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SERVICE REPORT CM1/31/26/0

February 1976

REMNANT ORE FROM MT GUNSON

THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES

Adelaide South Australia

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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

JE 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

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.

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 renmant 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.

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
   $= \frac{1200}{8}$ 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 in 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 say, 6, 4/3 Warmans; i.e., 6 x $1200	7,200	
(b)	Cementation Flow = 150 kl/h say, 2 x 4/3 Warman; i.e., 2 x $1200	2,400	
(c)	Barren liquor recycle Flow - 50 kl/h Say, 2 x 3/2 Warman at $1000 ea. 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.

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.

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Report OR2298/76
August, 1976

MT GUNSON ORE RESERVES

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750/75

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019

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11 August, 1976.

The Director
Department of Mines,
South Australia
PO Box 151
EASTWOOD SA 5063

Attention: Mr W.B. Robinson

FINAL SERVICE REPORT

MT GUNSON ORE RESERVES

Investigation and Report by: Dr B. Leach

Officer in Charge
Operations Research/Computer Services Section: Dr W.J. Howarth

for F.R. Hartley
Director.

(weighting them by length) and if the average is above the cut-off grade, the interval is accepted. If the average is below the cut-off grade, the scan is continued upward until the next above cut-off grade assay is found. The procedure is repeated until the interval exceeds the cut-off grade.

When there are multiple intervals of above cut-off grade material, the top interval is arbitrarily selected as the one to be used. Examination for one particular cut-off grade has shown that the top interval is, in most cases, the thickest.

The interval is called the ore thickness or the mineable width.

Two cut-off grades have been considered in this report. They are 1% Cu and 0.8% Cu. In the latter case, a cut-off of 1% Cu was used to find the top of the interval while the 0.8% Cu cut-off was used to find the bottom of the interval.

For each drillhole the following information is available :-

hole code
depth to start of interval
thickness of interval
% Cu by weighted average
% Zn by weighted average.

There are a number of drillholes with no values above the cut-off grade.

4. CONTOURING

The orebody is two-dimensional in structure. It consists of a large oval sheet of up to 10 metres in thickness covering an area of approximately two million square metres. Because the orebody is relatively thin, it is mined to its full depth at a single operation. This factor is reflected in the ore reserve calculation procedure that follows.

The important variables are the thickness of the orebody and the amount of Cu metal (= thickness x Cu grade) in the orebody at the point of each drillhole. Another trio of interesting variables is the ratio of the thickness of overburden to the thickness of ore (i.e., waste to ore ratio), the amount of Zn metal and the ratio of Zn to Cu.

All of these variables are contoured using inverse squares weighting. The procedure is to establish a mesh of grid points and calculate a value at each grid point. The value at the grid point is obtained by taking all drillholes inside a circle of influence about the grid point, calculating a weighted average of the drillhole values and assigning it to the grid point. Separate contour maps have been calculated for each of the five variables noted above.

MT GUNSON ORE RESERVES

1. INTRODUCTION

Following discussions with Mr W.B. Robinson of the South Australian Department of Mines, Amdel was asked to make a mathematical study of the Cattlegrid orebody currently being mined by Mt Gunson Limited, a subsidiary of CSR Limited.

The orebody is a horizontal layer occurring at the interface between the Whyalla Sandstone and the Pandurra formation at an average depth of 40 metres.

Amdel's brief is to prepare a computer model of the deposit so that a range of cut-off grades can be used in the calculation of ore reserves estimates. Flexibility of approach is to be the main aim in building the model.

The orebody is currently being mined by open pit to a cut-off of 1% Cu with an average grade of ore mined of 2%.

2. DATA

All drilling data has been transferred to punch cards according to the formats in Tables 1 and 2. A deck of cards for a single drillhole consists of a header card, a detail card for each consecutive interval down the drillhole, and a final blank card to terminate the drillhole. There are approximately 4,800 cards in the deck.

There is a total of 255 drillholes for the Cattlegrid area. All drillholes are vertical and drilled in two stages. Rotary drilling is used until the orebody is almost reached and then coring is used to the end of the drillhole.

3. DRILLHOLE ANALYSIS

The first step is to select from each drillhole the interval that is to be taken as ore for the specified cut-off grade of copper. This turns out to be a simple process because of the downhole grade profile.

When grade of copper is examined from the top of the drillhole downwards, there is a clear step from low values of less than 0.5% Cu to values of 1% Cu or greater. Thus for any cut-off grade in the range of 0.5% to 1% Cu, the top of the interval is clearly determined and the maximum change would be one sampling interval of 0.5 metres.

In the better drillholes, the plateau of plus 1% Cu assays is maintained for a few metres and then the grade falls away gradually to the limit of the drillhole. Even in falling grade tail there are high assays. The lower limit is taken initially as the first assay from the bottom of the drillhole that is above cut-off grade. All samples in the interval are then averaged

No contour plot has been drawn for percent Cu as it is not a continuous variable. Percent Cu can be obtained at any point as the ratio of amount of Cu metal to orebody thickness.

Four of the contour plots for the 1% Cu cut-off are presented here, having been traced from the computer printout format to contours on a base plan. The four contours are :-

Fig. 1: Mineable width
Fig. 2: Mineable width x Cu %
Fig. 3: Waste to ore ratio
Fig. 4: Ratio Zn% to Cu%.

The limit for contouring has been set as the limit of an acceptable density of drilling or else the termination of the orebody. This is done to limit the computer time and to allow a full-size contour map to be drawn on the computer line printer.

5. ORE RESERVES

An ore reserve is calculated by summing the values at the grid points. In fact, it is possible to calculate a range of ore reserves. The three variables that are used are ore thickness, amount of metal and waste to ore ratio.

At grid position I we have :-

ore thickness	T(I)
amount of metal	A(I)
waste to ore ratio	W(I).

Each T(I) is tested against the minimum mining width of 3 metres. Each W(I) is tested against the maximum value that is to be considered. If there are N grid points that satisfy the conditions on ore thickness and waste to ore ratio, the ore reserve estimate is obtained as follows :-

$$\text{volume of ore} = \sum_{I=1}^{N} T(I) \times G$$

$$\text{volume of overburden} = \sum_{I=1}^{N} T(I)\ W(I) \times G$$

$$\text{average grade} = \frac{\sum_{I=1}^{N} A(I)}{\sum_{I=1}^{N} T(I)}$$

The average grades for both Cu and Zn can be calculated using the grid values for amount of Cu and amount of Zn respectively.

The tonnage of overburden and ore can be obtained by multiplying the volume by the tonnage factor of 2.5 tonnes per cubic metre. The factor G is the area of an individual grid cell (6.35 x 10.5833 metres here).

The average waste to ore ratio can be obtained as the ratio of the volume of overburden to the volume of ore.

Results for the 1% Cu cut-off ore given in Table 3, and for the 0.8% Cu cut-off in Table 4.

The current mining limit of the 14 to 1 waste to ore ratio cut-off gives 37.05 m.t. of overburden for 4.92 m.t. of ore at an average grade of 2.01% Cu. Dropping the lower cut-off to 0.8% gives at a 12 to 1 waste to ore ratio cut-off 37.59 m.t. of overburden for 5.53 m.t. of ore at an average grade of 1.86% Cu.

6. CONCLUSIONS AND RECOMMENDATIONS

The calculated ore reserves of 2.01% Cu at an average waste to ore ratio of 7.53 to 1 give 4.92 m.t. of ore and agree well with the current mining results.

The overall impression gained from the contour plans is that the model agrees very well with the actual results.

The model developed can be used to test the various alternative proposed mining plans cheaply and easily.

TABLE 1: HEADER CARD

Columns	Field Description
1 - 10	hole code
11 - 20	northing (metres)
21 - 29	easting (metres)
30 - 35	elevation (metres)
36 - 41	total depth (metres)
42 - 46	bearing
47 - 51	dip
52 - 59	date drilling started
60 - 64	depth coring started
65 - 69	base of overburden
70 - 74	base of Whyalla Sandstone
75 - 79	base of Transition Zone
80	card type (H)

TABLE 2: DETAIL CARD

Columns	Field Description
1 - 10	hole code
15 - 20	bottom of interval
26 - 30	Cu (%)
33 - 37	Ag (ppm)
40 - 44	Pb (ppm)
47 - 51	Zn (ppm)
54 - 58	Co (ppm)
61 - 65	Bi (ppm)
80	card type (D)

NOTE: < is punched as - i.e., <2 becomes -2.

TABLE 3: ORE RESERVES FOR THE 1% Cu CUT-OFF

Waste to Ore Cut-Off	Overburden m.t.	Ore m.t.	% Cu	% Zn	Average Waste to Ore
6	7.85	1.68	1.95	0.39	4.66
8	17.21	3.04	2.01	0.39	5.66
10	24.70	3.88	2.03	0.39	6.36
12	31.21	4.47	2.02	0.38	6.98
14	37.05	4.92	2.01	0.37	7.53
All Ore	81.68	6.85	1.97	0.34	11.93

TABLE 4: ORE RESERVES FOR THE 0.8% Cu CUT-OFF

Waste to Ore Cut-Off	Overburden m.t.	Ore m.t.	% Cu	% Zn	Average Waste to Ore
6	11.17	2.43	1.87	0.37	4.60
8	20.29	3.77	1.89	0.37	5.39
10	29.04	4.75	1.90	0.36	6.12
12	37.59	5.53	1.88	0.35	6.80
14	45.52	6.14	1.86	0.34	7.42
All Ore	87.23	7.99	1.84	0.32	10.92

FIG. 1

amdel
The Australian Mineral Development Laboratories

Title
MOUNT GUNSON 1% COPPER CUT-OFF
MINEABLE WIDTH

2659-1

Drn G.S.	Appd	Scale 1:2500	Drg No 5162
Tcd	M Eng B.L.	Date 4·8·76	
Ckd			Sheet 1 of 1

FIG. 2

amdel
The Australian Mineral Development Laboratories

Title	MOUNT GUNSON 1% COPPER CUT-OFF MINEABLE WIDTH x Cu%		2659-2
Drn G.S.	Appd	Scale 1:2500	Drg No 5164
Tcd	M Eng B.L.	Date 4-8-76	Sheet 1 of 1
Ckd			

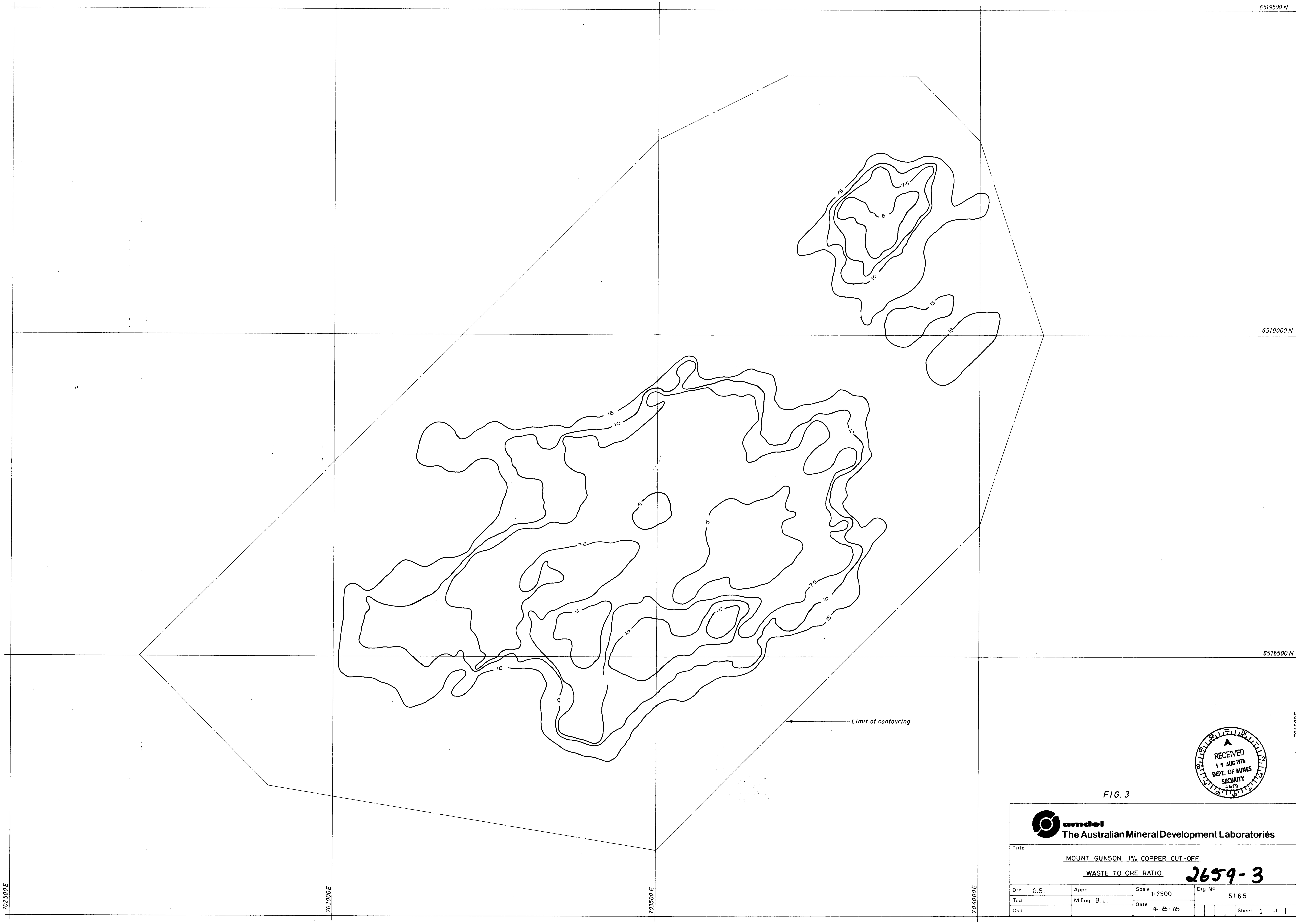

FIG. 3

amdel
The Australian Mineral Development Laboratories

Title
MOUNT GUNSON 1% COPPER CUT-OFF
WASTE TO ORE RATIO

2659-3

Drn G.S.	Appd M Eng B.L.	Scale 1:2500	Drg No 5165	
Tcd		Date 4·8·76		
Ckd				Sheet 1 of 1

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FIG. 4

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Title: MOUNT GUNSON 1% COPPER CUT-OFF RATIO Zn%/Cu%			2659-4
Drn G.S.	Appd	Scale 1:2500	Drg No 5163
Tcd.	M Eng B.L.	Date 4·8·76	
Ckd.			Sheet 1 of 1