MAP NO.: 115I 7	**ASSESSMENT REPORT** X **PROSPECTUS** **CONFIDENTIAL** X **OPEN FILE**	**DOCUMENT NO:** 093040 **MINING DISTRICT:** WHITEHORSE **TYPE OF WORK:** RERC SUBMISSION

REPORT FILED UNDER: WESTERN COPPER HOLDINGS

DATE PERFORMED: 1990-1991 **DATE FILED:** SUBMITTED MAY 1991

LOCATION: **LAT.:** 62°21'N **AREA:** WILLIAMS CREEK

LONG.: 136°41'W **VALUE $:**

CLAIM NAME & NO.: BOY 20 (Y51118), BOY 22 (Y51120), BOY 24 (Y51122), BOY 51-58 (Y51149-156)
BOY 83 (Y51181), BOY 85 (Y51183), DUN 1F-3F (Y59382-384), WAR 22 (Y59373)
W 1-49 (YB26708-756), W 50-58 (YB36249-257), W 59-80 (YB36452-YB473)
WAR 23-31 (YB36240-248), WAR 32-37 (YB36446-451)

WORK DONE BY: ARCHER CATHRO & ASSOCIATES

WORK DONE FOR: WESTERN COPPER HOLDINGS

DATE TO GOOD STANDING:

REMARKS: THIS REPORT IS THE INITIAL SUBMISSION FOR RERC.

Not sent to Mining Recorder's Office

WILLIAMS CREEK COPPER OXIDE PROJECT

YUKON TERRITORY

Initial Submission
Prepared for the
Regional Environmental Review Committee, Yukon Territory

Williams Creek Project
Managed By:

Archer Cathro & Associates (1981) Ltd. and
Silver Standard Resources Inc.

For:

WESTERN COPPER HOLDINGS LTD.
400 - 1199 West Hastings Street
Vancouver, B.C.
V6E 3T5

And

THERMAL EXPLORATION COMPANY
11525 Caroline Lane
Nevada City, California
U.S.A., 95959

May 1991

EXECUTIVE SUMMARY

The Williams Creek Project is located 200 kilometres northwest of Whitehorse and 48 road-kilometres northwest of Carmacks, Yukon Territory. The property is currently accessible by a two-wheel driveable road from Carmacks. Copper mineralization at **Williams Creek** was discovered in 1970 during a regional exploration program. Potential economic mineralization at **Williams Creek** occurs in the No. 1 Zone which is a northwest striking, steeply dipping (70° E) tabular zone which averages 29 metres in width, is exposed for 550 metres along strike and has been intersected in drill holes to a depth of 425 metres below surface. Mineralization consists of disseminations and fine veinlets of bornite, chalcopyrite, pyrite and minor molydenite. Due to the porosity of the schistose host, primary copper minerals have been oxidized to azurite, malachite and a copper-bearing limonite to depths of approximately 245 metres below surface. There are minor gold and silver credits associated with the copper minerals.

Drill-indicated reserves for the No. 1 Zone total 14.8 million tonnes grading 1.15% copper and 0.70 grams of gold per tonne. Fifty-six percent of this reserve is contained within the oxide zone which is equivalent to 8.7 million tonnes grading 1.2% copper. A preliminary open-pit ore reserve of 7.8 million tonnes grading 1.29% copper using a 0.5% copper cut-off grade has been calculated by William Hill Mining Consultants Ltd. (Appendix I). Allowing for 12% dilution, this is equivalent to 8.7 million tonnes grading 1.02% copper.

Williams Creek is currently being explored to assess its potential as a solvent extraction electro-winning copper oxide heap leach. Preliminary metallurgical work undertaken by Bacon Donaldson (Appendix III) showed recoveries of up to 86% after 33 days in column tests. It is expected that recoveries would be somewhat lower in a heap.

The property would be mined at a rate of 1 million tonnes of ore per year. The average strip over the 9 year mine life would be 3:1, waste:ore. Leaching would occur over an eight month period with four months of inactivity during the cooler winter months. The property would employ a total of 82 people including 30 in the mine, 35 in the plant and 17 office personnel. This would provide a Yukon based labour force with earnings of $3.5 to $4 million per year. It is expected that personnel would be housed on site and at Carmacks.

The property will require 50,000 kilowatts of power per day and Wrights Engineers (Appendix II) have estimated the cost for construction of a power line from the existing Yukon Power grid at Carmacks to the site at $1.4 million including transformers.

The project will be designed to minimize the impact on the environment. Baseline studies have already been initiated with respect to water quality and the project participants are looking forward to cooperating with the Yukon government and regional agencies with respect to collection of appropriate environmental data to undertake a proper assessment of the project prior to major construction expenditures.

There are three areas of environmental concern:

1. The transportation of sulphur to the property.
2. The use of sulphuric acid at the project.
3. Reclamation of the open-pit and heaps at cessation of mining activities.

It is the intent of the participants to use all means available to ensure that acid is transported safely to the project and that the project is engineered to ensure zero discharge to the environment. Reclamation plans will include the neutralization of the heaps at cessation of mining activities with contouring and covering of the same with soil and revegetation. The pit will be fenced and allowed to fill with water.

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Appendix II:	Williams Creek Property Review, May 1990, Project 1918, Wright Engineers Ltd.
Appendix III:	Metallurgical Investigation of Williams Creek Copper Oxide Ore, 1990 by Bacon Donaldson.
Appendix IV:	Preliminary Water Quality Investigation of Williams Creek 1989 by B.E. Burns and J.E. Gibson, Whitehorse, Yukon Territory, January 1990.
Appendix V:	Summary of Option Agreement with Archer Cathro & Associates (1981) Limited; Joint Venture Agreement between Silver Standard Resources Inc. and Thermal Exploration Company.

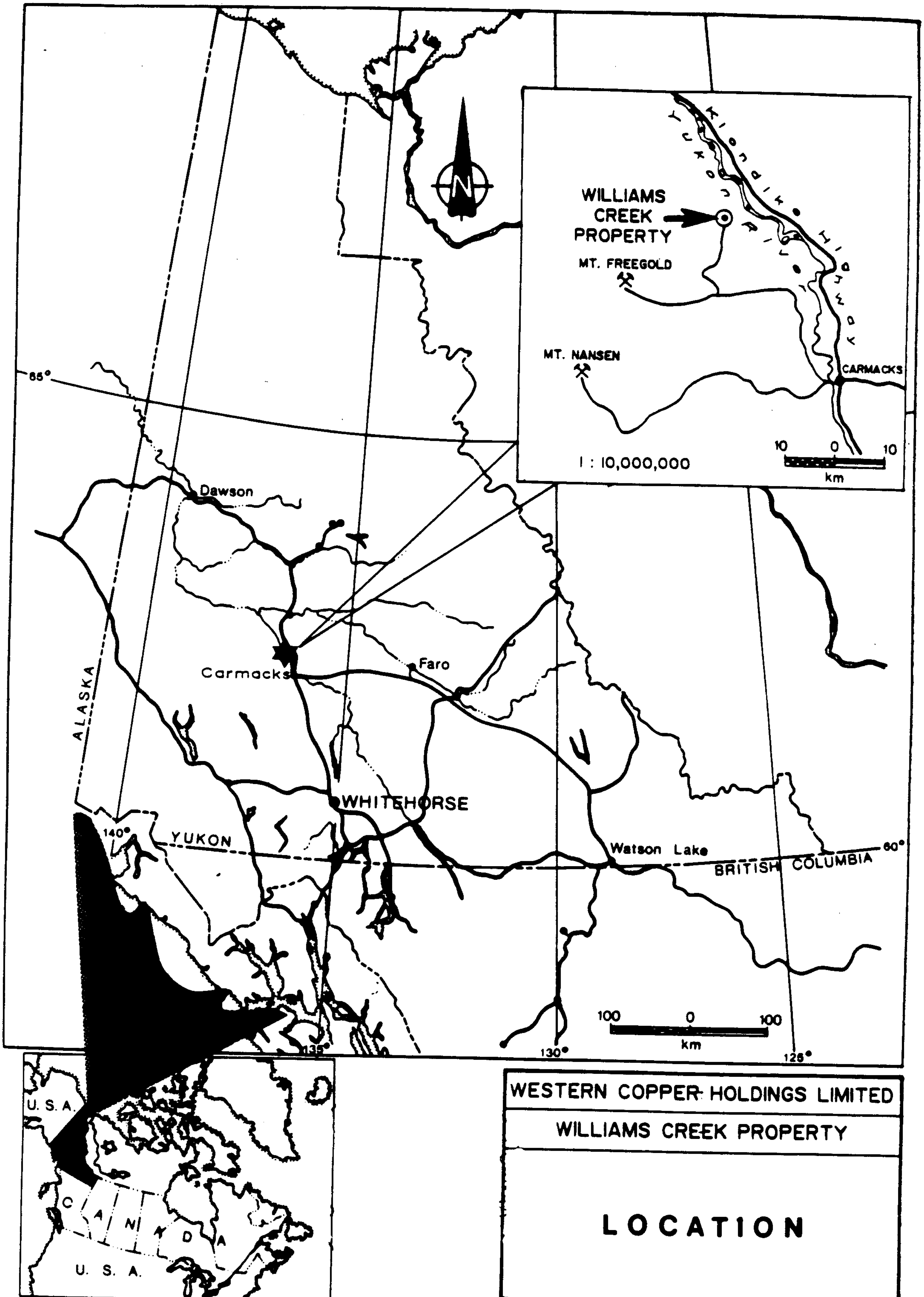
INTRODUCTION:

The Williams Creek copper property was discovered in the early 1970's and has received sporadic exploration since then. The recent advances in low-cost portable solvent-extraction electro-winning plants have made it possible to profitably develop copper projects which are amenable to leaching. A preliminary evaluation of the Williams Creek Project, which was undertaken in 1989, indicates that the property has the potential to be a profitable copper oxide heap leach. To this end, Western Copper Holdings Ltd. in joint venture with Thermal Exploration Company of Calgary have optioned the Williams Creek copper oxide project from Archer Cathro & Associates (1981) Ltd. and to date have spent in excess of $200,000 in carrying out initial test work, with a commitment in 1991 to expand an additional $700,000 to $900,000 in assessing the viability of this attractive property as a copper oxide leach.

It is management's belief that the property can be developed as an environmentally sound project with limited impact on the environment and a positive impact on the economy of the Yukon Territory particularly in and around Carmacks. This report sets out some of the preliminary parameters of the proposed project as a first stage in keeping the local and territorial government and associated agencies informed, to ensure the Williams Creek Project does not in any way compromise the environmental integrity of the Williams Creek ecosystem. We look forward to dialoguing with the Regional and Environmental Review Committee with respect to this project.

LOCATION:

The Williams Creek property is located 200 kilometres northwest of Whitehorse, Yukon (NTS 115 I/7) at 62°, 21' North latitude and 136°, 42' West longitude (Figure 1). The property is located near the head waters of Williams Creek within the Dawson Range, 38 kilometres northwest of the town of Carmacks (population approximately 500).



WESTERN COPPER HOLDINGS LIMITED	
WILLIAMS CREEK PROPERTY	
LOCATION	
Aurum Geological Consultants Inc.	Date April 1991
NTS 115-I/7 · Drawn by G.D.S.	Figure 1

HISTORY:

In the late 1960's, exploration for porphyry copper deposits in the Dawson Range lead to the discovery of the Casino porphyry copper deposit located 104 kilometres northwest of Williams Creek. The Williams Creek property was staked at this time by G. Wing of Whitehorse to cover copper occurrences he had found by prospecting prior to the Casino discovery. The Dawson Range Joint Venture (Straus Exploration Inc., Great Plains Developments of Canada Ltd., Trojan Consolidated Minerals Ltd., and Molybdenum Corporation of America) optioned the property and conducted reconnaissance prospecting and geochemical sampling. Archer, Cathro & Associates Limited acted as Manager and earned the right to acquire abandoned properties. Subsequent exploration located 13 mineralized zones which have been explored by bulldozer trenching, 8 of which have been drill-tested (Holstein, R., 1991)[1]. The best of these was called the No. 1 Zone.

Exploration of the No. 1 Zone consisted of bulldozer trenching and two X-ray diamond drillholes totalling 31.4 metres in 1970. In 1971 work programs consisted of soil sampling, geophysical surveys, bulldozer trenching, road construction and 25 diamond drillholes totalling 5,552.54 metres. In 1972, 8 additional drillholes totalling 1,530.7 metres were completed. This wide-spaced drilling was sufficient to identify a reserve of approximately 15 million tonnes grading 1% copper. With the downturn in copper prices in the early 1980s ownership of the property reverted to Archer Cathro & Associates Limited. The G. Wing residual interest was required by A. Arsenault in 1971 and 100% of the Arsenault interest is held under an Option Agreement to Archer, Cathro.

In 1989, the property including the rights to the Arsenault Option was optioned to Western Copper Holdings Ltd. who farmed-out a 50% property interest to Thermal Exploration Co. The companies then collected 3 tonnes of surface oxide material for testing of leaching characteristics by Bacon & Donaldson's laboratories in Vancouver, B.C. In addition, a preliminary economic analysis of the project was made by both

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[1] Holstein, R: Summary Report on the Williams Creek Property, Vancouver, B.C., Aurum Geological Consultants Inc., April 19, 1991, 16 pgs.

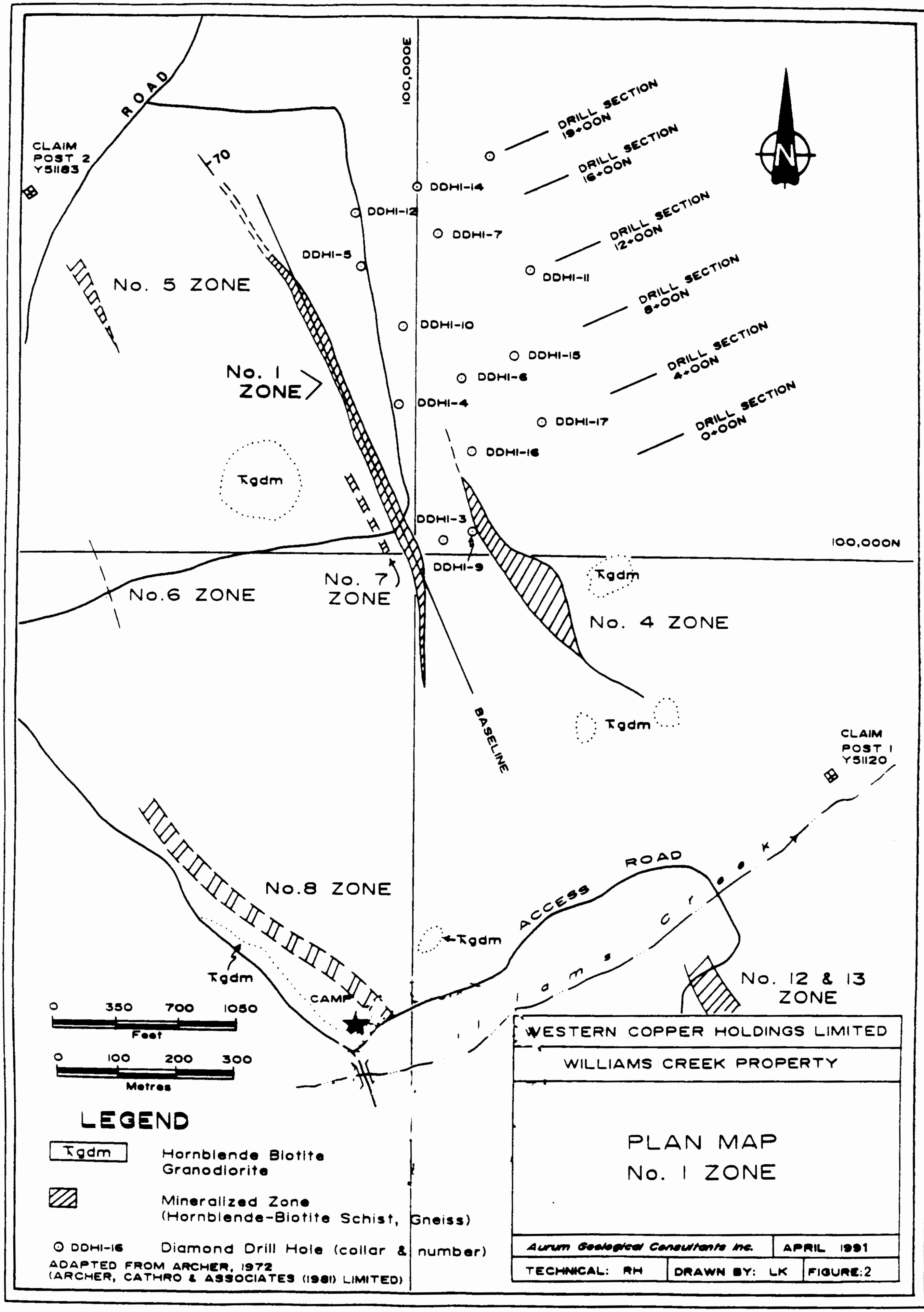
Wright's Engineers and William Hill Consultants Ltd. The company also undertook initial baseline studies using Burns & Gibson of Whitehorse and a road upgrade study by Boreal Consulting Services of Whitehorse.

ACCESS:

Direct access to the Williams Creek property is by helicopter from Whitehorse or from a seasonal base in Carmacks. The property is currently road accessible via the all-weather Mt. Freegold Road leading from the Klondike Highway No. 2 at Carmacks to a turnoff at Mile Post 22 along the Mt. Freegold Road (35 kilometres). A 13 kilometre two-wheel drive accessible road leads from the turn-off to the Williams Creek Project. Depending upon season this road may require four-wheel drive accessibility until it has been upgraded. Mile Post 22 on the Mt. Freegold Road the road has an elevation of 732 metres above sea level. From this point the Williams Creek access road heads north for a distance of 4 kilometres where it increases in elevation to 885 metres above sea level. It swings northeasterly for a distance of 9 kilometres, eventually dropping in elevation to 717 metres at the campsite at Williams Creek (Figure 1) (Holstein, R.).

ROAD UPGRADE:

Boreal Consulting Services Ltd. of Whitehorse was requested to study the current access road to the Williams Creek property and advise as to a two-phase program to upgrade this road. Phase A, estimated at $76,000, was sufficient to upgrade the road to exploration status. A portion of this upgrading was undertaken in the fall of 1990 in part funded by a grant from the Yukon Territorial government and further work is planned in late May 1991. If a mine is developed, the access road would have to be brought up to the standards of all-weather travel for a tractor-trailer operation (Phase B). It is difficult at this time to estimate the cost upgrade though it is expected that the cost will be in the order of $150,000.


ROAD
CLAIM POST 2 Y51183
100,000E
DRILL SECTION 19+00N
DRILL SECTION 16+00N
DRILL SECTION 12+00N
DRILL SECTION 8+00N
DRILL SECTION 4+00N
DRILL SECTION 0+00N
N
70
DDHI-14
DDHI-12
DDHI-7
DDHI-5
DDHI-11
DDHI-10
DDHI-15
DDHI-6
DDHI-4
DDHI-17
DDHI-16
DDHI-3
DDHI-9
No. 5 ZONE
No. 1 ZONE
Ҡgdm
No. 7 ZONE
No.6 ZONE
No. 4 ZONE
100,000N
BASELINE
CLAIM POST 1 Y51120
ACCESS ROAD
Williams Creek
No.8 ZONE
No. 12 & 13 ZONE
CAMP
0 350 700 1050
Feet
0 100 200 300
Metres
LEGEND
Ҡgdm Hornblende Biotite Granodiorite
Mineralized Zone (Hornblende-Biotite Schist, Gneiss)
DDHI-16 Diamond Drill Hole (collar & number)
ADAPTED FROM ARCHER, 1972
(ARCHER, CATHRO & ASSOCIATES (1981) LIMITED)
WESTERN COPPER HOLDINGS LIMITED
WILLIAMS CREEK PROPERTY
PLAN MAP
No. 1 ZONE
Aurum Geological Consultants Inc.
APRIL 1991
TECHNICAL: RH
DRAWN BY: LK
FIGURE: 2

TOPOGRAPHY AND CLIMATE:

The Williams Creek property is situated on the northeast flank of the unglaciated Dawson Range which is part of the Klondike Plateau, an uplifted erosion surface dissected by narrow valleys. The topography is subdued with local relief of approximately 300 metres and a maximum elevation of 915 metres above sea level. The property lies on the west side of the Yukon River Valley and is unglaciated with the exception of weak valley glaciation as a result of small lateral morraines and kame terraces. Overburden is generally less than 1 metre in thickness and consists of a number of centimetres of moss and organic material overlying 5 to 20 centimetres of white felsic volcanic ash with 10 centimetres of organics, all forming part of an A soil horizon with the underlying B soil horizon consisting of 15 to 50 centimetres of red-brown soil overlying bedrock.

Vegetation consists of dwarf willow and alder on south-facing slopes and moist valleys. White spruce and poplar prefer the dryer well drained soils of the hillside and there are a few growths of pine within the general area. Water and timber suitable for exploration and mining are available locally.

PROPERTY GEOLOGY:

The oldest rocks on the property are outcrops of biotite-hornblende-feldspar quartz schists presumably of the Yukon Tanana terrain. These are restricted to small isolated roof pendants within hornblende granodiorite of the Granite Mountain Batholith. Biotite and quartz content of the schists vary greatly within schist individual layers. Compositionally the shift is a hornblende-biotite-diorite. Foliation generally trends 145° with a 70° E dip (Figure 2). The hornblende granodiorite of the Granite Mountain Batholith is a weakly foliated course-grained hornblende and biotite-rich granodiorite containing large phenocrysts of white feldspar. The foliation generally trends northwest and dips deeply east. There are minor aplite and pegmatite dykes cutting the geology.

Thirteen mineralized zones have been discovered to date on and around the property. All zones have been explored by bulldozer trenching and 8 have been diamond drilled.

The No. 1 Zone is a tabular zone of mineralized biotite-hornblende-quartz-feldspar schist enclosed by granodiorite. The ore host has an unusually high porosity and primary copper sulphide minerals have been oxidized to a depth of approximately 245 metres below surface. To the north the deposit pinches out into granodiorite and terminates to the south by an assay cut-off due to increasing pyrite content and decreasing bornite and chalcopyrite. As most of the copper has been oxidized to 245 metres, reserves have been divided into oxide and sulfide and with a distinction between hanging-wall and footwall mineralization.

RESERVES:

Based on exploration work up to 1972 a reserve for the Williams Creek No. 1 Zone was calculated at 14.8 million tonnes grading 1.15% copper with 0.70 grams of gold per tonne. Based on a number of parameters, which are discussed in Section 3.3 of Appendix I by W. Hill Mining Consultant Ltd., a preliminary mineable ore reserve to a depth of 245 metres below surface has been calculated as 8.9 million tonnes grading 1.02% copper with a 3:1 strip. A detailed calculation is given in Exhibit 4 of Appendix I while Exhibit 5 of the same appendix lists the proposed production schedule for the nine year life of the mine.

PROPOSED 1991 EXPLORATION PROGRAM:

In 1991, the Joint Venture partners are planning a 2,500-metre drill program which will cost on the order of $450,000 and consist of a number of 80 - 180 metre length drillholes (approximately 26 holes) to better define the copper reserve for the Stage 1 Pit as proposed by William Hill (Appendix I). Drillholes will be spaced 45 - 60 metres apart and will define the Stage 1 Pit to a depth of 140 metres below surface. Drilling will commence in June and finish in August. In addition to the drilling, the Joint Venture will upgrade the Williams Creek access road, undertake additional surface geochemical sampling and trenching, and continue with baseline water and ecological studies as requested by the Regional Environmental Review Committee and other associated agencies. A preliminary layout of proposed drillholes is included in Figure 6 in Appendix I.

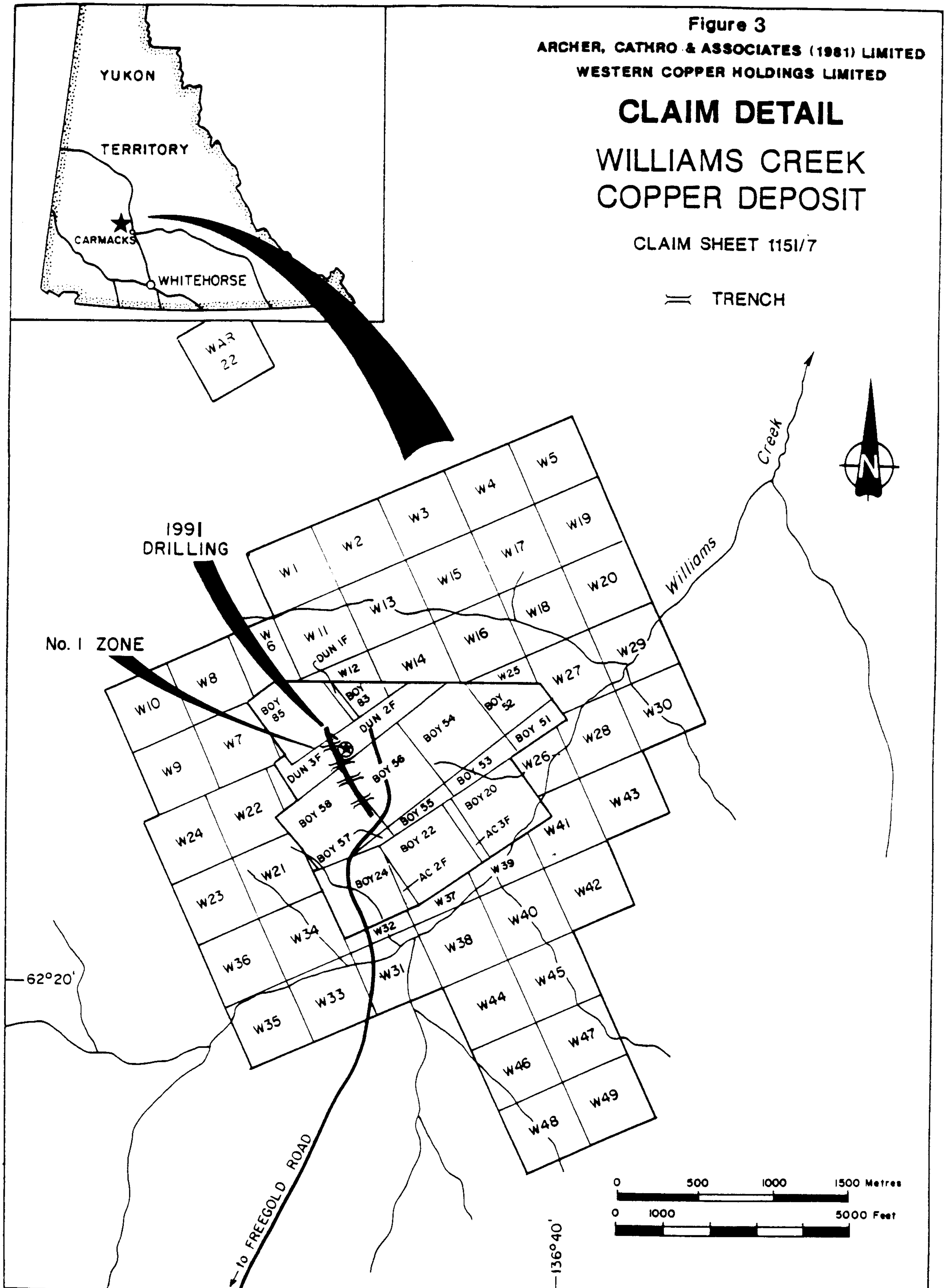


Figure 3
ARCHER, CATHRO & ASSOCIATES (1981) LIMITED
WESTERN COPPER HOLDINGS LIMITED
CLAIM DETAIL
WILLIAMS CREEK
COPPER DEPOSIT
CLAIM SHEET 115I/7
TRENCH
YUKON
TERRITORY
CARMACKS
WHITEHORSE
WAR 22
1991 DRILLING
No. 1 ZONE
Williams Creek
N
W1
W2
W3
W4
W5
W6
W7
W8
W9
W10
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W43
W44
W45
W46
W47
W48
W49
DUN 1F
DUN 2F
DUN 3F
BOY 83
BOY 85
BOY 54
BOY 52
BOY 51
BOY 56
BOY 53
BOY 58
BOY 55
BOY 20
BOY 57
BOY 22
BOY 24
AC 3F
AC 2F
62°20'
136°40'
to FREEGOLD ROAD
0 500 1000 1500 Metres
0 1000 5000 Feet

BASELINE STUDIES:

Recognizing that it is the objective of Western Copper and Thermal Exploration Co. to ensure that the Williams Creek Copper Oxide Project is environmentally sound with little negative impact on the William Creek area, the Joint Venture has contracted Burns & Gibson of Whitehorse to commence preliminary water studies for Williams Creek and associated tributaries. The results of the initial investigation in October 1989 are presented in Appendix IV with sample locations shown in Figure 2 of Appendix IV. Included with Appendix IV is a letter dated April 22, 1991 proposing additional sampling for the water quality survey at Williams Creek in 1991/92. Eight stations near the Williams Creek deposit and two at the confluence of Nancy Creek and Williams Creek and Williams Creek and the Yukon River will be studied. Analyses will be taken of the water and benthotic studies will be made of local aquatic life to establish baseline qualitative and quantitative data.

Burns & Gibson propose that each water quality station will be sampled and analyzed on a quarterly basis for the following:

Ions: ammonia, sulfate, nitrate, nitrite and total phosphorous.

Routine: alkalinity, turbidity, specific conductants, total solids, suspended solids and hardness.

Extractable Metals: 31 element ICP to ppb levels.

Dissolved metals: 31 element ICP to ppb levels.

In addition, field measurements for pH conductivity, stream flow and water temperature will be undertaken at each station. The May 1991 work has been completed. The Joint Venture partners look forward to additional comments from the RERC and associated agencies with respect to the Burns & Gibson proposal for establishing baseline criteria with respect to the water quality of Williams Creek.

With respect to fauna, the Joint Venture partners and employees will diarize observations with respect to fauna encountered within the area of work during the coming fieldseason.

METALLURGY:

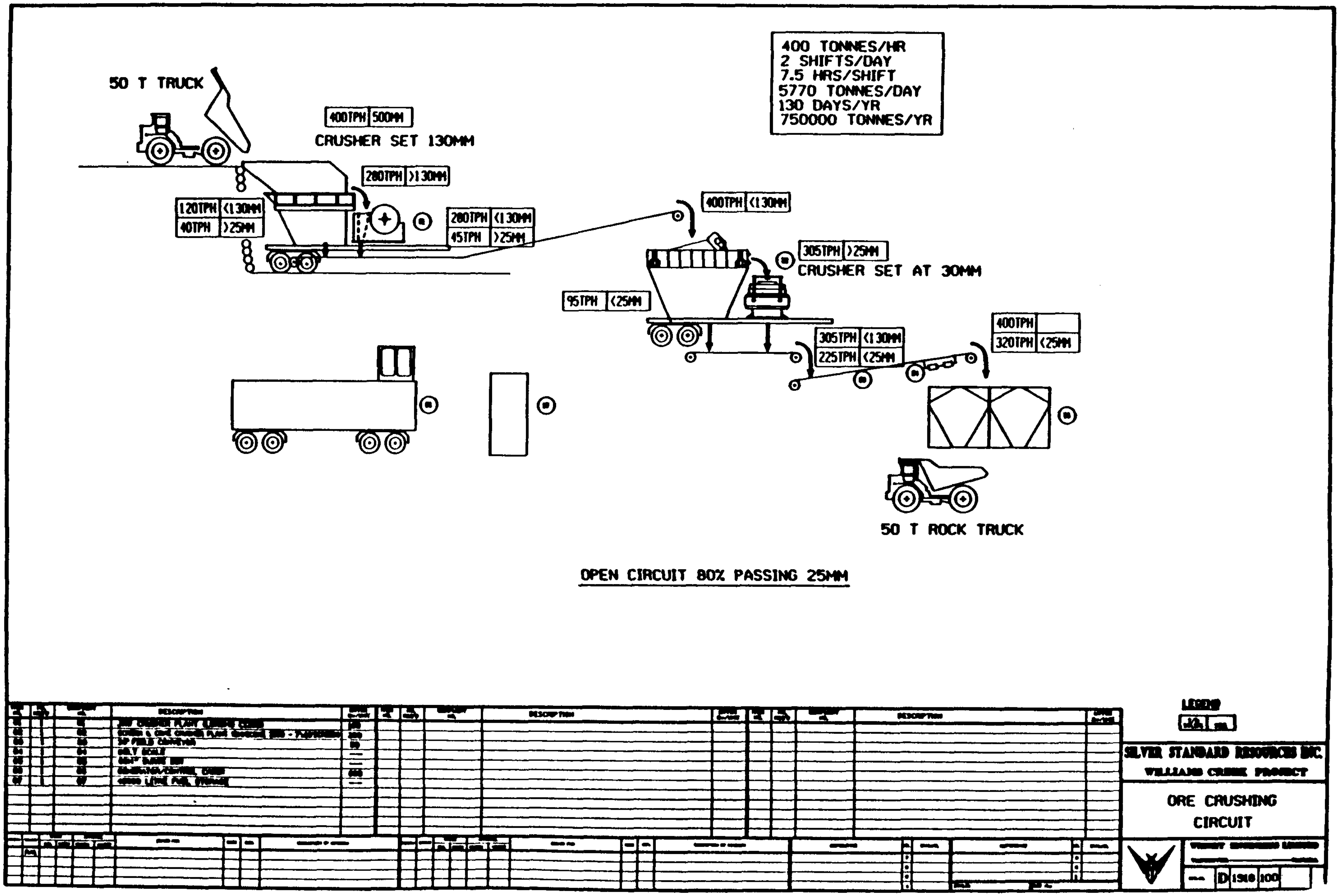
A number of metallurgical tests were carried out on samples of surface oxide material from Williams Creek. In the fall of 1989, three tonnes of oxide material was collected from 8 trenches on the Williams Creek property and delivered to Bacon Donald & Associates of Vancouver (Appendix III). Test work began on October 16, 1989 and included 6 bottle rolls of sulphuric acid leach tests, 1 bottle roll ammonia leach test and 2 larger scale sulphuric acid column leach tests. The bottle roll sulphuric acid leach tests indicated that copper extractions of 80 to 85% copper could be obtained with a grind of approximately -20 millimetres. Sulphuric acid consumption for these tests was 26 to 28 kilograms per tonne.

Based on the preliminary work, two column leach tests were conducted on a blend of material from across the property. The columns measured 3.5 metres by 30 centimetres in diameter loaded with -20 millimetres crushed material. The first test gave an 85% extraction of copper after 78 days. The leaching solution strength was increased to 20 grams per litre initially in the second test and reduced thereafter resulting in an 87% recovery of copper after 33 days. The second column had a marginally lower net sulphuric acid consumption of 48 kilograms per tonne as compared to 51 kilograms for column 1. These recoveries are very encouraging and it is expected that actual heap recoveries will be in the 75% to 80% range. ICI Specialty Chemicals, which manufactures reagents required for use in the solvent extraction electro-winning process, indicates that preliminary investigations of the copper-rich solutions from Williams Creek were positive.

MINING:

A detailed description of the pit operations and crushing is given in Section 4, page 10, of Appendix I by Hill and Section 3.4 - Proposed Mining Method on page 36 of

FIGURE 4

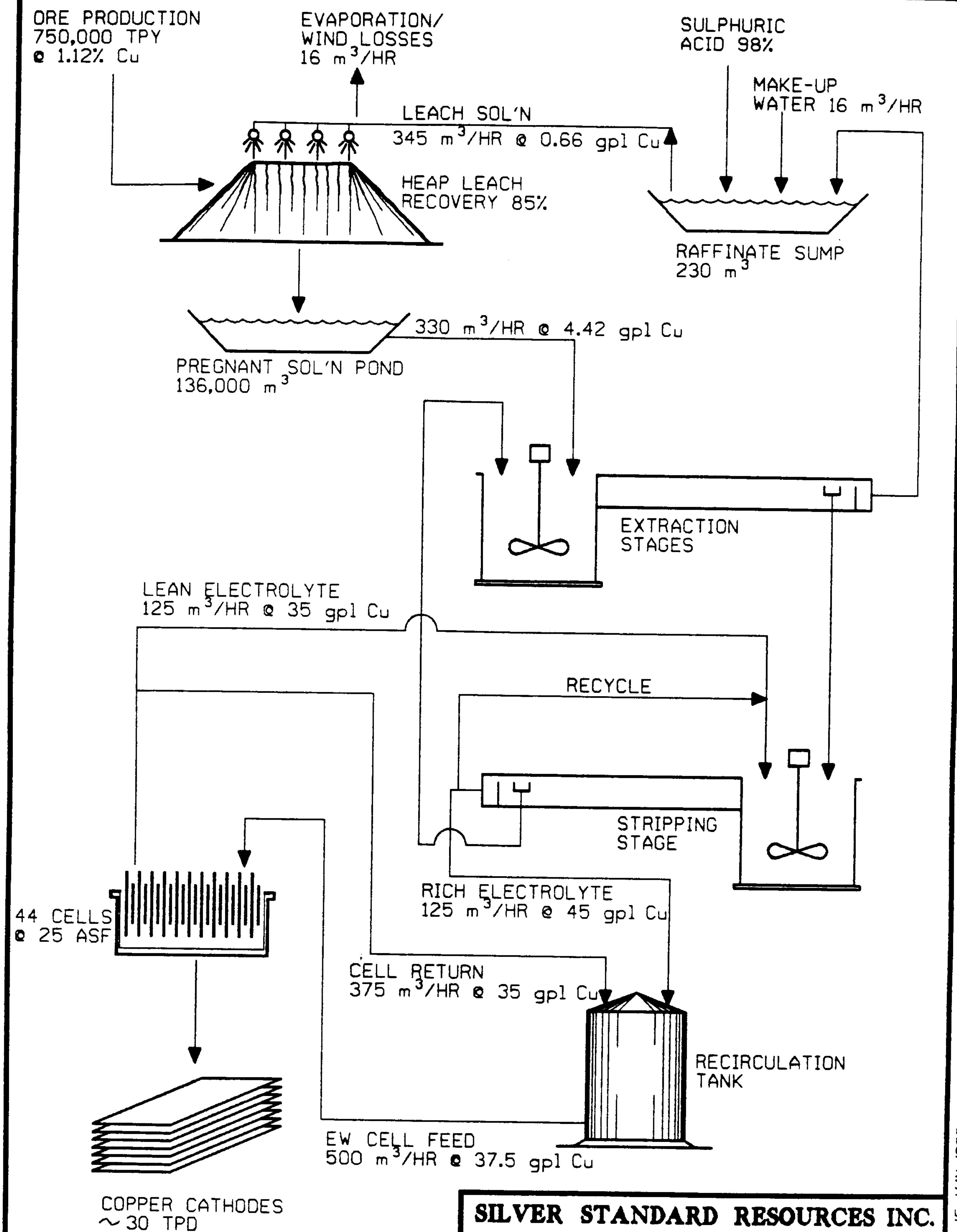

50 T TRUCK
400TPH 500MM
CRUSHER SET 130MM
280TPH >130MM
120TPH <130MM
40TPH >25MM
280TPH <130MM
45TPH >25MM
400TPH <130MM
305TPH >25MM
CRUSHER SET AT 30MM
95TPH <25MM
305TPH <130MM
225TPH <25MM
400TPH
320TPH <25MM
400 TONNES/HR
2 SHIFTS/DAY
7.5 HRS/SHIFT
5770 TONNES/DAY
130 DAYS/YR
750000 TONNES/YR
50 T ROCK TRUCK
OPEN CIRCUIT 80% PASSING 25MM
LEGEND
SILVER STANDARD RESOURCES INC.
WILLIAMS CREEK PROJECT
ORE CRUSHING CIRCUIT

the Wright Engineers study (Appendix II). The following is summarized from these two description reports.

Hill has recommended a four-stage mine development which will result in the ultimate mining of 8.2 million tonnes of ore grading 1.02% copper. Stage 1 will involve mining of higher grade material located at the north end of the deposit to a depth of 100 metres. Stage 2 will see deepening of the pit to 150 metres in the area of high grade, while stages 3 and 4 will see additional material removed for waste both in the hanging and foot walls and a deepening of the pit to its ultimate depth of 180 metres below surface. It should be pointed out that in the Hill study, at the end of year 9 there will be oxide reserves remaining below the ultimate pit depth which cannot be mined profitably by open pit methods.

Mining will be by conventional loader-truck methods. Wright's has estimated that there will be 130 operating days between May and October and production will be on the order of 17,000 to 20,000 tonnes per day to remove the required 1 million tonnes of ore and 3 million tonnes of waste per annum. The initial 1 million tonnes of waste which will be moved during the prestripping phase prior to mining, can be used for construction of leach pads, tailings dams, plant foundations, etc. In the pit, a bench height of 10 metres will be drilled and blasted. Depending on dilution, a lower bench height may be required, but this can only determined after production commences. Drill holes will be sampled and assayed for ore grade control.

A caterpillar 992C loader will operate in the pit and can three-pass load a 773B truck. Two 992C caterpillars will operate simultaneously loading both ore and waste. The 773B trucks, with a high power to weight ratio, can be transported on provincial highways without complete disassembly. Ore will be delivered to a crusher where it will be crushed to 80% passing -22 millimetres. Material will then be trucked to pads and stacked (Figure 4). Total capital costs are estimated at $41.7 million as indicated in Table 6 of the Wright's Engineering study. Actual costs are estimated at $33 million including $6 million for an acid generation plant. The additional $8 million is contingency monies due the fact that the Wright study is a first order estimate. Of this amount it is anticipated that $5 to $6 million will be used to purchase equipment and services available in the Yukon. This will include surface vehicles, preparation of



SILVER STANDARD RESOURCES INC.
WILLIAMS CREEK PROJECT
PROCESS FLOWSHEET Figure 5

/usr2/1918/1918_1.dgn

the road and mobilization of camp to the site, construction of housing, construction of buildings and a power line to the property.

PROCESS PLANT:

A 20 to 30 metric tonne per day SX-EW plant is expected to cost on the order of $10 million complete with building (Figure 5).

LABOUR:

The Williams Creek Copper Project will have a total manpower requirement of 82 persons with an annual payroll of $3 to $4 million. The reader is referred to Exhibits 7, 8 and 9 of the Hill study and Exhibits 7 and 8 of the Wrights study.

Of the 82 personnel, a majority will be individuals who will be trained and hired locally. The Joint Venture will make a commitment to hiring local natives who have equal abilities and skills. It is anticipated that 6 to 7 housing units will be built in Carmacks or purchased locally for managerial and support staff and that a 15-man bunkhouse will be built on site and that a more elaborate townhouse for up to 30 individuals will be constructed at Carmacks.

POWER:

It is anticipated over the 8 months of operation of the project from March through to November, power consumption will be 1,900 kilowatts per hour per tonne of copper produced. Thus on a daily basis, the operation will require 50,000 kw. It is expected that a power line sufficient to meet these requirements can be constructed from Carmacks to the property for $1.4 million including appropriate transformers.

ACID:

Acid is the single highest component of the Williams Creek operating costs as acid consumption is estimated at 45 kilograms per tonne of ore leached. At an estimated cost of $165 per tonne, total acid costs will be on the order of $5 to $6 million per annum. For this reason, Wright's has suggested the construction of an acid generating plant on site. Raw sulphur would be purchased at Fort St. John and transported to the site. A sulphur burning acid plant would be able to produce 165 tonnes per day of sulphuric acid which will be sufficient for the operation. The sulphur generating plant would be built with specifications which would have zero discharge to the environment. This would mean a considerable savings in operating costs and safety factor, as concentrated sulphuric acid will not be transported over the highways. The acid plant would have a 4 year payback.

SHIPPING:

The Williams Creek Project will end up producing copper cathodes which will be 99.9% copper. It is expected that the cathodes will be transported to Skagway and shipped from there. Shipping costs from the plant to Skagway are estimated at 17.5¢ per tonne mile. Shipments to Japan will be on the order of 4,000 tonnes from Skagway with 2 to 3 shipments made each year. Copper cathode will be transported from the plant to Skagway by road in 25 tonne shipments, with 6 shipments per week.

OPERATING COSTS:

Operating costs for the Williams Creek Project are estimated at 63¢ per pound of copper produced. This is broken down in Table 7 of the Wright's study. Of the estimated $8 million annual operating costs, $3.5 to $4 million will be payed in labour to residents of the Yukon. In addition, a number of expenses associated with equipment operation, maintenance of the SX-EW leaching facilities and shipping will be expended in the Yukon. Acid and organics will be acquired outside the Yukon. It is anticipated that somewhere between 70 to 80% of total operating costs will be expended directly in the Yukon Territory, which will amount to some $6.5 million per annum.

ENVIRONMENTAL STUDIES:

Western Copper and Thermal have demonstrated their concern regarding the environmental aspects and impact of this project, by initiating baseline studies in 1989 at a time when only a cursory examination of the project was being made. Even now the company has been paying for and setting up study programs ahead of actually completing drilling which will demonstrate whether or not the project is viable. The baseline studies which are being undertaken by Burns & Gibson in Appendix V should provide a sufficient understanding of the ecosystem of the Williams Creek area. The companies look forward to dialogue with the RERC on this matter, to ensure a sufficient data base is established prior to proceeding with development of the property.

PUBLIC INVOLVEMENT:

This report, which is cursory in nature, is being provided as a starting point for discussions with the RERC and various members of government and other concerned agencies who will want to have input into the development of the Williams Creek Project. The company will plan to meet over the summer with the RERC as the project proceeds. It will also update the local government officials and will plan to have at least one or two public meetings with the Mayor and Chief of Carmacks supplemented by property visits. The company will advise the individuals in Carmacks as to the possible job opportunities which may be available at the mine.

RECLAMATION PLANS:

At the cessation of mining activities some 10 years after the project has commenced, the Williams Creek Project site will consist of an open pit approximately 300 metres in length by 150 metres in width to a maximum depth of 150 metres. There will be a waste dump hosting some 20 million tonnes of waste rock and an extensive heap leach containing 8 million tonnes of 20 millimetres crushed material. The heap will have a maximum height on the order of 10 metres. At the cessation of mining, the heap will be neutralized by washing and contoured. The heaps would be covered with soil and seeded. The waste dumps will be contoured and seeded as well. With respect to the open pit, the same will be fenced and allowed to fill in with water.

TIME FRAME:

The following schedule is preliminary in that development of the Williams Creek property will be dependent upon copper price, the results of this year's drilling program and additional metallurgical testing. If all is positive the following time frame could be expected:

June to August:	2,500 metres of drilling.
September to November:	Additional 5,000 metres of drilling. Collection of metallurgical samples.
November to March:	Continued metallurgical testing of drill core and samples. Collection of water quality samples and counting of flaura and fauna both in the summer and fall by Burns & Gibson.
April 1992:	Preparation of a feasibility study.
June 1992:	Completion of feasibility study. Continuation of environmental baseline studies.
July/August 1992:	Additional exploration drilling.
September 1992:	Completion of bankable feasibility study and raising of capital for development purposes.
September to December 1992:	Raising of funds.
March 1993:	Construction and production.

This time frame should be to acquire all required data in order to carry out a systematic environmental study of the area and address all concerns which government and other interested agencies will have with respect to the Williams Creek Project.

RAQ:523

BENEFIT TO THE YUKON:

The Williams Creek Project will provide an investment of between $30 and $40 million of development capital directly into the Yukon Territory. Once in production, the project will provide on average $6 million per annum to the local Carmacks and Whitehorse economies. Direct employment will be 82 individuals, all living in the Carmacks area with substantial benefit to the local economies of Carmacks and to a lesser extent to Whitehorse. The project will provide some 700-man years of employment, see in excess of $60 million paid in local wages, and generation of gross sales of some $200 million, monies which will be taxed and find their ways back into the Yukon government of the life of the project. The power requirements may assist Yukon Power in justifying additional production, and the establishment of a viable operation at Williams Creek may in fact have a triggering effect by enhancing the production possibilities of the Mintor Copper property located 40 kilometres to the north and also the development of a number of smaller satellitic ore bodies in the Dawson Range located on the west side of the Yukon River.

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

JANUARY 1991

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M5H 2C7

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FAX: (416) 362-1933

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FIGURES

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EXHIBITS

APPENDICES

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 <u>Mineable Reserves</u>

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 (10,600 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

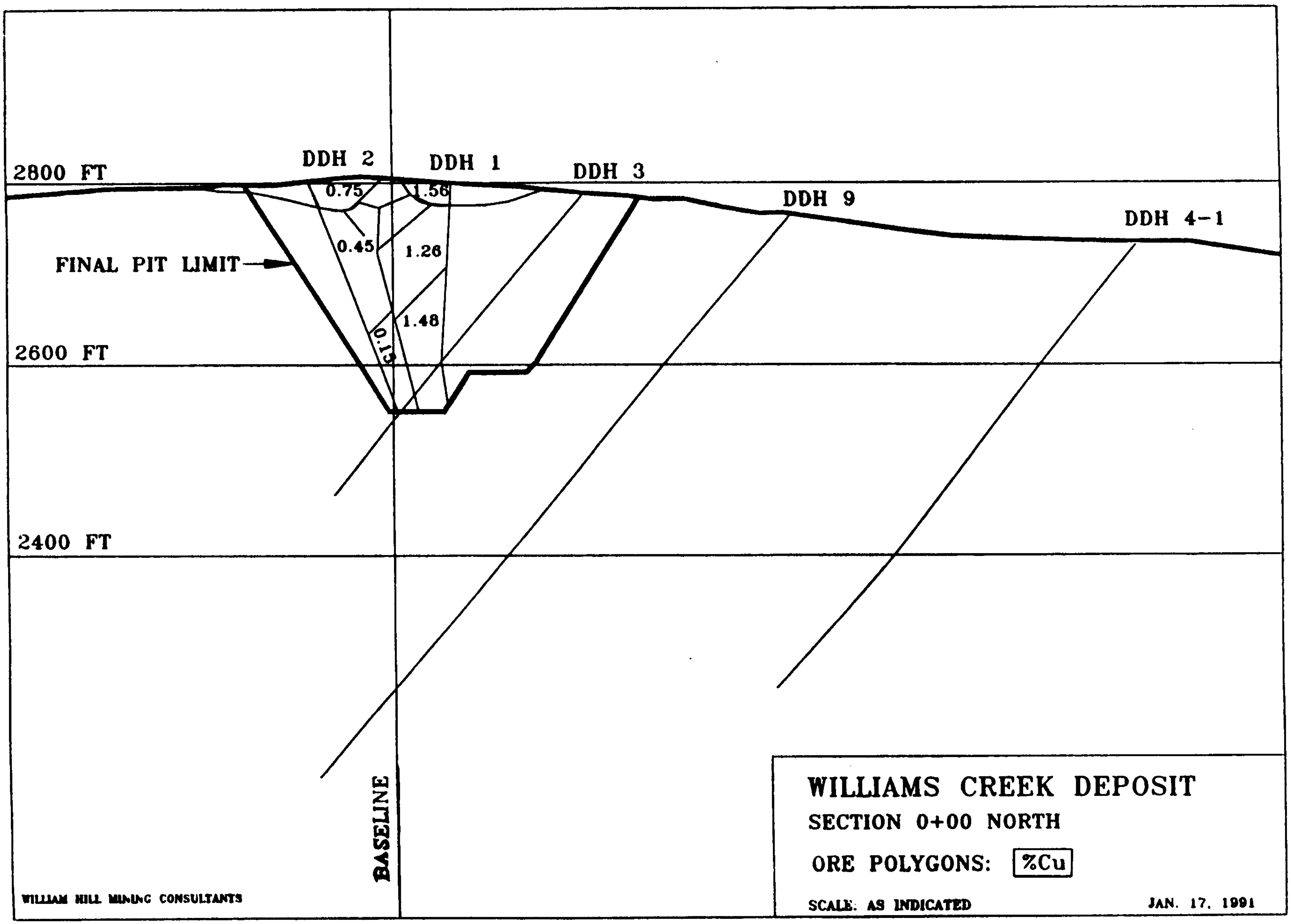
HMC has examined the capital cost estimates included in the Wright Engineers report of May 1990 and considers the figures to be realistic for the Williams Creek operation. In summary, the estimates for the operation are as follows:

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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2400 FT
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DDH 1
DDH 3
DDH 9
DDH 4-1
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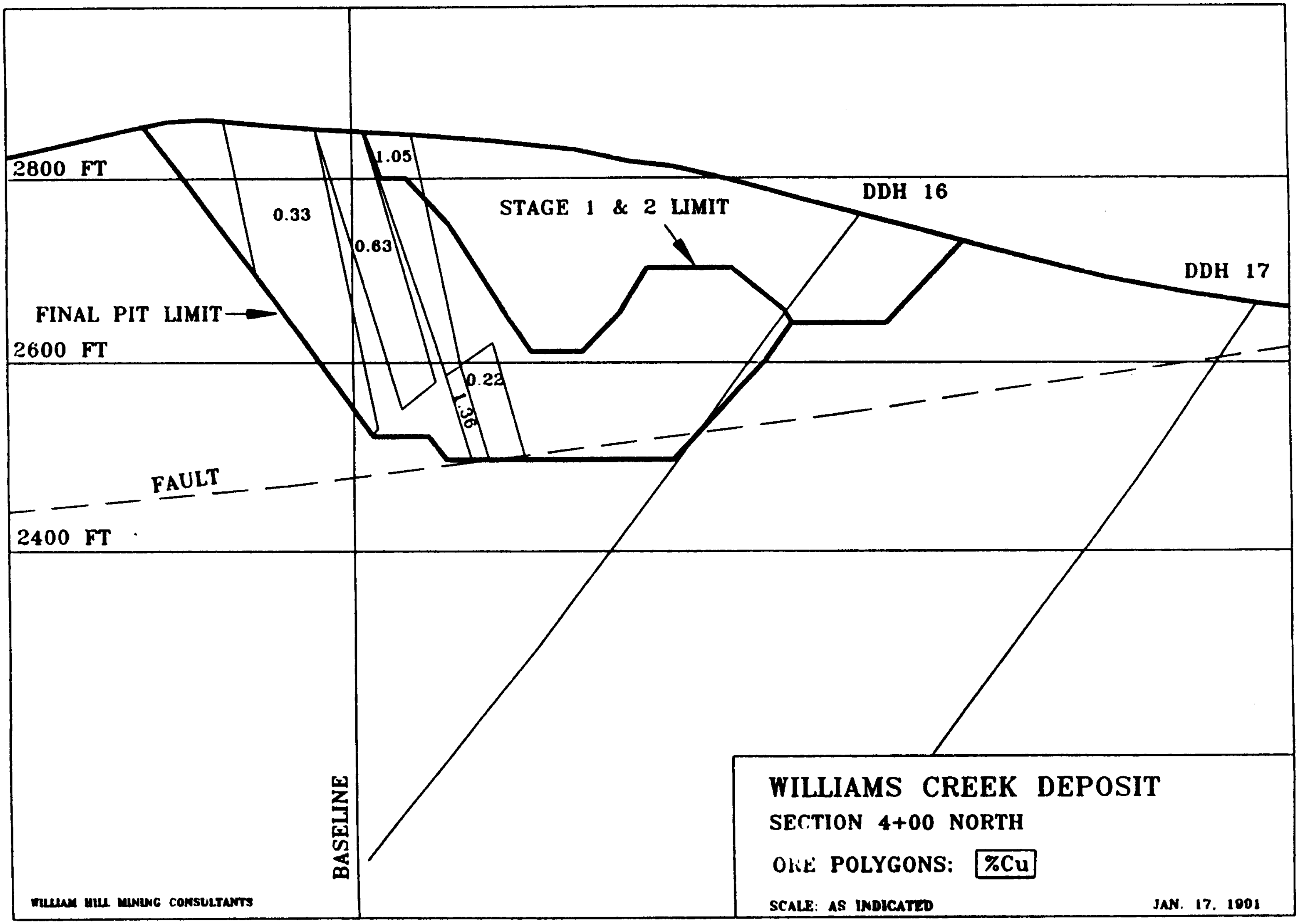
1.56
0.45
1.26
1.48
0.15
FINAL PIT LIMIT
BASELINE
WILLIAMS CREEK DEPOSIT
SECTION 0+00 NORTH
ORE POLYGONS: %Cu
SCALE: AS INDICATED
JAN. 17, 1991
WILLIAM HILL MINING CONSULTANTS

FIGURE 2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0.63
0.22
1.36
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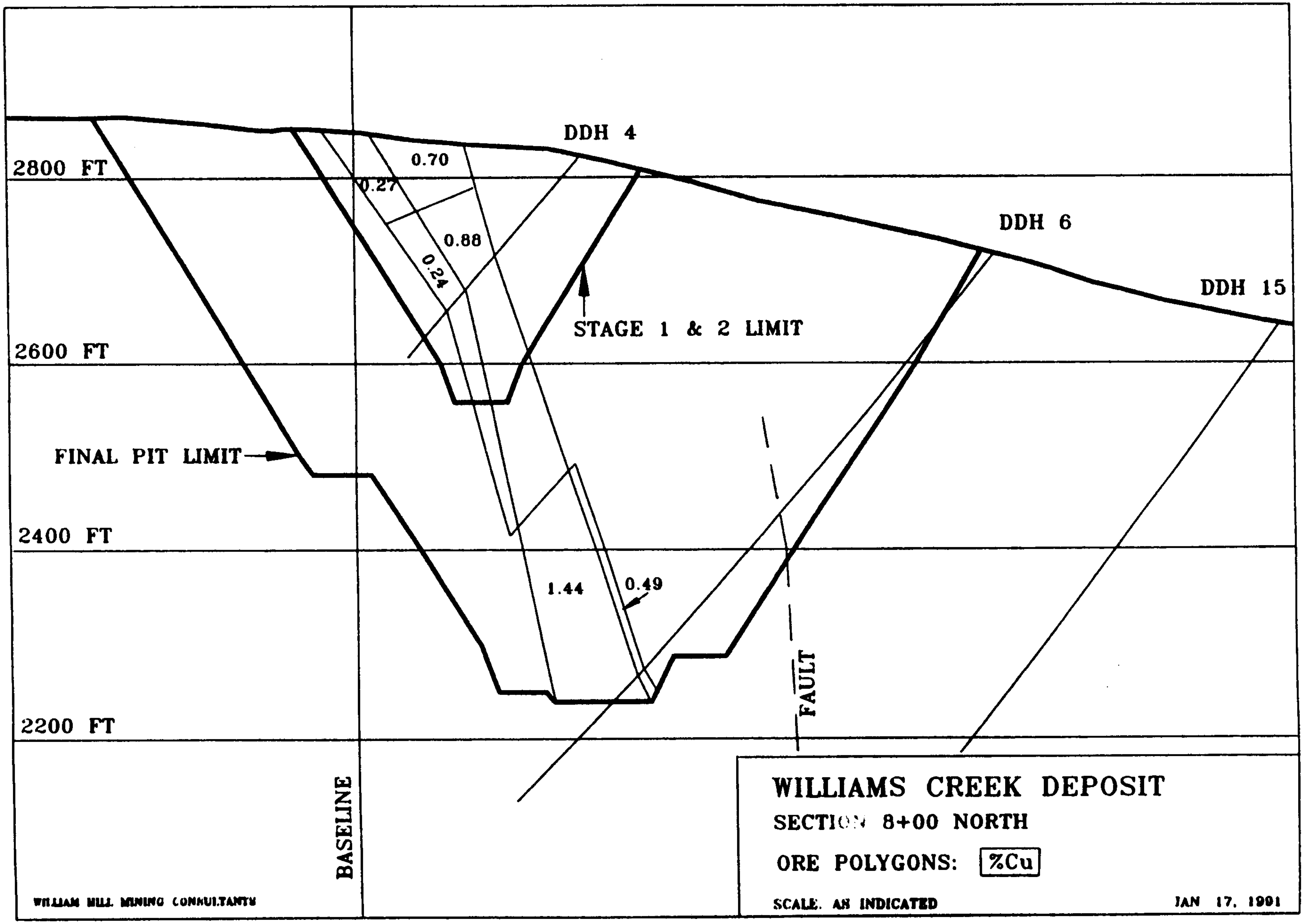
DDH 16
DDH 17
FINAL PIT LIMIT
FAULT
BASELINE
WILLIAMS CREEK DEPOSIT
SECTION 4+00 NORTH
ORE POLYGONS: %Cu
SCALE: AS INDICATED
JAN. 17, 1991
WILLIAM HILL MINING CONSULTANTS

FIGURE 3



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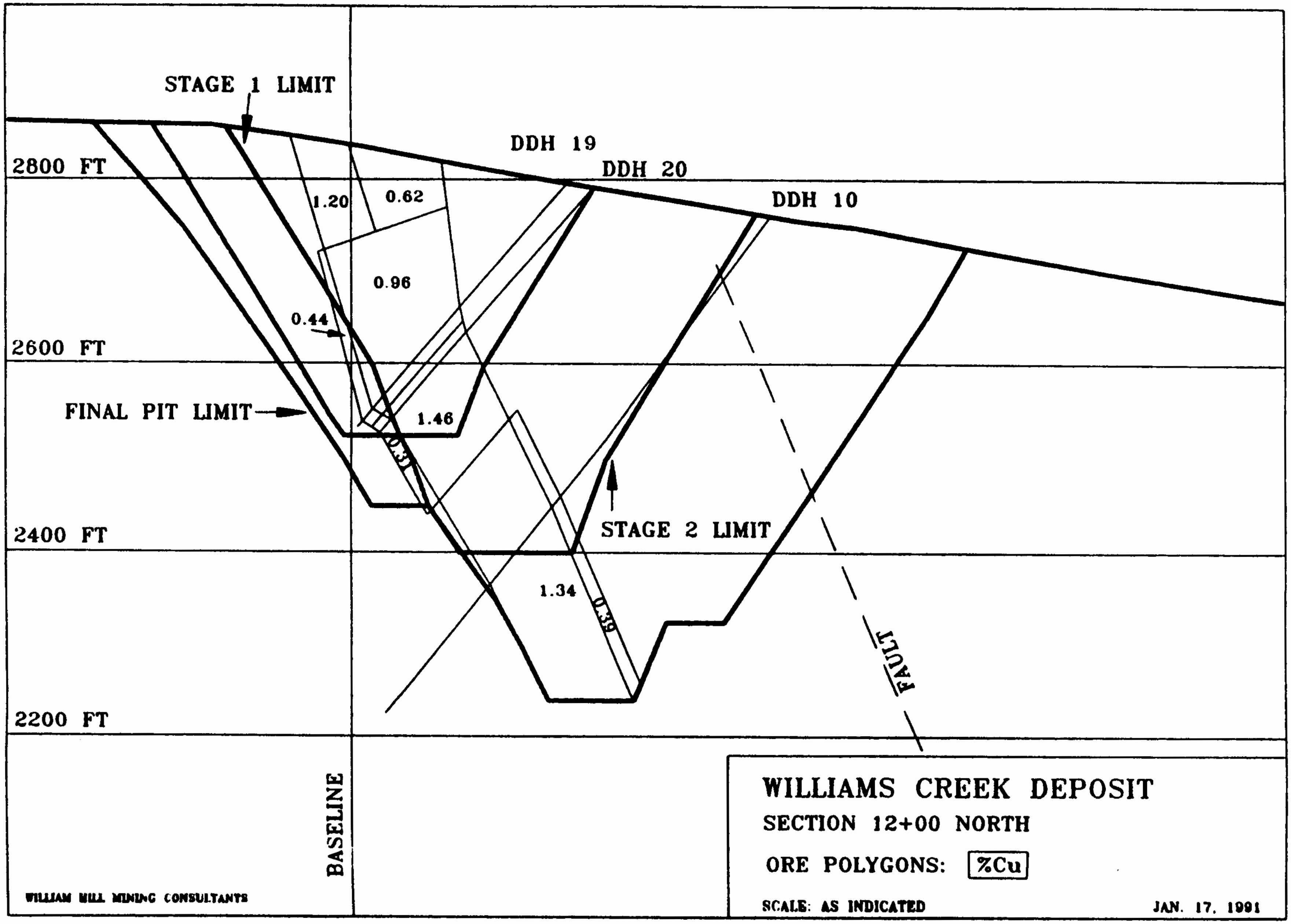
2800 FT
0.27
0.88
0.24
DDH 6
DDH 15
STAGE 1 & 2 LIMIT
2600 FT
FINAL PIT LIMIT
2400 FT
1.44
0.49
FAULT
2200 FT
BASELINE
WILLIAMS CREEK DEPOSIT
SECTION 8+00 NORTH
ORE POLYGONS: %Cu
SCALE. AS INDICATED
JAN 17, 1991
WILLIAM HILL MINING CONSULTANTS

FIGURE 4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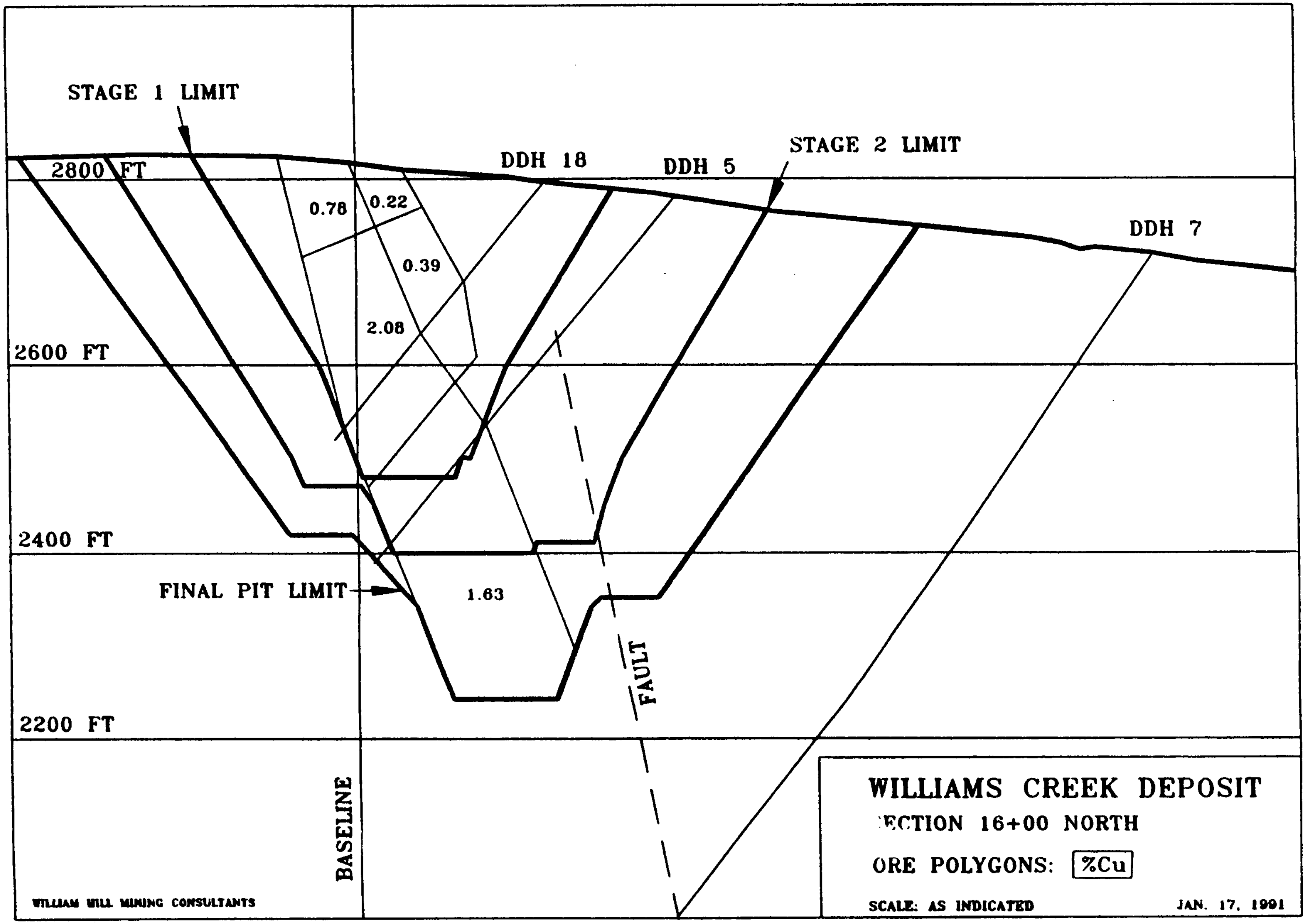
0.31
2400 FT
STAGE 2 LIMIT
1.34
0.39
FAULT
2200 FT
BASELINE
WILLIAM HILL MINING CONSULTANTS
WILLIAMS CREEK DEPOSIT
SECTION 12+00 NORTH
ORE POLYGONS: %Cu
SCALE: AS INDICATED
JAN. 17, 1991

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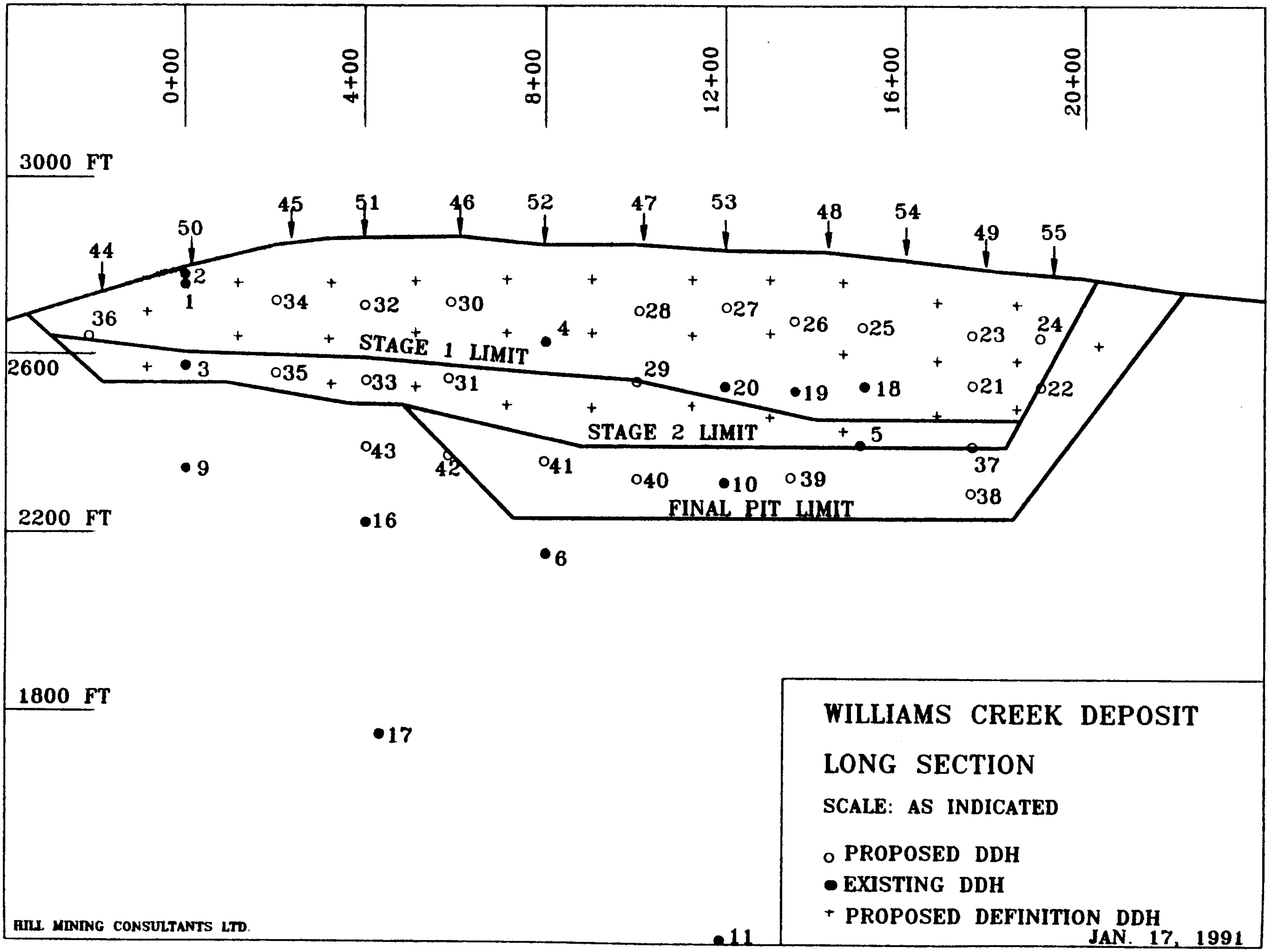
2400 FT
FINAL PIT LIMIT
1.63
FAULT
2200 FT
BASELINE
WILLIAM HILL MINING CONSULTANTS
WILLIAMS CREEK DEPOSIT
SECTION 16+00 NORTH
ORE POLYGONS: %Cu
SCALE: AS INDICATED
JAN. 17, 1991

FIGURE 6



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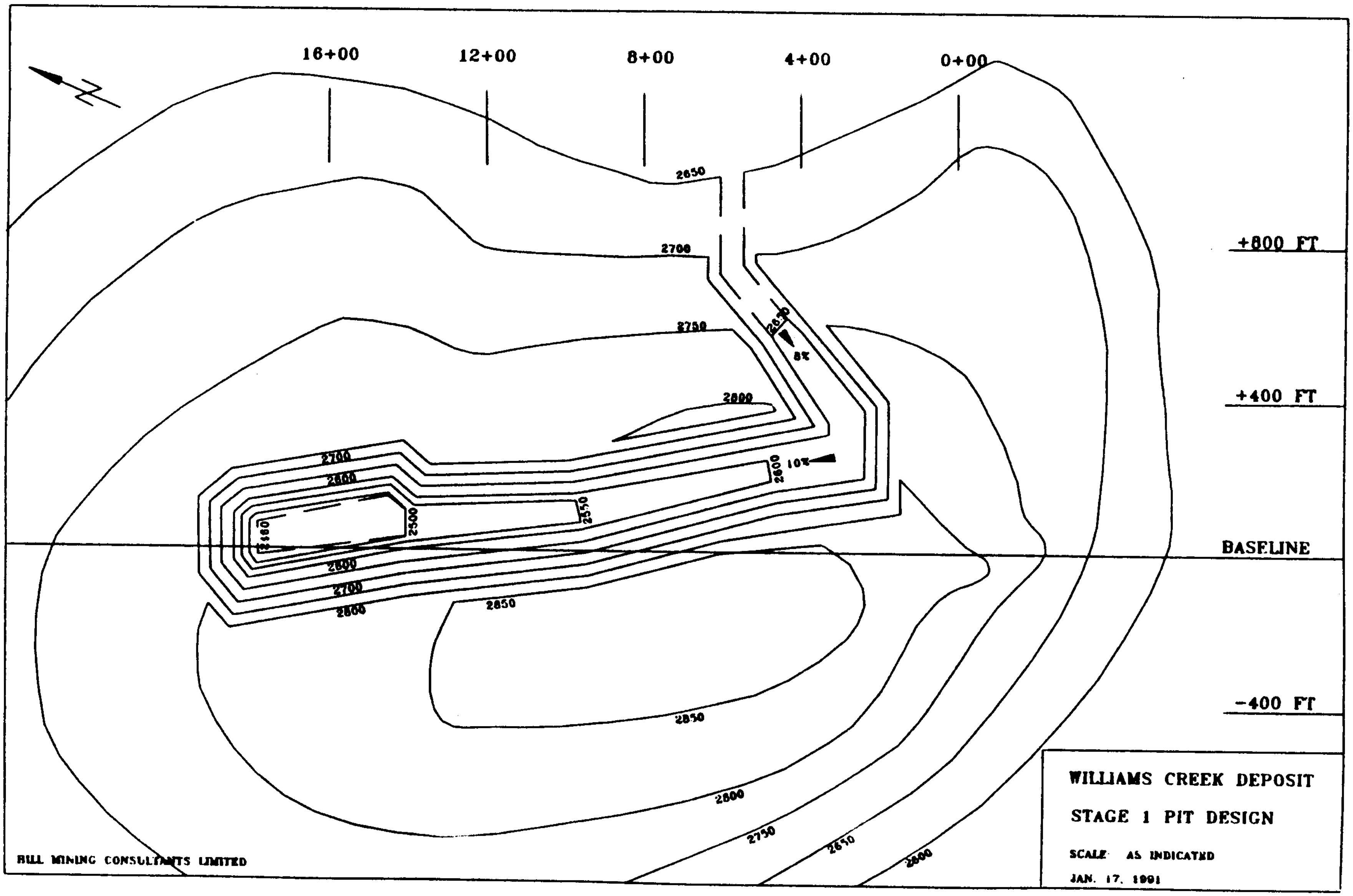
28
27
26
25
23
24
4
STAGE 1 LIMIT
3
35
33
31
29
20
19
18
21
22
STAGE 2 LIMIT
5
37
9
43
42
41
40
10
39
38
FINAL PIT LIMIT
16
6
17
11
WILLIAMS CREEK DEPOSIT
LONG SECTION
SCALE: AS INDICATED
PROPOSED DDH
EXISTING DDH
PROPOSED DEFINITION DDH
JAN. 17, 1991
HILL MINING CONSULTANTS LTD.

FIGURE 7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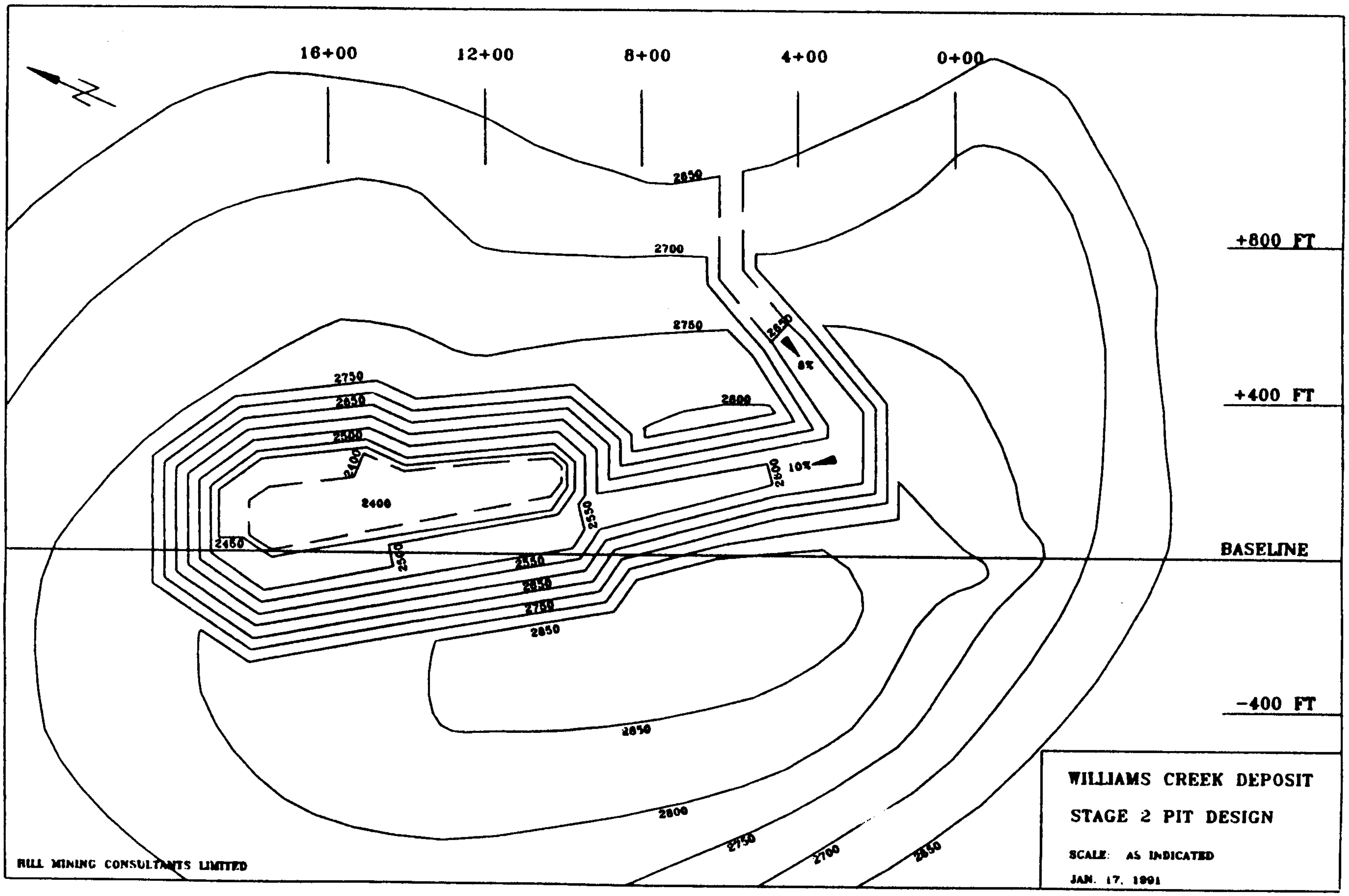
2700
2750
2800
8%
10%
2600
2550
2500
2850
WILLIAMS CREEK DEPOSIT
STAGE 1 PIT DESIGN
SCALE: AS INDICATED
JAN. 17, 1991
BULL MINING CONSULTANTS LIMITED

FIGURE 8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2700
2750
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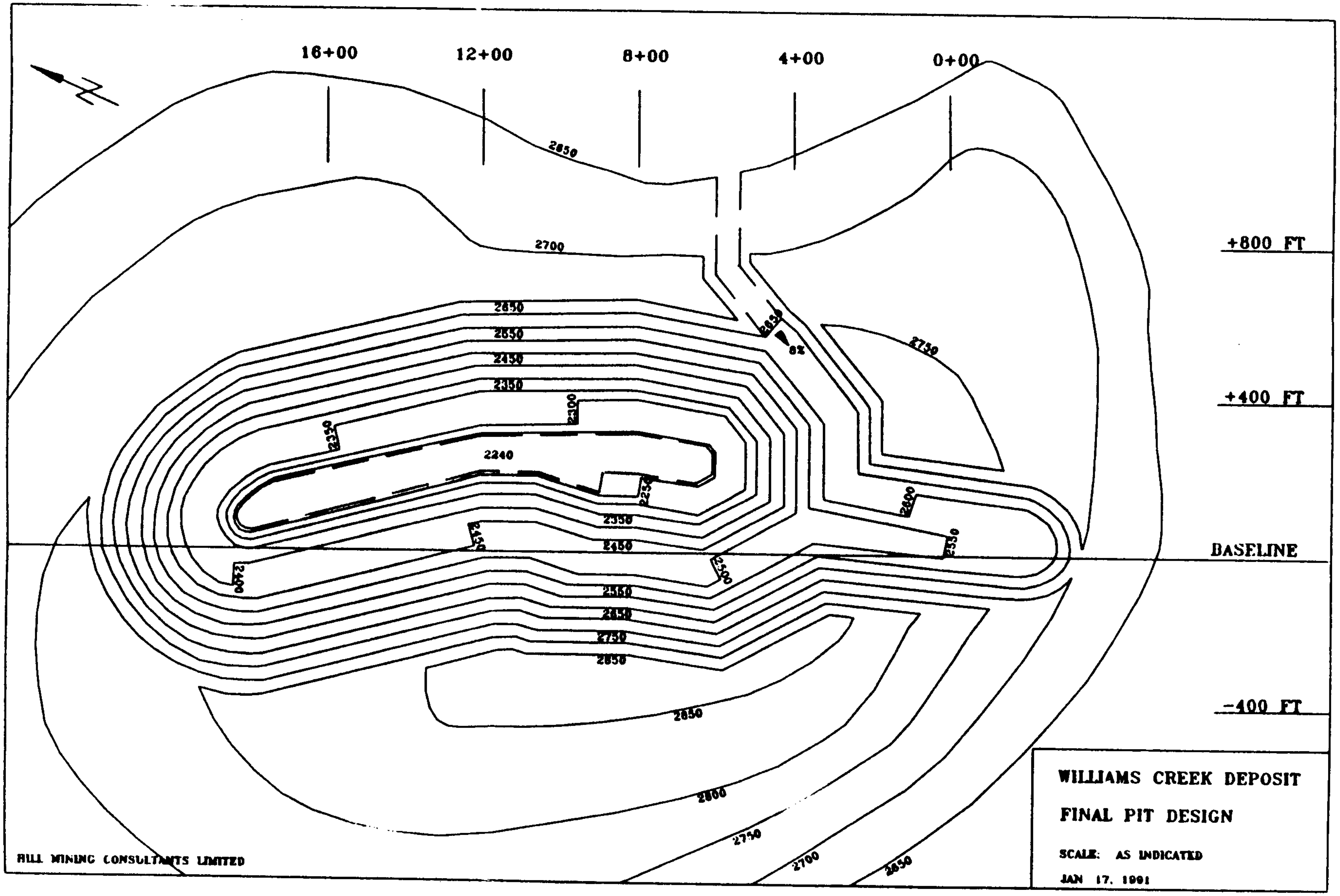
10%
2600
2650
2500
2400
2450
2550
2650
2750
2850
2850
2800
2750
2700
2650
RILL MINING CONSULTANTS LIMITED
WILLIAMS CREEK DEPOSIT
STAGE 2 PIT DESIGN
SCALE: AS INDICATED
JAN. 17, 1991

FIGURE 9



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8+00
4+00
0+00
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2700
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2550
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2350
2300
2350
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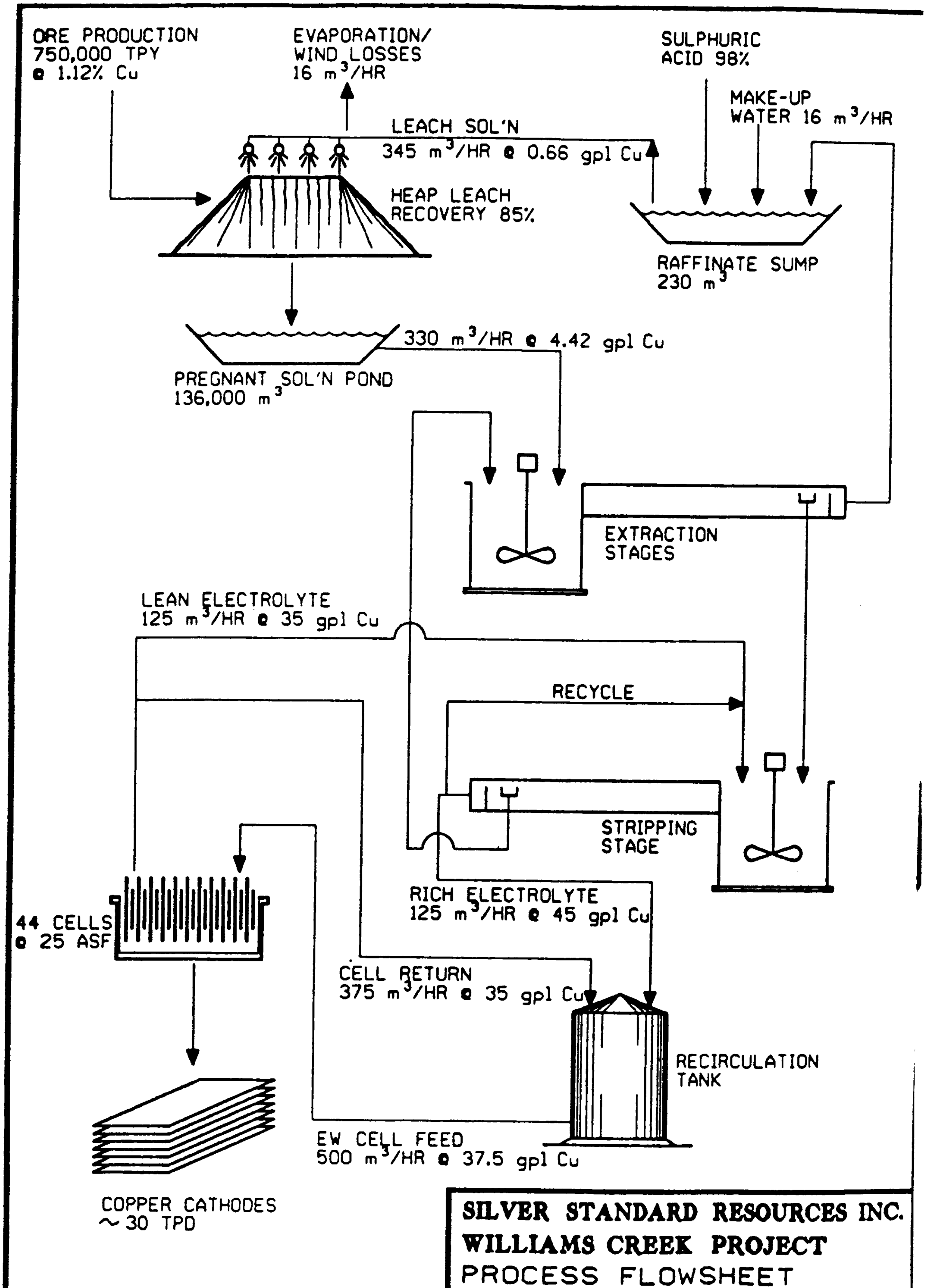
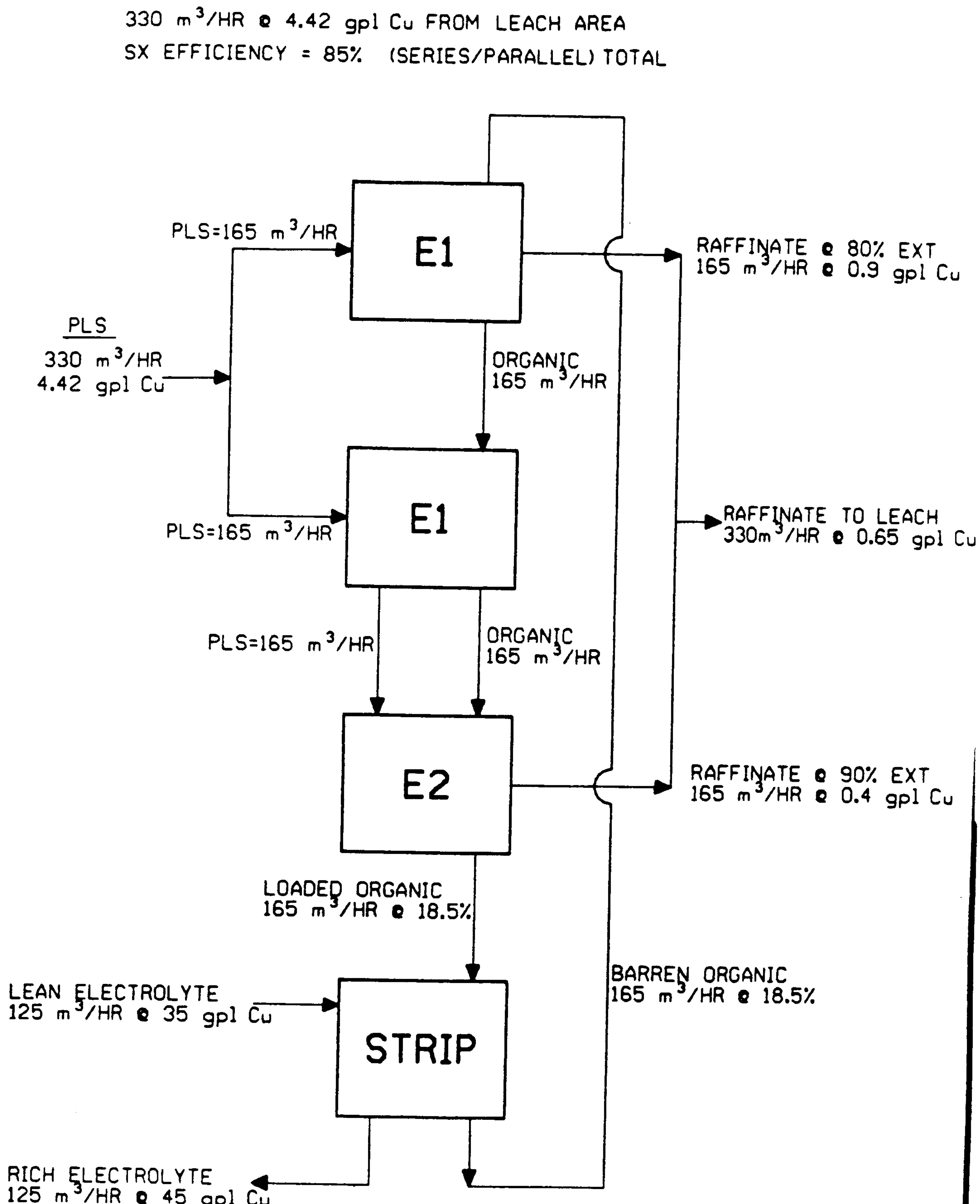
2450
2450
2400
2500
2550
2650
2750
2850
2850
2800
2750
2700
2650
2750
2600
2550
8%
+800 FT
+400 FT
BASELINE
-400 FT
WILLIAMS CREEK DEPOSIT
FINAL PIT DESIGN
SCALE: AS INDICATED
JAN 17, 1991
HILL MINING CONSULTANTS LIMITED

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

	+0.5% Cu		0 to 0.49% Cu		TOTAL	
SECTION	TONS	% CU	TONS	% CU	TONS	% 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	2,286	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
TRANSPORTATION @ $/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	$196,971
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	$36,708	$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.

WILLIAMS CREEK

STAGE 2

MATERIAL MOVEMENT

YEAR 4								YEAR 5			
LEVEL	TONS	% CU	WASTE	LEVEL	TONS	% CU	WASTE	LEVEL	TONS	% CU	WASTE
2600	26.0	0.64	1,034	2400	16.0	0.39	-				
2550	86.2	1.03	874	2400	196.0	1.34	-				
2500	200.0	1.13	652	2400	19.6	1.63	-				
2450	408.6	1.39	494					2850	20.0	0.33	114
2400	224.4	1.63	126					2800	224.0	0.44	604
								2750	441.4	0.69	1,480
	945.2	1.34						2700	28.2	1.26	
+ Dilution	54.8	0.67									
									945.2	0.79	
TOTAL	1,000	1.30	3,180					+ Dilution	54.8	0.395	
								TOTAL	1,000	0.77	2,198

Strip Ratio = 3.18:1

Strip Ratio = 2.20:1

WILLIAMS CREEK

STAGE 3

MATERIAL MOVEMENT

YEAR 6

LEVEL	TONS	% CU	WASTE
2700	429.8	0.66	-
2650	385.4	0.70	2,350
2600	40.0	0.15	7,556
2600	5.2	0.45	424
	10.0	0.22	
	10.8	0.33	
	945.2	0.70	
+ Dilution	54.8	0.35	
TOTAL	1,000	0.68	5,330

Strip Ratio = 5.33:1

YEAR 7

LEVEL	TONS	% CU	WASTE
2600	41.0	1.05	2,225
2600	72.0	0.63	
2600	109.2	0.33	
2550	386.7	0.68	2,266
2500	316.8	0.66	2,018
2450	19.0	1.44	
2450	0.5	0.49	
	945.2	0.66	
+ Dilution	54.8	0.33	
TOTAL	1,000	0.64	6,509

Strip Ratio = 6.51:1

YEAR 8

LEVEL	TONS	% CU	WASTE
2450	177.9	0.69	-
2400	146.8	1.15	1,448
2350	523.0	1.39	1,204
2300	97.5	1.44	870
	945.2	1.23	
+ Dilution	54.8	0.615	
TOTAL	1,000	1.19	3,522

Strip Ratio = 3.52:1

YEAR 9

LEVEL	TONS	% CU	WASTE
2300	434.5	1.38	456
2250	454.8	1.36	160
2240	25.0	1.63	
2240	30.9	1.44	
	945.2	1.38	
+ Dilution	54.8	0.69	
TOTAL	1,000	1.34	616

Strip Ratio = 0.62:1

YEAR 9

LEVEL	TONS	% CU	WASTE
2240	28.0	1.34	-
2240	1.4	1.44	
	29.4	1.34	
+ Dilution	1.6	0.67	
TOTAL	31.0	1.31	

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	DRILLTECH	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
ACCOMODATIONS					
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		10,600 ft

Preproduction Drilling

Holes marked with "+" (32 holes)

200 foot depth	300 ft holes	32 holes	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		350,000
TOTAL COST	$728,000	Say,	$750,000

Silver Standard Mines Limited

Williams Creek Property Review

Project No. 1918 May 14, 1990

DISCLAIMER

The following estimate/report is based in part on information not within Wright Engineers Limited control. It is believed that the estimates and conclusions contained therin will be reliable under the conditions and subject to the qualifications set forth. However, Wright Engineers does not warrant or guarantee their accuracy. Use of such estimates/reports shall, therefore, be at the user's sole risk. Such use shall constitute a release and agreement to defend and indemnify Wright Engineers from and against any liability (including but not limited to liability for special, indirect or consequential damages) in connection with such use, whether liability is asserted to arise in contract, negligence, strict liability or other theory of law.

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SECTION 1

INTRODUCTION

SECTION 1

INTRODUCTION

The Williams Creek property is located in the Yukon Territory approximately 30 miles by road northwest of the town of Carmacks. Preliminary exploration results by Archer Cathro have indicated the presence of oxide and carbonate copper mineralization outcropping on surface and at depth in diamond drill boreholes.

Wrights have been asked by Silver Standard Resources to review their documents relating to ore reserves, mining and metallurgy. The principle reasons for this review were to:

1. Generate an operating and capital cost estimate of this copper property for mining, followed by dump leach, solvent extraction and electrowinning of copper.

2. Make further recommendations with regards to metallurgical testing and exploration directed at mine design.

Drill logs and cross sections provided by Silver Standard were reviewed and recommendations were made with regards to further drilling requirements. The data do not warrant further reserve calculations until further exploration is completed. Basic criteria were established for the development of a mine design assuming that economic reserves could be proven by additional exploration. Mining capital and operating cost estimates were developed for a theoretical 750,000 tonne per year heap leach operation.

Metallurgical test results from bottle roll and column leach tests carried out by Bacon, Donaldson and Associates and Coastech Research Inc. have been reviewed. Based upon these very preliminary results order of magnitude capital and operating cost estimates have been prepared. Recommendations for further work have been made to improve the data base.

SECTION 2

SUMMARY

SECTION 2

SUMMARY

The geological data currently available are not adequate to produce a mineable reserve for the purposes of mine design or scheduling. To evaluate the Williams Creek prospect in terms of operating and capital costs associated with a heap leach copper operation, Wright have assumed that further drilling would provide an open pit mineable ore reserve of approximately 8.0 million tonnes grading 1.12% copper. This reserve will be mined and processed by leaching, solvent extraction and electrowinning at an annual rate of 750,000 tonnes ore at a constant waste to ore strip ratio of 2:1.

The capital cost to mine, crush and process the ore is $39.9 million. Given this operating scenario and 85% recovery Wright have generated an order of magnitude operating cost of approximately Cdn. $0.85/lb of copper produced.

A list of recommendations is provided for further metallurgical and geological testwork.

SECTION 3

REVIEW OF MINING

SECTION 3

REVIEW OF MINING

3.1 DRILLING AND ORE RESERVES

The Williams Creek property was trenched and drilled during the 1970's by Archer, Cathro and Associates on behalf of an exploration syndicate. The drill holes were "BQ" size.

The copper mineralization occurs in a Paleozoic schistose to gneissic xenolith typically 30 meters thick dipping easterly at approximately 70 degrees. The host rock is part of a cretaceous granodiorite pluton. Primary mineralization is chalcopyrite and bornite. Oxidation occurs to a depth of 180 meters. The alteration copper minerals predominant in the oxide zone include malachite and azurite. The mineralized zone is observed over a strike length of 610 meters.

Preliminary geological reserves have been prepared by Archer Cathro on cross sections along 488 meters of strike length. Reserves were categorized according to three ore types based upon oxidation - enrichment criteria as follows:

- Leached oxide - assumed 100% copper oxide
- Enriched oxide - cut-off grade applied to oxide assay
- Sulphide - cut-off applied to copper assay

The ore reserves were further subdivided to hanging wall and footwall zones. The hanging wall zone is clearly lower grade than the footwall zone.

These reserves are useful as an indication of the property potential but cannot be used for detail mine design. A major infill diamond drilling program is required to define the continuity of the grade within the mineralized structure.

3.2 GEOSTATISTICAL ANALYSIS

A preliminary analysis of the drillhole data for the Williams Creek project has been done to determine the degree of ore reserve definition possible from the existing data, and to define the needs for further sampling information for realistic project valuation.

This section presents the results of this analysis, and makes recommendations for further work to be done.

3.2.1 Sample Data Preparation

Downhole assay results in 3 meter sections were available for total % copper and for % copper oxide in fourteen inclined surface drillholes, giving a total of 211 assay values.

All drillholes were on an approximate 50 degree dip at an azimuth of 220 degrees, and downhole sample distances were desurveyed to produce a data file of assay values with X, Y and Z coordinates of sample top and bottom.

In order to perform further analysis on a constant sample support size, the information was composited into a series of horizontal 3 meter benches, with composited values being allocated only where there was a minimum of 50% of the bench intersection containing assayed values.

The compositing process resulted in a total of 167 sections in 3 meter horizontal levels.

3.2.2 Statistical Analysis

Basic statistics for the % total copper and % copper oxide for the composite values are presented in Figures 1(a) and 1(b). The histogram of copper oxide in Figure 1(b) shows that

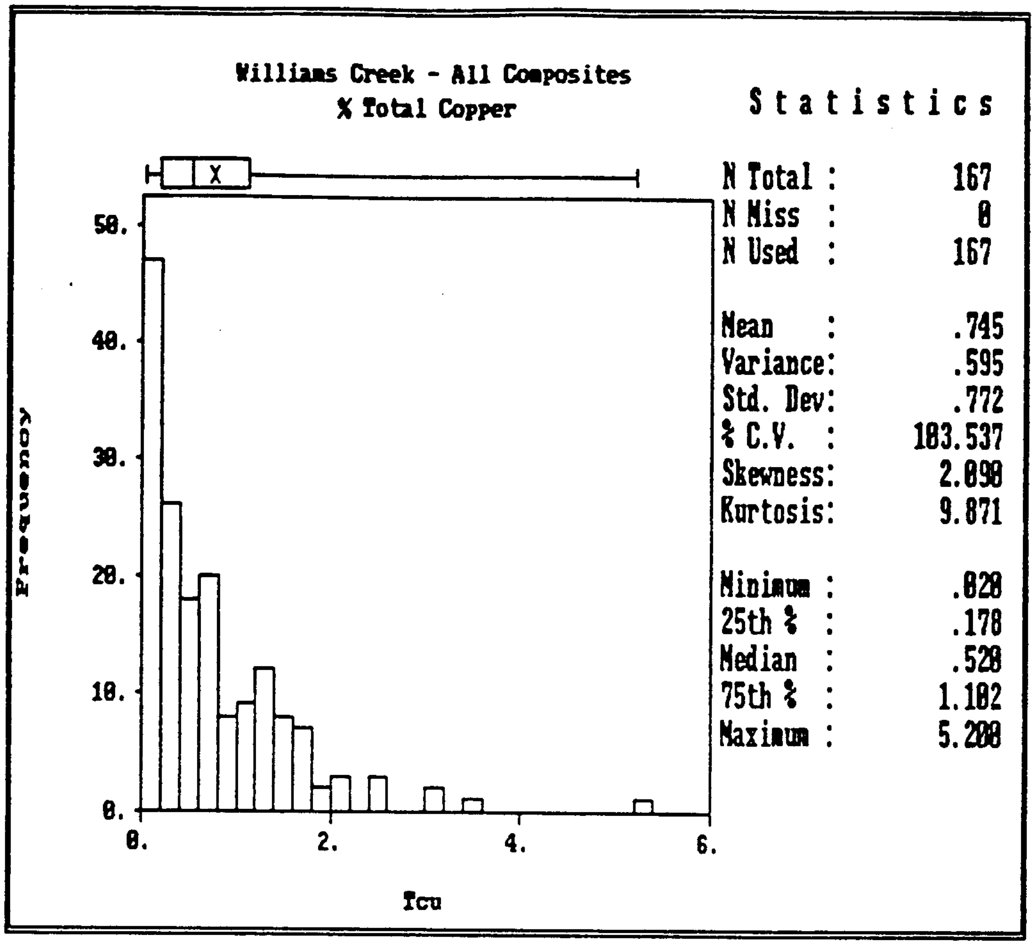


Figure 1(a)

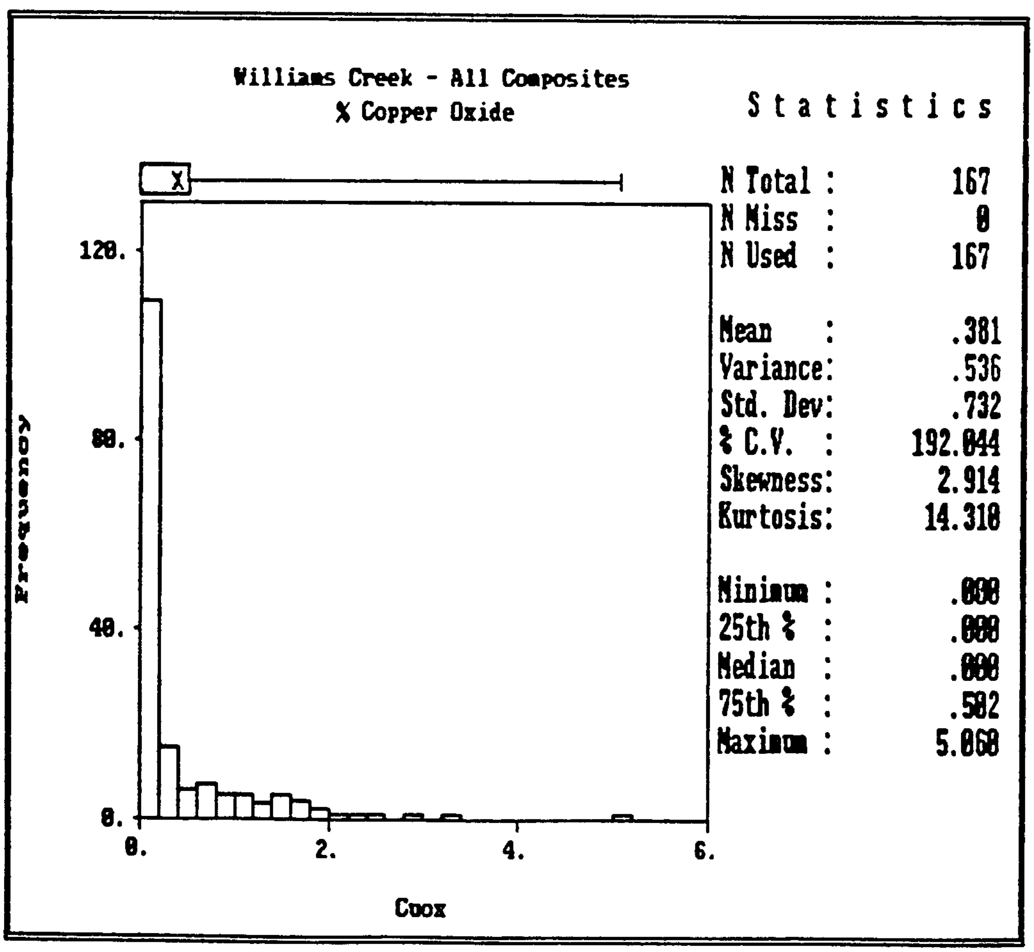


Figure 1(b)

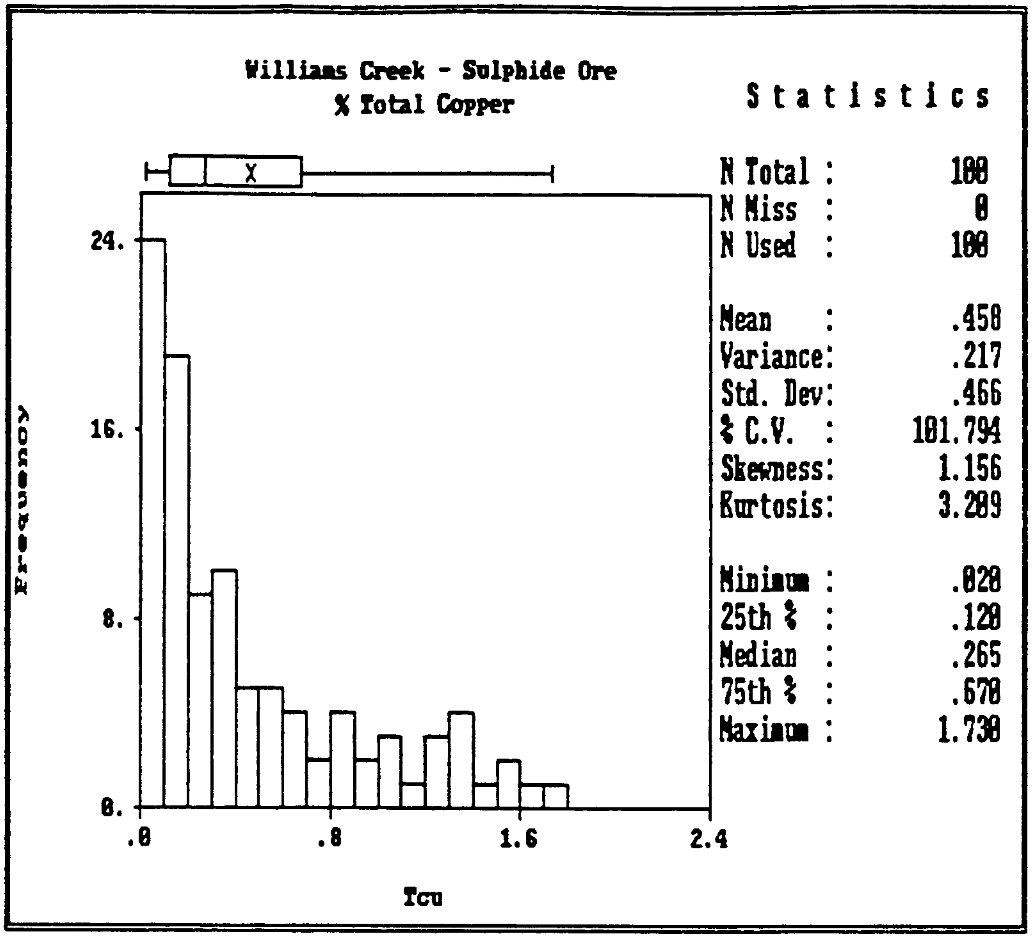


Figure 2(a)

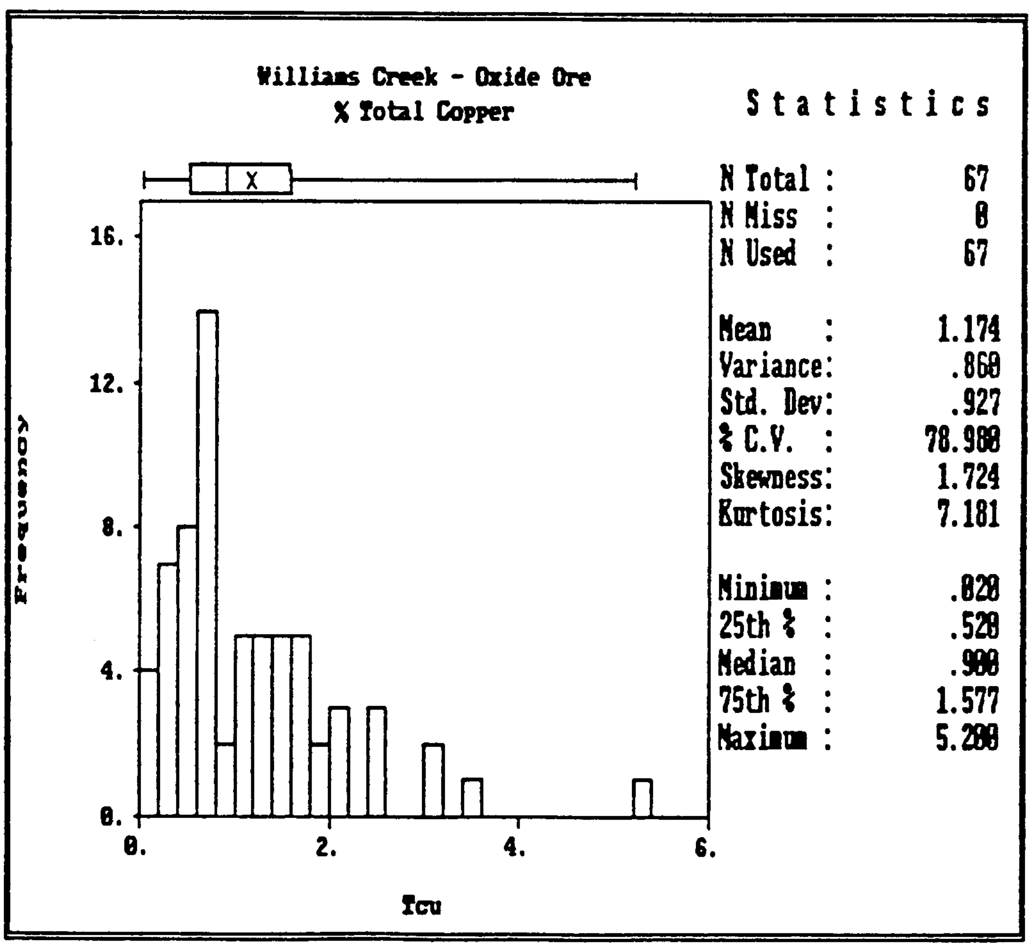


Figure 2(b)

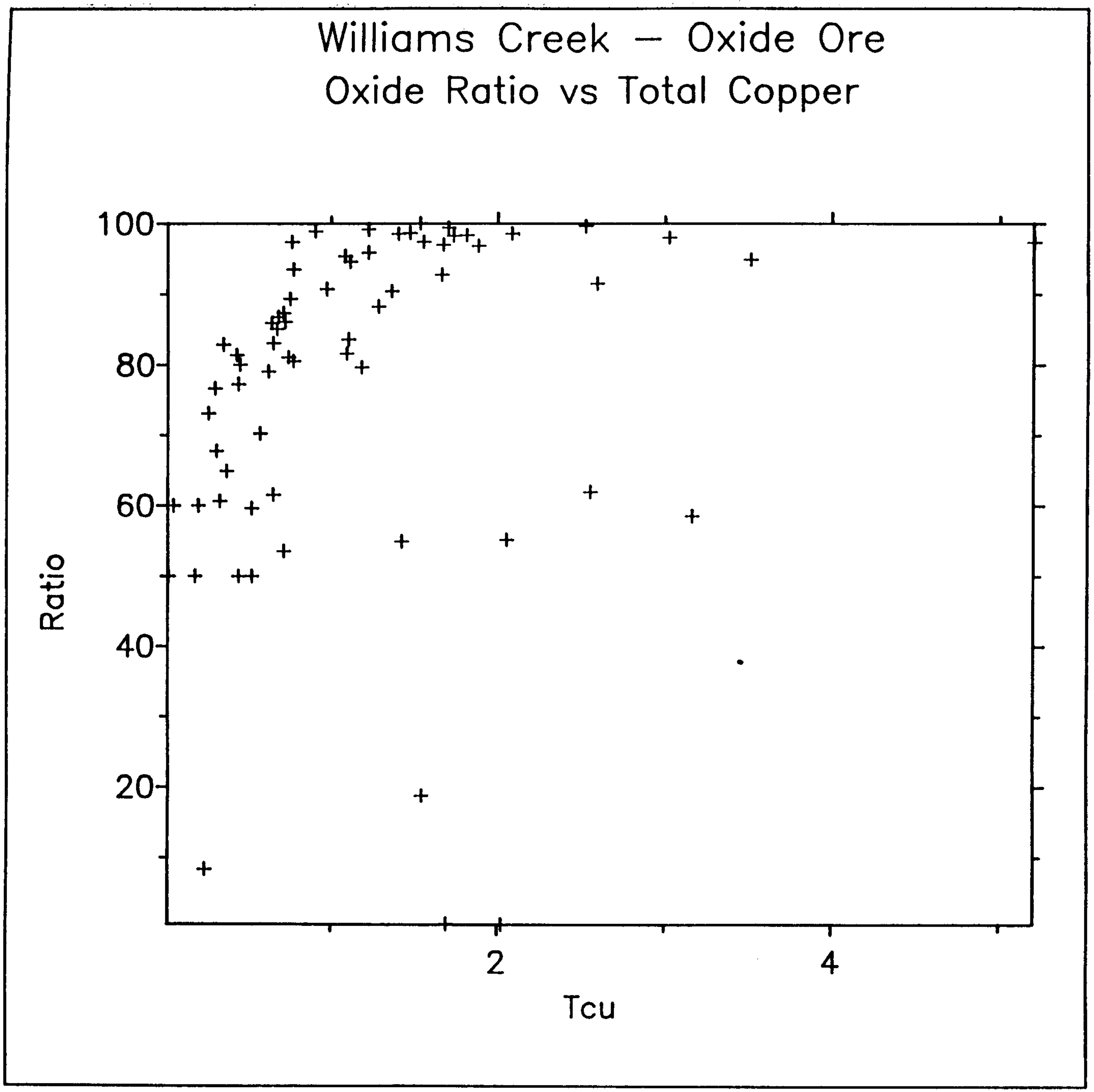

Williams Creek — Oxide Ore
Oxide Ratio vs Total Copper
100
80
60
40
20
Ratio
2
4
Tcu

Figure 3

more than half of the composite values have a zero copper oxide content, and there is clearly a distinct division between oxide copper and sulphide copper in the orebody.

After splitting the composite values into a sulphide zone, with zero copper oxide, and an oxide zone, the histograms of total copper for these two zones are shown in Figure 2(a) and (b). It is now seen that the oxide zone shows significantly higher assayed copper values, with an average value of 1.17% total copper as compared with 0.46% in the sulphide zone.

To determine the relationship between the total copper content and the degree of oxidation in the oxide zone, the scatter plot in Figure 3 shows that there is a tendency for higher total copper values to be associated with higher oxide ratios. Although this relationship is perhaps not statistically significant, it suggests that further emphasis should be given to establishing the possible metallurgical implications.

3.2.3 Spatial Data Analysis

To provide quick representations of the spatial distribution of total copper and oxide ratio, projected cross-sections of the composite values were produced. These are given in Figures 4(a) and (b), showing projected east/west and north/south cross sections respectively of the total copper composite values, and in Figures 5(a) and (b) for the oxide ratio.

Figures 4(a) and (b) show that higher copper values tend to be concentrated in the northern portion of the ore body, with a lower grade zone at depth towards the southeast. Initial potential for further ore definition would appear to be in a region bounded by 100530N and 101510N, as seen in Figure 4(b).

The projected sections of oxide ratio in Figures 5(a) and (b) show that the oxidized copper is confined to the shallower portions of the ore body, with little copper oxidation evident below an elevation of 640 meters.

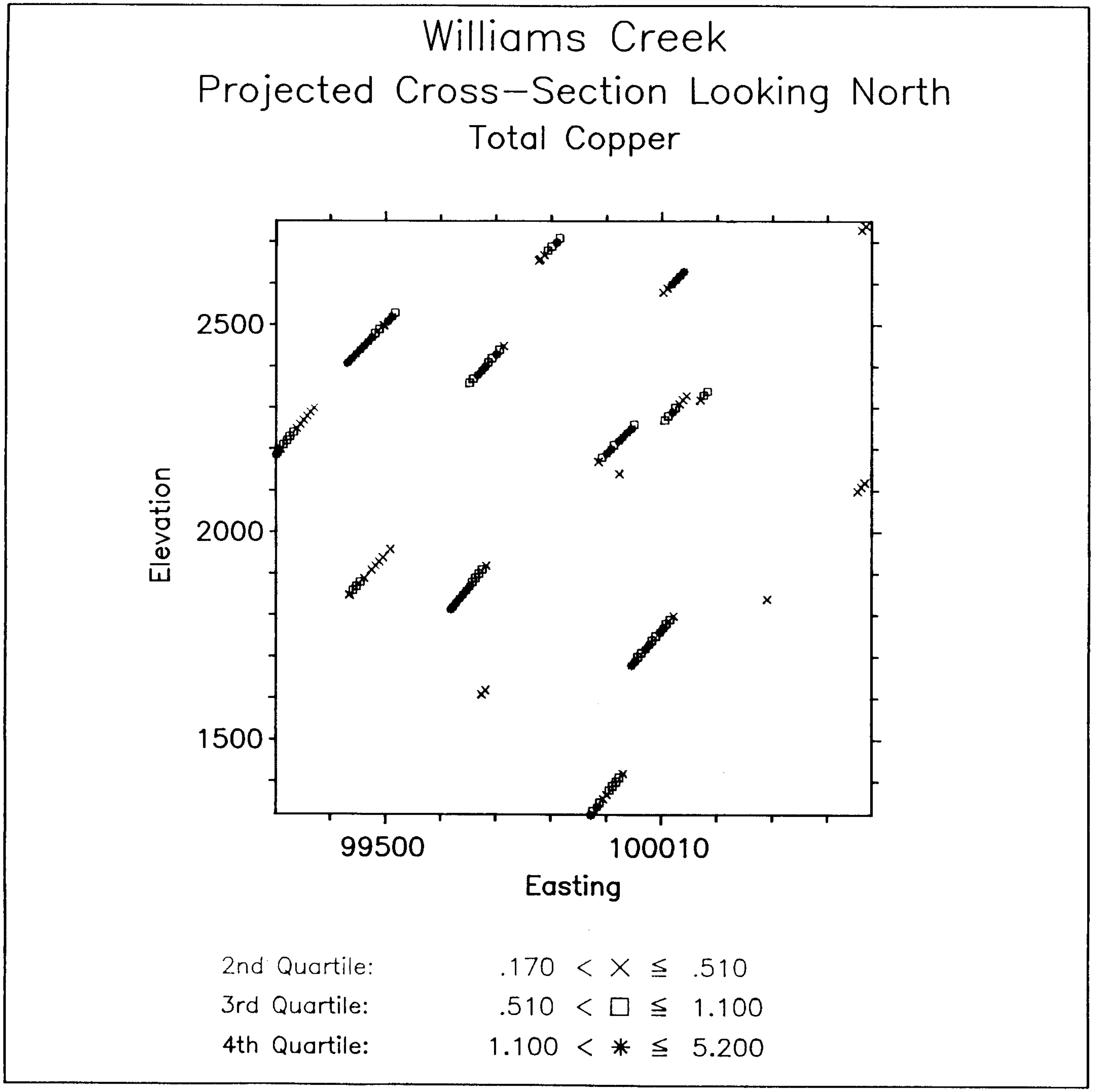
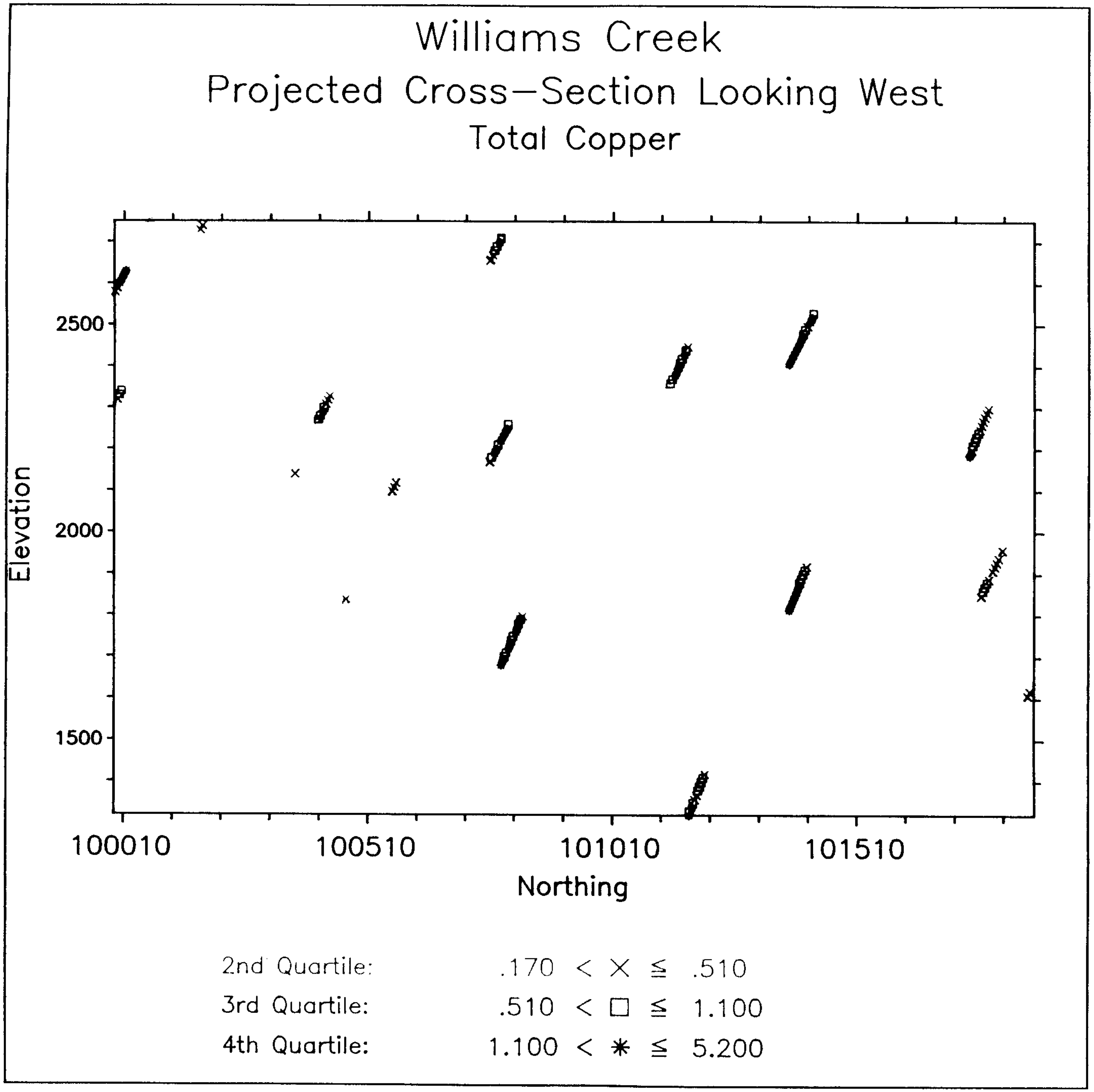

Williams Creek
Projected Cross-Section Looking North
Total Copper
2500
2000
1500
Elevation
99500
100010
Easting
2nd Quartile: .170 < X ≦ .510
3rd Quartile: .510 < □ ≦ 1.100
4th Quartile: 1.100 < ✱ ≦ 5.200

Figure 4(a)



Williams Creek
Projected Cross-Section Looking West
Total Copper
Elevation
2500
2000
1500
100010
100510
101010
101510
Northing
2nd Quartile: .170 < X ≤ .510
3rd Quartile: .510 < □ ≤ 1.100
4th Quartile: 1.100 < ✳ ≤ 5.200

Figure 4(b)

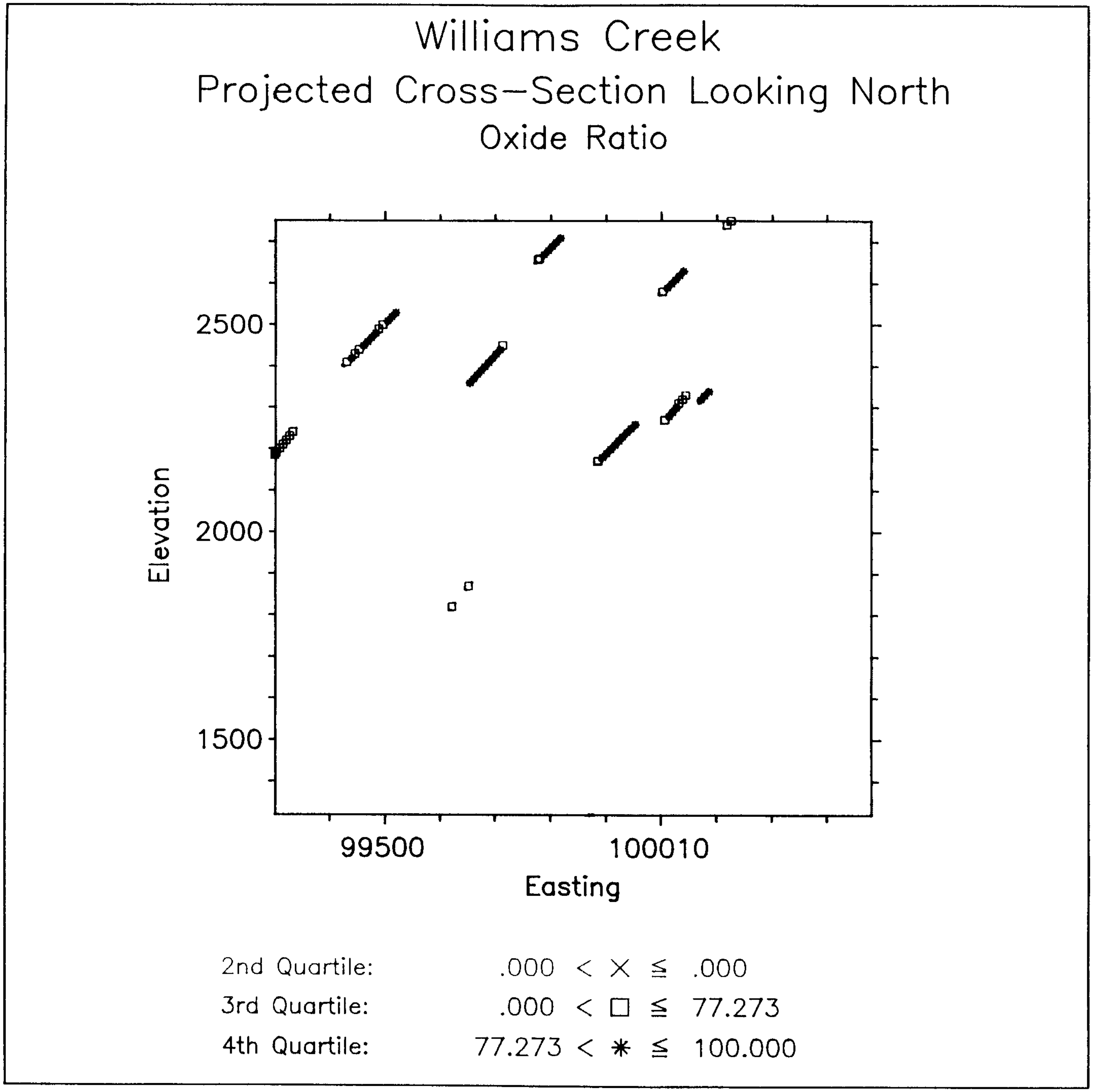
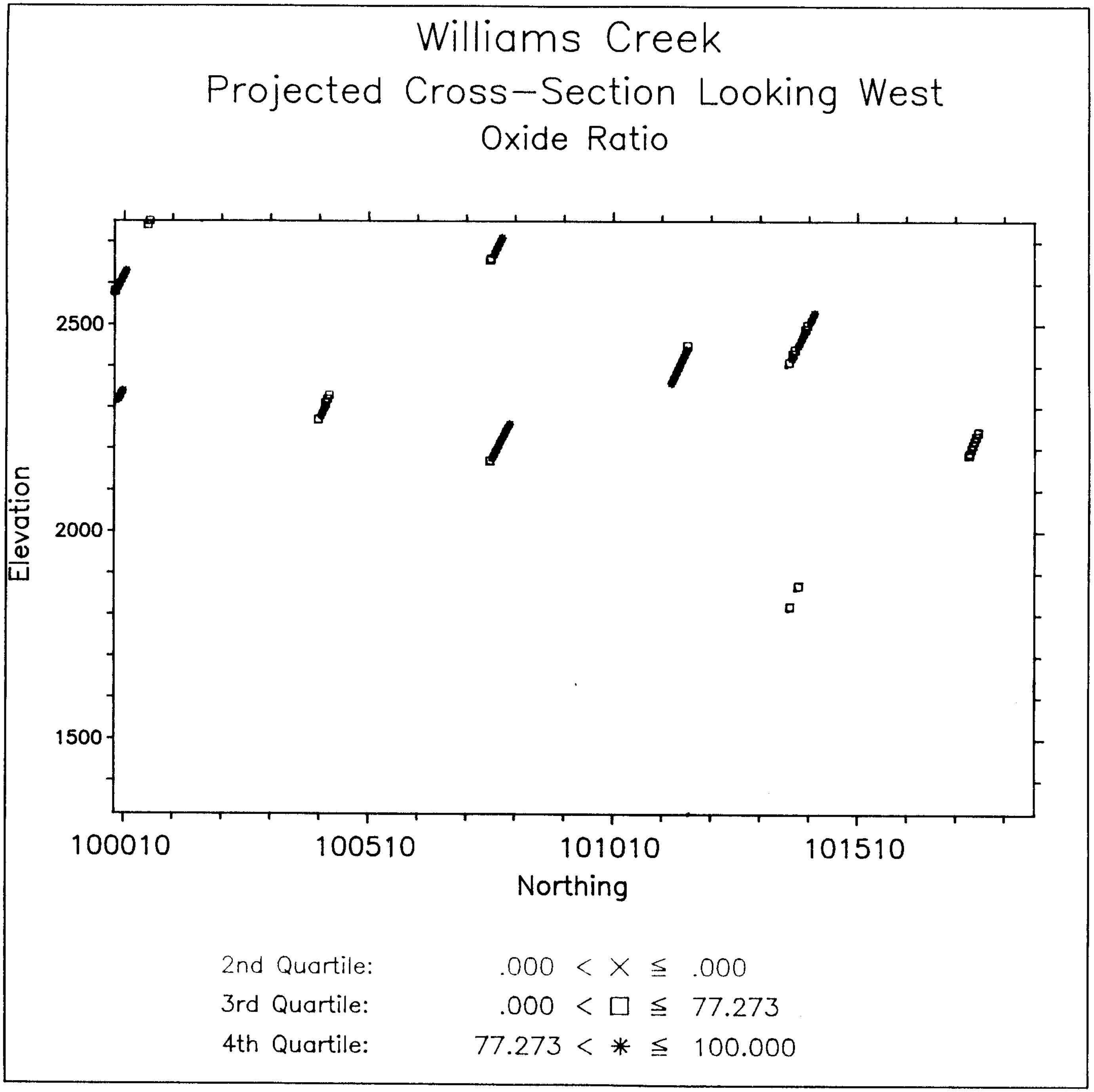

Williams Creek
Projected Cross–Section Looking North
Oxide Ratio
Elevation
2500
2000
1500
99500
100010
Easting
2nd Quartile: .000 < X ≦ .000
3rd Quartile: .000 < □ ≦ 77.273
4th Quartile: 77.273 < ✱ ≦ 100.000

Figure 5(a)



Williams Creek
Projected Cross-Section Looking West
Oxide Ratio
Elevation
2500
2000
1500
100010
100510
101010
101510
Northing
2nd Quartile: .000 < X ≦ .000
3rd Quartile: .000 < □ ≦ 77.273
4th Quartile: 77.273 < * ≦ 100.000

Figure 5(b)

WILLIAMS CREEK
Down-Hole Variogram - Total Cu (logs)

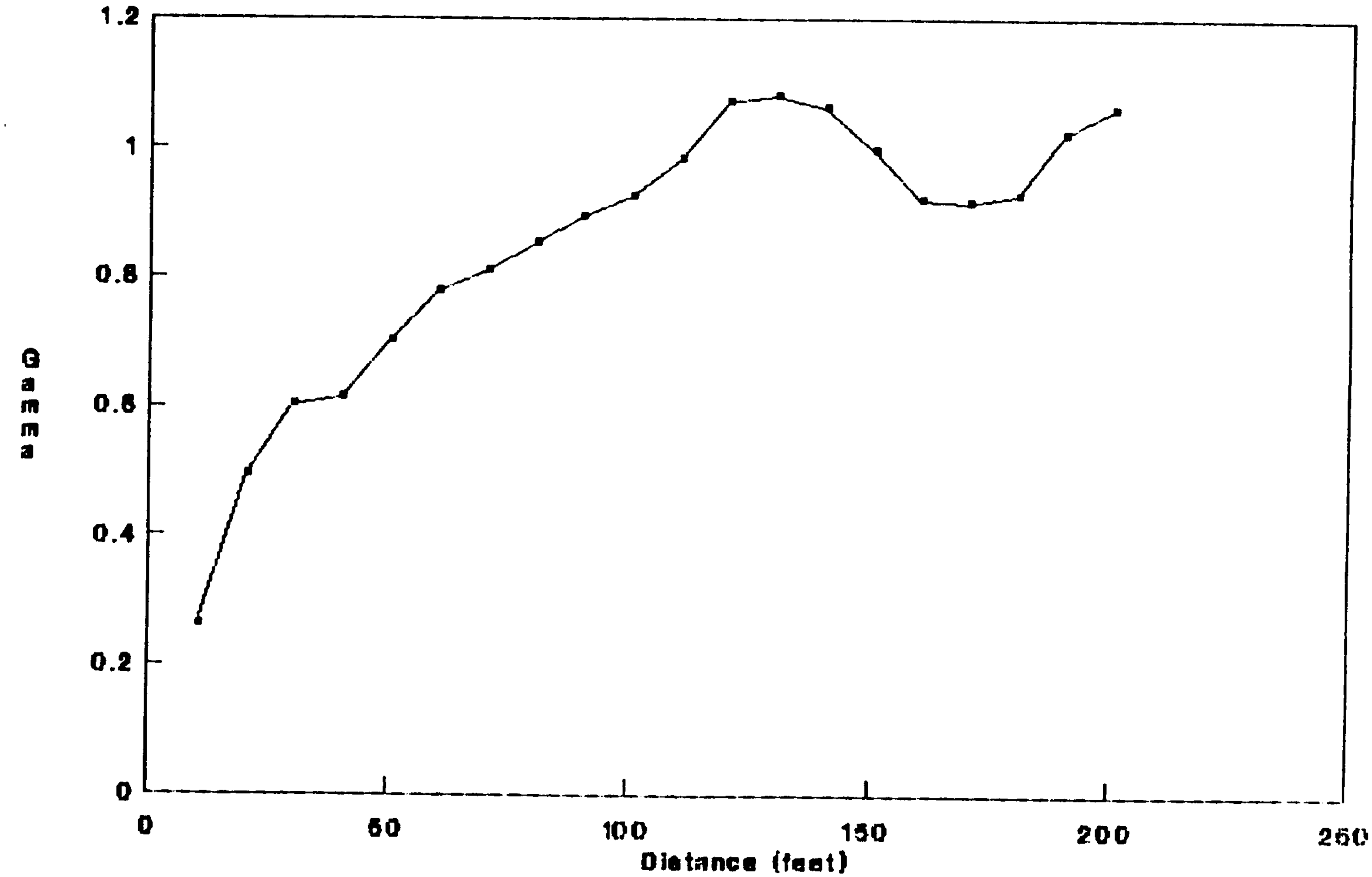


Figure 6

3.2.4 Variogram Analysis

In order to define the degree of spatial continuity in mineralization, an attempt was made to calculate directional variograms based on the composite data.

With the limited information available, it was not possible to obtain any clear definition of spatial correlational structure within the plane of the ore body. With a drillhole spacing of some 91 meters on strike, no distinct variogram was determined in the strike direction, and again sampling information was limited down dip.

Down hole variograms were computed on the 3 meter composites, and do show a distinct structure as shown in Figure 6. This variogram, calculated on logarithmically transformed total copper values, shows that there is a range of influence of some 36.6 meters, with a steady decrease in inter-sample correlation up to this distance.

It would be wrong to assume that the correlation structure seen in this direction, perpendicular to the plane of the orebody, is representative of the continuity in mineralization in other directions. Genetic control on mineralization determines the degree of variability in any particular direction, and lithologic and structural controls in the Williams Creek ore body will assist in defining any directions of preferential ore continuity.

However, the lack of any clear spatial structure within the plane of the ore body suggests that the sampling information is insufficient, and too widely spaced at this stage, to allow for any realistic grade estimation on a local basis. Further infill drilling is required, and based on the downhole variograms it is felt that a minimum hole spacing of 30 meters is required.

3.2.5 Recommendations

The sampling information available at this stage is insufficient for detailed ore body modelling or reserve estimation. Global estimates of in-situ geological reserves are possible, but no local definition would be realistic. No interpolation of grade values at this stage would be merited, and no detailed mine design should be attempted.

In order to define the ore body to an acceptable level of detail, further drilling on strike is required, and a minimum hole spacing of 30 meters is recommended at this stage. As further information becomes available, this can be re-evaluated.

It is suggested that the drilling be concentrated in the northern portion of the ore body, and that efforts be made to determine the boundary of oxidation.

3.3 REQUIREMENTS FOR DETAIL MINE DESIGN

To complete a final open pit design and associated production schedule a block model should be generated. The block model should incorporate known structural and lithological ore controls. It should be based upon a block size which can approximate the mineable selective unit. The block model should contain information regarding the following points:

- Copper grade - total and oxide
- Gold grade
- Lithologic units - rock type and alteration
- Rock quality determination - RQD, fracture frequency, etc.
- Specific gravity
- Pyrite - pyrhotite estimate for dump acid generation potential calculations

This information must come from the drill core and trench sample locations. The data should be available as computer files. The boreholes drilled for grade and geology information should be surveyed carefully and when appropriate logged for geotechnical information that may relate to the stability of the final wallslope.

Geostatistical techniques should be used to study the impact of loader selectivity at the digging face upon anticipated mine head grades.

3.4 PROPOSED MINING METHOD

The proposed mining is by conventional loader - truck methods. Alternate methods include combinations of hydraulic front shovels, in-pit crushers, conveyors and stackers. A contractor rate for mining and crushing was solicited.

CONTRACT MINING COST ESTIMATE

Mining	2,250,000 tonnes per year
Crushing	750,000 tonnes per year
Pit depth maximum	180 metres
Pit depth average	75 metres
Waste dump distance within	500 metres
Leach pad distance	500 metres
Contract mining	$1.75-$2.00/tonne
Contract crushing	$2.85/tonne

These preliminary estimates were considered high relative to the rates used by Silver Standard in their evaluation of the property. Wright have therefore produced a mining and crushing capital and operating cost estimate to evaluate the contractor rates.

The window of opportunity for mining has been defined as the period between May 15 - October 15. The production has been assumed at 750,000 tonnes at a 2:1 strip ratio. A total of 2,250,000 tonnes of material is mined annually. Waste stripping could in fact be carried on over an extended period but for this study it has been assumed that there are 130 days of operation available based upon 6 days per week operations utilizing two ten hour shifts. The employees will rotate out of the site as required on a monthly basis.

Production from the mine will be 17,300 tonnes per day or 8,650 tonnes per shift.

With the exception of the Caterpillar 992C, the equipment selected for mining at the Williams Creek property can all be moved by conventional tractor trailer or is self contained and transportable. Mobilization and set up of the entire plant can be accomplished in approximately three weeks. The 992C loaders must be erected on site.

3.4.1 Drilling and Blasting

The ore and waste will be drilled and blasted. For cost estimation purposes a bench height of 10 metres has been used. Further studies on dilution may dictate a lower bench height. For the production rates anticipated in the mine a single blasthole drill is required capable of single pass drilling 7-7/8" holes to 15 metres.

Drill holes will be sampled and assayed for ore grade control. Ore and waste contacts will be staked on the muck piles.

Pit conditions are anticipated to be relatively dry since it has been reported that diamond drill holes are losing water. The majority of the blasting operations will be carried out using AN/FO with liners used as required.

The oxidized ore zone is foliated and reported to be highly prefractured. The ore should be blasted to 100% passing 0.5 meters. All indications are that the ore will break well. A conservative powder factor has been included in the costs to ensure the plant feed is of a size suitable for size reduction in a moderate size jaw crusher.

Controlled blasting techniques will be required if maximum wall slope angles are to be attained. Presumably some form of trim blasting using the same drilling unit can be implemented. Geotechnical studies will be required to evaluate local and large scale stability factors and failure mechanisms.

A Bobcat skid steer loader has been included in the capital cost estimate to be used for blasthole stemming. This unit will also be useful around the crushing plant for cleanup around the conveyor transfer points and screening operations.

3.4.2 Loading

Loading will require two machines since ore and waste faces will be mined concurrently. Split faces will not always be available. The primary loading unit selected was a Caterpillar 992C. This machine has the capability to three pass load a 773B truck. A second loader, another Caterpillar 992C is required for loading ore. This second loader can also be used intermittently on the stockpile and around the plant area. Two large loaders represent a slight loader overcapacity but this ensures production will continue in the event of a major mechanical breakdown of the primary loading unit.

3.4.3 Hauling

Caterpillar 773B trucks were selected for mine haulage. These trucks have a high power to weight ratio, good manueverability and are transportable on provincial highways without complete dissassembly. With slight modifications to the hopper of the portable jaw crushing plant the 777B truck box size is suited to direct dumping.

With the limited information available with regards to locations of dumps and leach pads relative to the pit it is estimated that five trucks will be required initially to haul ore to the crusher, waste to the dump and product to the stockpile. The truck requirement will increase as the pit gets deeper unless the length of the operating season is extended.

3.4.4 General Support Equipment

Track mounted dozers will be required to maintain the leach pad as it is constructed in lifts and also in the mine pushing blast related backbreak, waste dump maintenance, and pit bench pioneering. Since these activities may be separated by a substantial distance a dozer has been proposed for the dump and another for the pit.

Road surfaces must be maintained properly to maximize truck speeds and to reduce operating costs. A used grader has been included in the capital cost estimate. This grader will also be used in the winter to maintain access to the plant site.

A single bay shop facility has been included in the capital cost. It is assumed that this would be a relatively simple structure on a concrete slab.

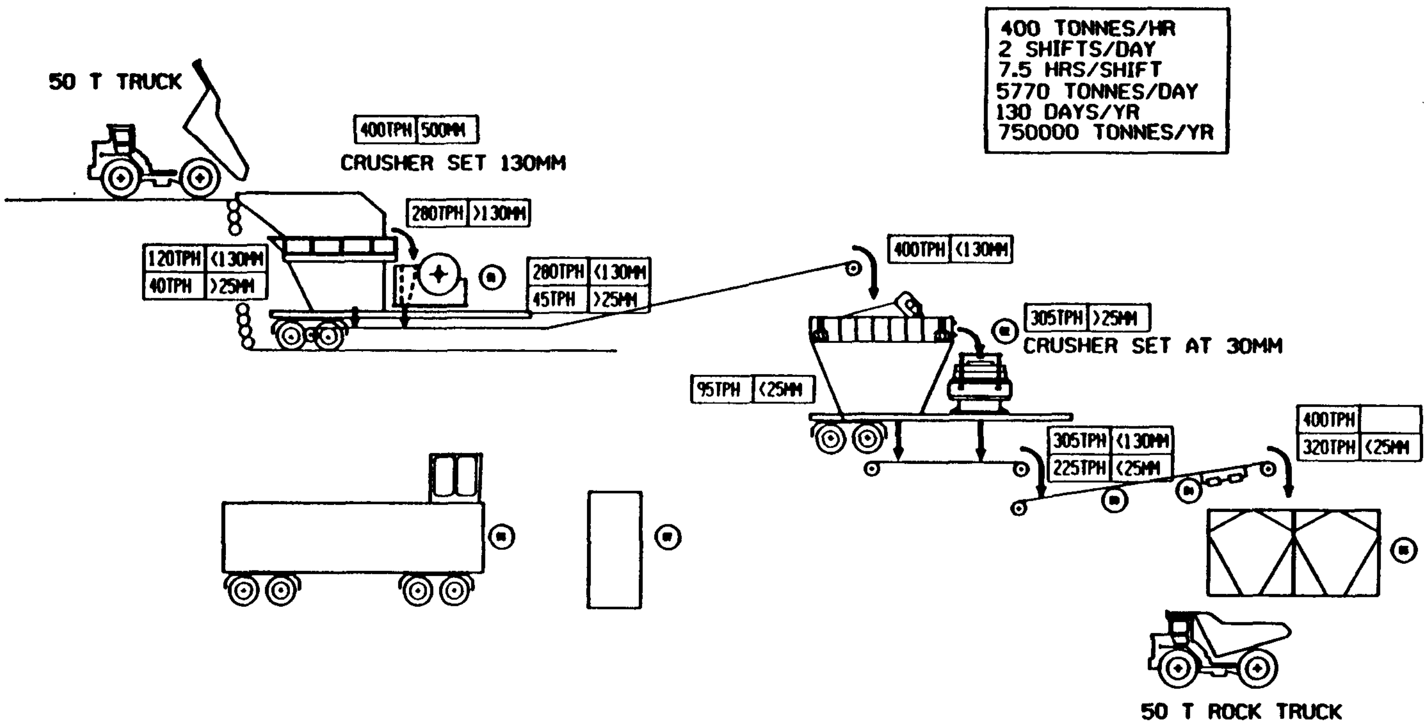
3.5 ORE CRUSHING

The metallurgical testing carried out to this point indicates copper recoveries in excess of 85% can be attained by crushing to 80% passing 25 mm.

A typical open circuit mobile crushing plant has been assembled for capital and operating cost estimation. A flowsheet of the process in shown on Figure 7. The first stage reduction is by jaw crusher. A LOKOMO C125 crusher has been selected. Maximum feed size is approximately 0.5 meters. A variable speed vibrating grizzly feeder with a spacing of 127 mm will be placed ahead of the jaw crusher. A retaining wall will be constructed at the primary crusher dump pocket. This wall will be a simple wing wall construction using vertical wide flange supports and locally available timber.

Product from the jaw crusher will pass over a 1.8 meter x 6.1 meter double deck screen. Material passing 25 mm will go to the product belt. Material retained on the screen will go directly to a 1560 OMNICONE crusher. This crusher will be equipped with coarse liners and set at approximately 29 mm. Product from the cone crusher will pass onto the final product belt.

A surge bin will be built to contain approximately 60 cubic metres of product. The bin will be elevated so as to allow a 773B truck to pass underneath. The trucks will be gravity



OPEN CIRCUIT 80% PASSING 25MM

01	1	01	JAW CRUSHER PLANT [illegible]	[illegible]
02	1	02	SCREEN & CONE CRUSHER PLANT [illegible]	300
03	1	03	36" FIELD CONVEYOR	[illegible]
04	1	04	BELT SCALE	—
05	1	05	[illegible]	—
06	1	06	GENERATOR/CONTROL [illegible]	[illegible]
07	1	07	[illegible] LITRE FUEL STORAGE	—

LEGEND

SILVER STANDARD RESOURCES INC.

WILLIAMS CREEK PROJECT

ORE CRUSHING CIRCUIT

D 1918 100

loaded by activating a hydraulically operated gate located in the bottom of the bin. An overflow spillway will be located on the side of the bin.

The crushing plant is supplied power from a 600 kW generator located in the control van. The entire plant is controlled by one operator located in the control booth of the generator - switch gear van. A second man will be required on the ground to clean up and to monitor areas out of sight of the operator. Trucks will control the loading operation. A belt weigh scale should be located on the product belt. A vertical 45,000 litre fuel storage tank will be located behind the van for the generator.

The generator set included with the crushing plant can be used for auxilliary backup power for the processing plant during the winter months.

SECTION 4

METALLURGY

SECTION 4

METALLURGY

The metallurgical test results from column and bottle roll leach tests conducted by Bacon, Donaldson and Associates Ltd. (BDA) and Coastech Research Inc. (Coastech) have been reviewed. Based on these preliminary tests the Williams Creek deposit appears to leach readily with a sulfuric acid lixiviant. If these tests are representative of the ore body in general, then a recoverable copper estimate of 85 percent (average) should be achievable in 30 days of leaching using ore crushed to 80% minus 25 mm.

4.1 TESTWORK

Two column leach tests conducted by BDA achieved 84.2 and 92.4 percent recovery in 75 and 26 days respectively. Both tests were continuing to recover copper at a rate of 0.24 percent per day when they were stopped. Minus 25 mm ore was leached in both columns with the only difference being leach solution application rate. A higher application rate significantly improves the initial leaching rate, leading to higher recoveries within a given period of time. This is very typical for oxidized copper deposits where a significant portion of the copper mineralization exists along fracture planes and faces in the host rock. Net acid consumption for the two tests averaged 51.8 and 46.9 kilograms per tonne of ore leached. A column test performed by Coastech using minus one inch ore recovered 73.7 percent of the copper contained in 22 days of leaching. Acid consumption for the test was 60 kilograms per tonne ore leached. The results of the Coastech tests indicates the potential variability in leaching performance.

Bottle roll testwork was also conducted by both BDA and Coastech. Various particle sizes were tested over 166 hours of leaching at BDA. Results are summarized in the following.

Facility	Size Fraction	% Cu Recovered	(Net) kg Acid/Tonne
Coastech	-200 mesh	92.4	77.6
BDA	1/4" -3/8"	80.3	23.8
BDA	1/4" -3/8"	86.6	26.5
BDA	3/4"	80.6	27.9
BDA	3/4"	82.3	26.4
BDA	1"-1.5"	74.7	29.3
BDA	1"-1.5"	66.3	28.0

In the BDA testwork, copper was still being leached at 166 hours. The Coastech recovery would probably be more indicative of ultimately leachable copper.

4.2 COPPER RECOVERY

The testwork conducted to date is very limited and utilized surface samples exclusively. What is not known is to what extent copper recoveries are affected with respect to location in the ore body. In general, recovery rates tend to decrease with depth in an ore body due to an increasing proportion of copper sulphide minerals. Overall recovery is very site specific and depends on many factors, such as:

- Host rock type(s)
- Mineralization (oxide, sulphide)
- Relative proportions of mineral types
- Geologic conditions
- Climatic conditions

Based on the physical characteristics of the drill work done to date, as described by R. Quartermain of Silver Standard, the ore body appears to be well oxidized with a very low occurence of sulphide minerals. All the material considered for leach at this time is contained in the oxidized zone with a sharply defined transitional zone between oxidized and undisturbed sulphide zones.

Allowing that some decrease in copper recovery will occur with depth and the probability of near ultimate copper recovery for surface material, an overall recovery of 85 percent should be achievable. This is based on at least 80 percent minus 25 mm ore using multiple lifts to an ultimate height of 32 meters in 4.5 meter increments. Recovery rates for sulphide copper minerals are dependent on the type and leaching conditions. A significant portion of the contained copper should be leached over time. A reasonable time frame for leaching to 85 percent should be 30 days over two cycles. An initial recovery of 80+ percent in the first leach cycle of 30 days should be achievable since the ore will be mined from the surface. Additional copper will be recovered as leaching solutions percolate through lifts of ore placed over previously leached material. Testwork on samples obtained at depth will be required to better define the potential recovery and rate of recovery for Williams Creek.

4.3 ACID CONSUMPTION

Acid consumption figures in this report are indicated as net. The net acid consumption is determined by subtracting the acid regenerated in the solvent extraction circuit from the total acid consumed as described by:

$$2\ RH + CuSO_4 \longrightarrow H_2SO_4 + R_2Cu$$

where R is the chelating organic extractant for copper. Acid required to reverse this reaction is generated in electrowinning as described by:

$$CuSO_4 + H_2O \xrightarrow{2.0\ V} Cu^{\circ} + H_2SO_4 + 1/2\ O_2$$

where Cu° is the copper metal plated.

Acid consumption described in this manner is representative of the acid that will be required to be purchased. It is therefore most relevant to examining costs.

Based on the testwork completed, acid consumptions range from 77.6 to 23.8 kilograms per tonne of ore leached. As with copper recoveries, acid consumption can also vary throughout the ore body and specifying an average consumption without benefit of testwork is highly speculative. For the purposes of this report an average consumption of 45 kilograms per tonne of ore was used. Dr. Morris Beattie of Bacon, Donaldson & Associates, who conducted the testwork, suggests that 45 kilograms per tonne of ore appears to be a reasonable figure.

The cost of acid is the single highest operating cost in producing copper at Williams Creek (nearly 25%). It is therefore imperative that additional testwork be conducted to better discern what acid consumption rates are likely. An acid consumption of 45 kilograms per tonne ore is about average for oxidized copper ore bodies. Sulphide minerals may also play a role in acid consumption. As the mineral is leached sulphur is made availabile for conversion to sulphuric acid, usually by bacterial action.

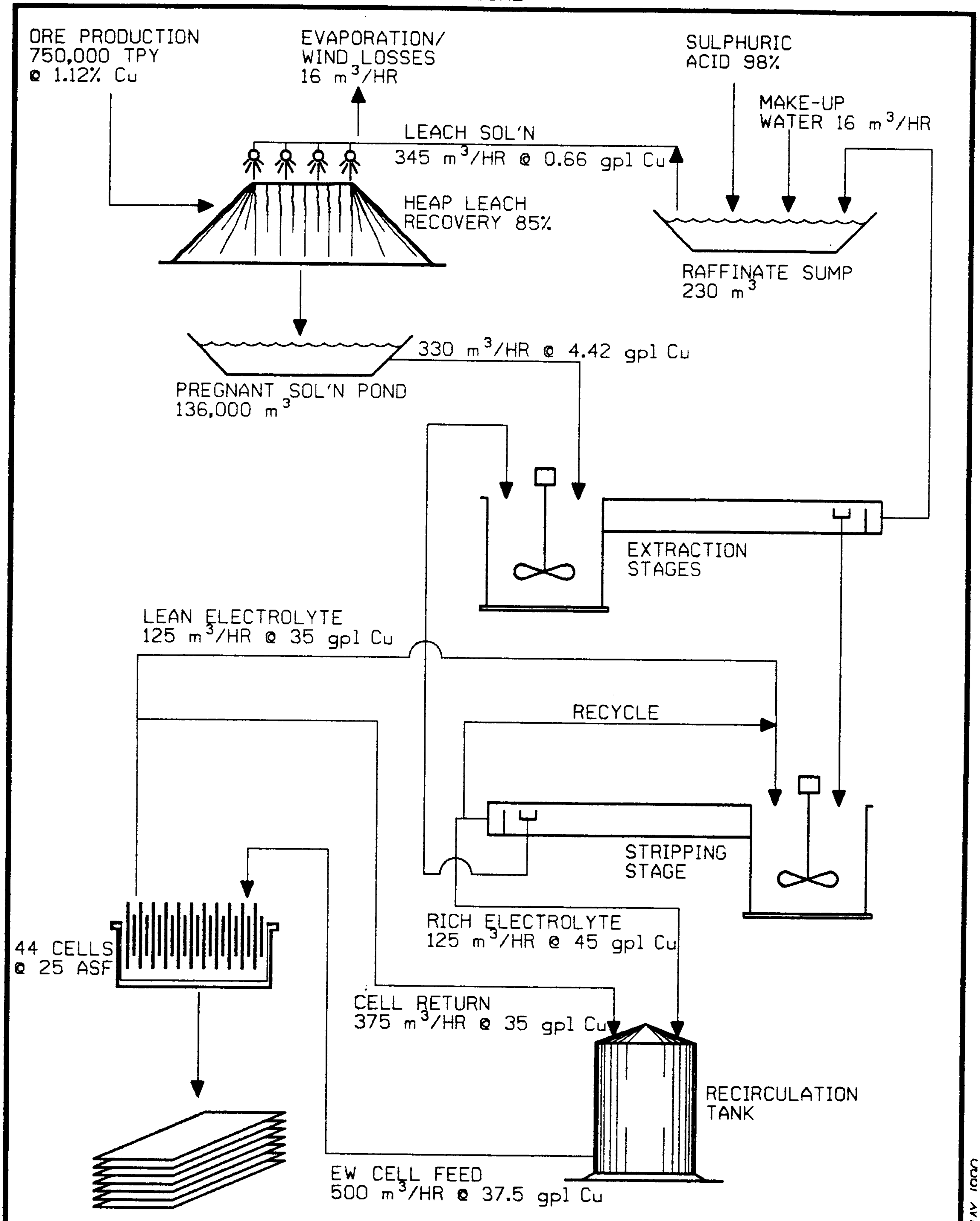
4.4 DESIGN CONCEPTS

The basic process flowsheet that is used for this report is shown in Figure 8. Only principal flow streams and basic process equipment are shown. The plant is designed to process 330 m^3/hr of pregnant leach solution with an average copper tenor of 4.42 grams per liter to produce approximately 30 tonnes of copper cathode per day. The plant is expected to operate 8 months per year on a seasonal basis. Total annual production is expected to be 7,140 tonnes of cathode copper.

4.4.1 Solvent Extraction

A block diagram for solvent extraction is shown in Figure 9. The solvent extraction plant is to process 330 m^3/hr of pregnant leach solution. A series-parallel flow (extraction) arrangement is proposed to reduce capital costs and physical size of the operation. Krebs type mixer/settler units are also proposed to further reduce capital costs and physical area required. An extraction efficiency of 85 percent is expected. The entire plant will be located indoors to minimize cold weather problems and facilitate seasonal operation.

FIGURE



SILVER STANDARD RESOURCES INC.
WILLIAMS CREEK PROJECT
PROCESS FLOWSHEET

PLOTTED 15 MAY 1990

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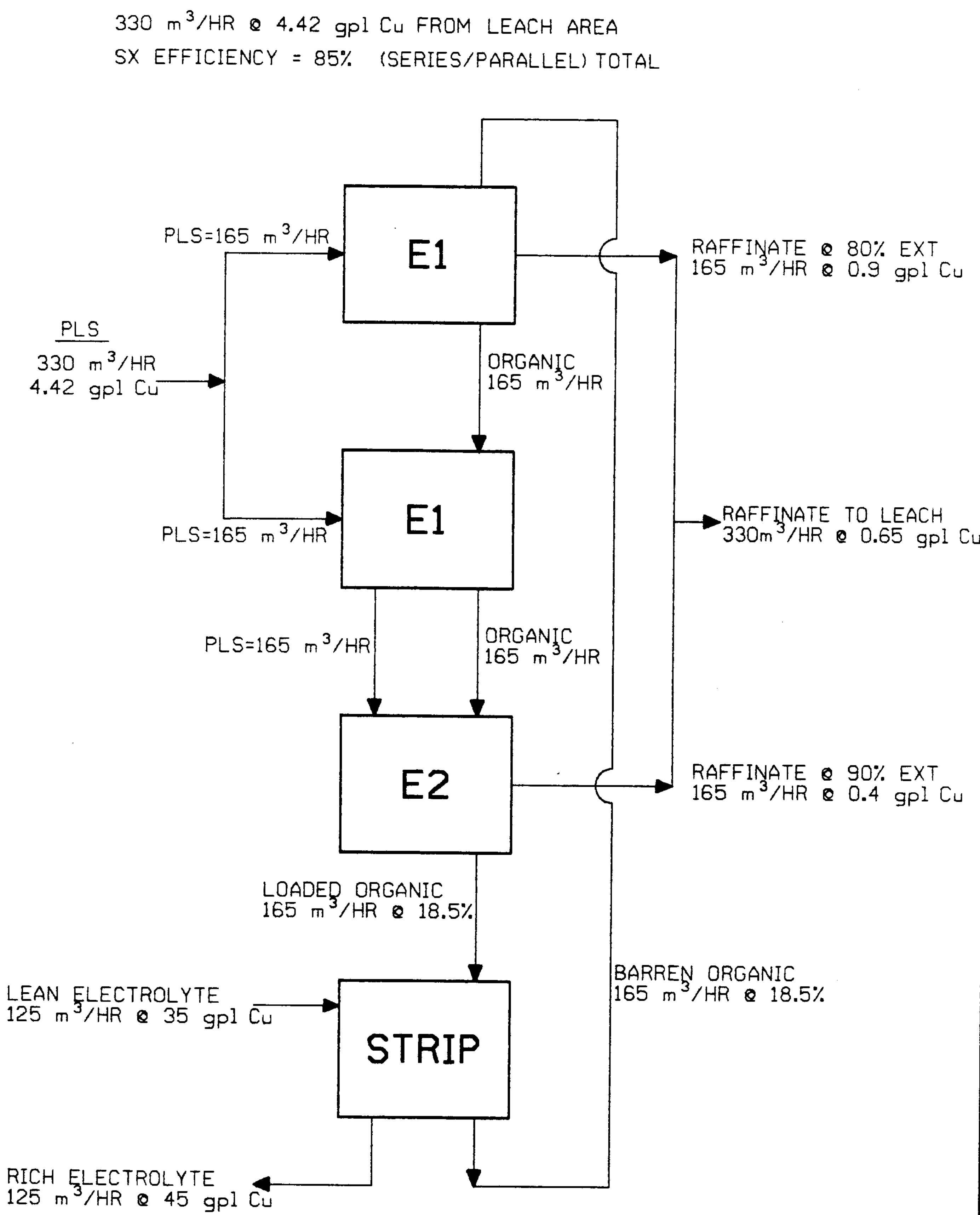
FIGURE 9



SILVER STANDARD RESOURCES INC.
WILLIAMS CREEK PROJECT
SOLVENT EXTRACTION

PLOTTED 15 MAY 1990

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4.4.2 Electrowinning

The electrowinning plant is to utilize the ISA process of permanent stainless steel cathode mother blanks. Cells are to be Cominco style polymer concrete construction to minimize maintenance, installation and support structure costs. A very simple version of the Wemnec stripping machine is proposed. A nominal current density of 270 amps per square meter of plating area is to be used with 44 cells containing 48 cathodes and 49 insoluble lead anodes. The annual production capacity based on 8 months operating is 7,140 tonnes per year with a maximum of 8,640 tonnes per year at 325 amps per square meter. The current efficiency is expected to be 92+ percent. The solvent extraction building is to be extended to house the electrowinning plant with a fire wall separating the two plants. All process tanks will also be located in the building at an elevation below ground level to make use of gravity flow where possible.

4.4.3 Cold Weather

Extreme cold weather can significantly hamper any mining operation. Heap leach/solvent extraction/electrowinning operations are no exception. The Williams Creek property is located in central Yukon in a semi-arid region. A seasonal operating scenario is currently proposed to avoid the severe cold period from December through March. The practicality of this type of operation will have to be carefully studied. Freezing of process solutions, especially electrolyte, will make re-starting the plant very difficult. Thawing of heap piles and solution ponds will also make start-up difficult. Since most of the process equipment is located indoors minimal damage would be expected.

The use of temporary heating of process solutions (i.e. portable boiler units, etc.) may facilitate start-up after winter. Since the Williams Creek property is small and process flowstreams are not excessively large it may be more practical to consider year round operations. The following are some key points that will have to be considered for year round operations or cold weather operations:

- Heap leaching
  - Minimize surface area of pond and insulate surface
  - Heat solutions as necessary to prevent freezing
  - Use buried drip emitters for leach solution application or similar equipment
  - Bury all pipelines and/or insulate

- Solvent extraction
  - Adequate settling time for increased phase disengagement times must be included in design
  - Heat solutions as necessary to prevent freezing
  - Heat building and insulate as required
  - Determination of affects on copper transfer kenetics/proper reagent selection
  - Mixing design modifications for kinetics if required
  - Ventilate building for dangerous fumes
  - Use of explosion proof electrical components

- Electrowinning
  - Heat process solution (electrolyte) to prevent poor cathode deposits and prevent sulfate crystallization.
  - Reduce acid concentrations to prevent crystallization. This will affect solvent extraction performance.
  - Minimize heat losses with mist suppression system enhancements.

- Miscellaneous
  - Provide for road maintenance or adequate storage for consumables during road closures.
  - Camp facilities for workforce and supplies.
  - Plant equipment and enclosures designed for cold temperatures.

Some or all of these measures may be required even if a seasonal operation is maintained. Local climatic conditions will dictate the required measures and subsequent costs. The occurence of sulphide minerals may also play a role in cold weather operations. The sulphuric acid reaction created by the oxidation of free sulphur is exothermic and can provide an additional heat source.

SECTION 5

CAPITAL COSTS

SECTION 5

CAPITAL COSTS

5.1 MINING

An exploration program designed to drill the pit area on 30 meter centres will result in approximately 65 holes - 10,670 meters of drilling. A core size of "NQ" is recommended. Drilling should be coordinated to provide maximum geotechnical information while drilling for grade and structural targets.

The capital cost estimate for mining equipment is based upon new prices for the majority of the equipment. As indicated previously an important criteria in the selection process has been the mobility of the various items. As a result the fleet will closely represent what will be required by a contract mining company. If Silver Standard chooses to mine the deposit themselves this type of equipment will be readily mobilized to another similar deposit or for resale in the construction or aggregate markets. Even after mining 8 million tonnes of ore and associated waste the equipment should have excellent salvage value.

Some items not critical to operations were included as used. Sensitivity studies should be carried out to include the effect of decreased availabilities and costs associated with used equipment.

Mobile mine equipment capital cost estimates are based upon dealer budget quotes FOB Vancouver. Mining capital costs are shown in Table 1. The estimated number of highway truck loads have been indicated. These are then used to provide an estimate of the mobilization cost for the mining equipment.

Mine and crusher set-up time will be minimal due to the portable nature of the equipment. Concrete supports to elevate the crushers and screens are recommended if only to facilitate clean-up operations around conveyor transfer points.

TABLE 1 – WILLIAMS CREEK MINING CAPITAL COST

TYPE	MAKE	MODEL	STATU	UNIT	LOADS	UNIT COST	TOTAL
LOADING							
LOADER	CAT	992C	NEW	2	6	$1,050,000	$2,100,000
HAULAGE & ROADS							
HAUL TRUCKS	CAT	773	NEW	5	1	$612,000	$3,060,000
TRACTORS	CAT	D8N	NEW	2	3	$427,000	$854,000
GRADER	CAT	14G	USED	1	1	$50,000	$50,000
WATER TRUCK			USED	1	1	$50,000	$50,000
DRILLING							
ROTARY DRILL	DRILLTECH	D60K	NEW	1	2	$550,000	$550,000
SKID STEER LOADER	BOBCAT	600	NEW	1		$40,000	$40,000
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							$7,131,000
CRUSHING							
JAW PLANT	NORDBERG	PC125	NEW	1	2	$325,000	$325,000
CONE/SCREEN PLANT	NORDBERG	1560	NEW	1	2	$620,000	$620,000
GENERATOR	NORDBERG	CONTROL	NEW	1	1	$300,000	$300,000
FIELD CONVEYOR	NORDBERG	36 "	NEW	1	1	$75,000	$75,000
WEIGH SCALE	RAMSEY REC		NEW	1		$10,000	$10,000
SURGE BIN	CUSTOM		NEW	1	1	$75,000	$75,000
FUEL TANKS	TIDY		NEW	1	0.5	$12,000	$12,000
TRAILER	FRUEHAUF		USED	2	2	$20,000	$40,000
RETAINING WALL	CUSTOM			1	1	$20,000	$20,000
CRUSHING TOTAL							$1,477,000
MOBILIZATION							
TRANSPORTATION				LOADS	28	$5,000	$140,000
CONCRETE & SERVICES							$30,000
SET UP COST							$50,000
MOBILIZATION TOTAL							$220,000
TOTAL MINING AND CRUSHING							$8,828,000

TABLE 1 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
ACCOMODATIONS					
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	$25,000	$75,000	
FORKLIFT	(units)	1	$50,000	$50,000	
MOBILE CRANE (15 tonne)	(units)	1	$250,000	$250,000	$375,000
PROCESS PLANT					
SOLVENT EXTRACTION				$1,000,000	
ELECTROWINNING				$7,600,000	
POWER LINE				$1,600,000	
PRIMARY POWER DISTRIBUTION				$400,000	
WATER HEATER				$150,000	
BUILDINGS	(feet^2)	23000	$150	$3,450,000	$14,200,000
WORKING CAPITAL/FIRST FILL				$2,225,000	$2,225,000
PROCESS PLANT SUBTOTAL					$23,570,000
MINE OPERATIONS					
MINING EQUIPMENT				$7,131,000	
CRUSHING EQUIPMENT				$1,477,000	
MOBILIZATION				$220,000	$8,828,000
MINE OPERATIONS SUBTOTAL					$8,828,000
INVENTORY					$250,000
ENVIRONMENTAL STUDIES					$150,000
GEOTECHNICAL ENGINEERING					$150,000
SUBTOTAL CAPITAL COSTS					$32,948,000
EPCM ON PROCESS PLANT @ 12%					$2,828,000
CONTINGENCY ON PROCESS PLANT @ 20%					$4,714,000
TOTAL PROJECT COST					$39,940,000

A single bay shop is included in the estimate as well as an allowance for shop tools to include items such as hydraulic jacks, compressor, torches and the fundamental hand tools not normally provided by skilled trandemen.

The mine is allocated three 4-wheel drive pick-up trucks, a 1 ton truck for the blasting crew and an ambulance.

The Yukon Territory has no equivalent of provincial taxes. Since delivery of equipment is to the Yukon, no taxes have been included. All costs are in second quarter, 1990 Canadian dollars.

5.2 METALLURGY

The estimate of capital costs determined by Wright has been based on the following:

- Consideration of increased construction costs applicable to the Yukon.
- Information provided by Silver Standard Mines Ltd.
- Utilization of capital cost factors obtained from a feasibility study currently in progress for a southwestern U.S. leach/SX/EW facility.
- Use of modular or premanufactured designs where possible.
- Vendor budgetary quotes as required.
- A U.S. to Canadian currency conversion of 1:1.2.
- Accepted and published industry cost factors.
- Incorporation of best available proven technology.

It is recognized that construction costs in the Yukon will be significantly higher due to increased labour, materials and transportation costs. Based on the 1990 Marshall and Swift Valuation Quarterly, a differential cost multiplier of 1.75 should be applied to the above stated components when comparing and applying cost factors determined for a southwestern U.S. facility. Equipment costs, delivered to the Williams Creek site, are also expected to be higher due to increased transportation costs. An estimate of 32 percent of the capital cost of the equipment has been used to determine the additional costs incurred over similar equipment delivered to a site in the southwest U.S. The reasons for selecting a southwest U.S. facility for comparison are:

- Most cost factors for SX/EW facilities are based on southwest U.S. plants.
- Wright Engineers most recent cost data are based on a proposed SX/EW plant in Arizona. Data was collected in December 1989.
- Vendor data are more accurate and duplication of cost contingencies are minimized.

Since a significant portion of the Williams Creek facility is comprised of equipment or modular pre-manufactured components, an overall cost factor of 1.60 has been used. This factor includes the U.S. to Canadian currency conversion.

The estimated capital costs for the Williams Creek property are summarized in Table 1. A brief explanation of each cost area and major component items are presented below. All figures are in second quarter, 1990 Canadian dollars.

<u>Exploration</u>

A budget of $150,000 has been allotted for metallurgical testwork. This amount is based on a reasonable number of column and bottle roll leach tests to determine the expected leachability of the Williams Creek deposit and identify the methods and conditions for the economical production of copper. Large scale test heaps and extensive column testing are not considered necessary at this time and are not included. The level of testwork required ultimately will be determined by Silver Standard Mines Ltd. with costs adjusted accordingly.

<u>Preproduction</u>

Silver Standard has indicated that the government will upgrade the road from Carmacks to the Williams Creek turn-off at no cost and provide 75% of the funding to construct the road to the property.

Heap Leach

Ore is to be stacked in 4.5 meter lifts to a height of 32 meters. A lined area of 460,000 square meters will be required to contain the expected 8,000,000 tonnes of ore to be mined. Due to the potential clearing and grubbing requirements, the installed cost of a single 60 mil high density polyethylene liner with a fabric under liner and sized rock and sand overliner is estimated to be $21.53 per square meter. A 114,000 cubic meter pregnant leach solution collection pond is required for containment of the heap leach pile drain down for seasonal operation. A cost of $26.91 per square meter has been used for a 6 meter deep pond constructed with a double 30 mil high density polyethylene liner. Geotechnical engineering will be required for the pad site.

Accommodation

Housing costs in Carmacks are relatively low. An average three bedroom home is currently $50,000 to $60,000. Six homes have been included for management personnel. A bunkhouse for 30 persons has been included. All of these facilities are to be located in or around Carmacks.

A camp for 11 people is also included and is to be constructed at the mine site. Silver Standard believes that the balance of the employees required will be obtained from communities in the surrounding area.

Vehicles

Three 4 x 4 pick-up trucks have been included in the process plant capital costs for management and operations use. A 3.5 ton forklift for material handling including cathode production is provided. A 15 ton mobile crane is included for maintenance needs for the entire mine site.

Process Plant

Solvent extraction costs are based on Krebs mixer/settler technology using a series parallel extraction scenario. Krebs units are to be of fiberglass modular construction. A preliminary budget price from Krebs for the proposed plant excluding piping, other than interstage requirements, process tanks for solution circulation and instrumentation was $750,000 fully installed.

Electrowinning costs are based on ISA process technology with polymer concrete cells. Costs factored from a recent study are approximately $725 per annual tonne. The total allowance for the Williams Creek property for SX/EW construction is approximately $827 per annual tonne capacity. The electrowinning capacity is designed at 30 tonnes per day or an annual capacity of 10,400 tonnes.

The powerline extension to the property is approximately 40 kilometers. The expected rating is approximately 25-30 kV. A cost of $40,000 per kilometer has been used for an unsubsidized installation. Silver Standard believes that a subsidized cost could be $25,000 per kilometer.

A boiler rated at 10 MM BTU/hr has been selected to provide process solution heating requirements.

A 2,135 square meter building is required to house the process plant, control room, offices and laboratory. The estimated cost is $1,600 per square meter. Lab equipment and office furnishings are included at $125,000.

First fill and a 60 day working capital provision has also been included.

Excluded from this cost estimate is the water supply system for the process requirements. Approximately 200 gpm will be required. Water availability and quality data does not currently exist. A potential source might be Williams Creek itself. The Yukon River is the most likely source requiring a pumping/piping distribution system of 4 to 8 kilometers.

SECTION 6

OPERATING COSTS

SECTION 6

OPERATING COSTS

6.1 MINING

The mining costs have been estimated for the Williams Creek property by Tercon Contractors. Their estimate was kindly provided based upon a very limited amount of information which can be summarized as follows:

	Site preparation:	
-	Road upgrading	\$25,000/km
-	Clear and grub	\$1,000/acre
		\$0.25/square meter
	Mining \$1.75-\$2.00/tonne:	
-	Annual quantity mined	2,250,000 tonnes
-	Maximum haul at pit crest	0.5 km
-	Maximum pit depth	200 meters
-	Pit depth - centre of gravity	80 meters
-	Mining during a 4-5 month period	
	Crushing and stockpiling \$2.75/tonne:	
-	Annual quantity crushed	750,000 tonnes
-	Crushing jaw-cone-screen	80% <25 mm
-	Loading haulage and stockpiling	

As more information and detail is developed regarding the nature of the mining operation a more accurate contract mining rate can certainly be developed. Clearly a long term contract will influence the unit cost. Mobilization expenses may not necessarily be incurred on an annual basis and the risk associated with equipment purchases will be spread over a greater volume of rock.

Wrights have prepared an estimate of mine and crusher operating costs which are summarized in Table 2.

TABLE 2

SUMMARY OF MINING AND SX-EW COSTS

Mining	$/Year	$/t Ore	$/lb Cu
Mining			
Equipment operation	2,089,000	2.79	0.13
Wages	1,101,000	1.47	0.07
Crushing			
Equipment operation	638,000	0.85	0.04
Wages	448,000	0.60	0.03
Total			
Equipment operation	2,727,000	3.64	0.17
Wages	1,549,000	2.07	0.10
Total Mining	4,276,000	5.70	0.27
SX-EW Leaching			
Labour			
Administration	556,250	0.74	0.04
Operations - Hourly	917,235	1.22	0.06
Maintenance - Hourly	251,790	0.34	0.02
Total	1,725,275	2.30	0.11
Power	848,232	1.13	0.05
Acid	3,769,920	5.03	0.24
Organics	659,736	0.88	0.04
Maintenance	549,780	0.73	0.03
Other costs	408,408	0.54	0.03
Shipping	801,108	1.07	0.05
Total	7,037,184	9.38	0.45
Total SX-EW	8,762,459	11.68	0.56
Housing and Camp	366,000	0.49	0.02
TOTAL SX-EW AND MINING	13,038,459	17.38	0.85

The annual wages have been calculated using a 25% loading factor. Mechanics have been scheduled for eight months. Operators were scheduled five months. Manpower requirements are detailed in Table 3.

Blasting consumable costs are calculated using budget quotes from Explosives Limited. Drill productivities do not include wall control measures. No allowance is made for serious water inflows to blastholes.

Equipment operating costs are shown in Table 5. Detail design of the mine is required. These operating costs provide a reasonable estimate assuming similar conditions to those described for contract mining. Equipment hours include allowance for mining crushing and stockpile construction.

6.2 PROCESS PLANT

The operating costs for the process plant have been estimated using expected loss or consumption rates typical for SX/EW facilities. The basis for copper production is 750,000 tonnes mined per year at 1.12 percent copper at 85 percent recovery. The average annual production will be 7,140 tonnes at 29.75 tonnes per day for 240 days of operation. A summary of these costs are shown in Table 2.

6.2.1 Labour

Process plant labour and administration costs are based on the manpower and wage rates shown in Table 3. These rates are typical of mining rates in Western Canada. More detailed evaluation should utilize definitive local rates.

6.2.2 Power

Yukon Power costs are approximately $0.06/kWh for commercial operations. Yukon Power has indicated to Silver Standard that they would have to start out at approximately

$0.09/kWh reducing to about $0.06/kWh with time. Yukon Power is currently charging $0.042/kWh to a zinc mining operation at Faro. For a first order estimate, $0.06/kWh is reasonable after negotiations, with an equal chance of being higher or lower at this time. Power consumption is based on 1900 kWh/tonne copper produced plus pumping and mixing costs for the leach/SX/EW plant and miscellaneous uses.

6.2.3 Acid

Acid is the single highest cost component to the Williams Creek operating costs. Acid consumption is estimated to be 45 kilograms per tonne of ore leached including an acid credit of 1.54 tonnes per tonne of cathode copper produced in electrowinning. The consumption rate has been estimated from very limited information and may or may not be representative of the ore body in general. Only additional testwork, as described in the recommendations in Section 7.2, will provide the necessary information to more accurately determine potential acid consumption rates.

The unit cost for sulphuric acid is $165 per tonne from either Fort St. John or Seattle, Washington. In the future it may be worth investigating the possibility of using acid plant waste or below specification acid. The additional trucking costs would have to be weighed against possible price concessions but may provide a means by which to reduce the overall cost of acid. Supply would also need to be investigated. For the purposes of this report $165/tonne has been used.

It is Wright's opinion that the incorporation of an acid production facility may provide a means to significantly improve the economics of the Williams Creek property. A sulphur burning acid plant capable of producing 165 tonnes per day of sulphuric acid will cost approximately $10 M - $12 M fully erected. The current price of liquid sulphur delivered to site is $138 per tonne. Approximately 3 tonnes of acid can be produced from 1 tonne of sulphur. The unit cost for acid would be reduced to approximately $46 per tonne which would reduce the unit cost to produce copper by 17¢ per pound. The net cost savings would be approximately $2.4 M per year at nominal production rates which would indicate a 4-5 year payback period. An

TABLE 3 – WILLIAMS CREEK SCHEDULE OF WAGE RATES – MINING & SX-EW

POSITION	HOURLY RATE ($/hr)	ANNUAL Salary	OWNERS COST ($/yr)/Man	MEN	TOTAL COST ($/YR)	$/T ORE	$/T MINED
ADMINISTRATION OF MINE OPERATIONS							
Mine Superintendent •		$75,000	$93,750	1	$93,750	$0.125	$0.042
Mine Foreman •		$55,000	$68,750	1	$68,750	$0.092	$0.031
Clerk		$35,000	$43,750	1	$43,750	$0.058	$0.019
Mine Geologist/Engineer •		$45,000	$56,250	1	$56,250	$0.075	$0.025
Surveyor - 6 Months		$20,000	$25,000	1	$25,000	$0.033	$0.011
Helper - 6 Months		$15,000	$18,750	1	$18,750	$0.025	$0.008
TOTAL MINING ADMINISTRATION				6	$306,250	$0.408	$0.136
MINE OPERATIONS - HOURLY							
Drillers	$18.00	$25,200	$31,500	2	$63,000	$0.084	$0.028
Truck Drivers	$17.00	$23,800	$29,750	8	$238,000	$0.317	$0.106
Wheeled Loaders	$18.00	$25,200	$31,500	4	$126,000	$0.168	$0.056
Tracked Dozers	$18.00	$25,200	$31,500	3	$94,500	$0.126	$0.042
Road Grader	$18.00	$25,200	$31,500	1	$31,500	$0.042	$0.014
Blasters	$18.00	$25,200	$31,500	1	$31,500	$0.042	$0.014
				19	$584,500	$0.779	$0.260
MINE MOBILE MAINTENANCE - HOURLY							
Shop Mechanics	$21.00	$29,400	$36,750	4	$147,000	$0.196	$0.065
Welder	$21.00	$29,400	$36,750	1	$36,750	$0.049	$0.016
Labourers	$15.00	$21,000	$26,250	1	$26,250	$0.035	$0.012
				6	$210,000	$0.280	$0.093
TOTAL MINING HOURLY				25	$794,500	$1.059	$0.353
CRUSHER OPERATIONS - HOURLY							
Crusherman I	$18.00	$25,200	$31,500	3	$94,500	$0.126	
Crusherman II	$16.00	$22,400	$28,000	1	$28,000	$0.037	
Labourers	$15.00	$21,000	$26,250	2	$52,500	$0.070	
				6	$175,000	$0.233	
CRUSHER MAINTENANCE - HOURLY							
Shop Mechanics	$21.00	$29,400	$36,750	4	$147,000	$0.196	
Welder	$21.00	$29,400	$36,750	2	$73,500	$0.098	
Labourers	$15.00	$21,000	$26,250	2	$52,500	$0.070	
				8	$273,000	$0.364	
TOTAL CRUSHING				14	$448,000	$0.597	
TOTAL MINING & CRUSHING HOURLY				39	$1,242,500	$1.657	
TOTAL MINING & CRUSHING				45	$1,548,750	$2.065	
UNIT COST $/T ORE 750000 TONNES					$2.065		
UNIT COST $/T MINED 2250000 TONNES					$0.688		
ADMINISTRATION OF SX-EW OPERATIONS							
Plant Manager •		$90,000	$112,500	1	$112,500		
Superintendent •		$70,000	$87,500	1	$87,500		
Shift Supervisors		$40,000	$50,000	4	$200,000		
Secretary/Clerk		$35,000	$43,750	1	$43,750		
Engineer •		$50,000	$62,500	1	$62,500		
Chemist		$40,000	$50,000	1	$50,000		
TOTAL WAGES ADMINISTRATION				9	$556,250	$0.742	
SX-EW OPERATIONS - HOURLY							
SX/EW	$18.00	$43,164	$53,955	8	$431,640		
Leaching	$18.00	$43,164	$53,955	7	$377,685		
Laboratory	$18.00	$43,164	$53,955	2	$107,910		
TOTAL WAGES OPERATIONS				17	$917,235	$1.223	
SX-EW MAINTENANCE - HOURLY							
Electricians/Instruments	$21.00	$50,358	$62,948	2	$125,895		
Mechanics	$21.00	$50,358	$62,948	2	$125,895		
TOTAL WAGES MAINTENANCE				4	$251,790	$0.336	
TOTAL WAGES HOURLY				21	$1,169,025	$1.559	
TOTAL WAGES HOURLY & SALARIED				30	$1,725,275	$2.300	
TOTAL MANPOWER COST MINING & SW-EW					$3,274,025	$4.365	
HOUSING COST					$60,000	$0.080	
CAMP COST					$368,800	$0.492	
TOTAL LABOUR COST					$3,702,825	$4.937	

TABLE 4 – WILLIAMS CREEK PRODUCTION BLASTHOLE DRILLING

ESTIMATE USING 7-7/8" PRODUCTION HOLES
EXPLOSIVES - AN/FO

		PRODUCTION HOLES 200 mm	COST NOT INCLUDING WAGES
DRILLING PARAMETERS			
COLLAR HOLE	minutes	2.0	
ADD STEEL	minutes	0.0	
REMOVE STEEL	minutes	0.5	
MOVE	minutes	2.0	
PENETRATION	metres/hr	22.0	
DRILLING	min./hole	37.2	
PRODUCTION	m/hr	19.3	
HOLE DEPTH	metres	12.0	
BURDEN	metres	5.2	
SPACING	metres	5.2	
BENCH HEIGHT	metres	10.0	
VOLUME	m^3	270.4	
S.G.	t/m^3	2.75	
TONNES/HOLE	tonnes	743.6	
EXPLOSIVES PARAMETERS			
CHARGE DIAMETER	mm	200	
COLLAR	metres	3.2	
LOADED LENGTH	metres	8.8	
S.G. POWDER	t/m^3	0.85	
VOLUME	m^3	0.28	
WEIGHT	kg	235.0	
POWDER FACTOR	kg/t	0.32	
POWDER FACTOR	kg/m^3	0.87	
POWDER AN/FO $100.00	$/100 kg	$234.99	
ACCESSORIES	$/hole	$20.00	
WAGES BLASTING	$/hr	$24.65	
TIME ALLOCATION	hr/hole	0.40	
WAGES BLASTING	$/hole	$9.86	
TOTAL BLASTING	$/hole	$264.85	$240.20
TOTAL BLAST COST	$/tonne	$0.356	$0.323
DRILLING COST CALCULATION			
DRILL BITS	each	$3,000	
EXPECTED LIFE	metres	1500	
COST	$/metre	$2.00	
FUEL COST	$/l	$0.280	
FUEL CONSUMPTION	l/hr	90	
FUEL COST	$/hr	$25.20	
R&M COSTS	$/hr	$40.00	
R&M/OPERATE RATIO		0.6	
WAGES R&M	$/hr	$28.00	
EFFECTIVE R&M LABOUR	$/hr	$16.80	
WAGES OPERATING	$/hr	$24.15	
TOTAL DRILL COST	$/hr	$144.83	$103.88
TOTAL DRILL COST	$/m	$7.49	
TOTAL DRILL COST	$/hole	$89.86	$64.45
TOTAL DRILL COST	$/tonne	$0.121	$0.087
COMBINED DRILLING AND BLASTING			
TOTAL DRILL & BLAST	$/tonne	$0.477	
DRILL FACTOR ESTIMATION			
DRILL FACTOR hrs/1000t		0.83439	

NOTE: Not enough is known about perimeter wall control blasting requirements to attach a cost to this aspect of the blasting operation.

TABLE 5 – MINE OPERATING COST ESTIMATE

EQUIPMENT	MAKE	TYPE	HRS/1000T	HOURS	$/HOUR INC WAGES	ANNUAL COST	COST/TONNE MINED	COST/TONNE ORE
LOADING								
LOADER	CAT	992C	1.75	2625	$82.00	$215,250	0.10	0.29
HAULAGE & ROADS								
HAUL TRUCKS	CAT	773	2.50	7500	$60.00	$450,000	0.20	0.60
TRACTORS	CAT	D8N	2.00	4500	$41.00	$184,500	0.08	0.25
GRADER	CAT	14G	0.50	1125	$21.00	$23,625	0.01	0.03
DRILLING								
ROTARY DRILL	DRILLTECH	D60K	0.83	1877	$104.00	$195,156	0.09	0.26
SKID STEER LOADER	BOBCAT	600		400	$20.00	$8,000	0.00	0.01
BLASTING AGENTS						$787,500	0.35	1.05
CAP MAGAZINE								
POWDER MAGAZINE								
POWDER/PUMP TRUCK						$5,000	0.00	0.01
MAINTENANCE								
SERVICE TRUCK	GMC					$5,000	0.00	0.01
WELDER	MILLER					$2,000	0.00	0.00
MISCELLANEOUS								
SINGLE BAY SHOP						$50,000	0.02	0.07
TOOLS						$2,000	0.00	0.00
FUEL TANKS	TIDY							
PICK UP TRUCKS	GMC					$35,000	0.02	0.05
AMBULANCE						$1,000	0.00	0.00
ENGINEERING								
COMMUNICATIONS						$20,000	0.01	0.03
SUPPLIES						$30,000	0.01	0.04
CONSULTANTS						$25,000	0.01	0.03
MISCELLANEOUS						$50,000	0.02	0.07
TOTAL MINING						$2,089,031	0.93	2.79
CRUSHING								
JAW PLANT			2.50	1875	$100.00	$187,500		0.25
CONE/SCREEN PLANT			2.50	1875	$100.00	$187,500		0.25
GENERATOR			2.50	1875	$120.00	$225,000		0.30
FIELD CONVEYOR			2.50	1875	$5.00	$9,375		0.01
WEIGH SCALE			2.50	1875	$2.00	$3,750		0.01
SURGE BIN			2.50	1875	$10.00	$18,750		0.03
						$1,000		0.00
						$5,000		0.01
TOTAL CRUSHING						$637,875		0.85

additional benefit could be realized by effective use of the nearly 20 M BTU per hour waste heat produced. The impact on potential capital and/or operating costs would require a more in-depth study.

6.2.4 Organics

Organic costs are based on a loss rate of 60 ppm in both raffinate and electrolyte flowstreams. Budget prices of $0.61 per liter for Shelsol 160 and $15.5 per kilogram for Acorga M5640 delivered to site were used.

6.2.5 Maintenance

The modular plant equipment in new condition should not require excessive maintenance. Normal maintenance costs are 2 cents per pound of copper for southwest U.S. plants especially with an ISA tankhouse. A conversion factor of 1.75 for the Yukon was used for increased cost and currency exchange.

6.2.6 Shipping

Shipping costs were estimated at 17.5¢ per ton mile from the plant to Skagway. A cost of U.S. $44 per tonne for cathode shipment to Japanese ports in 4,550 to 6,360 tonne lots was used. Prices were obtained from Gearbulk.

6.2.7 Other

Remaining costs were estimated at 20 percent of the maintenance and operating supply cost (excluding acid). Due to the limited operating time only minimal electrode replacement will be required.

Owner's costs and applicable taxes have not been included in this estimate.

SECTION 7

RECOMMENDATIONS

SECTION 7

RECOMMENDATIONS

7.1 MINING

7.1.1 Drilling and Block Model Development

Diamond drilling directed towards ore reserve estimation at Williams Creek should be carried out on 30 meter centers and used to provide the maximum possible information for block model development. A block model is recommended since the results will then be available for pit limit economic analysis. The core size recommended is NQ wireline. The data logs should include information about the following:

- Total copper grade
- Oxide copper grade
- Gold grade
- Carbonate data
- Lithology
- Density
- Specific gravity
- Sulphide content
- Jointing data - fracture frequency - rock quality determination
- Major structural features

A three dimensional block model should be generated using a computer. This model should contain all the information that can be interpolated from the diamond drill hole data and surface trenches. Structural domains should be identified and the hanging wall and footwall boundaries used to define blocks within the model that should be modelled using common data. Gold and copper should be modelled separately.

An effort should be made to verify that the leaching characteristics of the material tested from surface samples can be projected to depth in the mineralized zone.

7.1.2 Mine Design

The open pit mine should be designed using a floating cone or Lerchs Grossman optimizer. A starter pit should be located to provide an initial pit which maximizes cash flow during the early years of the mine life. This is generally accomplished by locating an area with a lower than average strip ratio or higher than average grade or perhaps a combination of the two parameters. Practical push backs on the starter pit should be staged to increase the mine size to the final economic limits.

Areas of high gold values should be identified and mining should incorporate a plan to stockpile this material in a segregated area.

7.1.3 Mining Method

Substantial savings in mine operating costs can be made by eliminating the crushing plant requirement. Metallurgical testing directed towards defining the economics of recovery versus particle size will have an impact on this cost. Run of mine leaching should be explored.

Contractor estimates will be better defined as the orebody model and mining plan is completed. Long term contracts with contract mining companies will probably result in a lower unit cost. This option should be explored.

Leasing of mining equipment should be investigated as an alternative to purchasing.

The used equipment market should be investigated as a way to reduce mine capital cost.

Conveyors, stackers and self propelled crushers are alternate materials handling methods worth considering to reduce unit operating costs. The current plan incorporates loaders and trucks to provide flexibility in the mine operating environment. Conveyors if applicable will be cheaper to operate.

7.2 METALLURGY

The following recommendations apply to the metallurgical testwork program. It is necessary that these items be completed prior to further feasibility work to adequately describe process design criteria.

- It is essential that information be obtained from material at depth. At least 4-6 "H" drill core holes must be drilled to provide material for leaching testwork. Material obtained should be as representative as possible of the ore body in general.

- Composite samples at regular intervals (3 meters) should be taken for bottle roll tests as conducted by Bacon, Donaldson & Associates. This should be conducted on all core samples.

- Material from at or near the expected "bottom" of the proposed pit should be leached under standard conditions determined for column test baseline.

- Analysis of head and tail samples for bottle roll tests should include determination of the mineralogy of the copper species.

- Various particle sizes should be tested in columns to determine the economic return of each crushing step. A good range would be 80 percent minus 100 mm material, 80 percent minus 50 mm material and 80 percent minus 25 mm material. Allow sufficient time to see if ultimate recoveries are impacted.

- Incorporation of an acid cure is typical for oxide copper leaching. This should be tested and optimized. An initial cure of 20 kg per tonne would be a starting point. The pH of the pregnant solution will determine limits.

- Optimization of flowrate per unit area. Typical ranges are 0.12 to 0.33 liters gpm per minute per square meter of area.

- Collection of pertinent climatic data for design considerations.

General recommendations for future design consideration are:

- Study economic benefits of incorporation of an acid plant complex. Sulphur supplies must be investigated, including below specification grade sources.

- Year round operations for SX/EW/leach facility. Acid plant waste heat recovery.

- Mining of sulphide material for acid and heat values. Additional copper recovery would also be realized.

- High grade mining in the first years would accelerate debt payback and reduce interest costs.

Gold recovery

- Gold mineralization appears to be of sufficient grade in a localized area to merit some testwork using neutralization/cyanide leach. This could be accomplished in practice by incorporating a segregated leach area for high gold content ore.

- Gold dissolution in acidic sulphate solutions with chloride may be possible. Gold recovery from copper leach solutions using an ion exchange resin is currently being piloted at an Arizona oxide copper leach facility.

- Ammonia leaching may still be worth investigating in a test column to determine if copper recovery can be achieved with additional time. The potential for gold recovery may merit consideration of finer crushing sizes and possibly vat leaching for copper and gold. Ammonia has also been used for leaching chalcopyrite and may allow economic recovery of sulphides.

- Thiourea leaching in an acidic environment may also be worth investigating.

It is generally expected that gold recovery is probably not economical with current technology. These recommendations are made with the idea that a limited high grade zone of 0.06 oz per ton may be treated separately.

SECTION 8

ADDENDUM

SECTION 8

ADDENDUM

At the request of Mr. Bob Quartermain of Silver Standard Resources Ltd. Wrights estimators reviewed the multipliers used to scale the capital costs for construction of an SX-EW plant in the Yukon Territory. They found no basis for changing these multipliers without more specific site related information.

Wrights have prepared an additional estimate of capital and operating costs based upon the following scenario:

1. Further testing may indicate that run of mine ore can be leached effectively. Copper recoveries are expected to be 75% as opposed to 85% used in the original study. Fragmentation of the ore will be adequate using the currently specified powder factors. The crushing plant capital cost is eliminated.

2. The electrowinning plant capital cost is reduced from $7.6 million to $6.7 million. The building size is reduced by 30 square metres.

3. The plant is operated 8 months of the year and placed in care and maintenance for 4 months. The workforce is reduced during winter months.

4. The mining equipment capital is reduced by purchasing more used mining equipment including rebuilt D8K tractors, used 773 trucks, used mobile crane, and one used 992 loader. Used mining equipment can be obtained at widely varying costs. The risk is that operating costs and availabilities become more difficult to predict and impact on the production capability of the fleet.

5. The power line costs were reduced to reflect verbal estimates from Yukon Power provided by Mr. Quartermain.

6. A 100 tonne capacity acid plant was included at a capital cost of $6 million with a resultant decrease in acid cost to $0.08/lb of copper produced.

The resultant capital cost summary is shown in Table 6. The associated operating cost summary is shown in Table 7. The wage schedule is shown in Table 8 with adjustments made to the operating periods.

In summary the operating costs are reduced to $0.63/lb of copper produced from $0.85/lb.

The total capital costs are increased to $41.7 million by the addition of an acid plant.

This exercise goes beyond the limits of the data provided upon which to base the property review but gives an indication of the impact various cost reducing methods can have upon the capital and operating costs. The basic exploration and metallurgical test work must be carried out to provide a foundation for this type of "what if?" analysis. Clearly management philosophies can have an impact on equipment, maintenance , and manpower costs. These factors are difficult to incorporate in a study of such a preliminary nature.

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	DRILLTECH	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
ACCOMODATIONS					
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

TABLE 7 – SUMMARY OF MINING & SX-EW COSTS RUN OF MINE

RECOVERY AT 75%

	$/YR	$/T ORE	$/lb Cu
MINING			
EQUIPMENT OPERATION	$2,089,000	2.79	0.15
WAGES	$1,101,000	1.47	0.08
TOTAL MINING	$3,190,000	4.25	0.23
SX-EW LEACHING			
LABOUR			
ADMINISTRATION	$541,700	0.72	0.04
OPERATIONS - HOURLY	$611,500	0.82	0.04
MAINTENANCE - HOURLY	$167,900	0.22	0.01
TOTAL	$1,321,100	1.76	0.10
POWER	$694,260	0.93	0.05
ACID	$1,110,816	1.48	0.08
ORGANICS	$555,408	0.74	0.04
MAINTENANCE	$416,556	0.56	0.03
OTHER COSTS	$408,408	0.54	0.03
SHIPPING	$694,260	0.93	0.05
TOTAL	$3,879,708	5.17	0.28
TOTAL SX-EW	$5,200,808	6.93	0.37
CAMP & HOUSING	$322,000	0.43	0.02
TOTAL SX-EW & MINING	$8,390,808	11.19	0.63

TABLE 8 – WILLIAMS CREEK SCHEDULE OF WAGE RATES – MINING & SX-EW

POSITION	HOURLY RATE ($/hr)	ANNUAL Salary	OWNERS COST ($/yr)/Man	MEN	TOTAL COST ($/YR)	$/T ORE	$/T MINED
ADMINISTRATION OF MINE OPERATIONS							
Mine Superintendent *		$75,000	$93,750	1	$93,750	$0.125	$0.042
Mine Foreman *		$55,000	$68,750	1	$68,750	$0.092	$0.031
Clerk		$35,000	$43,750	1	$43,750	$0.058	$0.019
Mine Geologist/Engineer *		$45,000	$56,250	1	$56,250	$0.075	$0.025
Surveyor - 6 Months		$20,000	$25,000	1	$25,000	$0.033	$0.011
Helper - 6 Months		$15,000	$18,750	1	$18,750	$0.025	$0.008
TOTAL MINING ADMINISTRATION				6	$306,250	$0.408	$0.136
MINE OPERATIONS - HOURLY							
Drillers	$18.00	$25,200	$31,500	2	$63,000	$0.084	$0.028
Truck Drivers	$17.00	$23,800	$29,750	8	$238,000	$0.317	$0.106
Wheeled Loaders	$18.00	$25,200	$31,500	4	$126,000	$0.168	$0.056
Tracked Dozers	$18.00	$25,200	$31,500	3	$94,500	$0.126	$0.042
Road Grader	$18.00	$25,200	$31,500	1	$31,500	$0.042	$0.014
Blasters	$18.00	$25,200	$31,500	1	$31,500	$0.042	$0.014
				19	$584,500	$0.779	$0.260
MINE MOBILE MAINTENANCE - HOURLY							
Shop Mechanics	$21.00	$29,400	$36,750	4	$147,000	$0.196	$0.065
Welder	$21.00	$29,400	$36,750	1	$36,750	$0.049	$0.016
Labourers	$15.00	$21,000	$26,250	1	$26,250	$0.035	$0.012
				6	$210,000	$0.280	$0.093
TOTAL MINING HOURLY				25	$794,500	$1.059	$0.353
TOTAL MINING				31	$1,100,750	$1.468	
UNIT COST $/T ORE 750000 TONNES					$1.468		
UNIT COST $/T MINED 2250000 TONNES					$0.489		
ADMINISTRATION OF SX-EW OPERATIONS							
Plant Manager *		$90,000	$112,500	1	$112,500		
Superintendent *		$70,000	$87,500	1	$87,500		
Shift Supervisors		$40,000	$50,000	4	$200,000		
Secretary/Clerk		$35,000	$29,167	1	$29,167		
Engineer *		$50,000	$62,500	1	$62,500		
Chemist		$40,000	$50,000	1	$50,000		
TOTAL WAGES ADMINISTRATION				9	$541,667	$0.722	
SX-EW OPERATIONS - HOURLY							
SX/EW	$18.00	$43,164	$35,970	8	$287,760		
Leaching	$18.00	$43,164	$35,970	7	$251,790		
Laboratory	$18.00	$43,164	$35,970	2	$71,940		
TOTAL WAGES OPERATIONS				17	$611,490	$0.815	
SX-EW MAINTENANCE - HOURLY							
Electricians/Instruments	$21.00	$50,358	$41,965	2	$83,930		
Mechanics	$21.00	$50,358	$41,965	2	$83,930		
TOTAL WAGES MAINTENANCE				4	$167,860	$0.224	
TOTAL WAGES HOURLY				21	$779,350	$1.039	
TOTAL WAGES HOURLY & SALARIED				30	$1,321,017	$1.761	
TOTAL MANPOWER COST MINING & SW-EW					$2,421,767	$3.229	
HOUSING COST					$60,000	$0.080	
CAMP COST					$261,600	$0.349	
TOTAL LABOUR COST					$2,743,367	$3.658	

TABLE 9 – MINE OPERATING COST ESTIMATE

EQUIPMENT	MAKE	TYPE	HRS/1000T	HOURS	$/HOUR INC WAGES	ANNUAL COST	COST/TONNE MINED	COST/TONNE ORE
LOADING								
LOADER	CAT	992C	1.75	2625	$82.00	$215,250	0.10	0.29
HAULAGE & ROADS								
HAUL TRUCKS	CAT	773	2.50	7500	$60.00	$450,000	0.20	0.60
TRACTORS	CAT	D8K	2.00	4500	$41.00	$184,500	0.08	0.25
GRADER	CAT	14G	0.50	1125	$21.00	$23,625	0.01	0.03
DRILLING								
ROTARY DRILL	DRILLTECH	D60K	0.83	1877	$104.00	$195,156	0.09	0.26
SKID STEER LOADER	BOBCAT	600		400	$20.00	$8,000	0.00	0.01
BLASTING AGENTS						$787,500	0.35	1.05
CAP MAGAZINE								
POWDER MAGAZINE								
POWDER/PUMP TRUCK						$5,000	0.00	0.01
MAINTENANCE								
SERVICE TRUCK	GMC					$5,000	0.00	0.01
WELDER	MILLER					$2,000	0.00	0.00
MISCELLANEOUS								
SINGLE BAY SHOP						$50,000	0.02	0.07
TOOLS						$2,000	0.00	0.00
FUEL TANKS	TIDY							
PICK UP TRUCKS	GMC					$35,000	0.02	0.05
AMBULANCE						$1,000	0.00	0.00
ENGINEERING								
COMMUNICATIONS						$20,000	0.01	0.03
SUPPLIES						$30,000	0.01	0.04
CONSULTANTS						$25,000	0.01	0.03
MISCELLANEOUS						$50,000	0.02	0.07
TOTAL MINING						$2,089,031	0.93	2.79

BACON, DONALDSON & ASSOCIATES LTD.

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METALLURGICAL INVESTIGATION OF WILLIAMS CREEK COPPER OXIDE ORE

Prepared for:

CONSOLIDATED SILVER STANDARD MINES LTD.
400 - 1199 W. Hastings Street
Vancouver, B.C.
V6E 3T5

Attention: Robert Quartermain

File Number: M90-001
May 15, 1990

Ed Henrioulle, B.A.Sc.

Dr. M.J.V. Beattie, P.Eng.

1.0 INTRODUCTION

At the request of Mr. Robert Quartermain of Consolidated Silver Standard Mines Ltd, metallurgical test work was carried out on bulk samples of oxide copper ore from the Williams Creek property. Two composites of run of mine sized ore were provided, with an average head grade of approximately 1.5% Cu. Total testwork in this investigation consisted of six bottle roll sulphuric acid leach tests, one bottle roll ammonia leach test, and two larger-scale sulphuric acid column leach tests. The purpose of the investigation was to provide detailed metallurgical information on the amenability of the Williams Creek ore to heap leaching with sulphuric acid.

This investigation was initiated by a phone call to Bacon, Donaldson & Associates (BDA) by Mr. Robert Quartermain on August 29, 1989. Further correspondence followed to define more fully the scope of the investigation. Actual testwork began on October 16, 1989. Throughout this investigation, testwork results were provided to Mr. Quartermain as soon as they became available.

2.0 SUMMARY

Bottle roll sulphuric acid leaching tests indicated that copper extractions of 80-85% Cu could be obtained for both ore composites at a fairly coarse crush (minus 3/4 inch). Net sulphuric acid consumption for these tests was 26-28 kg/tonne.

Two column leach tests conducted on 50:50 blends of composite Orange and composite Blue at a 3/4 inch crush yielded final copper extractions of 85.0% Cu (Column 1) and 86.7% Cu (Column 2). Both tests were identical, but Column 2 involved more intensive leaching in the beginning of the leach; solution strength was 20 g/l H_2SO_4 initially (compared to 15 g/l for Column 1), and solution flowrate was 100 ml/min initially (compared to 20 ml/min for Column 1). The more intensive leaching employed in Column 2 greatly increased the copper extraction rate, with Column 2 reaching 80% copper extraction within 15 days (compared to 60 days for Column 1).

With the shorter test duration, Column 2 had a marginally lower net sulphuric acid consumption; 48 kg/tonne compared to 51 kg/tonne for Column 1.

One bottle roll test utilizing ammonia leaching of the copper was performed. This test resulted in low copper extraction and this process route was not investigated further.

3.0 ORE CHARACTERIZATION

On September 6, 1989 approximately 2700 kg (6000 lbs) of damp, minus 15 cm (6 inch) copper oxide ore was received by Bacon, Donaldson and Associates at their facilities in Richmond, B.C. The shipment consisted of two separate ore composites: Composite Orange and Composite Blue. Approximately equal proportions of each composite were received.

The composites were jaw crushed separately to approximately minus 3 cm. It should be noted that some of the material crushed in a distinctly slabby manner, with pieces up to 10 or 12 cm along their long axis and less than 3 cm along their short axis. This was not a major occurrence however, with the total amount of slabby breakage estimated to be less than 1% by weight.

Sample cuts were taken from each composite for head analysis. This information is presented in Table 3.1 along with the average back calculated head grades from testwork.

Table 3.1
Head Assay Summary

Composite	Assay Type	Total Copper % Cu	Oxide Copper as % Cu	Total Iron % Fe
Orange	Assay	1.32	1.26	4.00
	Avg. Back-Calc.	1.54	1.43	3.84
Blue	Assay	1.68	1.60	4.60
	Avg. Back-Calc.	1.65	1.57	4.77
50:50 Mix	Assay	1.50	1.43	4.80
	Avg. Back-Calc.	1.56	-	4.11

4.0 BOTTLE ROLL LEACHING

In total, 7 bottle roll tests were conducted in this study. Six of these were sulphuric acid leaches conducted at various ore sizes to determine the optimum feed size for each composite, and to establish the plateau for copper extraction that could be expected in subsequent column leach testing. The final bottle roll test (L7) was an ammonia leach on a 50:50 mix of composites Orange and Blue to determine the general amenability of the ore to leaching with an ammonia solution. All bottle roll tests were conducted on 4 to 5 kg of ore at 50 to 55% solids. All leach solutions were changed and analyzed incrementally to provide extraction data.

4.1 Bottle Roll Sulphuric Acid Leaching

Six bottle roll sulphuric acid leach tests were conducted in this study. The tests investigated three different ore sizes on both Composite Orange and Composite Blue. All other test parameters were similar:

- 4000 - 5000 g dry ore
- 50 - 55% solids
- 15 g/l H_2SO_4 solution strength
- 166 hour duration (approximately 7 days)

The results of the six tests are summarized in Table 4.1 and the complete details are presented in Appendix I.

Table 4.1

Bottle Roll Sulphuric Acid Leaching

Test No.	Composite	Ore Size	Tailings $\%Cu^{Total}$	Copper Extraction %	Net Acid Consumption Kg/tonne
L1	Blue	1-1.5 inch	0.38	74.7	29.3
L3	Blue	3/4 inch	0.35	80.6	27.9
L5	Blue	1/4-3/8 inch	0.34	80.3	23.8
L2	Orange	1-1.5 inch	0.52	66.3	28.0
L4	Orange	3/4 inch	0.28	82.3	26.4
L6	Orange	1/4-3/8 inch	0.20	86.6	26.5

The results in Table 4.1 indicate that Composite Orange shows greater sensitivity to crush size than Composite Blue. However, for both composites, the 3/4 inch size is adequate to achieve 80% copper extraction

In Table 4.1, the acid consumption figures shown are the net consumptions. These are calculated from the gross acid consumptions shown in the detailed balances (Appendix I) by correcting for the acid which occurs as copper.

Figures 4.1 and 4.2 illustrate the copper extraction as a function of time for the six tests. The curves indicate that the extraction rates have all levelled off by 166 hours, and it is likely that the final numbers have approached the ultimate extractions for these test parameters.

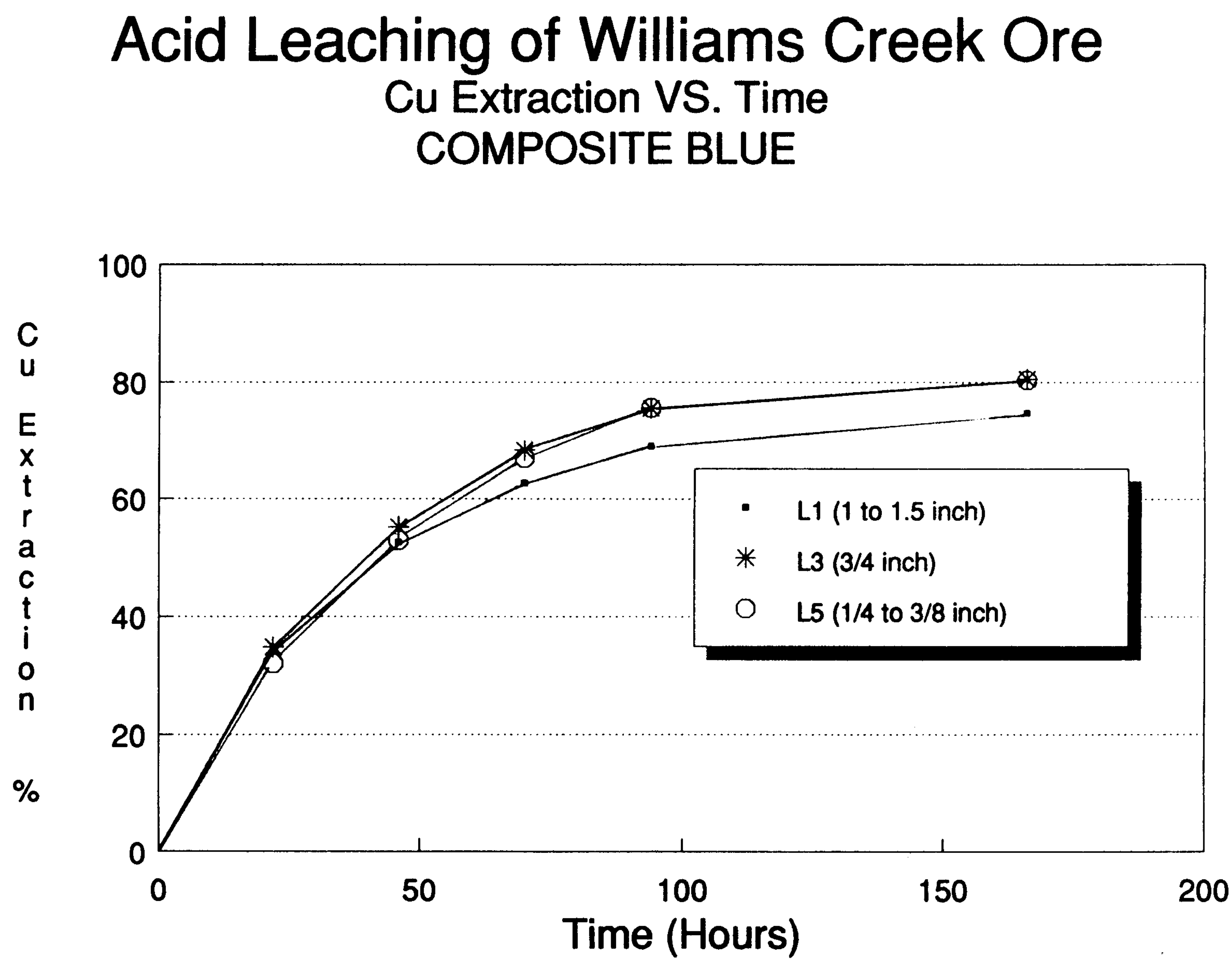

Acid Leaching of Williams Creek Ore
Cu Extraction VS. Time
COMPOSITE BLUE
Cu Extraction %
100
80
60
40
20
0
0
50
100
150
200
Time (Hours)
L1 (1 to 1.5 inch)
L3 (3/4 inch)
L5 (1/4 to 3/8 inch)

Figure 4.1

Acid Leaching of Williams Creek Ore

Cu Extraction VS. Time

COMPOSITE ORANGE

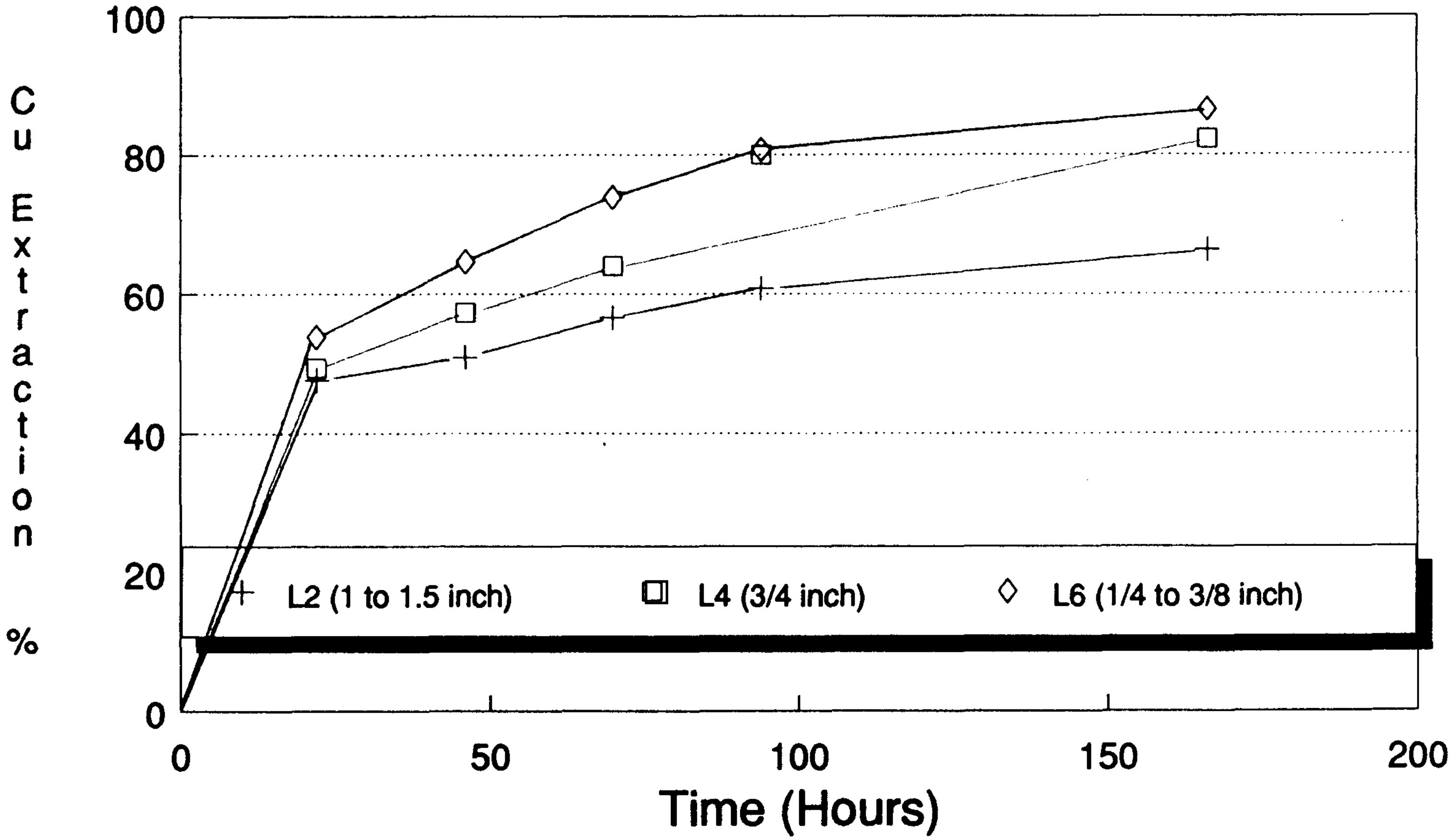


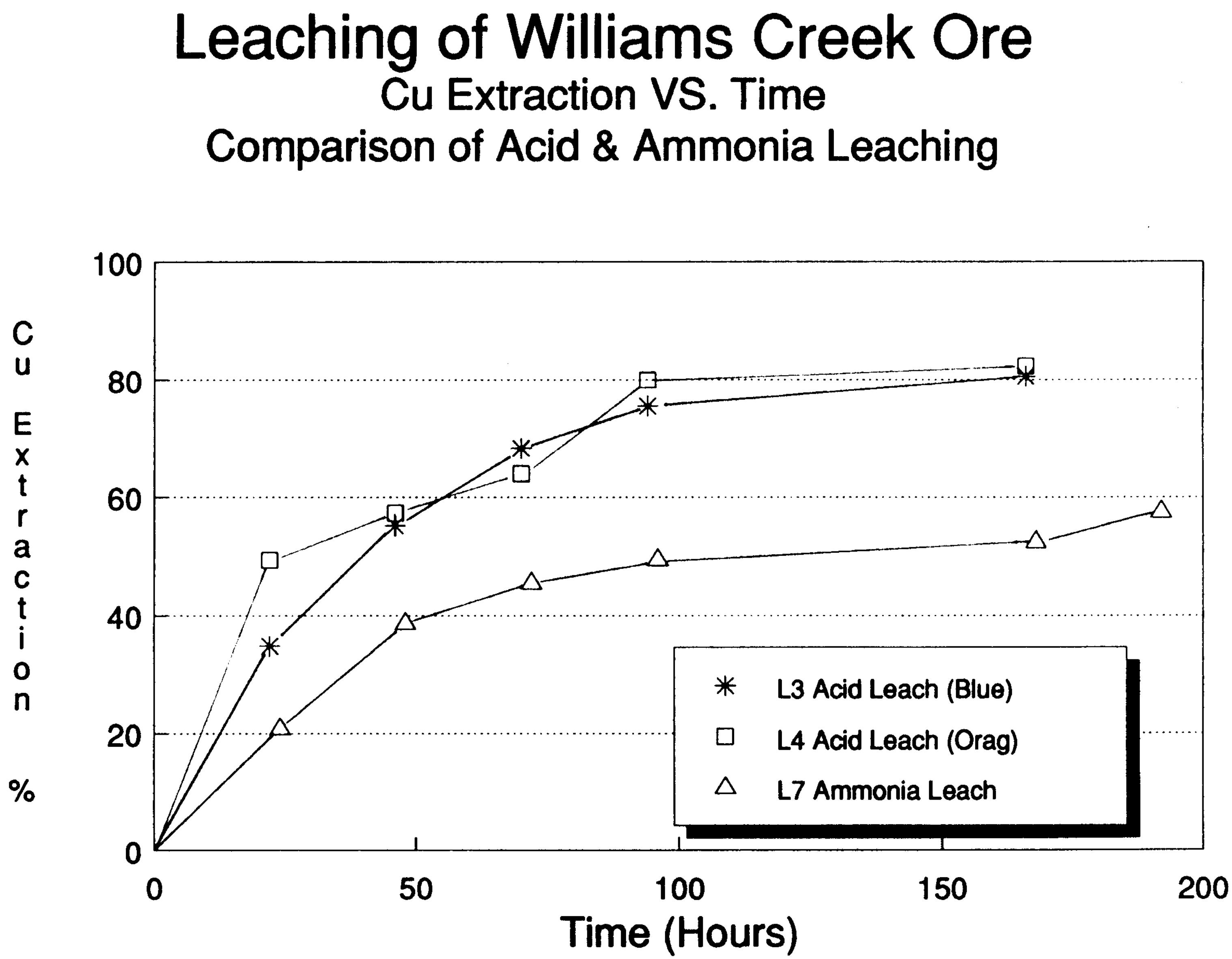
Figure 4.2

4.2 Bottle Roll Ammonia Leaching

A single bottle roll ammonia leach test was conducted on a 50:50 blend of Composite Orange and Composite Blue. The ore was crushed to 3/4 inch and leached for 10 days at 50% solids with a "0.25M NH_4OH + 0.25M $(NH_4)_2CO_3$" strength ammonia solution. The leach solution was changed daily for the first 96 hours and then after a total of 7 and 8 days.

Final extraction for this test was 57.7% copper and the final tailings assayed 0.70% Cu^T. These results are considerably worse than the comparable test (L3 and L4) with sulphuric acid. The ammonia leach extraction rate is compared to the rates of tests L3 and L4 in figure 4.3.

Since this test was preliminary in nature, no analyses were performed to establish reagent consumption. However, the motivation for the test was the fact that the contained iron and carbonate minerals would not be dissolved under the leach conditions.



Leaching of Williams Creek Ore
Cu Extraction VS. Time
Comparison of Acid & Ammonia Leaching
Cu Extraction %
100
80
60
40
20
0
0
50
100
150
200
Time (Hours)
L3 Acid Leach (Blue)
L4 Acid Leach (Orag)
L7 Ammonia Leach

Figure 4.3

5.0 COLUMN LEACHING

Two separate column leach tests were conducted on 50:50 blends of minus 3/4 inch Orange and Blue composites. Both tests utilized a 12 foot by 1 foot diameter PVC plastic pipe for a column container. Sulphuric acid leach solutions were pumped continuously by metering pumps to the drip point at the top of the column. A special drip arrangement ensured that the contents of the column were uniformly wetted through the cross sectional area. Both columns were loaded by pre-wetting the blended ore with an 18 g/l H_2SO_4 solution at a solution - solids ratio of 3%. Both columns contained approximately 375 kg of ore.

Regular maintenance of the columns consisted of collecting and weighing the pregnant (out) solution in its entirety. This solution was then sampled for assay, with a further 2 litres saved in a glass bottle for future reference. The remaining pregnant solution was neutralized and discarded. Feed solution (in) was topped up at the appropriate solution strength and the metering pumps were checked periodically to ensure they were delivering the correct flowrate.

To end the columns, feed solution was stopped and the columns were allowed to drain for a period of days. Following this, the columns were flushed twice with water at a high flow rate. The out solutions from the washes were collected and assayed. Finally, the columns were dumped, the solids examined visually to determine if there were indications of solution channelling, then the solids were dried and sampled for assay and size analysis.

The two column tests in this investigation differed in that the first column was leached slowly, with low solution strengths and flowrates. This test had a duration of 78 days. The second column was leached more intensely, with high solution strengths and high flowrates for the initial part of the test. This type of intensive leaching at the beginning of a test is known as "pugging". Column 2 had a duration of 33 days.

5.1 Column Test 1

Column test 1 was leached for 78 days (including draining and washing) and reached a final copper extraction of 85.0%. Figure 5.1 illustrates the extraction rate for copper over the duration of the test. The curve depicts a long climb up to the final extraction of 85% Cu; 60% extraction was not reached until day 32, and 80% until day 60.

Column 1 began leaching with a solution strength of 15 g/l H_2SO_4 at a flowrate of 20ml/min and this was continued until day 54 when the iron content of the out solution reached 3.0 g/l Fe. After day 54, the feed strength was cut back to 10 g/l H_2SO_4 and this was maintained for the duration of the test. The lower feed strength had an immediate effect on the amount of iron being leached; the next out solution assayed only 1.84 g/l Fe. Figure 5.2 illustrates the iron and copper concentration of the pregnant solution over time. The test was ended when the copper concentration of the pregnant solution had decreased to approximately 0.5 g/l.

Column 1 had a total acid consumption of 70.6 kg H_2SO_4 per tonne of ore. Of this, 19.1 kg/tonne was consumed by the copper and would be recovered in a solution processing plant. Correcting for the dissolved copper, the net acid consumption for column 1 was 51.5 kg/tonne.

When column 1 was dismantled, visual inspection of the tailings indicated that all the material had leached evenly with no indications of channelling. The tailings were dried, then sampled for assay and size analysis.

At the request of the client, gold and silver assays were also conducted on the tailings for column 1. The assay indicated values of 0.51 g/tonne Au and 4.53 g/tonne Ag.

Column Leaching of Williams Creek Ore

Cu Extraction vs Time

Column #1

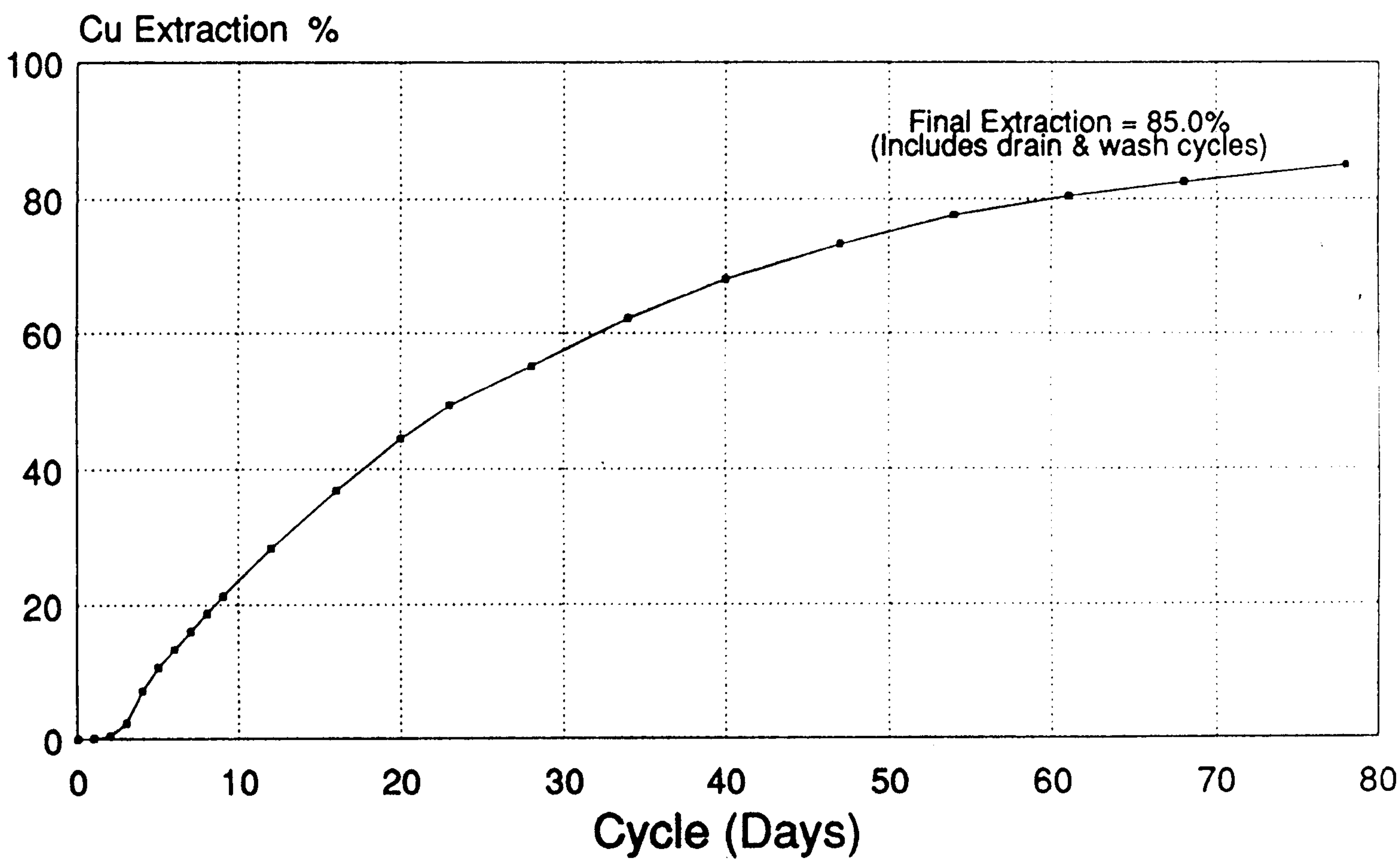


Figure 5.1

Column Leaching of Williams Creek Ore

Cu and Fe Concentration vs Time

Column #1

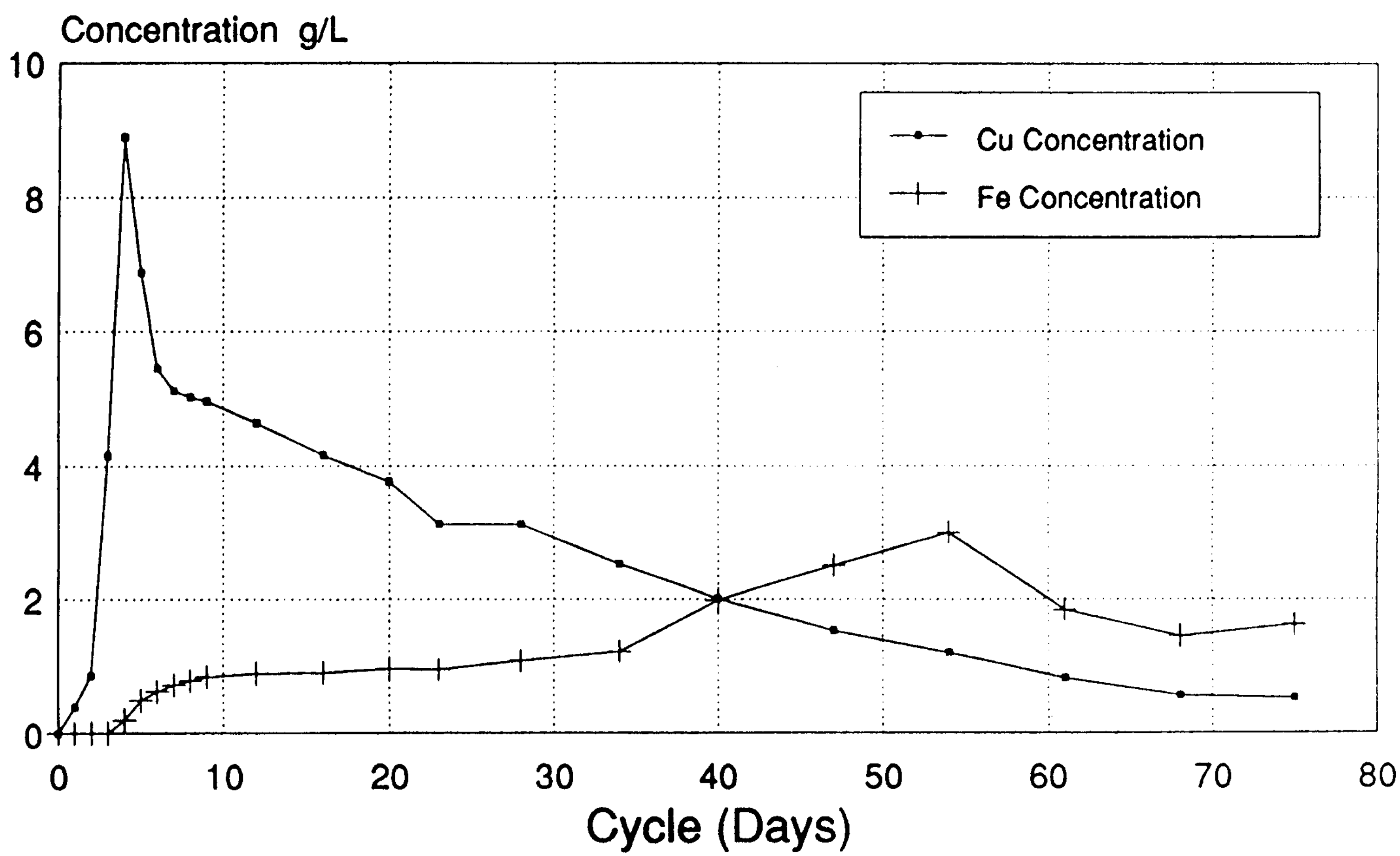


Figure 5.2

5.2 Column Test 2

Column test 2 was leached for 33 days (including draining and washing) and reached a final copper extraction of 86.7%. Figure 5.3 illustrates the rate of copper extraction over the course of the test. For this test, with its more intensive leaching, copper extraction progressed rapidly and reached 60% within 5 days, and 80% within 15 days.

Column 2 began leaching with a solution strength of 20 g/l H_2SO_4 and a flowrate of 100 ml/min (compared to 15 g/l H_2SO_4 and 20 ml/min for column 1) and maintained this until day 6 when the feed solution was cut back to 15 g/l H_2SO_4. Finally, on day 15 the feed strength was cut back to 10 g/l H_2SO_4. Throughout this test, iron content of the out solution did not exceed 2.0 g/l Fe. Flowrate of feed solution was cut back periodically through the course of the test and by the end of the test it was down to 20 ml/min. Figure 5.4 illustrates the copper and iron concentration of the pregnant solution over time. As for column 1, the test was ended when the copper concentration in the pregnant solution had decreased to approximately 0.5 g/L.

Column 2 had a total acid consumption of 69.1 kg H_2SO_4 per tonne of ore. Of this, 21.1 kg/tonne was consumed by this dissolving copper. After correcting for dissolved copper, <u>net</u> acid consumption for column 2 was 48 kg/tonne.

Upon dismantling of column 2, visual inspection of the tailings indicated even leaching with no apparent channelling. The dried tailings were sampled for assay and size analysis.

While column test 2 was in progress several pieces of the ore were placed in acid solution. This solution was changed periodically to ensure the availability of acid and to maximize the diffusion gradient for acid and copper. Upon the completion of column test 2 the pieces of rock were broken in half to observe the extent to which copper leaching had progressed. The copper oxide had been leached to a depth of 4 to 5mm. Inside this leached rim the rock contained visible copper "oxide" minerals.

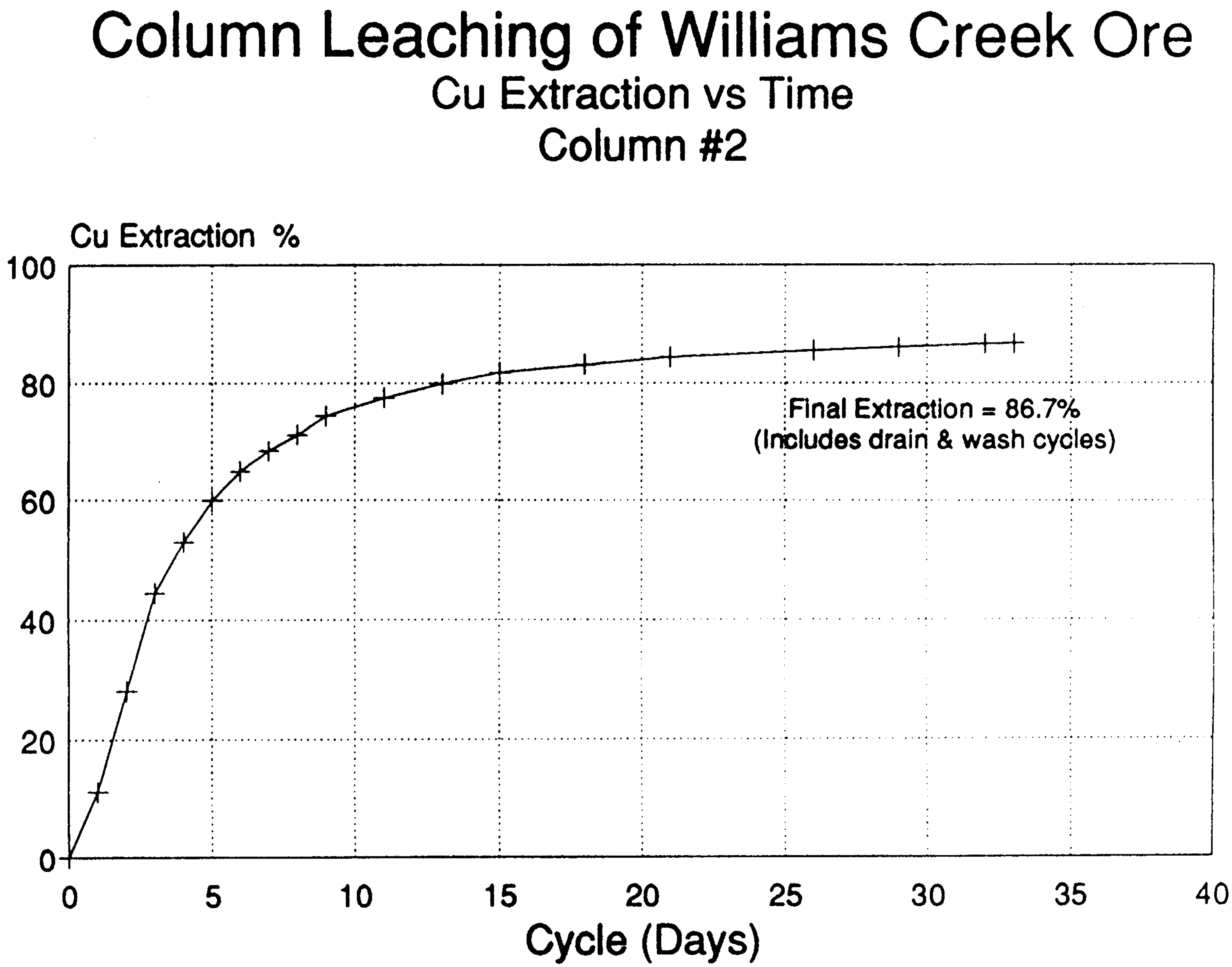

Column Leaching of Williams Creek Ore
Cu Extraction vs Time
Column #2
Cu Extraction %
100
80
60
40
20
0
Final Extraction = 86.7%
(Includes drain & wash cycles)
0
5
10
15
20
25
30
35
40
Cycle (Days)

Figure 5.3

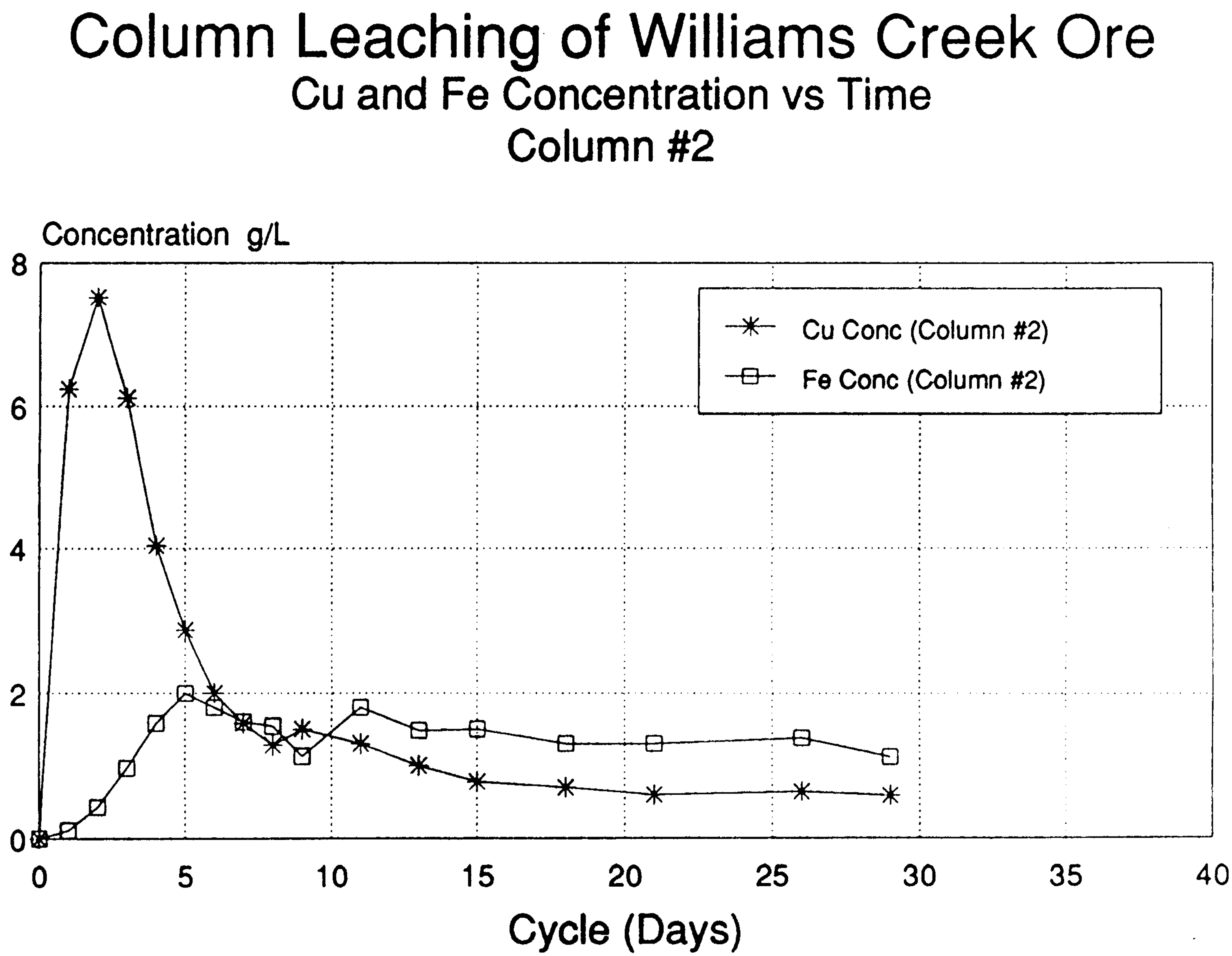

Column Leaching of Williams Creek Ore
Cu and Fe Concentration vs Time
Column #2
Concentration g/L
8
6
4
2
0
0
5
10
15
20
25
30
35
40
Cycle (Days)
Cu Conc (Column #2)
Fe Conc (Column #2)

Figure 5.4

5.3 Comparison: Column 1 vs Column 2

Table 5.1 summarizes the important information from both column tests.

Table 5.1

Comparison: Column 1 and Column 2

Test No.	Composite	Ore Size	Duration (Days)	Tailings %Cu^{Total}	Copper Extraction %	Net Acid Consumption Kg/tonne
Column 1	50:50 Mix	3/4"	78	0.22	85.0	51
Column 2	50:50 Mix	3/4"	33	0.21	86.7	48

Figure 5.5 illustrates the dramatic difference in copper extraction rates for the two tests.

It can be seen from the information in Table 5.1 and the graph in figure 5.5 that column 2 has given better results than column 1; copper extraction occurred much quicker and acid consumption was lower for column 2.

Column Leaching of Williams Creek Ore

Cu Extraction vs Time

Comparison of Column #1 & Column #2

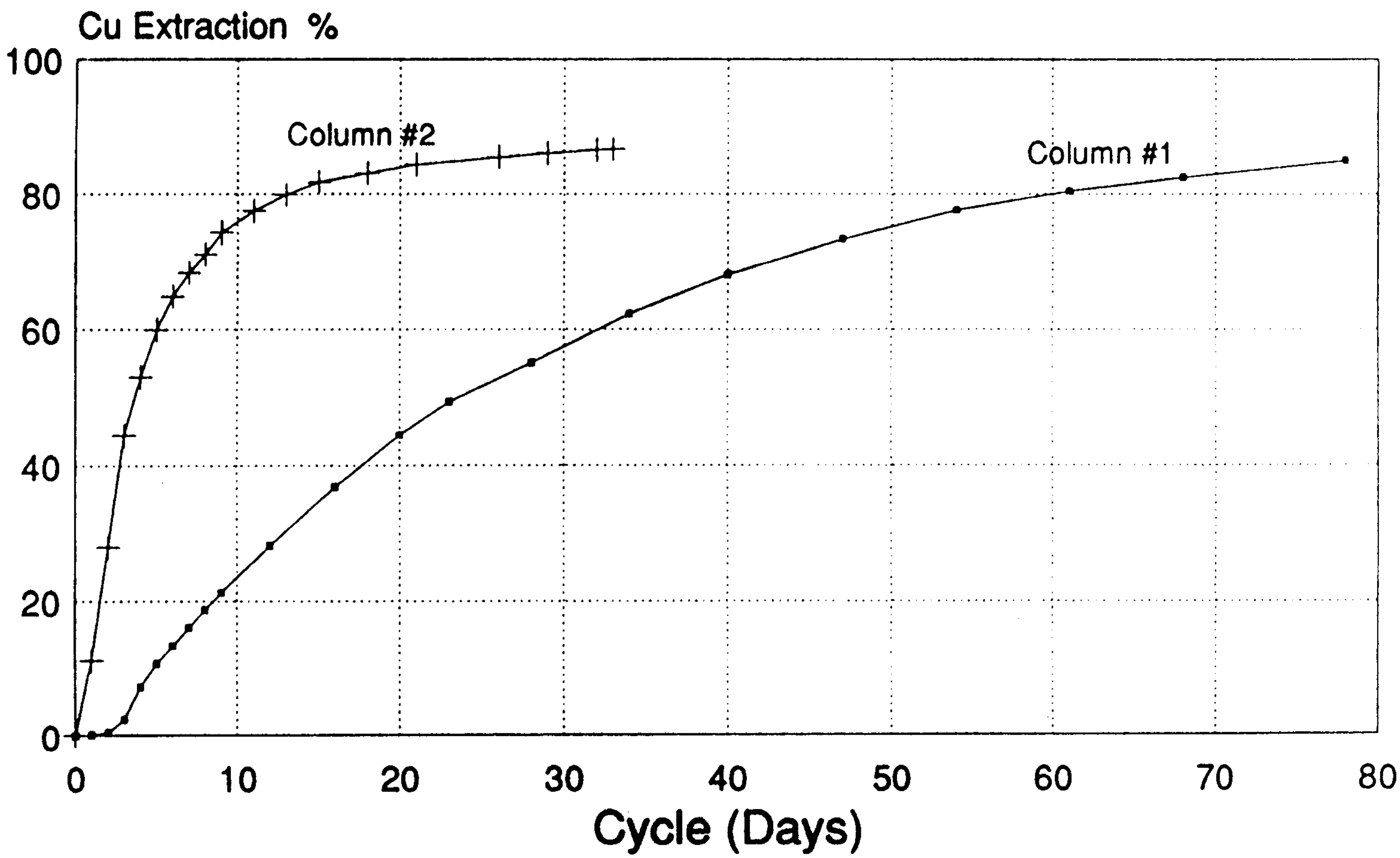


Figure 5.5

APPENDIX I
Bottle Roll Sulphuric Acid Leaching

BOTTLE ROLL SULPHURIC ACID LEACHING OF COPPER OXIDE ORES

File No: M90-001 Date: 16-Oct-89
Test No: L1
Sample Description: COMP BLUE (Size: 1 to 1.5 inch)

TEST CONDITIONS:

Solids: 4809.3 g
H2O: 4150.7 g
% Solids: 54%
Solution Strength: 15 g/l H2SO4
Test Duration: 166 hours

TEST DESCRIPTION:

-solids and acid solution combined in large bottle
-bottles placed on rollers
-each day, acid solution decanted and replaced with fresh solution
-decanted solution analyzed for Cu, Fe, free acid, and pH
-test ended after 166 hours
-solids washed and wash solutions analyzed
-final solids assayed for TOTAL COPPER, OXIDE COPPER, IRON

TEST RESULTS:

Solution Analyses:

TIME	SAMPLE SIZE	COPPER		IRON		H2SO4		pH
	(g)	(g/l)	(g)	(g/l)	(g)	(g/l)	(g)	
22	2919	5.00	14.60	0.03	0.09	0.29	0.85	2.9
46	2780	5.60	15.57	0.24	0.67	0.45	1.25	2.8
70	2622	3.60	9.44	0.55	1.44	0.71	1.86	2.6
94	2548	2.44	6.22	0.09	0.23	0.91	2.32	2.7
166	2319	1.92	4.45	1.33	3.08	0.79	1.83	2.5
wash	4411	0.83	3.66	0.81	3.57	1.07	4.72	2.2
	TOTAL		53.93		9.08		12.83	

Solids Analyses:

TIME	SAMPLE SIZE	TOTAL COPPER		OXIDE COPPER		IRON	
	(g)	(%)	(g)	(%)	(g)	(%)	(g)
166	4809.3	0.38	18.28	0.28	13.47	4.60	221.23
	TOTAL		18.28		13.47		221.23

CALCULATIONS:

TIME	COPPER EXTRACTION		IRON EXTRACTION		ACID CONSUMPTION	
	INDV. %	CUM. %	INDV. %	CUM. %	INDV. kg/tonne	CUM. kg/tonne
22	34.55%	34.55%	0.06%	0.06%	12.70	12.70
46	18.07%	52.62%	0.41%	0.47%	8.79	21.49
70	10.08%	62.70%	0.86%	1.33%	8.19	29.67
94	6.31%	69.00%	-0.21%	1.12%	7.62	37.29
166	5.59%	74.59%	2.35%	3.47%	7.57	44.86
wash	0.10%	74.69%	0.47%	3.94%	1.73	46.59
	TOTAL	74.69%		3.94%		46.59

HEAD GRADE:

	TOTAL COPPER	OXIDE COPPER	IRON
ASSAY HEAD	1.68%	1.60%	4.60%
CALCULATED HEAD	1.50%	1.40%	4.79%

SIZE DISTRIBUTION

SAMPLE NO. M90-001 L1

Bottle Roll Tails

Size Fraction (mesh)		Individual Percentage Retained %	Cumulative Percentage Passing %
	+ 1"	16.6	83.4
- 1"	+ 3/4"	12.7	70.5
- 3/4"	+ 1/2"	5.1	65.4
- 1/2"	+ 3/8"	2.7	62.7
- 3/8"	+ 4 mesh	2.6	60.1
- 4	+ 6	0.6	59.6
- 6	+ 8	0.5	59.1
- 8	+ 10	0.6	58.5
- 10	+ 14	0.4	58.0
- 14	+ 20	0.5	57.6
- 20	+ 28	0.6	57.0
- 28	+ 35	2.0	55.0
- 35	+ 48	1.4	53.6
- 48	+ 65	1.3	52.3
- 65	+ 100	1.2	51.1
- 100	+ 150	1.4	49.7
- 150	+ 200	1.9	47.8
- 200		47.8	

BOTTLE ROLL SULPHURIC ACID LEACHING OF COPPER OXIDE ORES

File No: M90-001 Date: 16-Oct-89
Test No: L2
Sample Description: COMP ORANGE (Size: 1 to 1.5 inch)

TEST CONDITIONS:

Solids: 4897 g
H2O: 4458 g
% Solids: 52%
Solution Strength: 15 g/l H2SO4
Test Duration: 166 hours

TEST DESCRIPTION:

-solids and acid solution combined in large bottle
-bottles placed on rollers
-each day, acid solution decanted and replaced with fresh solution
-decanted solution analyzed for Cu, Fe, free acid, and pH
-test ended after 166 hours
-solids washed and wash solutions analyzed
-final solids assayed for TOTAL COPPER, OXIDE COPPER, IRON

TEST RESULTS:

Solution Analyses:

TIME	SAMPLE SIZE	COPPER		IRON		H2SO4		pH
	(g)	(g/l)	(g)	(g/l)	(g)	(g/l)	(g)	
22	3193	6.76	21.58	0.33	1.05	0.33	1.05	2.7
46	3019	4.12	12.44	0.66	1.99	1.13	3.41	2.3
70	2838	2.12	6.02	1.49	4.23	2.73	7.75	2.2
94	2701	1.43	3.86	1.16	3.13	3.99	10.78	2.2
166	2506	1.50	3.76	2.10	5.26	1.91	4.79	2.2
wash	5054	0.49	2.48	0.78	3.94	2.67	13.49	2.2
	TOTAL		**50.14**		**19.61**		**41.27**	

Solids Analyses:

TIME	SAMPLE SIZE	TOTAL COPPER		OXIDE COPPER		IRON	
	(g)	(%)	(g)	(%)	(g)	(%)	(g)
166	4897	0.52	25.46	0.31	15.18	3.20	156.70
	TOTAL		**25.46**		**15.18**		**156.70**

CALCULATIONS:

TIME	COPPER EXTRACTION		IRON EXTRACTION		ACID CONSUMPTION	
	INDV. %	CUM. %	INDV. %	CUM. %	INDV. kg/tonne	CUM. kg/tonne
22	47.62%	47.62%	1.00%	1.00%	13.35	13.35
46	3.28%	50.90%	1.14%	2.13%	8.84	22.19
70	5.71%	56.61%	3.09%	5.23%	7.09	29.29
94	4.15%	60.76%	1.61%	6.84%	5.96	35.25
166	5.48%	66.24%	3.97%	10.81%	7.97	43.22
wash	0.08%	66.32%	0.32%	11.12%	0.56	43.78
TOTAL		**66.32%**		**11.12%**		**43.78**

HEAD GRADE:

	TOTAL COPPER	OXIDE COPPER	IRON
ASSAY HEAD	1.32%	1.26%	4.00%
CALCULATED HEAD	1.54%	1.33%	3.60%

SIZE DISTRIBUTION

SAMPLE NO. M90-001 L2

Bottle Roll Tails

Size Fraction (mesh)		Individual Percentage Retained %	Cumulative Percentage Passing %
	+ 1"	25.2	74.8
- 1"	+ 3/4"	18.8	56.1
- 3/4"	+ 1/2"	8.4	47.7
- 1/2"	+ 3/8"	4.6	43.1
- 3/8"	+ 4 mesh	3.4	39.7
- 4	+ 6	0.9	38.8
- 6	+ 8	0.6	38.2
- 8	+ 10	0.5	37.7
- 10	+ 14	0.3	37.4
- 14	+ 20	0.3	37.1
- 20	+ 28	0.1	37.0
- 28	+ 35	0.2	36.8
- 35	+ 48	0.1	36.7
- 48	+ 65	0.2	36.5
- 65	+ 100	0.2	36.3
- 100	+ 150	0.4	35.9
- 150	+ 200	0.7	35.2
- 200		35.2	

BOTTLE ROLL SULPHURIC ACID LEACHING OF COPPER OXIDE ORES

File No: M90-001 Date: 16-Oct-89
Test No: L3
Sample Description: COMP BLUE (Size: 3/4 inch)

TEST CONDITIONS:

Solids:	4538.2 g
H2O:	4450.8 g
% Solids:	50%
Solution Strength:	15 g/l H2SO4
Test Duration:	166 hours

TEST DESCRIPTION:

-solids and acid solution combined in large bottle
-bottles placed on rollers
-each day, acid solution decanted and replaced with fresh solution
-decanted solution analyzed for Cu, Fe, free acid, and pH
-test ended after 166 hours
-solids washed and wash solutions analyzed
-final solids assayed for TOTAL COPPER, OXIDE COPPER, IRON

TEST RESULTS:

Solution Analyses:

TIME	SAMPLE SIZE	COPPER		IRON		H2SO4		pH
	(g)	(g/l)	(g)	(g/l)	(g)	(g/l)	(g)	
22	2812	5.32	14.96	0.02	0.06	0.29	0.82	2.7
46	2733	6.68	18.26	0.12	0.33	0.33	0.90	2.9
70	2524	5.00	12.62	0.40	1.01	0.50	1.26	2.8
94	2424	3.52	8.53	0.07	0.17	0.66	1.60	2.7
166	2362	2.54	6.00	1.08	2.55	0.56	1.32	2.6
wash	3595	1.50	5.39	0.75	2.70	0.63	2.26	2.5
	TOTAL		65.76		6.81		8.17	

Solids Analyses:

TIME	SAMPLE SIZE	TOTAL COPPER		OXIDE COPPER		IRON	
	(g)	(%)	(g)	(%)	(g)	(%)	(g)
166	4538.2	0.35	15.88	0.23	10.44	4.20	190.60
	TOTAL		15.88		10.44		190.60

CALCULATIONS:

TIME	COPPER EXTRACTION		IRON EXTRACTION		ACID CONSUMPTION	
	INDV. %	CUM. %	INDV. %	CUM. %	INDV. kg/tonne	CUM. kg/tonne
22	34.81%	34.81%	0.05%	0.05%	14.43	14.43
46	20.38%	55.20%	0.25%	0.30%	9.08	23.50
70	13.09%	68.29%	0.81%	1.11%	8.67	32.17
94	7.26%	75.55%	-0.24%	0.87%	7.91	40.08
166	5.36%	80.91%	2.44%	3.31%	7.76	47.84
wash	-0.36%	80.55%	0.14%	3.45%	2.36	50.20
	TOTAL	80.55%		3.45%		50.20

HEAD GRADE:

	TOTAL COPPER	OXIDE COPPER	IRON
ASSAY HEAD	1.68%	1.60%	4.60%
CALCULATED HEAD	1.80%	1.68%	4.35%

SIZE DISTRIBUTION

SAMPLE NO. M90-001 L3

Bottle Roll Tails

Size Fraction (mesh)		Individual Percentage Retained %	Cumulative Percentage Passing %
	+ 1"	0	100.0
- 1"	+ 3/4"	2.0	98.0
- 3/4"	+ 1/2"	12.5	85.5
- 1/2"	+ 3/8"	6.2	79.3
- 3/8"	+ 4 mesh	7.5	71.8
- 4	+ 6	2.1	69.7
- 6	+ 8	1.6	68.1
- 8	+ 10	1.8	66.3
- 10	+ 14	1.4	65.0
- 14	+ 20	1.5	63.5
- 20	+ 28	1.0	62.5
- 28	+ 35	2.7	59.9
- 35	+ 48	2.4	57.4
- 48	+ 65	2.6	54.8
- 65	+ 100	2.6	52.2
- 100	+ 150	2.5	49.7
- 150	+ 200	2.2	47.4
- 200		47.4	

BOTTLE ROLL SULPHURIC ACID LEACHING OF COPPER OXIDE ORES

File No: M90-001 Date: 16-Oct-89
Test No: L4
Sample Description: COMP ORANGE (Size: 3/4 inch)

TEST CONDITIONS:

Solids: 4805.3 g
H2O: 4167.7 g
% Solids: 54%
Solution Strength: 15 g/l H2SO4
Test Duration: 166 hours

TEST DESCRIPTION:

-solids and acid solution combined in large bottle
-bottles placed on rollers
-each day, acid solution decanted and replaced with fresh solution
-decanted solution analyzed for Cu, Fe, free acid, and pH
-test ended after 166 hours
-solids washed and wash solutions analyzed
-final solids assayed for TOTAL COPPER, OXIDE COPPER, IRON

TEST RESULTS:

Solution Analyses:

TIME	SAMPLE SIZE	COPPER		IRON		H2SO4		pH
	(g)	(g/l)	(g)	(g/l)	(g)	(g/l)	(g)	
22	2927	7.45	21.81	0.29	0.85	0.27	0.79	2.8
46	2914	5.16	15.04	0.58	1.69	0.62	1.81	2.6
70	2813	2.80	7.88	0.93	2.62	1.37	3.85	2.3
94	2637	3.80	10.02	0.15	0.40	2.36	6.22	2.2
166	2263	1.80	4.07	1.58	3.58	1.36	3.08	2.2
wash	3362	1.09	3.66	1.14	3.83	1.21	4.07	2.3
	TOTAL		62.48		12.96		19.82	

Solids Analyses:

TIME	SAMPLE SIZE	TOTAL COPPER		OXIDE COPPER		IRON	
	(g)	(%)	(g)	(%)	(g)	(%)	(g)
166	4805.3	0.28	13.45	0.22	10.57	3.60	172.99
	TOTAL		13.45		10.57		172.99

CALCULATIONS:

TIME	COPPER EXTRACTION		IRON EXTRACTION		ACID CONSUMPTION	
	INDV. %	CUM. %	INDV. %	CUM. %	INDV. kg/tonne	CUM. kg/tonne
22	49.38%	49.38%	0.78%	0.78%	12.78	12.78
46	7.93%	57.30%	0.98%	1.77%	8.67	21.44
70	6.71%	64.01%	1.70%	3.47%	8.07	29.51
94	15.94%	79.95%	-0.36%	3.11%	7.12	36.63
166	2.08%	82.03%	3.43%	6.55%	7.80	44.44
wash	0.25%	82.28%	0.42%	6.97%	2.05	46.49
TOTAL		82.28%		6.97%		46.49

HEAD GRADE:

	TOTAL COPPER	OXIDE COPPER	IRON
ASSAY HEAD	1.32%	1.26%	4.00%
CALCULATED HEAD	1.58%	1.52%	3.87%

SIZE DISTRIBUTION

SAMPLE NO. M90-001 L4

Bottle Roll Tails

Size Fraction (mesh)		Individual Percentage Retained %	Cumulative Percentage Passing %
	+ 1"	0	100.0
- 1"	+ 3/4"	8.9	91.1
- 3/4"	+ 1/2"	25.5	65.6
- 1/2"	+ 3/8"	10.3	55.3
- 3/8"	+ 4 mesh	9.5	45.7
- 4	+ 6	2.0	43.8
- 6	+ 8	1.5	42.3
- 8	+ 10	1.5	40.8
- 10	+ 14	1.0	39.8
- 14	+ 20	1.0	38.9
- 20	+ 28	0.6	38.3
- 28	+ 35	1.3	37.0
- 35	+ 48	1.0	36.0
- 48	+ 65	0.9	35.1
- 65	+ 100	0.7	34.4
- 100	+ 150	0.6	33.8
- 150	+ 200	0.7	33.2
- 200		33.2	

BOTTLE ROLL SULPHURIC ACID LEACHING OF COPPER OXIDE ORES

File No: M90-001
Test No: L5
Sample Description: COMP BLUE (Size: 1/4 to 3/8 inch)

Date: 16-Oct-89

TEST CONDITIONS:

Solids: 4595.3 g
H2O: 4395.7 g
% Solids: 51%
Solution Strength: 15 g/l H2SO4
Test Duration: 166 hours

TEST DESCRIPTION:

-solids and acid solution combined in large bottle
-bottles placed on rollers
-each day, acid solution decanted and replaced with fresh solution
-decanted solution analyzed for Cu, Fe, free acid, and pH
-test ended after 166 hours
-solids washed and wash solutions analyzed
-final solids assayed for TOTAL COPPER, OXIDE COPPER, IRON

TEST RESULTS:

Solution Analyses:

TIME	SAMPLE SIZE	COPPER		IRON		H2SO4		pH
	(g)	(g/l)	(g)	(g/l)	(g)	(g/l)	(g)	
22	2750	4.80	13.20	0.03	0.08	0.17	0.47	3.4
46	2494	6.60	16.46	0.15	0.37	0.21	0.52	3.2
70	2430	5.36	13.02	0.39	0.95	0.38	0.92	2.9
94	2341	3.88	9.08	0.62	1.45	0.67	1.57	2.7
166	2166	2.88	6.24	1.03	2.23	0.44	0.95	2.7
wash	3387	1.70	5.76	0.69	2.34	0.36	1.22	2.7
	TOTAL		63.76		7.42		5.66	

Solids Analyses:

TIME	SAMPLE SIZE	TOTAL COPPER		OXIDE COPPER		IRON	
	(g)	(%)	(g)	(%)	(g)	(%)	(g)
166	4595.3	0.34	15.62	0.23	10.57	5.00	229.77
	TOTAL		15.62		10.57		229.77

CALCULATIONS:

TIME	COPPER EXTRACTION		IRON EXTRACTION		ACID CONSUMPTION	
	INDV. %	CUM. %	INDV. %	CUM. %	INDV. kg/tonne	CUM. kg/tonne
22	32.03%	32.03%	0.07%	0.07%	14.19	14.19
46	20.92%	52.95%	0.24%	0.31%	8.84	23.02
70	13.95%	66.90%	0.60%	0.91%	7.86	30.89
94	8.70%	75.60%	0.85%	1.76%	7.45	38.34
166	5.50%	81.10%	1.35%	3.11%	7.52	45.86
wash	-0.78%	80.32%	0.02%	3.13%	2.30	48.17
TOTAL		80.32%		3.13%		48.17

HEAD GRADE:

	TOTAL COPPER	OXIDE COPPER	IRON
ASSAY HEAD	1.68%	1.60%	4.60%
CALCULATED HEAD	1.73%	1.62%	5.16%

SIZE DISTRIBUTION

SAMPLE NO. M90-001 L5

Bottle Roll Tails

Size Fraction (mesh)		Individual Percentage Retained %	Cumulative Percentage Passing %
	+ 1"	0	100.0
- 1"	+ 3/4"	0	100.0
- 3/4"	+ 1/2"	0.9	99.1
- 1/2"	+ 3/8"	4.1	95.1
- 3/8"	+ 4 mesh	15.6	79.5
- 4	+ 6	4.5	75.0
- 6	+ 8	3.1	71.8
- 8	+ 10	3.3	68.5
- 10	+ 14	2.6	65.9
- 14	+ 20	2.8	63.1
- 20	+ 28	1.8	61.3
- 28	+ 35	4.2	57.1
- 35	+ 48	3.7	53.3
- 48	+ 65	4.0	49.4
- 65	+ 100	3.7	45.7
- 100	+ 150	3.4	42.3
- 150	+ 200	2.8	39.6
- 200		39.6	

BOTTLE ROLL SULPHURIC ACID LEACHING OF COPPER OXIDE ORES

File No: M90-001 Date: 16-Oct-89
Test No: L6
Sample Description: COMP ORANGE (Size: 1/4 to 3/8 inch)

TEST CONDITIONS:

Solids:	4787 g
H2O:	4150 g
% Solids:	54%
Solution Strength:	15 g/l H2SO4
Test Duration:	166 hours

TEST DESCRIPTION:

-solids and acid solution combined in large bottle
-bottles placed on rollers
-each day, acid solution decanted and replaced with fresh solution
-decanted solution analyzed for Cu, Fe, free acid, and pH
-test ended after 166 hours
-solids washed and wash solutions analyzed
-final solids assayed for TOTAL COPPER, OXIDE COPPER, IRON

TEST RESULTS:

Solution Analyses:

TIME	SAMPLE SIZE	COPPER		IRON		H2SO4		pH
	(g)	(g/l)	(g)	(g/l)	(g)	(g/l)	(g)	
22	2750	7.40	20.35	0.28	0.77	0.24	0.66	2.7
46	2863	6.08	17.41	0.57	1.63	0.29	0.83	2.9
70	2689	3.52	9.47	0.85	2.29	0.90	2.42	2.5
94	2488	2.44	6.07	0.12	0.30	1.51	3.76	2.3
166	2281	1.98	4.52	1.45	3.31	1.03	2.35	2.3
wash	3777	1.03	3.89	0.90	3.40	0.84	3.17	2.4
	TOTAL		**61.70**		**11.69**		**13.19**	

Solids Analyses:

TIME	SAMPLE SIZE	TOTAL COPPER		OXIDE COPPER		IRON	
	(g)	(%)	(g)	(%)	(g)	(%)	(g)
166	4787	0.2	9.57	0.15	7.18	3.80	181.91
	TOTAL		**9.57**		**7.18**		**181.91**

CALCULATIONS:

TIME	COPPER EXTRACTION		IRON EXTRACTION		ACID CONSUMPTION	
	INDV.	CUM.	INDV.	CUM.	INDV.	CUM.
	%	%	%	%	kg/tonne	kg/tonne
22	53.82%	53.82%	0.75%	0.75%	12.80	12.80
46	10.78%	64.60%	0.89%	1.64%	8.44	21.23
70	9.30%	73.90%	1.46%	3.10%	8.27	29.50
94	6.83%	80.73%	-0.42%	2.68%	7.39	36.89
166	6.01%	86.74%	3.12%	5.80%	7.43	44.32
wash	-0.17%	86.57%	0.24%	6.04%	2.12	46.44
TOTAL		**86.57%**		**6.04%**		**46.44**

HEAD GRADE:

	TOTAL COPPER	OXIDE COPPER	IRON
ASSAY HEAD	1.32%	1.26%	4.00%
CALCULATED HEAD	1.49%	1.44%	4.04%

SIZE DISTRIBUTION

SAMPLE NO. M90-001 L6

Bottle Roll Tails

Size Fraction (mesh)		Individual Percentage Retained %	Cumulative Percentage Passing %
	+ 1"	0	100.0
- 1"	+ 3/4"	0	100.0
- 3/4"	+ 1/2"	1.2	98.8
- 1/2"	+ 3/8"	4.4	94.4
- 3/8"	+ 4 mesh	30.1	64.3
- 4	+ 6	9.2	55.1
- 6	+ 8	5.0	50.1
- 8	+ 10	4.3	45.8
- 10	+ 14	2.6	43.3
- 14	+ 20	2.2	41.0
- 20	+ 28	1.4	39.7
- 28	+ 35	3.2	36.5
- 35	+ 48	2.8	33.7
- 48	+ 65	2.7	31.1
- 65	+ 100	2.3	28.8
- 100	+ 150	1.9	26.9
- 150	+ 200	1.4	25.5
- 200		25.5	

APPENDIX II

Bottle Roll Ammonia Leaching

BOTTLE ROLL AMMONIA LEACHING OF COPPER OXIDE ORES

File No: M90-001 Date: 12-Mar-90
Test No: L7
Sample Description: ORANGE ZONE:BLUE ZONE 50:50 COMPOSITE 3/4" CRUSH

TEST CONDITIONS:

Solids: 4000 g
H2O: 4000 g
% Solids: 50%
Solution Strength: .25 M NH4OH + .25 M (NH4)2CO3
Test Duration: 10 days

TEST DESCRIPTION:

-solids and ammonia solution combined in large bottle
-bottles placed on rollers
-each day, ammonia sol'n decanted and replaced with fresh sol'n
-decanted solution analyzed for Cu, Fe, and pH
-test ended after ten (10) days
-solids washed and wash solutions analyzed
-final solids assayed for TOTAL COPPER, OXIDE COPPER, IRON

TEST RESULTS:

Solution Analyses:

TIME	SAMPLE SIZE	COPPER		IRON		pH
hours	(g)	(g/l)	(g)	(g/l)	(g)	
24	3359	3.36	11.29	0.00	0.00	9.5
48	2986	3.48	10.39	0.00	0.00	9.3
72	2961	2.04	6.04	0.00	0.00	9.3
96	2833	1.20	3.40	0.00	0.00	9.4
168	2810	0.86	2.42	0.00	0.00	9.4
192	2710	0.98	2.66	0.00	0.00	9.4
wash	4835	0.34	1.64	0.00	0.00	9.2
	TOTAL		37.84		0.00	

Solids Analyses:

TIME	SAMPLE SIZE	TOTAL COPPER		OXIDE COPPER		IRON	
	(g)	(%)	(g)	(%)	(g)	(%)	(g)
192	4000	0.7	28.00	0.66	26.40	4.80	192.00
	TOTAL		28.00		26.40		192.00

CALCULATIONS:

TIME	COPPER EXTRACTION		IRON EXTRACTION	
	INDV.	CUM.	INDV.	CUM.
hours	%	%	%	%
24	20.90%	20.90%	0.00%	0.00%
48	17.89%	38.79%	0.00%	0.00%
72	6.83%	45.61%	0.00%	0.00%
96	3.95%	49.56%	0.00%	0.00%
168	3.07%	52.63%	0.00%	0.00%
192	4.41%	57.04%	0.00%	0.00%
wash	0.67%	57.70%	0.00%	0.00%
TOTAL		57.70%		0.00%

HEAD GRADE:

	TOTAL COPPER	OXIDE COPPER	IRON
ASSAY HEAD	1.50%	1.43%	4.80%
CALCULATED HEAD	1.65%	1.61%	4.80%

APPENDIX III

Column Leaching

M90-001: WILLIAMS CREEK COPPER COLUMN #1

DATE		LEACH CYCLE	LITRES IN	LITRES OUT	FLOW (ml/min) IN (Aim)	FLOW (ml/min) OUT (Actual)	H2SO4 (g/l) IN	H2SO4 (g/l) OUT	CONSUMPTION H2SO4 GRAMS IND.	CONSUMPTION H2SO4 GRAMS CUM.	CONSUMPTION H2SO4 Kg/tonne (TOTAL)	CONSUMPTION H2SO4 Kg/tonne (As Cu)	PREGNANT SOLUTION Fe (g/l)	PREGNANT SOLUTION COPPER (g/l)	PREGNANT SOLUTION COPPER (g total)	CALCULATED % COPPER EXTRACTED IND.	CALCULATED % COPPER EXTRACTED CUM.
Nov	30	1	29.43	10.05	20	7.0	14.79	0.05	638.8	638.8	1.69	0.02	0.00	0.39	3.92	0.1%	0.1%
Dec	1	2	26.70	24.11	20	16.7	14.79	0.09	392.6	1031.5	2.73	0.10	0.00	0.86	20.74	0.4%	0.4%
	2	3	27.96	25.78	20	17.9	14.79	0.13	410.2	1441.6	3.81	0.54	0.00	4.16	107.25	2.0%	2.4%
	3	4	27.96	29.80	20	20.7	14.79	0.17	408.5	1850.2	4.90	1.62	0.20	8.90	265.19	4.8%	7.2%
	4	5	28.27	27.46	20	19.1	14.79	0.03	417.3	2267.5	6.00	2.39	0.49	6.88	188.92	3.4%	10.7%
	5	6	27.75	27.98	20	19.4	14.79	0.05	408.9	2676.4	7.08	3.01	0.62	5.45	152.51	2.8%	13.4%
	6	7	28.86	28.19	20	19.6	14.79	0.14	422.9	3099.3	8.20	3.60	0.72	5.12	144.35	2.6%	16.1%
	7	8	28.54	28.77	20	20.0	14.79	0.16	417.5	3516.8	9.31	4.19	0.78	5.02	144.44	2.6%	18.7%
	8	9	27.71	27.81	20	19.3	14.79	0.20	404.3	3921.1	10.38	4.75	0.84	4.96	137.93	2.5%	21.2%
	11	12	83.65	84.13	20	19.5	14.79	0.09	1229.6	5150.6	13.63	6.34	0.89	4.64	390.34	7.1%	28.3%
	15	16	114.55	112.27	20	19.5	14.79	0.28	1662.7	6813.3	18.03	8.24	0.90	4.16	467.06	8.5%	36.8%
	19	20	114.01	111.14	20	19.3	14.79	0.43	1638.4	8451.7	22.37	9.95	0.96	3.76	417.87	7.6%	44.4%
	22	23	85.18	85.00	20	19.7	14.79	0.86	1186.7	9638.5	25.51	11.03	0.95	3.12	265.20	4.8%	49.2%
	27	28	106.08	101.70	20	14.1	14.79	0.92	1475.3	11113.7	29.41	12.32	1.08	3.12	317.31	5.8%	55.0%
Jan	2	34	163.60	157.59	20	18.2	14.79	0.96	2268.4	13382.2	35.41	13.94	1.22	2.52	397.12	7.2%	62.2%
	8	40	167.24	160.49	20	18.6	14.79	0.40	2409.2	15791.4	41.79	15.25	1.98	2.00	320.98	5.8%	68.0%
	15	47	196.22	192.61	20	19.1	14.79	0.29	2846.2	18637.6	49.32	16.44	2.50	1.52	292.76	5.3%	73.4%
	22	54	199.00	193.27	20	19.2	14.79	1.00	2749.9	21387.5	56.60	17.38	3.00	1.20	231.93	4.2%	77.6%
	29	61	197.36	192.24	20	19.1	9.86	0.98	1757.6	23145.1	61.25	18.03	1.84	0.82	157.64	2.9%	80.4%
Feb	5	68	200.94	197.60	20	19.6	9.86	0.82	1819.3	24964.3	66.06	18.48	1.45	0.56	110.65	2.0%	82.5%
	12	75	187.00	181.18	20	18.0	9.86	0.56	1742.4	26706.7	70.67	18.86	1.62	0.52	94.21	1.7%	84.2%
	13	76	0.00	7.42	0	5.2	0.00	0.56	-4.2	26702.5	70.66	18.88	1.38	0.66	4.89	0.1%	84.3%
Wash 1		77	100.00	95.26	50	66.2	0.00	0.31	-29.5	26673.0	70.58	19.03	0.50	0.38	36.58	0.7%	84.9%
Wash 2		78	100.00	103.24	50	71.7	0.00	0.11	-11.4	26661.6	70.55	19.05	0.10	0.05	5.37	0.1%	85.0%
															4675.17		

COLUMN TAILS:

DATE	CYCLE	TOTAL SOLIDS (g)	Total Copper %	Total Copper grams	Oxide Copper %	Oxide Copper grams	Iron %	Iron grams
Feb 16	78	377896	0.218	823.81	0.155	585.74	3.16	11941.51

CALCULATED HEAD GRADE: 1.46% Cu 3.99% Fe

CALCULATED TOTAL METAL: 5498.99 g Cu 15090.6 g Fe

SIZE DISTRIBUTION

SAMPLE NO. M90-001

Williams Creek Column Tails #1

Size Fraction (mesh)		Individual Percentage Retained %	Cumulative Percentage Passing %
	+ 1"	1.3	98.7
- 1"	+ 3/4"	20.3	78.4
- 3/4"	+ 1/2"	24.8	53.6
- 1/2"	+ 3/8"	9.7	43.9
- 3/8"	+ 3 mesh	6.5	37.4
- 3	+ 4 mesh	4.9	32.5
- 4	+ 6	3.2	29.3
- 6	+ 8	3.2	26.1
- 8	+ 10	2.6	23.5
- 10	+ 14	2.2	21.3
- 14	+ 20	2.4	18.9
- 20	+ 28	1.4	17.5
- 28	+ 35	4.0	13.5
- 35	+ 48	2.3	11.2
- 48	+ 65	2.0	9.2
- 65	+ 100	1.9	7.3
- 100	+ 150	1.5	5.8
- 150	+ 200	1.2	4.6
- 200	+ 325	1.2	3.4
- 325		3.4	

M90-001: WILLIAMS CREEK COPPER COLUMN #2

DATE	LEACH CYCLE	LITRES IN	LITRES OUT	FLOW (ml/min) IN (Aim)	FLOW (ml/min) OUT (Actual)	H2SO4 (g/l) IN	H2SO4 (g/l) OUT	CONSUMPTION H2SO4 GRAMS IND.	CONSUMPTION H2SO4 GRAMS CUM.	CONSUMPTION H2SO4 Kg/tonne (TOTAL)	CONSUMPTION H2SO4 Kg/tonne (As Cu)	PREGNANT SOLUTION Fe (g/l)	PREGNANT SOLUTION COPPER (g/l)	PREGNANT SOLUTION COPPER (g total)	CALCULATED % COPPER EXTRACTED IND.	CALCULATED % COPPER EXTRACTED CUM.
Mar 9	1	137.77	105.87	100	73.5	19.72	0.38	2887.1	2887.1	7.72	2.72	0.11	6.24	660.65	11.2%	11.2%
Mar 10	2	134.14	133.36	100	92.6	19.72	0.65	2558.5	5445.6	14.56	6.85	0.43	7.52	1002.90	17.0%	28.1%
11	3	149.27	156.85	100	108.9	19.72	1.21	2753.9	8199.5	21.93	10.81	0.97	6.12	959.93	16.2%	44.4%
12	4	133.32	125.20	100	86.9	19.72	3.41	2202.2	10401.7	27.82	12.89	1.58	4.04	505.80	8.6%	52.9%
13	5	136.23	144.16	100	100.1	19.72	5.05	1958.3	12360.1	33.06	14.60	2.00	2.88	415.18	7.0%	59.9%
14	6	143.94	143.80	100	99.9	14.79	4.18	1527.7	13887.8	37.14	15.78	1.80	2.00	287.59	4.9%	64.8%
15	7	131.43	131.01	100	91.0	14.79	4.05	1413.2	15301.0	40.92	16.64	1.60	1.58	206.99	3.5%	68.3%
16	8	130.28	130.96	100	90.9	14.79	4.57	1328.3	16629.4	44.48	17.33	1.54	1.28	167.63	2.8%	71.1%
17	9	128.15	130.01	100	90.3	14.79	4.91	1257.0	17886.4	47.84	18.13	1.13	1.50	195.01	3.3%	74.4%
19	11	141.49	139.99	50	48.6	14.79	2.84	1695.0	19581.4	52.37	18.88	1.80	1.30	181.98	3.1%	77.5%
21	13	141.94	143.07	50	49.7	14.79	3.11	1654.3	21235.7	56.80	19.47	0.48	1.00	143.07	2.4%	79.9%
23	15	142.25	141.40	50	49.1	9.86	2.16	1097.2	22332.9	59.73	19.92	1.50	0.78	110.29	1.9%	81.8%
26	18	113.00	114.27	30	26.5	9.86	1.11	987.3	23320.3	62.37	20.25	1.30	0.70	79.99	1.4%	83.1%
29	21	122.36	122.36	30	28.3	9.86	0.87	1100.1	24420.3	65.31	20.55	1.30	0.60	73.42	1.2%	84.4%
Apr 3	26	94.64	100.64	20	14.0	9.86	1.03	829.5	25249.8	67.53	20.82	1.38	0.64	64.41	1.1%	85.5%
Apr 6	29	71.27	65.09	20	15.1	9.86	0.47	672.2	25921.9	69.33	20.98	1.12	0.59	38.40	0.6%	86.1%
Wash 1	32	144.55	141.36	100	98.2	0.00	0.12	-17.0	25905.0	69.28	21.10	0.56	0.21	29.69	0.5%	86.6%
Wash 1	33	126.36	140.27	100	97.4	0.00	0.47	-65.9	25839.0	69.11	21.13	0.08	0.04	6.17	0.1%	86.7%

5129.11

COLUMN TAILS:

DATE	CYCLE	TOTAL SOLIDS (g)	Total Copper %	Total Copper grams	Oxide Copper %	Oxide Copper grams	Iron %	Iron grams
Apr 12	33	373897	0.21	785.18	0.16	598.24	2.84	10618.67

CALCULATED HEAD GRADE: 1.58% Cu 3.55% Fe

CALCULATED TOTAL METAL: 5914.30 g Cu 13284.58 g Fe

SIZE DISTRIBUTION

SAMPLE NO: M90-001 Cu Column #2 Tails

Size Fraction		Individual Percentage Retained	Cumulative Percentage Passing
	+3/4"	11.3	88.7
-3/4"	+1/2"	19.6	69.1
-1/2"	+3/8"	7.0	62.1
-3/8"	+ 3 mesh	5.1	57.0
- 3	+ 4 mesh	4.3	52.7
- 4	+ 6 mesh	4.5	48.2
- 6	+ 8 mesh	3.9	44.3
- 8	+ 10 mesh	4.3	40.0
- 10	+ 14 mesh	3.8	36.2
- 14	+ 20 mesh	4.5	31.7
- 20	+ 28 mesh	4.8	26.9
- 28	+ 35 mesh	5.3	21.5
- 35	+ 48 mesh	4.5	17.0
- 48	+ 65 mesh	3.7	13.3
- 65	+100 mesh	4.2	9.0
-100	+150 mesh	2.3	6.8
-150	+200 mesh	2.2	4.6
-200		4.6	

A PRELIMINARY WATER QUALITY INVESTIGATION OF WILLIAMS CREEK 1989

Prepared for

ARCHER CATHRO & ASSOCIATES (1981) LTD.

Prepared by

B.E. BURNS & J.E. GIBSON

January 1990

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1.0 INTRODUCTION

B.E. Burns and J.E. Gibson were contracted by Archer Cathro & Associates (1981) Ltd. to conduct a survey of the streams and creeks on the Williams Creek property, in October 1989.

The purpose of the survey was to obtain preliminary baseline water quality of the waters in the area of a copper/gold deposit which is expected to be developed as a mine.

2.0 STUDY AREA

Williams Creek is a north easterly flowing tributary to the Yukon River with its mouth located at 62°24' and 136°36', approximately 35 km upstream of the Village of Carmacks (Figure 1).

Williams Creek consists of a mainstem - right fork and a tributary - left fork (Nancy Lee Creek) flowing from the north west. The mainstem and tributary join 1.2 km above the creek mouth. The mineral target and therefore this water quality baseline study area, are located on the upper reach of the mainstem.

2.1 Area Description

Observations of the Williams Creek area include:

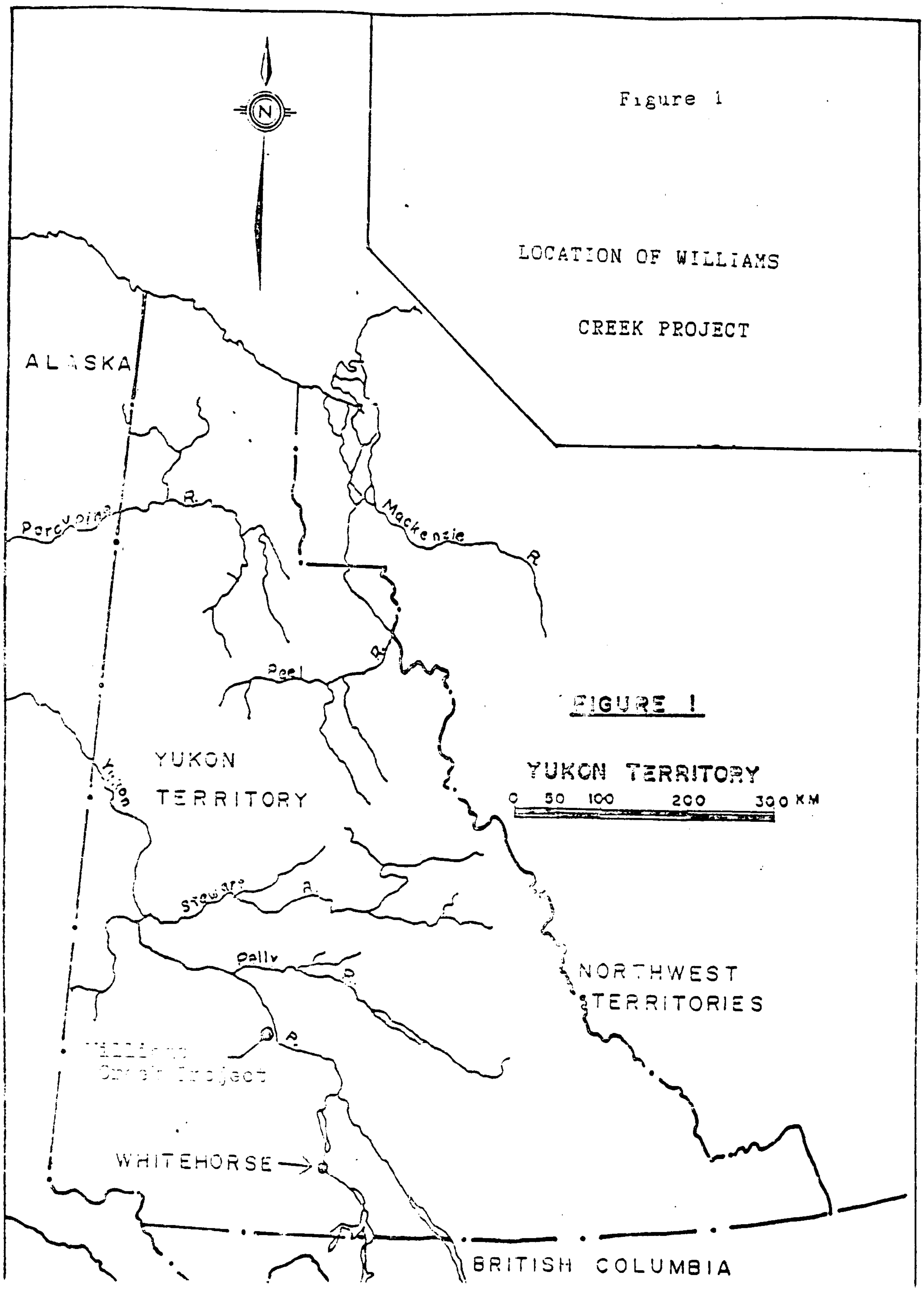


Figure 1
LOCATION OF WILLIAMS
CREEK PROJECT
ALASKA
Porcupine R.
Mackenzie R.
Peel R.
FIGURE 1
YUKON TERRITORY
0 50 100 200 300 KM
YUKON
TERRITORY
Yukon
Stewart R.
Pelly R.
NORTHWEST
TERRITORIES
WHITEHORSE
BRITISH COLUMBIA

Area topography is well rounded hill of less than 1070m height.

Mainstem headwaters are at approximately 830m elevation.

Creek confluence with the Yukon River is at 490m elevation

Channel lengths are:

mainstem - 11.4 km

mainstem above Station W4 - 6.1 km

Nancy Lee trib - 12.5 km

Channel slopes were calculated at:

mainstem - 2.9%

mainstem above Station W4 - 3.0%

Watershed areas were calculated at:

mainstem - 43.9 km2

mainstem above Station W4 - 29.4km2

Nancy Lee trib - 44.6 km2

Total area - 88.5 km2

Area vegetation is typical of the Freegold area. South facing slopes tend to be steeper, better drained and less eroded by water than the north facing slopes. South facing slopes are vegetated by aspen, poplar and pine with marginal undergrowth of grasses and sedges.

North facing slopes which have poorer drainage, lower slope, more snowpack accumulation and more melt water runoff, are vegetated by spruce,alder and willow with thick undergrowth of blueberry, buffaloberry, Laborador tea and moss. The moss presumably overlies zones of discontinous permafrost.

The mainstem channel of Williams Creek above 600m elevation is generally vegetated by species common to the north facing slopes. The valley bottom is also thick with large alder growth and firekill deadfall.

The stream channel in the study area is characterized by a straight channel with few meanders or side channels. The channel is consistently in mid-valley with heavy alder and willow growth along the bank crests.

Channel banks are lined by moss and undergrowth vegetation resulting in the assumption that the channel is not subject to high freshet flows in spring and early summer.

Channel dimensions are generally consistent at:

Width - 1m at the base

- 2m at the crest

Depth - 1 to 1.5m

Side slopes - 1:1 to 2:1 (horizontal : vertical)

There was little evidence of the study area being used by moose; no tracks were found and vegetation was not browsed. Tracks of rabbit, squirrel and wolverine were noted.

2.2 Sample Stations

The mineral target is located on a south facing slope adjacent to upper mainstem Williams Creek. Work thus far on the property consists of trenches on the south facing and east facing slopes of the hill. Water samples were taken from mainstem Williams Creek and all tributaries with surface flow entering the mainstem. Tributary channels with no surface flows were recorded as "dry". A total of 12 stations were established, four which were dry channels (Figure 2). Station locations and descriptions are:

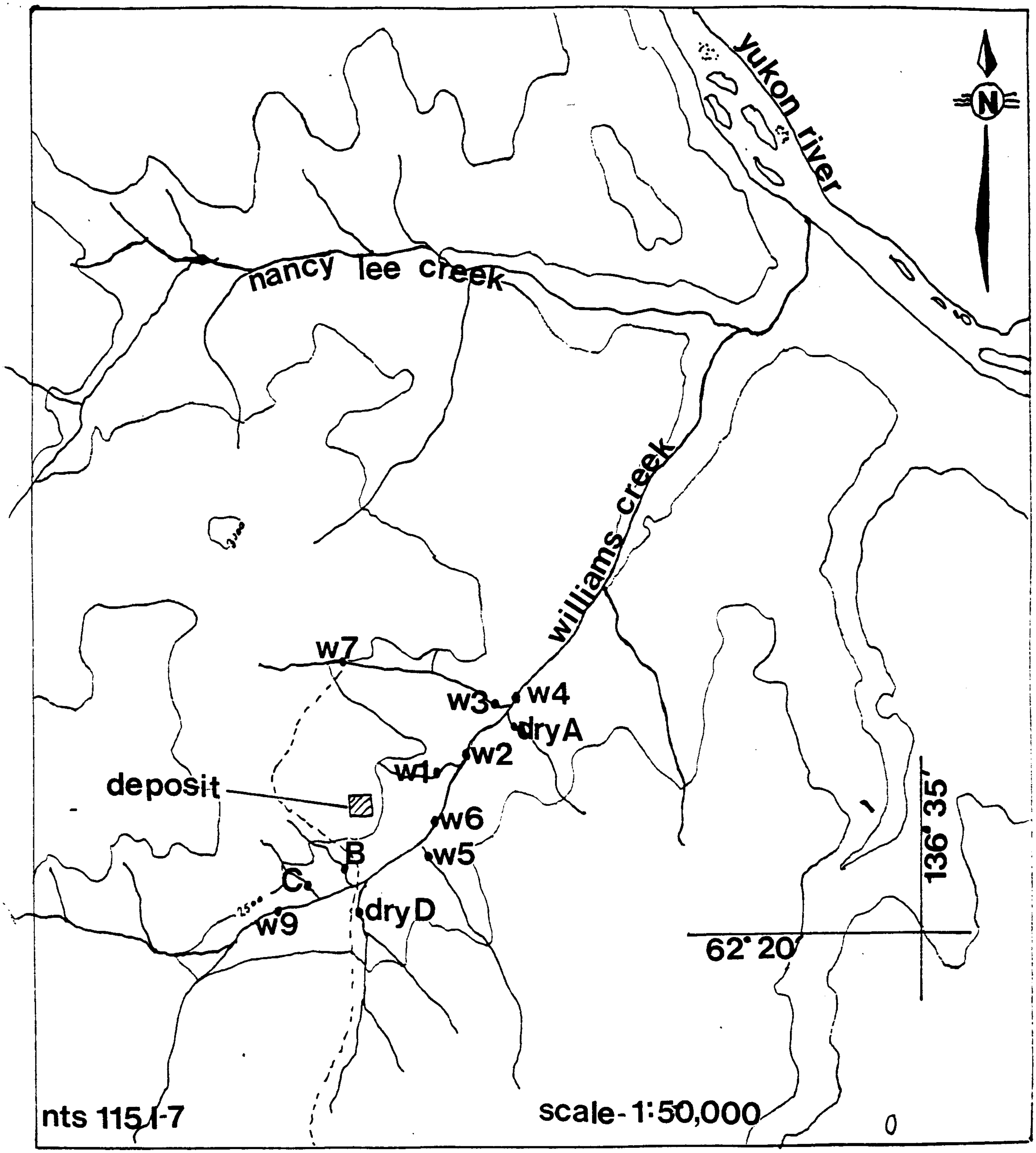
W-1 East flowing tributary to Williams Creek. Drains the south east hillside containing the mineral target. Sample site is located approximately 50m downstream of the slope toe.

W-2 Williams Creek 20m downstream of the confluence with W-1.

W-3 East flowing tributary to Williams Creek. Drains the north facing hillside containing the mineral target. Station W-7 is located 2 km upstream on this tributary.

W-4 Williams Creek 100m downstream of the confluence with W-3.

Figure 2. LOCATION of SAMPLE STATIONS



This station is located the farthest downstream.

Dry A Dry channel on the north facing slope immediately across from W-3.

W-5 Northwest flowing tributary to Williams Creek.

W-6 Williams Creek 50m downstream of the confluence with W-5.

W-7 East flowing tributary draining the north slope of mineral target. See notes on W-3.

Dry B South facing channel on the west side of mineral target.

Dry C South facing channel upstream of Dry B channel.

W-9 Williams Creek approximately 500m upstream of Dry D channel. This is the upstream station for background quality.

Dry D North facing channel immediately across from the mineral target.

Note: There is no Station W-8.

Each sample location with surface flow was marked by a blazed tree and a numbered white plywood marker (ie W-1) nailed to the blazed tree.

All stations with surface flow were under ice cover. Only Station W-1 and W-9 did not have anchor ice in the channel. Due to surface and anchor ice, flow measurements were not taken but were estimated. Flows ranged from 2 l/s at W-9 to 0.1 l/s at W-7. Flow at W-3 was intermittent and was an oxidized reddish brown in colour. Station W-3 also had algae and moss growth in the channel.

3.0 METHODS

Water quality samples were collected at each station using acid washed Nalgen sample bottles on October 13 and 14, 1989. Samples to be analyzed for ammonia, nitrite, phosphate and DNX anion were collected in a 2 litre bottle. A 1 litre sample was taken for suspended solids and laboratory pH, conductivity, and alkalinity. Samples to be analyzed for ICAP metals were collected in a 100 ml bottle and preserved with 0.5 ml of concentrated HNO_3. All samples were sent to Vancouver and analyzed by Quanta Trace.

Prior to sampling, ice cover was removed using an axe. As this often disturbed bed sediments, flows were allowed to clear before the sample was taken. Each sample bottle was rinsed three times prior to filling. Filled bottles were then capped and labelled with station number, location, date and analysis required. This data, along with estimated stream flow, ice conditions, insitu measurements and general observations were recorded in a field note book. Photographs of the site were also taken and are presented in Figures 3 to 8.

4.0 RESULTS

4.1 Field Measurements

Due to the -18 degree celsius air temperature on October 14, 1989, the pH and conductivity meters would not work for Stations W 1,2,3,4,5,6 and 9. Station W7 which was sampled on October 13 had a pH of 7.4, a conductivity of 183 umhos/cm and water temperature of 2°C. Water temperature at the seven other stations was 0°C or less on October 14. Flow measurements were estimated and are presented in Table 1.

4.2 Chemical Analysis

Water quality results are presented in Table 2 and no anomalies were apparent.

4.2.1 Alkalinity

Alkalinity is a measure of water's ability to neutralize acid. In aquatic systems, alkalinity usually results from the geology

TABLE 1

FLOW ESTIMATES

Station	Flow (litres/second)	Comments
W-1	1	surface ice only
W-2	1	surface and anchor ice
W-3	0.5	surface and anchor ice, intermittent flow
W-4	2	surface and anchor ice
W-5	1	surface ice only
W-6	2	surface ice only
W-7	0.1	surface ice, sulphur smell
W-9	2	surface ice only

TABLE 2

WATER QUALITY DATA

Sample Site	W1	W2	W3	W4	W5	W6	W7	W9
Date								
pH	7.7	7.7	7.5	7.7	7.5	7.9	7.7	7.8
Conductivity uS/cm	462	431	380	395	157	415	325	505
Suspended Solids mg/l	20	9	<5	<5	<5	<5	<5	<5
ICAP METALS mg/l								
Aluminum	0.03	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
Antimony	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005
Arsenic	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
Barium	0.146	0.035	0.037	0.031	0.013	0.034	0.037	0.043
Beryllium	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001
Boron	0.004	<0.001	<0.001	<0.001	<0.001	0.002	0.007	0.004
Cadmium	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002
Calcuim	55.3	44.2	52.4	40.5	18.8	43.8	38.2	44.2
Chromium	0.0066	<0.0002	0.0024	<0.0002	0.0016	0.0012	0.0081	0.0005
Cobalt	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005
Copper	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	0.0010	0.0090	<0.0005
Iron	0.181	0.371	0.84	0.519	0.458	0.687	0.195	0.199
Lead	<0.002	<0.002	<0.002	<0.002	<0.002	<0.002	0.003	<0.002
Lithium	0.26	0.30	0.34	0.35	0.35	0.30	0.36	0.29
Magnesium	11.2	12.8	8.89	10.7	4.16	13.9	6.84	18.2
Manganese	<0.001	0.068	0.570	0.077	0.046	0.101	0.026	0.015
Mercury	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005
Molybdenum	0.015	0.003	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
Nickel	0.0009	0.0015	0.0009	0.0014	0.0029	0.0018	0.0020	0.0012
Phosphorus	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
Potassium	1.1	0.9	0.8	0.8	<0.2	0.9	0.3	1.3
Selenuim	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005
Silicon	4.42	3.99	4.33	4.86	5.53	5.08	4.69	4.92
Silver	<0.002	<0.002	<0.002	<0.002	<0.002	<0.002	<0.002	<0.002
Sodium	8.37	12.8	7.32	11.2	5.57	14.3	7.21	19.1
Strontium	0.644	0.444	0.481	0.372	0.089	0.426	0.161	0.583
Thorium	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Titanium	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
Uranium	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
Vanadium	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002
Zinc	0.0476	0.0330	0.0523	0.0578	0.0661	0.0109	0.0185	0.0064
ANIONS mg/l								
Chloride	4.70	3.50	2.80	3.60	1.01	1.30	2.60	2.00
Fluoride	3.6	1.1	<1	<1	<1	<1	<1	<1
Nitrate	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Sulfate	129	78.0	21.0	47.0	3.29	51.0	15.1	54.0
Nitrite	<0.003	<0.003	<0.003	<0.003	<0.003	0.003	<0.003	<0.003
O-Phosphate	0.014	0.014	0.017	0.022	0.007	0.017	0.008	0.008
Ammonia	0.10	0.06	0.06	<0.05	<0.05	0.05	0.05	0.03
Alkalinity as CaCO3	53	100	180	120	68	140	120	170
Hardness as CaCO3	184.2	153.1	167.4	145.1	64	163.6	123.6	185.3

and geochemistry of the rocks and soils in the area. Alkalinity varies widely in Canadian surface waters and usually results from the presence of the bicarbonate ion. Alkalinity varied from a low of 58 mg/l as $CaCO_3$ at W-1 to a high of 170mg/l as $CaCO_3$ at W-9. Generally speaking, the best waters for aquatic life are those with a pH value between 7 and 8 and with a total alkalinity of 100 mg/l as $CaCO_3$ or more (Environment Canada, 1976). The alkalinity serves as a buffer preventing sudden pH change. All stations sampled fall in this catagory except for W-1 and W-5 with alkalinities of 58 mg/l and 68 mg/l as $CaCO_3$ respectively. Higher alkalinity values are usually associated with low discharge so during higher flows, the alkalinity concentrations could be less.

4.2.2 pH

The pH of Williams Creek was slightly alkaline with values ranging from 7.5 to 7.9.

4.2.3 Conductivity

Conductivity was relatively high at all stations. The lowest value of 157 was observed at W-5. The other stations ranged from 325 at W-7 to 505 at W-9. Conductivity increases in surface water when groundwater infiltrates the flow.

4.2.4 Suspended Solids

The suspended solids levels were very low which is typical of the time of year sampled. Staions W-3 through to W-9 were below the detection limit of 5 mg/l. W-1 had the highest value of 26 mg/l and W-2 had 8 mg/l.

4.2.5 Aluminum

Aluminum levels were below the detection limit of 0.02 mg/l except at W-1 which had a concentration of 0.03 mg/l. The parameters of antimony, arsenic, beryllium, cadmium, cobalt, mercury, phosporus, selenium, silver, thorium, titanium, uranium and vanadium were all below their respective detection limits.

4.2.6 Barium

Although barium is a common element in the earth's crust only trace levels are normally found in the aquatic system.[1] Concentrations of 1 mg/l are very rarely found. Barium ranged from a low of 0.013mg/l at W-5 to a high of 0.146 mg/l at W-1.

4.2.7 Boron

[1] Unless otherwise stated, all information on the parameters in this section is from Canadian Water Quality Guidelines March, 1987.

Boron concentrations in surface waters average 0.1 mg/l. Concentrations at all the sites sampled were very low; W-2, W-3, W-4 and W-5 were below the detection limit of 0.001 mg/l and the remaining stations ranged from 0.002 mg/l at W-6 to 0.007 mg/l at W-7.

4.2.8 Calcium

Calcium is one of the most abundant cations in surface waters and groundwaters. It is readily soluble in water and enters the environment through the weathering of rocks and from the soils through seepage and runoff. Therefore, concentrations of calcium in natural fresh waters vary according to the proximity of calcuim-rich geological formations. The calcium levels ranged from a low of 18.8 mg/l at W-5 to a high of 55.3 mg/l at W-1 indicating a presence of carbonate rocks in the area.

4.2.9 Chromium

Chromium enters aquatic systems through weathering and is generally present at low concentrations in Canadian surface waters. Levels were low at all stations sampled and ranged from below the detection limit of 0.0002 mg/l at W-2 and W-4 to a high of 0.0081 at W-7.

4.2.10 Copper

Copper concentrations were below the detection limit of 0.0005 mg/l at all stations except for W-6 and W-7 which had levels of 0.0010 mg/l and 0.0090 mg/l respectively.

4.2.11 Iron

Iron is the fourth most abundant element in the earth's crust. It is released naturally into the environment from the weathering of sulphide ores (pyrite, FeS_2). Iron concentrations ranged from a low of 0.18 mg/l at W-1 to a high of 0.84 mg/l at W-3.

4.2.12 Lead

Lead concentrations were below the detection limit of 0.002 mg/l at all stations sampled except W-7, which had a level of 0.003 mg/l.

4.2.13 Lithium

Lithium compounds are extremely soluble and tend to remain in solution. It is easily leached from rocks and sediments and ususally occurs with sodium in aqueous solutions. Environmental concentration ranges for lithium in Canadian surface waters in the Pacific region were <0.0005 mg/l to 0.0069 mg/l (NAQUADAT, 1985). Lithium ranges for Williams Creek were 0.26 mg/l at W-1 to 0.36 mg/l at W-5 and W-7. These levels are relatively high and could indicate influence from ground waters.

4.2.14 Magnesium

Magnesium is a common constituent of natural water and the concentrations found in Williams Creek are typical of Canadian surface waters (NAQUADAT, 1985). Magnesium ranged from a low of 4.16 mg/l at W-5 to a high of 18.2 mg/l at W-9.

4.2.15 Manganese

Environmental concentrations of manganese found in natural surface waters are ususally present in quantities of 0.2 mg/l or less. Concentrations ranged from below the detection limit of 0.001 mg/l at W-1 to 0.570 mg/l at W-3. The high concentration at W-3 could be indicative of some ground water influence.

4.2.16 Molybdenum

The weathering of igneous and sedimentary rocks (especially shales) constitutes an important natural source of molybdenum to the aquatic environment. Molybdenum was below the detection limit of 0.001 mg/l at all stations except at W-1 and W-2 which had concentrations of 0.015 mg/l and 0.003 mg/l respectively.

4.2.17 Nickel

Environmental concentration ranges for nickel in Canadian surface waters for the Pacific region were <0.001 mg/l to 0.003 mg/l (NAQUADAT, 1985). Nickel concentrations at Williams Creek ranged from a low of 0.009 mg/l at W-1 and W-3 to a high of 0.0029 mg/l at W-5.

4.2.18 Potassium

Potassium levels were fairly low ranging from 0.2 mg/l at W-5 to 1.3 mg/l at W-9. Potassium concentrations range from <0.1 mg/l to 9.3 mg/l in Canadian surface waters (NAQUADAT, 1985).

4.2.19 Silicon

Silicon is the second most abundant element in the earth's crust. The major sources of silica in water are the feldspars, micas and clay minerals. In the Pacific region silicon concentrations ranges from 0.3 mg/l to 25.4 mg/l in Canadian surface waters. In Williams Creek, silicon ranged from 3.99 mg/l at W-2 to 5.53 mg/l at W-5.

4.2.20 Sodium

All natural waters contain some sodium and the concentration varies considerably depending on the source of the sodium (ie weathering of salt deposits). Concentrations ranged from 5.57

mg/l at W-5 to 19.1 at W-9.

4.2.21 Strontium

Naturally occurring strontium is not radioactive. It is fairly soluble and the levels encountered in Williams Creek are not unusual. It ranged from a low of 0.089 at W-5 to a high of 0.644 mg/l at W-1.

4.2.22 Zinc

Zinc is usually present in trace amounts (<0.05 mg/l), in both ground and surface waters. Zinc ranged from a high of 0.0661 mg/l at W-5 to a low of 0.0064 mg/l at W-9.

4.2.23 Chloride

Natural sources of chloride to aquatic systems are through the leaching of sedimentary rocks and soils. In natural surface waters, chlorides are present in low concentrations, usually less than 10 mg/l (NAQUADAT, 1976). The waters sampled at Williams Creek fell within these limits ranging from 1.01 mg/l at W-5 to 4.70 mg/l at W-1.

4.2.24 Fluoride

Fluoride is present in trace amounts in soils and rocks but is most prevalent in volcanic regions. The concentration in most surface waters is less than 1 mg/l. Groundwater usually contains higher concentrations, often as high as 10 mg/l. Station W-1 had the highest level of 3.8 mg/l. Stations W-2 had a level of 1.1 mg/l and the remaining stations were below the detection limit of 1.0 mg/l.

4.2.25 Nitrate

Surface waters contain at least trace levels of nitrates usually less than 1 mg/l. All stations sampled were below the detection limit of 0.1 mg/l.

4.2.26 Nitrite

Nitrites are normally absent or present in minute quantities in surface waters. All the stations sampled were below or at the detection limit of 0.003 mg/l.

4.2.27 Sulphate

Sulphates may be leached from most sedimentary rocks. Elevated concentrations can be found in waters in areas of acid mine drainage. The levels encountered in Williams Creek are ususal. Higher concentrations were found in the creeks draining the area of the deposit. This reflects the fact that the deposit is a sulphide bearing ore.

4.2.28 Ammonia

Natural waters typically contain concentrations of ammonia below 0.1 mg/l. Station W-1 had the highest concentration of 0.10 mg/l. The lowest concentrations of less than 0.05 mg/l were observed at W-4 and W-5.

4.2.29 Hardness

Hardness is an important modifying factor in water quality as it can significantly influence the form and toxicity of numerous heavy metals (Alabaster and Lloyd, 1982). All of the stations sampled exhibited "hard" water (a hardness of 121 to 180 mg/l as $CaCO_3$) except for W-5 which had a hardness classification of moderately soft (61 to 120 mg/l as $CaCO_3$).

5.0 SUMMARY

There were no anomalies apparent in the water quality data.

There were virtually no heavy metals present as concentrations were either very low or below the detection limits.

The relatively high concentrations of conductivity and some of the alkali earth metals could possibly indicate some groundwater influence at some of the sampling sites.

The drilling test results show that the deposit is oxidized to 50 meters. Underlying this are unoxidized sulphides. The developing of this area has the potential to create acid mine drainage. The alkalintiy of the stream which drains the deposit, W-1, is low (58 mg/l as $CaCO_3$) and therefore may not adjust well should acid mine drainage occur in this area. However, the alkalinities of the receiving water,(mainstem Williams Creek W-2, W-4,and W-6) are high and indicate a good buffering capacity. The pH values of all the stations sampled were slightly alkaline which can also serve to assist neutralization of an acidic discharge.

Water is considered to be soft if it has a hardness of less than 120 mg/l as $CaCO_3$. Only one station, W-5, fitted this catagory, which drains the hillside opposite the deposit. The rest of the stations have hard water. Hardness exhibits a significant effect on the toxicity of heavy metals to aquatic life, expecially fish. Toxicity is greatly increased in soft water, therefore higher concentrations of heavy metals can be tolerated by fish in hard water.

6.0 RECOMMENDATIONS

This survey represents water quality at low flow conditions. For a more comprehensive baseline study, water quality should also be undertaken during high flows as concentrations of several parameters will increase and the background concentrations will be more accurately represented.

A survey of the benthic populations should be conducted prior to development of the property to determine the species etc. present. Changes to the population structure can then be monitored during and after mining to assess any influence mining discharge may have on the benthic community.

The resident fish populations should also be determined to ascertain which species if any, utilize this area for rearing, spawning etc.

REFERENCES

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April 22,1991

Al Archer
Archer, Cathro & Associates (1981) Limited
1016 - 510 West Hastings Street
Vancouver, B.C.
V6B 1L8

Dear Al:

**Re: 1991/92 Budget Proposal for Water Quality Survey
Williams Creek - Consolidated Silver Standard Mines Ltd.**

I enclose a water quality survey budget for the Williams Creek property for consideration by you and Consolidated Silver Standard. The budget is based on four sample events:

May 1991 - pre-freshet flow
July 1991 - summer high flow
September 1991 - summer low flow
March 1992 - base flow conditions

As it turns out, the attempted survey in late March,1991 was scheduled too late to sample base flow conditions and has been rescheduled for early March 1992.

The lab analysis costs have been adjusted to include dissolved metals in addition to extractable metals. The number of sample stations has also been changed from the eight station in October, 1989 to ten stations in May, 1991. The two additional stations are Nancy Lee Creek upstream of its confluence with Williams Creek and Williams Creek at the mouth (see attached map). These stations will be important in assessing potential impacts of development. A water quality station on the Yukon River upstream of Williams Creek is also being considered as a receiving water station. This station may not be necessary if the NAQUADAT Yukon River at Carmacks station data is pertinent and transferable - this is being looked at now.

Each water quality station will be sampled and analyzed for:

Ions - ammonia, sulfate, nitrate, nitrite and total phosphorus
Routine - alkalinity, turbidity, specific conductance, total solids, suspended solids and hardness
Extractable metals - 31 element ICP to ppb levels
Dissolved metals - 31 element ICP to ppb levels

Field measurements for pH, conductivity and water temperature will also be done at each station. Streamflow volume will be measured with a Price velocity meter.
All laboratory analysis will be done by Quanta Trace Laboratories of Burnaby B.C. Quanta Trace follows EPA protocols with 20% duplicates and SRM's.

The budget breakdown for each of the proposed sample events is on the following page.

RECEIVED MAY 1 1991

Water Quality Survey Budget - Williams Creek

1991 - 1992 (period ending March 31,1992)

May 1991

Lab analysis (10 stations)		$	2580.00
Wages - J.Gibson 25 hrs @ $35/hr			875.00
Transportation - 500 km @ $0.37/km			185.00
Equipment Rental (ATV)			200.00
Meals / Accomadations @ $60/day			120.00
Water Quality Report			800.00
Misc. (phone, maps, film, postal)			75.00
	Total	$	4835.00

July 1991

Lab analysis (10 stations)		$	2580.00
Wages - J.Gibson 25 hrs @ $35/hr			875.00
Transportation - 500 km @ $0.37/km			185.00
Equipment Rental (ATV)			200.00
Meals / Accomadations @ $60/day			120.00
Water Quality Report			800.00
Misc.			75.00
	Total	$	4835.00

September 1991

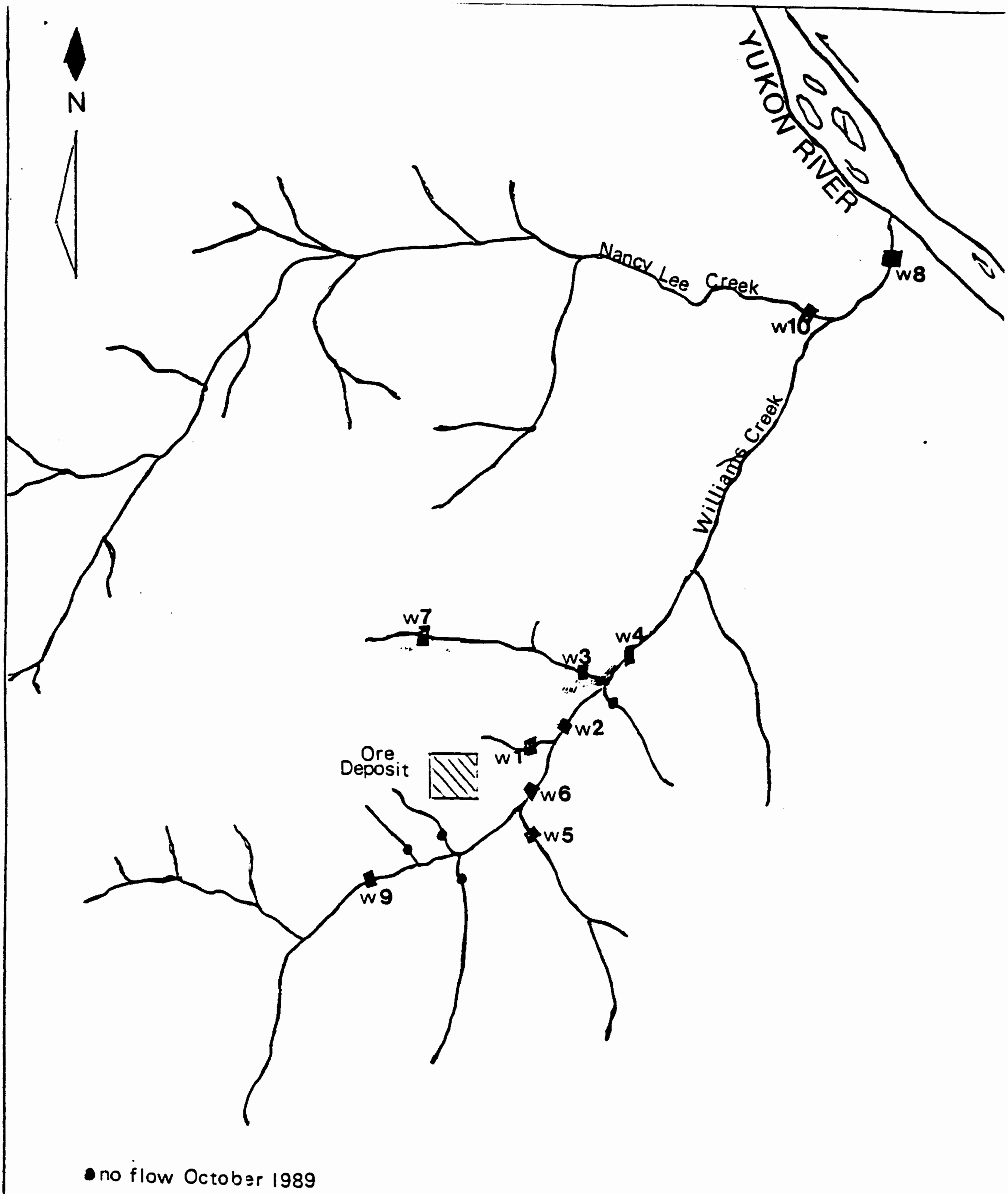
Lab analysis (10 stations)		$	2580.00
Wages - J.Gibson 25 hrs @ $35/hr			875.00
Transportation - 500 km @ $0.37/km			185.00
Equipment Rental (ATV)			200.00
Meals / Accomadations @ $60/day			120.00
Water Quality Report			800.00
Misc.			75.00
	Total	$	4835.00

March 1992

Lab analysis (10 stations)		$	2580.00
Wages - J.Gibson 30 hrs @ $35/hr			1050.00
- Assistant 25 hrs @ $25/hr			500.00
Transportation - 500 km @ $0.37/km			185.00
Equipment Rental (2 skidoos)			200.00
Meals / Accomadations @ $60/day			240.00
Water Quality Report			800.00
Misc.			75.00
	Total	$	5630.00

	Total 1991/92	$	20,135.00

GST not included in the above prices



WILLIAMS CREEK

WATER QUALITY STATIONS 1991

1:50 000

I have also drawn up a tentative schedule for initial site work on hydrology, benthic invertebrates and fisheries . The Environmental Assessment and Review Process (EARP) for the Yukon Territory will likely require an Initial Environmental Evaluation (IEE) be done as part of a Level 11 screening. The IEE requires a minimum of two years physical and biological data be included in the evaluation.

The site work is scheduled to tie in with water quality surveys to reduce costs. Contact with various government agencies is underway to establish acceptable procedures and methodology.

The tentative schedule has work in all three areas to begin in July, 1991. If Consolidated Silver Standard wishes to begin all or part of the initial site work, please notify me and I will prepare a budget.

July, 1991

Hydrology

Purpose - to establish a stage discharge relationship for Williams Creek in the project area.

- install a staff gauge at a water quality station
- initial gauge readings and flow measurements
- if staff are onsite, train them in gauge readings and flow measurements (will require purchase of velocity meter to remain onsite)

Benthics

Purpose - to establish baseline information on benthic species presence and diversity.

- identify benthic sampling sites (upper Williams Creek and Williams Creek at the mouth)
- install 6 week benthic traps at selected sites.

Fisheries

Purpose - to establish baseline quantitative and qualitative data on fish populations and habitat in the area.

- literature search to document and compile existing fisheries information on Williams Creek and adjacent Yukon River.
- identify and document spawning, rearing and over wintering capabilities in the area.
- survey for Chinook spawning activity in the Yukon River and in lower Williams Creek. Document utilization.
- sampling to determine fish species presence by angling, beach seining and electroshock.

September, 1991

Hydrology

- continue staff gauge readings and streamflow measurements.
- resurvey (level circuit) to determine changes/ corrections to "gauge zero" datum.

Benthics

- remove benthic traps
- identification of species
- data analysis to determine species diversity

Fisheries

- survey for Chum salmon spawning activity in the Yukon River and lower Williams Creek. Document utilization.
- identify possible over wintering sites on Williams Creek
and adjacent Yukon River for March 1992 sampling.

March, 1992

Fisheries

- sampling of overwintering sites for species presence.

SUMMARY

OPTION/JOINT VENTURE AGREEMENTS

The Williams Creek property (the "Property") is comprised of 68 claims located 25 miles by road northwest of Carmacks in the Whitehorse Mining District, Yukon Territory. The Property encompasses an area totalling 1,350 hectares (2,700 acres). The details of the claims are as follows:

Claim Name	Grant No.	Units	Expiry Date
Boy 20	Y51118	1	09 March 99
Boy 22	Y51120	1	09 March 99
Boy 24	Y51122	1	09 March 99
Boy 51-58	Y51149-Y51156	8	09 March 99
Boy 83	Y51181	1	09 March 99
Boy 85	Y51183	1	09 March 99
Dun 1F-3F	Y59382-Y59384	3	09 March 99
War 22	Y59373	1	09 March 99
AC 2F-3F	Y91722-Y91723	2	09 March 99
W1-W10	YB26708-YB26717	10	04 August 96
W11-W20	YB26718-YB26727	10	04 August 96
W21-W30	YB26728-YB26737	10	04 August 96
W31-W40	YB26738-YB26747	10	04 August 96
W41-W49	YB26748-YB26756	9	04 August 96

By an option agreement dated August 18, 1989, as amended by letter agreements dated September 21, 1990 and January 9, 1991 (collectively called the "Option"), the Issuer was granted an option by Archer, Cathro & Associates (1981) Limited ("Archer") of 1016 - 510 West Hastings Street, Vancouver, B.C., V6B 1L8, to acquire a 100% interest in the Property.

The Issuer acquired from Archer an option to acquire a 90% interest in the Property, free and clear of all charges, claims or encumbrances and an assignable option to acquire the remaining 10% net proceeds royalty interest in the property pursuant to an agreement dated December 8, 1987 between Archer and Armand Arsenault, Businessman, of Whitehorse, Yukon territory (the "Prospector's Agreement"). For as long as the agreement is in effect, Western is to keep the Prospector's Agreement in good standing by making the $5,000 annual payment due under the Prospector's Agreement to Armand Arsenault (due October 1st each year until 2001) in order to maintain the option on the 10% interest in the Property not owned by Archer in good standing. Upon commencement of production on the Property, the unpaid balance of the cumulative total of $75,000 is to be paid, in a lump sum, within sixty days, to Armand Arsenault pursuant to the terms of the Prospector's Agreement. Pursuant to the terms of the Option, the Company has agreed to the following cash payments:

$20,000 Initial payment on execution of agreement (paid)
$25,000 Option payment on or before January 10, 1991 (paid)
$10,000 Option payment on or before March 31, 1991 (paid)

and the following work expenditures:

(a) $50,000 firm commitment on or before March 31, 1990 (completed)
$150,000 in the aggregate on or before December 31, 1990 (completed)
$900,000 in the aggregate on or before December 31, 1991
$2,000,000 in the aggregate on or before December 31, 1992, or

(b) completion of a bankable feasibility study on the Property on or before December 31, 1992.

Delivery of said feasibility study will suspend any further obligations by the Company to complete the remaining aggregate cumulative work expenditures set out above, provided that the Company has incurred at least $1,600,000 in expenditures with respect to exploration and/or development on and for the Property.

Archer will retain either a 2.5% net smelter royalty interest, up to a maximum of $2.5 million, or a 15% net profits interest in the Property, at the election of the Issuer. In addition, Archer will receive advance royalty payments of $100,000 per annum commencing January 1, 1993 provided the average annual price of copper is greater than U.S. $1.10 per pound. If the annual price is less than U.S. $1.10 per pound, no advance royalty will be payable.

Should the Issuer fail to comply with the aggregate cumulative work expenditure requirements described above, the Issuer may satisfy such default by paying directly to Archer one-half of the deficiency thereof within 30 days of receiving notice from Archer of such failure and, upon doing so, the subject aggregate cumulative work expenditure requirement will be deemed to have been incurred for the purposes of the agreement.

Upon completion of payment and work expenditures and/or delivery of a bankable feasibility study on the Property, the Issuer shall have earned a 100% interest in the Property, subject to the royalty interest reserved in the Prospector's Agreement and the royalty interests reserved by Archer pursuant to the terms of the Option agreement.

Pursuant to the terms of the Option agreement, the Issuer has been granted an exclusive and irrevocable right of first refusal in respect of any offer to purchase or otherwise acquire Archer's interest in the Option agreement, the advance royalty payments or Archer's net profits royalty or net smelter royalty interest (collectively or separately, in whole or in part referred to as the "Owner's Interest"). In the event that Archer receives an offer (the "Offer") from a third party to purchase or otherwise acquire some or all of the Owner's Interest, Archer shall provide the Issuer with written notice setting out the terms of the Offer. The Issuer shall have 30 days from receipt of the Offer in which to advise Archer, in writing, of the Issuer's intention to purchase or otherwise acquire the said interest on such terms or their cash equivalent, as set out in the Offer. In the event that the Issuer does not advise Archer, within the 30 day period, Archer shall be entitled to accept the Offer of the third party. If the Offer is not accepted within 60 days of the termination of the

Issuer's 30 day period, then the Issuer's right of first refusal will be reinstated in respect of such Offer and any subsequent offer. Archer may only assign the Owner's Interest subject to the Issuer's right of first refusal and then only upon the assignee specifically agreeing in writing to be bound by the terms of the Option agreement. Western may assign its rights under the Option agreement in whole or in part to any third party.

The Issuer entered into an agreement dated August 18, 1989 with Thermal Explorations Ltd. ("Thermal") of 115-25 Caroline Lane, Nevada City, California, 95959 to jointly explore the Property (the "Thermal Agreement"). After jointly paying the above-noted initial payment and various other costs associated with staking, sampling and metallurgical testing totalling $90,000.00, the terms of the August 18, 1989 agreement provide that Thermal, if it elects to proceed, will have the exclusive right, up to May 1, 1991, to elect to pay the next $640,000 in exploration expenditures on the Property, such funds to be expended by no later than July 31, 1991. If Thermal does not elect to expend the $640,000 or, having elected to do so defaults in funding such $640,000 by July 31, 1991, it will:

a) forfeit all of its right, title and interest in and to the Option agreement and Property to the Issuer,

b) have no further rights or obligations under the Thermal Agreement, save and except where the Issuer delivers written notice to Thermal forthwith after July 31, 1991 stating that it will not keep the Option agreement in good standing, in which case each of the Issuer and Thermal will be jointly and severally liable with respect to the termination obligations provided for in the Option agreement, and

c) in the case of a default by Thermal in paying such $640,000 of Expenditures after having elected to do so pursuant to the terms of the Thermal Agreement, Thermal will owe to the Issuer, as liquidated damages for such default, the deficiency in expenditures not incurred by Thermal, such to be payable by Thermal to the Issuer forthwith upon demand, together with interest thereon at a per annum rate equal to the prime rate plus 2%.

If Thermal elects to expend the funds described above and expends such funds by July 31, 1991, it shall have earned 100% of the Issuer's rights and obligations under the Option agreement, subject to the Issuer's right to earn back a 50% interest.

The Issuer must commit to fund the next $960,000 in exploration expenditures on or before August 31, 1992 in order to reacquire a right to earn a 50% interest in the Property. Thereafter, Thermal and Western will share expenditures 50%/50% with the Issuer as operator.