

INTERIM REPORT ON THE LEACHABILITY OF  
MT. WASHINGTON ORES  
THE MWC 1 Fr TO MWC 526 MINERAL CLAIMS  
IN THE NANAIMO MINING DIVISION

Owned by  
MT. WASHINGTON COPPER COMPANY LIMITED

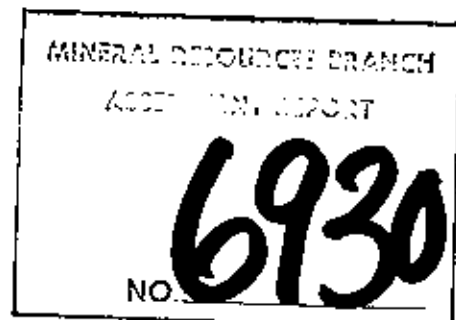
92 F 11 and 14

Lat.  $49^{\circ} 45'$  Long.  $125^{\circ} 17'$

For  
ESSO MINERALS CANADA  
IMPERIAL OIL LIMITED

By  
A. Bruynesteyn  
and countersigned by  
R. Somerville, P.Eng.

October 1978



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INTERIM REPORT ON THE  
LEACHABILITY OF MT. WASHINGTON ORES

INTRODUCTION

Location

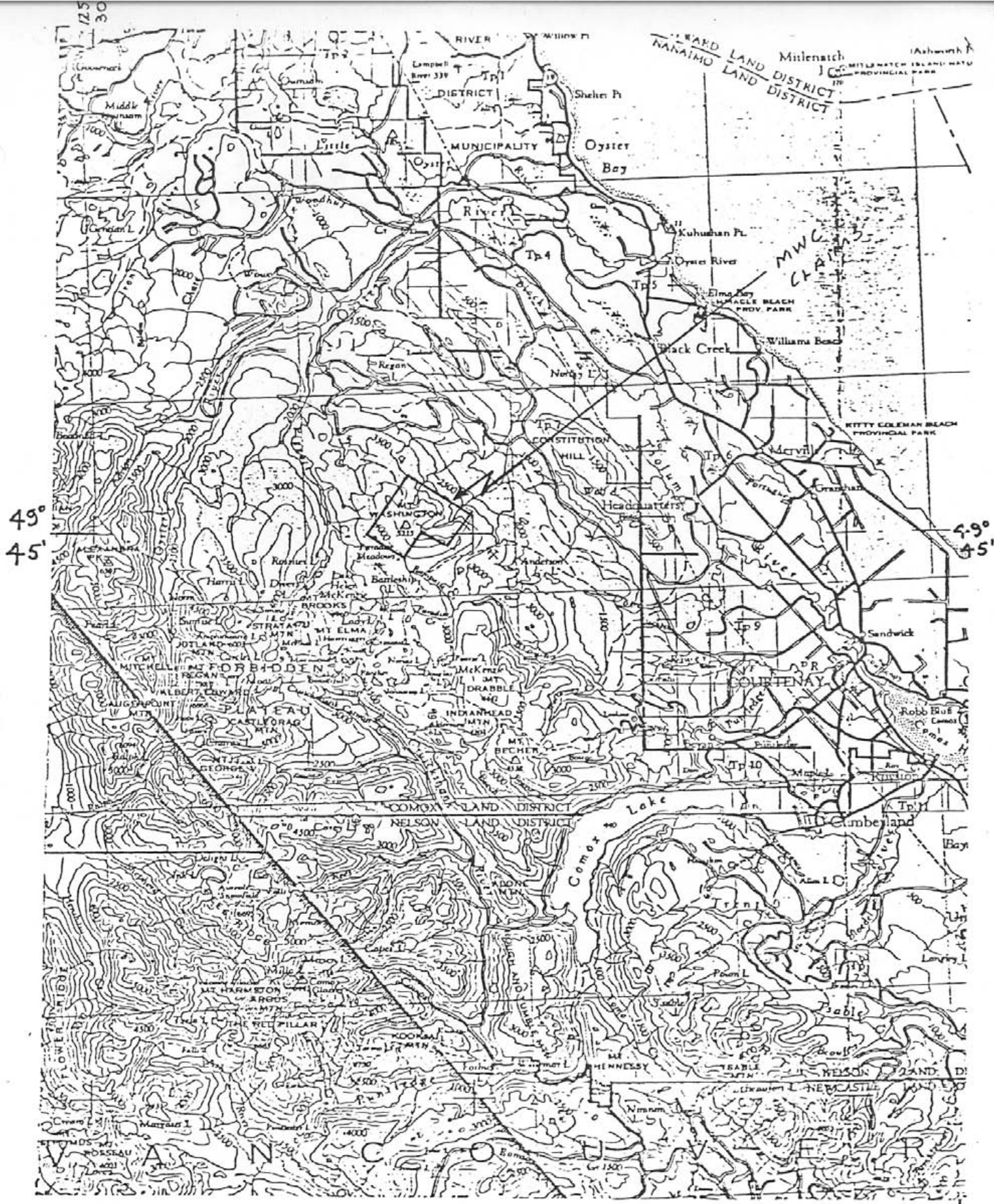
The MWC claims are indicated on Location Maps #1 and 2. This property is centered around the northeast facing slope of Mt. Washington.

Mt. Washington is located on Vancouver Island north west of Courtenay, B.C., some 17 road miles from tidewater. The access is by well kept logging roads maintained by Crown Zellerbach Ltd.

The Property

The Mt. Washington property consists of 163 MWC claims and 4 crown grants. The expiry dates subsequent to the filing of the assessment work in September 1978 will be as follows:

<u>Expiry Date</u>		<u>Claims</u>
1979	MWC	159-161, 262, 264, 266, 269, 270, 275, 276, 279, 289-292, 296, 298, 300, 522, 524, 526.
1980	MWC	129-131 Fr., 137-142 Fr., 157, 158, 170 Fr., 175-192, 201-202, 237-244, 247-252, 257, 258, 281-284.
1981	MWC	103-114, 116, 118-128, 133, 134, 173, 174, 211, 212, 222 Fr., 229, 230, 235, 236.
1982	MWC	132, 203, 205, 207-210, 215-218, 221, 223, 225, 227, 228.
1983	MWC	1 Fr., 101, 152, 224, 226, 280.
1984	MWC	102, 115, 117, 135, 136, 143-146, 149, 151, 153-156, 171, 172, 204, 206, 231-234, 271, 273.
1986	MWC	147, 148, 150, 213, 214, 272, 274 Fr., 294.

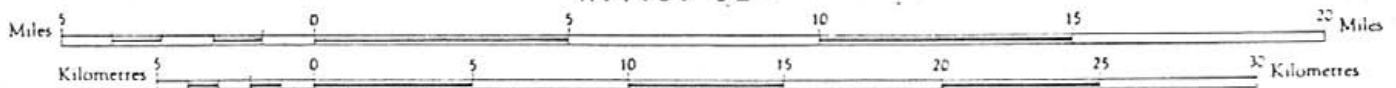


MWC CLAIMS - MT. WASHINGTON PROJECT

Scale 1:250,000

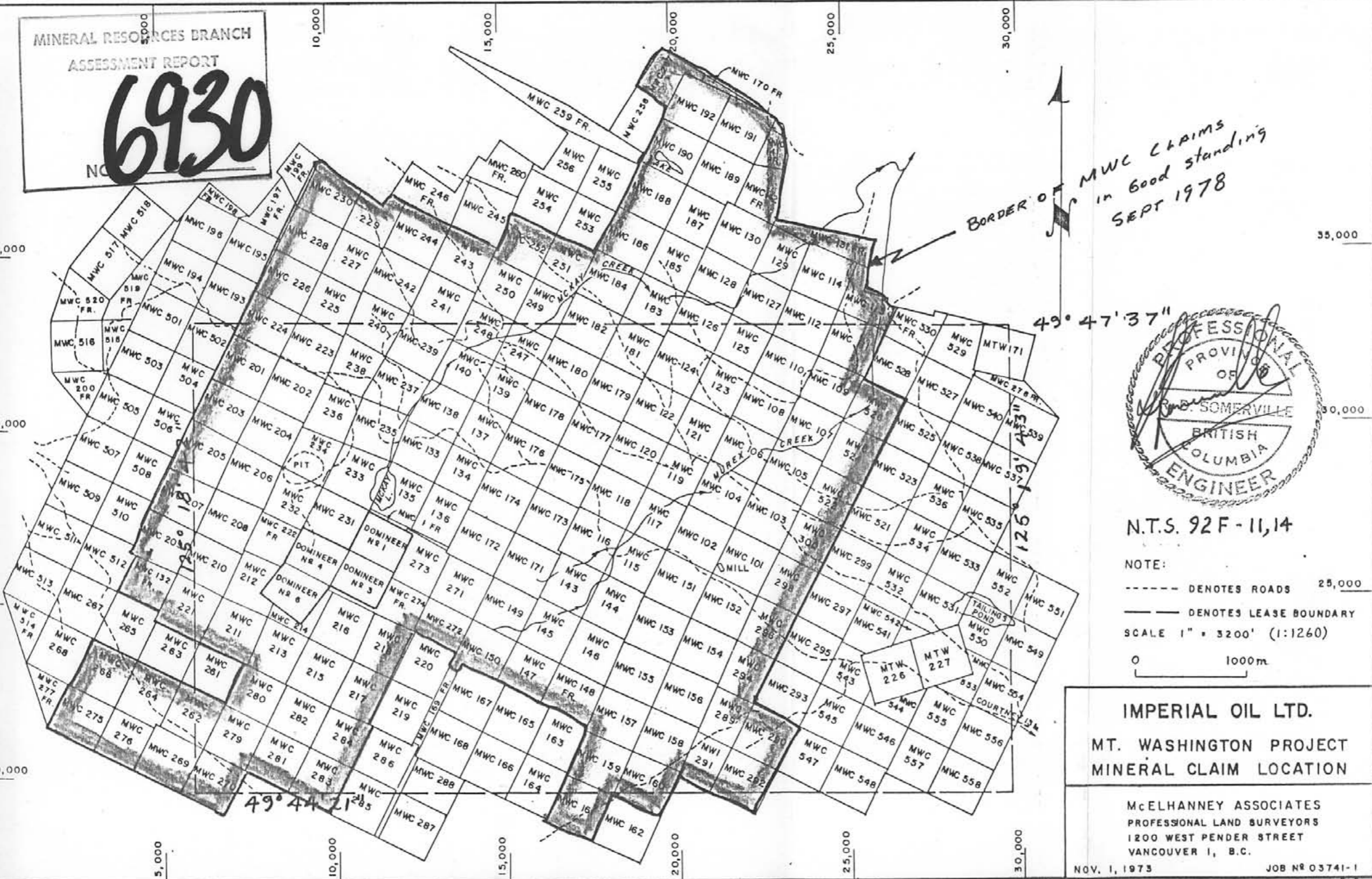
Location Map No. 1

N.T.S. 92-F



MINERAL RESOURCES BRANCH  
ASSESSMENT REPORT

6930



N.T.S. 92F-11,14

NOTE:  
 - - - - - DENOTES ROADS 25,000  
 ——— DENOTES LEASE BOUNDARY  
 SCALE 1" = 3200' (1:1260)  
 0 1000m

**IMPERIAL OIL LTD.**  
 MT. WASHINGTON PROJECT  
 MINERAL CLAIM LOCATION

McELHANNEY ASSOCIATES  
 PROFESSIONAL LAND SURVEYORS  
 1200 WEST PENDER STREET  
 VANCOUVER 1, B.C.  
 NOV. 1, 1973 JOB NO 03741-1

## Objective

To assess the leaching characteristics of run of the mine leach ores from the Mount Washington Mine.

## Background

The Minerals Division of Imperial Oil Limited has performed extensive exploration work on the Mount Washington Copper Company Limited property on Vancouver Island which indicated the presence of large quantities of sulfide minerals. Preliminary shakeflash tests performed at B.C. Research (reference 1), showed that the mineralization was amenable to chemical and biological leaching and at the request of Imperial Oil, B.C. Research has conducted a series of large column tests on four samples of mineralization from varying areas of the mine, to confirm the leachability of the samples. This report discusses the result of these column tests.

## SUMMARY

- . Mineral samples from 4 different locations on the property were leached for 180 days in large columns.
- . Of the 4 ore samples, only one sample No. 4 leached extremely well at an extraction rate of 29% per year, equivalent to a production of 3,500,000 kg copper per year per million tonnes of ore.
- . The No. 4 ore sample did not require any acid and was leached with tap water while still producing pregnant solution at a pH of 2.5 - 2.0.
- . The ore represented by sample No. 1 reached 8% extraction in 180 days but its acid consumption and pregnant solution pH was so high that it is indicated that most of this extraction was chemical rather than biological. However, the sample was probably preleached by natural leaching and is not representative of the dump.
- . The ores represented by samples No. 2 and 3 were highly acid consuming and leached poorly, reaching 1.1 and 2.1% extraction respectively.
- . Leach residues from the column showed that many ore particles were coated with a translucent coating of  $\text{CaSO}_4$ .

## Experimental Procedures

### Sample

On November 9, 1977, four samples of ore were received at B.C. Research representing different areas around the property.

Sample No. 1 Eastern Dump, sample taken from 1 ft. below surface.

Sample No. 2 Taken from just west of the dump from which sample 1 was taken.

Sample No. 3 Taken from the floor of the open pit. This sample was split in high grade 3A and low grade 3B portions.

Sample No. 4 Taken south of the main pit towards Glacier Lake.

Upon arrival at B.C. Research, each sample was homogenized, coned and quartered twice and a 1/16 portion set aside for particle size analyses and head grade determination:  $\frac{1}{2}$  of each size portion was crushed and pulverized and a portion split out for head assays.

### Column charging

The columns were charged with a homogeneous load of ore. There was sufficient ore of each sample to fill one column totally and a second column partially.

CL 1	- ore No. 1	charge	1871 lb	-	848.7 kg
CL 2	- " " "	"	3018 lb	-	1368.9 kg
CL 3	- ore No. 2	"	1767 lb	-	801.5 kg
CL 4	- " " "	"	2711 lb	-	1229.7 kg
CL 5	- ore No. 3	"	1891 lb	-	857.7 kg
CL 6	- " " "	"	2652 lb	-	1202.9 kg
CL 7	- ore No. 4	"	1360 lb	-	616.9 kg
CL 8	- " " "	"	2481 lb	-	1125.4 kg



Operating schedule

All columns were initially wetted with neutral pH water. Acid additions to the barren solution were then made according to the pH of the pregnant solutions produced. Ores No. 1, 2 and 3 required the addition of 2 g H<sub>2</sub>SO<sub>4</sub> per liter for some time until the pregnant pH lowered sufficiently to enable inoculation with the leaching bacteria. Ore No. 4 never required any acid and proved to be acid producing. The acid content of the barren solutions for the various columns is illustrated in Figure 1.

Barren solution was applied at a rate of 25 ml per minute, equivalent to a rate of 10/h/m<sup>2</sup>.

Analytical Methods

Dissolved metal values were determined with an atomic absorption spectrometer. pH values were measured on a Radiometer pH meter. Head and tail assays were performed by wet chemical digestion.

RESULTS

Extraction

Head assays on the four samples showed a significant variation in grade.

Ore 1	0.155% Cu
2	0.513% Cu
3	0.085% Cu
4	1.204% Cu

However, tail assays on the residues from the columns holding the first three ores samples gave the following residual values.

Ore 1	0.167% Cu
2	0.524% Cu
3	0.271% Cu

Repeat of the head assays on the original ore samples gave similar results so that it must be concluded that the headsample, particularly that of ore No. 3 was not representative of the ore in the column.

Although the tail assays are higher than the head assays for ore No. 1, 2 and 3, only the No. 3 ore shows an excessive difference. Since repeats of the head assays confirmed the earlier results, it is indicated that the sample taken from the No. 3 ore was not representative.

There is no evidence to suggest however that the head assays from the No. 1, 2 and 4 ores are not representative.

Based on the amount of copper dissolved during the leaching period and the tail assay values, head assays and extraction after 182 days of leaching were calculated.

Ore No. 1 charge 1368.9 kg

Residual 0.167% Cu = 2286.10 g  
 Extracted 213.58 g

Total 2499.68 g

Head assay  $\frac{2500}{13681} = 0.183\%$

Extraction:  $\frac{213.6}{25.00} = 8.54\%$

Ore No. 2 charge 1229.7 kg

Residual 0.524% Cu = 6443.6 g  
 Extracted 72.7 g

Total 6516.3 g

Head assay  $\frac{6516}{12297} = 0.530\%$

Extraction:  $\frac{72.7}{65.16} = 1.12\%$

Ore No. 3 charge 1202.9 kg

Residual 0.271% Cu = 3259.80 g  
 Extracted 70.25 g

Total 3330.05 g

Head assay  $\frac{3330}{12029} = 0.277\% \text{ Cu}$

Extraction  $\frac{70.25}{33.30} = 2.11\%$

The above extractions are reasonably accurate since they are based on tail assay and solution assays.

For the No. 4 ore, which is still under leach, we have only a head assay of 1.20% copper. Based on this head assay and the 2434 g extracted after 182 days, the extraction after 182 days was:

charge:	1125.4 kg	
Head assay	1.20% Cu = 13505 g	
Extracted	2434 g	
Extraction	$\frac{2434}{135.1}$	= 18.0%

The extraction profiles for the four ore types are illustrated in Figure 2.

The extraction process is normally evaluated as a 2 phase process. Phase 1 represents the extraction of the acid soluble copper and phase 2 the biological extraction process. Figure 2 shows that in contrast to ores No. 2 and 3, ores No. 1 and 4 have a significant acid soluble copper content and that their long term extraction should be calculated from the linear phase of their extraction curves.

Obviously, the performance of ores No. 2 and 3, at indicated extraction rates of approximately 2 and 4% per year is very poor and does not justify consideration of these ores for economic application.

Ore No. 4 however has leached well and has reached 18% extraction in one half year. The linear extraction rate is 8% in 100 days or 29% per year, representing a production of 3,500,000 kg/10<sup>6</sup> tonnes of 1.2% ore. A preliminary economic analyses (appendix) submitted on June 2/78 shows that this ore may be leached profitable. At a production of 6,750,000 lb of copper per year the yearly operating income will vary from \$2,186,000 with copper selling at 60¢/lb to \$4,946,000 with copper at 90¢/lb.

Ore No. 1 is leaching at a reasonable rate, considering its low copper content. Its linear extraction rate is approximately 15% per year. Due to its low head grade however, this rate represents a production of only 280,000 kg/10<sup>6</sup> tonnes/year. However, since this sample represents only the top surface of the dump, conceivably natural leaching has removed the more readily leachable copper minerals from the sample. The interior of the dump may contain higher grade ore which would produce better extraction.

### Acid consumptions

While ore No. 4 never consumed any acid, the other 3 ore types did. Ores 2 and 3 consumed very little acid once the initial gangue was satisfied. While the barren feed was applied at pH2, the pregnant solution pH reached as low as 2.2 at the termination of the test, indicating that eventually these two ores might become acid producers.

Ore No. 1 however had a high initial acid demand. It took 150 days of leaching with 2 g/l sulfuric acid before the pregnant pH reached below 3.0. At that time, approximately 8-10 kg of acid had been consumed and only 200 grams of copper extracted. Subsequently, the pregnant pH decreased to a value of 2.5 at the time of termination indicating that it is possible that during a long term leach the ore could have become acid producing. Because the sample was obtained from the surface of the dump, most of the acid producing minerals probably were removed by natural leaching processes. Since the dump is an acid producer, as proven by its acidic effluent, it is likely that ore from a lower elevation in the dump would leach considerably better than this sample.

### Particle breakdown

The results of the sieve analyses performed on the head sample and the column residue (Table 1) indicate that significant particle breakdown occurred during leaching on ore No. 1.

Ore No. 1 had 80.9% + 1" particles before leaching and 63.1% upon termination. Due to the particle breakdown, the -4 mesh fractions increased from 6.9% to 17.1%.

Ore No. 2 did not show a significant breakdown of large particles into smaller ones.

Ore No. 3, which had a very high percentage of +1" material (91.5%) showed some breakdown (84.3% in the residue) and an increase in -4 mesh particles from 1.2% to 3.1%.

It is indicated by these results that the amount of particle breakdown is related to the amount of leaching that took place. We expect to find a very high breakdown rate in the No. 4 ore which currently is still under leach.

### Pregnant solution grades

During operation of the column tests, percolation of barren solution through approximately 18 ft of ore produced pregnant solutions holding as little as 5 mg/l copper and as much as 5500 mg/l.

Ore No. 1 initially produced pregnant solutions holding up to 190 mg/l copper but they eventually levelled off at a steady 25-28 mg/l.

Ore No. 2 did not show an initial high solution grade. Its pregnant solution slowly increased from 2 to 18 mg/l over the first 70 days of leaching, after which it held constant at 15-20 mg/l.

Ore No. 3 did not have an initial high grade either and took 75 days to reach 10 mg/l copper in solution. Between days 75 and 180 the solution grade varied between 10 and 20 mg/l.

Ore No. 4 was outstandingly different from the other three ores. During the first day of leaching it produced pregnant solutions holding 5500 mg/l copper. During the first 6 days this grade decreased slowly to 1000 mg/l, then to 500 mg/l by 15 days of leaching. The grade held steady at 400-800 mg/l till 100 days, then slowly decreased to 250 mg/l by 160 days. This grade was then maintained for the duration of the test.

### Effect of rest cycles

Since ores No. 1, 2 and 3 produced grades from 18 ft of ore sufficiently low to suggest that 100 ft high leach dumps would not produce high enough pregnant solution grades to warrant recovery, a series of rest - leach cycles were used to increase solution grades.

When an actively leaching body of ore is put on rest, i.e. barren solution is not applied, the remaining liquid in the orebody often enables the leaching bacteria to continue the sulfide oxidation process. As a result, copper and iron sulfates are produced during a rest period and these salts can accumulate to high concentration in the remaining static water content of the orebody. Subsequent washing with barren solution then removes the salts produced and thus give a pregnant solution with high metal sulfate content.

Such leaching during rest cycles can only take place however if the bacterial sulfide oxidation produces sufficient sulfuric acid to offset the acid demand of the gangue minerals.

Ore No. 1, using a 4 day rest cycle followed by 3 days of washing produced a grade of 40 to 25 mg/l copper, over the 3 day wash period, from 848 kg of ore (11.1 ft ore column) while a column with 1369 kg of ore (18.0 ft) produced 30 mg/l pregnant solution over a representative time period. Thus, the rest cycle did serve to increase the average pregnant solution grade.

Similarly, 801 kg of No. 2 ore (11.7 ft ore column) produced pregnant solutions holding 36-20 mg/l copper while 1230 kg of ore produced 15-20 mg/l over a similar time period. Ore No. 3 gave similar results when subjected to rest periods but neither of these ores were able to produce high enough grades to warrant commercial application.

Overall extraction of all three ores, when subjected to rest periods, was less than when leached continuously.

Ore No. 4 did give significantly higher pregnant grades when leached with rest cycles then when leached continuously. 617 kg of No. 4 ore (9.9 ft ore column) gave pregnant grade between 700 and 300 mg/l while 1125 kg of No. 4 ore, when leached continuously, produced grades of 250-150 mg/l copper. Although initially the column subjected to rest cycles lacked behind, eventually the rested column produced copper at the same rate as the continuously leached column. This strongly suggests that the rest periods are not effective until most of the acid soluble metal content has been removed, which of course is a process dependent upon chemical principles and thus the supply of sulfuric acid.

#### Effect of added ferric iron

Dissolved ferric iron levels in the pregnant solution from ore No. 2 were less than 100 mg/l. To determine if increased ferric iron levels would improve extraction, on day 134 the No. 2 ore was leached with barren solution from a small can precipitation plant which treated pregnant solutions from ore No. 4. Dissolved ferric iron levels increased rapidly to as much as 1900 mg/l but over a period of 30 days, no significant improvement in extraction was noticed.

Similarly, ore No. 3 was leached with canplant barren between days 125 and 160. Ferric iron levels in the pregnant solutions increased from 150 mg/l to 2000 mg/l but a significant increase in extraction was not obtained.

Although ore No. 4 was leaching very well, its leach solution was changed from water to canplant barren between days 125 and 160 to determine if increased ferric iron levels would have any effect at all. This procedure raised ferric iron levels from 1100-2000 mg/l to up to 4000 mg/l. As with the other ore types the results of the test did not suggest a significant increase or decrease in leachrate.

### Shakeflask test on +4" ore samples

In early February 1978 the poor performance of the No. 2 ore suggested an additional investigation into the characteristics of the ores in the columns. Since all information at that time suggested that the +4" fractions of the ores had a head assay similar to the total ore samples, several 4" rock pieces of each ore type were crushed, pulverised and ballmilled to -400 mesh. Small samples of the ballmilled material were leached in shakeflasks.

Both ore No. 4 and No. 1 leached well, ore No. 4 became a strong acid producer after only 40 hours of leaching while ore No. 1 did not require acid after 46 hours but did not become a strong acid producer. The No. 1 ore sample had a head assay of 0.08% copper and gave 50% extraction at an acid consumption of 0.7 kg H<sub>2</sub>SO<sub>4</sub> per kg of copper extracted while the No. 4 ore sample contained 0.65% copper and gave 93% extraction at an acid consumption of 1 kg/kg of copper extracted.

Both ores No. 2 and 3 required constant acid additions during the first 370 hours of leaching and required 43 and 196 kg of acid per kg of copper extracted respectively. Ore No. 2 head assayed 0.19% copper and eventually gave 52% extraction, ore No. 3 contained 0.036% copper and gave 44% extraction.

### Leach residue observations

Upon completion of the test on the No. 1, 2 and 3 ores, the content of each column was visually evaluated. It was noticeable that all three ore types contained many particles with a lacquer type coating which was hard and appeared non penetratable to the leach solution. In many cases this coating covered the cracks and fractures in the rocks, thus possibly preventing the leach solution from entering the fracture plains.

Analyses with a scanning electron microscope by Mr. Arvid Lacis of the metallurgy department of U.B.C. proved this material to be calcium sulfate. (Figure 3A & 3B)

### Discussion

With the exception of ore No. 4, the extraction obtained during this 6 month long column test are sufficiently low to prevent the making of conclusions regarding the amenability of the ores to the microbiological leaching process. The very low extraction on ores No. 2 and 3 suggests however that commercial application of the biological leaching process is not feasible. The performance of ore No. 2 is particularly disappointing since the sample tested contained as much as 0.5% copper. The poor performance of ore No. 3, which contained approximate 0.27% copper was not expected either. The fact that ore No. 2 required 2g/l and the No. 3 ore 1-2 g/l acid in their barren solutions and produced pregnant solutions containing 100 mg/l or less total iron suggest that most, if not all the copper extracted was due to chemical leaching rather than biological leaching. This would also explain why the ore leached less well when subjected to rest cycles since no fresh acid is made available during the rest cycles. At no time was there any indication that this ore became, or was about to become, an acid producer.

The acid consuming character of these 2 ores, as further confirmed by the shakeflask tests on +4" material, may have made it impossible for the leaching bacteria to properly develop and grow on the ores. We have strong indications from other studies that the bacteria will not grow when acid consuming chemical reactions take place. Not until such acid consuming demand is satisfied will the bacteria take hold and actively start to oxidise sulfides and ferrous iron.

Although the No. 1 ore gave a better extraction than ores No. 2 and 3, it also proved to be a strong acid consumer and not until after 160 days did the pH of its pregnant solutions drop below 3.0. It is highly unlikely that a proper biological environment was ever established in the column. Since this sample was taken from 1 foot below the top surface of the dump, it is likely that most of the leachable sulfides, particularly the readily leachable ores, had already been leached by the rain fall over the many years this dump has been in existence. There must be less acid demanding material at lower elevation in the dump since acidic drainage emanates from its toe. Consequently, it appears that the sample tested was not truly representative of the mineralization in the dump. The significant particle breakdown that occurred during this test suggest that the ore should be amenable to the leaching process and consideration should be given to leaching the dump, probably on an experimental basis, or as an annex to an operation treating the No. 4 ore. The results of the column test on ores No. 1, 2 and 3 indicate that little biological action took place and that most of the copper solubilised was the result of chemical action rather than biological action. The calcium sulfate coating noticed on many rock particles is a direct result of the alkaline gangue content of the ores and probably prevented direct contact between part of the mineral surfaces and the leach solutions.

It may be possible to prevent the formation of calcium sulfate coatings by initially leaching the ore with a strong acid solution so that its acid demand can be satisfied rapidly. Subsequent inoculation with the leaching bacteria may then give improved extraction. Such a process can probably be readily evaluated on small quantities of crushed ore. Since the shakeflask test on the No. 1 ore showed that the No. 1 ore is amenable to biological leaching, a test on some crushed ore, preferably from inside the dump, should be made to determine how much acid will be required to initiate the biological leaching process.

The enhancing effect of rest cycles did not show up on ores No. 1, 2 and 3, mainly because of the limited biological activity during the test. Only when the bacteria actively oxidise sulfides does the rest cycle produce copper at a rate similar, or better, than when leached continuously. The noted high acid consumption of the ore during this 6 month test would have caused the remaining water film around the rock particles to lose sufficient acid to raise its pH to undesirable levels. It is possible that once the acid demand would have been satisfied, the rest periods would become effective.



Only on the No. 4 ore did the column using rest cycles produce similar extraction rates as the column leached continuously, but only after 150 days. Prior to that, the rest cycle produced a leachrate of only 80% of that of the continuously leached column.

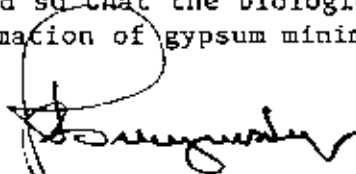
Ore No. 4 leached extremely well and produced copper at a rate of 8% per 100 days, equivalent to an extraction of 3.5 million kg of copper per year per million tonnes of ore. An preliminary economic feasibility study for leaching 15 million tonnes of this type of ore, assaying 0.3% copper, has been made and is appended to this report.

### Recommendations

It is recommended that the quantity of ore available at the mine represented by sample No. 4, be established. The amenability of the ore to the leaching process can be readily determined in a series of shakeflask tests in where the No. 4 ore types should leach fast and without acid consumption, in contrast to the other 3 ore types.

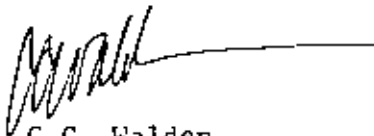
The two columns holding the No. 4 ore should be allowed to continue to leach until extraction deteriorates so that a proper extraction profile can be established for this ore.

Since it is likely that the No. 1 ore will be better amenable to the leaching process than indicated by the sample tested, a more representative sample should be obtained and leached under both standard conditions and after prior treatment with 5-10 g/l acid so that the biological environment can be established rapidly and the formation of gypsum minimised.



A. Bruynesteyn  
Manager, Mineral Leaching Studies  
Division of Applied Biology

On behalf of B.C. Research



C.C. Walden  
Head, Division of Applied Biology

## REFERENCE

Assessment of the leaching characteristics of two samples of leach ore from the Mount Washington Mine. Project Report No. 1, October 1977.

## TABLE

Screen, Head and Tail Assays.

## FIGURES

- 1 Acid Content of Barren Solution.
- 2 Extraction Profiles for the 4 Mount Washington Ores.
- 3A Scanning electron microscope picture of gypsum crystals.
- 3B Scanning electron microscope response graph.

## APPENDIX

Preliminary Economic Analyses of the Mount Washington Leaching Project.

T A B L E

TABLE 1  
 SCREEN, HEAD AND TAIL ASSAYS  
 Ore No. 1 (M.L.)

	Head Assay			Tail Assay		
	% b.w.	% Cu	Fraction	% b.w.	% Cu	Fraction
+4 in.	18.24	0.09	0.016	14.7	0.10	0.015
-4 in. +2 in.	40.19	0.15	0.060	29.7	0.15	0.044
-2 in. +1½ in.	12.54	0.14	0.018	8.9	0.14	0.013
-1½ in. +1 in.	9.96	0.18	0.018	10.3	0.19	0.020
-1 in. +¾ in.	4.45	0.26	0.011	5.2	0.26	0.014
-¾ in. +½ in.	3.66	0.16	0.006	5.7	0.16	0.009
-½ in. +⅜ in.	1.64	0.22	0.004	3.2	0.23	0.007
-⅜ in. +4 mesh	2.44	0.19	0.004	5.2	0.19	0.002
-4 mesh +8 mesh	1.29	0.20	0.003	3.3	0.20	0.007
-8 mesh +18 mesh	1.47	0.27	0.004	4.5	0.27	0.012
-18 mesh	4.12	0.26	<u>0.011</u>	9.3	0.26	<u>0.024</u>
Mean:			0.155			0.167

ORE No. 2 b.w.

	Head Assay			Tail Assay		
	% b.w.	% Cu	Fraction	% b.w.	% Cu	Fraction
+ 4 in.	24.41	0.23	0.056	12.5	0.22	0.028
-4 in. + 2 in.	28.73	0.37	0.106	39.4	0.51	0.201
-2 in. + 1½ in.	13.11	0.35	0.046	12.1	0.44	0.053
-1½ in. + 1 in.	10.00	0.80	0.080	12.1	0.44	0.053
-1 in. + ¾ in.	5.19	0.84	0.043	5.1	0.51	0.026
-¾ in. + ½ in.	5.09	0.77	0.030	5.1	0.62	0.032
-½ in. + ⅜ in.	2.40	0.71	0.017	2.2	0.76	0.017
-⅜ in. + 4 mesh	3.47	0.95	0.035	3.5	0.79	0.028
-4 mesh + 8 mesh	1.78	0.91	0.018	1.9	0.91	0.017
-8 mesh +18 mesh	2.02	1.02	0.021	2.2	0.84	0.018
-18 mesh	3.80	1.60	<u>0.061</u>	3.9	1.34	<u>0.052</u>
Mean:			0.513			0.524

ORE No. 3 b.w.

	Head Assay			Tail Assay		
	% b.w.	% Cu	Fraction	% b.w.	% Cu	Fraction
+ 4 in.	33.3	0.03	0.010	17.7	0.41	0.073
-4 in. + 2 in.	37.23	0.08	0.030	48.0	0.18	0.087
-2 in. + 1½ in.	13.0	0.08	0.010	9.0	0.17	0.015
-1½ in. + 1 in.	8.04	0.09	0.007	9.6	0.14	0.013
-1 in. + ¾ in.	3.35	0.08	0.003	4.5	0.26	0.012
-¾ in. + ½ in.	2.32	0.30	0.007	3.9	0.22	0.009
-½ in. + ⅜ in.	0.77	0.49	0.004	1.8	0.48	0.009
-⅜ in. + 4 mesh	0.84	0.23	0.002	2.4	0.35	0.008
-4 mesh + 8 mesh	0.30	0.69	0.002	0.8	0.78	0.006
-8 mesh + 18 mesh	0.25	0.88	0.002	0.6	1.43	0.009
-18 mesh	0.61	1.35	<u>0.008</u>	1.7	1.81	<u>0.031</u>
Mean:			0.085			0.271

F I G U R E S

Figure 1  
ACID CONTENT OF BARREN SOLUTION

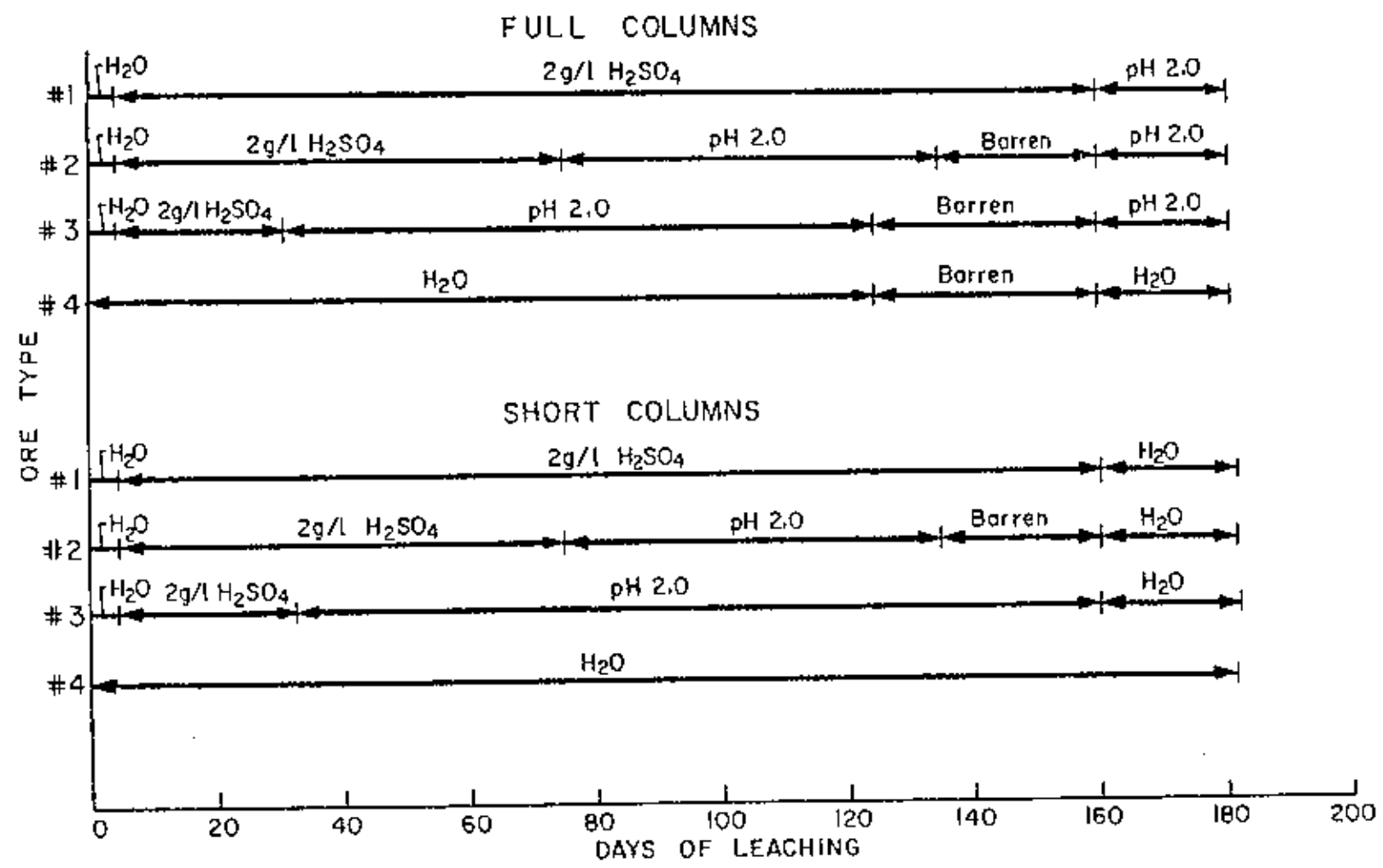
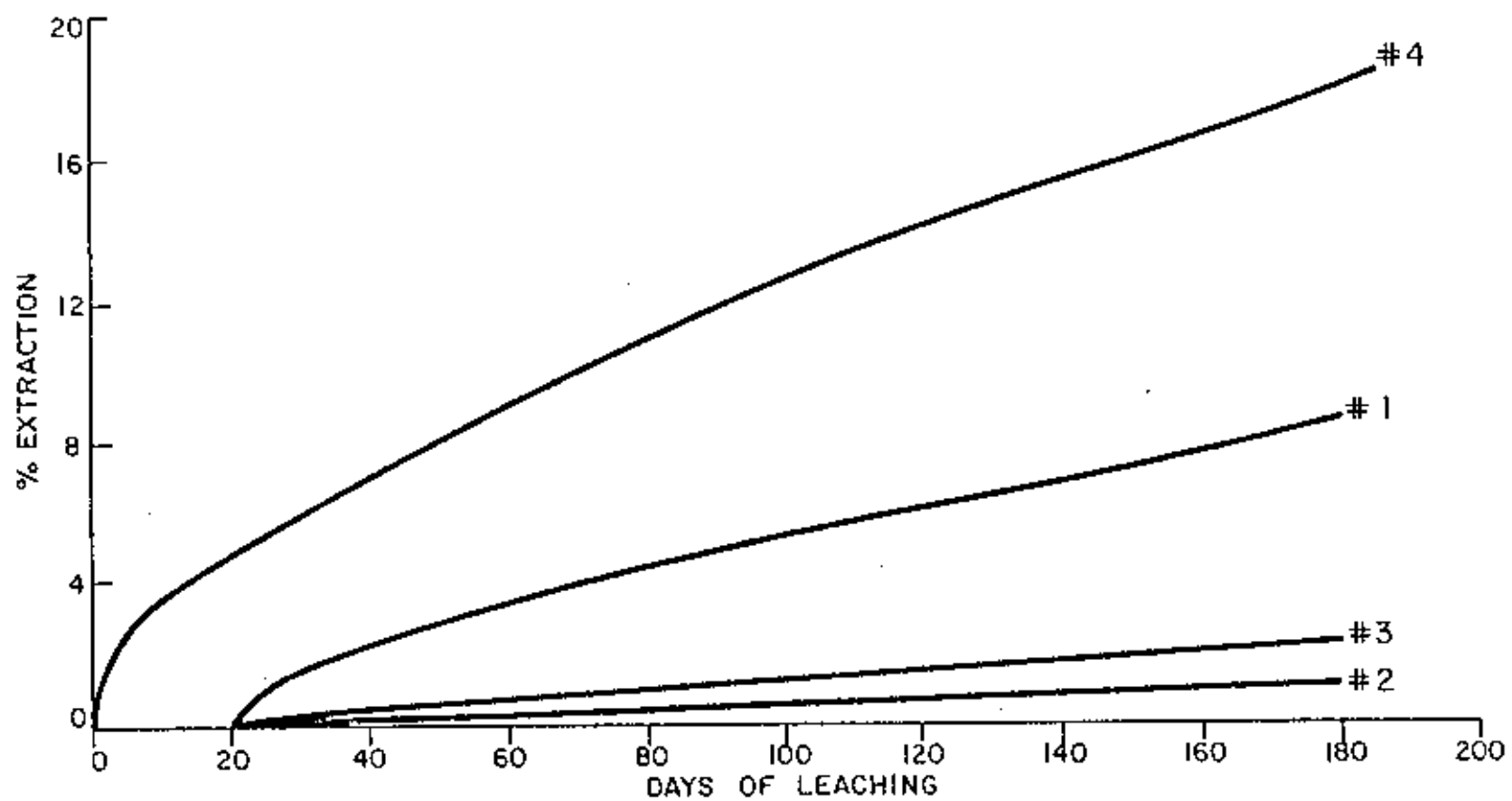




Figure 2  
EXTRACTION PROFILES FOR THE 4 MT. WASHINGTON ORES



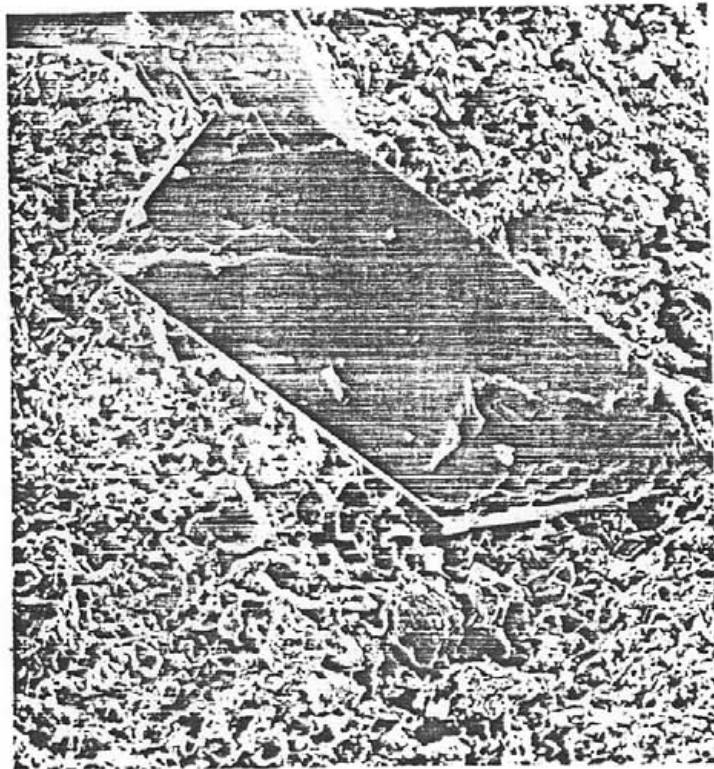


Figure 3A

Scanning electron microscope picture of gypsum crystals

Fig 3B. Scanning electron microscope  
response graph.

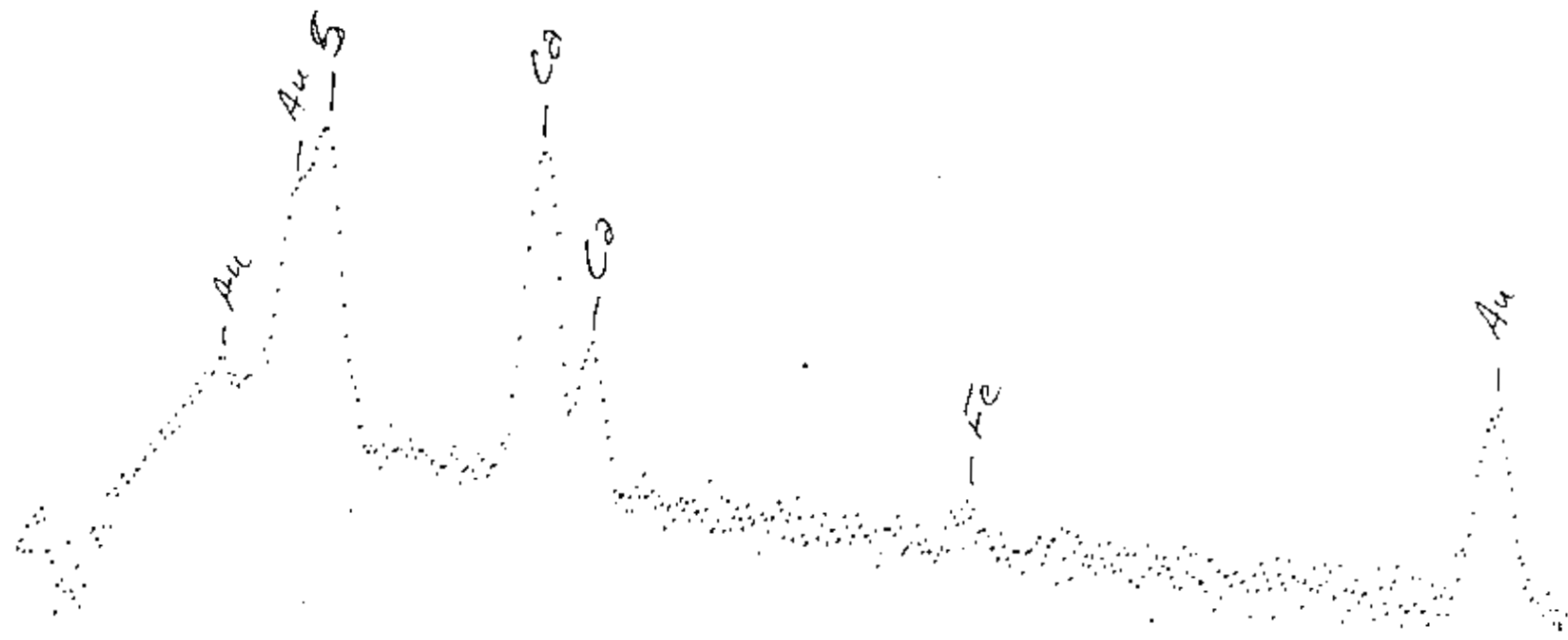
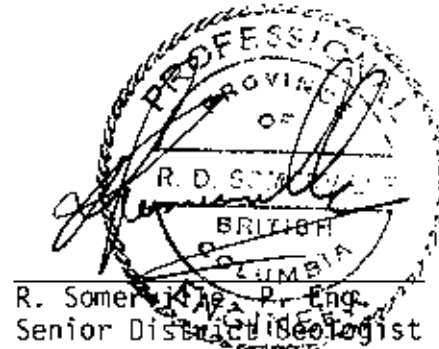


Fig 3B

## STATEMENT OF QUALIFICATIONS

The author of this report Mr. A. Bruynesteyn obtained his Bachelor's Degree in Chemical Engineering from the H.T.S. of Amsterdam in 1958. After nine years of engineering work in the distilling industry, Mr. Bruynesteyn went with B.C. Research to supervise studies in biological and chemical leaching of sulphide minerals. He is now the Manager of this Division of B.C. Research. His experience in the field is extensive and varied, involving large, practical research programs for such companies as Anaconda Copper Company and Cia Minera de Cananea. B.C. Research and Mr. Bruynesteyn are world recognized authorities on the subject of biological leaching.



R. Somerville, P. Eng.  
Senior District Geologist

COST STATEMENT

Re: Mt. Washington Prospect

Expenditures From Sept. 15, 1977 to Sept. 14, 1978

Wages:       Geologist (R. Somerville)

Office:	September	15,16,22,23,	1977		
Office:	October	11,12,17,21,	1977		
Field:	October	18, 19	1977		
Office:	November	9,10,14,23,	1977		
Office:	December	2,13,14,15,16,	1977	30 days @ \$200	\$ 6,000.00
Office:	April	10,11	1978		
Office:	May	23,24	1978		
Office:	June	1,2,	1978		
Office:	July	19,20,21,26	1978		
Office:	August	13,	1978		

Technologist

Office:	October	14,	1977		
Field:	October	16 to 24 incl.	1977	12 days @ \$85	1,020.00
Field:	November	9,10	1977		

Contracted Services:

October 18 to Oct. 23, 1977:					
Sample four sights- Drill; blast; sample					
Supply labour and equipment and misc.					
supplies					3,091.29
December 1, 1977 to August 31, 1978:					
Process and analyses of four samples					
(Re B.C. Research Council)					30,175.41

Food & Accommodation:

October 16 through Oct. 24, 1977					
28 man days @ \$26.78 per man day					749.77

Materials and Supplies:

Sampling supplies and equipment					532.07
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Cost Statement (cont'd.)

Transportation Costs:

Oct. 16 to Oct. 25, 1977		
Four wheel drive truck: 10 days	\$191.78	
Ferry Fees	27.00	
Gasoline and Oil	182.87	
Sample Transport-Courtenay to Vancouver	521.05	\$ 922.70
		<hr/>
Assessment Report Preparation (other than B.C. Research)		
R. Somerville, 1 day @ \$200		200.00
Stenographic, 1 day @ \$85		85.00
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Total Expenditures		\$ 42,776.24
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