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ALBERTON GOLDFIELD

E.L 23/82

ANNUAL REPORT

MAY, 1986

BY M.R. BENDALL

FOR OCEANIA TASMANIA PTY.LTD.

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THE ALBERTON GOLDFIELD

E.L 23/82

OCEANIA TASMANIA PTY.LTD.

INTRODUCTION: Included in this annual report is a summary of the lease details and also a report prepared by consultant geologist, Peter Goldner. From the report which varified previous geological interpretation as well as further assaying, a decision has been made to carry out trial sampling of the lodes. This will establish the exact recoverable amount of free as opposed to sulphide and amalgamated gold in the ore. A mobile mill, a plan of which is included in this report, will be used to process the ore.
As the assays are not yet available, they will be included in the next first quarterly report.

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EXPLORATION LICENCE ALBERTON GOLDFIELD

1. LEASE DATA

Licence No.	E.L. 23/82
Location	Alberton, N.E. Tas.
Granted	1982
Area	8 sq. km.
Land Status	95% Crown Land, 5% Private
Ownership	Oceania Tasmania Pty Ltd
Encumberances	Nil
Mineral	Gold

Licence Conditions:

Amount expended on Lease (2 years) $15,960

(plus geophysical prospecting and research by Geological Department of University of Tasmania)

Reduction of Area: This exploration licence must be reduced to 4 sq. km's by five years from date of granting (namely May, 1988)

Period of Licence: Issued for 12 months with renewals for further 12 month periods

Maximum tenure is 10 years (namely May, 1993)

Expenditure Requirements: (Annual) — $5,000 per annum, with provision for over-expenditure to be carried forward tomeet under-expenditure

2. EXISTING INFRASTRUCTURE

a) Access: Public road frontage

b) Electrical power: Along frontage

c) Water: Dorcett and New Rivers

3. CURRENT ACTIVITY ON ADJACENT LEASES

Nil

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4. SUMMARY OF SALIENT GEOLOGY

The Upper Devonian Granite is 5 kilometres east of the Lower Devonian Mathinna beds. These form a ridge which runs north of Mount Victoria containing all the mines. These reefs are of a fault fissure nature, have strikes of N.E.-S.W. and N.W.-S.E. relating to a N.N.W.-S.S.E. trending anticlynal structure in the Mathinna beds, dipping to the south. Saddle reefs and laminated quartz lodes characteristic of the Bendigo field are also present in the area. There is almost the complete Mathinna sequence represented at Alberton from the graphitic shales at the bottom to yellow argillaceous sandstones and brown slates, to blue grey slates with interbedded quartzites and sandstones, to the top sequence of quartzites, sandstones and feldspathic sandstones.

5. SUMMARY OF PREVIOUS MINING AND EXPLORATION

The goldfield was discovered in 1883 and was intermittently worked until 1925. Over 100 reefs were mined, five being on a major scale, (New River, Ringarooma United, Mercury, Long Struggle, and Mount Victoria Mines). All the workings are recorded on a map compiling details of underground production. The field was abandoned due to under-capitalisation, sulphide ores, the Depression and a fire which destroyed the six batteries in the area. A total of 15,000 ounce of gold was produced from approximately 30,000 tonnes of lode material from the field.

6. POTENTIAL TARGET

The old workings did not sink to any great depth on the lodes and even the deepest mine, the Ringarooma United, did not go below river level. The Ringarooma United Mine, a 60m deep lode of 1 metre width and 60m length, has three lodes converging at depth. These lodes assayed 1 ounce per ton. Re-opening of the prospect shaft would mean that mining could be carried out immediately. Also ore left in the mine under 12 dwts per ton could now be treated. This philosophy could be followed for the five major mines in the area producing at least 50,000 ounces of gold, provided the lodes continue at depth with similar grades and width.

7. WORK UNDERTAKEN BY MINSTOCK

Two annual reports have been produced, covering relocation and cleaning of old workings, assaying, track cutting and geophysical work.

6. PROPOSED FUTURE EXPLORATION AND COST

Completely re-opening the five major mines with channel sampling of lodes plus drilling to determine further depth of lodes,

AN EVALUATION OF THE ALBERTON GOLDFIELD

BY PETER GOLDNER AND ASSOCIATES

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FOR OCEANIA TASMANIA PTY.LTD.

4. PROSPECT EVALUATION

4.1 GOLD PROSPECTS

4.1.1 ALBERTON GOLDFIELD

Oceania Tasmania Pty. Ltd holds Exploration Licence 23/82 covering an area of eight (8) square kilometres over the Alberton Goldfield in northeast Tasmania. The area is in close proximity to power and water supplies and the main road between the village of Alberton and Ringarooma passes through the property.

Geology and Mineralisation

The licence area is underlain by a folded sequence of the Lower Proterozoic Mathinna Beds which consists of interbedded slates, phyllites, sandstones and quartzites. The lower part of the sequence, which hosts the richest gold mineralisation, is dominated by finer grained lithologies including graphitic slates and phyllites and minor fine grained sandstone.

Devonian granodiorites outcrop to the east and west of E.L. 23/82 and flat lying Permian sediments overlie the Mathinna Beds to the south and southeast. A small Jurassic dolerite has intruded the sediments at Mt. Victoria southeast of Alberton.

The Alberton goldfield lies on the western limb of a broad north northwest trending anticline. A later phase of cross folding has affected the area and the highest density of quartz veining typically occurs in areas where cross folding is most strongly developed.

The auriferous veins of the Alberton field are dominantly located in a broad regional north northwest trending shear zone, approximately one kilometre wide, which can be traced from New River in the north to ~~Mathinna~~ in the south.

Two vein sets can be recognised, the first occupies fissures paralleling the north northwesterly strike while the second and more dominant set, strikes in a northeasterly direction (030^{o}) and dips steeply to the southeast (Nye and Blake 1938).

Over one hundred individual gold bearing quartz lodes (not all of economic significance) are known from the field. The veins range from a few centimetres to 10 metres in width and from a few metres to over 300 metres in length. The auriferous quartz veins characteristically contain sulphides and gold occurs both within the sulphides and as free gold. Arsenopyrite appears to be the most common sulphide, however pyrite, chalcopyrite galena and sphalerite are also present in various proportions. Hills (1923) reports that the highest gold values were generally obtained from galena bearing quartz veins.

Mining History

Gold was first discovered in the Alberton (or Mt. Victoria) field in 1882 and the most intense mining activity occurred prior to 1900. Intermittent mining continued until 1939 mainly in the Ringarooma United Mine.

The three largest mines on the field were the Ringarooma United, Mercury and Long Struggle Mines with numerous other smaller mines being present.

Accurate production records for the Alberton field are not available. Hills (1923) reports the total production from the Alberton field up to 1923, as about 15,000 ounces of gold from 30,000 tonnes of ore. Morrison (1981) states that approximately 10,000 ounces of gold were produced jointly from the Ringarooma United and Mercury Mines.

Typically mines on the Alberton field worked more than one quartz vein. For example the Mercury Mine worked two parallel lodes from two main adits and a small open cut while seven lodes were tapped by the Ringarooma United Mine from at least six adits and several shafts. Records suggest that most workings were quite shallow, the deepest being the Ringarooma United Mine which reached a depth of about 119 metres (390 ft.).

Grade information is sparse, Hills (1923) indicates that most of the lodes worked to that date contained between 10 g/t gold and 90 g/t gold. The total production figures above indicate an average grade of approximately 15 g/t gold.

Recent Exploration

No serious exploration of the Alberton field has been undertaken in recent years. The Tasmanian Mines Department drilled a number of diamond drill holes in the past to test various specific lode horizons, with little success.

In 1932-33 four holes were drilled in the New River Mine area to test various lodes at depths of up to 60 metres below surface. Results were disappointing with the best intersection being 1.5 metres averaging 0.83 g/t gold and 7.2 g/t silver.

In 1933, three underground holes were drilled in the Long Struggle Mine to test the Long Struggle lode. Hole 1, drilled horizontally from the lower adit intersected two closely spaced, thin auriferous veins between 41.45 metres and 43.43 metres. The first vein averaged 49.5 g/t gold and 3.35 g/t silver over 0.6 metres while the second vein, 0.76 metres wide, contained 41.88 g/t gold and 3.83 g/t silver. The second hole, collared in the same location but drilled at an angle of 55^{o}, intersected the lode horizon which only contained a trace of gold and silver. The third underground hole was abandoned after drilling 11 metres.

Amdex Mining Limited undertook a brief assessment of the Alberton field during 1980 under an option agreement with Kibuka Mines, the then title holder. Work by Amdex consisted of a literature review and a field examination which included sampling of some of the old workings. Eleven samples collected from seven of the old mines were analysed with disappointing results. The highest value was obtained from the Hannah adit which returned a result of 1.8 g/t gold.

Very little work has been undertaken by Minstock to date and has been limited to the relocation and clearing out of some of the old workings. A number of grab samples of dump material were collected from the old workings and analysed for gold with a number of encouraging results being obtained as follows:

- Two samples from the Mercury mine yielded 2.96 g/t gold and 19.4 g/t gold.
- A sample from the Hannah Mine yielded 4.6 g/t gold. Two other samples contained less than 0.1 g/t gold.
- A sample from the Cross lode workings assayed 3.3 g/t gold. Of the three other samples collected from this mine the best value was 0.28 g/t gold.
- A sample from the Point adits yielded 9.1 g/t gold and a second sample returned 1.38 g/t gold.

Minstock also undertook some pan sampling of alluvial/eluvial material available within the licence area with visible gold noted in a number of pans. The alluvials occur in the New and Dorset Rivers and are reported to be over 10 metres in depth in places and over 2 kilometres in length. Accurate dimensions are not available.

The writer briefly inspected both the Ringarooma United and Mercury mine working as well as some of the surrounding areas.

Three veins were noted within the currently accessible portion of two of the Ringarooma adits. The vein channels were well defined and appeared to range in width from 1 to 1.5 metres and ranged from massive quartz veins to zones of brecciated quartz and black shales. In a number of cases a massive quartz vein ranging from 0.3 to 0.5 m wide occurred on the footwall side of the channel with the remainder of the zone consisting of quartz veined and brecciated sediments. Limonite staining after sulphides is a characteristic of the lode material. The vein channels strikes north northeasterly and dip steeply ($\approx 70^{\circ}$) to the east. The well defined lode channels are generally 10-20 metres apart and there is little visual evidence to suggest that the sediments between the lodes are mineralised, however, no detailed sampling to test this possibility has been undertaken.

Numerous parallel lodes, usually with exploratory workings or small shafts, occur up hill and northwesterly from the Ringarooma workings. This area is obscured by vegetation and it is difficult to gauge the dimensions of the lodes. From scattered talus material, they appear to range from narrow veinlets to more massive lodes up to 0.3 metres wide.

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The accessible portion of two adits in the Mercury Mine were visited. The area of the lode channel was partially obscured due to the collapse of the stoped area, the lode channel however appears to have been approximately 1 to 1.5 metres wide. The lode horizon is similar in appearance to the lodes in the Ringarooma Mine and consists of a massive quartz vein on the footwall with a zone of quartz veined sediments in the remainder of the lode channel.

Conclusions and Proposed Exploration Programme

Mineralisation in the Alberton goldfield lies within a defined regional shear zone with individual auriferous quartz veins either paralleling or cross cutting the regional structure.

Although the majority of veins were small, the vein density was comparatively high and most of the old mines were developed on a series of veins rather than a single lode. A significant number of veins were greater than a metre in thickness and over 100 metres in length.

The field has not been systematically investigated and mining records suggests the density of veining and the indicated grade are sufficiently encouraging to warrant detailed evaluation of the area held.

The proposed initial exploration programme would include the compilation and evaluation of the historical data, followed by mapping of the entire area to locate and identify as many of the old workings as possible. This would be followed by detailed tape and compass mapping and systematic sampling of surface and available underground exposures of all the quartz veins as well as the adjacent host sediments. The initial phase exploration programme is estimated to cost approximately $15,000 - $20,000. Further work will depend on results but is likely to involve geophysical surveys and ultimately diamond drilling. An evaluation of the alluvial potential would also be undertaken as part of the first phase exploration programme.

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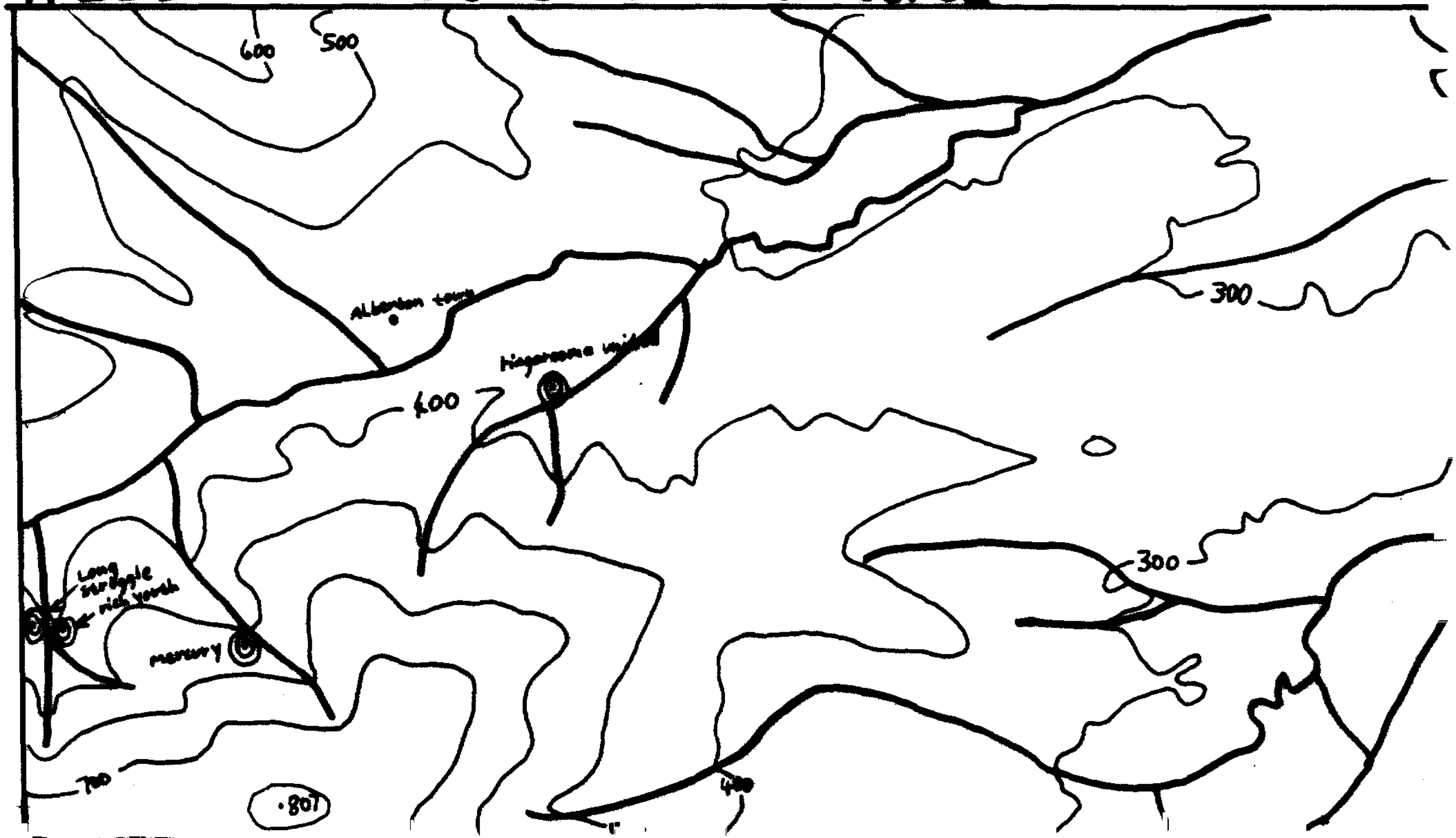
Positions for Sample Mining Alberton Goldfield ◎

ALBERTON GOLDFIELD EL23/82



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~~NOT RELEVANT~~

MINSTOCK SILVER/LEAD/ZINC PROJECT

PRELIMINARY PROCESS DESIGN

prepared for

LAWRENCE H. HOWROYD & ASSOCIATES

JULY 1984

by

LONGWORTH & McKENZIE PTY LIMITED

LONGWORTH & McKENZIE PTY. LIMITED

NOTE:

MODULES 1, 2 and 3 are the components specifically related to the gold sample plant for E L 23/82.

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MINSTOCK SILVER/LEAD/ZINC PROJECT

PRELIMINARY PROCESS DESIGN

prepared for

LAWRENCE H. HOWROYD & ASSOCIATES

JULY 1984

by

LONGWORTH & McKENZIE PTY LIMITED

REF: UMT0123/RBE/dmci

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1.0 INTRODUCTION

Lawrence H. Howroyd & Associates engaged Longworth & McKenzie Pty Limited (LM) to prepare a preliminary process design and equipment selection for the proposed Minstock Silver/Lead/Zinc Mining Venture. Minstock Mining is considering the development of several mineral deposits near Zeehan in Western Tasmania. Operations would include mining and treatment of ore to produce a blended concentrate. This study has been limited to the process plant, and includes a conceptual design, selection of major equipment, and indicative capital and operating costs. LM has prepared this report with the assistance of Michael J. Noakes, consultant metallurgist.

LM emphasizes that the plant design is very preliminary. Very little information has been available on the ore, and the process is based on a number of important assumptions, as discussed in the report. The cost estimates indicate the order of magnitude only for capital expenditure and plant operating costs. Both design and costs are heavily influenced by the ore characteristics, particularly grindability and grain size.

2.0 CONCEPTUAL DESIGN

2.1 Design Data

The following design parameters were established by the client :

- o Several ore deposits exist with potentially 70,000 - 100,000 tonnes of ore each at approximately 30% mineralisation.
- o Mining may involve only one deposit or perhaps several concurrently, in a number of small open cut operations.
- o The process plant is to be designed to produce 10,000 tonnes per annum of blended silver/lead/zinc concentrate at 60% mineralisation.

The information available included a geological report dated November 1983 by Summons Geoservices Pty. Ltd. and copies of several laboratory analyses on ore samples. The information pertained principally to the Comstock ore area. The Summons report estimates that "the best compromise" mineral assemblage for the Comstock area ores is 60% pyrite, 30% sphalerite and 10% galena.

2.2 Ore Grade and Specific Gravity

For the purposes of this study, it is assumed that the mineral assemblage above is representative of all the Minstock deposits. In a normal mining operation some non-sulphide dilution can be expected. Assuming 15% non-sulphide gangue, the typical ore feed grade to the process plant would be :

Feed Grade	7.8% Pb
	16.8% Zn
	600 g/t Ag
	23.8% Fe
	36.6% S
	15% Insolubles

The specific gravities of the minerals in the assumed feed analysis are :

	S.g.
Pyrite	5.0
Galena	7.5
Sphalerite	4.1
Insolubles	2.7

Accordingly, the specific gravity of the ore is calculated as 4.67.

2.3 Process Selection

Based on the above specific gravities, a gravity or heavy media process would remove only a small portion of the ore (insolubles), but would not reduce the major diluent, pyrite. Therefore it is proposed to consider a small demountable flotation plant, with associated crushing and grinding circuits.

Assuming the recoveries as listed below, the following product concentrate grade is calculated :

Concentrate Grade :	16.6% Pb
	35.8% Zn
	900 g/t Ag
	11.9% Fe
	33.7% S
	2.0% Insol.
Assumed Recoveries :	Lead 85%
	Zinc 85%
	Silver 60%
	Pyrite 20%
	Insol. 5%

The bulk rougher concentrate would be 40% of the feed weight (i.e. for a 100 tpd feed rate, 40 tpd of bulk concentrate would be produced).

2.4 Proposed Design Criteria

The following base design criteria were selected, based on the above considerations.

Feed grade :	7.8% Pb
	16.8% Zn
	600 g/t Ag
Ore Sg :	4.67
Crushability :	Soft - somewhat sticky
Work index (Bond) :	10.0 kwhr/short ton
ROM sizing :	100% - 900 mm
Ball mill feed size :	80% - 12.7 mm
Flotation feed size :	80% - 75 micron (Rosebery ores - very fine grained)
Reagent Addition :	$Na_2S_2O_5$ 400 g/t
	Na_2CO_3 800 g/t
	Lime 600 g/t
	Xanthate 130 g/t
	NaCN 65 g/t
	$CuSO_4$ 600 g/t
	Frother 60 g/t
Flotation pH =	9.0
Flotation Recovery :	Lead 85%
	Zinc 85%
	Silver 60%
	(Significant silver associated with tetrahedrite)
Flotation density :	40% solids
Flotation time :	20 minutes
Settling rate : (concentrates & tailings)	0.5 m^2/t/24 hrs
Final settled density	80% solids

It is assumed that the plant will operate 7 hours/day for 200 days per year, with an availability of 90%. Thus to produce 10,000 tpa of concentrate, the daily mill feed rate would be :

$$\frac{10{,}000 \text{ tpa product}}{\underset{\text{days}}{200} \times \underset{\text{conc. recovery}}{.4} \times \underset{\text{avail.}}{.9}} \text{ or 140 dry tonnes per day feed}$$

The conceptual process flowsheet is shown in Figure 1. The calculated mass balance and water balance are also shown, based on the assumed design criteria.

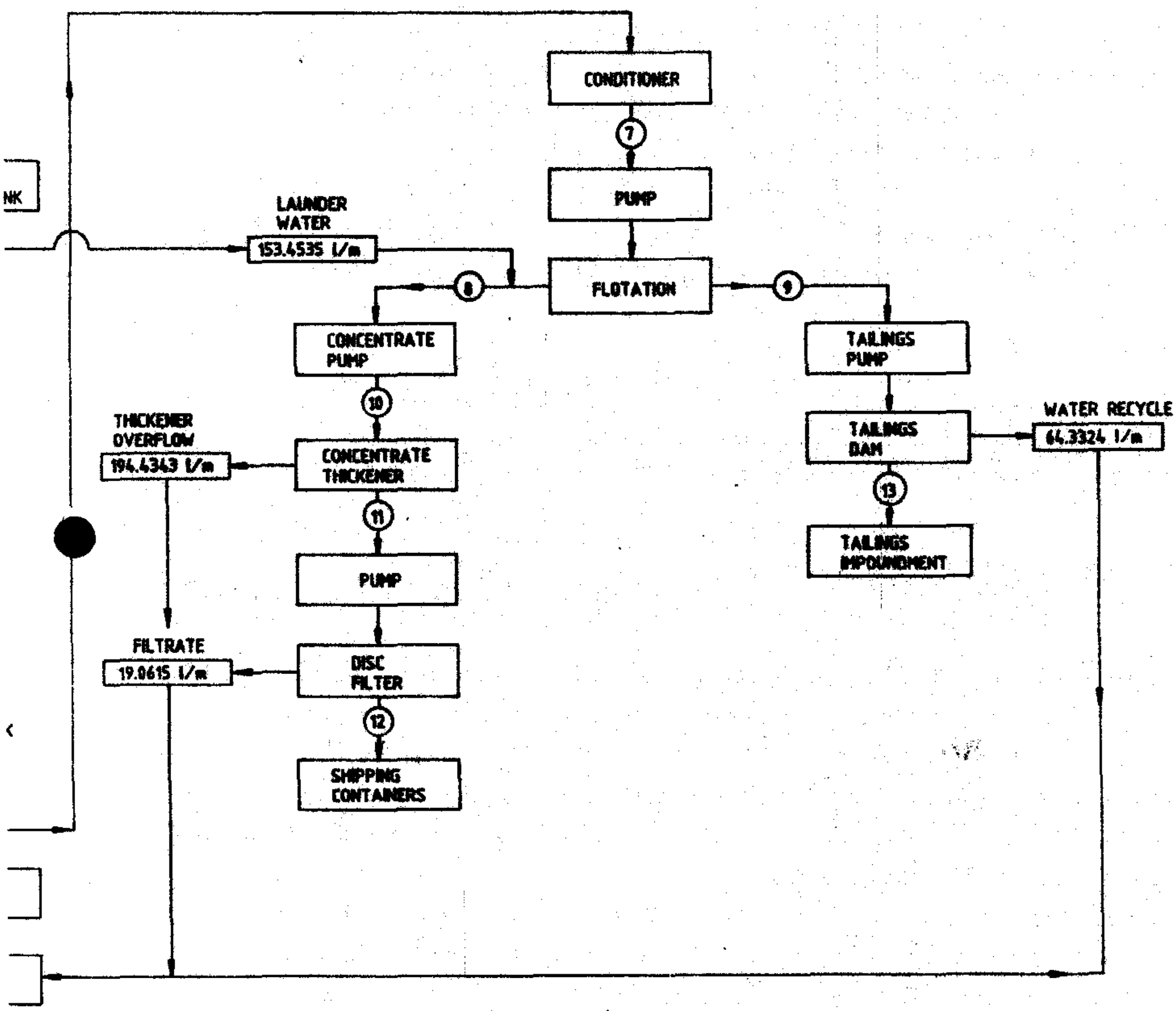
3.0 FUNCTIONAL DESCRIPTION

Based on the flowsheet in Figure 1, major equipment selections were made as listed in Appendix A. The following describes the plant operation.

Run-of-mine ore is reclaimed from the ROM stockpile using a 1.5 m^3 front end loader and fed to a hopper protected by an oversize grizzly. A 1.0 m width variable speed apron feeder withdraws the ore from the bin at the required rate and feeds it to a 300 x 900 mm single toggle jaw crusher for reduction to 85% passing 75 mm. The hopper, feeder and jaw crusher are mounted on a steel skid mounted module referred to as Module No. 1. The oversized crusher is chosen to handle the proposed topsize ROM ore of 900 mm.

Jaw crushed product is conveyed to a 1.2 x 3.6 m double-deck screen containing a 50 mm square mesh top deck protection screen and a 12.7 mm square mesh lower deck. The oversize from both decks falls into a 610 mm short head cone crusher whose product is conveyed back to the screen feed conveyor. It is estimated that 83% of the feedweight will circulate in this manner. The double-deck screen and cone crusher are steel skid mounted as Module No. 2.

FLOTATION CIRCUIT (MODULES 4 AND 5)



ALANCE

ED	5.1167	l/m
ILL ADDN.	140.6995	l/m
R ADDN.	153.4535	l/m
TOTAL	299.2697	l/m

ER OVERFLOW	194.4343	l/m
E	19.0615	l/m
S RECYCLE	64.3324	l/m
S IMPOUNDMENT	14.5820	l/m
CAKE	6.8595	l/m
TOTAL	299.2697	l/m

LANCED

)N (EXCLUDING EVAPOURATIVE
:) = 0.22 TONNES/TONNE ORE.

LONGWORTH & McKENZIE PTY. LIMITED		
DRAWN:	M.J.N.	MINSTOCK SILVER/LEAD/ZINC PROJECT ZEEHAN TASMANIA
REV. 1		
		CONCEPTUAL PROCESS FLOWSHEET FIGURE 1.
DATE:	14-6-84	

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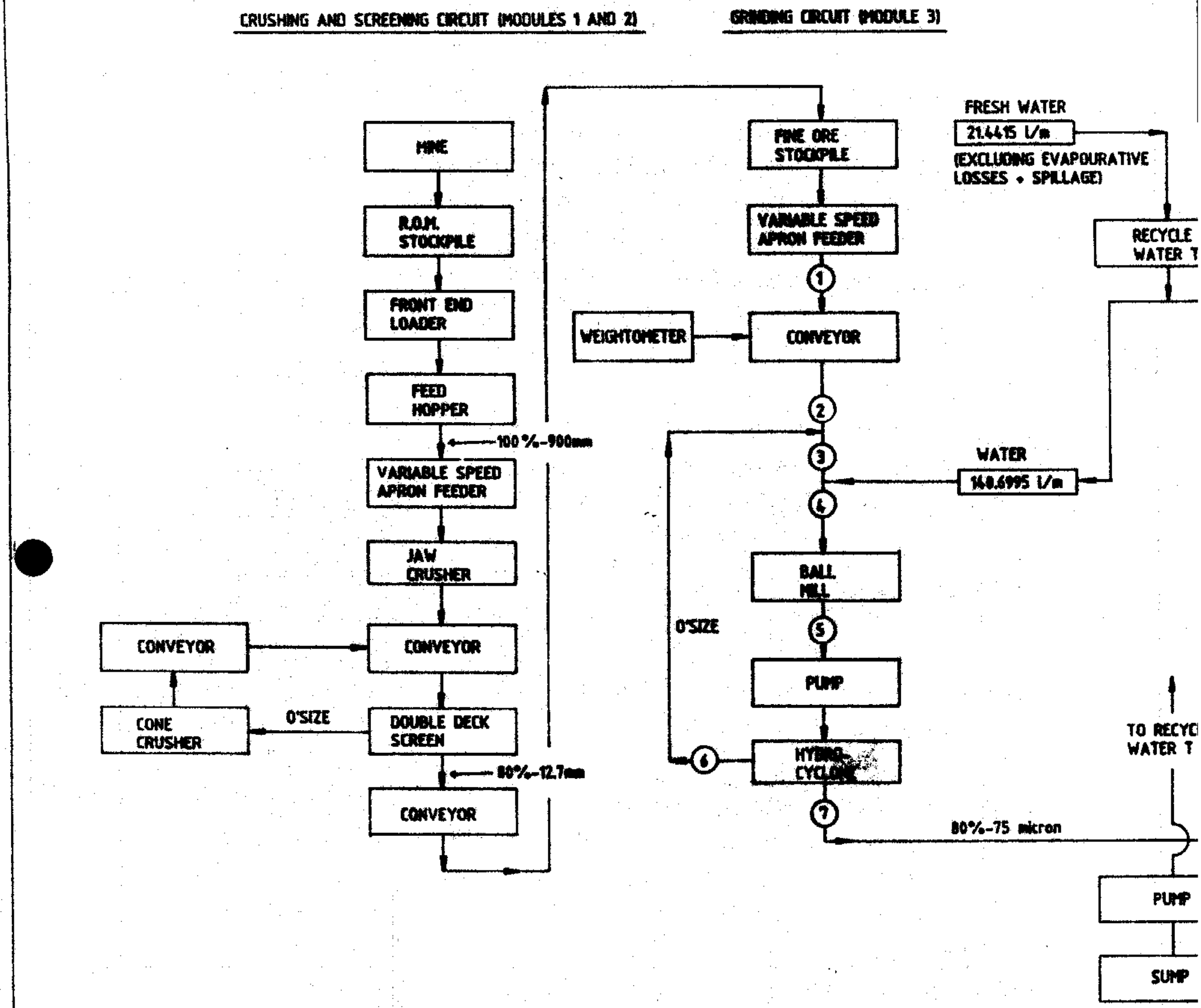

STREAM	MT/DAY ORE				PULP FLOWS			ASSAYS			% DISTRIBUTION		
	DRY MTPD	% SOLIDS	DENSITIES		PULP (l/m)	SOLID (l/m)	WATER (l/m)	% Pb	% Zn	g/t Ag	Pb	Zn	Ag
			SOLIDS	PULP									
1	140	95	4.67	3.9459	25.9333	20.8166	5.1167	7.8	16.4	600	100.0	100.0	100.0
2	140	95	4.67	3.9459	25.9333	20.8166	5.1167	7.8	16.4	600	100.0	100.0	100.0
3	420	74.9	4.67	2.4308	160.5598	62.4498	97.9100	7.8	16.4	600	100.0	100.0	100.0
4	420	55.0	4.67	1.7613	301.0593	62.4498	238.6095	7.8	16.4	600	100.0	100.0	100.0
5	420	55.0	4.67	1.7613	301.0593	62.4498	238.6095	7.8	16.4	600	100.0	100.0	100.0
6	280	67.7	4.67	2.1369	134.4265	41.6332	92.7933	7.8	16.4	600	100.0	100.0	100.0
7	140	40	4.67	1.4585	166.6328	20.8166	145.8162	7.8	16.4	600	100.0	100.0	100.0
8	56	36.8	5.46	1.4298	74.0264	7.1246	66.9018	16.6	35.0	900	85.13	85.24	60.0
9	84	42.5	4.26	1.4020	92.6064	13.6920	78.9144	1.93	4.13	400	14.87	14.76	40.0
10	56	15.0	5.46	1.1396	227.4799	7.1246	220.3553	16.6	35.0	900	85.13	85.24	60.0
11	56	60.0	5.46	1.9612	33.0456	7.1246	25.9210	16.6	35.0	900	85.13	85.24	60.0
12	56	85.0	5.46	3.2714	13.9841	7.1246	6.8595	16.6	35.0	900	85.13	85.24	60.0
13	84	80.0	4.26	2.5787	28.2740	13.6920	14.5820	1.93	4.13	400	14.87	14.76	40.0

WATER

WATER IN :

MILL
BALL
LAUN

WATER OUT :

THICK
FILTR
TAILI
TAILI
FILTE

WATER CONSUM
LOSSES + SPILL

The double-deck screen undersize sizing 80% passing 12.7 mm is conveyed to an open conical fine ore stockpile supported by 15 x 15 m concrete pad. The pad is above a tunnel and the fine ore passes through an opening in the pad to a 0.45 m width variable speed apron feeder to be withdrawn at the rquired rate of about 20 tph. The feeder discharges onto the mill feed conveyor, which is equipped with a weightometer for mill feed control.

Mill feed is washed into a 3 m diameter x 3 m long overflow type ball mill. The ball mill discharge is pumped to a 150 mm hydrocyclone classifier which recirculates an estimated 200% of the mill feed. The ball mill, pump and hydrocyclone are steel skid mounted as Module No. 3. This module is the largest unit by weight in the plant and represents a major portion of the capital cost. Significant capital savings could be realized if it could be shown that a coarser grind could be used than has been assumed.

The hydrocyclone overflow, which is estimated to contain 80% passing 75 micron material, flows by gravity to a 0.9 x 1.35 m reagent conditioning tank where the required reagents are added. This tank allows the pulp to be conditioned for 5 minutes. (It is noted that some reagents may be added to the ball mill or later in the circuit). The conditioned pulp is pumped to six 566 litre flotation cells where a bulk concentrate containing an estimated 16.6% Pb, 35.8% Zn and 900 g/t Ag is produced and is pumped to the concentrate thickener discussed below. The flotation tailings are pumped to a tailings dam for impoundment. The conditioner, pumps and flotation cells are mounted on a steel skid mounted module referred to as Module No. 4.

It is possible that, following metallurgical testwork, it may be necessary to add cleaner flotation cells to this module to produce a satisfactory concentrate grade.

Flotation tailings will be collected in a series of ponds, where the solids will be allowed to settle for impoundment, and the water will be recycled to the process plant.

In the conceptual model developed for this study the bulk concentrate is pumped to 6 m diameter x 3 m conventional thickener for dewatering up to 60% solids. Thickener overflow water is recycled by pump to the mill recycle water head tank.

The thickener underflow is pumped to an elevated steel skid mounted 4 x 1.8 m disc vacuum filter. The filter, vacuum pump, compressor, filtrate tank and pump are all mounted on this module, referred to as Module No. 5. Filter cake falls through the floor openings into a concentrate bay. Concentrate filter cake either falls directly into a container or haulage truck or is picked up by front end loader and transferred to its transportation mode.

While the plant is scheduled for 7 hours of operation/day, it is assumed that Module No. 5 will operate 24 hours/day. This is necessary for two reasons, vis :

* To cope with concentrate produced late in day shift.

* To reduce the filter size which would otherwise be quite large for a small throughput plant.

It is reasoned that the additional cost of 2-3 extra personnel more than offsets the additional capital cost required for very much larger thickening and filtration equipment.

The entire plant is powered by a 625 KVA diesel driven generator with associated switch gear and full tankage which is sized for normal operations. The major starting load requirement is for the ball mill motor (300 KW) which will have to be relay started and geared to prevent instantaneous overload. The generator and switch gear are steel skid mounted as Module No. 6.

Instrumentation is minimal with only pH control being used in the flotation section. Normal electrical interlocking and safety precautions are observed.

Sampling is by periodic hand samples collected by the laboratory technician and these samples along with mine, exploration, etc. samples are assayed by atomic absorption techniques in an onsite demountable laboratory which also houses the mill offices and toilet facilities.

There are no mill buildings budgeted but it is assumed that some rain shelter is provided over Modules 3, 4, 5 and 6.

The mill is managed by a qualified superintendent, who is assisted by a qualified chemist/foreman. A salaried secretary/clerk is employed to assist the superintendent and foreman. Hourly paid employees number eleven of whom seven are day shift only employees. The remaining four employees operate a 4 shift schedule over a 7 day week, to double as watchmen and product handling operators (the filtration and tailings disposal systems require 24 hour/day attendance).

4.0 CAPITAL COST ESTIMATE

The estimated capital costs are summarised in Table 1.

Capital costs are derived by sizing major units of equipment and calculating costs from in-house information or from cost information derived on other similar Australian operations. In some cases costs are adjusted for inflation to $A 1984. Certain equipment as follows are costed as used equipment and prices are taken from used equipment catalogues for good quality equipment :

Used equipment :	Front end loader
	Jaw crusher
	Cone crusher
	Ball mill

The remaining equipment is assumed to be new.

Installation costs such as steel fabrication, earthworks, foundation, piping, electrical, instrumentation, engineering, etc., are factored using the "Plant Component Cost Ratio Method" as proposed by Balfour and Papucciyan at the 4th Annual Meeting of Canadian Mineral Processors, January, 1972.

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TABLE 1

CAPITAL COST ESTIMATE SUMMARY

Item No.	Item	Major Equipment Included in Cost (Installed on Site)	Est. Capital Cost $A 1984
1	Front End Loader	F.E.L. only	75,000
2	Module No.1	Feed Hopper, Apron Feeder, Jaw Crusher, Motor, Conveyor Skid	95,500
3	Module No.2	D.D. Screen, Cone Crusher, Motor, 2 Conveyors, Skid	110,000
4	Fine Ore Stockpile	Concrete Pad	4,500
5	Fine Ore Apron Feeder	Feeder, Conveyor, Weightometer	30,350
6	Module No.3	Ball Mill, Motor, Discharge Pump, Hydrocyclone, Skid	350,750
7	Module No.4	Conditioner, Pump, Flotation Cells, Tailings Pump, Concentrate Pump, Skid	44,700
8	Water Head Tank	Tank, Pump	15,000
9	Concentrate Thickener	Thickener, Pump	28,000
10	Module No.5	Disc Filter, Vacuum Pump, Compressor, Filtrate Pump, Skid	80,000
11	Module No.6	Power Pack, Tankage, Skid	145,000
12	Miscellaneous Items	Laboratory/Office, Tailings Dam, Utility Vehicles, Roadworks, Site Clearing	110,000
13	Engineering & Project Management		125,000
		TOTAL MILL COST	$1,213,800

Note: No Allowance for Contingency

018025

5.0 OPERATING COST ESTIMATE

It is estimated that the mill operating cost is $24.30 tonne ore treated derived as summarized in Table 2 and detailed in Tables 3 and 4.

It is stressed that these figures are based on experience of similar Australian operations, with the Rosebery operations being used as a major influencing factor. Metallurgical testwork is required to define the true operating parameters.

TABLE 2

Operating Cost Estimate Summary

Item	Cost/Annum $A 1984	Cost/Tonne $/t
Manpower	433,700	17.35
Consumables		
Power	90,000	3.6
Water	1,100	0.04
Steel	25,750	1.03
Reagents	32,000	1.28
Spares	25,000	1.00
TOTAL	607,550	24.30

Note - No allowance for contingency

TABLE 3

Estimated Manpower Cost

Type Salaried (S) Hourly (H)	Requirement No of Shifts	 No of Days/week	Total No of Men	Annual Expense $A 1984
Superintendent (S)	-	-	1	$ 47,500
Foreman/chemist (S)	-	-	1	40,000
Secretary/Clerk (S)	-	-	1	16,200
Sub-Total (S)	-	-	3	$ 103,700
F.E.L. Driver (H)	1	5	1	$ 25,000
Crusher & Ball Mill Operator (H)	1	5	2	25,000
Flotation Operator	1	5	1	25,000
Prod. Hdlg./Watchman	4	7	4	100,000
Labourer (H)	1	5	1	25,000
Mechanic/Fitter (H)	1	5	1	25,000
Mechanic's Assistance (H)	1	5	1	25,000
Sample Bucker/Lab. Tech. (H)	1	5	1	25,000
Sub-Total (H)	-	-	11	$ 275,000
20% Absentee	-	-	-	$ 55,000
TOTAL ANNUAL	-	-	14	$ 433,700

TABLE 4

Estimated Consumables Cost

Item	Consumption Rate	Unit Cost	Annual Cost	Sub Totals
Power	30 KWhr/tonne	$0.12/KWhr	$ 90,000	$ 90,000
Water	0.44l/t ore	$0.10/t	$ 1,100	$ 1,100
Steel				
Crusher Liners	0.10 kg/t	$2000/t	$ 5,000	
Mill Liners	0.10 kg/t	$2000/t	5,000	
Steel Balls	0.70 kg/t	$900/t	15,750	$ 25,750
Reagents				
Sodium Bisulphite	400 g/t	$380/t	$ 3,800	
Sodium Carbonate	800 g/t	$240/t	4,800	
Lime	600 g/t	$150/t	2,250	
Xanthate	130 g/t	$2100/t	6,825	
Sodium Cyanide	65 g/t	$1450/t	2,356	
Copper Sulphate	600 g/t	$650/t	9,750	
Frother	60 g/t	$1500/t	2,250	$ 32,000
Spares (2½% Inst. Cap)	-	-	$ 25,000	$ 25,000
TOTAL ANNUAL				$ 173,850

* Current Estimated Prices Delivered Zeehan

6.0 DISCUSSION

The report presents a preliminary design and cost estimates for a small ore concentrator. The capital and operating costs are indicative only. The plant is designed for operation on day shift only (with the exception of the filtration section) to minimise labour costs.

Because the plant has a relatively low throughput, a modular mounting approach has been adopted for major systems and equipment. This modular approach provides flexibility in plant arrangement. The modules are demountable and the plant can be shifted to different sites if necessary, with a minimum of civil and earthworks required. Figure 2 is a photograph of a similar transportable ore concentrating plant. The proposed Minstock project requires a concentrate thickener and a tailings impoundment area, and these facilities may have to be reeconstructed for alternate sites, depending on location.

A disk type vacuum filter has been included in the preliminary design. The principal reason for the filter is to reduce the concentrate moisture to a level suitable for bulk solids handling (about 15% moisture). This assumes that the product will be transported to Burnie for ocean shipment to a smelter. Altervatively, it may be possible to negotiate with Electrolytic Zinc Company for handling and shipping the concentrate by rail from their Rosebery concentrator.

The moisture content of the product concentrate can also influence the reactivity of the material. Under certain conditions sulphide concentrates are susceptible to sportaneous combustion during storage and handling. This is most likely to occur if the material is allowed to dry to below 10%. This factor should be considered in the design of storage and handling facilities.

If the results of this current study together with other previous work indicate that the project is viable, the following investigative work is recommended :

(1) Further exploration work is needed to prove sufficient ore reserves of suitable quality.

(2) A preliminary mine plan, including cost estimates, should be prepared.

(3) The market for the concentrate should be investigated (tonnage, sales prices, and location).

(4) Concentrate storage and transport options should be considered to establish the most appropriate and lowest cost methods.

(5) Metallurgical testwork should be performed to allow more definitive process design to proceed. This could have significant effect on both capital and operating costs.

(6) Following the above items, an engineering feasibility study should be carried out to confirm project viability.

FIGURE 2

TYPICAL TRANSPORTABLE CONCENTRATING PLANT

APPENDIX A

LIST OF MAJOR EQUIPMENT

APPENDIX A

032 [illegible]STOCK - MAJOR EQUIPMENT LIST

Module No.	Major Equipment Description	Size or Model No.	No. of Units	Motor KW/Unit	Total Connected KW
	Front End Loader	1.5m³ Bucket	1	-	-
	Feed Hopper	2m x2m x 1m	1	-	-
1 (Primary Crushing Module)	Variable Speed Apron Feeder	3m x 1m	1	4	4
	Jaw Crusher	300 x 900mm	1	55	55
	Jaw Product Discharge Conveyor	0.45 x 15m	1	5.5	5.5
2 (Sec. Crusher & Screen Module)	Double Deck Screen-50mm/12.7m	1.2mm x 3.6mm	1	15	15
	Cone Crusher-Short Head+Fine Bowl	610mm	1	22	22
	Cone Crusher Discharge Conveyor	0.45 x 15m	1	5.5	5.5
	Screen Undersize Product Conveyor	0.45 x 15m	1	5.5	5.5
	Fine Ore Stockpile Pad	15m x 15m	1	-	-
	Variable Speed Apron Feeder	0.45 x 1.8m	1	4	4
	Mill Reed Conveyor	0.45 x 15m	1	5.5	5.5
	Weightometer	Ramsey 10-20A	1	-	-
3 (Grinding Module)	Ball Mill - Overflow Type	3m x 3m	1	-	-
	Ball Mill Motor	-	1	300	300
	Ball Mill Discharge Pump - Horizontal Centrifugal	62.5mm SRL	1	4	4
	Hydrocyclone	150mm	1	-	-
4 (Flotation Module)	Conditioner	900 x 1350mm	1	1.5	1.5
	Flotation Feed Pump - Vertical Sandle Centrifugal	50mm	1	2.2	2.2
	Flotation Cells	566 litre	3x2	4/pair	12
	Tailings Pump - Horizontal Centrifugal	62.5mm SRL	1	4	4
	Concentrate Pump - Vertical Spindle Pump	50mm	1	2.2	2.2
	Water Head Tank + Recycle Water Pump	6m x Ø x 3m	1	1.5	1.5
	Concentrate Thickener	6m x Ø x 3m	1	.75	.75
	Thickener Underflow Pump - Vertical Spindle	50mm	1	2.2	2.2
5 (Filtration Module)	4 Disc Filter + Vacuum Pump		1	15	15
	+ Compressor + Filtrate Pump	4 x 1.8m Ø	1	37	37
			1	4	4
			1	7.5	7.5
6	Power Pack - Generator Set	625 KVA	1	-	-

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APPENDIX B

CAPITAL COST BREAKDOWN

APPENDIX B

CAPITAL COST ESTIMATE

ITEM 1

Front End Loader		$A 1984
Second hand price from Thompson Plant P/L (good condition)		$ 75,000

ITEM 2

Module No.1

Feed Hopper	new allow	$ 5,000
Apron Feeder	new price allow	$ 15,000
Jaw Crusher (new price $56,700)	used allow	$ 30,000
75 H.P. Motor	new price	$ 7,500
Discharge Conveyor	new allow	$ 5,000
	Subtotal	$ 62,500

Module Construction

Steel Fabrication + Construction (material & labour)		$ 15,000
Electrical (material & labour)		$ 10,000
Instrumentation (material & labour)		$ 1,500
Engineering		$ 6,500
	No.1 Module cost	$ 95,500

ITEM 3

Module 2

Double Deck Screen	new price allow	$ 28,500
Cone Crusher (new price $57,860)	used allow	$ 30,000
30 HP Motor	new allow	$ 3,000
Cone Discharge Conveyor	new allow	$ 5,000
Screen Undersize Conveyor	new allow	$ 5,000
	Subtotal	$ 71,500

Module Construction

Steel Fabrication & constr. (material & labour)		$ 18,000
Electrical (material & labour)		$ 11,000
Instrumentation (material & labour)		$ 2,000
Engineering		$ 7,500
	No.2 Module Cost	**$ 110,000**

CAPITAL COST ESTIMATE
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ITEM 4

			$A 1984
Fine Ore Stockpile			
Concrete 225 m^2 at $20/$m^2$	allow	$	4,500

ITEM 5

Fine Ore Apron Feeder			
Apron Feeder	new allow	$	12,250
Installation		$	1,800
Earthworks		$	1,000
Electrical		$	1,800
	Total Feeder	$	16,850
Mill Feed Conveyor	new allow	$	5,000
Weightometer	new allow	$	8,500
	Fine Ore Apron Feeder Cost	**$**	**30,350**

ITEM 6

Module No. 3			
Ball Mill (new price $347,000)	used allow	$	175,000
400 HP Motor	new	$	40,000
Ball Mill Discharge Pump	new	$	2,000
Hydrocyclone	new	$	2,750
	Subtotal	$	219,750
Module Construction			
Steel fabrication & Constr. (material & labour)		$	55,000
Piping (material & labour)		$	15,000
Electrical		$	30,000
Instrumentation		$	6,000
Engineering		$	25,000
	No.3 Module cost	**$**	**350,750**

CAPITAL COST ESTIMATE

ITEM 7 $A 1984

Module No.4

Conditioner	new allow	$	3,200
Flotation Feed Pump	new	$	2,000
Flotation Cells (6x20ft^3) $2,600/cell	new	$	15,600
Tailings Pump	new	$	2,000
Concentrate Pump	new	$	1,500
		$	24,300

Module Construction

Steel Fabrication & Const (material & labour)		$	6,000
Piping (material & labour)		$	3,000
Electrical		$	4,000
Instrumentation		$	700
Low Pressure Blower & Plant Services		$	4,000
Engineering		$	2,700
	No.4 Module cost	**$**	**44,700**

ITEM 8

Water Head Tank (Installed)	**new allow**	**$**	**15,000**

ITEM 9

Concentrate Thickener

Price	new allow	$	18,500
Installation		$	2,700
Earthworks		$	1,300
Electrical		$	2,000
	Subtotal	$	24,500

Conc. Thickener U/F Pump

	new	$	1,500
Installation		$	1,000
Piping		$	1,000
	Subtotal	**$**	**3,500**
	Concentrate Thickener Total	**$**	**28,000**

CAPITAL COST ESTIMATE

		$A 1984
ITEM 10		
Module No.5		
4 Disc Filter	**new allow**	$ 50,000
Module Construction		
Steel Fab & Const.		$ 12,500
Piping		$ 3,500
Electrical		$ 7,000
Instrumentation		$ 1,500
Engineering		$ 5,500
	No.5 Module cost	**$ 80,000**
ITEM 11		
Module No.6		
Power Pack -	allow $200/KVA	$ 125,000
Miscellaneous	tankage	
	control panel	
	housing	$ 20,000
	skid mounting	
	No.6 Module Cost	**$ 145,000**
ITEM 12		
Miscellaneous Items		
Laboratory/Office	allow	$ 25,000
Tailings Dam	allow	$ 50,000
Utility Vehicles	allow	$ 10,000
Roadworks & Site clearing	allow	$ 25,000
	Miscellaneous Items	**$ 110,000**
ITEM 13		
Engineering & Project Management	**allow**	**$ 125,000**
TOTAL MILL COST		**$ 1,213,800**