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EL 73

MUTOOROO

MUTOOROO COPPER PROSPECT

ECONOMIC EVALUATION, JANUARY 1975

Submitted by
Noranda Australia Ltd and Electrolytic Zinc Co. of Australasia Ltd
1975

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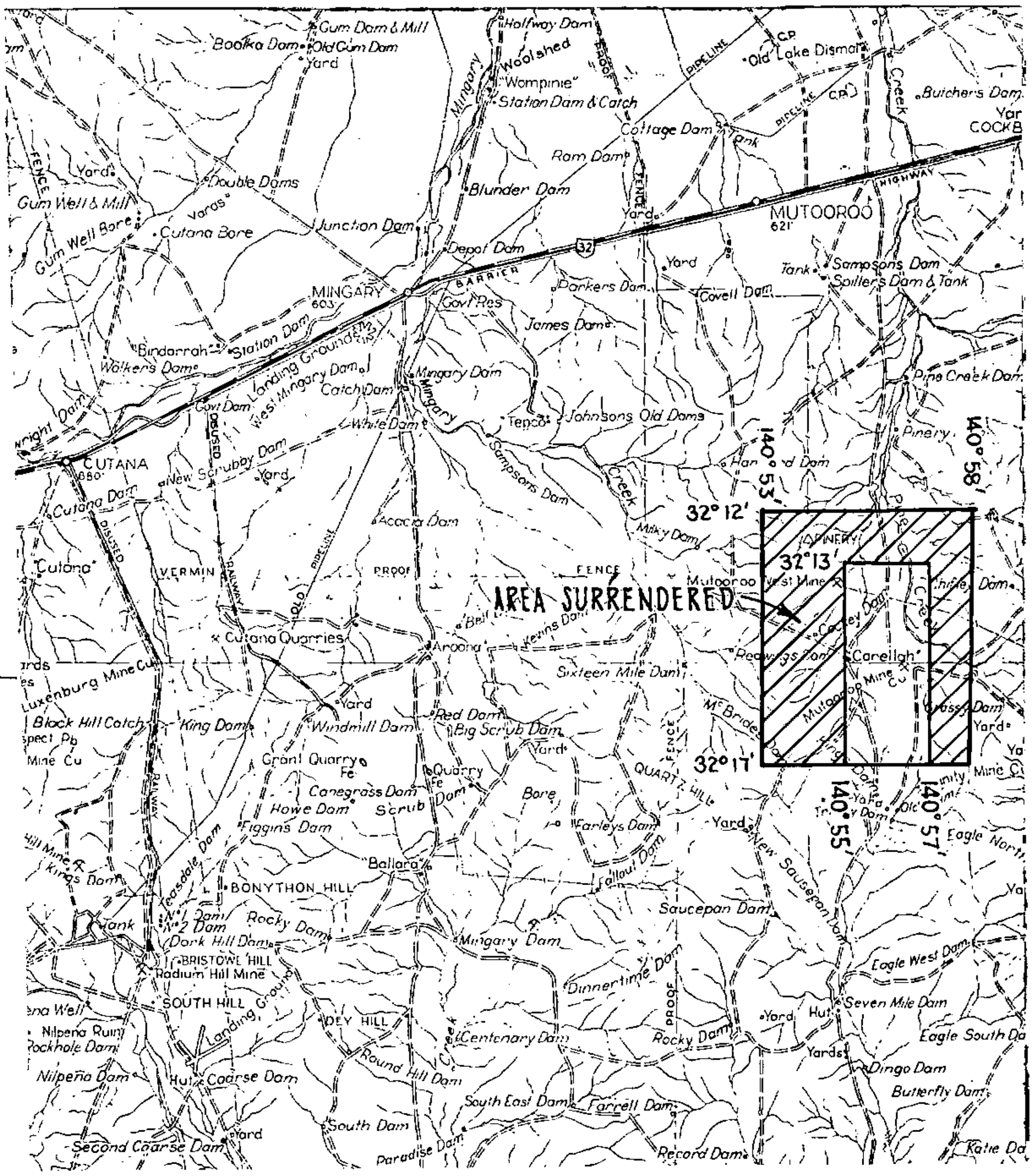
Government of South Australia
Department for Manufacturing, Innovation, Trade, Resources and Energy

SCHEDULE 'A'



MINES EXPLORATION PTY. LTD.

DOCKET DM. 469/73 AREA ~~73~~ km²

1:250000 PLANS OLARY REDUCED 24 km²

LOCALITY MUTOOROO MINE – APPROX. 10 Km S OF MUTOOROO

E.L.No. 73 . EXPIRY DATE ~~21-6-74~~

~~21-9-74~~ ~~21-3-75~~ 21-6-75

MUTOOROO COPPER PROSPECT

ECONOMIC EVALUATION

PRESENTED BY:

NORANDA AUSTRALIA LIMITED

ELECTROLYTIC ZINC COMPANY OF AUSTRALASIA LIMITED

JANUARY, 1975

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SUMMARY

Diamond drilling of the Mutooroo copper prospect near Cockburn in South Australia has outlined probable ore reserves of 9.2 million tonnes at 1.7% copper. The ore is massive sulphide and there are no associated metal values, (e.g. silver, gold, etc.).

The following study considers a proposal for mining the deposit using conventional mining and ore treatment techniques over a period of ten years to depletion. The mine production rate would be 1,000,000 tonnes per annum at a grade of 1.5% copper after allowing 10% mining dilution, and yielding 56,000 dry tonnes per annum of concentrate containing 25% copper.

Current cost estimates show that a capital investment of $45 million would be required to bring the deposit into production. This figure includes $6.6 million for a township to house 470 employees and their dependents.

Annual operating costs are estimated at $1,300 per tonne of copper contained in the concentrate product. This estimate includes distribution costs and smelter charges, but excludes taxation, royalties, depreciation and financing charges. The current market price for copper is $900 per tonne.

The copper price at which the project would break even operating at full capacity, and excluding interest charges and royalties, is $1,600 per tonne. This price has been exceeded on the London Metal Exchange only for a short period from December 1973 to June 1974. A significantly higher copper price would be required if the project is to break even at less than full capacity. At present there appears to be little likelihood that prices will reach the levels necessary for economic exploitation using conventional technology.

SECTION I

SUMMARY OF MUTOOROO MINE GEOLOGY

This section is based on information supplied by Mines Exploration Pty. Limited.

1. WALL ROCK STRUCTURE

Brooke (Honours Thesis, 1966) mapped the surface geology at Mutooroo Mines on a scale of 400 ft. Although he distinguished a number of rock types only the granite gneiss and amphibolites were considered to be reasonable persistent units.

Interpretation of the surface mapping shows the massive amphibolite unit to completely contain the lode and to pinch and swell along its length. The amphibolite appears to transgress a granite gneiss unit to the west. Because of the broadly spaced nature of the drilling and the difficulty in defining clear-cut margins very few bulges in the amphibolite can be correlated from level to level; in consequence no broad plunge can be deduced. Two tenuous lines of evidence may suggest a north plunge of the rock units:

(a) Brooke mapped one isoclinally folded quartzite band which plunged at 65° N.

(b) D.D.M.M. 13 appears to have intersected a wide amphibolite section outcropping to the south on a north plunge.

A plunge to either the north or south is necessary to explain the relatively narrow amphibolite intersection made by D.D.M.M. 10 down dip from the wide surface amphibolite development at M 25 S 5 E.

In cross section, the rock units dip to the west at $40^\circ - 50^\circ$. The D.D.M.M. 14 and D.D.M.M. 17B sections show a marked steepening of the units at depth.

2. LODE STRUCTURE

The old workings extend for 7000 ft with a great intensity between M 40 - 60 S. The surface indications of the lode are based mainly on material from dumps so that any surface strike direction deduced may not be entirely accurate. It is apparent that the lode is closely related to the massive amphibolite unit and largely contained within it while exhibiting a variable orientation to the margins.

In the northern section the lode is centrally within the body of the massive amphibolite unit while moving south at M 25 S it occupies a position in or near the hanging wall of the amphibolite. At M 25 S the lode again occurs within the body of the amphibolite (as evidenced by the RL - 450' holes D.D.'s M.M. 2, 21 - A and 20) and at M 50 S it is close to the footwall (D.D.'s M.M. 19, 18 and 1 intersections). At the southernmost workings a change in strike brings it into a hanging wall position.

The lode disposition on the RL - 450' geological interpretation plan largely reflects the surface relationships. The southern limit shows a "horse tailing" effect which persists at RL - 900'.

The RL - 900' geological interpretation plan shows the lode to be at its widest development, coinciding with the appearance of sillimanite and undifferentiated gneisses on its footwall. At the north the D.D.M.M. 3 intersection largely occurs with these gneisses. The D.D.M.M. 23 intersection consisted of pyrite with gneissic gangue. The D.D.'s M.M. 9 and 22 suggest the gneisses occur within the massive amphibolite unit.

At RL - 1300' the tendency for the development of two sub-parallel lode systems first apparent at RL - 900' has become more pronounced. The western system is contained wholly within amphibolite while the eastern occurs in undifferentiated gneisses and quartzites. In the latter case the lack of continuation of the important intersection in D.D. M.M. 7 through to D.D. M.M. 27 is noteworthy.

Information at RL - 1700' is scanty, however a single lode appears to transgress an amphibolite - gneiss contact between D.D.'s M.M. 15 and 8. The lode is wider within the amphibolite but higher grade within the gneiss. D.D. M.M. 16 defines its northern extent.

In summary the lode shows a close spatial relationship to the massive amphibolite unit. The majority of intersections have amphibolite as the footwall and hanging wall, however D.D.'s M.M. 7 and 8 (and to the lesser extent D.D. M.M. 3) in which the intersections were bounded by gneisses showed no diminution in width or grade.

Records of the primary lode in the shallow workings show extreme variations in width with many irregular bulges, e.g. 4' to 14' to 6' over a distance of 20'. The pattern of the drill hole intersections, although widely spaced, are in accord with these observations.

The lode dips at 40^{o} - 50^{o} with some local variations, e.g. the dip between D.D.'s M.M. 2 and 3 is 70^{o}. An unusual flattening is recorded at the Ironblow Shaft to 25^{o} - 30^{o}.

Pitch data on the lode is lacking. Discontinuous and probably oxidised ore shoots in the northern workings are reported to pitch 65^{o}N. The shoot-like nature of the mineralisation in the primary zone is suggested by D.D. M.M. 27 (1' of lode) measured against adjacent lode intersections. Various combinations of intersections can be connected on north pitch lines, e.g. the rich D.D.'s M.M. 6 and 8 intersections, however it is equally true that similar manipulations can be used to support the idea of a south pitch.

A line on the longitudinal projection defining the northern extent of lode development between D.D.'s M.M. 8 and 16, through D.D. M.M. 23 (fringe zone) and immediately south of D.D. M.M. 10 indicates a direction of vertical elongation of the total lode zone ("pitch") of 30^{o}S. Absence of a lode intersection in D.D. M.M. 17B defines the bottom of this pitching zone.

3. CONTROLS OF MINERALISATION

The discordant relationship of the lode to rock contacts proves an epigenetic origin. The abundance of open spaces and generally sharp contacts suggest the filling of a large open space or fracture. Brooke from studies of the nature of the minor gangue considered some replacement has occurred. In addition he showed that the ore has been deformed.

Three possible modes of origin are apparent:

(a) the lode was emplaced in a continuous planar fracture which subsequently attained its present shape through the deformation processes which affected the ore. In this case the quantity and grades of ore would be largely independent of structural features and any control would be essentially of depositional origin;

(b) the lode was emplaced after the present shape had been attained with the tendency for mineralisation to favour more open parts of the fracture. Pitches on structures would then indicate directions of continuity, and the deformation of the ore now apparent produced subsequent to the emplacement;

(c) if replacement was an important contributing mechanism then even with a vein-like form the mineralised shoots would be likely to be elongated in the direction of regional plunge.

The failure of two holes (D.D.'s M.M. 14 and 17B) to make lode intersections on steepening dips may point to some broad control such as the closure (or initial failure to develop) of the fracture system. However, it should be noted that other holes, e.g. D.D. M.M. 16 failed to make intersections in flat dipping sequences.

4. ORE POTENTIAL, MUTOOROO MINES

There are frequently two separate lode occurrences intersected in the drilling, of which the lower, or footwall zone appears to be the more significant.

All intersections of possible significance have been plotted on the ore potential long section (Fig. 1). No dilution has been incorporated in the narrow intersections. Where intersections are so narrow or so low grade that they would not be of economic interest, they have been omitted. Likewise, where lodes occur close together, it was sometimes more sensible to take a combination. In some holes, one zone was preferable to the other and it was obvious that, because of the low grade and small pillar distances involved, the second zone would then be unmineable. Sometimes it was possible to regard the two zones as separately mineable and they were calculated as such.

When plotted on the long section, the intersections can be grouped into three main categories: the main orebody; the western orebody; and a narrow southern zone which, when diluted to mineable widths, would be fairly low grade.

The results of the ore reserve calculations completed on August 20, 1973, by the Electrolytic Zinc Company of Australasia Ltd. is summarised below:-

Table 1: Probable Ore Reserves

Grade of cut-off (Cu%)	1%	1%
Minimum Stoping Width	10 ft	16 ft
Tonnage (Million tons)	8.7	9.1
Grade (Cu%)	1.8	1.7

7 levels. [illegible] plans

The calculation was undertaken by a blockwise assignment of grade and thickness to an unrolled plan of the orebody. The sub planar and gently flexured form of the orebody determined by footwall contouring was considered particularly amenable to unrolling.

The ore calculation was undertaken according to the definitions and requirements recommended by the Australasian Institute of Mining and Metallurgy (A.I.M.M.), and the Australian Mining Industry Council (A.M.I.C.) Joint Committee on ore reserves report of April, 1972, and incorporated in the official listing requirements of the Australian Associated Stock Exchanges (A.A.S.E.) 1st March, 1973.

The A.I.M.M./A.M.I.C. committee defines "ore" as "a solid naturally occurring aggregate from which one or more valuable constituents may be recovered and which is of sufficient economic interest to require estimation of tonnage and grade". Within this definition the Mutooroo mineralised body may be described as "ore".

The term "ore" is further qualified as "probable" and defined as "... ore that has been cut by drill holes too widely spaced to assure continuity".

The ore reserve calculation does not provide for dilution of ore by waste rock during mining extraction.

SECTION II

MINING

SUMMARY

Ore Reserves:	With allowance for dilution by waste rock the mine is expected to produce 10,000,000 tonnes of ore at an average grade of 1.5% Cu.
Mine Production:	3,500 tonnes per day 285 days per year at a head grade of 1.5% Cu. (1m tpa)
Mining Method:	Mechanised horizontal cut-and-fill with inclined access from the surface. Ore hoisting by vertical shaft.
Capital Estimate:	$10.1 million, January 1975.
Operating Cost Estimate:	$7.4 million p.a. = $7.40 per tonne of ore.

1. INTRODUCTION

Geological assessment of drilling completed to date indicates that the lode is closely related to a massive amphibolite unit and is largely contained within it. The lode dips at between 40 degrees and 50 degrees with some local variations with dips as steep as 70 degrees.

No detailed studies of core to estimate the strength of the amphibolite in the hanging wall have been conducted. Provided the amphibolite zone in the hanging wall is normally in excess of 1.5m thick and that overbreak during extraction of the orebody is well controlled by geological inspection, the well proven extraction technique of "Mechanised Horizontal Cut and Fill Stoping" should be satisfactory. If adequate ground control is to be obtained, a reasonably "close coupled" production support system is required, not however to the degree that low productivity square set stoping is the only solution.

On the basis that a minimum stoping width of 4.9m is chosen there is a wide range of diesel powered rubber tyred equipment available to achieve relatively high efficiency and high production rates from individual stopes or stope parties.

For the purposes of broad conceptual planning the orebody is divided into several production zones or blocks of broadly similar bulk grade and tonnage so as to allow the proper selection of development, blast-hole drilling and LHD mucking equipment to provide steady continuity of production during the life of the mine.

The degree of continuity accepted in the ore reserves estimates completed by Mines Exploration Pty. Ltd. is adopted for the purposes of conceptual mine layout. An exploration shaft and exploration drives will allow completion of a programme of diamond drill holes to ascertain orebody limits for purposes of detailed mine design.

Due to the relatively shallow depth at which initial production operations can commence and the choice of diesel-powered mobile equipment, the following broad conceptual plan was adopted:-

(a) equipment and personnel access to the mine is via a 4.6m x 3.7m decline at a gradient of 20-25%;

(b) ore is extracted using the mechanized horizontal cut-and-fill method. Close-patterned rock bolting of hanging walls using tensioned grout anchored rock bolts is required;

(c) ore from production blocks is hauled horizontally to a main ore pass system by 15 tonne capacity diesel dump trucks;

(d) ore gravitates to a primary crusher station for reduction to minus 125 mm size before hoisting to the surface storage bins;

(e) ore is hoisted via a 4.3m round concrete lined shaft fitted with two skips in balance. The shaft is equipped with a small man carrying conveyance for service personnel and for transportation of key maintenance personnel such as pump station attendants, skipmen, shift electricians. Main mine levels are developed at 150m (500'), 275m (900'), 400m (1300'), 525m (1700') and 650m (2100').

2. PRODUCTION SCHEDULE

The ore reserve of 9,100,000 long tons grading 1.7% copper based on a minimum mining width of 16' was adopted for development and production planning purposes. The above reserve did not provide for dilution by waste rock which for the mining method chosen is expected to be about 10% by volume. For the purposes of economic evaluation it is assumed that the mine will produce 10,000,000 tonnes of ore grading 1.5% Cu over a mine life of 10 years. Mine production will proceed 285 days per year at a production rate of 3,500 tonnes per day.

3. MINING OPERATIONS

3.1 DEVELOPMENT

Weekly production rates expected from the individual longitudinal cut and fill stopes of production blocks is about 5,000 tonnes per week. To achieve the designed production rate, four stopes are required to be in production. Each stope will operate as follows:

(a) each stope is divided into two sections of minimum length 120m;

(b) each section proceeds from an initial sill development some 10m above the main level;

(c) stope ore passes are developed in the footwall of the stoping block terminating in a chute designed to load 15 tonne capacity diesel haulage units (4 wheel drive to permit use in driving access decline from the surface);

(d) initial access to the first production main levels at 150m and 275m is from the surface via the decline, the decline being extended progressively to the lower production main levels;

(e) the ore haulage shaft is developed at co-ords 2800' (854m)S, 0300' (91.5m)W by raise boring a pilot raise then stripping and concrete lining. The final exhaust ventilation shaft is developed in a similar way. The exploration shaft is used as a temporary exhaust ventilation shaft;

(f) an ore crushing station at R.L. 360m houses a 1200mm x 915mm jaw crusher fed by a chain feeder. Main ore and mullock passes are developed from this crusher station to the 150m and 275m production levels. The crusher station is relocated to the 650m level to permit ore production from the 400m, 525m, and 650m levels after year 5;

(g) inflow of ground water into the mine is expected to be minimal based on the experience of mines at Radium Hill, Broken Hill and Cobar. Pumping facilities are provided in the concept to cope with drainage water predominantly issuing from stope back-fill.

3.2 STOPING

After completion of sill and chute raise development, ventilation raise and access raise development (where access to the main decline is not possible) the stope block is ready for production to commence.

3.2:1 DRILLING

It is envisaged that horizontal blast holes are required in some instances where back conditions are difficult. Normally blast holes are inclined forward at 70° to the horizontal and drilled parallel to the plane of the nearest wall to avoid over-break and resultant ore dilution. Blast hole lengths vary from 4.2m to 5.2m corresponding to variations in dip from 70° to 50°.

3.2:2 ORE BREAKING

The prepared ore is blasted for about half of the length of the stope using AN-FO as the primary blasting agent. Individual firing tonnages will be determined by ground conditions.

3.2:3 ORE LOADING

Broken ore is loaded by diesel LHD units with a bucket capacity of about $4m^3$ and transported to the footwall chute raises. Concurrently with loading the stope walls and back are scaled and rockbolted as required using mobile hydraulic scaling towers and drill platforms. For costing purposes the stope wall rockbolting pattern adopted is about 1.2m x 1.2m x 3m rockbolts.

3.2:4 FILLING

After loading of the broken ore the stope is prepared for filling by laying drainage pipes, building drainage towers and bulk-heading access cross cuts and ore pass raises. Footwall chute raises are extended for the next stope "lift". Equipment is moved to the remaining productive section of the stope to commence ore production. A temporary timber barricade is erected across the stope and hydraulic fill introduced "upstream" to provide a fill barricade between the "filling" and "production" stope sections. Hydraulic fill is then continuously placed until the horizontal surface of the fill is at the correct distance from the stope back for production drilling to commence. This sequence then continues in each stope block.

3.3 ORE HANDLING

3.3:1 Ore is transported by diesel truck from the stope chute raises to the main ore pass system terminating at each production level. A small grader is required for maintenance of level haulage cross cuts and drives which are gravel surfaced.

3.3:2 Ore gravitates through the main ore pass system to the crusher station, where the crusher reduces it to minus 125mm.

3.3:3 Crushed ore gravitates through an ore pass, enlarged to hold approximately 200 tonnes, to the shaft loading station. Here ore is loaded semi-automatically via a standard measuring flask arrangement into 8 tonne capacity Saunders-type bottom dump skips.

3.4 ORE HOISTING

Hoisting equipment is sized to permit a maximum hoisting rate of 300 tonnes per hour. The conventional double drum in-balance hoist winder operates at maximum skip speeds of 300m/s to avoid excessive power demands on the relatively small diesel power station. A multi-rope friction hoist would reduce further the cyclical power demands however this system does not readily allow for shaft extension.

3.5 MINE VENTILATION

Mine ventilation is conventional with exhaust of mine air via an exhaust shaft fitted with a 3m diameter vertical mounted exhaust fan. Dust collection from ore passes, crusher station and loading station is provided.

As the mine is essentially based on trackless diesel-powered equipment, at least 100 c.f.m. of air is required at each working place for each h.p. of installed diesel power. A total primary ventilation flow of 400,000 c.f.m. is required.

3.6 EQUIPMENT MAINTENANCE

All mobile equipment is driven to the surface workshops for major maintenance. Routine maintenance and servicing is conducted by mobile service crews on the job or at rudimentary service bays on each main level. The central workshop incorporates a rock drill maintenance and drill rod/bit sharpening facility.

4. ESTIMATE OF CAPITAL INVESTMENT

4.1 PRELIMINARY DEVELOPMENT

Confirmation of ore reserves and determination of the limits of the orebody for mine layout and design:

1.	Diamond drilling	$280,000
2.	Exploration shaft	220,000
3.	Exploration drives off shaft to drill sites and to obtain bulk samples for detail metallurgical testing	88,000
	Sub Total	$588,000

4.2 MINE DEVELOPMENT

1.	Surface civil works (preparation shaft collar and decline portal)	60,000
2.	Decline surface to 150m level	150,000
3.	Cross-cut connection to exploration (temporary exhaust) shaft	60,000
4.	Installation temporary exhaust fan at surface exploration shaft	10,000
5.	Decline 150m level to 275m level	175,000
6.	Bore hoisting shaft pilot raise 150m level to surface	30,000
7.	Bore main ventilation shaft pilot raise 150m level to surface	30,000
8.	Bore hoisting shaft pilot raise 275m level to 150m level	24,000

9.	Develop chute raise to pilot raise at 275m level	4,000
10.	Connection to exploration shaft 275m level	7,000
11.	Strip and concrete line hoisting shaft surface to 175m level	180,000
12.	Main level development 150m level incl. return airway	350,000
13.	Decline 275m level to 400m level incl. access to exhaust shaft	175,000
14.	Develop pump station including access, pump foundations, lining and settling chambers, installation of pipework electrical control equipment, pumps and pipework	24,000
15.	Bore exhaust shaft pilot raise 275m level to 400m level	24,000
16.	Bore hoist shaft pilot raise 275m level to 400m level	24,000
17.	Strip and concrete line hoisting shaft 275m level to 400m level	80,000
18.	Construct loading stage and install measuring flasks	60,000
19.	Construct hoisting head frame and install shaft steelwork including ladderways, pump and compressed air mains and electrical cabling	266,000
20.	Strip exhaust shaft to 3.7m diameter surface to 400m level and concrete line to about the 60m R.L.	184,000

Hoisting Shaft to 400m $ 666,000
63
730,000

21.	Develop 7.7m x 15.4m x 18m primary crushing station at 360m R.L. (2900m^3) and bore ore pass raises to 275m and 150m levels including a crusher station by-pass	63,000
22.	1200m x 915mm jaw crusher and motor $133,000 Foundations 10,000 Feeder 15,000 Installation 31,000	189,000
23.	Develop 275m level	315,000
24.	Develop 150m level stope sill horizon	140,000
25.	Develop 275m level stope sill horizon	140,000
26.	Bore ventilation raises, 8 of @ 120m	180,000
	Sub Total	$2,944,000

4.3 EQUIPMENT COSTS

4.3:1 DEVELOPMENT

4	Crawler mounted rocker shovels, 4 @ $60,000	240,000
8	4 wheel drive (Wagner type) 15 tonne diesel trucks, 8 @ $70,000	560,000
4	2 boom rubber tyred drill diesel jumbos, 4 @ $90,000	360,000
	Hand held drilling and ancilliary equipment	20,000
	Miscellaneous equipment (rock bolting wrenches, ladders, hand tools, etc.)	50,000

4.3:2 PRODUCTION

6	Production drilling jumbos 6 @ $90,000	$540,000
6	Diesel LHD units, 6 @ $100,000	600,000
	Diesel ore trucks, one only (mine development equipment will assume ore transportation role)	70,000
4	Diesel powered hydraulic scaling platform, 4 @ $20,000	80,000
	Miscellaneous production equipment	50,000

4.3:3 GENERAL SERVICE

Diesel powered general service vehicles, 6 @ $30,000	180,000
Sub Total	$2,750,000

4.4 ENGINEERING & CONSTRUCTION

20% of 4.1 and 4.2 items	706,500
5% of 4.3 items	137,000
	$ 844,000

4.5 CONTINGENCIES

20% of 4.1+4.2+4.3 items	$1,256,000

4.6 TOTAL COST TO DEVELOP & BRING MINE TO PRODUCTION

TOTAL COST TO DEVELOP & BRING MINE TO PRODUCTION	$8,382,000

Hoth

- 18/10

4.7	CAPITAL SPARES - allow	$140,000
	MINE DEVELOPMENT COSTS, YEAR 5	
	Extension of ventilation, hoisting shafts and decling involving relocation pump station, crusher station and loading stage	1,180,000
	Production equipment replacement	400,000

4.8 TOTAL, CAPITALISED DEVELOPMENT & FIXED CAPITAL

Year 0		8,522,000
Year 5		1,580,000
say	Year 0	8.5 million
	Year 5	1.6 million
	TOTAL	$10.1 million

5. ESTIMATE OF OPERATING COSTS

5.1 MANNING

5.1:1 SUPERVISION

	Annual Cost $ (Incl. on-costs)		manpower
Mine Manager	22,800		1
Secretary/Stenographer	6,600		1
Underground Superintendent	21,600		1
Mine Planning Engineer	20,000		1
Mining Engineer	16,200		1
Geologists (Senior)	16,200		1
Geologists	13,200		1
Draftsmen (2)	21,600		2
Surveyor (Senior)	13,200		1
Surveyors (2)	24,000		2
Mine Foremen (2)	26,400		2
Shift Supervisors (12)	151,200		12
Maintenance Engineer	16,000		1
Maintenance day foremen (3)	39,600		3
Maintenance Shift Foremen (4)	50,400		4
			34
Total Supervision	$459,000		

5.1:2 MINE LABOUR

Stope Development miners (14) @ $15,000	210,000		14
Main Development miners (12) ~~@ $14,000~~	-	*	(12)
Exploration Core drillers (6) @ $14,000	84,000		6
Production drillers (24) @ $13,000	312,000		24
L.H.D. operators (12) @ $13,000	156,000		12
Rockbolters/Scalers (24) @ $13,000	312,000		24
Fill Operators (6) @ $10,000	60,000		6
Blast Hole chargers (6) @ $13,000	78,000		6
Nippers 12? @ $7000	84,000		12?

Timbermen (24) @$13,000	312,000	24
Diesel Truck operators (18) @$12,000	216,000	18
General Labour (6) @$7,000	42,000	6
Winder driver (4) @$10,000	40,000	4
Skipman (2) @$10,000	20,000	2
Shaft maintenance (2)	20,000	2
Crusher attendant (2)	20,000	2
Surface Nippers (6) @$5,750	34,500	6
Safety (6) @$8,000	48,000	6
Hygiene (2) $8,000	16,000	2
Pumpstation (3) @$8,000	24,000	3
Change Room (6) @$7,000	42,000	6
Surveyors Chainmen (4) @$8000	32,000	4
Clerks (2) @$7,500	15,000	2
Relief operators (annual leave/sick leave, etc.) (15)	195,000	15
		108
Total Operating Labour	$2,372,000	

* Cost of this labour is incorporated into the Capital Estimate.

5.1:3 MAINTENANCE LABOUR

Tradesmen (42)	420,000	42
Tradesmen's Assistants (31)	263,500	31
Relief (6)	57,000	6
		79
Total Maintenance Labour	$740,500	

Supervision 34
Operating 108
Maint 79
221

5.2 OPERATING COSTS

5.2:1 MANNING	$ p.a.	$ p.a.
Supervision	459,000	
Labour	3,113,000	3,572,000
5.2:2 STORES		
Concrete/cement etc	400,000	
Air/water pipes	200,000	
Rockbolts/mesh	190,000	
Explosives	430,000	
Timber	475,000	
Misc. consumables	400,000	
Maintenance consumables	300,000	2,395,000
5.2:3 WATER		
0.5×10^6 l.p.d.	55,000	55,000
5.2:4 POWER		compare to Power pg 37.
4 MW	360,000	360,000
		$6,382,000 3.5 MW @ $548,000
5.2:5 CONTINGENCY - 15%		957,000
	Total	$7,339,000
	say	$7,400,000

SECTION III

ORE TREATMENT

SUMMARY

Mill Feed:	3,000 t.p.d. ore at 1.5% copper.
Product:	56,000 dry t.p.a. copper concentrate @ 25% copper.
Process:	Conventional crushing and grinding followed by flotation of chalcopyrite.
Capital Estimate:	$13.0 million, January, 1975.
Operating Cost Estimate:	$3.0 million p.a. = $3.00 per tonne ore.

Manpower

Supervision	12
Operating	24
Maint	10
	46

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1. INTRODUCTION

Bench-scale flotation testwork carried out on drill core by Broken Hill South Limited during 1967 is used as the basis for the following estimates.

The major mineralization consists of pyrrhotite, pyrite, chalcopyrite, amphibolite and quartz. Adequate liberation is assumed to require a grind of 80% passing 74 μ. The chalcopyrite is selectively floated from the siliceous material using ethyl xanthate, while the pyrrhotite and pyrite minerals are depressed using a combination of high alkalinity and sodium sulphite. The flotation selectively between the iron sulphides and the chalcopyrite will be critical and slow, and will be compounded by a high ratio of the former to the latter mineral in the circuit. It is anticipated that successful separation will require a steady controlled flotation feed, as well as large capacity cleaner and recleaner flotation banks.

The flowsheet employs conventional crushing plus rod mill/ball mill grinding, rather than autogenous grinding, mainly for the purpose of achieving a steady feed condition. Future testwork may establish a case for an autogenous approach, and could result in some level of cost savings.

Aside from the above aspects the flowsheet is conventional, and should be effective. Details are given in Fig. 3.

2. EQUIPMENT

2.1 CRUSHING

The crusher station is designed to operate in phase with the mine hoisting programme of two shifts per day for six days per week.

Minus 125 mm ore is discharged from the skips into a 500 t. capacity surge bin. Ore is withdrawn from the bin via a variable speed apron conveyor and is fed over a 50 mm rod deck screen into a 1.5 m standard cone crusher set at 50 mm.

The cone crusher discharge joins the undersize material from the rod deck screen at a 13 mm vibrating screen. Oversize material from this screen discharges into a 1.5m shorthead cone crusher set at 13 mm, and operating in closed circuit with the vibrating screen. Undersize material is transferred by conveyor to one of three 1,500 t. capacity fine ore storage bins.

2.2 GRINDING

Ore is withdrawn from the fine ore bins by variable speed hydraulic discharge feeders and fed via a collecting conveyor and weightometer to a 3 m x 4.3 m wet rod mill operating in open circuit. The rod mill discharge is expected to be 80% passing 1.17 mm, and is pumped to two 3.9 m x 4.3 m ball mills operating in parallel. Each ball mill operates in closed circuit with hydrocyclones to yield a final ground product of 80% passing 74 μ.

2.3 FLOTATION

Flotation cell volumes of 43 m^3 have been allowed for each of the rougher, scavenger, cleaner and recleaner flotation stages. This will allow an average flotation time of 10 minutes for rougher flotation and is based upon the Broken Hill South testwork. The others are assumed volumes, and would require laboratory confirmation. The rougher tailings are fed to the scavenger cells, while the scavenger concentrate and cleaner tailings are both recycled to the head of the rougher section. Recleaner tailings are recycled to the head of the cleaner section.

The final concentrate is the recleaner float product which is fed to a 15 m thickener followed by a 20 m^2 disc filter unit. The concentrate is loaded onto road trucks at around 8% moisture for transport to the rail siding.

2.4 TAILINGS DISPOSAL

Scavenger tailings are deslimed in hydrocyclones. The sands fraction is stored in one of two 3,000 t. unagitated tanks, similar to those at C.S.A., Cobar before pumping underground for fill. The slimes fraction is thickened in a 46m thickener before pumping to the slimes dam. A by-pass pump is provided for by-passing scavenger tailings direct to the slimes dam.

3. ESTIMATE OF FIXED CAPITAL INVESTMENT

A preliminary capital cost estimate is made using a factored estimating procedure developed by Parkinson and Mular. The procedure is based on estimated purchased costs of major equipment items. Details of the procedure are given in "Mineral Processing Equipment Costs and Preliminary Capital Cost Estimations", Special Volume 13, 1972, published by The Canadian Institute of Mining and Metallurgy. All costs have been taken in January 1975 terms.

3.1 PURCHASED COSTS OF MAJOR ITEMS

		Purchased Cost	
1.	Coarse ore bin, 500 t.	$ 96,000	$152,000
2.	Apron feeder	40,000	
3.	Rod deck screen, 50 mm	16,000	
4.	Standard cone crusher 1.5 m	117,000	$254,000
5.	Vibrating screen, 13 mm	17,000	
6.	Shorthead cone crusher, 1.5m	120,000	
7.	Fine ore conveyor	37,000	$2070,000
8.	Fine ore bins, 3 @ 1,500 t. ea.	564,000	
9.	Discharge feeders	102,000	
10.	Rod mill, 3 m x 4.3 m	290,000	
11.	Ball mills, 3.9m x 4.3m, 2 of.	1,040,000	
12.	Cyclone classifiers	37,000	
13.	Flotation equipment	376,000	$467,000
14.	Concentrate thickener, 15m	51,000	
15.	Disc filter, 20 m^2	40,000	

16.	Flotation tailings pumps	32,000
17.	Tailings cyclones	37,000
18.	Cyclone underflow pumps	20,000
19.	Sands storage tanks, 2 @ 3,000 t.	180,000
20.	Sands pumps	20,000
21.	Slimes thickener, 46m	236,000
22.	Slimes pumps	30,000
23.	Tailings by-pass pump	10,000
	Total	$3,508,000

$555,000
+20%

3.2 FIXED CAPITAL INVESTMENT BY FACTORS

1.	Total purchased equip. costs (from 3.1)	$3,508,000
2.	Installed equipment costs (143% of item I)	5,016,000
3.	Process Piping (10% of item 2)	502,000
4.	Instrumentation (10% of item 2)	502,000
5.	Buildings & site development: mill buildings, crusher building, laboratory, workshop, office (35% of item 2)	1,756,000
6.	Auxiliaries: water supply, diesel standby power (5% of item 2)	251,000
7.	Outside lines (8% of item 2)	402,000

$6,020,000
+20%

2,409,000
+20%

$8,429,000

8.	Total Physical Plant Costs (sum items 2 to 7)	$8,429,000
9.	Engineering & construction (25% of item 8)	2,107,000
10.	Contingencies (20% of item 8)	1,686,000
11.	Plant cost (sum items 8, 9 & 10)	12,222,000
12.	Capital spares, allow	200,000
13.	Residue dam, allow:	
	Year 0	300,000
	Year 5	300,000
14.	Total, Fixed Capital	
	Year 0	12,722,000
	Year 5	300,000
	say Year 0	12.7 million
	Year 5	0.3 million
	Total	$13.0 million

4. ESTIMATE OF OPERATING COSTS

4.1 MANNING

4.1:1 SUPERVISION

	Annual Cost $ (incl. on-costs)	Man Power
Superintendent	21,600	1
Metallurgist	16,200	1
Chemists (2)	26,400	2
Shift Foremen (4)	50,400	4
Clerk	7,500	1
Maintenance Engineer	16,000	1
Maintenance Foremen (2)	26,400	2
Total Supervision	$ 164,500	12

4.1:2 MILL LABOUR

Crushing (2)	13,800	2
Grinding (4)	27,600	4
Flotation (8)	55,200	8
Thickening & Filtration (4)	27,600	4
Relief Operators (3)	20,700	3
Labourers (2)	11,500	2
Sampler (1)	5,800	1
Total Operating Labour	$ 162,200	24

4.1:3 MAINTENANCE LABOUR

Tradesmen (5)	50,000	5
Assistants (4)	34,000	4
Relief (1)	9,500	1
Total Maintenance Labour	$ 93,500	10

Supr 12
Operat 24
Maint 10
46

4.2 OPERATING COSTS

	$ p.a.	$ p.a.
4.2:1 MANNING		
Supervision	164,500	
Operating Labour	162,200	326,700
4.2:2 CHEMICALS		
Lime	453,500	
Sodium Sulphite	204,200	
Ethyl Xanthate	36,000	
Cresylic Acid	7,500	
Miscellaneous	28,800	730,000
4.2:3 WATER		
2.2×10^6 l.p.d.		245,000
4.2:4 POWER		
3.5 MW		548,000
4.2:5 RODS & BALLS		250,000

	$ p.a.	$ p.a.
4.2:6 MAINTENANCE		
Labour	93,500	
Supplies	300,000	393,500
4.2:7 MISCELLANEOUS		50,000
Total		$2,543,200
4.2:8 CONTINGENCY - 15%		381,500
Total		$2,924,700
say		$3,000,000

SECTION IV

SERVICES & INFRASTRUCTURE

SUMMARY

Capital Estimates:

Power Supply	$ 3.6 million, January 1975
Water Supply	$ 3.4 million, January 1975
Engineering & Compressed Air	$ 2.2 million, January 1975
General Service Facilities	$ 1.2 million, January 1975
Township	$ 6.6 million, January 1975
Total	$17.0 million, January 1975.

Operating Cost (unallocated)	$2.0 million p.a. = $2.00 per tonne ore

78½ people

INTRODUCTION

The minesite is approximately 20 km to the south of the transcontinental railway line which would provide access to the port facilities at Port Pirie. A rail siding would be established and concentrates transported from the mine by trucks over a sealed access road.

The mine itself is isolated and would require a high level of self-sufficiency. A high level of engineering services would be required. Power would be generated on site.

A township would be established in the vicinity of the rail siding and adjacent to the Barrier Highway. A high level of community and recreational facilities would need to be provided. Bus services would be supplied between the township and the mine.

The nearest reliable water supply is Stevens Creek Reservoir, 85 km to the north-east in the Broken Hill area. Water would have to be piped to a dam at the mine to supply both the mine and township requirements.

1. MINE POWER GENERATION AND DISTRIBUTION

Estimated power requirements are as follows:

Mine	4.0 MW
Mill	3.5 MW
Other	0.5 MW
Total	8.0 MW

? Morgan Consultants to R Reed

This requirement could be supplied by five 2.5 MW units. Four would provide continuous output at a normal loading of 80%, with the fifth on standby or maintenance. The engines would be slow speed diesels with direct coupled alternators generating at 11KV. Cooling would be through forced draft cooling towers and closed circuit heat exchangers.

The power station building would be fully clad. It would also house the air compressor installation, and a minor maintenance workshop.

The main distribution voltage would be 11KV feeding substations at suitable locations. The mill substation would transform to 3.3KV for major drives, as well as 415 volts for minor drives and general purpose power.

2. COMPRESSED AIR

The compressed air requirement is 200 m^3/min. of free air, supplied at 8.5 Kg/cm^2. This requirement could be supplied by five units, each of 60 m^3/min. capacity, one being available for standby and maintenance requirements. Each unit would be driven by an electric motor, and would be arranged for automatic operation based on pressure controls. A reticulation grid would distribute the air to required points underground.

3. WATER SUPPLY

The mine and township water requirement would be of the order of 3.5 million litres per day. This would be supplied from the Stevens Creek Reservoir, 85 km to the north-east, through a steel pipeline to a dam at the minesite. Township requirements would be reticulated from the mine dam.

4. ENGINEERING FACILITIES

4.1 WORKSHOPS & OFFICES

A central workshop complex would be required to handle on-site maintenance. The complex would be required to provide the company with a high level of practical independence for all maintenance jobs, except for a few major items which would require shipment away from the site. The main building would be approximately 140 sq. in total floor area, and serviced by an overhead travelling crane, plus a number of monorail and swing hoists. Concrete hard standing areas would be constructed outside the building together with vehicle and plant parking areas.

Offices for the accommodation of engineering staff and foremen would form an annex to the main workshop building. Total floor area would be approximately 24 sq.

Minor special purpose workshops would be established underground, in the mill, at the power station, and in the township. A mobile workshop would also be required.

4.2 MOBILE PLANT

A mobile plant and equipment fleet would be provided to supplement the fixed engineering maintenance facilities and would permit the operation to be practically self-sustaining.

Equipment would include 5t. and 30t. capacity mobile cranes, a low loader, special purpose and general purpose trucks, special purpose trailers, a mobile compressor and a backhoe/ front-end loader. A number of personnel transport vehicles would also be allocated to the engineering section.

4.3 FUELLING & LUBRICATING FACILITIES

Dieselene would be reticulated to all major consumption points at the surface and underground. A single petrol fuelling point would be established at the fuel storage depot for vehicle servicing. Lubrication facilities would be established at the central and underground workshops, and in addition, a lubrication vehicle would be provided.

4.4 SEWERAGE

Toilets would be constructed throughout the surface facilities, and would be serviced by a single mine sewerage treatment plant of adequate capacity to handle both the surface and underground effluent. Underground facilities would be serviced by a pan system. Pans would be collected at regular intervals and emptied into the surface sewerage system.

4.5 CIVIL WORKS

All roads and car parks within the immediate plant area would be sealed. Roads to the tailings dam, sewerage plant, explosives store, and water storage dams, would be unsealed.

The site itself should be relatively flat, and the requirement for site preparation and grading for drainage should be small.

The town access road will have a sealed width of 6.5 metres over its full distance of 21 Km.

4.6 COMMUNICATIONS

Telephones would be installed throughout the mine complex, and operated through a private automatic exchange. A talk-back loud speaker system would be installed in the ore treatment plant as a supplement to the telephone system. Mobile radios would be installed in vehicles. P.M.G. trunk facilities would be available.

5. GENERAL SERVICE FACILITIES

5.1 GENERAL OFFICE

The general office would have a total floor area of approximately 80 sq., would be of ground level construction and fully air-conditioned.

It would accommodate the operations manager, departmental heads, mine services personnel, accounting staff and general services staff. It would include conference and interview rooms, stationary stores, library and central records. Accounting and office equipment would be included.

5.2 SAFETY, SECURITY AND FIRE

A safety officer would be appointed for the purpose of developing and maintaining a safe working environment. The estimates provide for the supply of protective clothing and equipment, rescue equipment, and signs. A fully equipped first-aid room would be provided and manned continuously. An ambulance would be provided for the removal of injured persons to the township hospital, or to Broken Hill in more serious cases.

Security fences would be constructed around the site, and a gatehouse would be constructed at the main entrance and manned continuously. Additional security fencing would be constructed around isolated installations.

Fire hydrants, hose units and portable extinguishers would be provided at appropriate locations.

5.3 TRANSPORT

The concentrate would be transported from the mine to a rail siding and loaded into rail trucks by a contractor. The company would provide the rail siding.

A common pool of transport and handling vehicles would be provided for general use. The fleet would include five 45 seat buses, two 12 seat buses, two forklifts, one front-end loader, plus a number of trucks and light personnel vehicles.

5.4 CHANGEHOUSE

A changehouse would be provided. This building would be located close to the decline entrance, and would be integral with the safety store and first-aid room.

5.5 STORES

The degree of isolation of the operation dictates the holding of relatively large stocks of operating and maintenance supplies on site. In general, stocks would need to be sufficient to support the operation for a period of three months. A warehouse of around 150 sq. would be required and designed to permit the handling of all items by forklift or mobile crane.

Separate facilities would be provided for storage of dieselene and explosives.

6. TOWNSHIP

The township would be located 21 km to the north of the mine, and on the Barrier Highway 60 km to the west of Broken Hill. The full workforce of 473 employees plus families would be accommodated. Total population would be approximately 1400 persons. 1400 ÷ 473 × 3

Housing would be provided for 190 employees, single quarters accommodation for 140, and caravan park facilities for 143.

The basic house would be a 3 bedroom, evaporative cooled, 12 sq. demountable building. The estimated cost for each of these units is $22,000 installed with land, fences, roadway and services.

A shopping complex and a wide range of community and recreation facilities would be provided. The community facilities would include a kindergarten, primary school, community hall, library, fire station, hospital and medical care centre. Recreation facilities would include a sports club, swimming pool, sports oval and golf course.

7. CAPITAL ESTIMATE - SERVICES & INFRASTRUCTURE

	$	$
7.1 POWER GENERATION & DISTRIBUTION		
Power Generation	2,310,000	?
Distribution	920,000	
Power station building	260,000	
Area lighting	90,000	3,580,000
7.2 COMPRESSED AIR		
Generation	300,000	
Reticulation	30,000	330,000
7.3 WATER		
Main Pipeline	2,650,000	
Township pipeline	500,000	
Pumping and storage	200,000	3,350,000
7.4 ENGINEERING FACILITIES		
Workshop & offices	600,000	
Mobile plant	100,000	
Fuel & lubrication facilities	40,000	
Sewerage	160,000	
Civil works - minesite	200,000	
Town access road	600,000	
Communications	270,000	1,970,000

7.5 GENERAL SERVICE FACILITIES

General office	190,000	
Safety, Security & Fire	200,000	
Transport vehicles	250,000	
Rail Siding	30,000	
Changehouse	100,000	
Warehouse building & equipment	220,000	
Open storage area & equipment	50,000	
Fuel storage	90,000	
Explosives storage	60,000	
		1,190,000
Total Mine Services		$10,420,000

7.6 TOWNSHIP

Manager's residence	30,000	
Senior staff residences (7)	182,000	
General residences (182)	4,004,000	$5,106,000
Single workers' quarters	640,000	
Central messing facility	100,000	
Caravan park	150,000	
Shopping centre	300,000	
Kindergarten	30,000	
Primary school	250,000	
Community hall	200,000	$1,466,000
Library	30,000	
Fire station	20,000	
Hospital & medical care	300,000	
Sports/social club & facilities	200,000	
Swimming pool	100,000	
Sports oval	14,000	
Golf course	20,000	
Total Township facilities		$6,570,000
Total Services & Infrastructure		$16,990,000

8. ESTIMATE OF OPERATING COSTS

8.1 MANNING, ENGINEERING

8.1:1 SUPERVISION

	Annual Cost $ (incl.on-costs)	Manpower
Chief Engineer	21,600	1
Secretary	6,600	1
Planning Engineer	16,200	1
Engineer, Surface & Township	16,200	1
Clerks (2)	18,000	2
Day Foreman (½)	6,400	½
	85,000	7½

8.1:2 MAINTENANCE LABOUR

	Annual Cost $ (incl.on-costs)	Manpower
Tradesmen (6)	60,000	6
Assistants (6)	41,200	6
Draftsmen (2)	22,000	2
Drivers (6)	44,900	6
Relief (1)	6,900	1
	175,000	21

8.2 MANNING, ACCOUNTING & STORES

8.2:1 SUPERVISION

Chief Accountant	21,600	1
Secretary	6,600	1
Accountants (2)	29,400	2
Paymaster	10,800	1
Purchasing Officer	10,800	1
Clerks (6)	54,000	6
Stores Officer	10,800	1
Typists (4)	24,000	4
	168,000	17

8.2:2 GENERAL LABOUR

Stores Assistants (4)	35,000	4

8.3 MANNING, ADMINISTRATION & GENERAL SERVICES

8.3:1 SUPERVISION

Manager, Operations	30,000	1
Superintendent, General Services	21,400	1
Secretaries (2)	13,200	2
Personnel Officer	13,200	1
Industrial Officer	13,200	1
Safety & Fire Control Officer	12,000	1
Community Services Officer	12,000	1
Clerks (2)	18,000	2
	133,000	10

8.3:2 GENERAL LABOUR

Watchmen (4)	34,800	4
First Aid Attendants (4)	34,800	4
Drivers (6)	44,900	6
Gardener (1)	6,300	1
Assistants (2)	12,600	2
Reliefs (2)	12,600	2
	146,000	19

8.4 OPERATING COSTS

8.4:1 MANNING

	$ p.a.	$ p.a.	
Supervision:			
Engineering	85,000		7½
Accounting	168,000		17
Administration	133,000		13
			37½
Labour:			
Engineering	175,000		21
Accounting	35,000		4
Administration	146,000	742,000	16 / 78½

8.4:2 SUPPLIES

Engineering	215,000		Does this include water charges to B.H. Water Authority? see over
Accounting & Stores	20,000		
Administration & Services	70,000	305,000	

8.4:3 UTILITIES

Power	78,000	
Water	75,000	153,000
		$1,200,000
??, ? — 8.4:4 OVERHEADS		500,000 [illegible]
Total		$1,700,000
8.4:5 CONTINGENCY, 15%		255,000
Total Cost =		$1,955,000

Overheads & contingency add 63%?

SECTION V

ECONOMIC EVALUATION

1. CAPITAL COSTS

1.1 FIXED CAPITAL

The following estimates of fixed capital investment required to bring the mine into production have been derived in the preceding sections of the report. All estimates are in January, 1975, terms.

			Operating costs
Mining	Year 0	$ 8.5 million	$7.4
	Year 5	$ 1.6 million	
Ore Treatment	Year 0	$12.7 million	$3.0
	Year 5	$ 0.3 million	
Mine Services	Year 0	$10.4 million	$2.0
Township	Year 0	$ 6.6 million	
Total	Year 0	$38.2 million	
Total	Year 5	$ 1.9 million	
		$40.1 million	
? Detailed Feasibility Study		$ 0.9 million	
Total Fixed Capital Expenditure		$41.0 million	

1.2 WORKING CAPITAL

As a first estimate, working capital would need to be sufficient to finance four months' operations, and approximately $4 million would be required.

Four months [illegible] concentrate be [illegible] every 3 x 4 months. [illegible]

1.3 TOTAL CAPITAL

From 1.1 and 1.2:

Fixed Capital	$41.0 million
Working Capital	$ 4.0 million
Total Capital	$45.0 million

2. OPERATING COSTS

Operating cost estimates have been detailed in the preceding sections of the report, and are summarised below in Table 2. Estimated total direct operating cost is $12,200,000 per annum, at full output which is equivalent to $12.20 per tonne of ore mined. This cost is exclusive of distribution costs, smelting and refining charges, taxation, royalties, depreciation and financing charges. It has been estimated in January, 1975, terms.

3. MARKETING

For the purposes of the study, it was assumed that the concentrates would be stockpiled on the wharf at Port Pirie for bulk shipment to Japan in three to four lots per year.

The total distribution cost, C.I.F. Japan, was estimated at $33.40 per dry tonne.

This seems low if it includes rail freight & handling charges at Pt Pirie?

Concentrates would realize on the copper content only, and payment was taken on the basis of current average terms of 95% of the copper content less $250 per tonne of copper paid for.

4. ECONOMIC EVALUATION

The economics of the deposit have been examined at each of the production levels of 100%, 75%, and 50% of the planned capacity of the plant and equipment costed in the preceding sections. At 25% copper in copper concentrate, these production levels are 14,000 tonnes, 10,500 tonnes, and 7,000 tonnes of contained copper respectively.

Table 3 attached sets out the costs of production, depreciation and interest charges at the different operating levels in January, 1975, terms. These costs are demonstrated graphically in Fig. 4 attached.

Examination of the project has found the deposit to be uneconomic at current copper prices and using existing mining techniques.

At full capacity, the costs of production, excluding royalties, depreciation, and interest charges, are $1,288.98 per tonne of contained copper. Including a 10% depreciation charge on fixed capital the costs rise to $1,581.84 per tonne. These figures exceed the break even copper price for world marginal production of around $1,240 per tonne (source: Chairman's address, latest meeting of Mt. Lyell Mining & Railway Company).

The minimum copper price at which the Mutooroo project would break even operating at full capacity and excluding interest charges and royalties would be $1,581.84. This price has only been exceeded for a short period from December 1973 to June 1974. The average L.M.E. copper price in December 1974 was $974.23. This is $607.61 below the break even price for the Mutooroo project excluding royalties and interest charges, and $800.47 below the break even price including interest charges.

A significantly higher copper price is required if the project is to break even at less than full capacity. Break even prices at different production levels are shown in Fig. 5.

Should mine production fall to 50% of capacity the break even price, excluding interest charges and royalties, would be \$2,366.34, i.e. £1,336.92 (Exchange rate: £1 = \$A 1.77). This is well in excess of the highest ever monthly average L.M.E. copper price, refer Fig. 6.

World consumption of refined copper is expected to remain stagnant for the next few years with a possible fall in consumption in 1975. World mine capacity on the other hand is expected to increase by 4.6% in 1975 creating a potentially substantial copper surplus. Prices should remain stable, but at a low level. There appears little likelihood that copper prices will reach the level necessary for economic exploitation of the Mutooroo deposit in the foreseeable future given the prohibitive costs associated with the project.

TABLE 2: DIRECT OPERATING COSTS

(Annual $'000)

	Salaries and Wages				Utilities		Operating Supplies	Maintenance Supplies	Overhead	Direct Cost	Contingency	Total Costs
	Supervision	Operations	Maintenance	General	Power	Water						
Mining	459	2,372	741		360	55	2,095	300		6,382	957	7,339
Metallurgy	165	162	94		548	245	1,030	300		2,544	382	2,926
Engineering	85		175				35	180		475	71	546
Accounting & Stores	168			35			10	10		223	34	257
Administration & General Services	133			146	78	75	50	20	500	1,002	150	1,152
Direct Cost	1,010	2,534	1,010	181	986	375	3,220	810	500	10,626		
Contingency	152	380	152	27	148	56	483	120	75		1,594	
Total Cost	1,162	2,915	1,162	208	1,134	431	3,703	930	575			12,220

TABLE 3: COSTS OF PRODUCTION AT VARIOUS LEVELS OF OPERATION

	Tonnes p.a. Copper in Concentrates					
	14,000		10,500		7,000	
	$000's	$ per tonne cont. Cu	$000's	$ per tonne cont. Cu	$000's	$ per tonne cont. Cu
Costs of Production						
Operating Costs:						
Fixed	6883	491.64	6883	655.52	6883	983.29
Variable	5337	381.24	4003	381.24	2669	381.24
Total Operating Costs	12220	872.88	10886	1036.76	9552	1364.53
Distribution Costs	1870	133.60	1403	133.60	935	133.60
Smelting & Refining Charges	3955	282.50	2966	282.50	1978	282.50
Total Costs of Production	18045	1288.98	15255	1452.86	12465	1780.63
Depreciation (10% of fixed cost @ $41 M)	4100	292.86	4100	390.48	4100	585.71
Cost of Production plus dep. charge	22145	1581.84	19355	1843.34	16565	2366.34
Interest (12% on 50% of total capital @ $45M)	2700	192.86	2700	257.14	2700	385.71
Total Costs, excluding royalties	24845	1774.70	22055	2100.48	19265	2752.05

wouldn't this reduce from year 1

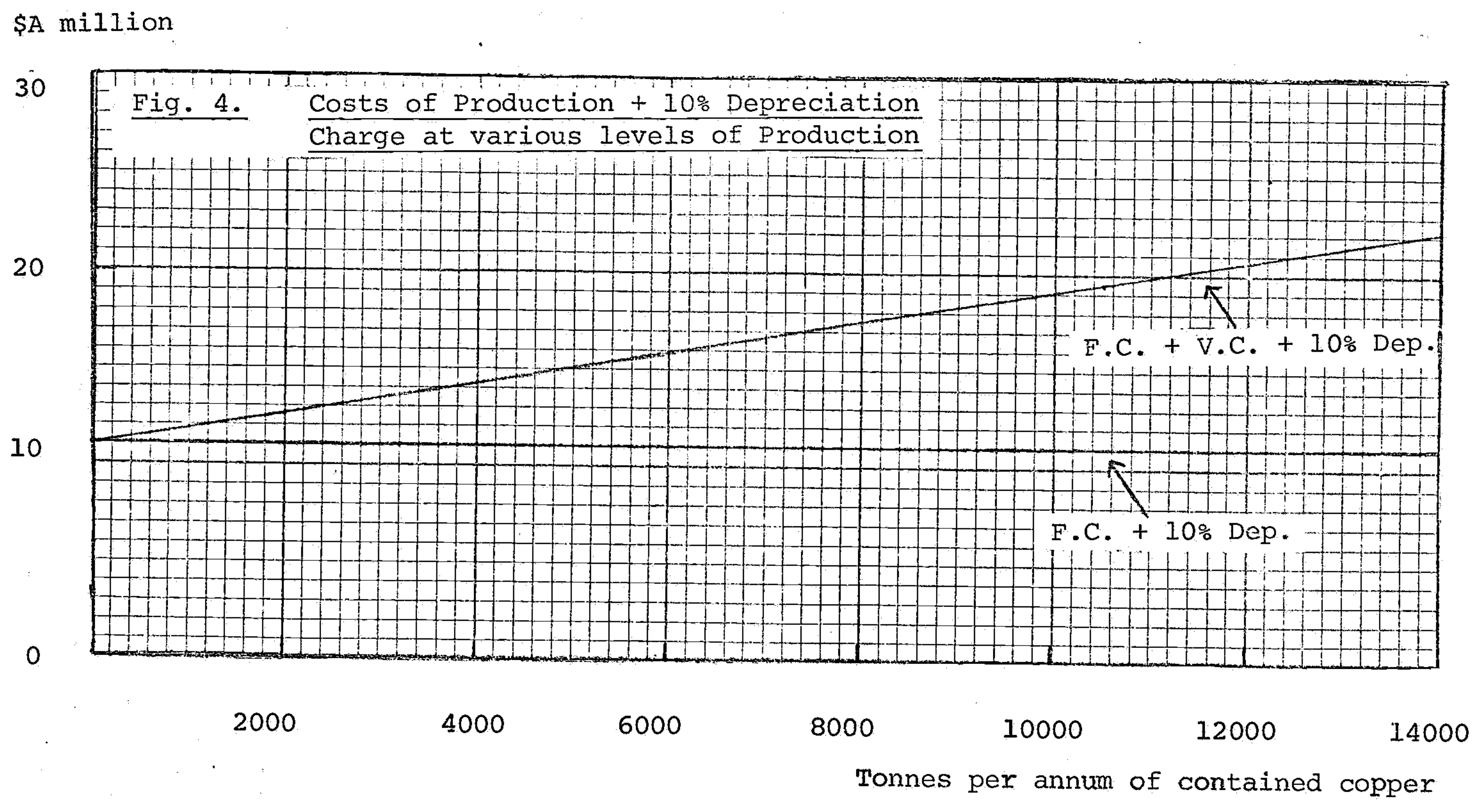


Fig. 4. Costs of Production + 10% Depreciation Charge at various levels of Production

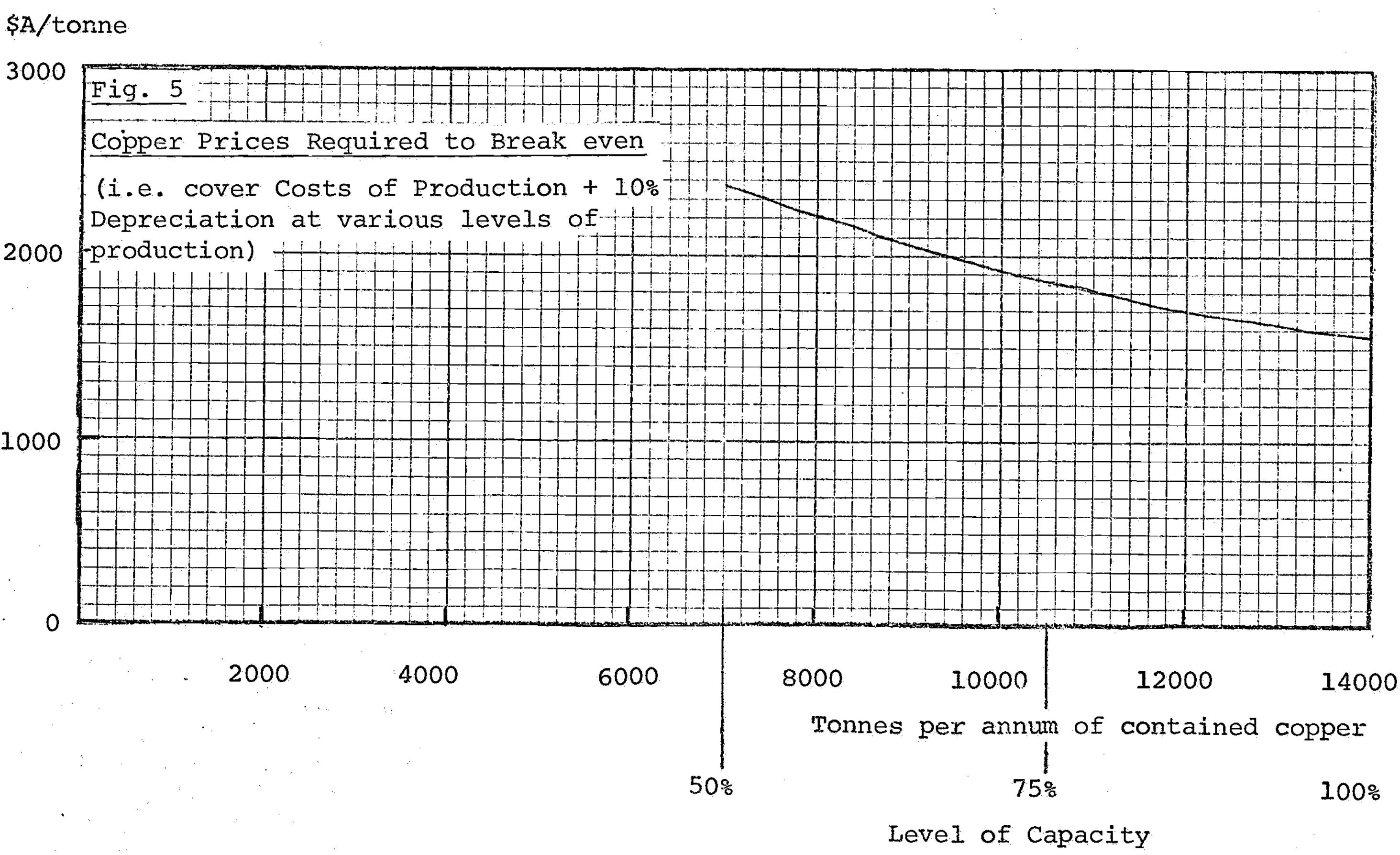


Fig. 5

Copper Prices Required to Break even

(i.e. cover Costs of Production + 10% Depreciation at various levels of production)

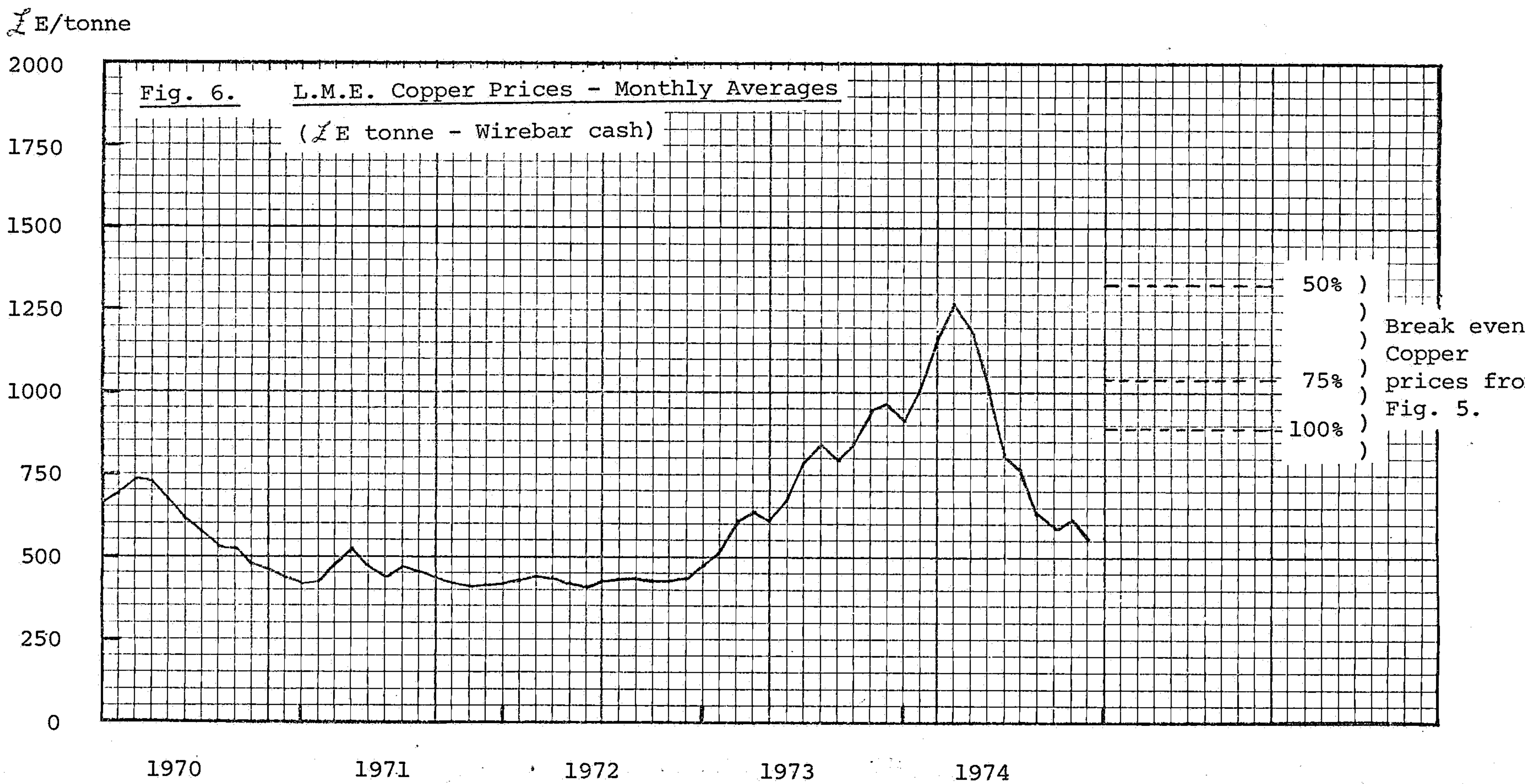
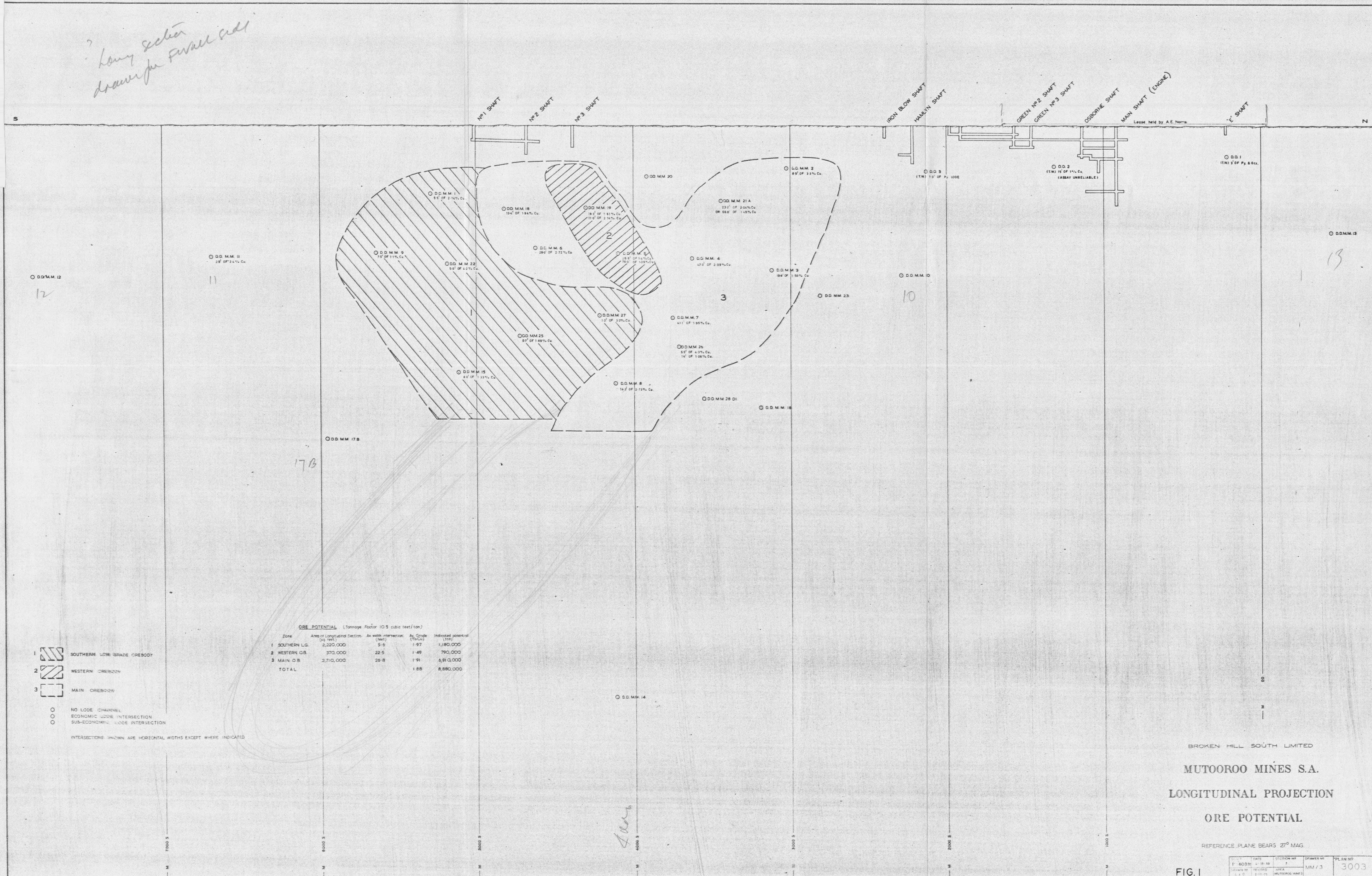


Fig. 6. L.M.E. Copper Prices - Monthly Averages

(£E tonne - Wirebar cash)



ORE POTENTIAL (Tonnage Factor 10.5 cubic feet/ton)

	Zone	Area of Longitudinal Section (sq. feet)	Av. width intersection (feet)	Av. Grade (%Cu)	Indicated potential (ton)
1	SOUTHERN L.G.	2,220,000	5·6	1·97	1,180,000
2	WESTERN O.B.	370,000	22·5	1·49	790,000
3	MAIN O.B.	2,710,000	26·8	1·91	6,910,000
	TOTAL			1·88	8,880,000

1 SOUTHERN LOW GRADE OREBODY
2 WESTERN OREBODY
3 MAIN OREBODY

NO LODE CHANNEL
ECONOMIC LODE INTERSECTION
SUB-ECONOMIC LODE INTERSECTION

INTERSECTIONS SHOWN ARE HORIZONTAL WIDTHS EXCEPT WHERE INDICATED

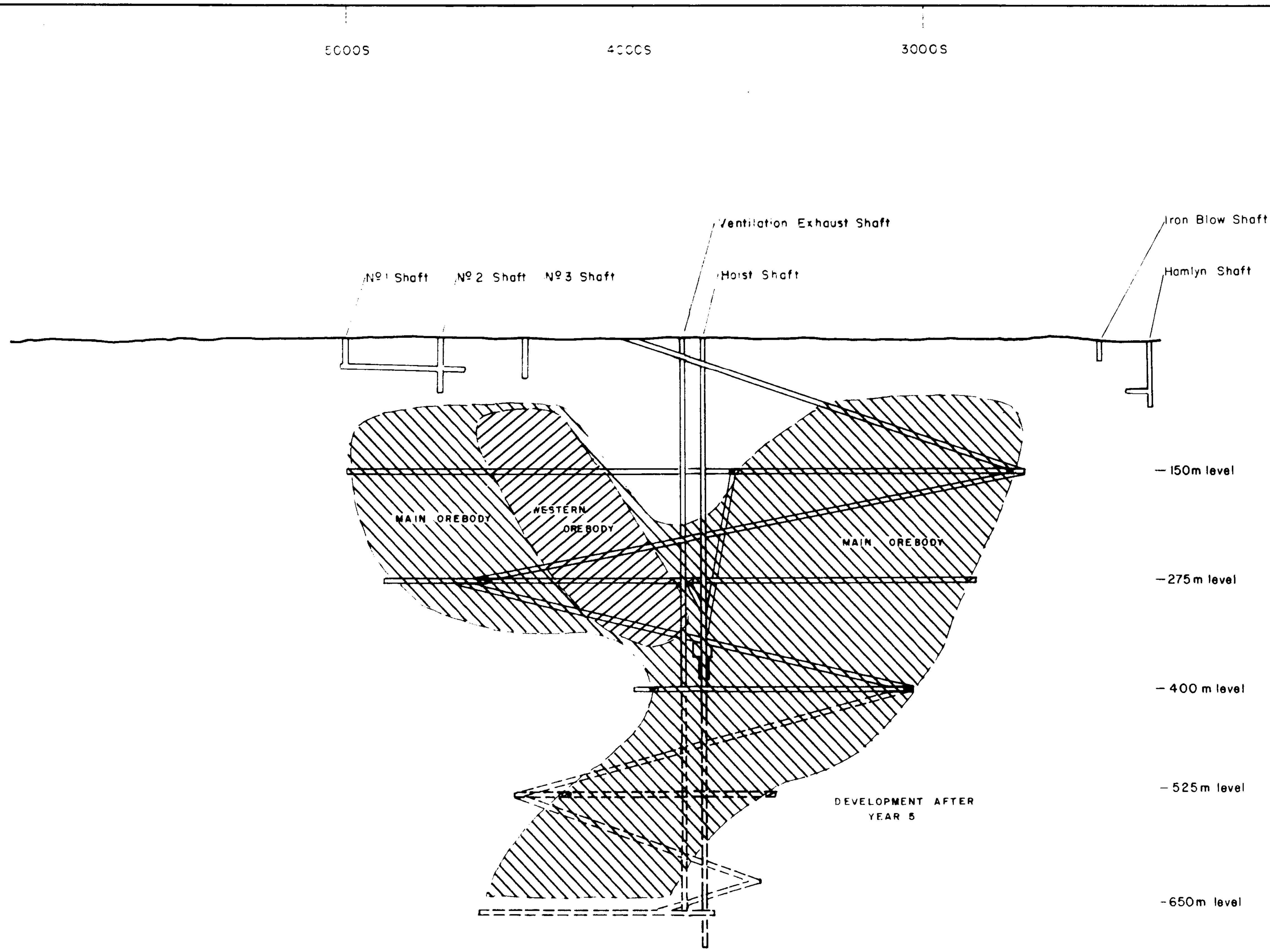
BROKEN HILL SOUTH LIMITED

MUTOOROO MINES S.A.
LONGITUDINAL PROJECTION
ORE POTENTIAL

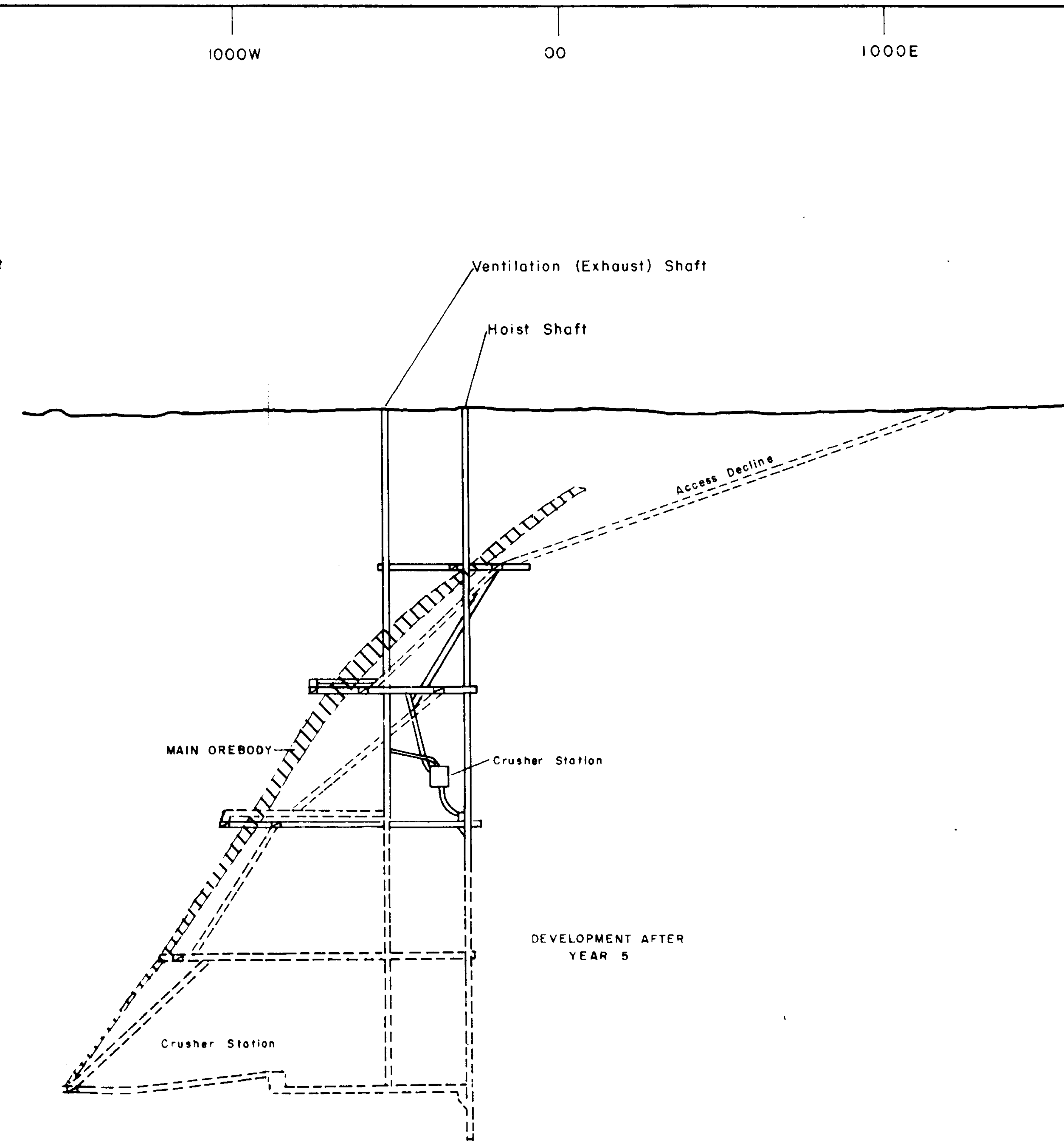
REFERENCE PLANE BEARS 27° MAG.

FIG. 1

SCALE	DATE	SECTION No	DRAWER No	PLAN No
1" 400 ft	4-10-59	1	MM / 3	3003
DRAWN BY	REVISED	AREA		
L.J.D.	2-11-71	MUTOOROO MINES		



LONGITUDINAL SECTION
MINE DEVELOPMENT



TYPICAL CROSS-SECTION
MINE DEVELOPMENT

NORANDA AUSTRALIA LTD

MUTOOROO MINE

CONCEPTUAL DEVELOPMENT LAYOUT

SCALE
4500

FIG. 2

DATE [illegible]	DRAWN
APPROVED	DRAWING No

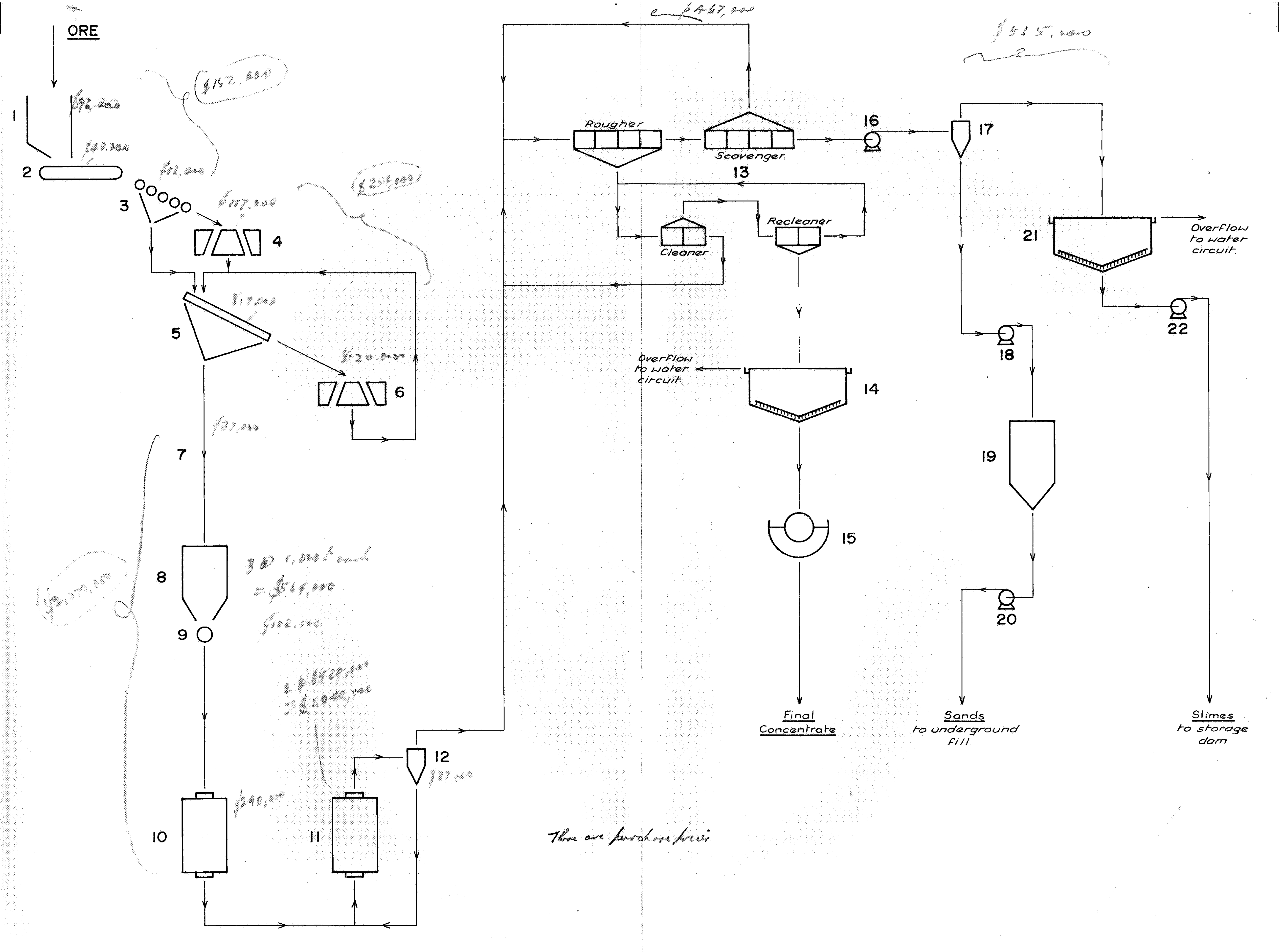


Fig. 3. MUTOOROO ORE TREATMENT FLOWSHEET

LEGEND FOR FIG. 3.

MUTOOROO ORE TREATMENT FLOWSHEET

1	Coarse Ore Bin, 500t
2	Apron Feeder
3	Rod Deck Screen, 50 mm
4	Standard Cone Crusher, 1.5 m
5	Vibrating Screen, 13 mm
6	Shorthead Cone Crusher, 1.5 m
7	Fine Ore Conveyor
8	Fine Ore Bins, 3 @ 1,500 t ea.
9	Discharge Feeders
10	Rod Mill, 3 m x 4.3 m
11	Ball Mills, 2 @ 3.9 m x 4.3 m
12	Cyclone Classifiers
13	Flotation
14	Concentrate Thickener, 15 m
15	Disc Filter, 20 m^2
16	Flotation Tailings Pumps
17	Tailings Cyclones
18	Cyclone Underflow Pumps
19	Sands Storage Tanks, 2 @ 3,000 t
20	Sands Pumps
21	Slimes Thickener, 46 m
22	Slimes Pumps

ENVELOPE 2384

TO THE DIRECTOR OF MINES:

Comments on the Economic Evaluation of the Mutooroo Copper Prospect (Jan. 1975)

The following comments are made on the Evaluation of the Mutooroo Copper Prospect submitted by Noranda Australia Ltd. and the E.Z. Company in January, 1975.

I. Summary of Evaluation

Prospect contains 9.4 million tonnes of massive sulphides, averaging 1.7% copper, with no associated metal values (e.g. silver, gold etc.), which can be mined with minimum width of 16' and 1% grade down to 650 metres depth.

The Method evaluated used conventional mining and ore treatment techniques to work the deposit to depletion in 10 years by mining 1,000,000 tonnes per year of 1.5% copper ore (after allowing 10% for dilution), and would produce 56,000 tonnes of 25% grade concentrates to be sold in Japan.

The project would require $45 million capital investment including $6.6 million for a township of 1,400 to be established on the Adelaide-Broken Hill road and railway 12 miles from the mine.

Operating costs including delivery costs to Japan and smelter charges but excluding taxation, royalty, depreciation and interest charges are estimated to be $1,300 per tonne of copper in the concentrates.

II. General Comments

1. Pessimistic Report - The report, after considering the cost of a conventional large scale method of operation, gives a very pessimistic outlook for the future of the prospect unless the price of copper holds double its present price of $960 per tonne for ten years.

   However the report really can only be considered as a first appreciation to see what is required; before the prospect is laid aside other factors should be considered and other methods of attack examined.

2. State Deposit - the group holding the E.L. had spent $540,000 by October 1971 and a small amount has been spent since. Noranda has accounted for 51%, Broken Hill South 25%, North Broken Hill 12% and the E.Z. Company 12%. The group has shown the deposit to be not large by present

standards but it is of considerable size for South Australia especially if it is looked at as a whole, because it is a massive sulphide deposit, not 1.7% copper in country rock. It was taken by W.J.L. Brooks in his 1966 report (page 57) to contain 35% pyrrhotite, 20% pyrite, 15% chalcopyrite, 5% magnetite and 25% silica by volume. This would give a composition of 5% copper (unfortunately further testing gave this as 1.7% copper), 34% sulphur, 44% iron and 20% silica.

This means that the 9 million tonnes considered for mining would contain, before dilution:-

150,000	tonnes of copper	@ 1.7% grade
3,000,000	tonnes of sulphur	@ 34% grade
4,000,000	tonnes of iron	@ 44% grade
1,800,000	tons of silica	@ 20%

The copper content is about 3 times the total copper produced from Burra in the last century and half the copper so far mined from Wallaroo-Moonta. This resource should be further examined to see if it can be developed by using other methods or at least enough found out about the deposit so its potential is known ready for change in circumstances favourable to its operation.

3. <u>Roasting and Leaching instead of Grinding and Flotation</u>

Because the copper is contained in massive sulphides, the possibility should be examined of extracting the copper by the old "Henderson Process" of roasting off most of the sulphur and then chloridise roasting the residue with salt to give a copper salt soluble in water. This and similar processes ~~was~~ were used extensively in Spain 100 years ago on pyrite containing 2-6% copper and was installed at Kapunda but the remnant ore there was too low grade only being ½% copper.

This could possibly save some of the $200 per tonne of copper allowed in the report for mill operating costs and $13m capital for the mill, as it may be possible to roast about 2" size lump-ore without further dressing or with most of the wall rock contamination removed by, say, heavy media separation. The massive sulphides would have an S.G. of over 4.2 against about 3 for amphibolite. If this method is used to dissolve the copper perhaps some form of refined copper could be produced in South Australia saving on the delivery cost of the concentrate to Japan of $136 per tonne of copper and the Japanese smelter charges of about $283 per tonne of copper (see Table 3 of Report).

If the mine is made into a smaller operation with a longer life, markets may be available for the sulphur as sulphuric acid (at present sulphur costs over $60 per tonne landed in Australia). Also some of the iron sinter may gain a market for cement manufacture etc.

4. Other Factors which were not taken into account but could be considered:-

(a) the Deposit is not "way out back" but is only 43 miles from Broken Hill, 30 miles of which are by good road and rail, so there should be no need to set up a town for 1400 people 12 miles from the mine. Broken Hill is slowing down and will have an oversupply of mine workers, houses, general facilities (M.M.M. is working on remnants and North B.H. Ltd. does not seem to be finding much ore).

Broken Hill may also have spare capacity of power, water workshop facilities.

(b) If the roasting idea is used some arrangements may be possible with B.H.A.S. at Port Pirie for the roasting to be carried out there.

(c) Slower mining may mean less need for initial capital at the commencement with the mine opened up more gradually and the underground crusher may not be economic. The mining method proposed for underground operation at Kanmantoo may be more suitable.

(d) If conventional treatment is to be used autogeneous grinding may be possible but pilot samples would be required to examine this.

III. Suggestion

It is suggested that the companies be given a further 6 months to consider these points and any others that they may think of and that they present a further report on those matters with the understanding that further work will be required in obtaining and testing bulk samples and advancing the knowledge of the deposit in the next period after the six months or the ground will be passed on to others to further upgrade the prospect.

JRA:JS
18th March, 1975.

MINERAL DEVELOPMENT ENGINEER

APPENDIX

Comments on some Details in the Evaluation

(a) Ore Reserves look reliable with the present knowledge but more should be found out about the extent of the wide area. The reserve figures may be a little conservative with the use of 10.5 cubic feet to the tonne for the massive sulphides.

(b) Mining Estimates

(i) No allowance appears to have been made for extraction losses due to level pillars etc. being left.

(ii) The ore body is rather flat lying in places and some difficulty may be experienced with hold-ups in the chutes if these follow the ore.

(iii) It appears that more stope openings may be required for access.

(iv) The development costs appear low to me. However the mining operating costs should be attained.

(c) Milling Costs

These capital costs seem high especially the $7m for the purchase installation, instresmutation and housing of the grinding section of the mill.

(d) Services and Infrastructure

The $6.6m does not seem necessary for the establishment of a township 12 miles from the mine and only 30 miles from Broken Hill.

The listed work force only adds up to 346 but the total work force is mentioned as 473. This is 37% extra, i.e. more than the 15% allowed for contingencies.

The $500,000 per year allowed for overheads page 53 seems high when no details are given.

DEPARTMENTAL MEMORANDUM

200blks200—11.64 1804

Date 24th June, 1975

From: W.B. ROBINSON

To: DIRECTOR OF MINES

Subject: E.L.73. Notes on the regranting of the E.L.

Docket Reference D.M. 205/75 Security File No.

Recommendations

I recommend that the E.L.73 be granted for a further period of one year with expenditure committments to be between $2,400 and $12,000; this expenditure to be for

1) further exploration - particularly south of main orebody;

or

2) a more detailed assessment of possible Government assistance in terms of infrastructure provision.

If at the end of one year no further exploration is recommended and the economic climate is not conducive to commencing work toward development then the company be advised to convert the area required to leases.

These leases should be granted for two years with general labour conditions suspended (as under Section 57(3)(b) of the Regulations) but with general feasibility work to continue.

If at the end of the expiration of the 2 year period, the company has made no serious attempt to do anything regarding the development of the mine the Minister should be advised not to renew the leases, unless the Company can show good reason why they should be renewed the company to take the risk of being plaintiff.

Discussions should also be held with the company with reference to the release of information on the deposit. As part of the agreement to renew the exploration licence either the Company or the Mines Department should write a report on the general geology of the deposit to be published and placed on open file.

Discussion

A. EXPENDITURE COMMITTMENTS ON E.L.73

The expenditure committment expressed in $/squ. kilometre is summarised for four differing conditions.

DM 117

DEPARTMENT OF MINES—SOUTH AUSTRALIA

DEPARTMENTAL MEMORANDUM

Date 24th June, 1975

800 lks200—11.64 1804

From :	**To :**
W.B. ROBINSON	DIRECTOR OF MINES

Subject: E.L. 73. Notes on the regranting of the E.L.

Docket Reference D.M. 205/75 **Security File No.**

[illegible]

1. $1,460 per squ. kilometre – when $35,000 (~~DM 21~~ recommendation) is expended on 24 squ. km, the present E.L.73 area.
   Total Expenditure $35,000

2. $460 per squ. kilometre – when $35,000 is expended on the combined areas of E.L.73 and E.L.141, a total of 76 squ. kilometres. (DM [illegible])
   Total Expenditure $35,000

3. $100 per squ. kilometre – if expenditure is at the general current rate asked of companies on exploration licences.
   Total Expenditure $2,400

4. $500 per squ. kilometre – if the area was converted to leases. The annual rental would be $5.00/hectare which is $500/squ. kilometre. This assumes ~~supervision~~ suspension of labour conditions.
   Total Expenditure on 24 squ. km. $12,000.

B. MINING ACT

Under Section 57(3)(b) of the Regulations lease "be worked in such a manner as may be approved by the Minister for any period not exceeding two years at any one time."

C. In order to encourage further exploration in the State serious thought should be given to encouraging Companies to release information on the results of their exploration.

In cases such as the present one and Wallaroo-Moonta, where exploration has been in progress for many years, I am of the opinion that publishing of the information would

1) not be harmful to the company – if they had had

DEPARTMENT OF MINES—SOUTH AUSTRALIA D.M. 117

DEPARTMENTAL MEMORANDUM

200b Ins200—11.64 1904

Date 24th June, 1975

From :	*To :*
W.B. ROBINSON	DIRECTOR OF MINES

Subject: E.L. 73. Notes on the regranting of the E.L.

Docket Reference D.M. 205/75 **Security File No.**

any new ideas from their exploration which would assist in other areas then presumably they would have followed up those ideas say in two to five years.

2) aid exploration within South Australia.

D. NORRIS' LEASE
South of E.L. 73.

Norris has applied three times in the last eighteen months for suspension of labour conditions on the ground that he is conducting negotiations with Mr. C. Bonython for the sale of the lease. Each application has been successful.

The last time he applied for suspension the Mining Warden summoned he and Bonython to appear in court to discuss the progress of negotiations. Mr. Norris did not appear in court due to a misunderstanding of the date.

The case is due to be heard again in the near future.

Unless Norris is manning the lease at the moment forfeiture of the leases could be applied for.

Company don't seem very interested. They have been asked to revamp the feasibility study and [illegible] also make disappear[illegible] of field geology.

R

W.B. ROBINSON