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PIT DESIGN AND EVALUATION
OF THE WILLIAMS CREEK COPPER PROPERTY
YUKON TERRITORY

JANUARY 1991

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1.0 INTRODUCTION

At the request of Mr Dale Corman, President of Thermal Exploration Inc. (Thermal), William Hill Mining Consultants Limited (HMC) has completed an engineering study of the Williams Creek copper property located near the town of Carmacks in the Yukon Territory.

The study has been directed particularly at designing an open pit and assessing the potential of the oxide ore reserves to support viable copper production using SX/EW processing. Specific terms of reference given to HMC were to:

(i) estimate the drill-inferred/indicated mineable reserves available for open pit mining and heap leaching of the oxide ore

(ii) design an open pit and sequence production to achieve optimum metal output during the early years of the operation

(iii) estimate preliminary operating costs and cash flows to the end of pit reserves

HMC's work is based on drill logs and sections provided by Thermal which were transferred into HMC's computer system for reserve estimation and pit design.

HMC has relied heavily on reports by Wright Engineers (May, 1990) and Bacon & Donaldson (May 1990) for background data, heap leach and SX/EW design, and capital costs to support this study.

Unless otherwise stated, all dollars are in 1990 constant Canadian terms.

2.0 SUMMARY

HMC has designed an open pit operation to mine the near surface oxide reserves contained in the Williams Creek copper deposit located near the town of Carmacks in the Yukon Territory. The deposit, consisting of oxide and sulphide copper mineralization, has been traced on surface and from diamond drilling for over 2400 ft and to a depth of over 1000 ft below surface; average thickness of the oxide ore is in the order of 100 ft.

HMC's oxide reserves for open pit mining have been calculated from diamond drill and surface trenching information on 5 sections at 400 ft spacing. Using a cutoff grade of 0.5% Cu (based on preliminary operating cost estimates and a metal price of US$1.00 per lb of copper), total in-situ reserves of oxide ore have been estimated at 8.6 million tons of 1.29% Cu down to the contact with sulphide mineralization at approximately 800 ft below surface. In addition, lower grade incremental ore found between the 0.5% cutoff and the limits of mineralization has been estimated at 2.6 million tons at 0.33% Cu.

From the total in-situ reserves of oxide ore, mineable reserves within HMC's designed pit have been estimated at 9.8 million tons at a grade of 1.02% Cu; this includes some 540,000 tons of diluting material, or almost 6% of the in-situ ore, and incremental material below the 0.5% cutoff.

Given the good continuity of copper mineralization found in the deposit, HMC is confident that the tonnages and grades estimated do realistically reflect the reserves available for open pit mining. Additional drilling will however be required to fully upgrade the reserves from the drill-inferred to the drill-indicated category, particularly for the initial stages of mining. HMC has estimated that a drilling program totalling 21,400 ft will be required with approximately 50% of this scheduled for Stages 1 and 2 of the open pit.

The open pit has been designed to mine at a rate of 1 million tons per year of ore at an overall stripping ratio of 2.94 to 1. Mining has been scheduled in two main stages with the first being aimed particularly at the highest grade blocks lying on Section 1600. Stripping ratios will be maintained at less than 1.5:1 during Years 1 and 2, and then at approximately 4:1 by providing a pre-stripping stage and maintaining steady waste removal every year to avoid excessive movements occurring in any particular period. Average production estimates over the life the open pit can be summarised as follows:

YEAR	TREATED (Thous Tons)	WASTE:ORE RATIO	COPPER RECOVERED (Thous Lbs)
Preproduction	279	3.58	1,168
1	1,000	1.09	15,980
2	1,000	1.29	22,100
3	1,000	3.00	25,253
4	1,476	4.19	24,770
5	1,000	4.20	13,090
6	1,000	4.33	11,560
7	1,000	5.51	10,880
8	1,000	3.52	20,230
9	1,031	0.60	23,486
TOTALS	9,786	2.94	168,535

Leaching operations will be maintained over an 8-month year compared to mining operations which should be possible over 10 1/2 months based on HMC's own experience at Curragh's Faro pit. Testwork completed in 1990 returned results indicating that a recovery of 85% over a 33 day cycle can be expected from heap leaching and SX/EW processing of the ore.

HMC has estimated that operating costs can be maintained at approximately US $0.50 per pound of copper produced over the first 4 years of operation and will provide payback of the $41 million capital cost within a 3 - 4 year period at a copper price of US $1.00 per pound. Total cash flow before repayment of capital at this price is estimated at $86.1 million over the 9 years of the pit life.

Using a price of US $1.25 per pound, total cash flow over the pit life is estimated at $135 million which will provide repayment of capital in less than 2½ years.

These cash flows only allow for recovery of the copper content of the ore but not the gold values since at this stage there is insufficient data for evaluation. However, preliminary estimates show that there is a probability that segregated high grade ores could contain up to 50,000 ounces of gold, of which a portion could be recoverable after copper leaching is finished.

3.0 ORE RESERVES

3.1 Geological Background

The Williams Creek copper deposit occurs as a steeply dipping, tabular zone within weakly schistose rock believed to represent a recrystallised roof pendant of volcanic or sedimentary origin. The surrounding country rock consists of granodiorite which provides a sharp footwall contact but a less well defined, gradational hangingwall. The deposit has been traced on surface and by drilling over a 2400 ft strike length and to a depth below surface of over 1000 ft; average thickness of the zone is indicated to be approximately 100 ft.

Mineralization consisting of disseminated bornite, chalcopyrite, and pyrite with the occasional irregular veinlet of bornite/chalcopyrite is best developed on the footwall of the deposit. Minor gold and silver values are also found associated with the copper minerals; molybdenum is found in small quantities associated with the copper mineralization.

3.2 In-situ Reserves

Ore reserves can be divided into sulphides at depth and, closer to surface, the oxide zone which can be further subdivided into:

- a leached zone closest to surface
- an enriched zone below the leached zone

In addition reserves can be further divided into footwall and hangingwall zones corresponding to higher and lower grade respectively.

HMC's oxide ore reserves have been calculated from DDH and surface trenching information on 5 sections as shown in Figures 1 to 5. The reserves were outlined at a 0.5% copper cutoff grade and also at a 0% cutoff to assess the reserve of marginal material available for incremental leaching. All the design, layouts, and calculations were completed using HMC's computer workstation consisting of a Compaq 386, 36*48 inch plotter, digitizer, etc. and Geostat software for reserve estimation using the sectional method.

The key parameters used to calculate the in-situ reserves were as follows:

- polygons around copper intersections at 0.5% copper extended half way to adjacent holes, or surface trench, on section

- on plan, half way rule also used between sections i.e. an average horizontal influence of 400 ft

- tonnage factors: 12.5 for ore and 12.0 for waste (in cu. ft. per ton)

- only oxide ore within any intersection or zone considered

In this way total in-situ reserves of 11.2 million tons at a grade of 1.07% Cu were calculated for the deposit as set out in Exhibit 1.

3.3 Mineable Reserves

Mineable reserves available for mining within the limits of the in-situ reserve were calculated from a pit design with the following basic parameters:

- pit bottom on each section established from incremental strip ratios set out in Exhibit 2 using operating costs taken from the Wright Engineers report of 1990. Therefore pit bottom set at elevation 2240 on sections 800, 1200, and 1600 and at elevation 2550 on section 0.

- overall pit slopes taken at 60 degrees steepening to 70 degrees over the bottom 100 ft of the pit

- ramp width of 60 ft used to allow ample passing room for two, 15 ft wide haul trucks

- ramp grade of 8% used

- a bench height of 50 ft used at this preliminary stage pending additional drill information

Dilution has been calculated by adding a 3ft skin around the ore zones at the 0.5% cutoff. In practice, the easily identifiable green malachite staining at the ore/waste contact will greatly assist in controlling dilution. Waste stringers of dyke rock within ore zones have not been separated out at this stage; in fact, HMC expects that with selective mining this waste material can easily be separated out from the ore. Exhibit 3 summarises dilution estimates.

With overall dilution of 5.8% estimated in this way, HMC's total mineable reserves within the pit design can be summarised as follows:

In-situ (within designed pit)	9.25 million tons 1.05% Cu
Mineable, diluted (within pit)	9.79 million tons 1.02% Cu

Exhibit 4 provides details of the mineable reserves by section.

3.4 Additional Drilling

Given the good continuity of copper mineralization found in the deposit, HMC is confident that the tonnages and grades estimated do realistically reflect the reserves available for open pit mining. Additional drilling will however be required to fully upgrade the reserves from the drill-inferred to the drill-indicated category, particularly for the initial stages of mining.

HMC recommends that a program of drilling be undertaken to provide this information. As a first step, RC holes could be put down next to three or four existing diamond drill holes for comparison of results which would then be followed by a complete program of in-fill RC drilling provided that satisfactory confirmation of DDH results have been obtained from the initial holes. RC drilling may also avoid any losses of copper values suspected from diamond drilling since the copper is found along fractures in the rock and may have been washed away with the drill water. The relatively low number of holes required may, however, justify continuing with diamond drilling.

Figure 6 shows HMC's recommended pattern of drilling to achieve the necessary information and is directed at improving the precision of two blocks of ground:

(i) The area between sections 1400 and 1800 which would be drilled on 200 ft centres with holes at 60 degrees. This block represents the initial, Stages 1 and 2, mining area and would require 29 holes for a total of 5,600 ft.

(ii) Over the balance of the pit ore from sections -200 to 1400, drilling at 200 ft centres should also be adequate for the initial, "first-pass", drilling an estimated at 5,200 ft in 8 holes.

Drilling for defining reserves to the limits of the final pit will then require some 37 holes (10,800 ft). Prior to production an additional 33 definition drill holes will have to be drilled for production control.

4.0 PIT OPERATIONS

Open pit mining of the oxide reserves within the limits of HMC's design will take place in four phases, namely pre-stripping followed by two Stage Pits before reaching the ultimate limits of the Final Pit.

HMC's estimates show that a total of 1 million tons of waste will be removed during the pre-stripping phase which will stretch over a period of 6 months. This material will be used for construction of the leach pads, tailings dams, plant foundations, etc. with any balance being disposed of in waste dump areas close to the pit limits. During this period an estimated 279,000 tons of low grade (<0.5% Cu) will be mined from the pit to provide the first material for crushing and leaching.

The aim of Stage 1 mining will be to access the high grade blocks of ore on Section 1600 which will provide feed for the leaching operation over the initial 3 years of mining. Mining will take place from south to north along strike of the deposit to provide access as quickly as possible to the Stage 1 area. This plan will be assisted by ramping at 10% for access to the deepest ore. Stripping for Stage 2 will commence in the final year of this opening pit during which some 345,000 tons of 0.34% copper will be stockpiled in Year 2 and 131,000 tons at 0.32% Cu in Year 3. In the winter months, a contractor will be hired to crush the stockpiled material and to place it on the pads.

In Stage 2, the pit will be deepened to further access the high grade reserves on section 1600 over the next two years of operation. Stripping will continue through Stage 2 to avoid any excessive requirements for waste movement in the later years of the pit.

The ultimate pit limits will be reached in Year 9 when the total mineable reserves will be exhausted.

HMC suggests that the remaining oxide reserves below the limit of this ultimate pit could be mined as an in-situ leaching operation. A 2200 ft long drift driven from the pit bottom at elevation 2300 down to elevation 2000 would provide undercut access to this ore block which could then be drilled off from surface, loosely blasted and then leached in place. The cost of the development workings underground is estimated at approximately $1.1 million based on a drifting cost of $500 per ft.

Exhibit 5 sets out the schedule of ore and waste moved over the 9-year mine life; details of the scheduling can be found in Appendix I.

Figures 7, 8 and 9 show plans for the 2 stages of pit mining and the final pit outline.

5.0 LEACHING OPERATIONS

The proposed heap leaching and electrowinning operation has been well described in the reports of Wright Engineers and Bacon Donaldson both completed in 1990.

The design concepts for the heap leach and SX/EW treatment are illustrated in Figures 10 and 11 taken from the Wright Engineers report.

Leaching of the oxide ore will take place over an 8 month season while mining is projected on 10 1/2 months operation which is consistent with HMC's experience at Curragh's nearby Faro open pit operation. Two sets of pads will be constructed one each for high and low grade with all the ore being crushed prior to agglomeration and heaping. HMC suggests that consideration be given to spraying concentrated acid onto the ore as it travels from the crusher to the surge stockpile so that there is the maximum possible mixing of acid prior to placement of ore on the pads. It is possible that the gold values in the ore can be recovered from the high grade pads once completely "clean" of recoverable copper (i.e. using standard cyanide leaching methods).

Based on the Bacon Donaldson bottle roll and column leach testwork, HMC has used a 85% recovery factor over a 33 day period for projecting copper metal production.

6.0 OPERATING COSTS

HMC has estimated operating costs based on background data from the Wright's report and estimated productivities, supply costs, etc. taken from HMC's own experience.

The costs have been divided into the following sections:

(i) Total labour for mining, processing, and administration for the site as a whole

(ii) Non-labour mining costs i.e. supplies and materials for equipment, maintenance, and operations

(iii) Consumables required for the heap leach and SX/EW facilities, principally power, acid, and other reagents

(iv) Other operating consumables for the SX/EW facilities

Exhibits 6 through 11 set out HMC's estimates for these operating costs as follows:

Exhibit 6:	Summary of labour costs for the site
7:	Mine department labour costs
8:	Treatment facilities labour costs
9:	Staff and administration labour costs
10:	Mine department materials and supplies
11:	Treatment facilities materials and supplies

In summary, mine operating costs (excluding labour) are expected to rise from $0.62 per ton moved in the initial years of operation to $0.67 in the mid years and then to $0.70 per ton moved until the exhaustion of reserves in year 9.

Total operating costs for the site including both labour and supplies for mining, treatment, and transportation of the cathode production is estimated at $11.2 per ton of ore treated over the 9 year life. The higher cost period will occur in years 6 and 7 when stripping demand will require that a contractor be brought in to handle the additional requirements.

A summary schedule of operating costs is shown in Exhibit 12 together with preliminary cash flows at two metal prices, $1.00 and $1.25 per pound of copper. This Exhibit shows that unit costs can be maintained in the US $0.50 per lb copper range in the first 4 years rising to US $1.00 per lb in Year 7 (the period of high stripping) before falling again below $0.50 per lb through to the end-of-mine life; average cost per lb is estimated at US $0.56 per lb over the 9 year life.

The results of the preliminary cash flow projections (in Thous CDN $) were as follows:

Copper Price (US $/lb)	$1.00	$1.25
Total Cash Flow	$86,066	$135,059
Payback Period[1] (Years)	4.1	2.25
Average Cost/lb (US $)	$0.56	$0.56

Note: 1. Based on a capital cost of $41.7 million (see over).

7.0 CAPITAL COSTS

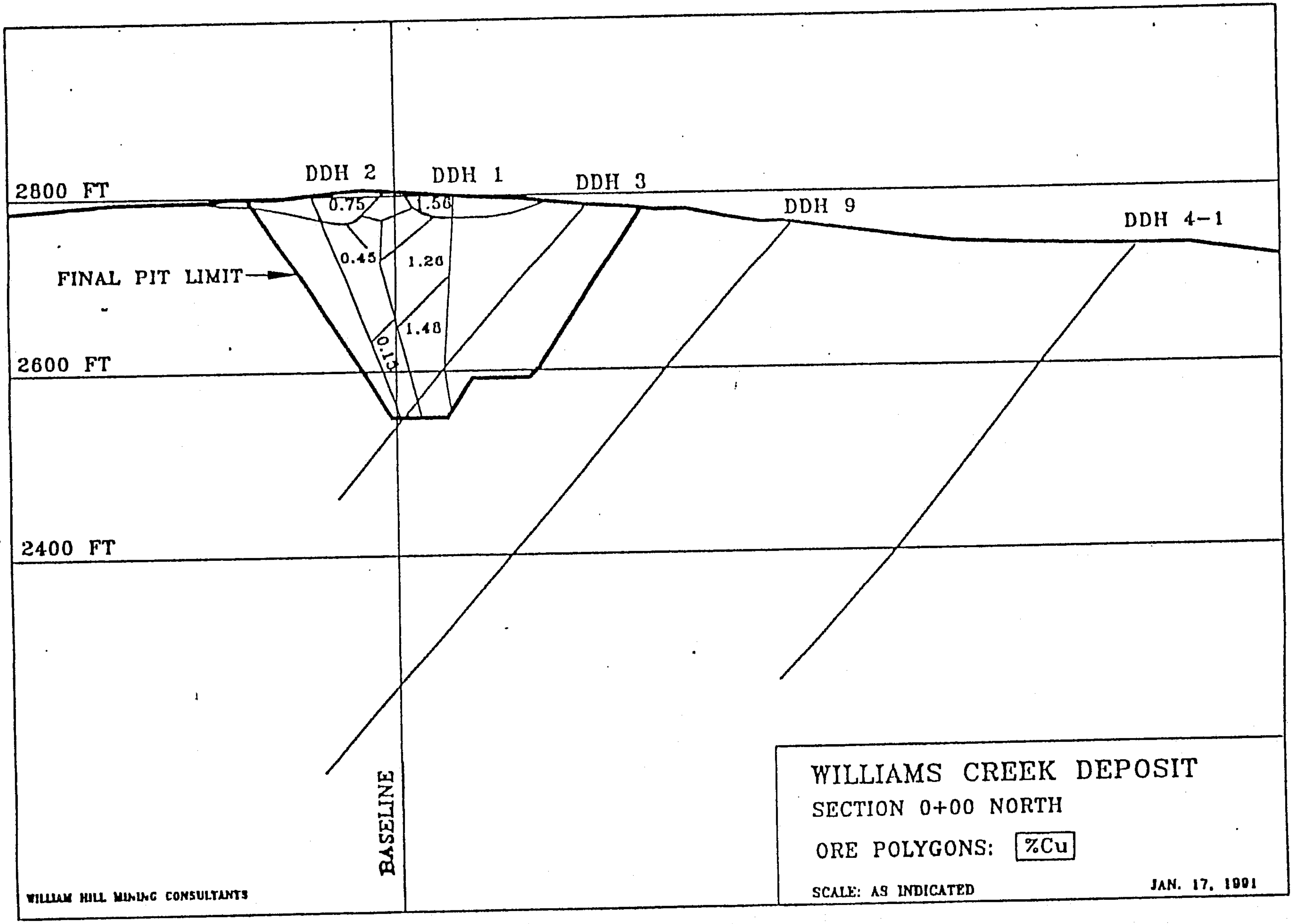
HMC has examined the capital cost estimates included in the Wright Engineers report of May 1990 and considers the figures to be realistic for the Williams Creek operation. In summary, the estimates for the operation are as follows:

Exploration & Site Preparation		$ 2,050,000
Mining	$ 4,807,000	
Mobilization	$ 90,000	
SubTotal - Mining		$ 4,897,000
Heap Construction	$ 3,850,000	
Accommodations	870,000	
Vehicles	229,000	
Processing Plant	18,655,000	
Working Capital	2,225,000	
SubTotal - Treatment		$25,829,000
Inventory & Other		$ 550,000
TOTAL CAPITAL COST		$32,948,000
+ Engineering, Contingency, etc.		$ 8,919,000
TOTAL CAPITAL REQUIREMENT		$41,695,000

Details of this capital cost estimate are included in Appendix II.

FIGURES

FIGURE 1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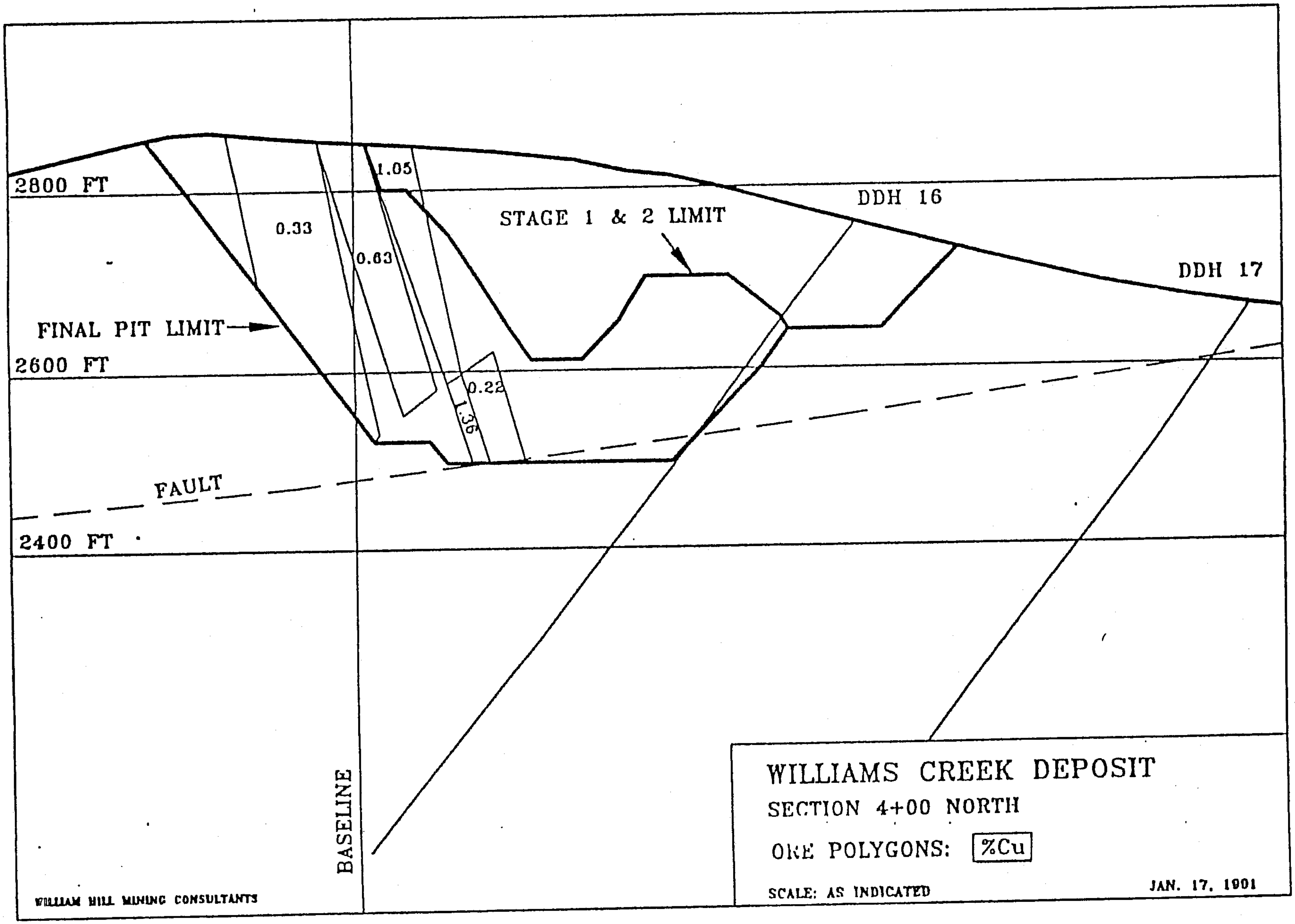
2600 FT
2400 FT
DDH 2
DDH 1
DDH 3
DDH 9
DDH 4-1
0.75
1.56
0.45
1.26
1.48
0.13
FINAL PIT LIMIT
BASELINE
WILLIAMS CREEK DEPOSIT
SECTION 0+00 NORTH
ORE POLYGONS: %Cu
SCALE: AS INDICATED
JAN. 17, 1991
WILLIAM HILL MINING CONSULTANTS

FIGURE 2



2800 FT
2600 FT
2400 FT
1.05
0.33
0.63
0.22
1.36
STAGE 1 & 2 LIMIT
FINAL PIT LIMIT
DDH 16
DDH 17
FAULT
BASELINE
WILLIAMS CREEK DEPOSIT
SECTION 4+00 NORTH
ORE POLYGONS: %Cu
SCALE: AS INDICATED
JAN. 17. 1991
WILLIAM HILL MINING CONSULTANTS

FIGURE 3

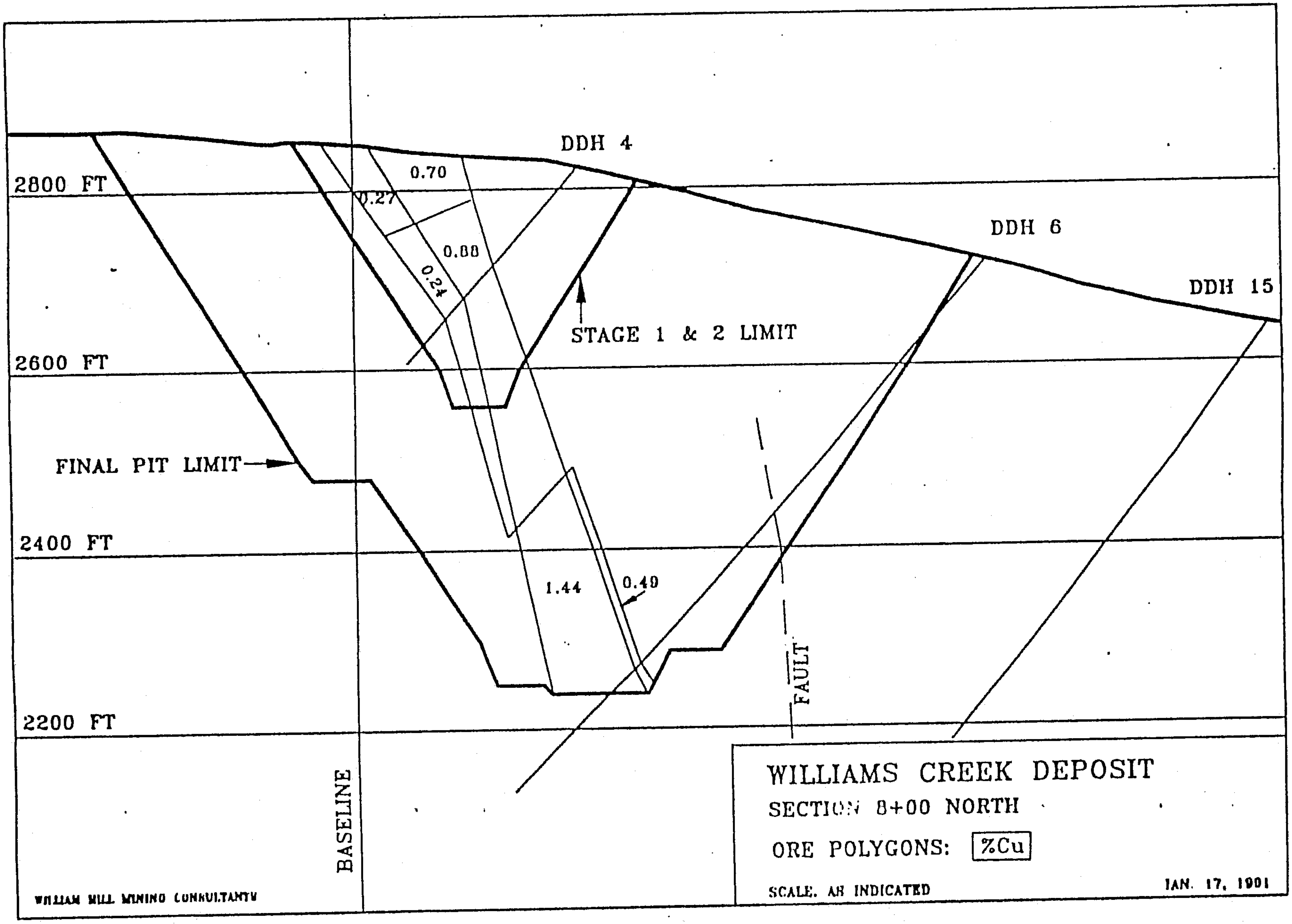
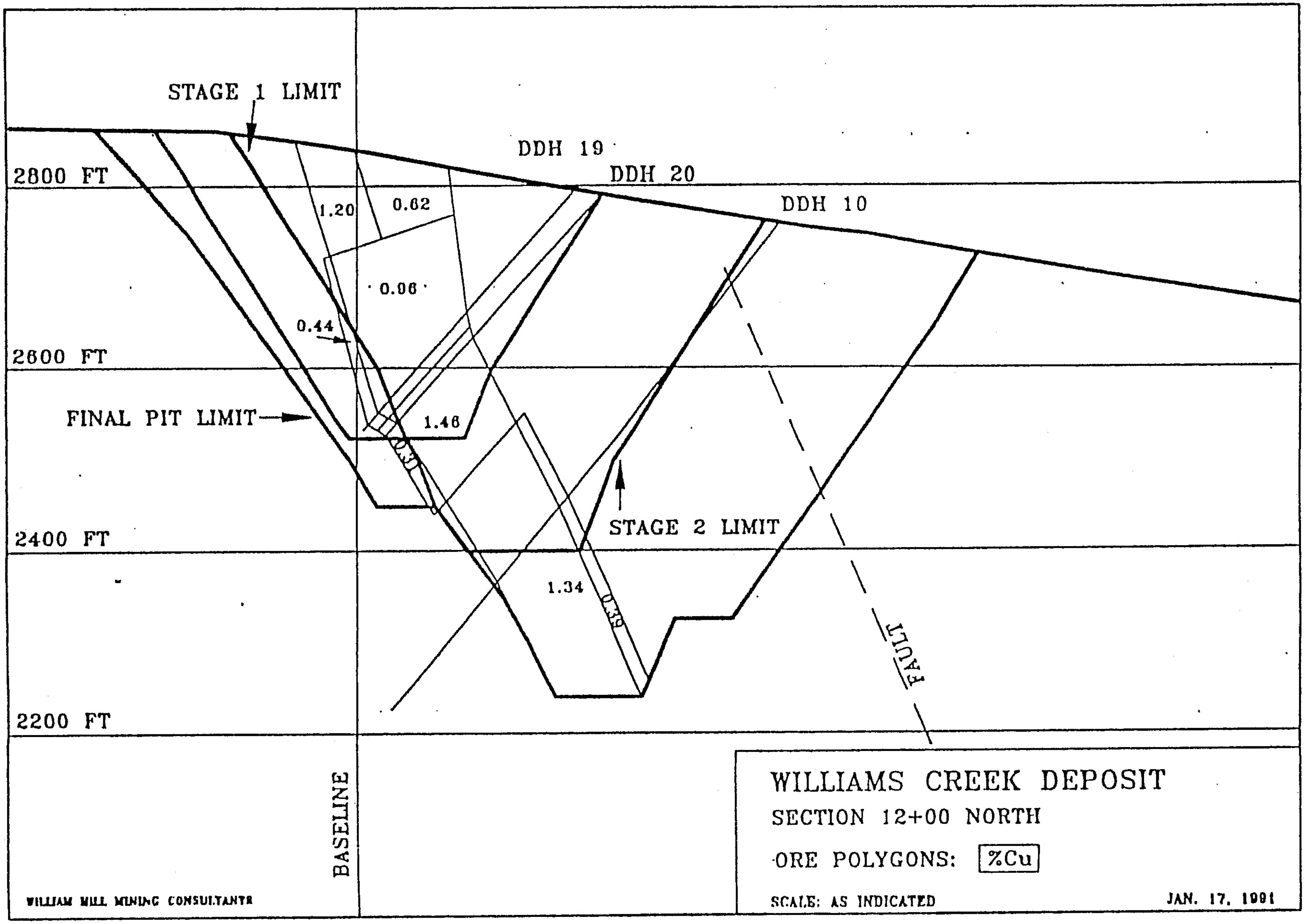

DDH 4
0.70
0.27
0.88
0.24
STAGE 1 & 2 LIMIT
DDH 6
DDH 15
2800 FT
2600 FT
2400 FT
2200 FT
FINAL PIT LIMIT
1.44
0.49
FAULT
BASELINE
WILLIAMS CREEK DEPOSIT
SECTION 8+00 NORTH
ORE POLYGONS: %Cu
SCALE: AS INDICATED
JAN. 17, 1991
WILLIAM HILL MINING CONSULTANTS

FIGURE 4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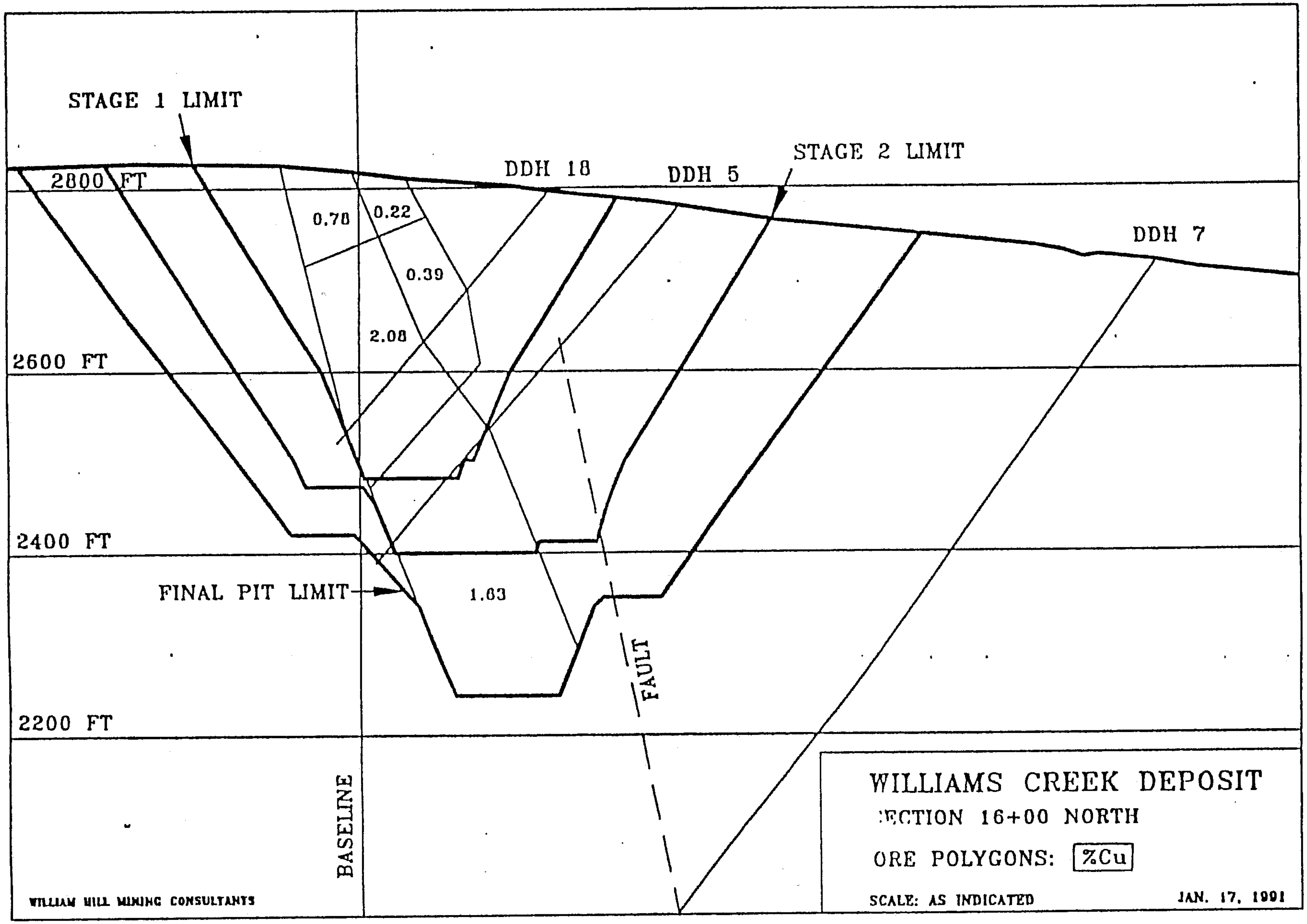
DDH 19
DDH 20
DDH 10
2800 FT
1.20
0.62
0.06
0.44
2600 FT
FINAL PIT LIMIT
1.46
0.31
STAGE 2 LIMIT
2400 FT
1.34
0.39
FAULT
2200 FT
BASELINE
WILLIAMS CREEK DEPOSIT
SECTION 12+00 NORTH
ORE POLYGONS: %Cu
SCALE: AS INDICATED
JAN. 17, 1991
WILLIAM HILL MINING CONSULTANTS

FIGURE 5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DDH 5
STAGE 2 LIMIT
DDH 7
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0.22
0.39
2.08
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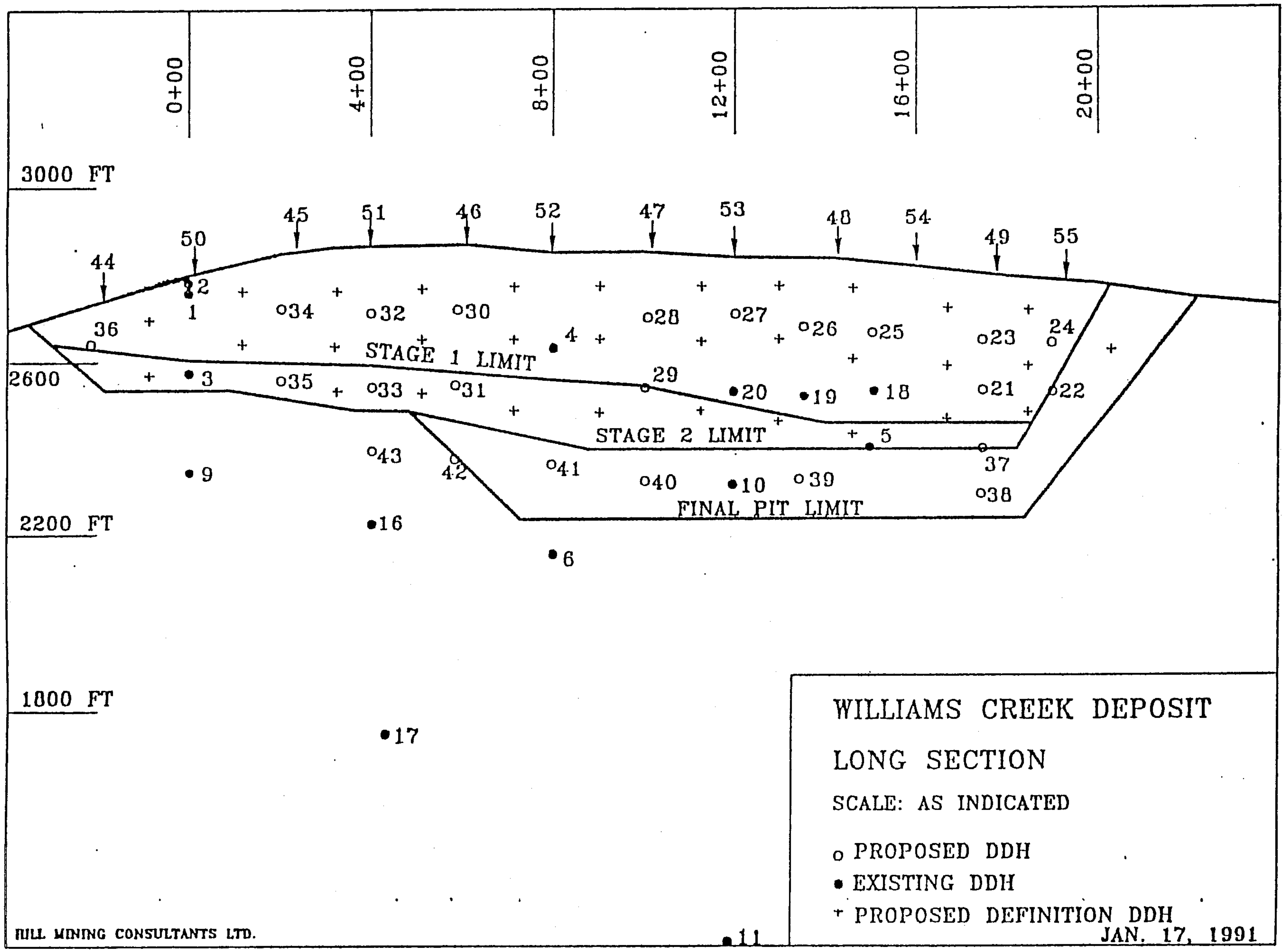
2400 FT
FINAL PIT LIMIT
1.63
FAULT
2200 FT
BASELINE
WILLIAMS CREEK DEPOSIT
SECTION 16+00 NORTH
ORE POLYGONS: %Cu
SCALE: AS INDICATED
JAN. 17, 1991
WILLIAM HILL MINING CONSULTANTS

FIGURE 6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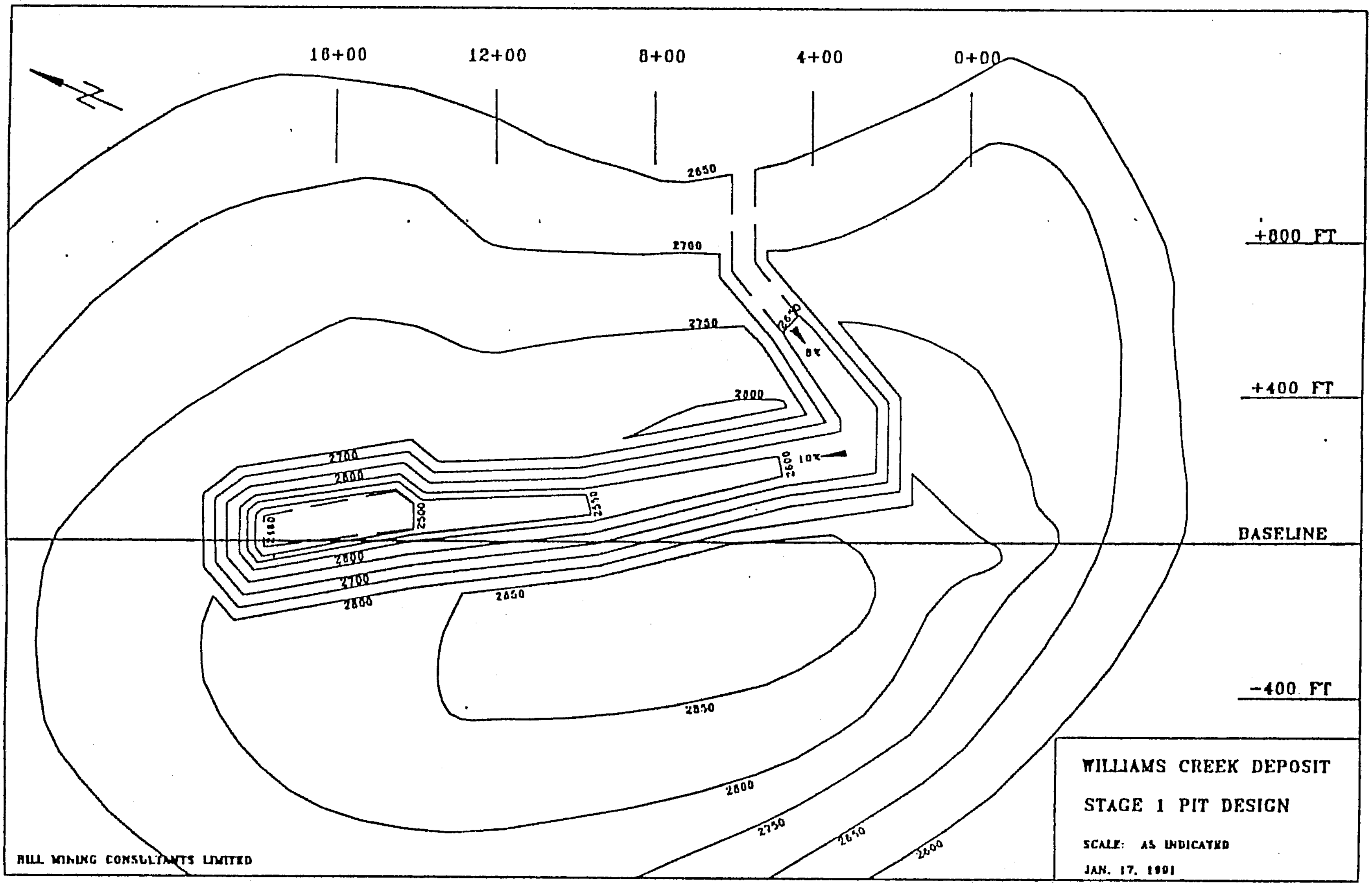
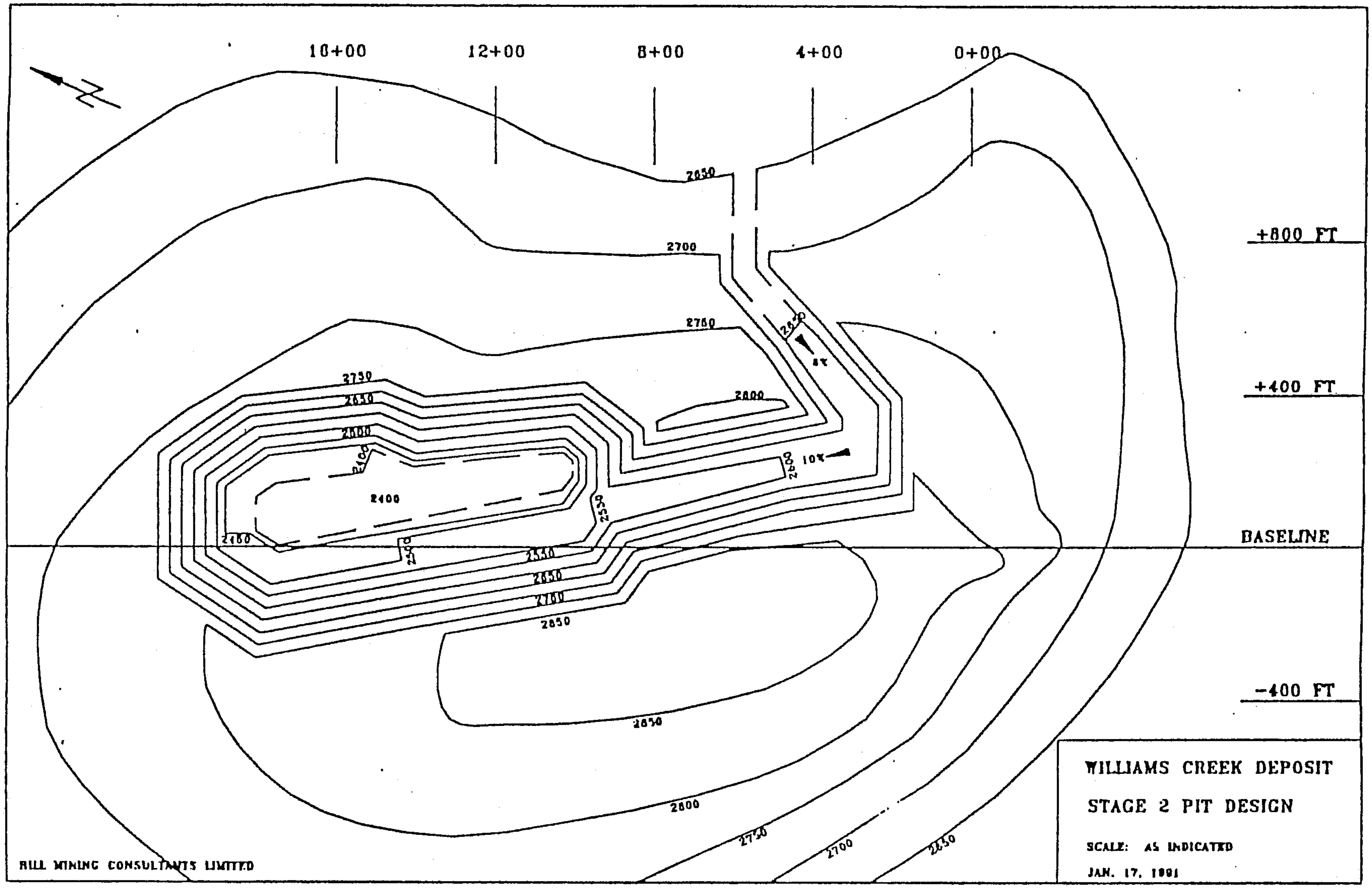
4+00
8+00
12+00
16+00
20+00
3000 FT
2600
2200 FT
1800 FT
STAGE 1 LIMIT
STAGE 2 LIMIT
FINAL PIT LIMIT
WILLIAMS CREEK DEPOSIT
LONG SECTION
SCALE: AS INDICATED
o PROPOSED DDH
• EXISTING DDH
+ PROPOSED DEFINITION DDH
JAN. 17, 1991
HILL MINING CONSULTANTS LTD.

FIGURE 7



16+00
12+00
8+00
4+00
0+00
2650
2700
2750
2800
2700
2600
2550
2500
10%
8%
+800 FT
+400 FT
BASELINE
-400 FT
2850
2800
2750
2650
2600
WILLIAMS CREEK DEPOSIT
STAGE 1 PIT DESIGN
SCALE: AS INDICATED
JAN. 17, 1991
HILL MINING CONSULTANTS LIMITED

FIGURE 8



16+00
12+00
8+00
4+00
0+00
+800 FT
+400 FT
BASELINE
-400 FT
2650
2700
2750
2650
2600
2550
2400
2800
2850
10%
8%
WILLIAMS CREEK DEPOSIT
STAGE 2 PIT DESIGN
SCALE: AS INDICATED
JAN. 17, 1991
HILL MINING CONSULTANTS LIMITED

FIGURE 9

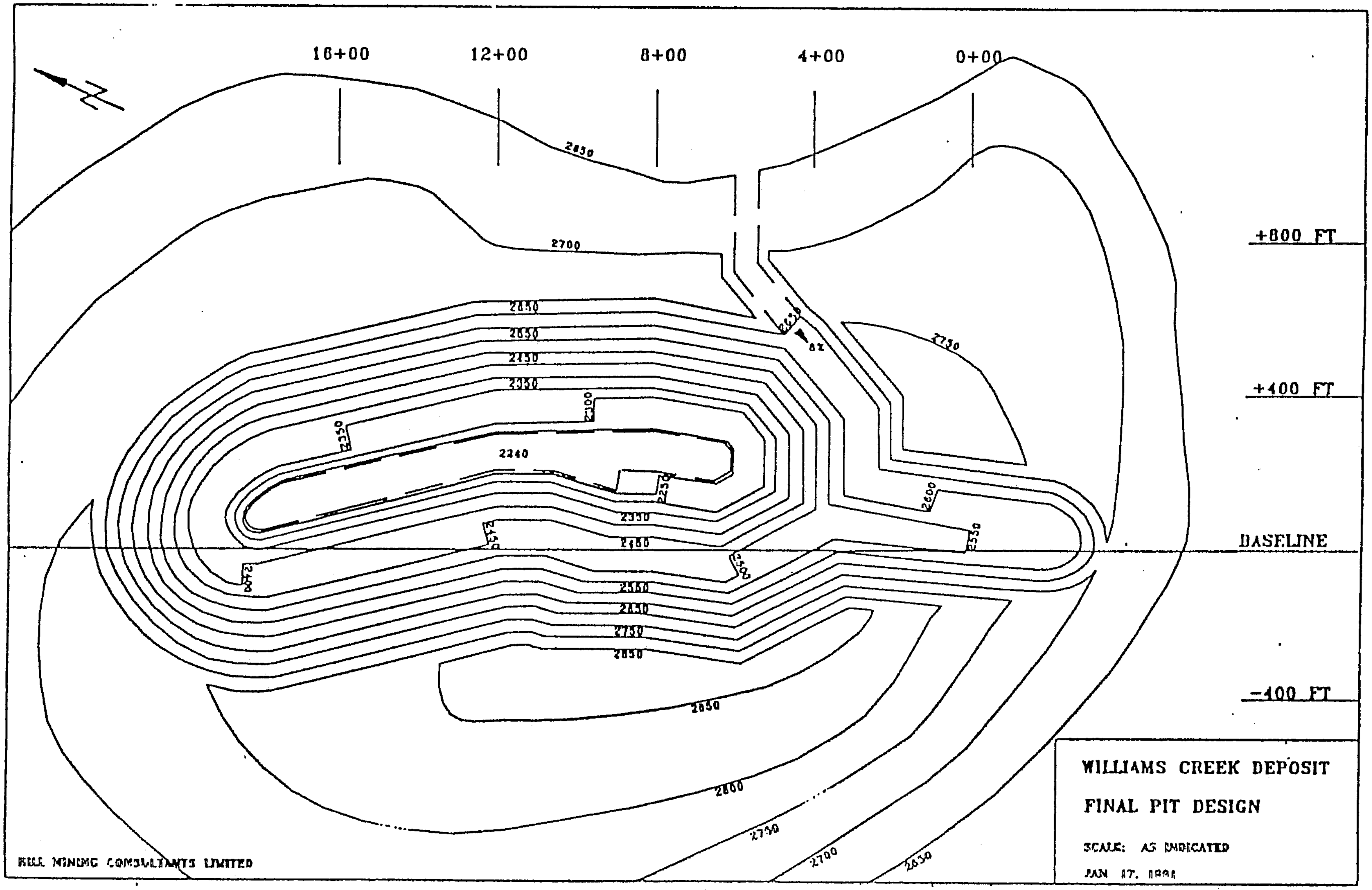
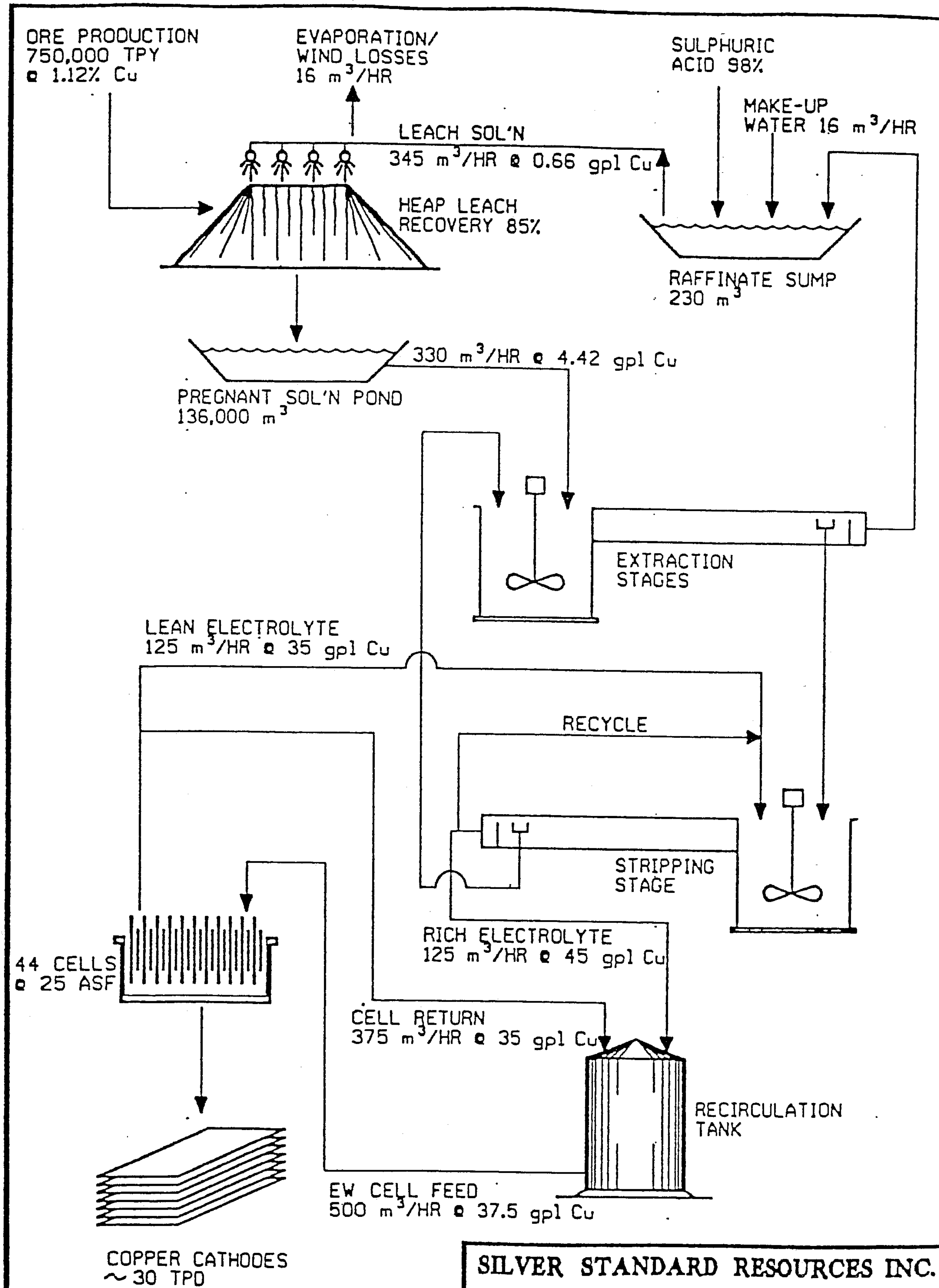
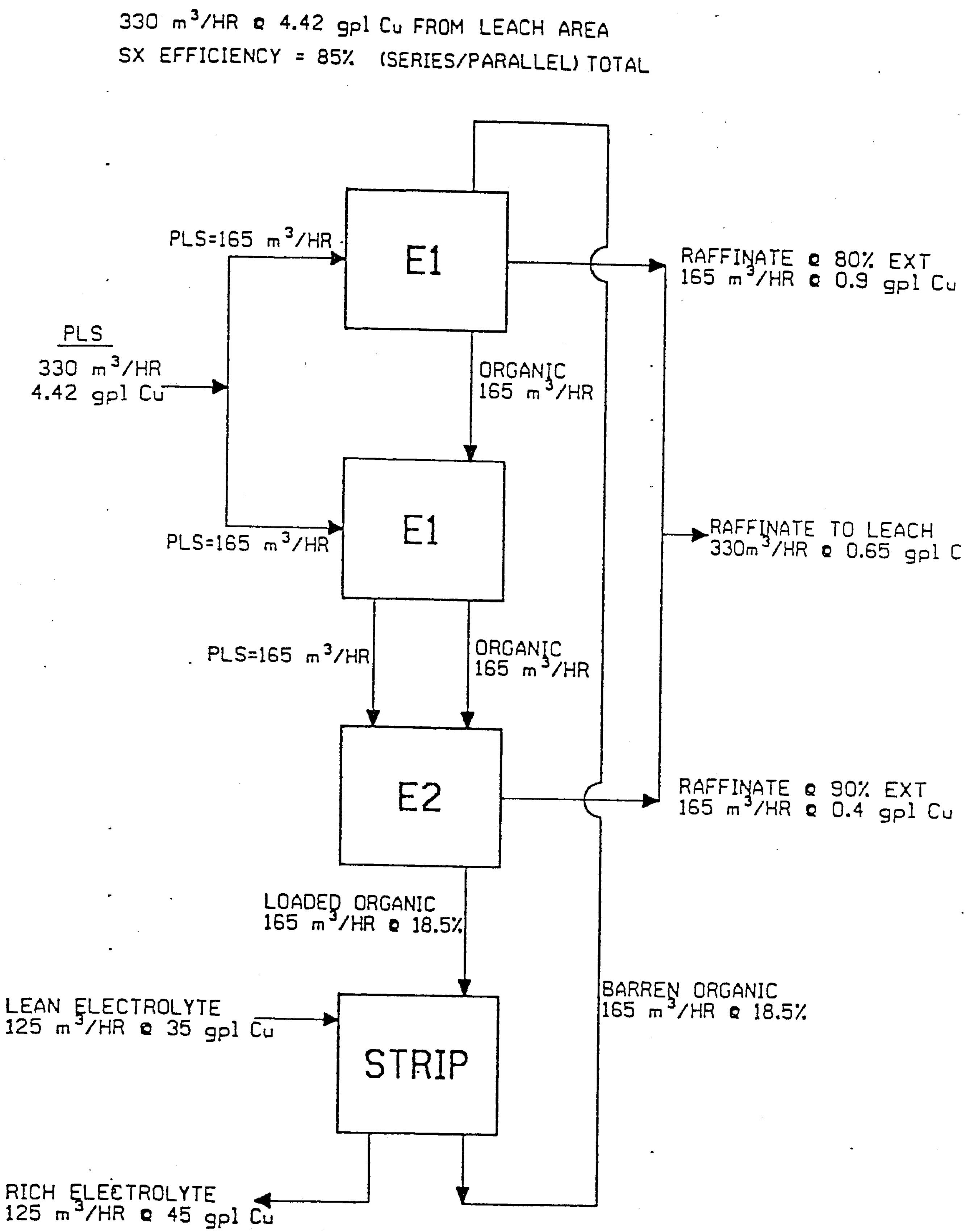

16+00
12+00
8+00
4+00
0+00
2850
2700
2650
2550
2450
2350
2300
2240
2250
2730
8%
2600
2550
2400
2500
+800 FT
+400 FT
BASELINE
-400 FT
WILLIAMS CREEK DEPOSIT
FINAL PIT DESIGN
SCALE: AS INDICATED
KILL MINING CONSULTANTS LIMITED

FIGURE 10



SILVER STANDARD RESOURCES INC.
WILLIAMS CREEK PROJECT
PROCESS FLOWSHEET

FIGURE 11



SILVER STANDARD RESOURCES INC.
WILLIAMS CREEK PROJECT
SOLVENT EXTRACTION

EXHIBITS

EXHIBIT 1

WILLIAMS CREEK

IN-SITU RESERVES

SECTION	+0.5% Cu TONS	+0.5% Cu % CU	0 to 0.49% Cu TONS	0 to 0.49% Cu % CU	TOTAL TONS	TOTAL % CU
1600	2,108,800	1.69	404,000	0.35	2,512,800	1.47
1200	2,830,400	1.22	288,000	0.38	3,118,400	1.14
800	1,956,800	1.19	427,200	0.29	2,384,000	1.03
400	1,034,240	0.97	1,236,800	0.31	2,271,040	0.61
0	670,800	1.17	266,400	0.40	937,200	0.95
TOTAL	8,601,040	1.29	2,622,400	0.33	11,223,440	1.07

EXHIBIT 2

WILLIAMS CREEK

INCREMENTAL STRIP RATIOS

Cost/Tonne Ore	=	$10.76
Price of Copper	=	$ 1.16/lb
1.0% Cu yields	=	22.04 lbs
	=	$25.57

Thus, Breakeven Ore Grade, assuming 75% recovery $= \dfrac{\$10.76}{(\$25.57 \times 0.75)}$

$= 0.56\%$

Cost to mine a ton of waste = $2.00

For Breakeven Strip Ratio of 1:1, the Ore Grade must be:

$$\frac{(\$10.76 + \$2.00)}{(\$25.57 \times 0.75)} = 0.67\%$$

ORE GRADE (% Cu)	INCREMENTAL STRIP RATIO
0.56	0:1[1]
0.67	1:1
0.77	2:1
0.87	3:1
0.98	4:1
1.08	5:1
1.19	6:1
1.29	7:1
1.39	8:1
1.50	9:1
1.60	10:1
1.71	11:1
1.81	12:1
1.91	13:1
2.02	14:1
2.12	15:1

Note: 1. Breakeven.

EXHIBIT 3

WILLIAMS CREEK

DILUTION

(Based on 3'Skin Around Ore)

SECTION	TONNAGE DILUTION	IN-SITU TONNAGE ORE WITHIN FINAL PIT		% DILUTION
		Tons	% Cu	
1600	124,100	2,512,800	1.48	4.9
1200	137,900	2,489,800	5.5	5.5
800	128,200	1,847,500	0.90	6.9
400	95,200	1,711,700	0.55	5.6
0	51,100	688,000	0.94	7.4
TOTAL	536,500	9,249,800	1.05	5.8

EXHIBIT 4

WILLIAMS CREEK

MINEABLE RESERVES

	SECTION 1600		SECTION 1200		SECTION 800		SECTION 400		SECTION 0	
	TONS ORE	% CU	TONS ORE	% CU	TONS ORE	% CU	TONS ORE	% CU	TONS ORE	% CU
	224,000	0.78	192,000	1.20	140,800	0.27	432,000	0.63	68,800	0.75
	96,000	0.22	200,000	0.62	220,800	0.24	291,200	1.05	17,600	1.56
	308,000	0.39	595,200	0.96	161,600	0.70	54,700	1.36	190,400	1.26
	724,800	2.08	369,600	1.46	672,000	0.88	793,000	0.33	160,000	1.48
	1,160,000	1.63	64,000	0.44	80,000	0.49	140,800	0.22	179,200	0.45
			70,400	0.31	572,300	1.44			72,000	0.15
			153,600	0.39						
			845,000	1.34						
	2,512,800	1.48	2,489,800	1.09	1,847,500	0.90	1,711,700	0.55	688,000	0.94
+ Dilution	145,700	0.74	144,400	0.545	107,200	0.45	99,300	0.275	39,900	0.47
TOTAL	2,658,500	1.44	2,634,200	1.06	1,954,700	0.87	1,811,000	0.54	727,900	0.91

Summary: Total Mineable Ore = 9,786,300 t @ 1.02 % Cu
In-situ Ore = 9,249,800 t @ 1.05 % Cu
Dilution = 536,500 t @ 0.525 % Cu

EXHIBIT 5

WILLIAMS CREEK

MINING SCHEDULE[1]

YEAR	TONS ORE (x 10^3)	GRADE (% Cu)	CONTAINED (Lbs Cu x 10^3)	RECOVERED[2] (Lbs Cu x 10^3)	TONS WASTE (x 10^3)	STRIP RATIO (Waste:Ore)	TONS MOVED (x 10^3)
Preproduction	279	0.25	1,395	1,186	1,000	3.58	1,279
1	1,000	0.94	18,800	15,980	1,094	1.09	2,094
2	1,000	1.30	26,000	22,100	1,286	1.29	2,286
3	1,000	1.49	29,709	25,253	3,000	3.00	4,000
4	1,000	1.30	26,000	22,100	4,188	4.19	5,188
4a	476	0.33	3,142	2,670	-	-	476
5	1,000	0.77	15,400	13,090	4,198	4.20	5,198
6	1,000	0.68	13,600	11,560	4,330	4.33	5,330
7	1,000	0.64	12,800	10,880	5,509	5.51	6,509
8	1,000	1.19	23,800	20,230	3,522	3.52	4,522
9	1,031	1.34	27,631	23,486	616	0.60	1,647
TOTAL	9,786	1.02	198,277	168,535	28,743	2.94	38,529

Note:
1. Smoothed compared to details shown in Appendix I.
2. 85% Recovery.

EXHIBIT 6

WILLIAMS CREEK

SUMMARY OF LABOUR & SUPERVISORY COSTS

YEAR	PREPRODUCTION	1 TO 3	4 TO 5	6 TO 9
Mine Labour	$526,000[1]	$1,052,000	$1,426,000	$1,426,000
Plant Labour	-	$1,133,000	$1,133,000	$1,133,000
Supervisory & Administration	-	$1,010,000	$1,010,000	$1,010,000
TOTALS	$526,000	$3,195,000	$3,569,000	$3,569,000

Note: 1. Cost capitalized. Supervised by construction supervisors.

EXHIBIT 7

WILLIAMS CREEK

MINE LABOUR COSTS

1. Operating Labour Costs

	MANPOWER REQUIREMENTS		
	PREPRODUCTION TO YEAR 3	YEARS 4 TO 5	YEARS 6 TO 9
Drilling	2	2	2
Blasting	1	1	1
Loading	3	4	4
Hauling	4	8	8
Dozers	3	4	4
Grader	1	1	1
Dumping & Leach Loading	2	2	2
Total Men	16	22	22
Yearly Cost/Man	$ 46,000	$ 46,000	$ 46,000
TOTAL COST PER YEAR	$736,000	$1,012,000	$1,012,000

2. Mine Maintenance Labour

	PREPRODUCTION TO YEAR 3	YEARS 4 TO 5	YEARS 6 TO 9
Men Required	6	8	8
Cost Per Year Per Man	$ 51,750	$ 51,750	$ 51,750
Total Cost Per Year	$316,500	$316,500	$316,500

3. Total Cost of Mining Labour

PERIOD	MINE	MAINTENANCE	TOTAL
Preproduction[1]	$ 368,000	$158,250	$ 526,250
Years 1 to 3	$ 736,000	$316,500	$1,052,500
Years 4 to 5	$1,012,000	$414,000	$1,426,000
Years 6 to 9	$1,012,000	$414,000	$1,426,000

Note: 1. For 6 months only.

EXHIBIT 8

WILLIAMS CREEK

TREATMENT FACILITIES LABOUR

(Based on 8 Months Operation/Year)

1. Plant Operating Labour

	MEN
Crusher	6
SX/EW Plant	8
Leaching	7
Laboratory	2
Total Men	23
Yearly Cost Per Man	$ 31,000
Total Cost Per Year	$ 713,000

2. Plant Maintenance Labour

Crusher	8
SX/EW Plant	4
Total Men	12
Yearly Cost Per Man	$ 35,000
Total Cost Per Year	$ 420,000
TOTAL COST	$1,133,000

EXHIBIT 9

WILLIAMS CREEK

ADMINISTRATION, SUPERVISION & TECHNICAL MANPOWER COSTS

		SALARY RATE	TOTAL COST
Mine Manager	1	$110,000	$ 110,000
Mine Superintendent	1	$ 80,000	$ 80,000
Mine Foreman	2	$ 70,000	$ 140,000
Engineers	3	$ 50,000	$ 150,000
Plant Superintendent	1	$ 80,000	$ 80,000
Foremen	4	$ 50,000	$ 200,000
Assayer	1	$ 50,000	$ 50,000
Clerical & Support Staff	4	$ 50,000	$ 200,000
TOTAL COST			$1,010,000

EXHIBIT 10

WILLIAMS CREEK

OPERATING EQUIPMENT & SUPPLY COSTS PER TON MOVED

	PREPRODUCTION	YEARS 1 TO 3	YEARS 4 TO 5	YEARS 6 TO 9
Drilling	0.09	0.09	0.09	0.09
Blasting	0.09	0.09	0.09	0.09
Loading	0.10	0.10	0.10	0.10
Hauling	0.14	0.14	0.18	0.20
Roads & Dumps	0.10	0.10	0.11	0.12
Leach Pads Service	0.05	0.05	0.05	0.05
SubTotal	0.57	0.57	0.62	0.65
Other	0.05	0.05	0.05	0.05
TOTAL	0.62	0.62	0.67	0.70

EXHIBIT 11

WILLIAMS CREEK

TREATMENT COSTS

1. Acids and Reagents

Bacon Donaldson tests indicate that about 3.5 pounds of acid will be required to extract each pound of copper.

It is proposed that sulphur will be purchased at Fort Nelson at a cost of about $60 per ton. Transport to site is about 400 miles at a cost of about 15¢ per ton mile. It is assumed that the acid will be about $200.00 delivered at site or roughly 3.5¢ per pound of acid based on 1 pound of sulphur generating 3 pounds of acid.

Acid will be a fixed cost of about 10¢ per pound of copper.

Reagents, primarily organics, will be about 4¢ per pound of copper.

2. Power

Each pound of copper will require about 1.0 kilowatt hour (KWH) of electricity per pound of copper in the electrowinning process, and about 0.6 KWH for all other functions including pumping of solution or roughly 1.6 KWH per pound total.

It is assumed that power costs at site will be about 6¢ per KWH, or roughly 10¢ per pound of copper.

3. Other Costs

The balance of the plant costs are estimated at about $1,000,000 per year as follows:

Maintenance	$ 500,000
Other & Miscellaneous	$ 500,000
	$1,000,000

4. Transport Costs (Copper Metal)

Transport to market is estimated at about $0.05 per pound of copper.

EXHIBIT 17

WILLIAMS CREEK PROPERTY

SCHEDULE OF PRODUCTION, COSTS, & CASH FLOW
(Thous C$ & Tons)

ACTIVITY		PRE-PRODUCTION	YEAR 1	2	3	4	5	6	7	8	9	TOTAL
TOTAL MOVED BY COMPANY		1,279	2,094	2,206	4,000	5,664	5,198	4,500	4,500	4,522	1,647	35,690
MOVED BY CONTRACTOR								830	2,009			2,839
TOTAL TREATED		279	1,000	1,000	1,000	1,476	1,000	1,000	1,000	1,000	1,031	9,786
TOTAL COPPER PROD'N	thous lbs		17,166	22,100	25,253	24,770	13,090	11,560	10,880	20,230	23,486	168,535
OPERATING COSTS:												
Mining Non-Labour Unit Cost, $/t		$0.62	$0.62	$0.62	$0.62	$0.67	$0.67	$0.67	$0.70	$0.70	$0.70	
MINING TOTAL NON-LABOUR		$793	$1,298	$1,417	$2,480	$3,795	$3,483	$3,015	$3,150	$3,165	$1,153	$23,749
CONTRACT MINING @ $/t	$1.10							$913	$2,210			$3,123
SX/EW Consumables @ $/lb	$0.20	$0	$3,433	$4,420	$5,051	$4,954	$2,618	$2,312	$2,176	$4,046	$4,697	$33,707
SX/EW Other Operating		$0	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$1,000	$9,000
SX/EW TOTAL NON-LABOUR		$0	$4,433	$5,420	$6,051	$5,954	$3,618	$3,312	$3,176	$5,046	$5,697	$42,707
TRANSFORTATION @ $/lb	$0.05	$0	$858	$1,105	$1,263	$1,239	$655	$578	$544	$1,012	$1,174	$8,427
MANPOWER (Total Site)		$526	$3,195	$3,195	$3,569	$3,569	$3,569	$3,569	$3,569	$3,569	$3,569	$31,899
TOTAL OPERATING COSTS		$1,319	$9,785	$11,137	$13,362	$14,556	$11,324	$11,387	$12,649	$12,792	$11,593	$109,905
Cost/Lb Copper		$0.00	$0.57	$0.50	$0.53	$0.59	$0.87	$0.99	$1.16	$0.63	$0.49	$0.65
In US$		$0.00	$0.49	$0.43	$0.46	$0.51	$0.74	$0.85	$1.00	$0.54	$0.42	$0.56
CASH FLOW PROJECTIONS												
REVENUE, Thous C$ @ US$	$1.00	$0	$19,960	$25,698	$29,364	$28,802	$15,221	$13,442	$12,651	$23,523	$27,309	$195,971
OPERATING COSTS		$1,319	$9,785	$11,137	$13,362	$14,556	$11,324	$11,387	$12,649	$12,792	$11,593	$109,905
CASH FLOW		($1,319)	$10,176	$14,560	$16,002	$14,246	$3,897	$2,055	$2	$10,731	$15,716	$86,066
REVENUE, Thous C$ @ US$	$1.25	$0	$24,951	$32,122	$36,705	$36,003	$19,026	$16,802	$15,814	$29,404	$34,137	$244,964
OPERATING COSTS		$1,319	$9,785	$11,137	$13,362	$14,556	$11,324	$11,387	$12,649	$12,792	$11,593	$109,905
CASH FLOW		($1,319)	$15,166	$20,985	$23,343	$21,447	$7,702	$5,415	$3,165	$16,612	$22,543	$135,059

APPENDICES

APPENDIX I

DETAILED SCHEDULES FOR ORE, WASTE,
AND STOCKPILES

WILLIAMS CREEK

STAGE 1

ORE SCHEDULE

LEVEL	SECTION 1600									
	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu
2800	54.0	0.78	26.0	0.22						
2750	128.0	0.78	70.0	0.22	16.0	0.39				
2700	42.0	0.78			124.0	0.39	95.4	2.08		
2650					96.0	0.39	144.0	2.08		
2600					72.0	0.39	176.0	2.08		
2550							176.0	2.08	24.8	1.63
2500							118.0	2.08	116.0	1.63
2450							15.4	2.08	52.0	1.63
TOTAL	224.0		96.0		308.0		724.8		192.8	

LEVEL	SECTION 1200									
	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu
2800	76.0	1.20	80.0	0.62						
2750	96.0	1.20	120.0	0.62	21.0	0.96				
2700	20.0	1.20			176.0	0.96			4.0	0.44
2650					192.0	0.96			9.0	0.44
2600					128.0	0.96	38.0	1.46		
2550					38.0	0.96	112.0	1.46		
2500							52.0	1.46		
TOTAL	192.0		200.0		555.0		202.0		13.0	

LEVEL	SECTION 800								SECTION 400	
	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu
2800	76.8	0.27	102.0	0.70					88.2	1.05
2750	64.0	0.27	59.6	0.70	11.0	0.24	54.0	0.88		
2700					50.0	0.24	80.0	0.88		
2650					43.0	0.24	64.0	0.88		
2600					32.0	0.24	72.0	0.88		
2550					20.0	0.24	25.0	0.88		
TOTAL	140.8		161.6		156.0		295.0		88.2	

Stage 1 - Ore Schedule (continued)

SUMMARY

LEVEL	TOTAL		STOCKPILE		HIGH GRADE	
	Tons	% Cu	Tons	% Cu	Tons	% Cu
2800	503.0	0.74	102.8	0.26	400.2	0.87
2750	639.6	0.69	161.0	0.26	478.6	0.83
2700	591.4	0.94	178.0	0.35	413.4	1.20
2650	548.0	1.08	148.0	0.35	400.0	1.35
2600	518.0	1.24	104.0	0.34	414.0	1.47
2550	395.8	1.60	20.0	0.24	375.8	1.67
2500	286.0	1.78	-	-	286.0	1.78
2450	67.4	1.73	-	-	67.4	1.73
TOTAL	3,549.2	1.09	713.8	0.31	2,835.4	1.28
+ Dilution	205.8	0.545	41.2	0.155	164.6	0.64
GRAND TOTAL	3,755.0	1.06	755.0	0.30	3,000.0	1.25

WILLIAMS CREEK

STAGE 2

ORE SCHEDULE

SECTION	1600		1200												800					
LEVEL	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu
2800																				
2750																				
2700																				
2650																				
2600			16.0	0.44	10.0	0.96														
2550			19.0	0.44	30.2	0.96			32.0	1.46					5.0	0.88				
2500			16.0	0.44			12.0	0.31	100.0	1.46	24.0	0.39	40.0	1.34	6.0	0.88	2.0	0.24		
2450	180.0	1.63					8.0	0.31	35.6	1.46	24.0	0.39	154.0	1.34	5.0	0.88	1.0	0.24	1.0	1.44
2400	244.0	1.63									16.0	0.39	196.0	1.34						
TOTAL	424.0		51.0		40.2		20.0		167.6		64.0		390.0		16.0		3.0		1.0	

Stage 2 - Ore Schedule (continued)

SECTION	400										0											
LEVEL	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu
2850					20.0	0.33																
2800			80.0	0.63	144.0	0.33																
2750	51.0	1.05	72.0	0.63	144.0	0.33					68.8	0.75	17.6	1.56	38.0	0.45	50.0	1.26				
2700	58.0	1.05	72.0	0.63	144.0	0.33									80.0	0.45	104.0	1.26				
2650	50.0	1.05	72.0	0.63	131.0	0.33									56.0	0.45	36.4	1.26	40.0	1.48		
2600	41.0	1.05	72.0	0.63	120.0	0.33			10.0	0.22					5.2	0.45			64.0	1.48	40.0	0.15
2550	3.0	1.05	64.0	0.63	70.0	0.33	22.7	1.36	64.0	0.22									56.0	1.48	32.0	0.15
2500					20.0	0.33	32.0	1.36	66.8	0.22												
2450																						
2400																						
TOTAL	203.0		432.0		793.0		54.7		140.8		68.8		17.6		179.2		190.4		160.0		72.0	

WILLIAMS CREEK

STAGE 3

ORE SCHEDULE

SECTION	1600		1200						800							
	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu	Tons	% Cu
2850																
2800																
2750																
2700																
2650																
2600																
2550									4.0	0.24	71.0	0.88				
2500			20.0	0.31					28.0	0.24	150.0	0.88				
2450			30.4	0.31					20.0	0.24	120.0	0.88	8.0	0.49	19.0	1.44
2400					3.0	0.39			9.8	0.24	20.0	0.88	18.0	0.49	96.0	1.44
2350	200.0	1.63			30.0	0.39	147.0	1.34					18.0	0.49	128.0	1.44
2300	200.0	1.63			30.0	0.39	140.0	1.34					18.0	0.49	144.0	1.44
2250	118.2	1.63			26.6	0.39	140.0	1.34					18.0	0.49	152.0	1.44
2240	25.0	1.63					28.0	1.34							32.3	1.44
TOTAL	543.2		50.4		89.6		455.0		61.8		361.0		80.0		571.3	

WILLIAMS CREEK

WASTE TONNAGE BY LEVEL
(Thous Tons)

Stage 1

LEVEL	1600	1200	800	400	0	TOTAL
2850	–	8	6	–		
2800	72	76	157	228		
2750	329	215	228	329		
2700	229	164	157	342		
2650	143	86	114	372		
2600	80	34	45	91		
2550	48	–	25	–		
2500	3	–	–	–		
2450	–	–	–	–		
TOTAL	904	583	732	1,362		3,581

Stage 2

LEVEL	1600	1200	800	400	0	TOTAL
2850	–	24	25			
2800	168	150	25			
2750	660	309	100			
2700	732	424	80			
2650	640	400	70			
2600	590	384	60			
2550	530	304	40			
2500	480	152	20			
2450	430	64	–			
2400	102	24	–			
TOTAL	4,332	2,235	420			6,987

Stage 3

LEVEL	1600	1200	800	400	0	TOTAL
2850	–	20	64	30	–	
2800	102	64	326	112	–	
2750	450	70	560	64	336	
2700	614	368	960	80	328	
2650	548	448	976	312	272	
2600	520	432	1,008	465	224	
2550	388	416	928	472	62	
2500	370	432	816	400	–	
2450	376	432	640	–	–	
2400	356	368	480	–	–	
2350	214	256	400	–	–	
2300	44	124	288	–	–	
2250	–	20	140	–	–	
2240	–	–	–	–	–	
TOTAL	3,982	3,450	7,586	1,935	1,222	18,175
TOTAL						28,743

WILLIAMS CREEK

STAGE 1

MATERIAL MOVEMENT

YEAR 1

Level	Tons	% Cu	STOCKPILE Tons	STOCKPILE % Cu	WASTE Tons
2800	353.2	0.84	102.8	0.26	547
2750	478.6	0.83	161.0	0.26	1,101
2700	95.4	2.08			446
2700	18.0	1.20			
	945.2	0.97	263.8	0.26	
Dilution	54.8	0.485	15.2	0.13	
TOTAL	1,000	0.94	279.0	0.25[1]	2,094[2]

Strip Ratio = 2.09:1[3]

YEAR 2

Level	Tons	% Cu	STOCKPILE Tons	STOCKPILE % Cu	WASTE Tons
2700	300.0	0.92	178.0	0.35	446
2650	400.0	1.35	148.0	0.35	715
2600	176.0	2.08			125
2600	38.0	1.46			
2600	31.2	0.96			
	945.2	1.34	326.0	0.35	
+ Dilution	54.8	0.67	19.0	0.175	
TOTAL	1,000	1.30	345.0	0.34	1,286

Strip Ratio = 1.29:1

YEAR 3

Level	Tons	% Cu	STOCKPILE Tons	STOCKPILE % Cu	WASTE Tons	Level	WASTE Tons
2600	168.8	0.93	104.0	0.34	125	2850	49
2550	422.8	1.58	20.0	0.24	73	2800	343
2500	286.0	1.78			3	2750	1,069
2450	67.4	1.73				2700	1,236
	945.2	1.54	124.0	0.32		2650	1,100
+ Dilution	54.8	0.77	7.0	0.16			
TOTAL	1,000	1.49	131.0	0.32	201		3,807

Strip Ratio = 4.01:1

Notes:
1. Mined in preproduction period.
2. 1,000 mined in preproduction stage.
3. The waste tonnages, and thus strip ratios, have been smoothed in the final schedule, Exhibit 5.

APPENDIX II

CAPITAL COST DETAILS

TABLE 6 – WILLIAMS CREEK MINING CAPITAL COST

TYPE	MAKE	MODEL	STATUS	UNIT	LOADS	UNIT COST	TOTAL
LOADING							
LOADER	CAT	992C	NEW	1	3	$1,050,000	$1,050,000
LOADER	CAT	992C	USED	1	3	$600,000	$600,000
HAULAGE & ROADS							
HAUL TRUCKS	CAT	773	USED	5	1	$350,000	$1,750,000
TRACTORS	CAT	D8K	USED	2	3	$150,000	$300,000
GRADER	CAT	14G	USED	1	1	$50,000	$50,000
WATER TRUCK			USED	1	1	$50,000	$50,000
DRILLING							
ROTARY DRILL	DRILLTECK	D60K	NEW	1	2	$550,000	$550,000
SKID STEER LOADER	BOBCAT	600	NEW	1		$30,000	$30,000
BLASTING							
CAP MAGAZINE			NEW	1	0.5	$10,000	$10,000
POWDER MAGAZINE			NEW	1	0.5	$50,000	$50,000
POWDER/PUMP TRUCK			NEW	1	0.25	$40,000	$40,000
MAINTENANCE							
SERVICE TRUCK	GMC		NEW	1	0.5	$45,000	$45,000
WELDER	MILLER		NEW	1		$10,000	$10,000
MISCELLANEOUS							
SINGLE BAY SHOP			NEW	1	1	$50,000	$50,000
TOOLS			NEW				$40,000
FUEL TANKS	TIDY		NEW	1	0.5	$12,000	$12,000
PICK UP TRUCKS	GMC		NEW	3		$25,000	$75,000
AMBULANCE			USED	1	0.25	$40,000	$40,000
ENGINEERING							
SURVEY INSTRUMENTS			NEW				$10,000
COMPUTER HARDWARE			NEW				$15,000
COMPUTER SOFTWARE			NEW				$5,000
OFFICE EQUIPMENT			NEW				$25,000
MINING TOTAL							$4,807,000
MOBILIZATION					18	$5,000	$90,000
TOTAL MINING							$4,897,000

TABLE 6 WILLIAMS CREEK – CAPITAL REQUIREMENT (continued)

COST CENTRE		UNITS	UNIT COST	SUBTOTALS	SUBTOTALS
EXPLORATION					
DRILLING "NQ" CORE	(feet)	35000	$40.00	$1,400,000	-
DRILLING "H" CORE	(feet)	2000	$75.00	$150,000	
METALLURGICAL TEST WORK				$150,000	$1,700,000
PREPRODUCTION SITE PREPARATION					
ROAD				$50,000	
STRIPPING/CLEARING				$200,000	
CAMP MOBILIZATION				$100,000	$350,000
HEAP CONSTRUCTION					
LINER DESIGN & CONSTRUCTION	(feet^2)	1500000	$2.00	$3,000,000	
POND	(gallons)	25000000		$550,000	
PIPING & IRRIGATION				$250,000	
PUMPS				$50,000	$3,850,000
ACCOMMODATIONS					
TOWN HOUSING FIVE FAMILIES	(houses)	6	$50,000	$300,000	
SITE CAMP	(man)	15	$18,000	$270,000	
TOWN BUNK HOUSE	(man)	30	$10,000	$300,000	$870,000
VEHICLES					
PICK-UP TRUCKS	(units)	3	$18,000	$54,000	
FORKLIFT (used)	(units)	1	$50,000	$50,000	
MOBILE CRANE (used) (15 tonne)	(units)	1	$125,000	$125,000	$229,000
PROCESS PLANT					
ACID PLANT 100 TPD CAPACITY				$6,000,000	
SOLVENT EXTRACTION				$1,000,000	
ELECTROWINNING				$6,700,000	
POWER LINE				$1,000,000	
PRIMARY POWER DISTRIBUTION				$400,000	
WATER HEATER				$150,000	
BUILDINGS	(feet^2)	22700	$150	$3,405,000	$18,655,000
WORKING CAPITAL/FIRST FILL				$2,225,000	$2,225,000
PROCESS PLANT SUBTOTAL					$27,879,000
MINE OPERATIONS					
MINING EQUIPMENT				$4,807,000	
MOBILIZATION				$90,000	$4,897,000
MINE OPERATIONS SUBTOTAL					$4,897,000
INVENTORY					$250,000
ENVIRONMENTAL STUDIES					$150,000
GEOTECHNICAL ENGINEERING					$150,000
SUBTOTAL CAPITAL COSTS					$33,326,000
EPCM @ 12%					$3,345,000
CONTINGENCY @ 20%					$5,574,000
TOTAL PROJECT COST					$41,695,000

APPENDIX III

EXPLORATION HOLES FOR DRILL-INDICATED RESERVES

Stages 1 & 2

Holes 21 - 37 inclusive (17 holes)

150 foot depth	200 ft holes	11 holes	2,000 ft
300 foot depth	400 ft holes	6 holes	2,400 ft

Holes 44 - 55 inclusive Surface Confirmation (12 holes)

50 to 75 foot depth	100 ft holes	12 holes	1,200 ft
	SUBTOTAL - Stages 1 & 2		5,600 ft

Stage 3

Holes 38 - 43 inclusive (8 holes)

500 foot depth	650 ft holes	8 holes	5,200 ft
	TOTAL		10,800 ft

Preproduction Drilling

Holes marked with "+" (32 holes)

200 foot depth	300 ft holes	32 holes	Say, 10,000 ft

Allow for $35.00 per foot of drilling:

Stages 1 & 2	$196,000	Say,	$200,000
Stages 3	182,000	Say,	200,000
Preproduction	350,000	Say,	350,000
TOTAL COST	$728,000	Say,	$750,000