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**AMERICAN BULLION**
MINERALS LTD.
886917

# RED CHRIS PROJECT

# EXECUTIVE SUMMARY

# 1998 PRE-FEASIBILITY STUDY

June, 1998

# EXECUTIVE SUMMARY

## 1. INTRODUCTION

The American Bullion Minerals ("ABP") Red Chris project is a large copper-gold deposit located in northern British Columbia, approximately 190 km north of Stewart, B.C. The Project is owned by ABP (80%) and Teck Corporation Ltd. ("Teck") (10% participating and 10% carried interest), both of Vancouver, B.C., Canada. ABP has a right subject to certain terms and conditions, to acquire Teck's 10% participating interest any time up to December 24, 1998 and half (or 5%) of Teck's carried interest in the property any time up to commercial production.

Since an initial Pre-Feasibility study was completed in 1996 by Fluor Daniel Wright Ltd. ("FDW") there has been a significant reassessment of the deposit geology, mine development, waste management options and commodity market conditions. By comparison, the 1998 Pre-Feasibility study (the "Study") defines a smaller, higher grade mine plan with lower overall capital cost requirements, more attractive cash flow and low cash costs of production.

The Study establishes the viability of an open pit mine, concentrator and infrastructure for the Red Chris Project in a design that takes advantage of both the topography to reduce mine costs and scheduling to access the high grade core in the deposit. The mine plan uses a system of ore passes and underground conveyors to move material from the open pit – creating a gravity flow of material from the mine to the concentrator and waste storage areas. The Study also contemplates two cases, carrying detailed designs and cost estimating for the 'Design Case' and a more conceptual evaluation of a reduced capital and operating cost scenario called the 'Mine-Mill' Enhanced Comminution Case. The latter considers the beneficial reduction in grinding requirements as a result of additional blasting effort expended in the mine operations, and natural (autogenous) comminution in ore passes.

## 2. PROJECT DESCRIPTION

The Red Chris deposit is a structurally controlled bulk-tonnage copper-gold porphyry deposit. The proposed mine plan is an open pit operation, mining the Red Chris deposit as two pits initially (the "East" and the "Main" Pits) that rapidly converge into a single open pit. Over the 21 year mine life, 225 million tonnes of copper-gold porphyry ore will be processed in a 30,000 tonne per day mill producing on average 151,000 tonnes per year of concentrate grading 26.9% copper, 14.8 g/t gold and 40.7 g/t silver.

The mine plan begins with a total of six ore and waste passes strategically located to optimize grade early in the mine schedule and to minimize pre-production stripping requirements. Benches are sloped toward the ore passes to improve productivity. Good blasting practices, communication and supervision will be required to ensure productivity during operations. Dozers have been chosen as the prime production unit for moving ore and waste to ore passes eliminating the need for a shovel/truck fleet and support equipment.

Based on experience in the coal mining industry, large dozers have proven to be the lowest cost method of moving material for distances of up to 150 m in certain applications; (i.e. limited haul distance and constant withdrawal of ore and waste such that the dozer does not push upwards). The use of dozers at the Red Chris project provides advantages of low capital cost, low overhead (i.e. no truck maintenance shops, shovel assembly, etc.), low operating cost, and no wide haul road. In addition, the technique allows higher grade to be mined earlier, steeper pit walls and very low pre-stripping requirements.

Ore will be crushed underground and conveyed through an adit to the mill, which is located about 1.5 km east of the pit. Waste rock is similarly crushed and conveyed to the combined tailings and waste rock storage area, and sequentially covered with alkaline tailings. Non-reactive waste rock will be used for construction. Low grade ore will also be temporarily stored within the tailings and waste rock storage facility.

In the Study, concentrate will be trucked in concentrate pods to the port at Stewart and loaded onto ocean going vessels for shipment to offshore smelters. The concentrate trucking system will also be used to backhaul supplies to the site.

The history of property development is summarized in the following:

| | |
|---|---|
| 1920 to 1950's | Independent exploration work in the district. |
| 1956 | Conwest Exploration Limited staked Windy claims on Todagin Plateau. |
| 1968 – 1972 | Staking and exploration by Great Plains Development Company Canada, Ltd. |
| 1970 – 1972 | Silver Standard Mines staked claims north and east of Chris claims. |
| 1973 | Ecstall Mining Limited (later Texasgulf Canada Limited) optioned Red claims and conducted drilling program. |
| 1974 | Joint venture agreement between Great Plains Development, Silver Standard Mines and Texasgulf Canada Limited (operator). |
| 1974-1980 | Exploration by Texasgulf Canada Limited identified the 'Main' and 'East' Zones. |

| | |
|---|---|
| 1994, January | Ownership of the property re-organized between Falconbridge (60%), Norcen Energy Resources (20%) and Teck Corporation (20%) |
| 1994 | Geological review by Rebagliati (1994) evaluating copper-gold geological resource and identifying high grade core zones open to depth and potentially along strike. |
| 1994 | American Bullion Minerals Ltd. (ABP) negotiated 80% interest in Red Chris deposit, with 20% interest held by Teck Corporation and a 1.8 % NSR held by Falconbridge. |
| 1994 | ABP staking and exploration programs comprising staking, land surveying, geochemical sampling, geophysical surveying, and diamond drilling of 58 holes totaling 21,417 m for assay and acid base account testing. |
| 1995 | ABP staking and exploration programs including baseline environmental, staking, land surveying, geochemical sampling, geological mapping, geotechnical diamond drilling, and exploration drilling of 112 holes totaling 36,770 m. Metallurgical testing and resource evaluation studies conducted. |
| 1996, May | Issued Pre-Feasibility Study by Fluor Daniel Wright for ABP. |
| 1997 | ABP initiates evaluation of mine development alternatives and re-evaluation of resource. |
| 1998, June | Revised Pre-Feasibility Study issued. |

# 3. PROJECT ECONOMICS

## 3.1 Capital Cost

A summary of the pre-production capital cost estimate by the major project areas is shown in Table A below. The estimated total pre-production capital cost is C$263.2 million (US$ 181.5 million). A chart showing a breakdown of capital costs into major categories is shown in Figure A.

Underground mine development costs include all rock excavation work and assume that the work will be done by a contractor. Construction labor rates and construction indirect costs are based on similar size projects in remote areas of B.C. The average cost of construction labor has been estimated at $48/hour.

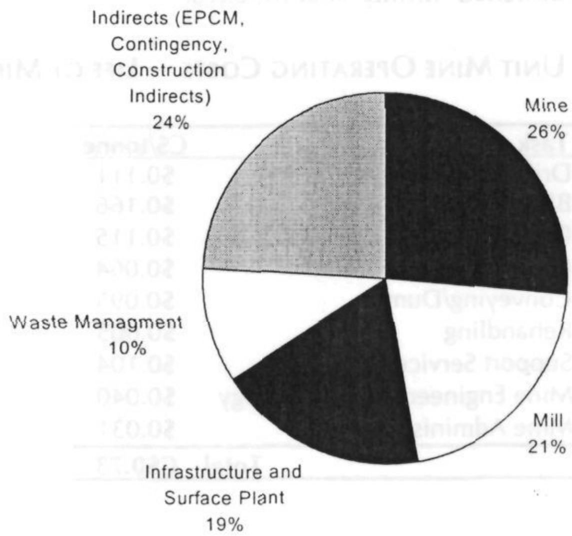
The costs for the majority of the major mechanical equipment is based on the availability of good quality, fully refurbished equipment and include the costs for equipment inspection, dismantling, refurbishment and transport to the project site. The estimates for earthworks, concrete, structural steel and platework are based on recent contractor experience in Northern B.C.

## TABLE A PRE-PRODUCTION CAPITAL COST ESTIMATE (DESIGN CASE)

| Area | Description | Total C$000's[1] |
|---|---|---|
| 06 | Underground Mine Development | 25,865 |
| 07 | Ore and Waste Handling System | 23,381 |
| 08 | Mine Services | 755 |
| 09 | Mobile Equipment | 13,946 |
| 10 | Pre-Stripping | 5,982 |
| 12 | Ore Storage | 4,406 |
| 20 | Concentrator Building | 15,595 |
| 21 | Grinding | 14,839 |
| 22 | Flotation | 8,660 |
| 23 | Dewatering & Storage | 1,359 |
| 24 | Reagents & Services | 1,034 |
| 25 | Process, Fresh and Fire Water | 1,763 |
| 26 | Tailings and Waste Rock | 25,989 |
| 27 | Plant Site Services | 5,297 |
| 28 | Service Buildings | 3,706 |
| 29 | Camp | 2,753 |
| 30 | Power Supply | 33,702 |
| 31 | Access Road | 3,720 |
| 34 | Spare Parts | 5,000 |
| 35 | Initial Fills | 2,000 |
| | Sub-Total Direct Cost | C$ 199,749 |
| 81 | Construction Indirects | 15,687 |
| 82 | EPCM Engineering Services | 12,460 |
| 92 | Contingency[2] | 35,268 |
| | Sub-Total Indirect Cost | 63,415 |
| | **TOTAL** | **C$ 263,164**<br>**(US$ 181,583)** |

Notes: 1. Costs by area are shown rounded to nearest thousand dollars. All totals and sub-totals are based on actual (not rounded) values.
2. A 15% contingency has been applied to the Mine/Concentrator capital costs, 30% contingency to the tailings and waste rock capital costs, and a 5% contingency to indirects and EPCM.

**FIGURE A PRE-PRODUCTION PROJECT CAPITAL COST SUMMARY**




## 3.2 Operating Costs

The following average unit costs have been estimated for the mining, processing and general and administration (G & A) for the life of the mine at Red Chris.

**TABLE B OPERATING COST SUMMARY (DESIGN CASE)**

| Task | C$ | US$ |
|---|---|---|
| Mining ($/tonne ore) | 1.74 | 1.20 |
| Process ($/tonne ore) | 3.05 | 2.11 |
| G & A ($/tonne ore) | 0.47 | 0.32 |
| **Total ($/tonne ore)** | **5.26** | **3.63** |

General hourly and staff wages are based on current wages for operating mines in B.C. A payroll burden of 30% has been added to salaries to cover costs of benefits, holiday pay, CPP, WCB, UIC, life and long-term disability and company pension plan.

The manpower costs are based on the following:

- Mine employees will be on a 2 week in, 2 week out rotation;
- Mill and surface employees will be on a 2 week in, 2 week out rotation; and,
- G&A and technical staff will be on a 4 day in, 3 day out rotation.

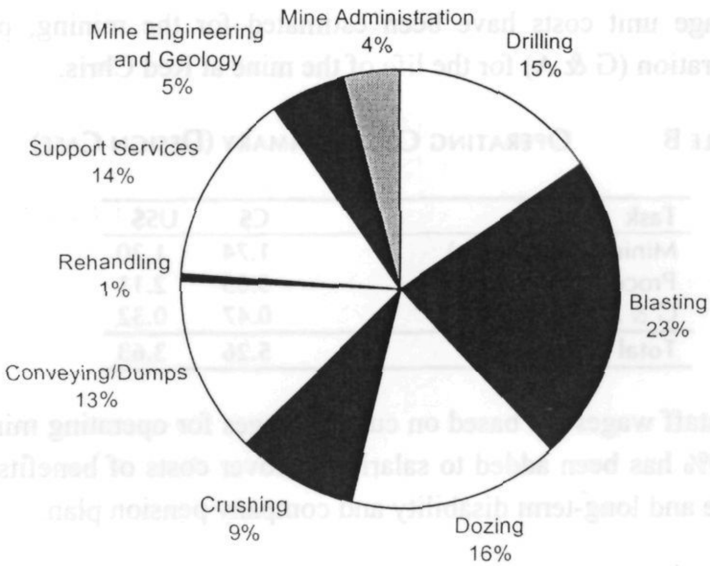
### 3.2.1 *Mine Operating Cost*

The operating cost estimates for the mine have been developed from first principles using equipment productivity consistent with industry standards. Costs for consumables were obtained from suppliers or published mining cost indexes.

TABLE C UNIT MINE OPERATING COSTS – LIFE OF MINE

| Task | C$/tonne |
|---|---|
| Drilling | $0.111 |
| Blasting | $0.166 |
| Dozing | $0.115 |
| Crushing | $0.064 |
| Conveying/Dumps | $0.095 |
| Rehandling | $0.005 |
| Support Services | $0.104 |
| Mine Engineering and Geology | $0.040 |
| Mine Administration | $0.031 |
| Total | C$0.73 |

FIGURE B MINE OPERATING COSTS BY FUNCTION



### *3.2.2 Process Operating Costs*

The processing (mill) costs cover all unit operations including crushed ore storage/reclaim, grinding, flotation, concentrate dewatering, tailings pumping, water treatment, concentrate loading and assay laboratory.

**TABLE D MILL OPERATING COSTS – LIFE OF MINE**

| Task | | C$/tonne |
|---|---|---|
| Labor | | $0.59 |
| Operating Supplies | | $1.88 |
| Power | | $0.58 |
| | Total | C$3.05 |

Processing costs also include all supplies and maintenance in processing circuits, plus reagent handling and mixing. A total mill work force of 90 employees will be required.

The project power consumption has been calculated from a detailed assessment of connected load, factored for expected operating load. Overall annual energy consumption is estimated to be 212 Gigawatts, with process facilities accounting for 181 Gigawatts.

The energy cost is based on power supply from BC Hydro at $0.039/kWhr. This rate includes demand charges. The province is contemplating reductions to this rate for operating mines in B.C.

## 3.3 Financial Analysis

Two cases were evaluated for the financial analysis of the Study; the Design Case and the Enhanced Comminution (crushing and grinding) Case. Both cases assume the same mine schedules (shown on Table E), development requirements and similar mining styles. The Enhanced Comminution case evaluates an estimated decrease in the grinding requirement for the mill to reflect the additional breakage of the ore as a result of the:

- high degree of drilling and blasting effort;
- impact of the prime mover (large dozers) in moving material; and,
- grinding that naturally occurs in the ore passes (demonstrated at operating mines).

Cost est. = US$0.55/lb Cu

1.5 Moz Au
lb Cu

## TABLE E MINING SCHEDULE AND CASH FLOW - DESIGN CASE

| MINING SCHEDULE & CASHFLOW – Design Case | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **DESCRIPTION/YEAR** | **UNITS** | **-2** | **-1** | **1** | **2** | **3** | **4** | **5** | **6** | **7** | **8** | **9** | **10** |
| **ORE MINED (NSR cutoff>C$3.86@0.75Cu/275 Au** | | | | | | | | | | | | | |
| Tonnes | tonnes ('000's) | | 2,779 | 10,479 | 9,659 | 12,538 | 18,263 | 10,781 | 10,245 | 10,663 | 7,433 | 5,089 | 10,613 |
| **LOW GRADE MINED** | | | | | | | | | | | | | |
| Tonnes | tonnes ('000's) | | | | | | | 1,138 | 1,088 | 419 | 268 | 151 | 251 |
| **WASTE MINED** | | | | | | | | | | | | | |
| Waste Mined | tonnes ('000's) | | 5,765 | 17,956 | 19,281 | 15,306 | 10,436 | 15,660 | 19,318 | 16,065 | 21,208 | 24,859 | 19,311 |
| Total ore (+LG) & waste mined | tonnes ('000's) | | 8,544 | 28,435 | 28,940 | 27,844 | 28,700 | 26,440 | 29,563 | 26,728 | 28,641 | 30,099 | [illegible] |
| Strip Ratio | | | 2.07 | 1.71 | 2.00 | 1.22 | 0.57 | 1.45 | 1.89 | 1.51 | 2.85 | 4.74 | 1.78 |
| Average daily production | tonnes | | 23,409 | 77,904 | 79,288 | 76,286 | 78,629 | 72,439 | 80,994 | 73,228 | 78,468 | 82,463 | 82,673 |
| **ORE MILLED** | | | | **28,710** | **30,000** | **30,001** | **30,001** | **29,999** | **30,000** | **30,001** | **29,999** | **30,002** | **30,000** |
| Tonnes | tonnes ('000's) | | | 10,479 | 10,950 | 10,950 | 10,950 | 10,950 | 10,950 | 10,950 | 10,950 | 10,951 | 10,950 |
| Net Smelter Return (N.S.R.) | C$/tonne ore | | | 10.33 | 8.39 | 6.79 | 11.38 | 8.49 | 5.44 | 7.37 | 8.35 | 5.39 | 6.56 |
| Copper grade | % | | | 0.582 | 0.472 | 0.396 | 0.641 | 0.472 | 0.327 | 0.421 | 0.452 | 0.294 | 0.344 |
| Gold grade | g/tonne | | | 0.407 | 0.331 | 0.246 | 0.448 | 0.345 | 0.182 | 0.280 | 0.359 | 0.228 | 0.301 |
| **ORE STOCKPILE** | | | **+** | | **-** | **+** | **+** | **-** | **-** | **-** | **-** | **-** | |
| Added or removed from stockpile | tonnes (000's) | | 2,799 | 0 | -1,291 | 1,588 | 7,313 | -169 | -705 | -287 | -3,517 | -5,711 | 0 |
| **LOW GRADE STOCKPILE** | | | | | | | | **+** | **+** | **+** | **+** | | **-** |
| Added or removed from stockpile | tonnes ('000's) | | | | | | | 1,138 | 1,088 | 419 | 268 | 0 | -86 |
| **CASH FLOW** | | | | | | | | | | | | | |
| Copper price | US$/lb. | **0.85** | **0.85** | **0.85** | **0.85** | **0.85** | **0.85** | **0.85** | **0.85** | **0.85** | **0.85** | **0.85** | **0.85** |
| Gold price | US$/oz. | **310** | **310** | **310** | **310** | **310** | **310** | **310** | **310** | **310** | **310** | **310** | **310** |
| Capital costs | C$millions | 61.38 | 201.78 | 19.02 | 1.68 | 7.71 | 9.18 | 10.32 | 1.32 | 0.64 | 0.44 | 11.21 | 4.96 |
| Mining cost | C$/tonne | | in capital | 0.70 | 0.70 | 0.70 | 0.70 | 0.72 | 0.70 | 0.72 | 0.72 | 0.72 | 0.69 |
| Processing, power and G&A cost | C$/tonne ore | | | 3.52 | 3.52 | 3.52 | 3.52 | 3.52 | 3.52 | 3.52 | 3.52 | 3.52 | 3.52 |
| Operating costs | C$ millions | | | 56.79 | 58.89 | 58.15 | 58.55 | 57.50 | 59.12 | 57.66 | 59.11 | 60.19 | [illegible] |
| Revenue | C$ millions | | | 130.70 | 110.86 | 89.91 | 150.45 | 112.07 | 72.18 | 97.42 | 110.17 | 71.09 | [illegible] |
| NSR royalty (@0.8%) | C$ millions | | | 1.05 | 0.89 | 0.72 | 1.20 | 0.90 | 0.58 | 0.78 | 0.88 | 0.57 | 0.69 |
| Undiscounted profit (pre-tax) | C$ millions | -61.38 | -201.78 | 53.84 | 49.41 | 23.33 | 81.52 | 43.35 | 11.16 | 38.34 | 49.74 | -0.88 | 21.45 |
| Undiscounted cumulative profit (pre-tax) | C$ millions | -61.38 | -263.16 | -209.32 | -159.91 | -136.58 | -55.06 | -11.71 | -0.55 | 37.79 | 87.53 | 86.65 | 108.10 |
| Discount factor @5% discount rate | % | 100% | 95% | 91% | 86% | 82% | 78% | 75% | 71% | 68% | 64% | 61% | 58% |
| Discounted profit (pre-tax) | C$ millions | -61.38 | -192.17 | 48.84 | 42.68 | 19.19 | 63.87 | 32.35 | 7.93 | 25.95 | 32.06 | -0.54 | 12.54 |
| Discount factor @ 8% discount rate | % | 100% | 93% | 86% | 79% | 74% | 68% | 63% | 58% | 54% | 50% | 46% | 43% |
| Discounted profit (pre-tax) | C$ millions | -61.38 | -186.83 | 46.16 | 39.22 | 17.15 | 55.48 | 27.32 | 6.51 | 20.72 | 24.88 | -0.41 | 9.20 |
| Discounted factor @ 10% discount rate | % | 100% | 91% | 83% | 75% | 68% | 62% | 56% | 51% | 47% | 42% | 39% | 35% |
| Discounted profit (pre-tax) | C$ millions | -61.38 | -183.44 | 44.50 | 37.12 | 15.94 | 50.62 | 24.47 | 5.73 | 17.89 | 21.09 | -0.34 | 7.52 |
| I.R.R. (pre-tax) | % | 13.6% | | | | | | | | | | | |

## TABLE E MINING SCHEDULE AND CASH FLOW - DESIGN CASE (CONT'D)

MINING SCHEDULE & CASHFLOW – Design Case

| DESCRIPTION/YEAR | UNITS | 11 | 12 | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | TOTAL |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **ORE MINED (NSR cutoff>C$3.86@0.75Cu/275 Au** | | | | | | | | | | | | | |
| Tonnes | tonnes ('000's) | 10,513 | 7,566 | 10,094 | 10,396 | 10,630 | 10,613 | 10,513 | 10,630 | 9,877 | 11,266 | 5,742 | 216,381 |
| **LOW GRADE MINED** | | | | | | | | | | | | | |
| Tonnes | tonnes ('000's) | 586 | 921 | 988 | 502 | 435 | 268 | 184 | 622 | 385 | 0 | 0 | 8,205 |
| **WASTE MINED** | | | | | | | | | | | | | |
| Waste Mined | tonnes ('000's) | 19,115 | 19,953 | 17,037 | 16,931 | 14,153 | 12,467 | 7,365 | 5,698 | 3,458 | 1,421 | 184 | 302,950 |
| Total ore (+LG) & waste mined | tonnes ('000's) | 30,214 | 28,441 | 28,119 | 27,829 | 25,218 | 23,348 | 18,062 | 16,950 | 13,720 | 12,688 | 5,926 | [illegible] |
| Strip Ratio | | 1.72 | 2.35 | 1.54 | 1.55 | 1.28 | 1.15 | 0.69 | 0.51 | 0.34 | 0.13 | 0.03 | [illegible] |
| Average daily production | tonnes | 82,778 | 77,919 | 77,038 | 76,244 | 69,090 | 63,968 | 49,484 | 46,439 | 37,589 | 34,760 | 16,236 | [illegible] |
| **ORE MILLED** | | **30,000** | **30,000** | **30,000** | **30,000** | **30,000** | **30,000** | **30,000** | **30,000** | **29,997** | **30,866** | **15,731** | |
| Tonnes | tonnes ('000's) | 10,950 | 10,950 | 10,950 | 10,950 | 10,950 | 10,950 | 10,950 | 10,950 | 10,949 | 11,266 | 5,742 | 224,587 |
| Net Smelter Return (N.S.R.) | C$/tonne ore | 10.83 | 6.85 | 5.42 | 5.57 | 6.35 | 7.14 | 7.90 | 7.32 | 6.91 | 8.54 | 12.73 | 7.73 |
| Copper grade | % | 0.546 | 0.370 | 0.312 | 0.312 | 0.338 | 0.377 | 0.418 | 0.384 | 0.354 | 0.434 | 0.637 | 0.419 |
| Gold grade | g/tonne | 0.533 | 0.295 | 0.202 | 0.223 | 0.282 | 0.322 | 0.354 | 0.334 | 0.331 | 0.414 | 0.633 | 0.330 |
| **ORE STOCKPILE** | | | | | | | | | | | | | |
| *Added or removed from stockpile* | tonnes (ooo's) | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 316 | -316 | |
| **LOW GRADE STOCKPILE** | | + | - | + | - | + | - | - | + | - | | | |
| Added or removed from stockpile | tonnes ('000's) | 149 | -2,463 | 132 | -52 | 115 | -69 | -253 | 302 | -687 | 0 | 0 | |
| **CASH FLOW** | | | | | | | | | | | | | |
| Copper price | US$/lb. | **0.85** | **0.85** | **0.85** | **0.85** | **0.85** | **0.85** | **0.85** | **0.85** | **0.85** | **0.85** | **0.85** | **0.85** |
| Gold price | US$/oz. | **310** | **310** | **310** | **310** | **310** | **310** | **310** | **310** | **310** | **310** | **310** | **310** |
| Capital costs | C$millions | 10.12 | 12.66 | 1.96 | 1.76 | 1.96 | 5.68 | 1.96 | 10.19 | 8.97 | 2.12 | 0.50 | 387.52 |
| *Mining cost* | *C$/tonne* | *0.73* | *0.74* | *0.70* | *0.70* | *0.72* | *0.74* | *0.85* | *0.81* | *0.85* | *0.88* | *0.87* | *0.73* |
| Processing, power and G&A cost | C$/tonne ore | 3.52 | 3.52 | 3.52 | 3.52 | 3.52 | 3.52 | 3.52 | 3.52 | 3.52 | 3.52 | 3.52 | 3.52 |
| Operating costs | C$ millions | 60.69 | 59.68 | 58.31 | 58.02 | 56.78 | 55.78 | 53.90 | 52.19 | 50.23 | 50.80 | 25.37 | 1,167.02 |
| Revenue | C$ millions | 142.39 | 90.30 | 71.78 | 73.62 | 83.68 | 94.02 | 104.05 | 96.38 | 90.93 | 115.58 | 87.70 | [illegible] |
| NSR royalty (@0.8%) | C$ millions | 1.14 | 0.72 | 0.57 | 0.59 | 0.67 | 0.75 | 0.83 | 0.77 | 0.73 | 0.92 | 0.70 | [illegible] |
| Undiscounted profit (pre-tax) | C$ millions | 70.44 | 17.25 | 10.94 | 13.25 | 24.28 | 31.82 | 47.37 | 33.23 | 31.00 | 61.73 | 61.13 | 510.53 |
| Undiscounted cumulative profit (pre-tax) | C$ millions | 178.55 | 195.79 | 206.73 | 219.98 | 244.25 | 276.07 | 323.44 | 356.66 | 387.67 | 449.40 | 510.53 | |
| *Discount factor @5% discount rate* | % | *56%* | *53%* | *51%* | *48%* | *46%* | *44%* | *42%* | *40%* | *38%* | *36%* | *34%* | |
| Discounted profit (pre-tax) | C$ millions | 39.22 | 9.15 | 5.52 | 6.37 | 11.12 | 13.88 | 19.68 | 13.15 | 11.68 | 22.16 | 20.90 | 204.17 |
| Discount factor @ 8% discount rate | % | 40% | 37% | 34% | 32% | 29% | 27% | 25% | 23% | 21% | 20% | 18% | |
| Discounted profit (pre-tax) | C$ millions | 27.97 | 6.34 | 3.72 | 4.18 | 7.09 | 8.60 | 11.85 | 7.70 | 6.65 | 12.26 | 11.24 | 105.63 |
| Discounted factor @ 10% discount rate | % | 32% | 29% | 26% | 24% | 22% | 20% | 18% | 16% | 15% | 14% | 12% | |
| Discounted profit (pre-tax) | C$ millions | 22.44 | 5.00 | 2.88 | 3.17 | 5.28 | 6.30 | 8.52 | 5.43 | 4.61 | 8.34 | 7.51 | 59.19 |
| I.R.R. (pre-tax) | % | | | | | | | | | | | | |

## TABLE F SUMMARY OF CASH FLOW ANALYSIS FOR THE DESIGN CASE

| Design Case 30,000 tpd | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Total 224,587 million tonnes milled | | | | | | | | | | | | | |
| Life of Mine Cu Grade (%) 0.419 | | | | | | | | | | | | | |
| Life of Mine Au Grade (g/t) 0.330 | | | | | | | | | | | | | |
| Prices Copper (US $/lb) | Gold (US $/oz) | NSR Life of Mine | | I.R.R. (pretax) | Net Present Value - Discount Rate (C$1 = US$0.69) millions | | | | | | | | Capital Payback (years) |
| | | | | | 0% | | 5% | | 8% | | 10% | | |
| | | (C$) | (US$) | (%) | C$ | US$ | C$ | US$ | C$ | US$ | C$ | US$ | |
| 0.75 | 275 | $7.84 | $5.41 | 5.5% | 184.3 | 127.2 | 9.7 | 6.7 | -44.4 | -30.6 | -69.2 | -47.8 | 12 |
| 0.80 | 300 | $8.64 | $5.96 | 10.1% | 363.1 | 250.5 | 116.0 | 80.1 | 37.6 | 25.9 | 0.9 | 0.6 | 7.5 |
| 0.85 | 310 | $9.31 | $6.42 | 13.6% | 510.5 | 352.3 | 204.2 | 140.9 | 105.6 | 72.9 | 59.2 | 40.8 | 6 |
| 0.90 | 325 | $10.02 | $6.91 | 17.3% | 668.4 | 461.2 | 298.4 | 205.9 | 178.3 | 123.0 | 121.5 | 83.8 | 4.5 |
| 0.95 | 350 | $10.83 | $7.47 | 21.2% | 847.2 | 584.6 | 404.7 | 279.2 | 260.2 | 179.6 | 191.6 | 132.2 | 4 |
| 1.00 | 375 | $11.63 | $8.02 | 25.0% | 1026.0 | 707.9 | 511.0 | 352.6 | 342.2 | 236.1 | 261.7 | 180.5 | 3.5 |

## TABLE G SUMMARY OF CASH FLOW ANALYSIS FOR THE MINE/MILL CASE

| Mine/Mill Case 30,000 tpd | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Total 224,587 million tonnes milled | | | | | | | | | | | | | |
| Life of Mine Cu Grade (%) 0.419 | | | | | | | | | | | | | |
| Life of Mine Au Grade (g/t) 0.330 | | | | | | | | | | | | | |
| Prices Copper (US $/lb) | Gold (US $/oz) | NSR Life of Mine | | I.R.R. (pretax) | Net Present Value - Discount Rate (C$1 = US$0.69) millions | | | | | | | | Capital Payback (years) |
| | | | | | 0% | | 5% | | 8% | | 10% | | |
| | | (C$) | (US$) | (%) | C$ | US$ | C$ | US$ | C$ | US$ | C$ | US$ | |
| 0.75 | 275 | $7.84 | $5.41 | 7.9% | 266.9 | 184.2 | 62.2 | 42.9 | -2.1 | -1.4 | -31.9 | -22.0 | 10 |
| 0.80 | 300 | $8.64 | $5.96 | 12.5% | 445.7 | 307.5 | 168.6 | 116.3 | 79.9 | 55.1 | 38.2 | 26.4 | 6.5 |
| 0.85 | 310 | $9.31 | $6.42 | 16.0% | 593.2 | 409.3 | 256.7 | 177.1 | 147.9 | 102.1 | 96.5 | 66.6 | 4 |
| 0.90 | 325 | $10.02 | $6.91 | 19.7% | 751.0 | 518.2 | 350.9 | 242.1 | 220.6 | 152.2 | 158.8 | 109.6 | 4 |
| 0.95 | 350 | $10.83 | $7.47 | 23.7% | 929.8 | 641.6 | 457.2 | 315.5 | 302.6 | 208.8 | 228.9 | 157.9 | 3.5 |
| 1.00 | 375 | $11.63 | $8.02 | 27.6% | 1108.6 | 764.9 | 563.6 | 388.9 | 384.5 | 265.3 | 299.0 | 206.3 | 3.2 |

A discounted cash flow analysis method involves projecting annual revenues and subtracting projected annual capital and operating costs. The resulting net annual cashflows are discounted and totaled to determine Net Present Values (NPV's) at various discount rates. The Internal Rate of Return (IRR) is the discount rate that gives a zero NPV.

The financial analysis for the Study is summarized in Table H and detailed for both cases in Table F and Table G. Throughout the Study, an exchange rate of C$1.00 = US$0.69 has been assumed. All of the cashflow analyses presented are based on pre-tax calculations and have been calculated based on a 224.5 million tonne unsmoothed pit.

**TABLE H FINANCIAL ANALYSES SUMMARY**

| | Design Case | Enhanced Comminution Case |
|---|---|---|
| Pre-production Capital Cost Estimate (including Contingency, EPCM[1], and Indirects) | C$263 M (US$ 182 M) | C$ 252 M (US $174 M) |
| Operating Cost – life of mine | C$5.26 (US$3.63) | C$4.94 (US$3.41) |
| IRR (pre-tax)@ US$0.85/lb Cu, US$310/oz Au | 13.6 % | 16.0% |
| Net Present Value @ 8% Discount | C$ 106 M (US$ 73 M) | C$ 148 M (US$ 102 M) |
| @ 10% Discount | C$ 59 M (US$ 41 M) | C$ 97 M (US$ 67 M) |

Note: 1. EPCM = Engineering, Procurement and Construction Management

The smelter terms and the concentrate transport costs used in the financial analysis are shown in Table I and Table J.

**TABLE I SMELTER TERMS**

| | |
|---|---|
| Copper concentrate pay factor | 1.0 unit deduction |
| Gold pay factor | 97% |
| Silver pay factor | 93% |
| Copper refining charge | U.S.$ 0.08/payable lb. |
| Gold refining charge | U.S.$ 6.00/payable troy ounce |
| Silver refining charge | U.S.$ 0.60/ounce |
| Base treatment charge | U.S.$ 80/D.M.T. |
| + price participation | 10% over U.S.$ 1.00/lb |
| Impurity penalties | Not applicable |

**TABLE J CONCENTRATE TRANSPORTATION COSTS**

| Task | Cost per Wet Metric Tonnes (W.M.T.) | |
|---|---|---|
| Truck from mine site to Stewart | C$ 23.00 | |
| Port and loading charges | C.$5.40 | |
| Subtotal | C.$28.40 | U.S.$ 19.60 |
| Ocean Freight | U.S.$ 20.00 | |
| TOTAL concentrate transport charges | **U.S.$ 39.60** | |

The design includes a 0.8% N.S.R. royalty payable on production. Taxes have not been calculated in the cash flow model and all rates of return and NPV calculations are 'Pre-Tax'. The design case financial analysis has been run on a 100% equity basis (no debt financing included).

The estimated annual net cash flows have been discounted to the beginning of Project Year -2 at real discount rates of 5%, 8% and 10%. Detailed cash cost estimates have been calculated for both case studies and are included in Appendix 6 of the Study. Table K summarizes the results of the analysis. All of the cash costs shown in Table K are calculated per pound of recovered copper with gold credits calculated using a gold price of US$310/oz.

**TABLE K ESTIMATED CASH COST SUMMARY**

| Case | Years | Cash Cost/lb Recovered Copper | Gold Credit at US$310/oz Gold |
|---|---|---|---|
| Design | 1 – 5 | US$0.43 | US$0.22 |
| | Life of Mine | US$0.48 | US$0.25 |
| 'Mine-Mill' | 1 – 5 | US$0.41 | US$0.22 |
| | Life of Mine | US$0.45 | US$0.25 |

## 4. GEOLOGY AND GEOLOGICAL RESOURCES

### 4.1 Geology

The Red Chris mineralization can best be described as a structurally-controlled bulk-tonnage copper-gold porphyry deposit. It is a continuous zone of copper-gold mineralization with an apparent east-northeasterly strike length of approximately 1,700 m and widths ranging from 250 to 700 m. It encompasses two zones of higher grade copper-gold mineralization, centered on the 'Main' and 'East' zones, that are the subjects of recent modeling and resource estimate studies. The closely associated Yellow Chris Deposit includes copper-gold mineralization at the Gully and Far West Deposit.

The Red Chris is hosted by the Red Stock intrusion, which is comprised of two major units. Approximately 80 % of the Red Stock is the quartz diorite porphyry 'Main Phase' unit that hosts most of the known copper-gold mineralization. The 'Late Phase' unit is similar in composition, usually barren of copper-gold mineralization, and represents approximately 10 to 18 percent of the stock.

Pyrite, chalcopyrite and lesser bornite, and covellite are the principal sulfide minerals. The copper sulphides occur as disseminations and fracture fillings within the stockwork. Gold, second in economic importance to copper, occurs as electrum. Microscopic gold grains are intimately associated with the copper sulfides. Silver values are geochemically significant but are of minor economic importance.

The gold-bearing mineralization is intimately associated with the copper mineralization. Copper to gold ratios tend to be constant with depth, but vary laterally decreasing from 1:0.8 to 1:4 (percent copper to grams gold per tonne) in a westward direction. This westward transition of copper-gold ratios is coincident with increased pyritization and decreased bornite to chalcopyrite in the copper mineralization. Thus, it appears that the alteration and mineralization are 'telescoped' along the axis of the Red Stock in a westward direction rather than being equidimensional and equidistant from a more intensely altered and mineralized core.

## 4.2 Geologic Data

The drill hole data base comprises 244 diamond drill holes covering 71,393 m of drilling. The database contains 22,951 copper assays spanning 68,993 m and 22,433 gold assays spanning 67,433 m. Most assay samples are approximately three meters in length. A total of 187 holes were drilled for the Main and East Zones in these zones.

## 4.3 Update of the Geological Resource

As a result of a manual audit of the 1996 block model and resource estimate, an update of the geological resource estimate was undertaken by Giroux and Minorex based on a refined geological model that recognizes the known structural and geological features which control the grade of mineralization within the Main Phase unit. The Main Phase mineralized unit was sub-divided into three separate sub-domains named "Inner Core", "Outer Shell" and "Main Phase" (referring to the remainder of the Main Phase Unit).

Statistical analyses confirmed that this revised modeling exercise provided a better representation of the actual grade distribution than the previous single domain approach in that "*...statistics for each variable (copper and gold) within each geologic domain showed predominately single mineralized populations with low coefficients of variation*". Giroux and Minorex (1998).

A summary of the updated total resource and the resource within each of the geological zones is shown in Table M and Table N. By comparison to the FDW model, the average grade of the Inner Core Zone (I) increased from the previous single Main Phase domain model, and the average grades of the Outer Shell (II) and Main Phase Zone (III) decreased as a result of more constrained grade assessments.

Significantly, the contained metal within the new model decreased relative to previous resource estimates because:

- a smaller area was modeled, and grades outside of this area were set to zero;
- smaller search distances were used than in the previous resource calculation; and,
- boundaries have been imposed on the model, which prevent samples on one side of a boundary being used to estimate blocks on the other side of the boundary (even when samples exist within the allowable search distances).

The results of the new analysis for the inner core zone indicated that the copper grade had increased by an estimated 19% and the gold grade by approximately 20%, as compared to the FDW model, over the same blocks defined by the Inner Core. While the Inner Core has significantly increased in grade, the new model and resource estimate are recognized as being a more accurate and conservative assessment of the global resource than previous estimates.

## 5. MINING

### 5.1 Pit Optimization

Pit optimization resulted in selection of an ultimate pit of 217 million tonnes of ore based on a fixed commodity price of US$0.75/lb copper, US$275/oz gold, and C$3.86 (US$2.35) cut-off NSR. Scheduling and pushback design was found to be critical to the NPV, demonstrating the importance of optimizing grades in early years of production.

**TABLE L IN PIT RESERVES**

| Classification | Million tonnes ore | Copper (%) | Gold (g/tonne) |
|---|---|---|---|
| Measured | 127.6 | 0.390 | 0.305 |
| Indicated | 87.6 | 0.485 | 0.389 |
| Inferred | 1.8 | 0.270 | 0.224 |
| Total | 217.0 | 0.427 | 0.338 |

The stripping ratio of the ultimate pit is 1.40. The total waste rock (plus low grade) in the ultimate pit is 306 million tonnes. Annual scheduling was based on processing 10.95 million tonnes/year ore processed (365 days @ 30,000 tonnes/day), with a maximum production of 80,000 tonnes/day (combined ore and waste), minimum of 3 ore/waste passes in use at all times, and maximum of 8 benches per sub-pit to be mined in one year.

## 5.2 Mine Plan

The Red Chris pit will be developed initially in two starter pits – the Main and East - centered on the higher grade central core of each in the early years and developed into a single pit with three pushbacks over the mine life. The mine plan uses conventional open pit and underground mining techniques, combined in a way to best take advantage of the deposit geometry and site topography. Rather than using conventional shovel and truck fleets to haul from the pit, rock will be conveyed through an adit at the 1200 m elevation from the pit.

The open pit has been designed to attain the highest grade possible in the early years of the mine life, and to accommodate ore pass locations and the corresponding mode of operation of large dozers. Dozers achieve this in the first 5 years of operation and yet hold annual stripping and pre-stripping requirements to a minimum. The technique utilized is relatively new to the industry, however it is directly applicable to the uniqueness of the Red Chris ore zone and geographic setting.

With ore passes within the central portion of each ore zone, dozers have the ability to move down within the ore zone at a rate of up to eight benches/year. This rapid rate of development and ore production can be achieved because:

CAT D11 → D12 (future) (now)

- ore passes are also used as pioneering sinking cuts rather than the traditional sinking cut requirements of open pit mining;
- two ore passes are planned within each (East and Main) ore zone such that development around an ore pass can occur while maximum production is occurring at other ore passes. Ore, low grade, and waste will be mined and moved on a

Eg. Grasberg - 6 ore passes 4.5m x 600m

Eg. Cadomin, Alberta (Ist) 3m x 300m - 2 ore passes campaign basis such that development will not interfere with daily production requirements; and,

- dozers do not need preparatory work prior to operating and therefore allow rapid development of the ore body.

In order to ensure that ore and waste production continuity is realized, the following design and operating characteristics are included:

- all ore and waste passes will be lined;
- there are four ore passes within the two central ore zones;
- a three month inventory of high grade ore is developed in the pre-stripping period to ensure that the mill schedule is protected; and,
- a significant portion of low grade is continuously developed and stockpiled which can be used as emergency feed.

Blasting and fragmentation are critical factors in the materials handling for the Red Chris mine plan. Drill and blast designs were based on achieving high fragmentation to enhance "dozeability" and crusher throughput. Analyses show that the higher costs of drilling and blasting to achieve the above listed priorities are offset by lower dozing, crushing and secondary blasting costs.

Working slopes have been designed with angles of 55° in all rock types except in the Bowser Sediments where a slope of 45° is used. Wall stability benefits from the enhanced dewatering from drainage through underground workings, horizontal drains and surface water management. Use of dozers in the pit provides the opportunity to scale walls and selectively remove wedges or unstable blocks for increased wall stabilization.

Weather conditions at the property are not viewed as being detrimental to normal operating practices. The project, being east of Highway 37, is within an area of moderate precipitation but more extreme temperatures. Winters will be dry and cold, particularly between mid-December and the end of February. The remainder of the year will be moderate.

It will be important to control precipitation and run-off within the pit area, and particularly snow or water that could be channeled into the ore passes. Where necessary, surface ditches will divert surface water away from the pit area. In winter, snow fences will be placed in appropriate locations to intercept drifting snow.

## TABLE M RED CHRIS RESOURCE ESTIMATE SUMMARY

| Cut-off grade Cu (%) | Measured | | | Indicated | | | Measured & Indicated | | | Inferred | | | All Categories | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Tonnes | % Cu | Au (g/t) | Tonnes | % Cu | Au (g/t) | Tonnes | %Cu | Au (g/t) | Tonnes | % Cu | Au (g/t) | Tonnes | % Cu | Au (g/t) |
| TOTAL RESOURCE (ALL 3 GEOLOGICAL ZONES) | | | | | | | | | | | | | | | |
| 0.0% | 230.9 | 0.319 | 0.243 | 379.0 | 0.308 | 0.244 | 609.9 | 0.312 | 0.244 | 243.5 | 0.117 | 0.095 | 853.4 | 0.256 | 0.201 |
| 0.2% | 201.0 | 0.348 | 0.262 | 300.9 | 0.359 | 0.281 | 501.9 | 0.355 | 0.273 | 20.8 | 0.296 | 0.229 | 522.7 | 0.352 | 0.272 |
| 0.4% | 37.1 | 0.644 | 0.502 | 79.5 | 0.583 | 0.479 | 116.6 | 0.602 | 0.486 | 2.8 | 0.497 | 0.426 | 119.3 | 0.600 | 0.485 |

## TABLE N RED CHRIS RESOURCE ESTIMATE BY ZONE

| Cut-off grade Cu (%) | Measured | | | Indicated | | | Measured & Indicated | | | Inferred | | | All Categories | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Tonnes | % Cu | Au (g/t) | Tonnes | % Cu | Au (g/t) | Tonnes | %Cu | Au (g/t) | Tonnes | % Cu | Au (g/t) | Tonnes | % Cu | Au (g/t) |
| INNER CORE ZONE (I) | | | | | | | | | | | | | | | |
| 0.0% | 31.9 | 0.665 | 0.52 | 87.1 | 0.554 | 0.452 | 119.0 | 0.584 | 0.470 | 7.4 | 0.375 | 0.318 | 126.4 | 0.571 | 0.461 |
| 0.2% | 31.8 | 0.667 | 0.521 | 87.1 | 0.554 | 0.452 | 118.9 | 0.584 | 0.470 | 6.6 | 0.398 | 0.338 | 125.5 | 0.574 | 0.463 |
| 0.4% | 28.7 | 0.701 | 0.554 | 72.2 | 0.597 | 0.499 | 101.0 | 0.627 | 0.515 | 2.6 | 0.497 | 0.439 | 103.6 | 0.623 | 0.513 |
| OUTER SHELL ZONE (II) | | | | | | | | | | | | | | | |
| 0.0% | 175.2 | 0.284 | 0.211 | 219.6 | 0.277 | 0.211 | 394.8 | 0.280 | 0.211 | 3.5 | 0.196 | 0.233 | 398.3 | 0.279 | 0.211 |
| 0.2% | 168.2 | 0.289 | 0.213 | 212.3 | 0.28 | 0.212 | 380.5 | 0.284 | 0.212 | 1.2 | 0.238 | 0.248 | 381.7 | 0.284 | 0.213 |
| 0.4% | 8.4 | 0.447 | 0.323 | 7.2 | 0.436 | 0.285 | 15.5 | 0.442 | 0.305 | 0.0 | 0 | 0 | 15.5 | 0.442 | 0.305 |
| MAIN PHASE ZONE (III) | | | | | | | | | | | | | | | |
| 0.0% | 23.9 | 0.114 | 0.11 | 72.2 | 0.108 | 0.094 | 96.1 | 0.109 | 0.098 | 232.6 | 0.107 | 0.086 | 328.7 | 0.108 | 0.090 |
| 0.2% | 1.0 | 0.216 | 0.19 | 1.5 | 0.256 | 0.183 | 2.5 | 0.240 | 0.186 | 12.9 | 0.248 | 0.171 | 15.4 | 0.247 | 0.173 |
| 0.4% | 0.0 | 0 | 0 | 0.1 | 0.549 | 0.261 | 0.1 | 0.549 | 0.261 | 0.2 | 0.497 | 0.253 | 0.3 | 0.514 | 0.256 |

Ditches located on pit wall berms will also collect and channel water to pre-determined drain hole locations and directed to the underground system prior to exiting the mine. All pit water will be collected in a sump for settling, prior to discharge to the mill or the tailings.

Snow that falls within the pit area will be dozed to pre-determined sump locations and the meltwater will be pumped to drain holes. Snow, rainfall and runoff will be controlled as practically as possible to minimize the amount entering operating ore passes.

## 6. METALLURGY AND PROCESS

The Red Chris mill is a conventional flotation mill, designed to process 11 million tonnes per year of ore containing copper, gold, and silver values. The flowsheet is a conventional flotation circuit recovering a copper-gold concentrate in a rougher and 4-stage cleaner/scavenger circuit. Concentrate is thickened and dried prior to shipment by truck to the port in Stewart.

The mill will operate 24 hours per day, 365 days per year with scheduled downtime for maintenance of equipment. Based on a 91% availability the mill will process an average of 30,000 tonnes of ore per calendar day. Design costing for this throughput rate is described as the "Design Case".

The average mill feed grade over the mine life is 0.42% Cu and 0.33 g/t Au. The average head grade over the first five years is higher, at 0.51% Cu and 0.35 g/t Au. Metallurgical testing shows that a concentrate recovering 88.4% of the copper, and 63.1% of the gold can be achieved from this mill feed of the combined East and West pit ores. The concentrate will, on average, grade 26.9% Cu, 14.8 g/t Au and 40.7 g/t Ag. The annual average dry concentrate production is estimated to be 150,800 tonnes of copper/gold concentrate.

Metallurgical testing was completed on composites of drill core from each of the East Pit and the Main Pit zones. Samples were selected from 21 drill holes in the Main and East Pits to represent a range of spatial distribution and assay. Table O summarizes the head grades of each of the two metallurgical composite samples used in the testing.

**TABLE O METALLURGICAL COMPOSITE AND ORE HEAD GRADES**

| Parameter | Unit | Ore Year 1- 5 | Ore Life of Mine | Metallurgical Composite Main Zone Composite | Metallurgical Composite East Zone Composite |
|---|---|---|---|---|---|
| Cu | % | 0.52 | 0.42 | 0.59 | 0.52 |
| Au | g/t | 0.36 | 0.33 | 0.38 | 0.42 |

A Bond ball mill work index of 17.1 kWh/t was used as the basis for design. The design grind for the rougher flotation feed is 80% passing ($P_{80}$) 110 microns.

The metallurgical balance that was used as the basis for design is a calculated average of the locked cycle results for the East and Main Pit composites. The ratio of East to Main pit ores varies over mine life and a locked cycle test on a combined feed test has not been completed to date. Thus, an average of the two locked cycle tests is used as the basis for design as shown in Table P. These calculated feed grades are similar to the projected average head grades for the first five years of 0.51% Cu and 0.35 g/t Au.

Table Q shows the adjusted values for the overall mine life average grades of 0.42% Cu and 0.33 g/t Au.

**TABLE P COMBINED FEED CYCLE TEST AVERAGE**

| Product | Weight | Grade | | Distribution | |
|---|---|---|---|---|---|
| | % | Cu % | Au g/t | % Cu | % Au |
| Feed | 100 | 0.52 | 0.4 | 100 | 100 |
| Rougher Conc. | 15.4 | 3.17 | 2.18 | 93.6 | 83.7 |
| Rougher Tail | 84.7 | 0.04 | 0.08 | 6.4 | 16.4 |
| Cleaner Conc. | 1.71 | 26.9 | 14.8 | 88.4 | 63.1 |
| Cleaner Tail | 13.6 | 0.2 | 0.6 | 5.2 | 20.6 |
| Total Tail | 98.29 | 0.062 | 0.15 | 11.6 | 36.9 |

**TABLE Q OVERALL AVERAGE FOR MINE LIFE**

| Product | Weight | Grade | | Distribution | |
|---|---|---|---|---|---|
| | % | Cu % | Au g/t | % Cu | % Au |
| Feed | 100 | 0.42 | 0.33 | 100 | 100 |
| Rougher Conc. | 15.4 | 2.56 | 1.80 | 93.6 | 83.7 |
| Rougher Tail | 84.7 | 0.03 | 0.07 | 6.4 | 16.4 |
| Cleaner Conc. | 1.4 | 26.9 | 14.8 | 88.4 | 63.1 |
| Cleaner Tail | 13.9 | 0.16 | 0.50 | 5.2 | 20.6 |
| Total Tail | 98.29 | 0.05 | 0.12 | 11.6 | 36.9 |

## 7. ENVIRONMENTAL

In the design of the mining and processing operations for the proposed Red Chris project, one of the key considerations is minimizing the long-term impact of this development on the local area. Thus, as each component was designed, consideration was also given to the decommissioning of the component. The project design principles are three-fold:

- to construct and operate the mine with the least possible environmental impact;
- to incorporate closure considerations from the start of operation so as to minimize the decommissioning required at the end of the mine life; and,
- to build, operate and decommission the mine so as to provide long-term sustainable land use.

Characterization of the existing environment in the project area provides the basis for assessing environmental impact and identifying sensitivities that must be considered in project design. Baseline studies of site and regional meteorology, hydrology, water quality, aquatic resources, wildlife and vegetation were initiated in 1994 and continue to date. Additional baseline programs in some areas will be required to refine the environmental impact assessment for feasibility and project permitting. These baseline studies have been developed from the requirements of the Red Chris Project Report Specifications issued by the B.C. Environmental Assessment Office, June 1996.

Baseline investigations to date show that there are no rare and endangered species that would be affected by the Red Chris project development, no key habitat would be disturbed and that the area of mine development can be limited to one watershed. A monoculture of rainbow trout is the primary fish species identified in the immediate project area. Additional aquatic resource studies are required at the feasibility and permitting level for environmental impact assessment. An additional important consideration is that none of the project developments encroach on the habitat of the important local Stone's sheep population nor on the exclusive archery hunting area for these sheep. However, access, hunting, poaching and displacement must be carefully monitored and controlled during project operation. Preliminary investigations show that land use by local people is primarily wilderness/recreational. ABP has had discussions with the local First Nations about the project development.

Environmental testing of tailings samples from the metallurgical program show that the tailings from the East Pit are net acid consuming with an NP:AP ratio of 4.5:1, however the Main Pit tailings are net acid generating at an NP:AP ratio of 0.6 to 1.

During operation, ore will be produced from both the West and East Pits and processed together such that for most years of operation, the combined tailings would be considered net acid consuming with an NP:AP ratio of 1.5 or greater. The upper layers of tailings in each cell and all exposed tailings surfaces at the end of mine life will have an NP:AP ratio of 2:1 or greater.

Static testing of 969 ore and waste rock samples, combined with the geological and mine development modelling, shows that 82 % of the waste rock is probably or possibly net acid generating at an NP:AP ratio of 3:1 or less. For the purposes of the Study, the conservative assumption has been made that an NP:AP ratio of 3:1 or less defines waste rock that is potentially reactive and requires control measures. Kinetic testing will be done to refine these ABA criteria for "reactive" (acid generating and/or metal leaching) and "non-reactive" waste.

A conceptual reclamation plan has been developed for the Study and continue to be refined during feasibility design and during mine operation. Where possible, reclamation of mine components will begin during operation, for example, cells 1, 2, and 3 of the tailings and waste rock storage facility. This will also allow monitoring of the closure measures, and particularly water quality, during the operational period for refinement of the closure plan.

At closure, all buildings and structures will be removed, foundations broken up or covered and all remaining structures made physically stable. In addition, the waste management facilities are designed to control water quality for the long term with encapsulation of reactive material with alkaline tailings. Assessment of closure for the open pit requires additional testing at the feasibility level. Throughout the mine site there will be reclamation, contouring, and establishment of vegetative cover to restore natural appearance of site area consistent with the surroundings and the existing wildlife and local community needs.

## 8. TAILINGS AND WASTE ROCK MANAGEMENT

The waste management plan for the Red Chris project is based on a combined tailings and waste rock storage facility located east of the pit and mill facility. The general design concept for the tailings and waste rock storage facility is to encapsulate the potentially reactive waste rock in non-acid generating tailings.

The waste containment area will be divided into cells lined with low permeability natural material. Waste rock will be deposited mechanically by stackers into the cells then covered by hydraulically deposited tailings.

A total of four cells have been designed to provide enough storage capacity for all of the tailings and the reactive waste rock, and to provide enough flexibility in the deposition sequence to allow a cover of 30 to 40 m of tailings on top of the waste rock. This represents a significant cover for the reactive waste rock. The tailings cover is expected to remain saturated by capillary rise and act as an oxygen barrier to limit oxidation and acid generation in the reactive waste rock.

For the first four years of operations, the tailings will be deposited in the first cell, while waste rock is placed in the base of Cell #2. After four years of operation, the deposition of waste rock in the second cell ceases, and a third cell is constructed to store waste rock. The tailings lines are then moved to the second cell, and a cover of 30 to 40 m of tailings are deposited on top of the waste rock. This process will be repeated for all subsequent cells. The production of waste rock ceases two years before the end of tailings production. The last cell was sized to ensure that a sufficient cover of tailings could be achieved with the last two years of tailings.

A total of six rock fill embankments or dams will be required as containment for each of the four cells described. All of the exterior dams will be constructed of either non-reactive compacted waste rock fill or cycloned sand using a staged method of construction. Each structure consists of a starter dam and a downstream shell that is raised annually using the centerline method of construction. Due to the pervious nature of the soils, till borrow will be used to line the initial cells, and non-acid generating fine grained sedimentary waste rock will be used when available.

The potential for instability as a result of a PGA of 0.12 g (corresponding to a 10,000 year return period, often considered to be the Maximum Credible Earthquake) is considered to be low. As such, seismicity is presently not considered likely to affect the stability of natural slopes, or man-made earthworks or pit slopes at Red Chris.

The water balance model indicates that with perimeter diversions constructed to minimize catchment areas the overall system (comprised of the tailings disposal areas, the waste rock storage areas, and the area to dam SR7) will generally be in a water deficit position for most conditions with reclaim rates of 80% to 90% achievable during average precipitation conditions. Reclaim to the mill will be maximized. Make-up water will be required with Kluea Lake and the Klappan River being possible sources. At closure, the final containment surface will be graded down from the north toward Kluea Lake.

## 9. ACCESS, ANCILLARY FACILITIES AND SERVICES

The Stewart-Cassiar Highway (Highway 37N) is the main access road north of Meziadin Junction to Dease Lake and the North, and provides connection to the deep water port of Stewart and to the main CN railway at Terrace. The road is, for the most part, paved and is maintained year-round by the Department of Highways. A main connector road of approximately 15 km will provide access from Highway 37 to the project site for transport of concentrates, for supplies and services for the mine, and access from the Iskut Airstrip.

BC Hydro will supply power to the site at 138 kV via a new 240 km line from Meziadin Junction. Current project demands are roughly 30 MW, however BC Hydro indicates that in excess of 40 MW may be available depending on specific line designs and control equipment. There will be a new circuit breaker required at Meziadin to provide service protection to the line to Stewart. The forecast demand loads for the life of mine based on the annual schedule are as follows:

**TABLE R POWER FORECAST SUMMARY**

| Year | Operating kW | Peak kW |
|---|---|---|
| 1 | 23920 | 28518 |
| 2 | 26467 | 29518 |
| 3 – 21 | 28345 | 30076 |

Costing also includes a single facility housing the mine services complex and mine dry, vehicle repair and warehouse facility, operations camp located approximately 1 km to the west of the main complex, the assay, metallurgical and environmental laboratories, and fuel, chemical and reagent storage and handling facilities.

## 10. SOCIOECONOMIC IMPACT

Joining several other successful mining ventures in the general project area, the proposed Red Chris Project will have predominantly positive socio-economic impacts on the project area.

Project construction is estimated to require 27 months to completion. It is anticipated that 420 person years of activity will be required over the construction period. In order to minimize impacts of this activity on local communities, the construction workforce will be housed on-site in a construction camp.

+250 indirect jobs

Operations are projected to require a total direct workforce of 238 people, with 38 in the General and Administration area, 90 in Mill Operations and 110 in Mine Operations. In addition, the project will generate an estimated 40 jobs required for concentrate haul and an estimated 10 additional jobs at the port facility.

Local employment will be maximized in order to optimize the benefits of the project to neighboring communities. The operations workforce will be housed on-site at a camp. Assuming that the company can attract local residents to fill 40% of the workforce (95 people) and can encourage the majority of the remainder of the workforce to commute, the local population should experience growth of approximately 129 people.

It is estimated that the project will generate $15 million annually in direct salaries. In addition, approximately $18 million in indirect and induced salaries will be generated annually. This will produce significant benefits to the local economy of the project region, as employees spend their salary dollars locally. It can also be anticipated that a significant proportion of the annual operational requirements of approximately $4.7 million will benefit the local regional economy.

## 11. PROJECT EXECUTION AND SCHEDULE

A staged project program is proposed for the pre-production development, engineering and construction activities to suit the seasonal construction restrictions and to tie in underground and ore pass development with the overall surface construction. The underground mine development has a duration of 27 months and the mill construction duration is forecast at 20 months. Construction is based on a 54-hour week, and the work hours reported in the capital cost estimate.

## 12. PROJECT OPPORTUNITIES

The Study includes a section on recommendations highlighting areas of additional investigation to further refine the project. The principal project opportunities are summarized below.

### *'Mine-Mill' Enhanced Comminution*

The conceptual mine-mill case is presented in the financial analysis as a result of very recent observations from operating mines around the world, primarily lead by the Julius Kruttschnitt Mineral Research Center, University of Queensland. Their work has shown that improvement in mining operations, specifically fragmentation and gradation, can significantly improve mill throughputs and costs. They have recently observed at several very large operating mines that blasting modifications have allowed mill throughputs to increase by over 25%.

These improvements can be somewhat limited at operating mines by the capital equipment and mine plan already in place – improved blasting may have much smaller improvements in mine costs or may actually increase the mine (as distinct from milling) operating costs. For the Red Chris Pre-Feasibility Study, additional mine operating requirements (and therefore costs) are already included to ensure that materials handling works effectively, that is, to facilitate dozeability, ore pass transfer and to reduce crushing prior to conveying. However, an estimate of the beneficial impact of this 'extra' work in the mine on grinding requirements at the mill has not been included in the Design Case. In the absence of specific methods to quantify the benefit, the conceptual 'Mine-Mill' Enhanced Comminution Case was developed based on the assumptions that the grinding circuit can be reduced by 33%, and, the required power and media for a 20,000 tonne per day grinding circuit will be sufficient to process 30,000 tonnes per day.

(Australia)

The Julius Kruttschnitt Center is compiling a large database of information from mining operations that is providing the basis for estimating techniques to evaluate the improvement in throughput that results from this approach. The Project Team believes that additional work in this area will assist in generating accurate estimates of significant improvement that may be expected in the project economics as a result of optimizing on capital and throughput rates.

### *Reduction in Underground Development and Capital Cost*

A significant degree of conservatism is built into the underground design to ensure the effectiveness of dozer productivity and of the underground materials transport system. The use of twin ore passes in both the East and Main pits for example, could be reduced to a single, perhaps larger capacity ore pass – thus reducing the development and equipment required to be installed.

Also, the selection of Caterpillar D11CD dozers was based on their availability (they are currently in the testing phase), and this has limited the 'reasonable' push distances to ore-passes to approximately 150m. Bigger machines are currently available from other manufacturers and CAT is designing the next larger size (D12) dozer. Internal studies have demonstrated that the dozer application has a positive impact on grade enhancement, pit wall angle, pre-stripping requirements, and grinding benefit. When treating these benefits as a value function coupled with the potential to use larger dozers, the push distances could be up to 400 meters. These larger distances would reduce the number of ore passes required since they could be spaced at greater distances.

### *Waste Management*

The capital cost of waste management for the project represents a significant life-of-mine cost. This cost is based on the design assumptions related to the amount of waste to be stored in the plan. An opportunity to reduce the overall cost may be realized by carrying out additional geochemical testing (kinetic tests) to refine the conservative criteria used currently to define 'reactive' waste rock.

### *Mineable Reserve Estimate*

Opportunities exist to increase the size of the mineable reserve by including the Yellow Chris and Gully Zone resources, and ore within the Red Chris at greater depth. For the purpose of the Study, only the Red Chris deposit was included in the optimization. Also, The Optimization itself was based on 30,000 tonne per day throughput, producing a 20.5 year project. Financial analyses of higher throughput rates would influence the relative value of a larger mineable reserve.

### *Metallurgical Performance*

The metallurgical testwork shows that the Red Chris ore can be processed in a conventional flotation circuit with good copper concentrate recovery and grade. Gold metallurgy has not been investigated to the same extent. The test results and comparisons with other sites clearly indicate that there are opportunities to improve gold recovery and, possibly, copper concentrate grade and recovery.

### *BC Rail Access*

The project is well situated by virtue of its proximity to Highway 37N for trucking of concentrate to the ocean port of Stewart, BC. It is equally well situated to take advantage of access via BC-Rail's Dease Lake Extension (an uncompleted rail grade), constructed in the 1970's. The route passes within 15 km of the project site and the rail head is approximately 150 km south of the area.

The opportunity to make use of the grade as means of transporting concentrate to rail head by truck may provide a low cost alternative to shipping on highway to Stewart. The grade may be considered an off-highway route and, with appropriate crushed rock road bedding, could accommodate trucks with loading in excess of the highway regulations. BC Rail also operates a trucking division, and an expression of interest has been received for complete shipping from the project site to North Vancouver, or connection to the CN line.

BC Rail and the province have made no commitments to complete the rail extension to Dease Lake, however they have contemplated shorter extensions, most recently to Mt. Klappan for access to the Mt. Klappan Coal Project. Other industrial rail users in the corridor may generate enough demand in the future for the completion of the rail line.

### Hydrometallurgical Technologies

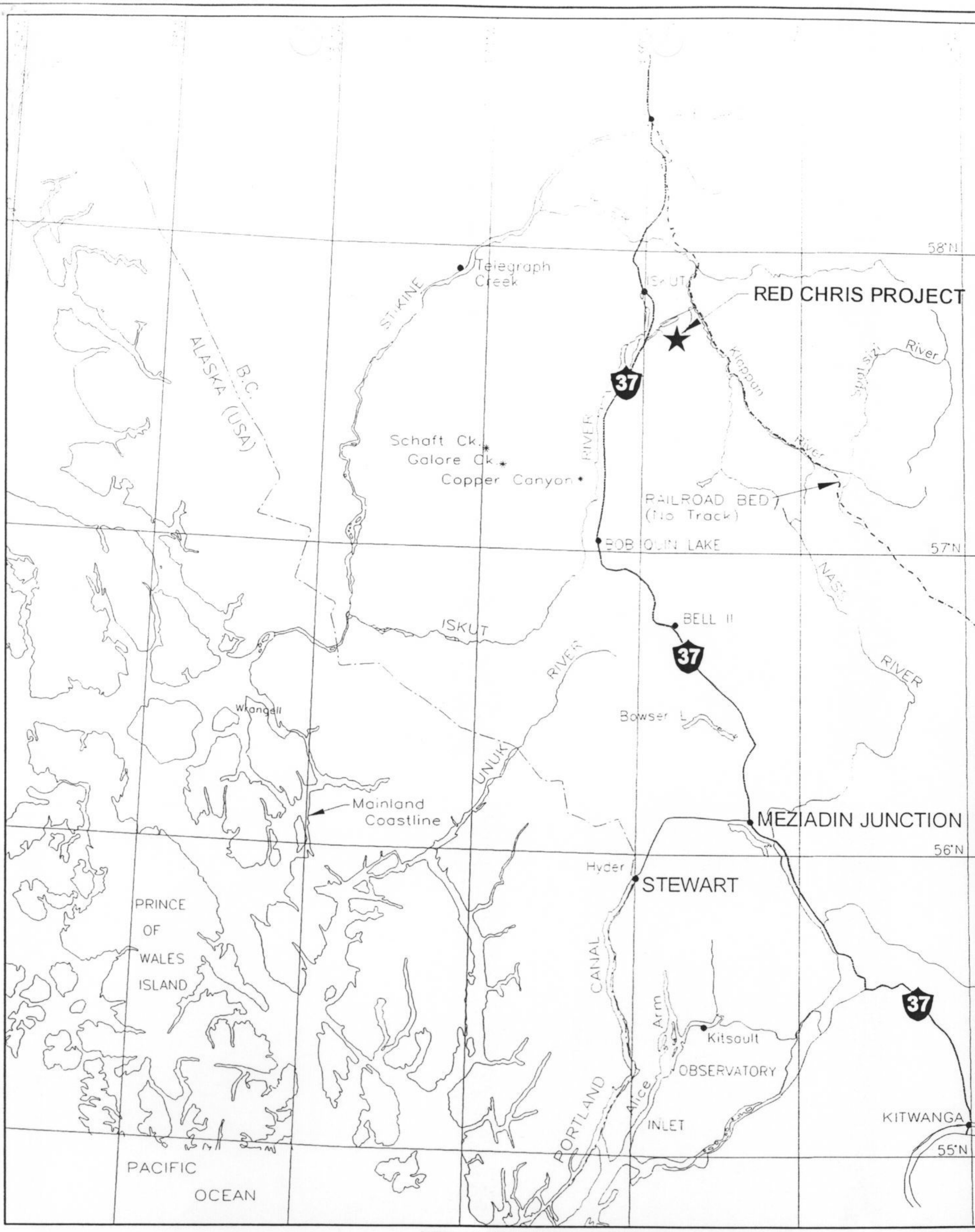
Several hydrometallurgical technologies are being advanced to, or are presently in, the pilot plant phase of their development. It is currently believed that these technologies could allow sulphide copper operations to produce LME (London Metals Exchange) grade copper products on site or at a central toll facility (rather than off-shore shipping to a conventional smelter refinery). Of the potential technologies, only the CESL (Cominco Engineering Services Limited) process and the INTEC process are currently at pilot plant scale. Both technologies could process lower grade concentrates, thereby increasing recovered copper and improving revenues from the mine.

At this time, only INTEC indicates that their plant is likely to be sufficiently 'scalable' to accommodate the amount of concentrate from an individual mine/concentrator like Red Chris. It is likely over the course of the next few years that one or more of these technologies will be put into place, either at, or central to, a supply of copper concentrate. The resulting reduction in shipping and refining costs and increased copper recovery will make significant changes to the cash flow projections for Red Chris.

### BC Rail and Hydrometallurgical Facilities

The potential for a centrally located hydrometallurgical 'toll' facility in the province requiring a source of copper concentrate could soon be a reality. With close proximity to BC Rail for transport within the province, these combined opportunities could significantly affect the project design and cost assumptions. It is expected that centrally located toll facilities (those accepting concentrates from a variety of producers) will be some of the first hydrometallurgical refineries established.

"Power For Jobs" -- proposal to cut costs in ½ (eg. $0.58 → $0.30/k




LEGEND

- • Town
- * Deposit

SCALE: (Approximate)
kilometers
0 40 80

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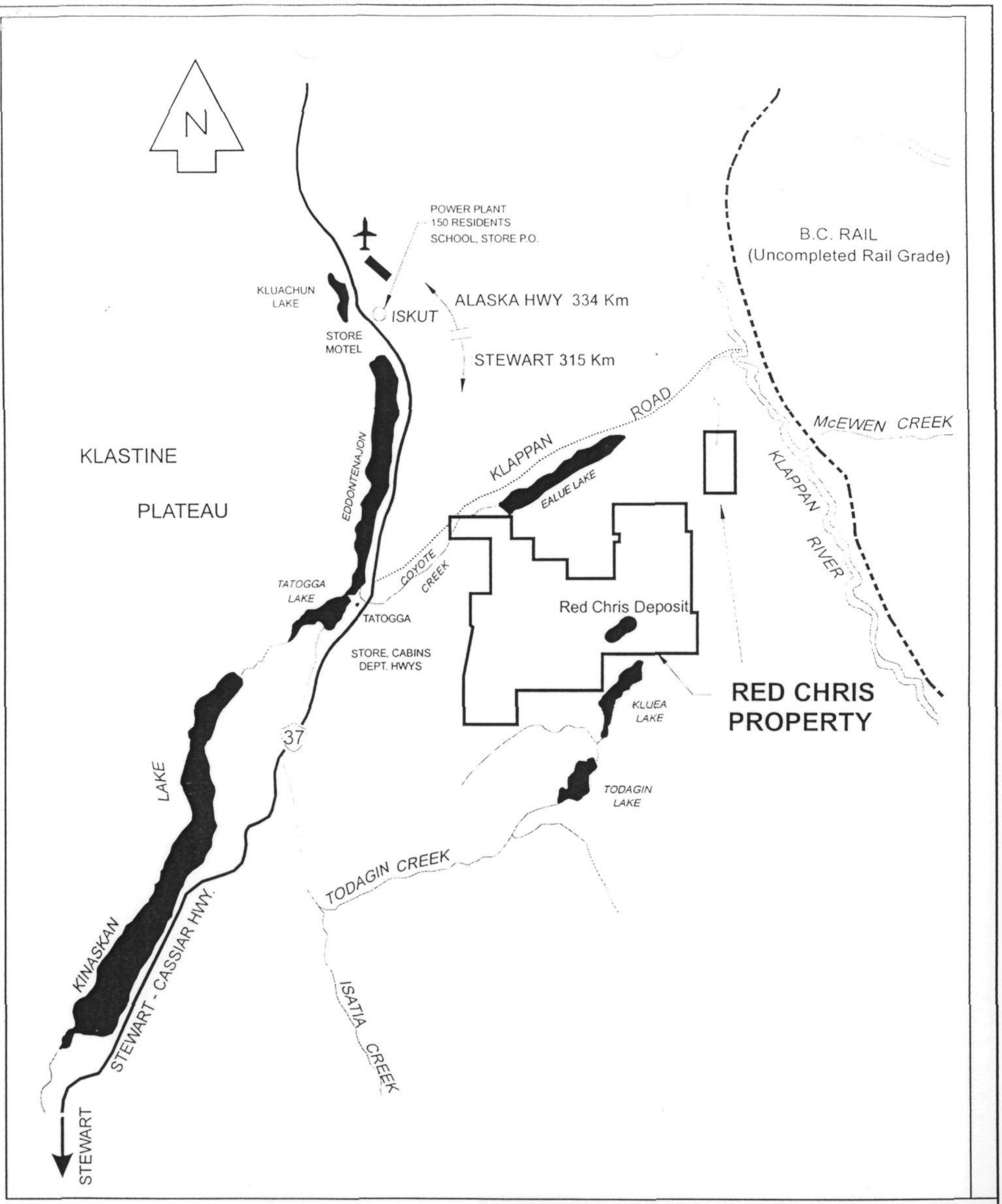
## RED CHRIS PROJECT
## LOCATION MAP

| | | |
|---|---|---|
| Designed By: S.B. | Date July 1998 | Fig: 1 |
| Modified By: hh | File | |




0 5 10
kilometres

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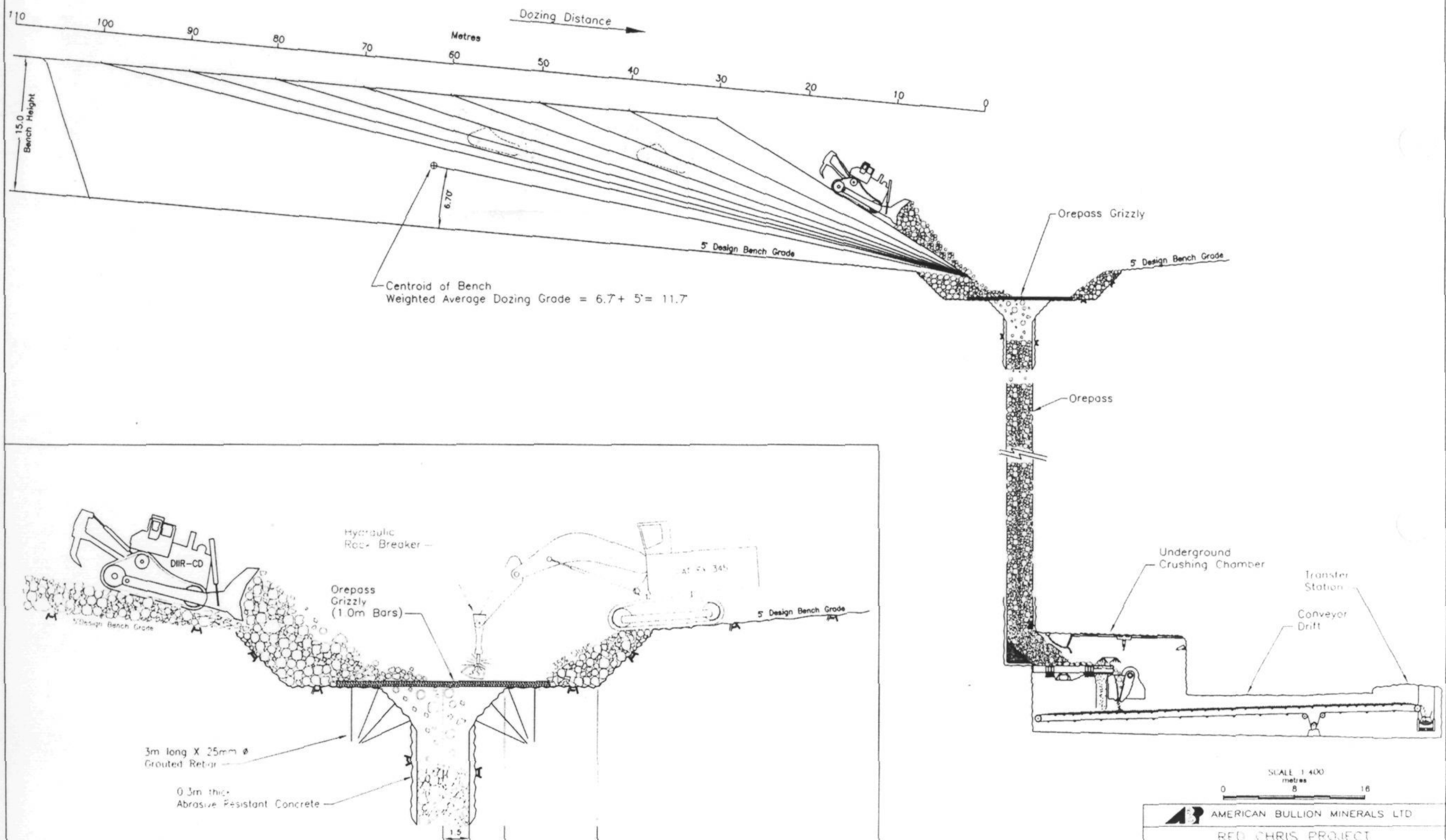
**RED CHRIS PROJECT**
**PROPERTY MAP**

| | | |
|---|---|---|
| Designed By: S.B. | Date: July 1998 | Fig: 2 |
| Modified By: hh | File: | |




NOTE:
1. Rockbreaker to be deployed if and when required.

NOTE:
All dimension are in meters unless otherwise indicated.

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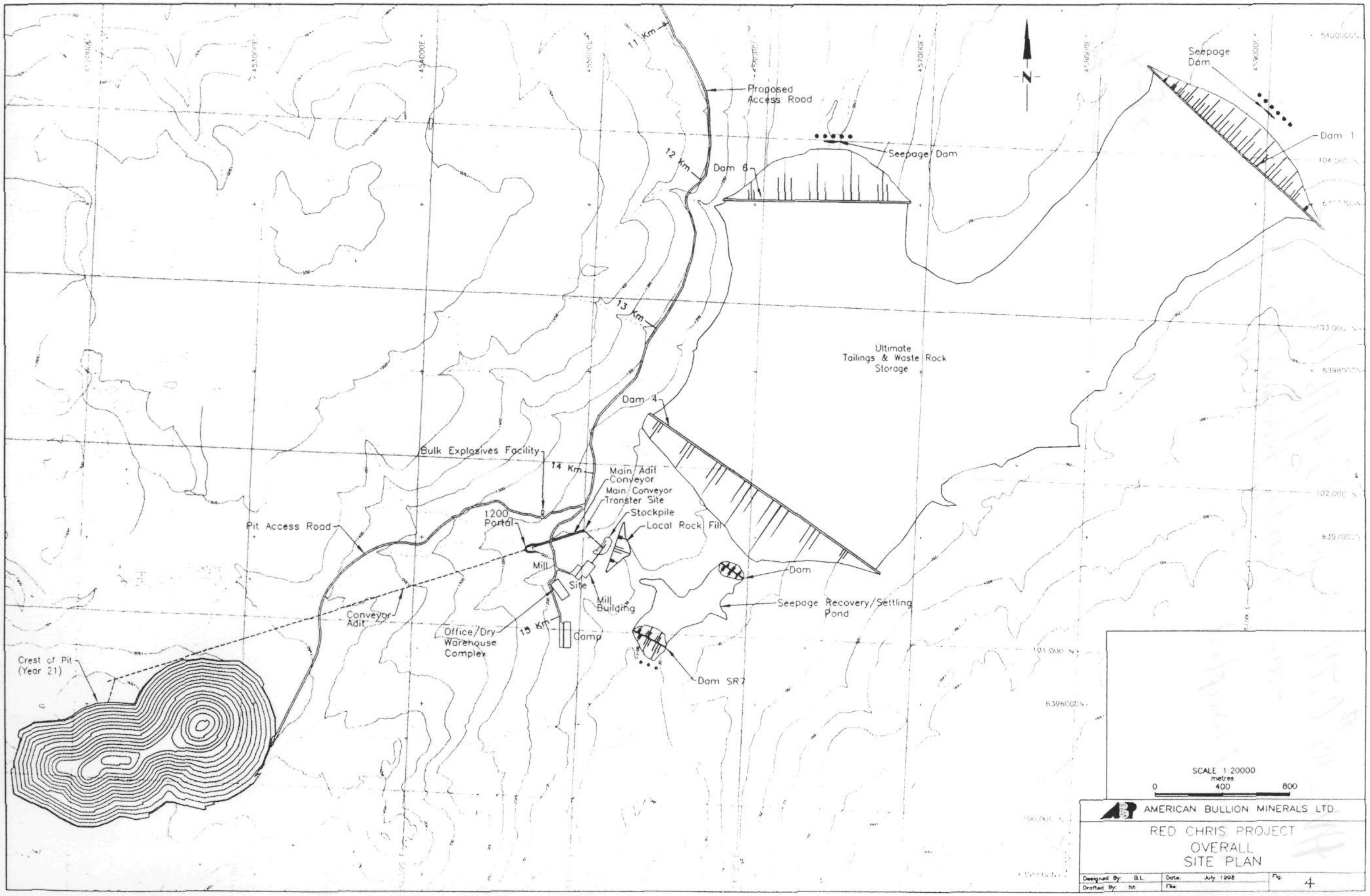
RED CHRIS PROJECT

DOZING TO ORE PASS SCHEMATIC

| | | | |
|---|---|---|---|
| Designed By: B.L. | Date: July 1998 | Fig. | 3 |
| Drafted By: bb | Chk. | | |


11 Km
Proposed Access Road
12 Km
Dam 6
Seepage Dam
Seepage Dam
Dam 1
N
13 Km
Ultimate Tailings & Waste Rock Storage
Dam 4
Bulk Explosives Facility
14 Km
Main Adit Conveyor
Main Conveyor Transfer Site
Stockpile
Local Rock Fill
1200 Portal
Pit Access Road
Mill
Site
Mill Building
Dam
Seepage Recovery/Settling Pond
Conveyor Adit
Office/Dry Warehouse Complex
15 Km
Camp
Dam SR7
Crest of Pit (Year 21)
SCALE 1:20000
metres
0
400
800
AMERICAN BULLION MINERALS LTD.
RED CHRIS PROJECT
OVERALL
SITE PLAN
Designed By: B.L.
Drafted By: hh
Date: July 1998
File:
Fig. 4

HVC - US $0.74 — US $0.62 ¢/lb Cu ✓
- new in-pit blasting technique (Barker)

scoping / conceptual

SEX Golf
→ Barker