GEOLOGICAL AND METALLURGICAL STUDIES

on the

GETTY CLAIMS

HIGHLAND VALLEY AREA

KAMLOOPS MINING DIVISION, B.C.

NTS 92 I/10W

LATITUDE 50°34'

LONGITUDE 121°00'

Prepared for

ROBAK INDUSTRIES

2520 Ashurst Avenue

Coquitlam, B.C.

V3K 5T4

by

GOWER, THOMPSON & ASSOCIATES LTD.

#360 - 522 Seventh Street

New Westminster, B. C.

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OCTOBER 24, 1988

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B.Sc., F.G.A.C.

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Bacon, Donaldson & Associates Ltd.

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FIGURE 3 - Getty Claims 1" = 1,320'

FIGURE 3.1 - Acid Leaching of Krain Oxide Ore (Consumption vs Time)

FIGURE 3.2 - Acid Leaching of Krain Oxide Ore Cu Conc. vs Time

FIGURE 4 - Krain Copper Column

SUMMARY

The Getty property is located approximately 650 metres (2,200 feet) east of North Forge Mountain about 10 kilometres (6 miles) north of Valley Copper Mine in the Highland Valley District, Kamloops, M.D., N.T.S. 92 I/10W, Latitude 50°34', Longitude 120°00'. The property consists of 24 two-post claims and one fractional claim. These claims are wholly owned by Robak Industries.

Access to the property is via the South Seas Trojan Road, which leaves the main Highland Valley road at the Old Bethlehem Mine. There are no buildings or equipment on the property.

The claim area is underlain by quartz diorites of the Highland Valley phase (Guichon variety). The mineral zone occurs within a northwest trending fracture system which also hosts the Bethlehem, Trojan and South-Seas deposits.

The mineral deposit consists of a zone of oxidized copper sulphides in the northern half of the mineral system overlying a primary sulphide mineral zone which extends to an additional depth of at least 150 metres (500 feet).

The present program concentrated on procuring two bulk samples across the oxide zone to test the leaching characteristics of the mineralization.

A program of reverse circulation drilling, metallurgical testing and bulldozer trenching is recommended to evaluate the feasibility of mining the oxide and primary deposits.

ESTIMATED COST OF RECOMMENDED PROGRAM

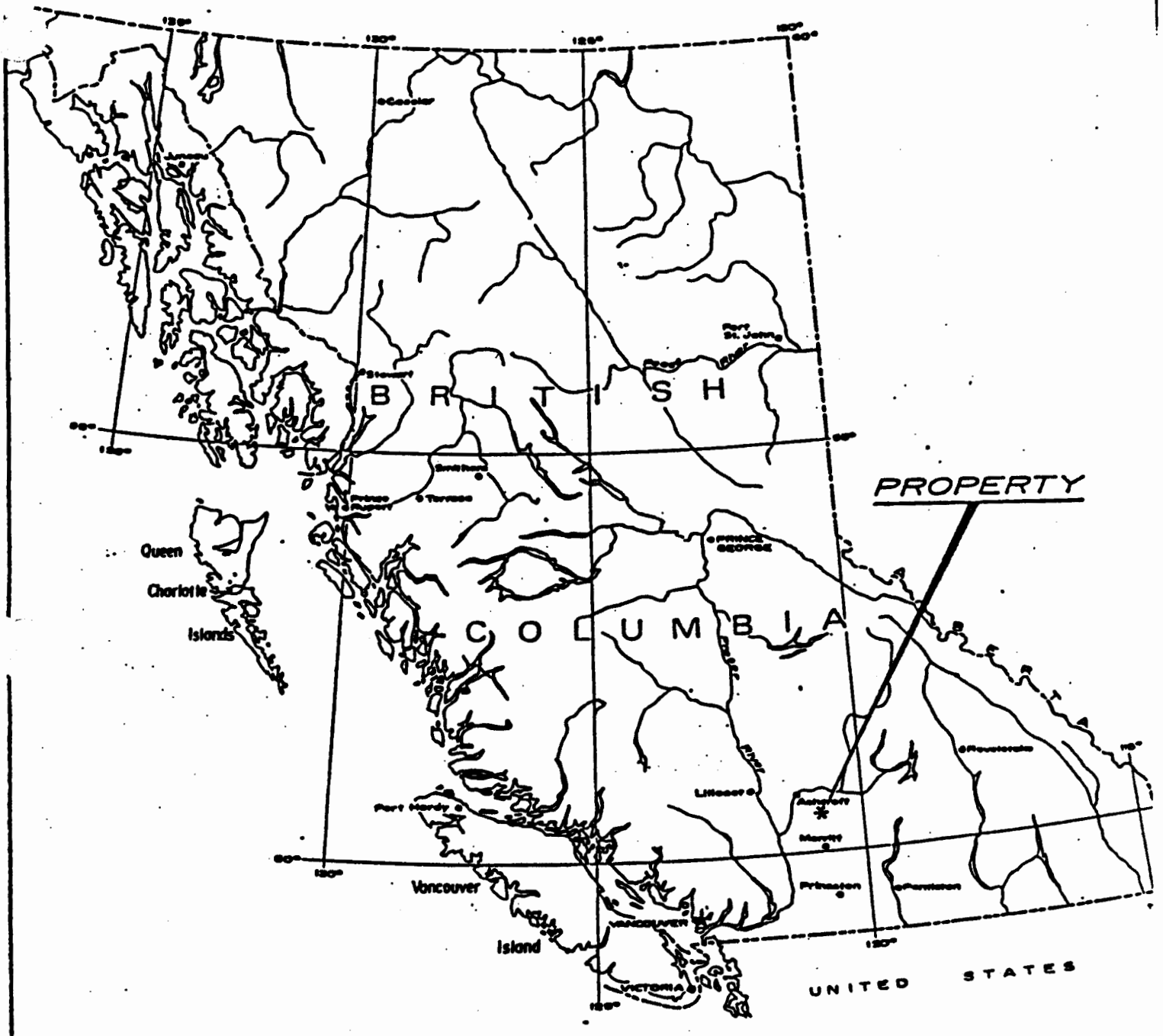
Reverse circulation drilling	
10,000 feet @ $22.00/foot	$ 220,000
Bulk sampling of surface oxide material	25,000
Metallurgical testing	150,000
Feasibility report	200,000
TOTAL ESTIMATED COST:	$ 595,000

STATEMENT OF COSTS

Field Personnel:	
S. C. Gower, Geologist	
June 3, 4, 5, 6 (4 days @ $300)	$ 1,200.00
E. M. Thompson, Blaster	
June 3, 4, 5, 6 (4 days @ $200)	800.00
Food and Accommodations	
(8 person-days @ $50/day)	400.00
Truck Rental	
4 days @ $55/day	220.00
Metallurgical Testing	
Bacon, Donaldson & Associates Ltd.	
(see Pro Forma Summary Job 7005)	15,355.24
Report preparation	800.00
TOTAL:	$ 18,775.24

Work performed during 1987-1988 Anniversary Year - $ 10,320.40

Work performed during 1988-1989 Anniversary Year - $ 8,454.84



PROPERTY

LOCATION MAP

HIGHLAND VALLEY, B.C.

SCALE
1" = 136 Miles

1 : 8,617,000

0 218 436 654 872
Kilometres

fig 1

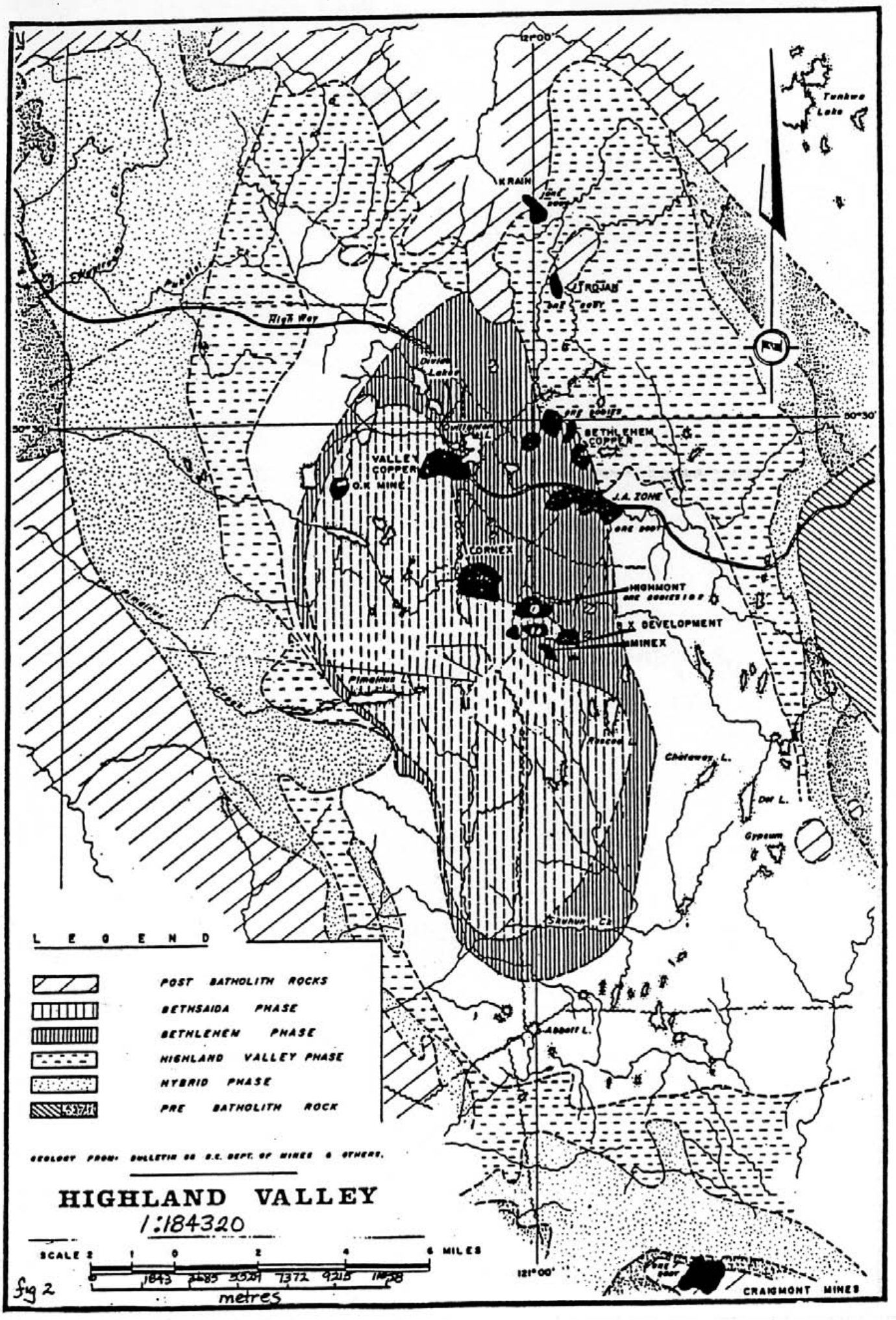
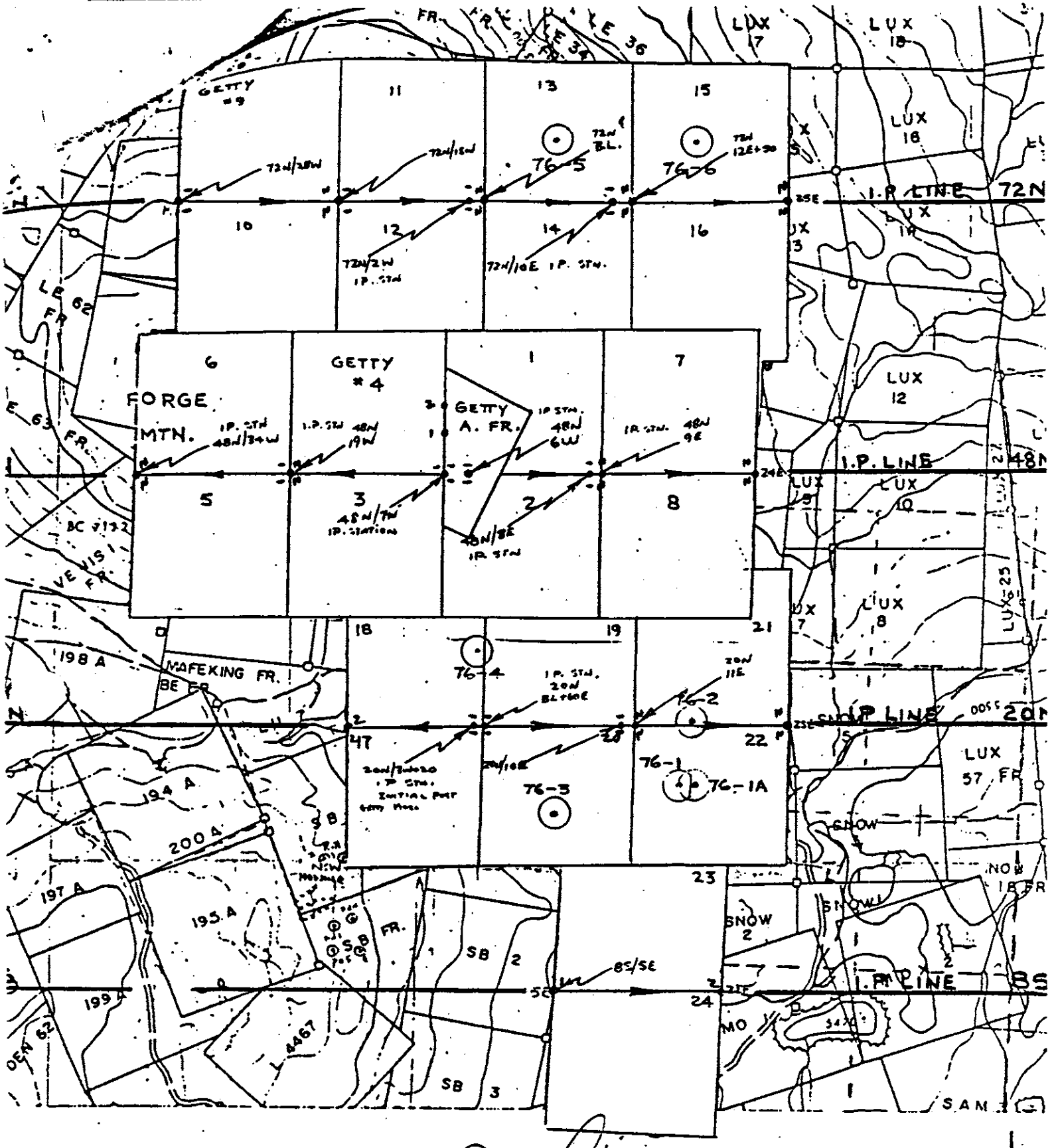

121°00'
50°30'
Tunkwa Lake
KRAIN
TROJAN
High Way
Nicola
Douglas Lake
BETHLEHEM COPPER
VALLEY COPPER
O.K. MINE
J.A. ZONE
ORE BODY
CORNEX
HIGHMONT
ORE BODIES 1 & 2
S.X. DEVELOPMENT
MINEX
Pimainus L.
Pimainus Creek
Chataway L.
Owl L.
Gypsum
Skuhun Cr.
Abbott L.
CRAIGMONT MINES
LEGEND
POST BATHOLITH ROCKS
BETHSAIDA PHASE
BETHLEHEM PHASE
HIGHLAND VALLEY PHASE
HYBRID PHASE
PRE BATHOLITH ROCK
GEOLOGY FROM BULLETIN 38 B.C. DEPT. OF MINES & OTHERS.
HIGHLAND VALLEY
1:184320
SCALE 2 1 0 2 4 6 MILES
1843 3685 5527 7372 9213
metres

fig 2



GETTY CLAIMS

#1-24 + "A" FRACTION

KAMLOOPS MINING DIVISION

HIGHLAND VALLEY, B.C.

0 168 336 504

Metres

1:15840

CLAIM MAPS

92I/11E

+

92I/10W

SCALE 1" = 1320'

N

fig 3

INTRODUCTION

Gower, Thompson & Associates Ltd. and Bacon, Donaldson & Associates Ltd. were contracted by Robak Industries to sample the outcropping oxide zone and conduct heap leaching tests to determine the economic viability of leaching the copper.

LOCATION

The centre of the property is located approximately 650 metres (2,145 feet) east of North Forge Mountain, about 10 kilometres (6 miles) north of Valley Copper Mine in the Highland Valley District, Latitude 50°34', Longitude 121°00'. Access to the property is via the South-Seas Trojan Road, which leaves the main Highland Valley Road at the Old Bethlehem Mine.

CLAIM STATUS

The property consists of the Getty 1-24, two-post claims and the A-fractional claim. These claims are wholly owned by Robak Industries.

- Getty 1-2, Record #128405-406 (Month of Record - August)
- Getty 3-24, Record #128545-566 (Month of Record - August)
- Getty A-Fraction, Record #128567 (Month of Record - August)

HISTORY

In 1955, the property was acquired by the Beaverlodge Uranimum - Farwest Tungsten group from Northlodge Copper Mines. Ltd. It was optioned in succeeding years to major and junior mining companies who have expended about $1,300,000 on the property. Robak Industries acquired the property in 1974.

A list of optioners since 1955 and a summary of work is as follows:

1955-1957: Beaverlodge Uranium and Farwest Tungsten optioned the property and carried out magnetometer and geochemical surveys, geological mapping, 600 metres of bulldozer trenching and 27 diamond drill holes totalling 2,937 metres DDH K-1 to K-27 inclusive.

1957-1959: The property was optioned to Kennecott Copper who carried out geological mapping, geochemical, magnetometer and I.P. surveys, 660 metres of bulldozer trenching and diamond drilled two holes totalling 340 metres DDH K-28, D-1. Option agreement was terminated by Kennecott and the property remained dormant until 1964.

1964-1965: North Pacific Mines acquired the property and diamond drilled eight holes totalling 2,344 metres and percussion drilled 17 holes totalling 800 metres. DDH 1-65 to 8-65 and P-1 to P-17.

HISTORY, contd.

1965-1966: Property was optioned to Canex Aerial Explorations Ltd. who carried out a soil sampling survey and a diamond drilling program totalling 2,000 metres. DDH 9-65 to 22-65 and 1-66, 2-66.

1967: The property was optioned to the Isaac Shulman Syndicate of Vancouver, who diamond drilled four holes totalling 846 metres. DDH S-30 to S-33.

1968: The property reverted to North Pacific Mines Ltd. who carried out some bulldozer trenching and an airborne magnetometer survey.

1968-1969: The property was optioned to Brameda Resources Ltd. who, in turn, optioned it to Noranda Explorations Ltd. Noranda carried out geochemical and I.P. surveys, geological mapping and follow-up diamond drilling, totalling 958 metres, DDH 1-69 to 3-69 and 9-69 to 12-69 of geophysical anomalies located at some distance from the main copper zone.

1970: The property again reverted to North Pacific who drilled 25 percussion holes totalling 1,150 metres, P1-70 to P18-70, P20-70 to P26-70.

1971-1972: The property was optioned to Getty Mines who carried I.P. surveys and drilled 16 percussion holes totalling 1,766 metres and three diamond drill holes totalling 625 metres. DDH 71-1 to 71-3.

HISTORY, contd.

1972-1973: The property was optioned to Quintana Minerals who drilled several percussion holes to test for an extension of the copper zone under volcanic cover to the northwest.

1974 to present: The property was acquired by Robak Industries Ltd. who performed claim staking, percussion drilling, silt and soil geochemistry, bulk sampling and preliminary metallurgical testing.

ECONOMIC ASSESSMENT

The Getty mineral deposit, based on about 125 diamond and percussion drill holes totalling 13,800 metres, occurs as an elongated and partially buried mineral deposit approximately 400 metres long, 200 metres wide and up to 450 metres deep.

Chalcopyrite, bornite and pyrite occur mainly as specks, disseminations and fracture fillings within the quartz diorite and breccia zones and near the shattered margins of the porphyry intrusive. An extensive zone of oxidation covers the northwestern portion of the deposit. Mineralization within the oxide zone consists of chrysocolla, malachite, azurite, cuprite, chalcocite, native copper, hematite and magnetite.

ECONOMIC ASSESSMENT, contd.

Based on diamond drilling done to 1973, reserves exploitable in a mineable zone from a combination of open pit and bulk caving consist of at least:

High Grade - 8,679,750 tons at 0.52% Copper
Low Grade - 12,453,000 tons at 0.20% Copper

In plan view, the copper sulphide deposit is triangular with the known apex to the southeast. The zone appears to be cut off by a fault to the northwest. The northeastern and southern boundaries are near vertical; however, the nose appears to have a steep plunge to the northwest.

GEOLOGY

The mineralization on the Getty property is hosted within quartz diorites of the Highland Valley phase (Guichon variety) of the Guichon batholeth, and within younger dykes and small stocks. The Getty deposit occurs within a northwest trending fracture system typlified by porphyry dykes, hydrothermal veins and fracture assemblages. This fracture system hosts the Bethlehem copper deposit and the Trojan - South-Seas deposit.

The primary ore controls on the Getty property are associated with an elongated 1000x200 metre stock which is unroofed at the centre of the deposit. This intrusive stock appears to be a cupola-like projection developed above the main body of the younger stock.

GEOLOGY, contd.

The apex of the stock plunges gently away from the high point at Getty to both the northwest and southeast. Fracturing, brecciation, alteration and mineralization are localized in and around the younger intrusive stock.

OXIDIZED ZONE

A zone of oxidized copper sulphides occurs in the northern half of the mineral system. This oxide mineralization forms a cap up to 100 metres thick which has been preserved beneath Early Tertiary cover. Copper enrichment has occurred with the oxide zone primarily consisting of Malachite, chrysocolla and a black, waxy copper oxide. These minerals occur as filling fractures and cavities. Only trace amounts of chalcocite have been observed in the oxide zone.

The copper oxide zone is thickest over the centre of the copper sulphide "pipe", and appears to increase in thickness and decrease in grade to the northwest. The southern and northwestern edges of the zone have been depleted in copper, leaving predominantly iron oxides. The average grade of the oxide copper material is probably 0.80% Cu based on studies carried out by the B. C. Research Council.

PRIMARY MINERALIZATION AND ALTERATION

Primary sulphide mineralization and silicate alteration form well defined zonal patterns around the younger stock. Within the stock, and in its margins, chalcopyrite-bornite occurs, associated with

PRIMARY MINERALIZATION AND ALTERATION, contd.

molybdenite bearing quartz veinlets. Adjacent to the core zone, chalcopyrite pyrite fracture fillings occur within quartz stockworks with copper content diminishing towards the outer margins. Sulphide content seldom exceeds 5% by volume, the maximum concentrations occurring within the 0.1% Cu limits of the mineral zone.

Associated silicate alteration is pervasive throughout the zone to its outer margins. A broad chlorite and epidote halo occurs beyond the 0.05% copper limits.

The copper sulphide zone has been tested to a depth of 500 feet for a distance of about 360 metres (1,200 feet). Holes deeper than 244 metres (800 feet) have been drilled only in the southeast nose of the zone, and most of these have copper sulphides grading better than 0.20% Cu to the bottom. Assays from the deepest hole, #2-65, indicate that 0.22% Cu is present 1,500 feet below the drill collar.

STRUCTURE

Copper and molybdenum mineralization is structurally controlled. The highest grades occupy areas of highest fracture density adjacent to the young stock. A strong, predominantly post-mineral north and northeasterly trending fault system crosses the property. Faulting within the Early Tertiary, Kamloops group rocks are restricted almost entirely to down-faulted blocks.

1988 PROGRAM

Two heap leaching tests were carried out by Bacon, Donaldson & Associates Ltd.- The first test was performed on rock material which had been pulverized. Copper extraction by leaching was satisfactory in this test. However, acid consumption was higher than anticipated. Further sample material was procured and a second test carried out on material averaging about 12 cm in size. Acid consumption was much lower in the second test.

HEAP LEACHING TEST #1

This test demonstrated that at least 90% of the copper can be extracted by acid leaching. A solution of pH of 2.0 or less had to be maintained to achieve this extraction. Under the Test #1 conditions, a total acid consumption of 57.5 kg/tonne was indicated for a bulk sample grading 1.0% copper when treating pulverized material. Of this acid consumption, 15.4 kg/tonne is associated with copper and will therefore be recovered, resulting in a net consumption of 42.1 kg H_2SO_4 per tonne treated. The cost of acid per tonne ore and per kg copper producted is summarized in Table 1.

TABLE 1

Summary of Acid Costs

Projections	- H_2SO_4 obtained from source for $30.00/tonne	
	- Transportation cost for acid to site @ $30.00/tonne	
	- Copper grade = 1% (grade of Bulk Sample Test #1)	
	- Copper recovery = 90%	
	- Acid Consumption = 42.1 kg/tonne	
Then	- Acid cost per tonne of ore	= $ 2.53
	- Acid cost per kg of Cu	= $ 0.28
	- Acid costs per pound of copper produced	= $ 0.13

HEAP LEACHING TEST #2

Description

A heap leaching test was performed on a second bulk sample of oxide copper ore obtained from the Krain deposit by S. C. Gower. The sample weighed 240 kilograms and had a top size of 12 cm. The entire sample was placed in a column, having a diameter of 30 cm. Sulphuric acid solution was pumped to the top of the column where a distribution head spread it over the surface of the column. The leach solution was collected from the bottom of the column and analyzed to determine copper extraction and acid consumption.

When the rate of copper extraction had decreased significantly, the acid flow was stopped and the column was flushed with water. The contents of the column were dumped out and crushed prior to being sampled for assay.

Results

The detailed solution analyses are summarized in the attached table. Over the course of the test, both the acid concentration and the solution flow rate were varied to observe the effect on copper leaching. A high concentration of copper in solution corresponded to high acid strength. Doubling the solution flow rate did not significantly lower the copper concentration in solution. As the copper concentration in the solution started to decrease (leach cycles 17 to 19), the acid concentration was decreased from 10 to 5 g/L.

HEAP LEACHING TEST #2, contd.

Results, contd.

Figure 1 shows copper extraction as a function of leach cycle. Most cycles represent a period of one day, but, in some cases, two cycles per day were monitored. The extraction rate was very high for the first ten days and then started to decrease as the acid concentration and remaining copper decreased.

Figure 2 shows the acid consumption as a function of leach cycle. The consumption occurred steadily over the leach period to a total of 28.6 kg/tonne as H_2SO_4. Of this total, 11.7 kg/tonne is accounted for as copper. This acid would be recycled from the subsequent copper recovery step so that the net acid consumption was 16.9 kg H_2SO_4/tonne ore.

The analysis of the solids after copper leaching indicated that 0.82% Cu remained. It was visually apparent when the column was emptied that this copper was concentrated in several sections of the column which had not become property wetted by the leach solutions. Increased solution flows together with a larger column diameter to maximum particle ratio would overcome this problem. A piece of copper-bearing ore from the leached product was placed in dilute acid and was shown to be readily leachable. Although only 57.1% of the copper was extracted from the feed to the column, there is no question that this low extraction was due only to poor percolation and that a copper extraction of 90% or better can be achieved.

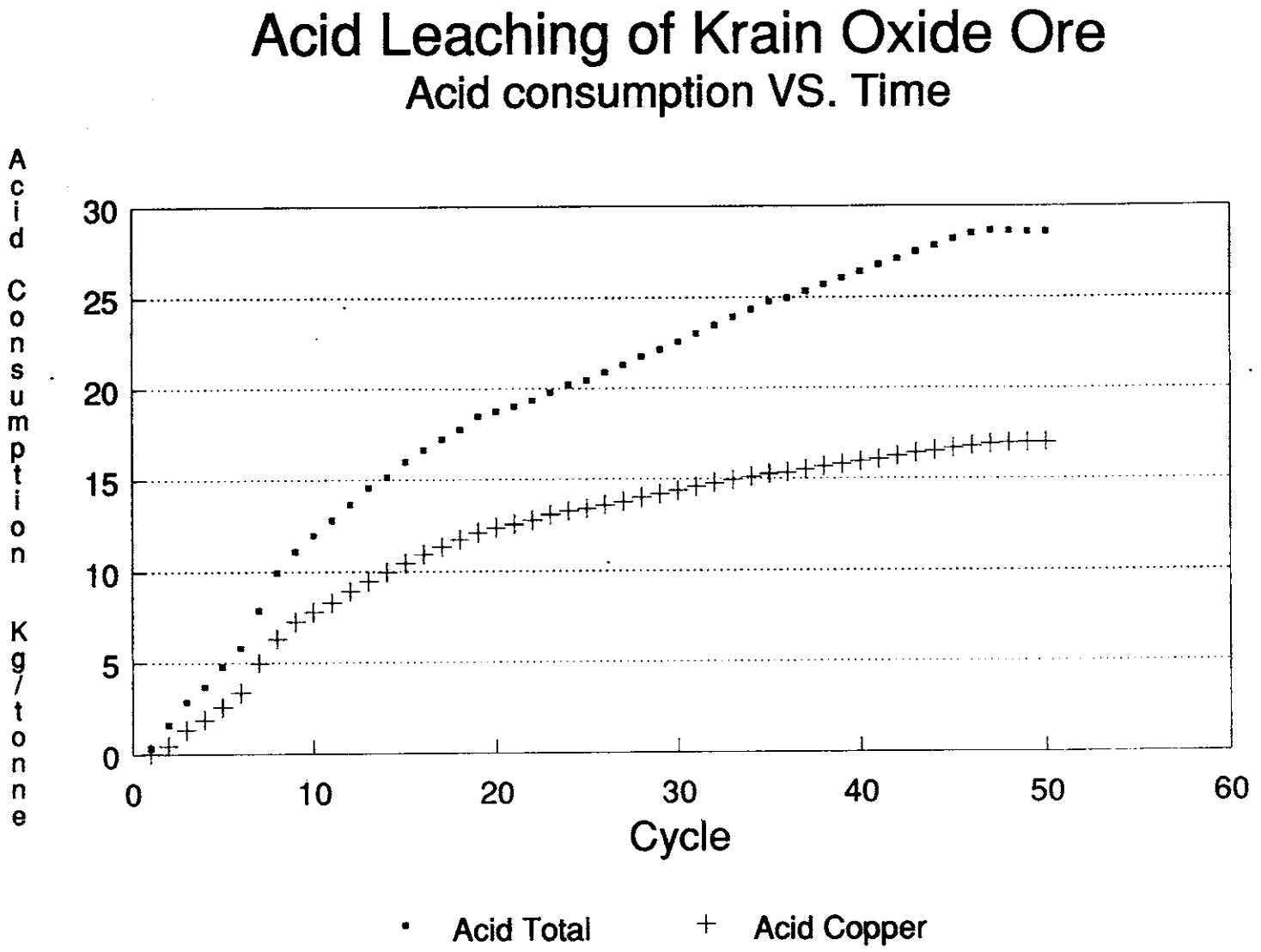

Acid Leaching of Krain Oxide Ore
Acid consumption VS. Time
Acid Consumption Kg/tonne
30
25
20
15
10
5
0
0
10
20
30
40
50
60
Cycle
Acid Total
Acid Copper

Figure 3.1

Acid Leaching of Krain Oxide Ore

Copper Concentration VS. Time

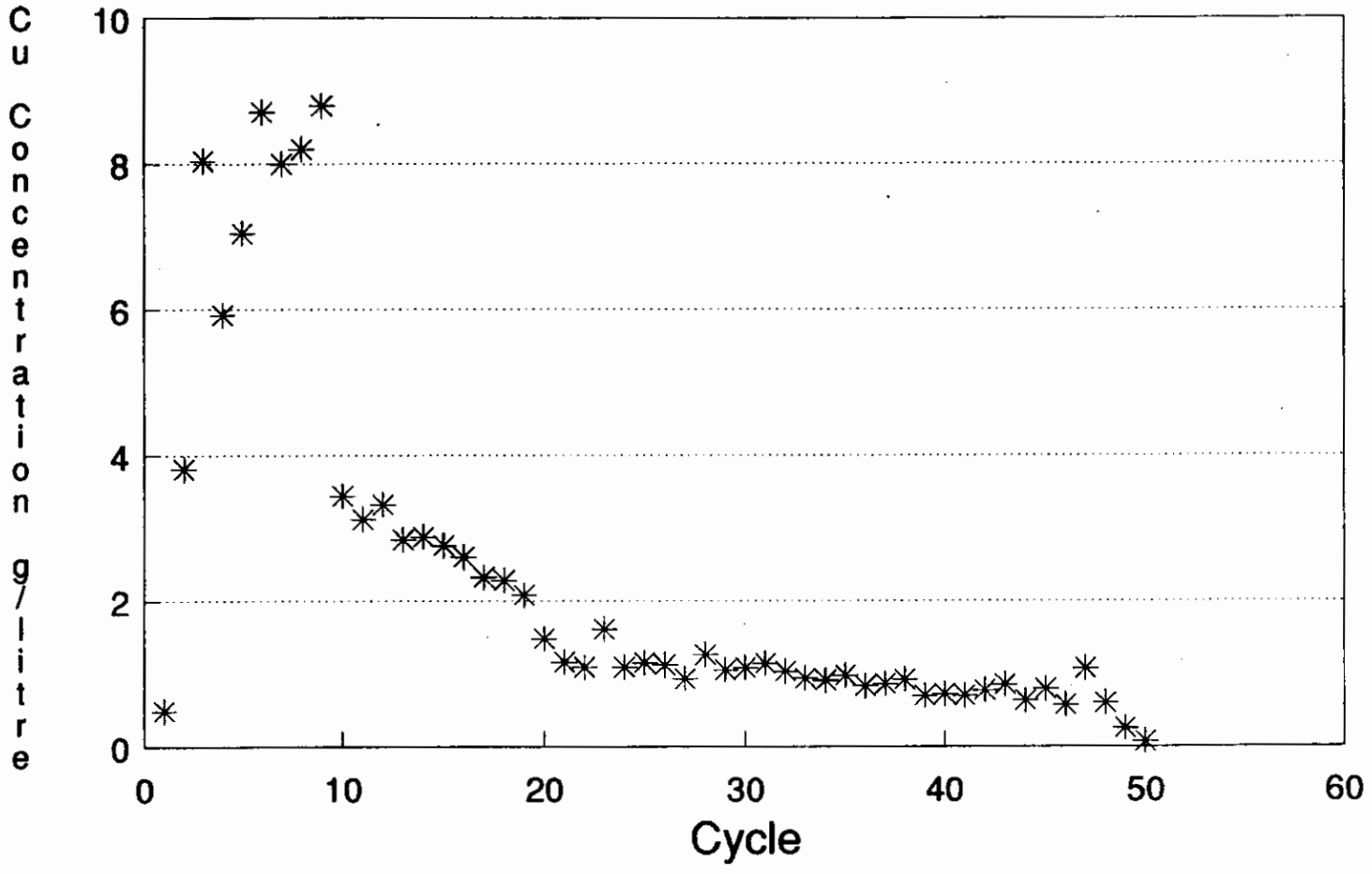


Figure 3.2

#7005: KRAIN COPPER COLUMN

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 GRAMS H2SO4 IND.	CONSUMPTION GRAMS H2SO4 CUM.	Fe (g/l)	COPPER g/l	COPPER g total	CALCULATED % COPPER EXTRACTED IND.	CALCULATED % COPPER EXTRACTED CUM.
June 23	1	17.45	13.00	10	9.0	5	0.71	78.0	78.0	0.040	0.49	6.37	0.1%	0.1%
24	2	16.82	16.64	10	11.6	20	2.67	292.0	370.0	0.006	3.81	63.40	1.4%	1.6%
25	3	16.27	16.09	10	11.2	20	2.41	286.6	656.6	0.052	8.04	129.36	2.9%	4.5%
26	4	13.64	13.82	10	9.6	20	5.81	192.5	849.1	0.162	5.92	81.81	1.8%	6.3%
27	5	14.27	14.82	10	10.3	20	2.34	250.7	1099.8	0.097	7.04	104.33	2.4%	8.7%
28	6	13.82	13.55	10	9.4	20	2.92	236.8	1336.7	0.181	8.71	118.02	2.7%	11.4%
29	7	29.18	30.09	20	20.9	20	3.70	472.3	1808.9	0.380	8.00	240.72	5.4%	16.8%
30	8	27.27	24.18	20	16.8	20	2.87	476.0	2284.9	0.290	8.20	198.28	4.5%	21.3%
July 4	9	15.82	15.91	20	11.0	20	3.19	265.6	2550.6	0.500	8.80	140.01	3.2%	24.4%
5	10	23.90	22.82	20	15.8	10	1.78	198.4	2749.0	0.185	3.44	78.50	1.8%	26.2%
6	11	24.27	24.27	20	16.9	10	2.18	189.8	2938.8	0.237	3.12	75.72	1.7%	27.9%
7	12	26.82	27.64	20	19.2	10	2.61	196.1	3134.8	0.268	3.32	91.76	2.1%	30.0%
8am	13	30.00	29.73	40	41.3	10	3.00	210.8	3345.6	0.275	2.84	83.41	1.9%	31.9%
8pm	14	24.36	25.00	40	34.8	10	4.43	132.9	3478.5	0.308	2.88	72.00	1.6%	33.5%
11	15	26.00	26.36	20	18.3	10	2.51	193.8	3672.3	0.310	2.76	72.75	1.6%	35.1%
12am	16	25.64	28.00	40	38.9	10	3.81	149.7	3822.0	0.300	2.60	72.80	1.6%	36.8%
12pm	17	26.45	25.73	40	42.9	10	5.13	132.5	3954.5	0.335	2.32	59.69	1.3%	38.1%
13am	18	26.18	26.27	40	36.4	10	5.30	122.6	4077.1	0.310	2.28	59.90	1.4%	39.5%
13pm	19	26.82	27.09	40	37.6	10	3.68	168.5	4245.6	0.230	2.08	56.35	1.3%	40.8%
14am	20	25.91	25.90	40	36.0	5	2.60	62.2	4307.8	0.180	1.48	38.33	0.9%	41.6%
14pm	21	25.00	24.91	40	34.6	5	2.46	63.7	4371.6	0.150	1.16	28.90	0.7%	42.3%
15am	22	26.82	27.64	40	37.3	5	2.31	70.3	4441.8	0.160	1.09	30.13	0.7%	43.0%
18	23	29.73	28.64	20	19.9	5	1.61	102.5	4544.3	0.200	1.61	46.11	1.0%	44.0%
19	24	26.64	25.82	20	17.9	5	1.43	96.3	4640.6	0.170	1.09	28.14	0.6%	44.6%
20	25	15.82	16.27	20	11.3	5	1.54	54.0	4694.7	0.190	1.15	18.71	0.4%	45.1%
21	26	30.00	28.09	20	19.5	5	1.72	101.7	4796.4	0.210	1.12	31.46	0.7%	45.8%
22	27	28.45	28.09	20	19.5	5	1.71	94.2	4890.6	0.190	0.93	26.12	0.6%	46.4%
28	28	28.09	26.51	20	18.4	5	1.38	103.9	4994.4	0.189	1.26	33.40	0.8%	47.1%
29	29	25.91	26.43	20	18.4	5	1.56	88.3	5082.8	0.186	1.05	27.75	0.6%	47.7%
30	30	29.73	27.53	20	19.1	5	1.77	99.9	5182.7	0.240	1.08	29.73	0.7%	48.4%
Aug 3	31	28.43	26.23	20	18.2	5	1.39	105.7	5288.4	0.175	1.14	29.90	0.7%	49.1%
5	32	29.41	27.93	20	19.4	5	1.58	102.9	5391.3	0.216	1.03	28.78	0.6%	49.7%
8	33	29.29	27.13	20	18.8	5	1.64	102.0	5493.2	0.210	0.94	25.50	0.6%	50.3%
9	34	29.01	25.40	20	17.6	5	1.96	95.3	5588.5	0.220	0.91	23.11	0.5%	50.8%
11	35	27.82	24.27	20	16.9	5	1.75	96.6	5685.1	0.220	0.98	23.78	0.5%	51.4%
12	36	14.91	14.09	20	9.8	5	2.01	46.2	5731.4	0.240	0.83	11.69	0.3%	51.6%
15	37	27.82	27.73	20	19.3	5	1.92	85.9	5817.2	0.240	0.86	23.85	0.5%	52.2%
16	38	27.00	27.09	20	18.8	5	1.88	84.1	5901.3	0.230	0.92	24.92	0.6%	52.7%
17	39	29.82	29.23	20	20.3	5	2.28	82.5	5983.8	0.200	0.70	20.46	0.5%	53.2%
18	40	29.36	29.09	20	20.2	5	2.31	79.6	6063.4	0.240	0.72	20.94	0.5%	53.7%
19	41	30.00	26.18	20	18.2	5	2.41	86.9	6150.3	0.210	0.70	18.33	0.4%	54.1%
22	42	29.82	33.73	20	23.4	5	2.31	71.2	6221.4	0.210	0.77	25.97	0.6%	54.7%
30	43	29.91	27.73	20	19.3	5	2.08	91.9	6313.3	0.180	0.85	23.57	0.5%	55.2%
31	44	29.82	30.27	20	21.0	5	2.50	73.4	6386.7	0.200	0.63	19.07	0.4%	55.6%
Sept 7	45	29.82	28.18	20	19.6	5	2.15	88.5	6475.3	0.170	0.79	22.26	0.5%	56.1%
9	46	29.82	29.73	20	20.6	5	2.69	69.1	6544.4	0.190	0.57	26.95	0.6%	56.8%
12	47	17.55	15.73	10	10.9	5	3.38	34.6	6579.0	0.310	1.07	16.83	0.4%	57.1%
WATER WASHES:														
Sept 20	48		15.00	20	10.4	0	0.60	-9.0	6570.0	1.070	0.60	9.00	0.2%	57.3%
22	49		15.55	20	10.8	0	0.22	-3.4	6566.5	0.100	0.25	3.89	0.1%	57.4%
23	50		30.09	20	20.9	0	0.04	-1.2	6565.3	0.080	0.07	2.11	0.0%	57.5%

COLUMN TAILS

DATE	LEACH CYCLE	TOTAL WEIGHT (g)	Cu %	Cu grams	Fe %	Fe grams	S (TOTAL) %	S (TOTAL) grams	S (-2) %	S (-2) grams
Sept 23	50	229628	0.82	1882.9	2.32	5327.4	0.01	22.96	0.01	22.96

TOTAL COPPER 4427.81 g = 1.93% 169 kg/t

ACID CONSUMPTION 28.59 Kg/tonne Ore

RECOMMENDATIONS

1. Ten thousand feet of reverse circulation drilling is required to confirm data presently available on tonnage and grade. Metallurgical tests can be carried out on drill cutting composites.

2. Bulk samples are required from surface for metallurgical testing. It is recommended that four one-ton samples be obtained. Three samples can be procured from trenches cutting different portions of the oxide zone and one from the existing adit. Column leaching tests should be carried out on each sample to establish acid requirement, leaching rate and ultimate extraction. Solvent extraction consisting of small scale continuous testing should be done on the pregnant solution from the column tests.

REFERENCES

CANEX PLACER; Intercompany Report; March 23, 1971.

GOWER, S.C.; Geological and Geochemical Assessment Report; October 31, 1984.

GRISWOLD, G. B.; Summary Report of Field Work, conducted by Getty Mines on the Krain Deposit, February 29, 1972; August 1972.

McMILLAN, W. J.; Preliminary Geological Map of the Highland Valley; B. C. Department of Mines; 1971.

NORTHCOTE, K. E.; Geology and Geochronology of the Guichon Batholith; 1969.

CERTIFICATE OF QUALIFICATIONS

I, Stephen C. Gower, of 985 Gatensbury Street, Coquitlam, B. C., do hereby certify that:

1. I have been practising as a geologist for a period of approximately 19 years for mining, exploration and consulting companies.

2. I obtained a B.Sc. in geology from U.B.C. in 1970 and have taken Masters courses at U.B.C. in property evaluation and exploration.

3. I am a fellow in the Geological Association of Canada.

4. The geological work in this report was carried out by Stephen C. Gower and Elaine M. Thompson during the period June 3 to 6, 1988.

5. I have no interest either directly or indirectly in the properties held by Robak Industries. It is possible at some time that I may be entering into a business arrangement concerning the Getty property.

October 1988

Stephen C. Gower

STATEMENT OF QUALIFICATIONS

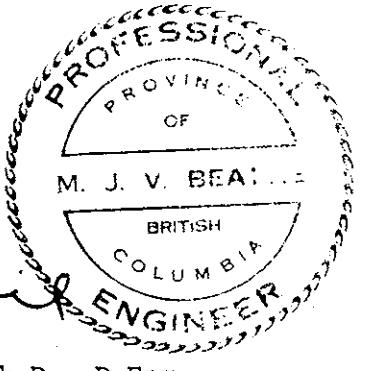
1. I received a B.A.Sc. in 1971, a M.A.Sc. in 1973 and a Ph.D. in 1983 from the University of British Columbia in Mineral Process Engineering.

2. I have been employed as a project engineer and later as Vice-President of Extractive Metallurgy with Bacon, Donaldson and Associates Ltd. since 1973.

3. I have been an adjunct professor of Mineral Process Engineering at the University of British Columbia since 1985.

4. I am registered to practice my profession as an engineer with the Association of Professional Engineers of British Columbia.

5. I reviewed the available metallurgical data on the Getty deposit and supervised the metallurgical testwork conducted by Bacon, Donaldson and Associates Ltd. in 1987. Based on this information, I believe that the information pertaining to metallurgy as presented in the October 24, 1988 report by Gower, Thompson and Associates Ltd. presents fairly the results to be achieved and the preliminary cost estimates.



October 1988

Morris J.V. Beattie, Ph.D., P.Eng.

APPENDIX A

PRO FORMA SUMMARY

FOR JOB 7005

BACON, DONALDSON & ASSOCIATES LTD.

PRO FORMA SUMMARY FOR JOB 7005

	TOTAL	INVOICED	BALANCE
SERVICES:	7,964.00	0.00	7,964.00
CHARGES:	7,077.00	0.00	7,077.00
DIRECTS:	314.24	0.00	314.24
ADVANCES:	0.00		
TOTAL:	15,355.24	0.00	15,355.24

Pro Forma Invoice for Job : 7005

Date In : 03/27/87

Client : KRAIN PROJECT PO :

Contact :
Phone : Fax :

Eng-in-Charge : mvb Nature : copper deposit
Department : min Cost Basis:
Status : a

Re : _____
_____
_____

Complete: Y N

Invoices

Job	InvNo	Date	Services	Charges	Directs	Total
** Subtotal **			0.00	0.00	0.00	0.00
*** Total ***			0.00	0.00	0.00	0.00

Advances

RN	Job	Date	Advances
** Subtotal **			0.00
*** Total ***			0.00

JOB NO: 7005

Hours Assignment

RN Job	Week	Norm	OT	Rate	OTS	Services
** Hours for bs						
1182 7005	42/88	3.0	0.0	60	7	180.00
1270 7005	43/88	9.5	0.5	60	7	603.50
** Subtotal **						
2402		12.5	0.5			783.50
** Hours for cr						
2852 7005	03/89	2.0	0.0	20	0	40.00
3006 7005	04/89	1.0	0.0	20	0	20.00
1261 7005	43/88	1.0	0.5	30	7	48.50
1657 7005	47/88	1.5	0.0	30	7	45.00
1824 7005	48/88	1.5	0.0	30	7	45.00
2081 7005	50/88	2.0	0.0	30	7	60.00
2164 7005	51/88	2.0	0.0	30	7	60.00
2234 7005	52/88	1.5	0.0	30	7	45.00
** Subtotal **						
17079		12.5	0.5			363.50
** Hours for db						
2439 7005	01/89	2.5	0.0	60	7	150.00
2784 7005	03/89	3.0	0.0	60	7	180.00
985 7005	41/88	1.0	0.0	60	7	60.00
1105 7005	42/88	0.5	0.0	60	7	30.00
1178 7005	43/88	6.5	3.0	60	7	591.00
1383 7005	44/88	6.0	2.0	60	7	494.00
1490 7005	45/88	6.0	2.0	60	7	494.00
1523 7005	46/88	2.5	4.0	60	7	418.00
1760 7005	48/88	2.0	0.0	60	7	120.00
1864 7005	49/88	2.0	0.0	60	7	120.00
1988 7005	50/88	1.5	0.0	60	7	90.00
2282 7005	52/88	1.5	0.0	60	7	90.00
2319 7005	53/88	1.0	0.0	60	7	60.00
** Subtotal **						
23095		36.0	11.0			2897.00
** Hours for en						
3352 7005	06/89	4.5	0.0	75	0	337.50
2010 7005	50/88	1.5	0.0	60	0	90.00
2121 7005	51/88	1.0	0.0	60	0	60.00
** Subtotal **						
7483		7.0	0.0			487.50
** Hours for jh						
2824 7005	03/89	1.0	0.0	60	7	60.00
1229 7005	43/88	1.0	0.5	60	7	93.50
1462 7005	45/88	7.5	0.0	60	7	450.00
1572 7005	46/88	7.5	0.0	60	7	450.00
1671 7005	47/88	9.5	0.0	60	7	570.00
1789 7005	48/88	1.0	0.0	60	7	60.00
2017 7005	50/88	0.5	0.0	60	7	30.00
2129 7005	51/88	3.0	0.0	60	7	180.00

** Subtotal **						
14693		31.0	0.5			1893.50
** Hours for mb						
2999 7005	04/89	4.0	0.5	30	7	138.50
1189 7005	43/88	0.5	1.5	30	7	70.50
1644 7005	47/88	1.0	0.0	30	7	30.00
** Subtotal **						
5832		5.5	2.0			239.00
** Hours for mvb						
279 7005	33/88	1.0	0.0	100	0	100.00
354 7005	34/88	4.5	0.0	100	0	450.00
507 7005	35/88	4.0	0.0	100	0	400.00
974 7005	40/88	0.5	0.0	100	0	50.00
1067 7005	41/88	1.0	0.0	100	0	100.00
1177 7005	43/88	1.0	0.0	100	0	100.00
1869 7005	49/88	1.0	0.0	100	0	100.00
** Subtotal **						
6227		13.0	0.0			1300.00
*** Total ***						
76811		117.5	14.5			7964.00

JOB NO: 7005

Charges

RN	Job	Date	Source	Details	Total
** n	7005				
591	7005	01/ /89	oasy	9 @ $10	90.00
9	7005	02/29/88	wip	total hrs. in $	5577.00
669	7005	03/ /89	oasy	5 @ $10	50.00
748	7005	04/ /89	oasy	10 @ $10	100.00
843	7005	06/ /88	oasy	2 @ $10	20.00
304	7005	43/ /88	oasy	2 @ $10	20.00
346	7005	44/ /88	oasy	11 @ $10	110.00
354	7005	45/ /88	oasy	25 @ $10	250.00
375	7005	46/ /88	oasy	22 @ $10	220.00
392	7005	47/ /88	oasy	19 @ $10	190.00
446	7005	48/ /88	oasy	2 @ $10	20.00
451	7005	49/ /88	oasy	9 @ $10	90.00
486	7005	50/ /88	oasy	11 @ $10	110.00
513	7005	51/ /88	oasy	17 @ $10	170.00
521	7005	52/ /88	oasy	4 @ $10	40.00
577	7005	53/ /88	oasy	2 @ $10	20.00
** Subtotal **					7077.00
*** Total ***					7077.00

Direct Expenses

RN Job	Date	Source	Details	Amount	%SC	Total
** n 7005						
204 7005	06/09/88	Intn'l Plastics	13' pipe & delivery	246.95	10	271.64
269 7005	06/28/88	BUTT & BOWES - petty	Buckets	38.73	10	42.60
** Subtotal **				285.68		314.24
*** Total ***				285.68		314.24