

| | | 500 T/Day | | | 1,000 T/Day | | |
|---|---|---|---|---|---|---|---|
| | | Size | H.P. | $ | Size | H.P. | $ |
| Grinding | Equipment | | | | | | |
| | Ball Mill | 8 x 10 ft | 350 | 55,000 | 10 x 12 ft | 700 | 85,000 |
| | Pump | 4/3-in | 5 | 1,000 | 6/4-in | 5 | 2,000 |
| | Cyclone | 24-in | - | 500 | 36-in | - | 1,500 |
| Flotation | Cells (incl. pumps etc.) | 14 x 48-in units | 130 | 25,000 | 28 x 48-in units | 200 | 53,000 |
| Concentrate | Thickener | 8 x 6 ft | 1 | 5,000 | 12 x 6 ft | 1 | 10,000 |
| | Filter Feed Pump | | 1 | 1,000 | | 1 | 1,000 |
| | Filter (incl. vacuum pumps, receivers etc) | 6 ft x 7-Disc | 20 | 30,000 | 6 ft x 4-Disc | 20 | 50,000 |
| Tailing | Thickener | 40 x 10 ft | 2 | 15,000 | 60 x 10 ft | 2 | 30,000 |
| | Filter feed pump | | 1 | 1,000 | | 1 | 1,000 |
| | Filter* | 6 ft x 8 Disc | 10 | 30,000 | 2 x 6ft x 8 Discs | 20 | 60,000 |
| | Disposal Pumps | 6/4-in | 10 | 2,500 | 2 x 6/4-in | 20 | 5,000 |
| Water | Piping | | 10 | 10,000 | | 10 | 10,000 |
| Pulp | Piping, Pump Sumps etc. | | - | 10,000 | | - | 15,000 |
| Reagent | Tanks, Feeders, Piping | | 5 | 10,000 | | 5 | 10,000 |
| General | Items | | | 13,000 | | | 16,500 |
| Total | | | 545 | 209,000 | | 985 | 350,000 |
| | Plus crushing | | 224 | 131,000 | | 224 | 131,000 |
| | | | 769 | 340,000 | | 1,209 | 481,500 |
| Installation cost (50% on) | | | | 170,000 | | | 240,500 |
| | | | | 510,000 | | | 721,500 |
| Electrical installation | | | | 40,000 | | | 60,000 |
| Mill Building | | | | 60,000 | | | 100,000 |
| | | | | 610,000 | | | 881,000 |

* This unit utilizes some auxilliaries allowed for in concentrate thickener.

## Operating Costs

| | Men | |
|---|---|---|
| 1. Manpower per shift:- | 500 T/Day | 1,000 T/Day |
| Crushing and Conveyors | 1 | 1 |
| Milling and Classification | 1 | 1 |
| Flotation | 1 | 1 |
| Thickening, Concentrate and tailing Filtration, and Tailing disposal | 1 | 1 |
| General Hand | 1 | 2 |
| Supervision | 1 | 1 |
| | $ | $ |
| Men per shift @ $50 per week | 250 | 300 |
| Supervisor @ $75 " " | 75 | 75 |
| | 325 | 375 |
| Total per 5 day week for 3 shifts | 975 | 1125 |
| Plus 25% for shift allowance, holiday and sick pay etc. | 245 | 281 |
| | 1210 | 1406 |
| Cost per day | 242 | 281 |
| Cost per ton | 0.48 | 0.28 |

| 2. Overheads | 500 T/Day<br>$/T | 1,000 T/Day<br>$/T |
|---|---|---|
| Assume 50% of direct operating labour cost | 0.24 | 0.14 |

| 3. Stores (including spares, grinding media, flotation reagents etc.) | 500 T/Day<br>$/T | 1,000 T/Day<br>$/T |
|---|---|---|
| Crushing and Grinding | 0.10 | 0.10 |
| Reagents* | 0.05 | 0.05 |
| Flotation operating | 0.05 | 0.05 |
| Miscellaneous | 0.05 | 0.05 |
| Total Cost | 0.25 | 0.25 |

* Reagent costs:-

| Reagent | lb/ton of feed | $/lb | $ per ton of feed |
|---|---|---|---|
| Collector-frother | 0.2 | 0.25 | 0.05 |

| 4. Power | 500 T/Day | 1,000 T/Day |
|---|---|---|
| Installed H.P. | 769 | 1,209 |
| Consumption HP/HR* | 630 | 1,125 |
| Cost @ 2c per unit, $ per ton | 0.45 | 0.40 |

* average over 24 hours

| 5. Water | 500 T/Day | 1,000 T/Day |
|---|---|---|
| Water required, gal per day | 336,000 | 672,000 |
| Less water recovered from tailing by thickening and filtration (assume 20% moisture cake) | 308,000 | 616,000 |
| Net water required | 28,000 | 56,000 |
| Cost @ $0.8 per 1000 gal, $ per ton | 0.04 | 0.04 |

| 6. Maintenance | 500 T/Day<br>$ per ton | 1,000 T/Day<br>$ per ton |
|---|---|---|
| Assume 8% of fixed capital - Cost | 0.30 | 0.23 |

| Total Operating Costs | 500 T/Day<br>$ per ton | 1,000 T/Day<br>$ per ton |
|---|---|---|
| Labour | 0.48 | 0.28 |
| Overheads | 0.24 | 0.14 |
| Stores | 0.25 | 0.25 |
| Power | 0.45 | 0.40 |
| Water | 0.04 | 0.04 |
| Maintenance | 0.30 | 0.23 |
| Total | 1.76 | 1.34 |

Taking the ore grade as 1.2% or 24 lb Cu per short ton, and the overall recovery as 90%, the operating costs per pound of copper in the product would be 8.1 cents or 6.2 cents for 500 or 1000 tons per day respectively.

## 1.2 Leaching and Recovery Plant

This plant accepts as feed either tailing from the flotation plant, or ore direct from the ball mill circuit. In either case the solid is leached at 20% solids using raffinate from the solvent extraction circuit as a source of acid. Fresh acid as required is added directly to the leaching stages. All costs detailed below are exclusive of crushing and grinding costs.

### Capital Costs

| | | $ |
|---|---|---|
| Leaching: | At 20% solids, 2 hour hold up in 3 stages based on Pt. Pirie P.H. Tanks for agitation Steel at $300/ton, lining $3/ft | 71,000 |
| C.C.D.: | Feed No. 1 20%, other 17%. Underflow 50% Using 0.15 lb float/ton, 3 x 40 ft. thickeners and drum filter Based on figures ex Eimco, Warman | 94,000 |

| | | |
|---|---|---|
| **Clarifier:** | Precoat drum filter + vacuum pump, etc. | 70,000 |
| **SX:** | 4 extraction stages | 48,000 |
| | 3 stripping stages | 30,000 |
| **Crystallisers:** | 4 x 40 ft Svenson Walker crystallisers | 25,000 |
| **Centrifuge:** | 40 inch suspended basket, rubber covered | 8,000 |
| **Product Dryer:** | rotary 4 ft x 25 ft | 20,000 |
| **Product Packaging Unit:** | weighing, bagging | 10,000 |
| **Tank farm:** | Acid, reagent tanks | 35,000 |
| **Process Pumps:** | U/F, liquor pumps | 15,000 |
| | Total purchased equipment cost: | 416,000 |

| | | | |
|---|---|---|---|
| **Additions:** | Installation cost, weighted | 40% | |
| | Instrumentation | 10% | |
| | Piping | 50% | |
| | Electrical | 30% | |
| | Buildings, etc. | 15% | |
| | Utilities | 20% | |
| | Design and engineering fees | 20% | |
| | Contingency | 25% | |
| | | 210% | 875,000 |
| **Solvent inventory** | | | 45,000 |
| | Total Fixed Capital | | $1.34m. |

## Operating Cost

**Labour:**

| | |
|---|---|
| Leaching<br>C.C.D.<br>Clarifier | 2 operators |
| SX | 1 operator |
| Crystallisers<br>Centrifuge<br>Dryer | 2 operators |
| Relief | 1 operator |

Total 6 Operators @ $50 p.w. x 4 shifts $62,300

| | | |
|---|---|---|
| | | b/f $62,300 |
| Packing, general duties 2 operators day shift only | | $5,200 |
| Increase shift super over p3 value - now require 4 supers @ $100 p.w. i.e. addition $125 p.w. | | 6,500 |
| General superintendent $750/month | | 9,000 |
| | | 83,000 |
| Plus 25% allowance for shift allowance, leave, sick pay, etc. | | 21,000 |
| Total Direct Labour | | $104,000 |
| **Power**: 156 h.p. at 75% load | | 13,000 |
| **Water**: Total available 72,000 g.p.d.<br>Required for mill 28,000<br>Leaving 44,000 g.p.d. at $0.80/1000 galls | | 11,600 |
| **Solvent**: using figures of Mr. De Ment | | |
| Lix 64 1100 galls @ $20 | $22,000 | |
| Kero 39,000 galls @ ¢0.20 | 7,800 | 30,000 |
| **Other Reagents**: including flocculant | | 25,000 |
| **Acid**: 40 pounds/short ton at $40 | | 132,000 |
| **Fuel Oil**: for dryer, 130 tons p.a. @ $20 | | 3,000 |
| **Maintenance**: 8% of fixed capital | | 103,000 |
| **Sundry O/heads**: 50% of direct labour | | 52,000 |
| Direct Operating Cost | | $474,000 |

## Direct Operating Costs, Leaching Plant, 500 tons/day

| | $ p.a. | $/ton ore |
|---|---|---|
| Labour | 104,000 | 0.63 |
| Power | 13,000 | 0.08 |
| Water | 11,600 | 0.07 |
| Solvent | 30,000 | 0.18 |
| Acid | 132,000 | 0.80 |
| Other reagents | 25,000 | 0.15 |
| Fuel oil | 3,000 | 0.02 |
| Maintenance | 103,000 | 0.63 |
| Overheads | 52,000 | 0.31 |
| | 473,600 | 2.88 |

## Operation at 1000 tons/day

The plant costed above for 500 ton/day would have little spare capacity and the size of each unit would have to be increased to handle 1000 tons/day. As a reasonable first approximation the fixed capital cost is taken to be 1.7 times that for the smaller plant, or $2.28m. Individual cost items except labour and water are doubled.

Direct Operating Costs, Leaching Plant, 1000 tons/day

| | $ p.a. | $/ton ore |
|---|---|---|
| Labour | 104,000 | 0.31 |
| Water | 11,600 | 0.04 |
| Maintenance | 175,000 | 0.53 |
| Others | 510,000 | 1.55 |
| | 800,600 | 2.43 |

Crushing and grinding charges are additional. Capital installation required is approximately $0.4m for 500 tpd. and $0.5m for 1000 tpd., these amounts being included in the flotation plant estimates earlier. Direct operating costs in this section of the plant are approximately $1.0 per ton of ore or 4 cents per pound of recovered copper.

## 1.3 Leaching of Flotation Middlings

Experimental work on flotation of oxidised (i.e. carbonate) ore have indicated that it is possible to prepare 3 products:

| | |
|---|---|
| Concentrate | 1.1% of ore by wt., containing 35% Cu representing 35% of total copper |
| Middling | 24% of ore by wt., containing 2.5% Cu representing 46% of total copper |
| Tailing | 75% of ore by wt., containing 0.3% Cu representing 20% of total copper. |

Another possible approach is therefore to subject the oxide ore to flotation, recovering a concentrate for sale as such. The middling fraction could be leached and treated by solvent extraction for copper sulphate production, while the tailing would be rejected. Overall recovery would be around 80%, considerably better than in flotation alone.

The estimates which follow are based on treatment of 1000 tons of ore daily by this route (with costs as in 1.1 above), followed by leaching of the 240 tons per day of 2.5% middling.

## Capital Costs

| | | |
|---|---|---|
| Leaching: | This is taken to be proportional to ore throughput | $36,000 |
| C.C.D.: | The estimate is scaled down from the 500 ton plant using a 0.7 power law | 79,000 |
| Clarifier: | Again based on 500 tons and 0.7 power law | 60,000 |
| Solvent Extraction: | Dependent on copper throughput, same at 500 tpd ore | 78,000 |
| Crystallisers: | As before | 25,000 |
| Centrifuge: | | 8,000 |
| Dryer: | | 20,000 |
| Packaging: | | 10,000 |
| Tankage: | | 35,000 |
| Pumps, etc: | | 15,000 |
| | Total purchased equipment cost | $366,000 |
| Additions: | As before, 210% | 705,000 |
| Solvent Inventory: | As before | 45,000 |
| | Total Fixed Capital | $1,116,000 |

## Operating Cost

| | | |
|---|---|---|
| Labour: | As before for 500 tpd. | 104,000 |
| Power: | As before | 13,000 |
| Water: | | 11,600 |
| Solvent: | | 30,000 |
| Reagents: | Reduced 50% | 12,500 |
| Fuel Oil: | As before | 3,000 |
| Acid: | Assumed 90lb/ton of middling at $40/ton | 162,000 |

| | | |
|---|---|---|
| Maintenance: | 8% of fixed capital | $85,600 |
| Sundry O/heads: | 50% of direct labour | 52,000 |
| | Direct Operating Cost | $473,000 |

Direct Operating Costs, Middling Leach, 1000 tons ore/day

| | $ p.a. | $/ton ore |
|---|---|---|
| Labour | 104,000 | 0.32 |
| Power | 13,000 | 0.04 |
| Water | 11,600 | 0.04 |
| Solvent | 30,000 | 0.09 |
| Acid | 162,000 | 0.49 |
| Other reagents | 12,500 | 0.04 |
| Fuel oil | 3,000 | 0.01 |
| Maintenance | 85,600 | 0.26 |
| Overheads | 52,000 | 0.16 |
| | 473,700 | 1.44 |

## 2. WORK IN HAND

No further experimental work is planned, although some flotation results remain to be reported in detail.

Preparation of the final report is in progress.

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3/259/2

Progress Report No. 3A
March 1967

2037

# MT. GUNSON ORE

THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES
Adelaide South Australia

RESEARCH FOR INDUSTRY

**THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES**

CONYNGHAM STREET - FREWVILLE - SOUTH AUSTRALIA
TELEPHONE 791662 · TELEGRAMS 'AMDEL' ADELAIDE

*Please quote this reference in your reply:*

IC 3/262/1

*Your reference:*

20th April, 1967

Mr. J.B. Evans,
Austminex Pty.Ltd.,
Suite 20,
541 St. Kilda Road,
MELBOURNE. Vic.

## MT. GUNSON ORE

## PROGRESS REPORT NO. 3A

To 31st March, 1967

Investigation and Report by: L.J. Weir

Officer in Charge, Industrial Chemistry Section: R.E. Wilmshurst

for P.A. Young
Director.

## 1. REVIEW OF PROGRESS

Flotation testing of oxidised copper minerals indicates a maximum recovery of approximately 80%, of which 35% reports in a high grade concentrate. Recycling of all flotation middlings resulted in a decrease in recovery to approximately 60%, at a concentrate grade of up to 35% Cu; however it is considered feasible that overall flotation recovery can be maintained at 80% by directing low-grade middlings containing 45% of the total copper in 25% by weight of the ore can be directed to leaching.

## 2. WORK IN HAND

All test work is completed, and a final report on the project is being prepared.

## 3. MATERIAL EXAMINED

As in Progress Report No. 1.

## 4. EQUIPMENT

As in Progress Report No. 1.

## 5. ANCILLARY MATERIALS

Phosphorous penta-sulphide, commercial grade.
Lime, commercial grade.
Sodium meta-silicate (METSO), commercial grade.
Secondary Butyl Xanthate (S.B.X.), Cyanamid Australia Pty.Ltd.
Aerofloat 208, Cyanamid Australia Pty.Ltd.
Aero depressant 633, Cyanamid Australia Pty.Ltd.
Reagent S3461, Cyanamid Australia Pty.Ltd.
Aerofroth 65, Cyanamid Australia Pty.Ltd.

## 6. EXPERIMENTAL PROCEDURE AND RESULTS

### 6.1 Flotation

### Tests Using Collector For Sulphide Mineral, and Activation of Oxidised Minerals

Further batch flotation tests were done similar to those reported in Progress Report No. 1. The variables investigated were:-

Use of gangue dispersants and depressants
Use of alternative collectors and activators
Extension of flotation time.

Results of the tests, and conditions used, are shown in Table 5B.

### Cyclic Tests

Batch tests were done, consisting of a number of cycles, in which middling products were re-treated in the succeeding cycle as shown in Figure 2.

Conditions for cyclic tests are shown in Tables 7 and 9, and results in Tables 8 and 10.

## 6.2 Size Analysis of Flotation Tailing

A sample of tailing from Test 13 on sample B4, was sized, and fractions assayed for copper.

The results are shown in Table 11.

## 6.3 Mineralogical examination of Flotation Tailing

Size fractions of tailing from flotation test 13 were examined microscopically, a report of which is included in Appendix A.

## 6.4 Analysis of Natural Brine-Water

Analysis of lagoon water from the mine area is shown in Appendix B.

## 7. DISCUSSION

Batch flotation tests at extended time of operation in the roughing stages increased the copper recovery significantly over that previously reported. A copper recovery of 82.5% was obtained on the composite ore sample (Test 27, Table 5B). Oxide concentrate grade was raised in this test to 36.8% Cu; however 56% of the copper was then contained in middling at a grade of 3.3% Cu. The weight of middling increased from approximately 10% in previous tests, to 23.5% in test 27. The increase in rougher concentrate weight floated in this and other tests at similar conditions, was probably due to the higher pH used using lime or soda ash addition.

The use of sodium silicate at low levels, which also produced a rise in pulp pH, as a gangue dispersant and depressant, gave the highest recovery obtained of 83.7% (Test 33, Table 5B). Concentrate grade was raised to 43.2% Cu.

The use of sodium silicate and sodium meta-silicate at amounts of from 2 to 12 lb/ton, and of an organic depressant, gave generally poorer grade/recovery figures (tests 36 to 46 and 49 to 51 inclusive, Table 5B).

Cyclic batch flotation tests, used a mixed collector found suitable in previous tests, with a low addition of sodium meta-silicate and without, failed to recover sufficient of the copper reporting in low-grade middlings to raise the concentrate recovery above the earlier predicted level of approximately 60%.

## 8. CONCLUSIONS AND RECOMMENDATIONS

The recovery predicted in plant flotation operation involving re-cycling of middlings on ore of the type investigated is approximately 60% in a concentrate containing approximately 25% Cu. It is possible that concentrate grade up to approximately 35% Cu may be attained with little change in recovery.

It is considered feasible that approximately 35% of the total copper could be obtained at a concentrate grade of 35%, and possibly up to 40% Cu, by flotation operation with no re-cycling of low-grade middlings. Middlings, consisting of approximately 25% by weight of flotation feed and containing 45% of the total copper at a grade of approximately 2.5% Cu could be directed to a leaching circuit, to give a gross recovery by flotation of 80%.

## TABLE 5B: INVESTIGATION OF GANGUE DEPRESSANTS AND FLOTATION TIME

| Sample No. and Test No. | Product | Weight % | Assay Cu % | Distribution Cu % | Conditions & Reagents | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | Grind Time min | $Na_2S$ | | Collector lb/ton | Flotation Time min | pH |
| | | | | | | lb/ton | Cond. Time min | | | |
| B4 + H11 | <u>Sulphide</u> | | | | | | | A25 | | |
| Test 20 | Rougher conc | 3.0 | 3.45 | 8.6) | 10 | | | 0.5 | 6 | 6.4 |
| | <u>Oxide</u> | | | | (with) | | | | | |
| | Cleaner conc | 0.5 | 36.2 | 15.0) 43.1 | (brine) | 5.0 | 5 | 0.5 | 7½ | 6.5 |
| | Cleaner tail | 4.9 | 4.8 | 19.5) | (Water) | | | | | 7.3 |
| | Rougher tail | 91.6 | 0.75(a) | 56.9 | | | | | | |
| | | 100.0 | 1.21 | 100.0 | | | | | | 7.1 |
| B4 + H11 | Re-cl conc | 0.2 | 41.6 | 6.4) | 10 | 1.0 | ½ | S3461 | 12 | n.d. |
| Test 22 | " " tail | 2.0 | 13.6 | 20.9) 59.6 | | 1.0 | ½ | | | |
| | Cl. tail | 7.1 | 5.9 | 32.3) | | 1.0 | ½ | 0.2 to | | |
| | Rougher tail | 90.7 | 0.58(a) | 40.4 | | | | mill | | |
| | | 100.0 | 1.30 | 100.0 | | | | | | |
| B4 + H11 | <u>Sulphide</u> | | | | | | | A25 | | |
| Test 24 | Rougher conc | 6.9 | 2.5 | 13.9) | 15 | 1.0 | ½ | 0.2 | 6 | n.d. |
| | <u>Oxide</u> | | | | | 1.0 | ½ | 0.1 | 12 | |
| | Cleaner conc | 0.6 | 45.2 | 21.7) 64.0 | | 1.0 | ½ | 0.1 | | |
| | Cleaner tail | 5.7 | 6.2 | 28.4) | | | | 0.1 | | |
| | Rougher tail | 86.8 | 0.52 | 36.0 | | | | | | |
| | Calc. feed | 100.0 | 1.25 | 100.0 | | | | | | |
| | | | | | | | | A25 | | |
| B4 + H11 | Cleaner conc | 1.6 | 31.2 | 37.5) 79.2 | 10 | 0.5 | ½ | 0.1 | 4 | 8.1 |
| Test 25 | Cleaner tail | 21.9 | 2.54 | 41.7) | | 0.5 | ½ | 0.1 | 25 | |
| | Rougher tail | 76.5 | 0.36(a) | 20.8 | | 0.5 | ½ | 0.1 | | 8.6 |
| | | 100.0 | 1.33 | 100.0 | | 0.5 | ½ | 0.1 | | |
| | | | | | | 0.5 | ½ | 0.1 | | |
| | | | | | | | | 0.1 | | |

TABLE 5B (Contd.)

| Sample No. and Test No. | Product | Weight % | Assay Cu % | Distribution Cu % | Conditions and Reagents | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | Grind Time min. | $Na_2S$ lb/ton | $Na_2S$ Cond Time min. | Collector lb/ton | Flotation Time Min. | pH |
| | | | | | | | | A25 | | |
| B4 + H11 | Cleaner conc. | 1.0 | 32.0 | 26.7) 78.7 | 10 | 0.5 | ½ | 0.1 | | 9.5 |
| Test 26 | Cleaner tail. | 23.6 | 2.65 | 52.0) | + 2.0 lb/t | 0.5 | ½ | 0.1 | 25 | |
| | Rougher tail. | 75.4 | 0.34 | 21.3 | soda ash | 0.5 | ½ | 0.1 | | |
| | | 100.0 | 1.20(a) | 100.0 | | 0.5 | ½ | 0.1 | | |
| | | | | | | | | 0.1 | | 8.9 |
| | | | | | | | | A25 | | |
| B4 + H11 | Cleaner conc. | 0.9 | 36.8 | 26.5) 82.5 | 10 | 0.5 | ½ | 0.1 | | 9.5 |
| Test 27 | Cleaner tail. | 23.5 | 3.3 | 56.0) | + 0.5 lb/t | 0.5 | ½ | 0.1 | 25 | |
| | Rougher tail. | 75.6 | 0.29 | 17.5 | lime | 0.5 | ½ | 0.1 | | |
| | | 100.0 | 1.25(a) | 100.0 | | 0.5 | ½ | 0.1 | | 9.0 |
| | | | | | | $P_2S_5$ | | A25 | | |
| B4 + H11 | Rougher conc. | 21.7 | 3.6 | 68.4 | 10 | 0.5 | ½ | 0.1 | | 9.5 |
| Test 28 | Rougher tail. | 78.3 | 0.46 | 31.6 | + 2.0 lb/t | 0.5 | ½ | 0.1 | 25 | |
| | | 100.0 | 1.14(a) | 100.0 | soda ash | 1.0 | 5 | 0.1 | | |
| | | | | | | 1.0 | 5 | 0.1 | | |
| | | | | | | | | 0.1 | | |
| | | | | | | $Na_2S$ | | SBX | | |
| B4 + H11 | Cleaner conc. | 1.2 | 35.2 | 34.4) 80.2 | 10 | 0.5 | ½ | | | |
| | Cleaner tail. | 23.0 | 2.45 | 45.8) | | 0.5 | ½ | 0.1 | 15 | 9.2 |
| | Rougher tail. | 75.8 | 0.32 | 19.8 | | | | 0.1 | 0.5 lb/ton | |
| | | 100.0 | 1.23(a) | 100.0 | | | | | lime at start | |
| | | | | | | | | A404 | | |
| B4 + H11 | Cleaner conc. | 1.2 | 32.4 | 31.5) 78.2 | 10 | 0.5 | ½ | 0.1 | | 9.5 |
| Test 30 | Cleaner tail. | 24.5 | 2.35 | 46.7) | + 1.0 lb/t | 0.5 | ½ | 0.1 | 25 | |
| | Rougher tail. | 74.5 | 0.36 | 21.8 | lime | 0.5 | ½ | 0.1 | | |
| | | 100.0 | 1.23(a) | 100.0 | | 0.5 | ½ | 0.1 | | |
| | | | | | | | | 0.1 | | 8.6 |

(a) calculated grades

TABLE 5B (Contd.)

| Sample No. and Test No. | Product | Weight % | Assay Cu % | Distribution Cu % | Grind Time min. | Na$_2$S lb/ton | Na$_2$S Cond. Time min. | Collector lb/ton | Flotation Time Min. | pH |
|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | | | A25 | | |
| B4 + H11 | Sulphide | | | | | | | | | |
| Test 33 | Rougher conc. | 20.0 | 2.10 | 33.2) | 15 | | | 0.2 | 6 | |
| | Oxide | | | )83.7 | | | | | | |
| | Cleaner conc. | 0.7 | 43.2 | 23.8) | | 1.0 | ½ | 0.1 | 20 | n.d. |
| | Cleaner tail. | 14.7 | 2.30 | 26.7 ) | | 1.0 | ½ | 0.1 | Sod.silicate | |
| | Rougher tail. | 64.6 | 0.32 | 16.3 | | 1.0 | ½ | 0.1 | 1.0 lb/ton | |
| | | 100.0 | 1.27 (a) | 100.0 | | 1.0 | ½ | 0.1 | added at start | |
| B4 + H11 | + 20 microns | | | | | | | A25 | | |
| Test 34 | Sulphide | | | | | | | 0.2 | 5 | |
| | Rougher conc. | 16.4 | 1.9 | 27.0 | | | | 0.2 | | |
| | Oxide | | | | 10 | 0.5 | ½ | 0.2 | 20 | n.d. |
| | Cleaner conc. | 0.6 | 43.2 | 22.4 | | 0.5 | ½ | 0.1 | Sod. silicate | |
| | Cleaner tail. | 10.4 | 2.3 | 20.7 | | 0.5 | ½ | 0.1 | 0.5 lb/ton | |
| | Rougher tail. | 67.5 | 0.29 | 17.0 | | 0.5 | ½ | 0.1 | added at start | |
| | | | | | | | | 0.1 | | |
| | - 20 microns | | | | | | | A25 | | |
| | Sulphide | | | | | | | 0.2 | | |
| | Rougher conc. | 2.5 | 3.15 | 6.8 | | | | 0.2 | | |
| | Oxide | | | | | | | 0.2 | | |
| | Rougher conc. | 1.8 | 3.00 | 4.7 | | 0.5 | ½ | 0.1 | | |
| | Rougher tail. | 0.8 | 2.15 | 1.4 | | 0.5 | ½ | 0.1 | | |
| | | 100.0 | 1.16 (a) | 100.0 | | | | | | |
| H11 | Sulphide | | | | | | | | | |
| Test 36 | Cleaner conc. | 2.3 | 2.10 | 6.6) | 10 | | | SBX 0.2 | 6 | 10.4 |
| | Cleaner tail. | 9.6 | 1.18 | 15.6) 79.3 | + 5 lb/ton | | | A25 0.2 | | |
| | Oxide | | | ) | Metso | 0.5 | ½ | | | |
| | Cleaner conc. | 1.0 | 36.8 | 50.9) | | 0.5 | ½ | SBX 0.4 | 12 | |
| | Cleaner tail. | 3.6 | 1.24 | 6.2 ) | | 0.5 | ½ | A25 0.4 | | 9.4 |
| | Rougher tail. | 83.5 | 0.18 | 20.7 | | 0.5 | ½ | | | |
| | | 100.0 | 0.72 (a) | 100.0 | | | | | | |

Note: Test 34 - Deslimed at 20 microns and fractions floated separately

(a) calculated grades

TABLE 5B (Contd.)

| Sample No. and Test No. | Product | Weight % | Assay Cu % | Distribution Cu % | Grind Time min. | Na2S lb/ton | Na2S Cond. Time min. | Collector lb/ton | Flotation Time min. | pH |
|---|---|---|---|---|---|---|---|---|---|---|
| H11 | Sulphide | | | | | | | | | |
| Test 37 | Rougher conc. | 9.3 | 1.46 | 18.3) | 10 | | | SBX 0.2 | 6 | 11.5 |
| | Oxide | | | )82.6 | + 12 lb/ton | | | A25 0.2 | | |
| | Cleaner conc. | 0.8 | 29.6 | 32.5) | Metso | 0.5 | ½ | | | |
| | Cleaner tail. | 19.3 | 1.2 | 31.8) | | 0.5 | ½ | SBX 0.4 | 12 | |
| | Rougher tail. | 70.6 | 0.18 | 17.4 | | 0.5 | ½ | A25 0.4 | | |
| | | 100.0 | 0.73 | 100.0 | | 0.5 | ½ | | | 9.7 |
| B4 | Sulphide | | | | | | | | | |
| Test 38 | Rougher conc. | 11.0 | 2.82 | 20.5) | 10 | | | SBX 0.2 | 6 | 10.3 |
| | Oxide | | | )69.3 | + 5 lb/ton | | | A25 0.2 | | |
| | Cleaner conc. | 1.2 | 40.0 | 31.7) | Metso | 0.5 | ½ | | | |
| | Cleaner tail. | 12.8 | 2.02 | 17.1) | | 0.5 | ½ | SBX 0.4 | | |
| | Rougher tail. | 75.0 | 0.62 | 30.7 | | 0.5 | ½ | A25 0.4 | 12 | |
| | | 100.0 | 1.51 (a) | 100.0 | | 0.5 | ½ | | | 9.4 |
| B4 | Sulphide | | | | | | | | | |
| Test 39 | Rougher conc. | 11.4 | 2.92 | 19.6) | 10 | | | SBX 0.2 | 6 | 11.3 |
| | Oxide | | | )71.7 | + 12 lb/ton | | | A25 0.2 | | |
| | Cleaner conc. | 1.6 | 38.8 | 36.6) | Metso | 0.5 | ½ | | | |
| | Cleaner tail. | 11.5 | 2.28 | 15.5) | | 0.5 | ½ | SBX 0.4 | | |
| | Rougher tail. | 75.5 | 0.64 | 28.3 | | 0.5 | ½ | A25 | 12 | 9.7 |
| | | 100.0 | 1.70 (a) | 100.0 | | 0.5 | ½ | | | |
| B4 + H11 | Sulphide | | | | | | | | | |
| Test 40 | Rougher conc. | 12.3 | 2.60 | 24.1) | 10 | | | SBX 0.2 | 10 | 10.5 |
| | Oxide | | | )74.2 | + 5 lb/ton | | | A25 0.2 | | |
| | Cleaner conc. | 1.0 | 43.2 | 32.5) | Metso | 0.5 | ½ | | | |
| | Cleaner tail. | 8.7 | 2.68 | 17.6) | | 0.5 | ½ | SBX 0.4 | 20 | |
| | Rougher tail. | 78.0 | 0.44 | 25.8 | | 0.5 | ½ | A25 0.4 | | |
| | | 100.0 | 1.33 (a) | 100.0 | | 0.5 | ½ | | | 8.9 |
| B4 + H11 | Sulphide | | | | | | | | | |
| Test 41 | Rougher conc. | 1.5 | 2.5 | 3.5) | 10 | | | SBX 0.2 | 10 | 11.4 |
| | Oxide | | | )57.0 | + 12 lb/ton | | | A25 0.2 | | |
| | Cleaner conc. | 0.9 | 42.0 | 35.3) | Metso | 0.5 | ½ | | | |
| | Cleaner tail. | 9.2 | 2.13 | 18.2) | | 0.5 | ½ | SBX 0.4 | | |
| | Rougher tail. | 88.4 | [illegible] | 43.0 | | 0.5 | ½ | A25 0.4 | 20 | |
| | | 100.0 | [illegible] | 100.0 | | 0.5 | ½ | | | 9.7 |

TABLE 5B (Contd.)

| Sample No. and Test No. | Product | Weight % | Assay Cu % | Distribution Cu % | Conditions and Reagents | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | Grind Time min. | $Na_2S$ lb/ton | $Na_2S$ Cond. Time min. | Collector lb/ton | Flotation Time min. | pH |
| B4 + H11 | <u>Sulphide</u> | | | | | | | | | |
| Test 42 | Rougher conc. | 0.3 | 2.75 | 0.8) | 10 | | | SBX 0.2 | 10 | 0.5 |
| | <u>Oxide</u> | | | )64.4 | + 5 lb/ton | | | A25 0.2 | | |
| | Cleaner conc. | 1.0 | 42.4 | 39.8) | Sodium | 0.5 | ½ | | | |
| | Cleaner tail. | 12.7 | 2.0 | 23.8) | Silicate | 0.5 | ½ | SBX 0.4 | 20 | |
| | Rougher tail. | 86.0 | 0.44 | 35.6 | | 0.5 | ½ | A25 0.4 | | |
| | | 100.0 | 1.06 (a) | 100.0 | | 0.5 | ½ | | | 9.1 |
| B4 + H11 | <u>Sulphide</u> | | | | | | | | | |
| Test 43 | Rougher conc. | 1.6 | 2.2 | 3.2 | 10 | | | SBX 0.2 | 10 | 9.9 |
| | <u>Oxide</u> | | | | + 12 lb/ton | | | A25 0.2 | | |
| | Cleaner conc. | 1.0 | 40.0 | 36.5 | Sodium | 0.5 | ½ | | | |
| | Cleaner tail. | 9.8 | 2.45 | 21.9 | Silicate | 0.5 | ½ | SBX 0.4 | 20 | |
| | Rougher tail. | 87.6 | 0.48 | 38.4 | | 0.5 | ½ | A25 0.4 | | |
| | | 100.0 | 1.10 (a) | 100.0 | | 0.5 | ½ | | | 9.2 |
| B4 + H11 | <u>Sulphide</u> | | | | | | | | | |
| Test 44 | Rougher conc. | 6.9 | 2.25 | 12.5) | 10 | | | KAX 0.2 | 10 | 9.7 |
| | <u>Oxide</u> | | | )62.8 | + 3 lb/ton | | | A208 0.2 | | |
| | Cleaner conc. | 1.0 | 42.5 | 34.3) | Metso | | | | | |
| | Cleaner tail. | 3.2 | 6.2 | 16.0) | | 1.0 | ½ | | | |
| | Rougher tail. | 88.9 | 0.52 | 37.2 | | 1.0 | ½ | KAX 0.3 | | |
| | | 100.0 | 1.24 (a) | 100.0 | | 1.0 | ½ | A208 0.3 | 15 | 9.7 |
| B4 | <u>Sulphide</u> | | | | | | | | | |
| Test 45 | Rougher conc. | 8.6 | 3.55 | 18.2) | 10 | | | KAX 0.2 | 10 | 9.6 |
| | <u>Oxide</u> | | | )61.2 | + 3 lb/ton | | | A208 0.2 | | |
| | Cleaner conc. | 1.2 | 43.2 | 30.8) | Metso | 1.0 | ½ | KAX 0.3 | | |
| | Cleaner tail. | 3.4 | 6.0 | 12.2 | | 1.0 | ½ | A208 0.3 | 15 | 9.5 |
| | Rougher tail. | 87.0 | 0.75 | 38.8 | | 1.0 | ½ | | | |
| | | 100.0 | 1.68 (a) | 100.0 | | | | | | |

(a) calculated grades

TABLE 5B (Contd.)

| Sample No. and Test No. | Product | Weight % | Assay Cu % | Distribution Cu % | Conditions and Reagents: Grind Time min. | $Na_2S$ lb/ton | $Na_2S$ Cond. Time min. | Collector lb/ton | Flotation Time min. | pH |
|---|---|---|---|---|---|---|---|---|---|---|
| H11 | Sulphide | | | | | | | | | |
| Test 46 | Rougher conc. | 7.6 | 1.75 | 12.2) | 10 | | | KAX 0.2 | 10 | 9.8 |
| | Oxide | | | )54.2 | + 3 lb/ton | | | A208 0.2 | | |
| | Cleaner conc. | 0.2 | 36.8 | 6.8) | Metso | 1.0 | ½ | | | |
| | Cleaner tail. | 6.1 | 6.3 | 35.2) | | 1.0 | ½ | KAX 0.3 | | |
| | Rougher tail. | 86.1 | 0.58 | 45.8 | | 1.0 | ½ | A208 0.3 | 15 | |
| | | 100.0 | 1.09 (a) | 100.0 | | | | | | 9.5 |
| B4 | Sulphide | | | | | | | | | |
| Test 49 | Rougher conc. | 6.3 | 3.7 | 13.9) | 10 | | | SBX 0.2 | 5 | 9.6 |
| | Oxide | | | )67.9 | + 3 lb/ton | | | A65 0.1 | | |
| | Cleaner conc. | 1.3 | 37.6 | 29.2) | Metso | 0.5 | ½ | | | |
| | Cleaner tail. | 8.3 | 5.0 | 24.8) | | 0.5 | ½ | SBX 0.4 | 20 | |
| | Rougher tail. | 84.1 | 0.64 | 32.1 | | 0.5 | ½ | A65 0.1 | | |
| | | 100.0 | 1.67 (a) | 100.0 | | | | | | 9.5 |
| B4 | Sulphide | | | | | | | | | |
| Test 50 | Rougher conc. | 9.0 | 3.30 | 17.1) | 10 | | | SBX 0.1 | 5 | n.d. |
| | Oxide | | | )71.6 | | | | A65 0.1 | | |
| | Cleaner conc. | 1.8 | 37.6 | 39.0) | | 0.5 | ½ | | | |
| | Cleaner tail. | 7.0 | 3.85 | 15.5) | | 0.5 | ½ | SBX 0.4 | 20 | |
| | Rougher tail. | 82.2 | 0.60 | 28.4 | | 0.5 | ½ | A65 0.1 | 0.4 lb/ton at start and | |
| | | 100.0 | 1.74 (a) | 100.0 | | 0.5 | ½ | | 0.1 in cleaning | |
| B4 + H11 | Sulphide | | | | | | | | | |
| Test 51 | Rougher conc. | 7.6 | 2.80 | 17.0) | 10 | | | SBX 0.2 | 10 | n.d. |
| | Oxide | | | )71.4 | | | | A65 0.1 | | |
| | Cleaner conc. | 1.3 | 36.0 | 37.3) | | 0.5 | ½ | SBX 0.4 | | |
| | Cleaner tail. | 5.8 | 3.7 | 17.1) | | 0.5 | ½ | A65 0.1 | 20 | |
| | Rougher tail. | 85.3 | 0.42 | 28.6 | | 0.5 | ½ | | 0.4 lb/ton | |
| | | 100.0 | 1.25 (a) | 100.0 | | 0.5 | ½ | | AC633 at start and 0.1 in cleaning | 8.7 |

(a) calculated grades

## TABLE 7: CONDITIONS FOR CYCLIC TEST 1

| Grind Time Min | $Na_2S$ lb/ton | $Na_2S$ Cond Time Min | Collector lb/ton | Flotation Time Min | pH |
|---|---|---|---|---|---|
| 10 | 0.5 | ½ | Sulphide S.B.X. 0.2 | 10 | 9.5 |
| + 1 lb/ton | 0.5 | ½ | A25 0.2 | | |
| Metso | | | A65 0.4 | | |
| | 0.5 | ½ | Oxide S.B.X. 0.4 | 20 | |
| | 0.5 | ½ | A25 0.4 | | 9.2 |
| | | | A65 0.2 | | |

## TABLE 8: RESULTS OF CYCLIC TEST 1

| Product | Wt % | | Assay Cu % | Distribution Cu % | |
|---|---|---|---|---|---|
| Rougher tailing | | | | | |
| Stage 1 | 21.5 | | 0.66 | 8.6 | |
| 2 | 23.0 | | 0.72 | 10.1 | |
| 3 | 23.7 | | 0.70 | 10.1 | |
| 4 | 24.1 | | 0.70 | 10.3 | |
| Cleaner Tailing | | 92.3 | | | 39.1 |
| Stage 4 | | 4.1 | 2.51 | | 6.3 |
| Bulk Cleaner Concentrate | | 3.6 | 25.0 | | 54.6 |
| | | 100.0 | 1.65[(a)] | | 100.0 |

## TABLE 9: CONDITIONS FOR CYCLIC TEST 2

| Grind Time Min | $Na_2S$ lb/ton | $Na_2S$ Cond Time Min | Collector | Flotation Time Min | pH |
|---|---|---|---|---|---|
| 10 | 0.5 | ½ | Sulphide S.B.X.0.2 | 10 | 8.2 |
| | 0.5 | ½ | A25 0.2 | | |
| | 0.5 | ½ | Oxide S.B.X.0.4 | 20 | |
| | 0.5 | ½ | A25 0.4 | | 8.5 |

## TABLE 10: RESULTS OF CYCLIC TEST 2

| Product | Wt % | | | Assay Cu % | | Distribution Cu % | | |
|---|---|---|---|---|---|---|---|---|
| Rougher tailing | | | | | | | | |
| Stage 1 | 20.4 | | | 0.34 | | 6.0 | | |
| 2 | 23.6 | | | 0.44 | | 9.0 | | |
| 3 | 23.9 | | | 0.47 | | 9.7 | | |
| 4 | 24.8 | | | 0.55 | | 11.7 | | |
| Cleaner Tailing | | 92.7 | | | | | 36.4 | |
| Stage 4 | | 4.5 | | 3.68 | | | 14.3 | |
| Bulk re-cleaner tailing(b) | | 1.5 | 2.8 | 7.00 | 24.0 | | 9.0 | 49.3 |
| Bulk re-cleaner concentrate | | 1.3 | | 36.8 | | | 40.3 | |
| | | 100.0 | | 1.16(a) | | | 100.0 | |

(a) Calculated Grade

(b) Bulk cleaner concentrate re-cleaned.

TABLE 11: SIZE ANALYSES OF FLOTATION TAILING, TEST 13

| Nominal Particle Size Microns | Wt % | Assay Cu % | Distribution Cu % |
|---|---|---|---|
| + 250 | 2.4 | 0.49 | 1.1 |
| -250 + 180 | 13.4 | 0.64 | 8.1 |
| -180 + 125 | 29.5 | 0.87 | 23.9 |
| -125 + 90 | 23.5 | 1.02 | 22.4 |
| -90 + 63 | 13.4 | 1.26 | 15.7 |
| -63 + 45 | 4.0 | 1.76 | 6.6 |
| -45 + 33.5 | 4.5 | 1.22 | 5.1 |
| -33.5 + 23.5 | 3.3 | 1.48 | 4.6 |
| -23.5 + 12.0 | 2.7 | 1.66 | 4.2 |
| - 12.0 | 3.3 | 2.70 | 8.3 |
| | 100.0 | 1.07(a) | 100.0 |

(a)

## APPENDIX A

## MINERALOGICAL EXAMINATION OF SIZE FRACTIONS OF TAILING FROM FLOTATION TEST 13

Examination of the fractions from 250 through to 12 microns showed that free malachite occurs in all except the finest fraction (minus 12 microns). There are also many particles which are pale green and consist of copper-stained gangue.

There seems no reason why the free malachite should not be recoverable. However, the copper-stained gangue is probably only amenable to leaching.

The finest fraction, containing 2.7% Cu, is a problem because no copper minerals could be identified with certainty. It is thought that very fine-grained cuprite (perhaps the chalcotrichite variety) is present. Some copper may also be present in the form of copper ions adsorbed onto clays. This would be difficult to establish, however.

Cuprite may be present in the coarser fractions also, but this is difficult to establish with certainty; it is very fine-grained (probably less than 0.01 mm across) and locked in quartz.

Investigation and Report by H.W. Fander, Officer in Charge, Mineralogy Section.

# APPENDIX B

## ANALYSIS OF WATER FROM MINE AREA

| | Parts per million |
|---|---|
| Chloride, Cl | 190100 |
| Sulphate, $SO_4$ | 16730 |
| Bicarbonate, $HCO_3$ | 60 |
| Nitrate, $NO_3$ | 35 |
| Sodium, Na | 92150 |
| Potassium, K | - |
| Calcium, Ca | 350 |
| Magnesium, Mg | 20510 |
| Total | 319935 |

## ASSUMED COMPOSITION OF SALTS

| | |
|---|---|
| Calcium bicarbonate | 80 |
| Calcium sulphate | 1120 |
| Calcium chloride | - |
| Magnesium bicarbonate | - |
| Magnesium sulphate | 19975 |
| Magnesium chloride | 64515 |
| Sodium bicarbonate | - |
| Sodium sulphate | - |
| Sodium chloride | 234200 |
| Sodium nitrate | 45 |

## HARDNESS (as Calcium Carbonate)

| | |
|---|---|
| Total | 85273 |
| Temporary | 50 |
| Permanent | 85223 |
| Due to calcium | 874 |
| Due to magnesium | 84399 |

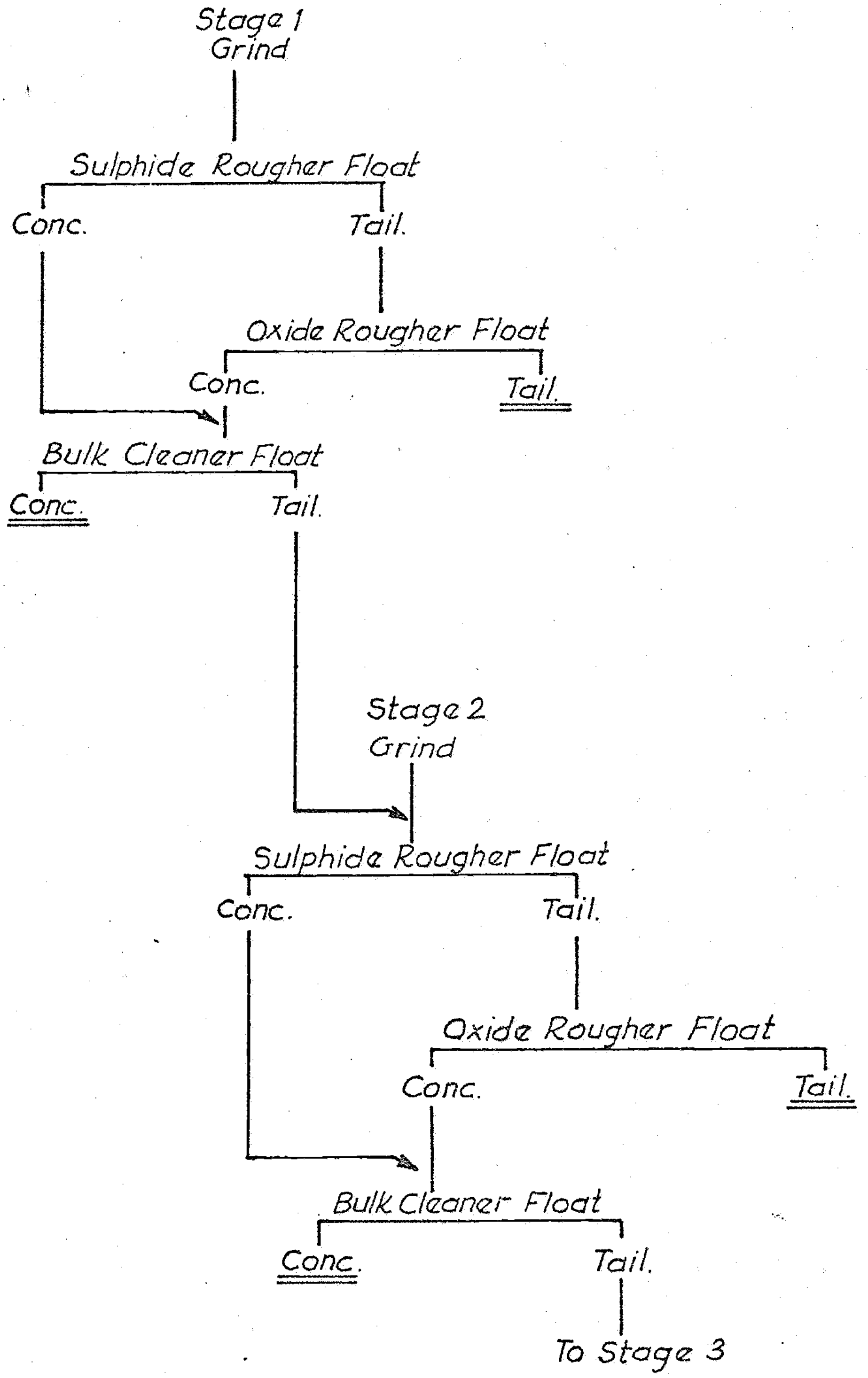

FIGURE - 2 : FLOWSHEET FOR CYCLIC BATCH FLOTATION TESTING.

000101

EXHIBIT NO. 8t

Stearns-Roger
CORPORATION
• ESTABLISHED 1885

• 660 BANNOCK STREET • P. O. BOX 5888, DENVER, COLORADO 80217
• PHONE 303-222-8484 • TWX 910-931-0453 • TELEX 045-540

April 7, 1967

Mr. J. B. Evans
Austminex Property Ltd.
Suite 27
67 Queens Road
Melbourne
Victoria, Australia SC3

Dear Mr. Evans:

Eldon DeMent of General Mills has asked us to prepare an Order of Magnitude Estimate of your proposed copper recovery plant. We understand the scope of this estimate will include a new solvent extraction plant with an electrolytic tank house to produce 5 tons copper per day.

The SX circuit is to be a four-stage extraction system with a mixer volume of 114 cubic feet, and a settler area of 280 square feet per stage. The strip section is to be three stages, the first two of which are to have a mixer volume of 192 cubic feet and a settler volume of 347 square feet per stage. The third stage mixer is to have 309 cubic feet while the settler is to have 578 square feet.

The tank house is to be designed to handle an electrolyte containing 50 grams per liter of copper at a current density of 30 amps per square foot. We believe such a plant should cost between $500,000 and $700,000 American Dollars. This figure assumes the following:

1. Electrical power is available at the site.
2. The SX circuit will be completely housed for weather protection.
3. Foundation conditions are normal, i.e., piles or special foundations will not be required.
4. The necessary surge tanks and settlers for organic recovery are included but would probably not be enclosed in the main structure.
5. Labor rates and labor productivity would be roughly comparable to U.S.A. practice.
6. It would not be required to establish a special camp or develop special water facilities or other utilities to make this a complete facility.

Mr. J. B. Evans
Victoria, Australia SC3
April 7, 1967

I am real pleased to hear of you again and hope that this information is what you required. I was down in Questa two months ago and they are having continuing problems.

May I hear from you if we can be of any service on this interesting project discussed above.

Very truly yours,

STEARNS-ROGER CORPORATION

G. T. Swanby

G. T. Swanby

GTS/mg
cc: Eldon DeMent

EXHIBIT NO. 8a

COMMONWEALTH OF AUSTRALIA

COMMONWEALTH SCIENTIFIC AND INDUSTRIAL RESEARCH ORGANIZATION

MINERAGRAPHIC INVESTIGATIONS

TELEPHONE: 34 1392

C/o GEOLOGY SCHOOL,
UNIVERSITY OF MELBOURNE,
PARKVILLE, N.2,
VIC.

# SUMMARY REPORT ON THE MINERALOGY OF THE MT. GUNSON COPPER MINERAL DEPOSITS, S.A.

The mineralogical study summarized has been made on specimens from various copper mineral deposits of the Mt. Gunson area, South Australia, in conjunction with recent exploration of this district by Austminex Ltd. The specimens examined may not comprehensively represent the variation within these deposits, but were selected to represent the major types of copper mineralization recognizable by visual inspection.

Two distinct types of sulphide deposits have been recognized in the study. One type appears to have been deposited syngenetic with quartzite, and may thus have formed at various positions within it elsewhere. The other type appears to have formed by deposition, below the water table, of copper sulphide from cupriferous ground water. This would occur where ground water accumulated in stagnant, anerobic environments, probably relatively recently, and thus be restricted to areas of appropriate drainage, for which the present topography would provide a guide.

The oxidized copper mineralization varies from that in which the oxidized copper minerals have formed from in situ oxidation of copper sulphides, together with dispersal of copper into adjoining sediments, to that in which there is no evidence of preceeding in situ copper sulphides, and which formed from cupriferous ground water which had migrated away from the source of the copper.

## 1. Syngenetic ? Sulphide Ore

The probable existence of syngenetic copper sulphides in the quartzite is revealed by pulverized samples from the bottom of percussion drill-holes C6, C9 and C12, which penetrated the base of the oxidized zone. Particles of sulphides in these were concentrated by heavy liquid separation.

The very small concentrate so obtained contains, in approximate order of abundance, djurleite ($Cu_{1.96}S$), covellite ($CuS$), digenite ($Cu_{1.8}S$), bornite (ca. $Cu_5FeS_4$), chalcopyrite ($CuFeS_2$) and sphalerite ($ZnS$).

Most features of the texture of the sulphides cannot be observed in a pulverized sample. However the bornite and chalcopyrite appear to be primary sulphides. Some bornite contains extremely fine exsolution lamella of chalcopyrite. This may be correlated with the exsolution of chalcopyrite obtained on gentle heating of sulphur-rich bornite, which has deposited under conditions of high sulphur activity, e.g. from appropriate aqueous solutions, at a temperature below that which produces the exsolution. Such bornite could have formed preceeding burial of the quartzite by overlying sediments, and exsolution developed by rise in temperature under burial, from the geothermal gradient. It is most likely syngenetic with the quartzite; the conditions producing its deposition could have occurred at various locations during deposition of the quartzite, so other similar ~~such~~ bodies may occur elsewhere in it.

Of the supergene sulphides djurleite, covellite, and digenite which are present, the most, abundant, djurleite, was only differentiated in 1962 as a mineral distinct from chalcocite ($Cu_{2.0}S$). Microscopically it is virtually identical with chalcocite, from which it is distinguished by its x-ray diffraction pattern. A few of the grains with the optical properties of chalcocite and djurleite, from which an x-ray powder pattern was obtained, were all djurleite; this does not however establish that there is no chalcocite present.

## 2. Isolated supergene sulphide ore

In the Lagoon deposit djurleite forms a solid matrix to a poorly sorted quartzite and sandy breccia. As noted above, the djurleite has been distinguished from chalcocite by x-ray diffraction patterns.

Very minor chalcopyrite occurs as curved concentric lamella in voids in quartzite near the surface of this deposit.

There is no texture or mineral indicative of primary sulphides, such as should be preserved in the unoxidized specimens from below the water table if primary sulphides were originally present.

Thus the ground water appears to have carried copper away from the source of this metal, to produce a supergene type sulphide deposit separate from any primary deposit. If so, other similar bodies may occur "down-stream" from a source of copper, in a restricted environment where stagnant ground water could be subject to non-oxidizing conditions.

## 3. Oxidized copper mineral deposits

In the specimens of oxidized ore examined copper occurs mainly as poorly crystalline chrysocolla (ca. $CuSiO_3.nH_2O$) from which the copper is readily leached by dilute acid, and malachite. An analysis just received of one specimen shows 1.6% Cl, suggesting some atacamite ($Cu_2(OH)_3Cl$) is present.

Oxidized ore of the old open cuts contains black residual nodules, and veinlets, with cores of sulphides, which persist to the upper part of the oxidized ore. Djurleite is the main sulphide, with some covellite, and minor sphalerite. The sulphides commonly alter in situ to malachite, with part of the copper dispersing into adjacent sediments and precipitating as chrysocolla. Cuprite ($Cu_2O$) and tenorite ($CuO$) occur locally.

The dispersed copper minerals occur along joints, as a matrix to the quartzite grains, and replacing cementing quartz and the rounded quartz grains of the quartzite.

In the H 11 slot, some 2,000 ft. WSW of the main open cut, oxidized copper minerals lack evidence of prexisting copper sulphides. The host massive quartzite has appreciably porosity, but apparently low permeability, and oxidized copper minerals are along joint planes, from which introduced copper has only very slightly penetrated into interstices of the quartzite.

In the nearby D.J. pit, oxidized copper minerals extensively replace the matrix of a poorly sorted sandstone, which overlies the quartzite at H 11. This rock is similar to that at the "Lagoon" deposit. The similarity in ore texture of these two deposits, with chrysocolla at the D.J. pit in equivalent texture to djurleite at the Lagoon deposit, reflects similarities of the host rocks, and in particular their relatively high permeability.

J McAndrew.

J. McAndrew,
Principal Research Scientist.

19 April 1967

V1

24 October 1966

Mr. J. Woodcock,
C.S.I.R.O. Ore Dressing Division,
Mining Department,
University of Melbourne,
Melbourne, Victoria

Dear Jim,

I am forwarding to you today two cartons each of which contain two parcels of five plastic bags containing samples of the material cut in the trenches here at Mt. Gunson. In all there are four 25 lb. samples as follows:

1.) Bags 1-5 from cut 24 from 5'-10'
2.) Bags 1-5 from cut 24 from 10'-15'
3.) Bags 1-5 from cut H11 from 5'-10'
4.) Bags 1-5 from cut H11 from 10'-15'

They were obtained from a ¼ split of the material in the cut after it had been all broken to 1"; the half split [illegible] half split; the sample was then further split through a Jones' splitter to give a 25 lb. sample.

I hope that this is satisfactory and that you will be able to get some good information from them. Samples for the [illegible] will follow in about a week.

Best regards to yourself and your wife.

Tony Robinson

✓|

**CSIRO**

ORE DRESSING INVESTIGATIONS, MELBOURNE

MELBOURNE UNIVERSITY ORE DRESSING LABORATORY, MINING DEPT., UNIVERSITY, PARKVILLE, N.2, VIC. TELEPHONE 341338, 342274

25th October, 1966

Mr. T. Robinson,
Austminex Pty. Ltd.,
c/- Woocalla P.O.,
SOUTH AUSTRALIA.

Dear Tony,

Thank you for your letter of 24th October and for the four samples from Mt. Gunson which were delivered today. We will start work on them within a few days.

Kind regards,

Yours sincerely,

Jim Woodcock

J.T. Woodcock
SENIOR RESEARCH SCIENTIST

COMMONWEALTH SCIENTIFIC AND INDUSTRIAL RESEARCH ORGANIZATION, AUSTRALIA

EXHIBIT NO. 8c

CSIRO

ORE DRESSING INVESTIGATIONS, MELBOURNE

MELBOURNE UNIVERSITY ORE DRESSING LABORATORY, MINING DEPT., UNIVERSITY, PARKVILLE, N.2, VIC. TELEPHONE 341338, 342274

KSB/MC

8th November, 1966

Mr D. Irving
Chapman Wood Griswold & Evans Pty. Ltd.
542 St. Kilda Road,
MELBOURNE.

Ref: 6615-F

Dear Doug,

### MOUNT GUNSON. SAMPLE OF SULPHIDES ~~OF~~ in DOLOMITE

I am collecting together in this letter the information I have on the piece of dolomite picked up in Pernatty Lagoon during our visit recently. I will not include observations by John McAndrew on the minerals present.

## HEAD SAMPLE

The head sample gave the following assays

| | |
|---|---|
| Total copper | 1.92% |
| Oxidized copper | 0.08% |
| CaO | 29.4% |
| Insol. | 4.4% |
| Gold | tr |
| Silver | 30.9 dwt/ton |

## FLOTATION

Grinding and flotation conditions were not closely recorded. They were typical for a copper sulphide ore.

The result obtained was as follows

| | | Assay | | % Recovery | |
|---|---|---|---|---|---|
| Product | % Wt | % Cu | Ag dwt/ton | Cu | Ag |
| Concentrate | 3.63 | 46.8 | 770.7 | 90.3 | 90.5* |
| Tailing | 96.4 | 0.19 | | 9.7 | |
| Calc Head | 100.0 | 1.88 | | 100.0 | |

* By calculation

COMMONWEALTH SCIENTIFIC AND INDUSTRIAL RESEARCH ORGANIZATION, AUSTRALIA

Both the copper and silver have floated quite readily. The flotation test was of a very preliminary nature and I am sure that both concentrate grade and recovery could be improved.

Yours sincerely

K. S. Blaskett

K. S. Blaskett
PRINCIPAL RESEARCH SCIENTIST

**CSIRO**

ORE DRESSING INVESTIGATIONS, MELBOURNE

MELBOURNE UNIVERSITY ORE DRESSING LABORATORY, MINING DEPT., UNIVERSITY, PARKVILLE, N.2, VIC. TELEPHONE 341338, 342274

✓/

000110

JTW/MC

21st December, 1966

Mr D. Irving
Chapman Wood Griswold & Evans Pty. Ltd.
Suite 20
541 St. Kilda Road
MELBOURNE S. C. 2 Vic.

Dear Mr Irving,

## Mount Gunson samples

Further to our recent conversation I have summarised below the results we have obtained to date on the samples of drill cuttings from below the water table at the Gunson mine.

The same test procedure was used for all samples. The whole of the sample received (140 - 250 g) was dry ground minus 35 mesh and then floated at natural pH in a Denver cell (250 g capacity) for 10 min using about 0.3 lb/ton secondary butyl xanthate and about 0.1 lb/ton Aerofroth 65. Each flotation tailing was filtered, dried, and then leached with dilute sulphuric acid for 2 hr at a liquid/solid ratio of 2/1. Acid was added to keep the pH below about 1.8. Copper assays have been conducted on the pregnant solutions and are to be done on the washed and dried leach residue.

The main results are summarised below.

The sulphides appeared to float readily, but we cannot estimate the overall recovery until we know the assay of the leach residue. The copper extraction by sulphuric acid leaching after flotation was, in general, very low, indicating a low proportion of oxidized copper minerals present. Some malachite was noted in Sample C8-1115 (but not in the other samples) and some of this malachite reported in the flotation concentrate. Some copper was extracted from this sample by leaching the flotation tailing.

Flotation concentrate grades were fairly low, partly because an attempt was made to obtain a high recovery.

| Sample No. | Flot. conc |  |  | Preg. soln. Cu g/1. | *Equiv Cu assay of feed reporting in |  |
|---|---|---|---|---|---|---|
|  | % Wt. | Assay |  |  | flot. conc. | preg. soln. |
|  |  | Cu % | Ag dwt/ton |  |  |  |
| D9-1069 | 4.2 | 19.9 | 197 10 | 0.08 | 0.85 | 0.02 |
| 1070 | 3.0 | 27.9 | 232 12 | 0.10 | 0.84 | 0.02 |
| 1071 | 2.9 | 25.8 | 194 10 | 0.13 | 0.75 | 0.03 |
| C8-1115 | 2.7 | 44.2 | 770 37 | 3.03 | 1.19 | 0.61 |
| 1116 | 5.3 | 52.5 | 359 18 | 0.18 | 2.78 | 0.04 |
| 1117 | 2.2 | 25.3 | 170 8 | 0.08 | 0.56 | 0.02 |
| 1118 | 2.1 | 24.5 | 260 13 | 0.15 | 0.51 | 0.03 |

Yours sincerely

J. Woodcock.

J. T. Woodcock
SENIOR RESEARCH SCIENTIST

CSIRO

ORE DRESSING INVESTIGATIONS, MELBOURNE

MELBOURNE UNIVERSITY ORE DRESSING LABORATORY, MINING DEPT., UNIVERSITY, PARKVILLE, N.2, VIC. TELEPHONE 341338, 342274

EXHIBIT NO. 8f

000112

1st February, 1967.

Mr. J.B. Evans,
Managing Director,
Chapman, Wood, Griswold & Evans Pty. Ltd.,
Suite 20,
541 St. Kilda Road,
MELBOURNE, S.C.2. : Victoria.

Dear Mr. Evans,

## Lagoon samples from Mount Gunson

We have now completed some preliminary tests on the samples taken from Pernatty Lagoon (point C4) and the results are given below.

### Sample preparation

The samples were received wet and were first air dried. This was a relatively slow process, probably because of the presence of some salt in the samples. After drying, one eighth of the samples was riffled out and crushed minus 14 mesh. The minus 14 mesh product was riffled into 500 g. lots for test work. One 500 g. lot was pulverized to provide a sample for assay.

### Head assays

The head assay of the samples was as follows.

| | Sample | Cu % | ox Cu % | S* % | Ag dwt/ton | Au dwt/ton |
|---|---|---|---|---|---|---|
| 6·28 | C4 (10 - 15 ft) | 0.99 | 0.04 | 0.88 | 6.5 | < 0.1 |
| 12·35 | C4 (15 - 20 ft) | 1.97 | 0.11 | 1.04 | 14.2 | < 0.1 |

* Total S ($SO_4$ assays to come)

2/.......

## Flotation tests

One rougher flotation test on each sample has been completed. Grinding was done wet in an iron ball mill and flotation was conducted in a Denver Cell. General conditions are set out below and are followed by the results.

| | C4 (10 - 15 ft) | C4 (15 - 20 ft) |
|---|---|---|
| Grind (min) | 10 | 10 |
| (% - 200 mesh) | 32 | 35 |
| Flotation time (min) | 15 | 15 |
| Reagents (lb/ton) | | |
| Sec. butyl xan. | 0.3 | 0.3 |
| Aerofroth 65 | 0.05 | 0.05 |
| pH at start | 7.0 | 7.3 |
| pH at finish | 7.4 | 7.8 |
| Test No. | 401 | 402 |

| Test No. | Product | % wt | Assay | | | % distrib. | |
|---|---|---|---|---|---|---|---|
| | | | Cu % | ox Cu % | Ag dwt/ton | Cu | Ag. |
| 401 | Flot. conc. | 2.5 | 31.7 | | 178.8 | 77.2 | 68.6 |
| | Flot. tail | 97.5 | 0.24 | 0.04 | 2.1 | 22.8 | 31.4 |
| | Calc. feed | 100.0 | 1.03 | | 6.5 | 100.0 | 100.0 |
| 402 | Flot. conc. | 3.6 | 46.6 | | 279.5 | 82.1 | 70.8 |
| | Flot. tail | 96.4 | 0.38 | 0.09 | 4.3 | 17.9 | 29.2 |
| | Calc. feed | 100.0 | 2.04 | | 14.2 | 100.0 | 100.0 |

## Discussion

The sample from 10 - 15 ft assayed about 1 per cent copper and 6 dwt/ton silver and the sample from 15 - 20 ft assayed about 2 per cent copper and 14 dwt/ton silver. Most of the copper is present as a copper sulphide but there is a little oxide copper present.

By straight forward rougher flotation, using a simple reagent combination, about 80 per cent of the copper and 70 per cent of the silver was recovered in a reasonably high grade concentrate from each sample. Recoveries and concentrate grades were a little better for the higher grade sample.

Some further work is planned to investigate ways of improving copper and silver recoveries.

Yours sincerely,

J. Woodcock.

J.T. Woodcock
SENIOR RESEARCH SCIENTIST

CSIRO

ORE DRESSING INVESTIGATIONS, MELBOURNE

MELBOURNE UNIVERSITY ORE DRESSING LABORATORY, MINING DEPT., UNIVERSITY, PARKVILLE, N.2, VIC. TELEPHONE 341338, 342274

EXHIBIT NO. 8e

JTW/MC 18th January 1967

Mr D. Irving
Chapman Wood Griswold & Evans Pty. Ltd.
Suite 20
541 St Kilda Road
MELBOURNE S.C.2 Vic.

Dear Mr Irving,

## Mount Gunson samples

Further to my letter of 21st December 1966 we have now completed some results on drill cuttings from below the water table at the Gunson mine.

## Head assays

The head assays supplied by you are set out below together with those calculated by us from assays of our test products.

| Sample<br>No | Your assay<br>% Cu | Our calc. assay<br>% Cu | Our calc. assay<br>Ag, dwt/ton |
|---|---|---|---|
| D9 1069 | 1.08 | 0.94 | 11.2 |
| 1070 | 0.96 | 0.99 | 9.2 |
| 1071 | 1.03 | 0.90 | 7.7 |
| C8 1115 | 1.67 | 1.86 | 24.2 |
| 1116 | 2.30 | 2.92 | 21.0 |
| 1117 | 0.81 | 0.62 | 4.8 |
| 1118 | 0.54 | 0.60 | 6.6 |

## Test results

Results of flotation and leaching are set out below. You will recall that each sample was treated by flotation, and each flotation tailing was then leached with sulphuric acid.

## Discussion

These results reveal several features of interest which may be summarized as follows.

1. Our calculated assays are in moderately good agreement with your assays but there are one or two discrepancies.

| Sample No | Product | % Wt. | Assay |  | % distrib. |  |
|---|---|---|---|---|---|---|
|  |  |  | Cu % | Ag dwt/ton | Cu | Ag |
| D9-1069 | Flot.conc. | 4.2 | 19.9 | 197 | 88.8 | 73.7 |
|  | Flot.tail | 95.8 | 0.11 | 3.1 | 11.2 | 26.3 |
|  | Preg.soln. | 191.6 | 0.008 | n.d. | 1.6 |  |
|  | Leach res. | 95.0 | 0.095 | 3.1 | 9.6 |  |
|  | Calc.feed | 100.0 | 0.94 | 11.2 | 100.0 | 100.0 |
| D9-1070 | Flot.conc. | 3.0 | 27.9 | 232 | 84.9 | 75.9 |
|  | Flot.tail | 97.0 | 0.15 | 2.3 | 15.1 | 24.1 |
|  | Preg.soln. | 194.0 | 0.01 | n.d. | 1.9 |  |
|  | Leach res. | 96.3 | 0.135 | 2.3 | 13.2 |  |
|  | Calc.feed | 100.0 | 0.99 | 9.2 | 100.0 | 100.0 |
| D9-1071 | Flot.conc. | 2.9 | 25.8 | 194 | 82.8 | 73.5 |
|  | Flot.tail | 97.1 | 0.16 | 2.1 | 17.2 | 26.5 |
|  | Preg.soln. | 194.2 | 0.013 | n.d. | 2.8 |  |
|  | Leach res. | 96.6 | 0.135 | 2.1 | 14.4 |  |
|  | Calc.feed | 100.0 | 0.90 | 7.7 | 100.0 | 100.0 |
| C8-1115 | Flot.conc. | 2.7 | 44.2 | 770 | 64.0 | 85.9 |
|  | Flot.tail | 97.3 | 0.69 | 3.5 | 36.0 | 14.1 |
|  | Preg.soln. | 194.6 | 0.303 | n.d. | 31.7 |  |
|  | Leach res. | 94.9 | 0.085 | 3.6 | 4.3 |  |
|  | Calc.feed | 100.0 | 1.86 | 24.2 | 100.0 | 100.0 |
| C8-1116 | Flot.conc. | 5.3 | 52.5 | 359 | 95.3 | 90.6 |
|  | Flot.tail | 94.7 | 0.14 | 2.1 | 4.7 | 9.4 |
|  | Preg.soln. | 189.4 | 0.018 | n.d. | 1.2 |  |
|  | Leach res. | 93.9 | 0.110 | 2.1 | 3.5 |  |
|  | Calc.feed | 100.0 | 2.92 | 21.0 | 100.0 | 100.0 |

(cont'd)

n.d. : not determined.

| Sample | Product | % Wt. | Assay | | % distrib. | |
|---|---|---|---|---|---|---|
| No | | | Cu % | Ag dwt/ton | Cu | Ag |
| C8-1117 | Flot.conc. | 2.2 | 25.3 | 170 | 89.7 | 77.8 |
| | Flot.tail | 97.8 | 0.065 | 1.1 | 10.3 | 22.2 |
| | Preg.soln. | 195.6 | 0.008 | n.d. | 2.6 | |
| | Leach res. | 96.9 | 0.050 | 1.1 | 7.7 | |
| | Calc.feed | 100.0 | 0.62 | 4.8 | 100.0 | 100.0 |
| C8-1118 | Flot.conc. | 2.1 | 24.5 | 260 | 85.5 | 82.5 |
| | Flot.tail | 97.9 | 0.09 | 1.2 | 14.5 | 17.5 |
| | Preg.soln. | 195.8 | 0.015 | n.d. | 4.8 | |
| | Leach res. | 96.8 | 0.060 | 1.2 | 9.7 | |
| | Calc.feed | 100.0 | 0.60 | 6.6 | 100.0 | 100.0 |

n.d. : not determined

2. The flotation tailings assay 0.1 - 0.15 per cent copper (except for C8-1115, where oxide copper was present). This represents a recovery in the concentrate of 85-95 per cent of the copper. Grinding and other conditions may not have been ideal in these tests which were performed using a standard set of conditions, and it would be reasonable to conclude that at least 90 per cent recovery can be obtained on average sulphide ore. Silver recoveries also varied but in general were lower than those of copper.

3. Concentrate grades varied, and were in general fairly low for a chalcocite ore, although the aim in the tests was to obtain a high recovery. Insoluble assays on the concentrates were in the range 23 - 53 per cent indicating (probably) that some free quartz reported in the concentrates. However some fairly high grades were achieved in the rougher stage and it is reasonable to conclude that concentrate grades of at least 40 per cent copper can be obtained. Silver assays of the concentrate could be about 300 dwt/ton, but this would depend on the silver content of the feed.

4. In most tests leaching of the flotation tailing extracted only a few per cent of the copper, and would not be economic. However on sample C8-1115, leaching was necessary as nearly one-third of the total copper reported in the pregnant solution. On mixed ore, containing sulphide and oxide minerals of copper, flotation plus leaching may be needed to obtain a satisfactory result.

## Conclusion

These preliminary results indicate that flotation of sulphide ore should not present much difficulty. All of the samples forwarded have been used and we will look forward to investigating a bulk sample of sulphide ore.

Yours sincerely

J. Woodcock.

J. T. Woodcock
SENIOR RESEARCH SCIENTIST

CSIRO

ORE DRESSING INVESTIGATIONS, MELBOURNE

EXHIBIT NO. 8g

MELBOURNE UNIVERSITY ORE DRESSING LABORATORY, MINING DEPT., UNIVERSITY, PARKVILLE, N.2, VIC. TELEPHONE 341338, 342274

15th February, 1967.

Mr. J.B. Evans,
Managing Director,
Chapman, Wood, Griswold & Evans Pty Ltd.,
Suite 20,
541 St. Kilda Road,
MELBOURNE, S.C.2. : Victoria.

Dear Mr. Evans,

Lagoon samples from Mount Gunson

Further to my letter of 1st February, 1967 we have now completed some additional flotation tests on the samples taken from Pernatty Lagoon (point C4). The general technique used was similar to that described previously, but finer grinding was used in the work reported here.

Grinding times and per cent minus 200 mesh are set out below.

| Sample | Test No | Grind min | % minus 200 |
|---|---|---|---|
| C4 (10 - 15) | 401 | 10 | 32 |
| | 403 | 15 | 43 |
| | 404 | 20 | 52 |
| C4 (15 - 20) | 402 | 10 | 35 |
| | 405 | 15 | 44 |
| | 406 | 20 | 52 |

Results of tests 401 and 402 were reported in my letter of 1st February, and results of the other tests are shown below.

| Test No. | Product | % wt | Assay | | % distrib. | |
|---|---|---|---|---|---|---|
| | | | Cu % | Ag dwt/ton | Cu | Ag |
| 403 | Flot. conc. | 2.7 | 33.4 | 188.7 | 84.5 | 73.4 |
| | Flot. tail. | 97.3 | 0.17 | 1.9 | 15.5 | 26.6 |
| | Calc. feed | 100.0 | 1.07 | 6.9 | 100.0 | 100.0 |
| 404 | Flot. conc. | 2.6 | 31.7 | 177.0 | 85.8 | 73.5 |
| | Flot. tail | 97.4 | 0.14 | 1.7 | 14.2 | 26.5 |
| | Calc. feed | 100.0 | 0.96 | 6.3 | 100.0 | 100.0 |
| 405 | Flot. conc. | 3.6 | 44.9 | 283.5 | 84.0 | 74.6 |
| | Flot. tail | 96.4 | 0.32 | 3.6 | 16.0 | 25.4 |
| | Calc. feed | 100.0 | 1.92 | 13.7 | 100.0 | 100.0 |
| 406 | Flot. conc. | 3.8 | 44.0 | 271.8 | 84.5 | 74.9 |
| | Flot. tail | 96.2 | 0.32 | 3.6 | 15.5 | 25.1 |
| | Calc. feed | 100.0 | 1.98 | 13.8 | 100.0 | 100.0 |

These results show that finer grinding (15 or 20 minutes, i.e. 43 or 52 per cent minus 200 mesh) has given an improvement in recovery of copper and of silver compared with flotation after a 10 minute grind (about 33 per cent minus 200 mesh). However the difference between 15 and 20 minutes grinding is small, and even though the finest grind used (52 per cent minus 200 mesh) is not particularly fine in terms of flotation plant practice, it does not seem that still finer grinding would be particularly beneficial.

The concentrate grades were not noticeably affected by variation in fineness of grinding. Typical grades were about 33 per cent copper and 180 dwt/ton silver on the sample from the 10 - 15 ft level and were about 44 per cent copper and 280 dwt/ton silver on the sample from the 15 - 20 ft level.

The recovery of about 85 per cent of the copper and about 75 per cent of the silver obtained on both samples leaves scope for improvement and some further tests are planned.

Yours sincerely,

J. Woodcock.

J.T. Woodcock
SENIOR RESEARCH SCIENTIST

CSIRO

ORE DRESSING INVESTIGATIONS, MELBOURNE

MELBOURNE UNIVERSITY ORE DRESSING LABORATORY, MINING DEPT., UNIVERSITY, PARKVILLE, N.2, VIC. TELEPHONE 341338, 342274

JTW/PMO

6626 9th March, 1967

Mr. J.B. Evans,
Managing Director,
Chapman, Wood, Griswold & Evans Pty. Ltd.,
Suite 27,
67 Queens Road,
MELBOURNE. S.C.3., Victoria

Dear Mr. Evans,

LAGOON SAMPLES FROM MOUNT GUNSON

Further to my letter of 15th February we have now completed some additional flotation tests on the samples taken from Pernatty Lagoon (point C4). The general conditions used were similar to those described previously, but in the work reported here tests were conducted in soda ash and in lime circuits.

Specific conditions for the tests are set out below. The pH values are the average of the values at the start and end of the test.

| Sample | Test No. | Grind min | Reagent $Na_2CO_3$ lb/ton | Reagent CaO lb/ton | pH |
|---|---|---|---|---|---|
| C4(10-15) | 407 | 15 | 1.0 | - | 7.8 |
| (A) | 408 | 15 | - | 1.0 | 8.8 |
| C4(15-20) | 409 | 15 | 1.5 | - | 7.7 |
| (B) | 410 | 15 | - | 1.0 | 8.4 |

Results of the tests are set out below.

| Test No. | Product | % wt | Assay Cu % | Assay Ag dwt/ton | % distrib Cu | % distrib Ag |
|---|---|---|---|---|---|---|
| 407 | Flot.conc. | 2.7 | 33.3 | 169.2 | 86.0 | 68.1 |
| | Flot.tail | 97.3 | 0.15 | 2.2 | 14.0 | 31.9 |
| | Calc.feed | 100.0 | 1.05 | 6.7 | 100.0 | 100.0 |
| 408 | Flot.conc. | 2.5 | 33.7 | 191.2 | 82.0 | 71.0 |
| | Flot.tail | 97.5 | 0.19 | 2.0 | 18.0 | 29.0 |
| | Calc.feed | 100.0 | 1.03 | 6.7 | 100.0 | 100.0 |
| 409 | Flot.conc. | 4.0 | 43.8 | 281.6 | 85.9 | 78.0 |
| | Flot.tail | 96.0 | 0.30 | 3.3 | 14.1 | 22.0 |
| | Calc.feed | 100.0 | 2.04 | 14.4 | 100.0 | 100.0 |
| 410 | Flot.conc | 4.0 | 43.5 | 266.0 | 89.2 | 81.0 |
| | Flot.tail | 96.0 | 0.22 | 2.6 | 10.8 | 19.0 |
| | Calc.feed | 100.0 | 1.95 | 13.1 | 100.0 | 100.0 |

...2/...

In these tests, particularly those in lime circuit, the sulphides tended to float more quickly than those in previous tests. Concentrate grades were not greatly affected, but some of the recoveries differ from those obtained previously.

For Sample A flotation in soda ash circuit (Test 407) gave an improved copper recovery but a poorer silver recovery compared with results obtained previously. In lime circuit (Test 408) recoveries were slightly poorer than those at natural pH.

For Sample B flotation in soda ash circuit gave slightly better recoveries of copper and of silver. However in lime circuit (Test 410) significantly better recoveries of copper and of silver were obtained. These are the best results to date.

There appears to be a definite difference between the two samples and we are planning to investigate this further.

Yours sincerely,

J. Woodcock.

J.T. Woodcock
SENIOR RESEARCH SCIENTIST

EXHIBIT NO. 8j

000123

**CSIRO**

ORE DRESSING INVESTIGATIONS, MELBOURNE

MELBOURNE UNIVERSITY ORE DRESSING LABORATORY, MINING DEPT., UNIVERSITY, PARKVILLE, N.2, VIC. TELEPHONE 341338, 342274

6615

21st April, 1967

Mr. J.B. Evans,
Chapman, Wood, Griswold & Evans Pty. Ltd.,
Suite 27,
67 Queens Road,
MELBOURNE. S.C.3

Dear Mr. Evans,

## Flotation of sulphide ore from Mount Gunson

We have completed two flotation tests on the composite sample of drill cuttings forwarded to us. This sample was a composite from holes C6, D7, D8, D9, D10, D11, E8, and E9.

The head assay was as follows:

| | |
|---|---|
| Copper | 1.71% |
| Oxide copper | 0.24% |
| Sulphur | 0.62% |
| Silver | 18.5 dwt/ton |
| Gold | > 0.1 dwt/ton |

One flotation test (410) was conducted at natural pH (about 7) and one test (411) at pH 10 (after a CaO addition of 1 lb/ton). Both tests were conducted after grinding to 63 per cent minus 200 mesh. In both tests a 15 minute flotation period was used and a total of 0.4 lb/ton secondary butyl xanthate and about 0.05 lb/ton Aerofroth 65 was added. The sulphides seemed to float more quickly in the lime circuit. Results of the tests are set out below.

| Test No. | Product | % wt. | Assay Cu % | Assay Ag dwt/ton | % distrib. Cu | % distrib. Ag |
|---|---|---|---|---|---|---|
| 410 | Flot. conc. | 3.8 | 35.5 | 365 | 78.9 | 77.4 |
| | Flot. tail | 96.2 | 0.37 | 4.2 | 21.1 | 22.6 |
| | Calc. feed | 100.0 | 1.71 | 17.9 | 100.0 | 100.0 |
| 411 | Flot. conc | 4.2 | 34.9 | 354 | 86.0 | 84.7 |
| | Flot. tail | 95.8 | 0.25 | 2.8 | 14.0 | 15.3 |
| | Calc. feed | 100.0 | 1.71 | 17.6 | 100.0 | 100.0 |

These results show that flotation in lime circuit was more effective than flotation at natural pH as a higher recovery of copper and of silver has been obtained with about the same concentrate grade.

In test 411 about 86 per cent of the copper and 85 per cent of the silver were recovered in a rougher concentrate assaying 35 per cent copper and 354 dwt/ton silver. It is likely that a higher concentrate grade could be obtained by cleaning and it may be possible to obtain a slightly higher recovery by variation in flotation conditions.

...2/...

In general these results are similar to those obtained on the lagoon sample indicating that ore from both sources could be treated together if this were desirable. However, as I have already mentioned to you, it is desirable that we investigate a sample of lump sulphide ore to check these results.

The flotation tailing from both the above tests was leached with dilute sulphuric acid and about half the copper present dissolved in the solution indicating that part of the copper lost in flotation tailing was present as oxide minerals. It may be possible to float some of the oxide minerals after the sulphide float and this would be worth investigating.

Yours sincerely,

J. Woodcock.

J.T. Woodcock
SENIOR RESEARCH SCIENTIST

CSIRO

ORE DRESSING INVESTIGATIONS, MELBOURNE

MELBOURNE UNIVERSITY ORE DRESSING LABORATORY, MINING DEPT., UNIVERSITY, PARKVILLE, N.2, VIC. TELEPHONE 341338, 342274

EXHIBIT NO. 8k

6615

24th April, 1967

Mr. J.B. Evans,
Chapman, Wood, Griswold & Evans Pty. Ltd.,
Suite 27,
67 Queens Road,
MELBOURNE. S.C.3., Victoria

Dear Mr. Evans,

Flotation of sulphide ore from Mount Gunson

Further to my letter of 21st April we have now calculated the distribution of oxide copper and sulphide copper in the two tests we reported. These calculations were based on the assumptions that all the copper dissolved by sulphuric acid leaching of flotation tailing was present as oxide copper and that the arithmetical difference between the total copper assay was equivalent to sulphide copper. The results obtained are shown below.

| Test No. | Product | % wt. | Assay (%) | | | % distrib. | | |
|---|---|---|---|---|---|---|---|---|
| | | | Total Cu | Ox Cu | Sulp Cu | Total Cu | Ox Cu | Sulp Cu |
| 410 | Flot.conc. | 3.8 | 35.5 | 1.6* | 33.9* | 78.9 | 25 | 87.8 |
| | Flot.tail | 96.2 | 0.37 | 0.18 | 0.19* | 21.1 | 75 | 12.2 |
| | Calc.feed | 100.0 | 1.71 | 0.24 | 1.47* | 100.0 | 100 | 100.0 |
| 411 | Flot.conc. | 4.2 | 34.9 | 2.6* | 32.3* | 86.0 | 46 | 92.5 |
| | Flot.tail | 95.8 | 0.25 | 0.14 | 0.11* | 14.0 | 54 | 7.5 |
| | Calc.feed | 100.0 | 1.71 | 0.24 | 1.47* | 100.0 | 100 | 100.0 |

* Calculated assay

Considering Test 411 as being the better of the two tests these results show that 46 per cent of the oxide copper and 92 per cent of the sulphide copper was recovered in a flotation test designed to float sulphide minerals. The overall recovery of copper from flotation feed was 86 per cent, and it may be possible to increase this to 90 per cent.

We have now determined the sulphur content of the sample as follows:

Sulphur 0.62%

Yours sincerely,

J. Woodcock.

J.T. Woodcock
SENIOR RESEARCH SCIENTIST

COMMONWEALTH SCIENTIFIC AND INDUSTRIAL RESEARCH ORGANIZATION, AUSTRALIA

RESEARCH FOR INDUSTRY

**THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES**

CONYNGHAM STREET - FREWVILLE - SOUTH AUSTRALIA
TELEPHONE 791662 · TELEGRAMS 'AMDEL' ADELAIDE 5063

*Please quote this reference in your reply:*
ML 3/262/0

14th August, 1967

*Your reference:*

Mr J.B. Evans,
Managing Director,
Austminex Pty Ltd,
Suite 20,
67, Queen's Road,
MELBOURNE, Victoria. 3000

## PART REPORT ML 3276/67

YOUR REFERENCE: Application dated 16/6/67

MATERIAL: Copper Ore

LOCALITY: Mount Gunson, S.A.

IDENTIFICATION: Samples B4, H11, C4

WORK REQUIRED: Grinding, settling and filtering tests

Investigation and Report by: L.J. Weir and G. Sinica

Officer in Charge, Metallurgy Section: P.K. Hosking

PK Hosking
P.A. Young
Director.

# GRINDING, THICKENING AND FILTRATION CHARACTERISTICS OF MT GUNSON ORE

## 1. INTRODUCTION

Following laboratory flotation feasibility tests and flotation plant capital and operating cost estimates described in Amdel Report 541, the following tests were requested to permit closer specification of equipment requirements, for cost estimation and plant design:

a. grindability of ore samples to flotation feed sizing

b. settling tests on ground ore to simulate thickening of flotation tailing

c. filtration tests on thickened material

## 2. MATERIAL EXAMINED

| | |
|---|---|
| Oxide Ore | Sample B4, minus 1-inch ore, received November, 1966<br>" H11, " " " " " " |
| Sulphide Ore | Sample C4, nominally minus 2-inch ore, received July, 1967 |

## 3. EQUIPMENT

British Standard Series screens (BSS)
Crushing rolls, 10-inch diameter x 6 inch
Grindability mill, after Bond(1)
Graduated cylinders, 500 ml, with 1 rpm rake stirrers
Filter test leaf, 1/10 sq.ft. area, with accessory vacuum pump and filtrate receiver

## 4. ANCILLARY MATERIALS

Sulphuric acid, commercial grade
Sodium sulphide, caustic flake grade (60% $Na_2S$)
Phosphorous Pentasulphide, commercial grade, Alginate Industries, London, supplied by Albright and Wilson (Aust.) Pty Ltd
Aerofloat 208, Cyanamid Australia Pty Ltd
Separan AP30, Dow Chemical Co., supplied by Swift Trading Co.
Superfloc 127, Cyanamid Australia Pty Ltd

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(1) BOND, F.C. (1949) "Standard Grindability Tests Tabulated" Trans. Amer. Min. Metall. Engng., Vol. 4

## 5. EXPERIMENTAL PROCEDURE AND RESULTS

### 5.1 Sample Preparation

The ore samples received were rolls-crushed in stages in closed circuit with a 5-mesh screen. The crushed product in each case was thoroughly mixed and riffled into aliquots of approximately one pound.

### 5.2 Grindability Tests

The grindability of each ore was determined according to the method of Bond. Charges of700 ml of ore were dry ground in a 12 x 12-inch mill, and the product screened wet and dry at the grindability mesh size. Oversize was made up to feed volume with new ore, and the batch re-ground for a number of mill revolutions calculated to give an oversize circulating load of 250%.

Grindabilities were determined at 72- and 150-mesh, aiming for products containing 80% passing 100- and 200-mesh respectively.

Total grindability product was retained, and filtered, dried and sampled for subsequent test-work. Feed samples and products from the latter grindability stages at equilibrium conditions were sized, results of which are shown in Figures 1 to 3 inclusive.

The results of grindability tests are shown in Tables 1 to 6 inclusive.

Grinding work indices were calculated after Holmes$^{(2)}$, from the formula:

$$\text{Work Index, } W_i = \frac{44.5}{P_1^{0.23}\, G_{bp}^{0.82} \left( \frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}} \right)}$$

where $P_1$ = grindability screen aperture, microns

$G_{bp}$ = ball mill grindability product, g/mill revolution

$P$ = product size, screen aperture in microns through which 80% of product passes

$F$ = feed size, screen aperture in microns through which 80% of feed passes

Data derived from grindability tests and calculated work indices are shown in Table 7.

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(2) HOLMES, J.A. (1957) "A Contribution to the Study of Comminution" Trans. Inst. Chem. Engrs., Vol. 35

Calculated work indices were plotted against product sizes in per cent passing 100- and 200-mesh respectively, as shown in Figure 4. The sizing of the anticipated plant flotation feed superimposed on this graph gives interpolated work indices for the three ore samples investigated:

| Sample | Plant Work-index, Wi kWh/short ton |
|---|---|
| B4 | 15.0 |
| H11 | 14.9 |
| C4 | 22.2 |

Mill power input can then be calculated from the expression

$$W = 10\ Wi\ \left(\frac{1}{\sqrt{P}} - \frac{1}{\sqrt{F}}\right)$$

Thus for an assumed plant feed, 80% of which passes 12,000 microns, aiming for a product size 80% passing 105 microns (derived from laboratory batch tests, Amdel Report 541) and using the work index for sample C4,

$$W = 10 \times 22.2\ \left(\frac{1}{\sqrt{105}} - \frac{1}{\sqrt{12,000}}\right)$$

$$= 222\ (0.0976 - 0.0091)$$

$$= 19.65\ \text{kWh/short ton}$$

Horsepower required is then given by

$$HP = W \times \frac{\text{tons/day}}{18}$$

$$= 19.65 \times \frac{1000}{18}$$

$$= 1090\ \text{HP}$$

$$\text{Installed HP} = 1090 + 10\%$$

$$= 1200\ \text{HP}$$

## 5.3 Settling Tests

Samples of product from grindability tests were adjusted to 30% solids in mains water to give pulp samples of 500 ml for settling tests.

### 5.3.1 Determination of Settling Area

The samples were agitated by means of a plunger stirrer, and allowed to settle, noting the height of the settled solids/supernatent liquor interface at suitable time intervals.

Settling tests were commenced with and without flotation collector and, in the case of Samples B4 and H11, sulphidising activator added to the pulp to simulate flotation tailing pulp conditions. It was apparent that, while pulps in mains water gave reasonable liquor clarification on settling, those containing sodium sulphide were well dispersed and did not give acceptable liquor clarity. The addition of flocculants was therefore investigated, as well as adjustment of pH from a value of 9.5 with a dosage of 2 lb of sodium sulphide per ton of solids, to below neutral. The use of an alternative sulphidising agent which gave no rise in pulp pH was also investigated.

Results of the free settling tests are shown in Figures 5 to 13 inclusive.

Settling area requirements were calculated from settling curves using the method of Couche and Goldney(3).

The method consists of constructing a tangent to the settling curve at the compression point to intersect the final settled pulp height line, and reading off a settling time at this point. The unit settling area required is then given by the expression:

$$A = \frac{0.3\ tu}{Co\ \ Ho}$$

where A = area, sq.ft/long ton of solids/24 hours
tu = settling time, minutes
Co = feed concentration g solid/ml
Ho = feed height, inches

Areas calculated from the settling tests are shown in Tables 8, 9, and 10.

### 5.3.2 Determination of Settling Depth

Samples of pulp treated with flocculant to give satisfactory liquor clarity and settling rate as described in 5.3.1 were re-agitated and allowed to settle whilst raking the settled solids layer with a 1 rpm motor-driven agitator to simulate a continuous thickener mechanism. The volume of the settled solids layer was recorded at suitable time intervals until maximum compaction of solids was reached. Dilutions and pulp specific gravity in the compression zone were then calculated, and are shown in Table 11.

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(3) COUCHE, R.A. AND GOLDNEY, L.H. (1959) "The Design of Continuous Thickeners for Flocculated Materials", Proc. Aus. I.M.M. No. 191

### 5.3.3 Selection of Thickener Tank Size

A tank size for each of the three ore samples tested was calculated, taking a mean settling area between the two sizes of feed tested, and applying this to mean values obtained from raked settling tests.

Tank size calculations are shown in Appendices A, B and C.

## 5.4 Filter Tests

Ore samples from grindability tests were thickened to the water-solids ratios indicated by settling tests (Table 11) and treated by the addition of flocculants and flotation reagents as shown:

| | Solids % | Reagent | lb/ton |
|---|---|---|---|
| Sample B4 | 66.5, - 72 mesh | Phosphorus Pentasulphide<br>Aerofloat 208<br>Separan AP30 | 2.0<br>0.2<br>0.0025 |
| | 63.2, - 150 mesh | Phosphorus Pentasulphide<br>Aerofloat 208<br>Separan AP30 | 2.0<br>0.2<br>0.0025 |
| Sample H11 | 71.0, - 72 mesh | Phosphorus Pentasulphide<br>Aerofloat 208<br>Separan AP30 | 2.0<br>0.2<br>0.0025 |
| | 67.8, - 150 mesh | Phosphorus Pentasulphide<br>Aerofloat 208<br>Separan AP30 | 2.0<br>0.2<br>0.0025 |
| Sample C4 | 65.6, - 72 mesh | Aerofloat 208<br>Superfloc 127 | 0.2<br>0.003 |
| | 64.6, - 150 mesh | Aerofloat 208<br>Superfloc 127 | 0.2<br>0.003 |

Slurry samples were agitated in a holding tank by means of a perforated-plate plunger. The test filter leaf, connected to the vacuum source through a filtrate receiving flask, was immersed into the pulp, vacuum was then applied instantaneously, and cake formation allowed to proceed for a measured time. The disc was then removed from the pulp, inverted, and the cake allowed to drain under vacuum for a measured time.

The drained cake was then discharged by blowing lightly into the filter leaf, the cake was weighed, dried and re-weighed. Heavy edge was trimmed from the cake, and where it was in excess of 10% by weight of the total cake, calculated filtrate gallonage was corrected accordingly.

A short series of tests was done using various filter cloths of cotton and synthetic fabrics. Little difference in filtration rate was noted with those cloths giving acceptable clarity of filtrate, and a 9 oz. multi-weave polypropylene (Eimco Popr 853S) cloth was selected for the main filter tests.

Applied vacuum in all tests was in the range 16-18 inches mercury.

### 5.4.1 Determination of Cake Forming Time.

A series of tests was done on several samples of size fractions of the three ores, investigating a range of cake form times between 5 and 30 seconds. Cake drainage time for all tests was three times form time. Results of the tests are shown in Table 12, and in Figure 14. All cakes formed were readily discharged from the filter.

### 5.4.2 Determination of Cake Draining Time

From the results of tests at variable form time, a time of 30 seconds was taken for filter design purposes. A series of tests was then done at this form time, and at variable drainage time to determine the effect on cake product moisture content.

Results of the tests are shown in Table 13 and in Figure 15.

### 5.4.3 Determination of Plant Filter Requirements

Filter form and drain times of 30 and 60 seconds respectively were chosen for plant conditions. Thus, leaving 30 seconds for cake discharge plus re-submergence, the total filter cycle time is two minutes.

Filter area requirements were calculated on this basis for the cake weights obtained in the laboratory tests. To the laboratory figure a factor of 0.65 was applied to scale-up to plant filtration rate, allowing for variation in vacuum, cloth blinding, and variations in feed pulp density, etc.

Calculated areas required are shown in Appendices D, E and F.

## 6. DISCUSSION OF RESULTS

### 6.1 Grindability

The three ores tested gave products of similar size distribution. Calculated work indices for samples B4 and H11 were similar, but those for Sample C4 were very much higher due mainly to a substantial difference in feed size distribution (see Figure 3).

Calculated mill size required is based on a feed size F of 1200 microns, taken from a sizing of impact-milled ore nominally passing ½-inch (see Figure 4, Report ML 3277/67).

### 6.2 Settling Tests and Thickener Design

Samples B4 and H11 gave reasonably fast settling rates when pulped to 30% solids, and treated with reagents to simulate flotation tailing pulp conditions, but produced unacceptable liquor clarity for thickener design. An anionic flocculant was used, primarily to promote liquor clarity and partly to improve settling rate. Sample C4 gave acceptable liquor clarity on settling, and a non-ionic flocculant proved more satisfactory to increase settling rate than an anionic type.

Thickener area requirements were based on the use of flocculant at dosages producing satisfactory settling rates, but the optimum dosages were not determined.

### 6.3 Filter Tests

Thickened slurries of samples B4 and C4 gave only moderate filtration rates. Sample H11 produced excellent cake thickness, with consequent high filtration rate.

All samples produced cakes which did not crack under drainage, and which readily discharged from the filter.

Filter area requirements were calculated using a safety factor suggested by a filter manufacturer (Dorr-Oliver Inc.).

## APPENDIX A

## CALCULATION OF THICKENER TANK SIZE

### SAMPLE B4

From Table 8, settling area requirements for flocculated pulps are:

| Size Fraction | Sq.ft./short ton/day |
|---|---|
| - 72 mesh | 1.11 |
| - 150 " | 4.55 |
| Mean | 2.83 |

i.e. assumed area for plant feed = 2.83 sq. ft./ton/day

For 1000 ton/day, tank diameter = 60 ft
Allow 20% safety factor = 12 ft
72 ft

From Table 11, compression zone data:

| Size Fraction | Dilution Average | Dilution Final | Specific Gravity Average | Time in Compression hours |
|---|---|---|---|---|
| - 72 mesh | 0.639 | 0.503 | 1.62 | 0.687 |
| - 150 " | 0.653 | 0.585 | 1.62 | 3.42 |
| Mean | 0.646 | 0.544 | 1.62 | 2.05 |

Pulp capacity required $= \frac{2000}{24 \times 2.83}$

= 29.4 lb/sq.ft./hr

To hold 29.4 lb solids for 2.05 hours = total solids storage of 60.4 lb

At dilution of 0.646:1 = 60.4 x 1.646
= 99.2 lb pulp

Therefore depth required in compression $= \frac{99.2}{62.5 \times 1.62}$ = 0.89 ft

Plus feed-well allowance 2 ft.

For 72 ft diameter tank, ineffective depth under rakes (two-thirds of total depth) at rake-arm slope of 1.75 in./ft. $= \frac{1.75}{12} \times \frac{72}{2} \times \frac{2}{3}$ = 3.5 ft

Total depth = 6.4 ft

Recommended thickener tank size = 72 ft x 7 ft

## APPENDIX B

## CALCULATION OF THICKENER TANK SIZE

### SAMPLE H11

From Table 9, settling area requirements for flocculated pulps are:

| Size Fraction | Sq.ft./short ton/day |
|---|---|
| - 72 mesh | 0.92 |
| - 150 " | 1.41 |
| Mean | 1.16 |

i.e. assumed area for plant feed = 1.16 sq.ft./ton/day

For 1000 ton/day, tank diameter = 38.4 ft
Allow 20% safety factor 7.6
46.0 ft

From Table 11, compression zone data:

| Size Fraction | Dilution | | Specific Gravity | Time in Compression |
|---|---|---|---|---|
| | Average | Final | Average | hours |
| - 72 mesh | 0.587 | 0.410 | 1.66 | 1.17 |
| - 150 " | 0.552 | 0.473 | 1.68 | 1.70 |
| Mean | 0.569 | 0.441 | 1.67 | 1.43 |

Pulp capacity required = $\frac{2000}{24 \times 1.16}$

= 72 lb/sq.ft./hour

To hold 72 lb solids for 1.43 hours = total solids storage of 103.5 lb

At dilution of 0.569:1 = 103.5 x 1.569

= 162.5 lb pulp

Therefore depth required in compression = $\frac{162.5}{62.5 \times 1.67}$ = 1.56 ft

Plus feed-well allowance 2 ft

For 46 ft diameter tank, ineffective depth under rakes (two-thirds of total depth) at rake-arm slope of 1.75 in./ft = $\frac{1.75}{12} \times \frac{46}{2} \times \frac{2}{3}$ = 2.24 ft

Total Depth = 5.80 ft

Recommended thickener tank size = 46 ft x 6 ft

# APPENDIX C

## CALCULATION OF THICKENER TANK SIZE

### SAMPLE C4

From Table 10, settling area requirements for flocculated pulps are:

| Size Fraction | Sq.ft./short ton/day |
|---|---|
| - 72 mesh | 2.19 |
| - 150 " | 2.89 |
| Mean | 2.54 |

i.e. assumed area for plant feed = 2.54 sq.ft./ton/day

For 1000 ton/day, tank diameter = 56.8 ft
Allow 20% safety factor 11.4
68.2 ft

From Table 11, compression zone data:

| Size Fraction | Dilution | | Specific Gravity | Time in Compression hours |
|---|---|---|---|---|
| | Average | Final | Average | |
| - 72 mesh | 0.636 | 0.525 | 1.63 | 1.38 |
| - 150 " | 0.648 | 0.549 | 1.62 | 1.97 |
| Mean | 0.642 | 0.537 | 1.63 | 1.68 |

Pulp capacity required $= \dfrac{2000}{24 \times 2.54}$

$= 32.8$ lb/sq.ft./hr

To hold 32.8 lb solids for 1.68 hr = total solids storage of 55 lb

At dilution of 0.642:1 $= 55 \times 1.642$

$= 90.4$ lb pulp

Therefore depth required in compression $= \dfrac{90.4}{62.5 \times 1.63} = 0.89$ ft

Plus feed-well allowance 2 ft

For 68 ft diameter tank, ineffective depth under rakes (two-thirds of total depth) at rake-arm slope of 1.75 in./ft. $= \dfrac{1.75}{12} \times \dfrac{68}{2} \times \dfrac{2}{3} = \underline{3.31}$ ft

Total Depth = 6.20 ft

Recommended thickener tank size = 68 ft x 6½ ft.

# APPENDIX D

## CALCULATION OF FILTER AREA REQUIREMENT

## SAMPLE B4

From Table 12, filtration rates for a 30 second form time are:

| Size Fraction | Dry Cake Weight g/1/10th sq.ft. |
|---|---|
| - 72 mesh | 111 |
| - 150 " | 95.5 |
| Mean | 103 |

For a two-minute filter cycle (30 sec. form, 60 sec. drain and 30 sec. discharge)

$$\text{Filtration rate} = \frac{103}{454} \times 10 \times \frac{60}{2}$$

$$= 68.1 \text{ lb/sq.ft/hr}$$

$$= 0.817 \text{ short tons/sq.ft/day}$$

Allowing safety factor of 0.65

$$= 0.531 \text{ tons/sq.ft/day}$$

For 1000 tons/day, filter area requirement

$$= 1{,}884 \text{ sq. ft.}$$

# APPENDIX E

## CALCULATION OF FILTER AREA REQUIREMENT

### SAMPLE H11

From Table 12, filtration rates for a 30-second form time are:

| Size Fraction | Dry Cake Weight g/$^{1}$/10th sq.ft. |
|---|---|
| - 72 mesh | 326 |
| - 150 " | 286 |
| Mean | 306 |

For a two-minute filter cycle

Filtration rate $= \frac{306}{454} \times 10 \times \frac{60}{2}$

= 202 lb/sq.ft/hr

= 2.42 short tons/sq.ft/day

Allowing safety factor of 0.65

= 1.58 tons/sq.ft/day

For 1000 tons/day, filter area requirement

= 635 sq.ft.

# APPENDIX F

## CALCULATION OF FILTER AREA REQUIREMENT

### SAMPLE C4

From Table 12, filtration rates for a 30-second form time are:

| Size Fraction | Dry Cake Weight g/1/10th sq.ft. |
|---|---|
| - 72 mesh | 92 |
| - 150 " | 109 |
| Mean | 100 |

For a two-minute filter cycle,

filtration rate $= \frac{100}{454} \times 10 \times \frac{60}{2}$

= 66.1 lb/sq.ft/hr

= 0.794 short tons/sq.ft/day

Allowing safety factor of 0.65

= 0.515 tons/sq.ft/day

For 1000 tons/day, filter area requirement

= 1,940 sq.ft.

## TABLE 1: GRINDABILITY RESULTS

### Grindability at 72 mesh
### Sample B4

Finished material in feed (F) = 17.9%

| Stage | A<br>Wt. of 700 feed (g) | $A_1$<br>Wt. of new feed (g) | Y<br>Mesh in feed (g) | X<br>Gross Product for 250% circ. load | R<br>Number of Grinding Revs.<br>Calc. | <br><br>Actual | B<br>Screen over-size (g) | C<br>Gross Product (g) | D /<br>Net Product (g) | G<br>Grind-ability g/rev. | L<br>Circulating load % |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | 1139 | | 204 | | 80 | | 794 | 345 | 141 | 1.76 | 230 |
| 2 | 1123.2 | 329.2 | 59 | 316 | 146 | 97 | 873.8 | 248.4 | 189.4 | 1.95 | 350 |
| 3 | 1126.4 | 252.6 | 42.8 | 349 | 157 | | 785 | 341.4 | 298.6 | 1.90 | 230 |
| 4 | 1108 | 223.0 | 58.8 | 314 | 134 | 147 | 770.2 | 337.9 | 294.0 | 1.99 | 230 |
| 5 | 1104.4 | 334.2 | 59.8 | 308 | 126 | 130 | 786.0 | 318.4 | 258.6 | 2.05<br>1.99 | 247 |
| 6 | 1108 | 322.0 | 57.6 | 314 | 129 | | 794.0 | 314.0 | 256.4 | 1.99 | 252 |
| 7 | 1108.4 | 314.4 | 58.7 | 317 | 130 | | 791.2 | 317.2 | 259.2 | 1.99 | 250 |
| 8 | 1124.0 | 332.8 | 59.6 | 316 | 129 | | 764.2 | 359.8 | 300.2 | 2.37 | 320 |
| 9 | 1101.4 | 337.4 | 60.4 | 306 | 106 | 124 | 792.8 | 308.6 | 248.4 | 2.34<br>2.00 | 256 |
| 10 | 1102.4 | 309.6 | 55.4 | 316 | 130 | | 781.2 | 321.2 | 260.8 | 2.00 | 243 |

Average of Stages 5, 6, 7 and 9 inclusive - 250% circ. load, 1.99 g/rev.
2.09

$A_1 = A - B_p$

$Y = \frac{A_1 \times F}{100}$

$X = \frac{100\ B_p}{250}$

$R = \frac{X - Y}{G}$

$C = A - B$

$D = C - Y$

$G = \frac{D}{R}$

$L = \frac{100\ B}{C}$

Note: $B_p$ = B from previous grinding stage

## TABLE 2: GRINDABILITY RESULTS

### Grindability at 150 mesh

### Sample B4

Finished material in feed (F) = 9.1%

| Stage | A<br>Wt. of 700 feed (g) | $A_1$<br>Wt. of new feed (g) | Y<br>Mesh in feed (g) | X<br>Gross Product for 250% circ. load | R<br>Number of Grinding Revs. |  | B<br>Screen over-size (g) | C<br>Gross Product (g) | D<br>Net Product (g) | G<br>Grind-ability g/rev. | L<br>Circulating load % |
|---|---|---|---|---|---|---|---|---|---|---|---|
|  |  |  |  |  | Calc. | Actual |  |  |  |  |  |
| 1 | 1145 |  | 104 |  | 160 |  | 921 | 224 | 120 | 0.75 |  |
| 2 | 1124 | 203 | 18.5 | 368 | 465 | 300 | 850 | 274 | 255.5 | 0.85 | 310 |
| 3 | 1094.4 | 244.4 | 22.2 | 340 | 370 |  | 771.6 | 322.8 | 300.6 | 0.81 | 240 |
| 4 | 1080.2 | 308.6 | 28.1 | 309 | 347 |  | 771.0 | 309.2 | 281.1 | 0.81 | 250 |
| 5 | 1081.0 | 310.0 | 28.2 | 309 | 346 |  | 752.2 | 328.8 | 300.6 | 0.87 | 230 |
| 6 | 1091.0 | 338.8 | 30.8 | 301 | 310 |  | 760.0 | 331.0 | 300.2 | 0.97 | 230 |
| 7 | 1084.2 | 324.2 | 29.4 | 304 | 282 |  | 816.2 | 268.0 | 238.6 | 0.85 | 300 |
| 8 | 1047.6 | 231.4 | 21.0 | 327 | 360 | 356 | 725.2 | 332.4 | 311.4 | 0.86 | 220 |
| 9 | 1074.0 | 348.8 | 31.8 | 290 | 300 | 318 | 765.0 | 309.0 | 277.2 | 0.87 | 247 |
| 10 | 1095.2 | 330.2 | 30.1 | 306 | 316 |  | 780.2 | 315.0 | 284.9 | 0.89 | 248 |

Average of stages 5-10 inclusive - 250% circ. load, 0.885 g/rev.

$A_1 = A - B_p$ $\qquad$ $C = A - B$

$Y = \frac{A_1 \times F}{100}$ $\qquad$ $D = C - Y$

$X = \frac{100B_p}{250}$ $\qquad$ $G = \frac{D}{R}$

$R = \frac{X - Y}{G}$ $\qquad$ $L = \frac{100B}{C}$ $\qquad$ Note $\qquad$ $B_p$ = B from previous grinding stage

## TABLE 3: GRINDABILITY RESULTS

### Grindability at 72 mesh

### Sample H11

Finished material in feed (F) = 12.5%

| Sample | A<br>Wt. of 700 feed<br>(g) | $A_1$<br>Wt. of new feed<br>(g) | Y<br>Mesh in feed<br>(g) | X<br>Gross Product for 250% circ. load | R<br>Number of Grinding Revs.<br>Calc. | <br><br>Actual | B<br>Screen over-size<br>(g) | C<br>Gross Product<br>(g) | D<br>Net Product<br>(g) | G<br>Grind-ability<br>g/rev. | L<br>Circulating load<br>% |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | 1127.6 | | 141 | | 130 | | 741.6 | 386 | 245 | 1.88 | |
| 2 | 1048.2 | 306.6 | 38.3 | 296 | 136 | | 798.4 | 249.8 | 211.2 | 1.55 | 320 |
| 3 | 1123.3 | 324.9 | 40.5 | 319 | 180 | 172 | 756.8 | 366.5 | 321.5 | 1.87 | 207 |
| 4 | 1118.4 | 361.6 | 45.4 | 303 | 138 | | 782.0 | 336.4 | 291.4 | 2.1 | 233 |
| 5 | 1143.0 | 361.0 | 43.4 | 313 | 128 | | 837.0 | 306.0 | 261.0 | 2.04 | 270 |
| 6 | 1128.0 | 291.6 | 36.4 | 330 | 147 | | 775.8 | 352.2 | 315.8 | 2.14 | 220 |
| 7 | 1139.2 | 363.4 | 43.6 | 310 | 125 | | 813.8 | 325.4 | 381.8 | 3.25 | 250 |
| 8 | 1139.0 | 325.2 | 40.6 | 325 | 126 | | 822.0 | 317.0 | 276.8 | 2.19 | 259 |
| 9 | 1146.0 | 324.0 | 40.1 | 329 | 132 | | 805.0 | 341.0 | 300.9 | 2.27 | 236 |
| 10 | 1122.6 | 317.6 | 39.7 | 322 | 124 | | 820.6 | 302.0 | 282.3 | 2.27 | 290 |

Average of stages 5-9 inclusive - 250% circ. load, 2.18 g/rev.

$A_1 = A - B_p$ $C = A - B$

$Y = \frac{A_1 \times F}{100}$ $D = C - Y$

$X = \frac{100\ B_p}{250}$ $G = \frac{D}{R}$

$R = \frac{X - Y}{G}$ $L = \frac{100B}{C}$

Note: $B_p$ = B from previous grinding stage

## TABLE 4: GRINDABILITY RESULTS

### Grindability at 150 mesh

### Sample H11

Finished material in feed (F) = 5.9%

| Sample | A<br>Wt. of 700 feed<br>(g) | $A_1$<br>Wt. of new feed<br>(g) | Y<br>Mesh in feed<br>(g) | X<br>Gross Product for 250% circ. load | R<br>Number of Grinding Revs.<br>Calc. | <br><br>Actual | B<br>Screen over-size<br>(g) | C<br>Gross Product<br>(g) | D<br>Net Product<br>(g) | G<br>Grind-ability<br>g/rev. | L<br>Circulating load<br>% |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | 1121.2 | | 66.5 | | 300 | | 821.4 | 305.8 | 239.2 | 0.79 | 268 |
| 2 | 1106.4 | 281 | 16.6 | 328 | 394 | 360 | 786.8 | 319.6 | 393.0 | 0.84 | 247 |
| 3 | 1108.0 | 321.2 | 19 | 315 | 352 | | 766.4 | 341.6 | 322.6 | 0.91 | 225 |
| 4 | 1089.2 | 321.8 | 19 | 307 | 318 | | 745.4 | 343.8 | 324.8 | 1.02 | 217 |
| 5 | 1089.2 | 343.8 | 20 | 298 | 250 | 235 | 813.0 | 276.2 | 253.2 | 1.07 | 270 |
| 6 | 1088.2 | 275.4 | 16.2 | 325 | 289 | | 790.8 | 297.4 | 281.2 | 0.97 | 260 |
| 7 | 1087.4 | 296.6 | 17.5 | 316 | 307 | | 642.6 | 444.8 | 427.3 | 1.37 | 145 |
| 8 | 1111.4 | 468.6 | 27.6 | 257 | 160 | 282 | 710.0 | 401.0 | 374.4 | 1.32 | 174 |
| 9 | 1100.0 | 390.0 | 23.0 | 267 | 183 | 200 | 881.0 | 219.0 | 196.0 | 0.48 | 401 |
| 10 | 1131.4 | 250.4 | 14.8 | 352 | 344 | 300 | 754.8 | 376.6 | 361.8 | 1.20 | 200 |
| 11 | 1089.4 | 324.6 | 19.1 | 302 | 236 | | 848.0 | 248.4 | 221.9 | 0.94 | 340 |
| 12 | 1108.2 | 260.2 | 15.3 | 389 | 343 | 290 | 805.2 | 303.0 | 287.1 | 0.99 | 265 |
| 13 | 1074.6 | 269.4 | 15.9 | 322 | 309 | 300 | 768.6 | 306.0 | 290.1 | 0.97 | 250 |
| 14 | 1097.0 | 328.4 | 19.4 | 307 | 296 | | 785.5 | 311.5 | 292.1 | 0.98 | 252 |
| 15 | 1091.4 | 306.1 | 18.0 | 310 | 298 | 301 | 775.4 | 316.0 | 298.0 | 0.99 | 246 |

Average of stages 12-15 inclusive - 250% circulating load, 0.99 g/rev

$A_1 = A - B_p$ $\quad C = A - B$

$Y = \dfrac{A_1 \times F}{100}$ $\quad D = C - Y$

$X = \dfrac{100B_p}{250}$ $\quad G = \dfrac{D}{R}$

$R = \dfrac{X - Y}{G}$ $\quad L = \dfrac{100B}{C}$

Note: $B_p$ = B from previous grinding stage

## TABLE 5: GRINDABILITY RESULTS

### Grindability at 72 mesh

### Sample C4

Finished material in feed (F) = 22.2%

| Sample | A<br>Wt. of 700 feed (g) | $A_1$<br>Wt. of new feed (g) | Y<br>Mesh in feed (g) | X<br>Gross Product for 250% circ. load | R<br>Number of Grinding Revs. Calc. | R<br>Number of Grinding Revs. Actual | B<br>Screen over-size (g) | C<br>Gross Product (g) | D<br>Net Product (g) | G<br>Grind-ability g/rev. | L<br>Circulating Load % |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | 1116.0 | | 247.8 | | 71 | | 788.0 | 328 | 80.2 | 1.13 | 290 |
| 2 | 1116.2 | 328.2 | 72.9 | 315 | 214 | 150 | 849.0 | 267.2 | 184.3 | 1.23 | 317 |
| 3 | 1116.0 | 267.0 | 49.3 | 339 | 235 | 220 | 738.8 | 377.2 | 327.9 | 1.49 | 197 |
| 4 | 1076.2 | 337.4 | 74.8 | 295 | 148 | 162 | 761.4 | 314.8 | 240.0 | 1.48 | 242 |
| 5 | 1081.2 | 319.8 | 71.0 | 304 | 157 | | 781.2 | 300.0 | 229.0 | 1.46 | 260 |
| 6 | 1097.2 | 316.0 | 70.2 | 312 | 165 | | 780.0 | 317.2 | 247.0 | 1.47 | 246 |
| 7 | 1091.4 | 311.4 | 69.2 | 312 | 165 | | 776.0 | 315.4 | 246.2 | 1.49 | 247 |
| 8 | 1091.2 | 315.2 | 70.0 | 311 | 162 | | 778.0 | 313.0 | 243.0 | 1.50 | 247 |
| 9 | 1092.6 | 314.6 | 69.7 | 311 | 162 | | 776.8 | 315.8 | 246.1 | 1.50 | 250 |
| 10 | 1095.2 | 318.4 | 70.6 | 310 | 158 | | 785.0 | 310.2 | 239.6 | 1.51 | 253 |

Average of stages 7-10 inclusive - 250% circ. load, 1.50 g/rev

$A_1 = A - B_p$ $\quad$ $C = A - B$

$Y = \frac{A_1 \times F}{100}$ $\quad$ $D = C - Y$

$X = \frac{100\ B_p}{250}$ $\quad$ $G = \frac{D}{R}$

$R = \frac{X - Y}{G}$ $\quad$ $L = \frac{100\ B}{C}$ $\quad$ Note: $B_p$ = B from previous grinding stage

# TABLE 6: GRINDABILITY RESULTS

## Grindability at 150 mesh

## Sample C4

Finished material in feed (F) = 14.8%

| Sample | A Wt. of 700 feed (g) | $A_1$ Wt. of new feed (g) | Y Mesh in feed (g) | X Gross Product for 250% circ. load | R Number of Grinding Revs. | | B Screen over-size (g) | C Gross Product (g) | D Net Product (g) | G Grind-ability g/rev. | L Circulating load % |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | Calc. | Actual | | | | | |
| 1 | 1109 | | 164.4 | | 200 | | 839 | 270 | 106 | 0.53 | |
| 2 | 1118.8 | 279.8 | 42.4 | 310 | 504 | 356 | 846 | 272.8 | 231.4 | 0.65 | 310 |
| 3 | 1080.4 | 234.9 | 34.7 | 338 | 466 | 420 | 713 | 367.4 | 332.7 | 0.83 | 194 |
| 4 | 1059.0 | 346.0 | 51.2 | 295 | 294 | 325 | 748 | 311.0 | 259.8 | 0.80 | 240 |
| 5 | 1064.4 | 316.4 | 46.8 | 300 | 316 | | 747 | 317.4 | 270.6 | 0.85 | 237 |
| 6 | 1062.2 | 315.2 | 46.6 | 300 | 300 | | 777.2 | 285.0 | 238.4 | 0.80 | 270 |
| 7 | 1069.4 | 312.2 | 46.2 | 310 | 330 | | 762.5 | 306.9 | 260.9 | 0.79 | 249 |
| 8 | 1062.4 | 300.0 | 44.4 | 305 | 330 | | 758.4 | 304.0 | 259.6 | 0.79 | 250 |
| 9 | 1064.2 | 305.8 | 45.2 | 303 | 328 | | 756.8 | 307.4 | 262.2 | 0.80 | 247 |
| 10 | 1067.0 | 310 | 40.0 | 303 | 320 | | | | | | |

Average of stages 6-10 inclusive - 250% circ. load, 0.805 g/rev.

$A_1 = A - B_p$ $\qquad$ $C = A - B$

$Y = \frac{A_1 \times F}{100}$ $\qquad$ $D = C - Y$

$X = \frac{100\ B}{250}$ $\qquad$ $G = \frac{D}{R}$

$R = \frac{X - Y}{G}$ $\qquad$ $L = \frac{100\ B}{C}$ $\qquad$ Note: $B_p$ = B from previous grinding stage

## TABLE 7: SUMMARY OF GRINDABILITY TEST RESULTS

|  | Sample |  |  |  |  |  |
|---|---|---|---|---|---|---|
|  | B4 |  | H11 |  | C4 |  |
|  | Mesh |  | Mesh |  | Mesh |  |
|  | 72 | 150 | 72 | 150 | 72 | 150 |
| $P_1$ | 210 | 105 | 210 | 105 | 210 | 105 |
| P | 165 | 78 | 175 | 84 | 175 | 90 |
| F | 2700 | 2700 | 3350 | 3350 | 1350 | 1350 |
| $G_{bp}$ | 2.09<br>1.99 | 0.885 | 2.18 | 0.99 | 1.50 | 0.80 |
| Wi | 12.1<br>12.6 | 18.0 | 11.8 | 16.8 | 19.2 | 23.6 |

## TABLE 8: SETTLING AREA REQUIREMENTS FOR SAMPLE B4

| Sample | Reagent | Addition lb/ton | Settling Time min. (tu) | Pulp Height inches (Ho) | Pulp Concn. g/ml (Co) | Calc. Settling Area Reqd. sq.ft/ton/24 hr |  |
|---|---|---|---|---|---|---|---|
|  |  |  |  |  |  | Long tons | Short tons |
| B4, - 72 mesh | Blank |  | 66 | 11.6 | 0.38 | 5.24 | 4.68 |
|  | Sod. Sulphide | 2.0 |  |  |  |  |  |
|  | Aerofloat 208 | 0.2 |  |  |  |  |  |
|  | $H_2SO_4$ | 0.2 |  |  |  |  |  |
|  | Separan AP30 | 0.0125 | 15 | 10.0 | 0.38 | 1.19 | 1.06 |
| do. | $P_2S_5$ | 2.0 |  |  |  |  |  |
|  | Aerofloat 208 | 0.2 |  |  |  |  |  |
|  | Separan AP30 | 0.0125 | 17.3 | 11.0 | 0.38 | 1.24 | 1.11 |
| B4, - 150 mesh | Blank |  | 82 | 11.0 | 0.38 | 5.87 | 5.24 |
| do. | Sod. sulphide | 2.0 |  |  |  |  |  |
|  | Aerofloat 208 | 0.2 |  |  |  |  |  |
|  | $H_2SO_4$ | 0.2 |  |  |  |  |  |
|  | Separan AP30 | 0.0125 | 31 | 10.0 | 0.38 | 2.45 | 2.19 |
| do. | $P_2S_5$ | 2.0 |  |  |  |  |  |
|  | Aerofloat 208 | 0.2 | 126 | 11.0 | 0.38 | 9.05 | 8.06 |
| do. | $P_2S_5$ | 2.0 |  |  |  |  |  |
|  | Aerofloat 208 | 0.2 |  |  |  |  |  |
|  | Separan AP30 | 0.0025 | 71 | 11.0 | 0.38 | 5.10 | 4.55 |

## TABLE 9: SETTLING AREA REQUIREMENTS FOR SAMPLE H11

| Sample | Reagent | Addition lb/ton | Settling Time min. (tu) | Pulp Height inches (Ho) | Pulp Concn. g/ml (Co) | Calc. Settling Area Reqd. sq.ft/ton/ 24 hr. | |
|---|---|---|---|---|---|---|---|
| | | | | | | Tons Long | Short |
| - 72 mesh | Blank | | 32 | 10 | 0.38 | 2.59 | 2.31 |
| do. | Sod. sulphide<br>Aerofloat 208<br>$H_2SO_4$<br>Separan AP30 | 2.0<br>0.2<br>0.2<br>0.0125 | 11 | 10 | 0.38 | 0.955 | 0.852 |
| do. | $P_2S_5$<br>Aerofloat 208 | 2.0<br>0.2 | 17 | 10 | 0.38 | 1.81 | 1.62 |
| do. | $P_2S_5$<br>Aerofloat 208<br>Separan AP30 | 2.0<br>0.2<br>0.0025 | 13 | 10 | 0.38 | 1.03 | 0.920 |
| - 150 mesh | $P_2S_5$<br>Aerofloat 208 | 2.0<br>0.2 | 26.5 | 10 | 0.38 | 2.09 | 1.87 |
| do. | $P_2S_5$<br>Aerofloat 208<br>Separan AP30 | 2.0<br>0.2<br>0.001 | 24.5 | 11 | 0.38 | 1.76 | 1.57 |
| do. | $P_2S_5$<br>Aerofloat 208<br>Separan AP30 | 2.0<br>0.2<br>0.002 | 22 | 11 | 0.38 | 1.58 | 1.41 |
| do. | $P_2S_5$<br>Aerofloat 208<br>Separan AP30 | 2.0<br>0.2<br>0.003 | 17.5 | 10 | 0.38 | 1.38 | 1.23 |

## TABLE 10: SETTLING AREA REQUIREMENTS FOR SAMPLE C4

| Sample | Reagent | Addition lb/ton | Settling Time min. (tu) | Pulp Height inches (Ho) | Pulp Concn. g/ml (Co) | Calc. Settling Area Reqd. sq.ft/ton/ 24 hr | |
|---|---|---|---|---|---|---|---|
| | | | | | | Long tons | Short tons |
| - 72 mesh | Aerofloat 208<br>Separan AP30 | 0.2<br>0.001 | 73 | 11 | 0.38 | 5.24 | 4.68 |
| do. | Aerofloat 208<br>Superfloc 127 | 0.2<br>0.001 | 34 | 11 | 0.38 | 2.44 | 2.18 |
| do. | Aerofloat 208<br>Superfloc 127 | 0.2<br>0.003 | 31 | 10 | 0.38 | 2.45 | 2.19 |
| - 150 mesh | Blank | | 79 | 10 | 0.38 | 6.23 | 5.56 |
| do. | Aerofloat 208<br>Separan AP30 | 0.2<br>0.002 | 51 | 10 | 0.38 | 4.03 | 3.60 |
| do. | Aerofloat 208<br>Superfloc 127 | 0.2<br>0.002 | 46 | 10 | 0.38 | 3.63 | 3.24 |
| do. | Aerofloat 208<br>Superfloc 127 | 0.2<br>0.003 | 41 | 10 | 0.38 | 3.24 | 2.89 |

## TABLE 11: SETTLING TEST COMPRESSION ZONE DATA

| Sample | Flocculant Dosage lb/ton | Pulp in Compression Zone | | | Time in Compression Hours |
|---|---|---|---|---|---|
| | | Dilution (parts water:1 part solid | | Specific Gravity Average | |
| | | Average | Final | | |
| B4, - 72 mesh | 0.0125 (a) | 0.639 | 0.503 | 1.62 | 0.687 |
| B4, - 150 " | 0.0025 (a) | 0.653 | 0.585 | 1.62 | 3.42 |
| H11, - 72 mesh | 0.0025 (a) | 0.587 | 0.410 | 1.66 | 1.17 |
| H11, - 150 " | 0.0025 (a) | 0.552 | 0.473 | 1.68 | 1.70 |
| C4, - 72 mesh | 0.003 (b) | 0.636 | 0.525 | 1.63 | 1.38 |
| C4, - 150 " | 0.003 (b) | 0.648 | 0.549 | 1.62 | 1.97 |

(a) Separan AP30, plus phosphorus pentasulphide and Aerofloat 208
(b) Superfloc 127, plus Aerofloat 208

$\frac{\text{water}}{\text{solids}}$ .53

TABLE 12: VARIATION OF FILTER FORM TIME

| Sample | Form Time sec. | Drain Time sec. | Cake Thickness approx. ins. | Cake Weight g Wet | Cake Weight g Dry | Cake Moisture % (wet basis) | Filtrate ml | Filtrate gal/sq.ft/hr | Dry Cake Filtratio Rate lb/sq.ft/hr |
|---|---|---|---|---|---|---|---|---|---|
| B4, -72 mesh | 10 | 30 | 1/8 | 73.0 | 58.4 | 20.1 | 20.0 | 19.0 | 463 |
| | 15 | 45 | 3/16 | 100.0 | 80.5 | 19.5 | 40.0 | 25.4 | 426 |
| | 20 | 60 | 1/4 | 128.5 | 101.3 | 21.3 | 44.0 | 21.0 | 402 |
| | 30 | 90 | 5/16 | 135.4 | 111.1 | 17.9 | 46.0 | 14.6 | 293 |
| | 30 (a) | 90 | 1/2 | 257.5 | 206.8 | 19.8 | 78.0 | 24.8 | 547 |
| B4, -150 mesh | 10 | 30 | 3/16 | 81.7 | 64.4 | 23.6 | 32.0 | 30.5 | 511 |
| | 15 | 45 | 1/4 | 84.6 | 67.7 | 22.7 | 35.0 | 21.9 | 360 |
| | 20 | 60 | 9/32 | 95.2 | 74.5 | 21.8 | 38.0 | 18.1 | 295 |
| | 30 | 90 | 5/16 | 120.2 | 95.5 | 20.6 | 45.0 | 14.3 | 253 |
| H11, -72(b) mesh | 30 | 90 | 5/16 | 390.2 | 320.4 | 16.3 | 130.0 | 37.9 | 863 |
| H11, -150 mesh | 10 | 30 | 1/2 | 186.5 | 151.0 | 19.0 | 55.0 | 52.4 | 1200 |
| | 15 | 45 | 9/16 | 220.6 | 173.3 | 21.4 | 58.0 | 36.8 | 916 |
| | 20 | 60 | 3/4 | 257.9 | 206.0 | 20.1 | 72.0 | 34.2 | 819 |
| | 30 | 90 | 7/8 | 348.0 | 285.5 | 18.0 | 116.0 | 34.8 | 753 |
| C4, -72 mesh | 10 | 30 | 1/8 | 81.3 | 65.7 | 19.1 | 22.0 | 20.9 | 525 |
| | 15 | 45 | 3/16 | 95.7 | 78.4 | 18.1 | 28.0 | 17.8 | 414 |
| | 20 | 60 | 3/16 | 103.4 | 84.8 | 18.0 | 31.0 | 14.8 | 337 |
| | 30 | 90 | 3/16 | 112.4 | 91.8 | 18.4 | 33.0 | 10.5 | 243 |
| | 30 (a) | 90 | 3/8 | 168.2 | 138.0 | 18.0 | 70.0 | 22.3 | 365 |
| C4, -150 mesh | 10 | 30 | 1/8 | 66.3 | 53.7 | 19.0 | 35.0 | 34.3 | 426 |
| | 15 | 45 | 1/8 | 75.1 | 61.6 | 18.0 | 43.0 | 26.6 | 326 |
| | 20 | 60 | 1/4 | 118.5 | 95.1 | 19.8 | 55.0 | 26.2 | 378 |
| | 30 | 90 | 5/16 | 134.4 | 109.0 | 18.9 | 62.0 | 19.7 | 288 |

(a) Using clean, washed filter cloth
(b) Insufficient sample to complete test series

## TABLE 13: VARIATION OF FILTER DRAIN TIME

| Sample | Form time sec. | Drain time sec. | Cake Weight g | | Cake Moisture % | Filtration Rate lb/sq.ft/hr |
|---|---|---|---|---|---|---|
| | | | Wet | Dry | | |
| B4, - 72 mesh | 30 | 90 | 135.4 | 111.1 | 17.9 | 293 |
| | 30 | 150 | 132.8 | 112.7 | 15.2 | 297 |
| B4, - 150 " | 30 | 90 | 120.2 | 95.5 | 20.6 | 253 |
| | 30 | 120 | 126.4 | 99.7 | 21.0 | 264 |
| | 30 | 150 | 122.7 | 97.9 | 20.2 | 258 |
| H11, - 72 " | 30 | 90 | 390.2 | 326.4 | 16.3 | 863 |
| | 30 | 150 | 381.4 | 324.1 | 15.0 | 859 |
| | 30 | 180 | 328.8 | 282.6 | 14.1 | 746 |
| H11, - 150 " | 30 | 90 | 348.0 | 285.5 | 18.0 | 753 |
| C4, - 72 " | 30 | 90 | 112.4 | 91.8 | 18.4 | 243 |
| | 30 | 150 | 116.7 | 97.4 | 15.7 | 257 |
| C4, - 150 " | 30 | 90 | 134.4 | 109.0 | 18.9 | 288 |
| | 30 | 120 | 125.0 | 103.9 | 16.9 | 274 |
| | 30 | 150 | 122.2 | 102.2 | 16.8 | 278 |
| | 30 | 180 | 110.8 | 93.2 | 15.9 | 246 |

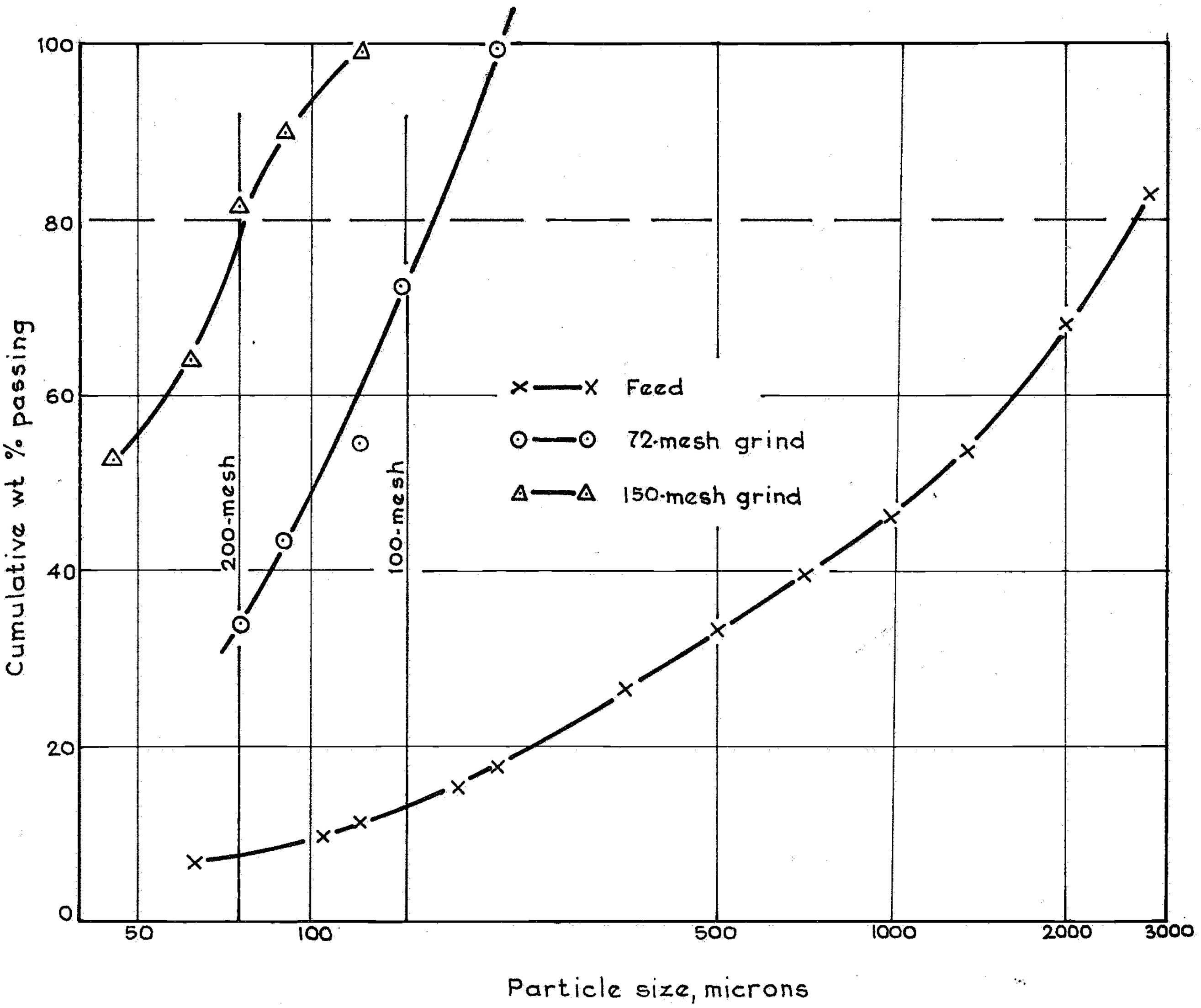

FIG.-I: GRINDABILITY OF GUNSON ORE SAMPLE B4

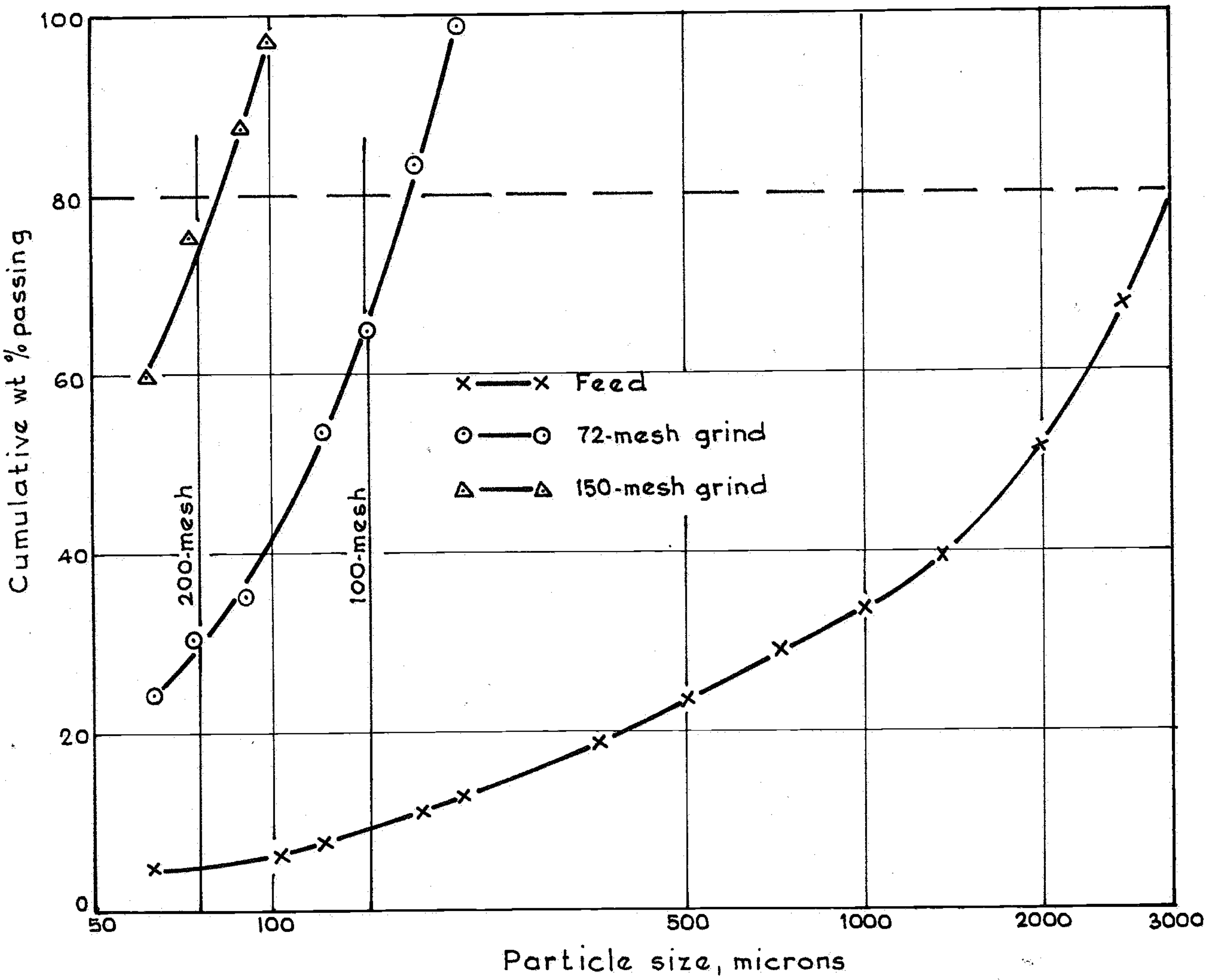

FIG.-2: GRINDABILITY OF GUNSON ORE SAMPLE H 11

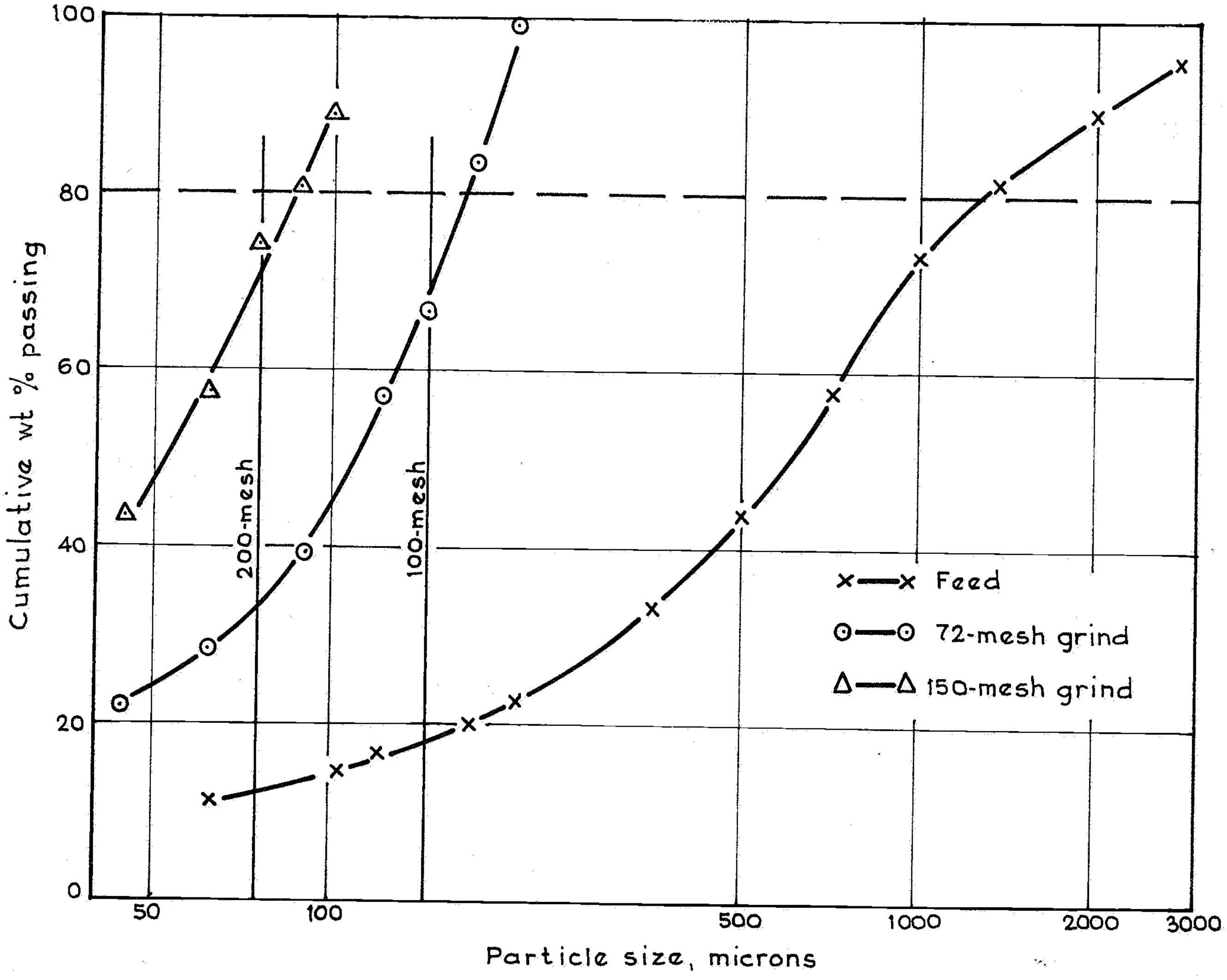

FIG.-3: GRINDABILITY OF GUNSON ORE SAMPLE C4

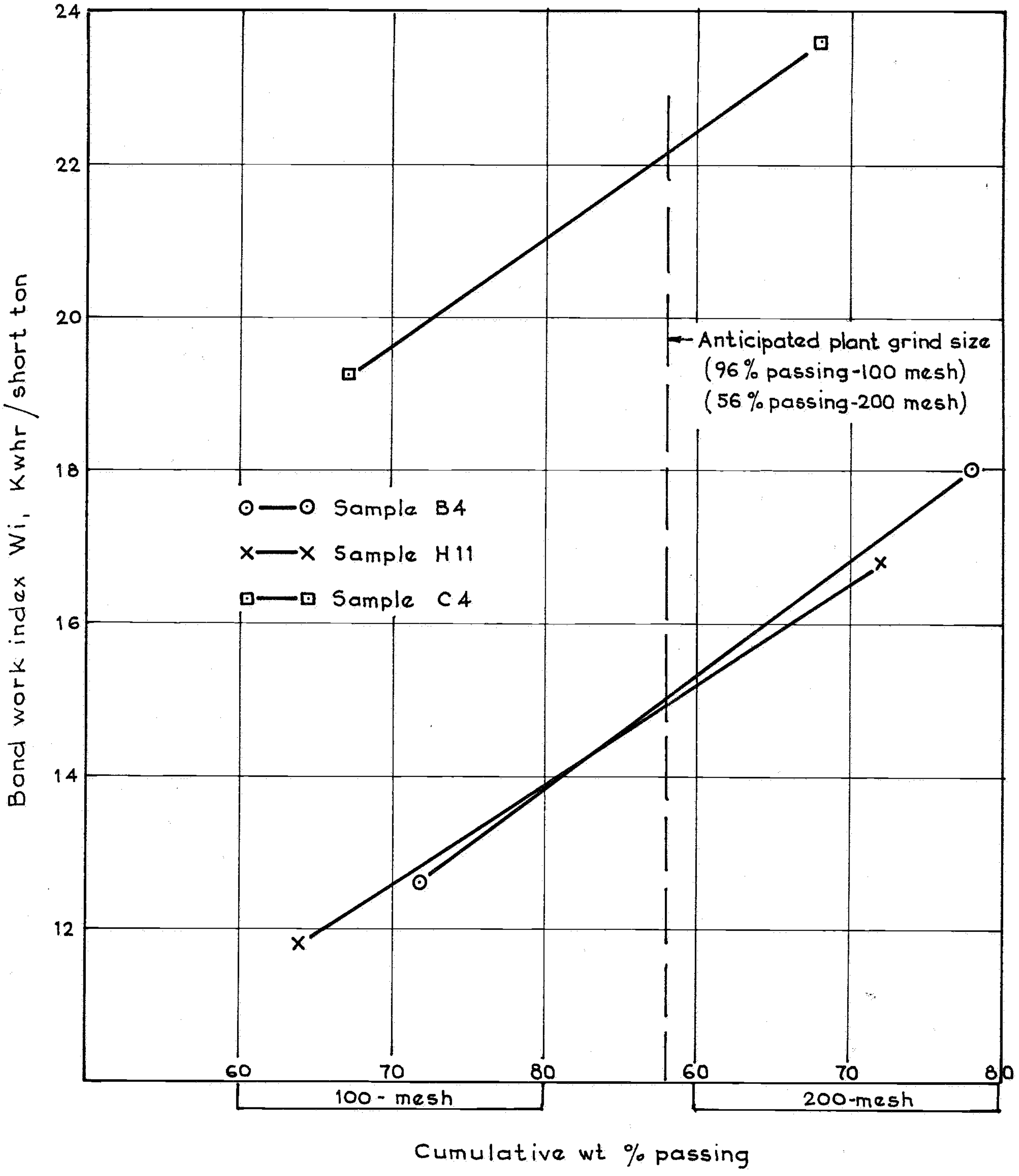

FIG.- 4: SUMMARY OF GRINDABILITY TESTS ON GUNSON ORES

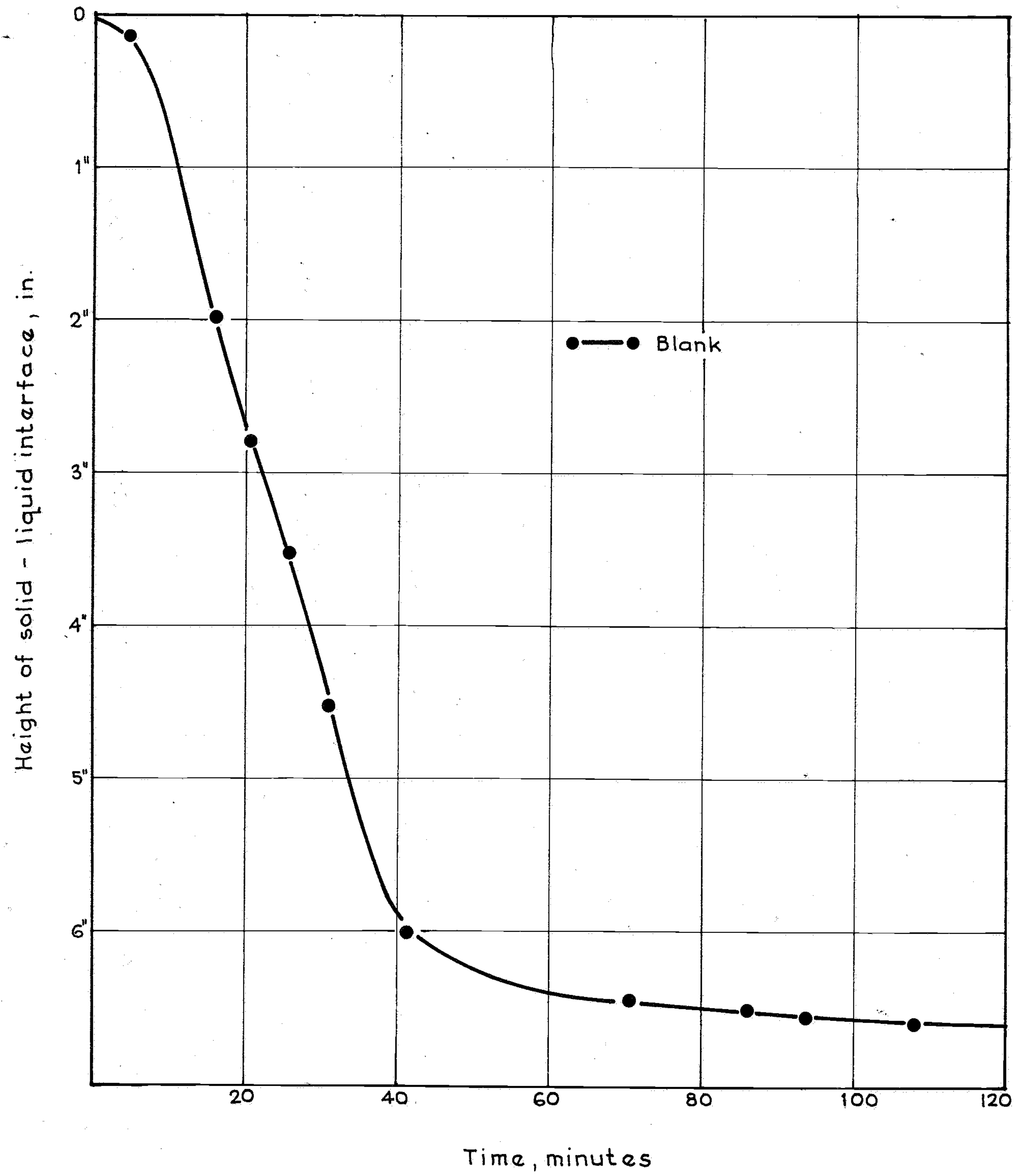

FIG.-5A: FREE SETTLING TESTS. SAMPLE B4 MINUS 72 MESH

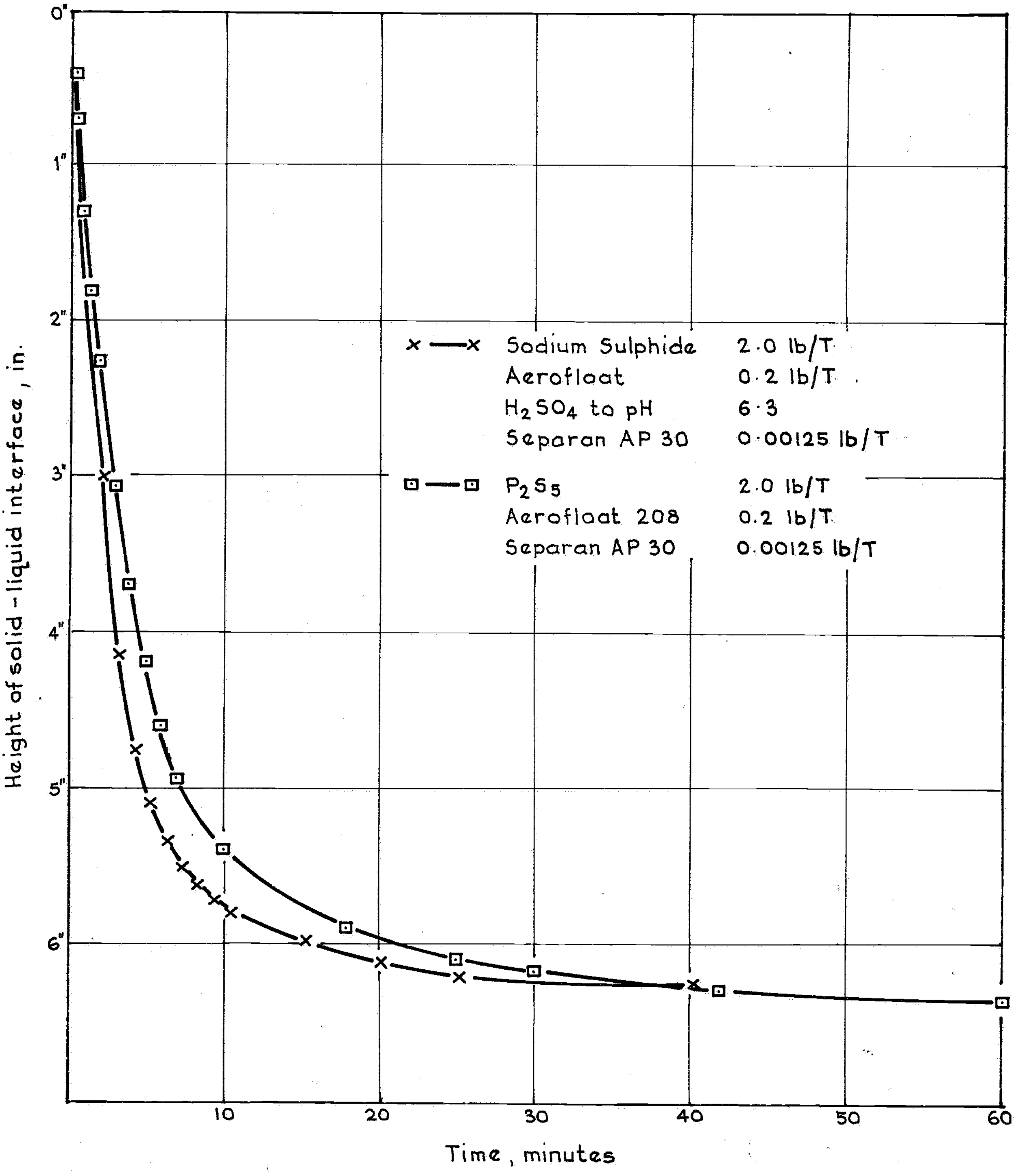

FIG.-5B: FREE SETTLING TESTS. SAMPLE B4 MINUS 72-MESH

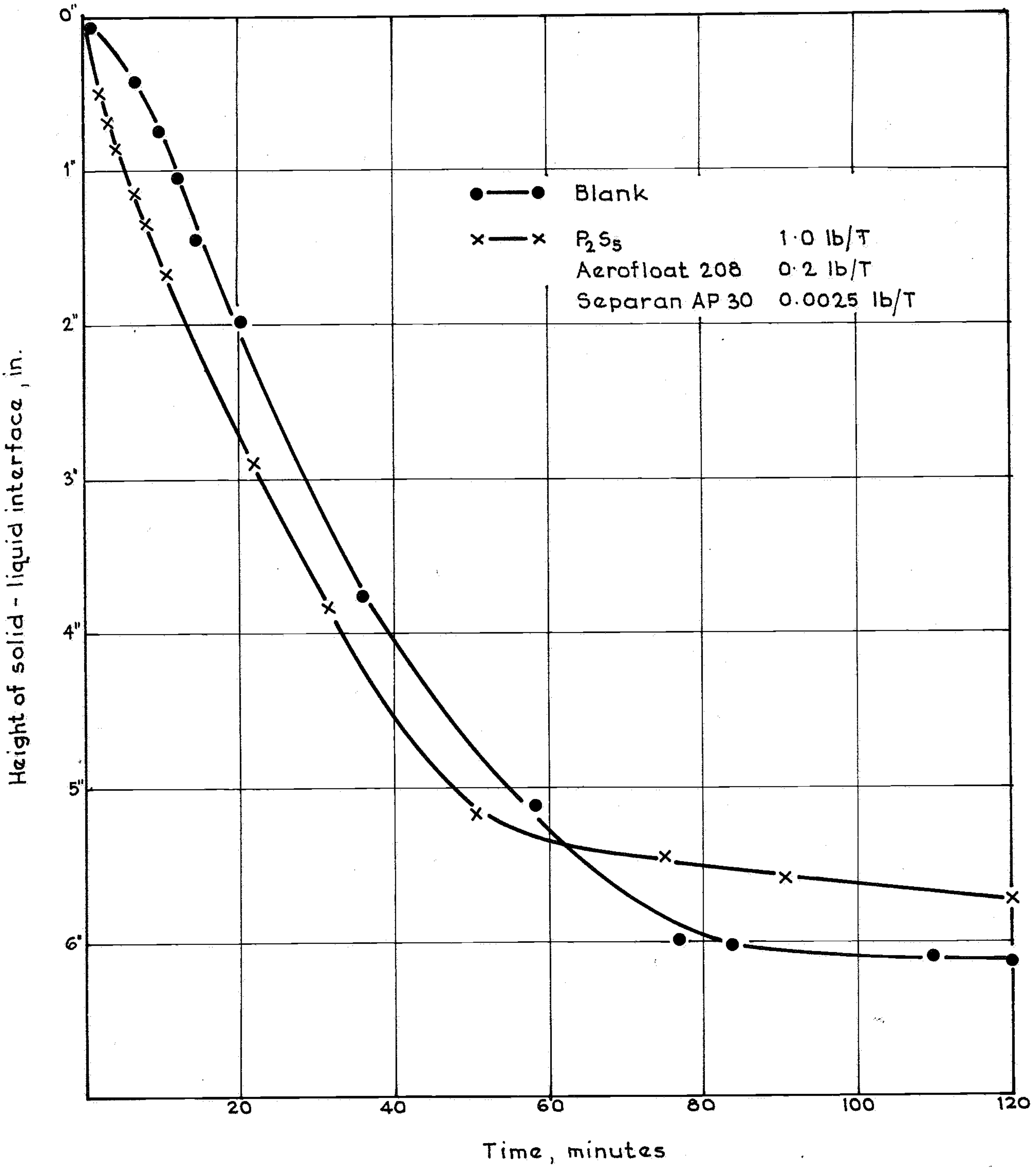

FIG.-6A: FREE SETTLING TESTS. SAMPLE B4 MINUS 150-MESH

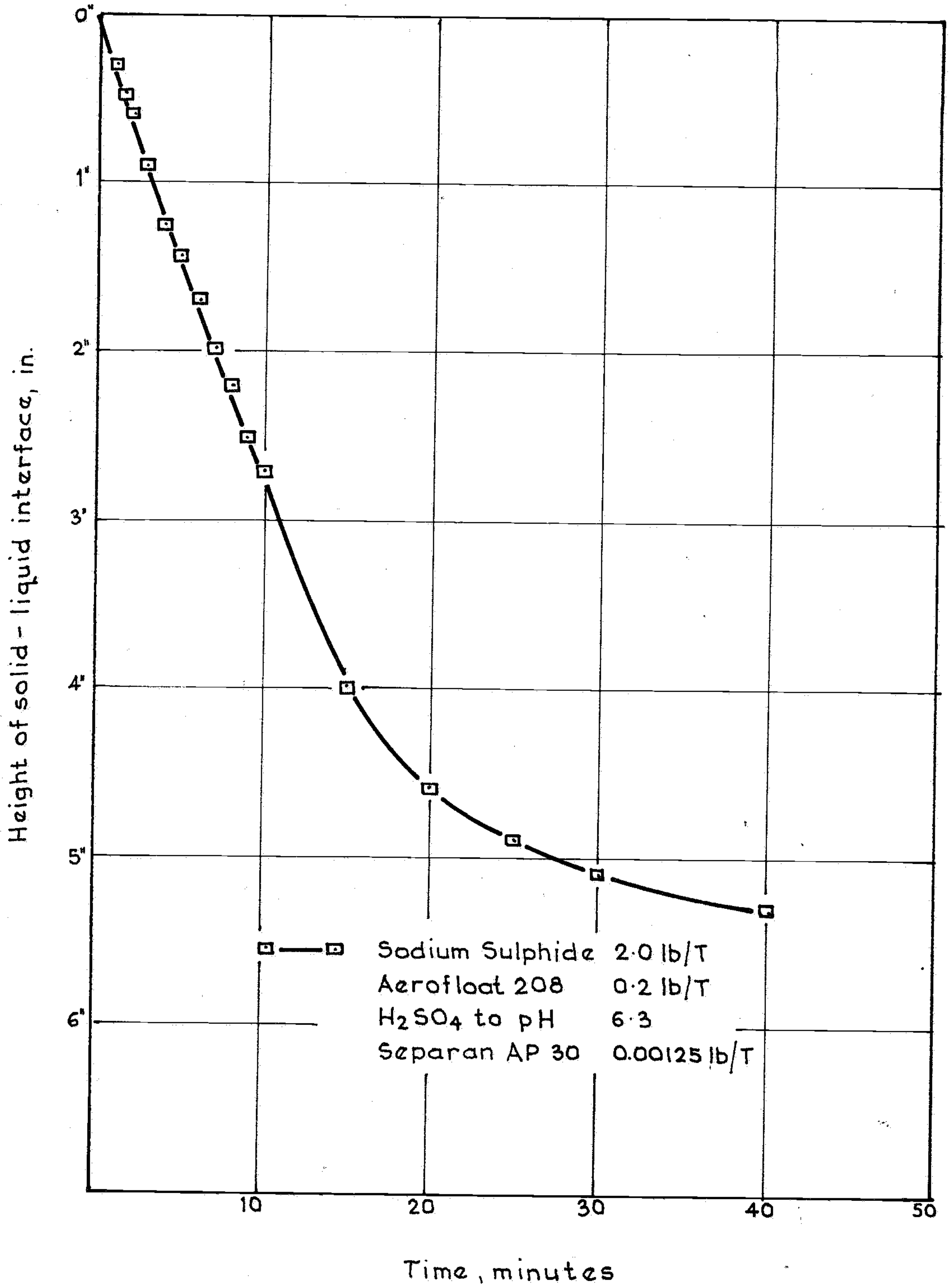

FIG.-6B: FREE SETTLING TESTS.
SAMPLE B4 MINUS 150-MESH

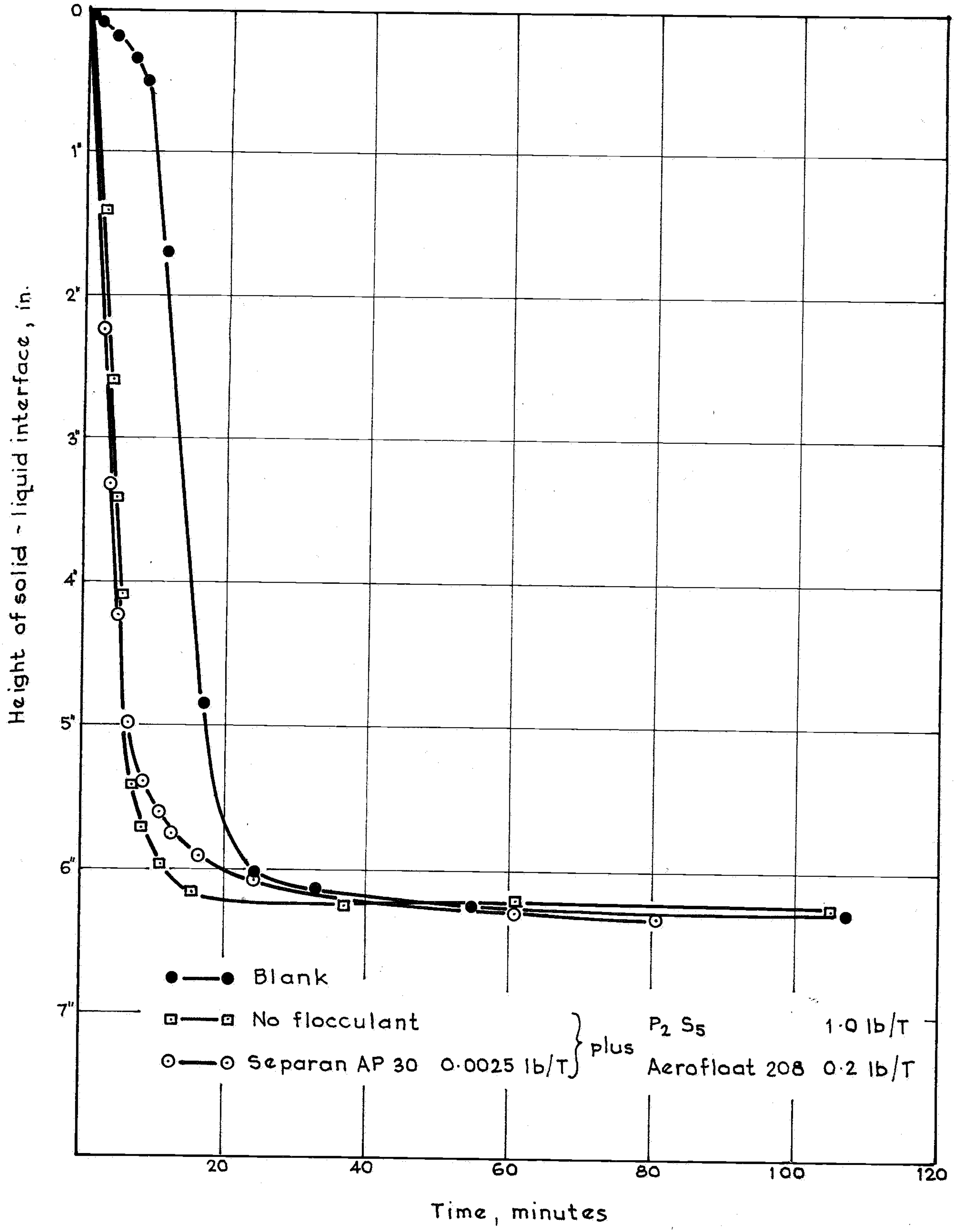

FIG.-7A: FREE SETTLING TESTS. SAMPLE H 11 MINUS 72-MESH

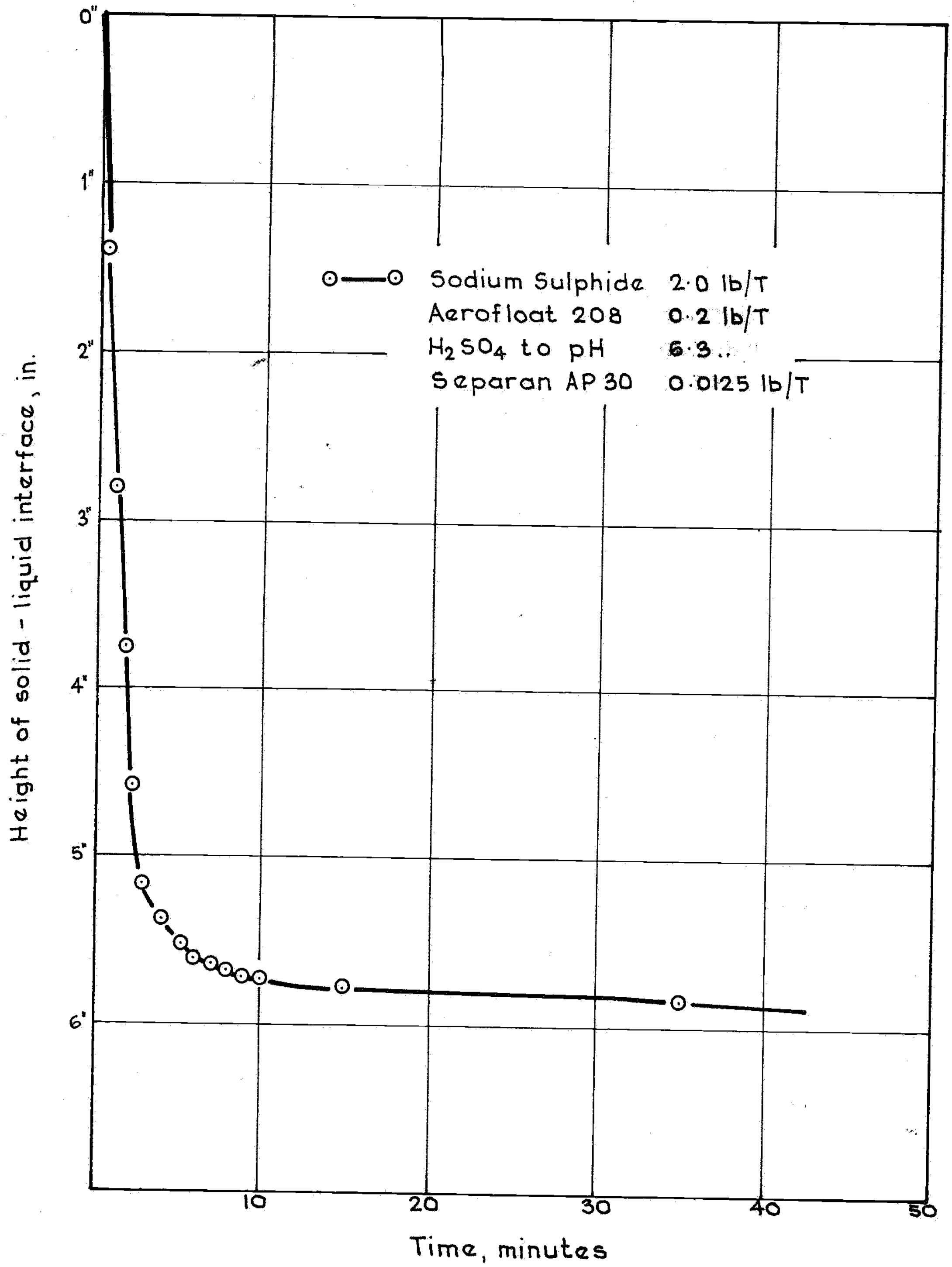

FIG.-7B: FREE SETTLING TESTS.
SAMPLE H11 MINUS 72-MESH

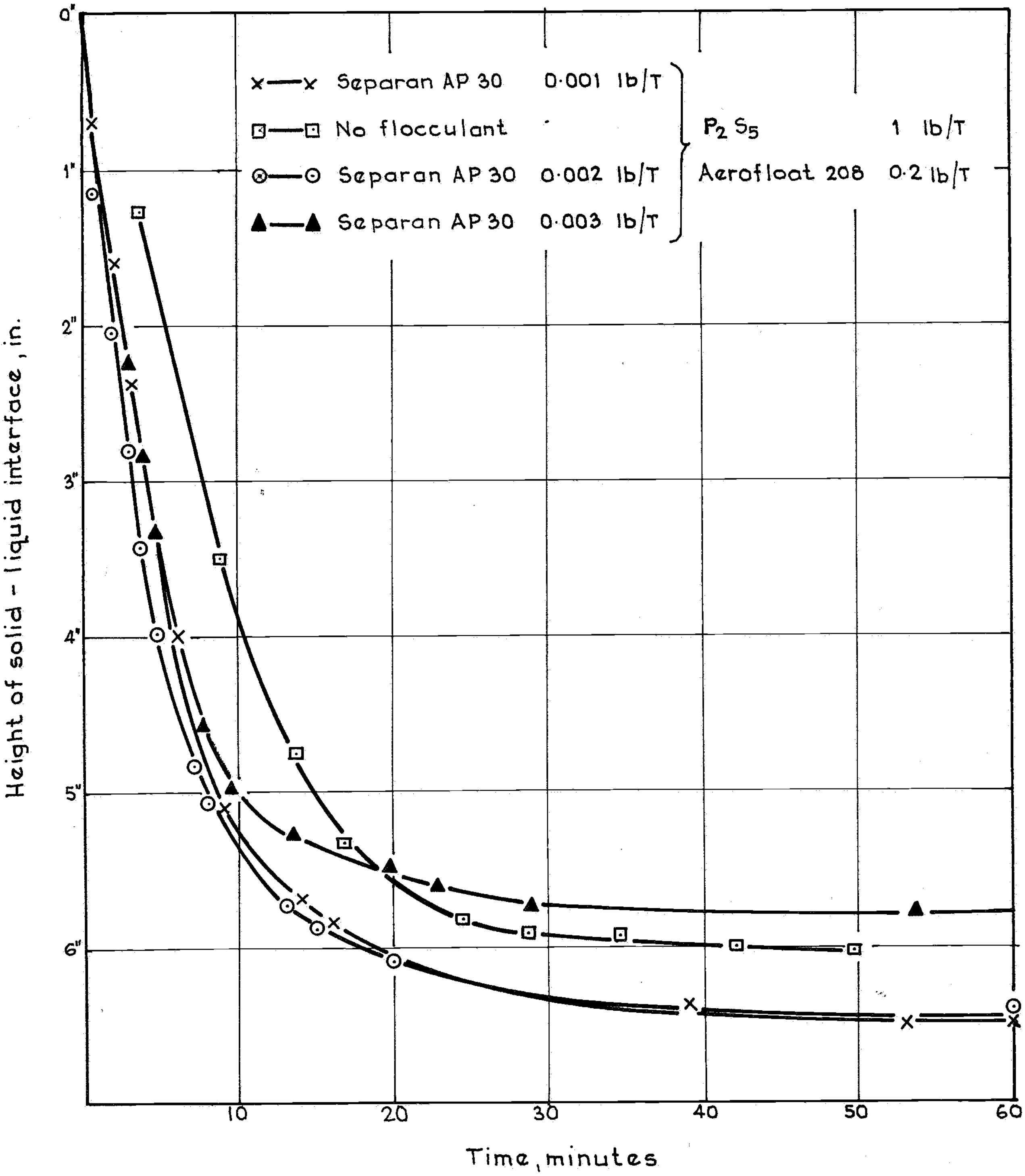

FIG.-8: FREE SETTLING TESTS. SAMPLE H11 MINUS 150-MESH

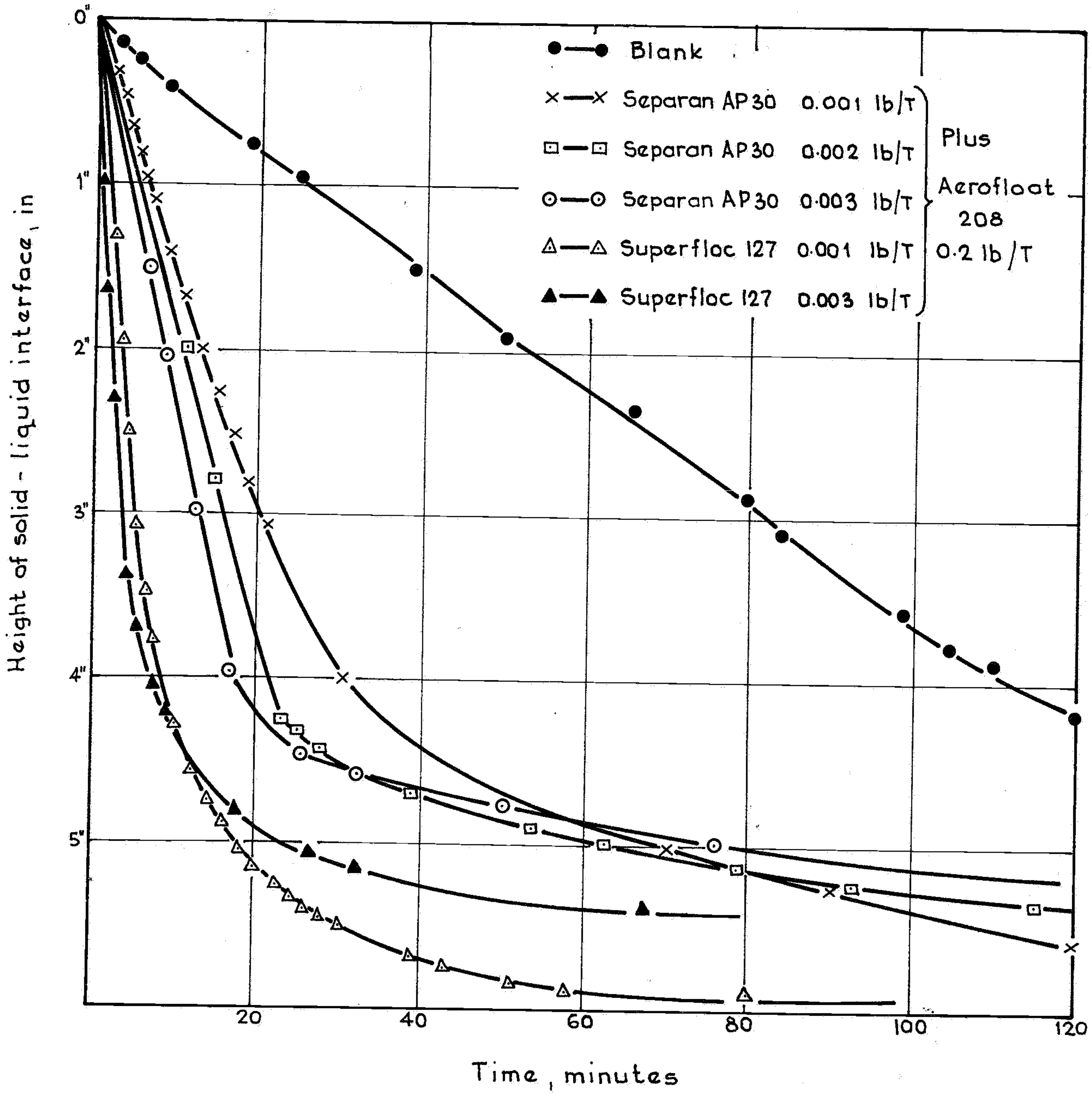

FIG.-9: FREE SETTLING TESTS. SAMPLE C4 MINUS 72-MESH

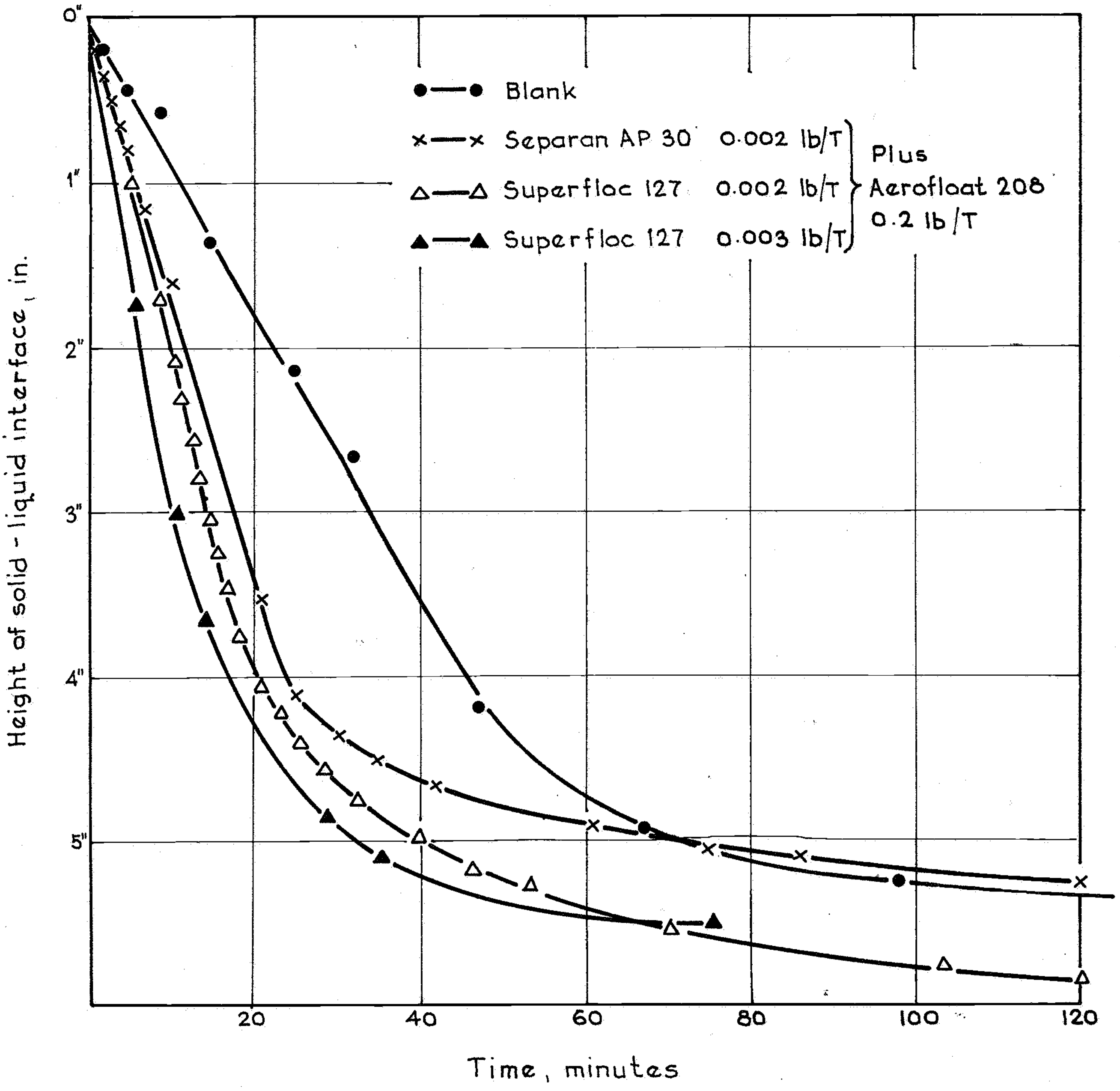

FIG.-10: FREE SETTLING TESTS. SAMPLE C4 MINUS 150-MESH

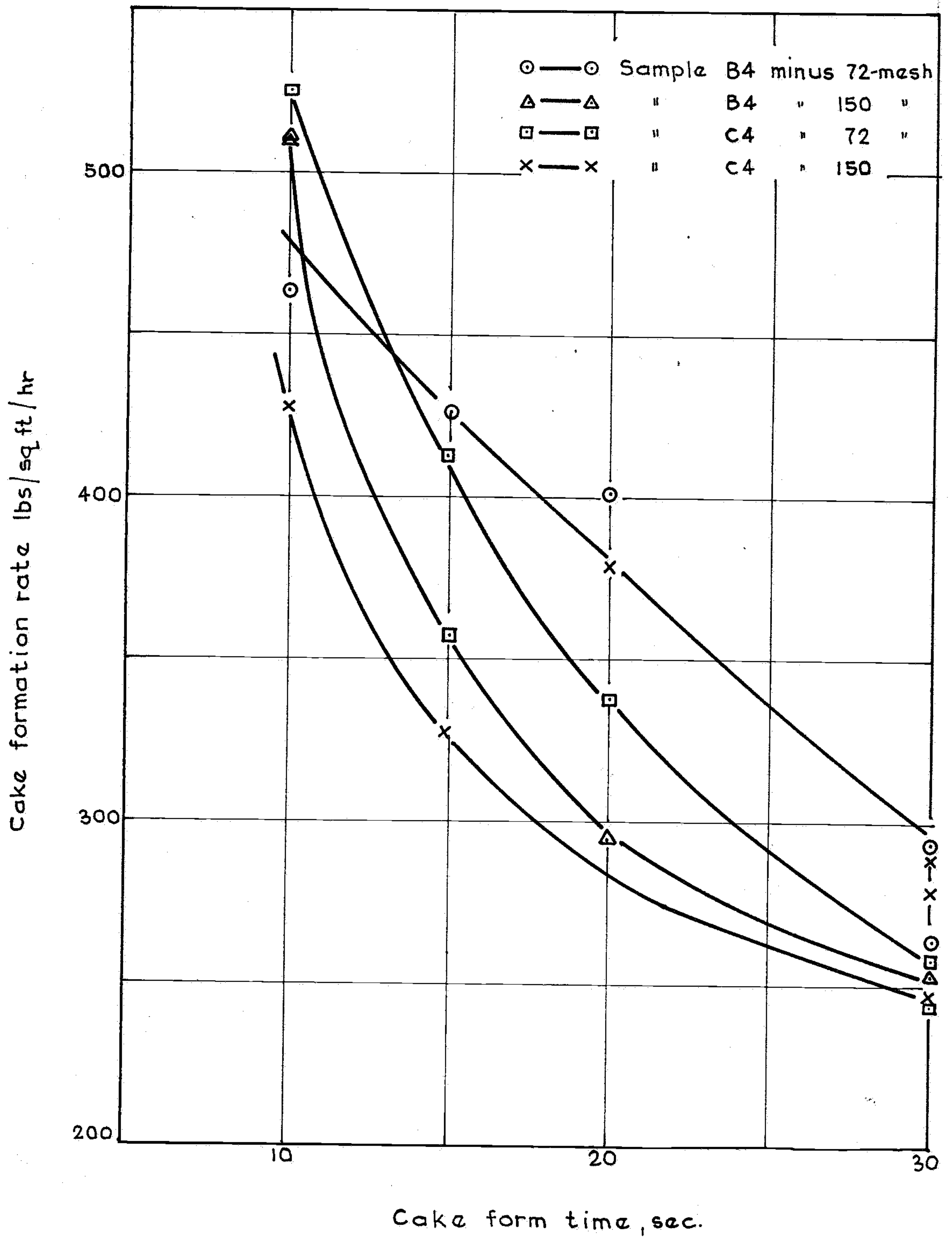

FIG. 11A: FILTRATION RATE OF GUNSON ORE

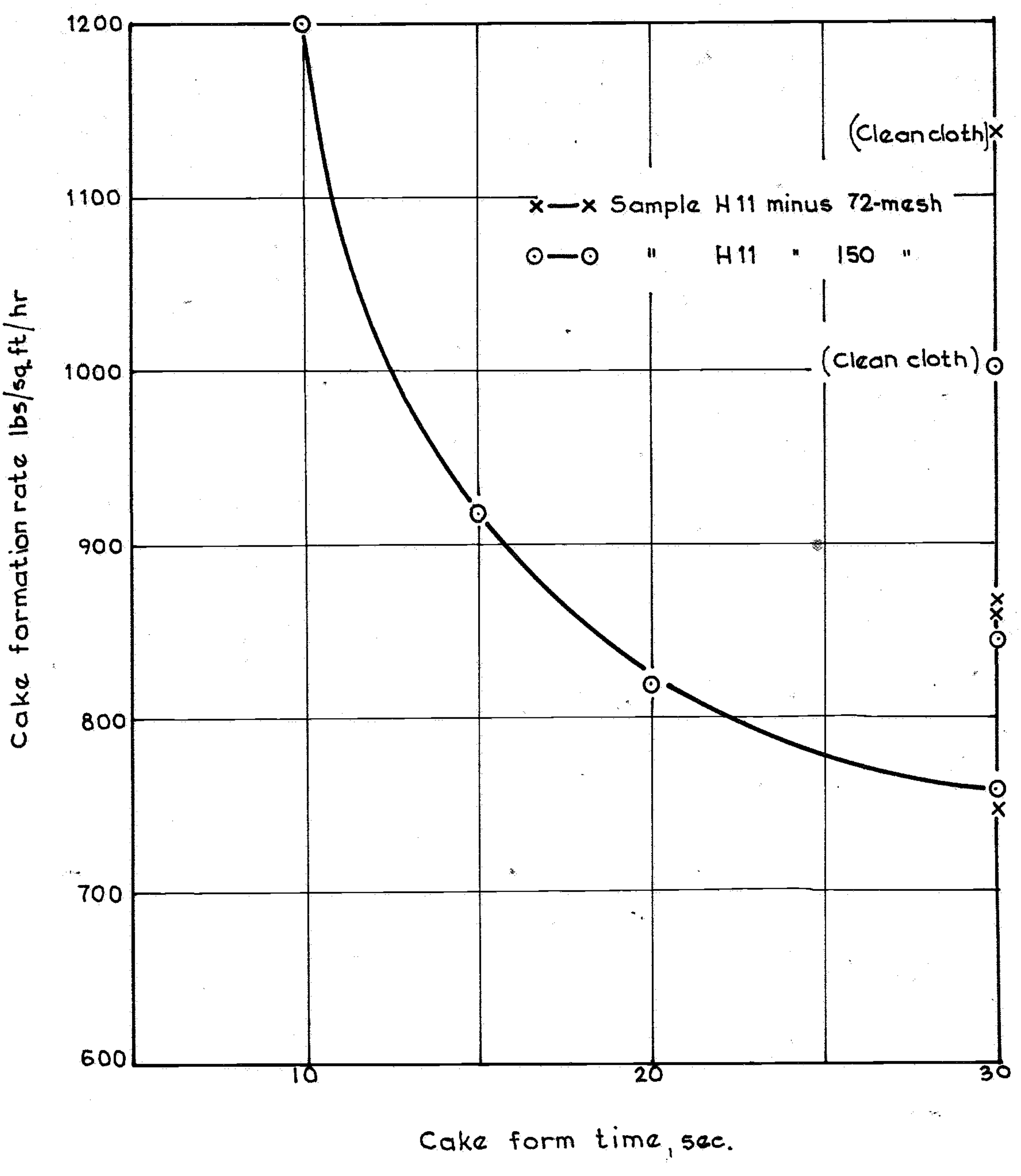

FIG-11B: FILTRATION RATE OF GUNSON ORE

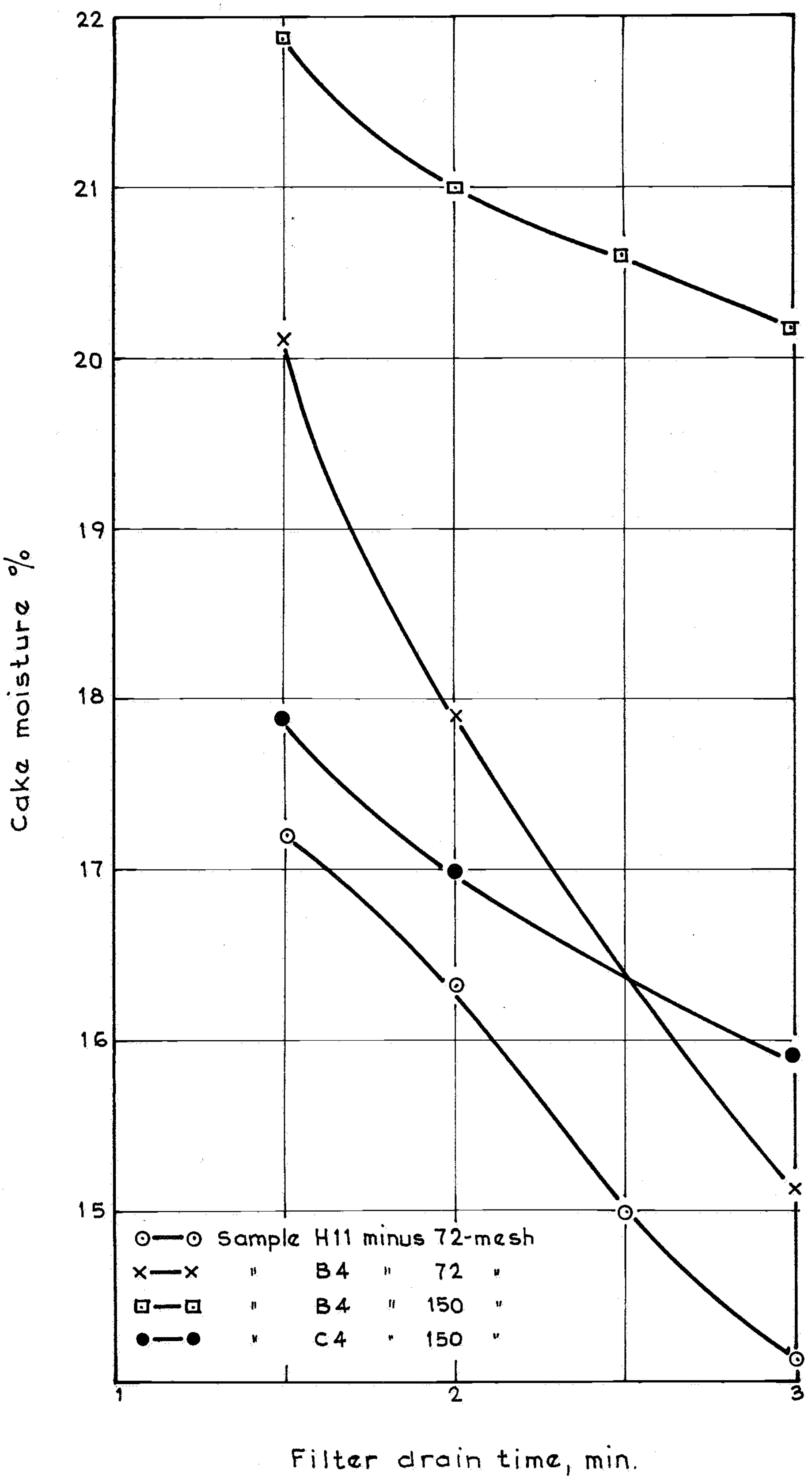

FIG.-12: MOISTURE CONTENT OF FILTER CAKES

RESEARCH FOR INDUSTRY

**THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES**

CONYNGHAM STREET - FREWVILLE - SOUTH AUSTRALIA
TELEPHONE 791662 · TELEGRAMS 'AMDEL' ADELAIDE

*Please quote this reference in your reply:*

ML 3/262/0

3rd October, 1967

*Your reference:*

Mr J.B. Evans,
Managing Director,
Austminex Pty Ltd,
Suite 20,
67 Queen's Road,
MELBOURNE, Vic, 3000.

## COMPLETED REPORT ML 3276-67

| | |
|---|---|
| YOUR REFERENCE: | Application dated 16/6/67 |
| MATERIAL: | Copper ore |
| LOCALITY: | Mt Gunson, S.A. |
| IDENTIFICATION: | Sample C4 (2nd Sample) |
| DATE RECEIVED: | 10/8/67 |
| WORK REQUIRED: | Grindability testing |

Investigation and Report by: L.J. Weir, J. Hofer

Officer in Charge, Metallurgy Section: P.K. Hosking

P.A. Young
Director.

# GRINDABILITY OF MT GUNSON ORE

## 1. INTRODUCTION

An earlier part report gave detailed results of grindability, thickening and filtration tests on 3 ore-samples from the Mt Gunson area.

Because of unexpectedly high grinding work indices on the sulphide-bearing ore, two further samples were submitted for confirmatory grindability, thickening and filter tests.

## 2. MATERIAL EXAMINED

| Ore | Locality |
|---|---|
| C4 (2nd sample) | Not stated |
| S | Grab sample from material trenched from lagoon after dozing to 12 feet and blasting 10 feet. |

## 3. EQUIPMENT

As in part report ML 3276-67.

## 4. ANCILLARY MATERIALS

As in part report ML 3276-67.

## 5. EXPERIMENTAL PROCEDURE AND RESULTS

### 5.1 Sample Preparation

The ore samples received were rolls-crushed in stages in closed circuit with a 5 mesh screen. Crushed products were thoroughly mixed and riffled into aliquots for grindability tests.

### 5.2 Grindability Tests

Grindabilities of the ores were determined at 72 and 150 mesh, according to the method of Bond as previously described.

The results of grindability tests are shown in Tables 1 to 4 and data derived from the tests, and calculated work indices are shown in Table 5. Sizings of grindability feed and products are shown graphically in Figure 1. A graphical interpolation of work-index for grinding in the anticipated flotation plant is shown in Figure 2.

Interpolation figures are:-

| Ore Sample | Plant Work Index, Wi kWh/short ton |
|---|---|
| C4 (2nd Sample) | 16.7 |
| S | 19.1 |

Mill power input can then be calculated from the expression

$$W = 10Wi \left( \frac{1}{P} - \frac{1}{F} \right)$$

Thus for an assumed plant feed, 80% of which passes 12000 microns, and aiming for a product size of 80% passing 105 microns,

For sample C4, (2nd sample)

$$W = 10 \times 16.7 \times \left( \frac{1}{105} - \frac{1}{12000} \right)$$

$$= 14.79 \text{ kWh/short ton}$$

For sample S,

$$W = 10 \times 19.1 \times \left( \frac{1}{105} - \frac{1}{12000} \right)$$

$$= 16.90 \text{ kWh/short ton}$$

Horsepower required is given by

$$\text{Installed HP} = \left( W \times \frac{\text{tons/day}}{18} \right) + 10\%$$

For Sample C4, (2nd Sample)

$$\text{Installed HP} = \frac{(14.79 \times 1000)}{18} + 10\%$$

$$= 902 \text{ HP}$$

For Sample S,

$$\text{Installed HP} = \left( 16.90 \times \frac{1000}{18} \right) + 10\%$$

$$= 1034 \text{ HP}$$

## 5.3 Settling Tests

Products from the latter stages of grindability tests at 72 and 150 mesh were combined in equal proportions to produce material of approximately the sizing of anticipated flotation plant tailing. Portions were diluted to 30% solids in mains water to give pulp samples of 500 ml for settling tests.

Free settling tests were done on pulp samples containing flotation reagents and flocculant, and settling curves drawn as previously described. Results of the tests are shown in Figures 3 and 4.

Calculated settling area requirements for the tests are shown in Table 6.

Calculated thickener tank diameters are shown in Appendix A.

Settling tests under raked conditions for determination of thickener depth requirements were not carried out.

## 5.4 Filter Tests

Samples from combined grindability products were treated with flotation reagents and flocculant addition as indicated by settling tests, and thickened by settling and decantation. Filter feed pulp conditions were as shown:-

| | Solids % | Reagent | lb/ton |
|---|---|---|---|
| Sample C4 (2nd Sample) | Approx. 62 | Phosphorus pentasulphide | 2.0 |
| | | Aerofloat 208 | 0.2 |
| | | Separan AP30 | 0.003 |
| Sample S | Approx. 57 | Phosphorus pentasulphide | 2.0 |
| | | Aerofloat 208 | 0.2 |
| | | Separan AP30 | 0.015 |

Filtration tests were done as previously described, using a 9 oz multi-weave polypropylene cloth. A filter form time of 30 seconds was adopted for the tests, and a range of drain times were used.

Excellent cakes were formed with Sample C4 (2nd Sample), but very poor filtration was obtained on Sample S. Results of the tests are shown in Table 7.

Calculations for plant filter area requirements are shown in Appendix B.

## 6. DISCUSSION

Grindability tests, settling tests on ground slurry and vacuum filter tests on thickened ground slurry gave results, for two samples submitted, which are at wide variance with those previously obtained on a sulphide ore from the Mt Gunson area.

# APPENDIX A

## CALCULATION OF THICKENER TANK DIAMETERS

### Sample C4 (2nd Sample)

From Table 6, settling area requirement for flocculated pulp = 1,73 sq ft/short ton/day

For 1000 ton/day, tank diameter = 46,8 ft
Allow 20% safety factor 9,7 ft
TANK DIAMETER 56,5 ft

### Sample S

From Table 6, settling area requirement for flocculated pulp = 2,36 ft/ short ton/day

For 1000 ton/day, tank diameter = 54,8 ft
Allow 20% safety factor 11,0 ft
TANK DIAMETER 65,8 ft

# APPENDIX B

## CALCULATION OF FILTER AREA REQUIREMENTS

**Sample C4 (2nd sample)**

From Table 7, filtration rates for a 30 second form time are:-

|  | Dry Cake Weight g/1/10th sq ft |
|---|---|
|  | 271.2 |
|  | 208.1 |
|  | 215.0 |
| mean | 231.4 |

For a 2-minute filter cycle (30 sec form, 60 sec, drain and 30 sec discharge)

Filtration rate $= \frac{231}{454} \times 10 \times \frac{60}{2}$

$= 152.9$ lb/sq ft/hr

$= 1.83$ short tons/sq ft/day

Allowing a scale up factor of 0.65

$= 1.19$ tons/sq ft/day

For 1000 tons/day, filter area required
$= 840$ sq ft.

**Sample S**

From Table 7, filtration rates for a 30 second form time are:-

|  | Dry Cake Weight g/1/10th sq ft |
|---|---|
|  | 98.0 |
|  | 91.0 |
|  | 77.0 |
| mean | 88.7 |

For a 2-minute filter cycle (30 sec form, 60 sec drain and 30 sec discharge)

Filtration rate $= \frac{88.7}{454} \times 10 \times \frac{60}{2}$

$= 58.6$ lbs/sq ft/hr

$= 0.704$ short tons/sq ft/day

Allowing a scale up factor of 0.65
$= 0.457$ tons/sq ft/day

For 1000 tons/day, filter area required
$= 2190$ sq ft.

## TABLE 1

## AMDEL

## GRINDABILITY RESULTS

Project No. ________

Sample C4 (2nd sample)

Test by ________

Date ________

Grindability at 72 mesh

Finished material in feed (F) = 14.0%

| Stage | A<br>Wt. of 700 feed (g) | $A_1$<br>Wt. of new feed (g) | Y<br>mesh in feed (g) | X<br>Gross Product for 250% circ.load | R<br>No. of Grinding revs | B<br>Screen Over-size (g) | C<br>Gross Product (g) | D<br>Net Product (g) | G<br>Grind-ability g/rev. | L<br>Circul-ating load % |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | 1145 | / | 150 | | 125 | 749 | 396 | 236 | 1.89 | |
| 2 | 1096 | 347 | 48.6 | 300 | 133 | 796.0 | 300 | 251.4 | 1.89 | 265 |
| 3 | 1101 | 305 | 42.6 | 318 | 140 | 780.4 | 320 | 278.0 | 1.98 | 244 |
| 4 | 1098.8 | 318.4 | 44.5 | 312 | 134 | 790.8 | 308 | 263.5 | 1.97 | 256 |
| 5 | 1097.0 | 306.2 | 42.8 | 316 | 138 | 783.0 | 314 | 271.2 | 1.97 | 249 |

Unit Volume 700 ml = g in mill equivalent to lb/cubic foot of broken ore.

Average of Stages 3, 4 & 5 inclusive - 250% circ. load, 1.97 g/rev.

$A_1 = A - B_p$

$Y = \frac{A_1 \times F}{100}$

$X = \frac{100 B_p}{250}$

$R = \frac{X - Y}{G}$

$C = A - B$

$D = C - Y$

$G = \frac{D}{R}$

$L = \frac{100 B}{C}$

Note $B_p$ = B from previous grinding stage

# TABLE 2

## AMDEL

## GRINDABILITY RESULTS

Project No. ____________

Sample C4 (2nd Sample)

Test by ____________

Date ____________

Grindability at 150 mesh

Finished material in feed (F) = 8.7%

| Stage | A<br>Wt. of 700 feed (g) | $A_1$<br>Wt. of new feed (g) | Y<br>mesh in feed (g) | X<br>Gross Product for 250% circ. load | R<br>No. of Grinding revs<br>Calc | <br><br>Act. | B<br>Screen Over-size (g) | C<br>Gross Product (g) | D<br>Net Product (g) | G<br>Grind-ability g/rev. | L<br>Circul-ating load % |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | 1144.8 | | 80 | | 315 | | 828 | 316.8 | 236.8 | 0.75 | |
| 2 | 1092.0 | 264 | 18.5 | 330 | 415 | (380) | 778 | 314.0 | 295.5 | 0.78 | 247 |
| 3 | 1094.6 | 316 | 22.1 | 311 | 370 | (375) | 763.2 | 331.4 | 319.3 | 0.85 | 230 |
| 4 | 1066.2 | 303 | 21.2 | 305 | 334 | | 770.0 | 296.2 | 275.0 | 0.82 | 260 |
| 5 | 1079.2 | 309 | 21.6 | 307 | 348 | | 743.0 | 336.2 | 314.6 | 0.90 | 221 |
| 6 | 1076.0 | 333 | 23.3 | 297 | 304 | (320) | 770.0 | 306.0 | 282.7 | 0.88 | 251 |
| 7 | 1079.0 | 309 | 21.6 | 307 | 324 | | 771.0 | 308.0 | 286.4 | 0.88 | 250 |

**Unit Volume** 700 ml = g in mill equivalent to lb/cubic foot of broken ore.

**Average of Stages** 6 & 7 inclusive - 250% circ. load, 0.88 g/rev.

$A_1 = A - B_p$

$Y = \frac{A_1 \times F}{100}$

$X = \frac{100\ B_p}{250}$

$R = \frac{X - Y}{G}$

$C = A - B$

$D = C - Y$

$G = \frac{D}{R}$

$L = \frac{100\ B}{C}$

**Note** $B_p$ = B from previous grinding stage

# TABLE 3

## AMDEL

## GRINDABILITY RESULTS

Project No. ____________

Sample S

Test by ____________

Date ____________

Grindability at 72 mesh

Finished material in feed (F) = 25.7%

| Stage | A<br>Wt. of 700 feed (g) | $A_1$<br>Wt. of new feed (g) | Y<br>mesh in feed (g) | X<br>Gross Product for 250% circ.load | R<br>No. of Grinding revs<br>Calc | <br><br>Actual | B<br>Screen Over-size (g) | C<br>Gross Product (g) | D<br>Net Product (g) | G<br>Grind-ability g/rev. | L<br>Circul-ating load % |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | 1147 | | 292 | | 120 | | 693.2 | 453.8 | 161.8 | 1.35 | |
| 2 | 1143.4 | 450.2 | 116 | 277 | 119 | | 825.0 | 318.4 | 202.4 | 1.70 | 259 |
| 3 | 1135.4 | 310.0 | 80 | 330 | 147 | (140) | 814.0 | 321.4 | 241.4 | 1.72 | 253 |
| 4 | 1140.5 | 326.0 | 84 | 325 | 140 | | 811.0 | 329.5 | 245.5 | 1.75 | 247 |
| 5 | 1131.4 | 320.6 | 82 | 324 | 138 | | 811.0 | 320.4 | 238.4 | 1.71 | 253 |
| 6 | 1140.4 | 329.4 | 85 | 324 | 140 | | 809.0 | 331.4 | 246.4 | 1.75 | 246 |

Unit Volume 700 ml = g in mill equivalent to lb/cubic foot of broken ore.

Average of Stages 3 - 6 inclusive - 250% circ. load, 1.73 g/rev.

$A_1 = A - B_p$

$Y = \frac{A_1 \times F}{100}$

$X = \frac{100\ B_p}{250}$

$R = \frac{X - Y}{G}$

$C = A - B$

$D = C - Y$

$G = \frac{D}{R}$

$L = \frac{100\ B}{C}$

Note $B_p$ = B from previous grinding stage

# TABLE 4

## AMDEL

## GRINDABILITY RESULTS

Project No. ____________

Sample S

Test by ____________

Date ____________

Grindability at 150 mesh

Finished material in feed (F) = 15.1%

| Stage | A<br>Wt. of 700 feed (g) | $A_1$<br>Wt. of new feed (g) | Y<br>mesh in feed (g) | X<br>Gross Product for 250% circ.load | R<br>No. of Grinding revs<br>Calc | R<br>Actual | B<br>Screen Over-size (g) | C<br>Gross Product (g) | D<br>Net Product (g) | G<br>Grind-ability g/rev. | L<br>Circul-ating load % |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | 1147.4 | | 173 | | 300 | | 803.2 | 394.2 | 141.2 | 0.47 | |
| 2 | 1082.4 | 219.2 | 42 | 320 | 590 | (400) | 755.0 | 327.4 | 285.0 | 0.71 | 232 |
| 3 | 1110.0 | 355 | 54 | 302 | 350 | | 811.0 | 299.0 | 245.0 | 0.70 | 271 |
| 4 | 1109.0 | 298 | 45 | 324 | 398 | | 776.0 | 333.0 | 288.0 | 0.72 | 233 |
| 5 | 1110 | 334 | 50 | 310 | 362 | (370) | 779.0 | 331.0 | 281.0 | 0.76 | 235 |
| 6 | 1111 | 332 | 50 | 312 | 344 | (350) | 795.0 | 316.0 | 266.0 | 0.76 | 251 |
| 7 | 1109.2 | 314 | 47.5 | 318 | 355 | | 795.0 | 314.0 | 266.5 | 0.75 | 253 |

Unit Volume 700 ml = g in mill equivalent to lb/cubic foot of broken ore.

Average of Stages 5-7 inclusive - 246% circ. load,
0.76 g/rev.

$A_1 = A - B_p$

$Y = \frac{A_1 \times F}{100}$

$X = \frac{100\ B_p}{250}$

$R = \frac{X - Y}{G}$

$C = A - B$

$D = C - Y$

$G = \frac{D}{R}$

$L = \frac{100\ B}{C}$

Note $B_p$ = B from previous grinding stage

## TABLE 5: SUMMARY OF GRINDABILITY TEST RESULTS

| | Sample | | | |
|---|---|---|---|---|
| | C4 (2nd Sample) mesh | | S mesh | |
| | 72 | 150 | 72 | 150 |
| $P_1$ | 210 | 105 | 210 | 105 |
| P | 181 | 81 | 178 | 74 |
| F | 2400 | 2400 | 1590 | 1590 |
| $G_{bp}$ | 1,97 | 0,88 | 1,73 | 0,76 |
| Wi | 13,8 | 18,6 | 16,6 | 21,0 |

## TABLE 6: SETTLING AREA REQUIREMENTS FOR FLOCCULATED PULPS

| Sample | Reagent | Addition lb/ton | Settling Time min (tu) | Pulp Height ins (Ho) | Pulp Concn g/ml (Co) | Calc Settling Area Requ, sq ft/ton/24 hr Long tons | Short tons |
|---|---|---|---|---|---|---|---|
| C4 (2nd Sample) | $P_2S_5$<br>Aerofloat 208<br>Separan AP30 | 2,0<br>0,2<br>0,003 | 27 | 11,0 | 0,38 | 1,94 | 1,73 |
| S | $P_2S_5$<br>Aerofloat 208<br>Separan AP30 | 2,0<br>0,2<br>0,015 | 37 | 11,0 | 0,38 | 2,65 | 2,36 |

TABLE 7: FILTRATION TESTS - VARIATION OF DRAIN TIME AT CONSTANT FORM TIME

| Sample | Form Time sec | Drain Time sec | Cake Thickness approx. ins | Cake Weight g |  | Cake Moisture % (Wet Basis) | ml | Filtrate gal/sq ft/hr | Dry Cake Filtration Rate lb/sq ft/hr |
|---|---|---|---|---|---|---|---|---|---|
|  |  |  |  | Wet | Dry |  |  |  |  |
| C4 | 30 | 90 | 7/8 | 318,2 | 271,2 | 14,8 | 212 | 23,7 | 717 |
| (2nd sample) | 30 | 120 | 11/16 | 241,8 | 208,1 | 13,9 | 160 | 19.1 | 551 |
|  | 30 | 150 | " | 241,0 | 215,0 | 12,1 | 144 | 19,1 | 568 |
| S | 30 | 90 | $\frac{1}{4}$ | 116,2 | 98,0 | 15,6 | 40,0 | 19,1 | 259 |
|  | 30 | 120 | $\frac{1}{4}$ | 106,5 | 91,0 | 14,6 | 40,0 | 19,1 | 241 |
|  | 30 | 150 | $\frac{1}{4}$ | 91,0 | 77,0 | 15,4 | 35,0 | 11,1 | 220 |

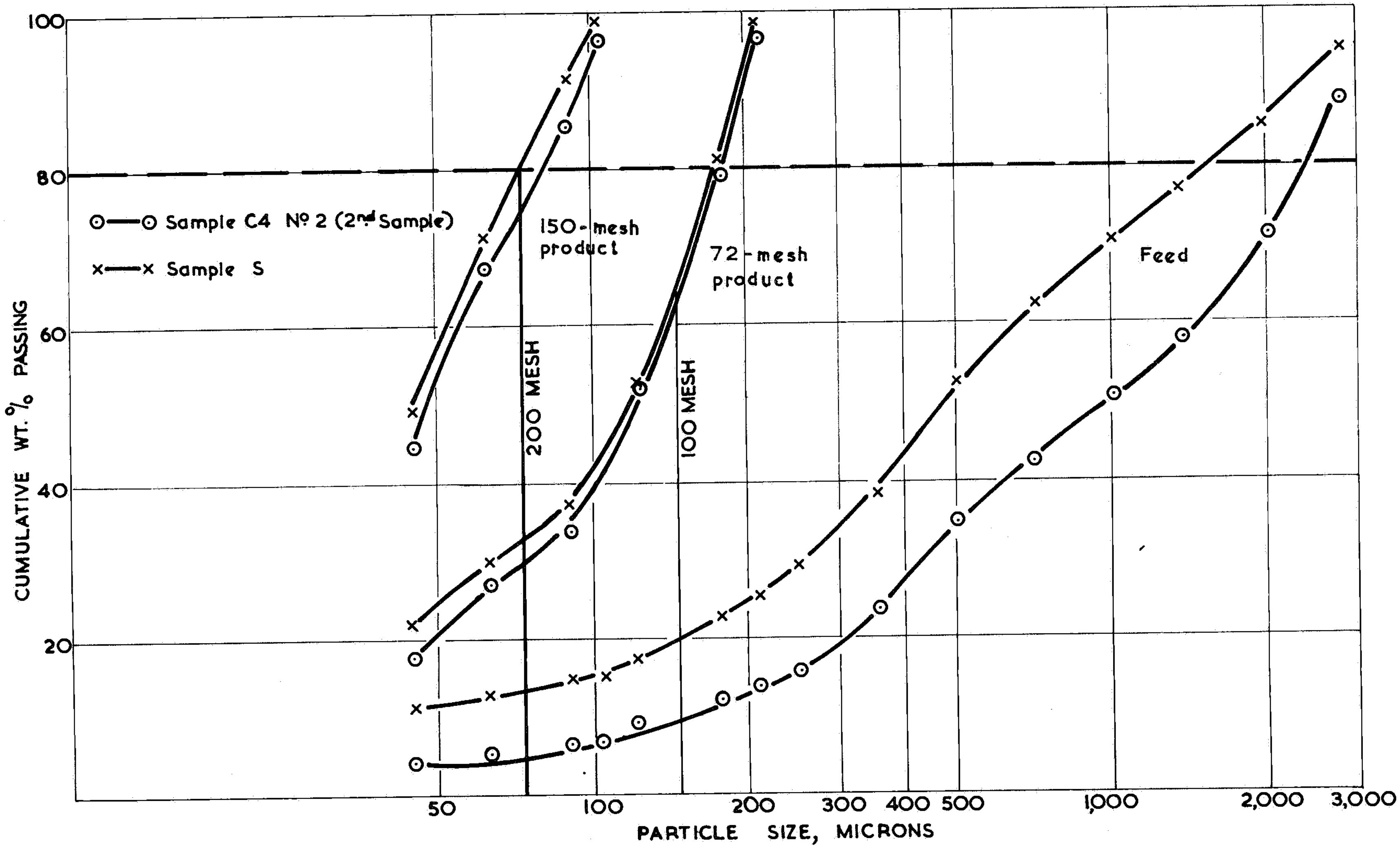

FIGURE I: SIZING OF GRINDABILITY FEED & PRODUCTS ON GUNSON ORES.

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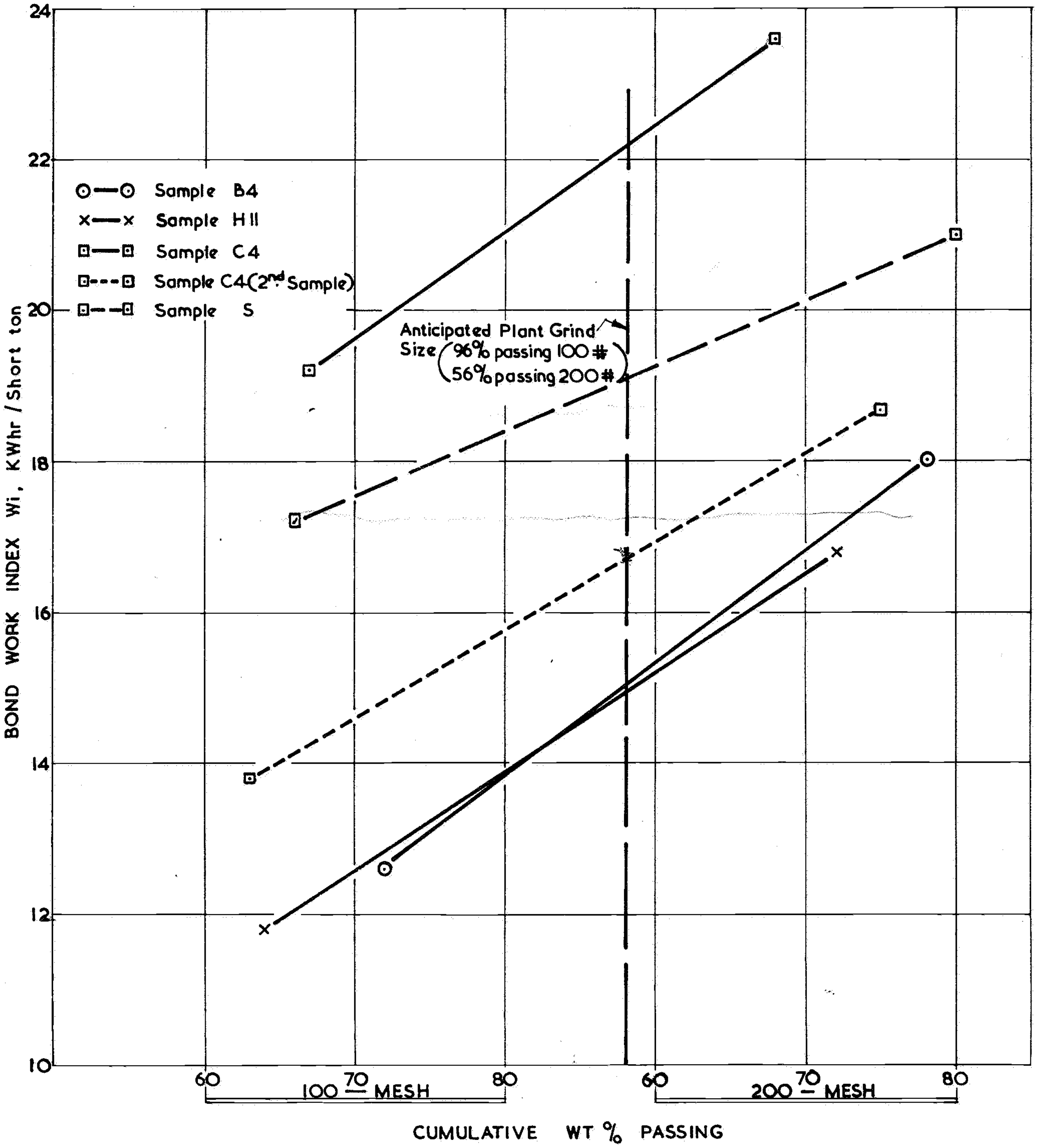

FIGURE 2: SUMMARY OF GRINDABILITY TESTS ON GUNSON ORES.

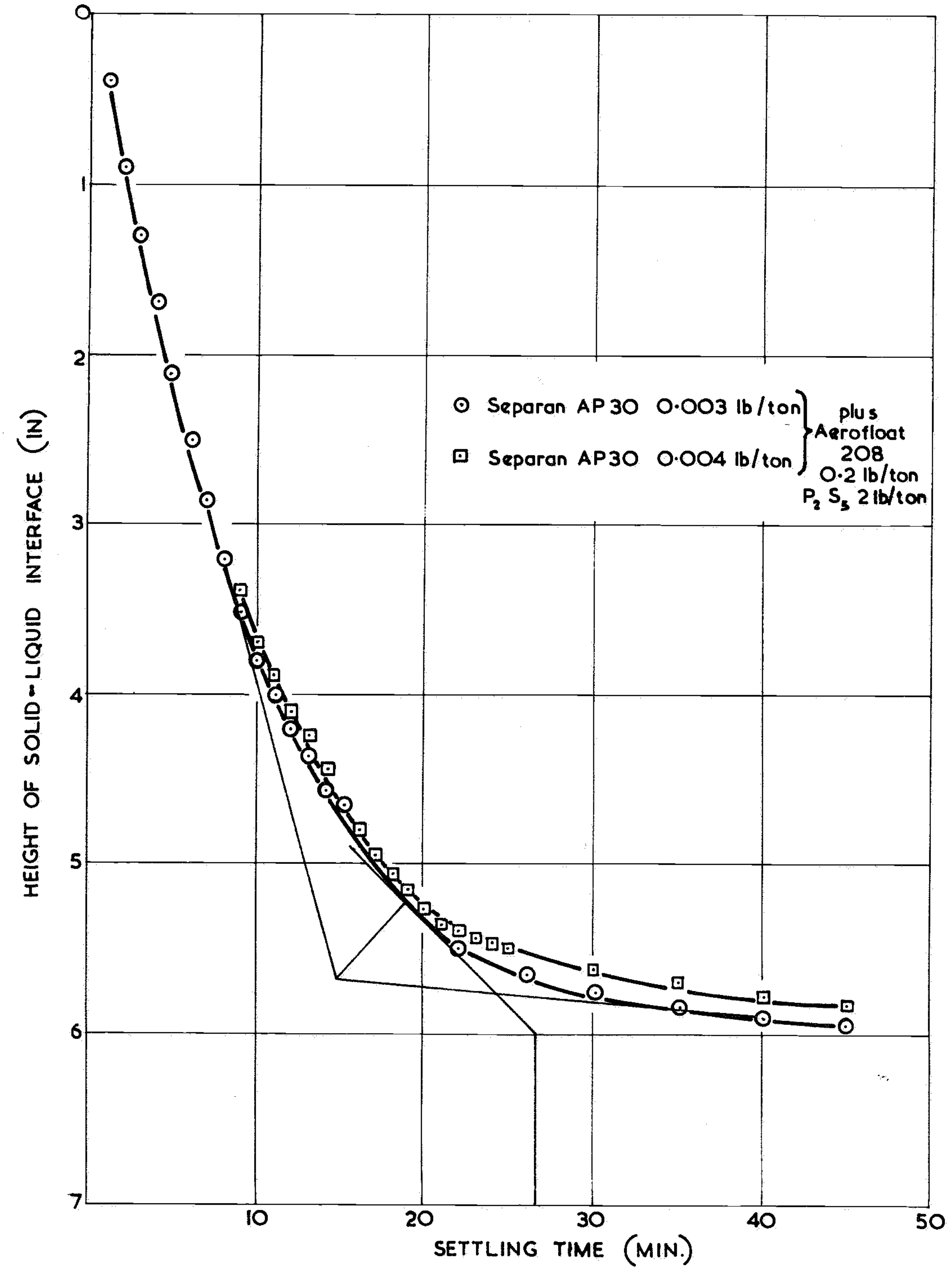

FIGURE 3: FREE SETTLING OF GUNSON ORES. SAMPLE C4 (2nd. Sample). COMBINED -72 & -150 MESH GRINDABILITY PRODUCTS.

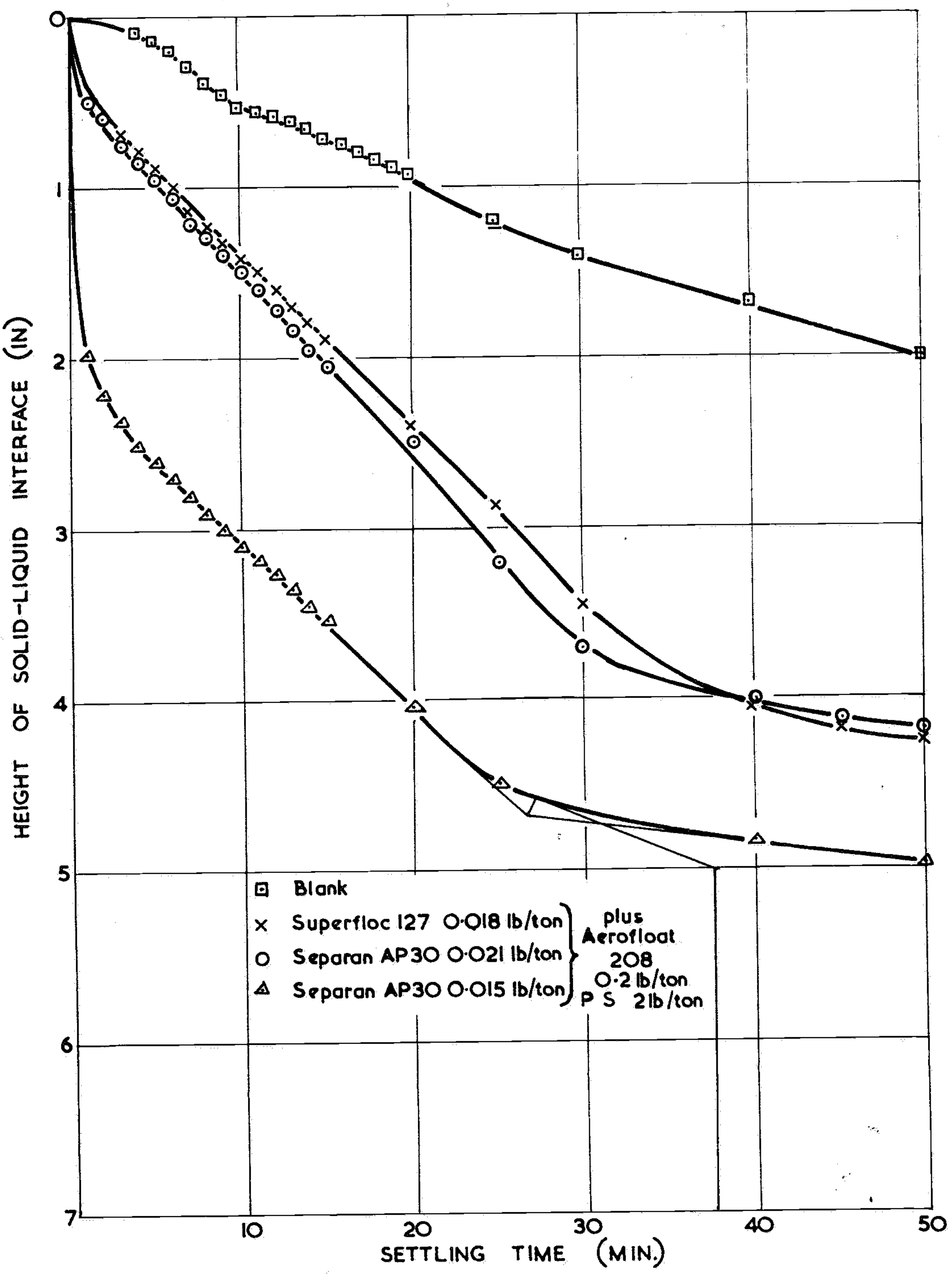

FIGURE 4: FREE SETTLING OF GUNSON ORES. SAMPLE S. COMBINED -72 & -150 MESH GRINDABILITY PRODUCTS.

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RESEARCH FOR INDUSTRY

**THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES**

CONYNGHAM STREET - FREWVILLE - SOUTH AUSTRALIA
TELEPHONE 791662 · TELEGRAMS 'AMDEL' ADELAIDE 5063

*Please quote this reference in your reply:*
ML 3/262/0

21st August, 1967

*Your reference:*

Mr J.B. Evans,
Managing Director,
Austminex Pty Limited,
Suite 20,
67 Queen's Road,
MELBOURNE, Vic. 3000.

## REPORT ML 3277-67

YOUR REFERENCE: Application dated 16/6/67

MATERIAL: Copper ore

LOCALITY: Mount Gunson, S.A.

IDENTIFICATION: Samples B4, H11, C4

DATE RECEIVED: 21/6/67

WORK REQUIRED: Crushing tests.

Investigation and Report by: L.J. Weir, C.V. Agate

Officer in Charge, Metallurgy Section: P.K. Hosking

PK Hosking
P.A. Young
Director.

# CRUSHING TESTS ON MT GUNSON ORES

## 1. INTRODUCTION

Laboratory beneficiation tests had demonstrated the feasibility of recovering copper by flotation from ground ore from the Mt Gunson deposits. Crushing tests were requested to assist in equipment selection for a proposed 1000 ton/day concentration plant.

## 2. MATERIAL EXAMINED

Selected samples of three ore types were received. The material was nominally 8 inch lump ore, with particles ranging from approximately 10 inch to 4 inch in maximum dimension.

## 3. EQUIPMENT

British Standard Series screens (B.S.S.) 18 inch diameter
Jaques jaw-crusher, 20 inch x 12 inch
Jaques Gyrex crusher, 14 inch diameter
Hazemag laboratory impact crusher.

## 4. EXPERIMENTAL PROCEDURE AND RESULTS

### 4.1 Jaw-Crushing

Each of the three samples supplied was fed to the jaw-crusher set to a close-size of nominally 3 inch. The crushed product was mixed and halved. One portion was held for gyratory crushing tests, and the other portion was sized by screening, results of which are shown in Table 1 and Figure 1. The largest pieces of ore in product were:-

| | |
|---|---|
| Sample H11 | 11 inch x 5½ inch x 2 inch |
| Sample C4 | 5 inch x 4½ inch x 2 inch |

### 4.2 Gyratory Crushing

Material previously jaw-crushed to nominally passing 3 inch was fed to a gyratory crusher set to a close-size of ½ inch. The crushed products were sized by screening, results of which are shown in Table 2 and Figure 2.

## 4.3 Impact Crushing

Fractions coarser than ½ inch from screening of jaw-crushed material were re-combined and hand-fed to a laboratory impact mill. The unit was fitted with a tared set of 4 manganese-steel blow-bars, and was run initially at a speed of 800 rpm. Further tests were done at speeds of 1500 and 2000 rpm, aiming for a product nominally passing ½ inch. It was necessary to hand-knap some pieces of ore to allow particles to enter the impact mill feed throat (approximately 4 inch square). Crushed products were sized, results of which are shown in Tables 3, 4 and 5 and in Figures 3, 4 and 5.

Blow-bar wear-rate, determined over the three speeds of operation, was 24 g for 130 pounds of ore fed, equivalent to 0.81 lb per short ton of ore.

## 4.4 Angle of Rill and Repose

Determinations were made on impact-milled product at a mill-speed of 1500 rpm on sample C4. Angle of rill was determined by hand-feeding ore from a height of approximately 3 feet onto a pile. Angle of repose was determined by holding material in a box approximately 1 foot square, removing one side of the box, and allowing the contained ore to slump.

Angles determined are shown:-

| | ° From Horizontal |
|---|---|
| Angle of rill | 30 |
| Angle of repose | 38 |

## 5. DISCUSSION

Crushing in jaw and gyratory crushers produced a considerable amount of slabby particles, especially on sample H11.

Impact milling produced regular cubic particles on all ores, particularly on Sample C4. Although wear-rate on impact milling was high over 3 speed runs, it is considered that a more acceptable level may be obtained by milling at minimum speed to obtain the desired product. The effect of scaling-up to a commercial-size mill is not known, but it is anticipated that wear-rate would decrease in a larger mill.

The use of alternatives to manganese-steel blow-bars may also reduce wear significantly.

## 6. CONCLUSIONS AND RECOMMENDATIONS

The following conclusions are drawn from the tests:-

A. The percentage of product passing nominal crusher settings with jaw and gyratory crushers is relatively low on all samples tested. Thus, close-circuiting of secondary crushing, and possibly primary crushing, by screening is indicated to allow satisfactory product sizings and to obviate the need for excessively restricted crusher settings.

B. Impact milling gives a better particle shape in product than does gyratory crushing on all samples tested, and a very much more satisfactory size distribution in product on sample C4 at moderate mill speed.

Wear-rate, although high over a 3 speed range on all samples, may be acceptable at plant-scale because of benefits in subsequent size reduction stages.

It is recommended that impact crushing trials in a plant or near-plant size mill be conducted to further evaluate this process as a possible alternative to primary and secondary crushing in more conventional units.

## TABLE 1: SCREEN SIZING OF JAW-CRUSHER PRODUCT

| Size Fraction | Sample B4 Wt. % | B4 Cum Wt. Passing % | H11 Wt. % | H11 Cum Wt. Passing % | C4 Wt. % | C4 Cum Wt. Passing % |
|---|---|---|---|---|---|---|
| 4 inch | Nil | 73.4 | 20.4 | 79.6 | 9.3 | 90.7 |
| 3 " | 26.6 | 56.2 | 15.8 | 63.8 | 28.7 | 62.0 |
| 2½ " | 17.2 | 29.6 | 15.0 | 48.8 | 15.6 | 46.4 |
| 2 " | 26.6 | 19.5 | 16.2 | 32.6 | 15.6 | 30.8 |
| 1½ " | 10.1 | 12.0 | 10.4 | 22.2 | 10.9 | 19.9 |
| 1 " | 7.5 | 6.3 | 9.9 | 12.3 | 7.5 | 12.4 |
| ½ " | 5.7 | 3.7 | 5.9 | 6.4 | 4.0 | 8.4 |
| ¼ " | 2.6 | 2.8 | 2.5 | 3.9 | 2.0 | 6.4 |
| ⅛ " | 0.9 | | 0.9 | 3.0 | 0.9 | 5.5 |
| -⅛ " | 2.8 | | 3.0 | | 5.5 | |
| | 100.0 | | 100.0 | | 100.0 | |

## TABLE 2: SCREEN SIZING OF GYRATORY-CRUSHER PRODUCT

| Size Fraction | Sample B4 Wt. % | B4 Cum Wt. Passing % | H11 Wt. % | H11 Cum Wt. Passing % | C4 Wt. % | C4 Cum Wt. Passing % |
|---|---|---|---|---|---|---|
| 1½ inch | 0.4 | 99.6 | 0.2 | 99.8 | 0.4 | 99.6 |
| 1¼ " | 1.7 | 97.9 | 3.0 | 96.8 | 0.8 | 98.8 |
| 1 " | 4.9 | 93.0 | 8.2 | 88.6 | 4.2 | 94.6 |
| ¾ " | 18.1 | 74.9 | 16.5 | 72.1 | 14.1 | 80.5 |
| ⅝ " | 16.6 | 58.3 | 17.8 | 54.3 | 14.5 | 66.0 |
| ½ " | 15.8 | 42.5 | 13.6 | 40.9 | 14.9 | 51.1 |
| ¼ " | 24.1 | 18.4 | 22.9 | 18.0 | 25.4 | 25.7 |
| ⅛ " | 7.7 | 10.7 | 7.4 | 10.6 | 8.5 | 17.2 |
| 6 mesh | | | | | | |
| 12 " | 3.3 | 7.4 | 2.8 | 7.8 | 3.8 | 13.4 |
| 16 " | 0.6 | 6.8 | 0.6 | 7.2 | 0.6 | 12.8 |
| 22 " | 1.5 | 5.3 | 1.1 | 6.1 | 1.7 | 11.1 |
| 30 " | | | | | | |
| 44 " | 1.6 | 3.7 | 1.7 | 4.4 | 5.5 | 5.6 |
| 60 " | 1.2 | 2.5 | 1.2 | 3.2 | 2.6 | 3.0 |
| -60 " | 2.5 | | 3.2 | | 3.0 | |
| | 100.0 | | 100.0 | | 100.0 | |

## TABLE 3: SCREEN SIZING OF IMPACT-MILL PRODUCT (800 rpm)

| Nominal Width of Aperture | BSS Equivalent | Sample B4 Weight % | Sample B4 Cum Wt. Passing % | Sample H11 Weight % | Sample H11 Cum Wt. Passing % | Sample C4 Weight % | Sample C4 Cum Wt. Passing % |
|---|---|---|---|---|---|---|---|
| | 4 inch | | | | | | |
| | 2 " | | | | | | |
| 12.70 mm | ½ " | 51.0 | 49.0 | 53.0 | 47.0 | 41.6 | 58.4 |
| 6.35 " | ¼ " | 19.7 | 29.3 | 22.1 | 24.9 | 15.8 | 42.6 |
| 3.18 " | ⅛ " | 10.1 | 19.2 | 10.2 | 14.7 | 9.1 | 33.5 |
| 1.40 " | 12 mesh | 5.8 | 13.4 | 4.6 | 10.1 | 5.9 | 27.6 |
| 1.00 " | 16 " | 1.1 | 12.3 | 0.8 | 9.3 | 1.4 | 26.2 |
| 710 microns | 22 " | 2.6 | 9.7 | 1.7 | 7.6 | 4.5 | 21.7 |
| 355 " | 44 " | 3.6 | 6.1 | 2.6 | 5.0 | 8.4 | 13.3 |
| 250 " | 60 " | 2.1 | 4.0 | 1.7 | 3.3 | 6.8 | 6.5 |
| 180 " | -60 " | 4.0 | | 3.3 | | 6.5 | |
| | | 100.0 | | 100.0 | | 100.0 | |

## TABLE 4: SCREEN SIZING OF IMPACT-MILL PRODUCT (1500 rpm)

| Nominal Width of Aperture | BSS Equivalent | Sample B4 Weight % | Sample B4 Cum Wt. Passing % | Sample H11 Weight % | Sample H11 Cum Wt. Passing % | Sample C4 Weight % | Sample C4 Cum Wt. Passing% |
|---|---|---|---|---|---|---|---|
| 12.70 mm | ½ inch | 10.7 | 89.3 | 14.2 | 85.8 | 3.4 | 96.6 |
| 6.35 " | ¼ " | 17.7 | 71.6 | 20.1 | 65.7 | 8.2 | 88.4 |
| 3.18 " | ⅛ " | 19.7 | 51.9 | 20.1 | 45.6 | 12.5 | 75.9 |
| 2.80 " | 6 mesh | | | | | | |
| 1.40 " | 12 " | 14.3 | 37.6 | 12.6 | 33.0 | 11.7 | 64.2 |
| 1.00 " | 16 " | 2.5 | 35.1 | 2.0 | 31.0 | 2.4 | 61.8 |
| 710 microns | 22 " | 6.7 | 28.4 | 5.1 | 25.9 | 6.7 | 55.1 |
| 355 " | 44 " | 9.7 | 18.7 | 8.2 | 17.7 | 20.7 | 34.4 |
| 250 " | 60 " | 6.9 | 11.8 | 6.1 | 11.6 | 19.9 | 14.5 |
| 180 " | -60 " | 11.8 | | 11.6 | | 14.5 | |
| | | 100.0 | | 100.0 | | 100.0 | |

TABLE 5: SCREEN SIZING OF IMPACT-MILL PRODUCT
(2000 rpm

| Nominal Width of Aperture | BSS Equivalent | Sample B4 Weight % | Sample B4 Cum Wt. Passing % | Sample H11 Weight % | Sample H11 Cum Wt. Passing % | Sample C4 Weight % | Sample C4 Cum Wt. Passing% |
|---|---|---|---|---|---|---|---|
| 12.70 mm | ½ inch | 0.9 | 99.1 | 0.6 | 99.4 | 0.7 | 99.3 |
| 6.35 " | ¼ " | 7.5 | 91.6 | 6.9 | 92.5 | 1.6 | 97.7 |
| 3.18 " | ⅛ " | 17.2 | 74.4 | 16.2 | 76.3 | 5.0 | 92.7 |
| 1.40 " | 12 mesh | 17.5 | 56.9 | 15.9 | 60.4 | 7.6 | 85.1 |
| 1.00 " | 16 " | 3.0 | 53.9 | 2.9 | 57.5 | 1.7 | 83.4 |
| 710 microns | 22 " | 8.2 | 45.7 | 8.6 | 48.9 | 8.8 | 74.6 |
| 355 " | 44 " | 14.3 | 31.4 | 14.7 | 34.2 | 27.6 | 47.0 |
| 250 " | 60 " | 11.5 | 19.9 | 12.5 | 21.7 | 28.3 | 18.7 |
| 180 " | -60 " | 19.9 | | 21.7 | | 18.7 | |
| | | 100.0 | | 100.0 | | 100.0 | |

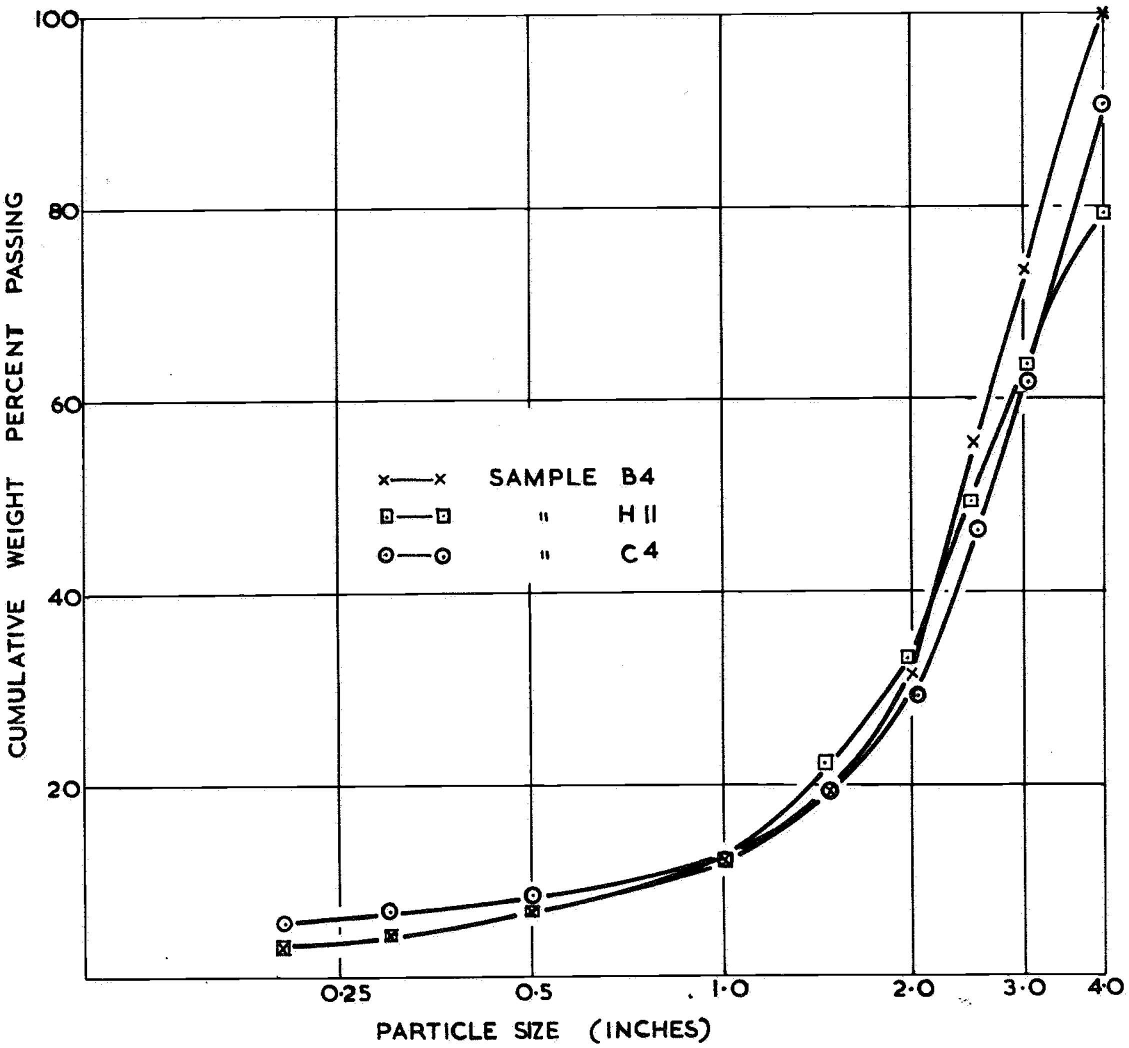


FIGURE 1: JAW CRUSHING MINUS 8 INCH GUNSON ORE.

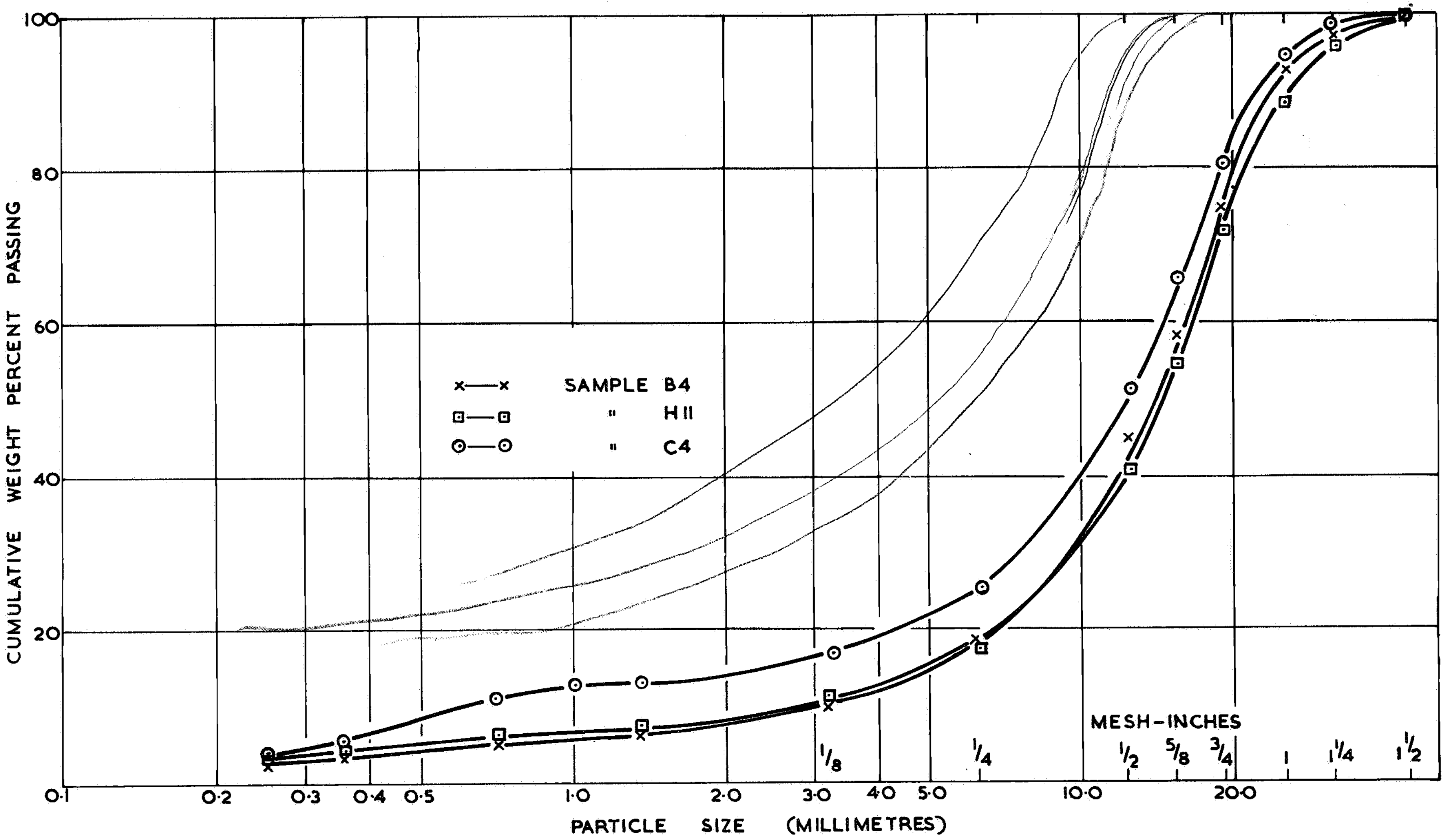
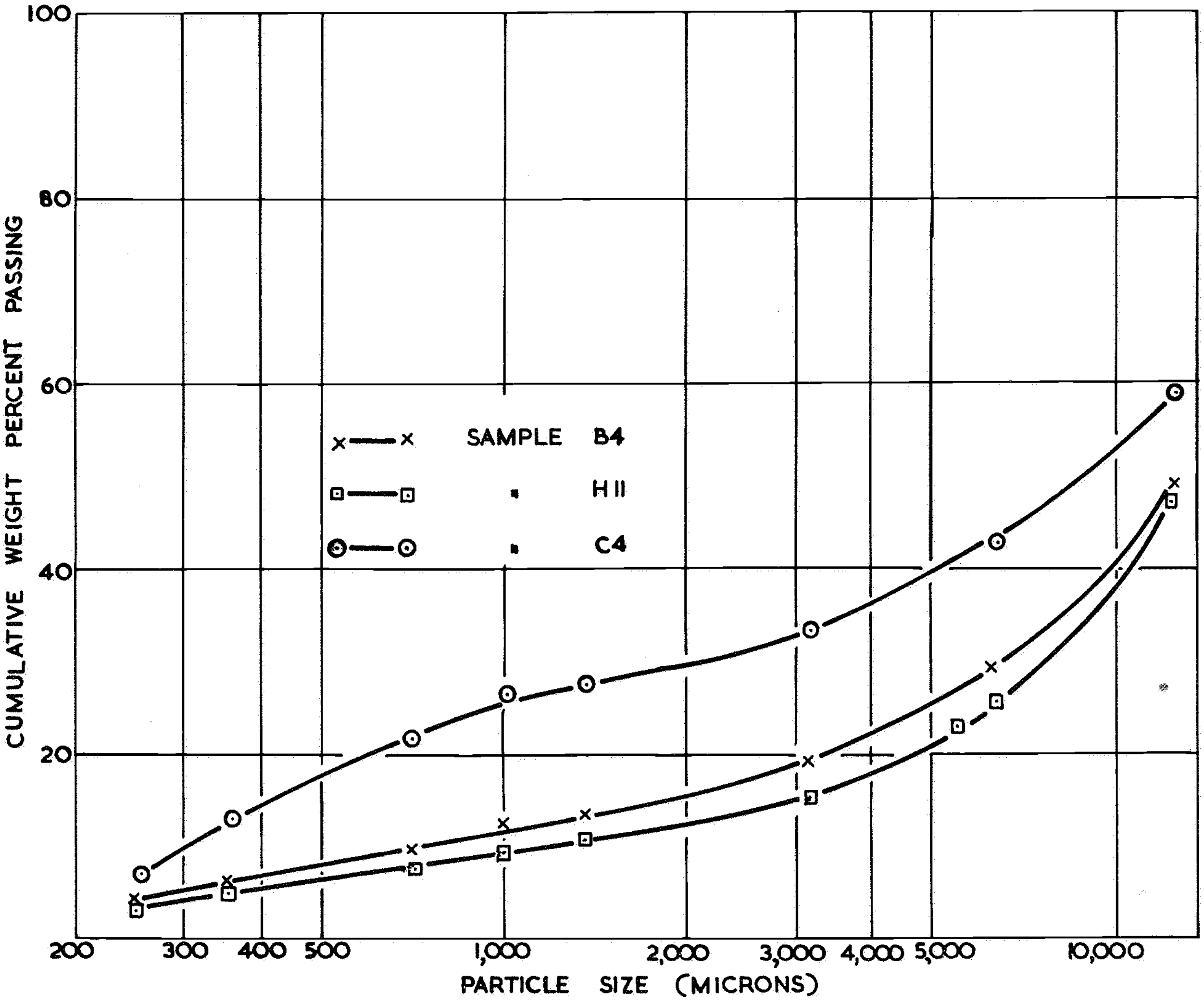


FIGURE 2: GRYRATORY CRUSHING OF MINUS 3 INCH GUNSON ORE.

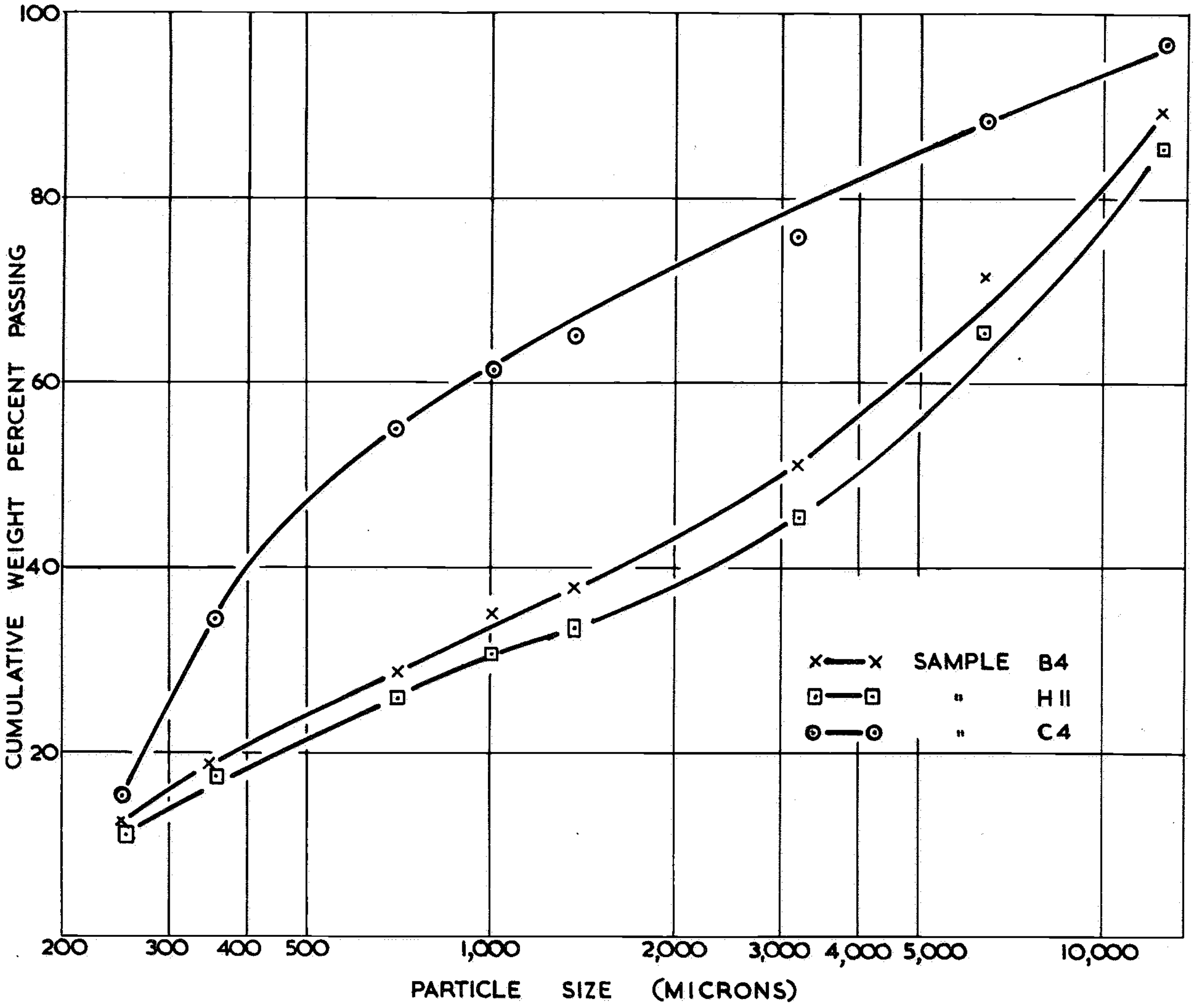
RG 2011



800 R.P.M.

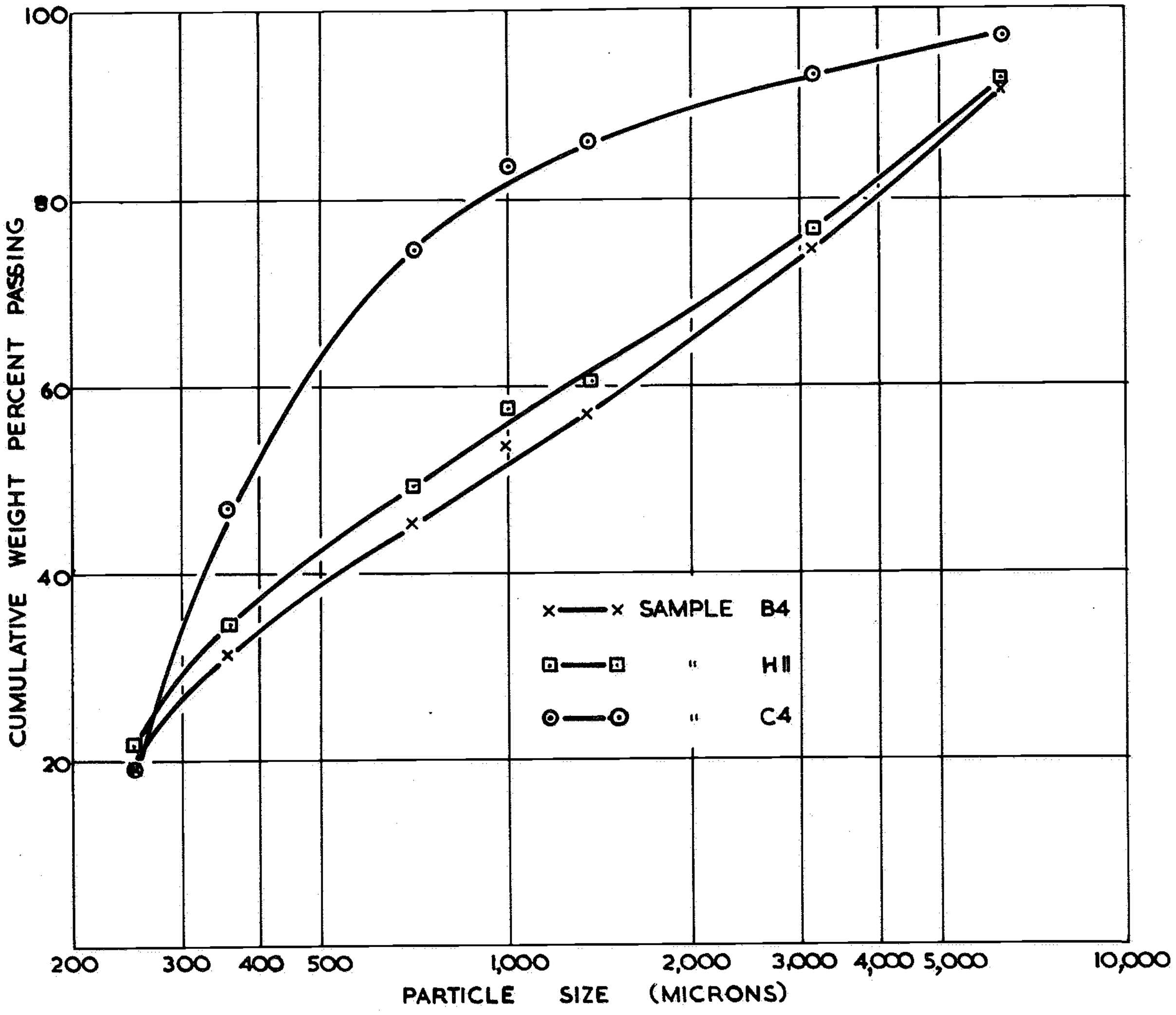
FIGURE 3: HAZEMAG IMPACT CRUSHING OF MINUS 3 INCH GUNSON ORE.

RG 2012



1,500 R.P.M.

FIGURE 4: HAZEMAG IMPACT CRUSHING OF MINUS 3 INCH GUNSON ORE.



2,000 R.P.M.

FIGURE 5: HAZEMAG IMPACT CRUSHING OF MINUS 3 INCH GUNSON ORE.

# REVIEW OF PROJECT FEASIBILITY

for

1,000 T.P.D MINE & MILL

## MT. GUNSON COPPER PROJECT

South Australia

WRIGHT ENGINEERS LIMITED
VANCOUVER ——— CANADA

PROJECT N° 496

SEPT 1968

WRIGHT ENGINEERS LIMITED

PHONE 684-9371 • CABLE "WRIGHTENG" • TELEX 04-50360

1101 WEST PENDER STREET • VANCOUVER 1, B.C., CANADA

September 12, 1968.

Ord, Minnett, T.J. Thompson & Partners,
33rd Floor, Australia Square,
Sydney, N.S.W. 2000.

Dear Sirs:

This report was prepared for inclusion in the prospectus of the Mount Gunson Copper Project.

## Introduction

Oxidized copper orebodies near Mount Gunson, South Australia, have been outlined by drilling and samples of copper ore from these orebodies have been tested metallurgically to determine the most suitable methods of treatment.

Proposals have been advanced to mine and treat the copper ore at a rate of 1000 tons daily. We have examined these proposals with respect to plant design, capital and operating costs and estimated ore reserves and our evaluation and recommendations are summarized in this report.

The site of the proposed mining and process plant has been inspected by Mr. L. F. Wright, of Wright Engineers Limited, and Mr. E. Macdonald, consulting mining engineer, of Sydney, N.S.W., who has reviewed the results of exploratory drilling and calculation of ore reserves.

We are satisfied that we have received full disclosure of all conditions that may affect the capital costs, operating costs, and revenue that would be attained by the proposed plant. *

## Process Plant Requirements

It is proposed to contract the mining of copper ore which would be hauled to a crushing plant and flotation concentrator at a rate of 1,000 tons of ore daily. Electric power and water required by this process plant will be drawn from power and waterlines which supply Woomera.

We have reviewed various estimates of capital cost of the process plant and related services and we consider the total capital cost will be as follows:

Cont'd...

| | |
|---|---|
| Pit Preparation | $ 50,000 |
| Crushing Plant and Concentrator | 1,364,000 |
| Electric Power | 70,000 |
| Water Supply | 319,000 |
| Preproduction Expenses | 215,000 |
| Inventory | 100,000 |
| Working Capital | 180,000 |
| Contingency | 180,000 |
| Expenses to June, 1968. | 640,000 |
| TOTAL | $3,118,000 |

Of this total, $200,000 of construction costs may be deferred for two months after the start of production of copper concentrates.

Plant Operation

The copper ore delivered to the process plant during the first seven years will contain copper sulphide minerals which are readily concentrated by flotation. During the last three years of operation, the ore delivered to the process plant will be chiefly oxidized copper ore which will be more costly to treat in the process plant and which will yield a low grade of copper concentrate. We estimate that the revenue, operating costs and operating profit during the estimated ten year life of the operation will be as follows:

| Year | Revenue from Sale of Copper Concentrates | Smelting Costs and Operating Costs | Operating Profit |
|---|---|---|---|
| 1 | $ 2,413,600 | $ 1,265,800 | $ 1,147,800 |
| 2 | 2,434,400 | 1,391,300 | 1,043,100 |
| 3 | 2,328,900 | 1,470,100 | 958,800 |
| 4 | 1,864,900 | 1,492,000 | 372,900 |
| 5 | 1,864,900 | 1,492,000 | 372,900 |
| 6 | 1,951,500 | 1,505,100 | 446,400 |
| 7 | 1,950,700 | 1,505,500 | 445,200 |
| 8 | 1,405,000 | 1,264,900 | 130,100 |
| 9 | 1,389,300 | 1,161,000 | 228,300 |
| 10 | 1,219,700 | 1,124,600 | 95,100 |
| TOTAL OPERATING PROFIT | | | $ 5,240,600 |

It should be noted that an additional capital expense of $18,000 will be required early in the eighth year of production for modifications to the process plant to permit treatment of oxidized copper ore.

## Ore Reserves

The four orebodies of the Mount Gunson copper deposit are estimated by Mr. E. Macdonald to contain the following tonnage and grade of copper ore:

| Orebody | Tons | Copper Grade | Silver Content |
|---|---|---|---|
| Lagoon (sulphide ore) | 952,000 | 1.19% | 0.52 oz/ton |
| House (oxide ore) | 126,000 | 0.82% | nil |
| House (sulphide ore) | 703,000 | 0.83% | 0.43 oz/ton |
| Main open cut (oxide ore) | 693,000 | 1.31% | nil |
| Main open cut (sulphide ore) | 644,000 | 0.88% | 0.37 oz/ton |
| Gunyot (oxide ore) | 102,000 | 1.13% | nil |
| TOTAL | 3,220,000 | 1.04% | 0.32 oz/ton |

A total of 4,030,000 tons of waste must be removed progressively from the Lagoon, House, Main open cut, and Gunyot orebodies to permit complete ore extraction over a ten year period.

The Lagoon orebody will be mined first because this orebody contains the highest grade sulphide ore and the lowest ratio of waste to ore. Sulphide ore would be mined from the House orebody and the Main open cut orebody between the fourth and seventh years of production. During this period, oxide ore would be stripped and stockpiled for processing during the last three years of production when the stockpiled oxide ore would be augmented by oxide ore mined from the Gunyot orebody.

We consider the proposed sequence of stripping waste, stockpiling oxide ore, and mining sulphide ore will yield the most favourable return of operating profits over the estimated ten year life of the property.

Respectfully submitted,

WRIGHT ENGINEERS LIMITED

L. F. Wright, P.Eng.,
Vice-President.

* Our estimate of electric power cost and water supply is based on verbal assurance of Mr. J. B. Evans of the cost reduction and availability of electric power and water from sources supplying Woomera. This verbal assurance will be confirmed in writing by Mr. J. B. Evans.

# MOUNT GUNSON COPPER DEPOSIT
# REVIEW OF PROJECT FEASIBILITY

## Introduction

On July 9, 1968, Wright Engineers Limited were requested by Ord, Minnett, T. J. Thompson and Partners, of Sydney, N.S.W., to review technical information that had been compiled on the Mount Gunson Copper Deposit and to check and make recommendations on the capital costs, plant design, ore reserves, overburden ratio, metallurgy, extraction rates and operating costs of the proposed Mount Gunson Copper Project.

## Capital Costs

We have reviewed the capital cost estimate prepared by Chapman, Wood, Griswold and Evans Pty., in their Evaluation Report dated January 12, 1968, together with additional and subsequent technical information compiled by others. We have discussed the effect of our recommendations and modifications of the plant design on capital costs of the plant with Mr. J. B. Evans on September 3, and 4, 1968, in Vancouver.

Our estimate of plant capital cost is shown in Table 1.

### TABLE 1 SUMMARY OF ESTIMATED CAPITAL COSTS

| | Cost A $ |
|---|---:|
| Pit Preproduction Preparation | 50,000 |
| Crushing Plant and Concentrator | 1,364,000 |
| Power | 70,000 |
| Water | 319,000 |
| Sub-Total | 1,803,000 |
| Contingency @ 10% | 180,000 |
| Working Capital | 180,000 |
| Preproduction Overhead | 80,000 |
| Preproduction Hiring | 25,000 |
| Preproduction Inventory | 100,000 |
| Preproduction Payments | 110,000 |
| Expenses to June, 1968. | 640,000 |
| Sub-Total | 3,118,000 |
| Deferred Capital Cost | 200,000 |
| TOTAL | 2,918,000 |

## NOTES ON CAPITAL COST ESTIMATE

### Pit Preproduction Preparation

Before the sulphide ore in the Lagoon East orebody is exposed for mining, 100,000 tons of overyling waste must be stripped.

Based on expected stripping costs of 40 cents per short ton, as per Contractor's prices, this waste removal will cost $40,000.

The cost of road work and other preparation work prior to stripping is estimated at $10,000.

### Crushing Plant and Concentrator

The revised design concept of the crushing plant and fine ore handling installation results in an estimated net increase in cost of $39,348.

A saving of $13,430 has resulted from changes made in the concentrator and an additional saving of $49,068 would be realized by eliminating the maintenance building and facilities, which would be provided by the Mining Contractor.

Our review of the capital cost estimate indicates that estimated costs for civil work covering excavation, steel and concrete are insufficient, and an allowance for increased cost has been included. An additional $15,000 is deemed necessary for surface vehicles as listed.

### Power

As recommended by Cor Stastra and Associates, November 29, 1967, power will be supplied by a 132 K.V. line from the Port Augusta - Woomera power line to a 132/33 K.V. sub-station containing a 3.5 M.V.A. transformer and associated switch gear. A 33 K.V. feeder from the sub-station to the site will terminate in suitable distribution transformers. The estimated total cost of transmission lines, transformers, and sub-stations is $160,000.

We have received verbal assurance, to be confirmed in writing, that the expenditure required will be for the power line only, totalling $70,000.

### Water

We accept the system of water supply as proposed by Cor Stastra and Associates on June 17, 1968. Total of estimated cost $285,000.

The proposed water supply will consist of 7.47 miles of 6" P.V.C. water line connecting the plant site with a take off point 7 miles upstream of the Woocalla pumping station. A 500,000 gallon water tank is required at the plant site. Capacity of the water supply system will be 200 g.p.m. at 50 ft. head at the mill.

An increase of the Port Augusta - Woomera water supply from 500 to 650 g.p.m., will be effected by fitting the three existing stations with new pumps and installing three booster stations.

Our estimate for engineering of the above installations is $34,000.

## Working Capital

Our estimate of working capital is based on two months operation, at an operating cost of $3.00 per ton, prior to receipt of revenue from the sale of concentrates.

## Preproduction Overhead

Preproduction overhead costs cover head office expenses from June 30, 1968, to October 31, 1969, incurred by four personnel at an estimated monthly cost of $5,000.

## Preproduction Hiring

$25,000 has been allowed to cover the wages of employees that will be hired one month earlier than the start of production.

## Preproduction Inventory

An allowance of $100,000 will be required for the purchase of operating supplies and vital spare parts prior to the start of production.

## Preproduction Payments

Reference Chapman, Wood, Griswold and Evans Pty. Ltd.'s Report June 19, 1968.

| | |
|---|---|
| Stamp Duty, Registration, Prospectus | $ 35,000 |
| Jervois Payment | $ 15,000 |
| Legal and Underwriting | $ 40,000 |
| Insurance and Inspection | $ 20,000 |

## Expenses to June 30, 1968.

Preproduction investigation expenses incurred to June 30, 1968, reference Chapman, Wood, Griswold and Evans Pty. Ltd.'s Report, June 19, 1968.

| | |
|---|---|
| Exploration, Ore Delineation, Metallurgical Testing | $515,000 |
| Purchase of Leases and Plant | $125,000 |

## Deferred Capital Cost

With a scheduled start-up on October 31, 1969, the production rates for November and December are estimated at 300 and 450 tons of concentrate respectively, valued at approximately $350. per ton. Assuming 150 tons will be in circuit inventory, saleable assets will amount to approximately $216,000. On this basis, $200,000 has been considered available at two months after start-up for deferred capital payments.

## PLANT DESIGN

### Scope of Study

We have reviewed the basic information provided to us and the proposed plant design as contained in the McKee - I.E.L. Report No. 102-1, dated September, 1967.

This review has involved a study of the design concept of the crushing and milling facilities and of the major process equipment incorporated in the design.

Details of mill design and specifications of minor equipment have not been thoroughly examined and comments and recommendations on these items are not included.

### Conclusions

Because the primary crushing operation will not be under the control of the plant operating staff, but will be the responsibility of an outside Contractor, we recommend that coarse and fine crushing should be designed as independent operations.

We consider the reduction ratio required by the fine crushing to be too large for one stage crushing by a 5-1/2 ft. shorthead cone crusher. This may be particularly apparent in this project where the primary crusher product size will not be under the strict control of the treatment plant and where a single stage Ball mill circuit without benefit of rod mill will require a fine crushed feed.

Problems may also be encountered in feeding a shorthead crusher with a primary crusher product that is oversized.

The one stage of wet grinding, followed by rougher flotation and two stages of cleaner flotation are considered to be the most economic and efficient methods of concentration and we concur with this proposal.

### Recommendations - (Reference Drawings No. 496-001-101 and 102)

1. A larger width jaw crusher is preferred to handle open pit ore and a 30" x 42" unit is recommended. A single toggle crusher would be suitable for this duty.

2. A 4' x 8' scalping screen is proposed to remove minus 1/2" material prior to stockpiling. In addition to minimizing the dust hazards and loss of values, the under size material will bypass the secondary crushing circuits and will be conveyed directly to the fine ore bin.

3. A 2,500 ton live coarse ore stockpile is recommended so that mining and ore delivery is independent of mill requirements. An open stockpile is especially applicable in arid conditions. An open stockpile can be readily enlarged and reduces expensive fine ore storage requirements.

4. A 4 ft. standard and a 3 ft. shorthead crusher are recommended for the secondary and tertiary crushing stages. The crusher and controls would be located within the grinding area where it would be available for operation on a demand basis by the grinding section operator. This crushing circuit will have a nominal capacity of 120 tons per hour and could be operated at lower rates if so desired.

5. A 1,000 ton fine bin bins is proposed, equipped with Mexican type feeders which are recommended for optimum control with the free running feed which is anticipated. This storage will provide 24 hour storage in the event of a major shut-down in the crushing plant.

6. We agree with the power selection of 850 HP for grinding, but suggest that a larger mill should be considered to ensure that the installed horsepower can be applied to grinding. Confirmation on optimum mill size should be sought from the manufacturer.

7. The flotation conditioner, sodium sulphide and aeropromotor reagent facilities have been eliminated as non-essential items for the sulphide ore treatment.

8. Although we agree with the proposed flotation capacity, we recommend substituting the two banks of No. 40 rougher cells with one bank of No. 60 cells. This would result in very definite savings in capital and installation costs, as well as providing a simplified circuit to operate.

9. Our experience would dictate the consideration of an infra-red type dryer, which would provide the desired flexibility and simplicity of design for the comparatively small tonnages involved. Final specifications of type and size would require detailed study. Our estimated pricing and power requirements are submitted from preliminary engineering studies.

## TABLE 2 DETAILS OF CHANGES IN CAPITAL EXPENDITURES CRUSHING PLANT AND CONCENTRATOR

| | Cost A $ | |
|---|---|---|
| Item | Deleted | Added |
| Jaw Crusher, 24" x 36" Double Toggle | 33,800 | |
| Jaw Crusher, 30" x 42" Single Toggle | | 32,800 |
| Shorthead Cone Crusher, 5-1/2 Ft. | 74,900 | |
| Standard Cone Crusher, 4 Ft. | | 35,000* |
| Shorthead Cone Crusher, 3 Ft. | | 27,000* |
| Scalping Screen, 4' x 8' | | 3,680 |
| Fine Screen, 5' x 16' | 4,500 | |
| Fine Screen, 5' x 10' | | 3,500 |
| Conveyors, 500 Ft. | | 50,000* |
| Stockpile Reclaim Tunnel | | 10,000 |
| Stockpile Reclaim Feeders | | 4,000* |
| Crusher Sump Pump | | 480 |
| Fine Ore Bins, 3 - 20' ø x 40' | 29,000 | |
| Fine Ore Bins, 1 - 30' ø x 40' | | 18,000 |
| Fine Ore Feeders, 6 - 24" x 30" | 5,800 | |
| Fine Ore Feeders, 5 - 8" Tube | | 2,888* |
| Flotation Conditioner, 10'-6" ø x 11' | 5,860** | |
| Flotation Distributor, 5' ø | 2,030 | |
| Flotation Cells, 36 No. 48 | 47,240 | |
| Flotation Cells, 14 No. 60, 6 No. 48 | | 35,400 |
| Concentrate Dryer | 19,500 | |
| Concentrate Bagger | 17,000 | |
| Concentrate Dryer, 16" ø x 32' Infra-red | | 15,000* |
| Concentrate Storage | | 4,000* |

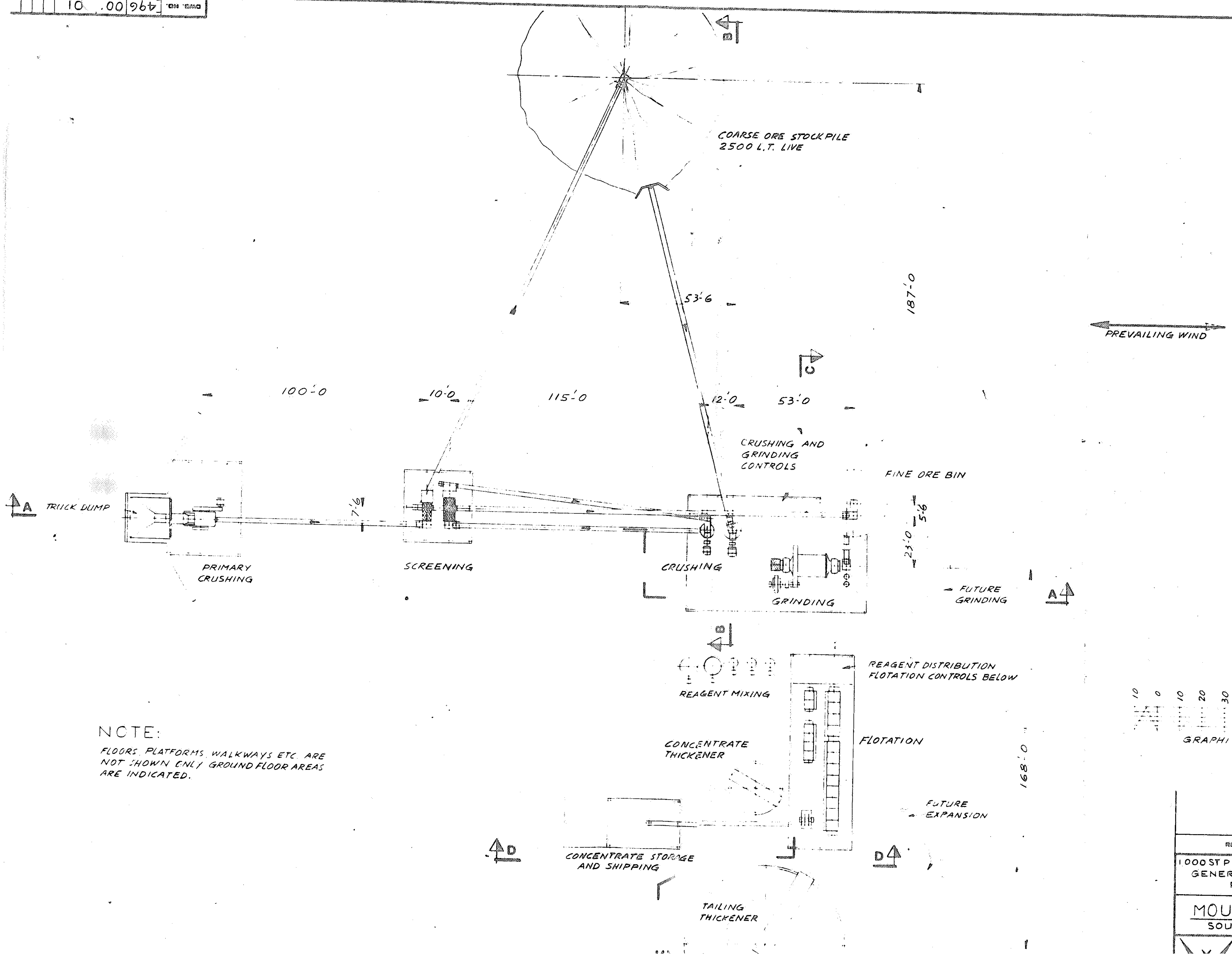
| Item | Cost A $ | |
|---|---|---|
| | Deleted | Added |
| Concentrate Weigh Scale | | 5,000* |
| Sodium Sulphide Mixer and Storage | 4,100** | |
| Aeropromotor Mixer and Storage | 4,100** | |
| Civil Work | | 10,000* |
| Surface Vehicles | | 10,000* |
| Front End Loader | | 5,000* |
| Tailing Dam Layout | | 2,000* |
| Maintenance Building, 50' x 95' | 17,200 | |
| Crane, 6 Ton O/H | 6,180 | |
| Lathe, 18" ∅ x 16' | 8,320 | |
| Drill, 6" Radial | 9,072 | |
| Welding m/c, 500 amp. | 2,400 | |
| Welding m/c, Diesel 350 amp. | 2,546 | |
| Grinder, 16" Pedestal | 350 | |
| Steam Cleaner | 3,000 | |
| TOTAL | $ 296,898 | $ 273,748 |
| Net Difference | 23,150 | |

| | |
|---|---|
| Tendered Capital Cost by Davy-Ashmore Pty. Ltd., (Revision May 10, 1965) | $ 1,387,150 |
| Less Net Difference | - 23,150 |
| Adjusted Estimated Capital Cost | $ 1,364,000 |

* Estimates by Wright Engineers Limited.

** The cost of these items, plus installation cost, which totals $18,000 will be incurred in the eighth year of production, when oxide ore will be processed.

000207



NOTE:

FLOORS, PLATFORMS, WALKWAYS ETC. ARE NOT SHOWN ONLY GROUND FLOOR AREAS ARE INDICATED.

SCALE: 1" = 20'-0

DESCRIPTION OF REVISION

ISSUED FOR

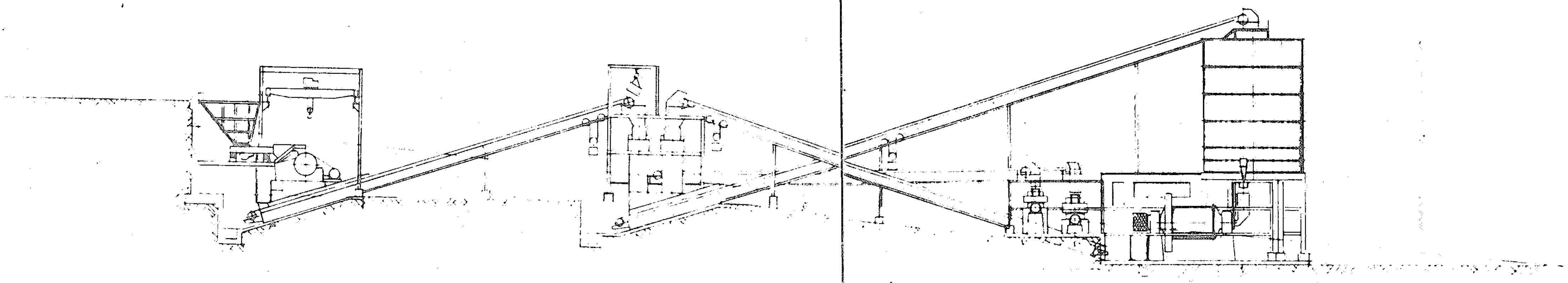
| REFERENCE | DWG. No. |
|---|---|

1000 ST P.D. Cu CONCENTRATOR
GENERAL ARRANGEMENT
PLAN VIEW

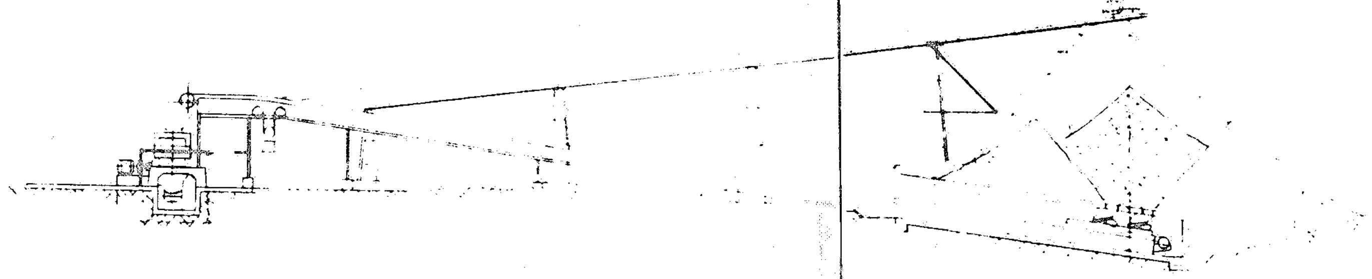
MOUNT GUNSON
SOUTH AUSTRALIA

APP. DATE 16.8.68
DESIGN ABG
DRAWN ABG
DIM. CHECK
DESIGN CHECK
FINAL CHECK
APPR.

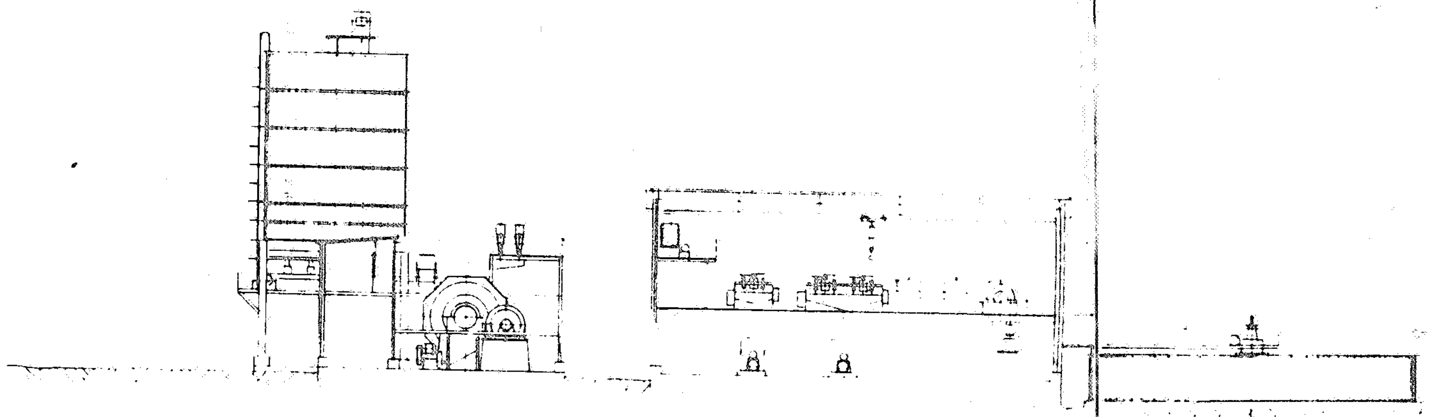
DWG. No. 496 001 101



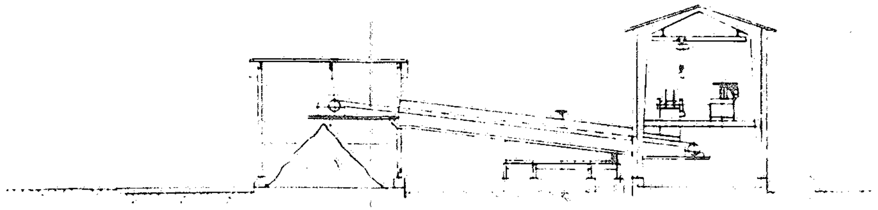
## SECTION A-A



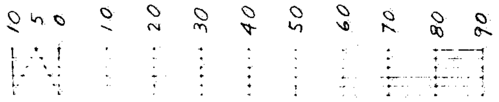
## SECTION B-B



## SECTION C-C



## SECTION D-D



GRAPHIC SCALE

SCALE: 1" = 20'-

DESCRIPTION OF REVISION

ISSUED FOR

APPD.

FINAL CHECK

DESIGN CHECK

DIM. CHECK

DRAWN ABG

DESIGN ABG

APP. DATE 16.8.68

REFERENCE | DWG. No.

1000 ST P.D Cu CONCENTRATOR
GENERAL ARRANGEMENT
SECTIONS

MOUNT GUNSON
SOUTH AUSTRALIA

DWG. No. 496 001 102

WRIGHT ENGINEERS LIMITED
VANCOUVER — CANADA

# METALLURGY

## Scope of Study

Reports on metallurgical test work that have been reviewed include the following:

Amdel Report No. 541, June, 1967.
Grinding tests and preliminary flotation tests.

Amdel Report No. ML 3276/67, August 14, 1967.
Grinding, settling and filtering tests.

Amdel Report No. ML 3277/67, August 21, 1967.
Crushing tests.

Amdel Report No. 3276/67, October 5, 1967.
Grindability testing.

Amdel Report, May, 1968.
Pilot Plant tests - Sulphide Ore.

Amdel Report, May, 1968.
Pilot Plant tests - Oxide Ore.

Metallurgical Summary by Keith Parsons,
United Uranium.

Progress Reports, No. 1-5, conducted at CSIRO, Melbourne,
May 31, 1967, - June 11, 1968, by Chapman, Wood, Griswold-
and Evans Pty. Ltd.

Our Study has included particular reference to:

Optimum grind for flotation.
Flotation methods and reagent requirements.
Anticipated recovery of values from sulphide ores
and oxidized ores.
Anticipated grades of concentrates from sulphide
ores and oxidized ores.

Our conclusions are as follows:

1. Grind requirements for design purposes should be 50% minus 200 mesh, corresponding to 80% minus 250 microns.

The above specification was reported in CSIRO Progress Report No. 2. However, the pilot plant achieved acceptable metallurgical results using a very coarse grind of 80% minus 250 microns which is probably too coarse for a continuous operation, which may select a grind between these limits. Reported determinations of the Work Index have varied from 14.9 to 22.2. A mean of 18.5 has been used in checking power requirements.

2. Flotation reagents and rates of consumption are as follows:

| | |
|---|---|
| Lime | 1.2 lbs/ton ore |
| Xanthate | 0.12 lbs/ton ore |
| Aerofloat | 0.06 lbs/ton ore |
| Sodium silicate | 0.10 lbs/ton ore |

CSIRO Progress Report No. 4 reduced reagent rates to these levels without apparent detrimental effects. The pilot plant consumed three times these rates probably because of the highly oxidized state of the ore. Freshly mined ore and recirculation of water should reduce rates to those tabulated above.

Test No. 408 in CSIRO Progress Report No. 4 indicates the sulphide mineral to be comparatively fast and free floating and conditioning does not appear warranted.

3. Recovery of the copper as sulphide minerals is predicted to be 94% with a corresponding 85% recovery of silver.

Recovery of the copper oxide minerals should be considered nil when treating such minerals by the copper sulphide beneficiation process.

Recovery of the copper as oxidized minerals by the sulphidization flotation process is predicted to be 53%.

A 2% loss in recovery is expected during the first year of treating sulphide ore and a 3% loss in recovery is expected when oxide ore is first treated during the eighth year of plant operation.

4. Estimated grades of sulphide concentrates and oxide concentrates are 57% copper and 35% copper, respectively.

The above conclusions are based on the following, as indicated by the metallurgical test reports submitted to us for study:

1. The recovery of the sulphide minerals is not affected by the presence of oxidized minerals.

2. Mineral particles are completely liberated at recommended grinds and no regrind stage is necessary.

3. Optimum pH of flotation is 10.

4. CSIRO Progress Reports Nos. 1, 2, and 3, in which sulphide ore from the Lagoon and House sulphide areas were tested, achieved:

93% Cu recovery with .62% Cu heads.
95% Cu recovery with 1.06% Cu heads.
96% Cu recovery with 1.59% Cu heads.

The pilot plant 24 hour test run recovered 96.6% of the sulphide Cu from 1.11% of sulphide Cu heads.

The CSIRO tests did not include cleaning flotation in all cases, but final grades were obtained, assaying 52.4% to 64.7% Cu on heads of 0.6 to 1.59% Cu.

The pilot plant reported 55.7% Cu concentrates.

Silver recovery should follow the copper recovery and both CSIRO (Test No. 411) and the pilot plant reported 85% recovery of silver.

5. Sulphidizing with sodium sulphide is effective in recovering a portion of the copper carbonate minerals. It is doubtful if copper as chrysocolla (copper silicates) can be recovered by this method, and the degree to which it is present will affect the ultimate metallurgy.

These effects were illustrated in CSIRO Progress Reports Nos. 1 and 2, in which copper recoveries ranged from a low of 17% to a high of 72%.

Amdel in testwork conducted in 1967 reported 60% recoveries with concentrate grades which varied from 25% copper to a maximum of 35% copper.

The pilot plant, which was experiencing considerable circuit difficulties, produced a copper concentrate grading 38% copper with 45% to 47% recovery. Results indicated that the recovery would have exceeded 50% if the test programme had been continued.

## TABLE 3 SUMMARY OF ESTIMATED DIRECT OPERATING COSTS

## CRUSHING AND MILLING

| | ¢ Per Ton Milled | |
|---|---|---|
| | Sulphide Ore | Oxide Ore |
| Direct Labour and Supervision | 22.7 | 22.7 |
| Power | 31.5 | 31.5 |
| Water | 18.1 | 18.1 |
| Maintenance | 24.1 | 24.1 |
| Operating Supplies | 31.0 | 52.0 |
| Concentrate Shipping | 3.8 | 3.8 |
| COST PER TON MILLED | $ 1.31 | $ 1.52 |

## NOTES ON ESTIMATED OPERATING COSTS

## CRUSHING AND MILLING

### Labour and Supervision

The labour costs as estimated in the McKee - I.E.L. Report No. 102-1, dated September 1967, have been reviewed and we believe these estimates to be basically correct. Crushing plant labour should be reduced because our proposed design permits the crushing plant to be operated by the grinding operator. However, we recommend no adjustment to the labour cost distribution to allow for a helper operator in training.

### Power

Our estimate for power consumption totals 21 KWH/ton ore, from an installed horsepower of 1675 HP. This includes a 160 K.W. infra-red dryer installation. A unit rate of 1.50¢ per K.W.H. has been calculated from basic rates as assessed by the Department of Supply, Feasibility Study, December, 1967.

### Water

Our study of the process, estimates that fresh water make up requirements will be 136 g.p.m. For computing operating costs, 150 g.p.m. has been used at a unit rate of 83.6¢ per 1,000 gallons, which was estimated by the Department of Supply, Feasibility Study, December, 1967.

### Maintenance

The breakdown of costs for maintenance as submitted by McKee - I.E.L. Pty. Ltd., has been examined and accepted as a realistic estimate. Our opinion is that the maintenance labour crew is rather high for this size of operation and maintance costs can be expected to decrease as preventative maintenance is put into effect.

## Operating Supplies

Steel consumption for crusher and mill liners have been estimated at 0.055 lbs/ton and 0.15 lbs/ton ore respectively, and 1.5 lbs/ton for grinding media. Using 30¢ per pound for liner steel and 10¢ per pound for balls, this represents a cost per ton ore of 22¢.

A review of the metallurgical test work confirms reagent consumptions as quoted by Chapman, Wood, Griswold and Evans Pty. Ltd., amounting to 9¢ and 30¢ for the sulphide ore respectively.

## Concentrate Shipping

This cost covers the rental charges for 5 ton concentrate containers, verbally expressed by Chapman, Wood, Griswold and Evans Pty. Ltd., as $210.00 per year per $1,000.00 capital cost. It is estimated that 240 containers costing $250.00 each, would be in use, consequently rental charges are based on a capital cost of $60,000.

## TABLE 4 SUMMARY OF INDIRECT OPERATING COSTS

Reference: Chapman, Wood, Griswold and Evans Pty. Ltd.
Evaluation Report, January, 1968.

| | ¢ Per Ton Milled |
|---|---|
| Ore Delineation | 8 |
| * Project Supervision | 29 |
| Vehicles and Incidentals | 6 |
| Housing and Transportation | 11 |
| Refurbishments | 21 |
| Insurance | 9 |
| Offsite Administration | 9 |
| + Miscellaneous | 10 |
| Total Indirect Operating Cost Per Ton Milled | A $ 1.03 |

\+ Estimated by Wright Engineers Limited to allow for costs of a general nature which are not included in above costs.

* This item has been increased by 4¢ as a result of indications that salary estimates for supervisory personnel are too low, which have been revised as follows:

| | A $/Month |
|---|---|
| Mine Manager | 1,250 |
| Mill Superintendent | 1,000 |
| Mechanical Superintendent | 750 |
| Chief Engineer | 750 |
| Geologist | 700 |
| Survey Helper | 400 |
| Assayer | 450 |
| Warehouseman | 350 |
| Clerks (2) | 600 |
| Total | 6,250 |
| Plus Indirect Charges @ 30% | 1,870 |
| Supervisory Labour Cost | $ 8,120 |

Per Ton Milled $$\$\ \frac{8,120 \times 12 \times 100}{330,000} = 29¢$$

## Ore Reserves

Ore reserves have been computed by Mr. E. Macdonald from the assays of percussion drill holes which have been generally drilled at 100 foot spacing on grid lines at 200 foot intervals along the orebodies.

Ore reserves have been grouped into two categories, in Table 5, depending on the degree of accuracy by which the computed tonnages and grades from drill hole assays are judged to reflect the actual tonnage and grade in each orebody. The measured ore category corresponds to ore which has been drilled at sufficiently close spacing and consistency of assay results so that the computed grade is judged to be within 5% of the actual grade. The indicated ore category corresponds to ore which is indicated partly by drilling and partly by reasonable projection on geological evidence beyond the limits of measured ore.

No ore reserve is shown for inferred ore where the continuity of ore may be assumed from broad geological evidence only and for which there are no specific physical evidence or assays confirming the existence of ore.

The measured and indicated ore reserves in each orebody are also classified in terms of the predominant copper mineralization. Ore which consists chiefly of copper sulphides which are readily amenable to flotation recovery is classified as sulphide ore. The grade of copper in these orebodies is computed from the assays of copper in the sulphide minerals only. Copper which is present as oxides or carbonate minerals is ignored because this copper content will not be readily recovered in the flotation process to be used during the first seven years of mill operation.

Ores which consist predominantly of copper oxides or copper carbonate minerals are classified as oxide ores. Because these minerals will require sulphidizing prior to treatment in the flotation process, the copper grade of these ores has been estimated from the combined copper content of copper oxide, copper carbonate and copper sulphide minerals.

The minimum cut-off grade of increments of ore that can be treated profitably was assumed to be as follows:

Oxidized ore containing at least 0.6% copper as copper oxide, copper sulphide, or copper carbonate.

Sulphide ore containing at least 0.4% copper as copper sulphide and typically overlaid by waste rock and oxidized ore.

Some variations in cut-off grades of increments of ore were accepted for inclusion in ore reserves when the increments of ore were adjoining higher grade ore, or when the amount of overlying waste and oxidized ore was low.

Surrounding each orebody is a layer of low grade waste and in mining practice it will be difficult to exclude some of this low grade waste from the orebody. We have assumed that where the outline of the orebodies merge with adjacent low grade waste, a 2 foot thick layer of low grade waste will be mined along with the ore for delivery to the mill.

# TABLE 5 MEASURED AND INDICATED ORE RESERVES

| | UNDILUTED ORE | | | DILUTION MATERIAL | | | | |
|---|---|---|---|---|---|---|---|---|
| Location | '000 tons | Cu % | Ag oz/ton | '000 tons | Cu % | Ag oz/ton | Tons Waste '000 | |
| **Lagoon E Sulphide** | | | | | | | | |
| Measured | 111 | 1.75 | 0.80 | | | | | |
| Indicated | 43 | 1.75 | 0.80 | | | | | |
| Sub-Total | 154 | 1.75 | 0.80 | 59 | 0.19 | 0.09 | 228 | |
| **Lagoon W Sulphide** | | | | | | | | |
| Measured | 456 | 1.35 | 0.65 | | | | | |
| Indicated | 90 | 1.35 | 0.65 | | | | | |
| Sub-Total | 546 | 1.35 | 0.65 | 193 | 0.15 | 0.08 | 548 | |
| **House Oxide** | | | | | | | | |
| Measured | 80 | 1.00 | - | | | | | |
| Indicated | 25 | 0.78 | - | | | | | |
| Sub-Total | 105 | 0.95 | - | 21 | 0.18 | - | 150 | |
| **House Sulphide** | | | | | | | | |
| Measured | 536 | 0.97 | 0.50 | | | | | |
| Indicated | 48 | 0.85 | 0.45 | | | | | |
| Sub-Total | 584 | 0.96 | 0.49 | 119 | 0.18 | 0.09 | 830 | |
| **Main Open Cut Oxide** | | | | | | | | |
| Measured | 536 | 1.65 | - | | | | | |
| Indicated | 0 | - | - | | | | | |
| Sub-Total | 536 | 1.65 | - | 157 | 0.19 | - | 1,088 | 2.0 |
| **Main Open Cut Sulphide** | | | | | | | | |
| Measured | 401 | 1.12 | 0.55 | | | | | |
| Indicated | 125 | 0.79 | 0.40 | | | | | |
| Sub-Total | 526 | 1.04 | 0.44 | 118 | 0.19 | 0.09 | 1,000 | 1.90 |
| **Gunyot Oxide** | | | | | | | | |
| Measured | 76 | 1.35 | - | | | | | |
| Indicated | 10 | 1.00 | - | | | | | |
| Sub-Total | 86 | 1.31 | - | 16 | 0.20 | - | 186 | |
| Total Measured Oxide | 692 | 1.53 | - | | | | | |
| Indicated Oxide | 35 | 0.85 | - | | | | | |
| Total Measured and Indicated Oxide | 727 | 1.49 | - | 194 | 0.19 | - | 1,424 | |
| Total Measured Sulphide | 1,504 | 1.18 | 0.58 | | | | | |
| Indicated Sulphide | 306 | 1.10 | 0.52 | | | | | |
| Total Measured and Indicated Sulphide | 1,810 | 1.17 | 0.55 | 489 | 0.17 | 0.08 | 2,606 | |
| Total Oxide and Sulphide | 2,537 | 1.26 | 0.39 | 683 | 0.18 | 0.06 | 4,030 | |

The diluted ore delivered to the mill, as shown in Table 6, will be greater in overall tonnage and lower in grade than the outlined ore in the measured and indicated categories.

## Sequence and Cost of Mining

Table 7 shows the proposed sequence of waste stripping, stockpiling of oxide ore and mining of sulphide ore during the 10 years of production. The overlying waste rock and oxide ore must be removed and dumped or stockpiled in advance of mining of sulphide ore, consequently there will be some lead time between removal of waste and mining the underlying sulphide ore. This lead time may vary depending on the shape and slope of the waste benches, but in general, an average of 4 months lead time has been assumed in the schedule shown in the Table. The tonnage of waste to be stripped has been estimated on the basis of a final pit wall slope of 60°.

The cost of mining ore and waste rock has been based on the costs quoted by Davis Contractors on July 3, 1968: 38¢ per short ton for stripping overburden and 58¢ per short ton for mining and hauling ore to the mill. The quote by Davis Contractors was based on an estimated total tonnage of ore and waste which was 14% more than our estimated tonnage and which assumed that maintenance facilities for earth moving equipment would be provided by Mount Gunson Mines.

We recommend that the contractor should provide his own maintenance facilities and our estimate of capital cost does not include maintenance facilities for the contractors earth moving equipment. We consider therefore, that the appropriate costs for stripping overburden should be 40¢ per short ton and the cost of mining and hauling ore should be 60¢ per ton on a contract basis if the contractor supplies his own maintenance facilities and the contract is based on our estimated smaller total tonnage of ore and waste. We have estimated that the cost of reclaiming and hauling oxide ore from stockpiles which will be located adjacent to each of the main orebodies will be approximately 25¢ per ton.

Table 8 shows the estimated mining cost by years. This table is based on the schedule of mining ore and waste shown in Table 7 and the estimated unit costs of mining ore and waste.

It will be noted that mining cost is maintained at a moderate level during the first three years when the easily accessible sulphide ore is being mined from the Lagoon East and Lagoon West orebodies. After the third year, the mining cost increases sharply because of the relatively large amounts of waste and oxide ore that must be removed from the House orebody and the main open cut orebody to prepare these orebodies for mining of sulphide ore. By the end of the seventh year, most of the waste will be removed and the oxide ore will be stockpiled for milling during the eighth, ninth and tenth years.

## TABLE 6 ORE RESERVES (AS MINED)

| Location | Tons | Cu % | Ag oz/ton | Copper Content (Lbs.) | Silver Content (Oz.) | Tons Waste | Strip Ratio |
|---|---|---|---|---|---|---|---|
| **Lagoon** | | | | | | | |
| East Sulphide Ore | 213,000 | 1.35 | 0.60 | 5,610,000 | 128,000 | 228,000 | 1.07 |
| West Sulphide Ore | 739,000 | 1.04 | 0.50 | 15,320,000 | 370,000 | 548,000 | 0.74 |
| Sub-Total Lagoon | 952,000 | 1.19 | 0.52 | 20,930,000 | 498,000 | 776,000 | 0.82 |
| **House** | | | | | | | |
| Oxide Ore | 126,000 | 0.82 | - | 2,060,000 | - | 150,000 | 1.19 |
| Sulphide Ore | 703,000 | 0.83 | 0.43 | 11,660,000 | 302,000 | 830,000 | 1.18 |
| Sub-Total House | 829,000 | 0.83 | 0.37 | 13,720,000 | 302,000 | 980,000 | 1.18 |
| **Main Open Cut** | | | | | | | |
| Oxide Ore | 693,000 | 1.31 | - | 18,280,000 | - | 1,088,000 | 1.57 |
| Sulphide Ore | 644,000 | 0.88 | 0.37 | 11,390,000 | 241,000 | 1,000,000 | 1.55 |
| Sub-Total M.O.C. | 1,337,000 | 1.11 | 0.18 | 29,670,000 | 241,000 | 2,088,000 | 1.56 |
| **Gunyot** | | | | | | | |
| Oxide Ore | 102,000 | 1.13 | - | 2,320,000 | - | 186,000 | 1.82 |
| Total Oxide Ore | 921,000 | 1.21 | - | 22,660,000 | - | 1,424,000 | 1.55 |
| Total Sulphide Ore | 2,299,000 | 0.96 | 0.43 | 43,980,000 | 1,041,000 | 2,606,000 | 1.13 |
| Total Oxide and Sulphide | 3,220,000 | 1.04 | 0.32 | 66,640,000 | 1,041,000 | 4,030,000 | 1.25 |

## TABLE 7 SCHEDULE OF ORE AND WASTE MINED

### Tons (Short) Mined by Years and Location

| Year | Ore Mined & Milled | Ore Mined & Stockpiled | Waste Mined & Dumped | Ore Reclaimed From Stockpile |
|---|---|---|---|---|
| 0 (Preproduction) | - | - | 100,000 (LE ) | |
| 1 | 213,000 (LE )<br>77,000 (LW ) | -<br>- | 128,000 (LE )<br>200,000 (LW ) | -<br>- |
| 2 | 330,000 (LW ) | - | 348,000 (LW) | - |
| 3 | 332,000 (LW ) | 126,000 (HO ) | 150,000 (HO )<br>74,000 (HS ) | - |
| 4 | 330,000 (HS ) | | 756,000 (HS )<br>44,000 (MOO) | - |
| 5 | 330,000 (HS) | 350,000 (MOO) | 450,000 (MOO) | - |
| 6 | 43,000 (HS)<br>287,000 (MOS) | 100,000 (MOO) | 594,000 (MOO)<br>106,000 (MOS) | - |
| 7 | 330,000 (MOS) | - | 800,000 (MOS) | - |
| 8 | 27,000 (MOS)<br>243,000 (MOO) | - | 94,000 (MOS) | 60,000 |
| 9 | - | - | - | 330,000 |
| 10 | 102,000 (GO ) | - | 186,000 (GO ) | 186,000 |
| Total | 2,299,000 (Sulphide)<br>345,000 (Oxide) | | | |
| | 2,644,000 | 576,000 | 3,930,000 | 576,000 |

Abbreviations for Orebodies:

LE = Lagoon East
HO = House Oxide
MOO = Main Open Cut Oxide
GO = Gunyot Oxide

LW = Lagoon West
HS = House Sulphide
MOS = Main Open Cut Sulphide

## TABLE 8 MINING COST BY YEARS

| Year | Cost of Mining Ore @ 60¢ | Cost of Stockpiling @ 40¢ | Cost of Waste Mined @ 40¢ | Cost of Reclaiming Ore @ 25¢ | Total Cost |
|---|---|---|---|---|---|
| 0 Preproduction | - | - | (40,000+25%) | | |
| 1 | 174,200<br>(290,000 tons) | - | 131,300 | - | 305,500 |
| 2 | 198,000<br>(330,000 tons) | - | 139,000 | - | 337,000 |
| 3 | 199,000<br>(332,000 tons) | 50,400 | 89,600 | - | 339,000 |
| 4 | 198,000<br>(330,000 tons) | | 320,000 | - | 518,000 |
| 5 | 198,000<br>(330,000 tons) | 140,000 | 180,000 | - | 518,000 |
| 6 | 198,000<br>(330,000 tons) | 40,000 | 280,000 | - | 518,000 |
| 7 | 198,000<br>(330,000 tons) | - | 320,000 | - | 518,000 |
| 8 | 162,200<br>(270,000 tons) | - | 37,600 | 15,000<br>( 60,000 tons) | 214,800 |
| 9 | - | - | - | 82,500<br>(330,000 tons) | 82,500 |
| 10 | 61,200<br>(102,000 tons) | - | 74,400 | 46,500<br>(186,000 tons) | 182,100 |
| Total | $ 1,586,600 | $ 230,400 | $ 1,571,900 | $ 144,000 | $ 3,532,900 |

## Production of Concentrates

Table 9 shows the grade of ore which will be treated during each year of production. It should be noted that the mill will be treating only sulphide ore during the first seven years and a small amount of sulphide ore in the eighth year. During the eighth year, the milling plant will be modified to treat oxide ore which will constitute most of the ore treated during the eighth and subsequent years. Although the oxide ore treated during the eighth, ninth and tenth years will be somewhat higher in copper grade than the sulphide ore treated during the first seven years, the expected recovery of copper from the oxide ore will be much lower than from the sulphide ore.

Table 10 shows the expected metal content of concentrates produced during the ten years of mill operation. Our review of metallurgical tests of copper ores from the Mount Gunson area indicates that 94% of the copper content and 85% of the silver content, should be recovered from the sulphide ores by the proposed mill treatment.

Copper recovery is expected to be only 53% when oxide ore is being treated. During the first year of treating sulphide ore and subsequently when treating the initial input of oxide ore, difficulties in adjusting mill operation may be expected to result in reduced recoveries of 92% for sulphide ore and 50% for oxide ore.

The expected grades of copper concentrate will be 57% copper when sulphide ores are treated and 35% copper when oxide ores are being treated. Although higher grades of copper concentrate have been attained in metallurgical test work, we do not expect that these ideal conditions will be maintained in milling operations.

## Revenue from sale of Concentrates

Table 11 shows our computation of smelter returns from copper concentrates grading 57% copper and 35% copper. The copper revenue has been based on copper prices of 40¢ (U.S.) and 38¢ (U.S.) per pound. We expect that a reasonable projection of future copper prices would be 40¢ per pound for the first two years of operation, thereafter 38¢ per pound. It is apparent that a copper price of higher than 40¢ per pound will significantly improve metal revenue, but we consider our projected copper price is suitably conservative for the purpose of estimating project feasibility.

Silver revenue has been based on an assumed silver price of $2.10 (U.S.) per ounce. We consider this assumed price is a reasonable projection of future silver prices.

After discussion with Mr. J. B. Evans on feasible methods of shipping copper concentrates in containers which can be unloaded at Port Pirie, the following freight costs have been estimated:

Contract trucking of containers 6-1/2 miles between the plant site and the railroad: $10,000 annually for an average of 5,290 tons annually = cost per short ton of $1.89.

Railroad freight between siding and Port Pirie = \$3.76 per short ton, plus return of empty containers @ 10% of \$3.76 = \$4.14 per short ton.

Shiploading at Port Pirie is estimated to be \$2.00 per short ton.

Cost of ocean shipping of concentrate to a copper smelter, including insurance will be about \$10.00 per short ton. In summary, the total freight cost per short ton of concentrate will be approximately (A) \$18.00.

The smelting charge of \$16.00 (U.S.) per long ton of dry solids and the copper revenue has been estimated from the smelter terms quoted by the British Metal Corporation in their letter of May 31, 1968. It should be noted, however, that final terms of a smelter contract may differ somewhat for the carbonate concentrates, and that an escalation clause on smelter charges may be established. For the purposes of feasibility estimation, however, modification of smelter terms would have only a minor effect on revenue, which would be insignificant compared with fluctuations in copper prices which would normally be expected to parallel changes in smelting costs.

The appended table shows the smelter revenue, smelting costs, freight costs and operating costs annually over the 10 year period of production. The figures shown are plotted graphically in sketch No. 496-001-1 to show the yearly changes in operating profit. It will be noted that the operating profit is about \$1,000,000 annually for the first three years, then drops to about \$400,000 for the next four years, and drops to about \$150,000 for the last three years of operation. The profit is high for the first three years chiefly because the grade of mill feed is high and because a 40¢ copper price is expected for the first two years. The lower profit level for the succeeding four years is due chiefly to the lower grade of mill feed and more costly mining when the House and the Main open cut sulphide orebodies are being mined. Profit is lowest during the final three years of operation because of low recovery from treatment of oxide copper. It should be noted that an additional capital expenditure of \$18,000 will be incurred at the beginning of the eighth year when the process plant will be modified for treatment of oxidized copper ores.

## TABLE 9 GRADE OF MILL FEED BY YEARS

| Year | Short Tons Milled | % Cu | Ag oz/ton | Cu Content in Lbs. | Ag Content in Oz. |
|---|---|---|---|---|---|
| 1 | 290,000 | 1.24 | 0.58 | 7,210,000 | 167,600 |
| 2 | 330,000 | 1.04 | 0.50 | 6,850,000 | 165,000 |
| 3 | 332,000 | 1.04 | 0.50 | 6,870,000 | 165,400 |
| 4 | 330,000 | 0.83 | 0.43 | 5,460,000 | 142,000 |
| 5 | 330,000 | 0.83 | 0.43 | 5,460,000 | 142,000 |
| 6 | 330,000 | 0.88 | 0.38 | 5,820,000 | 126,000 |
| 7 | 330,000 | 0.88 | 0.37 | 5,830,000 | 123,000 |
| 8 (Sulphide) | 27,000 | 0.89 | 0.37 | 480,000 | 10,000 |
| (Oxide) | 303,000 | 1.24 | - | 7,540,000 | - |
| Total | 330,000 | 1.22 | 0.03 | 8,020,000 | 10,000 |
| 9 (Oxide) | 330,000 | 1.24 | - | 8,200,000 | - |
| 10 (Oxide) | 288,000 | 1.20 | - | 6,920,000 | - |
| Total (Sulphide) | 2,299,000 | 0.96 | 0.45 | 43,980,000 | 1,041,000 |
| Total (Oxide) | 921,000 | 1.23 | - | 22,660,000 | - |
| GRAND TOTAL | 3,220,000 | 1.03 | 0.32 | 66,640,000 | 1,041,000 |

## TABLE 10 METAL CONTENT IN CONCENTRATE BY YEARS

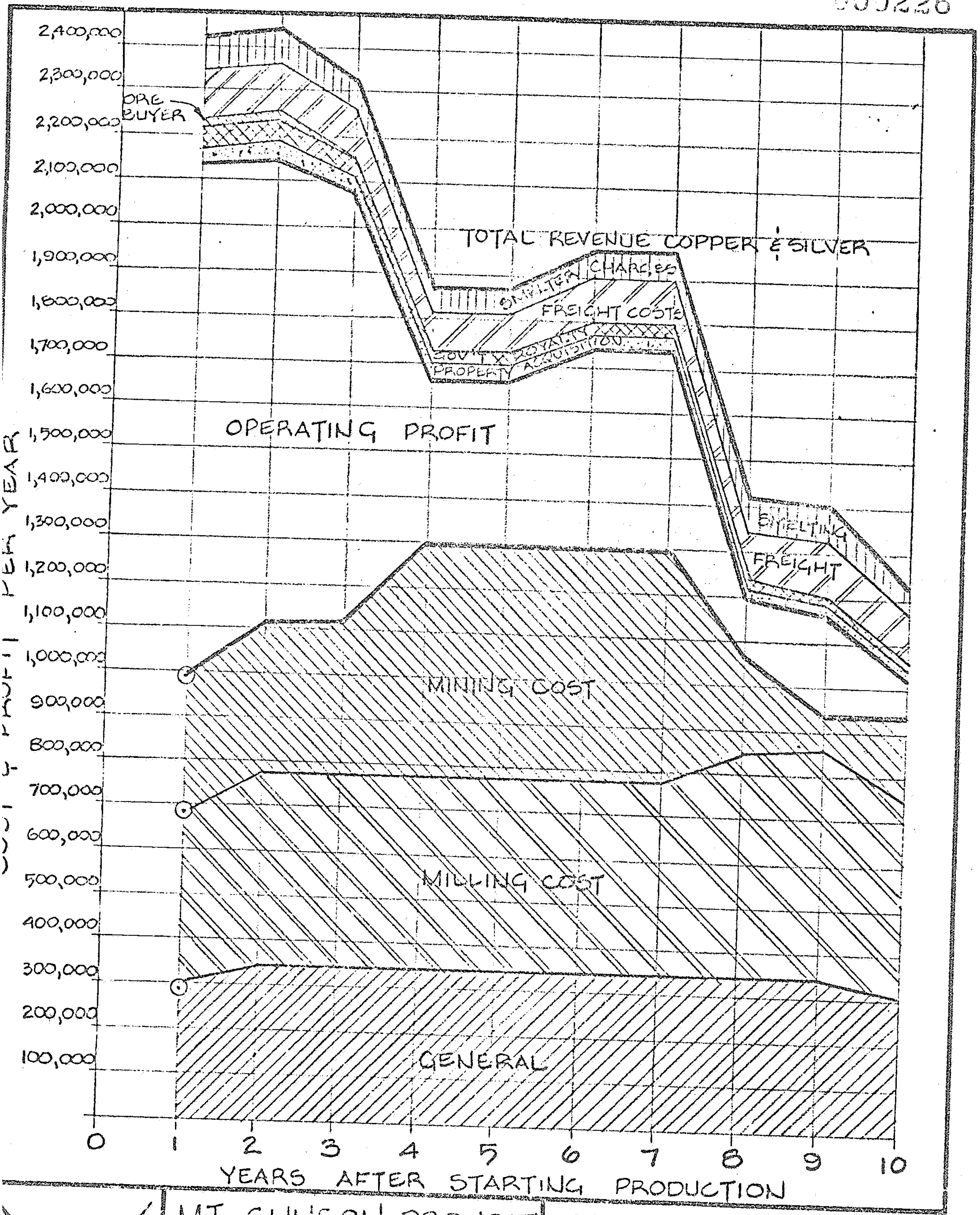
| Year | Dry Short Tons of Concentrate | Recovery of Copper | Recovered Copper in Lbs. | Recovery of Silver | Recovered Silver in Oz. |
|---|---|---|---|---|---|
| 1 | 5,820<br>(57% Cu, 24.0 oz. Ag) | 92% | 6,640,000 | 83% | 139,000 |
| 2 | 5,650<br>(57% Cu, 24.8 oz. Ag) | 94% | 6,440,000 | 85% | 140,300 |
| 3 | 5,670<br>(57% Cu, 24.8 oz. Ag) | 94% | 6,460,000 | 85% | 140,700 |
| 4 | 4,500<br>(57% Cu, 26.8 oz. Ag) | 94% | 5,130,000 | 85% | 120,900 |
| 5 | 4,500<br>(57% Cu, 26.8 oz. Ag) | 94% | 5,130,000 | 85% | 120,900 |
| 6 | 4,800<br>(57% Cu, 22.2 oz. Ag) | 94% | 5,470,000 | 85% | 107,000 |
| 7 | 4,810<br>(57% Cu, 21.8 oz. Ag) | 94% | 5,480,000 | 85% | 104,600 |
| 8 (Sulphide) | 400 | 94% | 450,000 | 85% | 8,500 |
| (Oxide) | 5,390 | 50% | 3,770,000 | - | - |
| (Sub-Total) | 5,790 | | 4,220,000 | - | 8,500 |
| 9 | 6,200<br>(35% Cu) | 53% | 4,340,000 | - | - |
| 10 | 5,250 | 53% | 3,670,000 | - | - |
| Total (Sulphide) | 36,150<br>(57% Cu, 24.4 oz. Ag) | | 41,200,000 | | 881,900 |
| Total (Oxide) | 16,840<br>(35% Cu) | | 11,780,000 | | - |
| Total (Oxide and Sulphide) | 52,990 | | 52,980,000 | | 881,900 |

## TABLE 11 CALCULATION OF SMELTER RETURN

(Based On One Short Dry Ton at Mill)

| | | | |
|---|---|---|---|
| Concentrate grade | 57% | 57% | 35% |
| Assumed copper price in U.S. ¢ per lb. | 40¢ | 38¢ | 38¢ |
| Weight of dry short ton at mill in lbs. | | 2,000.0 | |
| Weight of short dry ton (moistened to 6%) in lbs. | | 2,127.66 | |
| Estimated freight cost per short ton (in A $) | | $18.00 | |
| Freight cost per 2,127.66 lbs. (in A $) | | $19.149 | |
| Weight of copper per dry short ton at mill | 1,140 lbs. | 1,140 lbs. | 700 lbs. |
| Freight cost per lb. of copper shipped (in A ¢) 1.777¢ | 1.680¢ | 1.680¢ | 2.736¢ |
| Dusting loss en route to smelter | | 0.25% | |
| Weight of dry concentrate at smelter | | 1,995.0 lbs. | |
| Smelting charge per long ton (in $ U.S.) | | $16.00 | |
| Smelting charge per 1,995 lbs. dry concentrate (in $ U.S.) | | $14.250 | |
| Smelting charge per 1,995 lbs. dry concentrate (in A $) | | $12.723 | |
| Smelting charge per lb. of copper shipped (in A ¢) 0.946¢ | 1.116¢ | 1.116¢ | 1.818¢ |
| Weight of copper per 1,995 lbs. dry concentrate (in lbs.) | 1,137.15 | 1,137.15 | 698.25 |
| Less 1% of dry concentrate weight at smelter (in lbs.) | -19.95 | -19.95 | -19.95 |
| Weight of copper actually paid for | 1,117.20 | 1,117.20 | 678.30 |
| Copper paid at copper price less 1¢ (U.S.) | 39¢ (U.S.) | 37¢ (U.S.) | 37¢ (U.S.) |
| Copper revenue per dry short ton shipped (in $ U.S.) | $435.71 | $413.36 | $250.97 |
| Copper revenue per dry short ton shipped (in A $) | $389.03 | $369.07 | $224.08 |
| Copper revenue per lb. copper shipped (in A ¢) | 34.125¢ | 32.375¢ | 32.012¢ |
| Difference in copper revenue @ 40¢ and 38¢ Cu. price | 1.750¢ | | |
| Ore Buyer (@ 20% difference in revenue) 0.22 | 0.350¢ | | |
| Assumed silver price (in $ U.S. per oz.) | $2.10 | $2.10 | - |
| Silver revenue @ 90% (in A $ per oz.) | $1.688 | $1.688 | - |
| Copper price in A ¢ per lb. | 35.71¢ | 33.93¢ | 33.93¢ |
| Property acquisition @ 1.5% Cu. price (in A ¢/lb.) | 0.535¢ | 0.509¢ | 0.509¢ |

Deductns 2.95



MT. GUNSON PROJECT
SOUTH AUSTRALIA

OPERATING COSTS
REVENUE & PROFIT
BY YEARS

**WRIGHT ENGINEERS LIMITED**
VANCOUVER — CANADA

DRAWN BY J.M.
DATE 9/9/68

DRAWING NO. 496 001 1

061-WE

## TABLE METAL REVENUE AND OPERATING COSTS BY YEARS

(Australian currency unless otherwise noted)

| | Year 1 | Year 2 | Year 3 | Year 4 | Year 5 | Year 6 | Year 7 | Year 8 | | Year 9 | Year 10 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | | | Sulphide | Oxide | | |
| Lbs. copper produced | 6,640,000 | 6,440,000 | 6,460,000 | 5,130,000 | 5,130,000 | 5,470,000 | 5,480,000 | 450,000 | 3,770,000 | 4,340,000 | 3,670,000 |
| Lbs. copper deferred | - 230,000 | | | | | | | +230,000 | - 140,000 | | + 140,000 |
| Lbs. copper paid for | 6,410,000 | 6,440,000 | 6,460,000 | 5,130,000 | 5,130,000 | 5,470,000 | 5,480,000 | 680,000 | 3,630,000 | 4,340,000 | 3,810,000 |
| Oz. silver produced | 139,000 | 140,300 | 140,700 | 120,900 | 120,900 | 107,000 | 104,600 | 8,500 | - | - | - |
| Oz. silver deferred | - 5,000 | | | | | | | +5,000 | | | |
| Oz. silver paid for | 134,000 | 140,300 | 140,700 | 120,900 | 120,900 | 107,000 | 104,600 | 13,500 | - | - | - |
| Copp price/lb. (in ¢ U.S.) | 40¢ | 40¢ | 38¢ | 38¢ | 38¢ | 38¢ | 38¢ | 38¢ | 38¢ | 38¢ | 38¢ |
| Copper revenue/lb. (in A ¢ ) | 34.125 | 34.125 | 32.375 | 32.375 | 32.375 | 32.375 | 32.375 | 32.375 | 32.012 | 32.012 | 32.012 |
| Silver revenue/oz. (in A $ ) | 1.688 | 1.688 | 1.688 | 1.688 | 1.688 | 1.688 | 1.688 | 1.688 | - | - | - |
| Copper revenue (in A $) | 2,187,400 | 2,197,600 | 2,091,400 | 1,660,800 | 1,660,800 | 1,770,900 | 1,774,100 | 1,382,200 | | 1,389,300 | 1,219,700 |
| Silver revenue | 226,200 | 236,800 | 237,500 | 204,100 | 204,100 | 180,600 | 176,600 | 22,800 | | - | - |
| Smelting costs | 71,500 | 71,900 | 72,100 | 57,300 | 57,300 | 61,000 | 61,200 | 73,600 | | 78,900 | 69,300 |
| Freight costs | 107,700 | 108,200 | 108,500 | 86,200 | 86,200 | 91,900 | 92,100 | 110,700 | | 118,700 | 104,200 |
| Ore buyer payment | 22,400 | 22,500 | - | - | - | - | - | - | | - | - |
| Milling costs | 379,900 | 432,300 | 434,900 | 432,300 | 432,300 | 432,300 | 432,300 | 495,900 | | 501,600 | 437,800 |
| Net process revenue (N.P.R.) | 1,832,100 | 1,799,500 | 1,713,400 | 1,289,100 | 1,289,100 | 1,366,300 | 1,365,100 | 724,800 | | 690,100 | 608,400 |
| Government Royalty @ 2.5% N.P.R. | 45,800 | 45,000 | 42,800 | 32,200 | 32,200 | 34,200 | 34,100 | 18,100 | | 17,300 | 15,200 |
| Property acquisition | 34,300 | 34,500 | 32,900 | 26,100 | 26,100 | 27,800 | 27,900 | 21,900 | | 22,100 | 19,400 |
| Net revenue | 1,752,000 | 1,720,000 | 1,637,700 | 1,230,800 | 1,230,800 | 1,304,300 | 1,303,100 | 684,800 | | 650,700 | 573,800 |
| Mining costs | 305,500 | 337,000 | 339,00 | 518,000 | 518,000 | 518,000 | 518,000 | 214,800 | | 82,500 | 182,100 |
| General and Administrative costs | 298,700 | 339,900 | 339,900 | 339,900 | 339,900 | 339,900 | 339,900 | 339,900 | | 339,900 | 296,600 |
| Net Operating Profit | 1,147,800 | 1,043,100 | 958,800 | 372,900 | 372,900 | 446,400 | 445,200 | 130,100 | | 228,300 | 95,100 |

Total Operating Profit Over 10 Yrs. 5,240,600

SUBMITTED BY:

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VANCOUVER, BRITISH COLUMBIA.
SEPTEMBER, 1968.

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# C.S.R. RESEARCH LABORATORIES

## CONTRACT REPORT NO. 15

for

Mount Gunson Mines Pty. Limited

# THERMOANALYTICAL INVESTIGATION OF A SAMPLE OF MOUNT GUNSON COPPER CONCENTRATE

by

Mr. S.J. Craven
Dr. E.P. Crematy

Copy No. 4

May, 1970.

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## 1. SCOPE OF INVESTIGATION

The investigation consists only of the study of the oxidation of a single sample of copper concentrate stated by Mt. Gunson Mines Pty. Limited to have been supplied from its mine at Mount Gunson, South Australia.

## 2. INTRODUCTION

Mount Gunson Mines Pty. Limited requested (letter, 23rd March, 1970) that these Laboratories investigate the oxidation of Mt. Gunson copper (sulphides) concentrate. The purpose of this study was to ascertain whether the concentrate would oxidise vigorously under conditions likely to be experienced during transport (in standard 10' x 8' x 8' stainless steel containers) to smelters in Japan.

Sulphide concentrates are known to self heat as a result of oxidation - with the liberation of toxic sulphur dioxide gas. Disasters at sea have occurred as a result of such oxidation during transport[1,2]. The iron-containing concentrates are the most susceptible to oxidation, with the copper (low-level iron) sulphides reported to be amongst the most stable*.

The tendency of the more common concentrates towards oxidation decreases in the following order[1,6,9]:

Pyrrhotite (FeS) >arsenopyrite (FeAsS) >pyrite ($FeS_2$), marcasite ($FeS_2$) >chalcopytite ($CuFeS_2$) >sphalerite (Zns)> galena (PbS) >chalcocite ($Cu_2S$), covellite (Cus).

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*It has been reported[3] that pyrite ($FeS_2$) commences to oxidise at 50°C, whilst chalcocite ($Cu_2S$) is not significantly oxidized at temperatures less than 300°C[4,8].

The predominant sulphides in the sample of the Mt. Gunson concentrate supplied to the Research Laboratories are digenite ($Cu_{1.8}S$), covellite ($CuS$) and djurleite ($Cu_{1.96}S$). Sphalerite ($ZnS$), quartz and a small amount ($\sim$ 2%) of pyrite are also present. The oxidation of digenite and djurleite was expected to be comparable to that of chalcocite, and it was therefore anticipated that Mt. Gunson concentrate would be quite stable.

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[*] Based on mineralogical studies conducted by Dr. J.H. Patterson of these Laboratories.

## 3. ABSTRACT

As requested by Mt. Gunson Mines Pty. Limited (letter, 23rd March, 1970) a thermoanalytical investigation has been conducted on a sample of Mt. Gunson copper concentrate supplied by the mine. The study summarized in this report was based on the utilization of the following techniques :

(a) Differential Thermal Analysis
(b) Thermogravimetric Analysis
(c) Isothermal roasting of the concentrate - coupled with gas chromatographic analysis for sulphur dioxide.

The results of such studies have shown that the concentrate supplied is stable to oxidation and does not exhibit autogenous heating or sulphur dioxide evolution - at temperatures less than 190°C (374°F).

## 4. RECOMMENDATIONS

It should be noted that only a single sample was presented for analysis. Further, this sample had obviously been dried - probably by heating to temperatures a little above 100°C, which may have considerably lessened the propensity of the concentrate to further oxidation. In addition, the possibility of bacterial oxidation[10,13] of the concentrate has not been examined since any bacteria would have been destroyed by the heat-drying process. In view of the above the following actions are strongly recommended :

1. Samples of the first consignment of concentrate from Mt. Gunson should be forwarded to a responsible laboratory for the following studies :

   (a) X-ray diffraction analysis - in order to check that iron-containing sulphide minerals such as pyrite ($FeS_2$), chalcopyrite ($CuFeS_2$), marcasite ($FeS_2$) and pyrrhotite (FeS) are either absent or present at low concentrations (x-ray analysis has shown that the sample supplied to these Laboratories contained very little, if any, of these minerals).

   (b) Differential Thermal Analysis and if possible, though not essential, thermogravimetric analysis.

   (c) Isothermal roasting and sulphur dioxide analysis.

(d) Microbiological examination of the concentrate for sulphide-oxidizing bacteria (e.g. Thiobacillus ferrooxidatons and related species). If such are found then experiments should be carried out to determine the activity of said bacteria.

Samples of the concentrate should be obtained immediately prior to containerization, conveyed to the testing laboratory in hermetically sealed containers, and analysed as soon as possible.

2. As the experiments of this study have been of short duration - with very small quantities of concentrate, a long-term, large-scale experiment should be conducted. Such an experiment would simply involve monitoring the temperature of concentrate in a typical container, under conditions simulating those of transportation. The minimal duration of this experiment should be at least equal to the time of transportation from flotation plant to smelter.

3. All the usual fire-preventative procedures should be operational during the handling and transportation of the concentrate.

## 5. EXPERIMENTAL

### 5.1 Sample Preparation

In all the experiments described in this report the concentrate was ground to -200 mesh (76 microns, 0.003 in.), and used without further preparation.

### 5.2 Procedure

### 5.2.1 Differential Thermal Analysis (D.T.A.)

A Rigaku Thermoflex 8001 Thermal Analyser was used for the D.T.A. studies. Sample-holders were of platinum, with samples heated either in the presence of a fixed, though abundant, supply of air, or in a continuous air-stream. Heating-rates of 5°C, 10°C and 15°C/minute were used.

### 5.2.2 Thermo Gravimetric Analysis (T.G.A.)

A Stanton HT-D Automatic Thermo-Recording Balance was used for the T.G.A. studies. Samples were contained in platinum holders and heated in the presence of a fixed, though abundant, supply of air. A heating-rate of 3°C/minute was used throughout.

### 5.2.3 Isothermal Roasting of Concentrate with Gas Chromatographic Analysis for Sulphur Dioxide

In these experiments, the concentrate was heated for two hours at 50°C, 100°C and 150°C, with gas chromatographic analysis for any sulphur dioxide liberated at each of these temperatures.

The reaction vessel was basically a 10" long B40 Quickfit test-tube, fitted with an air-tight glass paddle-stirrer. Normally between 40 gm and 50 gm of the concentrate was roasted, with a 6:1 volume-ratio of air to concentrate.

In addition to being equipped for gas-flushing, the reaction vessel was fitted with a side-arm connected by polyethylene tubing to two anhydrous calcium sulphate water-absorbent traps connected in series and thence to a set of stainless steel condenser coils immersed in an alcohol/dry ice cooling bath (-72°C*). The coils were provided with special fittings by which they could be coupled to the inlet port of a F & M Model 500 Gas Chromatograph. At the completion of each experiment a stream of nitrogen gas was passed through the whole assembly to ensure that all gaseous products had been quantitatively conveyed to the coils. The coils were then disconnected from the apparatus and coupled to the inlet port of the gas chromatograph for sulphur dioxide analysis.

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*Sulphur dioxide gas liquifies at -10°C, with the liquid solidifying at -75.2°C.

# 6. RESULTS AND DISCUSSION

## 6.1 Differential Thermal Analysis

The D.T.A. thermogram of the concentrate sample is summarized in Table 1. As shown therein, there is only one exothermic peak ($217^{\circ}$C - $345^{\circ}$C ($\pm 17^{\circ}$C) in the whole thermogram. It should be noted that this peak is not strongly exothermic (12 ± 2 cal./gm.), and on no occasion during the course of the experiments was the concentrate observed to ignite.

Thermogravimetric analysis (discussed in the following Section) has shown that the sample, as supplied to the Laboratories, contained less than 0.1% moisture. As the concentrate produced by the flotation process at Mt. Gunson is expected to contain approximately 12% moisture, portions of the supplied concentrate were moistened to 4%, 8% and 12% and subjected to D.T.A.

Thermograms of the "wet" samples are quite similar to that described in Table 1 (for the dried concentrate). The only difference is, of course, that the water acts (through its vaporization) as a very efficient heat-dissipator, with the result that there is increased endothermic behaviour in the $80^{\circ}$C - $120^{\circ}$C range of the D.T.A. thermogram.

## 6.2 Thermogravimetric Analysis

The T.G.A. thermogram of the concentrate (used as supplied, except for grinding - vide supra) has shown that

## TABLE 1

## PEAK CHARACTERISTICS OF D.T.A. THERMOGRAM OF MT. GUNSON COPPER CONCENTRATE

| Peak Temperature Range (°C) | Peak Type | Maximum or Minimum Temperature (°C) | Remarks |
|---|---|---|---|
| 50 - 217 (± 8) | Endothermic (broad peak and one of the major peaks of the thermogram) | $T_{min}$ = 122 (± 1) | May be assigned to a combination of endothermic phase transitions[14] exhibited by copper sulphide constituents of the concentrate. The more important transitions may be :<br>(i) Digenite ⟶ High Digenite<br>(ii) Djurleite ⟶ High Digenite + Hexagonal Chalcocite |
| 217 - 345 (± 17) | Exothermic (broad and major) | $T_{max}$ = 302 (± 2) | Assigned to the exothermic oxidation of copper sulphides to oxides, with the evolution of sulphur dioxide. The oxides then react with some of the sulphur dioxide and with oxygen to form copper sulphate[4-8, 15-19, 24]. |
| 345 - 487 (± 10) | Endothermic (sharp and major) | $T_{min}$ = 430 (± 2) | This peak is, as yet, unassigned but is probably due to the endothermic dehydroxylation (water-loss) of either an unidentified clay impurity, or of malachite ($Cu_2CO_3.(OH)_2$), which need only be present at trace levels of concentration since its dehydroxylation is very strongly endothermic[20b]. |
| 569 (± 5) - 587 (± 5) | Endothermic (sharp and minor) | $T_{min}$ = 578°C (± 2°C) | Assigned to the quartz $\alpha \rightarrow \beta$ endothermic inversion[20a]. |
| 630 (± 20) - 1000 (± 20) | Endothermic (very broad and major) | (too broad to assign a $T_{min}$) | Assigned to the endothermic decomposition of copper sulphate to copper oxide, with the evolution of sulphur trioxide[4-8, 15-19]. |

has shown that there is no significant weight change of the sample on heating to 190°C (± 10°C). At temperatures above this threshold, the sample loses weight very slowly until 350°C (± 10°C), where it starts to rapidly gain weight. The weight loss from 190°C to 350°C is 2% (approximately). This initial weight loss may be attributed to the covellite-chalcocite interconversion[17]:

$4CuS$ (Covellite) ⟶ $2Cu_2S$ (Chalcocite) + $S_2$ (Sulphur(vapour))

This reaction is very weakly endothermic, and is, no doubt, followed by the exothermic oxidation of sulphur to sulphur dioxide - this reaction producing a weight loss.

Also within this 190°C - 350°C range (and probably at about 300°C), copper sulphide minerals of the concentrate commence to undergo oxidation to cuprous oxide, with the evolution of sulphur dioxide. The oxide reacts further with sulphur dioxide and oxygen to form sulphate[21-23].

The overall conversion of sulphides to sulphate results in a substantial weight gain. The weight gain that is found on the T.G.A. thermogram to commence at 350°C reaches a maximum of 30% at 620°C (± 10°C).

Above 620°C, the weight steadily decreases to a value which, at 1200°C, is 15% below that of the original weight of the sample. This weight loss is, of course, due to the decomposition of the copper sulphate to cupric oxide, with the evolution of sulphur dioxide and sulphur trioxide gases[21-23].

T.G.A. thermograms of moistened concentrate were found to be quite similar to that described above, except for the weight loss due to vaporization of water.

## 6.3 Isothermal Roasting of Concentrate with Gas Chromatographic Analysis for Sulphur Dioxide

The results of these experiments may be summarized by reporting that sulphur dioxide was not detected upon heating to 150°C either the concentrate, as supplied, or samples of it moistened to 4%, 8% and 12% water content. The limit of detection of sulphur dioxide with the analytical procedure adopted was better than $5 \times 10^{-7}$ mole - representing negligible decomposition of the sample.

## 7. CONCLUSION

Thermoanalytical investigation has shown that the dry concentrate supplied to these Laboratories is quite stable to oxidation - and therefore does not exhibit autogenous heating or sulphur dioxide evolution at temperatures less than 190°C (374°F). Samples of the concentrate moistened to 4%, 8% and 12% water content showed no added propensity to oxidation.

## 8. SCOPE OF CONCLUSION

Attention is particularly directed to the limited scope of the investigation (see from above) especially to the fact that only a single sample was available, which sample had been dried before received (section 4 above). The conclusion reached therefore cannot be regarded as applicable to concentrate produced and consigned under working conditions, and section 4 contains recommendations as to studies of the effect of actual transportation upon substantial qualities of concentrate.

## 9. ACKNOWLEDGEMENTS

It is a pleasure to thank Dr. J.H. Patterson for conducting the X-ray diffraction analyses of the concentrate, and Dr. S. Pash for his valuable assistance with the D.T.A. studies.

Thanks are also due to James Hardie and Co. Pty. Ltd., for the most generous use of the Rigaku Thermoflex 8001 Analyzer.

## 10. REFERENCES

1. Kirshenbaum, N.W., "Transportation and Handling of Sulphide Concentrates", Technomic Publishing Company, Inc., Connecticut, 1968.

2. Hikami, K., J. Min. Inst. Japan, 58, 458 (1942).

3. Pavlov, F.N. and Piev, N.P., Trudy Inst. Met., Akad. Nauk S.S.S.R. Ural. Filial, Sboinik Rabot 1957, (1), 22; Chem. Abstracts 53, 19750 (1959).

4. Morgulis, E.V. and Ponomorev, V.D., Izvest. Akad, Nauk Kazakh. S.S.R., Set. Met., Obogashchen. i Ogneuporov, 1958, (3), 9; Chem. Abstracts, 55, 3361 (1961).

5. Zawislak, L. and Crzebieluch, Z., Ruby Metale Niezelazne 12, 183 (1967); Chem. Abstracts, 67, 75248 (1967).

6. Collins, G.A. and Swan, A.C., Trans. Con. Inst. Mining Met. 57 (in Con. Mining Met. Bull. 508, 533) (1954).

7. Bayakhunov, A. Ya., Vdovenko, M.I. and Ershina, L.M., Izv. Akad. Nauk Kaz. S.S.R., Ser. Energ. 1962, (2), 63; Chem. Abstracts, 58, 11994 (1963).

8. Ionescu, V.T., Pincovschi, E. and Maxim, I., Z. anorg. u. allgem. Chem., 356, 207 (1968).

9. Klassen, V.I. and Mikrousov, V., "An Introduction to the Theory of Flotation", translated from the Russian by J. Leja and G.W. Poling, Butterworths, London, 1963 (Moscow, 1959), p.12.

10. Razzell, W.E. and Trussell, P.C., J. Bacteriol. <u>85</u>, 595 (1963).

11. Beck, J.V., Ibid., <u>79</u>, 502 (1960).

12. Bryner, L.D., Beck, J.V., Davis, D.B. and Wilson, D.G., Ind. Eng. Chem. <u>46</u>, 2587 (1954).

13. Colmer, A.R., J. Bacteriol. <u>83</u>, 761 (1962).

14. Roseboom, Jr., E.H., Econ. Geol. <u>61</u>, 641 (1966).

15. Wadsworth, M.E., Leiter, K.L., Porter, Wm. H., and Lewis, J.R., Trans. A.I.M.E., <u>218</u>, 519 (1960).

16 Lewis, J.R., Hamilton, J.H., Nixon, J.C. and Groversen, C.L. Am. Inst. Mining Met. Engrs., Nonferrous Met., Metals Technol. <u>15</u>, No. 4, Techn. Publ. No. 2388, 9 pp. (1948); Chem. Abstracts, <u>42</u>, 5816 (1948).

17. Ganguly, N.D. and Mukherjee, S.K., Chem. Eng. Sci., <u>22</u>, 1091 (1967).

18. Razouk. R.I., Farah, M.Y., Kikhail, R.A. and Kolta, G.A., J. Appl. Chem., <u>12</u>, 190 (1960).

19. McCabe, C.L. and Morgan, J.A., J. Metals, 8, Trans. A.I.M.E., <u>206</u>, 800A (1956).

20. "The Differential Thermal Investigation of Clays", edited by R.C. MacKenzie, Mineralogical Society, London, 1957.

(a) Grimshaw, R.W. and Roberts, A.L., Ch. XI.
(b) Webb, T.L. and Heystek, H., Ch. XIII.

21. Charrier, J., Compt. Rend. 79e Congr. socs. savantes Paris et depts., Sect. sci. 1953, 107; Chem. Abstracts, 49, 7465 (1955).

22. Kushima, I., Moriomoto, Y. and Moriyama, J., Suiyokwai-Shi, II, 522 (1950); Chem. Abstracts, 46, 1401 (1952).

23. Duval, C., Proc. Intern. Symposium Reactivity of Solids, Gottenburg, 1952, Pt 1, 511 (Pub. 1954); Chem. Abstracts, 48, 12528 (1954).

24. Maurel, C., Compt. Rend. 257, 2647, (1963); Chem. Abstracts, 60, 3875 (1964).

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# REPORT OF CONSULTING VISIT
# FEBRUARY 26, 1971

By

N. Draper

Samin Limited
G.P.O. Box 2452
ADELAIDE, 5001

March 4, 1971

# REPORT OF CONSULTING VISIT
## FEBRUARY 26, 1971

By

N. Draper
SAMIN LIMITED

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## INTRODUCTION

At the request of Mr L. Nicholls, General Manager of Mount Gunson Mines Pty Limited, a visit was made to the operations site on the 25th and 26th February, 1971. The purpose of the visit was to examine the operation of the treatment plant, determine the reason for unsatisfactory metallurgical recovery and recommend a suitable course of action.

It was stated that recovery of copper into flotation concentrate was currently of the order of 75-78% compared with near 90% obtained for several months initially following plant start up during 1970. It was desired to return to the earlier results by whatever modifications to plant practice necessary.

## PLANT FEATURES

A low grade copper ore is mined by open cut methods. Major copper mineralisation is chalcocite although it is apparent that some oxidised copper minerals are present. It is understood that the gangue minerals are mainly sandstone. The mill feed has a clean appearance and although it tends to give a viscous pulp at high densities is quite non-viscous at lower density. Liberation of copper minerals is high at quite coarse grain size and flotation plant feed is relatively coarse. No serious middlings problem is reported and does not seem apparent.

Concentration of copper minerals is carried out in a conventional treatment plant employing wet ball milling and a simple flotation circuit. Appendix I to this report shows the flowsheet. No attempt was made during the visit to record plant data or obtain a detailed description of the ore mineralisation, orebody or mining method. Unless aspects were considered of direct relevance to the problem they were briefly considered and then discarded in the interests of concentrating on the problem in the short visit made.

Of major importance to plant operation is the water system. Insufficient fresh water is available to operate the plant entirely from this source. Therefore used water is reclaimed by thickening plant tailings and concentrates and by pumping from the tailings dam pond. The reclaimed water is re-used and a further source of highly saline water from a seepage area is to be prepared for use.

Reagents in use at the time of the visit were as follows.

| Reagent | Feed Point & Dosage Rate (lbs/ton) | | |
|---|---|---|---|
| | Ball Mill | Rougher | Cleaner |
| Sod. Sec. Butyl Xanthate | 0.42 | 0.04 | 0.01 |
| Sodium Silicate | 0.53 | - | - |
| Aerofroth 65 | - | 0.16 | 0.08 |

No staging of reagents down the rougher/scavenger system is practiced and in view of this it is interesting to note the addition of collector and frother to the cleaners.

Since the ore is low grade and apparently clean the collector consumption is abnormally high and in fact has risen from about 0.1 lbs per ton when the plant was first operated.

No attempt was made to establish that a different ore type was being treated compared to earlier times when good results were achieved. It is considered most unlikely that the problem is related to change in ore, at least to any significant degree.

## CONSIDERATION OF PROBLEM

There is no reason to suspect that mineralisation in the ore has substantially changed. Therefore the drop in copper recovery is caused by a loss to tailings of liberated copper mineral which previously floated. The problem is one of flotation rate and the reason for the reduction in rate of flotation of apparently clean free sulphides had to be found. Examination of tailing assay sizing data accumulated by Mr A. Dobson indicated very low recovery of coarse copper with substantially better recovery of fine copper. Visual examination of the froth on the rougher/scavenger flotation machines revealed a thinly mineralised froth containing clear (sometimes large) unmineralised windows. Very poor recovery of coarse mineral is frequently associated with interference to collector adhesion by slime particles. The thinly mineralised froth and slow rate of flotation of even fine particles agreed with a theory that collector adhesion had become inefficient. The possibilities that an inadequate air supply was available to the flotation machines or collector supplied was sour were quickly examined and discarded.

It was concluded that slime interference was only a minor factor in affecting coarse mineral recovery and that another factor was present in the flotation circuit causing poor and slow collector adhesion. The fact that collector consumption had been increased to try and maintain at least some recovery under lined the nature of the problem and represented the only short term remedy possible.

## CAUSE OF PROBLEM

After a preliminary session on the flotation floor it was clear that the cause of the problem could not be ignored in favour of finding a quick solution. Fortunately a laboratory flotation machine was available and set up in the plant area and this was used to carry out a series of qualitative and quantitative tests.

The froth condition was similar to one which the writer had experienced on spasmodic occasions in the oxidised copper circuit at Mount Isa Mines Limited. Such a froth was caused by an over-reagented condition in the pulp and mineral recovery dropped away. The only solution was to remove all reagents, purge the plant of pulp and then restart with lowered reagent additions. The reagent system at Mt Isa was more complex than at Mt Gunson and it appeared likely that the problem at Mt Gunson was being caused by excessive sodium silicate.

A sample of flotation feed (without returns) was put into the laboratory cell and found to be moderately viscous. Dilution with water gave a fluid non-viscous pulp. Addition of further sodium silicate tended to increase viscosity and in the immediate area of addition induced flocculation. These unexpected reactions were the first evidence that sodium silicate at Mount Gunson was not acting in a normal beneficial manner.

A further sample of ball mill cyclone overflow was placed in the laboratory cell and the air turned on. A poorly mineralised froth was obtained similar to that in the plant. Further additions of silicate were then made with the air left on. After each addition the froth became less mineralised until a point was reached where a virtually barren froth was achieved. It was apparent that increasing silicate made an already bad condition worse.

It was concluded at this point that the problem at Mount Gunson is caused by an excess of sodium silicate in the system. It is believed that the recirculated water has built up a concentration of silicate which is far in excess of that normally needed for dispersion or gangue depression. In fact the situation has been reached where chalcocite particles are being surface conditioned by silicate rendering them wetted to the point where the attainment of hydro-phobicity by xanthate has become difficult at usual concentration levels. The result is a dirty, grey poorly mineralised froth in place of the normal deep black, heavily mineralised froth.

A series of tests were then carried out in the laboratory cell to determine whether the problem could be solved simply by removing silicate additions to the grinding mill. At the same time tests were included to explore the possible advantages of separate sand and slime flotation as it is still possible that slime interference is preventing maximum coarse mineral recovery.

The results of these tests are shown in Appendix II. The following procedure was used.

For Tests 1 and 2 the normal plant mill cyclone overflow was taken. This pulp therefore had been ground with 0.42 lbs per ton of xanthate and 0.53 lbs per ton of silicate. Test 1 was floated for 14 minutes in the laboratory cell after addition of 0.12 lbs per ton of xanthate. Pulp for test 2 was roughly separated into sands and slimes by passing through a 200 mesh screen. Sand was re-pulped with mill spray water and floated for 5 minutes with 0.11 lbs per ton of xanthate. Slimes were floated for 10 minutes with 0.11 lbs/ton of xanthate. For Tests 3 and 4 the sodium silicate was removed from the mill and the samples taken one hour later. This pulp then only contained 0.42 lbs per ton of xanthate. Test 3 was floated for 14 minutes with addition of .13 lbs per ton of xanthate at the start and a further 0.13 lbs per ton after 7 minutes. Test 4 was separated into sands and slimes as for Test 2 and the sands floated for 5 minutes with 0.12 lbs per ton of xanthate after re-pulping with mill spray water. Slimes were floated for 10 minutes with 0.12 lbs per ton of xanthate.

In Tests 1 and 3 the copper was visually slow to float with Test 3 perhaps being slightly faster. The froth in both tests was dirty, grey and not well mineralised. In Tests 2 and 4 the froth in the slime floats was again dirty, grey, poorly mineralised and the copper sluggish. On the other hand the sand floats gave a clean black well mineralised froth and in Test 4 the flotation was virtually complete after 1 minute. The sands of Test 2 took slightly longer but this was probably due to the higher head value.

Because of the small number of tests carried out, the large variation in head values and the different flotation conditions used positive conclusions are difficult to draw from the results. However, two very significant factors are apparent.

1. Simple removal of sodium silicate from the ball mill is not in itself the solution to the problem. The problem exists in the pulp irrespective of the amount of make-up sodium silicate.

2. Tailings from all tests are quite low indicating that the copper will float (note, xanthate additions were high). The problem therefore is not insoluble. The only requirement is to find the right recipe.

Prior to carrying out the above discussed tests it had been concluded that an excess of sodium silicate was the cause of low flotation rates. The tests did not disprove this conclusion but did prove that simple removal of silicate was not the answer. It can therefore be concluded that the return water is carrying such a build up of silicate that that alone is sufficient to hinder flotation. A further qualitative test was carried out as follows. Fresh ore from the belt was admixed with fresh Murray water and the slimes decanted off. The decanted slimes were placed in the laboratory flotation cell, contacted with xanthate and frother and the air turned on. Only a small amount of copper mineral was present but what there was floated on clean bubbles with a pronounced black colour.

A further and final piece of evidence was supplied by Mr A. Dobson when he pointed out that plant spray water was clean Murray water. This factor undoubtedly contributed to the excellent flotation of sands after washing on the 200 mesh screen with Murray water and re-pulping with Murray water with no silicate present.

Therefore it can safely be concluded that the problem is a buildup of excess silicate in the plant water system causing inefficient adhesion of collector to mineral surfaces.

## SOLUTION TO PROBLEM

The only solution to this problem that the author can see is to thoroughly purge the plant water system of silicate and if possible operate in future without silicate. This would be a simple matter but for two factors.

1. There is a serious shortage of fresh make up water at the mine site.

2. Silicate may be needed for concentrate grade control.

Ideally silicate should be removed from the grinding mill and return of water from the tailings dam should be discontinued until arrangements can be made to isolate the existing pool of silicate saturated water from the tailings dam return water pumps. This will require discarding a volume of water from future use.

There is some evidence that removing silicate from the ball mill may result in low grade concentrate. A two day run in January, 1971, with no silicate on the mill resulted in concentrate grades of less than 40% Cu. Tests 3 and 4 results also show a reduction in concentrate grade with silicate removed fmn the mill although these are only roughing tests.

It must also be pointed out that the writer's conclusions have been based on qualitative evidence only and it may well be desirable to carry out confirmatory tests to prove the effect of silicate saturated waters.

A short term partial solution to the problem will be to use a massive dosage of xanthate in an attempt to alleviate the poor collector adhesion problem. This approach was tried just prior to the writer's departure and gave visually improved froth mineralisation. Subsequent shift results using approximately 0.66 lbs per ton of xanthate to the mill showed that low tailings could be made this way at least under some circumstances.

Therefore to play safe until more is known of the ramifications of the problem the following method of approach is recommended.

1. Collect a representative sample of 50 lbs of mill ore feed and a representative sample of 20 litres each of Murray water, tailings thickener O/F and tailings dam return water. Include also 20 litres of the seepage water proposed as a further mill supply. Despatch these samples to the Australian Mineral Development Laboratories and request that they carry out standard roughing, scavenging and

and cleaning tests on the ore with the different waters. Each water should be tested at two levels of collector addition and silicate should be added to the mill in each case. One further series should be carried using Murray water without silicate. Each test should be duplicated. A total of 20 tests is required. A careful note should be taken of froth condition and colour and apparent flotation rate on each of the tests.

2. Xanthate addition to the ball mill should be maintained as high as is considered economically or practically possible.

3. Silicate should be removed from the ball mill. The effect on concentrate grade should be carefully observed. If grade drops below an acceptable level then add a small amount, say 0.02 - 0.05 lbs/ton of ore, to the cleaning circuit.

4. When, as expected they will, the Amdel results confirm the silicate problem decide whether to immediately isolate the available pool of silicate saturated water at the tailings dam and make provision for a new pool area or carry on with (3) above in the expectation that time will eventually purge the system.

At some stage it will be necessary to decide whether silicate is needed on this ore. My own opinion is that it should not be needed for pulp dispersion but may play some specific role in gangue depression. If this is found to be the case then it will need to be used in such a way as to eliminate the chance of the current problem redeveloping. This may well be possible by adding only small doses to the cleaning circuit. Alternatively, other forms of silicate such as the neutral P84, or the organic dyes can be evaluated.

## GENERAL

During the visit from discussion with company staff and personal observation several matters relevant to the operation were noted and are briefly discussed as follows.

1. From time to time lime is used on the ore by addition to the ball mill. During the visit, at the writer's request lime was added for a test period. Froth mineralisation was visibly worsened. I can see no place for the use of lime in the plant under present circumstances.

2. Although oxide copper minerals are not present in major quantities they were observed and will appear in the plant feed in varying amounts. At least a daily tailing composite should be assayed for oxide copper by an accepted method. Ideally each shift tailing should be so assayed. Intelligent metallurgical analysis of daily results will never be possible unless the factor of unfloated oxidised mineral is quantitatively understood.

3. It was noted that in an endeavour to recover maximum water from tailings the thickener slime level is run very high. At times slime overflows and contaminates the water system. When this happens plant results will invariably deteriorate and under no circumstances should slime be allowed to overflow the thickener. If oxidised copper is to be floated then this matter is even more important. A simple air probe should be installed on the thickener to measure slime level.

4. Frequently froth and scum on the thickener surface blows into the clean water launder contaminating the water. A froth ring (remnants of which exist) should be installed on the thickener.

5. No measurement of circulating load within the flotation circuit is taken. Should the load build up its subsequent release could result in some abnormally high tailings assays. An hourly density of cleaner tailings should be recorded and used as an operational control indicator.

6. Some experimentation was carried out on frother dosage rate. The conclusion was that it could be substantially reduced. Plant testing should be carried out to determine a minimum rate.

7. There is some evidence that sand mineral flotation is inefficient due to slime interference. When the major immediate problem is eliminated the advantages of separate sand/slime flotation should be explored. This ore should yield a recovery well in excess of 90% and I am confident it can do so.

8. Facilities for first class investigation work at Mt Gunson are inadequate. Consideration should be given to equipping an area adjacent to the plant but away from major noise as a basic testing laboratory.

## CONCLUSIONS

1. The problem at Mt Gunson is one of inefficient collector adhesion to mineral surfaces resulting in loss of recovery and high collector consumption.

2. The cause of the problem is the depressant effect of excess sodium silicate by virtue of its ability to wet mineral surfaces.

3. Other lesser problems exist and have been noted in this report.

4. The ore is capable of very high recovery under correct flotation conditions.

## RECOMMENDATIONS

1. Carry out the programme proposed and recommended in the section of this report headed "Solution to Problem".

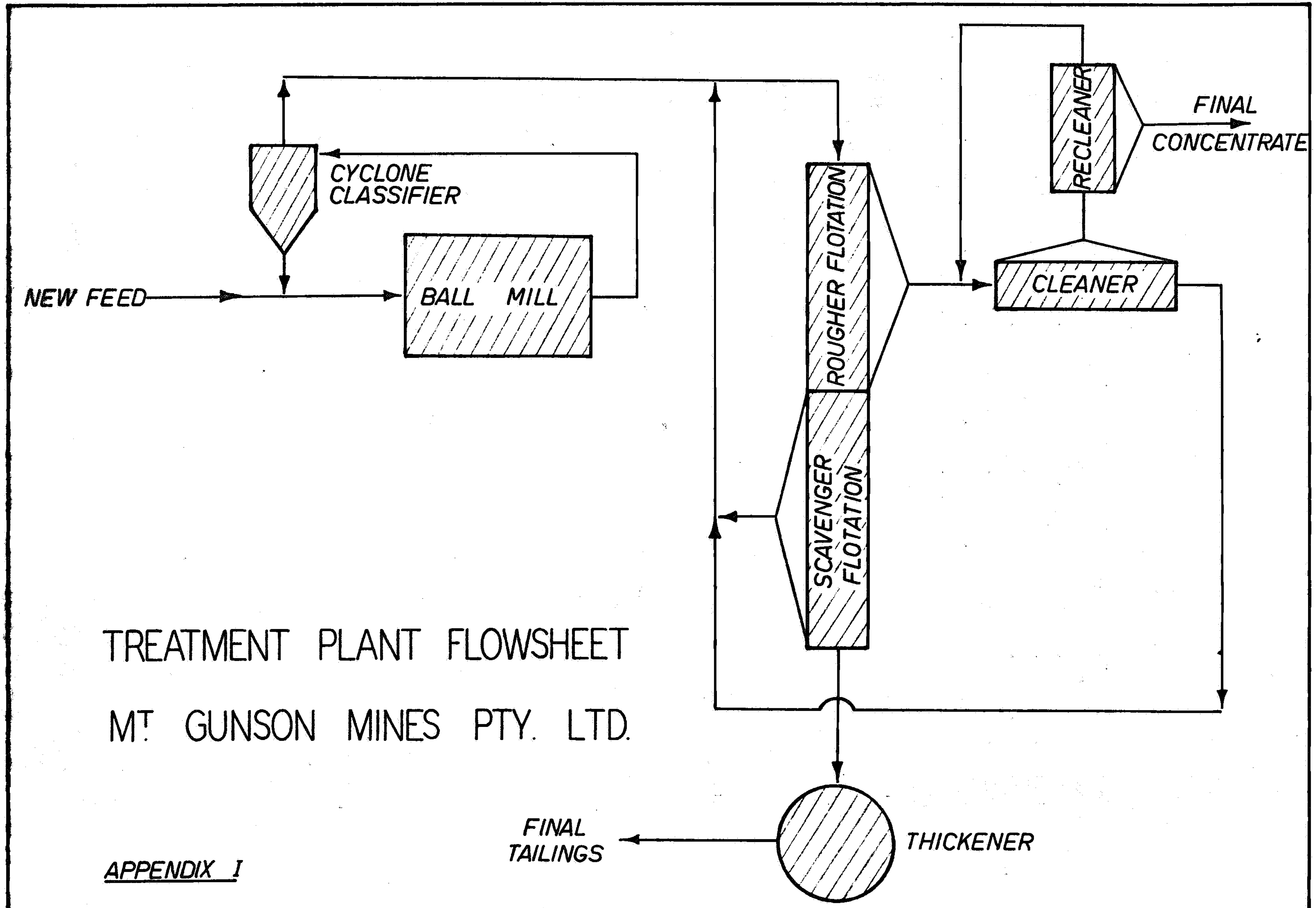
2. Investigate other improvements discussed in the report.

## ACKNOWLEDGEMENTS

The writer acknowledges with thanks the cooperation and assistance given freely by all the Mount Gunson Mines Pty Ltd staff during the visit. In particular the assistance given by the Mill Superintendent, Mr A. Dobson, made it possible to carry out a substantial work programme. The time given by Mr Dobson allowed a continuing dialogue during the visit which materially assisted in allowing the writer to reach a conclusion in what was a relatively short visit.

ORIGINAL SIGNED BY
N. DRAPER

N. Draper March 4, 1971



# TREATMENT PLANT FLOWSHEET

# M^T GUNSON MINES PTY. LTD.

APPENDIX I

## APPENDIX II

## LABORATORY TEST RESULTS

| Test No.<br>Product | Weight<br>% | Assay<br>% Cu | Distbn<br>% Cu |
|---|---|---|---|
| **Test 1:** | | | |
| Concentrate | 5.0 | 25.10 | 94.9 |
| Tailing | 95.0 | 0.07 | 5.1 |
| Feed | 100.0 | 1.32 | 100.0 |
| **Test 2:** | | | |
| Sand Concentrate | 2.0 | 41.00 | 51.3 |
| Sand Tailing | 66.5 | 0.01 | 0.4 |
| Sand Feed | 68.5 | 1.21 | 51.7 |
| Slime Concentrate | 3.6 | 20.10 | 45.3 |
| Slime Tailing | 27.9 | 0.17 | 3 0 |
| Slime Feed | 31.5 | 2.45 | 48.3 |
| Combined Concentrate | 5.6 | 27.50 | 96.6 |
| Combined Tailing | 94.4 | 0.06 | 3.4 |
| Total Feed | 100.0 | 1.60 | 100.0 |
| **Test 3:** | | | |
| Concentrate | 7.1 | 22.50 | 97.8 |
| Tailing | 92.9 | 0.04 | 2.2 |
| Feed | 100.0 | 1.64 | 100.0 |
| **Test 4:** | | | |
| Sand Concentrate | 1.4 | 39.00 | 55.6 |
| Sand Tailing | 63.8 | 0.01 | 0.6 |
| Sand Feed | 65.2 | 0.85 | 56.2 |
| Slime Concentrate | 4.2 | 9.30 | 39.[illegible] |
| Slime Tailing | 30.6 | 0.13 | 4.1 |
| Slime Feed | 34.8 | 1.24 | 43.8 |
| Combined Concentrate | 5.6 | 16.80 | 95.3 |
| Combined Tailing | 94.4 | 0.05 | 4.7 |
| Total Feed | 100.0 | 0.98 | 100.0 |

Additional copy in case original lost in post.

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The Colonial Sugar Refining Company Ltd.

TELEPHONE 660-0533
TELEGRAMS & CABLES
CENTRALAB PYRMONT

Central Laboratory

70 JOHN ST.
PYRMONT
N.S.W. 2009

NDH/he

16th September, 1971

The Manager,
Mount Gunson Mines Pty Ltd,
P.O. Box 246,
Woomera S.A 5720.

Dear Sir,

Attached is a report concerning eight copper ore agitation leaching trials carried out at Central Laboratory. The work was done by Mr. N.D. Harradine according to a requisition from Mt. Gunson Mines (No. 2155) and a letter from Mr. R.N.Dudgeon dated 27th August, 1971.

Yours faithfully,

ACTING MANAGER

c.c Mr. R.N. Selman
Mr. R.N. Dudgeon

The Colonial Sugar Refining Company Ltd.

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Central Laboratory

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N.S.W. 2009

NDH/he

16th September, 1971

The Manager,
Mount Gunson Mines Pty Ltd,
P.O. Box 246,
WOOMERA S.A 5720.

Dear Sir,

## AGITATION LEACHING TRIALS ON COPPER ORE

We now report the results of acid leaching tests carried out on eight samples of copper ore from the Mount Gunson area. The trials were done according to a method supplied by Mr. R.N. Dudgeon in a letter dated 27th August, 1971.

### 1. Preparation of Samples

Seven of the samples received contained only a small percentage of material larger than 14 mesh and so required only minimal crushing to achieve the required maximum particle size. The sample of "Oxide Ore" consisted of larger lumps and required considerably more grinding to achieve minus 14 mesh.

### 2. Agitation Leaching Trials

100 Grams of each ore was stirred at room temperature (20°C) with distilled water at a solid/liquid ratio of 1:3. The initial pH of the solution was taken and then a calculated amount of sulphuric acid was added. The amount was calculated as enough to dissolve copper present as oxide and silicate plus a 10 per cent excess. The pH of each solution was measured at the specified times and small samples of leach liquor taken at the times required. These samples were analysed for Cu by the method of Atomic Absorption Spectrophotometry. During the three hour leaching period the samples were agitated using blade stirrers. The rate of stirring was kept as uniform as possible during the trials, this rate being chosen to avoid settling of even the coarsest particles.

The sandy material in each ore was observed to be highly abrasive. In preliminary tests we used teflon-coated magnets as stirrers and this resulted in rapid removal of the teflon coating due to the abrasive action of the sand. Another interesting observation was the manner in which the ores settled when stirring

/2...

ceased. Two distinct layers formed - a coarse bottom layer and a very fine upper layer.

An examination of the residues indicated that these contained some blue and green particles. The "Oxide Ore" contained more of these than any of the others. These particles dissolved slowly in dilute sulphuric acid with effervescence and probably consisted mainly of malachite and azurite.

## 3. Results of leaching trials

Table I shows the copper analyses of the various ores as supplied by Mr. Dudgeon, and the analyses of the leached residues determined at Central Laboratory by A.A.S. The analyses of the residues show the expected trends when compared with the ore analyses, i.e. the difference between total copper and copper as oxide and silicate is very approximately the same as the amount of copper in the residue.

Table II shows the changes in pH and copper extraction with respect to time (see also graph I). Two of the trials, those on areas C and D, required more acid than the calculated quantity. Table IV shows that these are the two trials with the highest consumption of acid by gangue minerals.

Except for the trials on areas B and G, it appears, on examination of the copper concentrations in Table II, that extraction of copper is still continuing after the 3 hour period of agitation. The residue analyses in Table I tend to support this observation in as much as most of the residues appear to cotain slightly more Cu than would be expected from a determination of the difference between total Cu in the ore and copper as oxide and silicate in the ore.

Table III shows the percentages of copper extracted with respect to time. The values calculated on copper as oxide and silicate show a fair variation - from 68.0% in the case of area B to 106.5% in the case of area G. In two cases the extraction has risen to over 100% of the oxide and silicate copper (Areas A and G) and it appears that copper in other forms is actually being leached. However the analysis of the area G sample shows that all the copper exists as oxide and silicate. We have carried out a quick re-analysis of this sample by A.A.S and have obtained a total copper content of 2.28% compared with the value of 1.80% supplied to us. Using 2.28% Cu gives a considerably lower value of percent extracted (84.1% at 180 minutes). We are currently repeating this analysis by a wet chemical (Japanese Industrial Standard) method. As well, we plan to develop, at our own

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expense, a method for determining oxide and silicate copper.

Table IV contains the material consumption values calculated from these trials. The range of acid consumptions is quite considerable. Areas A, B, F and G show considerably lower consumptions by gangue minerals than the other samples. However A, B and F are the three samples with the lowest ratio of copper as oxide and silicate to total copper. On this basis sample G seems the most suitable in this group for an extraction process of this type.

## 4. Conclusion

On the basis of acid consumption and percentage of copper extracted, the samples from areas E and G appear to be the most suitable for this process. The Oxide Ore and the samples from areas C and D seem to have a fairly high acid consumption by gangue minerals, while the remaining samples show rather low extraction of total copper.

The presence of extractable copper after the three hour leach period suggests that the grind was perhaps too coarse at minus 14 mesh. In most cases, the extraction was near to completion before the first sampling at 30 minutes and leaching then increased slowly over the remaining period.

Yours faithfully,

N.D. Harradine

N.D. HARRADINE

c.c Mr. R.N.Selman
Mr. R.N.Dudgeon

## TABLE I

## ANALYSES OF SAMPLES AND RESIDUES

| Sample | % Total Cu | % Oxide and Silicate Cu | % Cu in Residue |
|---|---|---|---|
| Area A | 1.75 | 0.75 | 1.15 |
| Area B | 2.95 | 0.30 | 2.76 |
| Area C | 1.75 | 1.45 | 0.27 |
| Area D | 2.10 | 2.10 | 0.28 |
| Area E | 1.95 | 1.95 | 0.21 |
| Area F | 2.25 | 1.10 | 1.32 |
| Area G | 1.80 | 1.80 | 0.28 |
| Oxide Ore | 2.45 | 2.10 | 0.41 |

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report as lb/lb Cu extracted. ~~may~~ [illegible] (as well).

is this cummulative or not?

?

TABLE II

LEACHING TRIAL DATA

| Time (Mins) | OXIDE ORE | | | AREA A | | | AREA B | | | AREA C | | | AREA D | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | pH | Acid addn-lb/ ton ore | Cu g/l | pH | Acid addn-lb/ ton ore | Cu g/l | pH | Acid addn-lb/ ton ore | Cu g/l | pH | Acid addn-lb/ ton ore | Cu g/l | pH | Acid addn-lb/ ton ore | Cu g/l |
| 0 | 7.70 | 80.8 | | 7.25 | 28.5 | | 7.10 | 11.5 | | 8.20 | 54.8 | | 7.60 | 79.7 | |
| 5 | 1.42 | | | 1.78 | | | 1.90 | | | 2.30 | | | 2.10 | | |
| 15 | 1.60 | | | 1.85 | | | 1.95 | | | 3.40<br>1.94* | 8.2 | | 2.60 | | |
| 30 | 1.87 | | 5.22 | 1.92 | | 2.31 | 2.10 | | 0.71 | 2.90<br>2.10* | 4.1 | 3.90 | 3.10<br>2.15* | 4.1 | 5.48 |
| 45 | 2.10 | | | 2.02 | | | 2.08 | | | 2.50 | | | 2.80<br>2.05* | 4.1 | |
| 60 | 2.05 | | 5.73 | 2.05 | | 2.37 | 2.10 | | 0.68 | 2.92<br>2.20* | 4.1 | 4.10 | 2.48 | | 5.58 |
| 90 | 2.20 | | 5.94 | 2.10 | | 2.50 | 2.18 | | 0.71 | 2.70<br>2.12* | 4.1 | 4.42 | 2.95<br>2.10* | 4.1 | 5.90 |
| 120 | 2.30 | | 6.06 | 2.15 | | 2.50 | 2.18 | | 0.68 | 2.43<br>2.20* | 2.05 | 4.62 | 2.46 | | 6.02 |
| 180 | 2.45 | | 6.15 | 2.20 | | 2.61 | 2.20 | | 0.68 | 2.55 | | 4.82 | 2.85 | | 6.31 |
| - | Total Acid = 80.8 | | | Total Acid = 28.5 | | | Total Acid = 11.5 | | | Total Acid = 77.4 | | | Total Acid = 92.0 | | |

* pH value immediately after addition of extra acid

cumulative or not?

some redissolved maybe ?

if cumulati

**TABLE II (Cont'd)**

**LEACHING TRIAL DATA**

| Time (Mins) | AREA E | | | AREA F | | | AREA G | | |
|---|---|---|---|---|---|---|---|---|---|
| | pH | Acid addn-lb/ ton ore | Cu g/l | pH | Acid addn-lb/ ton ore | Cu g/l | pH | Acid addn-lb/ ton ore | Cu g/l |
| 0 | 7.70 | 73.9 | | 6.75 | 41.8 | | 6.95 | 68.3 | |
| 5 | 1.60 | | | 1.80 | | | 1.53 | | |
| 15 | 1.75 | | | 1.97 | | | 1.65 | | |
| 30 | 1.90 | | 4.94 | 1.98 | | 3.31 | 1.98 | | 5.60 |
| 45 | 2.01 | | | 2.12 | | | 1.99 | | |
| 60 | 2.10 | | 5.36 | 2.18 | | 3.36 | 2.00 | | 6.21 |
| 90 | 2.22 | | 5.58 | 2.30 | | 3.48 | 2.10 | | 6.39 |
| 120 | 2.21 | | 5.36 | 2.32 | | 3.48 | 2.20 | | 6.54 |
| 180 | 2.42 | | 5.90 | 2.41 | | 3.61 | 2.45 | | 6.39 |
| | Total Acid = 73.9 | | | Total Acid = 41.8 | | | Total Acid = 68.3 | | |

## TABLE III

## EXTRACTION RESULTS

| TIME (mins) | OXIDE ORE | | AREA A | | AREA B | | AREA C | | AREA D | | AREA E | | AREA F | | AREA G | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | % Cu Extracted | | % Cu Extracted | | % Cu Extracted | | % Cu Extracted | | % Cu Extracted | | % Cu Extracted | | % Cu Extracted | | % Cu Extracted | |
| | Total | Oxide & Silicate | Total | Oxide & Silicate | Total | Oxide & Silicate | Total | Oxide & Silicate | Total | Oxide & Silicate | Total | Oxide & Silicate | Total | Oxide & Silicate | Total | Oxide & Silicate |
| 30 | 63.9 | 74.6 | 39.6 | 92.4 | 7.2 | 71.0 | 66.9 | 80.7 | 78.3 | 78.3 | 76.0 | 76.0 | 44.1 | 90.3 | 93.3 | 93.3 |
| 60 | 70.2 | 81.9 | 40.6 | 94.8 | 6.9 | 68.0 | 70.3 | 84.8 | 79.7 | 79.7 | 82.5 | 82.5 | 44.8 | 91.6 | 103.5 | 103.5 |
| 90 | 72.7 | 84.9 | 42.9 | 100.0 | 7.2 | 71.0 | 75.8 | 91.4 | 84.3 | 84.3 | 85.8 | 85.8 | 46.4 | 94.9 | 106.5 | 106.5 |
| 120 | 74.2 | 86.6 | 42.9 | 100.0 | 6.9 | 68.0 | 79.2 | 95.6 | 86.0 | 86.0 | 82.5 | 82.5 | 46.4 | 94.9 | 109.0 | 109.0 |
| 180 | 75.3 | 87.9 | 44.7 | 104.4 | 6.9 | 68.0 | 82.6 | 99.7 | 90.1 | 90.1 | 90.8 | 90.8 | 48.1 | 98.5 | 106.5 | 106.5 |

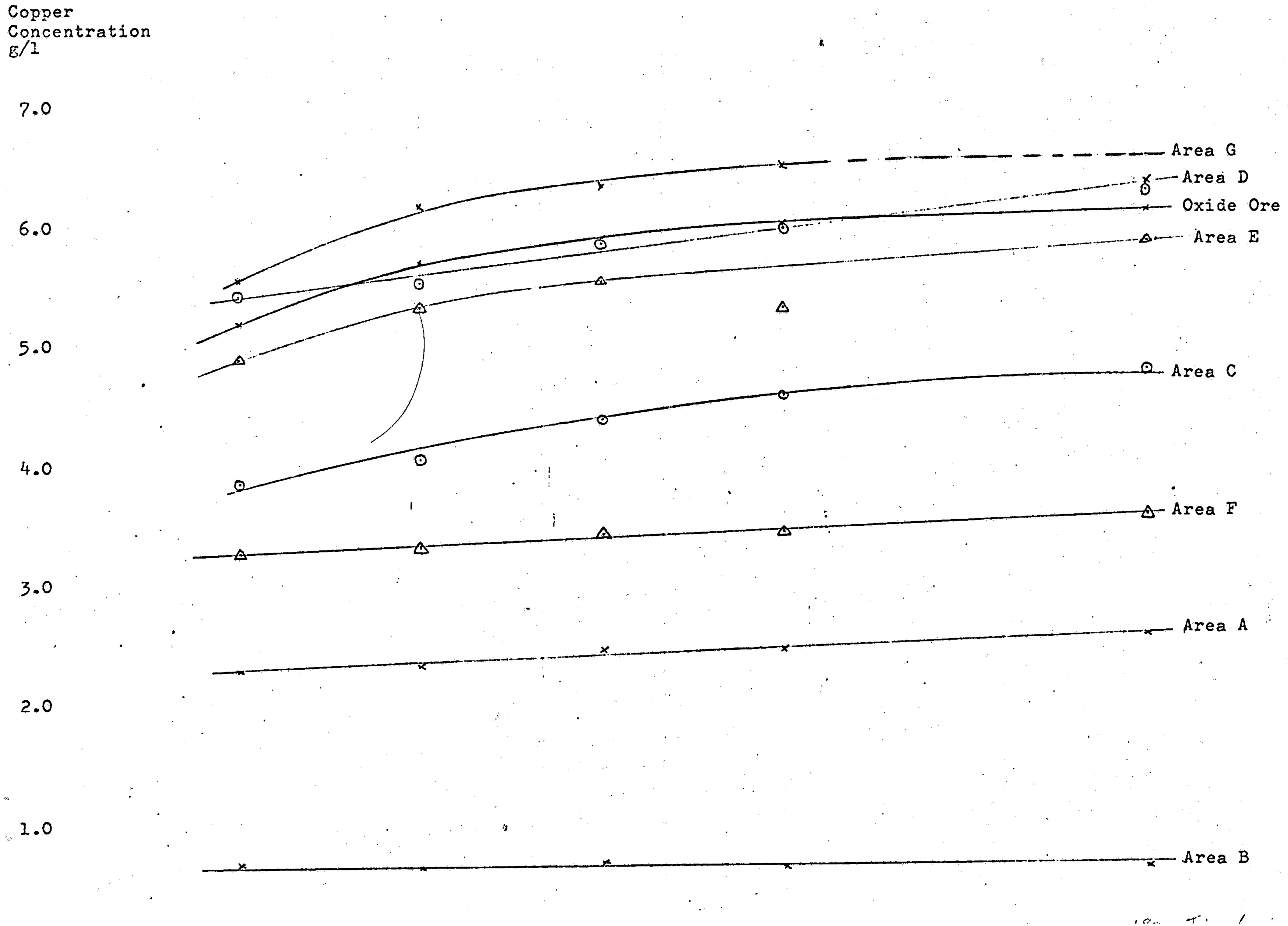
## TABLE IV

## MATERIAL CONSUMPTIONS AND LOSSES

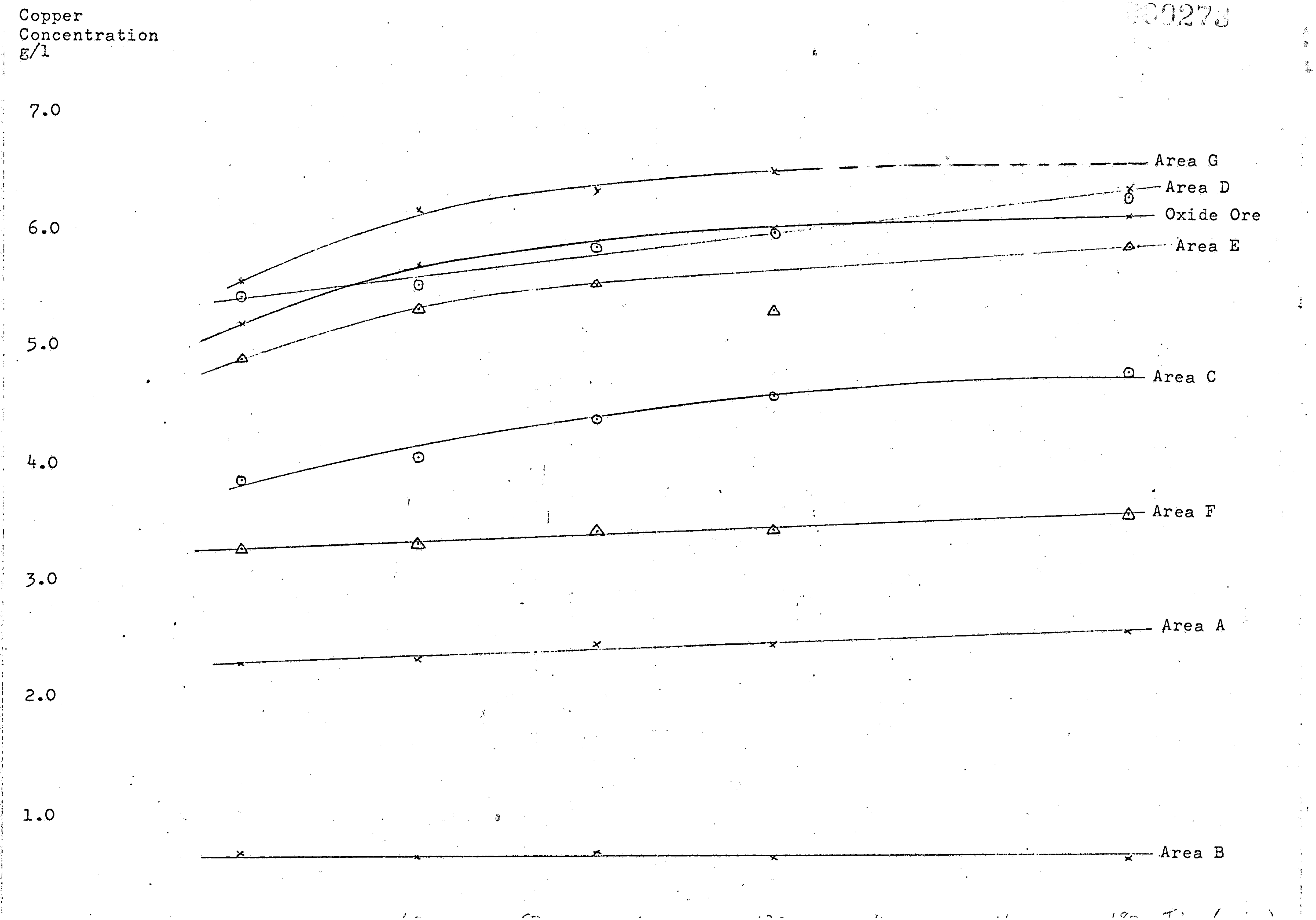
| | OXIDE ORE | AREA A | AREA B | AREA C | AREA D | AREA E | AREA F | AREA G |
|---|---|---|---|---|---|---|---|---|
| Total acid addition lb/ton ore | 80.8 | 28.5 | 11.5 | 77.4 | 92.0 | 73.8 | 41.8 | 68.3 |
| Actual acid consumption lb/ton ore | 79.6 | 26.4 | 9.5 | 76.5 | 90.5 | 72.5 | 40.5 | 67.1 |
| Acid consumed by Copper lb/ton ore | 63.8 | 27.1 | 7.1 | 50.0 | 65.4 | 61.2 | 37.4 | 66.3 |
| Acid consumed by gangue minerals lb/ton ore | 15.8 | -0.7* | 2.4 | 26.5 | 26.1 | 11.4 | 3.1 | 0.8 |
| Final acid concentration g/l | 0.17 | 0.31 | 0.31 | 0.14 | 0.07 | 0.19 | 0.19 | 0.17 |
| Weight of residue (g) | 94.4 | 96.1 | 97.7 | 94.3 | 94.5 | 94.9 | 96.0 | 92.6 |
| % Loss of Copper in residue | 16.7 | 65.7 | 93.6 | 15.4 | 13.3 | 10.8 | 58.7 | 15.6 |
| Calculated Copper assay of feed | 2.26 | 1.93 | 2.96 | 1.72 | 2.17 | 1.98 | 2.40 | 2.20 |

* Slight negative discrepancy probably due to precision of analyses etc.

GRAPH I : RATE OF COPPER EXTRACTION



## GRAPH I : RATE OF COPPER EXTRACTION



G. Livingstone

MG 26.49

**PACMINEX PTY. LIMITED**

**MEMORANDUM TO:** THE GENERAL MANAGER,
MOUNT GUNSON MINES PTY. LTD.

**OUR REF:** RND:LDR

**FROM:** R.N. DUDGEON

**DATE:** 18.4.1972

## PLANT TRIALS MAIN OPEN CUT SULPHIDE ORE DECEMBER, 1971.

Approximately 2,200 m.t. of sulphide ore was delivered to the plant during December, 1971. For a number of operating reasons it was not possible to keep this ore completely separate from West Lagoon Ore and so a proportion of it was processed mixed with West Lagoon Ore. Main Open Cut Ore was processed on December 7th and 10th and assays and recoveries are reported for periods during which only this ore was passing through the mill unmixed with Lagoon ore. This amounted to 1,437 m.t.

The initial delivery of M.O.C. ore was made on the afternoon of 2nd December. This ore was very sticky and contained a high proportion of clay. This introduced problems in crushing and screening. The feed rate to the Fine Ore Bin was about 25 T.P.H. whereas plant throughput was set at 50 T.P.H. Thus the material had to be mixed to maintain throughput. There was practically no froth generated in the flotation cells even by increasing the frother addition and air to the flotation cells. This could have been due to either contamination by oil from the plant or a contaminant in the ore. Frother problems continued during the following day. Reagent additions were as for West Lagoon Ore.

A further delivery of M.O.C. ore was made to the plant on 7th December. This material also contained clays which caused blinding of screens and restricted crushing. The ore appeared to process reasonably well with no frother problems.

. . /2

The recovery in excess of 75 per cent appears reasonable for a feed high in clays.

Laboratory flotation tests were carried out on grab sample of Rougher Feed taken at 12 p.m. on December, 7th. Results were discouraging and the test was repeated with a sulphidizing collector, AF 425 in place of xanthate with some apparent improvement. However, this material frothed excessively even with reduced air to the cell. The sulphides were slow in floating and during the latter period of the test excessive slime material was carried over with the froth which contributed to the low grade of the concentrate. This grab sample was not representative of the ore body, or even the feed to the plant during that shift but does point to the fact that the ore body is extremely variable and the need to blend feed. Results are tabulated in Table 1.

The variation in the calculated head grade between the two tests did suggest that such tests should be repeated, however, as the material was not representative this did not appear worthwhile.

M.O.C. ore received by the plant assayed 0.58% copper of which 0.11% was 'oxide' copper. The plant tails assay at the time was 0.19% Cu. Plant recovery was 76% which was in line with the grab tests.

The plant was operated normally using sod. sec. butyl xanthate as collector (approx. 0.3 lb/ton ore) and AF 65 as frother.

On Wednesday 8th December, due to the high proportion of partly oxidized feed from West Lagoon ore stockpiles, a sulphidizing reagent, AP 425 was added to the ball mill feed giving what appeared to be a marginal improvement in flotation.

. . /3

With the great variations in feed it is difficult to quantify the effect of such changes, however, concentrate grades did appear to improve.

On Friday 10th December, M.O.C. ore was delivered to the plant and no further changes were made in reagent additions - the AF425 was added to the ball mill at a rate of approximately 0.25 lb/ton ore. M.O.C. ore was processed undiluted and the physical condition of the material was such that there were no major problems in crushing and screening. Care was taken during these tests not to mix in West Lagoon Ore from the coarse ore stockpile. The type of froth produced was not all to be desired but was adequate. The pulp appeared muddy due to the higher than normal proportions of clay in the feed. Copper recovery varied from 73 to 87% with an improvement in concentrate grade.

On Saturday 11th a visit was made to the M.O.C. There appeared to be good ore on the floor of the pit although the walls revealed that some of the higher horizons contained clay and some oxidized ore.

On Monday 13th the floor of the M.O.C. pit was scraped and a further tonnage of ore was delivered to the plant and mixed with stockpile ore. It had no deleterious effects except that it was rather wet and caused some hanging-up in the F.O.B.

Results of the test runs are tabulated in Table 2.

## CONCLUSION:

The bulk of the M.O.C. ore as processed was probably not typical of the deposit in that it contained a considerable proportion of clay and oxidized material from higher horizons. Only a small amount of high grade ore from the lower horizons

. . /4

was processed and it appears that there is no real difficulty in processing this material. However, because of the variability within the deposit it would appear essential that the ore be mixed and blended before processing in an endeavour to produce a more consistent feed both for copper grade and the presence of clays and other materials which can cause processing difficulties. It is essential to accurately delineate this ore body and if possible carry out some laboratory test work to determine the optimum combination of flotation reagents and type of flocculant needed. The mining plan should be so designed to produce a consistent feed and every endeavour must be made to minimize big fluctuations in ore grade. It may be necessary to mine on more than one face simultaneously or alternatively stockpile ore and blend. The additional costs involved in either of these operations could be justified by improved metallurgical efficiency of the plant.

Test work and plant experience to date indicate that M.O.C. ore in the area designated Area A can be processed to give a satisfactory grade of concentrate and recovery. The variability in the ore will from time to time result in plant difficulties which can be minimized by blending and judicious alterations to reagent additions.

R.N. DUDGEON

18th April, 1972.

## TABLE 1.

## LABORATORY FLOTATION TEST - M.O.C. ORE

## GRAB SAMPLE ROUGHER FLOTATION FEED 7.12.1971.

| TEST NO. | REAGENT | CONS. GRADE | TAILS | RECOVERY % Cu | CALC. HEAD GRADE |
|---|---|---|---|---|---|
| R 80 | S.S.B. Xanthate | 3.6<br>(0.7 'oxide') | 0.12 | 76.9 | 0.47 |
| R 81 | AP 425 | 6.0<br>(1.0 'oxide') | 0.17 | 79.1 | 0.74 * |

NOTE: * Samples were taken consecutively.
Actual head 0.46.

## TABLE 2.

## PLANT TEST M.O.C. ORE

| DATE | TIME | TONNES | ASSAYS % Cu | | | RECOVERY |
|---|---|---|---|---|---|---|
| | | | HEAD | TAILS | CONS | |
| 7 | D/S | 337 | 0.67 | 0.18 | 52.0 | 75.8 |
| | A/S | 300 | 0.41 | 0.10 | 52.5 | 76.4 |
| 10 | D/S 8-12 | 382 | 1.09 | 0.26 | 62.0 | 76.5 |
| | 12- 4 | | 3.20 | 0.43 | 62.0 | 87.2 |
| | A/S 4- 8 | 418 | 1.15 | | Not applicable | |
| | 8-12 | | 1.10 | 0.35 | 57.5 | 73.4 |

NOTE: Mechanical problems A/S 10/12 4-8 p.m. resulted in 45% recovery.

REAGENTS: 7-12-71 Xanthate additions to ball mill and flotation cells
10-12-71 AP 425 addition to ball mill and Xanthate to flotation cells.

Assays per cent Cu Grab Samples 12 P.M. 7-12-71

Rougher Floats - 27.0
Scavenger Floats - 2.1
Final Concentrates 57.5
Final Tails - 0.14
M.O.C. Feed - 0.46

# MT. GUNSON MINES PTY LTD

# LEACHING PROJECT FEASIBILITY STUDY

Dwyer

J.G. Dwyer

Sydney 27.7.1971

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## I INTRODUCTION

The examination of the feasibility of either in-situ leaching or heap leaching of copper bearing material at Mt. Gunson has been initiated owing to the fact that the bulk of the remaining reserves around the several open pits appears to be oxidised material.

The types of mineral occurrences appear to be copper oxide, carbonate, silicate and finely divided sulphide all intermingled. Conventional grinding and flotation methods of combined oxidised and sulphide material is usually quite inefficient and the continuation of the present method will not be very economic.

Leaching has been suggested as one method of recovery at Mt. Gunson but this was only after mining, crushing, grinding and flotation i.e. treatment of the tailings of a normal type operation (Amdel Report No. 541, June 1967 for Austminex P/L). Such a method incorporates excessive metallurgical processes, when either in situ or heap leaching may provide means of economic recovery on its own, without the complex metallurgical treatment.

It is estimated that the capital expenditure and operating costs will be substantially reduced (operating costs may not be reduced as much in the case of heap leaching). It is expected that some of the existing equipment could be used for a leaching operation, although no account has been made of such consideration.

It should be realised that problems remain to be solved regarding a leaching operation (almost entirely dependent upon an experimental approach) but it is felt that if relatively small deposits similar to the ones at Mt. Gunson are found and developed under the conventional approach as practised, the operation may not be as economic as a leaching type method.

## II SUMMARY

The East and West Lagoon orebodies were examined in order to formulate an idea of how economic a leaching operation may have been. This involved a reassessment of ore reserves to

include some of the material which was not mined in the normal operation. Two leaching methods were examined viz: -

(i) heap leaching and
(ii) in-situ leaching

The study of leaching methods necessitated some assumptions to be made. These are detailed in the body of this report and are almost entirely dependent on experimental results. For this reason it is emphasised that the study is a preliminary investigation to see if field experimentation is warranted.

On this basis and understanding the following present values (15% compound interest) can be compared with actual operations.

| Copper Price £$/MTON | 400 | 450 | 500 |
|---|---|---|---|
| Heap Leaching A$ | -814, 100 | -602, 200 | -390, 400 |
| In-situ Leaching A$ | -160, 600 | - 20, 600 | 116, 000 |

It is to be noted that: -

(i) The studies have used the fact that Davis contractors have been and will continue to be miners and hence no capital expenditure on the part of Mt. Gunson will be incurred in this respect.

(ii) The studies take no account of any prior commitments either financial or contractural, except those included in the above.

(iii) No account of plant available from existing operations has been made. However once it is known what plant is available, a more detailed study could be undertaken.

## III CONCLUSIONS AND RECOMMENDATIONS

The present value studies indicate that the in-situ leaching operation, by virtue of its low capital and operating costs, provides a more economic method of recovery when based on the assumptions stated herein. Both operations do not appear attractive but it must be pointed out that with regard to the possibility of using these methods for the remaining copper reserves: -

(i) The East and West Lagoon orebodies are considered as basically sulphide minerals and as such are not as amenable to leaching as are oxidised minerals such as are in the House, Main and Gunyot areas.

(ii) The amount of copper to be recovered may be considerably more than previously stated in view of the fact that oxide mineralisation can now be considered as ore. Previously, economically recoverable mineralisation (i.e. "ore") was limited to the sulphides whereas now both oxides and sulphides constitute minerals from which copper may be recovered.

In view of the above mentioned considerations it is recommended that: -

(i) Reassessment of the ore reserve situation be carried out to include oxidised material as ore,

(ii) Contact be made with, and visits arranged to producers from similar operations in order to understand more thoroughly the complex parameters involved and their experimental determination.

(iii) Subject to the above recommendations being met and the medium term price for copper being around £$450 to £$500/MTon or higher, detailed field experimentation is warranted.

A more detailed and accurate feasibility study should then be made of the House, Main and Gunyot mineralisation's amenability to leaching.

It is highly desirable that expertise be developed in this field in view of the fact that in the near future lower grade deposits will have to be developed, and in-situ leaching is the ideal method (with qualifications) since it cuts down considerably on capital and operating costs.

Leaching is also applicable for the recovery of uranium from solution and other minerals are under study in other parts of the world for recovery by this method.

## IV DISCUSSION

### A. Ore Reserves

The "geological" ore reserves have been used in this case since virtually the whole delineated mineralisation is considered as ore, i.e. there is no strict economic limit as we know it in the conventional mining method. The grade of material that could have been worked is much lower than that which was mined, primarily because of the low costs of extraction of copper metal for the studied methods. This brings home the distinction between "geological ore" (strictly just mineralisation above a certain low, cut-off grade) and mineable ore reserves (i.e. economically recoverable mineralisation).

The following "geological" reserves calculated by Pacminex were used in this study -

| | | |
|---|---|---|
| East Lagoon: | | 201,720 MTons @ 1.16% Cu |
| West Lagoon: | | 360,000 MTons @ 0.98% Cu |
| Total: | (Say) | 560,000 MTons @ 1.00% Cu |

Thus, the contained copper is approximately 5,600 MTons. Not all of this can be recovered.

### B. Extraction Methods

#### 1. Heap Leaching

Basically the orebody is mined in the conventional manner and then stockpiled in heaps in winnow fashion. Ponds are developed on the top of the heaps to which an acidic liquor is applied allowing it to percolate through the heap. The liquor dissolves the copper and then the solution is collected in a dam as run-off from the heaps. See schematic diagram No. 1.

#### 2. In-situ Leaching

In this method, the orebody is not mined but is fractured considerably by drilling and blasting on a closely spaced

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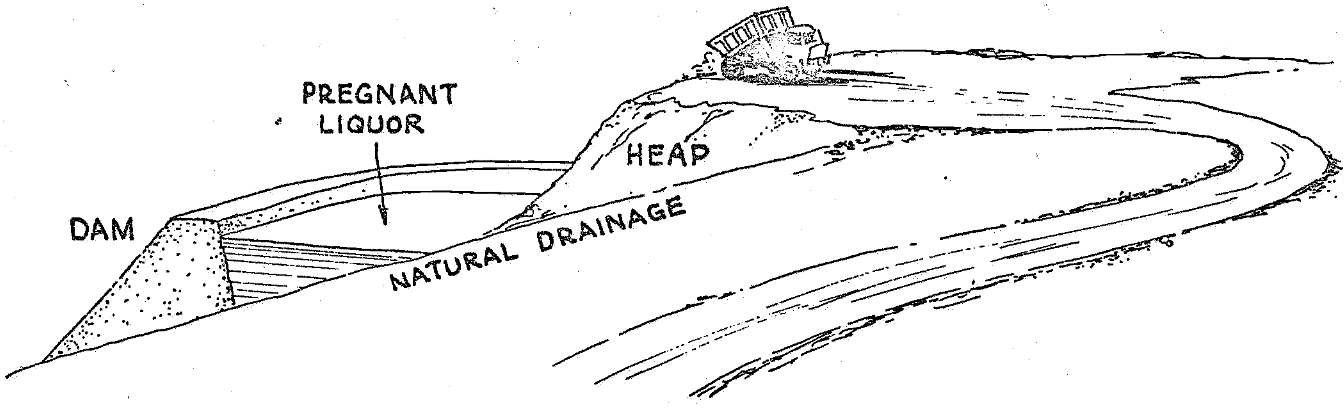
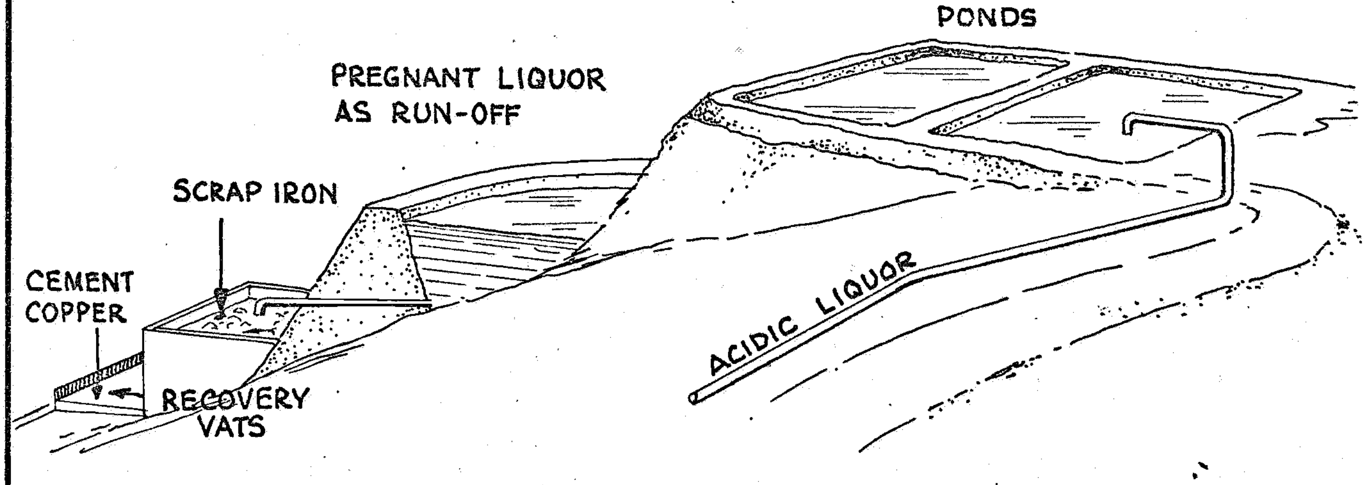


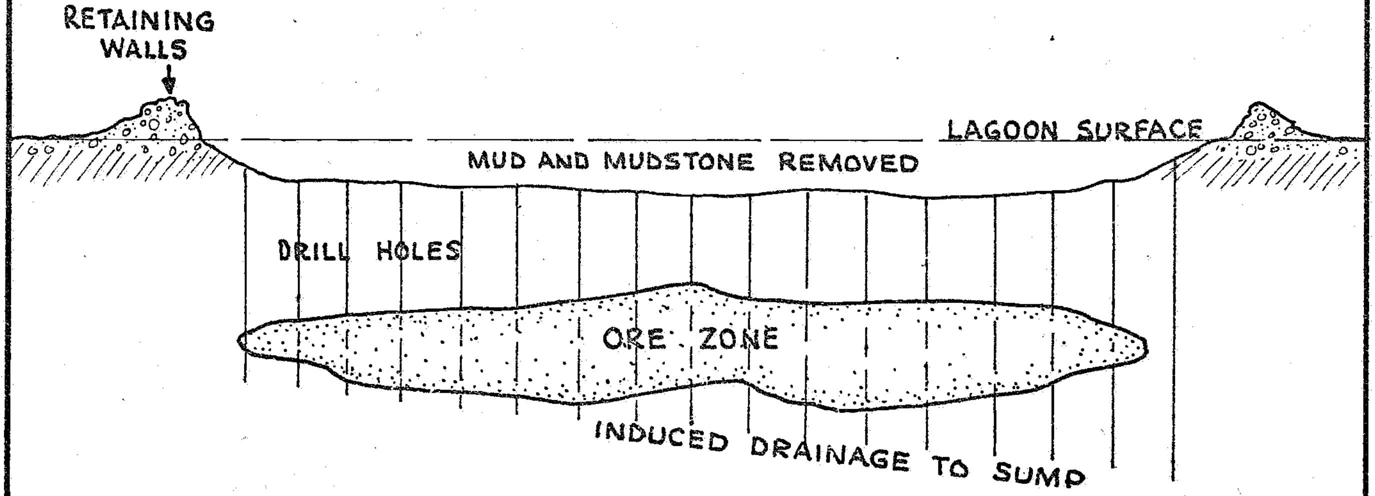
HEAP BUILDING



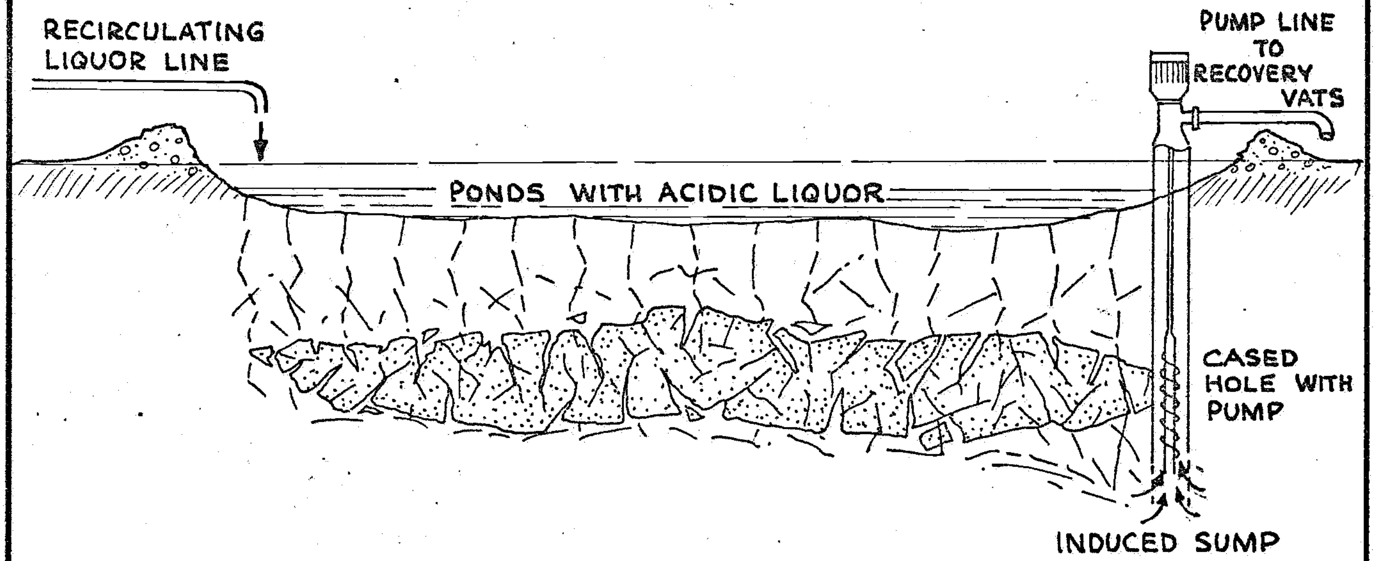
LEACHING IN PROGRESS

# 1 · HEAP LEACHING

| | | | |
|---|---|---|---|
| **PACMINEX PTY. LIMITED** | DRAWN | C.J.L. | SCALE |
| **MT. GUNSON** | DATE | JULY '71 | DWG. NO. Q 33 |
| | CHECKED | | |



PREPARE SURFACE - DRILL AND BLAST
ORE ZONE 25'x 25' GRID



# 2·IN-SITU LEACHING

| | | | |
|---|---|---|---|
| **PACMINEX PTY. LIMITED** | DRAWN | C.J.L. | SCALE |
| **MT. GUNSON** | DATE | JULY '71 | DWG. NO. Q 34 |
| | CHECKED | | |

grid. Acidic liquor is allowed to percolate through the fractured orebody and is collected in a sump which is formed by the gradual deepening of the drill holes towards one end of the orebody. The resultant copper pregnant solution is pumped out through a small diameter, cased shaft to the recovery plant. See schematic diagram No. 2.

## C. Recovery

The copper pregnant solution is pumped out into vats where cement copper (i.e. precipitated metal) is recovered. The "cement" is typically about 80% copper metal of approximately 98% purity. Iron replaces the copper in solution.

The recovery of Copper metal is the most difficult aspect of the operation to estimate and requires experimentation on the actual ores before an accurate assessment can be made.

For the purpose of this preliminary study however, figures from similar type operations in Australia and the U.S.A. have been examined, as well as the result of a leach percolation test on an unrepresentative sample of Mt. Gunson oxide ore, and what is beleived "reasonable" and "conservative" parameters employed.

From the foregoing studies and with the understanding that experimentation is imperative for an accurate assessment, it is assumed that: -

(i) Ultimate recovery of the copper from the Lagoon sulphide ores would be 50%. Tests on the Mt. Gunson unrepresentative sample of high grade <u>oxide</u> material gave 90% recovery after 3 days; - See graph of result in Appendix A.

The unrepresentative sample was first of all oxide but also of minus ½ in material and thus contact with the copper minerals would be much greater than for an in-situ operation after blasting on 25' x 25' grid, or for a heap leaching operation. Also it is to be remembered that blinding of the shattered orebody and the heaps by fine particles and iron salts etc. over time will reduce percolation rates and contact with minerals. Channelling of the liquid within the orebody will also reduce contact. These factors are virtually intangible at the present time, but an indication of their effects would be gained by controlled experimentation in the field.

(ii) Re-drilling and re-blasting of the in-situ orebody would be required once a year to keep the percolation rates high enough for envisaged solution flow rate. The blasting would have the added advantage of forming new flow patterns, thereby increasing solution contact with the copper minerals.

(iii) Minimal surface preparation will be necessary for the heap leaching operation.

(iv) Time for recovery of the 50% of the contained copper for both orebodies would be approximately 3 years. It is anticipated that up to 1000 MTons of copper metal will be recovered each year with recovery slowing down in the last year for a total recovered copper content of 2800 MTons. With both the heap leaching and in-situ leaching operations, cyclic wetting and drying (controlled by liquid application and lack of...) will be used to enhance dissolution and recovery. For the cyclic operation, three separate areas or zones need to be developed. Another reason for the separate zones is to minimise water losses from evaporation since make up water supply is limited. Thus the only exposed pondage required will be that for pumping of the return leach liquor and this will be relatively small.

(v) Ground water would be of pH 7 approximately and hence would not decrease considerably the effect of acid on the dissolution of the copper.

(vi) The percolating liquor would contain approximately 2 grams/litre (recoverable) of copper and is circulated at up to 300 gallons/minute. This gives a maximum copper recovery of just over 1000 MTons/year with pump efficiency at approximately 90% working 6 days a week for 50 weeks a year. Once again, the recovery rate and copper content of the leach liquor have been "averaged" from similar operations, and consideration of the following factors: -

(a) tons, and grade especially, of the material to be leached. (leaching of dumps (i.e. waste dumps and not heaps) of grades as low as 0.15% Cu has been successful in the United States).

(b) types of mineral occurrence; chalcocite has one of the highest dissolution rates and recoveries of the copper sulphides - see Table 1 (after Woodcock) page 9.

(c) geologic setting; the in-situ orebodies are underlain by hard, jointed but generally impermeable quartzites so that liquor losses will be kept to a minimum. Experimentation would enlighten the situation further. It is felt that horizontal (strata) losses of liquid will be kept to a minimum by (1) placing the sump towards the centre of the lagoon to enhance natural drainage, and (2) the effect of pumping which will set up its own drainage pattern.

(vii) Reagent consumption would be as follows: -

(a) $H_2SO_4$, 3lb/lb copper produced
(b) Scrap Iron, 2lb/lb copper produced

These figures have been increased slightly from the "average" of those studied at other operations since the actual consumption per lb of copper metal is unknown and can only be determined by controlled experimentation. Acid consumption would increase if it came in contact with calcerious or dolomitic material. Also more acid would be required if exess iron (in the form of ferric sulphate) was present. Acid suppresses the precipitation of iron salts in the orebody or heap where blinding would be deleterious.

(viii) Iron would replace copper in solution readily. Cement copper from iron exchange with scrap iron is by far the cheapest method of copper recovery if the process is applicable. Experimentation would determine if this is a viable method of recovery. The possibility of employing an electrowinning process has been ruled out because of the short life and the large capital expense involved. Electrowinning needs a decent power source and a fairly highly concentrated leach solution. To attain the high concentration involves a liquid concentration plant and an organic liquid stripping plant and thus further expense. One big advantage however is the fact that virtually no iron salts are present to blind the heaps or orebody. This factor is important when long heap/orebody life is envisaged.

(ix) The amount of copper in the "cement" precipitated by the iron in exchange would be about 80%. The remaining 20% would consist of moisture, iron salts, metallic iron, slimes from the heaps or orebodies etc...This figure of 80% is typical according to Woodcock* Copper values in precipitates range from about 30% to 95% depending on several factors including the nature of scrap iron, pH of solution, amount of iron salts etc...

* Woodcock, J.T. Copper Waste Dump Leaching Proc. Aust. I.M.M. No. 224, Dec. 1967.

## TABLE 1

## RELATIVE DISSOLUTION RATES OF COMMON COPPER MINERALS IN DUMP LEACHING SOLUTION

| Mineral | Aqueous Solvent | Dissolution rate [1] |
|---|---|---|
| Azurite | $H_2SO_4$ | 100% in 1 hour |
| Bornite | $H_2SO_4 + O_2$<br>$H_2SO_4 + Fe_2(SO_4)_3$<br>$H_2SO_4$ + bacteria | 27% in 24 days<br>95% in 12 days<br>100% in 20 days |
| Chalcocite | $H_2SO_4 + O_2$<br>$H_2SO_4 + Fe_2(SO_4)_3$<br>$H_2SO_4$ + bacteria | 30% in 14 days<br>95% in 12 days<br>90% in 30 days |
| Chalcopyrite | $H_2SO_4 + O_2$<br>$H_2SO_4 + Fe_2(SO_4)_3$<br>$H_2SO_4$ + bacteria | Low<br>30% in 40 days<br>100% in 26 days |
| Chrysocolla | $H_2SO_4$ | 100% in 1 day |
| Copper (native) | $H_2SO_4 + O_2$<br>$H_2SO_4 + Fe_2(SO_4)_3$ | 100% in 3 days<br>100% in 1 hour |
| Covellite | $H_2SO_4 + O_2$<br>$H_2SO_4 + Fe_2(SO_4)_3$<br>$H_2SO_4$ + bacteria | 26% in 35 days<br>60% in 24 days<br>50% in 76 days |
| Cuprite | $H_2SO_4$<br>$H_2SO_4 + O_2$<br>$H_2SO_4 + Fe_2(SO_4)_3$ | 50% in 1 hour [2]<br>100% in 3 days<br>100% in 1 hour |
| Malachite | $H_2SO_4$ | 100% in 1 hour |

[1] Dissolution rate is a function of many variables (such as size of mineral, reagent concentration, temperature, method of agitation, and type of rate control process) so that the figures shown are nominal only and simply give an indication of the relative time involved. As far as possible the rates are for the maximum total dissolution reported for room temperature or 35°C. The curves showing per cent dissolved with time are curved, and so extrapolation from the figures shown in impracticable.

[2] This is the maximum dissolution in oxidant-free solution.

## D. Capital and Operating Costs

Allow one month from the start of collection of liquor to elapse in order to accumulate enough copper for transportation. Allow a further 3 months to elapse before income from the sale is received. Capital costs are defined as all expenses incurred up to the time of income cash flow.

Thus, the time schedules for the present value studies are different from the basic production schedules because of the lapse between production and payment. Year one for the present value studies starts from the day the first cash income from sales is received.

## I HEAP LEACHINGS - EAST AND WEST LAGOONS

### A PRE-PRODUCTION EXPENDITURE

| | | A$ |
|---|---|---|
| 1 | Strip 70,000 MTons @ $0.52/MTon | 36,400 |
| 2 | Pumps 2 @ $1,300 each | 2,600 |
| 3 | Pipelines 3,500 ft @ $2/ft. | 7,000 |
| 4 | Storage Dam | 9,500 |
| 5 | Recovery Vats | 6,000 |
| 6 | Pump Station and Temporary Water Storage | 5,000 |
| 7 | Acid Storage and Metering | 10,000 |
| 8 | Water Supply Line | 2,000 |
| 9 | Electrics | 10,000 |
| | A Sub-Total | 88,000 |

### B WORKING CAPITAL

| | | A$ |
|---|---|---|
| 1 | Strip 507,500 M/Tons @ $0.52/MTon | 264,000 |
| 2 | Mine 186,000 MTons @ $0.87/MTon | 162,000 |
| 3 | Pump Maintenance | 1,000 |
| 4 | Pipeline Maintenance | 1,000 |
| 5 | Reagents for 296 MTons Cu(rec) | |
| | (i) $H_2SO_4$, 3lb/lb Cu; $50/MTon $H_2SO_4$ | 44,000 |
| | (ii) Scrap Iron; 2lb/lb Cu; $30/MTon Fe | 17,800 |
| 6 | Water; 14 mm galls. @ $0.90/1000 gall | 12,600 |
| 7 | Labour; 6 @ $5,000 | 30,000 |
| 8 | Supervision; 1 @ $10,000 and 2 @ $6,000 | 22,000 |
| 9 | Power, Insurance etc 2% of A | 1,000 |
| | B Sub-Total | 555,400 |
| | Totals | 643,900 |
| | Contingency @ 25% | 161,000 |
| | Total | 804,900 |
| | | 805,000 |

## II IN-SITU LEACHING - EAST AND WEST LAGOONS.

### A PRE-PRODUCTION EXPENDITURE

|  |  | A$ |
|---|---|---|
| 1 | Strip 453,000 MTons mud and mudstone @ $0.52/MTon | 236,000 |
| 2 | Drill 16,700 ft. @ $2.00 ft. | 33,400 |
| 3 | Blast East Lagoon only | 7,100 |
| 4 | Sink and Case Pump Hole | 2,000 |
| 5 | Water 4 MM galls @ $0.90/1000 gall | 3,600 |
| 6 | Recovery vats | 6,000 |
| 7 | Liquor Pipeline 5800 ft. @ $2.00/ft. | 11,600 |
| 8 | Return Liquor Pipeline | 11,600 |
| 9 | Distribution Lines (at Orebodies) | 22,000 |
| 10 | Acid Storage and Metering | 10,000 |
| 11 | Water Supply Line | 5,500 |
| 12 | Electrics | 10,000 |
|  | A Sub-Total | 358,800 |

### B WORKING CAPITAL

|  |  |  |
|---|---|---|
| 1 | Drill 17,140 ft. @ $2.00/ft. | 34,300 |
| 2 | Water; 8MM galls @ $0.90/1000 gall. | 7,200 |
| 3 | Pump Maintenance | 1,000 |
| 4 | Pipeline Maintenance | 1,000 |
| 5 | Reagents for 122 MTons Cu(rec) |  |
|  | (i) $H_2SO_4$; 3lb/lb Cu $50/MTon $H_2SO_4$. | 18,000 |
|  | (ii) Scrap Iron; 2lb/lb Cu; $30/MTon Fe | 7,300 |
| 6 | Labour; 6 @ $5,000 | 30,000 |
| 7 | Supervision; 1 @ $10,000; 2 @ $6,000 | 22,000 |
| 8 | Power, insurance etc. 2% of A | 7,200 |
|  | B Sub-Total | 128,300 |
|  | Totals | 487,100 |
|  | Contingency @ 25% | 121,800 |
|  | Total | 608,900 |
|  | (Say) | 609,000 |

## I HEAP LEACHING - EAST/WEST LAGOON

## OPERATING COSTS

| | | Year 1 | Year 2 | Year 3 |
|---|---|---|---|---|
| | Production M/Tons Cu | 884 | 884 | 736 |
| 1. | Strip, MTons | 506,000 | 421,500 | - |
| | @ $0.52/MTon. Cost | $263,000 | $220,000 | - |
| 2. | Mine, MTons | 186,000 | 186,000 | - |
| | @ $0.87/MTon Cost | $162,500 | $162,500 | - |
| 3. | Pumping Maintenance | $ 1,000 | $ 1,000 | $ 1,000 |
| 4. | Pipeline Maintenance | $ 1,000 | $ 1,000 | $ 1,000 |
| 5. | Reagents | | | |
| | (i) $H_2SO_4$, MTons | 2,650 | 2,650 | 2, 208 |
| | Cost | $132,500 | $132,500 | $110,400 |
| | (ii) Scrap Iron, MTons | 1,768 | 1,768 | 1,472 |
| | Cost | $ 53,000 | $ 53,000 | $ 44,200 |
| 6. | Water - 2MM galls/Month | $ 21,600 | $ 21,600 | $ 18,000 |
| 7. | Labour | $ 30,000 | $ 30,000 | $ 30,000 |
| 8. | Supervision | $ 22,000 | $ 22,000 | $ 22,000 |
| 9. | Power and Insurance etc. | $ 1,800 | $ 1,800 | $ 1,800 |
| | Sub-Total | $688,400 | $645,400 | $228,400 |
| | Contingency @ 15% | $103,300 | $ 96,800 | $ 34,300 |
| | Total | $791,700 | $742,200 | $262,700 |
| | (Say) | $792,000 | $742,000 | $263,000 |

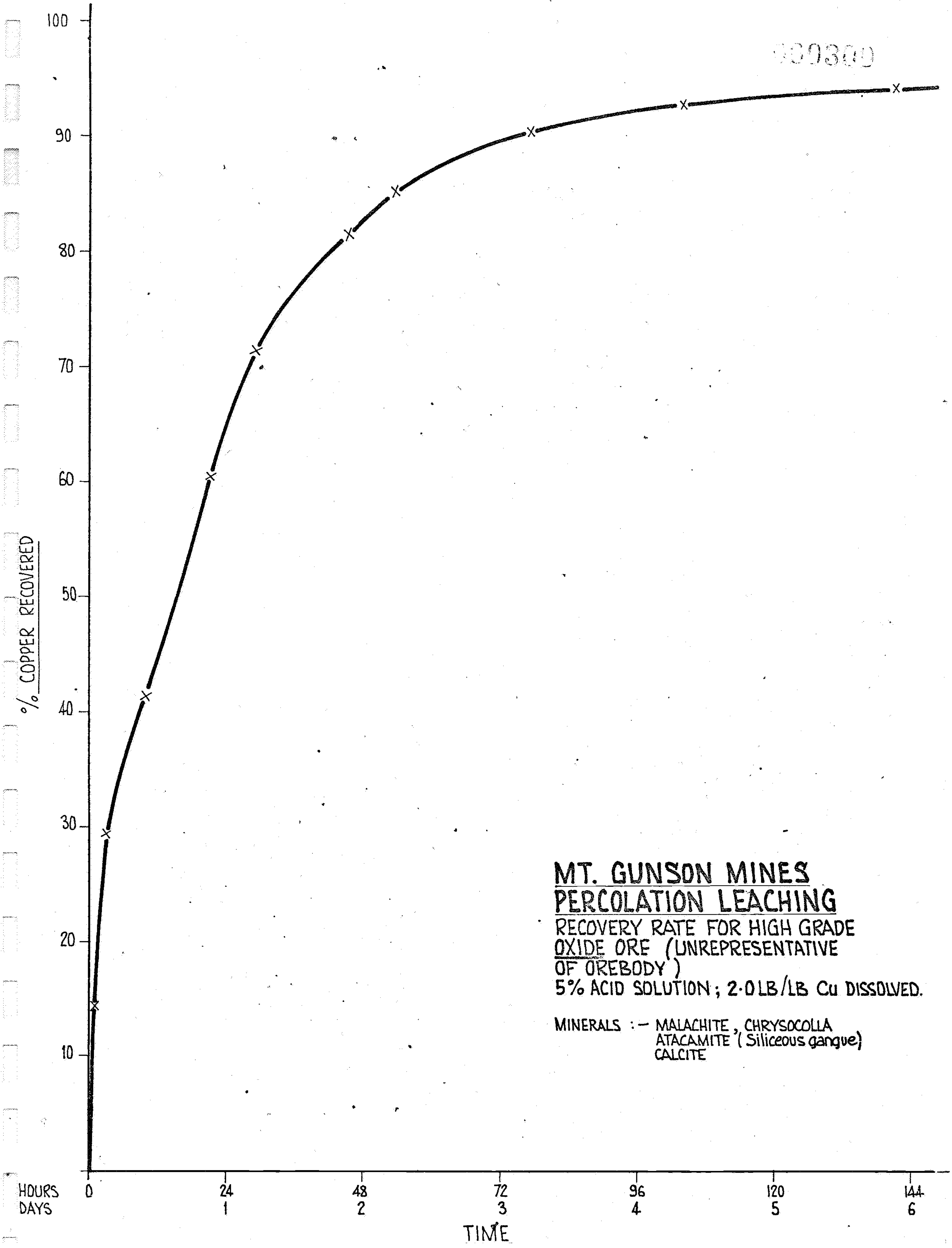
## II IN-SITU LEACHING - EAST/WEST LAGOON

## OPERATING COSTS

| | | Year 1 | Year 2 | Year 3 |
|---|---|---|---|---|
| | Production M/Tons Cu | 830 | 970 | 868 |
| 1. | Drill, (West Lagoon only) | 5,600ft. | - | - |
| | Cost | $ 11,200 | - | - |
| 2. | Blast, (West Lagoon only) | $ 12,000 | - | - |
| 3. | Re-Drill and Blast (at end of each year up to second year) (East and West Lagoons) | $ 90,000 | $ 90,000 | - |
| 4. | Sink pump sump (West Lagoon only) | $ 2,000 | - | - |
| 5. | Pumping Maintenance | $ 1,000 | $ 1,000 | $ 1,000 |
| 6. | Pipeline Maintenance | $ 1,000 | $ 1,000 | $ 1,000 |
| 7. | Reagents | | | |
| | (i) $H_2SO_4$, MTons | 2,490 | 2,910 | 2,604 |
| | Cost | $124,500 | $145,500 | $130,200 |
| | (ii) Scrap Iron, MTons | 1,660 | 1,940 | 1,736 |
| | Cost | $ 48,800 | $ 58,200 | $ 52,100 |
| 8. | Water | $ 21,600 | $ 21,600 | $ 21,600 |
| 9. | Labour | $ 30,000 | $ 30,000 | $ 30,000 |
| 10. | Supervision | $ 22,000 | $ 22,000 | $ 22,000 |
| 11. | Power and Insurance etc. | $ 7,200 | $ 7,200 | $ 7,200 |
| | Sub-Total | $372,300 | $376,500 | $265,100 |
| | Contingency @ 15% | $ 55,800 | $ 56,500 | $ 39,000 |
| | Total | $428,100 | $433,000 | $304,900 |
| | (Say) | $428,000 | $433,000 | $305,000 |

## APPENDIX A

## GRAPHICAL RESULT OF PERCOLATION LEACH OF HIGH GRADE MT. GUNSON OXIDE ORE (UNREPRESENTATIVE OF OREBODY)



MT. GUNSON MINES
PERCOLATION LEACHING
RECOVERY RATE FOR HIGH GRADE
OXIDE ORE (UNREPRESENTATIVE
OF OREBODY)
5% ACID SOLUTION; 2·0 LB/LB Cu DISSOLVED.

MINERALS :- MALACHITE, CHRYSOCOLLA
ATACAMITE (Siliceous gangue)
CALCITE

# APPENDIX B

# PRODUCTION SCHEDULES

## PRODUCTION SCHEDULES

The production schedules were drawn up and based on the following assumptions: -

1. Heap Leaching
    1. For 230 days/year and on a one shift basis, stripping of the overburden at 2250 M.Tons/day would proceed concurrently with mining of the ore at 750 M.Tons/day after an initial 70,000 M. Tons of overburden were removed. This gives a movement of material at approximately 3000 M.Tons/day. The cost rates used are those at present operating in the contract with Davis Contractors.

    2. One heap of 100,000 M.Tons of ore would be needed to provide enough copper bearing material for leaching to continue whilst the mining and heaping of the rest of the orebody proceeded.

    3. Water requirements would be built up to 10 million gallons for the initial heap and from then on only make up water would be required plus water for the camps etc... and general purpose. A cyclic application of water would be employed to provide (1) aeration and wetting cycles, and (2) conservation of supply.

    4. Collection of percolated liquor would start 2 months after application and would proceed from the various dumps until completion, 4 years after initiating the project, i.e. a liquor collection life of approximately 3 years is contemplated.

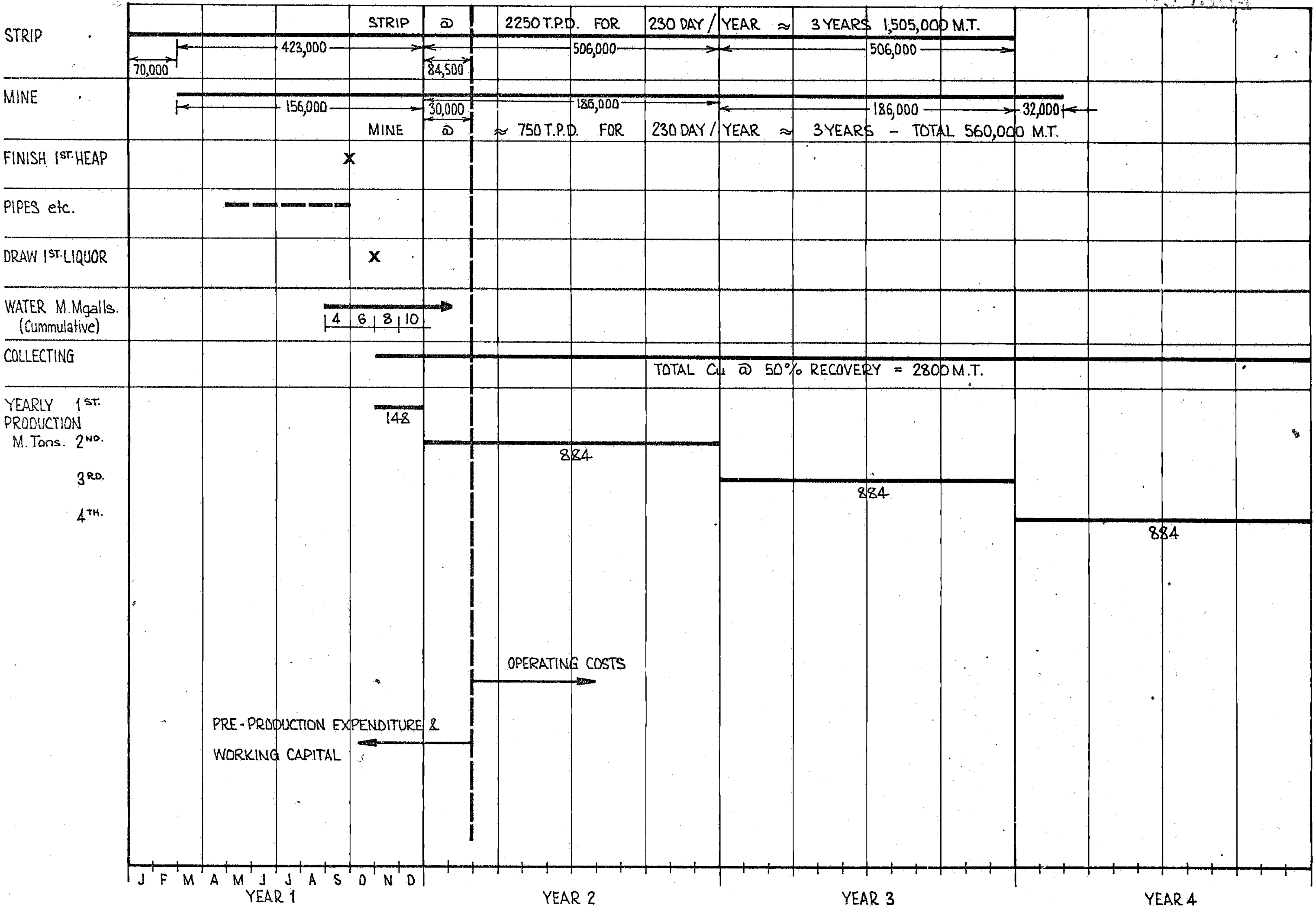
2. In-Situ Leaching
    1. Removal of 2 feet of mud and several feet of mudstone from the areas overlying the orebodies would be necessary in order to ensure adequate percolation of the liquid into the orebody. This would also remove from the lagoon surface any acid consuming material such as gypsum and calcereous deposits. The earth moving operation would

incorporate the building of retaining walls to keep out mud and liquid from the lagoon.

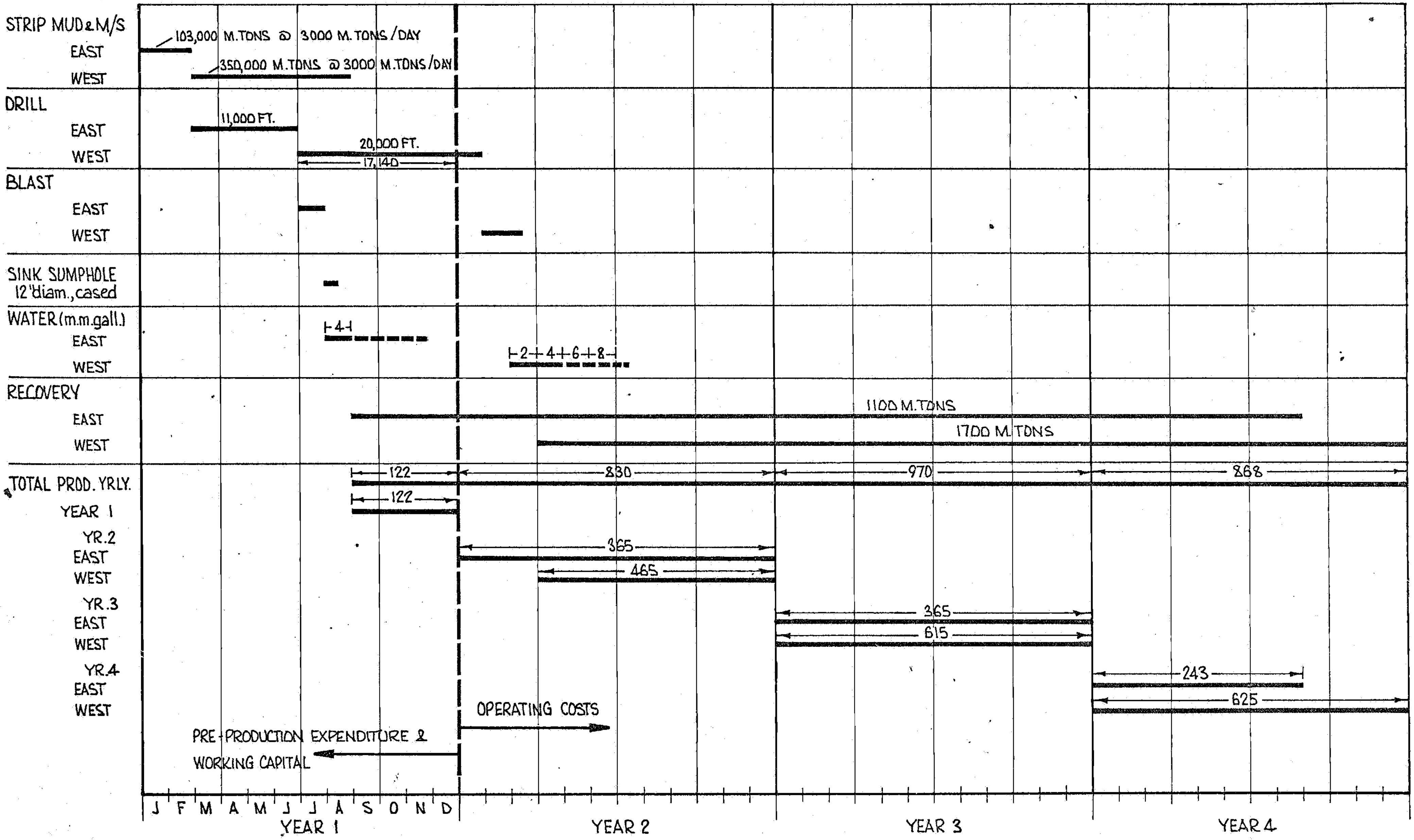
2. Drilling of one zone would commence immediately after the mud and mudstone were removed and would proceed at the rate of approximately 150 ft/day.

3. Water would be required to fill up the pore space in the shattered orebodies with an initial volume of 4 million gallons. Thereafter, make up water only is required for this and other zones as a cyclic wetting and aeration method would be employed.

4. Recovery would proceed after the initial one month wetting period (during which most of the 4 million gallons will have been supplied) and continue uninterrupted until all the zones have been treated completely. A life of just over 3 years is contemplated during which time 2800 M.Tons of copper metal would have been recovered.



HEAP LEACHING PRODUCTION SCHEDULES EAST/WEST LAGOON



INSITU LEACHING PRODUCTION SCHEDULES EAST/WEST LAGOON

# APPENDIX C

# PRESENT VALUE STUDIES

### A. Deductions for selling of a marketable product

A brief study by Tennant Trading Company suggests the copper in precipitate form would bring a better price from the European Market than if it were transported to E.R. & S. at Port Kembla for treatment and subsequent marketing.

A reasonable deduction would then be A$130.00/MTon of copper metal + 2.0% toll and marketing expense at the LME settlement price.

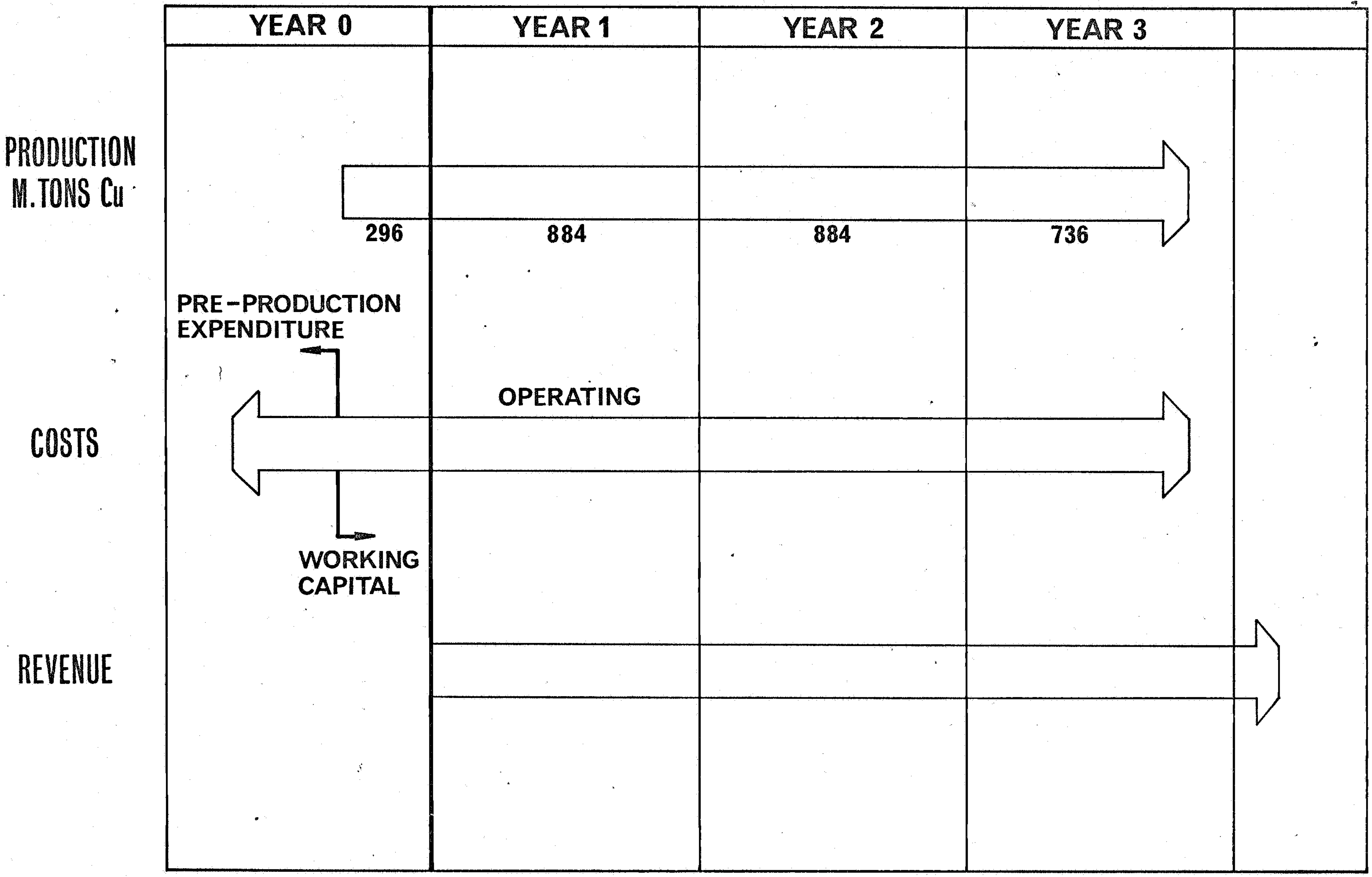
This figure assumes the following: -

(i) cement copper will be loaded in geetainers.
(ii) road freighted to Adelaide.
(iii) sea freighted to Europe.
(iv) treated and marketed in Europe.

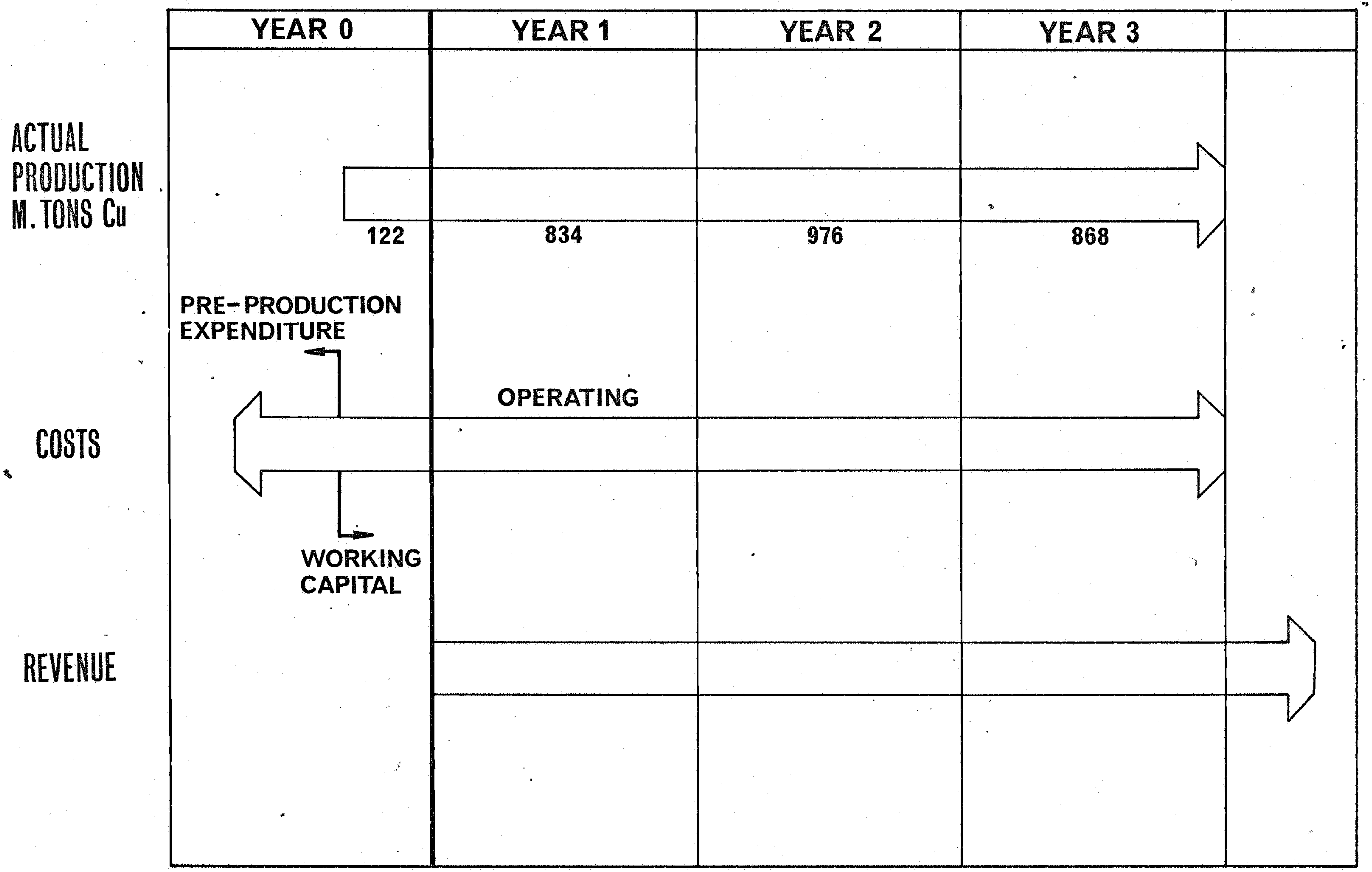
The deduction does not account for security arrangements, dust losses and loading costs at the mine etc. but includes insurance (geetainer, plus cargo) hire rates, all treatment costs and all transport costs.

### B. Production/Revenue relationships and Calculations

These follow and are in diagramatic and tabular form.



HEAP LEACHING ; PRODUCTION / REVENUE RELATIONSHIPS



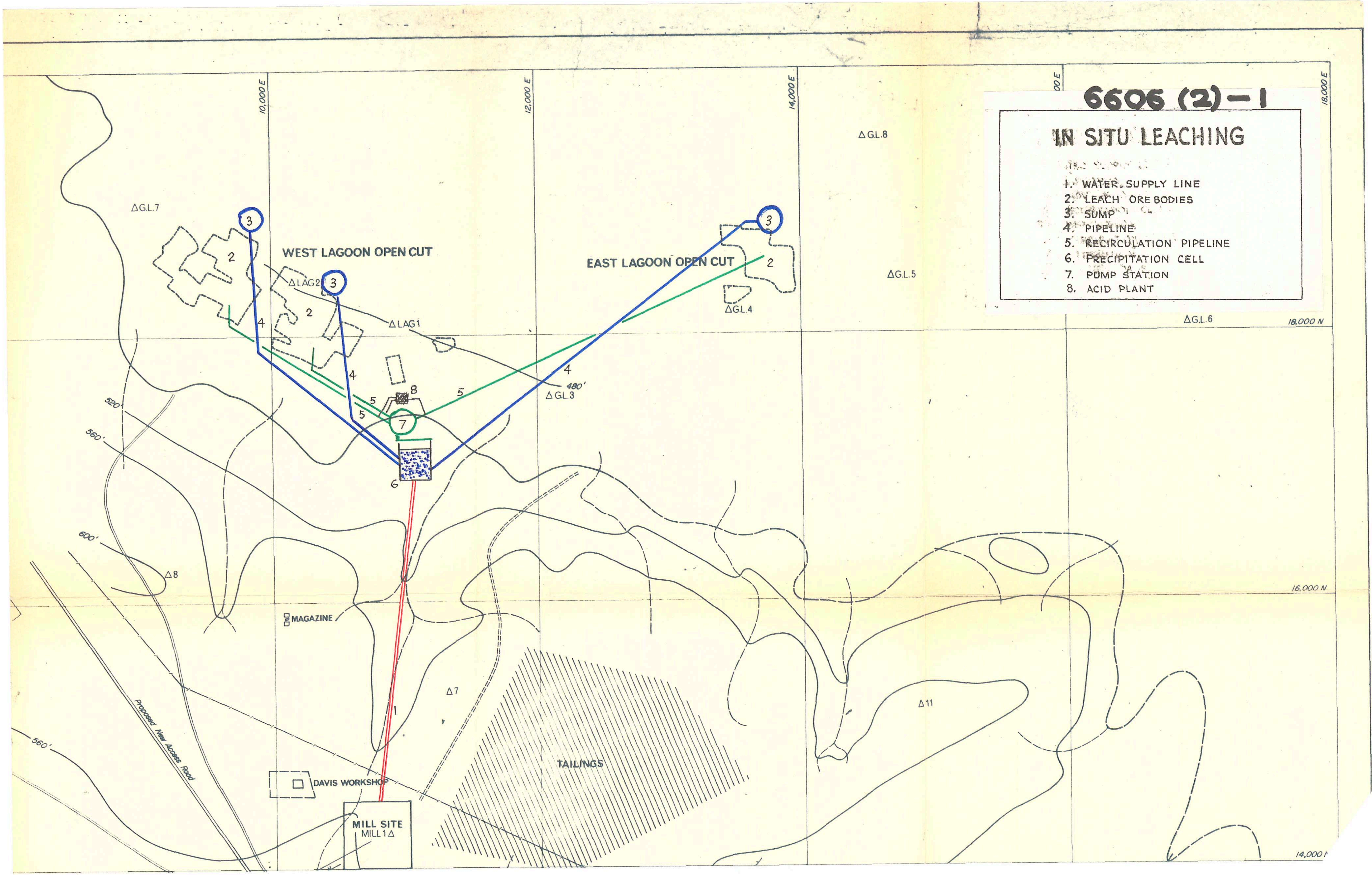
INSITU LEACHING ; PRODUCTION/ REVENUE RELATIONSHIPS

PRESENT VALUE I HEAP LEACHING OPERATION - EAST/WEST LAGOON

| COPPER PRICE £$/MTon | YEAR 0 | | | YEAR 1 | | | YEAR 2 | | | YEAR 3 | | | YEAR 4 | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | 400 | 450 | 500 | 400 | 450 | 500 | 400 | 450 | 500 | 400 | 450 | 500 | 400 | 450 | 500 |
| A$/MTon | 856 | 963 | 1070 | 856 | 963 | 1070 | 856 | 963 | 1070 | 856 | 963 | 1070 | 856 | 963 | 1070 |
| 1. PRODUCTION MTons Cu metal | | 296 | | | 884 | | | 884 | | | 736 | | | - | |
| Payment in MTons metal | | - | | | 884 | | | 884 | | | 884 | | | 148 | |
| 2. REVENUE A$ | | - | | 756700 | 851300 | 945900 | 756700 | 851300 | 945900 | 756700 | 851300 | 945900 | 126700 | 142500 | 158400 |
| - deductions | 43500 | 44100 | 44700 | 130000 | 131700 | 133500 | 130000 | 131700 | 133500 | 108200 | 109700 | 111100 | - | - | - |
| 3. NET REVENUE | - | - | - | 626700 | 719600 | 812400 | 626700 | 719600 | 812400 | 648500 | 741600 | 834800 | 126700 | 142500 | 158400 |
| - Royalties | | | | | | | | | | | | | | | |
| Jervois 1½% | - | - | - | 9400 | 10800 | 12200 | 9400 | 10800 | 12200 | 9700 | 11100 | 12500 | 1900 | 2100 | 2300 |
| S.A. Govt. 2½% | - | - | - | 15700 | 18000 | 20300 | 15700 | 18000 | 20300 | 16200 | 18500 | 20900 | 3200 | 3600 | 4000 |
| - Production Costs | | | | | 792000 | | | 742000 | | | 263000 | | | - | |
| 4. NET OPERATING REVENUE | - | - | - | - | - | - | - | - | 37900 | 359600 | 449000 | 538400 | 121600 | 136800 | 152100 |
| - Tax* (20% exempt income for Cu producers). | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| 5. CASH FLOW (OPERATIONS) | -43500 | -44100 | -44700 | -139400 | -101200 | -12100 | -140400 | -51200 | 37900 | 359600 | 449000 | 538400 | 121600 | 136800 | 152100 |
| - Equity | | 805000 | | | | | | | | | | | | | |
| 6. NET CASH FLOW | -848500 | -849100 | -849700 | -190400 | -101200 | - 12100 | -140400 | -51200 | 37900 | 359600 | 449000 | 538400 | 121600 | 136800 | 152100 |
| P.V. Factor at 15% | | 1.0 | | | 0.869 | | | 0.756 | | | 0.658 | | | 0.571 | |
| 7. PRESENT VALUE | -848500 | -849100 | -849700 | -165500 | -87900 | -10500 | -106100 | -38700 | 28700 | 236600 | 295400 | 354300 | 69400 | 78100 | 86800 |
| Cummulative P.V. | | | | -1014000 | -937000 | -860200 | -1120100 | -975700 | -831500 | -883500 | -680300 | -477200 | -814100 | -602200 | -390400 |
| 8. NET PRESENT VALUE | -814,100 | -602200 | -390400 | | | | | | | | | | | | |
| * Tax Calculation | | | | | | | | | | | | | | | |
| Net Op. Revenue | | | | | | | | | 37900 | 359600 | 449000 | 538400 | 121600 | 136800 | 152100 |
| Deprec. | | | | | | | | | 37900 | 359600 | 449000 | 538400 | 121600 | 136800 | 152100 |
| Taxable Income | | | | | | | | | - | - | - | - | - | - | - |
| TAX (38%). | | | | | | | | | - | - | - | - | - | - | - |

PRESENT VALUE II INSITU LEACHING OPERATION - EAST/WEST LAGOON

| | YEAR 0 | | | YEAR 1 | | | YEAR 2 | | | YEAR 3 | | | YEAR 4 | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Copper Price £$/MTon | 400 | 450 | 500 | 400 | 450 | 500 | 400 | 450 | 500 | 400 | 450 | 500 | 400 | 450 | 500 |
| A$/MTon | 856 | 963 | 1070 | 856 | 963 | 1070 | 856 | 963 | 1070 | 856 | 963 | 1070 | 856 | 963 | 1070 |
| 1. PRODUCTION MTons Cu metal | | 122 | | | 834 | | | 976 | | | 868 | | | - | |
| Payment in MTons metal | | - | | | 678 | | | 926 | | | 904 | | | 292 | |
| 2. REVENUE A$ | - | - | - | 580400 | 652900 | 725500 | 792700 | 891700 | 990800 | 773800 | 870600 | 967300 | 250000 | 281200 | 312400 |
| - deductions | 17900 | 18200 | 18400 | 122600 | 124300 | 125900 | 143500 | 145400 | 147400 | 127600 | 129300 | 131100 | - | - | - |
| 3. NET REVENUE | - | - | - | 457800 | 528600 | 599600 | 649200 | 746300 | 843400 | 646200 | 741300 | 836200 | 250000 | 281200 | 312400 |
| - Royalties | | | | | | | | | | | | | | | |
| Jervois 1½% | - | - | - | 6900 | 7900 | 9000 | 9700 | 11200 | 12700 | 9700 | 11100 | 12500 | 3800 | 4200 | 4700 |
| S.A. Govt 2½% | - | - | - | 11400 | 13200 | 15000 | 16200 | 18700 | 21100 | 16200 | 18500 | 20900 | 6300 | 7000 | 7800 |
| - Production Costs | | | | | 428000 | | | 433000 | | | 305000 | | | - | |
| 4. NET OPERATING REVENUE | - | - | - | 11500 | 79500 | 147600 | 190300 | 283400 | 376600 | 315300 | 406700 | 497800 | 239900 | 270000 | 299900 |
| - Tax* (20% exempt income for Cu productions) | - | - | - | - | - | - | - | - | - | - | 60800 | 156900 | 56200 | 102600 | 114000 |
| 5. CASH FLOW (OPERATIONS) | -17900 | -18200 | -18400 | 11500 | 79500 | 147600 | 190300 | 283400 | 376600 | 315300 | 345900 | 340900 | 183700 | 167400 | 185900 |
| - Equity | | 609000 | | | | | | | | | | | | | |
| 6. NET CASH FLOW | -626900 | -627200 | -627400 | 11500 | 79500 | 147600 | 190300 | 283400 | 376600 | 315300 | 345900 | 340900 | 183700 | 167400 | 185900 |
| P.V. Factor at 15%. | | 1.00 | | | 0.869 | | | 0.756 | | | 0.658 | | | 0.571 | |
| 7. PRESENT VALUE | -626900 | -627200 | -627400 | 10000 | 69100 | 128300 | 143900 | 214300 | 284700 | 207500 | 227600 | 224300 | 104900 | 95600 | 106100 |
| Cummulative P.V. | | | | -616900 | -558100 | -499100 | -473000 | -343800 | -214400 | -265500 | -116200 | 9900 | -160600 | -20600 | 116000 |
| 8. NET PRESENT VALUE | -160600 | -20600 | 116000 | | | | | | | | | | | | |
| | | | | | | | | | | | | | | | |
| *Tax Calculation | | | | | | | | | | | | | | | |
| Net Op. Revenue | | | | 11500 | 79500 | 147600 | 190300 | 283400 | 376600 | 315300 | 406200 | 497800 | 239900 | 270000 | 299900 |
| Deprec. | | | | 11500 | 79500 | 147600 | 190300 | 283400 | 376600 | 315300 | 246100 | 84800 | 91900 | - | - |
| Taxable INCOME | | | | - | - | - | - | - | - | - | 160100 | 413000 | 148000 | 270000 | 299900 |
| TAX (38%) | | | | - | - | - | - | - | - | - | 60800 | 156900 | 56200 | 102600 | 114000 |



6606 (2)-1

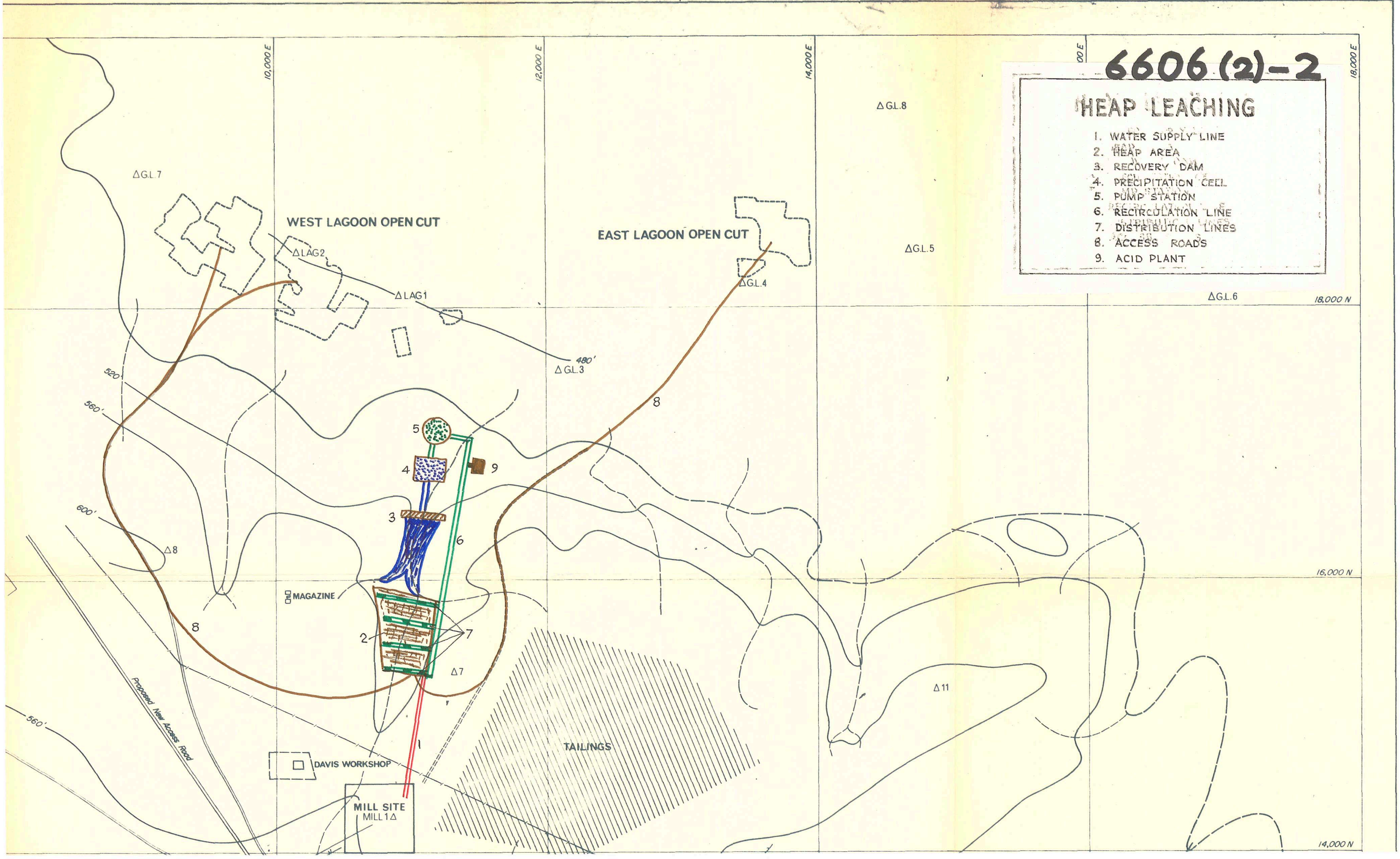
# IN SITU LEACHING

1. WATER SUPPLY LINE
2. LEACH ORE BODIES
3. SUMP
4. PIPELINE
5. RECIRCULATION PIPELINE
6. PRECIPITATION CELL
7. PUMP STATION
8. ACID PLANT

6606(2)-2

# HEAP LEACHING

1. WATER SUPPLY LINE
2. HEAP AREA
3. RECOVERY DAM
4. PRECIPITATION CELL
5. PUMP STATION
6. RECIRCULATION LINE
7. DISTRIBUTION LINES
8. ACCESS ROADS
9. ACID PLANT



**PACMINEX PTY. LIMITED**

**MEMORANDUM TO:** THE GENERAL MANAGER,
MOUNT GUNSON MINES PTY. LTD.

**OUR REF:** RND:LDR

**FROM:** R.N. DUDGEON

**DATE:** 18.4.1972

# PLANT TRIALS MAIN OPEN CUT SULPHIDE ORE DECEMBER, 1971.

Approximately 2,200 m.t. of sulphide ore was delivered to the plant during December, 1971. For a number of operating reasons it was not possible to keep this ore completely separate from West Lagoon Ore and so a proportion of it was processed mixed with West Lagoon Ore. Main Open Cut Ore was processed on December 7th and 10th and assays and recoveries are reported for periods during which only this ore was passing through the mill unmixed with Lagoon ore. This amounted to 1,437 m.t.

The initial delivery of M.O.C. ore was made on the afternoon of 2nd December. This ore was very sticky and contained a high proportion of clay. This introduced problems in crushing and screening. The feed rate to the Fine Ore Bin was about 25 T.P.H. whereas plant throughput was set at 50 T.P.H. Thus the material had to be mixed to maintain throughput. There was practically no froth generated in the flotation cells even by increasing the frother addition and air to the flotation cells. This could have been due to either contamination by oil from the plant or a contaminant in the ore. Frother problems continued during the following day. Reagent additions were as for West Lagoon Ore.

A further delivery of M.O.C. ore was made to the plant on 7th December. This material also contained clays which caused blinding of screens and restricted crushing. The ore appeared to process reasonably well with no frother problems.

. . . /2

The recovery in excess of 75 per cent appears reasonable for a feed high in clays.

Laboratory flotation tests were carried out on grab sample of Rougher Feed taken at 12 p.m. on December, 7th. Results were discouraging and the test was repeated with a sulphidizing collector, AF 425 in place of xanthate with some apparent improvement. However, this material frothed excessively even with reduced air to the cell. The sulphides were slow in floating and during the latter period of the test excessive slime material was carried over with the froth which contributed to the low grade of the concentrate. This grab sample was not representative of the ore body, or even the feed to the plant during that shift but does point to the fact that the ore body is extremely variable and the need to blend feed. Results are tabulated in Table 1.

The variation in the calculated head grade between the two tests did suggest that such tests should be repeated, however, as the material was not representative this did not appear worthwhile.

M.O.C. ore received by the plant assayed 0.58% copper of which 0.11% was 'oxide' copper. The plant tails assay at the time was 0.19% Cu. Plant recovery was 76% which was in line with the grab tests.

The plant was operated normally using sod. sec. butyl xanthate as collector (approx. 0.3 lb/ton ore) and AF 65 as frother.

On Wednesday 8th December, due to the high proportion of partly oxidized feed from West Lagoon ore stockpiles, a sulphidizing reagent, AP 425 was added to the ball mill feed giving what appeared to be a marginal improvement in flotation.

. . /3

With the great variations in feed it is difficult to quantify the effect of such changes, however, concentrate grades did appear to improve.

On Friday 10th December, M.O.C. ore was delivered to the plant and no further changes were made in reagent additions - the AF425 was added to the ball mill at a rate of approximately 0.25 lb/ton ore. M.O.C. ore was processed undiluted and the physical condition of the material was such that there were no major problems in crushing and screening. Care was taken during these tests not to mix in West Lagoon Ore from the coarse ore stockpile. The type of froth produced was not all to be desired but was adequate. The pulp appeared muddy due to the higher than normal proportions of clay in the feed. Copper recovery varied from 73 to 87% with an improvement in concentrate grade.

On Saturday 11th a visit was made to the M.O.C. There appeared to be good ore on the floor of the pit although the walls revealed that some of the higher horizons contained clay and some oxidized ore.

On Monday 13th the floor of the M.O.C. pit was scraped and a further tonnage of ore was delivered to the plant and mixed with stockpile ore. It had no deleterious effects except that it was rather wet and caused some hanging-up in the F.O.B.

Results of the test runs are tabulated in Table 2.

## CONCLUSION:

The bulk of the M.O.C. ore as processed was probably not typical of the deposit in that it contained a considerable proportion of clay and oxidized material from higher horizons. Only a small amount of high grade ore from the lower horizons

. . /4

was processed and it appears that there is no real difficulty in processing this material. However, because of the variability within the deposit it would appear essential that the ore be mixed and blended before processing in an endeavour to produce a more consistent feed both for copper grade and the presence of clays and other materials which can cause processing difficulties. It is essential to accurately delineate this ore body and if possible carry out some laboratory test work to determine the optimum combination of flotation reagents and type of flocculant needed. The mining plan should be so designed to produce a consistent feed and every endeavour must be made to minimize big fluctuations in ore grade. It may be necessary to mine on more than one face simultaneously or alternatively stockpile ore and blend. The additional costs involved in either of these operations could be justified by improved metallurgical efficiency of the plant.

Test work and plant experience to date indicate that M.O.C. ore in the area designated Area A can be processed to give a satisfactory grade of concentrate and recovery. The variability in the ore will from time to time result in plant difficulties which can be minimized by blending and judicious alterations to reagent additions.

R.N. DUDGEON

18th April, 1972.

## TABLE 1.

## LABORATORY FLOTATION TEST - M.O.C. ORE

## GRAB SAMPLE ROUGHER FLOTATION FEED 7.12.1971.

| TEST NO. | REAGENT | CONS. GRADE | TAILS | RECOVERY % Cu | CALC. HEAD GRADE |
|---|---|---|---|---|---|
| R 80 | S.S.B. Xanthate | 3.6<br>(0.7 'oxide') | 0.12 | 76.9 | 0.47 |
| R 81 | AP 425 | 6.0<br>(1.0 'oxide') | 0.17 | 79.1 | 0.74 * |

<u>NOTE</u>: * Samples were taken consecutively.
Actual head 0.46.

# TABLE 2.

## PLANT TEST M.O.C. ORE

| DATE | TIME | TONNES | ASSAYS % Cu | | | RECOVERY |
|---|---|---|---|---|---|---|
| | | | HEAD | TAILS | CONS | |
| 7 | D/S | 337 | 0.67 | 0.18 | 52.0 | 75.8 |
| | A/S | 300 | 0.41 | 0.10 | 52.5 | 76.4 |
| 10 | D/S 8-12 | 382 | 1.09 | 0.26 | 62.0 | 76.5 |
| | 12- 4 | | 3.20 | 0.43 | 62.0 | 87.2 |
| | A/S 4- 8 | 418 | 1.15 | | Not applicable | |
| | 8-12 | | 1.10 | 0.35 | 57.5 | 73.4 |

**NOTE:** Mechanical problems A/S 10/12 4-8 p.m. resulted in 45% recovery.

**REAGENTS:** 7-12-71 Xanthate additions to ball mill and flotation cells
10-12-71 AP 425 addition to ball mill and Xanthate to flotation cells.

Assays per cent Cu Grab Samples 12 P.M. 7-12-71

| | | |
|---|---|---|
| Rougher Floats | - | 27.0 |
| Scavenger Floats | - | 2.1 |
| Final Concentrates | | 57.5 |
| Final Tails | - | 0.14 |
| M.O.C. Feed | - | 0.46 |

# MOUNT GUNSON MINES PTY. LTD.

# REPORT ON CONCENTRATING PLANT AT MOUNT GUNSON

D.N. LARKIN
12.12.1972

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## Section 1

## INTRODUCTION

The first section of this report covers an investigation into the cost of the work required to prepare the concentrating plant at Mt. Gunson for operation without modification. All work covered in this section of the report is essential and a detailed estimate of cost has been prepared.

The second section of this report covers a study of :

1. the downtime and high labour areas in the plant

and

2. the accommodation facilities.

I have then itemised possible modifications which have been suggested and made a lump sum estimate of their cost.

## SUMMARY

The estimated cost of preparing the concentrating plant at Mt. Gunson for operation at a projected throughput not more than 30,000 tonnes/month is $90,000. This does not include work required on Tailings Dam or concentrate handling facilities.

I do not recommend starting operating again without some minor and major modifications to the crushing circuit, concentrate handling facilities, tailings disposal system and accommodation facilities.

The lump sum estimate of the essential modifications suggested is $850,000.

These modifications should reduce the plant down-time and therefore increase the annual tonnage of ore treated.

## RECOMMENDATIONS

1. No more detailed design work be done on the recommended modifications until geological reserves and feasibility study justify.

2. At least the work covered in Section 1 and all the modifications under the heading "Essential" be done before or within 3 months of a startup of operations at Mt. Gunson.

   An amount of at least 1 million dollars be allowed for this work in any feasibility study of the operation.

3. Estimates should be escalated to cover annual increases from December, 1972.

## MOUNT GUNSON CONCENTRATOR

### Section 1.

This section of the report covers an investigation into the cost of the work required to prepare the plant at Mount Gunson for operations.

For the purposes of this section it has been assumed that no modifications would be made to the circuit and that we would have a projected throughput of not more than 30,000 tonnes/month (average monthly tonnage to-date 24,300). The plant availability without modification would be unsatisfactory. In costing this work all labour has been charged at $5.00/hour; (assuming 10 hour 6 day week at $2.00/hour ordinary time + 100% overhead); material and spares have been included at cost even if they are at present on site as spares, stock would have to be replenished before commencement. A handling charge of 5% has been added to the total cost of materials.

The work involved has been divided into two major sections :

(i) Crushing Plant

and

(ii) Concentrating Plant

and set out in table form. A contingency allowance of 10% has been made on the total.

Two areas of operation have not been included in this estimate. They are :

(i) Concentrate handling after filtration. This is dependent on at least two unknowns viz. where the concentrate is to be sent and the type of container that will be used to ship it etc.

and

(ii) Tailings Dam. The present tailings disposal system is unsatisfactory because :

(a) Heavy tailings are pumped uphill to the dam whilst lighter return water flows back by gravity.

(b) The tailings line runs through the middle of the crushing circuit and is buried under a haul road. This creates an impossible situation when leaks occur.

(c) The mild steel line has been patched in many places and must be completely replaced before startup.

(d) The walls of the present dam would require considerable earthworks if they are to contain the discharge of tailings at startup.

(e) The clear water pond in the dam is completely silted up.

(f) The dam surface is almost flat. Water tends to soak in or evaporate rather than flow down to the clear water pond giving insufficient return water.

(g) The creek bed below the thickener is choked with tailings that have been discharged while pipe or pump repairs were done.

The tailings disposal system will require considerable investigation when factors such as the following are known :

(a) Annual tonnage,

(b) Expected life,

(c) Location of haul and access roads etc.

The total cost of the work covered in the tables following is estimated at $90,000. It would be preferable to engage our own team of tradesmen to perform some of this work. It may then be possible to select from these men, a good maintenance team to be kept on when the plant is operational. This has the advantage that the maintenance team is familiar with the plant before any operating problems occur.

I do not recommend restarting the concentrating plant without major alterations to the crushing circuit, concentrate handling and tailings disposal system.

(See section 2 of report)

## CRUSHING CIRCUIT

| EQUIPMENT NO. | DESCRIPTION OF WORK | LABOUR | MATERIAL | TOTAL |
|---|---|---|---|---|
| 1010<br>Grizzly Feeder | 1. Repair motor holding down bolts | 90 | 20 | 110 |
| | 2. Renew bearings in Electric motor | 65 | 25 | 90 |
| | 3. Refit motor and connect. | 25 | 5 | 30 |
| | 4. Repair and refit belt guard. | 45 | 5 | 50 |
| | 5. Renew V-belts. | 5 | 10 | 15 |
| | 6. Change oil in drive gear case. | 5 | 5 | 10 |
| | 7. Repair and rebuil~~t~~d steelwork around discharge to Jaw crusher | 450 | 70 | 520 |
| | | 685 | 140 | 825 |
| 1035<br>Secondary Screen. | 1. Refit drive motor and connect. | 10 | 5 | 15 |
| | 2. Repair side walls where worn to prevent spillage | 150 | 10 | 160 |
| | 3. Renew V-belts | 5 | 10 | 15 |
| | 4. Repair and refit belt guard | 15 | 5 | 20 |
| | 5. Renew screen deck. | 60 | 270 | 330 |
| | 6. Renew wear plates and part of body of discharge chute. | 200 | 40 | 240 |
| | 7. Renew linatex in surge bin under screen. | 40 | 90 | 130 |
| | | 480 | 430 | 910 |
| 1090<br>Tertiary Screen. | 1. Refit drive motor and connect. | 10 | 5 | 15 |
| | 2. Renew motor rails | 10 | 20 | 30 |
| | 3. Renew V-belts | 5 | 10 | 15 |
| | 4. Repair and refit belt guard. | 10 | 5 | 15 |
| | 5. Renew screen deck. | 20 | 80 | 100 |
| | 6. Renew linatex in surge bin under screen. | 40 | 90 | 130 |
| | 7. Fit new fines bypass chute. | 225 | 30 | 255 |
| | | 320 | 240 | 560 |
| 1015<br>O.H.T. Crane over primary Crusher. | 1. Renew brass cable guides. | 230 | 80 | 310 |
| | 2. Repair limit switching gear. | 60 | 20 | 80 |
| | 3. Clean all chain drives and regrease. | 25 | 5 | 30 |
| | | 315 | 105 | 420 |

## CRUSHING CIRCUIT

| EQUIPMENT NO. | DESCRIPTION OF WORK | LABOUR | MATERIAL | TOTAL |
|---|---|---|---|---|
| 1020<br>Primary Crusher | 1. Reline Jaws of crusher. | 320 | 3,500 | 3,820 |
| | 2. Renew cheek plate liners. | 180 | 700 | 880 |
| | 3. Refit and connect motor, test run. | 35 | 5 | 40 |
| | 4. Replace V-belts (7 x D250) | 15 | 70 | 85 |
| | 5. Repair Belt Guard. | 40 | 20 | 60 |
| | 6. Change oil in tank.<br>Clean and check oil pump. | 30 | 25 | 55 |
| | 7. Check toggle seat alignment. | 5 | – | 5 |
| | 8. Repair walkway above crusher | 80 | 5 | 85 |
| | | 705 | 4,325 | 5,030 |
| 1075<br>Secondary Crusher. | 1. Refit and connect motor, test run. | 45 | 5 | 50 |
| | 2. Replace V-Belts. | 5 | – | 5 |
| | 3. Repair oilpump motor and gearbox unit. | 55 | 25 | 80 |
| | 4. Change oil in crusher and oil tank. | 80 | 55 | 135 |
| | 5. Repair breather on oil tank. | 10 | 5 | 15 |
| | 6. Free adjusting mechanism | 20 | – | 20 |
| | 7. Renew wear bars and plates in discharge chute<br>Repair conveyor skirt plates | 370 | 45 | 415 |
| | | 585 | 135 | 720 |
| 1095<br>Tertiary Crusher. | 1. Refit and reconnect motor, test run. | 45 | 5 | 50 |
| | 2. Replace V-belts | 5 | – | 5 |
| | 3. Change oil in crusher and oil tank. | 80 | 55 | 135 |
| | 4. Repair breather on oil tank | 10 | 5 | 15 |
| | 5. Free adjusting mechanism. | 20 | – | 20 |
| | 6. Renew wear bars and plates in discharge chute | 370 | 45 | 415 |
| | | 530 | 110 | 640 |

## CRUSHING CIRCUIT

| EQUIPMENT NO. | DESCRIPTION OF WORK | LABOUR | MATERIAL | TOTAL |
|---|---|---|---|---|
| 1025<br>No. 1 Conveyor | 1. Refit conveyor belt and extend belt (2 Vulcanised joints) | 140 | 100 | 240 |
| | 2. Renew Flaseals in drive gearbox and change oil. | 90 | 15 | 105 |
| | 3. Refit electric motor and connect | 30 | 5 | 35 |
| | 4. Renew Poly-Vee belt | 5 | 7 | 12 |
| | 5. Fit new rubbers to impact and return idlers. | 120 | 200 | 320 |
| | 6. Clean and regrease<br>Head, Tail and Takeup pulley bearings. | 50 | 5 | 55 |
| | 7. Fit new skirting. | 90 | 60 | 150 |
| | 8. Renew cable on trip wire switch | 50 | 15 | 65 |
| | 9. Repair discharge chute to Secondary screen | 50 | 18 | 68 |
| | | 625 | 425 | 1,050 |
| 1045<br>No. 2 Conveyor | 1. Refit conveyor belt and extend belt | 160 | 100 | 260 |
| | 2. Repair small cuts in belt. | 40 | 20 | 60 |
| | 3. Renew Flaseals in drive gearbox and change oil | 120 | 15 | 135 |
| | 4. Refit electric motor and connect | 30 | 5 | 35 |
| | 5. Refit Poly-Vee belt and guard | 5 | - | 5 |
| | 6. Fit new rubbers to impact and return idlers | 160 | 200 | 360 |
| | 7. Clean and regrease Head Tail and Take up Pulleys bearings | 60 | 5 | 65 |
| | 8. Fit new skirting. | 70 | 40 | 110 |
| | 9. Complete modification of discharge chute. | 120 | 20 | 140 |
| | | 765 | 405 | 1,170 |

## CRUSHING CIRCUIT

| EQUIPMENT NO. | DESCRIPTION OF WORK | LABOUR | MATERIAL | TOTAL |
|---|---|---|---|---|
| 1065<br>No. 3 Conveyor | 1. Refit conveyor belt. (1 Vulcanized joint) | 140 | 100 | 240 |
| | 2. Renew Flaseals in drive gearbox and change oil. | 60 | 15 | 75 |
| | 3. Refit electric motor and connect | 30 | 5 | 35 |
| | 4. Refit poly-vee belt and guard | 10 | 5 | 15 |
| | 5. Fit new rubbers to impact and return idlers. | 250 | 200 | 450 |
| | 6. Clean and regrease head, tail and takeup pulleys bearings | 50 | 5 | 55 |
| | 7. Fit new skirting. | 220 | 180 | 400 |
| | | 760 | 510 | 1,270 |
| 1080<br>No. 4 Conveyor | 1. Renew conveyor belt. | 130 | 1,300 | 1,430 |
| | 2. Renew Flaseals in drive gearbox and change oil. | 60 | 15 | 75 |
| | 3. Refit electric motor and connect | 30 | 5 | 35 |
| | 4. Replace poly-vee belt and guard | 5 | 10 | 15 |
| | 5. Fit new rubbers to impact and return idlers. | 90 | 200 | 290 |
| | 6. Clean and regrease head, tail and takeup pulleys | 50 | 5 | 55 |
| | 7. Fit new skirting | 120 | 60 | 180 |
| | | 485 | 1,595 | 2,080 |
| 1100<br>No. 5 Conveyor | 1. Refit conveyor belt (1 Vulcanized joint) | 120 | 100 | 220 |
| | 2. Repair cuts in belt | 25 | 20 | 45 |
| | 3. Renew flaseals in drive gearbox and change oil | 80 | 15 | 95 |
| | 4. Refit electric motor and connect | 30 | 5 | 35 |
| | 5. Replace poly-vee belt and guard | 10 | 10 | 20 |
| | 6. Fit new rubbers to impact and return idlers | 90 | 200 | 290 |
| | 7. Clean and regrease head, tail and take up pulleys | 50 | 5 | 55 |
| | 8. Fit new skirting. | 120 | 60 | 180 |
| | | 525 | 415 | 940 |

## CRUSHING CIRCUIT

| EQUIPMENT NO. | DESCRIPTION OF WORK | LABOUR | MATERIAL | TOTAL |
|---|---|---|---|---|
| 1110<br>No. 6 Conveyor | 1. Refit conveyor belt (1 vulcanised joint) | 160 | 100 | 260 |
| | 2. Renew flaseals in drive gearbox | 120 | 15 | 135 |
| | 3. Refit electric motor and connect | 30 | 5 | 35 |
| | 4. Replace poly-vee belt and guard | 10 | 10 | 20 |
| | 5. Fit new rubbers to impact and return idlers | 100 | 200 | 300 |
| | 6. Clean and regrease head, tail and take-up pulleys | 60 | 5 | 65 |
| | 7. Fit new skirting | 50 | 40 | 90 |
| | 8. Clean and repair belt underspeed | 20 | 5 | 25 |
| | 9. Renew discharge chute bottom and wear plates in top of chute | 650 | 150 | 800 |
| | | 1,200 | 530 | 1,730 |
| 2005<br>No. 7 Conveyor | 1. Renew conveyor belt (retain old belt as spare) | 80 | 1,300 | 1,380 |
| | 2. Clean and regrease head and tail pulleys | 20 | 5 | 25 |
| | 3. Fit new skirting | 40 | 60 | 100 |
| | 4. Polish conveyor table | 50 | 5 | 55 |
| | 5. Complete repair of drive control mechanism | 45 | 5 | 50 |
| | 6. Fit new chain to drive and repair chain guard | 95 | 120 | 215 |
| | | 330 | 1,495 | 1,825 |
| 2010<br>No. 8 Conveyor | 1. Refit conveyor belt | 140 | 100 | 240 |
| | 2. Replace flaseals in drive gearbox | 200 | 15 | 215 |
| | 3. Check electric motor and lineup | 10 | 5 | 15 |
| | 4. Fit new rubbers to impact and return idlers | 110 | 200 | 310 |
| | 5. Clean and regrease head, tail and takeup pulleys | 50 | 5 | 55 |
| | 6. Fit new skirting | 45 | 30 | 75 |
| | | 555 | 355 | 910 |

## CRUSHING CIRCUIT

| EQUIPMENT NO. | DESCRIPTION OF WORK | LABOUR | MATERIAL | TOTAL |
|---|---|---|---|---|
| 5060<br>Lime Storage Bin and Feeder | 1. Refit and connect electric motor | 20 | 5 | 25 |
| | 2. Repair drive chain and sprockets | 20 | 5 | 25 |
| | 3. Make up cover for top of bin | 10 | 5 | 15 |
| | | 50 | 15 | 65 |
| 1055<br>Stockpile Armco Tunnel | 1. Clear windblown dirt from tunnel | 45 | – | 45 |
| | 2. Repair light fittings in tunnel | 30 | 20 | 50 |
| | | 75 | 20 | 95 |
| 1067<br>Stockpile Discharge Feeders | 1. Weld cracks in feeder chutes | 80 | 5 | 85 |
| | 2. Renew wear bars and plates in chutes | 380 | 25 | 405 |
| | 3. Check operation of feeders and repair as necessary | 30 | 5 | 35 |
| | | 490 | 35 | 525 |
| 1066<br>Metal Detector | 1. Replace broken glass in control box door with perspex | 5 | 5 | 10 |
| | 2. Check operation of unit and adjust as necessary | 10 | – | 10 |
| | | 15 | 5 | 20 |
| 2015<br>Belt Weigher | 1. Check weightometer | 50 | – | 50 |
| | | 50 | – | 50 |
| Fine Ore Bin. | 1. Repair steelwork above No. 7 Conveyor | 150 | 10 | 160 |
| | 2. Make up and fit manhole cover and ladder to hole on east side | 180 | 10 | 190 |
| | | 330 | 20 | 350 |

## CRUSHING CIRCUIT

| EQUIPMENT NO. | DESCRIPTION OF WORK | LABOUR | MATERIAL | TOTAL |
|---|---|---|---|---|
| General Electrical | 1. Inspect and regrease all electric motors before refilling | 250 | 15 | 265 |
| | 2. Replace air conditioners in crusher control rooms. (now fitted to geology office) | 40 | 600 | 640 |
| | 3. Protect all push-button stations and emergency stop-switches, from spillage | 150 | 15 | 165 |
| | 4. Check and correct all switchboard wiring (see "Report on Electrical Installation at Mt. Gunson" by T.C. Stafford 2.3.71) | 800 | 150 | 950 |
| | 5. Check and rewire as necessary all welding outlets | 200 | - | 200 |
| | 6. Prepare a minimum spare parts list for electrical spares ) | 250 | 250 | 500 |
| | 7. Purchase into store parts as recommended in (6). ) | | | |
| | 8. W.R.E. to reconnect Transformer | 150 | - | 150 |
| | 9. Repair broken light fittings | 80 | 50 | 130 |
| | 10. Complete wiring of floodlight lighting petrol storage area | 40 | 20 | 60 |
| | | 1,960 | 1,100 | 3,060 |
| Mechanical | 1. Repair all damaged handrails | 90 | 5 | 95 |
| | 2. Wirebrush and paint corroded steelwork (10% of initial painting MG/38) | 3,700 | - | 3,700 |
| | 3. Clean up of windblown fines and vegetation etc. | 180 | - | 180 |
| | 4. Remove temporary earth walls put up to divert rainwater during shutdown. | 45 | - | 45 |
| | 5. Repair damaged sheeting in mill building and warehouse walls | 100 | 50 | 150 |
| | 6. Repair watermeter - Camp waterline | 25 | 5 | 30 |
| | 7. Replace canvas firehoses | 40 | 150 | 190 |
| | 8. Recharge all fire extinguishers | 60 | 40 | 100 |
| | 9. Grade roads in mill area | 100 | - | 100 |
| | 10. Water flow meters, instruments etc.<br>Service and check. Repair as necessary. Connect remaining instruments. | 180 | 50 | 230 |
| | 11. Purchase steel stock for wear plates etc. | - | 500 | 500 |
| | | 4,520 | 800 | 5,320 |

## CRUSHING CIRCUIT

| EQUIPMENT NO. | DESCRIPTION OF WORK | LABOUR | MATERIAL | TOTAL |
|---|---|---|---|---|
| | Cost of Work in Crushing Circuit | | | |
| | Labour | 16,355 | | |
| | Material | | 13,210 | |
| | + 5% Handling Charges | | 660 | |
| | | | 13,870 | |
| | Total | | | 30,225 |
| | + 10% Contingency | | | 3,022 |
| | | | | 33,247 |
| | Allow $35,000 | | | |
| | Time required for work on Crushing Circuit | | | |
| | 3,270 Man hours | | | |
| | or 55 man weeks (10 hours/day 6 days/week) | | | |

## CONCENTRATING CIRCUIT

| EQUIPMENT NO. | DESCRIPTION OF WORK | LABOUR | MATERIAL | TOTAL |
|---|---|---|---|---|
| 3000<br>Ball Mill and Ancilliary Equipment. | 1. Repair top section of feed chute | 90 | 15 | 105 |
| | 2. Repair makeup water line and replace control value | 30 | 10 | 40 |
| | 3. Clean bearings and lower mill into position | 600 | 15 | 615 |
| | 4. Drain oil from both trunnions flush and refill | 100 | 60 | 160 |
| | 5. Refit bearing sealing rings | 100 | - | 100 |
| | 6. Repair oil line. Pressure gauge to feed end bearing | 10 | 5 | 15 |
| | 7. Overhaul Miller high pressure booster system (bearings) | 450 | 30 | 480 |
| | 8. Overhaul Farval spray lubrication system (drive gears) | 180 | 30 | 210 |
| | 9. Change oil in drive gearbox | 40 | 20 | 60 |
| | 10. Replace Ball Mill coupling and bearing housings (by Perry Engineering) | 600 | 500 | 1,100 |
| | 11. Replace Inner Ring feed end liners | 220 | 2,000 | 2,220 |
| | 12. Replace 3 broken liner bolts | 60 | 15 | 75 |
| | 13. Replace liner, hardwood block, and manhole cover | 90 | 45 | 135 |
| | 14. Recharge with balls (27 ton old and approx. 30 ton new) | 220 | 3,800 | 4,020 |
| | 15. Remove unnecessary pipework from discharge hopper | 50 | 5 | 55 |
| | 16. Remove worn linatex lining, patch steelwork and fit new linatex to discharge hopper | 230 | 220 | 450 |
| | 17. Protect B.M. control switchbox from spillage | 60 | 10 | 70 |
| | 18. Check electrical circuit and test run. Ball Mill. | 140 | - | 140 |
| | 19. Clean up Oil and grease spillages | 50 | 20 | 70 |
| | 20. Wire brush and paint corroded steelwork - feed chute support and discharge hopper | 45 | 10 | 55 |
| | 21. Recondition Dezurik Knife gate valve, discharge hopper | 90 | 5 | 95 |
| | 22. Protect feed end trunion from spillage | 50 | 10 | 60 |
| | 23. Replace rubber diaphragm in Airflex clutch also hold spare in stock | 50 | 1,100 | 1,150 |
| | 24. Assemble clutch and line up motor | 230 | 10 | 240 |
| | | 3,785 | 7,935 | 11,720 |

## CONCENTRATING CIRCUIT

| EQUIPMENT NO. | DESCRIPTION OF WORK | LABOUR | MATERIAL | TOTAL |
|---|---|---|---|---|
| 3010<br>Cyclones and pipework | 1. Open and Inspect Cyclones and repair as necessary | 60 | 200 | 260 |
| | 2. Replace pipework with linatex lined pipe and linatex hose | 390 | 620 | 1,010 |
| | 3. Replace Saunders type "A" valves with type "KB" Rubber lined valves | 60 | 380 | 440 |
| | | 510 | 1,200 | 1,710 |
| 3005<br>Cyclone Feed Pumps | 1. Open up and inspect both pumps - repair as necessary | 100 | 900 | 1,000 |
| | 2. Replace vee-belts | 30 | 40 | 70 |
| | 3. Check electric motors and regrease bearings | 80 | 5 | 85 |
| | | 210 | 945 | 1,155 |
| Sundry Mill Pipework | 1. Clear pipes and free valves | 90 | - | 90 |
| | 2. Replace eroded pipework | 180 | 40 | 220 |
| | 3. Repair leaking valves | 90 | 25 | 115 |
| | 4. Replace temporary pipe hangers and brackets | 90 | 25 | 115 |
| | | 450 | 90 | 540 |
| 5000<br>Tailings Thickener | 1. Repair thickener, raise and lower mechanism | 160 | 30 | 190 |
| | 2. Lower feed well and fit new screen deck | 300 | 250 | 550 |
| | 3. Connect alarm on torque overload | 55 | 10 | 65 |
| | 4. Change oil in reduction unit and drive gearbox | 30 | 25 | 55 |
| | 5. Refit and connect drive motor and coupling | 40 | 10 | 50 |
| | 6. Repair decking on thickener bridge | 20 | 10 | 30 |
| | 7. Tidy up electrical wiring | 30 | 5 | 35 |
| | 8. Level thickener overflow weir (after startup) | 180 | - | 180 |
| | 9. Replace worn out tails thickener feed pipe with launder | 450 | 550 | 1,000 |
| | Repair tailings drop box below float cells | 180 | 40 | 220 |
| | 10. Bulldoze tails spillage from creekbed to level lower than tunnel | 160 | - | 160 |

## CONCENTRATING CIRCUIT

| EQUIPMENT NO. | DESCRIPTION OF WORK | LABOUR | MATERIAL | TOTAL |
|---|---|---|---|---|
| 5000<br>Tailings Thickener (Contd.) | | | | |
| | 11. Build retaining walls to protect mouth of tunnel | 400 | 20 | 420 |
| | 12. Replace part of thickener underflow pipework; also possibly cone | 180 | 250 | 430 |
| | 13. Recondition | | | |
| | 2/6" Eureka gate valves | 250 | 60 | 310 |
| | 1/4" Crane gate valve | | | |
| | 1/4" John gate valve | | | |
| | 2/1" Saunders diaphragm Valves from underflow pipework | | | |
| | 14. Replace valve seats, valves and possibly diaphragms on duplex pumps | 135 | 360 | 495 |
| | 15. Refit and connect electric motor to duplex diaphragm pump drive | 40 | 5 | 45 |
| | 16. Replace broken handwheels on duplex pump | 15 | 70 | 85 |
| | 17. Repair broken light fittings in tunnel | 30 | 25 | 55 |
| | | 2,655 | 1,720 | 4,375 |
| 5010<br>Concentrate Thickener | 1. Replace main body casting of drive gearbox | 135 | 250 | 385 |
| | 2. Replace drive chain | 10 | 40 | 50 |
| | 3. Fit chain guard | 70 | 10 | 80 |
| | 4. Free raise and lower mechanism | 240 | 20 | 260 |
| | 5. Reconnect drive motor | 15 | 10 | 25 |
| | 6. Replace discharge valve | 10 | 35 | 45 |
| | 7. Repair feed pipes and support brackets | 50 | 10 | 60 |
| | 8. Clear accumulated flake rust from inside thickener | 25 | - | 25 |
| | 9. Cart and load approx. 60 tons of sand for thickener bed. | 350 | - | 350 |
| | 10. Clear underflow pipework | 35 | 5 | 40 |
| | 11. Clean jets on watersprays | 30 | - | 30 |
| | | 970 | 380 | 1,350 |

## CONCENTRATING CIRCUIT

| EQUIPMENT NO. | DESCRIPTION OF WORK | LABOUR | MATERIAL | TOTAL |
|---|---|---|---|---|
| 4015<br>4035<br>Flotation Cells | 1. Repair float cell feed pipes (to be lined with linatex) and splitter box | 320 | 215 | 535 |
| | 2. Fit linatex to cell feed box | 70 | 120 | 190 |
| | 3. Clean launder water sprays | 55 | - | 55 |
| | 4. Replace corroded water spray pipe and support brackets | 150 | 30 | 180 |
| | 5. Repair worn discharge box | 45 | 10 | 55 |
| | 6. Replace flexible rubber air hoses where necessary | 30 | 30 | 60 |
| | 7. Repair discharge valve and pipework | 180 | 420 | 600 |
| | 8. Repair steelwork at centre weir | 50 | 5 | 55 |
| | 9. Refit drain plugs to cells | 20 | 5 | 25 |
| | 10. Clean pipes - launders to hoppers | 100 | 5 | 105 |
| | 11. Alter position of grease nipples | 80 | 80 | 160 |
| | | 1,100 | 920 | 2,020 |
| 4055<br>Float Cell Air Blower | 1. Check electric motor and regrease | 30 | 5 | 35 |
| | 2. Check fan blades | 25 | - | 25 |
| | | 55 | 5 | 60 |
| 5040<br>5055<br>5060<br>5075<br>5080<br>5090<br>Reagent Feeders | 1. Reconnect reagent feeders | 60 | 10 | 70 |
| | 2. Clean reagent feed pipework | 75 | 20 | 95 |
| | 3. New pipework for flocculant feed to tails thickener | 40 | 40 | 80 |
| | 4. Free all plastic valves etc. | 40 | - | 40 |
| | | 215 | 70 | 285 |
| 5031, 5046<br>5081<br>Reagent Tanks | 1. Make up and fit new lids to tanks | 100 | 35 | 135 |
| | 2. Replace and connect Agitators | 30 | 5 | 35 |
| | | 130 | 40 | 170 |
| 5030, 5045,<br>5080<br>Reagent Transfer Pumps | 1. Open up and inspect pumps repair as necessary (new seals etc). | 40 | 95 | 135 |
| | 2. Replace and connect electric motors | 40 | 5 | 45 |
| | 3. Repair pipework from pumps to tanks | 15 | 10 | 25 |
| | | 95 | 110 | 205 |

## CONCENTRATING CIRCUIT

| EQUIPMENT NO. | DESCRIPTION OF WORK | LABOUR | MATERIAL | TOTAL |
|---|---|---|---|---|
| 4005<br>4020<br>4025<br>4045<br>Samplers | 1. Clean lubricate and test operation of samplers | 20 | 5 | 25 |
| | 2. Repair discharge pipework | 120 | 15 | 135 |
| | 3. Rougher Cons. sampler to be removed and held as spare (not used) | 15 | - | 15 |
| | | 155 | 20 | 175 |
| Standby Pulford Compressor | 1. Complete overhaul of compressor | 150 | 400 | 550 |
| | | 150 | 400 | 550 |
| 5020<br>Disc. Filter and Ancilliary Equipment | 1. Rebuild drive mechanism | 95 | 5 | 100 |
| | 2. Replace filter bags | 45 | 50 | 95 |
| | 3. Replace wodden segments | 45 | 650 | 695 |
| | 4. Clean out pipes to/from filters | 60 | 5 | 65 |
| | 5. Fit new rubber scrapers | 15 | 10 | 25 |
| Vacuum Pump | 6. Fit rebuilt Nash Vacuum pump in place of existing pump. Recondition removed pump as spare | 120 | 650 | 770 |
| | 7. Clean out all vacuum pipework | 90 | 5 | 95 |
| Filtrate Pump | 8. Open and inspect filtrate pump. Repair as necessary | 40 | 10 | 50 |
| | 9. Repair glandwater piping on filtrate pump. | 25 | 15 | 40 |
| | 10. Check air supply to filter bags | 10 | - | 10 |
| | 11. Supply and fit new mufflers to vacuum pump | 90 | 25 | 115 |
| | 12. Tidy up pipework and pipe support brackets | 65 | 10 | 75 |
| | | 700 | 1,435 | 2,135 |
| O.D.S. Pump and ball valves | 1. Clean out all lines from thickener to O.D.S. pump to filter bowl | 65 | 5 | 70 |
| | 2. Open up and inspect Diaphragm pump and ball valves. Repair as necessary | 40 | 50 | 90 |
| | 3. Recondition solenoid control valve/reducing valve, mist lubricator and strainer | 60 | 20 | 80 |
| | 4. Recondition water and air control valves in feed line | 40 | 10 | 50 |
| | | 205 | 85 | 290 |

## CONCENTRATING CIRCUIT

| EQUIPMENT NO. | DESCRIPTION OF WORK | LABOUR | MATERIAL | TOTAL |
|---|---|---|---|---|
| 5005 Tailings Pumps | 1. Check motors and regrease bearings | 50 | 5 | 55 |
| | 2. Open up and inspect pumps repair as necessary | 150 | 280 | 430 |
| | 3. Replace pumps in position. Reconnect to power | 270 | 20 | 290 |
| | 4. Repair gland water pumps | 45 | 40 | 85 |
| | 5. Repair gland water pipework | 35 | 25 | 60 |
| | 6. Replace adaptor pipes on each pump | 160 | 90 | 250 |
| | 7. Replace linatex hoses for connecting pumps | 60 | 300 | 360 |
| | 8. Repair feed hopper discharge valve | 35 | 10 | 45 |
| | 9. Replace 5" steel feed pipe with Nylex polypipe. Fix supports for pipe | 180 | 50 | 230 |
| | 10. Recondition discharge valves to tailings line. Replace gauges | 60 | 50 | 110 |
| | 11. Replace overflow pipework | 60 | 10 | 70 |
| | | 1,105 | 880 | 1,985 |
| Tailings Line 6 | 1. Replace existing worn out 5" M.S. pipe with 4" Nylex polypipe (approx. 2,000') or rubber lined steel pipe in 10 ft. lengths | 750 | 3,500 | 4,250 |
| | 2. Provide suitable tunnel under haulroad to allow access for repairs of pipe | 1,500 | 1,500 | 3,000 |
| | ** | 2,250 | 5,000 | 7,250 |
| Men's Change Room Toilet etc. | 1. Replace rusted out lockers | 10 | 70 | 80 |
| | 2. Clear drain | 40 | – | 40 |
| | | 50 | 70 | 120 |
| Air Compressor 4060 | 1. Change oil in compressor | 5 | 2 | 7 |
| | 2. Recondition oil pump and repair leaking oil lines | 25 | 15 | 40 |
| | 3. Clean air cleaner | 2 | – | 2 |
| | 4. Check and reset safety valves on receivers | 15 | – | 15 |
| | 5. Instal Air cooler | 250 | 50 | 300 |
| | 6. Check Motor and regrease bearings, check electrical circuit | 25 | 5 | 30 |
| | | 322 | 72 | 394 |

** Above subject to position of Tailings Diamond Type of pipe used.

## CONCENTRATING CIRCUIT

| EQUIPMENT NO. | DESCRIPTION OF WORK | LABOUR | MATERIAL | TOTAL |
|---|---|---|---|---|
| 4030<br>Rougher conc. sump and pump | 1. Check electric motor and regrease bearings | 15 | 5 | 20 |
| | 2. Open and inspect pump and repair as necessary | 15 | 70 | 85 |
| | 3. Weld broken lug on pump casing | 5 | – | 5 |
| | 4. Renew pipe and victaulic fittings between sump and pump | 15 | 10 | 25 |
| | 5. Replace brackets on feed pipes | 15 | 5 | 20 |
| | | 65 | 90 | 155 |
| 4040<br>Clean Tails Scavenger Cons. sump and pump | 1. Check electric motor and regrease bearings | 15 | 5 | 20 |
| | 2. Open up and inspect pump and repair as necessary | 25 | 160 | 185 |
| | 3. Renew pipe and replace victaulic fittings between sump and pump | 15 | 10 | 25 |
| | | 55 | 175 | 230 |
| 4050<br>cons. sump and pump | 1. Check electric motor and regrease bearings | 15 | 5 | 20 |
| | 2. Rewire stop/start switch to motor (repair switch) | 10 | 5 | 15 |
| | 3. Open up and inspect pump and repair as necessary | 15 | 70 | 85 |
| | 4. Renew pipe and replace victaulic fitting between sump and pump | 15 | 10 | 25 |
| | | 55 | 90 | 145 |
| 4065<br>Concentrator Building Sump Pump | 1. Check electric motor and regrease bearings | 15 | 5 | 20 |
| | 2. Open up and inspect pump and repair as necessary | 30 | 170 | 200 |
| | 3. Alter framework and guides so pump reaches bottom of sump | 65 | 20 | 85 |
| | 4. Renew discharge pipework | 45 | 15 | 60 |
| | 5. Renew screen around sump | 25 | scrap | 25 |
| | | 180 | 210 | 390 |

## CONCENTRATING CIRCUIT

| EQUIPMENT NO. | DESCRIPTION OF WORK | LABOUR | MATERIAL | TOTAL |
|---|---|---|---|---|
| Potable Water Pumps<br>6027 | 1.Check electric motors and regrease bearings | 20 | 5 | 25 |
| | 2.Open up and inspect pumps and repair as necessary | 20 | 35 | 55 |
| | 3.Recondition valves | 25 | 5 | 30 |
| | 4. Provide drain around pump bases | 45 | 30 | 75 |
| | | 110 | 75 | 185 |
| Fire Pump | 1. Check electric motor and regrease bearings | 15 | 5 | 20 |
| | 2. Open up and inspect pump and repair as necessary | 30 | 40 | 70 |
| | | 45 | 45 | 90 |
| 6028<br>Reclaim Water Pumps and Tank | 1. Check electric motors and regrease bearings | 55 | 5 | 60 |
| | 2. Clear area around control box and pump bases | 40 | – | 40 |
| | 3. Open up and inspect one pump and repair as necessary (other reconditioned in April 1972) | 180 | 340 | 520 |
| | 4. Recondition valves controlling reclaimed water | 110 | 50 | 160 |
| | 5. Repair top of tank | 30 | 15 | 45 |
| | | 415 | 410 | 825 |
| General Cleanup<br>7600 | 1. Clean up of area | | | |
| | 2. Removal of exploration gear to other storage | 2 men<br>7 days | | 630 |
| | | 630 | – | 630 |
| 7700<br>Mobile Crane Front End Loader | 1. Repairs to Albion Chieftan Capcrane engine by Southcott Pty. Ltd. (possibly Second-hand engine would be cheaper) | – | 2,000 | 2,000 |
| | 2. Replace engine in Truck etc by us | 450 | 30 | 480 |
| | 3. Complete service, change oils filters etc. in transmission and hydraulic circuit | 40 | 10 | 50 |
| | 4. Repair bucket of F.E. loader | 55 | 10 | 65 |
| | 5. Repair front swivel pin suspension | 30 | 50 | 80 |
| | 6. Recondition hydraulic pump valves, cylinders and hoses | 200 | 300 | 500 |
| | 7. Renew seat | 10 | 60 | 70 |
| | 8. Complete service of engine and transmission | 20 | 5 | 25 |
| | | 805 | 2,465 | 3,270 |

## CONCENTRATING CIRCUIT

| EQUIPMENT NO. | DESCRIPTION OF WORK | LABOUR | MATERIAL | TOTAL |
|---|---|---|---|---|
| 6030<br>Maintenance Equipment | 1. Improve storage of hand tools | 400 | - | 400 |
| | 2. Increase size range and types of tools in store | | 600 | 600 |
| | 3. Instal tool issue docket (as at Pyrmont) system | | 25 | 25 |
| | | 400 | 625 | 1,025 |
| | Further study in organising workshop facilities should be done once we know who will be doing the mining and with what equipment. | | | |
| 4070<br>Mill Building Crane | 1. Renew brass cable guides | 140 | 80 | 220 |
| | 2. Repair limit switching gear | 35 | 20 | 55 |
| | 3. Clean all chain drives and regrease | 25 | 5 | 30 |
| | 4. Check all electric motors and regrease | 40 | 5 | 45 |
| | 5. Adjust brake | 10 | - | 10 |
| | | 250 | 110 | 360 |
| Painting | 1. Wire brush and paint corroded steel work (10% of initial painting MG/38) | 3,700 | - | 3,700 |
| | <u>Cost of Work in Concentrating Circuit</u> | | | |
| | Labour | 21,822 | | |
| | Materials | | 25,672 | |
| | + 5% handling charge | | 1,284 | |
| | Total | | 26,956 | 48,778 |
| | + 10% Contingency | | | 4,878 |
| | Allow $55,000 | | | 53,656 |

<u>Time required for work on concentrating circuit</u> i.e. 4,364 man days
73 man weeks (10 hours/day 6 days/week)

<u>Mount Gunson Concentrator</u>

<u>Total Cost</u> of Work in Tables

| | |
|---|---|
| Cost of work on Crushing Circuit | $ 35,000 |
| Cost of work on concentrating circuit | $ 55,000 |
| | $ 90,000 |

<u>Total Time</u> required for work in tables :

i.e. 7,634 man hours at 10 hours/day
128 man weeks at 6 days/week.

## MOUNT GUNSON CONCENTRATOR

### Section II.

This section of the report covers a study of the downtime and high labour areas in the plant. The information has been gained from limited personal operating experience (6 weeks), shift logs production sheets, from reports by L.T. Nicholls, W.J. Rule, D. Baker, T.C. Stafford and J. Hemphill and from discussions with R.N. Dudgeon.

All the problems encountered and possible modifications to overcome them, have been discussed in detail in one or other of the above reports. This report has been confined to presenting the suggested modifications in table form.

Lump sum estimates have been made of the cost of these modifications under three headings - Essential, Necessary and Desirable. By carrying out the modifications under the "Essential" heading a double gain will be made. Projected throughput can be increased to 30,000 tonnes/month and plant availability should increase to around 95%. This would give an annual increase in the tonnage of ore treated and may reduce the manpower required. By carrying out the other modifications an even greater throughput may be possible.

The total cost of the essential modifications covered in the following tables is estimated at $850,000. This is a lump sum estimated only as the modifications have not as yet been fully investigated or detailed. Costs have been "guestimated", using Plant Register values of similar equipment.

The economic feasibility of any modification cannot be determined until we know the following :

1. The total ore reserve.
2. The head grade expected.
3. The nature of the ore.
4. The type and quanity of reagents needed.
5. The grade of concentrate produced.
6. The type of container and method of shipment of concentrates.
7. The planned annual throughput.
8. The expected life of the operation.

| | | Essential | Necessary | Desirable |
|---|---|---|---|---|
| 1. | Redesign crushing circuit to eliminate coarse stockpile and include closed circuit tertiary crushing | | | 450,000 allow |
| 2. | Rubber coating of conveyor pulleys to stop buildup of wet ore (29 pulleys) | | 3,500 | |
| 3. | Re-instal belt scrapers with rubber tips to clear wet ore | 2,400 | | |
| 4. | Concrete pads around crusher motors steelwork etc. to facilitate cleanup | 4,000 | | |
| 5. | Modify belt skirt clamps to improve skirt operation | 1,200 | | |
| 6. | Covers over crusher pits to deflect spillage and instal dust exhaust fan | 3,300 | | |
| 7. | Modify wind covers to facilitate cleanup of conveyor tables and eliminate damage to belts and rollers | 5,000 | | |
| 8. | Extend jaw crusher crane travel 12 ft. and provide bridge over pit to enable parts to be loaded from crane onto truck | 1,800 | | |
| 9. | Provide walkway from West end of F.O.B. tunnel to ground level for easy access from mill | 350 | | |
| 10. | Replace mexican feeder with two mechanical apron feeders under F.O.B. to avoid shutdown due to feeder belt failure and to increase live capacity of bin | | 40,000 | |
| 11. | Provide another F.O.B. of similar size to give total <u>live</u> capacity of 1,000 tons. This will avoid concentrator shutdown because of lack of feed due to mine or crushing circuit problems. (Present capacity only 6-8 hours F.O. storage) and allow crushing circuit operation on day shift only | | 80,000 | |
| 12. | Instal apron feeders and transfer conveyor belts to new F.O.B. | | 95,000 | |
| 13. | Redesign No. 3 Conveyor discharge chute to reduce time taken to reline secondary crusher. | 2,000 | | |
| 14. | Fit extra impact idlers and extend skirt on No. 2 conveyor to reduce spillage on feed end | 900 | | |
| 15. | Investigate rubber screen deck for secondary screen to reduce maintenance and improve screening of sticky ore | | 1,300 | |
| 16. | Redesign tertiary screen fines chute to reduce blockages | 2,200 | | |

| | | Essential | Necessary | Desirable |
|---|---|---|---|---|
| 17. | Replace grizzly feeder to jaw crusher with apron feeder to give more positive control | 30,000 | | |
| 18. | Instal closed circuit tertiary crushing to properly condition ball mill feed | 65,000 | | |
| 19. | Prove <u>small</u> Bob-Cat loader to clean up spillages | | 3,800 | |
| 20. | Extend ore storage area near jaw crusher to allow for blending of feed etc. | | 2,000 | |
| 21. | Protect No. 7 belt speed control unit from spillage (see 10) | 200 | | |
| 22. | Redesign reagent mixing tanks and platform to reduce operator time and improve safety | 7,000 | | |
| 23. | Fit separate larger potable water pump to reagent mixing circuit to reduce delays while filling tanks | 850 | | |
| 24. | Larger reagent mixing and or storage tanks to enable reagent mixing day shift only. (Possibly storage tanks on roof) | 2,800 | | |
| 25. | Provide concrete drains in various locations in mill area to stop nuisance flooding of store, switch rooms and mill building | 1,000 | | |
| 26. | Fit guttering to mill, office and warehouse buildings and store run-off water | 1,200 | | |
| 27. | Fixed pipe work from crusher pit sumps to facilitate pump out | - | | |
| 28. | Instal further half million gallon water storage tank to contain water from bores and pit and overflow during shutdowns etc. | 80,000 | | |
| 29. | Fit 4 apron feeders to Coarse Ore stockpile discharge instead of vibratory feeders to give more positive feed | 48,000 | | |
| 30. | Redesign and instal ball loading to reduce time taken and improve safety | 7,000 | | |
| 31. | Increase platform area at discharge end of ball mill | | 450 | |
| 32. | Slow down cyclone feed pumps to eliminate surge feed to cyclone | 500 | | |
| 33. | Raise waterline feeding discharge hopper to improve access to and operation of control valve | 335 | | |
| 34. | Make up set of spare pipe to speed up repair of quick wearing pipe | 15,000 | | |
| 35. | Design and instal walkway from flotation floor and ground floor to tailings thickener bridge to reduce time taken to check on tailings thickener | 4,500 | | |

| | | Essential | Necessary | Desirable |
|---|---|---|---|---|
| 36. | Move float cell blower outside mill building to reduce noise | 450 | | |
| 37. | Further reclaim water storage | 5,000 | | |
| 38. | Provide platform to facilitate repairs to cyclone underflow pipework | 900 | | |
| 39. | High and low level alarms on reagent tanks | 450 | | |
| 40. | Torque overload alarm on thickeners | 250 | | |
| 41. | Raise concentrate thickener and provide concrete slab under. Move O.D.S. pump to under centre of thickener. This will improve O.D.S. pump and filter operation | 7,000 | | |
| 42. | Design and fit hydraulic raise and lower mechanism to conc. thickener for ease of operation | 1,500 | | |
| 43. | Design and instal conc. storage tank, agitator and transfer pump beside conc. thickener to allow time for routine maintenance to filter and thickener | | 16,000 | |
| 44. | Move ball mill drive motor west to facilitate repairs to clutch | | 850 | |
| 45. | Provide 4" sykes or similar portable pump to reclaim water from gullies below existing tailings dam and any other storages from time to time | 5,000 | | |
| 46. | Duplicate tailings line to save running tailings down creek while repairing line. Also allow for regular rotation of line | 4,500 | | |
| 47. | Filter capacity must match up with head feed. Present filter has no surplus capacity. Provision for duplicating filter | 20,000 | | |
| 48. | Supervisors office in mill building to include control instruments and cribroom division for 4 operators | 1,000 | | |
| 49. | Move existing upstairs cribroom outside mill building for day labour and maintenance staff | 1,500 | | |
| 50. | Space obtained by move in (49) to be used for additional float cells which will possibly be required subject to metallurgical considerations | | 40,000 | |
| 51. | Float cell feed conditioning tank above supervisors office with provision for reagent addition and mechanical splitters. Conditioning feed will save on reagents | 7,000 | | |
| 52. | Scat handling facilities (if no modifications to crushing circuit are made) | | 1,500 | |
| 53. | Extend platform around tailings diaphragm pump to facilitate repairs | | 450 | |

| | | Essential | Necessary | Desirable |
|---|---|---|---|---|
| 54. | Provide enclosed outside stores area and racks for steel plate and sections | 1,800 | | |
| 55. | Air supply to crushing circuit to clean motors, switches and pulleys of wind blown fines including relocate standby compressor away from possible spillage | | 2,000 | |
| 56. | Purchase of medium sized second-hand engine lathe to reclaim worm pump parts, etc. This should be cheaper than new parts in many instances. | | 5,000 | |
| 57. | Provide Ambulance as required by Mining Regulations | 4,000 | | |
| 58. | Raise return waterline above ground and reroute around crushing circuit providing tunnels under roads. (Depends on action on tailings dam) | 4,000 | | |
| 59. | Investigate alternate sites/methods for tailings disposal<br>Instal new tailings disposal system. | 100,000 | | |
| 60. | Investigate alternate systems of concentrate handling, drying and transport. Instal new concentrate facilities after results of above | 70,000 | | |
| 61. | Investigate communications requirements e.g. Telex, Private automatic exchange house to house to mill to mine, radio contact with mine.<br>Instal systems after results of above | 16,000 | | |
| 62. | Provision for initial plant testing, operator training, startup inefficiencies and rectifications<br>i.e. Pre-production expenses | 40,000 | | |
| 63. | Provide one standard utility and one 4 wheel drive utility for mill shift workers and mill maintenance crew | 11,000 | | |
| 64. | Rebuild Men's change room outside mill building to keep clean and dry | 4,000 | | |

| | Essential | Necessary | Desirable |
|---|---|---|---|
| **ACCOMMODATION** | | | |
| Some major alterations to staff accommodation will be required so that the operating staff can be "on hand" when required. | | | |
| This will involve the following expenses. | | | |
| 1. Provide 9 more houses to accommodate married staff and 2 small flats for single senior staff. Landscaping etc. | 200,000 | | |
| 2. Minor repairs and cleaning of camp rooms and facilities. Fit air conditioners to three rooms | 2,500 | | |
| 3. Construction of incinerator and provision of pit for rubbish disposal from camp and village | 850 | | |
| 4. Fire protection of camp and village | 5,000 | | |
| 5. Provide (i) school and playground | 12,000 | | |
| (ii) small store | 5,000 | | |
| (iii) swimming pool and recreation area | | 18,000 | |
| 6. Eighteen passenger bus for school and shopping trips, non-operations staff transport to and from work and evening social trips to Woomera etc. | 8,000 | | |
| 7. Provision for larger standby power generating equipment | 20,000 | | |
| 8. Company provided caravan park facilities to allow some men to live with family on site in <u>own</u> caravan, for drilling contractors etc. or for emergency extra accommodation for M.G.M. | | 12,000 | |
| Further investigation of accommodation facilities will be needed when size of labour force is known. | | | |
| **SUMMARY OF ESTIMATES** | | | |
| Total cost of modifications to plant | 591,885 | 291,850 | 450,000 |
| Total cost of additions to accommodation facilities | 253,350 | 30,000 | |
| Total of Estimates | 845,235 | 321,850 | 450,000 |
| Allow | 850,000 | 322,000 | 450,000 |

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# ASSESSMENT OF THE MOUNT GUNSON MINES PTY. LTD. OPERATION

R.N. Dudgeon
Pacminex Pty. Ltd.
December 15, 1972

# ASSESSMENT OF THE MOUNT GUNSON MINES PTY. LTD. OPERATION

PAGE

PREFACE

## PREFACE

This report on the operation of the concentrator is based on the experience gained by the writer while working at Mount Gunson over an eight month period during 1971.

Problem situations associated with the operation of the concentrator are discussed. Recommendations as to the modifications needed to put the plant into a satisfactory operating condition for a start-up are discussed. Detailed engineering and cost recommendations are not given. Geology and mining is only discussed insofar as the processing of ore to concentrates is affected.

The combination of circumstances and problems turned what looked to be a potentially profitable venture into an unprofitable operation. The problem was aggravated by the drop in copper prices during the period of operation.

The purpose of this report is to review these matters with a view to making available the experience gained and lessons learnt. Such information could be applicable to other mineral processing projects which Pacminex Pty. Ltd. may become involved in.

## 1. INTRODUCTION

The project comprised both mine and concentrator. The planned throughput of the concentrator was 1,130 metric tons/day of ore. Production of concentrates commenced in April, 1970 and the production was suspended in December, 1971. During that period 484,000 tons of ore was mined, with 933,000 tons of waste; the quantity of ore milled was 466,500 tons. The tonnage of concentrate sold was 5,891 tons of an average assay of 57.34%. The average feed head assay was 0.85% and the recovery rate 85.6%.

Production was suspended in December, 1971 due to a lack of ore at a high enough grade for an economic operation at the low copper prices prevailing late in 1971. When the Mount Gunson Mine commenced operation, the LME price for copper was £S600/long ton.. The price in December, 1971 was around £S410.

## 2. MINERALOGY

### 2.1 Sulphide Ore - Pernatty Lagoon

The economic mineral contained in the ore mined from Pernatty Lagoon was chalcocite ($Cu_2S$). Small amounts of covellite ($CuS$), bornite ($Cu_5,FeS_4$), chalcopyrite ($CuFeS_2$) and native copper have been seen in ore received at the plant. Associated with the copper minerals was a small amount of silver. Galena ($PbS$) and sphalerite ($ZnS$) has been identified but the concentrations were not significant. Some pyrite ($FeS_2$) has also been noticed.

The deposit is of a secondary nature. The mineralization is usally found disseminated within the sandstone overlying the hard quartzitic beds and within the joint planes. It appears that reprecipitation of copper as sulphide took place below the water table following a drainage pattern above the quartzitic beds. Liberation of the chalcocite for separation by flotation was not difficult. In general the chalcocite was found in a coarse form deposited in fractures within the quartzite or as coatings between sandstone particles. Within the sandstone beds are found irregular layers of clays - two specific types were noticed, one apparently a red siliceous material formed by the decomposition of the quartzite and the other a yellow to white variety. The existence of these clays was not appreciated when plant design was being undertaken. Difficulties were experienced in the crushing circuit due to the presence of wet clays in the plant feed.

The lake surface is essentially several feet of mud, highly saline, which during dry periods becomes sufficiently hard enough to carry wide tracked vehicles and drilling on the lake is an extremely difficult operation.

Grade of ore supplied to the concentrator from both the East and West Lagoon deposits averaged 0.85% Cu.

## 2.2 Oxide and Sulphide Ore - Main Workings

Oxidized ore is found in the joint and bedding planes of quartzitic sandstone above the prevailing water table. The copper is present as the silicate mineral, chrysocolla; the carbonate, malachite, the chlorides; atacamite and paratacamite, with traces of other copper minerals.

The following is a mineralogical assay of a composite sample from two thousand tons of oxide ore mined for plant testing during May, 1971.

| | per cent |
|---|---|
| Quartz | 85 - 90 |
| Muscovite/Sericite | 2 - 4 |
| Gypsum | Trace |
| Paratacamite | 1 - 2 |
| Malachite | 1 - 2 |
| Chalcocite | Trace - ½ |
| Chrysocolla | 2 - 4 |
| Calculated Head Assay | 1.9% Cu |

Below the water table is found sulphide ore (chalcocite) with associations similar to the ore found in Pernatty Lagoon. In some of the softer horizons manganese "wad" containing as much as 30 per cent copper has been found.

In a test-mined pit the chalcocite was easily liberated to flotation sizes on milling, however mineralogical examination of some percussion cuttings has indicated that some of the chalcocite is extremely finely disseminated and could be difficult to liberate. Silver minerals have not been identified.

A ridge of dolomite traverses the area and some sulphide is found disseminated through part of it.

## 2.3 Comment

There appears to be considerable variation in the mineralogical associations throughout the deposits. This is to be expected as these deposits were put down under low temperature conditions.

A useful and effective mineralogical study of the deposits was not made prior to development of the flow sheet for the Mount Gunson concentrator. A two page description had been prepared by Dr. John McAndrew of the C.S.I.R.O. Mineragraphic Section, however, this report was based on the examination of only a few samples of material and thus could not be regarded as representative of the deposits.

One major criticism of the processing tests carried out by AMDEL to develop a flow sheet for the Mount Gunson concentrator is that a proper mineralogical examination of the ores tested was not done.

If this had been done then it would have been realised that sulphidizing of the "oxide" ore from the Main Workings followed by flotation to produce a concentrate could not have been successful due to the presence of silicate and chloride copper minerals which are not amenable to this treatment. The suggestion to sulphidize must have been based on the assumption that copper existed as "malachite" (the carbonate). Apparently in earlier mining operations high grade pockets of malachite were taken out. Also, it seems that there was some disagreement between AMDEL and officers of the C.S.I.R.O. Ore Dressing section as to whether such ore could be processed by flotation. A careful mineralogical examination would have indicated the futility of test work to develop a processs using sulphidization and flotation. It would have indicated that ammonia leaching was not possible but would have suggested that an acid leaching process may be a way of extracting copper from this ore.

It seems that clays were absent in the bulk sample used for pilot plant testing, however an examination of drill core material could have indicated their presence. The processing circuit could have been so designed as to minimise difficulties due to their presence.

The key to designing a successful processing circuit is an understanding of the mineralogy of the ore to be mined and processed. Essential information that is required:-

(a) an approximate mineralogical assay,

(b) a description of mineral associations,

(c) the size to which the ore must be crushed and ground to effectively liberate the wanted minerals,

(d) the presence of other elements which may add to the value of the concentrate produced or alternatively may result in a penalty.

## 3. PLANT DESIGN

Initial test work on samples was done by J. Woodcock at the C.S.I.R.O. Ore Dressing Laboratory, Melbourne University. The staff and equipment from this laboratory is now dispersed and the laboratory no longer operating. Mr. Woodcock reported results of test work in letters to J.B. Evans of Austimex Pty. Ltd. Further test work and pilot plant studies were carried out at AMDEL, Adelaide. One major criticism of the test work reported by AMDEL for the processing of Mount Gunson ores is that the lack of mineralogical information which is the basis of process selection. This criticism no longer applies to AMDEL test work programs and in fact it seems that one main criticism of current work done at AMDEL is that too much and quite often unnecessary effort is being spent on obtaining non essential mineralogical data.

McKee - I.E.L. were given the original design and construction contract. Their performance was far from satisfactory and eventually Central Engineering Services of C.S.R. took over and completed the project. C.E.S. engineers worked efficiently and conscientiously to bring the plant into operation using the design provided by McKee - I.E.L.

Very little consideration was given by McKee to maintenance in the plant layout, particularly in the crushing section. It was unfortunate that because of time limitations C.E.S. officers after taking over were not able to visit existing operating plants to discuss these matters and so make changes to the layout which would have materially assisted in maintenance and thus in the productivity of the plant.

Prior to any future start-up of the Gunson plant, the engineers responsible for putting it back into working condition should have the opportunity of discussing maintenance and general engineering problems with their counterparts at other mineral processing plants. For example, a similar ball mill to that at M.G.M. exists at the Roseberry concentrator of The Electrolytic Zinc Co. of Australia Ltd. An exchange of information would be useful.

Before any major modifications are made or pieces of equipment bought it is essential to obtain as much information as possible from other operators who have had similar problems. In the Australian mining industry this interchange of engineering information is almost standard practice.

One major problem associated with the design, construction and operation of the Mount Gunson project was that the total responsibility and co-ordination of the project was **not the** responsibility of any particular person. There should have been someone working full time charged with the overall responsibility for the project. It was not until the operation was almost ready for production that a General Manager was appointed. Unfortunately, the metallurgist - mill superintendent designate left after the operation had been in operation for only three months and a replacement was not available. Several crucial months in the life of the project elapsed before an appointment was made. Although there were reasons for such a situation the problems resulting worked against the project. If the Mount Gunson project is to reactivated in the future it is the opinion of the writer that it is utterly essential that the responsibility should be in the hands of one person who devotes his full time to the project. Essentially his job would be to co-ordinate the work of all groups involved - covering geology, design, construction, operation and marketing. He would call on the services of others both inside and outside Pacminex. Quite likely he would not be the on-site Manager of the operation when the plant starts up. A person who is a good plant operator does not necessarily possess the requisite skills needed to develop a project. A man with a wide experience in the industry would be needed to fill the position of project manager.

## 4. MINING

An experienced contractor was used for mining. One of the major problems at Gunson was the lack of liaison and understanding between the mining contractor and the management of Mount Gunson Mines. Part of the problem goes back to the interpretation of the Mining Contract.

The pit design and mining operation was planned in detail prior to the commencement of mining. However, during the initial stages the grade of ore delivered to the concentrator was far below that anticipated and was certainly at an uneconomic level.

Eventually a mining engineer was taken on by Mount Gunson Mines and worked closely with the mining contractor. Once the problem was understood and there was a better understanding of the nature of the deposit, the whole mining plan was changed and this resulted in an improvement of the grade of material delivered to the mill.

By working only on one face at a time, the result was a great variability in the nature of the ore delivered to the plant, both from the grade an in physical properties. Some of these problems could of been minimised by blending ore. However, this would have necessitated double handling at the ore bin feeding the jaw crusher and, although it would of been to his advantage, the contractor believed that the extra costs involved should be charged to Mount Gunson Mines. The result was that quite often sticky ore would be fed to the crushing circuit resulting in blockages in the jaw crusher and in chutes below the screens. Once a blockage formed the crushing circuit had to be closed down, the blockage removed and quite often screens lifted and the secondary and tertiary crushers opened up to allow such material to pass through the circuit.

With judicious blending this problem could have been minimised. As it happens, in the last months of the operation the contractor was convinced of the advantages to himself as well as to Mount Gunson Mines of blending ore and this was done to a limited extent, particularly with ore containing clays. In any future operation the mining must be controlled by the operating company. Additional costs involved in mining such as double handling of the ore for mixing, are often more than saved by averting problems in the processing circuit.

It was only in the last months of the operation that it was possible to maintain the rated throughput through the plant and in these circumstances the mining contractor had difficulty in maintaining the supply of ore to the concentrator.

There is some uncertainty as to whether the contractor provided suitable equipment for this mining operation. The method of mining and the use of equipment should be reported on by a mining engineer.

## 5. PROCESSING

### 5.1 Introduction

The plant flow sheet is essentially simple. The ore was crushed in three stages and fed to a ball mill in closed circuit with a hydrocyclone. The overflow from the hydrocyclone was pumped to a feed box where it was split into two streams and fed to rougher and scavenger flotation cells. The tailings from the flotation section were discharged to a thickener where flocculant was added. Clear overflow water was returned to the mill circuit. The thickened tailings were pumped to a tailings dam at a level higher than the plant. Water was reclaimed from this dam.

The sulphide concentrates from the scavenger circuit were returned to the rougher circuit and the concentrates from the rougher cells were sent to cleaner and re-cleaner cells. The final concentrate was pumped to a concentrate thickener and the discharge from the thickener to a Dorr-Oliver leaf filter. The filter discharge was picked up in the concentrate shed below and shipped to Japan in Geetainers.

In general there was no real difficulty in achieving a concentrate grade of +55 per cent copper.

### 5.2 Crushing

Ore was brought up from the mine in 13½ ton capacity Foden dump trucks and dumped directly into a 30 ton capacity coarse ore bin. On occasions when the jaw crusher was out of action, ore would be dumped on the pad adjacent to this bin and at a later stage fed into the bin by means of a front end loader.

On some occasions when ore brought up from the mine was particularly wet and sticky it was also dumped on this pad and allowed to dry out, or alternatively, was blended in with dry feed. This was very necessary otherwise blockages would occur between the jaws of the crusher and material would have to be barred out. Also wet material would build up on the screen and in the chutes from the screens and again cause blockages.

Another potential trouble spot, when wet ore was fed through the crushing circuit, was with the tertiary crusher. This was a particularly difficult area to get into and clean out. Wet ore, if it did pass through the circuit, would also hang up in the fine ore bin.

From discussions with the previous mill superintendent I believe it was never anticipated that wet, sticky ore or clays would be brought into the mill. The crushing circuit was designed to take material with a maximum of 5% moisture. Most problems occured in the crushing circuit. For the type of ore it was expected to handle it is a far from suitable flowsheet.

One of the major fundamental mistakes with the crushing circuit is that after ore had been through the jaw crusher it was screened, over-size ore taken to the coarse ore stock pile and the under-sized fed directly to the fine ore bin. Mining in general took place at the interface of the sandstone and the hard quartzite and this operation of screening had the effect of segregating two different types of ore. This meant that the softer sandstone and sometimes clays were fed directly to the fine ore bin during day shift whereas the hard quartzitic material went to the coarse ore stock pile. The problem was minimized by crushing both materials simultaneously. The mixture of crushed quartzite with the clays and sandstones was far less susceptible to hang-ups.

Another problem on day shift was that the softer material tended to be overground in the ball mill. There were also flocculation problems due to the presence of clays.

During the night shift when the feed came solely from the coarse ore stock pile, there were problems with what is known as "scats" being discharged from the ball mill. "Scats" is the term to designate over-size material and the quantity of this material produced did limit the throughput.

The problem of "scats" was due to the fact that there was far too great a size range of material fed to the ball mill, partly due to drawing solely from the coarse ore stock pile and partly due to the nature of the crushing circuit. It was interesting however that when ore from the coarse ore stock pile was being fed to the plant, both recovery of copper and the performance of the thickener were improved.

It is suggested that the circuit for the Mount Gunson operation should have been a jaw crusher followed by a crushed ore stock pile (without screening). Ore would be reclaimed from the stock pile, screened, undersize to the tertiary crusher and oversize to the secondary crusher. Screening devices should have been set in closed circuit with either the secondary crusher or the tertiary crusher. The fine ore would be fed directly to fine ore bin. For maximum efficiency in a concentrator it is essential to give the utmost attention to feed preparation - that is the crushing and grinding circuits.

The existing fine ore bin at Mount Gunson had an effective capacity of about 400 tons, insufficient for afternoon and night shift, and it was necessary to crush from the coarse ore stock pile on night shift and generally also on part of afternoon shift. To obtain sufficient working capacity for fine ore another bin of equal size is required if the operation is to be recommenced with a throughput of 1,250 tons per day.

The Mexican feeder used in conjunction with the fine ore bin proved troublesome and was far from ideal. During May, June and July when neither the secondary or tertiary crushers were in circuit, considerable amount of hard, oversized material was fed to the fine ore bin and some of this material would find its way between the steel running plate and the conveyor belt of the feeder. The belt was very susceptible to tearing. When using the Mexican-type feeder it is utterly essential that only screened fine ore be put in the bin.

There was some discussion as to whether the tertiary crusher is really needed. The plant did operate for periods with this crusher out of action. However, not having it in the circuit did contribute to the "scats" problem.

In order to keep material flowing from the fine ore bin, it was, in the earlier days of the operation, the practice to use explosives to bring down material which was hung-up in the fine ore bin. This I regard as being dangerous and unnecessary and certainly such hang-ups occurred infrequently when mixed ore was fed to the fine ore bin. Mixed ore was produced by drawing off the coarse ore stock pile and crushing simultaneously with new feed coming into the circuit.

The wind covers for the conveyors in the crushing circuit caused problems. With these covers on there was great difficulty in clearing spillages and blockages. It is suggested that rather than have covers, a better method to keep belts from being blown off would be to have bars across the belt. Serious damage was done to a number of rollers jammed by spilt rock. There has been a suggestion that the covers do protect the ore on the belt and in high winds ore can be blown off the belt. Only on one occasion during 1971 was an uncovered belt (No. 1 conveyor) put out of action by a high wind. It is likely that during this period of about 2 hours, ore would have been lost from the belts had the plant been operating.

The pits below both the secondary and tertiary crushers were a constant problem. Chutes under the screens feeding the conveyor belts were a major problem with buildups of sticky ore. Basically the problem was a design one. Other problem areas were the chutes to the vibrator feeders under the coarse ore stock pile. Very often coarse ore would not flow down to these feeders. Explosives were sometimes used to bring material down. Usually however, such material could be brought down by barring. The other method of keeping the coarse ore stock pile moving was the use of a loader from time to time to push the material into the chutes. Because of the nature of the stock pile it seemed essential that if feed to the secondary crusher was to be maintained it was necessary to use a front end loader at irregular intervals to push ore over the chutes. Half an hours push up with the loader would save many hours work of barring and make the use of explosives, which is dangerous, unnecessary. On one instance the explosive charge fell back on to the feeder and blew up the feeder.

It is recommended that before re-opening the Mount Gunson concentrator the crushing circuit be re-designed to handle the type of ore it is planned to mine and process.

With the present lack of capacity in fine ore storage and the need to make alterations in the crushing circuit to accomodate variations in feed it is suggested, even assuming that the crushing plant was in AI condition, there would be unavoidable down time and it would be optimistic to expect much above eighty per cent running time for the plant.

## 5.3 Grinding

Feed to the ball mill was weighed by a Ramsay electronic weightometer which performed well. A weekly calibration check was necessary. This was normally done by the plant electrician but really is the responsibility of the mill superintendent and/or plant metallurgist who should supervise such a test and be personally satisfied. The amount of feed to the mill was automatically controlled from the Ramsay unit.

On occasions blockages of the feed chute to the ball mill with lumps of mud occurred. The result was usually a spillage and an overflow of mill water which represented a real risk to the trunnion bearing and the air compressor below. A cover was eventually fitted to prevent ingress of pulp into the trunnion bearing and the air compressor moved outside to a clear location.

Cast and forged 4 inch balls were charged to the mill daily. The method used was primitive and dangerous. Up to September, 1971 this job was done by the two maintenance fitters. After this date it was done by plant operators and the fitters were more profitably used on engineering and maintenance.

It is recommended that consideration be given to developing a safe and effective ball charging mechanism.

The need to use 4 inch balls indicates that the feed to the mill contained excessive oversize which probably could be eliminated by the use of screening devices in closed circuit with the crushers. It is realized that blinding of fine screens could be a problem and a thorough investigation as to the most effective type of screening device for this application is needed.

Provision should be made for the return of plant "scats" (oversize from the ball mill) to the mill circuit rather than to waste.

By using a sized feed to the ball mill the throughput of 50 ton per hour could be significantly increased.

The ball mill was operated in closed circuit with hydro-cyclones and no major problems were associated with the operation of the cyclones other than that there was excessive wear in the feed lines, particularly at bends. It would seem necessary to use rubber lined pipes. Leakage from worn piping at the hydro-cyclones on to the reagent mixing area below caused problems. The reagent mixing equipment should be moved to a more suitable location.

### 5.4 Flotation

Although this was the least troublesome section of the plant there were many problems associated with its operation.

The fixed splitter box at the head of the rougher banks of cells was not entirely effective. It should be replaced by a mechanical splitter and reagents could be added at the inlet. Such a unit would, as well as dividing the flows equally, also mix pulp and reagents. It would seem that the flotation cells installed at Gunson were of a type designed to have automatic level control for pulps. However, such controls were not incorporated and the existing arrangement of an overflow weir for the final discharge of tailings was troublesome. This arrangement was modified and this modification proved effective. One useful modification to the re-cleaner and cleaner flotation cells was the use of wooden bearings in place of metal bearings for the wipers.

The flotation section of the Gunson mill was plagued with the problem of excessive reagent consumption. No conditioning tanks were incorporated in the circuit. Ideally flotation reagents should be added in conditioning tanks prior to the flotation. It was the practice to add xanthate at the ball mill feed. However, this meant xanthate was in closed circuit and, being unstable under certain conditions, it appeared that the xanthate would probably be consumed under the turbulent and oxidizing conditions in the ball mill.

Further xanthate was added at the head of the flotation cells but this does not allow for conditioning. However, it was found that by reducing the xanthate addition to the ball mill feed there was little change in the flotation of the chalcocite. A preferable position to add reagents would have been at the feed box to the flotation cells. Unfortunately this could not be done unless a new type of reagent feeding arrangement using pumps was installed. The desirable modification would be to include conditioning tanks in the circuit.

The Clarkson feeders in use at Gunson were quite effective, however, they were positioned in a most awkward spot, and rely on gravity for flow.

Concentrates were discharged from the flotation launders to sumps on the floor below. The pulps were then pumped according to the circuit. One job that had not been done was to size pumps - some were running too fast. This requires considerable effort. The change of speed is achieved by a change of pulley size. It may have been possible to run some pumps at a lower speed and substitute wear resistant rubber impellers and liners. Overflowing of sumps took place far too often resulting in a dirty plant and unnecessary recirculation of material. A sizing of pumps and sumps will be necessary before any future plant start-up.

## 5.5 Concentrate Thickening, Filtration and Drying

Final concentrates were fed to a concentrate thickener. During 1971 this thickener never functioned effectively and the reason why was never fully understood until 1972 when the thickener was cleaned out. Spanners, rags and other items were extracted from the discharge pipe. The pump feeding the filter behaved erratically and the reasons never fully understood. It was often dismantled, rather difficult, as there was little head room below this thickener. Serious consideration should be given to raising this thickener in order to provide sufficient headroom. The thickener was also used as a concentrate storage when the filter was being serviced. There were periods when concentrate had to be discharged to an outdoors dam, through lack of capacity, a very unsatisfactory arrangement. It is recommended that a tank with agitator be installed adjacent to the concentrate thickener (as at Kanmantoo Mines) to hold surplus concentrate slurry when either the thickener or filter is being maintained.

The leaf filter required constant attention but this could be to a large extent attributed to the continual malfunction of the concentrate thickener discharge and consequent changes in pulp density. It is essential to maintain a constant pulp density in the boot of the filter. With the plant operating at a rated throughput of 50 ton per hour ore at about 0.9% Cu the filter was hard pressed to cope with the output of concentrates. With a greater output of concentrates it is doubtful whether the existing filter could have coped.

The damp concentrates were discharged into a drying shed below. Possibly, by improving the performance of the grinding section, particularly to reduce overgrinding and the generation of mineral slimes, and by the addition of additives it should have been possible to produce continuously a concentrate of ten per cent or less moisture. No means was available to dry excessively damp concentrates except by outdoor drying which was not satisfactory. A means to redirect sloppy concentrates to a drying area was needed. Ideally it would appear logical for concentrates to be discharged directly into the shipping containers rather than be double handled. Geetainers were not entirely satisfactory for shipping concentrates and a study of alternative methods should be made.

## 5.6 Tailings Thickening and Disposal

The tailings from the flotation cells after being sampled were discharged into a Denver thickener of 90 feet diameter. Originally this thickener had twin drives. However, there were many problems associated with this arrangement and eventually the arrangement was discarded and replaced with a single drive.

Flocculant was added to the feed well. This is not good. It is preferable to spray flocculant into a launder which then flows into the feed well. By judicious management there appeared to be no real difficulty in maintaining a discharge density from the thickener of 55% solids. Tailings were pumped from the thickener to the tailings dam through a 6 inch line by two Warman 4/3 pumps. These pumps sat on a concrete pad. This was a particularly dirty and messy area and almost every time any maintenance work was needed on the pumps it would be necessary to remove slime and sand, and generally clean up the area before work commenced. These pumps and motors should have been protected and maintained in a much better condition.

Clear overflow water from the thickener went into a mixing tank where fresh water was also added. On a number of occasions slime in the water overflowed from the thickener into this tank and eventually settled out. These slimy materials were particularly abrasive as far as the water pumps were concerned and caused many problems.

There were many problems associated with the tailings dam. The tailings line passed under a haul road. It was partly protected by an ARMCO tunnel which eventually subsided and there would have been a real problem if the line had to be replaced. As it happened, replacements had to be made near the entrance of the tunnel. It is essential that all sections of a tailings line be accessible at all times. Individual lengths of pipe should not exceed 20 feet. These lengths even in steel can be manhandled as a crane is not always available. Victaulic joints are preferable. Pipes should be regularly turned to produce even internal wear.

An emergency tailings line should always be available. To shut down the concentrator because of a break in the line is an unnecessary and expensive shut down. Discharge of tailings directly below the thickener resulted in the thickener tunnel becoming permanently blocked up with mud and should never have been allowed.

The existing tailings dam was situated on a flat area. The dam base was not sealed. There were big water losses due to evaporation and seepage. Reclaimed water flowed by gravity back to the plant. Five to fifteen per cent of the plant water requirements were normally obtained from water reclaimed from the tailings dam.

It is suggested that a better site for the dam would have been the valley down which the haul road to West Lagoon runs. With such a dam the water depth would have been greater and consequently losses by evaporation minimized. Evaporation is a function of surface area. Indeed, if there were no seepage losses the tailings could be pumped directly to the dam. A thickener is not essential if there are minimal water losses due to seepage. (Kanmantoo Mines do not thicken tailings). Tailings can be desanded at the dam site with cyclones - the sands can be used to build the wall. Fines settle out in the dam. During periods of heavy rain a tailings dam in a valley can be used to collect run-off water and so augment the return water supply.

## 5.7 Plant Water Requirements

Assuming the use of return water a good figure to use is 200 gallons of fresh water per ton of ore milled. Under good operating conditions with moderate weather this can be reduced to 180 gallons per ton of ore milled.

Assuming a throughput of 8,000 tons of ore per week, the weekly requirement of water would be 1.6 million gallons of fresh water. This means that the plant requirements were, if operating efficiently, sixty per cent above the guaranteed water supply. It is only during winter months and periods of water surplus that this quantity may become available from the Woomera pipeline. To augment the water supply and also reduce costs several things must be done before a plant start-up:-

(1) Build a new tailings dam designed to minimize losses by seepage and evaporation,

(2) Assuming a suitable location make some arrangements for run off water to be collected in the tailings dam,

(3) Investigate the possibility of augmenting the water supply from bores, wells and small dams.

(4) Provide an additional 500,000 gallon fresh water storage at the plant. The additional storage could be useful during short term periods of water restriction on the Woomera pipeline.

During the latter part of 1971 when water consumption was around 180 gallons per ton of ore there was a build up of soluble salts in the mill circuit water to almost 4 per cent dissolved solids. The mill circuit water had a salinity equivalent to sea water. Much of this salt may have come from rather muddy material from West Lagoon processed during that period. Suggestions have been made that saline water from the worked out pits could be used to augment the supply. This water is saturated, about 40 per cent total dissolved salts and thus is of no real use. Excessive quantities of dissolved salts (sodium and calcium) in mill water will have deleterious effects upon the flotation process. Also with the plant there is an added corrosion problem. Excessive salt in the concentrates would certainly invoke a penalty from smelters. If salt in concentrates becomes a problem it may be necessary to wash on the filter which would require some redesign of this section of the plant.

## 5.8 Plant Layout

If the plant was to operate again then some modifications are needed. These would include:-

(1) a walkway from the mill to the tailings thickener,

(2) a toilet and locker block outside but adjacent to the existing mill building,

(3) a covered reagent preparation area in a new position away from possible sources of contamination,

(4) a crib room inside the mill with glass windows similar to the mill supervisor's office.

A major change would be the redesign of the crushing circuit.

A report on necessary modifications has been prepared by D. Larkin.

## 6. MILL CONTROL AND ASSAYING

Samples were regularly taken for mill feed, tailings, and final concentrates. Also a sample was taken from the feed belt to the ball mill for feed moisture. Sampling devices were installed at the feed box to the flotation cells, discharge box for the tailings, and the concentrates line from the recleaner cells. This was the final concentrate.

The sample for moisture was taken by hand hourly. Samples were collected in buckets and taken by the operators to the sample preparation room at the end of the shift. The arrangement was a little slaphappy and there is no doubt that at times samples were contaminated.

It is essential that the samples be rapidly processed and results are made readily available. It seems essential that the assayer and assistant live on site. Assays should be done daily while the plant is operating, which includes weekends.

Attention should be given to the development of rapid assay methods which can be carried out by the leading hands or operators in the plant. Such methods are most useful when problems appear and when it is essential to obtain rapid answers. The laboratory facilities were barely adequate.

The emphasis on assaying appeared to be to obtain information for material balances and product quality rather than plant control. Both types of information are necessary. More effort should have been made to obtain information required to optimize the operation of the plant and so maintain throughput, grade and recovery. Information indicating a change in feed grade is vital to operators. Ideally there should not be any sudden changes over a short period. Processing plants in general are not easy to operate when there are wide fluctuations in head grades as was the case at Mount Gunson.

Such fluctuations usually result in lower than normal recoveries. Information that is needed on a running basis:-

copper assay for tails (a rough colorimetric method was used);

pulp densities;

sizing analysis of cyclone overflow;

and residual xanthate concentration in tailings pulps (this can be used to control the addition of xanthate);

feed rate;

tailings flocculation test made when needed and as a check on thickner performance.

Accurate and conscientiously kept shift log sheets with all pertinent information are essential and no shift supervisor should take over from another without first looking over the log sheet of the previous shift. In general this practice was adherred to.

## 7. DEVELOPMENT AND TEST WORK

"Mother Nature" makes life very interesting for the mineral processing engineer. No two ore bodies are exactly alike and there are often wide variations within a particular deposit.

Thus it is essential that samples of ore to be processed should be examined in the laboratory before being fed into the plant. These samples may come from drill core material or from mined material awaiting processing. Thus there is a warning of any possible processing difficulties and action can be taken. It is essential that someone on the staff be capable of conducting such test work in order to cope with emergencies as they occur. There is not always a good correlation between results from drill core material and "as mined" material - this often accounts for unforeseen problems encountered during the past operation of the Gunson concentrator.

Development work covers both short term and long term projects. A longer term project for Mount Gunson Mines Pty. Ltd. would be the processing of "oxide" ore.

## 8. PLANT MAINTENANCE

Lack of maintenance in the early days of the operation of the Mount Gunson concentrator as much as anything else contributed to the poor performance of the plant.

It is essential to have a practical engineer supported by a trained mechanical crew (not large) living on site and following a program of planned maintenance.

During the 18 months of operation at Mount Gunson only breakdown maintenance was carried out.

Plant tidiness and cleanliness was of a low order and resulted in a "don't care" attitude on the part of operators. Unfortunately some areas were prone to spillages and floodings. Every effort should have been made to remove the causes, although some of these problems, as in the case of the crushing circuit were a function of design and could not easily be rectified without heavy capital expenditure.

Vehicle maintenance was of a low order. The motor mechanic was just not able to keep up with the work. Part of the problem was due to a "Rafferty's rules" as to who drove vehicles. Also, the Volkswagon vehicles in use were not suited to the conditions prevailing prior to the construction of a road from the mine to the Woomera - Port Augusta highway.

## 9. STORES

Adequate spare parts are essential to keep the operation moving. Because of this there often is a tendency to over order and maintain stocks which can become redundant.

The store at Mount Gunson was not organized. There was a succession of store keepers. This is an essential part of any operation. The issue of tools was haphazard and no doubt a number of fitters left with more tools than they brought.

An outdoor fenced store yard is needed. At the present time materials which can be left outside are stored on an area adjacent to the power lines. It has been realized that this could be a hazard. Before any future start-up a new site for the store yard must be selected well away from these power lines. A recent accident indicates that this is important.

## 10. PLANT MANNING AND ADMINISTRATION

Throughout the operation there were manning problems due largely to the constant turnover of operators. The numbers employed were reasonable for that particular size of operation.

It seems vitally necessary that all staff and operators concerned with continuous operation of the mill should live on the mine site. Under the conditions existing at Gunson only the hourly paid labour lived on the mine site. For each shift three operators are required. These men should be trained for all positions both inside and outside the mill. The system which was used at Gunson of designating one man for flotation, one for ball mill operation, did lead to demarkation. It is also felt that the size of the operation does not warrant having staff shift supervisors, particularly if the mill foreman lives on site. Normally one operator is sufficient to control the crushing circuit. On day shift two men are needed for outside work on tailings etc; one man for concentrate handling, one labourer for clean-up and one man for general cleaning within the plant. For the stores it is recommended that a storeman purchasing officer be supported by an assistant.

It is essential that all operators be carefully trained for their positions and with this in mind no operator should be put into a position without having worked for at least one week on shift under direct supervision. The manning should always include sufficient additional men to cover leave requirements, sickness and sudden departures.

Mill maintenance should be under the supervision of a practical engineer. Two fitters and a boiler maker seem an absolute minimum. A good electrician is essential and it may be necessary to provide a house on site if the electrician is married. Although there was little in the way of recreation available at Gunson, it does seem essential that all workers should have at least one day a week off work.

It is recommended that work shifts proceeds for six days a week, one day off, with a long break once a month of five or six days. This appears necessary as a large proportion of the men employed at Gunson and living in the single mens quarters were in fact married, and had homes at Port Augusta or Adelaide and thus required the time off to visit families.

It appeared that single men also look forward to a long break once a month. Under no account should men be worked more than twelve hours per day. On numerous occasions operators were working a sixteen hour day and the quality of work suffered.

Greater care should be exercised in hiring operators. It appeared that many who came to Gunson were totally unsuited for the position and there is no doubt that an untrained operator can cause many problems and much expense. Concentrator staff needed would be mill superintendent, junior metallurgist, and mill foreman.

The administration of the operation should be under the control of a manager. Other office staff required would include accountant, clerk, typist/telephonist, female clerk. A mining engineer is needed to control mining operations and a geological staff for ore delineation and exploration. The size of the geological staff is somewhat dependent upon the exploration program.

Other needs are a surveyor and assistant, draftsman, chemist, assistant and two sample preparation men.

The size of the mining staff in the future depends as to whether M.G.M. does the mining or whether it employs a contractor.

## 11. CAMP

The camp at Mount Gunson was one of the best run and most comfortable of mining camps of which the writer has lived in. The provision of T.V. and films two or three times a week did much to relieve the boredom and the films to some extent reduced drinking. It is felt that regularly monthly breaks away from the operation are needed by all and would help to maintain morale and interest in the work.

The office of the mining contractor was located at a considerable distance away from the Mount Gunson Mine office which meant that there was little liaison between Mount Gunson mines and Davis Contractors. It is essential that under such conditions the two groups work closely together and, to encourage this, it is essential that both office buildings should be at least adjacent.

Considering all the problems which occurred while the writer was on site at Mount Gunson it seemed essential that the Mine Manager, Mill Superintendent, Mill Foreman and Shift Supervisors live on site in preferably a small townsite about ½ mile away from the single mens quarters. As the plant operates 24 hours a day, seven days a week it is necessary that those concerned with its continuous operation live on site.

There are both advantages and disadvantages for other staff persons such as accountant, personnel officer, draughtsman, geologists etc. to live in Woomera. Transport to and from Woomera was always a problem. It is admitted that there would be advantages for several staff members to live in town, if just to collect parcels, mail etc. and maintain a contact with the W.R.E. The social contacts have, in the past been most helpful.

Amenities needed would probably include a swimming pool which could also double as a water storage for fire protection.

## 12. MARKETING

The marketing of the concentrates produced is most crucial to any operation. However, it is utterly essential that when contracts are being negotiated realistic specifications be written with bonuses for producing material of higher grade than anticipated. In many instances in which the writer has been associated with sales, specifications have been written on the basis of an analysis derived from a pilot plant test run, often not necessarily a fair sample of what will be produced. Variations in the grade of concentrates do occur due to changes in the mineralogy of the deposit and often nothing can be done to change the situation when material goes off specification for that reason. Blending is often the only solution.

With the Mount Gunson operation there were small changes in concentrate grades at different times, some due to mineralogy and others due to plant problems. Generally, the maintenance of grade (+55 per cent copper) was not difficult. Constant liaison between the mine staff and the marketing agents is essential.

There were problems in shipping concentrates and these have been reported elsewhere.

## 13. EXPLORATION AND THE FUTURE

The life of any mine is limited. Some effort must be continually expended to prove up and expand ore reserves as well as locate and delineate new ore bodies.

Also, those responsible should look for other economic minerals which may co-exist on the lease being mined. This is particularly important with copper. In the case of Mount Gunson operation silver was found associated with copper.

It was unfortunate that more effort was not expended initially to expand and prove up reserves.

Exploration has continued since suspension of operations and the indications are that this work should continue. The locating of further mineralization gives promise to the future, however, the type of mineralization found in the new "Cattle Grid Area" is more complex than that under Pernatty Lagoon. Detailed mineralogy is needed in order to assess the economic worth of this deposit.

## 14. CONCLUSION

This report is based on the experiences gained by the writer during eight months working at the Mount Gunson Mines. It has been written with a bias towards the future. Hopefully, it is assumed that new ore bodies will be proved up, that the copper price will move upwards, and so operations can once more commence.

There are many problems to face.

More detailed reporting on some aspects will be needed before consideration to a plant start-up is made. The existing plant is not entirely satisfactory. If it was proposed to process higher grade ore in the plant some alterations would be necessary. The crushing circuit needs redesign in any event.

The effort put into detailed planning would be amply repaid in a more efficient operation. Time may be lost doing this but the overall gain would warrant the effort.

The essential point is to design to operate continuously with a steady output of on-grade product with a good recovery. Only two elements, an on-grade product and a reasonable recovery, were present with the Mount Gunson operation.

All three elements are needed to make an operation successful. There are other elements such as operating and capital costs, and the price which is obtained for the product which make an operation economically successful. All elements are interelated and each must be studied in detail.

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MEMO TO: MR. H. VAN HERK

FROM: M. L. HOUSTON

## MT. GUNSON

## ECONOMICS OF MINING AND LEACHING LOW-GRADE ORE

### 1. INTRODUCTION

A zone of low-grade ore has been identified, below the high-grade ore zone in the Cattle Grid orebody.

The object of this study was to determine the economics of mining and treating this low-grade ore.

The orebody upon which mining plans are currently being based has an assay boundary in the footwall. Mineralisation continues down to variable depths and a decision has to be made, before mining starts, on whether or not any attempt should be made to recover this mineralised ground. Because the planned mining method involves the return of overburden into mined out areas, once the ultimate mining depth is decided, the decision is irrevocable.

only those areas that have been filled. We can decide some time later (during the pla… mining phase) to start extraction of low grade

The decision to recover low-grade ore is influenced by Econ… ? philosophical and practical factors.

In general ~~one~~ we should try to maximise the total recovery from ~~an~~ the orebody. However, ~~one~~ we ~~must have~~ need a practical and viable method of treating ore which is of below cut-off grade, and ~~or~~ of recovering some, if not all, of the contained minerals.

Several alternatives present themselves:-

(a) The low-grade ore could be treated in the concentrator. This is not acceptable, because, without expanding the capacity of the concentrator, this would mean low-grade ore would prevent high-grade ore from being treated, with an adverse effect on the economics of the operation.

① a loss is made on every ton. below "cut-off"

②

(b) The low-grade if above cut-off ore could be stockpiled and blended with super-high-grade ore, as it presents itself from time to time, in order to preserve a uniform grade of feed to the concentrator. This is a legitimate alternative but would X/o have a limited application.

ⓐ increased ⓑ or lowered costs

This is OK if the Cu price ~~not~~ lowers the cut-off below ~~to~~ what is stockpiled. Otherwise loss is made on each ton

However oxidation will take place

(c) The low-grade ore could be stockpiled and leached to recover the copper. The leach solutions could be stripped of their copper by cementation with scrap iron. This is a well established practice and has formed the basis of the study presented on the following pages.

Yes, if ~~economics~~ the ~~revenue from this exceeds the capital cost~~ a gross surplus from this operation provides adequate return on the capital outlay required

## 2. SUMMARY AND CONCLUSIONS

An examination of the assay results from the 20 holes included in the 1.0% cut-off pit outline indicates that there is a reserve of approximately 2,500,000 tonnes of ore with a mean grade of 0.58% Cu

check 3.9 @ 2.3 ~~2.5@~~

lying beneath the ore zone planned for mining.

The cost of mining and leaching this low-grade ore has been estimated and the effects on the economics of the project are summarised in the table below:-

## Table 1

## Summary of Economic Effects of Mining and Dump Leaching Low-Grade Ore

(Monetary values in millions of dollars)

| | High-Grade Ore Only | High & Low Grade Ore | Increase | % Increase |
|---|---|---|---|---|
| Total Cash In | 63.12 | 69.35 | 6.23 | 9.9 |
| Total Cash Out | 58.14 | 63.80 | 5.66 | 9.7 |
| Net Cash Flow | 4.98 | 5.~~6~~55 | 0.57 | 11.4 |
| D.C.F. Rate | 15% | 16% | 1 | 6.7 |

It will be noted from this table that although the increase in total net cash flow is 11.4%, the increase in the D.C.F. rate is only 6.7%. This is due to the delay in additional income provided by the dump leaching of the low-grade ore. (and immediate cost of ~~sc~~ implementing the system)

On the basis of this study, there is a slight economic advantage to be gained by mining and leaching the low grade ore. [There is a practical advantage also in having a stockpile of low-grade ore for blending purposes. ?] (oxidised ?)

We must not blend below cut-off material with ~~[illegible]~~ above cut-off material

However, the leaching process would be a major water consumer, 3,420 gallons per hour, and would only be feasible if water in these quantities were available.

~~Most~~ ~~Probably~~ the most adverse aspect of the scheme

24X7X3,420
= ~~574~~
575,000
gp week

This is what we are searching for to start up Cannon

The course of action recommended is therefore to postpone making a decision on the ultimate mining

Could delay [illegible] [illegible] postpone it indefinitely

If water is available Shouldn't we increase the mill capacity and therefore lower the C/O grade and increase ore reserves

000393

depth until we have satisfied ourselves that:-

(a) The low-grade ore zone indicated by preliminary drilling is in fact an identifiable zone of mineralisation. It should be (by assay) ~~Even if by assay~~

(b) The low-grade ore is amenable to dump leaching and that (economic) viable recoveries may be obtained. Shouldn't it be amenable?

(c) Water is available in sufficient quantity to enable a successful leaching operation to be carried out.

and (d) The preliminary cost and income calculations of this study are confirmed by subsequent work.

① Only marginal increase in income (return)
② By Gunson standards tremendous water demand } not recommended at this stage
③ Propose bulk test during mining – to test scheme – to assess recoveries
④

Can always start later with this proposal. [illegible]

## 3. NATURE OF THE LOW-GRADE ORE ZONE

The low-grade ore zone appears to consist of (? only? ? vertical to sub-vertical) joints in the quartzite, containing varying concentrations of sulphides.

In assessing the available quantity of this material, the top of the zone was taken as 0.5 metres below the depth of the high-grade zone defined by the 1.0% cut-off analysis. To what depth (a) or (b) minimum grade 0.3%

The mean thickness of the zone is 2.74 metres and the mean grade is 0.58% Cu.

Using the area of the pit floor in the 1.0% cut-off design and a specific gravity of 2.4, the tonnage of material available = 2.74 x 388,682 x 2.4

= 2,556,000 tonnes.

The details of the assays are shown in Table 2.

## Table 2

### Low-Grade Ore Zone - Details of Analyses

| Hole | Low-Grade Ore Zone | | | |
|---|---|---|---|---|
| No. | From<br>metres | To<br>metres | Thickness<br>metres | Grade<br>% Cu |
| 1 | 29.67 | 34.67 | 5.00 | 0.87 |
| 2 | 37.52 | 44.22 | 6.70 | 0.44 |
| 3 | 48.14 | 49.48 | 1.34 | 0.46 |
| 6 | 43.00 | 46.70 | 2.00 | 0.63 |
| 13 | 36.50 | 41.00 | 4.50 | 0.64 |
| 14 | 50.93 | 51.50 | 0.57 | 0.53 |
| 21 | 41.00 | 43.50 | 2.50 | 0.46 |
| 24 | 38.50 | 41.50 | 3.00 | 0.46 |
| 26 | 43.10 | 44.60 | 1.50 | 0.66 |
| 27 | 36.00 | 37.50 | 1.50 | 0.50 |
| 28 | 35.50 | 38.50 | 3.00 | 0.21 |
| 31 | 44.50 | 46.00 | 1.50 | 0.50 |
| 32 | 44.00 | 47.00 | 3.00 | 0.50 |
| 33 | 39.50 | 42.00 | 2.50 | 0.39 |
| 36 | 46.30 | 48.00 | 1.70 | 0.45 |
| 59 | 37.50 | 40.03 | 2.53 | 0.42 |
| 60 | 37.00 | 42.00 | 5.00 | 0.68 |
| 61 | 45.00 | 48.00 | 3.00 | 0.88 |
| 70 | 48.50 | 50.06 | 1.56 | 0.48 |
| 72 | 43.50 | 46.00 | 2.50 | 1.00 |

Weighted Average Thickness and Grade of 20 holes 2.74 metres 0.58 % Cu

(equal area of zone of influence assumed)

↑ ~~Weighted~~?

(~~areas of influence~~?

~~depth~~ length of intersection?)

## 4. THEORY OF DUMP LEACHING

Australian experience with dump leaching has been described by Woodcock[1], Anderson and Allman[2] and Moss and Anderson[3].

Current U.S. practice is described by Robinson[4] and Malouf[5, 6].

1. Woodcock, J. T., 1967, "Copper Waste Dump Leaching". Proc. Aus. I.M.M., No. 224, 47-66.

2. Anderson, J. E., and Allman, M. B., 1968, "Some Operational Aspects of Heap Leaching at Rum Jungle". Proc. Aus. I.M.M., No. 225, 27-31.

3. Moss, F. J., and Anderson, J. H., 1968, "The Effects on Environment of Bacterial Leaching Rates". Proc. Aus. I.M.M., No. 22~~3~~5, 15-27.

4. Robinson, W. J., 1972, "Finger Dump Preliminaries Promise Improved Copper Leaching at Butte". Min. Eng., Sept. 1972, 47-49.

5. Malouf, E. E., 1972, "Current Copper Leaching Practices", Min. Eng., Aug. 1972, 58-60.

6. Malouf, E. E., 1971, "The Role of Micro-organisms in Chemical Mining". Min. Eng., Nov. 1971, 43-46.

....../6

The results of these workers may be summarised as follows:-

The process involves circulating water through the dumps, which may contain from 0.1 to 1.0% Cu, oxidising sulphide minerals to soluble sulphates and then precipitating the dissolved copper onto scrap iron.

The dissolution of copper sulphides is a slow process, involving in general three oxidising agents, oxygen, ferric sulphate and bacteria, all in conjunction with dilute sulphuric acid. The acid is generated in the dissolution process, ferric sulphate is generated if sufficient pyrite or other iron sulphide (pyrrhotite, chalcopyrite) is present, oxygen is present if the dump is permeable enough and bacteria, e.g. thiobacillus ferroxidans, are generally present under these conditions, but the leach solution may be innoculated with the strain if they are not.

In general, the requirements for a successful leaching operation are:-

(a) The dump should consist of siliceous rocks, with little acid-consuming mineral present and should not decompose during leaching. If the dumps contain acid-consuming minerals, acid may have to be added, and if the rocks decompose percolation may cease.

(b) There should be an ample supply of water available, but there should be no heavy rainstorms. An inadequate water supply restricts leaching but heavy rain dilutes the solution

Don't we get these?

(0.15) 0.175

unnecessarily. Water circulation should be between 0.1 and 0.25 gallons per hour per square foot of dump surface ($1.36 \times 10^{-3}$ - $3.40 \times 10^{-3}$ l/sec/m$^2$).

what figure do you we use?

(c) Chalcocite and bornite are the preferred copper sulphides, plus a little pyrite to provide free acid and ferric iron. Covellite and chalcopyrite dissolve less readily than chalcocite and bornite and require more careful control of temperature, bacteria and ferric iron concentration in solution. Too much pyrite leads to a high iron consumption in cementation.

We have lots of pyrite See report P. Curtis

(d) Good fragmentation during mining, with a consequent good exposure of the copper minerals is desirable. Minerals occuring as aggregates in large boulders naturally require a longer leaching time.

Precipitation is usually carried out in launders, divided into sections. Iron, usually in the form of crumpled cans, is stacked on a wooden grid in the launder and the pregnant solution gravitates through the system. At appropriate intervals, each section is drained and the metallic copper washed with high pressure hoses through the grid into a settling pond. Addtional metallic iron is then charged to the cells and the flow of solution resumed. Iron for cementation should have a high surface area per unit of mass and be relatively pure.

/9

Cement copper produced usually contains between 30 and 95% Cu. The main impurities are basic iron salts, metallic iron and slimes derived from the dumps. It may be dried, packaged and shipped as a separate product or mixed with flotation concentrates to upgrade the latter.

From the foregoing comments, it will be noted that, apart from the high water requirement, the conditions at Mt. Gunson appear to favour a dump leaching operation.

## 5. ASSUMPTIONS

The following assumptions have been made with respect to a leach and precipitate operation at Mt. Gunson:-

- (a) That we will mine 500,000 tonnes per year of high grade

(a) That the low-grade ore will be mined concurrently with the high-grade ore and at a proportional rate, e.g. for 500,000 tonnes per year of high-grade ore 314,000 tonnes per year of low-grade ore will be mined.

(b) That the low-grade ore will be stockpiled in finger dumps, each dump equivalent to one year's supply of ore.

(c) That the low-grade ore is amenable to heap leaching and that a time lag of 12 months will occur between the start of making dump formation and the production of cement copper.

(is this reasonable if so please state)

i.e. 2 years after production of rock begins or after start of dump formation (1 year to end ew)

(d) Operating paramaters will be:-

| | |
|---|---|
| Concentrates grade | 70% Cu |
| Dump recovery | 60% |
| Precipitation plant recovery | 90% |
| Overall recovery of copper in cons. | 54%. |

Reasonable?

## 6. DESIGN

### 6.1 General Layout

The finger dumps will be generally as shown in Fig. 1. (where please?)

The volume of each dump is $160,000m^3$. ✓

The approximate dimensions are:-

| | |
|---|---|
| Top of dump | 42m x 252m |
| Bottom of dump | 70m x 280m ✓ |
| Height of dump | 10m |
| Distances between dumps | 10m. |

Water will be pumped from the sump at the precipitation plant through a main distribution pipe with branches to the top of each dump.

Local reticulation will be by four perforated pipes running longitudinally down each dump. These will permit a continuous trickle over the surface of the dump.

....../11

Cont?

The dumps will be laid down on an impervious prepared base. Pregnant solution will gravitate from the bottom of each dump to a collection box and thence through a main collection pipe to the precipitation plant.

The lengths of piping required are:-

| | |
|---|---|
| Main distribution pipe | 1,000m |
| Main collection pipe | 600m |
| Local reticulation pipe | 1,000m/dump |
| Local collection pipe | 30m/dump. |

diagram?
— 4 runs?

6.2 Dump and Pipe Sizes

Assume volume flow required = $2.0 \times 10^{-3}$ l/sec/m$^2$ (0.15 gal/hr/sq.ft.). why not $\frac{1}{2} \times (0.1 + 0.25)$ = 0.175 gal [illegible]

For a dump area of $\overset{(280 \times 70)?}{252 \times 42}$ = 10,584m$^2$. Volume flow = $2.0 \times 10^{-3} \times 10{,}584$ = 21.17 $\overset{\text{litre}}{\text{l}}$/sec. ✓

4.65 gals/sec
0.74 cu ft/sec
Vel = $\frac{0.74}{0.2}$ = [illegible]
[illegible] ft/sec

For a 150mm (6") pipe of 1,000m. in length, the friction head, at this rate of flow is 9m. ✓

For the local reticulation pipe, 75mm (3" diameter), the friction head is 18m.

The vertical lift is 10m, so the total head is 9 + $\overset{18}{(18)}$ + 10 = 37m. Allowing 3m for losses in bends and fittings, we would require a pump capable of delivering 21.17l/sec against a head of 40m. (280 gal/min @ 130 ft. head).

this is confusing (litre)

$$\text{Theoretical water HP} = \frac{280 \times 130 \times 10}{33{,}000}$$

$$= 11.0 \checkmark$$

$$= 8.2 \text{ KW.}$$

Assuming an efficiency of 68% (50%), motor power = 12 KW (16 KW)

....../12

## 7. COST ESTIMATES

### 7.1 Capital Costs

7.1.1 Alternative 1 - Mining High Grade Ore Only. The following capital cost estimates have been assumed for the base case:-

| | 000's $ | |
|---|---|---|
| Exploration and feasibility | 1,000.0 | make 500, This is a high [illegible] |
| Pre-production development | 525.0 ~~625.0~~ | 1000000 × 25 = 250000 - make 250.0 |
| Overburden mining equipment | 250 1,000.0 | |
| Ore mining equipment | 600.0 | |
| Concentrator modification and housing | 1,000.0 | |
| Sub-total | 4,125.0 | [illegible] |
| Contingency 10% (built in above) | 412.5 | |
| Total | 4,537.5 | Too high |

It is assumed, for the purposes of cash flow calculations, that this is all spent in the year immediately prior to production commencing. ✓

7.1.2 Alternative 2 - ~~Mining~~ as above, but with including and Dump Leaching of Low-Grade Ore. The capital costs ~~for the base case~~ will apply. as ~~for the~~ above and

In addition, the following expenses will be incurred:-

| | | 000's $ | |
|---|---|---|---|
| Additional mining equipment | | 261.0 | |
| Dump preparation - | | | |
| Bulldozing 300 hrs. @ $15.00/hr. | $ 4,500 | | |
| Clay, 50,000m$^3$ @ $2.00/m$^3$ | $100,000 | | 1/3rd of the dump volume?<br>too low<br>plus cost of obtaining material |
| Compaction, 300 hrs. @ $12.00/ hr. | $ 3,600 | 108.1 | |
| Pumps and pipework - | | | |
| Pump | $ 3,500 | | ? |
| 1,000m main distribution pipe @ $14.00/~~hr.~~m | $ 14,000 | | ← (6" pipe $2/ft) |
| Fittings for above | $ 1,400 | | |
| 8,000m local reticulation pipe @ $3.20/m | $ 25,600 | | |
| Fittings for above | $ 2,600 | | |
| 8 collection boxes @ $500 | $ 4,000 | | |
| 600m main collection pipe @ $18.00/m | $ 10,800 | | |
| Fittings for above | $ 1,100 | | |
| Sub-total | $ 63,000 | | |
| Installation @ 25% | $ 15,800 | 78.8 | |
| Precipitation plant - | | | |
| Building | $ 10,000 | | (size?) |
| Tanks and launders | $ 5,000 | | |
| Wash-down equipment | $ 2,000 | | |
| Drying pads | $ 2,000 | 19.0 | |

? This for the base
or only for the [illegible]

....../14

| | |
|---|---|
| Total additional capital | 466.9 (000's $) |

For the purpose of cash flow calculations, it will be assumed that part of this cost will be incurred in the first year of production and the balance in equal increments during the remaining seven years of operation.

| Initial cost | 000's $ |
|---|---|
| Mine equipment | 261.0 |
| 25% of dump preparation | 27.0 |
| Pump | 3.5 |
| Main distribution pipe and fittings | 15.4 |
| Main collection pipe and fittings | 11.9 |
| 12.5% of local reticulation pipe, etc. | 4.0 |
| Installation of pump and pipe work | 15.8 |
| Precipitation plant | 19.0 |
| Sub-total | 357.6 |
| Contingency @ 15% | 53.6 |
| Total initial cost | 411.2 |

| Annual Cost | 000's $/year |
|---|---|
| Balance of dump preparation $81,100 | 11.6 |
| Balance of local reticulation etc. $28,200 | 4.0 |
| Sub-total | 15.6 |
| Contingency @ 10% | 1.6 |
| Total annual cost | 17.2 |

divided by 7?

## 7.2 Operating Costs

7.2.1 Alternative 1 - High-Grade Ore Mining Only. Cost estimates are based on the following production estimates:-

| | | |
|---|---|---|
| Ore production | 500,000 tonnes | |
| Concentrates produced | 21,928 tonnes | 500,000 × 2.3% × .72 |
| Contained copper | 8,771 tonnes | 8280 |
| Copper sold | 7,984 tonnes | make 8000 |

The annual costs will be:-

| | | 000's $/year | |
|---|---|---|---|
| **Mining** | | | |
| Ore 500,000 tonnes @ 50¢/tonne | $250,000 | | |
| Waste 500,000 tonnes @ 50¢/ tonne | $250,000 | | |
| Waste 4,500,000 tonnes @ 25¢/ tonne | $1,125,000 | 1,625.0 | ✓ |
| Geology & Ore Delineation | | 162.1 | ✓ |
| **Concentrating** | | | |
| 500,000 tonnes ore @ $2.42/ tonne | 2.36 × 1.12 = 2.64 @ 360,000 | 1,212.0 | ? |
| Administration | | ~~412.6~~ | 336 ? |
| **Selling** | | | |
| 21,928 tonnes cons. @ $68.00/ tonne | | 1,491.1 | |
| **Treatment** | | | |
| 8,771 tonnes Cu @ $116.00/ tonne | | 1,017.4 | |
| Sub-total | | 5,920.2 | |

| | | |
|---|---|---|
| Contingency | 5% | 296.0 |
| Total annual cost | | 6,216.2 |

a bit high for 500,000. (This is the cost for 380,000 tpy) (1974) $5,500,000 is more likely

For cash flow purposes, it is assumed that six months waste removal will be undertaken in the pre-production period so that in the final year of operation only 2,500,000 tonnes of waste will have to be removed.

The mining cost in the final year will be - $1,000,000. good point!

The total cost will accordingly be - $5,560,000. (say $4,800,000)

### 7.2.2 Alternative 2 - Mining and Dump Leaching of Low Grade Ore.

Cost estimates are based on the following estimates of quantities:-

| | |
|---|---|
| High-grade ore production | 500,000 tonnes |
| Concentrates produced | 21,928 tonnes |
| Concentrates grade | 40.0% Cu |
| Contained copper | 8,771 tonnes |
| Low-grade ore production | 314,000 tonnes |
| Concentrates produced | 1,404 tonnes |
| Concentrates grade | 70.0% Cu |
| Contained copper | 983 tonnes ✓ |
| Total concentrates production | 23,332 tonnes |
| Mean concentrates grade | 43.6% Cu |
| Contained copper | 9,754 tonnes |
| Copper sold | 8,779 tonnes |

Can they be mixed? What about the iron scrap remnants!

The estimated annual costs ~~will be~~ are:-

| | 000's $/year |
|---|---|
| Mining | |
| High grade ore 500,000 tonnes @ 50¢/tonne | 250.0 |
| Low grade ore 314,000 tonnes @ 50¢/tonne | 157.0 |
| Waste 500,000 tonnes @ 50¢/tonne | 250.0 |
| Waste 4,500,000 tonnes @ 25¢/tonne | 1,125.0 |
| | 1,782.0 |
| Geology and ore delineation | 162.1 |
| Concentrating | |
| 500,000 tonnes @ $2.42/tonne | 1,212.0 ✓ |
| Dump leaching (see Appendix 1) | 234.9 ✓ |
| Administration | 412.6 — Too high |
| Selling | |
| 23,332 tonnes @ $68.00/tonne | 1,586.6 |
| Treatment | |
| 9,754 tonnes @ $116.00/tonne | 1,131.5 |
| Sub-total | 6,521.7 |
| Contingency @ 5% | 326.1 |
| | 6,847.8 |

note
157/1625 = 10% increase in mining cost.
2½% of Total.

6848
6216
632
about 10% increase in costs

This is the cost for a normal year of operation. However, in the first year of production it is assumed that no leach expenses are incurred and no production of precipitation plant concentrates is undertaken.

....../19

The annual costs for this year are reduced to $6,381,100.

Also in the final year of production only half the overburden has to be removed and the annual costs for this year are reduced to $6,191,500.

### 7.3 Income

A copper price of $1,000/tonne has been assumed throughout.

The annual income in Alternative 1 is therefore $7,894,000 and in Alternative 2 $8,779,000.

A ~~four~~ 5 months lag has been allowed between cost of production and realisation on sales. should be 5 months

No allowance has been made for payment for secondary minerals. ✓

## 8. CASH FLOW

The cash flow calculations are shown in Table 3.

The discounted cash flow rate for Alternative 1 is 15% and the rate for Alternative 2 is 16%.

The economic advantage of mining and dump leaching the low grade ore is therefore marginal.

The most critical factors in this exercise are the availability of water and the recovery rate to be

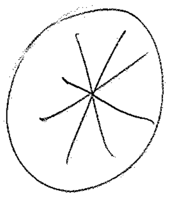
....../20

expected.

If water in sufficient quantities cannot be found the whole concept fails. If, however, the required water does become available, maximisation of leach recovery should be investigated thoroughly.

Should it be possible to raise the overall mine-site recovery rate from 54% (as used in this exercise) to 65%, the D.C.F. rate would be increased from 16% to 22%.

is this possible where?



........................
M. L. HOUSTON

## TABLE 3

## COMPARISON OF CASH FLOWS

(All monetary values in millions of dollars)

| **Year** | | 0 | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | TOTAL |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Alternative 1 - Mining of High-Grade Ore Only** | | | | | | | | | | | | |
| Quantities | | | | | | | | | | | | |
| Overburden Removed | 000's tonnes | 2,500 | 5,000 | 5,000 | 5,000 | 5,000 | 5,000 | 5,000 | 5,000 | 2,500 | - | 40,000 |
| Ore Mined | 000's tonnes | - | 500 | 500 | 500 | 500 | 500 | 500 | 500 | 500 | - | 4,000 |
| Concentrates Produced | tonnes | - | 21,928 | 21,928 | 21,928 | 21,928 | 21,928 | 21,928 | 21,928 | 21,928 | - | 175,424 |
| Contained Copper | tonnes | - | 8,771 | 8,771 | 8,771 | 8,771 | 8,771 | 8,771 | 8,771 | 8,771 | - | 70,168 |
| Copper Sold | tonnes | - | 5,263 | 7,894 | 7,894 | 7,894 | 7,894 | 7,894 | 7,894 | 7,894 | 2,631 | 63,152 |
| Cash In | | | | | | | | | | | | |
| Copper Sales @ $1,000/tonne | | - | 5.26 | 7.89 | 7.89 | 7.89 | 7.89 | 7.89 | 7.89 | 7.89 | 2.63 | 63.12 |
| Cash Out | | 4.0 | | | | | | | | | | |
| Capital Expenditure | | 4.54 | 55 | - | - | - | - | - | - | - | - | 4.54 |
| Operating Costs | | - | 6.22 | 6.22 | 6.22 | 6.22 | 6.22 | 6.22 | 6.22 | 5.56 | - | 49.10 |
| Tax | | - | - | - | - | - | 0.56 | 0.79 | 0.79 | 1.11 | 1.25 | 4.50 |
| Total Cash Out | | 4.54 | 6.22 | 6.22 | 6.22 | 6.22 | 6.78 | 7.01 | 7.01 | 6.67 | 1.25 | 58.14 |
| Net Cash Flow | | (4.54) | (0.96) | 1.67 | 1.67 | 1.67 | 1.11 | 0.88 | 0.88 | 1.22 | 1.38 | 4.98 |
| **Alternative 2 - Mining of Low-Grade Ore and Dump Leaching** | | | | | | | | | | | | |
| Quantities | | | | | | | | | | | | |
| Overburden Removed | 000's tonnes | 2,500 | 5,000 | 5,000 | 5,000 | 5,000 | 5,000 | 5,000 | 5,000 | 2,500 | - | 40,000 |
| High-Grade Ore Mined | 000's tonnes | - | 500 | 500 | 500 | 500 | 500 | 500 | 500 | 500 | - | 4,000 |
| Low-Grade Ore Mined | 000's tonnes | - | 314 | 314 | 314 | 314 | 314 | 314 | 314 | 314 | - | 2,512 |
| Concentrates Produced | tonnes | - | 21,928 | 23,332 | 23,332 | 23,332 | 23,332 | 23,332 | 23,332 | 23,332 | - | 185,252 |
| Contained Copper | " | - | 8,771 | 9,754 | 9,754 | 9,754 | 9,754 | 9,754 | 9,754 | 9,754 | - | 77,049 |
| Copper Sold | " | - | 5,263 | 8,779 | 8,779 | 8,779 | 8,779 | 8,779 | 8,779 | 8,779 | 2,631 | 69,347 |
| Cash In | | | | | | | | | | | | |
| Copper Sales @ $1,000/tonne | | - | 5.26 | 8.78 | 8.78 | 8.78 | 8.78 | 8.78 | 8.78 | 8.78 | 2.63 | 69.35 |
| Cash Out | | | | | | | | | | | | |
| Capital Expenditure | | 4.54 | 0.41 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | - | 5.09 |
| Operating Costs | | - | 6.38 | 6.85 | 6.85 | 6.85 | 6.85 | 6.85 | 6.85 | 6.19 | - | 53.67 |
| Tax | | - | - 5.6 | - | - | - | 0.75 | 0.91 | 0.91 | 1.22 | 1.25 | 5.04 |
| Total Cash Out | | 4.54 | 6.79 | 6.87 | 6.87 | 6.87 | 7.62 | 7.78 | 7.78 | 7.43 | 1.25 | 63.80 |
| Net Cash Flow | | (4.54) | (1.53) | 1.91 | 1.91 | 1.91 | 1.16 | 1.00 | 1.00 | 1.35 | 1.38 | 5.55 |

DCFA08 ?

## APPENDIX 1

## CALCULATION OF LEACHING AND PRECIPITATION COSTS

is this a 24hr. operation or 2 shifts

| | $/year | |
|---|---|---|
| **Labour and Supervision** | | |
| 2 operators in precipitation plant @ $5,000 p.a. | 10,000 | |
| 1 operator on dumps @ $5,000 p.a. | 5,000 | |
| ½ maintenance man @ $6,000 p.a. | 3,000 | |
| ½ supervisor/technical officer @ $8,000 p.a. | 4,000 | |
| Total direct labour | 22,000 | |
| Add 16% for labour on-costs | 3,500 | 7000 |
| Total labour costs | 25,500 | (30,000) |

30% At Gov!

**Scrap Iron**

Assume iron consumption is 2KgFe/KgCu.

Annual copper production in concentrates is 983 tonnes or 983,000 Kg.

Annual iron consumption is therefore 1,966,000 Kg.

| | | |
|---|---|---|
| Annual cost @ 6.5¢/Kg | 127,790 | |

**Power**

| | |
|---|---|
| Main pump, 12KW continuous service, 103,700 KWhrs/year @ 2¢/KwHr. | 2,100 |
| Incidential power, @ 10% of above | 200 |
| | 2,300 |

**Water**

This is a crucial %. In an arid region like Gunson it could be 70% or more

Assume 20% of the total water circulating is lost in evaporation, washing and drying.

Total make-up water requirements = 0.20 x 17,100 gal/hr.

~~Do we~~ After 3 years would we still be using water on the 1st dump?

Past operation indicates only 23% return water (i.e 72% water loss)

...../2

| | $/year | |
|---|---|---|
| = 3.42 thousand gal/hr. (3,420X3.5= 11,970 gph = 200gpm) | | |
| = 29,500 thousand gal/year. | | |
| Cost @ 97¢/thousand gal. Saltywater. | 28,700 | X3.5 = 100,000 |

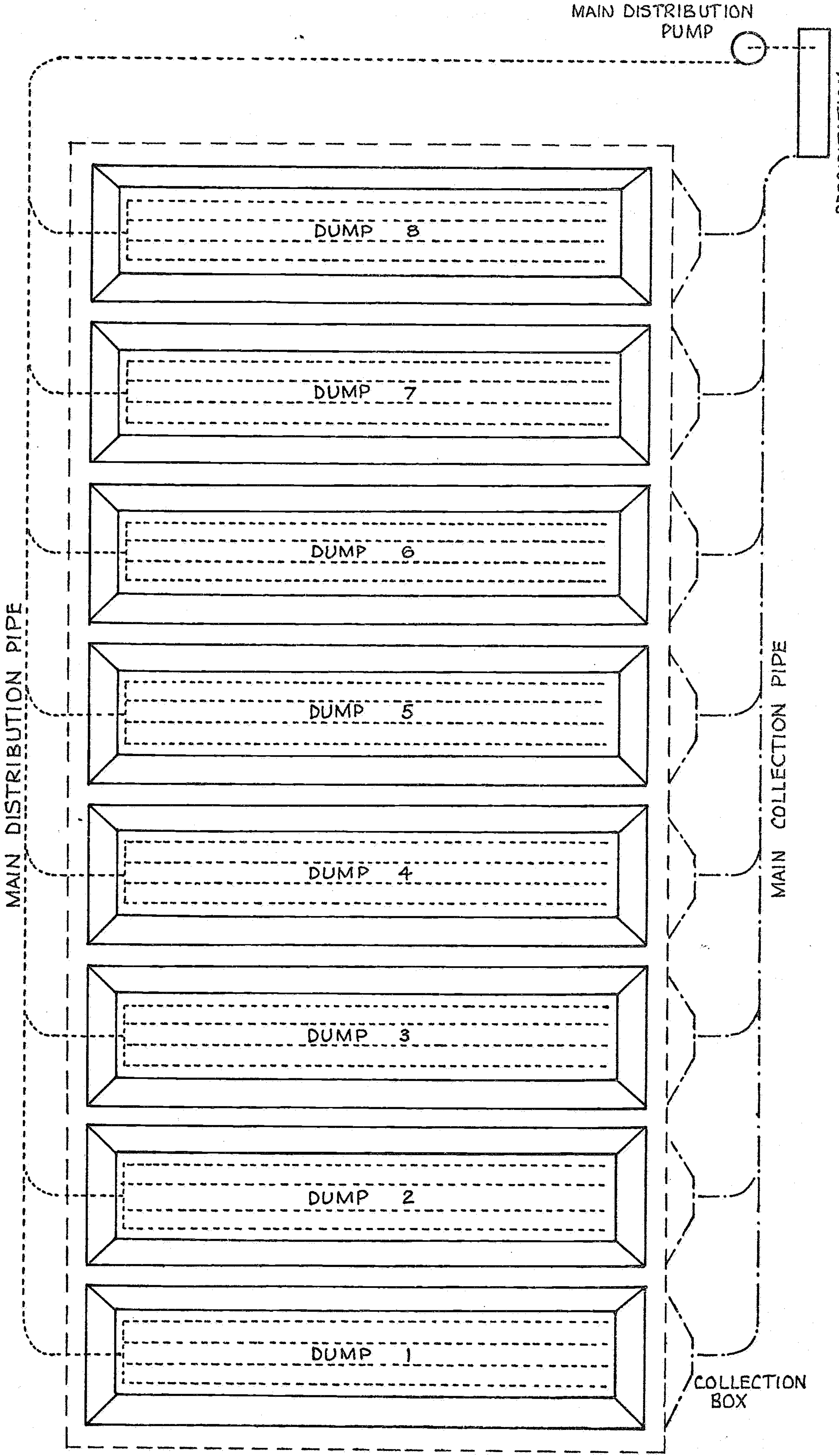
Maintenance

Assume all pumps and pipework are replaced once during the life of the operation.

| | | |
|---|---|---|
| Mean annual cost = $\frac{78,800}{8}$ | 9,850 | ✓ |

Summary

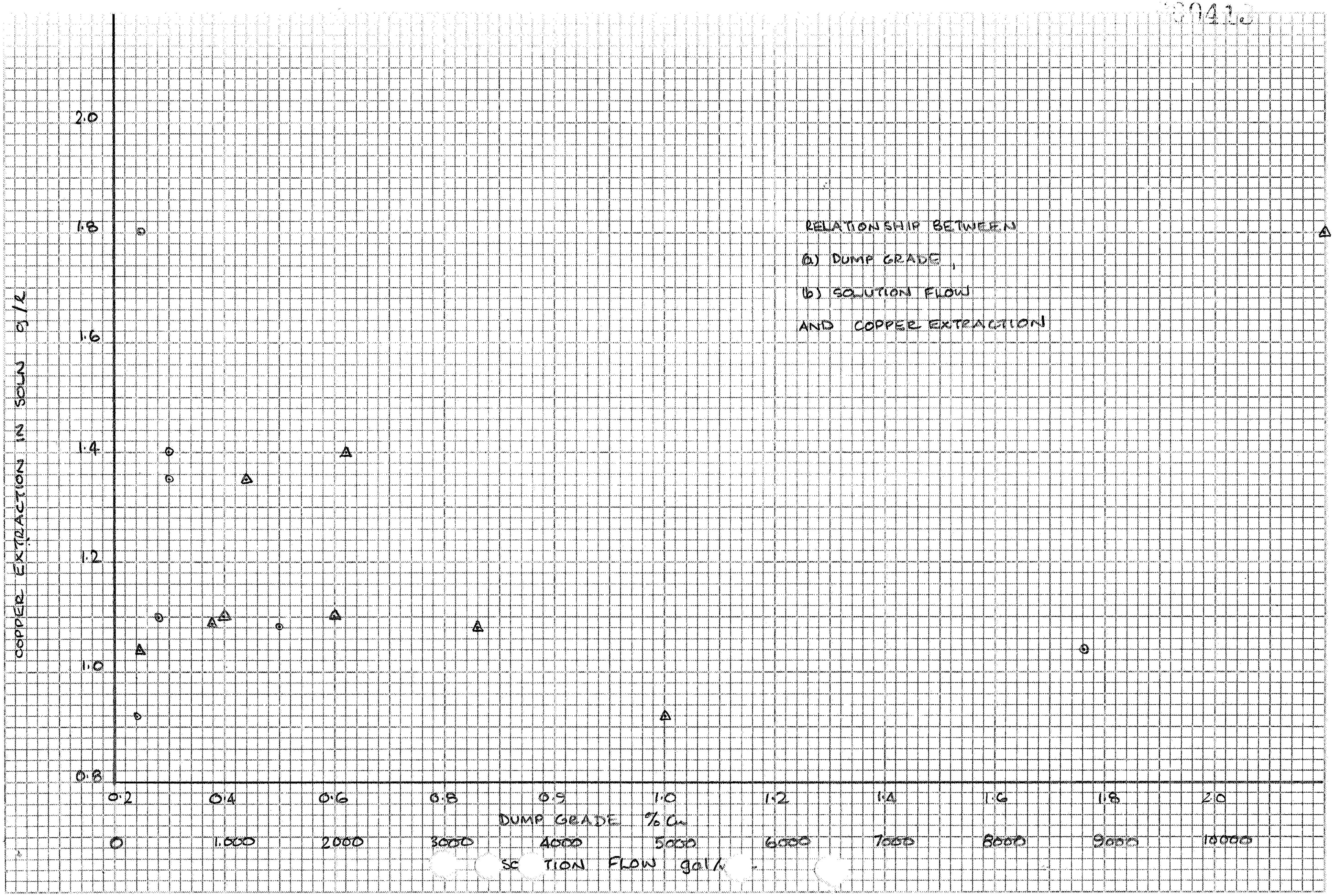
| | | |
|---|---|---|
| Labour | $ 25,500 | + 5000 |
| Scrap iron | 127,790 | ✓ |
| Power | 2,300 | |
| Water | 28,700 | + 70,000 |
| Maintenance | 9,850 | |
| Sub-total | 194,140 | 270,000 |
| Site overheads @ 10% of above | 19,414 | 27,0 |
| Sub-total | 213,554 | 297,000. |
| Contingency @ 10% of above | 21,355 | 30,000 |
| Total annual costs | $ 234,909 | 330,000 |



## LAYOUT OF DUMPS & LIQUOR CIRCULATION SYSTEM

| | | | | |
|---|---|---|---|---|
| **PACMINEX PTY. LIMITED** | DRAWN | M.J.H. | SCALE | 1 : 2,500 |
| | DATE | MARCH '73 | DWG. NO. | FIG. 1 |
| | CHECKED | | | |

20412


RELATIONSHIP BETWEEN
(a) DUMP GRADE,
(b) SOLUTION FLOW
AND COPPER EXTRACTION
COPPER EXTRACTION IN SOLN g/l
2.0
1.8
1.6
1.4
1.2
1.0
0.8
0.2
0.4
0.6
0.8
0.9
1.0
1.2
1.4
1.6
1.8
2.0
DUMP GRADE %Cu
0
1,000
2000
3000
4000
5000
6000
7000
8000
9000
10000
SC TION FLOW gal/

Sampling — picked up

H. Van Herk

FILE NOTE 23rd November, 1972.

Extracts from Amdel Report CM 1.1.138 To S.A. Department of Mines.

## PROGRESS REPORT NO. 2. SEPTEMBER, 1972 - TREATMENT OF LOW GRADE COPPER ORES.

Assumptions:

1. Operation near a small town.
2. On ETSA grid Power at 2¢/KWH.
3. Water at 50¢/1000 gals.
4. No capital for working capital offices etc.
5. No expense for tailings dam.
6. Ore grade 2%. (40 lb/cu per short ton).
7. Ore by open/cut - mine overburden.
8. Ore mined at rate of 10,000 short tons/year.
9. Labour requirements -

| | |
|---|---|
| Mining and carting | 1 man |
| Crushing | ½ man |
| Leaching | 1½ men |
| Cementation | 1 man |
| | 4 men |

10. Capital depreciated over 3 years.
11. 8 hr. day 6 day week 270 day/year single shift.
12. Cu as cement copper (70% Cu) at 33¢/lb. at plant site (Market price 42¢/lb). The payment for 70% cement at Port Kembla after deductions and Tariff is 34¢/lb. of contained Cu and freight on cement copper is $14/ton (.1¢/lb of contained Cu. (280 miles @ 5¢ ?)
13. Agitation leaching gives 80% efficiency.
14. Percolation leaching (operating or crushed ore minus ¼ inch, plus 50 mesh B.S.S.) uses 80% of ore and this is 80% efficient, i.e. overall efficiency 64%.
15. Wages for operators are $5,000 per year per man.
16. Scrap iron is 2.5lb Fe/1lb Cu and costs 2¢/lb. Fe.

17. Acid consumption for both agitation leaching and percolation leaching is 200% theoretical and acid costs $40/ton $H_2SO_4$.

## 6. Discussion and Conclusion

For a plant of 10,000 t/year operating over 3 years ore of at least 4% grade is required. For 20,000 t/year over 3 years at least 3% grade is needed. The minimum economic plant size approximately halves if the price of copper rises by 20%. The effect of increasing plant life to 4 years has about half this effect.

Percolation leaching is marginally more attractive than agitation leaching.

Table 1. Operating Costs.

| | Percolation<br>$ | Agistation<br>$ |
|---|---|---|
| Capital Depreciation (over 3 years) | 47,000 | 78,000 |
| Acid at $40/ton $H_2SO_4$ consumption 200% theoretical | 16,000 | 20,000 |
| Labour 4 men at $5000 per year | 20,000 | 20,000 |
| Scrap Fe 5¢/lb Cu. app 2.5 lb Fe/lb. Cu | 13,000 | 16,000 |
| Mining $2 ton | 20,000 | 20,000 |
| Water (50¢/1000 gallons) | 1,000 | 4,000 |
| Power (2¢/KWH) | 2,000 | 4,000 |
| Minor maintenance and overheads | 10,000 | 10,000 |
| Total operating costs | 129,000 | 172,000 |
| Product value at 33¢/lb. | 85,000 | 106,000 |
| Loss | 44,000 | 76,000 |

Table 2. 3% Copper Ore

| | Percolation<br>$ | Agitation<br>$ |
|---|---|---|
| Capital Cost | 56,000 | 71,000 |
| Installed Capital Cost | 140,000 | 178,000 |

## Titles of Reports sighted on visit to S.A. Dept. Mines October, 1972.

### AMDEL REPORT NO. 459. CUSTOM COPPER LEACHING PLANT.

by A.A. Hejja
December, 1965

Hydrometallurgy of Copper Sulphide
Electrowinning of Copper
Economics of Hydrometallurgical Recovery of Copper from Small deposits of Oxidized Ore.
Economics of Copper Metallurgy - R.F. Wilmhurst
Evaluation of Kapunda Ore.
1st July, 1971.
Progress Report No. 4.
Visits overseas.

Application of the Gycerol Process to Mineral Recovery -

R.N. DUDGEON
23.11.1972.

MLH

MEMO TO : MR. K.H. STACEY

FROM : M.L. HOUSTON

## TREATMENT OF LOW GRADE ORE IN FLOOR OF PIT

The following personnel from the S.A. Department of Mines visited Mount Gunson on Wednesday 19th November for the purpose of discussing what might be done with the low grade ore remaining in the floor of the pit after mining of "commercial" ore.

| | |
|---|---|
| Mr Jack Minogue | Chief Inspector of Mines |
| Mr Bob Adam | Mining Engineer |
| Mr Bob Wilding | Scientific Officer (trained in geology and metallurgy) |
| Mr Bruce Robinson | Geologist |

and Mr John Lackey of AMDEL

The Department of Mines, while appreciating our motives for leaving the low-grade 'ore' behind and burying it in the backfill operation, is concerned about the political implications of permitting $12 m worth of copper to be left in the ground in a situation where no one could economically recover it. The reason for their visit was to establish what alternative methods were available for recovering the copper and to investigate the economics of these alternatives.

The first point to be established was that we are unable to precisely define the tonnage and grade of material which falls into the category of "low-grade ore".

My original estimates were done in March 1973 and were based on the results of 20 diamond drill holes. I simply took into account the material that lay between the 1.00% and 0.50% Cu cut-off. On this basis, the reserve of low-grade material was 2.5m tonnes at 0.58% Cu.

It was decided that, to fully understand the problem, it would be necessary to recalculate the tonnage and grade and, if possible, to classify the reserve into tonnages of diminishing grades with appropriate grade and thickness distributions.

For example a tabulation of the data as depicted hereunder would be useful for describing the dimensions of the problem.

| Cut-off Interval | Tonnage | Grade & Limits | Mean Thickness |
|---|---|---|---|
| 0.99% - 0.70% | a | x ± x' | u |
| 0.69% - 0.40% | b | y ± y' | v |
| 0.39% - 0.10% | c | z ± z' | w |

It was conceeded that to carry out these calculations on site by manual methods would create an unreasonable workload for our staff. Mr. Robinson agreed to discuss with AMDEL the cost and feasibility of performing the calculations by computer using existing AMDEL programmes. To assist him in his enquiries Mr. Robinson was given copies of 3 sample drill logs and a plan showing the location and areas of influence of diamond drill holes.

From this starting point, discussion developed around the possible alternatives for treating the material in question. I nominated these to be:

1. In situ leaching of the ore after fracturing and backfilling.

   This would be the cheapest alternative to implement but the most difficult to control.

2. Mining of the material, carting to the surface and dump leaching.

   This would have comparatively favourable economics and would have a reasonably practical chance of success. However a large stockpile would eventually have to be disposed of.

3. Mining of the material and leaching in dumps on the floor of the pit.

   This would be cheaper than Alternative 2, would have the same control characteristics and would avoid the disposal problem. However an inordinate amount of pit floor-space would be required and this would inhibit backfilling operations.

..3

4. Mining of the material, carting to the surface and stockpiling for future use, either as a blend with high grade feed or as a feed for separate low grade milling operation at the end of the life of the current operation.

   This would involve a considerable capital investment in the stockpile (of the order of $2m) and suffers from the disadvantage that much of the ore would oxidize, with consequently poor recoveries during future flotation operations.

5. Expansion of the mill by 50% to treat increased tonnages of lower grade material and drop the cut-off grade to, say, 0.5% Cu.

   This suffers from the obvious disadvantages of high capital costs, only marginal improvements in copper production, due to lower recoveries with low grade ore, and increased costs per tonne of copper produced.

6. Take all waste to external dumps, leaving the pit floor exposed for future remnant mining operations, should the appropriate economic circumstances arise.

   This would pose severe environmental problems associated with the creation of a 50m tonne waste dump and would increase the cost of the waste removal operation by about 50%.

Additional to the disadvantages listed above, Alternatives 1, 2, 3 and 5 would require substantial increases in the supply of water to the operation, which could prove to be very expensive.

The Department of Mines, in conjunction with AMDEL, will undertake the investigation of these alternatives, at no cost to us. I have agreed to supply them with any information they require, either in terms of physical performance or operating costs.

The only work which we will undertake is to estimate the incremental costs associated with Alternative 6. I have already asked Mr. Collins to prepare these estimates.

M.L.H.
20/11/75

c.c. G.T.T.
G.L.
D.B.T.
J.L.C.
W.A.B.

file leaching

000421

MEMO TO : MR. D.C. McLAUGHLAN

FROM : M.L. HOUSTON

## COMMENTS ON AMDEL REPORT ON LEACHING OF LOW GRADE ORE

This report has been studied and the following comments are made:

Case 1 Heap Leaching.

1. Quantity of Copper Produced

The quantity of copper produced per year would be close to that predicted in the report, but arrived at as follows

Most likely: $200,000 \times \frac{0.58}{100} \times 0.54 = 626$ tonnes

(ore x grade x recovery)

Report: $150,000 \times \frac{0.64}{100} \times 0.65 = 624$ tonnes

2. Circulating & Make up Water

The figures for quantity of circulating and make-up water look reasonable.

3. Operating Costs

Mining Most likely: 200,000 x 0.695 = $139,100

(tonnes ore x unit cost)

Report: 150,000 x 0.55 = $82,000

Transport of Waste

Most likely: $1.76 \times 10^6 \times 0.033 =$ \$57,300

(tonnes waste x unit cost)

Report: $0.30 \times 10^6 \times 0.15 =$ \$45,000

Direct Labour

Most likely: 5 x 18,000 = $90,000

(No men x annual cost)

Report: 5 x 9,000 = $45,000

Cost differential = (286,400 - 172,000) = $114,400

All other costs being equal, a copper price of $A1,500/tonne would be required to generate a profit on this investment.

Case 2 In-Situ Leaching

1. <u>Quantity of Copper Produced</u>

   Again the ore tonnage will be higher, the grade lower and the recovery lower than that predicted in the report. The most likely estimate of copper production is:

   $$200,000 \times \frac{0.58}{100} \times 0.40 = 464 \text{ tonnes.}$$

2. <u>Operating Costs</u>

   Ore preparation - unit cost is probably OK but a higher tonnage will have to be prepared, ie. annual cost is more likely to be 200,000 x 0.35 = $70,000, not $52,000.

   Direct Labour - cost per manyear will be more like $18,000 so direct labour costs will be $90,000 not $45,000.

   Cost differential = (160,000 - 97,000)
   = $63,000/year.

Again, all other costs being equal, a zero profit Cu price of $A1,500/tonne would be required.

<u>General Comments</u>

The overall systems described and mechanics of operation are similar to a proposal I prepared for Mr. Van Herk in March 1973. However it was agreed then that if it were possible to provide all the additional water required for the leaching operation, more favourable economics would be provided by expanding the Mill and treating the lower grade ore in the Mill. The additional revenues from higher copper recoveries would more than offset the higher capital investment required.

M.L.H.
22/3/76
c.c. J.L.C.
G.L.

DM 750/75
Env 2657
000423

amdel
The Australian Mineral Development Laboratories

Flemington Street, Frewville, South Australia 5063
Phone Adelaide 79 1662, telex AA82520

Please address all correspondence to Frewville.
In reply quote: CM 1/31/26/0

3 February 1976

The Director,
South Australian Department of Mines,
139 Greenhill Road,
PARKSIDE SA 5063
Attention: Mr J.R. Adam

## REPORT: CM 1605/76

| | |
|---|---|
| YOUR REFERENCE: | DM 750/75 RLW:JS |
| MATERIAL: | Remnant Ore |
| LOCALITY: | Mt Gunson |
| DATE RECEIVED: | 5 December 1975 |
| WORK REQUIRED: | Economic Analysis of heap and in situ leaching |

Investigation and Report by: J.A. Lackey

Officer in Charge, Chemical Metallurgy Section: J.E.A. Gooden

J Gooden
for F.R. Hartley
Director

# 1. INTRODUCTION

A visit was made to Mt Gunson on 19 November 1975 by Messrs J. Minogue, J.R. Adam, R.L. Wildy and W.B. Robinson of SADM, and Dr J.A. Lackey of Amdel, to examine the remnant ore and collect information regarding its possible treatment. Estimates indicate that there is about 2 million tonnes of this low-grade material averaging about 0.64% Cu. This problem requires urgent consideration as Pacminex Pty Ltd has embarked on a programme of covering this low-grade ore with overburden.

Discussions held with Pacminex staff at Mt Gunson resulted in four possible alternatives being considered (in addition to Pacminex's present scheme):

(1) Mine as a total orebody with a cut-off grade of 0.5% Cu (instead of the present 1.0%).

(2) Mine the low-grade material as a separate exercise and treat in the mill following mining out of the 'economic orebody'.

(3) Heap, or in situ leaching treatment of the remnant ore.

(4) Dump the overburden outside the pit leaving the low-grade ore readily exposed for treatment at a later date.

There are a number of economic and environmental constraints that will make some of the above alternatives infeasible. For this reason, it is desirable to make a preliminary economic study before embarking on a detailed investigation of any of the above programmes.

Thus it was decided that Amdel should carry out a first-order, preliminary costing of the feasibility of leaching of this remnant ore, considering the following alternatives:

(1) Heap leaching of the ore inside the open pit;

(2) Heap leaching of the ore outside the open pit;

(3) In situ leaching of the broken ore with overburden on top.

Capital and operating costs have been estimated for the above cases taking an optimistic point of view (i.e., relatively high copper recoveries, low acid and scrap iron consumptions). In addition, the effect of the price of copper on the overall economics has been estimated.

# 2. HEAP LEACHING

## 2.1 Process Description

It is assumed that about 150,000 tonnes of remnant ore is being uncovered annually. This is to be mined and carted to an area of the pit floor where the bottom is well consolidated and of low permeability. Because this low-grade ore is being treated in the open pit area, the overburden that would have been placed in this area had the remnant ore remained in situ, will have to be transported to a higher area. This overburden will be moved to the periphery of the open pit or completely outside of it. The assumption is made that 300 000 t/a of overburden will have to be moved to the higher location.

Five heaps are to be constructed annually, each of 30 000 tonnes. Three of these heaps are to be leached simultaneously with the other two either drying out or new heaps being constructed. Once the area of the pit floor has been covered with heaps, new heaps will be constructed on top of the old heaps.

The heaps are to be about 6 m high x 30 m wide x 100 m long with a total surface area at the top of about 2200 $m^2$. With three heaps being operated simultaneously, there will be a total solution flow of 49 kl/h (7.5 $l/h/m^2$ of area). It is assumed that there is a 20% solution loss (by evaporation, soakage, etc.) and a borefield to provide this 10 kl/h of make-up has to be developed (as all available water resources are being used at present).

It is assumed that 65% of the 0.64% contained copper (i.e., 0.416% of 150 000 t/y or 624 tonnes Cu/y) is recovered from the heaps. Copper is recovered from the pregnant liquor by cementation on scrap iron in launders. This produces a 75% Cu cement copper product that is air dried before shipment to the smelter at Port Kembla.

## 2.2 Operating Details

The study was based on the following assumptions:

(1) Production rate of remnant ore 150 000 t/a of grade 0.64% Cu.

(2) Liquor circulation operated 24 h/d, 5d/wk for 45 wks/annum.

(3) Cementation launders operated 8 h/d, 5 d/wk for 45 wks/annum.

(4) Overall copper recovery - 65%.

(5) Acid consumption (from Pacminex heap leaching tests ) 2 kg $H_2SO_4$/kg Cu.

(6) Iron consumption in cementation launders 1.5 kg Fe/kg Cu.

(7) Labour requirements: heap placement, liquor distribution system: 2 men; cementation: 2 men; drying and packaging: 1 man; Total direct labour of 5 men.

## 2.3 Capital Costs

Capital cost details are given in Appendix A. A summary of the items is given below:

| | | Installed Cost, $ |
|---|---|---|
| (1) | Cementation | 70,000 |
| (2) | Pumps | 49,000 |
| (3) | Liquor holding/tanks | 50,000 |
| (4) | Fork lift | 30,000 |
| (5) | Concrete drying pad | 2,000 |
| (6) | Borefield | 200,000 |
| | | 401,600 |

Say, $400,000

000428

## 2.4 Operating Costs

Appendix B gives the operating cost details for mining, process materials, utilities and direct labour. Production costs are summarised below:

| | Item | Annual Cost, $ |
|---|---|---|
| 1. | Mining | 82,000 |
| 2. | Preparation of Heaps | 45,000 |
| 3. | Transportation of Overburden | 45,000 |
| 4. | Process Materials | 150,000 |
| 5. | Utilities | 26,000 |
| 6. | Direct Labour | 45,000 |
| 7. | Maintenance, 6% F.C. | 24,000 |
| 8. | Supervision, 20% of (6) | 9,000 |
| 9. | Operating Supplies, 10% of (6) | 5,000 |
| 10. | Direct Manufacturing Cost (Items 1 to 9) | 431,000 |
| 11. | Payroll overhead 20% of (6) | 9,000 |
| 12. | Plant overhead 100% of (6) | 45,000 |
| 13. | Process Control 20% of (6) | 9,000 |
| 14. | Packaging 3% of (10) | 13,000 |
| 15. | Indirect Manufacturing Cost (Items 11 to 14) | 76,000 |
| 16. | Depreciation 10% F.C. | 40,000 |
| 17. | Property taxes, rent and insurance 1% F.C. | 4,000 |
| 18. | Fixed Manufacturing cost (Items 16 and 17) | 44,000 |
| 19. | Manufacturing Cost (Items 10, 15 and 18) | 551,000 |
| 20. | Administrative Expenses 3% of (19) | 17,000 |
| 21. | Distribution and marketing expenses 3% of (19) | 17,000 |
| 22. | Non-manufacturing Cost (Items 20 and 21) | 34,000 |
| 23. | Total Annual Production Cost (Items 19 and 22) | 585,000 |

## 2.5 Economid Analysis

The on-site values of the 75% Cu cement copper product are calculated in Appendix C. Summarised below is an economic analysis of the proposed operation:

| Copper price $ | 900 | 1000 | 1200 | 1500 |
|---|---|---|---|---|
| Value of cement copper on site, $000's | 400 | 459 | 577 | 754 |
| Total Production Cost $000's | 585 | 585 | 585 | 585 |
| Surplus/Deficit | -185 | -126 | -8 | 169 |
| Return on Investment before tax, % | - | - | - | 42% |

From the above analysis, the Australian copper price would have to be about $1200/tonne to break even (no return on investment) and $1400/tonne for the project to give a return of 30%.

However, the present Australian price of copper is about $940/tonne and this would result in an annual cash deficit of $160,000.

Note – time value of costs – probably no significance [illegible] [illegible].
from [illegible]

An alternative to heap leaching in the open pit area would be heap leaching outside the pit area. This would save the incremental cost of transporting overburden ($45,000/a) but would result in additional expenditure for transportation of the remnant ore ($1.00/tonne instead of 55¢/tonne - i.e., an additional $68,000/a), base preparation for the heaps (approximately $22,000/a) and the cost of rehabilitation. Thus, this alternative would be at least $45,000/a more expensive with no advantages over heap leaching in the pit area.

In conclusion, while it is very difficult to forecast future copper prices, in the short term, at least, copper prices are likely to remain at a level which makes this project uneconomic.

## 3. IN SITU LEACHING

### 3.1 Process Description

As with heap leaching, it is assumed that about 150,000 tonnes of remnant ore is being uncovered annually. This ore is to be drilled and blasted, and broken sufficiently to allow solution to pass through the rocks. It is proposed that overburden will be dumped on the broken ore so that the operation will not result in overburden being carted out of the open pit area. However, a solution distribution system will have to be laid in place through this overburden material.

Because of the small depth of the remnant, it will be necessary to circulate liquor through it at a greater rate to ensure good liquor distribution. Liquor circulation rate is assumed to be 4 times that of the heap leaching operation, i.e., 200 kl/h. With the liquor being pumped below the surface, solution loss by evaporation will be much lower (assumed to be 5% or 10 kl/h). As with heap leaching, this make-up water will have to be obtained by developing a new borefield. Most of the liquor will be recirculated before pumping to the cementation launders.

It is assumed that 50% of the 0.64% contained copper (i.e., 0.32% of 150,000 t/a or 480 tonnes Cu/a) is recovered from the operation. As with heap leaching, copper is to be recovered from the pregnant liquor by cementation on scrap iron in launders. This produces a 75% Cu cement copper product that is air dried before shipment to the smelter at Port Kembla.

### 3.2 Operating Details

The study was based on the following assumptions:

(1) Production rate of remnant ore 150,000 t/a of grade 0.64% Cu.

(2) Liquor circulation operated 24 h/d, 5 d/wk for 45 wks/annum.

(3) Cementation launders operated 8 h/d, 5 d/wk for 45 wks/annum.

(4) Overall copper recovery 50%

(5) Acid concumption 2 kg $H_2SO_4$/kg Cu

(6) Iron consumption in cementation launders 1.5 kg Fe/kg Cu.

(7) Labour requirements: Liquor distribution system: 2 men; cementation 2 men; drying and packaging: 1 man. Total direct labour of 5 men.

### 3.3 Capital Costs

Capital cost details are given in Appendix D. A summary of the items is given below:

| | | Installed Cost, $ |
|---|---|---|
| (1) | Cementation | 70,000 |
| (2) | Pumps | 46,400 |
| (3) | Liquor holding/tanks | 50,000 |
| (4) | Fork lift | 30,000 |
| (5) | Concrete drying pad | 2,000 |
| (6) | Borefield | 200,000 |
| | | 398,400 |

Say, $400,000

### 3.4 Operating Cost

Appendix E gives the operating cost details for ore preparation, process materials, utilities and direct labour. Production costs are summarised below:

| | Item | Annual Cost, $ |
|---|---|---|
| 1. | Ore preparation | 52,000 |
| 2. | Preparation of liquor distribution system | 45,000 |
| 3. | Process materials | 116,000 |
| 4. | Utilities | 38,000 |
| 5. | Direct labour | 45,000 |
| 6. | Maintenance, 6% F.C. | 24,000 |
| 7. | Supervision, 20% of (5) | 9,000 |
| 8. | Operating supplies, 10% of (5) | 5,000 |
| 9. | Direct Manufacturing Cost (Items 1 to 8) | 334,000 |
| 10. | Payroll overhead, 20% of (5) | 9,000 |
| 11. | Plant overhead, 100% of (5) | 45,000 |
| 12. | Process control, 20% of (5) | 9,000 |
| 13. | Packaging, 3% of (9) | 10,000 |
| 14. | Indirect manufacturing cost (Items 10 to 13) | 73,000 |
| 15. | Depreciation, 10% F.C. | 40,000 |
| 16. | Property taxes, rent, insurance 1% F.C. | 4,000 |
| 17. | Fixed manufacturing cost (Items 15 and 16) | 44,000 |
| 18. | Manufacturing cost (Items 9, 14 and 17) | 451,000 |
| 19. | Administrative expenses 3% of (18) | 14,000 |
| 20. | Distribution and marketing expenses 3% of (18) | 14,000 |
| 21. | Non-manufacturing Cost | 28,000 |
| 22. | Total annual production cost (Items 18 and 21) | 479,000 |

### 3.5 Economic Analysis

The on-site values of the 75% Cu cement copper product are calculated in Appendix C. Summarised below is an economic analysis of the proposed operation:

| Copper price $ | 900 | 1000 | 1200 | 1500 |
|---|---|---|---|---|
| Value of cement copper on site, $000's | 308 | 353 | 444 | 580 |
| Total Production Cost $000's | 479 | 479 | 479 | 479 |
| Surplus/Deficit, $000's | -171 | -126 | -35 | 101 |
| Return on investment before tax, % | - | - | - | 25% |

From the above analysis, the Australian copper price would have to be about $1300/tonne to break even (no return on investment) and $1600/tonne for the project to give a return of 30%.

However, the present Australian price of copper is about $940/tonne and this would result in an annual cash deficit of $155,000.

In conclusion, solution mining appears to be less economically viable than a heap leaching operation. However, either operation would result in a substantial cash deficit with present low copper prices.

## 4. DISCUSSIONS AND CONCLUSIONS

From the economic analysis presented in this report, neither heap leaching nor in situ leaching would be an economically viable treatment method for the remnant ore at Mt Gunson with the present low copper prices.

At present copper prices, both heap leaching in the open pit area and in situ leaching would result in an annual cost deficit of about $155,000 - $160,000/a. However, if copper prices were to rise, the heap leaching operation would break even at about $1200/t and make a profit of 30% (before tax) at about $1400/t, whereas the in situ leaching operation would break even at about $1300/t and make a profit of 30% (before tax) at about $1600/t.

## 5. RECOMMENDATIONS

With the present low price of copper, leaching cannot be recommended as a viable method of recovering copper from the remnant ore.

However, in the longer term, if copper prices rise sufficiently, then it would be recommended that a further economic study be made of heap and in situ leaching as possible methods of treating the Mt Gunson remnant ore.

## APPENDIX A

## EQUIPMENT SIZING AND CAPITAL COSTS — HEAP LEACHING OPERATION

1. Preparation of heaps. The preparation of heaps including purchase and laying of PVC pipes, valves etc., for liquor distribution, is taken as an operating cost.

2. Cementation launders. Operating one shift only.

Production rate of liquor = 50 kl/h x 24
= 1200 kl/day

Liquor treatment in launders = 1200 kl/day

$$= \frac{1200}{8} \text{ kl/h}$$

= 150 kl/h

For a retention time of 2 h in the launders, a volume of 300 $m^3$ is required. Use 15 launders each of 20 $m^2$ capacity. With 1.2 m (operating height 1 m) x 1 m sections, the total wall area = 1056 $m^2$ (including ends).

Costed at \$65/$m^2$ (installed), total \$70,000

3. Pumps.

(1) Liquor transfer in percolation heaps with 3 heaps to be operated simultaneously, i.e., 3 x 16 kl/h. Pumps, say, Warman 3/2, have 3 pumping to heaps, 3 out of heaps and 2 spare i.e., 8 @ \$1000 ea. \$8000

(2) Cementation. Flow = 150 kl/h
Say, 2 x 4/3 Warman @ \$1200 ea. \$2400

(3) Barren liquor - recycle. Flow = 50 kl/h
Say, 2 x 3/2 Warman @ \$1000 ea. \$2000

Total (purchased) \$12400

4. Liquor holding/tanks

(1) Lined dams.
Liquor treated per day = 1200 kl
Assume a storage capacity of 2500 kl in each of 2 dams (barren or recycle liquor storage and pregnant liquor storage).

a. Excavation of 500 $m^3$ material for construction of dam walls @ 60¢/$m^3$
= \$300

b. Lining with polythene for 1200 $m^2$ (2-2.5 m deep) at 3.50/$m^2$
= \$4200

c. Replacement of layer of earth on top of lining
= \$500

Total \$5000 ea. (installed) \$10,000.

(2) Acid storage.
100-tonne acid tank (purchased) $5000

(3) Solution mixing tank.
5 kl capacity. Rubber-lined with stainless steel agitator (purchased)
$5000

5. Fork lift for cementation operation $30000
(equivalent to installed)

6. Concrete pad for drying of cement copper (installed)
$2000

7. Borefield. At present, all known water resources are being fully utilised and to supply the 10 kl/h make-up required for the heap leaching operation, new underground water aquifers will have to be located and developed. To supply the 20 kl/h borefield already developed cost approximately $250,000 so one can expect that finding and supplying of about ½ this size might cost as much as $200,000 (in 1976 costs).

Items (3) 4(2) and 4(3) are purchased item costs and have to be converted to installed costs. Multiplication of these purchased item costs by four is the usual procedure for converting them to installed costs, and allows for installation, instrumentation, piping, electrical, buildings (where required), foundations and structures, land, yard improvements and provision of utilities.

In summary:

| | Installed Cost, $ |
|---|---|
| Cementation | 70,000 |
| Pumps | 49,600 |
| Liquor holding/tanks | 50,000 |
| Fork lift | 30,000 |
| Concrete drying pad | 2,000 |
| Borefield | 200,000 |
| Total: | 401,600 |

say, $400,000

## APPENDIX B

## OPERATING COSTS - HEAP LEACHING

Mining. Cost of mining the low-grade ore and transportation to the site of the heap leaching operation is 55¢/tonne. That is, 150,000 x 0.55 = $82,000.

Preparation of Heaps. It is assumed that preparation of the heaps and purchase cost and laying of PVC pipework will amount to 30¢/tonne ore. That is, 150,000 x 0.30 = $45,000.

Transportation of Overburden. Transportation of overburden within the pit should cost about 30¢/tonne whereas an additional 15¢/tonne will be required to remove this material out of the pit area. That is, 300,000 x 0.15 = $45,000.

Process Materials. Acid - on-site cost of $60/tonne. Acid consumption is 2 kg/kg Cu extracted, i.e., 2 x 624 x 60 = $75,000.
Scrap Iron - on-site cost of $80/tonne. Scrap iron consumption is 1.5 kg/kg Cu i.e., 1.5 x 624 x 80 = $75,000.

Utilities. Water consumption of 10 kl/h for 5400 h/annum at 23¢/kl; i.e., 10 x 5400 x 0.23 = $12,000.
Electricity - Installed capacity approx. 65 kw; 5400 h/annum at 4¢/kwh; i.e., 65 x 5400 x 0.04 = $14,000.

Direct Labour. Operators = 5 at $9000/annum; i.e., 5 x 9000 = $45,000.

## APPENDIX C

## PRODUCT VALUES

For the sale of a 75% Cu cement product transported to Port Kembla, the following payments and charges apply (assuming dry concentrate and no loss or deductions due to fineness):

| | | | | | |
|---|---|---|---|---|---|
| 1. | Australian Copper Price per tonne Cu | $900 | $1000 | $1200 | $1500 |
| 2. | Payment per tonne of copper cement for contained Cu, less 1.3 (tonne % units) at 96% of Aust. Cu price | 637 | 707 | 849 | 1061 |
| 3. | Deduct $15/tonne material (sampling and smelting charges) | -15 | -15 | -15 | -15 |
| | | 622 | 692 | 834 | 1046 |
| 4. | Deduct $160/tonne Cu paid for (0.737 x 160 = $118) (Refining and Realisation charges) | -118 | -118 | -118 | -118 |
| 5. | Value of 1 tonne material | 504 | 574 | 716 | 928 |
| 6. | Value of 1 tonne contained Cu at smelter (Price/0.75) | 672 | 766 | 955 | 1238 |
| 7. | Deduct $30/tonne contained Cu for transport from Mt Gunson to Port Kembla | -30 | -30 | -30 | -30 |
| 8. | On-site value per tonne contained Cu. | 642 | 736 | 925 | 1208 |

## APPENDIX D

## EQUIPMENT SIZING AND CAPITAL COSTS — SOLUTION MINING OPERATION

1. Preparation of Area to be Treated by Solution Mining.

The preparation of this area, including blasting or breaking, solution distribution circuit etc., has been taken as an operating cost.

2. Cementation.

As for heap leaching: (installed cost) $70,000

3. Pumps.

| | | | |
|---|---|---|---|
| (a) | Liquor 200 kl/h capacity<br>say, 6, 4/3 Warmans;<br>i.e., 6 x $1200 | 7,200 | |
| (b) | Cementation Flow = 150 kl/h<br>say, 2 x 4/3 Warman;<br>i.e., 2 x $1200 | 2,400 | |
| (c) | Barren liquor recycle<br>Flow - 50 kl/h<br>Say, 2 x 3/2 Warman at $1000 ea.<br>i.e., | 2,000 | |
| | Total (purchased) | | $11,600 |

4. Liquor Holding/Tanks.

Capacities as for heap leaching,
i.e., (installed cost) 50,000

5. Fork lift.

as for heap leaching,
i.e., (installed cost) 30,000

6. Concrete pad.

For drying of cement copper, as for heap leaching;
i.e., installed 2,000

7. Borefield.

As for heap leaching,
i.e., installed 200,000

The pumps are purchased item costs and have to be converted to installed costs. Multiplication of these purchased items by four will convert them to installed costs.

In summary:

| Item | Installed Cost, $ |
|---|---|
| 1. Cementation | 70,000 |
| 2. Pumps | 46,400 |
| 3. Liquor holding/tanks | 50,000 |
| 4. Fork lift | 30,000 |
| 5. Concrete drying pad | 2,000 |
| 6. Borefield | 200,000 |
| | 398,400 |

Say, $400,000

## APPENDIX E

## OPERATING COSTS - IN SITU LEACHING

Ore preparation. Cost of drilling and blasting to break the ore for in situ leaching is 35¢/tonne; i.e., 150,000 x 0.35 = $52,000. 70,000

Preparation of liquor distribution system. It is assumed that the cost and laying of PVC pipework and maintenance of the liquor distribution system will cost ∿30¢/tonne ore. That is: 150,000 x 0.30 = $45,000.

Process materials. Acid - on-site cost of $60/tonne.
Acid consumption os 2 kg/kg Cu extracted, i.e., 2 x 480 x 60 = $58,000
Scrap Iron - on-site cost of $80/tonne. Consumption is 1.5 kg/kg Cu; i.e., 1.5 x 480 x 80 = $58,000.

Utilities. Water - consumption of 10 kl/h for 5400 h/annum at 23¢/kl; i.e., 10 x 5400 x 0.23 = $12,000
Electricity - Installed capacity approx. 120 kw, 5400 h/annum at 4¢/kwh; i.e., 120 x 5400 x 0.04 = $26,000.

Direct labour. Operations = 5 at $9000/annum; i.e., 5 x 9000 = $45,000.

## DIRECT LEACHING OF CONCENTRATE

## SUMMARY OF LABORATORY TEST WORK TO 10.4.1978

### INTRODUCTION

This ~~request~~ report attempts to summarise the main findings of laboratory testwork on the direct leaching of concentrate carried out from 10th January to 10th April, 1978.

Aspects of chloride and sulphate leaching, recovery of copper from solution, and residue treatment have been investigated. The likely costs of the processes described, and the marketability or otherwise of the products formed have not been considered.

The behaviour of copper metal and cobalt is emphasised in the sections that follow. Until continuous leaching etc. is carried out it is not possible to ascertain the behaviour of metals such as Pb, Ag, Bi etc.

### SECTION I

### LEACHING OF CONCENTRATE

It has been established that both ferric chloride and ferric sulphate are suitable leachants for unground Gunson concentrate. Under fairly mild conditions over 90% of the copper is dissolved, with only 5-10% of the cobalt. A typical set of leach conditions which yield these dissolution levels is set out in Table I. Although the leaching conditions have not been fully optimised, it can be seen that only mild temperatures and leach-times are required, and that leaching can be performed in open-vessels. The conditions set out in Table I represent only one set of satisfactory leach conditions. Similar results, can be achieved in only 3-4 hours if the leach temperature is raised to 85°C.

TABLE I

TYPICAL LEACH CONDITIONS TO PRODUCE >90% Cu DISSOLUTION & < 10% Co DISSOLUTION

| LEACH CONDITIONS (TYPICAL) | LEACHANT | |
|---|---|---|
| | FERRIC CHLORIDE | FERRIC SULPHATE |
| Temperature (°C) | 50-60 | 50-60 |
| Time (hr) | 36-48 | 36-48 |
| Pressure | Open Vessel | Open Vessel |
| Fe:Cu Ratio | 2.7:1 | - *1 |
| Impeller Speed (rpm) *2 | 300 | 300 |
| Solution Conc. (%) | 30 | 35 |
| Condition of Feed | Moist Unground | Moist Unground |

NOTES *1 : Ratio not fully determined, but likely to be approximately 4:1 or greater.

*2 : Impeller speeds have been sufficient to keep solids in suspension. No detailed testing performed.

For both the chloride and sulphate systems the dissolution of the elements follows a similar pattern with the exception of lead (- relatively insoluble in a sulphate leachant). Detailed discussion of the rates and levels of dissolution of the elements Zn, Bi, Pb, Ag, Fe etc. is not included. (Batch-testing using fresh leachant does not allow realistic assumptions to be made as to how these elements will behave when a recycled leach liquor is used).

There is no reason, at this stage of testing, to exclude either the chloride or sulphate leachant from on-going leaching testwork. Some comment about the comparative suitability of the leachants to solvent extraction and regeneration is made in subsequent sections.

The residue produced from the two systems is likely to be of different composition, and test-work to date has only been carried out on the chloride residue (See Section IV). Final choice of leachant will thus depend on residue characteristics, and amenability to filtration and further processing.

## SECTION II

## RECOVERY OF COPPER FROM SOLUTION

### (i) Direct Cementation

Direct cementation of copper from the leach solution of either system is feasible (: some pH control may be necessary in order to minimise iron consumption). It is clear that an impure cement copper product will result from direct cementation. A typical analysis is shown in Table 2.

TABLE 2

ANALYSIS OF CEMENT COPPER PRODUCED DIRECTLY FROM CHLORIDE LEACH LIQUOR

| SAMPLE | Cu % | Bi ppm | Ag ppm | Pb % | Zn % | Co % | Ni ppm | Fe % |
|---|---|---|---|---|---|---|---|---|
| Cℓ Cement Copper | 90-95 | 1200 | 400 | 0.12 | 0.06 | 0 | 400 | 5-10 |

Iron consumption in the few tests carried out to date has been low (approx. 1.2 kg Fe/kg Cu), although no tests have been carried out using sulphate liquor.

The suitability of the cement copper as feed for further treatment processes has not been determined. If direct cementation were to be utilised then the dissolution of minor metals would be similar to that obtained using 'fresh' leachant. Iron removal during or prior to regeneration of the leachant would be necessary.

(ii) Solvent Extractions (SX)

Although only very limited laboratory testing has been undertaken it appears that solvent extraction is technically feasible as a step to purifying the leach liquor. Solvent extraction of sulphate liquors is well proven and relatively straight forward. However, little is known on the chloride system. At least one manufacturer of solvents believes that good selectivities can be achieved in the latter system.

It would be improper to report the results of the SX testing carried out at MGM to date. In summary, however, it has been found that:

- SX can be used in the chloride system only if the ferrous : ferric iron ratio is ν high (pH)

- pH control is extremely important for good selectivity

- The sulphate system enables high loading and faster extraction than that available with the chloride liquor.

To provide a comparison between cement copper produced following SX and direct cement copper (chloride liquor) a SX test was performed with extraction pH of 1.4. The final stripping acid (15% $H_2SO_4$) was treated to produce cement copper. Analysis of the product is given in Table 3.

TABLE 3

ANALYSIS OF CEMENT COPPER PRODUCED AFTER SOLVENT EXTRACTION* OF CHLORIDE LEACH LIQUOR

| SAMPLE | Cu % | Bi ppm | Ag ppm | Pb % | Zn % | Co % | Ni ppm | Fe % |
|---|---|---|---|---|---|---|---|---|
| SX Cement Copper | 96 | 60 | 0 | 0.10 | 0 | 0 | 40 | 0.4 |

Note: * SX using AGORCA P-5100 - Results are indicative only.

One important consequence of using SX to purify the resulting 'electrolyte' is that the zinc content of the recycling leach liquor will build up, to a point where zinc removal will be necessary. It is yet to be determined if this is practicable. (If solvent extraction were to be used to remove zinc, then additional plant would be required. The zinc electrolyte could possibly be treated with $Na_2S$ to precipitate zinc sulphide).

## SECTION III

## REGENERATION OF LEACH LIQUOR

It ahs been shown that simple aeration of the ferrous chloride liquor (at elevated temp.) can adequately regenerate the ferric ion. Results have been reported previously.

U.S. researchers have noted that goethite ($Fe_2O_3.x.H_2O$) produced during regeneration (of the chloride liquor) is readily filterable at $\gtrsim$ 80°C. Excess iron is thus eliminated in a convenient form.

Experimentation with the sulphate system is underway.

Other methods of regeneration have not beeen considered due to the simplicity and low relative cost of regeneration by air.

## SECTION IV

## TREATMENT OF LEACH RESIDUE

The chloride leach method produces elemental sulphur as the major constituent of the residue. The sulphur is extremely fine and often bonded to the sulphide particles. The sulphides present in the residue have been indentified as chalcopyrite, sphalerite, carrollite and pyrite. Significant acid insoluble gangue is also present. The amount of minor element present (such as Pb, Ag, Bi etc) will depend naturally on whether continuous leaching is used. Elemental sulphur represents 50% w/w of the residue. This can be readily dissolved in warm perchloroethylene or, reportedly, in cool ammonium sulphide solution.

Flotation of the residue can be utilised to remove insolubles. Typical analyses of the above-mentioned products are given in Table 4.

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## TABLE 4

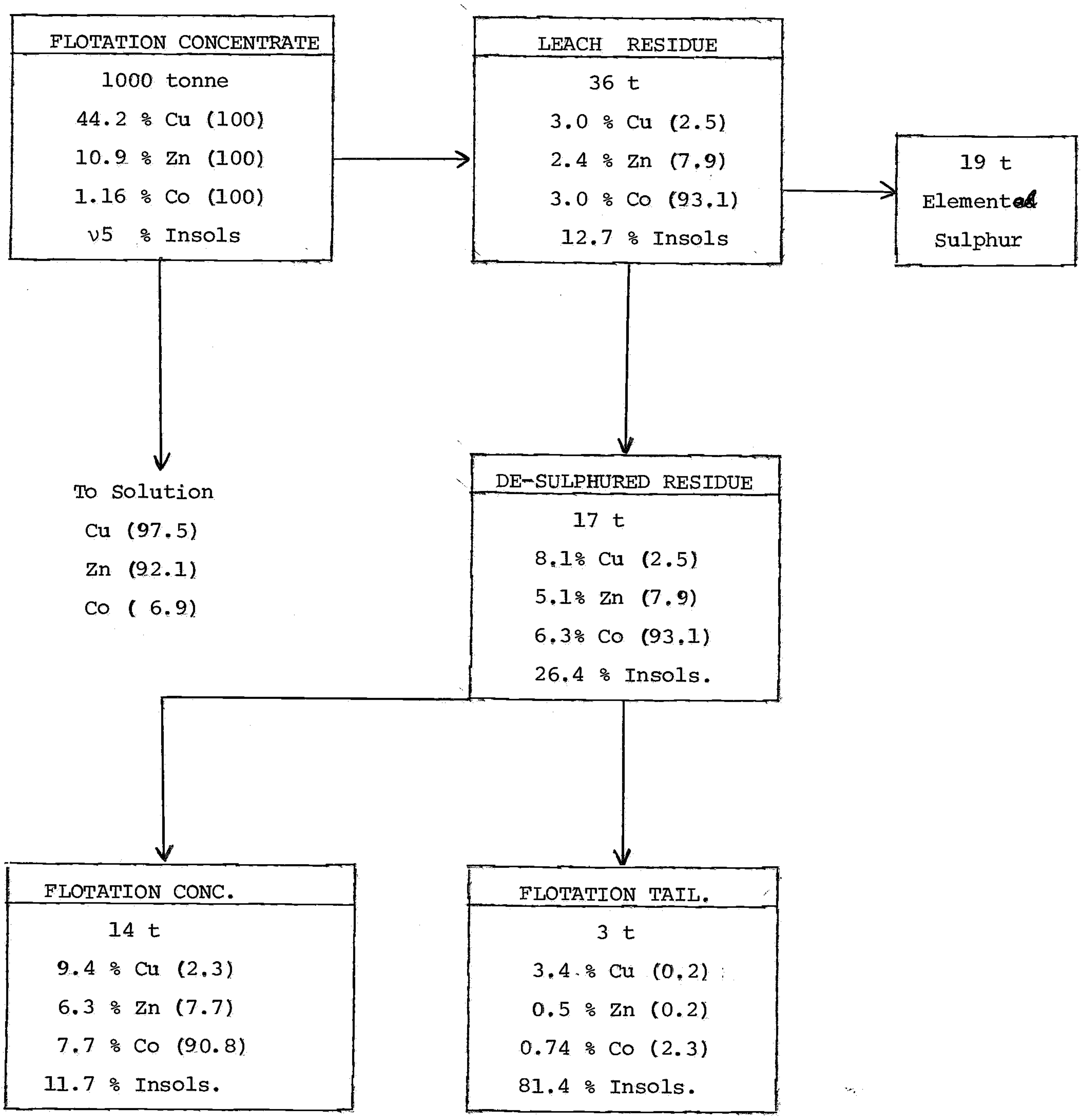
ANALYSES OF VARIOUS RESIDUES

| SAMPLE | Mass % | Cu % | Zn % | Co % | Ni ppm | Fe % | Insols % | $S^o$ % |
|---|---|---|---|---|---|---|---|---|
| Untreated Residue | 100 | 4.1 | 2.5 | 3.15 | 2400 | 7.0 | 13.0 | 50 |
| De-Sulphured Residue | 50 | 8.2 | 5.0 | 6.30 | 4800 | 14.0 | 26.0 | - |
| Flotation 'Conc' | 40 | 9.4 | 6.3 | 7.74 | 6000 | 17.2 | 11.2 | - |
| Flotation 'Tail' | 10 | 3.4 | 0.5 | 0.74 | 650 | 1.7 | 81.4 | - |

An overall mass and element balance for a possible chloride leach and residue treatment process is shown in Fig.1

Treatment of a sulphate leach residue has not yet been attempted.

R.L. STAINLAY
25th April, 1978.



( ) = Distribution - %

Figure 4 : Mass and Elemental Balance for a Chloride Leach & Residue Treatment (Results from Test 50/13/85 7 Mar 78

Pontifex & Associates Pty. Ltd.

~~[illegible]~~ for info

00443

TEL. 332 6744
A.H. 31 3816

26 KENSINGTON ROAD, ROSE PARK
SOUTH AUSTRALIA

P.O. BOX 91, NORWOOD
SOUTH AUSTRALIA 5067

# MINERALOGICAL REPORT NO. 2370

26th May, 1978

TO:

Mr. Ross Stainley,
Mount Gunson Mines Ltd.,
P.O. Box 246
WOOMERA S.A. 5270

YOUR REFERENCE: Your letter dated 3/5/78

MATERIAL: "oxide-ore"

IDENTIFICATION: * "M.O.C. ore - April 78"

WORK REQUESTED: Polished section examination with comments as specified

SAMPLE & SECTION: Retained

PONTIFEX & ASSOCIATES PTY. LTD.

* Main open cut ore:
Sample obtained from old dozer cut (Moranda Cut).
by M. Trevaskis / A.G. Turnbull in March 79.
leaching tests were carried out on ore
similar to this.

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## M.O.C. Sample - April 1978

To determine the relative abundance of translucent and opaque, ore and gangue minerals, a composite sample of this material was examined in thin and polished section. One thousand grains were counted to provide the following list of components and their corresponding mode. These figures refer to number of grains, irrespective of grain size and thus does not precisely represent exact volume %, although most minerals are represented through the grain size variations. The size range is 0.01 to 4 mm in maximum dimension, mean about 0.75 mm.

Comments on composition of the various grain types follow the table below.

| | approx % abundance |
|---|---|
| quartzose grains (including sandstone fragments) minor impurities adhering or as matrix | 72% |
| single green grains of secondary copper of which an estimated 70% is malachite (see note below) and ?chrysocolla | 5% |
| green grains of secondary copper (mainly malachite) intimately intergrown with clays and/or quartz sand material | 7.5% |
| virtually single grains of clay-sericite, (i.e. mica product) some turbid with ultrafine carbonate limonite and rare malachite | 6% |
| clay-sericite (± ultrafine carbonate) intergrown with sandstone (fragments) ± limonite and trace malachite stain | 9% |
| (opaque) chalcocite, intergrown with and partly enveloped by malachite | 1.5% |

malachite >> chrysocolla >>> atacamite
azurite.

..../

Most quartz grains are single crystal, having broken free from the original sandstone rock, and these grains commonly contain relicts of authigenic quartz overgrowths and/or minor adhering clays. Minor composite quartz grains (fragments of sandstone) are present, with siliceous cement-matrix intact, generally with minor clay-sericite, and to a far lesser extent minor green copper minerals through intergranular areas.

The grains designated as clay-sericite, consist of ultrafine ? kaolinitic clays ± more clearly defined admixed flaky sericite. Most of these materials are permeated by limonite-stain and rare malachite stain and some contain turbid, ultrafine carbonate. These grains appear to be altered felspar and/or pelitic-lithic fragments.

The green copper mineral grains appear under binocular microscope and in thin section to consist of malachite and chrysocolla, with trace associated azurite. Other secondary green copper minerals may be present, such as dioptase (hydrous copper silicate) and atacamite (hydrated oxy-chloride of copper), however the lack of distinctive optical properties prevents these minerals from being positively identified. Indeed even in thin section, malachite and chrysocolla are seen to be intimately intergrown, and the similarity in their optical properties makes it difficult to accurately determine their relative abundance. The petrography, and reaction to acid of the green grains mounted in araldite, as viewed under binocular microscope, indicates that about 60% of all green grains is malachite, the remainder of these grains (which are non-reactive to acid) are identified, albeit tentatively, as chrysocolla. Considering the relatively minor amount of other carbonate material, generally admixed with clays (? 3%), it may be possible to determine the approximate abundance of malachite from a chemical analysis of the carbonate radical in this sample.

..../

Malachite commonly, partly surrounds, and occurs through partings within the (opaque) chalcocite grains.

With the exception of small amounts of malachite more or less locked into parts of intergranular matrix within coarser sandstone fragments, there are no inert coatings of the secondary green copper minerals, or of the chalcocite which would prevent access to, and reaction with acid leach solutions.

Copper carbonate and sulphides are the most abundant acid-consuming phases in the sample. The very minor ultrafine carbonate admixed with clays, and to a lesser extent the widespread limonite, may be expected to consume relatively minor amounts of acid leach solutions, (but may be roughly estimated to be of the order of say 10% of total use of acid applied to the sample.)

No genuine copper oxide minerals were positively identified in the thin or polished section. Trace earthy cuprite may accompany some malachite and limonite grains, however in this form and mode of occurrence it is extremely difficult to optically differentiate from earthy limonite.

Rare-trace, minute inclusions of chalcopyrite and bornite occur in some quartz grains.

# LABORATORY REPORT: M.O.C. OXIDE ORE LEACHING

By - R.L. STAINLAY
D.S. BURNS

CONTENTS PAGE

## SUMMARY

A series of tests on various size fraction of M.O.C. oxide ore indicates that maximum copper recovery achievable by acid leaching is controlled by particle size. For a nominal minus 400mm ore sample, maximum oxide copper recoveries of the the order of 70% could be expected. For minus 50mm ore, recovery could be around 90%.

The tests were carried out with the aim of determining maximum copper recovery for each particle size range. The results achievable in a practical heap lead situation would be somewhat lower than those reported.

The sample was predominantly malachite/chryscolla in a quartzite gangue and is not necessarily representative of the oxide ore body.

## INTRODUCTION

To obtain basic information on the leaching charateristics of main-open-cut (M.O.C.) oxide ore a series of bench scale leaching tests were conducted. The aim of the testwork was to determine maximum copper recovery that could be achieved in separate size fractions of the ore.

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The ore sample was obtained from a shallow trench (Cut B) immediately to the east of the old smelter rubble (see Fig.1). The ore minerals were predominantly malachite/chryscolla in a quartz/sericite gangue. The oxides occur as both fracture fillings and grain coatings. Later examination showed that oxides were present within large, supposedly competant lumps of quartzite. A mineralogy report is attached as Appendix 1.

The sample was hand sorted to provide the required size fractions. Head grade varied from 3 - 8% total copper.

## EXPERIMENTAL PROCEDURE

The ore was split into nine size fractions, from 2mm to 400mm, as shown in Table 1. No feed sub-sampling could be carried out, and thus the feed samples were weighed only, with oxide copper content assumed to be ∽6%. The samples were placed in drums and submerged in "strong" sulphuric acid solution. Total acid addition was approximately three times the theoretical amount required, at a concentration of around 150-200 g/l. The containers were then left covered, with only occasional stirring. Samples of the solution were taken daily, and copper and acid concentration determined. After increments of copper concentrations became marginal, the solution was poured off, and the residue washed and dried. It was then crushed and sampled for analysis. Leach times varied from 9 to 26 days.

## RESULTS

Results are detailed in Tables 1 & 2.

## DISCUSSION OF EXPERIMENTAL RESULTS

### 1. Copper Recovery and Rates.

The oxide copper recovery achieved through the series of tests should be considered as a target maximum. This is because saturation leaching was carried out using very strong acid. The results show excellent copper recovery for the minus 45mm size fractions (over 90% in 9 days). It should be noted that the larger particles in this size range broke down during leaching, and produced a much finer residue.

Recovery dropped as particle size increased. For example, 40% oxide copper recovery in 24 days for plus 400mm ore. The lower recovery in the coarser fractions is clearly due to the imperviousness of the quartzite,

../3

thereby denying access to the oxides intermixed with the quartzite. Rate of recovery is also lower for the coarse fractions.

Fig.2 depicts graphically the rate of copper extraction.

In all calculations it was assumed that none of the sulphide mineral (i.e. chalcocite) was leached. Without carrying out feed analyses it is not possible to confirm that the chalcocite remained unattacked. (Although it is highly likely that this in fact was the case).

2. Acid Consumption.

High concentrations of acid will lead to abnormally high acid consumption rates. This has been proven experimentally elsewhere. Theoretical acid consumption is 1.54 kg/kg copper leached. The tests gave values of around 1.8 to 2.3 kg/kg. (See Table 1). The excess acid consumption, expressed as kg acid/tonne of ore, as shown in Table 1, is atypical, due to the high head grades. They would represent a 'worst case' situation. The acid consumption figures should be treated with caution.

3. General Observations.

3.1 As mentioned above, some degradation of "coarse" particles occurred during leaching. It is postulated that the oxide (or gangue) cementing material was dissolved during leaching, leading to particle break-down. The degraded fraction was not of slime size, however, and the effect should be a beneficial one in a heap leaching situation.

3.2 When drying and crushing the residue, crystals of copper sulphate pentahydrate were noticed. Although normally quite soluble, it may have been possible for these crystals to have grown under the quiescent conditions; perhaps seeded by native copper produced from the leaching of cuprite.

3.3 The results obtained in the series of tests are applicable to the sample tested, and do not necessarily reflect the leaching behaviour of the remainder of the oxide ore body. Because of the known close proximity of dolomite it is essential that tests be conducted on samples of ore obtained from other sections of the ore body

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## 4. Prediction of Heap Leaching Behaviour

The experimental data applies to discreet size fractions of ore. If an assumption regarding the likely size distribution of ripped or crushed ore is made, then it is possible to predict maximum recovery from a composite that consists of various proportions of each size fraction. This has been done, and the results appear in Tables (3 & 4). It can be seen that maximum theoretical recovery ranges from 77% for minus 400mm (16") to over 90% for minus 50mm ore (2").

To draw conclusive data on acid consumption in a similar manner would be improper. However, as a guide, a figure of 2.2 kg/kg copper leached could be employed for the ore treated.

## CONCLUSION

Size-by-size test leaching of a sample of fractured M.O.C. ore has shown that good recovery of copper can be achieved. Maximum recovery depends on particle size. Expected recovery in an operating heap leach situation would be around 60-70% (of oxide copper) for minus 400mm ore, increasing to around 85-90% for minus 50mm ore.

Acid consumption is greater than theoretical, but exact figures cannot be derived from this series of tests.

The results are only applicable to the type of sample treated i.e. a predominantly malachite/quartzite ore.

A.L.S.
12 Jan 78

## TABLE 1

## RESULTS OF LEACH TESTS ON DIFFERENT SIZE FRACTIONS OF M.O.C. ORE

| | Col A | B | C | D | E | F | G | H | I | J | K | L | M |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| PARTICLE SIZE RANGE -MM- | INITIAL ACID CONC *1 -g/l- | SAMPLE MASS -kg- | CALCULATED FEED % TOTAL CU *2 | CALCULATED FEED % OXIDE CU | RESIDUE % TOTAL CU | RESIDUE % OXIDE CU | RESIDUE MASS -kg- | FINAL CU IN SOLUTION -g- | LEACH TIME -days- | OXIDE CU RECOVERY IN LEACH TIME INDICATED -%- *3 | ACID CONSUMPTION -kg/kg Cu leached- | PROBABLE MAX OXIDE CU RECOVERY -%- IN ( ) DAYS | EXCESS ACID CONSUMPTION -kg/tonne ore *4 |
| -2 | 169 | .840 | 4.9 | 4.5 | 0.65 | 0.25 | .762 | 36 | 9 | 95 | 2.5 | All 90-95% in (10) Days | 42 |
| -5.5+2 | 153 | .757 | 6.9 | 6.1 | 0.97 | 0.10 | .660 | 45 | 9 | 99 | 1.8 | | 17 |
| -11+5.5 | 120 | .628 | 6.8 | 5.8 | 1.30 | 0.07 | .559 | 36 | 9 | 98 | 1.9 | | 17 |
| -22.5+11 | 187 | .871 | 6.7 | 5.2 | 2.10 | 0.42 | .781 | 42 | 9 | 92 | 1.9 | | 15 |
| -45+22.5 | 209 | 9.63 | 7.1 | 5.8 | 2.17 | 0.67 | 8.34 | 506 | 9 | 90 | 2.0 | | 24 |
| -100+45 | 210 | 10.46 | 3.5 | 3.5 | 0.40 | 0.38 | 9.92 | 328 | 26 | 90 | 2.3 | 90/(20) | 25 |
| -200+100 | 210 | 21.08 | 5.1 | 4.9 | 1.88 | 1.68 | 20.12 | 689 | 26 | 67 | 2.3 | 70/(25) | 24 |
| -400+200 | 238 | 59.52 | 6.3 | 6.15 | 2.62 | 2.48 | 55.61 | 2286 | 25 | 63 | 2.1 | 70/(35) | 20 |
| +400 | 185 | 62.40 | 8.2 | 6.8 | 5.90 | 4.44 | 57.59 | 1716 | 24 | 40 | 1.8 | 45/(40) | 8 |

**NOTES:**

- Above calculations assume all copper leached was oxide copper

*1 - Total volume of acid added was approximately 3 times theoretical acid consumption, assuming all samples 6% Cu

*2 - Feeds calculated from residue analysis and final solution analysis.

*3 - Assuming only oxide copper was leached. Total Cu recovery is approximately 5-10% lower for this material.

*4 - "Excess" acid is that acid consumed in excess of theoretical amount required to dissolve leached copper (1.54 kg/kg). These figures are possibly distorted, due to very high head grades.

## TABLE 2

## DAILY SOLUTION ANALYSES

## COPPER & ACID CONCENTRATIONS IN GRAMS/LITRE

| INITIAL H2S04 | ORE SIZE-MM | DAY 1 | | DAY 2 | | DAY 3 | | DAY 4 | | DAY 5 | | DAY 6 | | DAY 7 | | DAY 8 | | DAY 9 | | DAY 10 | | DAY 11 | | DAY 12 | | DAY 13 | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Cu | H2S04 | Cu | H2S04 | Cu | H2S04 | Cu | H2S04 | Cu | H2S04 | Cu | H2S04 | Cu | H2S04 | Cu | H2S04 | Cu | H2S04 | Cu | H2S04 | Cu | H2S04 | Cu | H2S04 | Cu | H2S04 |
| 169 | -2 | | | 21.5 | 122 | 26.5 | 103 | 30.5 | 94 | 31 | 93 | | | 30.5 | 88 | | | 29.5 | 87.5 | | | | | | | | |
| 152.5 | 2→5.5 | | | 24.5 | 113 | 36 | 88.5 | 37 | 85 | 37.5 | 84.5 | | | 38.5 | 77 | | | 38 | 76 | | | | | | | | |
| 120 | 5.5→11 | | | 19.5 | 89.5 | 28 | 72.5 | 28.5 | 68.5 | 28 | 69 | | | 29 | 67 | | | 28.5 | 64.5 | | | | | | | | |
| 187 | 11→22.5 | | | 26.5 | 132 | 33.5 | 122 | 34 | 118.5 | 35 | 118 | | | 36 | 113.5 | | | 35.5 | 107.5 | | | | | | | | |
| 209 | 22.5→45 | | | 39.0 | 141.5 | 42.5 | 127.5 | 43 | 124 | 44.5 | 123 | | | 46.5 | 118.5 | | | 46 | 112.5 | | | | | | | | |
| 210 | 45→100 | 9.5 | 191 | 12.3 | 174.5 | 15.8 | 162.5 | | | 18.4 | 160.5 | | | 20 | 152 | 21.4 | 152 | 22.3 | 156.5 | 22.9 | 151 | 23.5 | 153.5 | 24.0 | 151 | 24.4 | 149.5 |
| 210 | 100→200 | 3.5 | 198 | 12.4 | 177 | 14.8 | 173.5 | | | 18.3 | 168.5 | | | 19.7 | 153 | 20.8 | 150.5 | 22 | 152 | 22.4 | 150 | 23.1 | 150.5 | 23.8 | 148.5 | 24.0 | 147 |
| 238 | 200→400 | 7.0 | 217.5 | 12.2 | 205.5 | | | 19.2 | 20.3 | | | 23.6 | 182 | 26 | 185 | 27.2 | 180.5 | 28.3 | 180 | 29.2 | 175 | 30.1 | 174.5 | 30.7 | 173.5 | 30.9 | 171.5 |
| 185 | +400 | 3.7 | 167.5 | | | 8.1 | 182 | | | 13.6 | 150 | 15.5 | 150.5 | 16.5 | 148.5 | 17.8 | 145.5 | 18.3 | 146 | 19.4 | 145 | 20.4 | 143.5 | 21.0 | 141.5 | 21.5 | 141.5 |

| ORE SIZE-MM | DAY 14 | | DAY 15 | | DAY 16 | | DAY 17 | | DAY 18 | | DAY 19 | | DAY 20 | | DAY 21 | | DAY 22 | | DAY 23 | | DAY 24 | | DAY 25 | | DAY 26 | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Cu | H2S04 | Cu | H2S04 | Cu | H2S04 | Cu | H2S04 | Cu | H2S04 | Cu | H2S04 | Cu | H2S04 | Cu | H2S04 | Cu | H2S04 | Cu | H2S04 | Cu | H2S04 | Cu | H2S04 | Cu | H2S04 |
| -2 | | | | | | | | | | | | | | | | | | | | | | | | | | |
| 2→5.5 | | | | | | | | | | | | | | | | | | | | | | | | | | |
| 5.5→11 | | | | | | | | | | | | | | | | | | | | | | | | | | |
| 11→22.5 | | | | | | | | | | | | | | | | | | | | | | | | | | |
| 22.5→45 | | | | | | | | | | | | | | | | | | | | | | | | | | |
| 45→100 | 24.7 | 148 | 25.0 | 148 | 24.3 | 150 | 26.5 | 151 | | | | | | | | | | | | | | | 27.1 | 146 | 27.3 | 146 |
| 100→200 | 25.1 | 146 | 26.1 | 148 | 25.8 | 148 | 27.7 | 148 | | | | | | | | | | | | | | | 28.4 | 144 | 28.7 | 144.5 |
| 200→400 | 31.8 | 170.5 | 31.6 | 169.5 | 34.3 | 169 | | | | | | | | | | | | | | | 37.6 | 161 | 38.1 | 159 | | |
| +400 | 22.4 | 135.5 | 24.0 | 140 | | | | | | | | | | | | | | | 28.15 | 133 | 28.6 | 133 | | | | |

## TABLE 3

## PARTICLE SIZE DISTRIBUTIONS (ASSUMED)*
## FOR NOMINAL TOP SIZE SHOWN

| SCREEN SIZE -MM- | TOP SIZE - 400MM | | TOP SIZE - 200MM | | TOP SIZE - 100MM | | TOP SIZE - 50MM | |
|---|---|---|---|---|---|---|---|---|
| | MASS % RETAINED | CUM.% PASSING | MASS % RETAINED | CUM.% PASSING | MASS % RETAINED | CUM.% PASSING | MASS % RETAINED | CUM.% PASSING |
| +400 | 5 | 95 | | | | | | |
| +200 | 25 | 70 | 5 | 95 | | | | |
| +100 | 30 | 40 | 11 | 84 | 5 | 95 | | |
| + 45 | 22 | 18 | 36 | 48 | 29 | 66 | 8 | 92 |
| +22.5 | 11 | 7 | 25 | 23 | 31 | 35 | 26 | 66 |
| + 11 | 3 | 4 | 12 | 11 | 19 | 16 | 32 | 34 |
| +5.5 | 1 | 3 | 6 | 5 | 9 | 7 | 18 | 16 |
| + 2 | 1 | 2 | 2 | 3 | 2 | 5 | 10 | 6 |
| - 2 | 2 | 0 | 3 | 0 | 5 | 0 | 6 | 0 |

NOTE:

* Size distributions obtained by applying data obtained from secondary crusher product sizing, and assuming constant slope on Rosin-Rammler distribution curve.

## TABLE 4

## CALCULATED OXIDE COPPER RECOVERY FOR COMPOSITE SAMPLE, WITH SIZE DISTRIBUTION AS PER TABLE

| | NOMINAL TOP SIZE OF ORE -MM- | | | |
|---|---|---|---|---|
| | 400 | 200 | 100 | 50 |
| CALCULATED MAXIMUM OXIDE CU RECOVERY | 77 | 87 | 89 | 90+ |

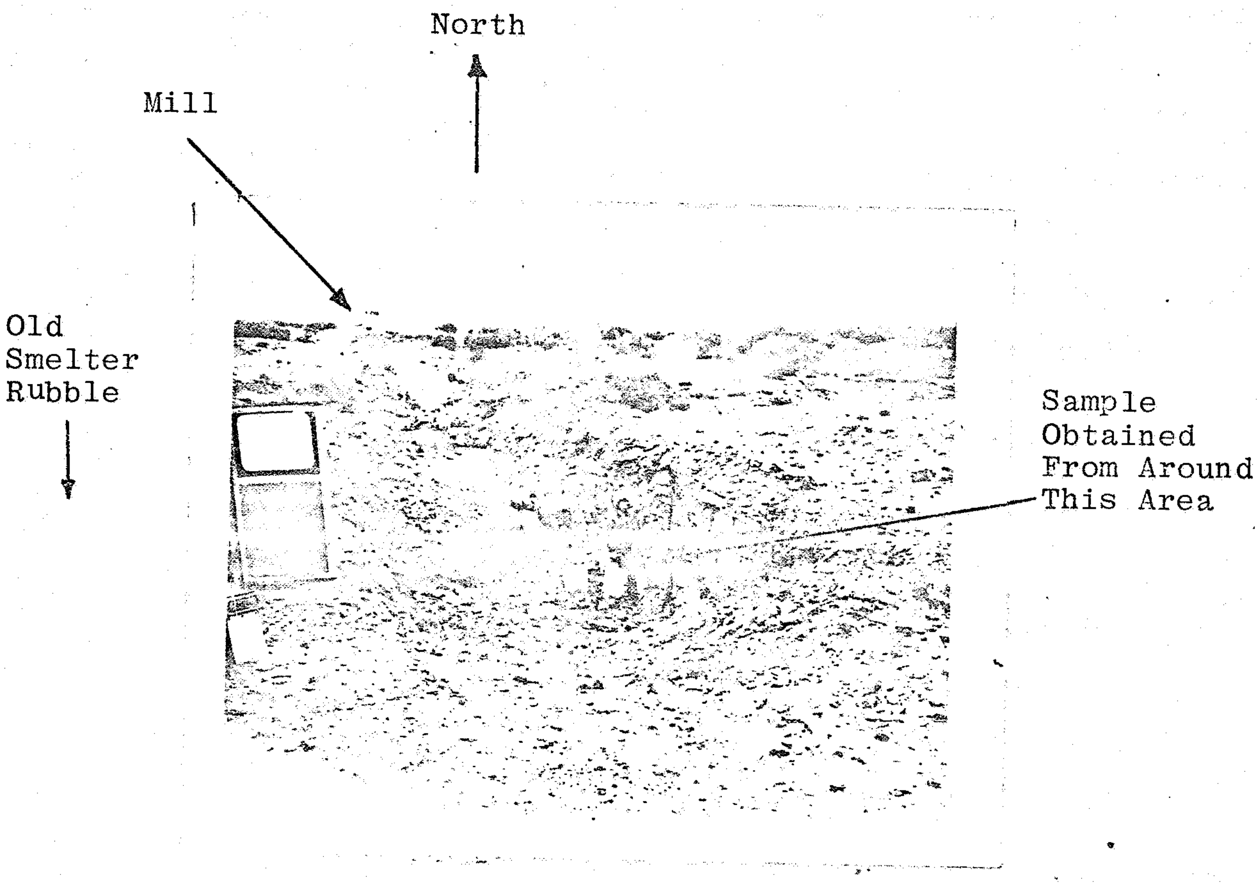

Fig 1. LOCATION OF OXIDE SAMPLE USED FOR LABORATORY TESTING.

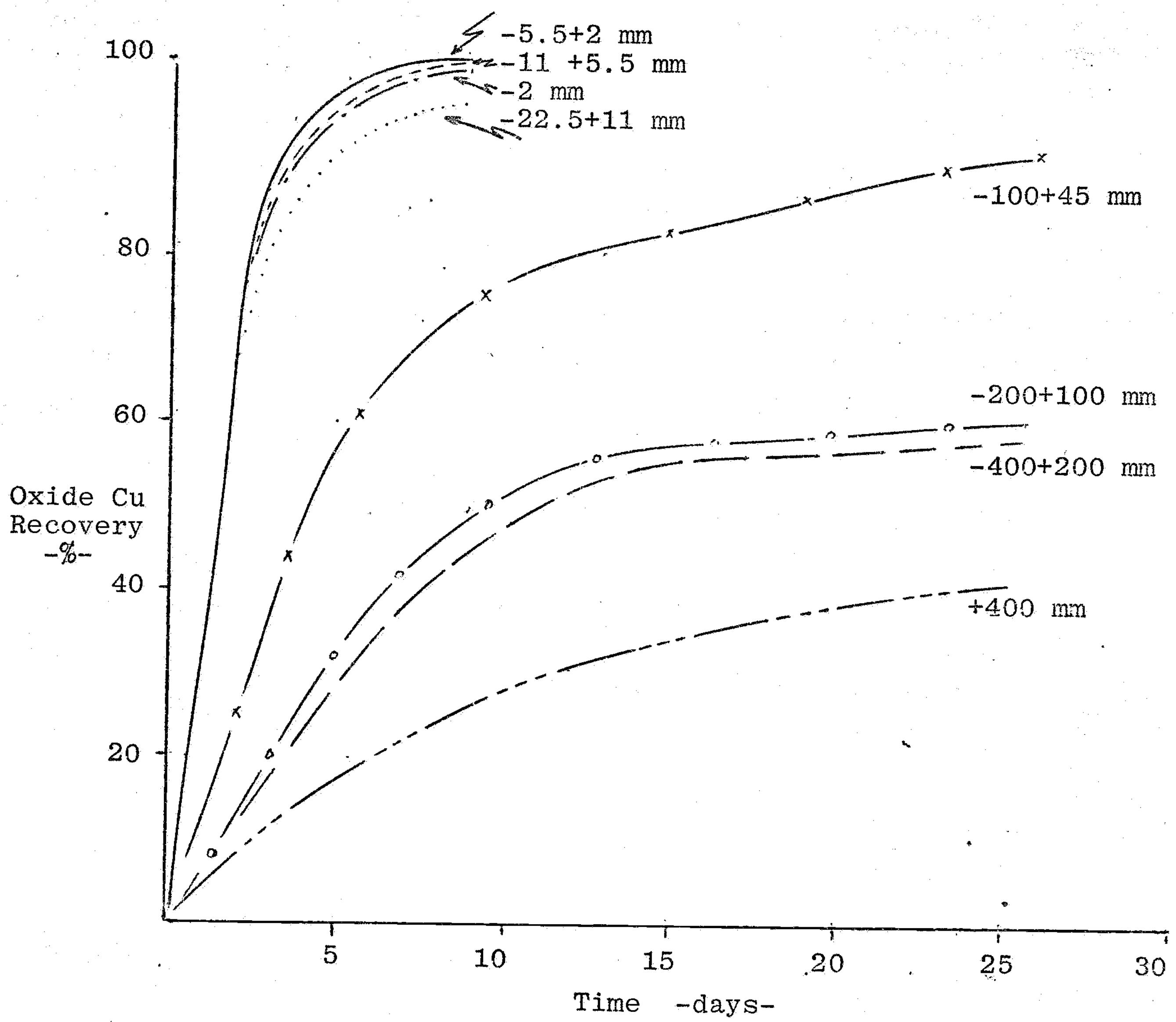

Fig.2. Oxide Copper Recovery With Time For Various Particle Sizes (As Shown).

APPENDIX 1

**Pontifex & Associates Pty. Ltd.**

TEL. 332 6744
A.H. 31 3816

26 KENSINGTON ROAD, ROSE PARK
SOUTH AUSTRALIA

P.O. BOX 91, NORWOOD
SOUTH AUSTRALIA 5067

## MINERALOGICAL REPORT NO. 2370

26th May, 1978

TO:

Mr. Ross Stainley,
Mount Gunson Mines Ltd.,
P.O. Box 246
WOOMERA S.A. 5270

YOUR REFERENCE: Your letter dated 3/5/78

MATERIAL: "oxide-ore"

IDENTIFICATION: "M.O.C. ore - April 78"

WORK REQUESTED: Polished section examination with comments as specified

SAMPLE & SECTION: Retained

PONTIFEX & ASSOCIATES PTY. LTD.

M.C.C. Sample - April 1978

To determine the relative abundance of translucent and opaque, ore and gangue minerals, a composite sample of this material was examined in thin and polished section. One thousand grains were counted to provide the following list of components and their corresponding mode. These figures refer to number of grains, irrespective of grain size and thus does not precisely represent exact volume %, although most minerals are represented through the grain size variations. The size range is 0.01 to 4 mm in maximum dimension, mean about 0.75 mm.

Comments on composition of the various grain types follow the table below.

| | approx % abundance |
|---|---|
| quartzose grains (including sandstone fragments) minor impurities adhering or as matrix | 72% |
| single green grains of secondary copper of which an estimated 70% is malachite (see note below) and ?chrysocolla | 5% |
| green grains of secondary copper (mainly malachite) intimately intergrown with clays and/or quartz sand material | 7.5% |
| virtually single grains of clay-sericite, some turbid with ultrafine carbonate limonite and rare malachite | 6% |
| clay-sericite (± ultrafine carbonate) intergrown with sandstone (fragments) ± limonite and trace malachite stain | 9% |
| (opaque) chalcocite, intergrown with and partly enveloped by malachite | 1.5% |

..../

Most quartz grains are single crystal, having broken free from the original sandstone rock, and these grains commonly contain relicts of authigenic quartz overgrowths and/or minor adhering clays. Minor composite quartz grains (fragments of sandstone) are present, with siliceous cement-matrix intact, generally with minor clay-sericite, and to a far lesser extent minor green copper minerals through intergranular areas.

The grains designated as clay-sericite, consist of ultrafine ? kaolinitic clays ± more clearly defined admixed flaky sericite. Most of these materials are permeated by limonite-stain and rare malachite stain and some contain turbid, ultrafine carbonate. These grains appear to be altered felspar and/or pelitic-lithic fragments.

The green copper mineral grains appear under binocular microscope and in thin section to consist of malachite and chrysocolla, with trace associated azurite. Other secondary green copper minerals may be present, such as dioptase (hydrous copper silicate) and atacamite (hydrated oxy-chloride of copper), however the lack of distinctive optical properties prevents these minerals from being positively identified. Indeed even in thin section, malachite and chrysocolla are seen to be intimately intergrown, and the similarity in their optical properties makes it difficult to accurately determine their relative abundance. The petrography, and reaction to acid of the green grains mounted in araldite, as viewed under binocular microscope, indicates that about 60% of all green grains is malachite, the remainder of these grains (which are non-reactive to acid) are identified, albeit tentatively, as chrysocolla. Considering the relatively minor amount of other carbonate material, generally admixed with clays (? 3%), it may be possible to determine the approximate abundance of malachite from a chemical analysis of the carbonate radical in this sample.

..../

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1/1/236
3/8/13/1

May 1980

SOUTH AUSTRALIAN DEPARTMENT OF MINES AND ENERGY
MOUNT GUNSON MINES PTY LTD

AMDEL Report

No. 1333

# HEAP LEACHING OXIDISED ORE - MOUNT GUNSON

by

L.H. Goldney

Investigated by: Operations Division

Manager, Operations Division: Bruce E. Ashton

Norton Jackson, Managing Director

THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES

Flemington Street, Frewville South Australia 5063

## CONTENTS

10461

amdel

**The Australian Mineral Development Laboratories**

Flemington Street, Frewville,
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Please address all correspondence to
P.O. Box 114 Eastwood
SA 5063
In reply quote:

30 May 1980

OD 3/8/13/1

Mr John Davies
Mt Gunson Mines Pty Ltd
PO Box 246
WOOMERA SA 5720

Dear Sir,

Enclosed are four copies of AMDEL Report No. 1333 entitled 'Heap Leaching Oxidised Ore - Mount Gunson', dated May 1980. A further five copies have been forwarded to South Australian Department of Mines and Energy.

Copper extraction in percolation leaching was relatively slow because of the low porosity of the large rocks in the as-ripped ore. To obtain reasonable extraction rates, size reduction will be required.

Agitation leaching of ground ore gave copper extractions in excess of 90%. This process could be readily applied to Mount Gunson oxidised ore but with a higher capital requirement.

Yours faithfully,
The Australian Mineral Development Laboratories

Norton Jackson

Norton Jackson,
Managing Director

Encls. 4 copies of AR No. 1333
AESIS Information Sheet
AMF Data Transmission Sheet
Quality Service Card

LHG:rp

Pilot Plant: Osman Place
Thebarton S.A.
Telephone 43 8053
Branch Laboratory: Perth

# SUMMARY

## Background

During a visit to Mount Gunson by Messrs D.W. Allen and R.J. Allen of AMDEL, the heap leaching of oxidised ore was discussed. AMDEL subsequently submitted a proposal dated 7 February 1979 for pilot scale percolation of as-ripped Mount Gunson ore in columns 3 metres high.

Initially 2 acid leaching tests were proposed, one with closed circuit recycle of leach liquor to produce a pregnant liquor with a high copper concentration and the other with leaching and intermediate removal of copper by cementation before liquor recycle. Each test was to run for approximately 60 days. Following discussions it was agreed that the most likely practical process for a relatively small scale operation would be the leach-cementation approach and initial testing proceeded with this circuit. The percolation time had to be extended to obtain meaningful data and the second test was eventually deleted.

## Objective

The objective was to obtain data which could be applied to large scale heap leaching at Mount Gunson.

## Summary of Work

The percolation column was loaded with 3.55 tonne of as-received ore and acid leaching proceeded for a period of 149 days, with intermediate copper recovery by cementation, followed by a washing period of 12 days. The initial operating pH was approximately 1.5 for the first 20 days but as the leaching rate was slow, this was decreased to pH 1.0 for the next 68 days and finally to pH 0.75 for the remainder of the period.

After leaching and washing, the ore was dried and screened into 5 size fractions which were analysed for residual copper content. From the extraction and residue results, the copper extraction was 55.3% with an acid consumption of 5.7 kg $H_2SO_4$/kg Cu extracted.

A separate 4.5 tonne head sample was analysed for size and copper distributions.

Preliminary agitation leaching tests were made with ground ore giving copper extractions of 92 to 93% with operation at pH 1.0.

## Conclusion

The as-ripped ore is not amenable to short term heap leaching because of the low porosity of the large rocks.

Copper extraction in excess of 90% is readily obtained by agitation leaching of the ground ore.

## Recommendations

Agitation leaching of the ground ore should be investigated in more detail to provide data for economic assessment of the process.

If heap leaching is considered, the degree of size reduction required to obtain reasonable copper extraction rates should be established before proceeding with large scale operations.

# 1. INTRODUCTION

During a visit to Mount Gunson by Messrs D.W. Allen and R.J. Allen of AMDEL, the heap leaching of oxidised ore was discussed. AMDEL subsequently submitted a proposal dated 7 February 1979 for pilot scale percolation leaching of as-ripped Mount Gunson ore in columns 3 metres high to obtain data which could be applied to large scale heap leaching at Mount Gunson.

Initially 2 acid leaching tests were proposed, one with closed circuit recycle of leach liquor to produce a pregnant liquor with a high copper concentration and the other with leaching and intermediate removal of copper by cementation before liquor recycle. Each test was to run for approximately 60 days. The aim of the first test would be to produce a high grade liquor from which copper recovery could be made by methods other than cementation. Following discussions it was agreed that the most likely practical process for a relatively small scale operation would be the leach-cementation approach and initial testing proceeded with this circuit.

The rate of leaching was slower than expected and after 70 days only about 30% of the total copper had been extracted and recovered by cementation. At this stage the programme was discussed with the Project Sponsors and it was agreed to extend this test to a total of 120 days and, because of limited finance, delete the second test. The results were assessed after operating 120 days and it was agreed that a further time extension was required. The eventual total leaching time was 149 days followed by column draining and washing over a further period of 12 days.

## 2. MATERIAL EXAMINED

A bulk sample of as-ripped ore was received from Mount Gunson in April 1979 with the following sample details:

| | |
|---|---|
| Weight: | 13.2 wet tonnes |
| Moisture: | 4.3% |
| Total Cu: | 1.7 to 1.9% |

In size the ore ranged from rocks up to approximately 30 cm to fines less than 1 mm. With rocks of this size and with the small tonnages involved, exact sample representativity was not obtainable in sample splitting.

A 3.55 tonne (dry weight basis) sample was used for percolation leaching. From the extraction and residue results obtained the calculated head grade was 1.78% Cu.

The weighted analyses of size fractions of a 4.5 tonne head sample taken from the remainder of the ore gave a calculated head grade of 1.57% Cu.

## 3. PERCOLATION LEACHING

The percolation flowsheet is shown in Fig. 1.

### 3.1 Equipment

| | |
|---|---|
| Leaching: | rubber lined conical bottom tank 1.05 m dia. × approximately 3 m operating ore height. |
| Liquor Holding: | 2 × 900 litre rubber lined tanks. |
| Liquor Distribution: | 1 rpm Type 316 stainless steel rotating arm with 17 × 1.6 mm dia. holes all on one side of arm and spaced at intervals to give approximately equal area cover per hole. |
| Liquor Recycle: | variable speed Type 316 stainless steel Mono pump. Operating flow rate 2 *l*/min. |
| Acid Addition: | pH controlled Watson Marlow tube pump (on-off control) adding 50% $H_2SO_4$ to a 30 litre Type 316 stainless steel tank feeding liquor distributor. |
| Cementation: | 1350 litre rubber lined tank with agitator and suspended stainless steel basket holding steel plate for cementation. |

### 3.2 Procedure and Results

The leaching tank was loaded with 3.55 tonne (dry weight basis) of ore giving an ore height of 3.10 m. Larger rocks, 15 to 30 cm, were selected from this ore batch and packed by hand in the tank cone to give a graded support bed for the total charge. The remainder of the ore, mixed fines and rocks up to approximately 15 cm, was tipped directly into the tank from an overhead crane. After circulating liquor for 24 hours, the ore height had reduced by 15 cm but thereafter there was little change. The operating packed ore volume was 2.07 $m^3$ i.e. 1.7 $t/m^3$.

Water containing approximately 15% NaCl from East Lagoon, Mount Gunson was diluted by a factor of 10 to give 1360 litres of circuit water containing the following soluble salts (in g/*l*):

| Mg | Na | Cl | $SO_4$ | Total Dissolved Solids |
|---|---|---|---|---|
| 0.5 | 5.3 | 9.5 | 0.75 | 16.8 |

The circuit water was pumped at a flow rate of 2 $l$/min. (2.3 $l$/min./m$^2$), initially without acid addition. The time taken for liquor to exit from the base of the ore after commencing liquor feed was 186 min., giving a hold-up volume of 372 litres. After circulating liquor continuously for 2 days the hold-up volume had increased to 430 litres.

The initial aim in leaching, as discussed with Mr Ross Stainley, Mount Gunson was to produce an effluent liquor at pH 2 for cementation. After some experimentation it was established that this required an input pH of approximately 1.7. At this pH copper extraction was obviously slow and the input pH was re-set at 1.5. The average pH during the first 20 days of leaching was near 1.5.. After discussion with Mount Gunson at this stage, the input pH was set at a nominal 1.0 and this was maintained until Day 85. As the total copper extraction at Day 85 was only 37% it was agreed during discussions with Mount Gunson and the Department of Mines and Energy that a higher acidity be tried for the remainder of the leaching. The input liquor pH was set at 0.75 on Day 86 and maintained until Day 149. The column was then drained and given 2 circulating water washes prior to unloading.

After draining the column was unloaded by hand, to avoid breaking the rocks, and the leached ore was dried prior to sizing. The individual size fractions were crushed to minus 12 mm before taking representative fractions for chemical analyses.

During the operation, copper was recovered by cementation when the pregnant liquor copper concentration was approximately 4 g/$l$ Cu and the 'barren' liquor was recycled to leaching. A total of 13 batch cementations were made. Iron consumption was approximately 2.2 kg Fe/1 kg Cu. Some liquor loss occurred with the cement copper slurry and on Day 105, 145 litres of fresh water was added to the leaching circuit.

Copper extraction over the total period was 55.3%, calculated from liquor volumes and Cu analyses before and after cementation and wash water Cu analyses. The copper head grade, calculated from the total Cu extracted plus Cu remaining in the leached solids residue fractions, was 1.78% Cu.

After completing the leaching programme, a 4.5 tonne sample of the remaining head ore was screened for sizing and copper distribution analyses. The size analysis was similar to the leached ore sample but the copper content of the larger rocks (+100 mm) was lower.

The results are presented in Tables 1 to 6 and Fig. 2 as under:

| | |
|---|---|
| Copper extraction: | Table 1 and Fig. 2 |
| Leached ore size and copper distributions: | Table 2 |
| Calculated head of leached ore: | Table 3 |
| Leach liquor analyses: | Table 4 |
| Acid consumption: | Table 5 and Fig. 2 |
| Head ore size and copper distributions: | Table 6 |

During unloading of the leached ore there was no evidence of 'dry' areas - all solids had been contacted with leach liquor. All large rocks were intact.

Large rocks, broken after sizing the leached residue, showed only minor liquor penetration, except on fracture planes. Penetration of hard rocks ranged from 1 to 6 mm but some softer rocks had penetration up to approximately 12 mm. Extraction of copper was therefore limited to a relatively shallow surface band on the hard material.

Plates 1 to 4 are photographs of intact and hand broken rocks after leaching:

| | |
|---|---|
| Plate 1: | Large rocks after leaching. |
| Plates 2 and 3: | Broken 15 to 20 cm, hard rocks showing minimum penetration with unleached copper minerals close to ore surface. |
| Plate 4: | Chipped softer rocks showing increased penetration (up to approximately 12 mm). |

## 4. AGITATION LEACHING

During operation of the percolation column, 2 size fractions of Mount Gunson ore were crushed and agitation leached with sulphuric acid to determine the effect of pH on copper extraction rate.

One 200 litre drum of as received ore, but with the plus 100 mm rocks removed, was taken at random and screened at 6 mm, giving the following analyses:

| | Plus 6 mm | Minus 6 mm | Weighted Average |
|---|---|---|---|
| Weight kg | 182.2 | 137.4 | |
| Weight % | 57 | 43 | |
| Moisture % | 1.5 | 6.9 | 3.8 |
| Cu % | 0.69 | 0.83 | 0.75 |

The 2 fractions were crushed to minus 500 µm (30 mesh BSS) prior to leaching for 7 hours at 20°C with slurry concentrations of 33% solids w/w. The conditions and results were:

| | Original +6 mm | | Original -6 mm | |
|---|---|---|---|---|
| | pH 1.0 | pH 1.5 | pH 1.0 | pH 1.5 |
| kg $H_2SO_4$/t ore | 45 | 19 | 65 | 27 |
| kg $H_2SO_4$/kg Cu extracted | 7.0 | 10 | 8.5 | 10 |
| Head % Cu | 0.69 | 0.69 | 0.83 | 0.83 |
| Tail % Cu | 0.05 | 0.35 | 0.07 | 0.37 |
| Cu Extraction % | 93 | 49 | 92 | 55 |

Although this sample was not representative in grade of the total ore, the results showed the significance of acid concentration in leaching. Operation near pH 1.0 was required for reasonable copper extraction rates. With the lower head grade of this sample, acid consumption (as kg $H_2SO_4$/kg Cu extracted) was relatively higher than in the percolation test.

## 5. DISCUSSION

The copper extraction rate in percolation leaching was lower than expected but the reason was apparent when the larger rocks were examined after the 161 day leaching and washing period. As shown in Plates 2 to 4, liquor penetration was minimal. Nearly 50% of the total residual copper was contained in the plus 100 mm fraction which was only 15% of the total residue weight.

Lack of penetration was probably a combination of factors:

(1) Low initial rock porosity.

(2) Precipitation of insoluble salts (e.g. gypsum) in the surface layers giving a further reduction in porosity.

Acid concentration is an important factor, as shown in the latter agitation leaching tests. The preferred pH of operation is near 1.0. In the percolation test the lower acid concentration applied (near pH 1.5) in the initial stages did not significantly decrease the overall percolation performance as this concentration was only maintained for a limited period (20 days in a total of 149 days). No apparent increase in leaching rate was obtained by decreasing the pH to 0.75 but acid consumption increased at this higher concentration.

With the near 'closed circuit' condition in leaching and cementation, the dissolved salts content of the recycling liquor increased over the total period but this factor did not appear to be detrimental in either leaching or cementation.

The acid consumption figures reported include acid consumed in cementation by reaction with metallic iron. Acid and iron consumption in cementation would be lower in normal plant operation where the acid concentration in pregnant liquors would be lower after passing the liquors through several heaps in series. A detailed study of cementation conditions was not included in this project programme.

## 6. CONCLUSIONS

The as-ripped ore is not amenable to short term heap leaching because of the low porosity of the large rocks.

Copper extraction in excess of 90% is readily obtained by agitation leaching of the ground ore.

## 7. RECOMMENDATIONS

Agitation leaching of ground ore should be investigated in more detail to provide data for economic assessment of the process.

If heap leaching is considered, the degree of size reduction required to obtain reasonable copper extraction rates should be established before proceeding with large scale operations.

## 8. ACKNOWLEDGEMENT

The co-operation and contributions of Messrs D.R.S. Morriss and L.A. Pilgrim in preparation and operation of the percolation pilot plant are acknowledged.

TABLES 1 to 6

FIGS 1 and 2

PLATES 1 to 4

## TABLE 1: COPPER EXTRACTION - CEMENTATION AND WASHING

| Cement No. | Day | Liquor Vol. *l* | Cu g/*l* |  | pH |  | Cement Copper |  |
|---|---|---|---|---|---|---|---|---|
|  |  |  | In | Out | In | Out | kg | kg Cum. |
| 1 | 19 | 692 | 3.32 | 0.40 |  |  | 2.02 | 2.02 |
| 2 | 29 | 616 | 4.36 | 0.18 |  |  | 2.57 | 4.59 |
| 3 | 37 | 635 | 4.45 | 0.59 | 1.1 | 1.4 | 2.45 | 7.04 |
| 4 | 48 | 526 | 3.79 | 0.40 | 1.1 | 1.5 | 1.78 | 8.82 |
| 5 | 56 | 579 | 5.8 | 0.67 | 1.1 | 1.5 | 2.97 | 11.79 |
| 6 | 62 | 500 | 5.9 | 1.50 | 1.1 | 1.7 | 2.20 | 13.99 |
| 7 | 70 | 505 | 6.2 | 0.80 | 1.1 | 1.5 | 2.73 | 16.72 |
| 8 | 81 | 487 | 5.8 | 0.70 | 1.1 | 1.6 | 2.48 | 19.20 |
| 9 | 91 | 500 | 5.0 | 0.68 | 0.8 | 1.3 | 2.16 | 21.36 |
| 10 | 99 | 487 | 4.6 | 0.92 | 0.8 | 1.2 | 1.79 | 23.15 |
| 11 | 113 | 627 | 4.6 | 0.50 | 0.8 | 1.0 | 2.57 | 25.72 |
| 12 | 127 | 690 | 4.4 | 0.40 | 0.8 | 1.0 | 2.76 | 28.48 |
| 13 | 145 | 741 | 4.15 | 0.44 | 0.8 | 1.0 | 2.75 | 31.23 |

| Total Copper Extraction | kg |
|---|---|
| Cementation Nos 1 to 13 | 31.23 |
| Leaching Days 146 to 150, then draining Days 150 to 153<br>756 litres at 2.6 g/*l* Cu | 1.97 |
| Wash No. 1, Days 155 to 159<br>855 litres at 1.3 g/*l* Cu (pH 1.4) | 1.11 |
| Wash No. 2, Days 160 to 161<br>731 litres at 0.90 g/*l* Cu (pH 1.8) | 0.70 |
| Total | 35.01 |

## TABLE 2: LEACHED ORE - SIZE AND COPPER DISTRIBUTIONS

| | Fraction | | Copper | |
|---|---|---|---|---|
| Size in. | Weight kg | Weight % | Cu % | Distribution % |
| +4 | 506 | 15.0 | 2.65 | 47.4 |
| -4 +2 | 515 | 15.3 | 1.00 | 18.2 |
| -2 +1 | 465 | 13.8 | 0.52 | 8.5 |
| -1 +½ | 454 | 13.5 | 0.33 | 5.3 |
| -½ | 1426 | 42.4 | 0.41 | 20.6 |
| | 3366 | 100.0 | 0.84 (a) | 100.0 |

(a) Weighted residue assay.

Note: The minus ½ in. fraction contained original fines plus small fragments containing relatively high grade copper which broke from the larger rocks during residue handling and screening.

## TABLE 3: CALCULATED HEAD AND COPPER EXTRACTION

| | Tonnes | | |
|---|---|---|---|
| Residue dry weight | 3.37 | | |
| Cu extracted | 0.035 | | |
| Unaccounted (fe, Al etc leached) | 0.145 | | |
| Original ore dry weight | 3.55 | | |
| Residual copper: 3.37 t 0.84% Cu | 28.31 kg | = | 44.7% |
| Extracted copper: from Table 1 | 35.01 kg | = | 55.3% |
| Total copper | 63.32 kg | | |
| Calculated Head 63.32 kg/3.55 t | 1.78% Cu | | |
| Cu Extraction 55.3% | | | |

TABLE 4: LEACH LIQUOR ANALYSES

| Day | Analyses | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| | Cu g/l | Ca g/l | Mg g/l | Al g/l | Total Fe g/l | Cd mg/l | Zn mg/l | Mn mg/l | TDS g/l |
| 19 | 3.3 | 0.66 | 1.29 | 0.23 | | | | | |
| 29 | 4.4 | | | | | 70 | 60 | 0.2 | |
| 37 | 4.5 | 0.60 | 1.70 | 0.44 | 4.0 | | | | 60.0 |
| 70 | 6.2 | 0.56 | 2.28 | 1.10 | 13.9 | | | | 105 |
| 81 | 5.8 | | | | | 170 | 160 | 0.4 | |
| 127 | 4.4 | | | | | 180 | 170 | 0.6 | |
| 147 | 4.15 | 0.56 | 2.46 | 2.12 | 29.7 | | | | 150 |
| 147 | | | | | 36.3(a) | | | | 150(a) |

Note: All analyses with the exception of (a) were made on pregnant liquors prior to cementation.

TABLE 5: ACID CONSUMPTION

| Day | Sulphuric Acid (100% $H_2SO_4$) | | |
|---|---|---|---|
| | Cum. Addition kg | kg/t Ore | kg/kg Cu Leached(a) |
| 19 | 32.8 | 9.2 | 6.5 |
| 29 | 46.0 | 13.0 | 6.1 |
| 37 | 58.3 | 16.4 | 5.8 |
| 48 | 71.7 | 20.2 | 6.1 |
| 56 | 78.1 | 22.0 | 5.3 |
| 62 | 89.5 | 25.2 | 5.3 |
| 70 | 98.3 | 27.7 | 5.0 |
| 81 | 109.0 | 30.7 | 4.9 |
| 91 | 128.7 | 36.3 | 5.3 |
| 99 | 141.7 | 39.9 | 5.4 |
| 113 | 158.1 | 44.5 | 5.5 |
| 127 | 173.4 | 48.8 | 5.5 |
| 145 | 191.7 | 54.0 | 5.6 |
| 149 | 199.2 | 56.1 | 5.7 |

(a) The copper extraction figures for this calculation are based on the cumulative cement copper figures in Table 1 with the addition of 3 kg Cu to give the total copper leached at that time. The liquor held up in the leach column and in recycle liquor tanks contained varying concentrations of copper but generally in the range of 2.5 to 3.5 kg contained Cu.

TABLE 6: HEAD SAMPLE - SIZE AND COPPER DISTRIBUTIONS

| Fraction | | | Copper | | |
|---|---|---|---|---|---|
| Size in. | Weight kg | Weight % | Oxide Cu % | Total Cu % | Total Cu Distribution % |
| +4 | 622 | 14.0 | 1.95 | 2.00 | 17.9 |
| -4 +2 | 797 | 17.9 | 2.30 | 2.30 | 26.2 |
| -2 +1 | 610 | 13.7 | 1.60 | 1.60 | 14.0 |
| -1 +½ | 573 | 12.8 | 1.40 | 1.40 | 11.4 |
| -½ | 1856 | 41.6 | 1.00 | 1.15 | 30.5 |
| | 4458 | 100.0 | | 1.57 (a) | 100.0 |

(a) Weighted head assay.

Tank Volumes - litres

| ① | ② | ③ | ④ | ⑤ |
|---|---|---|---|---|
| 30 | 2250 | 900 | 900 | 1350 |

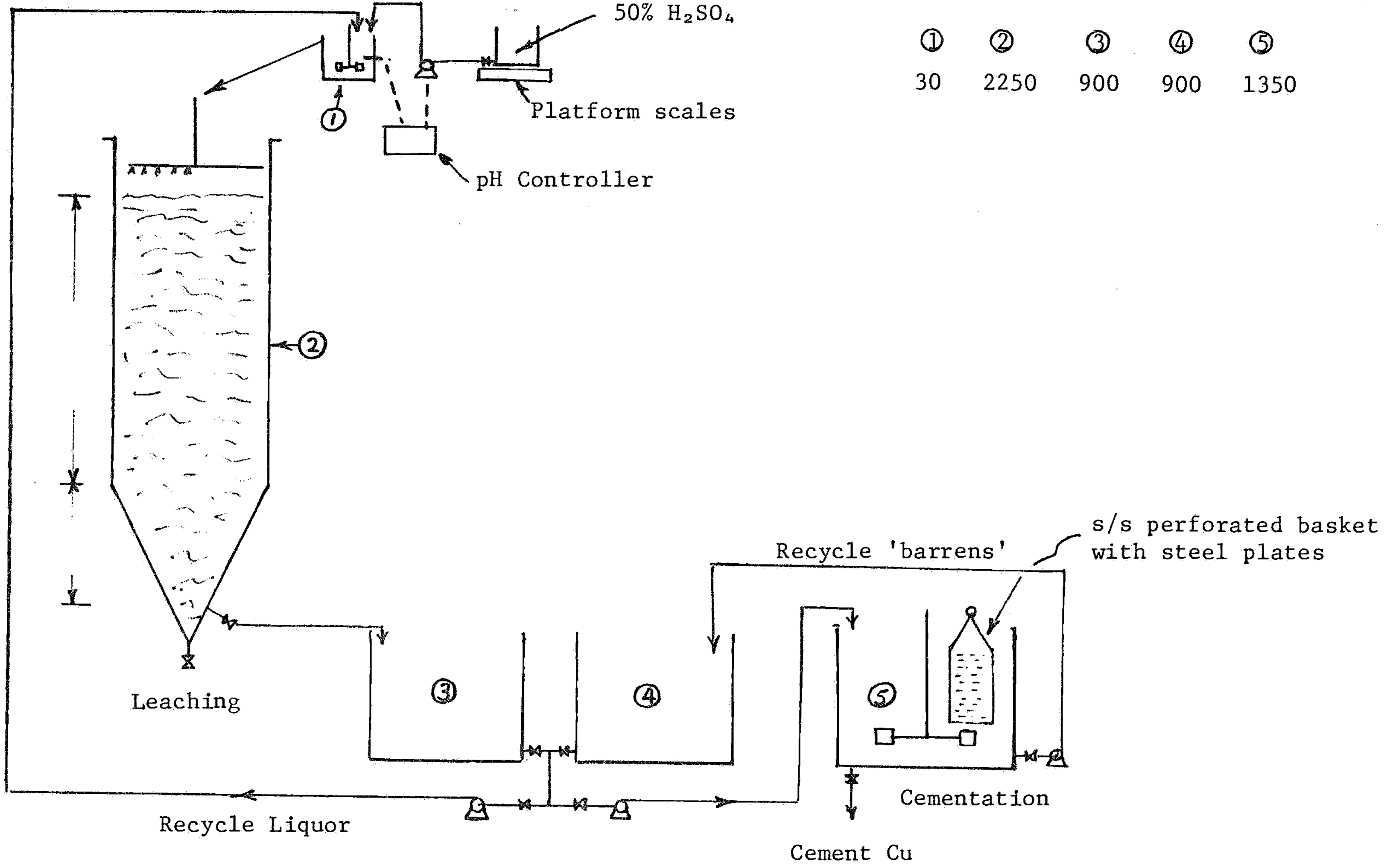

FIG. 1: PERCOLATION FLOWSHEET

000481

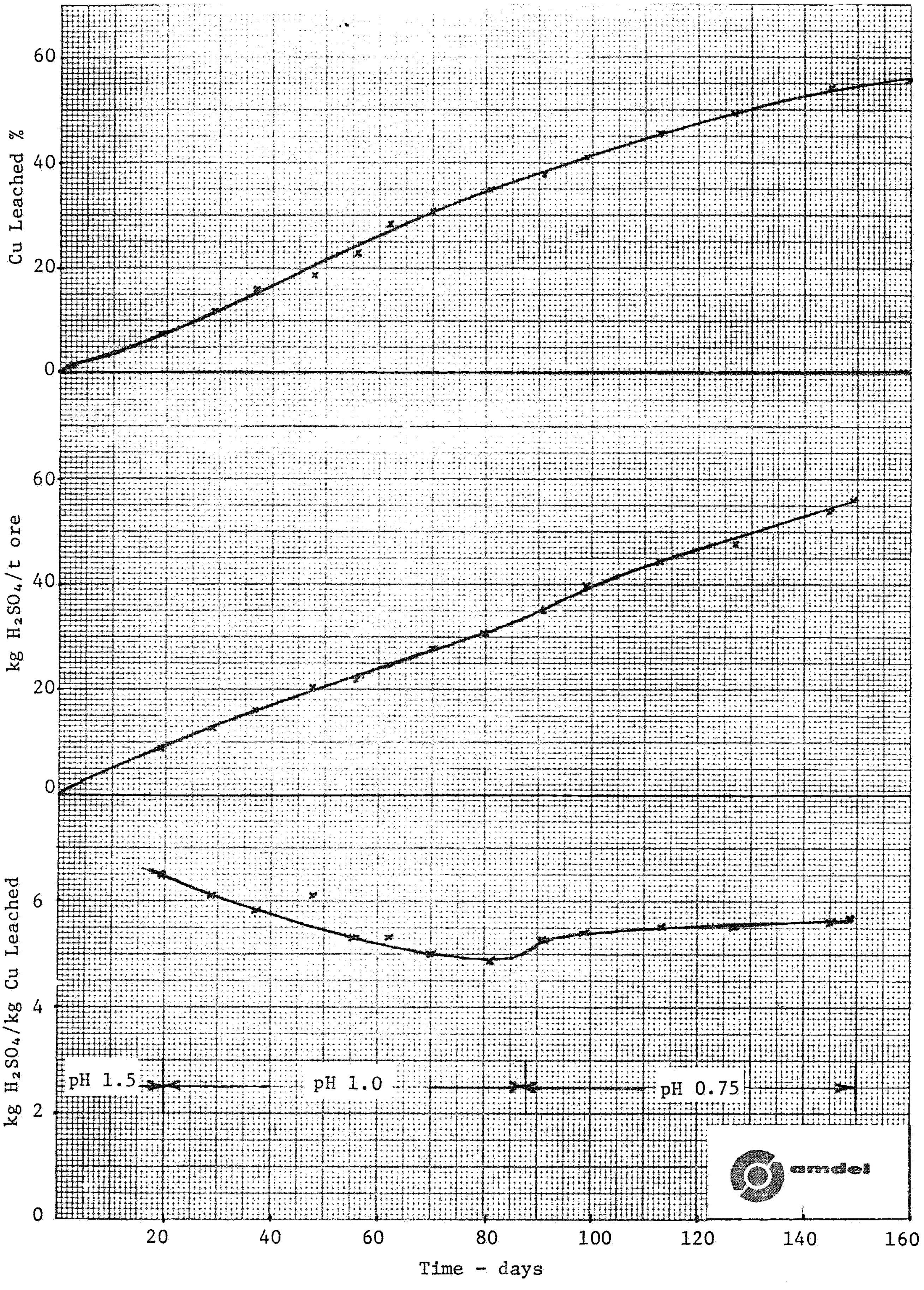

FIG. 2: COPPER EXTRACTION AND ACID CONSUMPTION

PLATE 1: LARGE ROCKS AFTER LEACHING

PLATE 2: BROKEN HARD ROCK

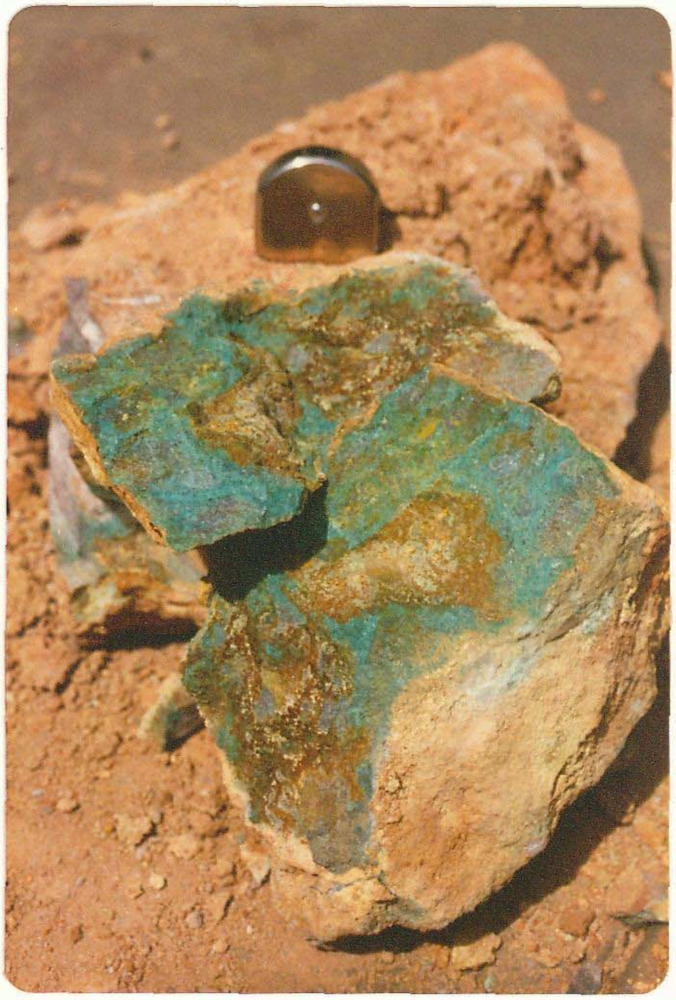

PLATE 3: BROKEN HARD ROCK

PLATE 4: CHIPPED SOFTER ROCKS

MEMO TO: B.W. Flood
FROM: C.A. Vanderdoes

SUBJECT: METALLURGICAL EVALUATION OF MAIN OPEN CUT DEPOSIT (M.O.C.)

## A. FLOTATION

The area known as the Main Open Cut Workings (M.O.C.) can be described as a mixed sulphide - oxidised orebody, although some areas are predominantly sulphides, while others are mainly "oxidised".

Seven distinct areas based on geographical and geological factors are delineated A to G and only Area B could be economically considered with anticipated recoveries of 85 - 90% depending on degree of selectivity in mining as dilutions with oxidised mineralisation will inevitably decrease overall recovery. Tonnage throughput must at this stage be viewed with caution untill settling tests confirm characteristics similar to Cattle Grid ore. It may be necessary to use both thickeners for tailings disposal.

No modifications are expected to be required in present concentrator.

Silver content appears to be appreciably higher in M.O.C. are with values reported up to 93 ppm.

## B. OXIDE FLOTATION

Historically, test work on flotation of M.O.C. oxidised ores has failed to evolve economically acceptable recoveries, due to presence of crysocolla and atacamite which do not respond to flotation techniques.

It must be accepted that flotation of M.O.C. oxidised ores is not feasible.

## C. ACID LEACHING

Test work conducted by several laboratories invariably confirmed M.O.C. oxidised copper mineralisation to be suitable for treatment by acid leaching resulting in recoveries up to 90% with low acid consumption. Ore particle size is critical due to impermeability of host rock and fine grinding is necessary for optimum recoveries.

The viability of an acid-leach process is dependent on an economic assessment based on copper price primarily and secondly on mining selectively as dilutions with sulphide bearing mineralisation will decrease recoveries.

### ORE RESERVES

320 000 Tonnes @ 1.6% copper.
620 000 Tonnes overburden (1.95:1 strip ratio)

### BASIC PROCESS

Crushing/grinding using existing plant followed by 2 stage cascade agitation leach, solid/liquid separation using c.c.d. thickeners, tailings disposal with existing plant. Copper recovery by cementation with scrap iron.

### CAPITAL COSTS

Estimate $1 million.

../2

OPERATING COSTS are expected to be similar to flotation, depending on cost of consumables of acid and scrap-iron less cost of flotation reagents, freight reductions and possibly improved realisation of cement copper produced if sold to Adelaide & Wallaroo Fertilisers, BURRA.

## D. FLOTATION/ACID LEACHING

The excellent response of sulphide mineralisation to conventional flotation and the response of oxidised material to acid leaching presents an attractive proposition to assess the M.O.C./House workings as a whole entity by flotation to recover sulphides and silver, followed by acid leaching of the tailings or middlings.

An overall recovery of 85-90% may be expected at total costs of estimated flotation proposal or marginally above, depending on cost of acid and scrap iron.

Ore reserves would be at least doubled and oxidised/sulphide mineralisation may be blended to balance concentrator circuit.

## RECOMMENDATIONS

Sufficient data is available on flotation and acid leaching M.O.C. copper mineralisation. Viability is directly related to copper price and effectiveness of selective mining techniques.

For optimum exploitation of M.O.C./House deposits it is recommended that some drilling be carried out in "mixed" oxidised/sulphide areas to provide samples for metallurgical evaluation of the feasibility of a flotation/acid leaching process.

CAVanderdoes

C.A. VANDERDOES

(Metallurgist)

25th January, 1982

c.c. J.L. Davies

3/8/13/2

August 1984

MOUNT GUNSON MINES PTY LIMITED

# AMDEL Report

## No. 1544

# METALLURGICAL TESTWORK - MG14 DEPOSIT

by

G.A. Dunlop

Investigated by: Operations Division

Manager, Operations Division: Bruce E. Ashton

Brian S. Hickman, Managing Director

THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES
Flemington Street, Frewville, South Australia, 5063

## CONTENTS

# SUMMARY

## Background

Following a metallurgical appraisal by Mr P.C. Bradshaw of Mount Gunson Mines Pty Limited in August 1983 using results of earlier work on ore samples from the MG14 Deposit, discussions were held at AMDEL resulting in proposal OD 5/0/4057 of 25 October 1983 for a test programme involving:

Part 1 - Sampling and analysis of drill core
2 - Metallurgical testing
3 - Pilot plant operation (if required)

Initially no action being required on Part 3, samples for the test programme were to come from newly drilled core.

Following a programme review between our Mr Dunlop and Mr Bradshaw, the proposal was amended to:

Part 1 - Preparation of the sample for metallurgical testwork
2 - Metallurgical testwork (including mineralogy) on ore composite sample from MG14 plus a comparative grind on a current mill sample from the Cattlegrid deposit.

Flexibility of the programme was to be maintained, so costing was to be on an as-worked basis rather than a lump sum contract.

Approval for this amended test programme was given by telex of 16 November 1983.

Samples were received in the week ending 16 December 1983 and after the Christmas closure work on these commenced on resumption of activities in week ending 9 January 1984.

Copies of worksheets were progressively supplied to Mr Bradshaw as each section was finished. Review meetings were held and the work programme re-arranged based on results obtained.

## Objectives

The client defined the overall objectives of this work as:

(a) to determine by assay and mineralogy, the distribution of the major copper minerals in a suitably ground sample assuming the current grind in the concentrator operating on Cattlegrid ore.

(b) to obtain the highest copper grade and recovery in the simplest possible circuit on the basis this deposit (MG14) was of limited tonnage and would be treated through the existing Mount Gunson concentrator with possibly minor equipment changes.

Final selection and optimisation of reagent types was not an objective of this programme.

## Summary of Work Done

Two separate samples (MG14 and Cattlegrid) were received for testwork. Each was prepared to 100% -1.7 mm by stage crushing and screening at AMDEL.

A site water sample was also received but on Mr Bradshaw's instruction was not used for the early stages of testwork.

The laboratory testwork dealt only with the MG14 sample covering:

(a) grinding tests
(b) mineralogy on sized fractions of a coarsely ground product
(c) comprehensive analysis of a head sample
(d) mineralogy on selected size fractions magnetically treated
(e) 8 flotation tests

Test sheets were progressively sent to Mr Bradshaws (usually in hand written format).

At Mr Bradshaw's request no work was done:

(a) on the Cattlegrid sample
(b) cleaning of concentrates produced in flotation tests

This report gives the results of all work completed and included all test data generated and metallurgical results calculated.

## Conclusions

The following conclusions are drawn from the test programme so far completed on this sample from the MG14 deposit.

(1) Mineralogically:

(a) the major (≥ 0.1%) minerals present are:

| | | |
|---|---|---|
| non-sulphide gangue | siderite | 30-40% |
| | quartz | 20-30% |
| | mica/illite | 15-20% |
| | dolomite | 10-15% |
| | chlorite | 3-5% |
| copper | chalcocite | 1.6% |
| | bornite | 1.5% |
| | chalcopyrite | 0.2% |
| other sulphides | sphalerite | 0.3% |
| | pyrite | 0.1% |
| | carollite | 0.1% |

(b) liberation of copper sulphide minerals is very poor at a grind of P80 ≃ 210 µm chosen to be close to current plant operations knowing mineralisation in MG14 is finer grained.
In fact liberation at this grind is almost negligible down to 75 µm and even in the -18+9 µm size fraction is only about 50% complete. ⊛

(c) Very fine grinding is required for liberation of copper sulphides. A grind to 100% -45 µm would be unlikely to produce good liberation of these minerals even in the -45+9 µm size range.

(d) For the grind of P80 ≃ 210 µm, a size by size analysis shows both the Cu and Ag uniformly distributed down the size ranges according to % weight with the -12 µm fraction (slimes) holding 21.5% and 18.2% respectively.

Heavy liquid separations (3.3 sp.gr.) reveal almost constant Cu values in the light fraction (<3.3) down the size ranges to 45 µm below which there is a slight decrease i.e. there is a 'background' of very finely disseminated copper minerals in the light gangue. The reverse is true for silver values and while bismuth follows the trend for silver, cobalt shows a marked concentration in the coarse size ranges and apparently good liberation from silicates even in these coarser sizes.

(e) Since with magnetic separation very little of the copper and silver is associated with particles containing a moderate to high proportion of siderite, the silicates contain this 'background' copper and are the hardest to grind for effective liberation.

(2) In grinding the ore is soft but at Mr Bradshaw's request no comparative grinds were done with the Cattlegrid sample supplied.

| F80 (μm) | P80 (μm) | Rod Load* | Grinding Time (mins) |
|---|---|---|---|
| 1250 | - | 0 | 0 |
| | 127 | 17 | 3 |
| | 78 | | 5 |
| | 51 | | 8 |
| | 210 | 12 | 3 |
| | 133 | | 5 |
| | 87 | | 8 |

Such steep reduction stages in minimum time means that the softer minerals are easily ground, only quartz and harder silicates remaining for grinding. Liberation of the 'background' copper mineral content is therefore a real problem as overgrinding is a distinct possibility in attempting maximum copper liberation.

(3) Viscosity problems are not apparent at any stage of treatment.

(4) In flotation:

(a) Cu recovery improves with fineness of grind with the Cu grade of a bulk rougher/scavenger concentrate remaining relatively constant. At a P80 ≃ 115 μm, Cu recovery after rougher and 3 scavenger stages approaches 75% at 18-19% Cu grades.

Cu recovery increases markedly as the grind becomes finer at 115 μm and sliming is not a problem.

Because the requirement was to stay within range of the concentrator grinding capacity, still finer grinds were not investigated.

---

*Normal rod load is 17 rods.

(b) Natural pH of flotation feed is 8.6-8.7 in Adelaide tap water. Changing the pH to 9.1 with soda ash was disastrous.

(c) Heavy reagent additions are required to achieve the Cu recovery down to the P80 ≃ 115 μm grind. Collector additions ranged from 250 to 450 g/t but recovery is lost at the lower levels although concentrate grades are higher.
Of those tried, PAX is the better collector and no float resulted with T401, MIBC being used successfully as frother.
Obviously at these grinds, the heavier additions are pulling composite particles.
With a still finer grind and adequate Cu mineral liberation, a trade off situation between collector requirements and degree of grind used will exist for maximum recovery.

(d) There is no logical metallurgical reason to go to a sand/slime circuit after grinding as the same result can be achieved in a straight through float using less equipment.

A financial analysis will decide whether it is worth treating only the sand fraction after desliming flotation feed and rejecting slimes straight to tailings. Bulk concentrate from such a sand float was 23.9% Cu for 64.9% total Cu recovery, the sand fraction representing 79.3% by weight and holding 86.5% of the total Cu. (i.e. 75% Cu recovery in sand float feed). This analysis will also be dependent on the efficiency in hydrocyclone operation achievable at plant scale.

(e) At these coarse grinds, total flotation times are long (15 to 22 minutes) for maximised Cu recovery.

(5) Realistic pulp densities were in use in flotation stages. Any attempt to decrease these would increase flotation times and considerably worsen the dewatering characteristics of plant products. Laboratory tailings could not be settled or filtered.

This problem was not investigated further.

(6) The best test result is from Test MG14 - F4, a straight through float producing a bulk rougher and scavenger concentrates under conditions of:

P80 ≃ 115 µm
PAX ≃ 400 g/t
Float time ≃ 15 minutes (3 rougher, 4-1st scavenger, 8-2nd scavenger)
Result ≃ 19.4% Cu grade for 71% Cu recovery

Based on other tests and an additional scavenger stage it is projected that maximum recovery will approach 75% Cu in 22 minutes flotation time and 450 g/t PAX for 18+% Cu grade.

## Recommendation

The next step recommended in the overall programme is to complete a financial analysis using results from this work and from earlier work on other samples.

It will be necessary to take the following into account assuming that there is a limit to capital expenditure on plant changes/expansion to treat what is a relatively small tonnage of MG14 ore:

(a) the power requirements for grinding MG14 samples for adequate copper mineral liberation

(b) a straight roughing/scavenging circuit at conventional pulp densities with necessary cleaning stages to follow

(c) high reagent additions for maximum copper recovery. There may be a trade off between grinding requirements and collector additions depending on just how soft the MG14 ore is in comparison to the currently treated Cattlegrid ore

(d) whether by (b) it is better to treat the sand fraction after desliming the flotation feed and rejecting slimes to tailing. This may allow maximum tonnage through existing plant with minimum capital outlay - one desliming cyclone setup - particularly if MG14 ore is considerably softer to grind than Cattlegrid ore.
Either way flotation times and reagent additions are similar.

Further testwork can then be programmed.

# 1. INTRODUCTION

Earlier work on ore samples from the MG14 deposit in South Australia was undertaken by Pacminex Pty Limited in 1975. Following a metallurgical appraisal by Mr P.C. Bradshaw of Mount Gunson Mines Pty Limited in August 1983, AMDEL were requested to prepare a suitable test programme (with cost estimate) to undertake follow-up metallurgical testwork on a new bulk sample. This work was restricted to laboratory scale only.

After reviewing previous testwork(1), mineralogy on selected samples(2) and Mr Bradshaw's appraisal(3) following a meeting to discuss possible programmes, AMDEL proposal OD 5/0/4057 of 25 October 1983 was issued in three parts:

Part 1 - Sampling and analysis of drill core
Part 2 - Metallurgical testing
Part 3 - Pilot plant operations (if required)

Initially no action was required on Part 3, so the issued detailed programme covered only Parts 1 and 2.

Samples for these test programmes were to come from newly drilled core.

This test programme having been reviewed between Mr G.A. Dunlop (AMDEL) and Mr Peter Bradshaw, AMDEL's proposal was amended to:

Part 1 - Preparation of the sample for metallurgical testwork
Part 2 - Metallurgical testwork (including mineralogy) on ore composite sample from MG14 plus a comparative grind on a current mill sample from the Cattlegrid deposit.

The whole programme was to be kept flexible and changed as results indicated. Therefore costing was to be on an 'as-worked' basis rather than a lump sum contract.

Approval for this test programme (Parts 1 and 2 amended) was given in telex of 16 November 1983.

Samples were received in the week ending 16 December 1983 and work on these commenced on resumption of activities in week ending 9 January 1984.

Copies of worksheets were progressively supplied to Mr Bradshaw and the work reviewed at meetings in AMDEL to determine which format programme would follow based on results obtained.

## 2. OBJECTIVES

The client defined the overall objectives of this work as:

(a) to determine by assay and mineralogy, the distribution of the major copper minerals in a suitably ground sample assuming the current grind in the concentrator operating on Cattlegrid ore.

(b) to obtain the highest copper grade and recovery in the simplest possible circuit on the basis this deposit (MG14) was of limited tonnage and would be treated through the existing Mount Gunson concentrator with possibly minor equipment changes.

Final selection and optimisation of reagent types was not an objective of this programme.

## 3. EQUIPMENT

Equipment in use was restricted to standard laboratory crushing, screening, grinding and flotation equipment and ancillaries.

The grinding unit in use was a laboratory stainless steel rod mill 305 × 238 mm with the required number of 30 mm stainless steel rods.

The flotation cell in use was the Agitair 3-in-1 LA-500 machine.

## 4. MATERIAL EXAMINED

Two separate ore samples and a water sample were received for testwork.

### 4.1 Ore Samples

These are designated in this report as:

(a) MG14 Deposit - MG14

This was received as bags of composite sample already crushed and subsampled at Mount Gunson for analysis of individual footages for Cu.

(b) Cattlegrid orebody - CGC

This was received as a crushed composite sample ex concentrator operations.

### 4.2 Water Sample

This was received as a composite water sample typical of that used in concentrator operations. This site water sample was to be used in the latter stages of testwork once test conditions had been decided upon.

## 5. ANCILLARY MATERIALS

### 5.1 Water Supply

Preliminary tests in the series used Adelaide mains tap water.

### 5.2 Flotation Reagents

The following flotation reagents were used:

| Reagent | Abbreviation | Supplier |
|---|---|---|
| Sodium Ethyl Xanthate | SEX | Union Carbide |
| Potassium Amyl Xanthate | PAX | Union Carbide |
| Substitute Z200 | SF323 | Mount Gunson Concentrator |
| Tervcic 401 Frother | T401 | ICI |
| Methyl Isobutyl Carbinol | MIBC | Shell |
| Soda Ash | $Na_2CO_3$ | ICI |
| Aerofloat 238 | AF238 | Cyanamid |

All reagents used are commercial (mining) grade.

# 6. EXPERIMENTAL PROCEDURES AND RESULTS

Each stage of the testwork when completed was discussed with Mr Bradshaw and subsequent stages amended or deleted depending on results obtained.

## 6.1 Sample Preparation for Testwork

### 6.1.1 Cattlegrid Composite - CGC

All bags of sample supplied were combined, thoroughly mixed by cross riffling and ¼ cut out as reserve sample to storage.

The remaining $^3/_4$ was again thoroughly mixed and carefully screened and stage crushed to 100% minus 1.7 mm.

This -1.7 mm material after mixing was riffled down to 10 by 2 kg charges for testwork and the remainder stored.

### 6.1.2 MG14 Deposit Sample - MG14

With uncertainty as to whether all bags received were uniform in content, each bag supplied was carefully screened and stage crushed to 100% minus 1.7 mm.

This -1.7 mm material after mixing was riffled down and ¼ held in storage. The remaining $^3/_4$ from each bag were then composited into one sample, thoroughly mixed and riffled down to 2 kg charges for testwork.

## 6.2 Preparation of Samples for Assay

In all parts of this work, all samples were prepared for assay by pulverising in the Sieb Technik.

## 6.3 Grinding Tests

All primary grinding was carried out in the rod mill at a pulp density of 60% solids by weight. Charge weight of minus 1.7 mm material was 2000 g.

Adelaide tap water was used in all grinds.

### 6.3.1 MG14 Sample

Initially grinds were completed at various times and rod loads to determine a satisfactory P80-grinding time relationship which would straddle an assumed P80 range of 200-250 µm from Mount Gunson Concentrator operations i.e. marginally finer recognising the finer grained nature of this sample.

Details and results of these tests are given in Fig. 1 which also contains the P80-grinding time relationship. For comparative grinding purposes, sizing of the feed to grind is also given in Fig. 1. Results are tabulated in Tables 1 to 4.

### 6.3.2 CGC Sample

Following the meeting with Mr Bradshaw on 29 February 1984, the decision was made not to work on this sample until the results of work on MG14 had been completed.

## 6.4 Head Samples

From grinding test products, head samples for assay were cut out from dried samples by riffling. Samples were combined, thoroughly mixed, subsampled by riffling and submitted for assay.

### 6.4.1 Assay

Results of the head assay on sample MG14 are given in Table 5.

### 6.4.2 Mineralogy

Having selected a suitable P80-grinding time relationship from Fig. 1 at 3 minutes with 12 rods for 210 μm, a trial grind was completed producing finished product for sizing, assay of size fractions and mineralogy on selected groups of size fractions to determine:

(a) major minerals present and their proportions

(b) liberation and locking characteristics mainly of the sulphide minerals

(c) association of sulphide minerals with siderite and possibilities of magnetic separation (resulting from preliminary examination in (a) and (b) above).

The results of the sizing and assay of the size fractions is given in Table 6 whilst the mineralogy is given in Appendix A (parts (a) and (b)) and B (part (c)). Photomicrographs of relevant important structure are included in Appendix A to illustrate problem areas in treating this ore.

## 6.5 Flotation Tests

All flotation testing was done in the Agitair LA-500 machine. Unless otherwise stated the impeller speed was maintained at 1000 rev./min.

Testing was done on 2000 g charges.

Adelaide tap water was used in every test.

Reagents used were all mining stock - commercial grades.

The initial rougher pulp density was nominally 30% solids, the cell being made up to a 5.7 litre mark each time.

Any makeup water required was normally added only between flotation stages. Cell sides were washed down by fine spray water as required during each test.

Frother in all tests was added to give a stable froth condition. T401 was used in early tests following concentrator practice but gave such poor froth condition, MIBC was substituted. This addition was recorded in drops and no conversion to g/t made as on a continuous basis, consumption of MIBC in a full scale mill with water recirculation will be lower than in these tests.

pH measurements were taken at every major step in the circuit and whenever a pH change was made.

Flotation times are recorded from froth stability to end of removal of concentrates.

Any conditioning stages required were done in the flotation cell.

Desliming operations were carried out using a 30 mm hydrocyclone rig. The sizings of the sand/slime split on a 4.5 mm grind are given in Fig. 2.

Details of the flowsheets and conditions used in each test completed together with comments and calculated results, are given in copies of the original worksheets W1 to W8 in Appendix C. An index of these tests is included in this Appendix.

For comparative purposes, results of these tests are summarised in Fig. 3.

As well as being totalled on the worksheets W1 to W8, reagent additions, nominal P80's and flotation times are summarised in Table 7.

At Mr Bradshaw's request (meeting 29 February 1984), all tests were restricted to roughing and scavenging stages only. No attempts were therefore made to clean concentrates to determine limiting concentrate grades.

## 6.6 Sizings of Selected Test Products

The products from the sand float in test MG14-F8 at a 6 min. grind (P80 ≈ 115 µm) were sized. Results are given in Fig. 4.

## 7. DISCUSSION OF PROCEDURES AND RESULTS

Whenever possible observations from tests and mineralogy are included along with pertinent comments which may have a bearing on the data used for design changes to the operating plant.

### 7.1 Head Samples and Sample Preparation

Moisture content of the samples averaged 1% W/W. In crushing for preparation for the test series, the samples prepared were very soft and very dusty.

There is excellent agreement between actual head assay (2.30% Cu) and the arithmetic average of the calculated test head grades (2.29% Cu).

### 7.2 Mineralogy

As shown in Appendix A, mineralogically liberation of copper sulphide minerals is very poor at a grind of P80 ≃ 210 μm. In fact liberation at this grind is almost negligible down to 75 μm and even in the -18+9 μm size fraction is only about 50% complete.

Very fine grinding is required for liberation of copper sulphides. It is estimated that a grind to 100% -45 μm would be unlikely to produce good liberation of copper minerals in the -45+9 μm size range - the best range for flotation. Therefore unless grinds better than this are achieved, tailings losses are likely to be high.

Further evidence of this is given in Table 6 where a size by size analysis of this coarse grind shows the copper fairly uniformly scattered down the size ranges with 21.5% in the -12 μm fraction (slimes).

Similarly in evaluating the heavy liquid fractions (3.3 sp.gr.) the light fraction (<3.3 sp.gr.) containing the non-sulphide gangue minerals shows constant Cu values down to 45 μm below which there is a slight decrease i.e. there is a 'background' of very finely disseminated copper minerals in the light gangue.

For silver, this latter trend is reversed with increasing liberation of silver by size (i.e. decreasing in the <3.3 sp.gr. fractions) but again the silver distribution overall follows the size fractions.

Bismuth follows the trend for silver whilst cobalt shows a marked concentration in the coarse size fractions and apparently good liberation from silicates even in these coarser sizes.

Major (≥0.1%) minerals present are:

| | | |
|---|---|---|
| Non-sulphide gangue minerals | siderite | 30-40% |
| | quartz | 20-30% |
| | mica/illite | 15-20% |
| | dolomite | 10-15% |
| | chlorite | 3-5% |
| Copper minerals | chalcocite | 1.6% (includes digenite) |
| | bornite | 1.5% |
| | chalcopyrite | 0.2% |
| Other sulphides | pyrite | 0.1% |
| | sphalerite | 0.3% |
| | carollite | 0.1% |

The major copper minerals present are chalcocite (including digenite) and bornite. Concentrates produced can be expected to be high grade but digenite slimes easily and in this state can be a very difficult copper mineral to recover by flotation.

In discussions with Mr Bradshaw, the presence of so much siderite raised the possibility of magnetic separation particularly if it were necessary to further clean copper concentrates. This was investigated and results in Appendix B show that very little of the copper and silver is associated with particles containing a moderate to high proportion of siderite. This situation would be further improved by grinding to finer sizes with subsequent less efficient magnetic separation.

Compared to earlier work in the references available to AMDEL (Section 10), this mineral distribution is quite different to that of previously tested samples. This time chalcocite (and digenite) dominate the copper minerals, siderite dominates in the non-sulphide gangue minerals and liberation at the coarse grind approaching current plant practice is extremely poor.

For maximum recovery by flotation, the following points need to be considered:

(a) the correct grind is needed which liberates the copper sulphides sufficiently without sliming the softer minerals (particularly digenite)

(b) the siderite/dolomite/chlorite content means that the bulk of the material will grind very easily

(c) since the 'background' copper mineral content of the <3.3 sp.gr. fraction must be with the harder quartz and other silicates (?), tailings losses can be expected to be high

(d) the 'sheet silicate'/clay content may pose viscosity problems but certainly will affect concentrate grades if not controlled.

## 7.3 Grinding

No power consumptions were determined as neither full Bond Grindabilities nor comparative grindabilities with the Cattlegrid sample supplied were carried out at Mr Bradshaw's request.

From the grinding tests done, the MG14 sample tested is extremely soft as shown from the following tabulation of 80% passing size with grinding time and a varying rod load:

| Sample | 80% Passing | Size (µm) | Rod Load* (No. of Rods) | Grinding Time (mins) |
|---|---|---|---|---|
| MG14 | F80 | 1250 | 0 | 0 |
| | P80 | 127 | 17 | 3 |
| | | 210 | 12 | |
| | | 78 | 17 | 5 |
| | | 133 | 12 | |
| | | 87 | 12 | 8 |
| | | 51 | 17 | |

i.e. a very steep reduction stage occurs in minimum time (3 minutes with light rod load) and continues even at double that time.

This almost certainly means that the softer minerals are easily ground and in the longer time scale, only the quartz and harder silicates remain for grinding.

This immediately poses a problem in liberating the 'background' copper mineral content of these in the grinding circuit as the 'heavy media' effect in cyclone classification will tend to overgrind liberated minerals thereby worsening the slime problems in flotation.

Viscosity problems if present posed no restrictions on materials handling at the 60% pulp density used. This may not be the case at normal plant levels and needs to be checked out.

## 7.4 Flotation

### 7.4.1 Individual Test Details and Discussion

#### Test F1

This was the initial test exploring a coarse grind (P80 ≃ 210 µm) to follow through on the mineralogy (at this grind) and the reagent regime practised in the concentrator.

Immediately flotation began problems were apparent in that:

---

*17 rods is normal full load.

(a) no froth was obtained with T401 until a large quantity of MIBC was substituted

(b) little concentrate floated in spite of preguessing that collector addition required would be higher than the YTD concentrator figures of SEX = 0.014 kg/t and SF323 = 0.005 kg/t. In fact heavy additions of PAX were required to obtain concentrates in any quantity.

The test was abandoned once levels of reagents for flotation were established.

Natural pH of the flotation feed is 8.6-8.7 in Adelaide tap water.

Test F2

With the same grind (P80 ≈ 210 µm) and heavy reagent additions of:

PAX - 450 g/t - staged to grind and floats
MIBC- to suit float

staged concentrates were taken off at various times (2, 4, 6 and 8 minutes). Good sulphides floated rapidly to the rougher concentrate (34.8% Cu for 36.8% Cu recovery) but remaining sulphides floated slowly and continuously until test end.

Bulk concentrate grade produced was 19.9% Cu for 66% Cu recovery. Although not tried this would easily clean up to high grade but the slow flotating nature of the scavenged material probably means a significant drop in recovery at this grind size i.e. composites will drop out or control final grade at maximum recovery.

Test F3

With a finer grind (P80 ≈ 142 µm) and same reagent and flotation conditions of Test F2, staged concentrates were again taken off.

Sulphides floated rapidly into the rougher concentrate (32.3% Cu for 41.1% Cu recovery) but then slowed in the scavengers to give a bulk concentrate of 18.5% Cu for 68.9% Cu recovery.

Test F4

In tests F2 and F3 the 3rd scavenger stage recovered very little Cu in 8 minutes (3.6-3.2% Cu), so this was dropped in this test with a still finer grind of P80 ≈ 115 µm and other conditions constant.

Sulphides floated very quickly into the rougher concentrate (32.5% Cu for 47.7% Cu recovery) and visually appeared faster in the two scavenger concentrates to give a bulk concentrate of 19.4% Cu for 71% Cu recovery in a 3 stage float (15 mins cmf 22 mins).

## Summary Tests F2 to F4

Heavy reagent additions are required for good copper recovery.

In spite of leaving out the 3rd scavenger in Test F4, equivalent recovery at similar Cu grades approaches 75% at P80 ≈ 115 μm.

In Fig. 5 (recovery Vs P80), recovery increases markedly as the grind becomes finer. Between 210 μm and 140 μm change is minor but is accelerating towards 115 μm and beyond i.e. copper minerals are increasingly liberated with the finest grind used without affecting bulk concentrate grade and at this point sliming is not a problem.

Recovery increases by additional flotation stages are unlikely as the lines in Fig. 5 for scavenger concentrates 2 and 3 are very close together and almost parallel. At 22 minutes, this is beyond the existing capacity of the concentrator at 30-35% solids range.

Still finer grinds will increase recovery and probably won't decrease bulk concentrate grade.

Because the requirement was to stay within range of the concentrator grinding capacity, still finer grinds were not investigated (Meeting 29 February, 1984, Mr Bradshaw).

## Test F5

Using the intermediate grind of Test F3 (P80 ≈ 142 μm), pH of flotation feed was increased from natural of 8.6/8.7 to 9.1 with 500 g/t $Na_2CO_3$ aiming for increased dispersion and an effect from pH.

Results are inferior to those obtained at natural pH even at the coarsest grind.

## Test F6

This test followed the general procedure of Test F3 (P80 ≈ 142 μm) but with significantly less PAX (250 Vs 450 g/t) and introduced 30 g/t SF323 staged through the float.

As might be expected, concentrate grades are significantly higher but overall recovery is some 15% lower (bulk concentrate grade 27.5% Cu at 55.2% Cu recovery).

## Summary Tests T5 and F6

Change in pH with $Na_2CO_3$ is disastrous.

Lower collector additions are possible when an adequate grind for maximised copper mineral liberation is obtained. Obviously the higher collector additions in Tests F2 to F4 are pulling composites.

There is a tradeoff situation between collector requirements and degree of grind used for maximum recovery.

## Test F7

This test examined separate flotation of sand and slime fractions after grinding (P80 ≃ 142 μm) using the general procedure for the total float on each fraction.

No SF323 was used on the sand float. Overall PAX addition is lower at 300 g/t.

The sand float gives very high concentrate grades with the overall sand bulk concentrate running 31.3% Cu for 53.8% Cu recovery (73.1% Cu recovery in sand float representing 63.5% by weight holding 73.6% of total copper).

The D50 of the hydrocyclone split is nominally 10 μm.

Recovery of copper in the slime float is poor with slow uniformly floating copper in each stage to a bulk concentrate of 7.1% Cu for an additional 8.7% Cu recovery (32.8% Cu recovery in slime float from 36.5% by weight holding 26.4% of total copper).

## Test F8

This test examined the possibility of desliming after grinding, rejecting the slime fraction and floating only the sands. On Mr Bradshaw's instructions overall grind was finer at 115 μm. Flotation time was also restricted to 15 minutes.

Again sulphides floated rapidly and easily in the rougher stage but dragged on through the two scavenger stages used.

Bulk concentrate grade was 23.9% Cu for 64.9% total Cu recovery in the sand fraction representing 79.3% by weight and holding 86.5% of the total Cu. On the sand float alone this result represents a 75% recovery of Cu in sand float feed.

Losses in the deslimed fraction were 13.5% Cu in 20.7% by weight which together with the sand float tail makes the total test tail 93.8% by weight running 0.86% Cu for 35.1% Cu distribution (loss).

Although the D50 was not checked out at this grind, the hydrocyclone operation was more efficient (produced less slime from a finer grind) than in Test F7.

## Summary Tests F7 and F8

If the results on the slime float in Test F7 are applied to the deslimed fraction in F8, then overall performance at the finer grind of P80 ≃ 115 μm using separate sand and slime floats can be approximated as:

| Product: | % Wgt | % Cu | % Cu Distribution | Basis |
|---|---|---|---|---|
| Bulk Sand Conc. | 6.23 | 23.9 | 64.9 | F8 |
| Bulk Slime Conc. | 1.73 | 6.0 | 1/3 × 13.5 | F7/F8 |
| Total Conc. | 7.96 | 20.0 | 69.4 | |

i.e. very close to the best test F4 in the series so far.

From all the results to date on this sample, there is no logical metallurgical reason to go to a sand/slime circuit as the same result can be achieved in a straight through float in less equipment.

For a final decision, financial analysis is required to decide on $ returns in treating only the sand fraction, throwing slimes away and this is also dependent on the efficiency in hydrocyclone separation which can be achieved at plant scale.

### 7.4.2 Materials Handling Properties

At the pulp densities used, no problems were experienced with pulp viscosity. Tailings (and feed samples) did not settle even overnight. In fact samples from the cell were evaporated to dryness before sample separation as they could neither be settled or filtered.

Magnafloc 351 was initially tried on float tailings as a settling aid but without success (>200 g/t).

For plant water recovery through the tailings system, a separate test programme on tailings etc. is required.

## 8. CONCLUSIONS

The following conclusions are drawn from the test programme so far completed on this sample from the MG14 deposit.

(1) Mineralogically:

(a) the major (≥0.1%) minerals present are:

| | | |
|---|---|---|
| Non-sulphide gangue | Siderite | 30-40% |
| | Quartz | 20-30% |
| | Mica/Illite | 15-20% |
| | Dolomite | 10-15% |
| | Chlorite | 3-5% |
| Copper | Chalcocite | 1.6% |
| | Bornite | 1.5% |
| | Chalcopyrite | 0.2% |
| Other Sulphides | Sphalerite | 0.3% |
| | Pyrite | 0.1% |
| | Carollite | 0.1% |

(b) liberation of copper sulphide minerals is very poor at a grind of P80 ≃ 210 μm chosen to be close to current plant operations knowing mineralisation in MG14 is finer grained.
In fact liberation at this grind is almost negligible down to 75 μm and even in the -18+9 μm size fraction is only about 50% complete.

(c) Very fine grinding is required for liberation of copper sulphides. A grind to 100% -45 μm would be unlikely to produce good liberation of these minerals even in the -45+9 μm size range.

(d) For the grind of P80 ≃ 210 μm. A size by size analysis shows both the Cu and Ag uniformly distributed down the size ranges according to % weight with the -12 μm fraction (slimes) holding 21.5% and 18.2% respectively.
Heavy liquid separations (3.3 sp.gr.) reveal almost constant Cu values in the light fraction (<3.3) down the size ranges to 45 μm below which there is a slight decrease i.e. there is a 'background' of very finely disseminated copper minerals in the light gangue. The reverse is true for silver values and while bismuth follows the trend for silver, cobalt shows a marked concentration in the coarse size ranges and apparently good liberation from silicates even in these coarser sizes.

Conclusions (Continued)

(e) Since with magnetic separation very little of the copper and silver is associated with particles containing a moderate to high proportion of siderite, the silicates contain this 'background' copper and are the hardest to grind for effective liberation.

(2) In grinding the ore is soft but at Mr Bradshaw's request, no comparative grinds were done with the Cattlegrid sample supplied.

| F80 (µm) | P80 (µm) | Rod Load* | Grinding Time (min) |
|---|---|---|---|
| 1250 | - | 0 | 0 |
| | 127 | 17 | 3 |
| | 78 | | 5 |
| | 51 | 12 | 8 |
| | 210 | | 3 |
| | 133 | | 5 |
| | 87 | | 8 |

Such steep reduction stages in minimum time mean that the softer minerals are easily ground, only quartz and harder silicates remaining for grinding. Liberation of the 'background' copper mineral content is therefore a real problem as over-grinding is a distinct possibility in attempting maximum copper liberation.

(3) Viscosity problems are not apparent at any stage of treatment.

(4) In flotation:

(a) Cu recovery improves with fineness of grind with the Cu grade of a bulk rougher/scavenger concentrate remaining relatively constant. At P80 ≈ 115 µm, Cu recovery after rougher and 3 scavenging stages approaches 75% at 18-19% Cu grade.
Cu recovery increases markedly as the grind becomes finer at 115 µm and sliming is not a problem.
Because the requirement was to stay within range of the concentrator grinding capacity, still finer grinds were not investigated.

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*Normal rod load is 17 rods.

Conclusions (Continued)

(b) Natural pH of flotation feed is 8.6-8.7 in Adelaide tap water. Changing the pH to 9.1 with soda ash was disastrous.

(c) Heavy reagent additions are required to achieve the Cu recovery down to the P80 ≈ 115 µm grind. Collector additions ranged from 250 to 450 g/t but recovery is lost at the lower levels although concentrate grades are higher.
Of those tried, PAX is the better collector and no float resulted with T401, MIBC being used successfully as frother.
Obviously at these grinds, the heavier additions are pulling composite particles.
With a still finer grind and adequate Cu mineral liberation, a trade off situation between collector requirements and degree of grind used will exist for maximum recovery.

(d) There is no logical metallurgical reason to go to a sand/slime circuit after grinding as the same result can be achieved in a straight through float using less equipment.
A financial analysis will decide whether it is worth treating only the sand fraction after desliming flotation feed and rejecting slimes straight to tailings. Bulk concentrate from such a sand float was 23.9% Cu for 64.9% total Cu recovery, the sand fraction representing 79.3% by weight and holding 86.5% of the total Cu. (i.e. 75% Cu recovery in sand float feed). This analysis will also be dependent on the efficiency in hydrocyclone operation achievable at plant scale.

(e) At these coarse grinds, total flotation times are long (15 to 22 mins) for maximised Cu recovery.

(5) Realistic pulp densities were in use in flotation stages. Any attempt to decrease these would increase flotation times and considerably worsen the dewatering characteristics of plant products. Laboratory tailings could not be settled or filtered. This problem was not investigated.

(6) The best test result is from Test MG14 - F4, a straight through float producing a bulk rougher and scavenger concentrates under conditions of:

P80 ≈ 115 μm
PAX ≈ 400 g/t
Float time ≈ 15 minutes (3 rougher, 4-1st scavenger, 8-2nd scavenger)
result ≈ 19.4% Cu grade for 71% Cu recovery.

Based on other tests and an additional scavenger stage it is projected that maximum recovery will approach 75% Cu in 22 minutes flotation time and 450 g/t PAX for 18+% Cu grade.

## 9. RECOMMENDATION

The next step recommended in the overall programme is to complete a financial analysis using results from this work and from earlier work on other samples.

It will be necessary to take the following into account assuming that there is a limit to capital expenditure on plant changes/expansion to treat what is a relatively small tonnage of MG14 ore:

(a) the power requirements for grinding MG14 samples for adequate copper mineral liberation.

(b) a straight roughing/scavenging circuit at conventional pulp densities with necessary cleaning stages to follow

(c) high reagent additions for maximum copper recovery. There may be a trade off between grinding requirements and collector additions depending on just how soft the MG14 ore is in conparison to the currently treated Cattlegrid ore.

(d) whether by (b) it is better to treat the sand fraction after desliming the flotation feed and rejecting slimes to tailing. This may allow maximum tonnage through existing plant with minimum capital outlay - one desliming cyclone setup - particularly if MG14 ore is considerably softer to grind than Cattlegrid ore.

Either way flotation times and reagnet additions are similar. Further testwork can then be programmed.

## 10. REFERENCES

(1) REPORT ON PROCESSING TEST WORK - MG14 AREA, MT GUNSON. Memorandum R.N. Dudgeon - R.L. Stainlay, January 1976.

(2) (a) PRELIMINARY APPRAISAL OF CORE SAMPLES FROM THE MG14 AREA PLUS ATTACHMENTS. Memorandum R.L. Muller - R.N. Dudgeon, 30 October, 1974.

(b) PETROGRAPHIC AND MINERAGRAPHIC EXAMINATION OF GULLY AND MG14 AREAS DRILL CORE SAMPLES MT GUNSON PMR 138/35 - RELEVANT MG14 SECTION PACMINEX PTY LIMITED, P.J. Curtis, September 1975.

(3) THE MG14 DEPOSIT - A METALLURGICAL APPRAISAL. P.G. Bradshaw, August 1983.

## The Australian Mineral Development Laboratories

TABLE 1

### Sizing Report 19/ 1/84

Client/Project No. 6191/84
Job No.
Sample MG 14 3MIN GRIND 12 RODS
Run No. Date
Submitted by GAD Date

Screen Set GS
Wet Screened on 45. microns
Screening Time 15.0 mins
Sample Weight 399.93 g
Tested by NG on 19.1.84

### Screen Sizing

| BSS Mesh | Nominal Size | Wt. (g) Retained | Wt. % Retained | Cumulative Percent Retained | Cumulative Percent Passing |
|---|---|---|---|---|---|
| 22 | 710 microns | 0.50 | 0.13 | 0.13 | 99.87 |
| 30 | 500 | 5.75 | 1.44 | 1.57 | 98.43 |
| 44 | 355 | 22.65 | 5.66 | 7.23 | 92.77 |
| 60 | 250 | 30.57 | 7.64 | 14.87 | 85.13 |
| 85 | 180 | 35.82 | 8.96 | 23.83 | 76.17 |
| 120 | 125 | 33.94 | 8.49 | 32.32 | 67.68 |
| 170 | 90 | 25.83 | 6.46 | 38.78 | 61.22 |
| 200 | 75 | 14.71 | 3.68 | 42.46 | 57.54 |
| 240 | 63 | 12.30 | 3.08 | 45.54 | 54.46 |
| 300 | 53 | 7.22 | 1.81 | 47.35 | 52.65 |
| 350 | 45 | 7.50 | 1.88 | 49.23 | 50.77 |
| -350 | -45 | | | 100.00 | |

Undersize Wt. Recovered 203.14 g
Undersize Wt. by Diff. 203.14 g
Unaccounted loss percent -0.00

### Cyclosizing

Sample Weight 105.70 g

| Nominal Size Microns | Corrected Size Microns | | | | |
|---|---|---|---|---|---|
| 45.0 | 34.9 | 4.28 | 2.06 | 51.29 | 48.71 |
| 33.5 | 26.0 | 8.80 | 4.23 | 55.52 | 44.48 |
| 23.5 | 18.2 | 13.81 | 6.64 | 62.16 | 37.84 |
| 15.5 | 12.0 | 15.82 | 7.60 | 69.76 | 30.24 |
| 12.0 | 9.3 | 11.00 | 5.29 | 75.05 | 24.95 |
| -12.0 | -9.3 | 51.99 | 24.98 | 100.00 | |

| | | Correction Factor |
|---|---|---|
| Temperature | 24.0 deg C | 0.954 |
| Elutriation Time | 15 mins | 0.939 |
| Flowmeter Reading | 180 mm | 1.000 |
| Sample S.G. | 3.2 | 0.866 |
| | | |
| Overall Correction Factor | | 0.776 |

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TABLE 2

## Sizing Report 5/ 3/84

Client/Project No. 6191/84
Job No.
Sample MG14 4.5MIN. GRIND 12 Rods
Run No. Date
Submitted by GAD Date

Screen Set GS
Wet Screened on 45. microns
Screening Time 15.0 mins
Sample Weight 119.00 g
Tested by NG on 5.3.84

### Screen Sizing

| BSS Mesh | Nominal Size | Wt. (g) Retained | Wt. % Retained | Cumulative Percent Retained | Passing |
|---|---|---|---|---|---|
| 44 | 355 microns | 2.68 | 2.25 | 2.25 | 97.75 |
| 60 | 250 | 5.82 | 4.89 | 7.14 | 92.86 |
| 72 | 212 | 3.85 | 3.24 | 10.38 | 89.62 |
| 85 | 180 | 3.75 | 3.15 | 13.53 | 86.47 |
| 100 | 150 | 5.74 | 4.82 | 18.35 | 81.65 |
| 120 | 125 | 3.08 | 2.59 | 20.94 | 79.06 |
| 150 | 106 | 4.86 | 4.08 | 25.02 | 74.98 |
| 170 | 90 | 3.47 | 2.92 | 27.94 | 72.06 |
| 200 | 75 | 4.64 | 3.90 | 31.84 | 68.16 |
| 240 | 63 | 3.90 | 3.28 | 35.12 | 64.88 |
| 300 | 53 | 3.07 | 2.58 | 37.70 | 62.30 |
| 350 | 45 | 1.90 | 1.60 | 39.30 | 60.70 |
| -350 | -45 | | | 100.00 | |

## The Australian Mineral Development Laboratories

TABLE 3

### Sizing Report 5/ 3/84

Client/Project No. 6191/84
Job No.
Sample MG14 6MIN. GRIND 12 Rods
Run No. Date
Submitted by GAD Date

Screen Set GS
Wet Screened on 45. microns
Screening Time 15.0 mins
Sample Weight 113.96 g
Tested by NG on 5.3.84

### Screen Sizing

| BSS Mesh | Nominal Size | Wt. (g) Retained | Wt. % Retained | Cumulative Percent Retained | Passing |
|---|---|---|---|---|---|
| 44 | 355 microns | 0.75 | 0.66 | 0.66 | 99.34 |
| 60 | 250 | 3.25 | 2.85 | 3.51 | 96.49 |
| 72 | 212 | 2.73 | 2.40 | 5.91 | 94.09 |
| 85 | 180 | 3.11 | 2.73 | 8.64 | 91.36 |
| 100 | 150 | 4.80 | 4.21 | 12.85 | 87.15 |
| 120 | 125 | 3.54 | 3.11 | 15.96 | 84.04 |
| 150 | 106 | 4.70 | 4.12 | 20.08 | 79.92 |
| 170 | 90 | 3.53 | 3.10 | 23.18 | 76.82 |
| 200 | 75 | 4.51 | 3.96 | 27.14 | 72.86 |
| 240 | 63 | 4.04 | 3.55 | 30.69 | 69.31 |
| 300 | 53 | 3.11 | 2.73 | 33.42 | 66.58 |
| 350 | 45 | 1.98 | 1.74 | 35.16 | 64.84 |
| -350 | -45 | | | 100.00 | |

## The Australian Mineral Development Laboratories

### TABLE 4 Sizing Report 19/ 1/84

Client/Project No. 6191/84
Job No.
Sample MG 14 8 MIN GRIND 12 ROD
Run No. Date
Submitted by GAD Date

Screen Set GS
Wet Screened on 45. microns
Screening Time 15.0 mins
Sample Weight 228.04 g
Tested by NG on 19.1.84

### Screen Sizing

| BSS Mesh | Nominal Size | Wt. (g) Retained | Wt. % Retained | Cumulative Percent Retained | Cumulative Percent Passing |
|---|---|---|---|---|---|
| 30 | 500 microns | 0.04 | 0.02 | 0.02 | 99.98 |
| 44 | 355 | 0.65 | 0.29 | 0.31 | 99.69 |
| 60 | 250 | 2.94 | 1.29 | 1.60 | 98.40 |
| 85 | 180 | 8.70 | 3.82 | 5.42 | 94.58 |
| 120 | 125 | 14.70 | 6.45 | 11.87 | 88.13 |
| 170 | 90 | 16.01 | 7.02 | 18.89 | 81.11 |
| 200 | 75 | 10.51 | 4.61 | 23.50 | 76.50 |
| 240 | 63 | 9.12 | 4.00 | 27.50 | 72.50 |
| 300 | 53 | 5.80 | 2.54 | 30.04 | 69.96 |
| 350 | 45 | 5.64 | 2.47 | 32.51 | 67.49 |
| -350 | -45 |  |  | 100.00 |  |

Undersize Wt. Recovered 153.93 g
Undersize Wt. by Diff. 153.93 g
Unaccounted loss percent 0.00

# TABLE 5: MG14 HEAD ASSAY

| Element | | Assay |
|---|---|---|
| % | Cu | 2.30 |
| % | Pb | 0.07 |
| % | Zn | 0.17 |
| % | Fe | 13.2 |
| % | S | 1.14 |
| % | Co | 0.05 |
| % | Ni | 0.01 |
| % | Bi | 0.02 |
| g/t | Ag | 22 |
| g/t | Au | 0.06 (In Triplicate Portions) |

Emission Scan

Semi-Quantitative Spectrographic Analysis Schemes A1,A2,A3,A4,A5,A7 & A9
Results in ppm unless otherwise stated. Detection limits in brackets

| ELEMENT | | Value | | | | | | | ELEMENT | | Value | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| A1 * | Ba (200) | X | | | | | | | A2 * | In (10) | X | | | | | | |
| | Be (1) | 3 | | | | | | | | Pb (1) | 400 | | | | | | |
| | Ce (300) | X | | | | | | | | Sb (30) | X | | | | | | |
| | Co (5) | 750 | | | | | | | | Sn (1) | X | | | | | | |
| | Cr (20) | 80 | | | | | | | | Zn (20) | 1500 | | | | | | |
| | La (50) | X | | | | | | | | | | | | | | | |
| | Mn (10) | >10000 | | | | | | | A3 * | Au (3) | X | | | | | | |
| | Mo (3) | 10 | | | | | | | | P (100) | 100 | | | | | | |
| | Nb (20) | X | | | | | | | | Te (20) | X | | | | | | |
| | Ni (5) | 150 | | | | | | | | Tl (1) | X | | | | | | |
| | Sc (3) | 10 | | | | | | | | | | | | | | | |
| | Sr (50) | 100 | | | | | | | A4 * | Li (1) | 60 | | | | | | |
| | Ta (100) | X | | | | | | | | Na (50) | 1500 | | | | | | |
| | Th (100) | X | | | | | | | | | | | | | | | |
| | Ti (100) | 4000 | | | | | | | A5 * | Cs (30) | X | | | | | | |
| | V (10) | 80 | | | | | | | | K (5) | >10000 | | | | | | |
| | W (50) | X | | | | | | | | Rb (10) | 400 | | | | | | |
| | Y (10) | 10 | | | | | | | | | | | | | | | |
| | Yb (1) | 1 | | | | | | | | | | | | | | | |
| | Zr (10) | 150 | | | | | | | | | | | | | | | |
| | | | | | | | | | A7 | B (3) | nd | | | | | | |
| A2 * | Ag (0.1) | 20. | | | | | | | | | | | | | | | |
| | As (50) | X | | | | | | | A9 | Al (100) | nd | | | | | | |
| | Bi (1) | 80 | | | | | | | | Ca (100) | nd | | | | | | |
| | Cd (3) | X | | | | | | | | Fe (100) | nd | | | | | | |
| | Cu (1) | >10000 | | | | | | | | Mg (100) | nd | | | | | | |
| | Ga (1) | 10 | | | | | | | | Si (100) | nd | | | | | | |
| | Ge (1) | X | | | | | | | | | | | | | | | |

Results are semi-quantitative. Elements apparently present in concentrations of economic interest should be redetermined by an appropriate accurate analytical technique. X = Not detected at limit quoted

TABLE 6: TEST MG14 - FEED, GROUND 3 MIN.
METALLURGICAL BALANCE

| Product µm | Weight | | Assay % or g/t* | | | | Unit | | | | Distribution % | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | CUM. Ret. | | | | CUM. Ret. | | | | CUM. Ret. | |
| XXXXXXXXXXXX | % | CUM % | Cu | Ag* | Cu | Ag | Cu | Ag | Cu | Ag | Cu | Ag | Cu | Ag |
| + 500. | 1.57 | | 1.27 | 12 | | | 1.99 | 18.84 | | | 0.9 | 0.8 | | |
| − 500 + 355 | 5.66 | 7.23 | 1.64 | 16 | 1.56 | 15 | 9.29 | 90.56 | 11.28 | 109.40 | 4.0 | 3.9 | 4.9 | 4.7 |
| − 355 + 250 | 7.64 | 14.87 | 1.82 | 17 | 1.69 | 16 | 13.91 | 129.88 | 25.19 | 239.28 | 6.1 | 5.7 | 11.0 | 10.4 |
| − 250 + 180 | 8.96 | 23.83 | 1.97 | 18 | 1.80 | 17 | 17.65 | 161.28 | 42.84 | 400.56 | 7.6 | 7.0 | 18.6 | 17.4 |
| − 180 + 125 | 8.49 | 32.32 | 2.35 | 21 | 1.94 | 18 | 19.45 | 178.29 | 62.79 | 578.85 | 8.7 | 7.8 | 27.3 | 25.2 |
| − 125 + 90 | 6.46 | 38.78 | 2.75 | 25 | 2.08 | 19 | 17.77 | 161.50 | 80.56 | 740.35 | 7.7 | 7.0 | 35.0 | 32.2 |
| − 90 + 75 | 3.68 | 42.46 | 3.0 | 26 | 2.16 | 20 | 11.04 | 95.68 | 91.60 | 836.03 | 4.8 | 4.2 | 39.8 | 36.4 |
| − 75 + 63 | 3.08 | 45.54 | 3.25 | 29 | 2.23 | 20 | 10.01 | 89.32 | 101.61 | 925.35 | 4.4 | 3.9 | 44.2 | 40.3 |
| − 63 + 53 | 1.81 | 47.35 | 3.60 | 33 | 2.28 | 21 | 6.52 | 59.73 | 108.13 | 985.08 | 2.8 | 2.6 | 47.0 | 42.9 |
| − 53 + 45 | 1.88 | 49.23 | 3.80 | 34 | 2.34 | 21 | 7.14 | 63.92 | 115.27 | 1049.00 | 3.1 | 2.7 | 50.1 | 45.6 |
| (Calc + 45). | (49.23) | | (2.34) | (21) | | | (115.27) | (1049.00) | | | (50.1) | (45.6) | | |
| Actual − 45. | 50.77 | | 2.35 | 25 | | | 119.31 | 1269.25 | | | (49.9) | (54.4) | | |
| (Calc Head) | (100.00) | | (2.35) | 23 | | | (234.58) | (2318.25) | | | 100.0 | 100.0 | | |
| −45 + C1 | 2.06 | 51.29 | 4.10 | 74 | 2.41 | 23 | 8.45 | 152.44 | 123.72 | 1201.44 | 3.7 | 6.6 | 53.8 | 52.2 |
| C2 | 4.23 | 55.52 | 2.8 | 35 | 2.44 | 24 | 11.84 | 148.05 | 135.56 | 1349.49 | 5.1 | 6.4 | 58.9 | 58.6 |
| C3 | 6.64 | 62.16 | 2.85 | 27 | 2.49 | 25 | 18.92 | 179.28 | 154.48 | 1528.77 | 8.3 | 7.8 | 67.2 | 66.4 |
| C4 | 7.60 | 69.76 | 2.20 | 20 | 2.45 | 24 | 16.72 | 152.00 | 171.20 | 1680.77 | 7.2 | 7.5 | 74.4 | 73.9 |
| C5 | 5.29 | 75.05 | 1.79 | 18 | 2.41 | 24 | 9.47 | 95.22 | 180.67 | 1775.99 | 4.1 | 7.9 | 78.5 | 81.8 |
| −C5* | 24.96 | | 1.48 | 23 | | | 36.96 | 574.08 | | | — | — | | |
| (Calc −C5) | (24.96) | | (1.98) | (21). | | | (49.33) | (574.01) | | | (21.5) | (18.2) | | |
| Calc Head*. | (100.00) | | (2.18) | (24). | | | (217.63) | (2350.01) | | | — | — | | |
| Actual Head. | | | 2.30 | 22. | | | | | | | | | | |
| Average Head | | | 2.32 | 23. | | | 232.00 | 2300.0 | | | 100.0. | 100.0 | | |

* On the Cyclosizer recovery of −C5 is not 100% — Fraction submitted contains only those fines recoverable in centrifuge. ∴ Back calculated actual −C5 using information from head assay and ± 45µm fractions — the Average head value.

TABLE 7: REAGENT ADDITIONS, P80's(1) AND FLOAT TIMES

| Test No. | P80 (µm) | Reagent Additions (g/t) | | | | | | Times (min) | |
|---|---|---|---|---|---|---|---|---|---|
| | | $Na_2CO_3$ | PAX | AF238 | SF323 | T401 | MIBC | Ro | Scav. |
| MG14 - F1 | 210 | .................. Abandoned Test .................. | | | | | | | |
| MG14 - F2 | 210 | - | 450 | - | - | Tried | ✓ | 2 | 20 |
| MG14 - F3 | 142 | - | 450 | - | - | - | ✓ | 2 | 20 |
| MG14 - F4 | 115 | - | 400 | - | - | - | ✓ | 3 | 12 |
| MG14 - F5 | 142 | 500 | 450 | (2) | - | - | ✓ | 2 | 20 |
| MG14 - F6 | 142 | - | 250 | - | 30 | - | ✓ | 2 | 20 |
| MG14 - F7 | 142 | - | 300 | - | 15 | - | ✓ | 2+2(3) | 18+20 |
| MG14 - F8 | 115 | - | 405 | - | - | - | ✓ | 3+0(4) | 12+0 |

Notes:
(1) From Fig. 1 using P80 - Time Relationship for 12 rods in charge

(2) Tried Additional $Na_2CO_3$ (500 g/t to pH ≈ 9.5), 50 g/t AF238 and 150 g/t PAX after last normal scavenger stage

(3) Sand float + slime float

(4) Sand float, deslimed

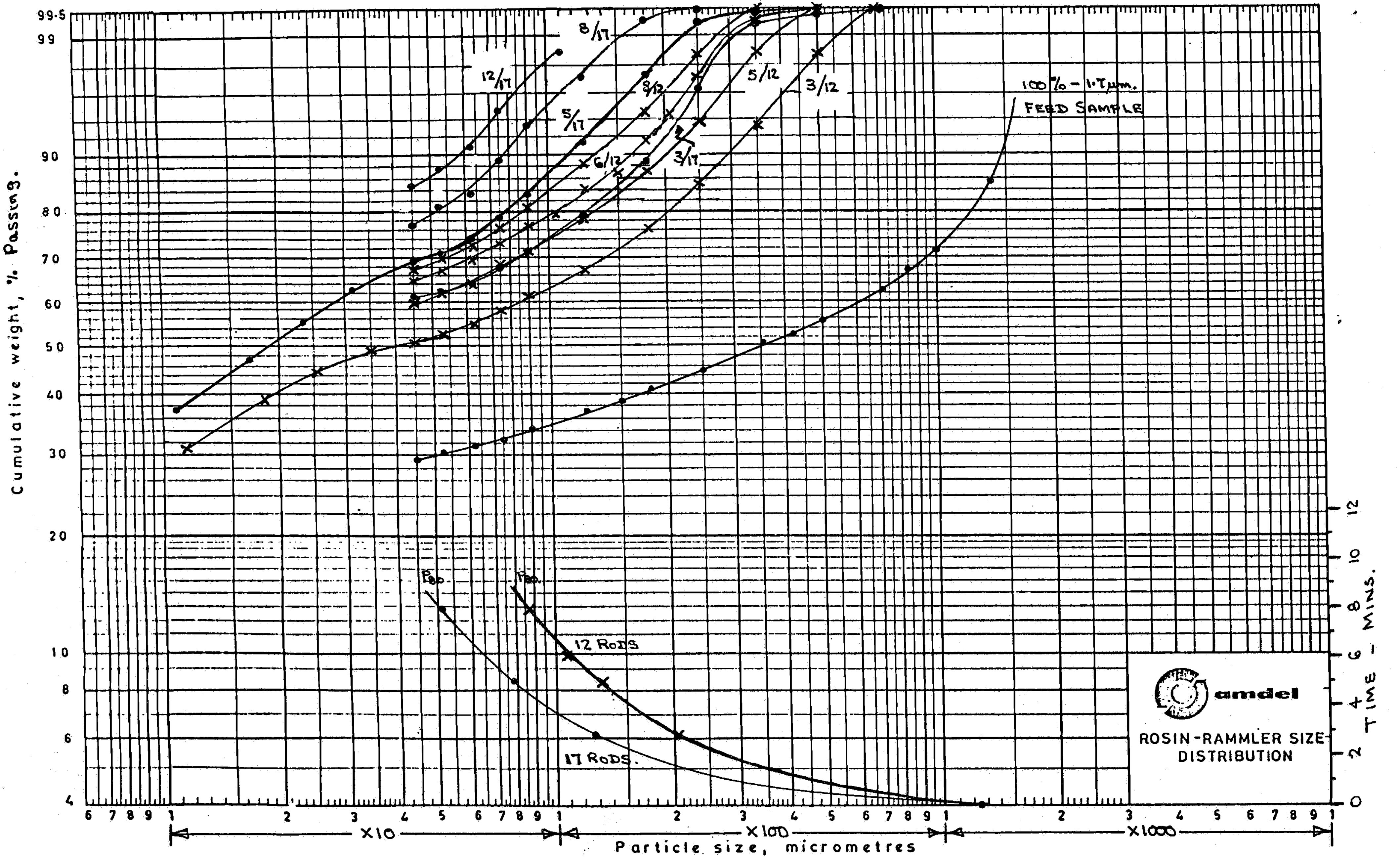


FIG. 1 : GRIND SIZES AT VARIOUS GRINDING TIMES _ P80 CURVES PLOTTED.
SAMPLE MG14 COMPOSITE _ 2 SETS OF ROD LOADS, 12 RODS (X) AND 17 RODS (•).

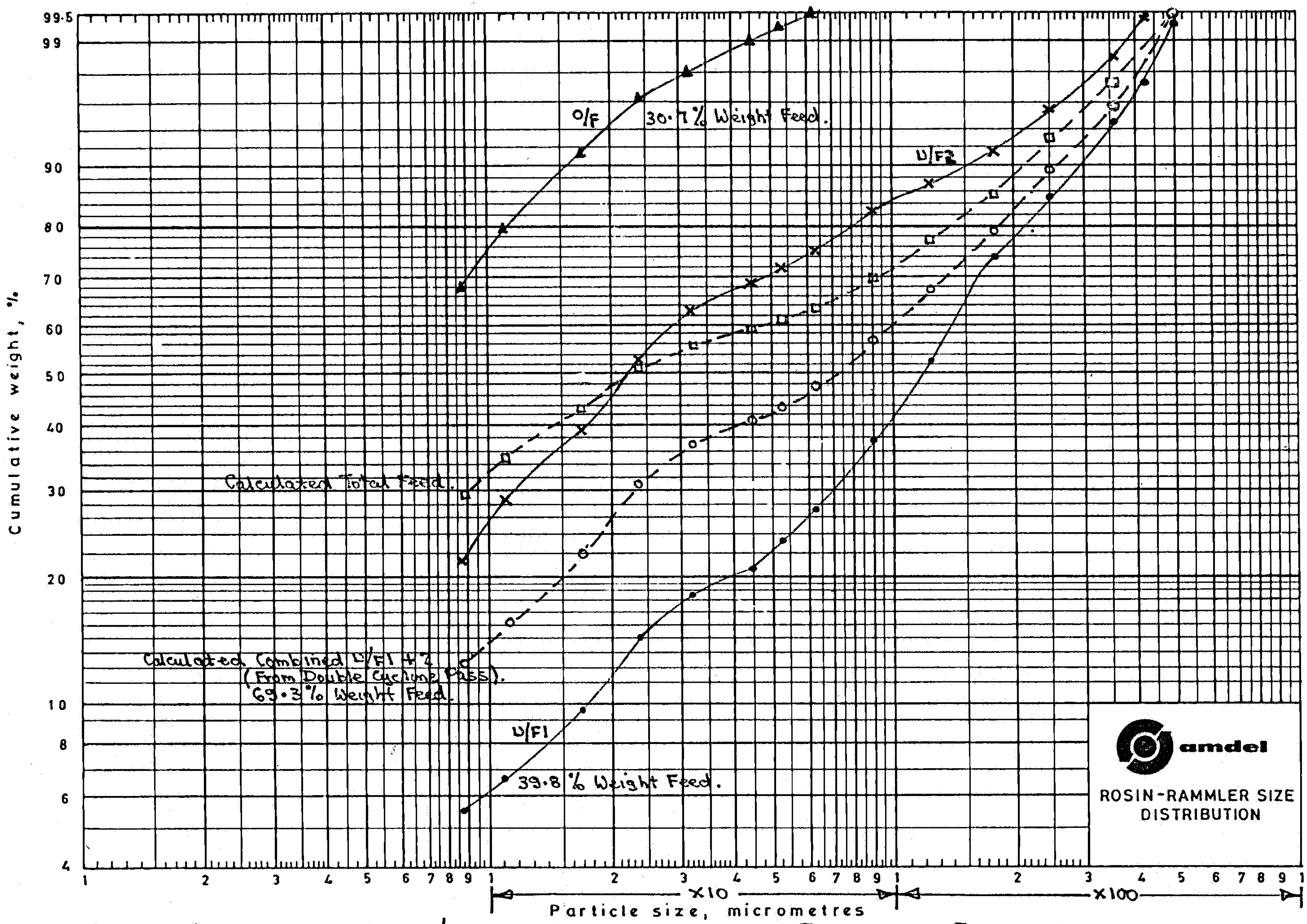


FIG. 2 : SIZINGS _ SAND/SLIME SPLIT ON GROUND FLOTATION FEED _ GRIND 4·5 MIN.
Calculated D50 = 10μm.

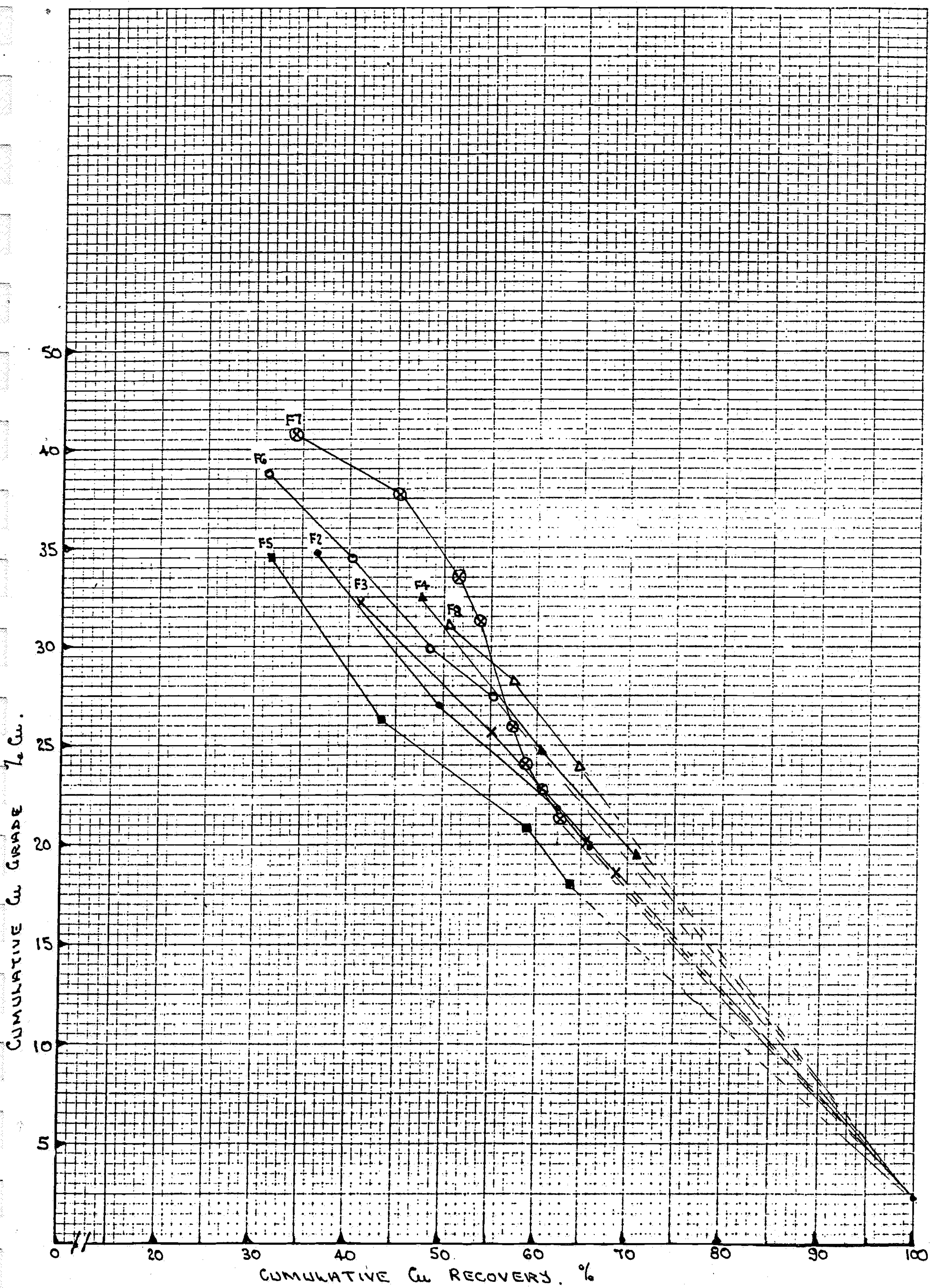


FIG. 3: METALLURGICAL PERFORMANCE – SAMPLE MG14.

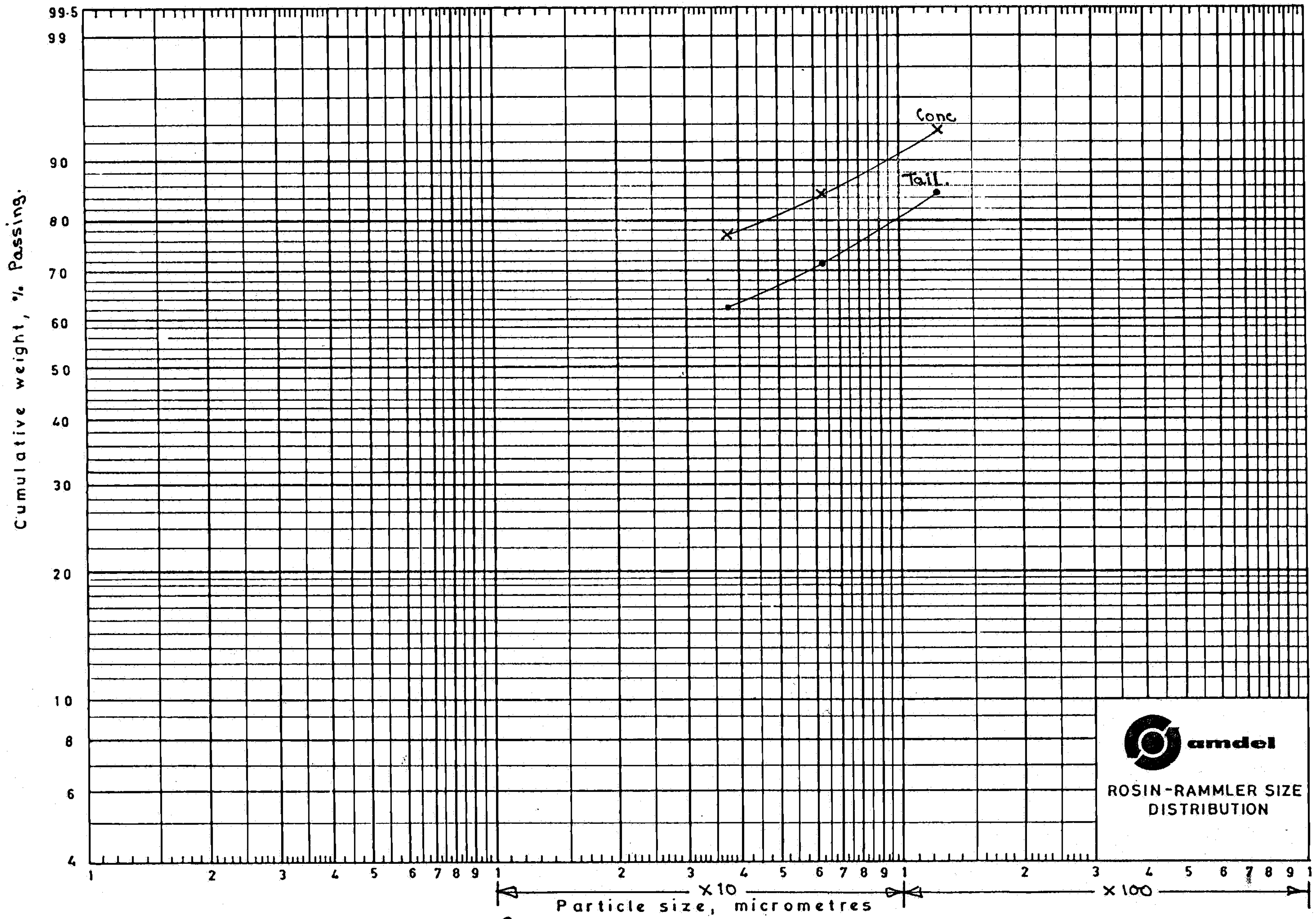


FIG. 4 : SIZINGS COMBINED CONC AND SCAVENGER TAILING FROM SAND FLOAT - TEST MG14-F8.

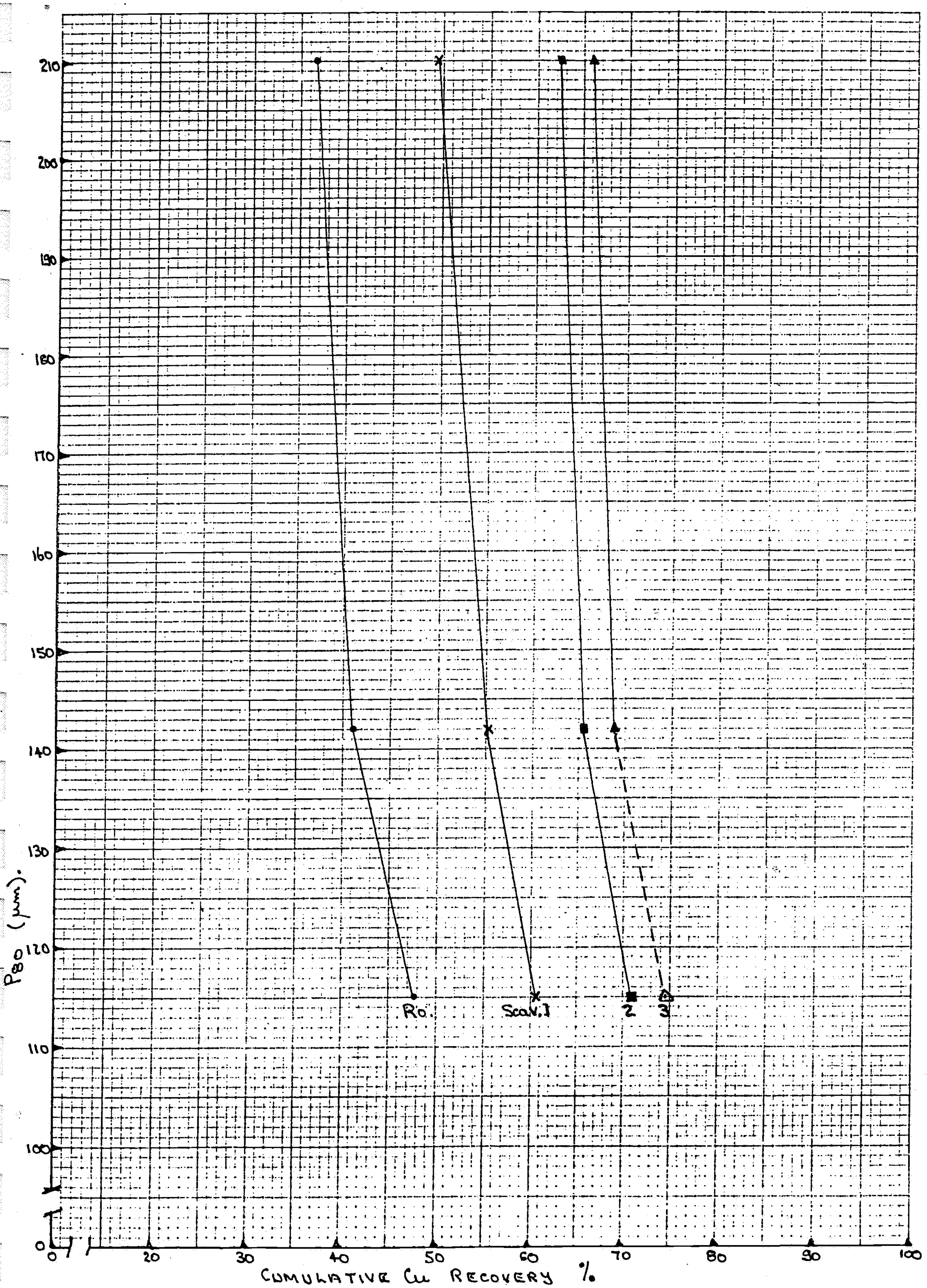


FIG. 5: Cu RECOVERY AT VARIOUS GRINDS.

# APPENDIX A

## ORE-DRESSING MINERALOGY OF ORE FROM THE MG14 DEPOSIT

### A1 INTRODUCTION

Seven grouped fractions of Sample MG14 ground for 3 minutes with 12 rods were received and the following work requested.

(a) Mineralogy

(b) Liberation/locking characteristics of the economically important minerals

(c) Analysis for Cu, Ag, Zn, Bi, Co and Pb

(d) Photomicrography

### A2 PROCEDURE

A riffled portion of each of the six +9 μm size fractions was separated in methylene iodide (Sp.Gr. 3.3), the +75 μm fractions statically and the -75+9 μm fractions centrifugally using a density gradient. Riffled portions of the -9 μm fraction and the <3.3 and >3.3 sp.gr. products of the +9 μm fractions were analysed for Cu, Ag, Zn, Bi, Co and Pb by atomic absorption spectrometry and the results were computer processed. Polished sections (PS32439-32444) were prepared of the +9 μm >3.3 and <3.3 Sp.Gr. products and the >3.3 Sp.Gr. products gross counted (at least 300 sulphide-bearing particles counted). Selected minerals were analysed using an electron probe microanalyser and colour photomicrographs of typical textures were prepared. Selected fractions and products were analysed by X-ray powder diffractometry.

### A3 GENERAL COMMENTS ON INTERPRETATION OF DATA

The distributions of Cu, Ag, Zn, Bi, Co and Pb with specific gravity and particle size are given in Tables A1 to A5. Interpreting these tables (e.g. Table A1) the following points should be noted:

(1) In column 2 ('Spec. Grav. Fraction') <3.30 refers to the <3.3 Sp.Gr. material, and >3.30 refers to >3.3 Sp.Gr. material.

(2) Columns 3-5 ('In Size Fractions') give the data for each size fraction - with weight and elemental distributions calculated to 100%. 'Totals for Above Size Fractions' refers to the total +9 μm material. 'Overall Totals' refers to the weight and element distributions between 'sands' (i.e. +9 μm material) and 'slimes' (-9 μm material).

(3) Columns 6-7 ('In Size Fractions') give the data on the basis of the head as 100%.

(4) Columns 8-10 ('Cumulative Data') give the cumulated totals by Sp.Gr. product. Thus, in the -125+75 μm size range for example, the <3.3 Sp.Gr. data refer to the total material with Sp.Gr. <3.3 in the three +75 μm size fractions.

The mineralogical data are presented in tabular and descriptive form. Most attention was paid to the Cu-bearing minerals, particularly the degree of liberation and locking characteristics of the >3.3 Sp.Gr. products.

## A4 HEAD MINERALOGY

The head mineralogy, as calculated from assays and estimated from X-ray diffraction analysis, is as follows:

| Mineral | % |
|---|---|
| Siderite | 30-40 |
| Quartz | 20-30 |
| Mica/Illite | 15-20 |
| Dolomite | 10-15 |
| Chlorite | 3-5 |
| Chalcocite* | 1.6 |
| Bornite | 1.5 |
| Sphalerite | 0.3 |
| Chalcopyrite | 0.2 |
| Carollite | 0.1 |
| Pyrite | 0.1 |
| Galena | 0.07 |
| Emplectite/Wittichenite | 0.02 |
| Native Gold | rare |

*Including digenite.

Analysis by an electron probe microanalyser indicates that carollite contains 17% Cu and 3% Ni, sphalerite is essentially Fe-free and native gold contains no detectable Ag. A copper bismuth sulphide mineral was identified and is probably either emplectite ($CuBiS_2$) or wittichenite ($Cu_3BiS_3$).

## A5 ELEMENT DISTRIBUTIONS

### A5.1 Copper Distribution (Table 1)

There is a progressive increase in the size fraction assays with decreasing particle size down to 18 μm, indicating preferential grinding of copper minerals. The Cu assays of the <3.3 Sp.Gr. products are relatively constant down to 45 μm, below which there is a slight decrease. These two features suggest the presence of a 'background' of very finely disseminated copper minerals in silicates in addition to liberated copper sulphides which are preferentially comminuted during grinding. There is a progressive increase with decreasing particle size in the proportion of Cu reporting into the >3.3 Sp.Gr. product (which would contain any liberated copper sulphide grains) and this is largely related to the increasing Cu contents of the size fractions caused by the preferential grinding of copper sulphides.

Overall in the +9 μm fractions 31.5% of the Cu reports in the <3.3 Sp.Gr. products and 68.5% in the >3.3 Sp.Gr. products.

### A5.2 Silver Distribution (Table A2)

Unlike Cu, Ag shows no preferential grinding and the Ag assays of the <3.3 Sp.Gr. products decrease progressively with decreasing particle size, reflecting increasing liberation of Ag from silicate gangue. The proportion of Ag locked with silicates (<3.3 Sp.Gr.) decreases from 68% in the +250 μm fraction to 17% in the -45+18 μm fraction. Overall in the +9 μm fractions, 46.4% of the Ag reports in the <3.3 Sp.Gr. products and 53.6% in the >3.3 Sp.Gr. products.

### A5.3 Zinc Distribution (Table A3)

Like Cu, the Zn assays for the <3.3 Sp.Gr. products remain virtually constant down to 45 μm, indicating the presence of some Zn finely disseminated in silicates (20-40% of the Zn in each size fraction).

### A5.4 Bismuth Distribution (Table A4)

Bi, similar to Cu, shows some preferential grinding, although this is not as marked as the case of Cu. There is a progressive decrease in the Bi assays of the <3.3 Sp.Gr. products with decreasing particle size, reflecting increasing liberation of Bi from silicates.

### A5.5 Cobalt Distribution (Table A5)

Unlike Cu, Co shows a marked concentration in the coarser size fractions, with the -250+125 μm fraction assaying 95 ppm Co and the -9 μm fraction assaying only 70 ppm Co. The -9 μm fraction, which contains 25% if the weight, contains only 3.7% of the total Co in the sample. Liberation of Co from silicate gangue is good, even in the +250 μm fraction (21% of the Co reporting in the <3.3 Sp.Gr. product).

### A5.6 Lead Distribution (Table A6)

The Pb distribution is generally similar to that of Cu.

## A6 MINERALOGY OF THE SEPARATION PRODUCTS

### A6.1 >3.3 Sp.Gr. Products

The approximate mineralogical compositions of the >3.3 Sp.Gr. products are summarised in Table A7 and the locking/liberation characteristics of Cu, Zn and non-opaques in Tables A8 and A9. The most common mineral is siderite, which with dolomite and quartz represents 90 to 95% of the >3.3 Sp.Gr. product. These minerals occur mainly either as liberated particles or locked with copper sulphides, either bornite (Fig. A1) or chalcocite (Fig. A2). Although almost all ternary particles (i.e. particles composed of three or more minerals) contain non-opaques, the proportion is generally minor (figs. A7 to A12).

Chalcocite and bornite occur in approximately equal amounts in the sample but chalcocite is relatively concentrated in the coarser particle sizes, possibly due to its larger grain size. Bornite is relatively concentrated in the finer particle sizes due to its overall smaller grain size. Although bornite and chalcocite occur as coarse grains (Figs. A1 and A2), most of the bornite occurs as grains intricately locked with sphalerite (Figs. A4 and A7) and pyrite (Fig. A11) and locked with non-opaques. This is reflected in the higher liberation of chalcocite than bornite (60% and 25% respectively in -45+18 µm >3.3 Sp.Gr. product). Chalcopyrite is a much less common copper sulphide, occurring in an intimate mixture with bornite, locked with non-opaques or as relatively coarse grains locked with carrollite patches in ternary particles.

Sphalerite occurs as grains intricately locked with copper sulphides (Figs. A3, A4, A7, A8, A9) or pyrite (Fig. A10) or as grains disseminated in non-opaques. Sphalerite is much less liberated than the copper sulphides, due to its very fine grain size and intricate locking. Carrollite occurs as relatively large grains almost always with inclusions of bornite (Fig. A7), chalcocite, non-opaques and rarely, emplectite. Pyrite, and possibly carrollite, is intimately locked with bornite, sphalerite (Fig. A11) and, less commonly, chalcocite and chalcopyrite (Fig. A5, A12). The pyrite grain size does not exceed 10 µm. Galena most commonly occurs as belbs and patches in chalcocite (Fig. A2) and as irregular patches in bornite-pyrite particles (Fig. A11) andsphlerite-pyrite particles. Emplectite/wittichenite occurs as inclusions in carrollite and as subhedral crystals in sphalerite-pyrite particles (Fig. A10).

A liberated native gold grain, 14 µm in size, was detected in the -45+18 µm fraction. However, no silver-bearing minerals were detected and further work would be required to locate the silver.

### A6.2 <3.3 Sp.Gr. Products

Quartz, dolomite, siderite and mica/illite were identified by X-ray powder diffractometry. By contrast the mica/illite content of the >3.3 Sp.Gr. products is negligible. The siderite content decreases with particle size. The copper sulphides occur as disseminated grains in the non-opaque particles.

### A6.3 -9 µm Fraction

Mica/illite, siderite and quartz were identified by X-ray powder diffractometry as dominant components. Dolomite, chlorite and ?kaolinite occur in minor amounts.

## A7 OVERALL LIBERATION AND LOCKING

Integration of the mineralogical data with the element distribution data allows the overall liberation and locking to be calculated. Table A10 gives the overall liberation and locking of the copper sulphides and it can be seen that liberation is almost negligible down to 75 μm and even in the -18+9 μm fraction is only about 50%.

The data for sphalerite are even worse than those for Cu, with sphalerite only approaching 20% liberation in the -45 μm fractions.

Clearly very fine grinding will be required prior to flotation of this ore and even grinding to 100% -45 μm would be unlikely to produce good liberation of Cu in the -45+9 μm size range.

TABLE A1 COPPER DISTRIBUTION WITH SIZE AND SPECIFIC GRAVITY.

| SIZE FRACTION (MICRONS) | SPEC GRAV FRACTION | --- IN SIZE FRACTIONS --- WT % | --- CU -- ASSAY % | CU -- DISTN % | IN HEAD SAMPLE WT % | CU DISTN % | CUMULATIVE DATA WEIGHT % | ASSAY % | DISTN % |
|---|---|---|---|---|---|---|---|---|---|
| + 250. | < 3.30 | 71.1 | 1.16 | 51.9 | 10.58 | 5.38 | 10.58 | 1.16 | 5.38 |
| | > 3.30 | 28.9 | 2.65 | 48.1 | 4.29 | 4.98 | 4.29 | 2.65 | 4.98 |
| | TOTAL | 100.0( | 1.59) | 100.0 | 14.87 | 10.36 | 14.87 | 1.59 | 10.36 |
| - 250. + 125. | < 3.30 | 73.8 | 1.33 | 43.4 | 12.88 | 7.51 | 23.46 | 1.25 | 12.89 |
| | > 3.30 | 26.2 | 4.90 | 56.6 | 4.56 | 9.81 | 8.85 | 3.81 | 14.79 |
| | TOTAL | 100.0( | 2.26) | 100.0 | 17.44 | 17.32 | 32.31 | 1.95 | 27.68 |
| - 125. + 75. | < 3.30 | 74.3 | 1.37 | 35.5 | 7.53 | 4.52 | 30.99 | 1.28 | 17.41 |
| | > 3.30 | 25.7 | 7.20 | 64.5 | 2.61 | 8.23 | 11.46 | 4.58 | 23.02 |
| | TOTAL | 100.0( | 2.87) | 100.0 | 10.14 | 12.75 | 42.45 | 2.17 | 40.43 |
| - 75. + 45. | < 3.30 | 69.0 | 1.39 | 27.0 | 4.67 | 2.85 | 35.66 | 1.30 | 20.26 |
| | > 3.30 | 31.0 | 8.40 | 73.0 | 2.10 | 7.72 | 13.55 | 5.17 | 30.73 |
| | TOTAL | 100.0( | 3.56) | 100.0 | 6.77 | 10.56 | 49.22 | 2.36 | 50.99 |
| - 45. + 18. | < 3.30 | 49.8 | 0.97 | 12.0 | 6.44 | 2.74 | 42.10 | 1.25 | 23.00 |
| | > 3.30 | 50.2 | 7.10 | 88.0 | 6.48 | 20.18 | 20.04 | 5.80 | 50.91 |
| | TOTAL | 100.0( | 4.04) | 100.0 | 12.93 | 22.92 | 62.14 | 2.71 | 73.91 |
| - 18. + 9. | < 3.30 | 70.4 | 0.98 | 34.2 | 9.08 | 3.90 | 51.18 | 1.20 | 26.90 |
| | > 3.30 | 29.6 | 4.50 | 65.8 | 3.81 | 7.52 | 23.85 | 5.59 | 58.43 |
| | TOTAL | 100.0( | 2.02) | 100.0 | 12.89 | 11.42 | 75.03 | 2.59 | 85.33 |
| TOTALS FOR ABOVE | < 3.30 | 68.2( | 1.20) | 31.5 | | | | | |
| SIZE FRACTIONS | > 3.30 | 31.8( | 5.59) | 68.5 | | | | | |
| OVERALL TOTALS.. | SANDS | 75.0( | 2.59) | 85.3 | | | | | |
| | SLIMES | 25.0 | 1.34 | 14.7 | | | | | |
| | TOTAL | 100.0( | 2.28) | 100.0 | | | | | |

CALCULATED HEAD ASSAY= 2.28 %

TABLE A2 SILVER DISTRIBUTION WITH SIZE AND SPECIFIC GRAVITY.

| SIZE FRACTION (MICRONS) | SPEC GRAV FRACTION | --- IN SIZE FRACTIONS --- WT % | ASSAY PPM | AG DISTN % | IN HEAD SAMPLE WT % | AG DISTN % | CUMULATIVE DATA WEIGHT % | ASSAY PPM | DISTN % |
|---|---|---|---|---|---|---|---|---|---|
| + 250. | < 3.30 | 71.1 | 30.00 | 67.9 | 10.58 | 11.59 | 10.58 | 30.00 | 11.59 |
| | > 3.30 | 28.9 | 35.00 | 32.1 | 4.29 | 5.48 | 4.29 | 35.00 | 5.48 |
| | TOTAL | 100.0( | 31.44) | 100.0 | 14.87 | 17.08 | 14.87 | 31.44 | 17.08 |
| - 250. + 125. | < 3.30 | 73.8 | 18.00 | 62.9 | 12.88 | 8.47 | 23.46 | 23.41 | 20.06 |
| | > 3.30 | 26.2 | 30.00 | 37.1 | 4.56 | 5.00 | 8.85 | 32.42 | 10.49 |
| | TOTAL | 100.0( | 21.14) | 100.0 | 17.44 | 13.47 | 32.31 | 25.88 | 30.55 |
| - 125. + 75. | < 3.30 | 74.3 | 22.00 | 64.5 | 7.53 | 6.05 | 30.99 | 23.07 | 26.11 |
| | > 3.30 | 25.7 | 35.00 | 35.5 | 2.61 | 3.33 | 11.46 | 33.01 | 13.82 |
| | TOTAL | 100.0( | 25.34) | 100.0 | 10.14 | 9.38 | 42.45 | 25.75 | 39.93 |
| - 75. + 45. | < 3.30 | 69.0 | 16.00 | 39.4 | 4.67 | 2.73 | 35.66 | 22.14 | 28.85 |
| | > 3.30 | 31.0 | 55.00 | 60.6 | 2.10 | 4.21 | 13.55 | 36.41 | 18.03 |
| | TOTAL | 100.0( | 28.07) | 100.0 | 6.77 | 6.94 | 49.22 | 26.07 | 46.87 |
| - 45. + 18. | < 3.30 | 49.8 | 16.00 | 16.6 | 6.44 | 3.77 | 42.10 | 21.20 | 32.61 |
| | > 3.30 | 50.2 | 80.00 | 83.4 | 6.48 | 18.95 | 20.04 | 50.51 | 36.97 |
| | TOTAL | 100.0( | 48.10) | 100.0 | 12.93 | 22.71 | 62.14 | 30.65 | 69.59 |
| - 18. + 9. | < 3.30 | 70.4 | 11.00 | 42.8 | 9.08 | 3.65 | 51.18 | 19.39 | 36.26 |
| | > 3.30 | 29.6 | 35.00 | 57.2 | 3.81 | 4.87 | 23.85 | 48.03 | 41.85 |
| | TOTAL | 100.0( | 18.10) | 100.0 | 12.89 | 8.52 | 75.03 | 28.50 | 78.10 |
| TOTALS FOR ABOVE | < 3.30 | 68.2( | 19.39) | 46.4 | | | | | |
| SIZE FRACTIONS | > 3.30 | 31.8( | 48.03) | 53.6 | | | | | |
| OVERALL TOTALS.. | SANDS | 75.0( | 28.50) | 78.1 | | | | | |
| | SLIMES | 25.0 | 24.00 | 21.9 | | | | | |
| | TOTAL | 100.0( | 27.37) | 100.0 | | | | | |

CALCULATED HEAD ASSAY= 27.37 PPM

## TABLEA3 ZINC DISTRIBUTION WITH SIZE AND SPECIFIC GRAVITY.

| SIZE FRACTION (MICRONS) | SPEC GRAV FRACTION | --- IN SIZE FRACTIONS --- , | | | IN HEAD SAMPLE | | CUMULATIVE DATA | | |
|---|---|---|---|---|---|---|---|---|---|
| | | WT % | --- ZN -- ASSAY % | DISTN % | WT % | ZN DISTN % | WEIGHT % | ASSAY % | DISTN % |
| + 250. | < 3.30 | 71.1 | 0.11 | 41.0 | 10.58 | 5.22 | 10.58 | 0.11 | 5.22 |
| | > 3.30 | 28.9 | 0.39 | 59.0 | 4.29 | 7.50 | 4.29 | 0.39 | 7.50 |
| | TOTAL | 100.0( | 0.19) | 100.0 | 14.87 | 12.72 | 14.87 | 0.19 | 12.72 |
| - 250. + 125. | < 3.30 | 73.8 | 0.11 | 32.0 | 12.88 | 6.36 | 23.46 | 0.11 | 11.58 |
| | > 3.30 | 26.2 | 0.66 | 68.0 | 4.56 | 13.52 | 8.85 | 0.53 | 21.02 |
| | TOTAL | 100.0( | 0.25) | 100.0 | 17.44 | 19.87 | 32.31 | 0.22 | 32.60 |
| - 125. + 75. | < 3.30 | 74.3 | 0.11 | 27.7 | 7.53 | 3.72 | 30.99 | 0.11 | 15.29 |
| | > 3.30 | 25.7 | 0.83 | 72.3 | 2.61 | 9.70 | 11.46 | 0.60 | 30.73 |
| | TOTAL | 100.0( | 0.30) | 100.0 | 10.14 | 13.42 | 42.45 | 0.24 | 46.02 |
| - 75. + 45. | < 3.30 | 69.0 | 0.10 | 23.2 | 4.67 | 2.10 | 35.66 | 0.11 | 17.39 |
| | > 3.30 | 31.0 | 0.74 | 76.8 | 2.10 | 6.96 | 13.55 | 0.62 | 37.68 |
| | TOTAL | 100.0( | 0.30) | 100.0 | 6.77 | 9.05 | 49.22 | 0.25 | 55.07 |
| - 45. + 18. | < 3.30 | 49.8 | 0.05 | 11.3 | 6.44 | 1.45 | 42.10 | 0.10 | 18.84 |
| | > 3.30 | 50.2 | 0.39 | 88.7 | 6.48 | 11.34 | 20.04 | 0.55 | 49.03 |
| | TOTAL | 100.0( | 0.22) | 100.0 | 12.93 | 12.79 | 62.14 | 0.24 | 67.86 |
| - 18. + 9. | < 3.30 | 70.4 | 0.03 | 29.6 | 9.08 | 1.22 | 51.18 | 0.09 | 20.06 |
| | > 3.30 | 29.6 | 0.17 | 70.4 | 3.81 | 2.91 | 23.85 | 0.49 | 51.93 |
| | TOTAL | 100.0( | 0.07) | 100.0 | 12.89 | 4.13 | 75.03 | 0.21 | 71.99 |
| TOTALS FOR ABOVE SIZE FRACTIONS | < 3.30 | 68.2( | 0.09) | 27.9 | | | | | |
| | > 3.30 | 31.8( | 0.49) | 72.1 | | | | | |
| OVERALL TOTALS.. | SANDS | 75.0( | 0.21) | 72.0 | | | | | |
| | SLIMES | 25.0 | 0.25 | 28.0 | | | | | |
| | TOTAL | 100.0( | 0.22) | 100.0 | | | | | |

CALCULATED HEAD ASSAY= 0.22 %

## TABLE A4 BISMUTH DISTRIBUTION WITH SIZE AND SPECIFIC GRAVITY.

| SIZE FRACTION (MICRONS) | SPEC GRAV FRACTION | --- IN SIZE FRACTIONS --- , | | | IN HEAD SAMPLE | | CUMULATIVE DATA | | |
|---|---|---|---|---|---|---|---|---|---|
| | | WT % | --- ASSAY PPM | BI -- DISTN % | WT % | BI DISTN % | WEIGHT % | ASSAY PPM | DISTN % |
| + 250. | < 3.30 | 71.1 | 90.00 | 52.6 | 10.58 | 7.85 | 10.58 | 90.00 | 7.85 |
| | > 3.30 | 28.9 | 200.00 | 47.4 | 4.29 | 7.07 | 4.29 | 200.00 | 7.07 |
| | TOTAL | 100.0( | 121.74) | 100.0 | 14.87 | 14.92 | 14.87 | 121.74 | 14.92 |
| - 250. + 125. | < 3.30 | 73.8 | 75.00 | 33.5 | 12.88 | 7.96 | 23.46 | 81.76 | 15.81 |
| | > 3.30 | 26.2 | 420.00 | 66.5 | 4.56 | 15.81 | 8.85 | 313.42 | 22.88 |
| | TOTAL | 100.0( | 165.27) | 100.0 | 17.44 | 23.77 | 32.31 | 145.24 | 38.69 |
| - 125. + 75. | < 3.30 | 74.3 | 65.00 | 28.6 | 7.53 | 4.04 | 30.99 | 77.69 | 19.85 |
| | > 3.30 | 25.7 | 470.00 | 71.4 | 2.61 | 10.10 | 11.46 | 349.03 | 32.98 |
| | TOTAL | 100.0( | 169.12) | 100.0 | 10.14 | 14.13 | 42.45 | 150.94 | 52.83 |
| - 75. + 45. | < 3.30 | 69.0 | 60.00 | 23.7 | 4.67 | 2.31 | 35.66 | 75.37 | 22.16 |
| | > 3.30 | 31.0 | 430.00 | 76.3 | 2.10 | 7.43 | 13.55 | 361.55 | 40.40 |
| | TOTAL | 100.0( | 174.54) | 100.0 | 6.77 | 9.74 | 49.22 | 154.19 | 62.56 |
| - 45. + 18. | < 3.30 | 49.8 | 50.00 | 13.4 | 6.44 | 2.66 | 42.10 | 71.49 | 24.82 |
| | > 3.30 | 50.2 | 320.00 | 86.6 | 6.48 | 17.10 | 20.04 | 348.10 | 57.51 |
| | TOTAL | 100.0( | 185.41) | 100.0 | 12.93 | 19.76 | 62.14 | 160.68 | 82.32 |
| - 18. + 9. | < 3.30 | 70.4 | 45.00 | 53.0 | 9.08 | 3.37 | 51.18 | 66.79 | 28.18 |
| | > 3.30 | 29.6 | 95.00 | 47.0 | 3.81 | 2.98 | 23.85 | 307.67 | 60.49 |
| | TOTAL | 100.0( | 59.78) | 100.0 | 12.89 | 6.35 | 75.03 | 143.35 | 88.68 |
| TOTALS FOR ABOVE | < 3.30 | 68.2( | 66.79) | 31.8 | | | | | |
| SIZE FRACTIONS | > 3.30 | 31.8( | 307.67) | 68.2 | | | | | |
| OVERALL TOTALS.. | SANDS | 75.0( | 143.35) | 88.7 | | | | | |
| | SLIMES | 25.0 | 55.00 | 11.3 | | | | | |
| | TOTAL | 100.0( | 121.29) | 100.0 | | | | | |

CALCULATED HEAD ASSAY= 121.29 PPM

TABLE A5 COBALT DISTRIBUTION WITH SIZE AND SPECIFIC GRAVITY.

| SIZE FRACTION (MICRONS) | SPEC GRAV FRACTION | --- IN SIZE FRACTIONS --- WT % | ASSAY PPM | CO DISTN % | IN HEAD SAMPLE WT % | CO DISTN % | CUMULATIVE DATA WEIGHT % | ASSAY PPM | DISTN % |
|---|---|---|---|---|---|---|---|---|---|
| + 250. | < 3.30 | 71.1 | 200.00 | 20.6 | 10.58 | 4.49 | 10.58 | 200.00 | 4.49 |
| | > 3.30 | 28.9 | 1900.00 | 79.4 | 4.29 | 17.31 | 4.29 | 1900.00 | 17.31 |
| | TOTAL | 100.0( | 690.48) | 100.0 | 14.87 | 21.80 | 14.87 | 690.48 | 21.80 |
| - 250. + 125. | < 3.30 | 73.8 | 200.00 | 15.4 | 12.88 | 5.47 | 23.46 | 200.00 | 9.96 |
| | > 3.30 | 26.2 | 3100.00 | 84.6 | 4.56 | 30.05 | 8.85 | 2518.67 | 47.36 |
| | TOTAL | 100.0( | 958.80) | 100.0 | 17.44 | 35.53 | 32.31 | 835.35 | 57.33 |
| - 125. + 75. | < 3.30 | 74.3 | 100.00 | 13.8 | 7.53 | 1.60 | 30.99 | 175.70 | 11.56 |
| | > 3.30 | 25.7 | 1800.00 | 86.2 | 2.61 | 9.96 | 11.46 | 2355.23 | 57.33 |
| | TOTAL | 100.0( | 537.05) | 100.0 | 10.14 | 11.56 | 42.45 | 764.11 | 68.89 |
| - 75. + 45. | < 3.30 | 69.0 | 70.00 | 6.4 | 4.67 | 0.69 | 35.66 | 161.85 | 12.26 |
| | > 3.30 | 31.0 | 2300.00 | 93.6 | 2.10 | 10.24 | 13.55 | 2346.70 | 67.56 |
| | TOTAL | 100.0( | 760.34) | 100.0 | 6.77 | 10.93 | 49.22 | 763.59 | 79.82 |
| - 45. + 18. | < 3.30 | 49.8 | 50.00 | 5.2 | 6.44 | 0.68 | 42.10 | 144.73 | 12.94 |
| | > 3.30 | 50.2 | 900.00 | 94.8 | 6.48 | 12.39 | 20.04 | 1878.64 | 79.95 |
| | TOTAL | 100.0( | 476.30) | 100.0 | 12.93 | 13.08 | 62.14 | 703.83 | 92.90 |
| - 18. + 9. | < 3.30 | 70.4 | 50.00 | 28.4 | 9.08 | 0.96 | 51.18 | 127.93 | 13.91 |
| | > 3.30 | 29.6 | 300.00 | 71.6 | 3.81 | 2.43 | 23.85 | 1626.45 | 82.38 |
| | TOTAL | 100.0( | 123.91) | 100.0 | 12.89 | 3.39 | 75.03 | 604.23 | 96.29 |
| TOTALS FOR ABOVE SIZE FRACTIONS | < 3.30 | 68.2( | 127.93) | 14.4 | | | | | |
| | > 3.30 | 31.8( | 1626.45) | 85.6 | | | | | |
| OVERALL TOTALS.. | SANDS | 75.0( | 604.23) | 96.3 | | | | | |
| | SLIMES | 25.0 | 70.00 | 3.7 | | | | | |
| | TOTAL | 100.0( | 470.82) | 100.0 | | | | | |

CALCULATED HEAD ASSAY= 470.82 PPM

TABLE A6 LEAD DISTRIBUTION WITH SIZE AND SPECIFIC GRAVITY.

| SIZE FRACTION (MICRONS) | SPEC GRAV FRACTION | --- IN SIZE FRACTIONS --- , | | | IN HEAD SAMPLE | | CUMULATIVE DATA | | |
|---|---|---|---|---|---|---|---|---|---|
| | | WT % | --- ASSAY % | PB -- DISTN % | WT % | PB DISTN % | WEIGHT % | ASSAY % | DISTN % |
| + 250. | < 3.30 | 71.1 | 0.05 | 60.6 | 10.58 | 9.18 | 10.58 | 0.05 | 9.18 |
| | > 3.30 | 28.9 | 0.08 | 39.4 | 4.29 | 5.95 | 4.29 | 0.08 | 5.95 |
| | TOTAL | 100.0( | 0.06) | 100.0 | 14.87 | 15.13 | 14.87 | 0.06 | 15.13 |
| - 250. + 125. | < 3.30 | 73.8 | 0.04 | 42.9 | 12.88 | 8.94 | 23.46 | 0.04 | 18.12 |
| | > 3.30 | 26.2 | 0.15 | 57.1 | 4.56 | 11.88 | 8.85 | 0.12 | 17.84 |
| | TOTAL | 100.0( | 0.07) | 100.0 | 17.44 | 20.82 | 32.31 | 0.06 | 35.95 |
| - 125. + 75. | < 3.30 | 74.3 | 0.04 | 30.8 | 7.53 | 5.23 | 30.99 | 0.04 | 23.35 |
| | > 3.30 | 25.7 | 0.26 | 69.2 | 2.61 | 11.76 | 11.46 | 0.15 | 29.59 |
| | TOTAL | 100.0( | 0.10) | 100.0 | 10.14 | 16.99 | 42.45 | 0.07 | 52.94 |
| - 75. + 45. | < 3.30 | 69.0 | 0.03 | 19.3 | 4.67 | 2.43 | 35.66 | 0.04 | 25.78 |
| | > 3.30 | 31.0 | 0.28 | 80.7 | 2.10 | 10.18 | 13.55 | 0.17 | 39.78 |
| | TOTAL | 100.0( | 0.11) | 100.0 | 6.77 | 12.61 | 49.22 | 0.08 | 65.55 |
| - 45. + 18. | < 3.30 | 49.8 | 0.02 | 11.1 | 6.44 | 2.24 | 42.10 | 0.04 | 28.01 |
| | > 3.30 | 50.2 | 0.16 | 88.9 | 6.48 | 18.00 | 20.04 | 0.17 | 57.78 |
| | TOTAL | 100.0( | 0.09) | 100.0 | 12.93 | 20.24 | 62.14 | 0.08 | 85.79 |
| - 18. + 9. | < 3.30 | 70.4 | 0.01 | 28.4 | 9.08 | 1.58 | 51.18 | 0.03 | 29.59 |
| | > 3.30 | 29.6 | 0.06 | 71.6 | 3.81 | 3.97 | 23.85 | 0.15 | 61.74 |
| | TOTAL | 100.0( | 0.02) | 100.0 | 12.89 | 5.54 | 75.03 | 0.07 | 91.33 |
| TOTALS FOR ABOVE SIZE FRACTIONS | < 3.30 | 68.2( | 0.03) | 32.4 | | | | | |
| | > 3.30 | 31.8( | 0.15) | 67.6 | | | | | |
| OVERALL TOTALS.. | SANDS | 75.0( | 0.07) | 91.3 | | | | | |
| | SLIMES | 25.0 | 0.02 | 8.7 | | | | | |
| | TOTAL | 100.0( | 0.06) | 100.0 | | | | | |

CALCULATED HEAD ASSAY= 0.06 %

## TABLE A7: APPROXIMATE MINERALOGICAL COMPOSITIONS (WT %) OF THE >3.3 SP.GR. FRACTIONS

| | Size Fraction (μm) | | | | | |
|---|---|---|---|---|---|---|
| | +250 | -250 +125 | -125 + 75 | -75 +45 | -45 +18 | -18 + 9 |
| Bornite | 1.2 | 1.8 | 3.2 | 4.6 | 6.3 | 3.8 |
| Chalcocite* | 2.1 | 4.5 | 6.1 | 6.2 | 3.5 | 2.5 |
| Chalcopyrite | 0.4 | 0.4 | 0.8 | 1.3 | 0.6 | 0.4 |
| Sphalerite | 0.6 | 1.0 | 1.2 | 1.1 | 0.6 | 0.2 |
| Carollite | 0.5 | 0.9 | 0.5 | 0.7 | 0.3 | 0.1 |
| Other Sulphides | 0.2 | 0.3 | 0.5 | 0.5 | 0.4 | 0.2 |
| Siderite | 80 | 80 | 80 | 80 | 80 | 80 |
| Dolomite | 5 | 5 | <5 | <5 | 5 | 5 |
| Silicates | 10 | 5 | 5 | 5 | 5 | 5 |
| Total | 100 | 100 | 100 | 100 | 100 | 100 |

*Chalcocite is used in this report to include both chalcocite and digenite.

TABLE A8: LIBERATION AND LOCKING CHARACTERISTICS OF COPPER SULPHIDES IN THE >3.3 SP.GR. PRODUCTS

| Mineral | | % Locked With* | | | | |
|---|---|---|---|---|---|---|
| | Liberated[§] % | Carr/Py | Sp. | Ga/Emp. | N/O[¶] | T. |
| (a) +250 μm Fraction | | | | | | |
| Bornite | 5 | - | - | - | 25 | 70 |
| Chalcocite | 20 | - | - | - | 15 | 65 |
| Chalcopyrite | - | - | - | - | 50 | 50 |
| Total Copper Sulphides | 10 | - | - | - | 25 | 65 |
| (b) -250+125 μm Fraction | | | | | | |
| Bornite | 5 | - | - | - | 35 | 60 |
| Chalcocite | 10 | - | - | - | 50 | 40 |
| Chalcopyrite | - | - | - | - | 60 | 40 |
| Total Copper Sulphides | 8 | - | - | - | 47 | 45 |
| (c) -125+75 μm Fraction | | | | | | |
| Bornite | 10 | <5 | <5 | - | 45 | 40 |
| Chalcocite | 10 | <5 | <5 | <1 | 50 | 35 |
| Chalcopyrite | 15 | 5 | - | - | 35 | 45 |
| Total Copper Sulphides | 10 | <5 | <5 | <1 | 45 | 40 |
| (d) -75+45 μm Fraction | | | | | | |
| Bornite | 10 | 5 | 5 | <1 | 55 | 25 |
| Chalcocite | 25 | 5 | 5 | <1 | 50 | 15 |
| Chalcopyrite | 30 | - | <1 | - | 30 | 40 |
| Total Copper Sulphides | 20 | 5 | 5 | <1 | 50 | 20 |
| (e) -45+18 μm Fraction | | | | | | |
| Bornite | 25 | 5 | <5 | <5 | 50 | 15 |
| Chalcocite | 60 | <5 | - | - | 35 | 5 |
| Chalcopyrite | 45 | 5 | 5 | - | 20 | 25 |
| Total Copper Sulphides | 37 | <5 | <5 | <5 | 42 | 10 |

...2/

TABLE A8: (Continued)

| Mineral | Liberated[§] % | % Locked With* | | | | |
|---|---|---|---|---|---|---|
| | | Carr/Py | Sp. | Ga/Emp. | N/O[¶] | T. |
| (f) <u>-18+9 µm Fraction</u> | | | | | | |
| Bornite | 70 | 15 | <1 | <1 | 10 | 5 |
| Chalcocite | 85 | <5 | <5 | - | 10 | - |
| Chalcopyrite | 80 | <5 | 15 | - | <5 | <5 |
| Total Copper Sulphides | 75 | 10 | <5 | <1 | 10 | <5 |

*Carr = Carollite, Emp = Emplectite, Ga = Galena, N/O = Non-opaques, Py = Pyrite, Sp = Sphalerite, T = Two or more minerals.

[§] Includes locking with other copper sulphides.

[¶] Includes locking with other copper sulphides in non-opaque particles.

## TABLE A9: LIBERATION AND LOCKING CHARACTERISTICS OF SPHALERITE AND NON-OPAQUES IN THE >3.3 SP.GR. PRODUCTS

(a) Sphalerite

| Size Fraction (μm) | Liberated % | % Locked With* | | | | |
|---|---|---|---|---|---|---|
| | | Bo | Cc | Cp | N/O | T. |
| +250 | - | - | - | - | - | 100 |
| -250+125 | - | - | - | - | - | 100 |
| -125+ 75 | 5 | - | - | - | 5 | 90 |
| - 75+ 45 | <5 | 15 | 5 | - | <5 | 75 |
| - 45+ 18 | 25 | 15 | 5 | 5 | 10 | 40 |
| - 18+ 9 | 30 | 25 | 5 | 5 | 10 | 25 |

(b) Non-Opaques

| Size Fraction (μm) | Liberated % | % Locked With* | | | | | |
|---|---|---|---|---|---|---|---|
| | | Bo | Cc | Cp | Sp | Carr/Py | T. |
| +250 | 20 | 25 | 25 | <1 | 10 | - | 20 |
| -250+125 | 45 | 20 | 15 | - | 5 | <1 | 15 |
| -125+ 75 | 70 | 5 | 15 | - | - | 5 | 5 |
| - 75+ 45 | 75 | <5 | 15 | <1 | - | <5 | 5 |
| - 45+ 18 | 80 | 5 | <5 | <1 | <1 | <1 | 10 |
| - 18+ 9 | >95 | <5 | <5 | <1 | <1 | - | <5 |

*Bo = Bornite, Carr/Py = Carollite/Pyrite, Cc = Chalcocite, Cp = Chalcopyrite, Sp = Sphalerite, T = Ternary.

## TABLE A10: OVERALL LIBERATION AND LOCKING OF COPPER SULPHIDES

| Size Fraction (μm) | Liberated % | % Locked Largely With | |
|---|---|---|---|
| | | Silicates/Siderite ± Other Sulphides | Other Sulphides Only |
| +250 | 5 | 95 | - |
| -250+125 | 5 | 95 | - |
| -125+ 75 | 6 | 92 | 2 |
| - 75+ 45 | 15 | 78 | 7 |
| - 45+ 18 | 32 | 58 | 10 |
| - 18+ 9 | 50 | 41 | 9 |

## TABLE A11: OVERALL LIBERATION AND LOCKING OF SPHALERITE

| Size Fraction (μm) | Liberated % | % Locked Largely with | |
|---|---|---|---|
| | | Silicates/Siderite ± Other Sulphides | Other Sulphides Only |
| +250 | - | 100 | - |
| -250+125 | - | 100 | - |
| -125+ 75 | 2 | 98 | - |
| - 75+ 45 | 3 | 85 | 12 |
| - 45+ 18 | 21 | 58 | 21 |
| - 18+ 9 | 17 | 63 | 20 |

# KEY TO PHOTOMICROGRAPHS

The minerals in Figures A1 to A12 have been labelled as follows:

| | |
|---|---|
| Bo | Bornite |
| Car | Carollite |
| Cc | Chalcocite |
| Cp | Chalcopyrite |
| E | Emplectite/Wittichenite |
| Ga | Galena |
| P | Pyrite |
| Sp | Sphalerite |

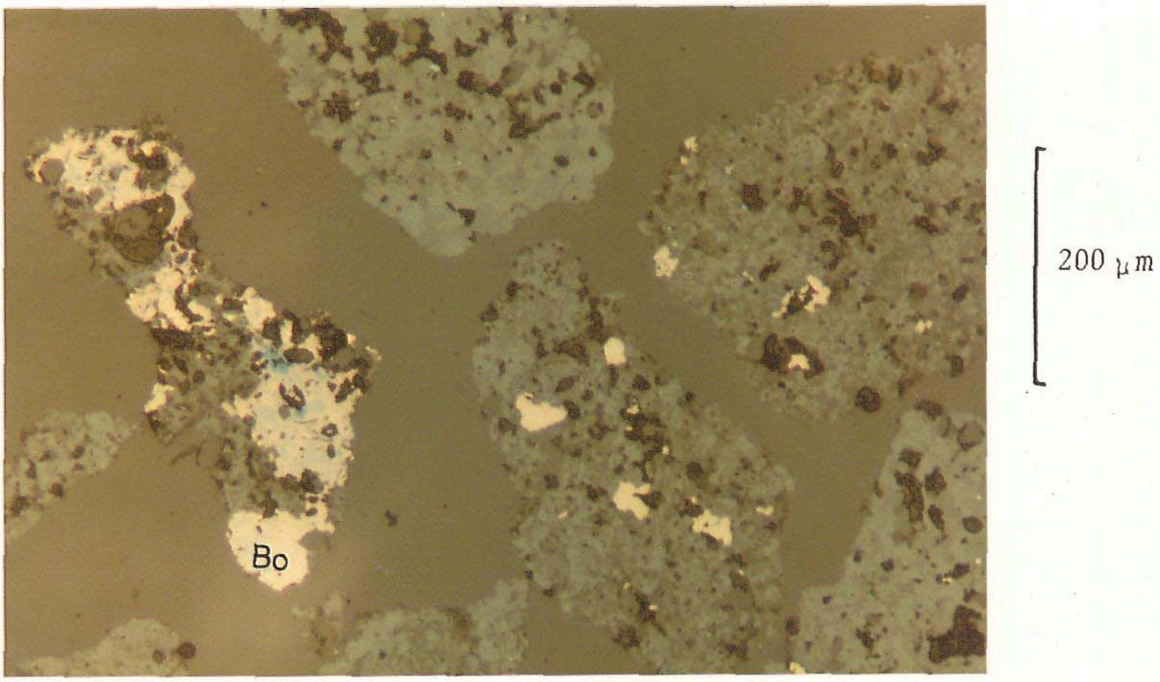


FIG. A1: +250 μm >3.3 Sp.Gr. Typical locking of bornite with non-opaques. Minor alteration to covellite (blue) is present.

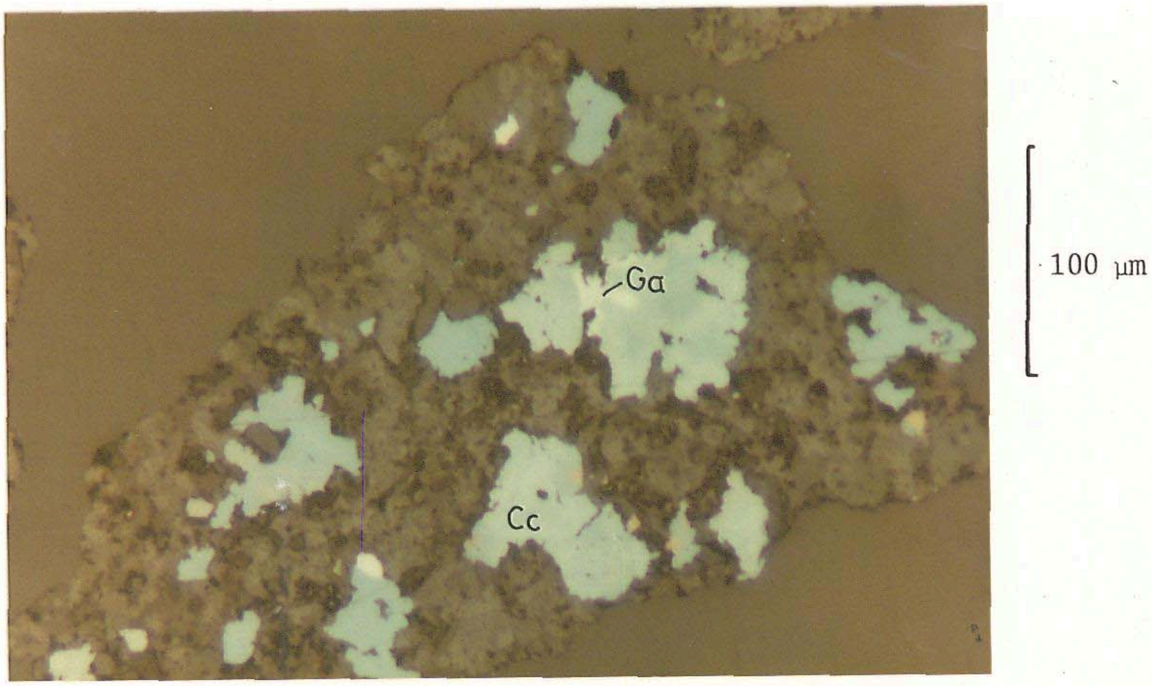


FIG. A2: -250+125 μm >3.3 Sp.Gr. Typical locking of chalcocite with non-opaques. Chalcocite contains inclusions of galena

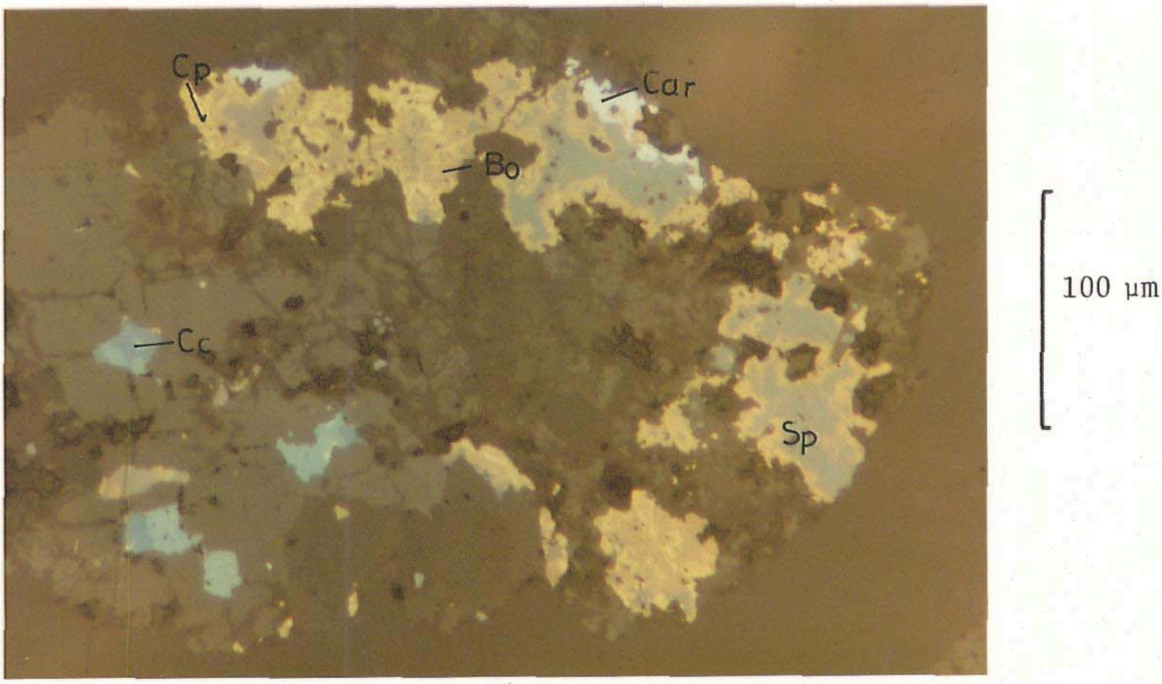


FIG. A3: +250 μm >3.3 Sp.Gr. Typical sphalerite/copper sulphide locking with non-opaques, with sphalerite core and bornite-chalcopyrite rim. The chalcocite contains a few blebs of galena.

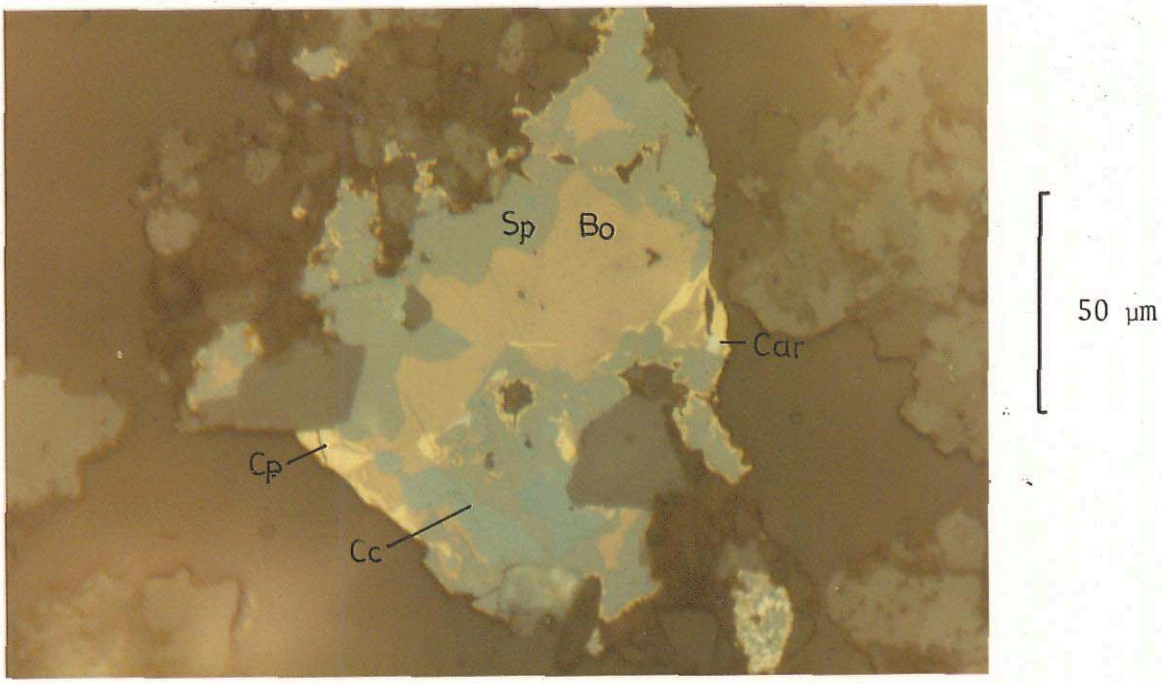


FIG. A4: -125+75 μm >3.3 Sp.Gr. Complex locking between sphalerite and copper sulphides.

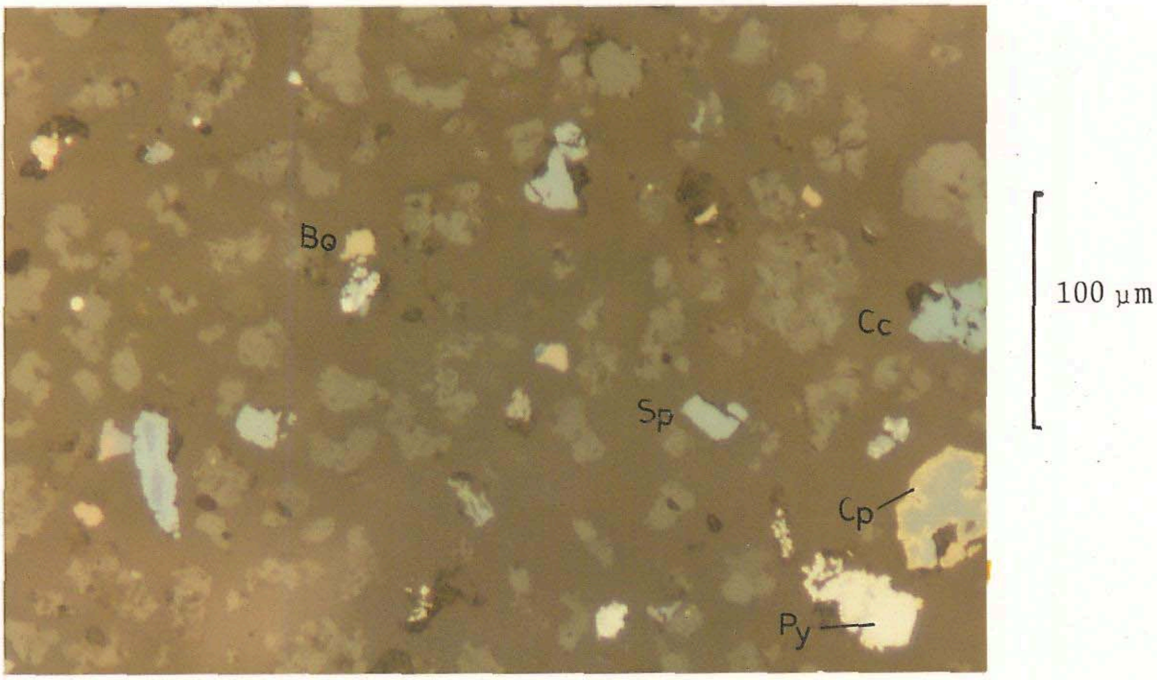


FIG. A5: -45+18 μm >3.3 Sp.Gr. Liberated non-opaque particles, and non-opaques locked with bornite, pyrite and chalcocite. Sphalerite occurs as liberated particles and locked with chalcopyrite/bornite and pyrite. The lighter-shaded non-opaques are siderite and the darker-shaded non-opaques are quartz.

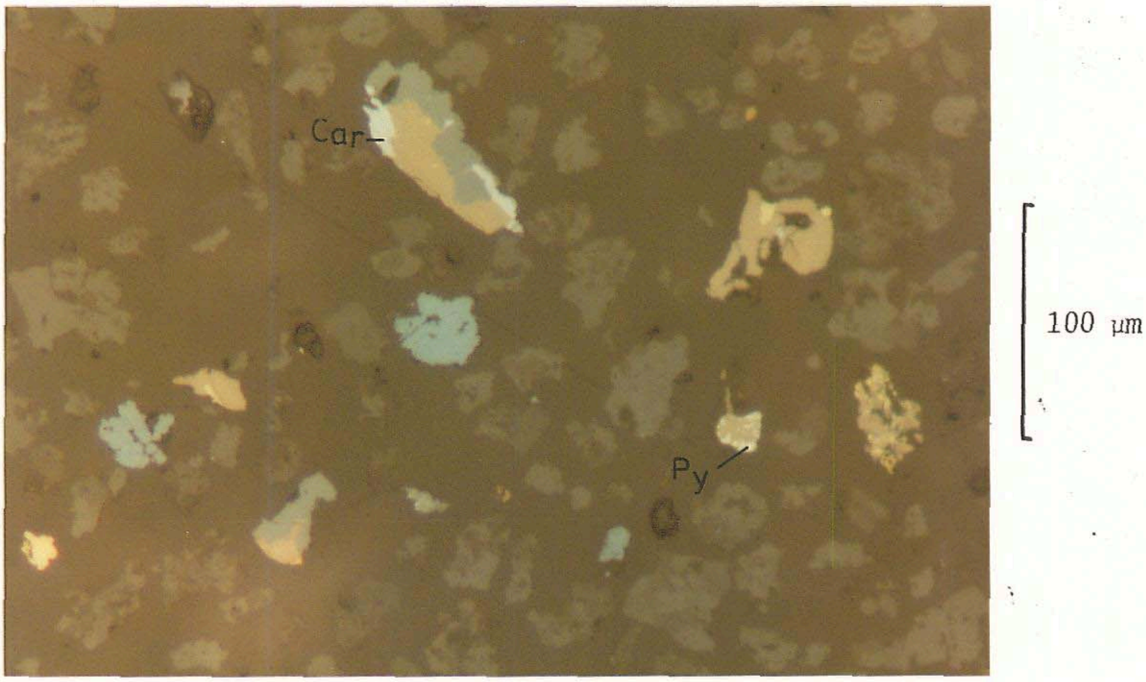


FIG. A6: -45+18 μm >3.3 Sp.Gr. Essentially liberated non-opaque particles. Bornite locked with sphalerite, non-opaques and in ternary particles.

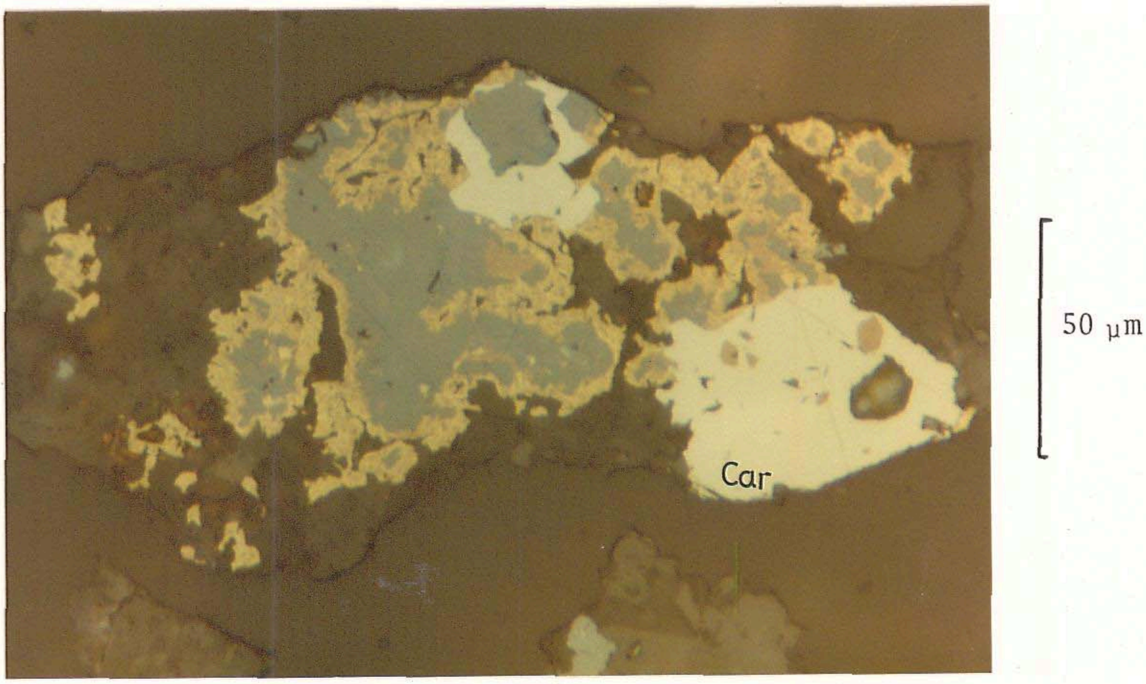


FIG. A7: -125+75 μm >3.3 Sp.Gr. Zonation of sphalerite and bornite/chalcopyrite. Carollite typically contains inclusions of bornite.

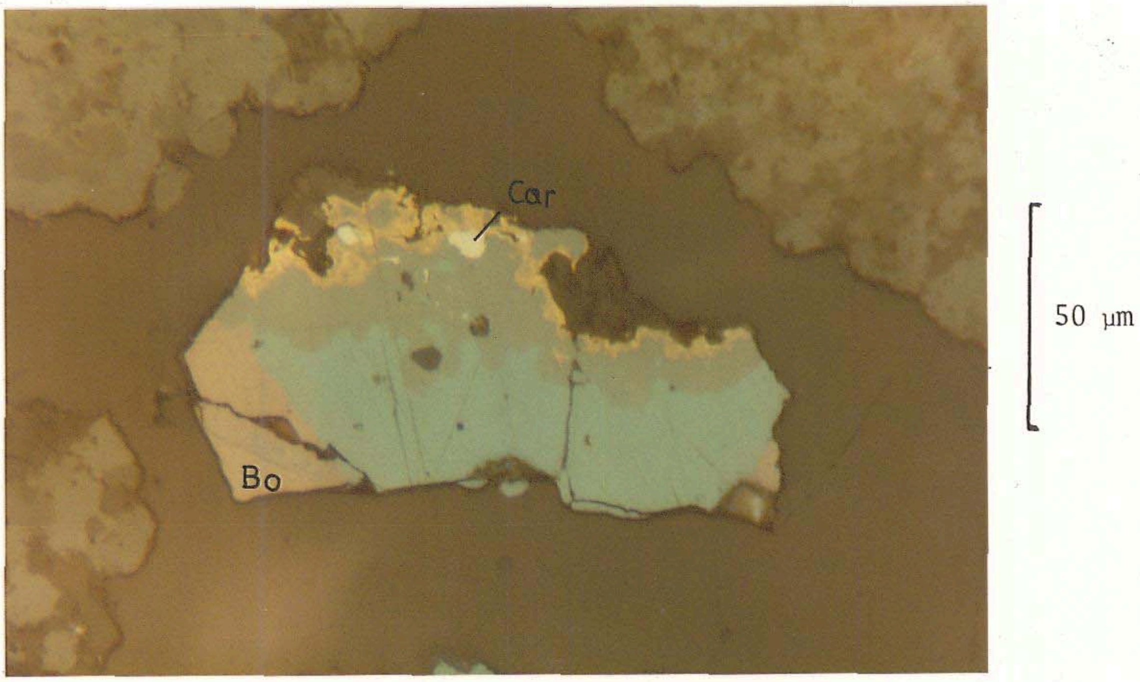


FIG. A8: -125+75 μm >3.3 Sp.Gr. Zonation of chalcocite-sphalerite with rim of bornite/chalcopyrite.

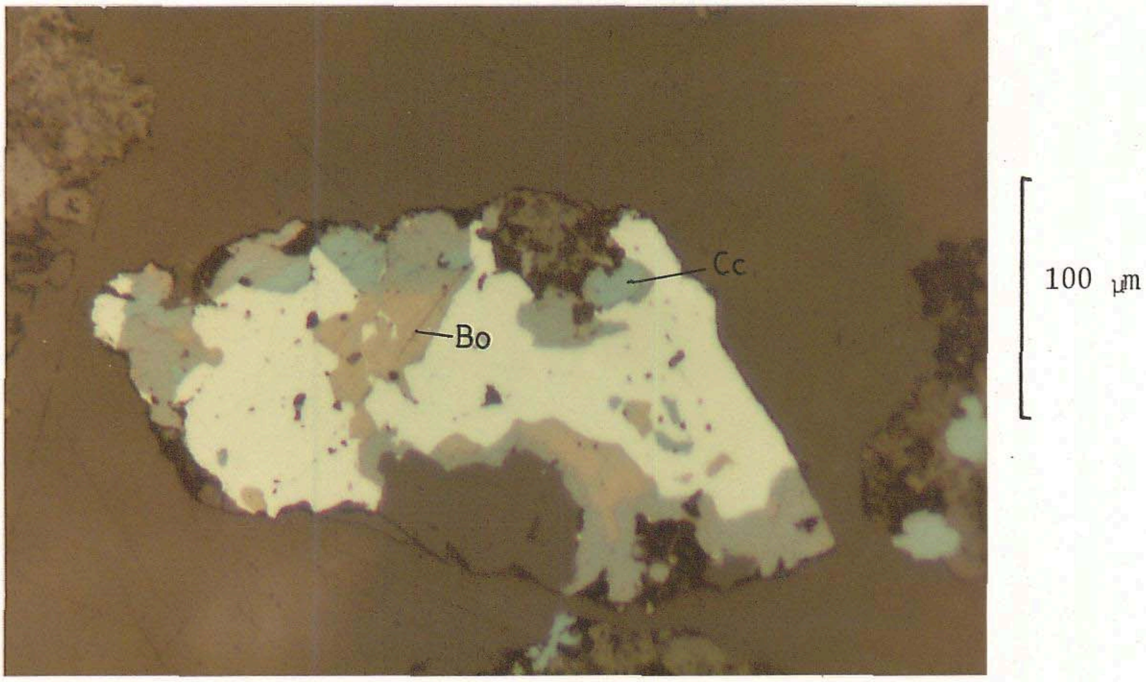


FIG. A9: +250 μm >3.3 Sp.Gr. Typical locking of carollite with copper sulphides (with sphalerite). Carollite occurs as relatively coarse grains.

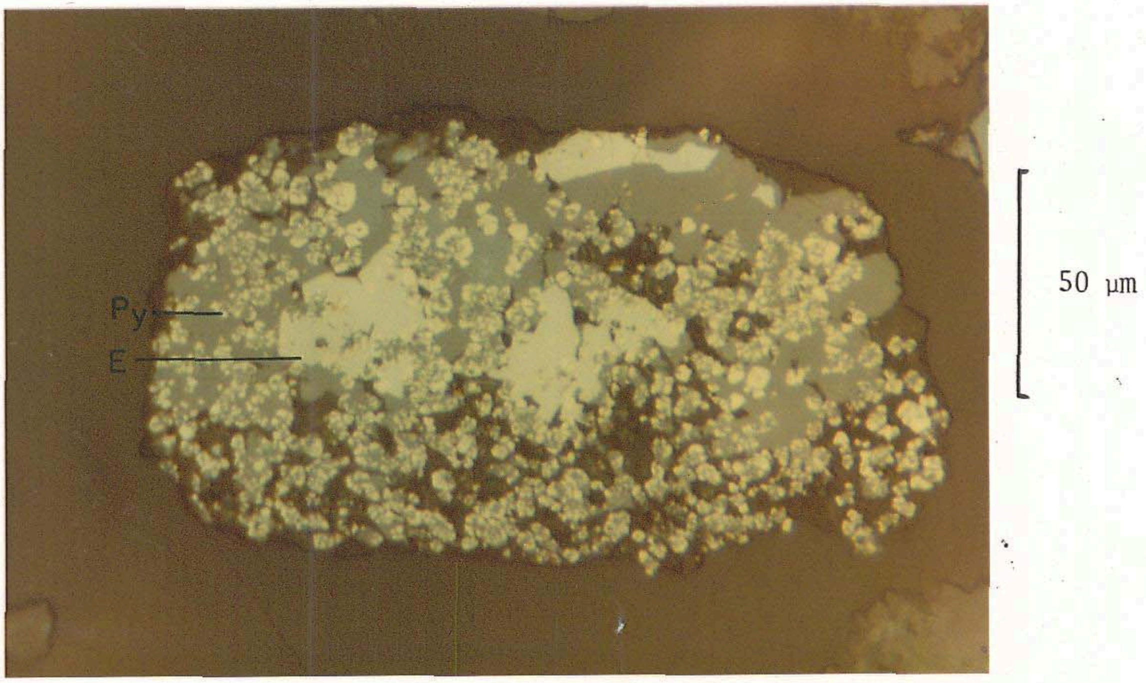


FIG. A10: -125+75 μm >3.3 Sp.Gr. Sphalerite containing pyrite crystals and emplectite/wittichenite patches.

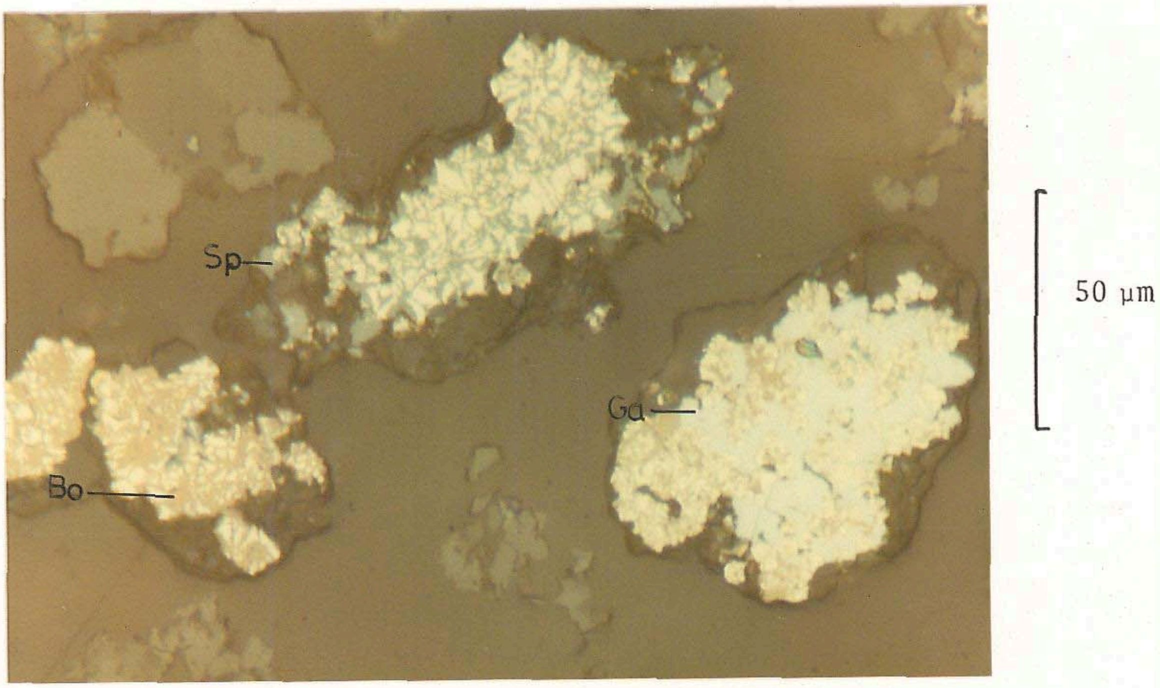


FIG. A11: -75+45 µm >3.3 Sp.Gr. Complex intergrowth of pyrite with bornite and sphalerite. Complex intergrowth of galena with bornite/pyrite.

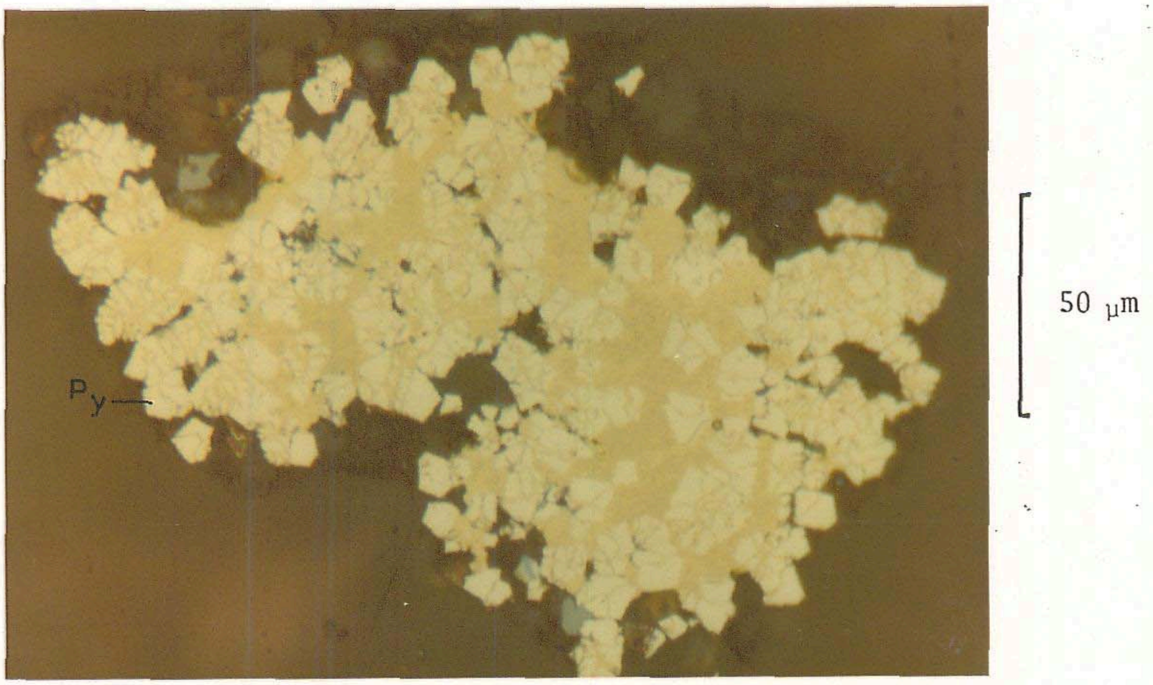


FIG. A12: -125+75 µm >3.3 Sp.Gr. Complex intergrowth of pyrite and chalcopyrite. This type of intergrowth is uncommon in this sample.

# APPENDIX B

## ADDITIONAL MINERALOGY EXAMINING POSSIBLE MAGNETIC SEPARATION ROUTES - MG14 DEPOSIT

### B1 INTRODUCTION

In discussions with Mr P. Bradshaw of Mt Gunson Mines Pty Limited, on 29 February, 1984 magnetic separation and chemical analysis was requested on selected size fractions of the MG14 deposit. Of particular interest was the extent of locking of valuable minerals with siderite.

### B2 PROCEDURE

The <3.3 and >3.3 Sp.Gr. products of the -75+45 μm and -250+125 μm fractions were separated with a hand magnet and with a Frantz isodynamic magnetic separator. The magnetic fractions were analysed for Cu and Ag by atomic absorption spectrometry.

### B3 RESULTS

The weight distribution and Cu and Ag assays and distributions of the magnetic fractions of the -75+45 μm and -250+125 μm specific gravity products are given in Tables B1 and B2 respectively. In the >3.3 Sp.Gr. product particles consisting mainly of siderite report into the 0.2A magnetic fraction, siderite-silicate composite particles report into the 0.2 to 0.4A fractions and particles consisting mainly of copper sulphides report into the 0.4A non-magnetic fraction. In the <3.3 Sp.Gr. product there was virtually no 0.2A magnetic material and hence 0.3A magnetic was selected to discriminate between particles with a moderagely high and moderately low proportion of siderite. The 0.4A non-magnetic material consists mainly of low-density silicates.

There is little association of Cu with particles containing a moderate to high proportion of siderite. For example in the -75+45 μm fraction these particles account for 29.1% of the weight yet only 4.2% of the copper. In the -75+45 μm fraction, 64.3% of the copper is concentrated in the >3.3 Sp.Gr. 0.4A non-mags, which assays 43.2% Cu and accounts for only 6.3% of the weight.

The location of the Ag is less easily characterised than that of Cu. However, only a minor proportion of the Ag appears to be associated with siderite and siderite-rich particles.

## TABLE B1: Cu DISTRIBUTION IN SPECIFIC GRAVITY AND MAGNETIC SEPARATION PRODUCTS OF THE -75+45 μm AND -250+125 μm FRACTIONS

| Particle Size (μm) | Sp.Gr. Product | Magnetic Fraction | Wt Distribution (%) in |  | Cu |  |  |
|---|---|---|---|---|---|---|---|
|  |  |  | Sp.Gr. Product | Size Fraction* | Assay, % | Distribution (%) in |  |
|  |  |  |  |  |  | Sp.Gr. Product | Size Fraction* |
| -250+125 | <3.3 | 0.3A Mags | 11.6 | 8.6 | 0.68 | 6.5 | 2.8 |
|  |  | 0.3-0.4A | 16.1 | 11.9 | 0.74 | 9.7 | 4.2 |
|  |  | 0.4A Non-Mags | 72.3 | 53.3 | 1.43 | 83.8 | 36.4 |
|  |  | Total | 100.0 | 73.8 | (1.23) | 100.0 | 43.4 |
|  | >3.3 | 0.2A Mags | 75.0 | 19.6 | 0.72 | 11.7 | 6.6 |
|  |  | 0.2-0.4A | 17.4 | 4.6 | 6.65 | 25.1 | 14.2 |
|  |  | 0.4A Non-Mags | 7.6 | 2.0 | 38.4 | 63.2 | 35.8 |
|  |  | Total | 100.0 | 26.2 | (4.62) | 100.0 | 56.6 |
| - 75+ 45 | <3.3 | 0.3A Mags | 13.2 | 9.1 | 0.49 | 5.0 | 1.4 |
|  |  | 0.3-0.4A | 10.3 | 7.1 | 0.70 | 5.6 | 1.5 |
|  |  | 0.4A Non-Mags | 76.5 | 52.8 | 1.50 | 89.4 | 24.1 |
|  |  | Total | 100.0 | 69.0 | (1.28) | 100.0 | 27.0 |
|  | >3.3 | 0.2A Mags | 64.6 | 20.0 | 0.60 | 3.4 | 2.8 |
|  |  | 0.2-0.4A | 15.0 | 4.7 | 5.40 | 8.1 | 5.9 |
|  |  | 0.4A Non-Mags | 20.4 | 6.3 | 43.2 | 88.0 | 64.3 |
|  |  | Total | 100.0 | 31.0 | (10.0) | 100.0 | 73.0 |

*Calculated using data from Table 1, AMDEL Report 6191/84#1.

## TABLE B2: Ag DISTRIBUTION IN SPECIFIC GRAVITY AND MAGNETIC SEPARATION PRODUCTS OF THE -75+45 µm AND -250+125 µm FRACTIONS

| Particle Size (µm) | Sp.Gr. Product | Magnetic Fraction | Wt Distribution (%) in | | Ag | | |
|---|---|---|---|---|---|---|---|
| | | | | | | Distribution (%) in | |
| | | | Sp.Gr. Product | Size Fraction* | Assay, ppm | Sp.Gr. Product | Size Fraction* |
| -250+125 | <3.3 | 0.3A Mags | 11.6 | 8.6 | 11 | 11.5 | 7.2 |
| | | 0.3-0.4A | 16.1 | 11.9 | 7 | 10.2 | 6.4 |
| | | 0.4A Non-Mags | 72.3 | 53.3 | 12 | 78.3 | 49.0 |
| | | Total | 100.0 | 73.8 | (11) | 100.0 | 62.6 |
| | >3.3 | 0.2A Mags | 75.0 | 19.6 | 10 | 20.7 | 7.7 |
| | | 0.2-0.4A | 17.4 | 4.6 | 70 | 33.6 | 12.6 |
| | | 0.4A Non-Mags | 7.6 | 2.0 | 210 | 45.7 | 17.1 |
| | | Total | 100.0 | 26.2 | (36) | 100.0 | 37.4 |
| - 75+ 45 | <3.3 | 0.3A Mags | 13.2 | 9.1 | 20 | 33.4 | 5.5 |
| | | 0.3-0.4A | 10.3 | 7.1 | 14 | 18.2 | 3.0 |
| | | 0.4A Non-Mags | 76.5 | 52.8 | 5 | 48.4 | 8.1 |
| | | Total | 100.0 | 69.0 | (8) | 100.0 | 16.6 |
| | >3.3 | 0.2A Mags | 64.6 | 20.0 | 15 | 16.9 | 13.9 |
| | | 0.2-0.4A | 15.0 | 4.7 | 45 | 11.8 | 9.7 |
| | | 0.4A Non-Mags | 20.4 | 6.3 | 200 | 71.3 | 58.8 |
| | | Total | 100.0 | 31.0 | (57) | 100.0 | 82.4 |

*Calculated using data from Table 2, AMDEL Report 6191/84#1.

# APPENDIX C

## LABORATORY TEST WORKSHEETS

INDEX

| Test No. | Worksheet No. | Grind P80 µm* | Brief Test Description |
|---|---|---|---|
| MG14-F1 | W1 | 210 | Initial test with grind approximating required plant operations - heavy reagent additions required and abandoned when no froth with T401 etc. |
| MG14-F2 | W2 | 210 | Repeat of first test - reagent additions adjusted. |
| MG14-F3 | W3 | 142 | Repeated MG14-F2 with finer grind and same reagent regime. |
| MG14-F4 | W4 | 115 | Repeated MG14-F2 with still finer grind and same reagent regime - cut test short as faster float occurring. |
| MG14-F5 | W5 | 142 | Repeated MG14-F3 with Soda Ash in grind to give Rougher pH ≈ 9.0. Tried also replacing some PAX with AF238 after normal float without any affect. |
| MG14-F6 | W6 | 142 | Repeated MG14-F3 with considerably reduced additions PAX and added small amounts SF323 as in plant operations. |
| MG14-F7 | W7 | 142 | After grinding Split Float Feed into sand and slime fractions by hydrocyclone and separately floated sand and slimes. Used PAX on sands and PAX SF323 on slimes. |
| MG14-F8 | W8 | 115 | Repeated Finest Grind of MG14-F4 followed by desliming step by hydrocyclone and floating sand fraction only with PAX. |

*From Fig. 1 using P80 - Time Relationship for 12 Rods in Charge.

PROJECT...OD. 3|8|13|2.......

SAMPLE.....MG.14.............

TEST..MG14 - F1.................

DATE...12-3-84..............

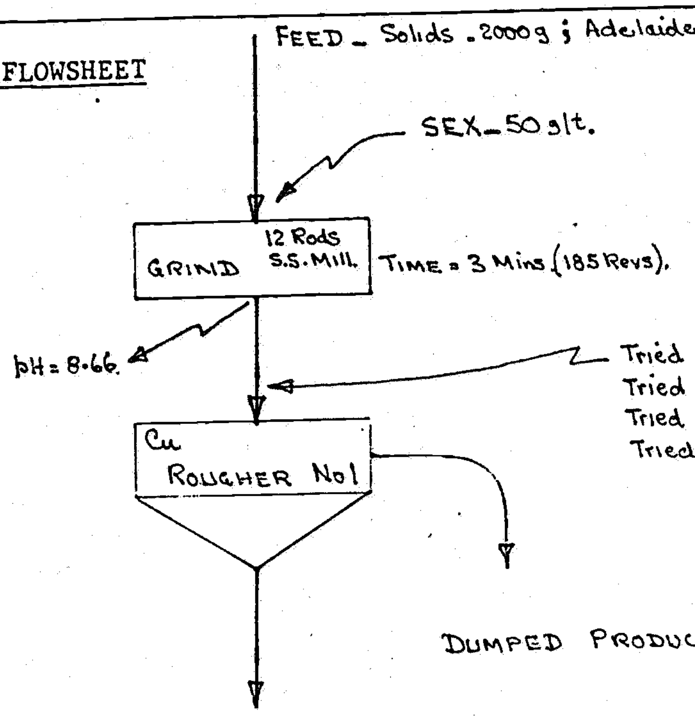
TESTED BY. G. Dunlop........N. Gray.

OBJECTIVE. Sighter Test Using Grind Abbroximating Plant Operations.

30556

WORKSHEET – W1.

**FLOWSHEET**



Tried 2d T401 – No Float, Very Little Sulphides Evident.
Tried 50 g/t SEX, 2d T401 – No Float, Some Sulphides Showing.
Tried 100 g/t PAX, 4d MIBC – No Float, Sulphides Showing.
Tried 100 g/t PAX, 4d MIBC – Obtained Froth But By Now Skin Float So Too Much Collector. In The First Stage of the Float.

DUMPED PRODUCTS – TEST ABORTED.

REAGENTS :
PAX – 300 g/t.
T401 – Abandoned, No Froth
MIBC – To Suit Float.

COMMENTS

Very Difficult Float – Not Only Did The T401 Not Give Froth But Initial Collector Additions Showed Few Signs Of Sulphides Floating.

METALLURGICAL BALANCE

| Product | Weight | | Assay % or g/t* | | | | Unit | | | | Distribution % | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | g | % | | | | | | | | | | | | |
| | | | | | | | | | | | | | | |
| | | | | | | | | | | | | | | |
| | | | | | | | | | | | | | | |
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| | | | | | | | | | | | | | | |
| | | | | | TEST ABORTED. | | | | | | | | | |
| | | | | | | | | | | | | | | |
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| | | | | | | | | | | | | | | |
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| | | | | | | | | | | | | | | |

PROJECT....OD. 3/8/13/3......

SAMPLE.....MG 14.............

TEST..MG14 – F2..............

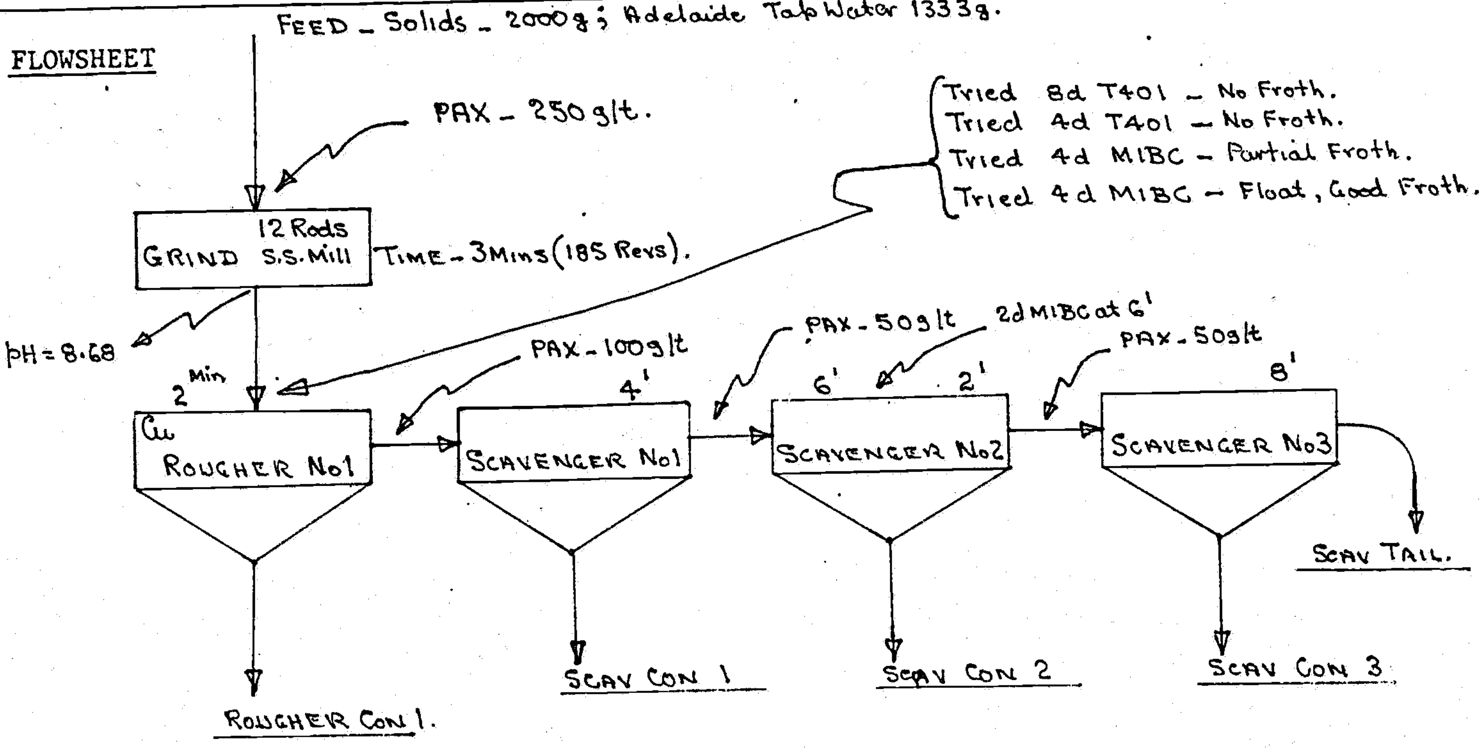
DATE...12-3-84...............

TESTED BY..G. Dunlop....N. Gray....

OBJECTIVE..Initial Test_ Adjusted reagents to suit float requirements.

WORKSHEET – W2.

FLOWSHEET



REAGENTS:
PAX _ 450 g/t.
T401 – Abandoned – No Froth.
MIBC – To Suit Float.

COMMENTS: Once Past The Initial Froth Problem, Good Sulphides Floated To Ro Con With Slowly Floating Sulphides Thereafter _ It Just Kept Floating With Time.

METALLURGICAL BALANCE

| Product | Weight | | Assay % or g/t * | | | | Unit | | | | Distribution % | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | g | % | Cu | | | | Cu | | | | Cu | | | |
| Cu Ro Con | 45.20 | 2.28 | 34.8 | | | | 79.34 | | | | 36.8 | | | |
| Cu Scav Con 1 | 33.12 | 1.67 | 16.3 | | | | 27.22 | | | | 12.6 | | | |
| | | (3.95) | (27.0) | | | | (106.56) | | | | (49.4) | | | |
| Cu Scav Con 2 | 44.08 | 2.22 | 12.7 | | | | 28.19 | | | | 13.0 | | | |
| | | (6.17) | (21.8) | | | | (134.75) | | | | (62.4) | | | |
| Cu Scav Con 3 | 19.59 | 0.99 | 7.75 | | | | 7.67 | | | | 3.6 | | | |
| | | (7.16) | (19.9) | | | | (142.42) | | | | (66.0) | | | |
| Cu Scav Tail | 1840.4 | 92.84 | 0.79 | | | | 73.34 | | | | 34.0 | | | |
| Calc Head. | 1982.25 | 100.00 | 2.16 | | | | 215.76 | | | | 100.0 | | | |
| | | | | | | | | | | | | | | |
| | | | | | | | | | | | | | | |
| | | | | | | | | | | | | | | |

PROJECT..OD 3/8/13/2........

SAMPLE...MG 14..............

TEST..MG14 - F3.............

DATE....12-3-84.............

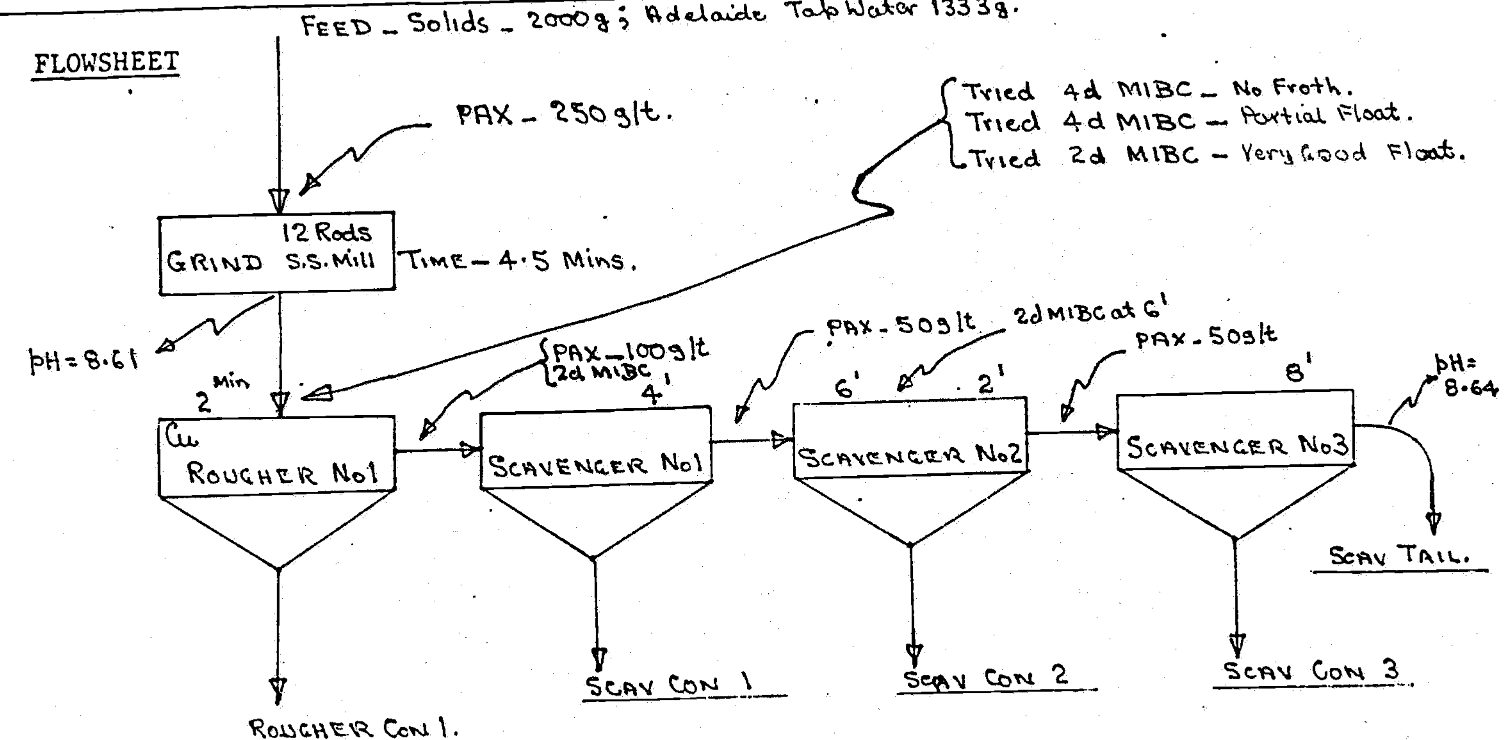
TESTED BY. G. Dunlop....... N. Gray...

OBJECTIVE. Follows. Test. MG14-F2. But With Finer Grind. Frother Changed To MIBC As No Response In Earlier Test With T401.

000558

WORKSHEET W3

FLOWSHEET



REAGENTS : PAX _ 450 g/t
MIBC _ To Suit Float.

COMMENTS : MIBC Gives A Good Froth. Finer Grind Marginally Increases Recovery.

METALLURGICAL BALANCE

| Product | Weight |  | Assay % or g/t* |  |  |  | Unit |  |  |  | Distribution % |  |  |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
|  | g | % | Cu |  |  |  | Cu |  |  |  | Cu |  |  |
| Cu Ro Con | 57.31 | 2.88 | 32.3 |  |  |  | 93.02 |  |  |  | 41.1 |  |  |
| Cu Scav Con 1 | 39.68 | 2.00 | 15.9 |  |  |  | 31.80 |  |  |  | 14.0 |  |  |
|  |  | (4.88) | (25.6) |  |  |  | (124.82) |  |  |  | (55.1) |  |  |
| Cu Scav Con 2 | 50.11 | 2.53 | 9.45 |  |  |  | 23.91 |  |  |  | 10.6 |  |  |
|  |  | (7.41) | (20.1) |  |  |  | (148.73) |  |  |  | (65.7) |  |  |
| Cu Scav Con 3 | 20.03 | 1.01 | 7.25 |  |  |  | 7.32 |  |  |  | 3.2 |  |  |
|  |  | (8.42) | (18.5) |  |  |  | (156.05) |  |  |  | (68.9) |  |  |
| Cu Scav Tail | 1817.1 | 91.57 | 0.77 |  |  |  | 70.51 |  |  |  | 31.1 |  |  |
| Calc Head | 1984.23 | 100.00 | 2.27 |  |  |  | 226.56 |  |  |  | 100.0 |  |  |
|  |  |  |  |  |  |  |  |  |  |  |  |  |  |
|  |  |  |  |  |  |  |  |  |  |  |  |  |  |
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PROJECT..OD 3/8/13/2......

SAMPLE.....MG 14.............

TEST. MG14 - F4..............

DATE...12-3-84..............

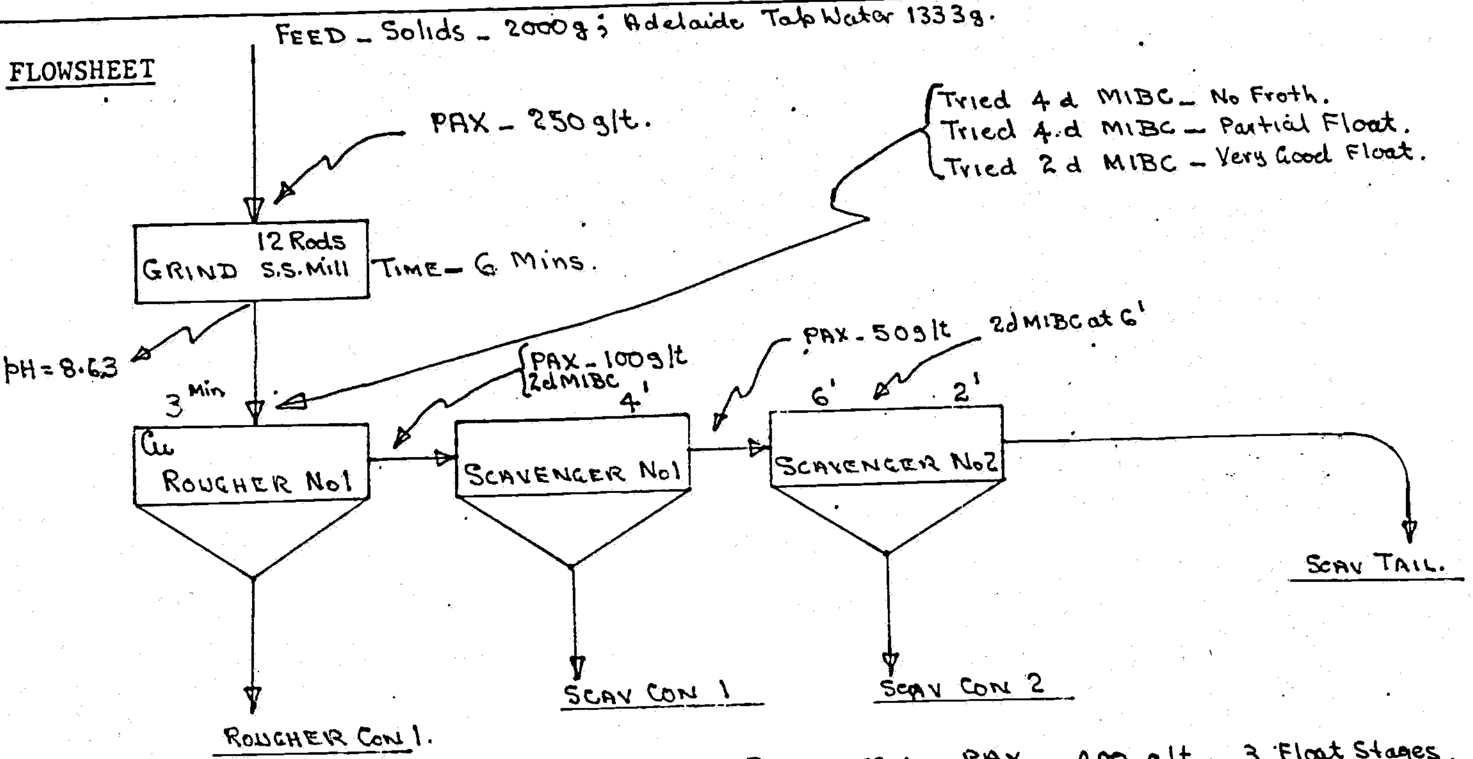
TESTED BY. C. Dunlop....N. Gray...

OBJECTIVE. Follows Test MG14-F3 But With Finer Grind, Deleted One Scavenging Stage.

009557

WORKSHEET W4.

**FLOWSHEET**



REAGENTS: PAX - 400 g/t - 3 Float Stages.
MIBC - To Suit Float.

COMMENTS: Finer Grind Gives Improved Recovery cmf Test MG14 F2, With Usual 4 Float Stages of Other Tests Could Have Expected > 75 % Cu Recovery (See Fig ).

METALLURGICAL BALANCE

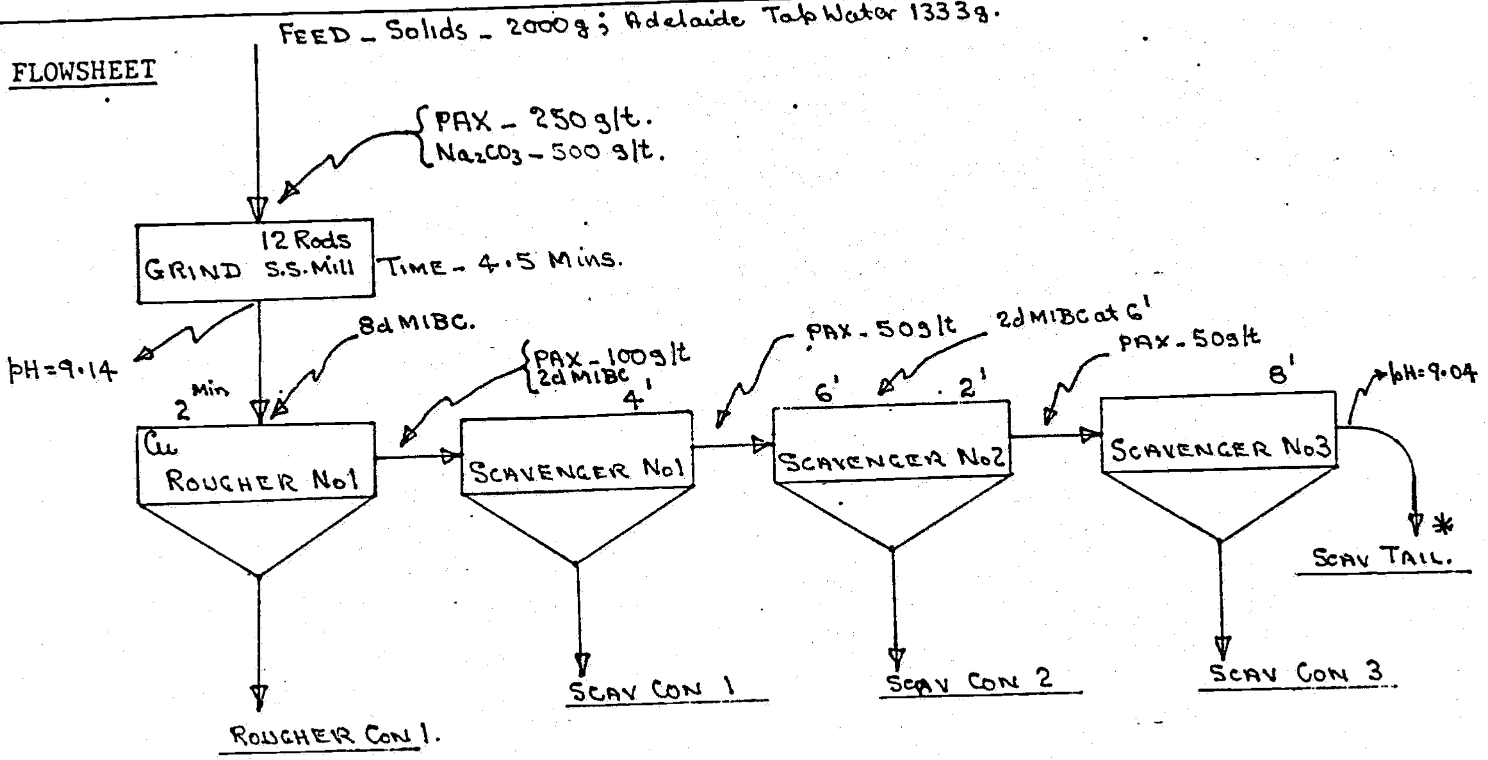
| Product | Weight |  | Assay % or g/t* |  |  |  | Unit |  |  |  | Distribution % |  |  |  |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
|  | g | % | Cu |  |  |  | Cu |  |  |  | Cu |  |  |  |
| Cu Ro Con | 64.85 | 3.27 | 32.5 |  |  |  | 106.27 |  |  |  | 47.7 |  |  |  |
| Cu Scav Con 1 | 43.57 | 2.20 | 13.1 |  |  |  | 28.82 |  |  |  | 12.9 |  |  |  |
|  |  | (5.47) | (24.7) |  |  |  | (135.09) |  |  |  | (60.6) |  |  |  |
| Cu Scav Con 2 | 53.64 | 2.71 | 8.55 |  |  |  | 23.17 |  |  |  | 10.4 |  |  |  |
|  |  | (8.18) | (19.4) |  |  |  | (158.26) |  |  |  | (71.0) |  |  |  |
|  |  |  |  |  |  |  |  |  |  |  |  |  |  |  |
|  |  |  |  |  |  |  |  |  |  |  |  |  |  |  |
| Cu Scav Tail | 1821.7 | 91.92 | 0.70 |  |  |  | 64.34 |  |  |  | 29.0 |  |  |  |
| Calc Head. | 1981.76 | 100.00 | 2.23 |  |  |  | 222.60 |  |  |  | 100.0 |  |  |  |
|  |  |  |  |  |  |  |  |  |  |  |  |  |  |  |
|  |  |  |  |  |  |  |  |  |  |  |  |  |  |  |

PROJECT..OD. 3|8 |13|2.......
SAMPLE...MG 14..........
TEST..MG14 - F5..........
DATE...12- 3- 84..........

TESTED BY. G. Dunlop...... N. Gray.
OBJECTIVE. Use Na$_2$CO$_3$ To Provide Some Dispersion And A Higher pH. Same Grind As Test MG14-F3.

WORKSHEET W 5.

FLOWSHEET



* Tried
- 500 g/t Na$_2$CO$_3$ To pH = 9.57
- 50 g/t AF238
- 150 g/t PAX
- 2d MIBC

No Further Material Floated.

**REAGENTS :**
PAX - 450 g/t.
Na$_2$CO$_3$ - 500 g/t.
MIBC - To Suit Float.

**COMMENTS :** cmf Test MG14 - F3 At Same Grind - Soda Ash For Partial Dispersion Has Not Helped Recovery - Down 5% at End, 10% Initially In Ro Con.

METALLURGICAL BALANCE

| Product | Weight |  | Assay % or g/t* |  |  |  | Unit |  |  |  | Distribution % |  |  |  |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
|  | g | % | Cu |  |  |  | Cu |  |  |  | Cu |  |  |  |
| Cu Ro Con | 41.92 | 2.12 | 34.5 |  |  |  | 73.14 |  |  |  | 31.7 |  |  |  |
| Cu Scav Con. | 42.45 | 2.14 | 18.1 |  |  |  | 38.73 |  |  |  | 16.8 |  |  |  |
|  |  | (4.26) | (26.3) |  |  |  | (111.87) |  |  |  | (48.5) |  |  |  |
| Cu Scav Con 2 | 45.50 | 2.30 | 10.6 |  |  |  | 24.38 |  |  |  | 10.6 |  |  |  |
|  |  | (6.56) | (20.8) |  |  |  | (136.25) |  |  |  | (59.1) |  |  |  |
| Cu Scav Con 3 | 32.00 | 1.62 | 6.80 |  |  |  | 11.02 |  |  |  | 4.8 |  |  |  |
|  |  | (8.18) | (18.0) |  |  |  | (147.27) |  |  |  | (63.9) |  |  |  |
| Cu Scav Tail | 1818.9 | 91.83 | 0.91 |  |  |  | 83.57 |  |  |  | 36.1 |  |  |  |
| Calc Head. | 1980.77 | 100.00 | 2.31 |  |  |  | 230.84 |  |  |  | 100.0 |  |  |  |

PROJECT OD 3|8|13|2

SAMPLE MG14

TEST MG14 - F6

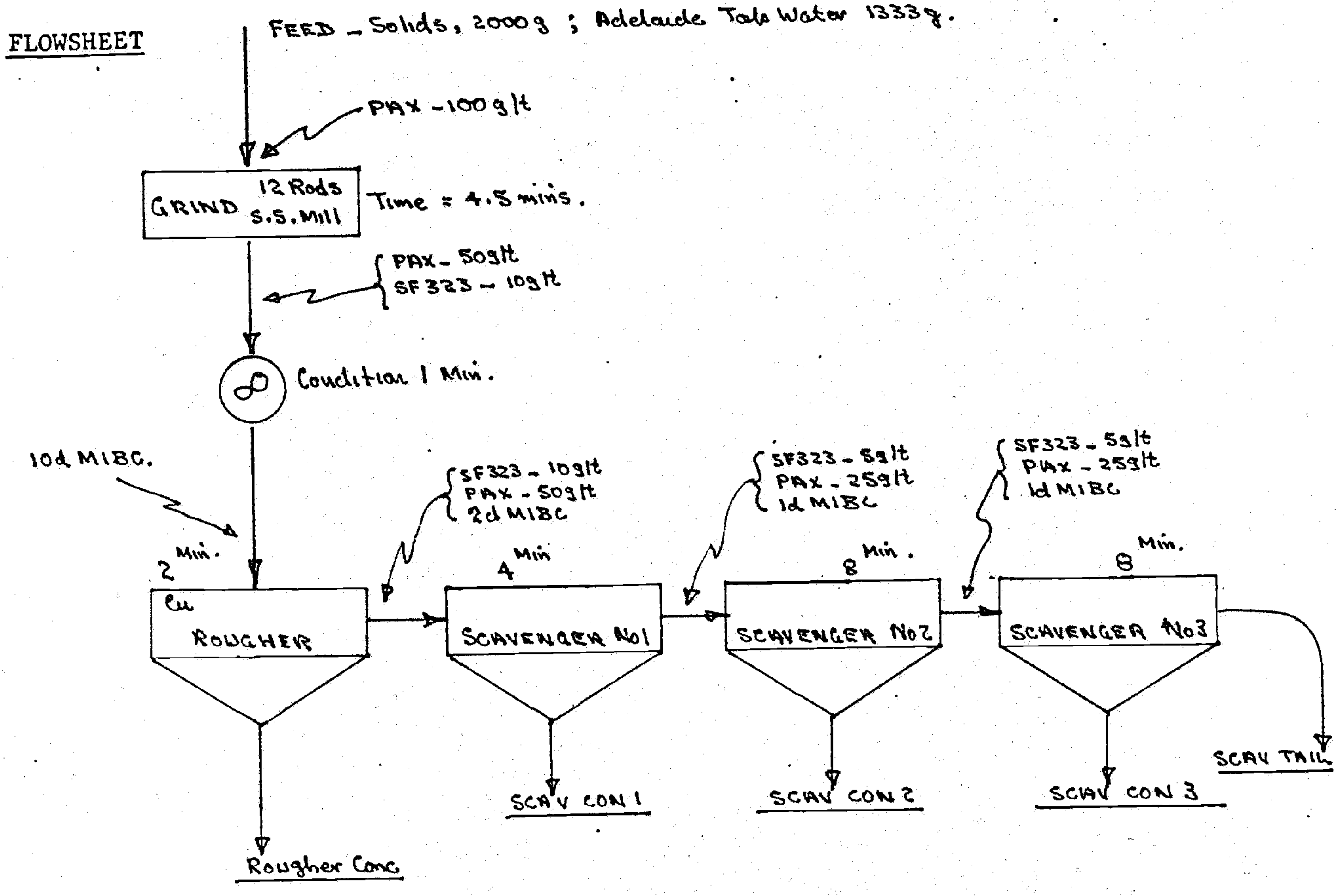
DATE 25-4-84

TESTED BY G. Dunlop - T. Wu.

OBJECTIVE Test With Reagent SF323 In Use Following Test 3 But With Considerably Reduced Reagent Additions And A Short Conditioning Stage Added.

WORKSHEET W6.

FLOWSHEET



REAGENTS : PAX - 250g/t
SF323 - 30g/t
MIBC - To Suit Float.

METALLURGICAL BALANCE

| Product | Weight | | Assay % or g/t* | | | | Unit | | | | Distribution % | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | g | % | Cu | | | | Cu | | | | Cu | | | |
| Cu Ro Con | 37.63 | 1.90 | 38.9 | | | | 73.74 | | | | 31.2 | | | |
| Cu Scav Con 1 | 17.04 | 0.86 | 24.4 | | | | 20.94 | | | | 8.9 | | | |
| | | (2.76) | (34.4) | | | | (94.68) | | | | (40.1) | | | |
| Cu Scav Con 2 | 21.48 | 1.08 | 18.5 | | | | 20.02 | | | | 8.5 | | | |
| | | (3.84) | (29.9) | | | | (114.70) | | | | (48.6) | | | |
| Cu Scav Con 3 | 18.16 | 0.91 | 17.2 | | | | 15.73 | | | | 6.7 | | | |
| | | (4.75) | (27.5) | | | | (130.43) | | | | (55.2) | | | |
| Cu Scav Tail | 1890.9 | 95.25 | 1.11 | | | | 105.73 | | | | 44.8 | | | |
| | | | | | | | | | | | | | | |
| Calc. Head | 1985.21 | 100.00 | 2.36 | | | | 236.16 | | | | 100.0 | | | |
| | | | | | | | | | | | | | | |
| | | | | | | | | | | | | | | |

PROJECT ..OD 3|8|13|2.......... TESTED BY .G. Dunlop - T. Wu.

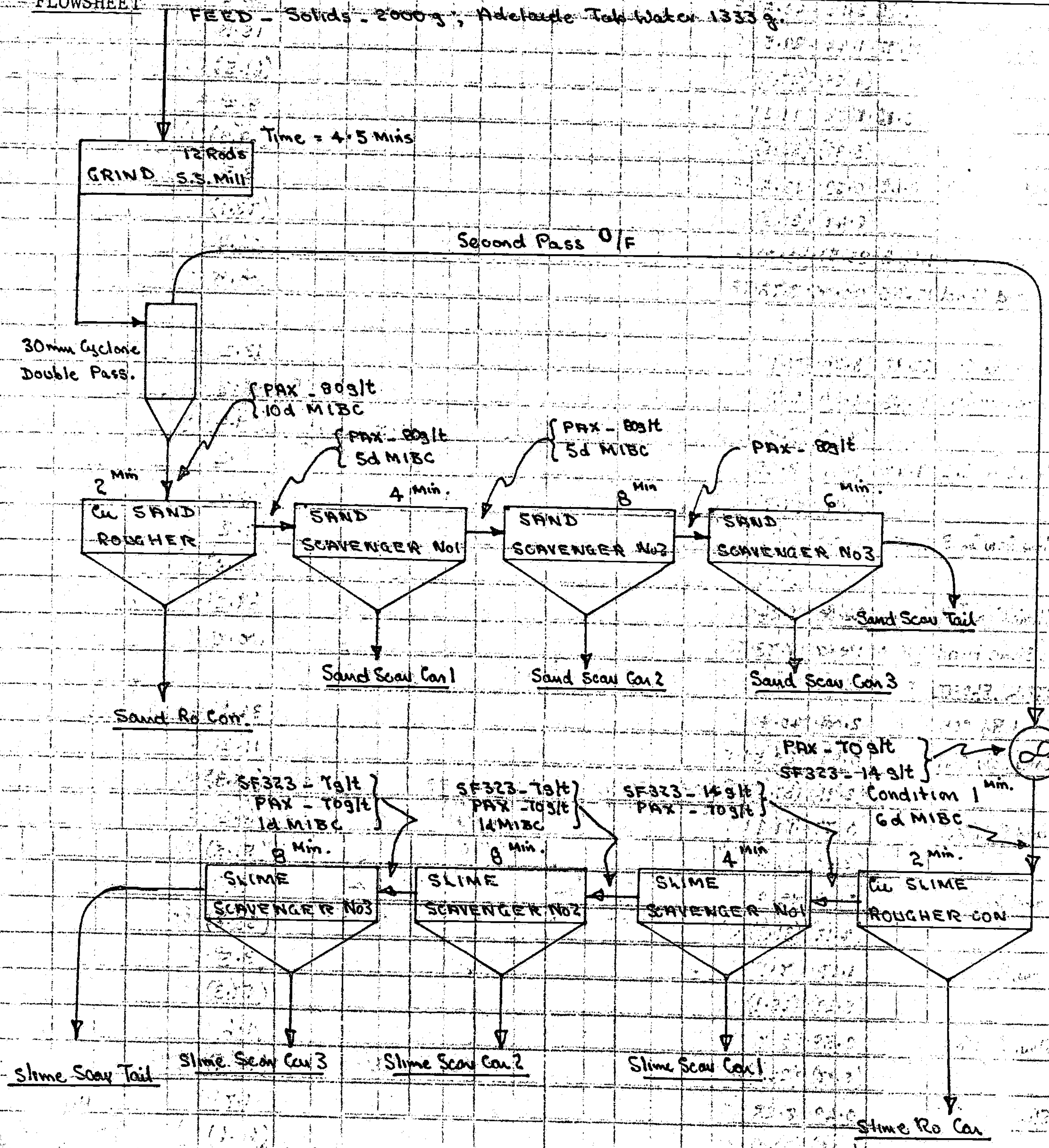
SAMPLE ...MG 14........ OBJECTIVE ..Sand - Slime Flotation Employed.

TEST ...MG14 - F7.

DATE ..25 - 4 - 84.

WORKSHEET W7.

FLOWSHEET



REAGENTS : Given In Per Tonne Of Feed To :

| (g/t) | Sand | Slime | Total Float. |
|---|---|---|---|
| PAX | 320 | 280 | 300 |
| SF323 | - | 42 | 15 |
| MIBC. | To Suit | To Suit | To Suit. |

NB: Reagent Additions on The Flowsheet are g/t of Float Section Feed.

TEST MG14 – F7.

METALLURGICAL BALANCE

| Product | Weight | | Assay % or g/t* | | | | Unit | | | | Distribution % | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| XXXXXXXXXXXX | g | % | Cu | | | | Cu | | | | Cu | | | |
| SAND. | | | | | | | | | | | | | | |
| Cu Sand Ro Con | 39.24 | 3.14 | 40.8 | | | | | | | | 46.2 | | | |
| Cu Sand Scav Con1 | 17.92 | 1.44 | 29.5 | | | | | | | | 15.3 | | | |
| | | (4.58) | (37.3) | | | | | | | | (61.5) | | | |
| Cu Sand Scav Con2 | 15.26 | 1.22 | 19.2 | | | | | | | | 8.4 | | | |
| | | (5.80) | (33.5) | | | | | | | | (69.9) | | | |
| Cu Sand Scav Con3 | 8.62 | 0.69 | 12.8 | | | | | | | | 3.2 | | | |
| | | (6.49) | (31.3) | | | | | | | | (73.1) | | | |
| Cu Sand Scav Tail | 1166.6 | 93.51 | 0.8 | | | | | | | | 26.9 | | | |
| Calc Sand Head | 1247.5 | 100.00 | 2.78 | | | | | | | | 100.0 | | | |
| SLIME | | | | | | | | | | | | | | |
| Cu Slime Ro Con | 22.98 | 3.20 | 7.1 | | | | | | | | 13.2 | | | |
| Cu Slime Scav Con1 | 11.46 | 1.60 | 6.7 | | | | | | | | 6.2 | | | |
| | | (4.80) | (7.0) | | | | | | | | (19.4) | | | |
| Cu Slime Scav Con2 | 9.44 | 1.31 | 8.25 | | | | | | | | 6.2 | | | |
| | | (6.11) | (7.2) | | | | | | | | (25.6) | | | |
| Cu Slime Scav Con 3 | 13.53 | 1.89 | 6.6 | | | | | | | | 7.2 | | | |
| | | (8.00) | (7.1) | | | | | | | | (32.8) | | | |
| Cu Slime Scav Tail | 660.6 | 92.00 | 1.26 | | | | | | | | 67.2 | | | |
| Calc Slime Head | 718.0 | 100.00 | 1.73 | | | | | | | | 100.0 | | | |
| TOTAL FLOAT. | | | | | | | | | | | | | | |
| Cu Sand Ro CON | | 2.00 | 40.8 | | | | | | | | 34.1 | | | |
| Cu Sand Scav Con1 | | 0.91 | 29.5 | | | | | | | | 11.2 | | | |
| | | (2.91) | (37.3) | | | | | | | | (45.3) | | | |
| Cu Sand Scav Con2 | | 0.77 | 19.2 | | | | | | | | 6.2 | | | |
| | | (3.68) | (33.5) | | | | | | | | (51.5) | | | |
| Cu Sand Scav Con3 | | 0.44 | 12.8 | | | | | | | | 2.3 | | | |
| | | (4.12) | (31.3) | | | | | | | | (53.8) | | | |
| Cu Slime Ro Con | | 1.17 | 7.1 | | | | | | | | 3.5 | | | |
| | | (5.29) | (25.9) | | | | | | | | (57.3) | | | |
| Cu Slime Scav Con1 | | 0.58 | 6.7 | | | | | | | | 1.6 | | | |
| | | (5.87) | (24.6) | | | | | | | | (58.9) | | | |
| Cu Slime Scav Con2 | | 0.48 | 8.25 | | | | | | | | 1.7 | | Heads | |
| | | (6.35) | (22.8) | | | | | | | | (60.6) | | Cu Slime | Cu Sand |
| Cu Slime Scav Con3 | | 0.69 | 6.6 | | | | | | | | 1.9 | % Wgt | 36.53 | 63.47 |
| | | (7.04) | (21.25) | | | | | | | | (62.5) | % Cu | 1.73 | 2.78 |
| | | | | | | | | | | | | Dist % | 26.4 | 73.6 |
| Cu Slime Scav Tail | | 33.61 | 1.26 | | | | | | | | 17.7 | | | |
| Cu Sand Scav Tail | | 59.35 | 0.80 | | | | | | | | 19.8 | | | |
| (Cu Total Tail) | | (92.96) | (0.97) | | | | | | | | (37.5) | | | |

PROJECT OD 3/8/13/2

SAMPLE MG 14

TEST MG 14 – F8

DATE 10-5-84

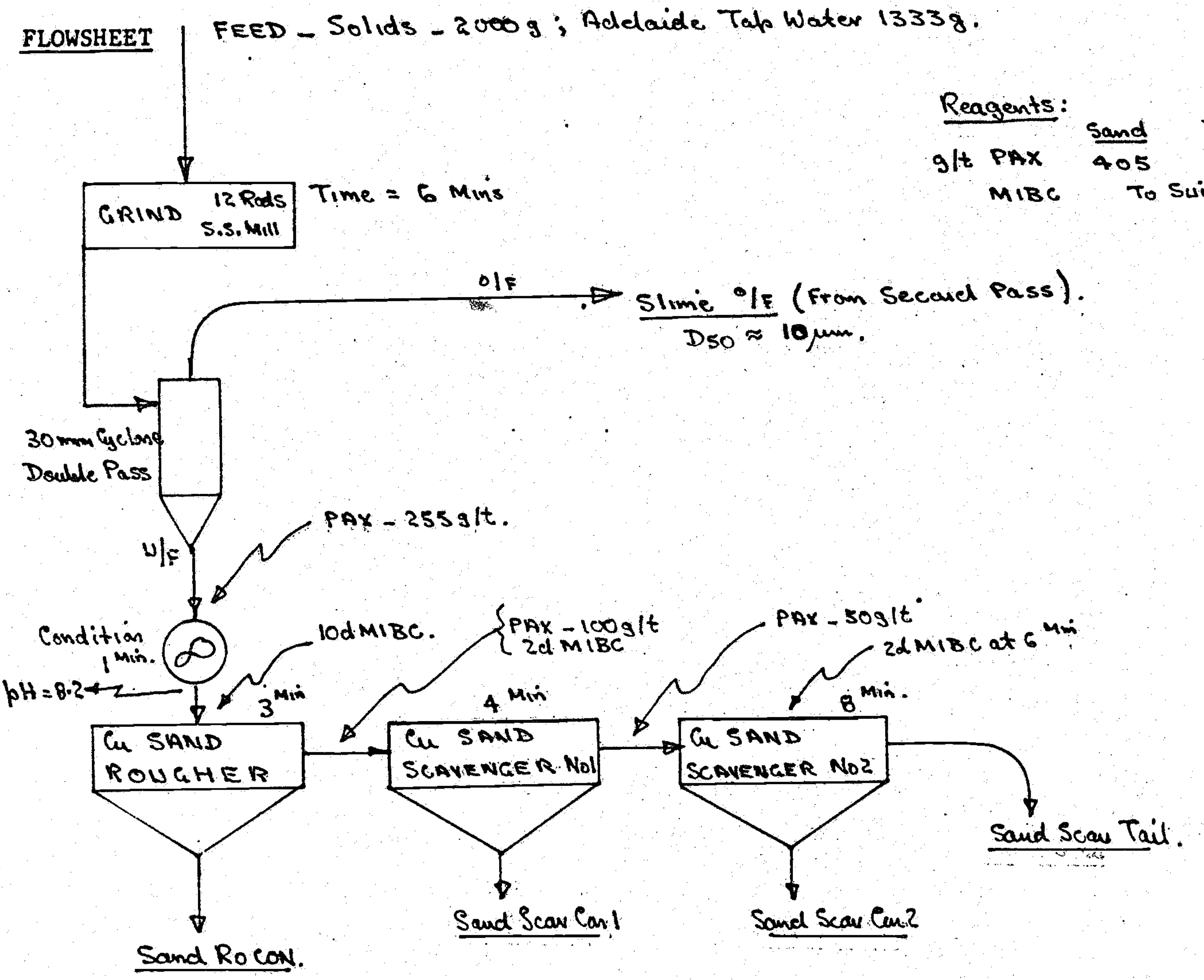
TESTED BY G. Dunlop – T. Wu

OBJECTIVE Flotation of Deslimed Feed Using A Fine Grind (Ex Test 4).

30562

WORKSHEET W8.

**FLOWSHEET**



Reagents:

|  | Sand | Total Feed |
|---|---|---|
| g/t PAX | 405 | 320 |
| MIBC | To Suit Floats. |  |

METALLURGICAL BALANCE

| Product | Weight |  | Assay % or g/t |  |  |  | Unit |  |  |  | Distribution % |  |  |  |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
|  | g | % | Cu |  |  |  | Cu |  |  |  | Cu |  |  |  |
| Cu Sand Ro Con | 69.4 | 3.76 | 31.0 |  |  |  | 116.56 |  |  |  | 50.7 |  |  |  |
| Cu Sand Scav Con 1 | 17.0 | 0.92 | 16.7 |  |  |  | 15.34 |  |  |  | 6.7 |  |  |  |
|  |  | (4.68) | (28.2) |  |  |  | (131.92) |  |  |  | (57.4) |  |  |  |
| Cu Sand Scav Con 2 | 28.7 | 1.55 | 11.1 |  |  |  | 17.21 |  |  |  | 7.5 |  |  |  |
|  |  | (6.23) | (23.9) |  |  |  | (149.13) |  |  |  | (64.9) |  |  |  |
| Cu Sand Scav Tail | 1349.5 | 73.06 | 0.68 |  |  |  | 49.68 |  |  |  | 21.6 |  |  |  |
| alc Head Sands |  | (79.29) | (2.5) |  |  |  | (198.81) |  |  |  | (86.5) |  |  |  |
|  |  |  |  |  |  |  |  |  |  |  |  |  |  |  |
| Slime O/F | 382.5 | 20.71 | 1.50 |  |  |  | 31.07 |  |  |  | 13.5 |  |  |  |
| Calc Combined Tail |  | (93.77) | (0.86) |  |  |  | (80.75) |  |  |  | (35.1) |  |  |  |
| Calc Total Head | 1847.1 | 100.00 | 2.30 |  |  |  | 229.88 |  |  |  | 100.0 |  |  |  |