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# 1979 REVISION OF CAPITAL AND OPERATING COSTS BEACONSFIELD MINE - ALLSTATE EXPLORATIONS N.L.

EL 17/73



20 March 1979

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## 1.0 INTRODUCTION

Mr. A. Silver in conjunction with Terence Willsteed and Associates have been requested by Allstate Explorations NL to review and update a "Report on the Feasibility of Reopening and Operating the Tasmania Gold Mine at Beaconsfield Tasmania". The date of that report is 1975.

The principal objective of this review is to revise Capital Operating Costs, and Mine Revenue taking cognisance of inflation since the report was prepared, and also increased gold prices.

The Company has requested that any major factors, such as current availability of equipment vis a vis the 1975 estimate be reported on, in so far as these factors influence the project viability.

## 2.0 REPORT SUMMARY AND CONCLUSIONS

The report reviews in summary form the geology, and the workings of the old Tasmania Gold Mine located at Beaconsfield, Tasmania.

The old mine was developed to the 1500 feet level, and ceased operations in 1914. Development on the lowest levels indicated gold values of 0.65 ozs per ton over 7 feet width.

Subsequent drilling by Mines Department and Allstate Explorations have given five intersections below the old workings. This has enabled indicated and possible mineralisation estimates to be made, in total 601,000 tonnes at 12.85 dwts of Au.

The mine is inundated to the shaft collar, and the first objective required is to dewater the mine by a system of pumps, to enable underground drilling to be undertaken.

Subsequent to the successful drilling of the mine, surface and underground installations would be made capable of processing 100,000 tonnes of ore per year.

The principal objective of this Report Revision is to update the Capital and Operating Costs for the Project.

This has been done on the basis of statistical indexes relating to labour and material costs, and in a number of cases, new quotations from suppliers.

The new estimate for Capital for the total project in 1979 dollars is now \$8,912,000. Allowing for inflation over a 2 year period, this would escalate to \$10,195,000.

The estimated Capital Cost for the first stage, that is dewatering, is \$2,161,000. This would on the present concept of purchasing new equipment for power supply and winders, reduce to \$1,761,000, by sale of equipment at the completion of the dewatering.

The dewatering phase, because of various unknowns on the water ingrěss to the mine, and shaft conditions is regarded as a high risk stage. It may overrun the estimate in both expenditure and time, and it can not be said with certainty that the proposal is achievable.

Assuming the mine does proceed to full development, then a revision of operating costs indicate a range of costs between \$30 - 35 per tonne milled, (1979 \$'s), at an annual production rate of 100,000 tonnes.

Accepting the gold values as postulated in indicated reserves and mineralisation, then the value of recovered gold per tonne of milled is \$113.60 on current gold price. To this is added some copper value (.83% Cu) to make a total of \$121 per tonne.

This would project a mine surplus on 1979 Costs and Gold prices of \$85 per tonne of ore treated. At an annual production rate of 100,000 tonnes, the mine cash flow would be \$8.5 million per year. It is considered that gold would remain untaxed. This would give a payback period of capital in approximately 1.2 years.

## 3.0 SUMMARY OF DATA - THE TASMANIA GOLD MINE

This report is to be read in conjunction with the two sections, Part A and B of the "Report on the Feasibility of Reopening and operating the Tasmania Gold Mine (August 1975).

### 3.1 Location

The Beaconsfield deposit in northern Tasmania is located adjacent to the Beaconsfield township, 39 kilometers north-west of Launceston, and 3 kilometers west of the Tamar Estuary.

### 3.2 Geology

The Tasmania reef is a gold bearing reef with quartz emplaced in a preexistant fault structure. The reef traverses Ordovician sediments, comprising of basal conglomerates, overlain by sandstones,and in turn by the Gordon limestones. The transition beds are approximately 400 metres thick.

The strike length of the reef is 400 metres, striking N 50$^{\circ}$ E, and the reef dips 50$^{\circ}$ to 60$^{\circ}$ to the S.East. Width varies from 1 metre to 8 metres, with occasional splitting into a hanging and footwall branch. The reef has been faulted in the upper horizons, with a fault reported on the 715 feet level. No subsequent faulting has been reported.

Gold quality is reported to have changed with depth, with the upper levels consisting mainly of free milling auriferous quartz. Changes occurred in mineralisation below the 400 foot level (120 m) with the presence of pyrite, chalcopyrite and increasing proportion of gold intimately associated with sulphides. (Reference 1/8 Geological Survey Report - Gee and Legge).

### 3.3 Mine Workings

Details on previous mine workings, grade, production statistics and processing are fully documented in the two sections of the 1975 Report.

It is proposed to briefly summarise for the purpose of this review the main aspects of the Mine. The mine closed in 1914, at which stage the 1500 feet level was partly developed to a length of 940 feet (285 metres).

Sampled ore values over the developed length of 940 feet of the 1500 feet level reported average of 13 dwts. (0.65 ozs) per ton, over an average width of 7 feet. Additionally a winze sunk 20 feet below the 1500' level reported a width of 5 feet of reef with an assay of 1 oz per ton. (Refer Plan 4/6).

## 3.4 Shafts

The mine was at the time of closure serviced by two shafts, the Harts Shaft (17' x 7') and Grubb Shaft (32' x 8'). Harts shaft served as the haulage shaft from the 1370 feet level to the surface. Harts' Shaft below the 1370 feet level had a winze connection to the 1500 feet level, and between these two horizons the winze cut the reef, at approximately 1470 feet, - the shaft from this horizon would be in the footwall of the lode. (Reference 1/5 Part B).

Grubb Shaft at the time of closure needed constant timber repairs. It had been collared in the Tertiary Deep lead sediments which extend to 500 feet below the collar, and although the shaft was satisfactory for pumping, it could not be used as a haulage and man shaft.

The present status of the Grubb Shaft collar is that there has been a complete surface collapse. The 1975 Report did not advocate any re-entry to mine workings from this shaft, and that view would persist today. At some later stage, the shaft will need to be permanently sealed off from workings.

The Harts shaft was completely serviceable at the time of closure. There has been minor collapse of the shaft collar, to depth of 15 feet below surface, with concrete masses from the collar blocking the shaft timber. The proximity of the collar collapse to the massive brick winder house walls is of some concern when any future works is undertaken. Pearson Bridge advocated shoring up on the winder walls, and driving sheet piling around the shaft collar with steel bracing for support. This may still be necessary.

In the concept of mine dewatering, as outlined in the 1975 Report as a preliminary stage to drilling and development of the mine, it was proposed that dewatering be attempted via Harts' Shaft. This concept has been persisted with in the present capital cost review.

### 3.5 Indicated Ore Reserves and Potential Mineralisation

The basis for the calculated "Indicated ore reserves" and possible mineralisation is detailed in Section 5 of the Feasibility Report. This has been done on the past records of the Tasmania Gold Mine, and the five drill intersections of the lode that have been made since the mine closure; the reserve calculations have been prepared by Mine Consultants, and verified by previous Allstate staff. As no further drilling has been performed subsequent to the previous report, these figures on reserves pertain to this review.

The data on reserves is extracted from the 1975 Report, with tons converted to metric tonnes. The figures quoted allow for a 20% dilution factor. (Refer section 5).

Indicated Ore Reserves -

224,000 tonnes at a grade of 14.46 dwts Au per tonne.

Possible Mineralisation -

377,000 tonnes at a grade of 11.9 dwts Au per tonne.

Combined Ore and Mineralisation -

601,000 tonnes at 12.85 dwts Au per tonne. (Weighted average).

### 3.6 Metallurgy - Process Design

Metallurgical testing on all the drill case intersections was conducted at the Launceston Laboratories of the Mines Department during 1974. A metallurgical Consultant was engaged to review the test work, to report on anticipated recoveries, and to design a mill flow sheet. On the basis of this flow sheet Sala Australia Ltd. submitted price estimates for the Mill covering supply and installation of equipment.

The mill flow sheet is fully detailed in Section 3 of the Part B Report.

The plant was specified at a 14 T.P.H. treatment rate or equivalent to 100,000 tonnes per year, after allowances (10%) for maintenance requirements and downtime.

In broad detail, the plant circuit involved coarse crushing, screening, and then two stage reduction with cone crushers, with a 500 tonne storage capacity bin for crushed ore.

Grinding was to be by ball mills, with jigs and for concentration of coarse gold. Jig tailings were to be treated by two stage flotation to remove some gangue minerals, and then followed by a flotation of a copper concentrate, carrying high gold values (approximately 10 ozs Au/tonne of concentrate).

The tailings from the copper concentrates were then floated for remaining sulphides, with a further cleaning stage, then the pyrite flotation concentrate was to be treated by direct cyanidation for gold recovery. Pregnant gold solution was then to be treated by precipitation, with conventional roasting and retorting of the gold from the calcine.

The flow sheet as developed in 1975 has not been modified, and it has been possible to go back to Sala Australia for a revision of their 1975 budget estimate. The plant has been re-estimated by Sala to cost $2.2 million dollars in 1979 values.

### 3.7 Mining Method

The mining method as postulated in the Feasibility Report involved overhand cut and fill, using light Drill jumbos in stoping, and loading with Cavo-load haul dump units to chutes.

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Each stoping block was planned at 400 feet, serviced by raises and ore chutes, with sequential extraction of 8 feet lifts over the stope, followed by filling with mill residues.

Two production levels were planned to achieve the anticipated annual output of 100,000 tonnes.

3.8 Infrastructure considerations

Because of the mine location at Beaconsfield, limited infrastructure provisions were allowed in the Feasibility Report. Power was to be provided by HEC; although purchase of Diesel sets was allowed for during the initial dewatering. Beyond a change-house, a Mine Administration building, and limited Engineering facilities and staff housing, no other service facilities were deemed necessary.

This situation is assumed to prevail in 1979.

## 4.0 MINE REOPENING

### 4.1 Basis of Mine Dewatering

The concept recommended for the dewatering of the Tasmania Mine is summarised from the 1975 Report as follows: -

1. Construct and seal a new watercourse for the Blyth Creek - this is a suspected water inflow source to the mine.
2. Effect Shaft collar repairs to Harts Shaft.
3. Install a temporary stage winder, and a fast sinking winder at Harts Shaft.
4. Provide power to the site - the lack of adequate HEC power required the purchase of a diesel generating station (3000 kw).
5. Install 3 Pleuger type submersible pumps in the shaft compartments with a combined capacity of 6 million gallons per day against a total head of 635 feet.
6. Effect minimal shaft repairs as the water level lowers, and dewater to the 635' crosscut (pump chamber) at Harts shaft.
7. Utilise the crosscut as a water storage horizon and install 3 centrifugal pumps, with rising mains to the surface.
8. Continue dewatering to the 1000 feet level, using the submersible pumps to lift shaft water to the centrifugal pumps.
9. Having exposed the 1000 feet level, install 2 centrifugal pumps, capable of pumping 6 million gallons to 1000 feet level, in conjunction with a water reservoir on the 1000 feet level.
10. Continue to dewater the Mine to the 1500 feet level via the submersible and centrifugal pumps.

The programme envisaged a dewatering period of 13 weeks from the delivery date of the submersible pumps, during which time about

800 million gallons would be pumped from the mine. The basis of 6 million gallons per day was predicted on previous maximum pumping rates for the Tasmania Gold Mine, and the fact that the "normal" mine inflow was approximately 2 million gallons per day. Minimal shaft timber repairs were to be carried out during this phase.

The principal objective, apart from the physical reopening of the mine, was to establish a drill site horizon on the 1370' level from where confirmatory drilling could be carried out to substantiate ore reserves.

This concept has been followed in establishing revised Capital Costs for the mine dewatering phase.

4.2 Subsequent Development

Again in summary form the subsequent Mine and Surface programme involved: -

1. Retimbering of Harts Shaft to provide a 2 compartment Cage-Skip haulage system.
2. Sinking Harts Shaft to 2000'.
3. The development of the 1500' level followed by two other levels.
4. Equipping the mine to produce initially 74,000 tonnes per year.
5. Construction of Mill facilities to process 100,000 tonnes per year.
6. Providing the necessary infrastructure facilities to support the mine.

Approximately 2 years was estimated for the total programme at a Capital Cost $5.43 million dollars (1975 values) which figure was increased to $7 million to cover inflation during the 2 year development period.

It is on these bases that this revised Capital estimate has been prepared.

5.0 REVISION OF CAPITAL ESTIMATE

For the revised capital estimate the following research has been undertaken: -

5.1 Labour

Labour rates applicable to mine personnel have been revised on the current Metalliferous Mining and Processing Award (December 1978).

The percentage increases since 1975 for mine personnel range from 44% to 52%. Table 12.1 of the 1975 Report sets out base rates, plus an estimate for over award payments and overhead allowances.

The percentage increases have been applied to the previous annual cost of labour, as scheduled in the 1975 Report. Appendix 1 sets out the calculations.

5.2 Construction, Engineering and Equipment

The Commonwealth Bureau of Statistics data has been the basis of revising construction and equipment supply, except where direct supply quotations have been sought from manufacturers; in the latter cases, the current quotations have been applied.

The following Statistical percentage increases are quoted using 1975 as a base.

| | % Increase |
|---|---|
| Metallic Materials | 48% |
| Electrical Installation Materials | 32% |
| Engineering Metals Labour Index | 44% |
| Industrial Machinery | 40% |
| Construction Building Labour Index | 41% |

A factor of 43% increase has been applied to items on which direct quotations were not obtained.

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5.3 Major Quotation Items

The principal items of plant on which quotations, by way of budget estimates were obtained cover: -

Mill Concentrating Plant.

Mine Equipment - Rock drills, Cavo loaders, Drill Steel Compressors, Rocker Shovels, Scrapers.

Submersible pumps.

Diesel Generating plant.

Main Winder.

Electric power HEC and Diesel.

Timber for Shaft Work.

Mine Buildings.

Changehouse.

Gold Room.

Additionally, from mining sources, current development costs were obtained, as were Shaft sinking costs.

5.4 Revised Capital Estimate

The revised estimate is prepared in a manner that enables comparison with Table 11.1 of the Feasibility Report.

5.4.1 Dewater Mine and Confirmatory Drilling

| Items | 1979 Estimate $ |
|---|---|
| Shaft Collar Repair, and Creek diversion | 99,000 |
| Purchase New Diesel Plant (6x475 kva ) | 560,000 |
| Purchase two Winders | 240,000 |
| Air & Water Pipes | 117,000 |
| Mine Labour | 61,000 |
| Purchase 3 Submersible Pumps, 5 Centrifugal Pumps | 376,000 |
| Cables and Installation | 105,000 |
| C/F | $ 1558,000 |

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| Items | 1979 Estimate |
|---|---|
| B/F | $ 1558,000 |
| Switchgear & Transformers | 142,000 |
| Power Cost, Diesel | 177,000 |
| Power HEC | 38,000 |
| Pump Installations, U/G | 18,000 |
| Diamond Drilling | 100,000 |
| Assays and Lab. Work | 10,000 |
| Site Supervision | 40,000 |
| Design Component | 57,000 |
| Miscellaneous (Items, Tools, Equipment Hire) | 21,000 |
| | $ 2,161,000 |

This compares with an estimate in 1975 $ values, of $1,017,000.

The major variations are due to: -

1. Purchase of New Diesel Sets - secondhand units are unavailable.
2. Purchase of new sinking winders.
3. Increase in oil prices. The cost of generating power by diesel plant has increased from 1.6 cents per kwh to 4.04 cents per kwh.
4. In total these variations amount to $613,000.

It is considered that the Diesel plant and the Winders would have a 50% resale value, thereby reducing the Net Cost of Stage 1 to $1,761,000 in 1979 costs.

5.4.2 Stage 2 - Complete Mine Development

| Items | 1979 Estimate $ |
| --- | --- |
| Pumps - 1500' Level & Cables | 181,000 |
| Mine Transformers | 118,000 |
| Switchgear | 35,000 |
| Power Usage - Diesel | 505,000 |
| Power Usage - HEC | 76,000 |
| Retimber Shaft | 158,000 |
| Install Main Winder, Headframe, Cables, Winder Building, Ore Bin | 416,000 |
| Compressors | 78,000 |
| U/G Work (Sumps etc.) | 56,000 |
| Sink Shaft 1500' - 1700' | 180,000 |
| Mine Equipment | 155,000 |
| Mine Supervision | 90,000 |
| Mine Development (790 metres) | 325,000 |
| Winder Drivers | 78,000 |
| Design Component | 12,000 |
| | $ 2,463,000 |

This compares with a cost of $1,509,000, estimated for 1975.

The major variations are due to: -

1. increased power costs from diesel generators
2. Purchase of a new winder
3. increased Mine Development costs.

These variations account for $624,000 increase.

## Stage 3 - Mine Plant and Equipment

| Items | 1979 Estimate $ |
|---|---|
| Purchase Drills, Cavo's, Loco's, Trucks, Skips, Loading Station, and Aux.Equipment | $321,000 |

## Stage 4 - Surface Plant and Buildings

| Items | 1979 Estimate $ |
|---|---|
| Surface Plant & Buildings, Administration Bldg. | 200,000 |
| Change Room | 100,000 |
| Workshop Store, Services | 50,000 |
| Houses | 150,000 |
| Vehicles | 20,000 |
| | $ 520,000 |

## Stage 5 - Mill

| Items | 1979 Estimate $ |
|---|---|
| Ore Processing Plant 100,000 T.P.A. | 2,200,000 |
| Mill Switchboard | 50,000 |
| General site works and concrete | 117,000 |
| Assay Office and Equipment | 150,000 |
| Gold Room | 100,000 |
| Tailing Dam and E.I.S. | 50,000 |
| | $ 2,667,000 |

This compares with $1.92 million estimated in 1975, and is based on Salas revised estimates.

Approximately $200,000 extra is provided in this review for Assay facilities and gold room security.

Stage 5, 6, 7 and 8

These items cover working capital, the capital costs for equipping a second level and the sinking of Harts Shaft.

| Items | 1979 Estimate |
|---|---|
| Working Capital | $ 400,000 |
| Equip 2nd level with mine plant | 155,000 |
| Sink shaft 300 ft. | 275,000 |
| Lump Sum Contingency | 350,000 |
| | $1,180,000 |

A summary of the Capital Cost, 1975 compared with 1979, is given in Table 1. Assuming the dewatering construction and development takes two years from June 1979, an estimate is made at the project completion date June 1981 at 8% inflation per year.

TABLE 1

SUMMARY OF CAPITAL COSTS

($000's)

| Stage No. | STAGE | 1975 Cost | Feb. 1979 Cost | Estimated Cost at 8% Inflation p.a. June 1981 |
|---|---|---|---|---|
| 1 | Dewater and Drill | 1,017 | 1,761 | 1,937 |
| 2 | Complete mine, Development | 1,509 | 2,463 | 2,832 |
| 3 | Mine Plant and Equipment | 230 | 321 | 369 |
| 4 | Surface Plant and Buildings | 163 | 520 | 598 |
| 5 | Mill | 1,920 | 2,667 | 3,067 |
| 6 7 & 8 | Working Capital, Capital Costs Year 3, Contingency | 591 | 1,180 | 1,392 |
| | TOTAL ($000's) | 5,430 | 8,912 | 10,195 |

## 6.0 OPERATING COSTS

The operating costs for the Mine, and the manning schedule, as at June 1975, are set out in following table.

| | Cost/Tonne $ | Manning | |
|---|---|---|---|
| | | Employees | Staff |
| MINE | 10.70 | 55 | 6 |
| MILL | 6.04 | 22 | 7 |
| ADMINISTRATION | 3.03 | | 12 |
| TOTAL | $ 19.76 | 77 | 25 |

The basis of developing costs was on labour costs projected for the operation, and prices for consumables pertaining at that time.

A check was made with Central Norseman Gold costs and Carr Boyd as to overall accuracy. The scale of Central Norseman operations was of the same order as projected for Beaconsfield, and the report quotes that "the operating costs for Beaconsfield are of the right order".

Applying the escalation factors developed for the previous section of this report. 1979 operating costs would be: -

| | | Cost/Tonne $ |
|---|---|---|
| MINING | | 15.30 |
| MILLING | | 8.65 |
| ADMINISTRATION | | 4.35 |
| TOTAL | = | $ 28.80 |

One can only make valued judgements in relation to these costs.

The Milling and Administration costs are of the right order. A recently commissioned gold mine, where total feed is cyanided, and labour rates are of the order of $20,000 to $24,000 per year has the following costs: -

| | |
|---|---|
| Milling | $ 5.40/Tonne |
| Administration | $ 3.50/Tonne |

The mining costs at $15.30 per tonne would be considered low.

For comparison, two highly mechanised mines in Tasmania, using large diesel loaders, run at $8.50 and $12.70 per tonne for mining.

Comparing output per man year employed underground the following figures are quoted: -

| | | |
|---|---|---|
| Renison Ltd. | 9000 | Tonnes |
| Cleveland | 4800 | Tonnes |
| Aberfoyle | 400 | Tonnes |
| Beaconsfield | 1780 | Tonnes |

It could be expected that Beaconsfield would fall between Cleveland and Aberfoyle range in terms of mining costs.

Assuming that an extra ten men need to be employed underground this would place a surcharge on costs of approximately $2.00/ tonne.

Again, assuming that the constant dewatering rate is 4 million gallons per day, then this would virtually double power costs. Using HEC power, this would increase costs by a further $2/tonne.

It is considered advisable to quote a range for the mining costs, between $15 - 20 per tonne, and the range for total costs between $30- and $35 per tonne milled, in 1979 values.
The higher figure of $35/tonne of ore milled is suggested for project viability analysis.

## 7.0 REVENUE AND PROJECT VIABILITY

### 7.1 Gold Revenue/Tonne Milled

The calculations made on gold revenue per tonne of ore milled, as per the 1975 Report were based on: -

- Gold recovery for the first two years of operations at 82.5%, and thereafter at 85%.
- Gold price at $200 US, equivalent to $A150 per ounce.

The test work performed on the three core samples demonstrated that 86.6% recovery could be achieved from a combination of gravity, flotation and cyanidation. The metallurgical consultant, as a result of his review, and taking cognisance of the high head grade of gold in core samples, projected that "the recovery of gold, as free gold, gold in cyamide solution, and gold in copper concentrate should total 80%".

Further the consultant indicated that on past records "gold recovered as free gold should be approximately 40%".

For the purpose of this review it is considered that a recovery of 82.5% of gold in ore be adopted. The major factors affecting revenue calculations have been the lift in gold price since the 1975 Report, and also the variation in exchange ratio due to devaluation of the Australian dollar.

For the purpose of updating Revenue projections it is suggested that the current gold price of $US 240 be adopted, at the present exchange rate of $1.12 US = $ 1 Aus.

This gives a gold price = $214 Aus. per ounce of gold in 1979 values.

The recovered gold value per tonne of ore milled, with a head grade of 12.85 dwts of Au per tonne is calculated at $113.6o Aus. per tonne.

7.2 Copper Revenue per Tonne Milled

The 1975 Report indicated a head grade of 0.83% Copper per ton, equivalent to 18.3 lbs of Copper per tonne.

Accepting the previous projections for copper recovery at 60%, at a copper concentrate grade of 25% Cu, it is calculated that on current copper prices, ($1700 per tonne Mt. Isa quote) the revenue from copper is $6.50 per tonne milled.

The total Revenue from gold and copper per tonne milled is therefore $120 Aus.

7.3 Project Viability

It is proposed to examine project viability on the following assumptions: -

Capital Cost: The total capital cost of the project is $8.912 million in 1979 values, and with a 2 year exploration and development programme would escalate capital cost to $10.195 million by 1981.

Revenue Estimate: The revenue per tonne of ore milled, is estimated at $120 per tonne. No escalation is applied to gold prices over the 2 year period.

Operating Costs: The operating cost per tonne of ore milled is taken at $35 per tonne as suggested in Section 6 of the report. Again no escalation is applied to operating costs, on the basic premise that gold prices will increase in the two year development programme sufficient to cover escalation in operating costs.

Taxation: The present position of no taxation on Gold Mining will persist over the life of the Mine. Historically since I.A.C. Report in July 1975 Government has not acted on the I.A.C. recommendation. Economically, and politically various factors have emerged since 1975 to strengthen the argument against taxing gold profits. Included in these factors are, the reopening of gold mines by Kalgoorlie Lake View Associates, where the breakeven price required is $212 Aus per ounce, the general employment position in Australia, and the components of gold in the calculation of Australia's Overseas Reserves.

Mine Surplus from Operations and Viability

The mine surplus per tonne of ore milled, at an annual treatment rate of 100,000 tonnes, is calculated as: -

| | | |
|---|---|---|
| Surplus per tonne of Ore Milled | = | $ 85/tonne. |
| Annual Surplus at 100,000 tonnes | = | $ 8.5 million |
| Total Mine Surplus based on 601,000 Tonnes of to be proved reserves | = | $ 51 million |
| Capital Cost for Project | = | $ 10.2 million |
| Cash flow generated over 6-7 years of Operations | = | $ 40.8 million |
| Payback Period on Capital | = | 1.2 Years |
| Annual Rate of Return on investment | = | 83% |

It is not considered that D.CF. calculations, or detailed sensitivity analyses are beyond the ambit of this report.

It can be stated that with a Capital Cost of $20 million and with operating costs as projected at $35/tonne, the cash flow generated subject to the reserves being proved up, is approximately $30 million.

Alternatively with Capital Cost constant at $10.2 million, operating costs at approximately $68 per tonne milled could be supported, to give a cash flow of $30 million over the life of Mine.

These examples demonstrate only what Allstate Directors have been aware of, namely the risk element is dominantly in the dewatering and proving up of the tonnage and grade of gold.

It must be emphasised that minimal engineering investigation work has to date been carried out on -

- pumping rates to dewater the mine
- the condition of the shaft
- the retimbering and refurbishing of the shaft
- ground conditions surface, and underground

and the Directors must appreciate that within the context of this report, all previous assumptions have been accepted in preparing this revision.

# APPENDIX 1

## METALLIFEROUS MINING & PROCESSING AWARD

## LABOUR RATE COMPARISON 1979 - 1975

| CLASSIFICATION | BASE RATE | | % |
|---|---|---|---|
| | 1979 $/Week | 1975 | Increase 1975 = 100 |
| MINER - WINZES RAISES | 161.40 | 112 | 144% |
| CONTRACT MINER * | | 170 | |
| FITTER/ELECTRICIAN | 163.00 | 113 | 144% |
| AVERAGE MILLHAND | 139.60 | 92 | 152% |
| WINDER DRIVER | 158.80 | 112 | 142% |
| BRACEMAN | 141.20 | 95 | 149% |

* Estimated to earn 1/3 above Award

## ANNUAL EARNING APPLIED TO FEASIBILITY STUDY

| CLASSIFICATION | ANNUAL RATE 1975 | % INCREASE | CALCULATED ANNUAL-1979 RATE |
|---|---|---|---|
| MINER- WINZES RAISES | $ 9,500 | 144 | 13,700 |
| CONTRACT MINER | $11,500 | 144 | 16,600 |
| FITTER ELECTRICIAN | $ 9,500 | 144 | 13,700 |
| AVERAGE MILLHAND | $ 7,000 | 152 | 10,640 |
| WINDER DRIVER | $ 9,500 | 142 | 13,500 |
| BRACEMAN | $ 8,000 | 149 | 11,920 |

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APPENDIX 2

REVISION OF PROCEDURES AND ESTIMATES WITH HEC POWER REPLACING DIESEL POWER IN YEARS 1 AND 2

## 1. INTRODUCTION

The course followed by the original report in 1975 in respect to the procedures for year 1 and year 2 took cognizance of three factors.

(i) Several sources of supply of suitable second hand diesel generators and switchgear were available at that time at reasonable prices.

(ii) The cost per KWH of diesel generated electricity and HEC electricity were similar.

(iii) The HEC required a capital bond of about $250,000 before agreeing to provide power and required a construction period over 12 months.

The situation today has changed in respect of the first two factors.

(i) Less second hand diesel sets are available and the prices have increased to the extent that the use of new sets must be assumed. The initial capital outlay for these sets would be greater than the bond for HEC power and the write off in the event of the mine not proceeding overlap in the same order as the bond for HEC power.

(ii) The fuel cost per KWH for diesel generated electricity has increased by a factor of 2.5 and the future price is uncertain. The cost of HEC power has increased by a factor of 1.13.

Consequent to the above it was felt that a revision of the original procedures should be made on the basis of using HEC power for all stages of the mine reopening and development and use.

It is emphasised that this review is done in brief to identify the possible order of cost and time. The time for this brief does not permit the same depth of analysis as for the original report.

## 2. ASSUMPTION RE HEC POWER

It is unlikely that there would be technical impediment to the HEC making high voltage power available for dewatering in 26 weeks. The conditions of supply and the willingness of the Authorities to provide power in this time is largely a matter of politics. In the present employment climate it is probable that the Tasmanian Government would strongly encourage the mine reopening and therefore assist in the provision of power.

Considering the foregoing the programme assumes 500 Kw of low voltage power available at the project start (as previously) with high voltage power available 26 weeks later.

## 3. REVISION OF CAPITAL ESTIMATES

Revision of capital estimates for Stage I and Stage II and the Summary are attached. The 2nd estimate for the total project decreases from $10,195,000 to $9,317,000.

## 4. REVISION OF PROGRAMME - MINE REOPENING

The concept of reopening the mine has been revised on the basis of HEC power use as follows: -

(1) Design of the submersible pump installation to the stage of finalising price and data for order placement of pumps and cables.

(2) Design the HEC installation and High voltage switchgear to the stage of issuing instructions to proceed. Some investigation of the alternate power requirements will be needed in this phase of design.

(3) Design the construction winder system and rig for handling the submersible pumps sufficient to enable the pump designs to be finalised and to enable the specifications of the construction stage winder to be established. If a winder has to be manufactured the delivery of this would be on the critical path.

All of the steps (1), (2) and (3) above are interelated and must be carried out concurrently. The time period of three months should be allowed.

(4) Manufacture and delivery of submersible pumps 6 months.

(5) Manufacture and installation of HEC power and high voltage switchgear 6 months.

(6) Detail design of stage winder installations and pump lifting rig and manufacture and installation of all equipment 6 months.

All of the steps (4), (5) and (6) are concurrent and are equally critical. They are consequent to steps (1), (2) and (3).

(7) Clear the top of the Shaft and make safer, inspect the shaft for blockages and condition of the walls. Time period 2 months.

(8) Design the new collar for the shaft and install. Time period 6 months. This work is consequent to part of steps (6) and (7).

(9) Construct and seal new watercourse at Blyth Creek - time period 4 months.

(10) Design pumping installations for each level and procure pumps and piping and cables - time period 7 months.

(11) Design of mine winder to the stage of order placement to ensure that construction details are fully known, time period 5 months.

(12) Design of Shaft timbering or repairs, time period 1 month.

(13) Design underground electrical supply, mains, time period 1 month.

All of items 10,11,12 and 13 are consequent to the inspection of the shaft item 7, and each is dependent on the other.

(14) Install submersible pumps followed by pumps at 630 ft level and dewater to 1500 feet, time period 3 months.

(15) Carry out test drillings at the 1370 feet level and analyse results.

The estimated total period to the commencement of item 15 is 12 months.

## (5) COMMENTS ON PROGRAMME

The dewatering operations are critical to the timing of the mine reopening. The order of engineering cost of items 1, 2 and 3 in the foregoing programme is $10,000. If this work were done prior to project start, the time period for the commencement of test drilling (item 15 on the foregoing programme) will reduce to 9 months which is in line with the original programme submitted.

In year 1 it will be required to design the mill flow sheet on the basis of known ore data and make selections of basic equipment up to the stage of preliminary order dates. This work could be done by consultants in conjunction with equipment suppliers.

In year 2 proceed with the detailed design of the mill, mill buildings and support buildings. This work would be done by a projects contractor.

REVISED CAPITAL ESTIMATE - HEC POWER

The revised estimate is prepared in a manner that enables comparison with Table 11.1 of the Feasibility Report.

Dewater mine and confirmatory drilling.

| Items | 1979 Estimate $ |
|---|---|
| Shaft collar repair, and Creek diversion | $ 99,000 |
| Mine Transformers Substation | 118,000 |
| Purchase two winders | 240, 00 |
| Air & Water Pipes | 117,000 |
| Mine Labour | 61,000 |
| Purchase 3 Submersible pumps 5 Centrifugal pumps | 376,000 |
| Cables and Installation | 105,000 |
| Switchgear & Transformers | 92,000 |
| Security Deposit H.E.C. | 300,000 |
| Power HEC | 94,000 |
| Pump Installations U/G | 18,000 |
| Diamond Drilling | 100,000 |
| Assays and Lab.Work | 10,000 |
| Site Supervision | 40,000 |
| Design Component | 57,000 |
| Miscellaneous Items, Tools and Equipment Hire | 21,000 |
| | $1,848,000 |
| Less Refund of HEC Security and 50% of Winder Cost | $ 420,000 |
| NET COST | $1,428,000 |

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## STAGE 2 - COMPLETE MINE DEVELOPMENT

| Items | 1979 Estimate 1979 |
|---|---|
| Pumps and Cable - 1500' level | $ 181,000 |
| Switchgear | 35,000 |
| Power Usage HEC | 252,000 |
| Retimber Shaft | 158,000 |
| Install Main Winder, Head Frame, Cages, Winder Building, Ore Bin | 416,000 |
| Compressors | 78,000 |
| U'G. Work (Sumps etc.) | 56,000 |
| Sink Shaft 1500' - 1700' | 180,000 |
| Mine Equipment | 155,000 |
| Mine Supervision | 90,000 |
| Mine Development (790 metres) | 325,000 |
| Winder Drivers | 78,000 |
| Design Component | 12,000 |
| | $ 2,016,000 |

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SUMMARY OF CAPITAL COSTS

| Stage No. | Stage | 1975 Cost | Feb 1979 Cost | Estimated Cost at 8% Inflation p.a. June 1981 |
|---|---|---|---|---|
| | | $000's | | |
| 1 | Dewater and Drill | 1,017 | 1,428 | 1,571 |
| 2 | Complete mine, Development | 1,509 | 2,016 | 2,320 |
| 3 | Mine Plant and Equipment | 230 | 321 | 369 |
| 4 | Surface Plant and Buildings | 163 | 520 | 598 |
| 5 | Mill | 1,920 | 2,667 | 3,067 |
| 6, 7 & 8 | Working Capital, Capital Costs, Year 3 Contingency | 591 | 1,180 | 1,392 |
| | TOTAL ($000's) | 5,430 | 8,132 | 9,317 |