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PRE-FEASIBILITY EVALUATION

LARA PROJECT

Prepared for:

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64301

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1. SUMMARY

1.1 Introduction

The following report concerns an evaluation of the Lara polymetallic deposit situated 15 km from Chemainus on Vancouver Island.

The objective of the evaluation was to provide justification and security for a program of underground exploration. The evaluation has therefore been limited to a pre-feasibility level in which order of magnitude capital and operating costs have been estimated. In addition operating concepts have been examined to identify those factors which may influence project viability.

This report has, purposely, been kept brief and the technical sections are appended separately under the following headings:

- ore reserve; geometry, distribution and geotechnical
- mining system
- metallurgical
- mining waste disposal

The metallurgical section has been produced by Proton Systems Ltd.

The scope of the project has been limited to the following:

- base case: 500 tonnes per day production for an annual production of 175,000 tonnes

- alternative: 200 tonnes per day, or 70,000 tonnes per year, factored from the base case

A detailed analysis of the ore reserve has not been attempted although an approximate assessment has been made for the purposes of evaluating the mining system.

The accuracy of cost estimates is no better than ± 25%.

1.2 Discussion

Reserves

- 500 tonne/day option - total mineable reserve = 554,000 tonnes
- 200 tonne/day option - total mineable reserve = 359,000 tonnes

The development of a mineable reserve within the $65 cut-off contour assumes that the ore grade is continuous within that cut-off. This is a gross assumption and most unlikely to be the case. Even so the proportion of waste development necessary is approximately 37% of the ore tonnage. The development constitutes 26% of the overall mining costs. If, as is likely, the orebody is less continuous the grade should improve, the reserve tonnage drop and the development cost will increase in proportion to the reduction in reserve tonnage.

An indication of the ratio of grade to tonnage is given in the latest Abermin estimates of reserves:

864,000 tonnes: C $ 140/tonne
304,000 tonnes: C $ 173/tonne
193,000 tonnes: C $ 206/tonne

The tonnage decreases faster than the grade increases. Assuming that the same total length of development is required the difference in cost in going from $140 value ore to one of $206 might be in the region of $38/tonne of ore. The difference between the net smelter return of the lower grade and the higher grades is approximately $40. Continuity is critical and 'high-grading' is not, necessarily, a viable option unless the high grade sulphide zones can be found more frequently than presently indicated.

Mining System

A mechanized cut-and-fill method is viable. Ground conditions will be variable and a proportion of the stoping will require bolts and screens. This has been taken into account in the costing analysis.

The ramp access is 20% but four-wheel drive haulage trucks should operate effectively so long as the footwall is properly maintained.

Maximum ventilation requirements in a mining area would be 50 m^3/s at fairly low pressures.

The 200 tonne/day case is based on a form of shrinkage with small pillars to decrease the 'open' span. The viability of such a method has not been assessed. The lower production rate has been included simply to estimate the influence on overall costs and viability.

Costs

500 tonnes/day

Item	Capital	Operating
Mining	$3.4 million	$32.5/tonne
Metallurgical/Site	$11.8 "	$10.0/"
Tailings	$1.7 "	$0.47/"
	C $16.9 million	C $42.97/"

200 tonnes/day

Item	Capital	Operating
Mining	$2.2 million	$43.4/tonne
Metallurgical	$8.7 "	$14.7/"
Tailings	$1.7 "	.47/"
	C $12.6 million	C $58.57/"

Capital costs are based on new plant and equipment. Mining capital costs could be considered as optimistic depending on contractor prices.

Productivity

500 tonne/day - 14.3 tonnes/manshift - all employed in mining
200 tonne/day - 7.14 tonne/manshift - all employed in mining

There is scope to improve the productivity.

Metallurgical

The Lara sulphides are associated intimately with considerable intergrowth of one mineral in another. Very fine grinding is necessary to complete liberation, though a barren tailing can be produced at a coarser grind (55% minus 200 mesh).

Further testwork is required to determine grades of the final concentrates. Without having achieved salable final grades, precise estimates of metal recoveries are not possible. Based on the information available, recoveries are predicted to be:

Gold	85 - 90% overall
Silver	80 - 90% overall
Lead	80 - 85% in lead concentrate
Copper	75 - 85% in copper concentrate
Zinc	80 - 85% in zinc concentrate

Concentrate grades are likely to be low. Assuming the interlocking of the Lara sulphides is similar in fineness and complexity to the Westmin ores, expected grades are 40 - 50% Pb, 20 - 25% Cu and 45 - 40% Zn. These low grades combined with the relatively small quantity of concentrate produced will restrict marketing options.

Capital and operating cost estimates were developed for 200 t/day and 500 t/day operations.

	200 t/day	500 t/day
Capital cost $	8,682,000	11,792,000
Operating cost $/ton	14.71	9.98

Tailings Disposal

There are no ideal sites. There are two suitable sites both of which would provide substantially more than 5 years of storage. No obvious flaws are apparent.

Viability

The 500 and 200 tonne/day cases were examined to estimate the grade of ore which would give a zero return at a 10% discount rate:

- o 500 tonne/day = $134/tonne 'head grade' 122/t
- o 200 tonne/day = $173/tonne 'head grade'

It is not clear whether these reserve grades are available within the 'mineable' tonnages used in this study. There is 'room for' the tonnage, the concern is whether the grades can be improved within the mineable areas. This can only be achieved with much closer spaced drilling. It is interesting to note that the Lynx mine, which Lara is said to resemble, very seldom has more than one to two years of reserves and is essentially a very conventional vein mining exercise. Suggesting that even close spaced drilling might not produce the required reserves for a production decision, even though the reserves may actually be present.

The use of a cut-and-fill method may be basically incorrect as it assumes a continuity of reserve which may not be available. If continuity is absent it is unlikely that a 500 tonne per day operation will be feasible.

2. ORE RESERVE

2.1 Mineable Reserve

The ore occurrence is characterized by very high grade sulphide zones surrounded by areas of much lower grade. The continuity and extent of the economic grades cannot be evaluated at the present drill spacing except in very localized areas. Indications are that the high grade sulphide zones may be limited in extent and the grade could vary considerably over fairly short distances.

The 'stoping' areas used in this evaluation are shown in the attached sketches, Figure 1. An arbitrary C$ 65 cut-off at 2 m thickness has been used. There are some very high grade occurrences within the zones but these have not been taken into account as continuity is unclear. The contours are based on 1986 metal prices.

The table below is a very approximate estimate of mineable reserves.

<u>Mineable Reserve Summary</u>

Case: 500 tonne/day

Mining Area	Reserve Tonnage	Reserve Grade C $/tonne	Assumptions: 3 m min. mining width
A + B	119,000	136	
C	134,000	105	
D	108,000	149	
E	<u>104,000</u>	<u>89</u>	
	465,000	120	

Dilution for cut and fill operation: 6 % sandfill
10% waste

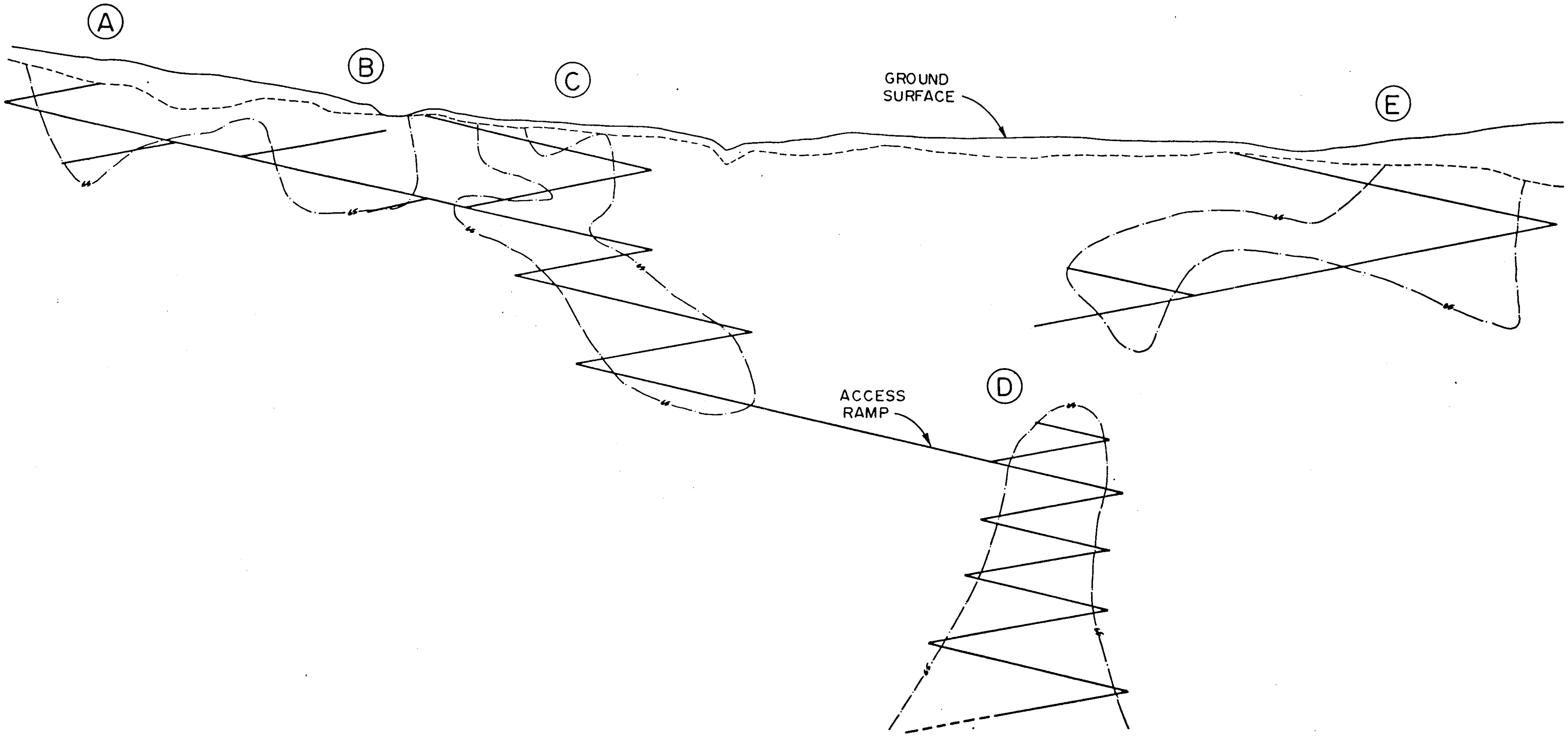
Extraction	= 554,000 tonnes
Mined Grade	= $101/tonne
1987 Prices	= $123/tonne
	= $99/tonne at 80% concentrator recovery

Case: 200 tonne/day

Mining Area	Reserve Tonnage	Reserve Grade C $/tonne	Adjustment from 500 tonne case: 2 m min. mining width 30% left in pillars in 'wide'areas; 10% in narrow areas
A + B	83,000	150	
C	94,000	116	
D	81,000	179	
E	<u>68,000</u>	<u>122</u>	
	326,400	142	

Dilution: 10 % waste

Extraction	= 359,000 tonnes
Mined Grade	= $128/tonne
1987 Prices	= $125 at 80% core recovery



ABERMIN CORPORATION	DATE SEPT. 1987
'MINEABLE' RESERVES WITHIN $65 CUT-OFF VALUE; WITH RAMP ACCESS SHOWN	PROJ. NO. 64301
	APPROVED
	NO. 1
STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers	

It is concluded that these estimates may be pessimistic regarding grade but optimistic with respect to tonnage. The reserve summary can be compared with the latest Abermin estimates of C $192/tonne and 175,000 tonnes including 20% dilution and 1987 metal prices. The Abermin reserve estimate is based on a polygon assessment which assumes that high grade zones have the same continuity as low grade. The major difference in the present assessment is the use of a 3 m minimum mining width whilst the reserve value contours were developed with a 2 m minimum mining width. The average dip of the orebody is approximately 60° and it would be very difficult to use a mechanized method at a 2 m limit. The 3 m limit also makes allowance for the local variation in dip and thickness which will, no doubt, be encountered.

It should be noted that the object of re-evaluating reserves was for the purpose of outlining mineable reserves to estimate the developmant required.

It is noticeable that there is very little 'intermediate' grade and that intersections tend to be 'high' or 'low' within a fairly continuous ore 'zone'. Distribution and continuity are very uncertain.

2.2 Geotechnical

Six 'representative' drill cores were logged approximately 30 m into the footwall, through the ore zone and 10 m into the hanging wall. The outcrop in the two trenches was also examined.

The overriding structural control is foliation. Jointing is present but it tends to be steeply dipping with little infill and joint surfaces are rough and irregular.

The foliation tends to be sub-parallel to the dip but can vary considerably. The foliation surfaces are often striated and polished and frequently contain sericite which is a soft low friction infilling. The degree of foliation varies and where it is more frequent the rock mass is significantly weaker and easily 'crumbles' into thin slivers. It would appear that the frequency of foliation is a function of disturbance and the presence of faulting. Where the rock mass is more foliated there are more frequent zones of shearing and weak infilling. Therefore the RQD values which have been recorded should be a good relative indication of rock mass competency. The deeper more eastern holes seem to be significantly less competent than the shallower more western holes. The difference in foliation intensity is illustrated in the photographs of holes 139 and 187, Figure 2.

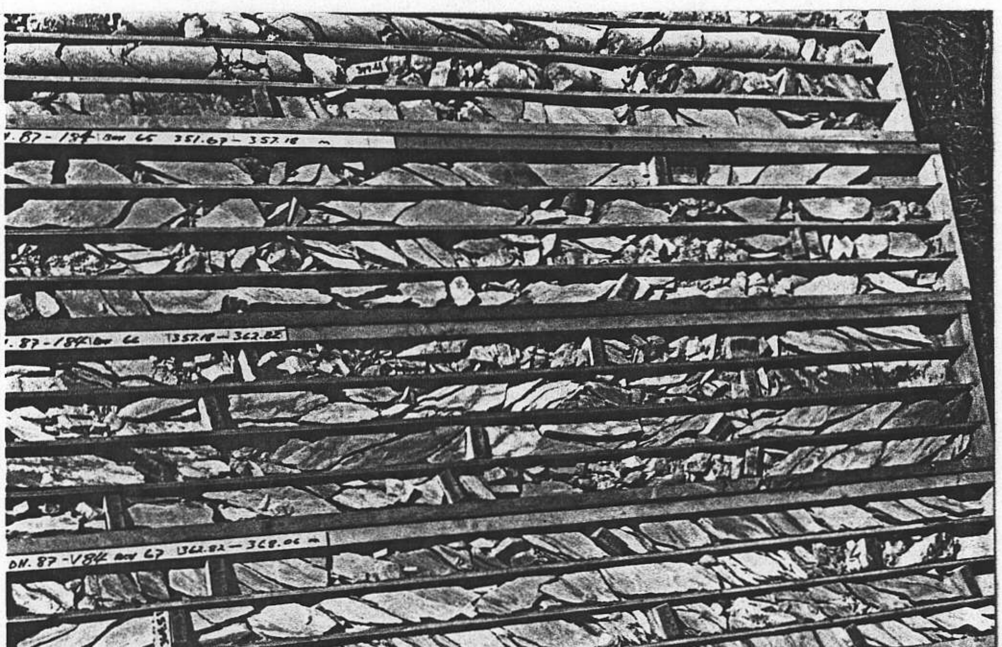
The foliation and jointing is steeply dipping and therefore wall control will be the most critical and cross-cuts will be far more stable than drifts. The footwall side of the drifts may be less stable than the hanging wall side.

The Rock Mass Rating (RMR) varies from an extreme low of 10 to a high of 46 (out of 100) with the median of results tending to be above 35. The footwall tends to be the most competent. Adjusted RMR to take into account the different mining directions brings the values for drifts down to an average of 25. The significance of this value is illustrated in Figure 3 with a 'hydraulic radius' of 2, or a stable drift of 4 m width down to a depth of 750 m (Rock Mass Strength = 70 x (35-8)/80 x .8) = 19 MPa). There is possibly more than 20% of the footwall rocks which would require fairly intensive support (bolts/shotcrete) and the rest, precautionary bolt support.

The orebody excavations (cut-and-fill) will be temporary but, dependent on orebody extent, may produce sufficient stress to fail the rock mass. There are few 'stress release' features to accept the strain and therefore some 50% of the orebody may require bolts and mesh as a temporary support measure. Unplanned dilution may occur in the footwall where the excavation is mined to a rectangular shape.

A proportion of the orebody may present difficulties - say 20% - requiring more intense support.

HOLE 139



HOLE 184

FIGURE 2 ILLUSTRATION OF VARIATION IN FOLIATION

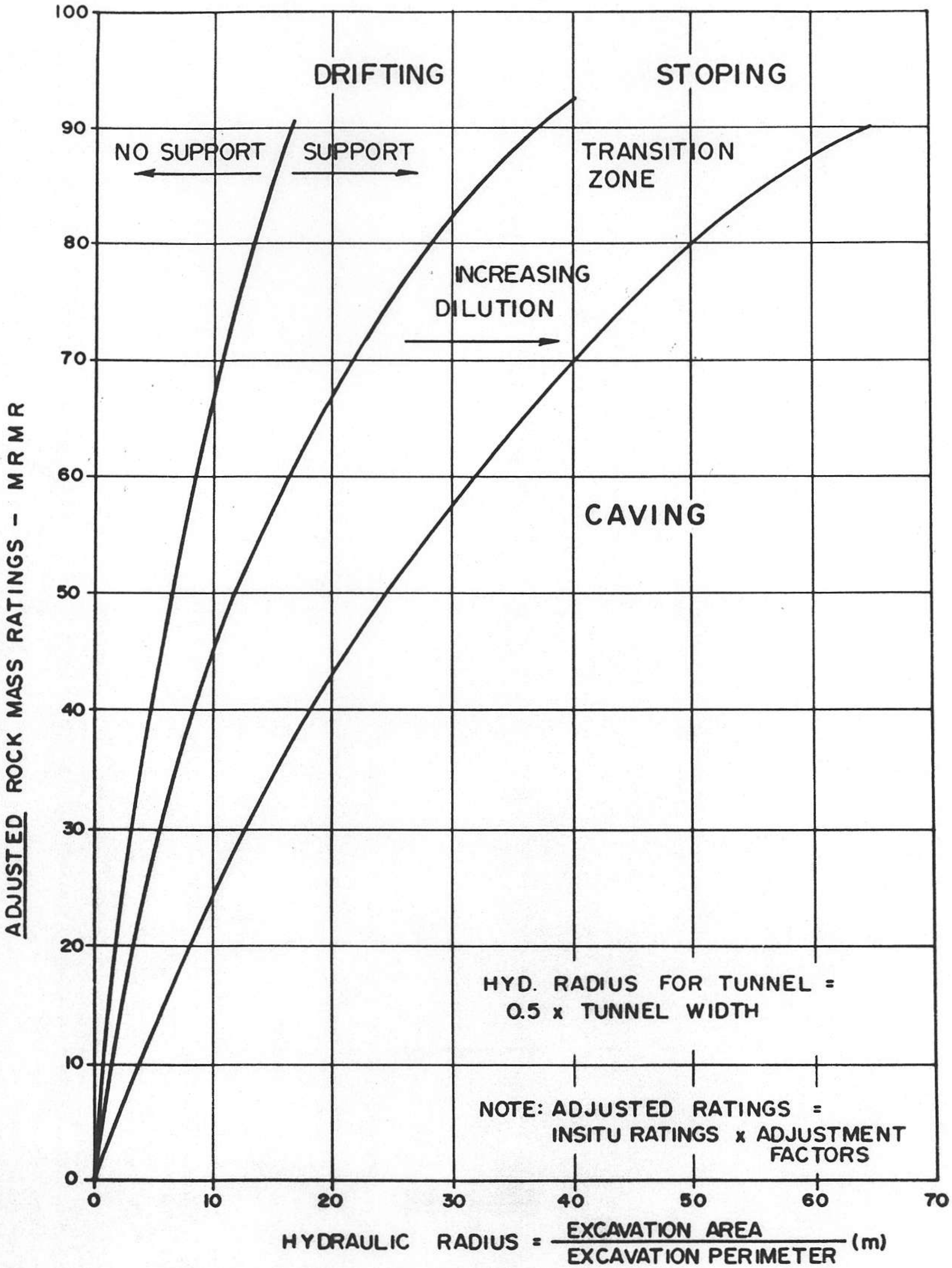


FIGURE 3 ESTIMATE OF EXCAVATION STABILITY

3. MINING

3.1 Methods

The ore is irregular and relatively thin. The following methods may be applicable:

- open stoping
- shrinkage
- cut-and-fill

The indicated irregularity of the grade precludes the use of long hole, remote mining methods unless a high proportion of sub-grade material is included in the mineable reserve. There is some doubt regarding the stability of the hanging wall over significant spans where foliation is more frequent.

Although better selectivity could be achieved with shrinkage, it is likely that the walls of the stoping excavation would slough to a more regular shape and high dilution could result.

The logical method for a pre-feasibility evaluation is cut-and-fill. Approximately 75% of the plant feed will go to tailings in the first stage of concentration with 55% at minus 200 mesh. A sufficient quantity of free draining fill should be available. Underground waste will also be used as backfill.

3.2 Case 1: 500 tonne/day

The 500 tonne per day scenario has been based on a mechanized, ramp access method with access to every mining cut.

The equipment list is as follows:

	Cost per Unit
. 3 x 15 ton Jarvis Clark JDT-415 trucks	157,000
. 2 x 2.2 yd Jarvis Clark JDT-220 (or 3 1/2 yd.) LHD's	135,000
. 2 x 1 boom hydraulic jumbo Maclean Eng. MEM-827	175,000
. 5 x service vehicles	25,000
. ventilation - estimate of total	400,000
. electrical - estimate of total	400,000
. pumping - (dirty water, submersible)	65,000
. grader	75,000
Total equipment capital	$2,156,000

The ramp has been evaluated at 20% which is relatively steep for Canada but for which 4 wheel drive trucks are suitable if good footwall conditions are maintained. A more conventional 15% would add 30% to the costs of the ramp and since the tonnage per metre developed is fairly small it does not warrant the extra capital cost to reduce the operating costs. The ramp size is 4.0 m by 3.5 m and has been costed at $1000/m ($71 per cubic metre). Access to the stopes is 3.0 m by 3.0 m and gives access to three slices by slashing the roof. An appropriate mining cost of $\$50/m^3$ has been used.

Pre-Production Capital Development

Pre-production development costs to bring into production A, B and the top part of C:

.	120 m vertical lift of ramp and raise	$1,062,000
.	construction piping, cable, etc. at 30% of mining cost	$ 320,000
	TOTAL	$1,382,000

Operating Costs

Productivity and operating costs have been assessed in two ways; from first principles and by using a recent, detailed study of cut-and-fill in Manitoba:

- 14.3 tonnes/manshift; 500 tonne/day case, all employed underground
- operating costs:

- equipment	$4.0/tonne
- materials	$5.2/tonne
- backfill	$1.0/tonne
- development	$8.6/tonne
- labour	$12.9/tonne ($23/hr. overall)
- power	$0.8/tonne
	$32.5/tonne

The 'development' has been costed and scoped as a separate item and is not included in the equipment and productivity evaluation as it is based on competitive contractor rates.

Costs apply to a mining depth of 160 m.

3.3 Case 2: 200 tonne/day

The 200 tonne per day evaluation is based on some form of shrinkage/open stoping with pillars and much less mechanization than for case 1. It is included as an estimate of variation should a lower tonnage option be necessary. The viability of the concept is not certain and the evaluation of the method is far less accurate than case 1. Ramp cross-section has been reduced to 3.5 by 3.0 m at a mining cost of $825/m ($78.6/m3).

- equipment cost - $1,411,000
- pre-production capital - $773,000
- operating cost $43.35/tonne (productivity 7.14 tonnes/man shift)

4. TAILINGS DISPOSAL

4.1 General

Tailings disposal for the Lara Project will likely consist of a land-based wet disposal system. An embankment will be constructed at an appropriate location using primarily soil and rock obtained from local sources. Slurried tailings will be piped to the impoundment and discharged behind this embankment.

Excess water associated with the slurried tailings will collect on the surface at the impounded tailings and be pumped back to the mill for recycling. In the event that most of this water cannot be recycled, the impoundment will gradually collect water which will have to be stored or treated, if necessary, and discharged to the environment downstream of the tailings impoundment.

A seepage return structure downstream of the tailings impoundment will collect water which seeps through the embankment. Depending on the quality of this seepage, it will either be pumped back into the tailings impoundment or treated, if necessary, and discharged into the natural channels downstream of the seepage return structure.

When the mine finally closes, the tailings impoundment will be reclaimed to a condition which is consistent with the surrounding terrain.

4.2 Design Parameters

- 500 tonnes of tailings per day for 350 days per year at a settled dry density of 1.4 tonnes per cubic metre (90 pcf): 125,000 cubic metres of tailings per year

- Two mine-life scenarios:

    - 1.5 years; 187,500 cubic metres of tailings to be stored
    - 5.0 years; 625,000 cubic metres of tailings to be stored

- Tailings are confirmed non-acid producers based on results of net acid/base accounting test and biological test carried out by Coastech Research Inc. (Further testing to confirm the non acid generating potential of the tailings is probably warranted.)

- Mill process apparently will utilize cyanide but will have a cyanide destruct circuit; process will be the same as that used by Westmin at their mine near Buttle Lake

    - multi-stage circuit
    - first stage: grind to 60% passing No. 200 sieve, froth flotation, discharge a portion of the tailings
    - second stage: regrind to 95% passing No. 200 sieve, froth flotation with cyanide, cyanide destruct, discharge remaining tailings

4.3 Site Selection

- Site selection process was carried out using topographic maps and a one-day site reconnaissance

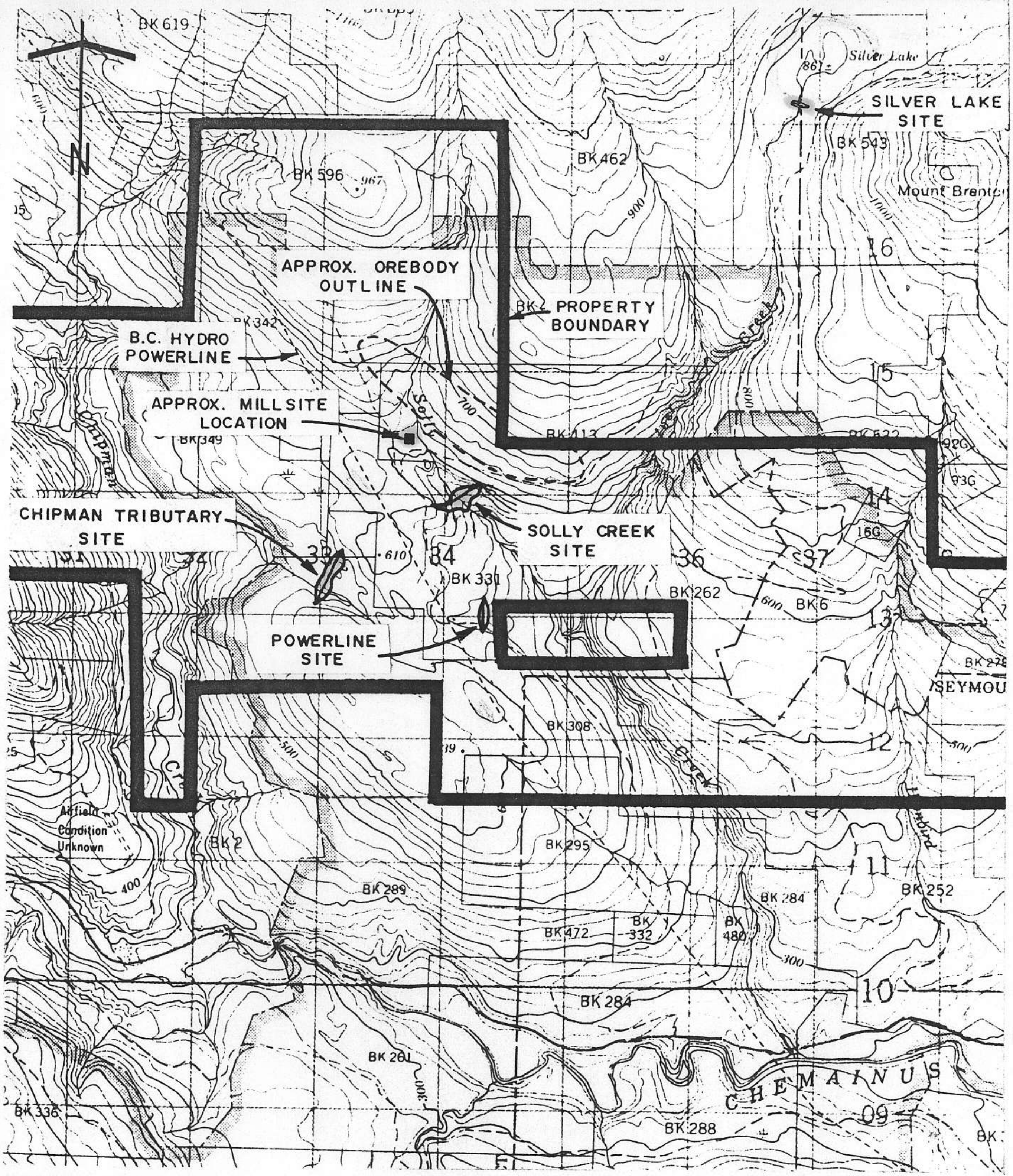
- As shown on Figure 4, four sites were identified as potential tailings disposal sites on the basis of topographic features. The four sites which provide reasonably efficient storage characteristics are labelled as follows:

    - Solly Creek Site: situated 0.5 km southwest of the proposed mill site (el. 600 m)

    - Chipman Tributary Site: 1.0 km southwest of the proposed mill site

    - Powerline Site: 1.2 km south of the proposed mill site

    - Silver Lake Site: 4.7 km northwest of the proposed mill site



SCALE - 1:50 000

ABERMIN CORPORATION	DATE SEPT. 1987
GENERAL SITE LAYOUT	PROJ. NO. 64301
	APPROVED
	NO. 4
STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers	

4.3.1 Site Descriptions

<u>Solly Creek Site</u>

- Impoundment is centred about Solly Creek.
- Solly Creek flows into the Chemainus River.
- Uniform, gentle gradient (4°) in creek channel (Figure 5).
- Slopes on either side are typically about 5 to 9°; a steep (32°) segment about 5 m high occurs near the west edge of the creek along the proposed embankment axis.
- The site is covered by sparse to dense second growth evergreens of two ages; one about 10 to 20 years old and one is much older.

<u>Chipman Tributary Site</u>

- Impoundment is centred about a double valley in which intermittent streams are believed to flow (Figure 5).
- These streams are tributary to Chipman Creek which, in turn, flows into the Chemainus River.
- Gradients in the double valley are about 2 to 3°.
- Slopes on either side of the valley and in the vicinity of the ridge which separates the two channels are about 0 to 11°.
- The site is covered by relatively mature second growth.

<u>Powerline Site</u>

- Impoundment is situated in a gully which encroaches on a B.C. Hydro powerline right-of-way (Figure 5).
- Water is believed to flow in the channel intermittently.
- The gradient in the channel is about 2°.
- The side slopes vary but are about 6 to 10° at the optimum embankment location.
- The powerline right-of-way is cleared; reasonably mature second growth occupies the area east of the powerline right-of-way.

<u>Silver Lake Site</u>

- Impoundment is situated in Silver Lake at approximately el. 860 m; requires in-lake disposal.
- Silver Lake flows into Silver Creek which, in turn, flows into Solly Creek.

- Lake is believed to be shallow.

- Reconnaissance of Silver Lake not done during 1-day site reconnaissance.

4.4 Results of Site Selection

- Both the Solly Creek Site and the Chipman Tributary Site are suitable for disposal of tailings and have the capacity to store tailings from a mill operating at a rate of 500 tpd for 5 or more years.

- The Powerline Site, as drawn on Figure 5, has the capacity to store only 96,000 cubic metres of tailings (0.8 years of production). Even at this volumetric capacity, the impounded tailings will encroach on the B.C. Hydro powerline right-of-way unless otherwise confined by an embankment. Therefore, this site has been eliminated from further consideration as a tailings disposal site.

- The Silver Lake Site is relatively distant, on land not controlled by Abermin Corporation and at a much higher elevation than the mill site. In addition the lake is apparently relatively shallow. As a result, the capital costs are likely to be significantly less expensive than at the other sites but, unlike the other sites, the operating costs are likely to be extremely high. In total, this site appears to offer certain cost advantages. However, in-lake disposal is a very controversial technique that may well generate more public opposition than any of the other sites. In addition, the land at Silver Lake is not controlled by the Abermin Corporation. Therefore, the Silver Lake site has been eliminated from further consideration as a tailings disposal site.

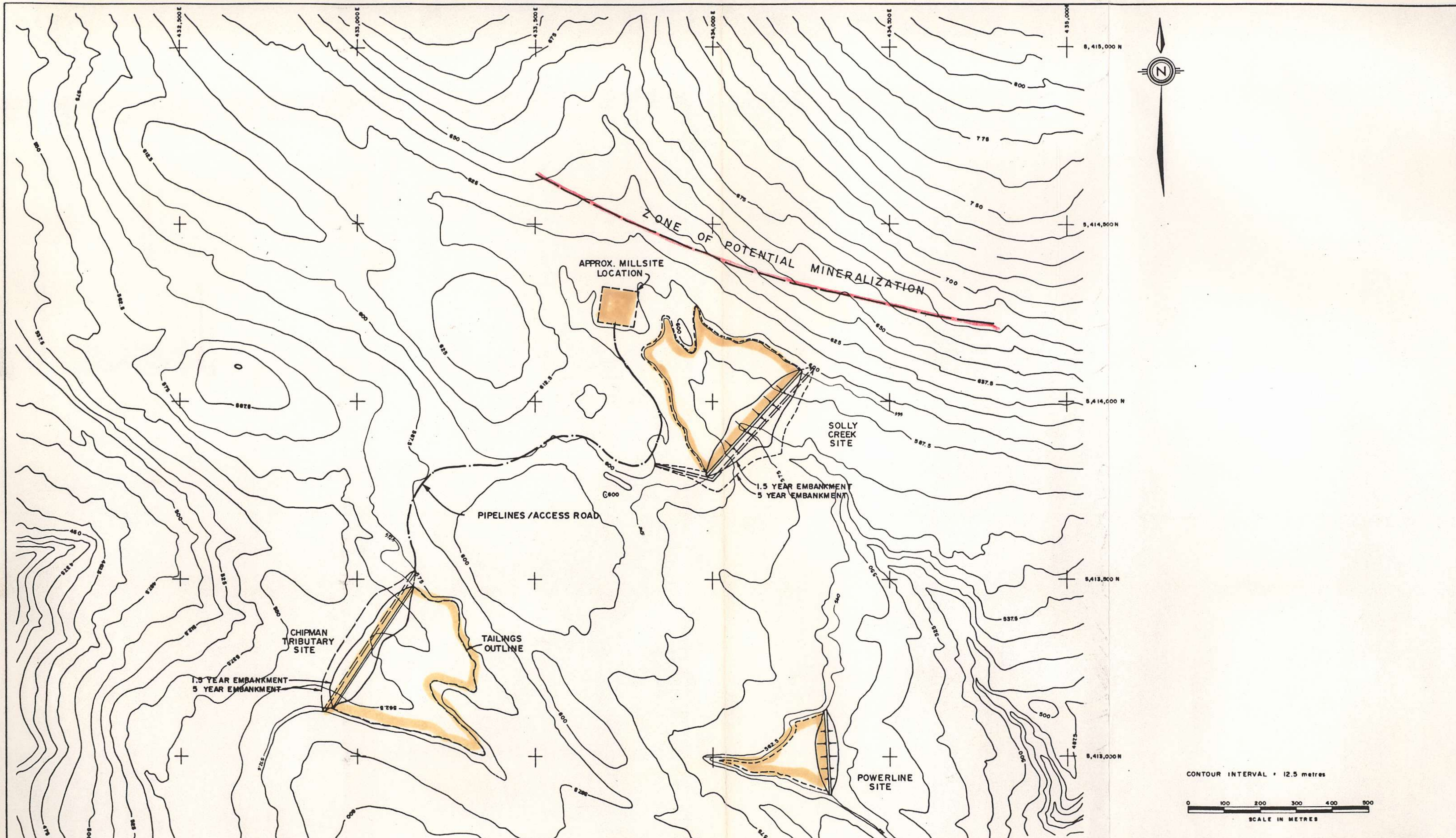
The following comparison has been made between the Solly Creek Site and the Chipman Tributary site:

	Solly Creek Site	Chipman Tributary Site
1.5-yr. embankment volume	190,000 m^3	178,000 m^3
1.5-yr. crest elevation	595 m	570.5 m
1.5-yr. embankment height	21 m	14 m
5-yr. embankment volume	448,000 m^3	378,000 m^3
5-yr. crest elevation	600 m	575 m
5-yr. embankment height	26 m	19 m
Catchment area	620 ha + 30%?	60 ha

- In terms of smaller volumetric capacity (for 1.5 years of storage), the two sites are very similar. However, for larger volumetric capacities, the Chipman Tributary site has substantially better storage characteristics.

- The drainage catchment areas for the two sites are very different, which means substantially different water management concepts for each of the two sites. It should be noted that essentially no water management studies have been carried out for this aspect of the project.

- The Solly Creek Site has an extremely large catchment; the Chipman Tributary Site, a relatively small catchment.



ABERMIN CORPORATION

POTENTIAL TAILINGS SITES

STEFFEN ROBERTSON & KIRSTEN, Consulting Engineers

DATE	SEPT. 1987
PROJ. NO	64301
APPROVED	
NO.	5

- Based on capital cost, considerations for an operation approaching 5 years, the best site will be the Chipman tributary Site. However, the Solly Creek Site will be cheaper in terms of operating costs. In terms of combined capital and operating costs, the two sites are relatively similar, with a slight edge going to the Chipman Tributary site. In addition, water management will be much easier at the Chipman Tributary Site.

  Order-of-magnitude capital cost and operating cost estimates have been prepared for the construction and operation of a tailings impoundment at the Chipman Tributary Site.

- Final site selection should be based on more site information and should incorporate environmental issues such as water quality, fisheries impacts and a better understanding of the mill process and the acid generation potential of the tailings.

4.5 Basis of Cost Estimates

- Order-of-magnitude cost estimates have been prepared on the basis of the following assumptions concerning design, layout and operation of the tailings disposal facilities at the Chipman Tributary Site.

- An earthfill embankment will be constructed to impound tailings produced from an operation which runs 3.2 years.

- The embankment will be homogenous embankment constructed mainly of local till and some quarried rock (for drains).

- A spillway will be constructed in one abutment to pass the major floods.

- Diversion ditches will be constructed to minimize the flow of water into the impoundment.

- An embankment, referred to as a seepage return dam, and spillway will be constructed downstream of the tailings impoundment to collect seepage so it can be pumped back into the tailings impoundment.

- Tailings will be pumped (to overcome friction) to the tailings impoundment.

- A barge and pump will be floated on the pond at the tailings impoundment.

- A return water line will run from the barge/pump to the mill and will be used to pump water (254 gpm) back to the mill.

- Fresh makeup water (approx. 14 gpm) will be extracted from Solly Creek.

The order-of-magnitude costs for the tailings impoundment based on these items are shown in Tables 1A and 1B.

4.6 Summary

- Based on a brief site selection study, four potential sites for tailings disposal were identified. Two were rejected, and after comparing the remaining two sites, the Chipman Tributary Site 1.2 km, southwest of the proposed mill site was identified as the most economic site, by a slight margin.

- The tailings will be pumped in a slurry pipeline and discharged behind an earthfill embankment.
- Diversion catches will minimize the amount of flow into the tailings impoundment. A spillway will pass major flood events.
- Water will be recycled to the mill in sufficiently large quantities that only a nominal amount of make-up water will be required. Makeup water will be obtained from Solly Creek.
- A seepage return structure downstream of the tailings impoundment will be used to catch seepage which will then be pumped back into the impoundment or treated, if necessary, and discharged.
- Order-of-magnitude cost estimates have been prepared for the Chipman Tributary site based on the design concepts identified about and production rate of 500 tonnes per day for a mine life of 3.2 years.
- The order-of-magnitude capital cost estimate and operation cost estimate are $1.76 million and $0.47 per tonne, respectively.

Table 1A

Order-of-Magnitude Capital Costs

Embankment	275,000 m^3	\$5/m^3	\$1,375,000
Clearing	20 ha	5000/ha	100,000
Drains	1500 m^3	20/m^3	30,000
Pipeline	1500 m	30/m	45,000
Access Road	1500 m	30/m	45,000
Reclaim Water Line	1500 m	20/m	30,000
Barge & Pump	one	10,000	10,000
Spillway	2000 m	8/m^3	16,000
Diversion Ditches	(1600 x 1.5x2)m^3	5/m^3	24,000
Seepage Return Structure (incl. pumps & hose)	10,000 m^3	8/m^3	80,000
TOTAL			\$1,755,000

Table 1B

Order-of-Magnitude Operating Costs

Tailings pumps & lines (maintenance, supervision, power)	\$0.20/tonne
Return Water Pumping (Maintenance, Supervision, power)	0.20/tonne
Seepage Water Pumping (Maintenance, supervision, power)	0.07/tonne
TOTAL	\$0.47/tonne

Reclamation costs not included.

PRELIMINARY STUDY FOR THE LARA PROCESS PLANT AND ANCILLARY SURFACE FACILITIES

SUBMITTED BY:

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September 1987

PRELIMINARY STUDY FOR THE

LARA PROCESS PLANT

AND ANCILLARY SURFACE FACILITIES

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1.0 INTRODUCTION

1.0 INTRODUCTION

Abermin Corporation required a brief overview of the Lara property on Vancouver Island to assist them in planning their future strategy for development of the site. Proton Systems Ltd. (Proton) were retained to estimate the requirements of the process plant and all other plant site services and facilities with the exception of the tailings pond.

The Lara property is located approximately 13.5 km WSW of the town of Chemainus. The terrain is generally rugged but a relatively level area, at approximately 600 m elevation is available for surface facilities. A 250 KV B.C. Hydro power line passes the site a few hundred metres to the west. Drainage from the property is in a southerly direction into the Chemainus River, which is a major salmon spawning waterway.

Overall the mineralogy has been compared to that of Westmin Resources Limited, Lynx mine. Mineralization consists of massive to desseminated sulphides. Sphalerite is the most common economic sulphide with chalcopyrite, galena and tetrahedrite present in lesser quantities. Gold and Silver are present. Intergrowths and inclusions of one mineral with another are common.

Proton was requested to review the metallurgical testwork and develop a conceptual plant for treatment of the material. The design of the 500 t/day plant was used to derive capital and operating costs. Subsequently, Proton was asked to also provide capital and operating costs for a 200 t/day plant.

2.0 SUMMARY

2.0 SUMMARY

The Lara sulphides are associated intimately with considerable intergrowth of one mineral in another. Very fine grinding is necessary to complete liberation, though a barren tailing can be produced at a coarser grind (55% - 200 mesh).

Further testwork is required to determine grades of the final concentrates. Without having achieved saleable final grades, precise estimates of metal recoveries are not possible. Based on the information available, recoveries are predicted to be:

Gold	85 - 90%	overall
Silver	80 - 90%	overall
Lead	80 - 85%	in lead concentrate
Copper	75 - 85%	in copper concentrate
Zinc	80 - 85%	in zinc concentrate

Concentrate grades are likely to be low. Assuming the interlocking of the Lara sulphides is similar in fineness and complexity to the Westmin ores, expected grades are 40 - 50% Pb, 20 - 25% Cu and 45 - 40% Zn. These low grades combined with the relatively small quantity of concentrate produced will restrict marketing options.

Capital and operating cost estimates were developed for 200 t/day and 500 t/day operations.

	200 t/day	500 t/day
Capital cost $	8,682,000	11,792,000
Operating cost $/ton	14.71	9.98

3.0 METALLURGY

3.0 METALLURGY

3.1 Status Of Testwork

Preliminary metallurgical testwork has been done by Coastech Research Inc. Three tests at different grinds on each of the two separate samples of significantly different head grade demonstrated that flotation of the sulphides can be achieved. Two further tests were run to obtain flotation rate information, while the remaining four tests were run with modified reagent levels. The testwork was conceived for the development of a plant to produce separate copper, lead and zinc concentrates. However, with the exception of a zinc concentrate in one test, no material was produced which came close to saleable grade.

Several of the preliminary tests used different grinds, either for the primary grind or reqrind. Size determination of the products, which were not reported, would permit better analysis of the practicality of achieving three saleable concentrates.

The process simulated had several intermediate products. All tests were performed in open circuit tests without recirculation of these intermediate products.

The test results indicate that the silver and gold reports with the copper. Precious metal recoveries were excellent (overall in the order of 95%) but concentrate selectivity was insufficient to be certain that no significant precious metal content was associated with the lead or zinc sulphides. Certainly this hypothesis is contradicted by the microscopic inclusions in sphalerite while silver was detected in galena by means of quantitative electronic microprobe analysis.

3.2 Metallurgical Projections

Precise predictions of metal recoveries are not possible at this stage. In the development of a process for plant cost estimating, the recoveries of copper, lead and zinc in their respective concentrates were assumed in the range 80% - 90%. Overall gold and silver recoveries can be estimated with even less certainty, though 80% for both would be conservative. (If gold recovery were less than 80%, economics would determine a different process e.g. sale of a bulk concentrate or leaching of a flotation concentrate).

Assuming the metallurgy at Lara can be developed to a similar level as Westmin the following tables would represent an optimistic metallurgical balance. Lead and zinc concentrate grades will be low, with the former in particular containing significant levels of the other base metals.

ASSAYS

	Au oz/ton	Ag oz/ton	% Cu	% Pb	% Zn
Feed	0.12	3.00	1.11	0.67	5.69
Pb conc.	2.46	86.07	7.3	45.0	23.3
Cu conc.	1.48	37.09	25.0	1.3	6.3
Zn conc.	0.25	3.10	0.6	0.5	50.0
Tails	0.01	0.35	0.06	0.05	0.4

Percent Distribution

	Au	Ag	Cu	Pb	Zn
Feed	100	100	100	100	100
Pb conc.	25	35	8	82	5
Cu. conc.	45	45	82	7	4
Zn. conc.	20	10	5	7	85
Tails	10	10	5	6	6

Factors which would mitigate against such a balance include:

- Head grade significantly different.
- Intergrowths of one mineral in another too fine for separation without causing excessive sliming or economic grinding.

At present metal prices the gold and zinc represent the two major values in the ore. Because the zinc concentrate is likely to be lean in precious metals, the degree of freedom for optimizing metallurgy may be limited. Ideal conditions for one metal may be counter-productive with regard to the other.

3.3 Recommendations For Future Testwork

It is recommended that further testwork be conducted aimed at defining metallurgical performance. Areas which require work are:

- Copper-lead separation from bulk concentrate
- Cleaning of copper, lead and zinc concentrates

- Determination of optimum size for the primary grind to enable the discharge of a final tailing.
- Determination of optimum size after regrinding
- Locked cycle tests to evaluate into which final products the minerals in the intermediate streams will collect.
- Reagent optimization tests.
- Determination of ore grindability.

Further testwork would be required for detailed engineering. These tests may be more efficiently done concurrently with the other testwork but are not essential for making an investment decision. Such tests would include:

- Settling characteristics of each concentrate.
- Filtration characteristics of each concentrate.
- Complete chemical analysis of each concentrate to complete smelter contract negotiations.

4.0 PROCESS PLANT

4.0 PROCESS PLANT

Though the size of the equipment would be smaller for a 200 t/day plant, the number of separate process steps is expected to be the same for a 500 t/day and a 200 t/day plant.

Run of mine ore would be crushed in a jaw crusher. The crusher discharge would feed a screen located on top of the fine ore bin. Screen oversize would be further crushed in a cone crusher which would discharge onto the same conveyor belt system feeding the screen. Dust collection would be provided at the crushers and all transfer points but the conveyor galleries and crushers would not be enclosed.

The crushing facilities would operate on the same schedule as the mine i.e. two shifts per day, 5 days per week. The concentrator would operate continuously, seven days per week.

Grinding to 55% passing 200 mesh would be achieved using a single ball mill. The plant was conceived as using flotation to produce a bulk copper-lead rougher concentrate. This concentrate would be reground and subjected to differential flotation to produce separate lead and copper rougher concentrates. These concentrates would be cleaned by flotation (5 stages for the lead and 2 stages for the copper) and fed to separate stock tanks. A single automatic belt press would be used to dewater both materials with discharge into separate stockpile storage.

The tailings from the bulk copper-lead float and the tailings from the copper flotation would be floated separately to produce zinc concentrates. These concentrates would be further cleaned and then dewatered in a second automatic belt press.

The grinding, flotation and dewatering circuits would be housed in a simple rectangular building. The stock tanks would be located outside the building.

A modest level of instrumentation was assumed for plant control. In view of the complexity of the metallurgical separation, an on-stream analyzer could be necessary for optimum results. The capital cost estimate makes no provision for such equipment.

An allowance has been included for a backfill plant based on a detailed estimate for a similar project. Detailed information regarding the mining schedule would be required to develop a more precise cost estimate.

5.0 UTILITIES

5.0 UTILITIES

5.1 Water Supply

With no recycling, an estimated 72 m^3/h of process water would be required for a 500 t/day plant. Westmin recycle an estimated 20-30% of their process water. and, assuming a similar percent recycle, the process water requirement could be reduced to 54 m^3/hr. A 200 t/day plant would required 40% of these quantities though the preference is to avoid recycling in smaller plants. To obtain additional water reuse would be expensive since a tailings effluent storage area sufficient for several months retention would be required. Depending on the watershed associated with such a pond, such a system would permit recycle of up to 95% of total requirements.

For this exercise the assumption was made that sufficient water for the plant would be available year round from Solly Creek.

5.2 Power Supply

B.C. Hydro require further information from Abermin before they will commit to a 25 KV mine power line can be run along their existing right-of-way. Our understanding of the location of the Sahtlam substation indicates the length of the new line from the substation to the minesite would be 11 km rather than 13.5 km as indicated in earlier communications. A brief study clearly showed that even for a 200 t/day operation, the use of power generators would be uneconomic compared with paying an estimated $550,000 for a new power line and purchasing power from Hydro.

Consequently both cost estimates are based on a new 11 km line along the B.C. Hydro right-of-way, but built on separate power poles.

Power required for the process and backfill plants has been estimated at 700 KW for a 500 t/day plant and 390 KW for a 200 t/day plant. For the 500 t/day plant the secondary crusher and ball mills would require 150 KW and 190 KW respectively, but all motors would be 56 KW or smaller. The ball mill for a 200 t/day mill would require 75 KW and the crusher 56 KW.

5.3 Ancillary Surface Facilities

A single building was assumed for the office, warehouse, maintenance shop and mine dry. For a 500 t/day operation, a mine dry sized to handle 100 persons was assumed. This resulted in the complete building having a floor area of 450 m^2. No specific building size was developed for the lower operating rate but the probable building size is of the order of 300 m^2. An assay laboratory and combined first aid/gatehouse were assumed to be housed in pre-fab trailers.

6.0 CAPITAL COST ESTIMATES

6.0 CAPITAL COST ESTIMATES

6.1 Summary of Costs

	200 t/day	500t/day
Site, utilities and general	$1,430,000	$1,628,000
Crushing/mill building and structures	1,401,000	1,621,000
Process equipment and services	2,162,000	3,746,000
Ancillary facilities	770,000	1,100,000
Construction Indirects	1,104,000	1,227,000
EPCM and Contingency	1,819,000	2,470,000
Total	$8,682,000	$11,792,000

6.2 Basis of Costs

The cost estimates for the 200 t/day and 500 t/day process plants were not derived by the same method.

Equipment suitable for handling the larger throughput was selected and sized, and the building size for the process plant was derived from a layout of this equipment. An estimate of the cost of each major item of equipment and the components of the building was derived using data from other projects. The cost for the other surface facilities, power supply and water supply were also developed from experience of similar sized projects.

Costs were in summer 1987 dollars using a labour rate of $30.00 per hour. This hourly rate is the field rate for a job we are currently managing, based on a 60 hr. week.

The estimate for the 200 t/day operation was derived by factoring costs derived for the 500 t/day case. Thus the level of accuracy of the estimate for the smaller plant can not be greater than "order of magnitude" i.e. ± 35%.

7.0 OPERATING COST ESTIMATE

7.0 OPERATING COST ESTIMATE

7.1 Summary

	200 t/day		500 t/day	
	Dollars	$/ton	Dollars	$/ton
Labour & Supervision	598,000	8.54	797,000	4.55
Power	131,000	1.87	235,000	1.34
Grinding Media	52,000	0.74	129,000	0.74
Reagents	192,000	2.74	479,000	2.74
Operating Supplies & Repair Parts	57,000	0.81	107,000	0.61
	1,030,000	14.71	1,747,000	9.98

Annual ore production rates of 70,000 tons and 175,000 tons were assumed.

7.2 Personnel

The following table shows the manpower assumed to manage and operate the two different sized plants.

	Rate	200t/day	500t/day
Superintendent	60,000/yr.	1	1
Assayer	40,000/yr.	1	1
Crushing operators	13.50/hr.	2	2
Mill operators	13.50/hr.	8	12
Backfill plant o.p.	13.50/hr.	-	2
Maintenance	15.00/hr.	3	3
		15	21

These hourly rates are lower than those of Westmin Resources at the Lynx mine but are comparable with those currently at Mascot Gold Mines. A burden of 30% was applied to all rates for fringe benefits.

7.3 Power

A rate of $0.04/kw hr. was assumed for Hydro power. The estimated power requirements for the 200t/day and 500t/day process plants were 390 KW and 700 KW respectively.

7.4 Operating Supplies and Repair Parts

Because only preliminary metallurgical testwork has been completed, minimal testwork was done to optimize either the reagent selection or rates of consumption. In estimating the operating cost the assumed reagent levels were a compendium taken from several test runs rather than one specific test. Although Aerofloat 208 was used in most tests, the benefit was minimal at best and the assumption made was that Aerofloat 208 would not be used in a plant. All other reagents used in the testwork were assumed required. The consumption rates of reagents and grinding media are given below:

Potassium amyl zanthate	120 g/ton
Zinc sulphate	1000 g/ton
Sodium sulphite	200 g/ton
Sodium cyanide	200 g/ton
Lime	3000 g/ton
Soda ash	75 g/ton
Copper sulphate	600 g/ton
Frother	10 g/ton

Grinding media	1000 g/ton

Annual repair parts were estimated at 4% of the equipment capital cost.

8.0 MARKETING

8.0 MARKETING

Low concentrate grades, in particular for the lead and zinc concentrates, are to be expected from Lara. Consequently the selection of smelters willing to accept these products is limited. Since freight costs per unit of contained metal would be relatively high, a nearby smelter would be advantageous. For these reasons Lara would probably ship its lead and zinc concentrates to the Trail operations of Cominco.

Since the Tacoma smelter shut down, there is no copper smelter in the west coast of the U.S. or Canada. Ocean shipment to a smelter in Japan or further afield would not be particularly attractive because the small production rate would result in time periods of several months between shipments. Payments would be correspondingly intermittent. Consequently, Flin Flon could be the most likely destination.

Another possibility is an agreement with Westmin Resources. Provided the copper and precious metal grades were similar to or better than equivalent values in the copper concentrates being produced by Westmin, Westmin may be willing to ship Lara concentrate to Japan with their own concentrates.

9.0 ACKNOWLEDGEMENTS

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