MINSTOCK SILVER/LEAD/ZINC PROJECT

PRELIMINARY PROCESS DESIGN

prepared for

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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 :

- Several ore deposits exist with potentially 70,000 - 100,000 tonnes of ore each at approximately 30% mineralisation.
- Mining may involve only one deposit or perhaps several concurrently, in a number of small open cut operations.
- 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

Shows how poorly they were informed by client.

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.

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 catelogues 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
›	Module No.3	Ball Mill, Motor, Discharge Pump, Hydrocyclone, Skid	350,750
	Module No.4	Conditioner, Pump, Flotation Cells, Tailings Pump, Concentrate Pump, Skid	44,700
	Water Head Tank	Tank, Pump	15,000
	Concentrate Thickener	Thickener, Pump	28,000
0	Module No.5	Disc Filter, Vacuum Pump, Compressor, Filtrate Pump, Skid	80,000
1	Module No.6	Power Pack, Tankage, Skid	145,000
2	Miscellaneous Items	Laboratory/Office, Tailings Dam, Utility Vehicles, Roadworks, Site Clearing	110,000
3	Engineering & Project Management		125,000
		TOTAL MILL COST	$1,213,800

ote: No Allowance for Contingency

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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	0.441/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.