86-807-15369

REPORT

ON

PRELIMINARY SAMPLING
AND METALLURGICAL TESTING

OF

TAILINGS AND STOCKPILES
ON THE
LAKE FR. CROWN GRANT

&

SURF INLET PROPERTY
PRINCESS ROYAL ISLAND, BRITISH COLUMBIA

MINING DIVISION: SKEENA
NTS 103H/2W
LATITUDE: 53°05.6'
LONGITUDE: 128°53'

OWNER(S)
MATACHEWAN CONSOLIDATED
MINES, LIMITED

OPERATOR(S)
TRM ENGINEERING LTD.

BY:
J.T. SHEARER, M.Sc., FGAC
G. HAWTHORN, P.Eng.
J.F. HARRIS, Ph.D.
S. GARDINER, B.Sc., FGAC

DATE SUBMITTED:
DECEMBER 16, 198[illegible]

WORK CARRIED OUT:
Field Work
November 6 - November 11, 1985
Metallurgical Testwork
December 10, 1985 - February 4, 1986
Mineralogical Study
January 8-21, 1986
Reports compiled: September 15-16, 1986

CONTENTS

LIST OF ILLUSTRATIONS

SUMMARY

1. The Surf Inlet Area is on Princess Royal Island, approximately 160 km southeast of Prince Rupert and has been a major source of gold, silver and copper in the past. The main period of operation was between 1917 and 1926.

2. The Surf and Pugsley mines produced to the end of 1942 a total of 1,091,131 tons (495,068 tonnes) from which were recovered 382,351 ounces (11,891,116 g) of gold, 208,752 ounces (6,492,187 g) of silver and 6,314,341 pounds (2,864,185 kg) of copper. The average head grade was 0.425 ounces per ton (13.8 g/tonne) gold. Mill recoveries averaged between 88% and 92%.

3. The ore was mined from underground workings. Access below 900 level is from internal inclined shafts. The entire operation was electrified from a nearby lowhead hydroelectric plant.

4. Gold mineralization is localized along an extensive, complex shear system that cuts gneiss and diorite. Gold associated with pyrite occurs in quartz-ankerite-sericite-sulphide veins.

5. Low grade stockpiles are located outside the 550 level of the Surf Mine. Two large, shallow trench samples were collected during the present program. These assayed; West dump: 0.151 oz/ton (4.90 g/tonne) Au and East dump: 0.067 oz/ton (2.18 g/tonne) Au. There are approximately 400,000 tons (181,488 tonnes) of dump material around the 550 level portal as estimated by Freeze (1981).

6. Sampling by Cominco in 1981 on the 550 level stockpiles gave the following results: West dump: 0.102 oz/ton (3.31 g/tonne) Au, East Dump: 0.051 oz/ton (1.66 g/tonne) Au.

7. Several very large samples were collected of the tailings from previous milling operations of which the average assay result of nineteen large samples was 0.061 oz/ton (1.98 g/tonne) gold.

8. The samples collected during the present program were submitted for metallurgical testing and petrographic examination to evaluate gold recoveries by various techniques. A summary of the results are:

SUMMARY (cont'd)

(i) the sulphide mineralogy consists predominantly of pyrite with minor chalcopyrite;
(ii) gold occurs primarily in a homogenously dispersed, submicroscopic form in pyrite;
(iii) flotation will produce a concentrate containing 4 to 6 oz gold/ton (129.92 to 194.88 g/tonne);
(iv) poor recovery is achieved by cyanidation, which reflects the mode of occurrence of gold. Alternative hydro metallurgical techniques should be investigated to determine whether overall recovery relative to cyanidation can be improved.

9. A large volume of stockpile and tailings material containing important gold values is present at the Surf Inlet Minesite. More detailed sampling is warranted on both the tailings and stockpiles.

10. The size and distribution tailings area and contained gold values should be evaluated more fully by:

(a) air photograph analysis of the Paradise Creek delta and braided lower creek areas;
(b) shallow coring of the delta and lower creek on a regular grid basis using a lightweight core drill or auger;
(c) shallow coring of the swamplands adjacent to the lower reaches of Paradise Creek.

11. The size and distribution of 550 level Surf Mine stockpiles and contained gold values should be evaluated more fully by:

(a) excavating close spaced hand trenches along the top and edges of each dump, ensuring large, unbiased samples are collected;
(b) accurately surveying the extent of each dump and estimating the probable thickness;
(c) evaluating the railbed sections between Surf and Pugsley Mines.

12. The existing 1:20,000 air photography should be compiled into a 1:1,000 orthophoto base map with 2 m contours.

INTRODUCTION

A preliminary sampling program was conducted in November 1985 to obtain representative material from mill tailings and mine stockpiles at the Surf Inlet Mine on Princess Royal Island.

The Surf Inlet-Pugsley Mines are the seventh largest source of gold in British Columbia, having produced a total of 1,091,131 tons (495,068 tonnes) of ore with overall recoveries of 0.350 ounces (11.37 g) gold, 0.18 ounces (5.85 g) silver and 0.29% copper per ton. Mill recoveries were approximately 88%-92%. Only Bralorne-Pioneer, Rossland, Premier, Nickel Plate-Mascot, Island Mountain-Cariboo Gold Quartz and Phoenix mines have produced more gold.

Very little published geological information is available on the Surf Inlet area or the ore bodies which were mainly mined in the period 1917-1926. However, a detailed collection of private reports, plans and sections documenting the geological and extraction processes have been saved by the present owners; Matachewan Consolidated Mines Ltd.

Gold mineralization is localized along an extensive, complicated shear system that has developed in intrusive and gneissic volcanics and metasediments of the Coast Plutonic Complex (see map in pocket). Gold associated with pyrite occurs in quartz-ankerite-sericite-sulphide veins. Distribution of ore shoots within the veins depend on late stage fault adjustments and flexures during which veins along certain shear surfaces and zones were fractured and mineralized.

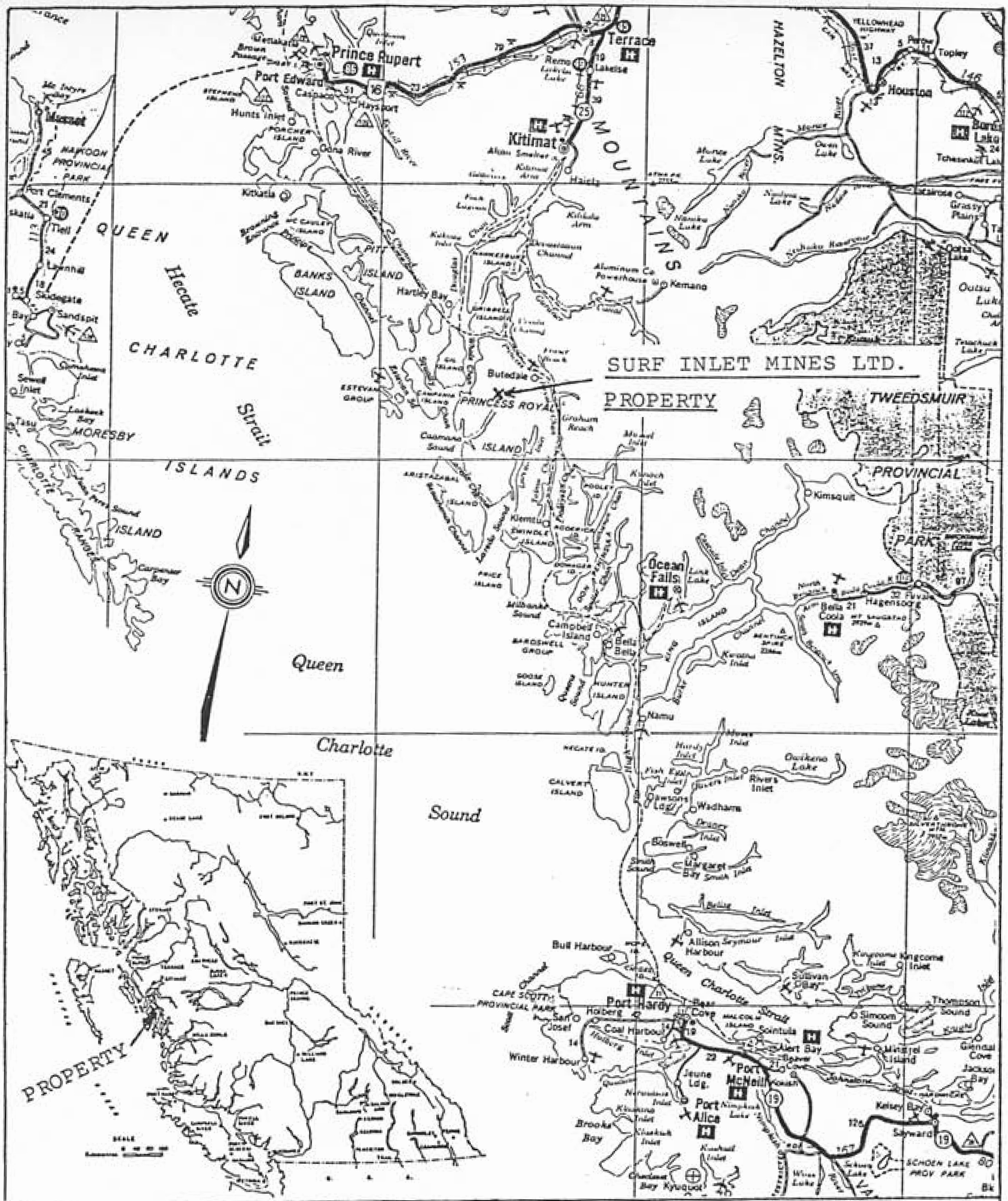
Current interest is focussed on applying modern metallurgical leaching processes for enhancing gold recovery from dump material, exploration of down-dip extensions of the Pugsley and Surf ore bodies and the possibility of new ore zones south of the Pugsley workings as suggested by surface mineralization.

LOCATION AND ACCESS

The Surf and Pugsley Mines are located near the head of Surf Inlet on Princess Royal Island approximately 160 kilometers southeast of the main supply base at Prince Rupert. The property is at 53° 05' N latitude and 128° 53' W longitude in mapsheet NTS 103 H/2W about 105 km southwest of Kitimat and 115 km northwest of Bella Bella. The nearest sizeable community is Hartley Bay, 44 km northeast. The docking facility at Butedale on the east coast of Princess Royal Island is a port of call for ships travelling the "Inside Passage" between Vancouver and Prince Rupert. Butedale is 16 km east of the Surf Inlet minesite. Ocean-going ships were able to call on the wharf at the head of Surf Inlet when the mines were in production. Currently the most active center of mineral exploration near Surf Inlet is Trader Resource Corp.'s gold project on Banks Island, 90 km to the northwest.

The Surf and Pugsley ore bodies, located on the north and south sides of Paradise Creek, are 11 km from the wharf and hydro-electric power site at the outlet of Cougar Lake. In the past, electric tramways and barges formed the supply link from the mines to tidewater. A tug and barge carrying fifteen 1-ton mine cars operated on the lake. At the mouth of Paradise Creek an overhead trolly electric railroad ran to the camp on an even grade. An incline from the ocean dock to the lake, a distance of 314 feet, and equipped with an electric hoist completed the transportation. Fixed wing aircraft with floats can land on Paradise Lake and a short foot-trail connects Paradise Lake to the minesites.

Topography in the area is very rugged with steep sided peaks rising to a maximum elevation of 1100 m ASL. The lowest level in the Pugsley Mine is the 1500 level which is 500 feet (152 m) below sea level. The lowest level on the Surf Mine is the 1400 level and is 275 feet (84 m) below sea level.



SURF INLET MINES LTD.

SURF INLET PROPERTY, B. C.

LOCATION MAP

FIGURE 1

Scale 1 : 2,400,000

PROPERTY AND TITLE

The property, as shown in Figure 2, consists of the following mineral tenure:

(a) Crown granted mineral claims (a total of 21 claims have been optioned from owner, Matachewan Consolidated Mines, Limited)

Bee	Lot 1915	Lake Fr.	Lot 32
Bench	35	Lakeview	229
Bluebell	2485	Marcia	2484
Bluff	34	Mountain Fr.	37
Cassie	228	Olive	227
DLS	31	Princess Royal	7
Excelsior	9	Sadie	8
Granite	1916	Sea Fr.	1914
Gulch	33	Twin Peaks	38
Independence Fr.	222	UTA Fr.	36
La Quivree	39		

(b) Optioned mineral claims (optioned from owner, Matachewan Consolidated Mines, Limited)

Claims	Units	Rec. Numbers	Expiry Date
Bear 1	15	2221	April 16, 1987
Bear 2	15	2222	April 16, 1987
Bear 3	20	2223	April 16, 1987

Total = 50 units

(c) Optioned mineral claims (optioned from owner, Placer Development Ltd.)

Claims	Units	Rec. Numbers	Expiry Date
Jen 1	20	2693	Nov. 27, 1986
Jen 2	20	2694	Nov. 27, 1986
Jen 3	10	2695	Nov. 27, 1986
Jen 4	20	2696	Nov. 27, 1986

Total = 70 units

PROPERTY AND TITLE (cont'd)

(d) Optioned Reverted Crown granted mineral claims (a total of 11 claims have been optioned from owner, Placer Development Ltd.)

Claims	Lot No.	Rec. Numbers	Expiry Date
Sheet Anchor Fr.	2105	1979	Jan. 14, 1987
Summit	226	1980	Jan. 14, 1987
Bonanza	224	1981	Jan. 14, 1987
Anaconda	223	1982	Jan. 14, 1987
Turner Fr.	221	1983	Jan. 14, 1987
Homestake	21	1984	Jan. 14, 1987
Seagull	2097	1985	Jan. 14, 1987
Little Tomy Fr.	2098	1986	Jan. 14, 1987
Brown Bear	2099	1987	Jan. 14, 1987
Sunlight Fr.	2103	1988	Jan. 14, 1987
Sea Lion Fr.	2104	1989	Jan. 14, 1987

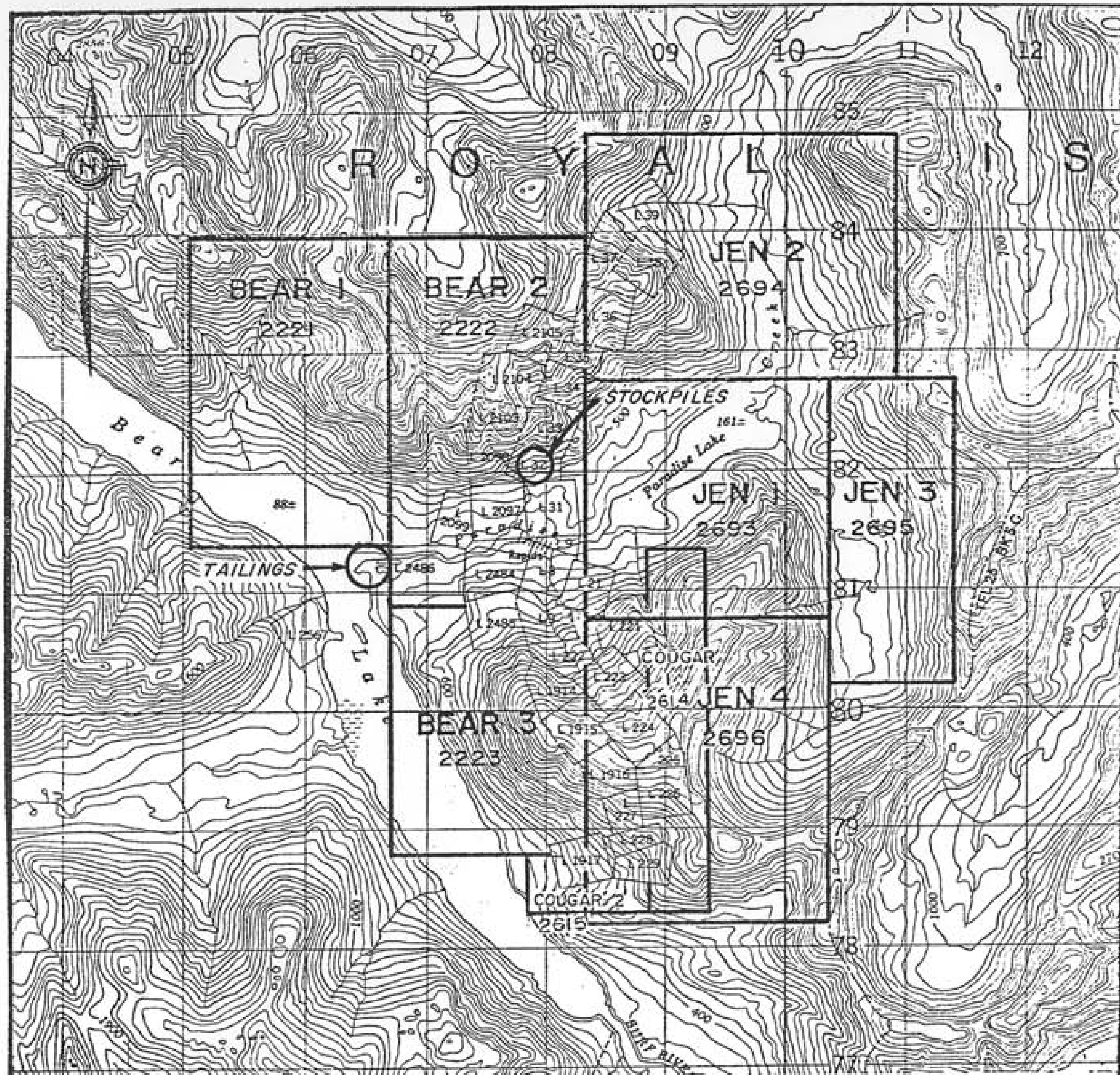
(e) Optioned mineral claims (optioned from owner, Coastoro Resources Limited)

Claims	Units	Rec. Numbers	Expiry Date
Cougar 1	6	2614	October 1, 1986
Cougar 2	2	2615	October 1, 1986

Total = 8 units

Mineral claims in sections (a) to (d) are held under option by T. van Wollen and M. McClaren. The Cougar claims (in section e) are held under option by Fleet Developments Ltd. All work on the claims was done for or by TRM Engineering Ltd.

On February 28, 1986, two 2-post claims and one modified grid system claim, Surf One, Surf Two and Surf Three respectively, were staked. Surf One and Surf Two were staked to cover the tailings area.



SCALE 1 : 50,000

1000 500 0 1000 2000 3000 M

CLAIM & INDEX MAP

PROJECT : SURF INLET

ENG. : TRM ENGINEERING LTD.

FIGURE 2

HISTORY

The original discovery of gold in the Surf Inlet area was made in the late 1800's by tracing white quartz float from the bottom of the valley which enters Bear Lake from the east, up to where the vein outcrop on the north and south sides of the valley. The first claims were located in 1898 and are the oldest in the Skeena Mining Division exclusive of the Queen Charlotte Islands (McConnell, 1914).

Trial shipments of the ore were first made in 1902, and although these yielded excellent values in gold (about 5 oz per ton) and copper (about 3%), subsequent work was discouraging (Roddick, 1970). There is no record of the tonnage or value produced in this period and some doubt arose as to the average grade of the ore. Activity on the property remained at a low level until 1912 when a more vigorous development program began. The property was initially known as the "D.L.S. Group" and was owned by Surf Inlet Mines Limited who optioned them to the Belmont Canadian Mines Ltd. in March 1914. The Belmont Canadian Mines Ltd., a subsidiary of Tonopah-Belmont Development Company, developed and bought the property by reorganizing into the Belmont-Surf Inlet Mines Ltd. The property produced continuously from September 1, 1917 to June 30, 1926. Records show that 848,883 tons (385,156 tonnes) of ore were produced from which 322,297 oz (10,023,437 g) of gold, 176,734 oz (5,496,427 g) of silver and 5,244,772 pounds (2,379,030 kg) of copper were recovered (Dolmage, 1946).

The 1918 Minister of Mines Annual Report indicates a mill recovery of 92%. Dolmage (1946) reports for the period 1916-1926:

> During that period, 848,883 tons of ore were mined, of which 57,632 came from the Pugsley. The average grade of this ore was 0.425 ounces of gold, 0.30 ounces of silver and 6 pounds of copper per ton. The maximum daily production was 400 tons and the average operating costs were $5.20 per ton. To the end of 1925, detail records show that from 822,233 tons of ore mined, 307,452.9 ounces of gold; 169,348 ounces of silver and 5,083,530 pounds of copper were recovered.[1]

[1] The above figures are taken from reports by Charles Mentzel.

HISTORY (cont'd)

The figures quoted by Dolmage indicate approximate gold recoveries of 88% assuming an average head grade of 0.425 oz/ton (13.80 g/tonne). The operators felt that there was no remaining ore when the mine closed in 1926.

In 1934, after the price of gold was raised, a new company was formed, Princess Royal Gold Mines by J.B. Woodworth, to acquire, rehabilitate and operate the property. This attempt failed and in 1935 the mine was again closed. The company was refinanced in 1936 and its name changed to Surf Inlet Consolidated Gold Mines Ltd. The old mill was originally rated at 300 tons per day but much of the machinery was removed prior to 1934 or had become obsolete. Milling resumed at 50 tons per day in 1936 and was gradually stepped up to a little over 100 tons per day by 1940 (Honsberger, 1973).

Overall, to the end of 1942 when the mine was closed by a scarcity of labour and general war conditions, total recorded production from the property amounted to 1,091,131 tons (495,068 tonnes), of which 169,886 tons (77,080 tonnes) came from the Pugsley and the remaining 921,245 tons (417,988 tonnes) from the Surf ore body. From this ore were recovered 382,351 ounces (11,891,116 g) of gold, 208,752 ounces (6,492,187 g) of silver and 6,314,341 pounds (2,864,185 kg) of copper (Dolmage, 1946).

When the mine was in operation, power was obtained from an efficient low head hydro-electric plant constructed in 1916 using a reinforced concrete dam of the Ambursen patent type. The dam is high enough to raise the level of the lower lake to make a continuous waterway from the head of the dam to the foot of the mountain, about 1.6 km from the mine.

In 1981 Cominco Ltd., in joint venture with Placer Development Ltd., carried out mapping, sampling and diamond drilling programs on the Surf property. Preliminary sampling of the surface stockpiles was also done. The 550 level mine dumps from the Surf Inlet mine were estimated to contain 400,000 tons (181,488 tonnes) at an average grade of 0.087 oz/ton (2.83 g/tonne) gold.

WORK DONE

Twenty-two samples of tailings and mine dump materials were collected by J. Shearer and two assistants during a property visit in November, 1985. The samples were sent to CDN Resource Laboratories and fire assayed for gold and silver. A composite of the tailings material was made and this and the stockpile samples were tested by floatation and cyanidation techniques by G. Hawthorn as described in Appendix I.

Products from the metallurgical testing were sent to J. Harris. Five samples were prepared as polished thin sections for mineralogical study. Analyses of these samples for tellurium, sulphur and fire assay for gold were done by Chemex and Cominco Ltd. laboratories. Results of this study are discussed in Appendix II.

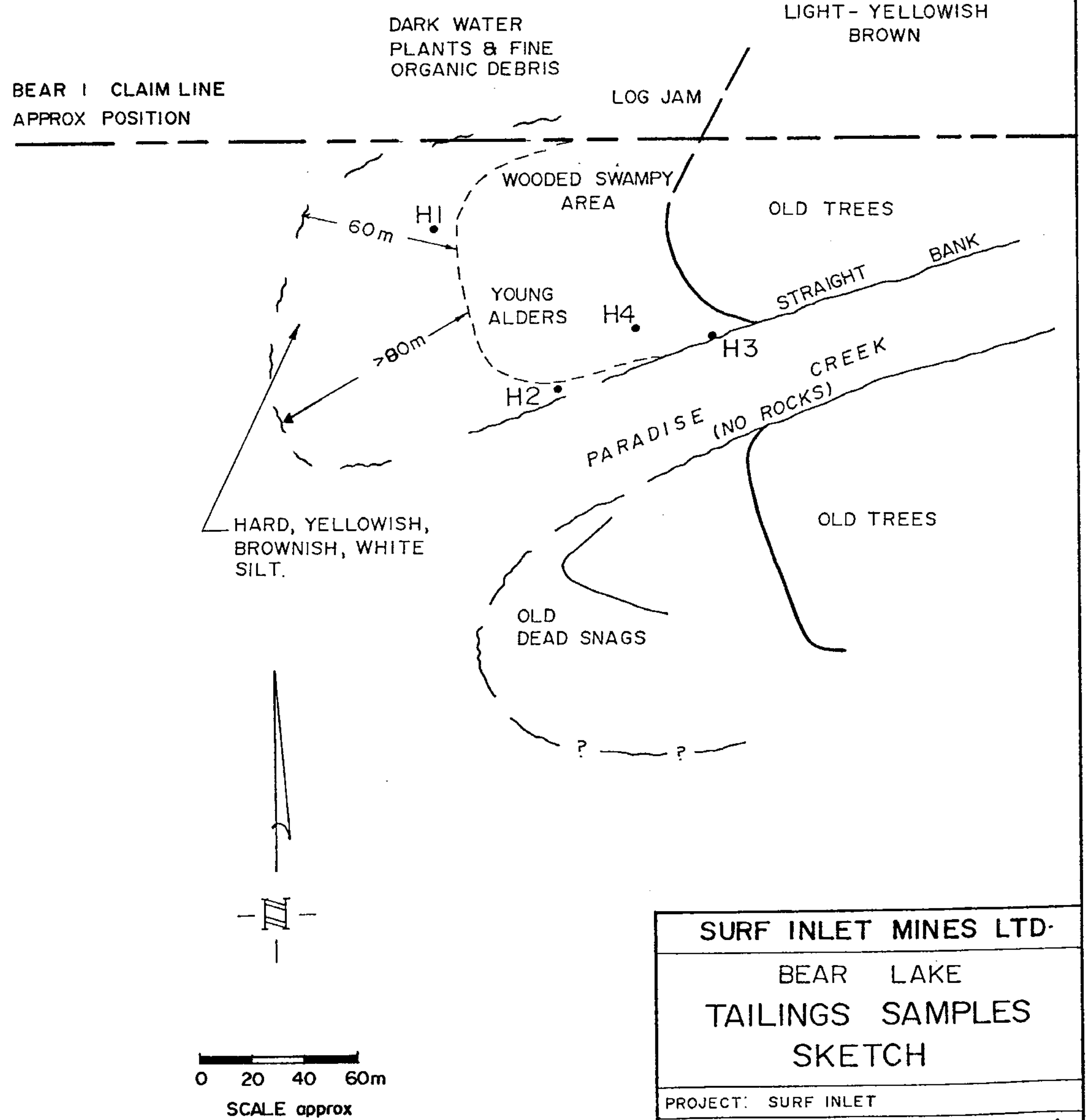
Laboratory certificates for analyses done accompany the reports in Appendix I and Appendix II.

TAILINGS SAMPLINGS

Samples were collected of typical tailings material from the recently built delta of Paradise Creek (Figure 4). The tailings are usually light brown, sandy and well compacted but easy to dig with the shovel. Each sample group is described below with corresponding assay results:

				Assay Results		
H1 at northwest side of delta			Gold oz/ton	Gold g/tonne	Silver oz/ton	Silver g/tonne
H1 A	10 cm depth		0.049	1.59	0.09	2.92
B	25 cm	some orange banding	0.055	1.79	0.06	1.95
C	40 cm	some orange banding organic material	0.055	1.79	0.03	0.97
D	60 cm	in water table green-grey colour	0.053	1.72	0.09	2.92
E	75 cm	green-grey colour	0.055	1.79	0.06	1.95
F	80 cm	grey-white colour	0.055	1.79	0.06	1.95
G	90 cm	grey-white colour	0.052	1.69	0.03	0.97
H	100 cm	grey-white colour	0.052	1.69	0.06	1.95
I	110 cm	grey-white colour	0.055	1.79	0.06	1.95
H2 at southwest side north side of creek underwater						
A	0 cm		0.058	1.88	0.09	2.92
B	20 cm	grey	0.065	2.11	0.06	1.95
C	30 cm		0.064	2.08	0.06	1.95
D	40 cm		0.064	2.08	0.09	2.92
E	50 cm	white-grey with streaks	0.064	2.08	0.09	2.92
H3 60 meters upstream from H2						
A	0-15 cm		0.067	2.18	0.09	2.92
B			0.084	2.73	0.06	1.95
C		5 bags all from	0.064	2.08	0.06	1.95
D		same level	0.075	2.45	0.03	0.97
E			0.075	2.45	0.06	1.95
		average	0.061	1.98		
H4 near H3 but on land Dirt - perhaps 150 year old delta Large cedar trees			0.046	1.49	0.03	0.97

The samples represent relatively fine grained, homogenous material with small inherent sampling errors.



SURF INLET MINES LTD.

BEAR LAKE

TAILINGS SAMPLES

SKETCH

PROJECT: SURF INLET

WORK BY: J.S. JAN. 15/86 FIG. 4

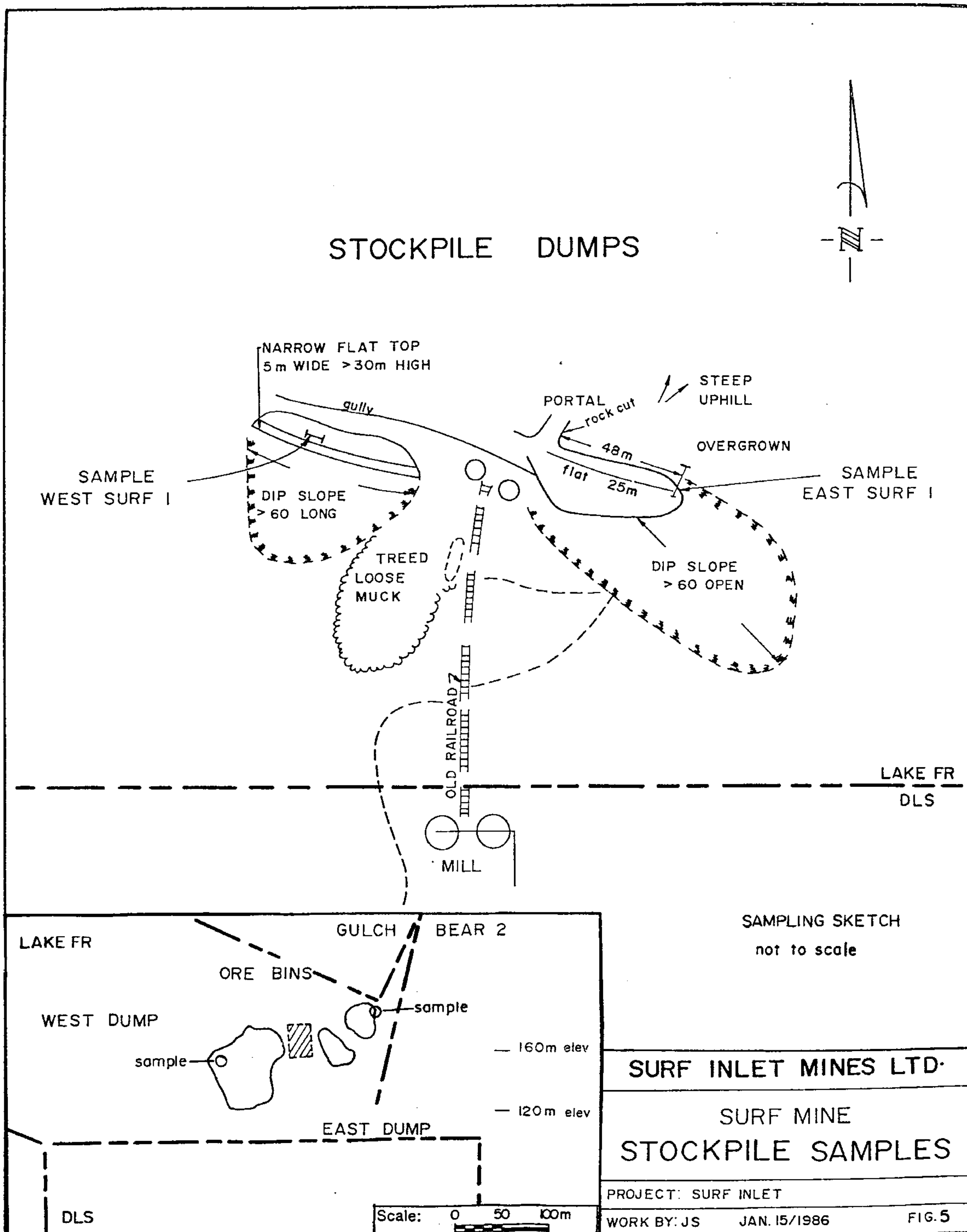
STOCKPILE SAMPLING

The Surf Mine 550 level stockpile dumps were briefly investigated to obtain a sample for metallurgical testing and to confirm the more extensive sampling carried out by Cominco in 1981.

The samples in the present program were collected along the edge of the stockpile, Figure 5, as continuous chips of all rock types encountered in a 5 meter interval. The samples and corresponding assay values are described below:

	Assay Results			
	Gold oz/ton	Gold g/tonne	Silver oz/ton	Silver g/tonne
East Surf I Mostly chloritic fine-grained altered rock minor medium crystalline diorite relatively abundant white quartz one piece of pyrite + quartz one piece of pyrite + MoS + quartz	0.067	2.18	0.29	9.42
Chip along 5 meters at lip of dump				
West Surf I 20-30% abundant rusty pyritic quartz; white bull quartz common; gneissic diorite - 50% chloritic alteration phase - 10%	0.151	4.90	0.20	6.50
Chip along 5 meters of dump				

Due to the relatively coarse grain size of these stockpiles, special care must be taken in estimation of average grades.



SURF INLET MINES LTD.

SURF MINE

STOCKPILE SAMPLES

PROJECT: SURF INLET

WORK BY: JS JAN. 15/1986 FIG. 5

CONCLUSIONS

A large volume of stockpiled mineralized broken rock containing important gold values is present outside the 550 level of the Surf Mine. Tonnage estimates by Freeze (1981) are at least 400,000 tons (181,488 tonnes). Previous estimates of the stockpile average grade was 0.087 oz/ton (2.83 g/tonne) gold which was calculated by combining the 0.102 oz/ton (3.31 g/tonne) gold assay from the west dump with the substantially lower grade of 0.051 oz/ton (1.66 g/tonne) gold from the east dump.

These preliminary results by Cominco are confirmed in the present sampling program with results of 0.151 oz/ton (4.90 g/tonne) gold from the west dump and 0.067 oz/ton (2.18 g/tonne) gold from the east dump. More sampling is warranted to accurately define the gold content of the stockpiles.

The resource potential of the old tailings has been ignored or not recognized by previous owners. Preliminary sampling by TRM Engineering Ltd. in November 1985 has indicated a large area underlain by tailings near the delta and lower portion of Paradise Creek. Nineteen samples of tailings from various depths and locations gave an average of 0.061 oz/ton gold. Interestingly, the highest grade samples come from farthest upstream.

Grab samples of massive pyrite located over 400 meters west of the Surf Mine yielded results of 2.18 oz/ton (70.81 g/tonne) Au and 1.04 oz/ton (33.78 g/tonne) Ag. Further evaluation of parallel and subsidiary mineralized zones should be undertaken.

Any economic appraisal of the Surf Inlet Area should include the following items:

(a) stockpile resource;
(b) tailings resource;
(c) deep level primary ore reserves;
(d) south exploration potential;
(e) exploration potential between Surf and Pugsley Mines;
(f) deep level exploration potential;
(g) potential for parallel sulphide zones to the west.

RECOMMENDATIONS

A systematic sampling program should be conducted over the stockpiles. Special efforts must be made to ensure representative samples are collected. This will involve excavating large trenches through the stockpiles at regular intervals. An accurate transit survey is required to estimate the thickness of each stockpile.

Sampling of the tailings area should be extended on a regular grid using a power-auger or light-weight core drill. The distribution of tailings along the braided lower parts of Paradise Creek and within the delta-swamp complex should be determined by an air photo interpretation and careful ground mapping. All results should be plotted on a 1:1,000 scale plan map with 2 m elevation contours which can be manufactured from the 1:20,000 air photo coverage.

J.T. Shearer
January 15, 1986

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APPENDIX I

SURF INLET GOLD PROJECT

PRINCESS ROYAL ISLAND, B.C.

EXTRACTION OF GOLD FROM MILL TAILINGS AND MINE WASTE: PRELIMINARY TESTING

METALLURGICAL ASSESSMENT

BY

GARY HAWTHORN, P.ENG.

CONTENTS

1. SUMMARY

The mine waste dumps and the mill tailing at Surf Inlet probably contain in excess of $10 million in recoverable gold and silver.

The testing indicates that flotation will produce a concentrate containing 4-6 oz/t (129.92-194.88 g/tonne) Au. This material may be either sold directly or leached on-site.

Additional testing will be required to determine the optimum metallurgical operating conditions, or the relative economics of flotation and cyanidation.

Future testing will investigate whether alternative hydrometallurgical techniques will enhance the overall recovery relative to cyanidation.

2. INTRODUCTION

The Surf Inlet property was previously mined in the periods of 1917 to 1926 and 1940 to 1943, leaving 1.0 x 10^6 tons of flotation tailing grading .03-.05 oz/ton Au and mine waste (estimate 400,000 tons) grading approximately 0.1 oz/ton.*

Processing during these periods consisted of crushing/grinding/gravity/flotation to produce an auriferous pyrite concentrate for sale to smelters.

Available literature suggests that:

- feed grade was 0.3 - 0.6 oz/ton Au;
- recovery was approximately 92%;
- tailing losses were 0.03 - 0.05 oz/ton Au;
- the concentrate contained approximately 2.5% Cu and 6 oz/ton Au;
- the concentrate represented approximately 9 - 11% of the original feed weight;
- the mineralization was refractory to the cyanide process.

* See Shearer, J.T.; Jan. 15/86: "Report on Preliminary Sampling of Tailings and Stockpiles at the Surf Inlet Mine".

3. DESCRIPTION OF THE SAMPLES

Nominal Description	Number of Samples	Sample Identification
Mill Tailing	20	See attached Assay Cert.
Mine Waste Rock	3	East Surf 1 - A to C
Mine Waste Rock	3	West Surf 1 - A to C

Average Sample Weight 8 Kg

Sample Preparation

Tailing

Equal volumes were removed from each of the twenty sample bags and composited to provide a laboratory feed sample of approximately 15 Kg.

Mine Waste Rock

All six samples were jaw crushed to -6 mm (1/4") and two composites were prepared (East Surf/West Surf), utilizing the entire samples.

4. DISCUSSION OF TEST RESULTS

Sample	Compo. Grade (oz/t) Au	Ag
A - PLANT TAILING	* .060	.063
B - E SURF COMPO 1-A/C	.067	.29
C - W SURF COMPO 1-A/C	.151	.20

* During the periods 1917-1926 and 1940-1943, the ore was ground to 90% -220 mesh and 45-55% -200 mesh, respectively. The plant tailing was reported to be .03-.05 oz/t Au.

Field sampling of the plant tailing in Nov. 1985 (20 samples .046-.084 oz/t, average .060 oz/t Au) indicate that natural classification of the tailing in the disposal system has left an enriched coarse portion adjacent to the outfall.

The portion of the tailing which flowed into the lake will likely be both finer and lower grade than the sample material.

SUMMARY OF TEST RESULTS

Sample	Test	Tailing oz/t		Recovery (1)		Rougher Conc Grade		TLG. Sizing % -200 M
		Au	Ag	Au	Ag	Au	Ag	
A	F	0.47	.03	23.5	27.3	1.09	0.85	10.0
A	C	0.42	.084	30.0	nil	-	-	10.0
B	F	0.18	.080	78.8	57.9	1.03	1.69	66.5
B	C	0.47	.33	29.9	nil	-	-	56.7
C	F	0.31	.02	83.2	88.5	3.33	3.34	44.1

F - FLOTATION C - CYANIDATION

* Note: (1) recoveries in this summary have been computed using assays of the composites and the test tailings.

COMMENTS

1. The rougher concentrate grades ranged 1.0 to 3.3 ounces of gold per ton. It is expected that the final concentrate would duplicate 4 to 6 ounces of gold per ton achieved by the previous operation.

2. There is a reasonably consistent gold to suphide ratio which resulted in good quality flotation concentrate grades.

3. Due to a very fine gold diśtribution in a sulphide matrix and/or discreet gold minerals, the flotation concentration method is more effective than cyanidation.

4. It may be possible to process the flotation concentrate by an alternative hydrometallurgical technique. This may enhance the economics of the operation by producing a bullion on-site rather than having to incur the high cost of transporting and marketing a flotation concentrate.

   Techniques which may be worthy of consideration include: thiourea leaching, bio-oxidation/cyanidation. These are still in the experimental stage and plant scale technical and financial feasibility have not been demonstrated.

5. Flotation concentration is sensitive to the degree of grinding, as evidenced by the following comparison:

GRIND % -200 M	FLOT. TAILS Au- oz/t
10	.047 LAB
44	.031 "
66	.018 "
46	.029 PLANT 1940

6. Recovery by cyanidation was technically not as attractive as was flotation concentration. However, the sales revenue from marketing bullion is appreciably higher than that obtained from flotation concentrate: typically 99% vs 85%.

LABORATORY CONCENTRATE ANALYSIS

NOTE: Rougher Concentrate Only

	WEST SURF COMP.	EAST SURF COMP.
S	22.4%	8.0%
Cu	1.39%	Not Assayed
Fe	25.5%	13.7%
INSOL	37.4%	64.4%
Au	3.33 oz/t	1.03 oz/t
Ag	3.34 oz/t	1.69 oz/t

5. PROCESSING ASSESSMENT

A preliminary financial evaluation has been prepared based on the following assumptions:

1) Flotation concentrate only
2) Feed Grade .1 oz/T
3) NSR = 85% contained gold value
4) Capital Cost in the $2,000,000 range
5) Mining Rate 200 tons/day
6) Gold Price $450 CAN

Operating Cost Estimate:

	FUNCTION	$/TON	$/MONTH
	Stockpile Recovery and Haulage	2.00	12,000
	Supervision	4.00	24,000
	Mill Operators	1.00	6,000
	Miscellaneous Labour	0.50	3,000
*	Power (40 kwh/t @ $.25/kwh)	10.00	60,000
	Grinding Media	1.25	7,500
	Miscellaneous Supplies	1.50	9,000
	Tailing Disposal	1.00	6,000
		21.25	127,500

Projected Cash Flow:

Assumed Mine Waste (tons)	200,000	400,000	600,000
Life (years)	2.7	5.4	8.2
NSR ($/ton)	28.7	28.7	28.7
Operating Cost ($)	21.3	21.3	21.3
Capital Cost Allowance ($)	10.0	5.0	3.3
GROSS REVENUE; (Before Taxes/Interest) ($/ton)	-2.30	2.40	4.10
TOTAL ($)	-500,000	1,000,000	2,500,000

* NOTE: Power costs were based upon typical data from diesel-electric operation in remote locations. This high cost may provide some incentive to rehabilitate the hydro-electric facilities.

There is no suggestion that 200 t/d is an optimum rate.

6. CONCLUSIONS AND RECOMMENDATIONS

The testwork shows that flotation concentration is technically feasible for the extraction of gold from mill tailing and the mine waste dumps.

The percentage recovery obtained by straight cyanidation was not as high as that obtained by flotation. (Flotation 23.5% to 83.2%/Cyanidation 30%). Some combination of flotation followed by cyanidation after regrinding may enhance the economics of the operation. This will be the subject of future testing.

Additional laboratory testing must be undertaken to determine:

- Whether flotation followed by regrinding and cyanidation is economically superior to producing a flotation concentrate for sale.

- The relationship between the grind and recovery in flotation.

- The optimum conditions for cyanidation.

If the revenue from flotation and cyanidation is low relative to the contained Au content, it may be appropriate to perform additional leaching tests using thiourea and bio-oxidation.

7. METALLURGICAL INVESTIGATION

7.1 Summary of Test Results

Sample	Composite Au	Grade Ag
A) Plant Tailing	.060	.063
B) E. Surf Composition 1-A/C	.067	.29
C) W. Surf Composition 1-A/C	.151	.20

Sample	Test	Tailing oz/ton Au	Tailing oz/ton Ag	Recovery % Au	Recovery % Ag	Concentrate Grade Au	Concentrate Grade Ag	Tailing Sizing (%-200M)
A	F	.047	.03	23.5	27.3	1.09	0.85	10.0
A	C	.042	.084	30.0	nil	-	-	10.0
B	F	.018	.080	78.8	57.9	1.03	1.69	66.5
B	C	.047	.33	29.9	nil	-	-	56.7
C	F	.031	.02	83.2	88.5	3.33	3.34	44.1

F - Flotation - rougher flotation
C - Cyanidation

7.2 Metallurgical Investigation - Details

Plant Tailing - Cyanidation

GRIND: Nil

LEACH: 500 g.24 hr/33% Solids

TIME	ADDITION (gm) NaCN	$Ca(OH)_2$	NaCN CONC. g/1	PH Initial	Final
0	1.0	0.2	1.0	9.4	
2	-	-	0.9	11.8	
24	-	-	0.8	11.7	

REAGENT CONSUMPTION

	Kg/t
NaCN	0.4
$Ca(OH)_2$	0.4

NOTE: - High natural pH of 9.4
- In future testing add only 0.1 g $Ca(OH)_2$ per 500 g sample.

METALLURGY

	Au	Ag
TAILING	.042 oz/t	.084 oz/t
FEED	.060 oz/t	.063 oz/t
EXTRACTION	30%	nil

FRACTIONAL SCREEN ANALYSIS (Test Tailing)

MESH	% Wt.	Oz/Ton Au	Oz/Ton Ag
35			
	19.5	.029	.17
65			
	34.9	.034	.05
100			
	15.9	.032	.12
150			
	19.7	.029	.02
200			
	10.0	.032	.10
	100.0	(.042)	(.084)

7.3 Plant Tailing - Flotation

GRIND: Nil

CONDITION: 2 Minutes

FLOAT: To completion 8 Minutes

REAGENT: Potassium Anyl Xanthate
AF 65
Natural pH

METALLURGY

	% Wt.	Oz/Ton		Dist. - 1%	
		Au	Ag	Au	Ag
CONC.	1.3	1.092	.85	23.5	27.3
TAILING	98.7	.047	.03	76.5	72.7
FEED	100.0	(.061) .060	(.041) .063	100.0	100.0

REAGENTS	ADDITION POINT	ADDITION g/1
PAX	COND	130
	FLOAT	390
AF 65	COND	250

7.4 East Surf 1A to C Composite

CYANIDATION

GRIND: 500 g/4 minutes 50% Solids

LEACH: 24 Hr/33% Solids

TIME	ADDITION (gm)		NaCN CONC.	pH	
	NaCN	$Ca(OH)_2$	g/1	Initial	Final
0	1.0	0.1	1.0	9.6	
24	-	-	0.8	10.2	

REAGENT CONSUMPTION	Kg/t
NaCN	.4
$Ca(OH)_2$	.2

METALLURGY		Oz/t		Dist.	
	g	Au	Ag	Au	Ag
PREG SOL'N	1100	.028	.035	55.4	21.5
TAILING	463.5	.047	.33	44.6	78.5
FEED	463.5	(.121) .067	(.42) .29	100.0	100.0

( ) DENOTES CALCULATED VALUE

FRACTIONAL SCREEN ANALYSIS - TAILING

MESH	%Wt.	Oz/ton Au	Oz/ton Ag
65	4.4	.031	.16
100	13.5	.034	.17
150	25.4	.046	.06
200	56.7	.052	.51
	100.0	(.047)	(.33)

7.5 East Surf Composite - Flotation

GRIND: 1000 g/12 minutes/50% Solids

CONDITION: 2 minutes

FLOAT: To Completion/6 minutes

REAGENTS:
- Potassium Amyl Xanthate
- Cyanimid A.F. 65
- Natural pH 9.4

METALLURGY

	% Wt.	Oz/ton		Dist.	
		Au	Ag	Au	Ag
CONC.	6.1	1.032	1.69	78.8	57.9
TAILING	93.9	.018	.080	21.2	42.1
FEED	100.0	(.080) .067	(.18) .29	100.0	100.0

REAGENTS

REAGENT	ADDITION POINT	QUANTITY g/t
PAX	CONC.	130
	FLOAT.	390
AF 65	COND.	200

FRACTIONAL SCREEN ANALYSIS - TAILING

MESH	% Wt.	Oz/ton	
		Au	Ag
65			
	2.3	.036	.28
100			
	6.9	.006	.02
150			
	24.3	.006	.02
200			
	35.7	.015	.06
325			
	30.8	.032	.15
	100.0	(.018)	(.080)

7.6 West Surf Composite - Flotation

GRIND: 1000 G/6 minutes/50% Solids

CONDITION: 3 minutes

FLOTATION: To Completion: 8 Minutes

REAGENTS: PAX
AF 65
Natural pH

METALLURGY

	% Wt.	Oz/ton		Dist. - 1%	
		Au	Ag	Au	Ag
CONC.	4.41	3.33	3.34	83.2	88.5
TAILING	95.59	.031	.02	16.8	11.5
FEED	100.0	(.177) .151	(.166) .20	100.0	100.0

SCREENING: 44.1% -200 Mesh

Note that the E. Surf Compo was ground for 12 minutes to 66.5% -200 Mesh

REAGENTS: As in 5.4

8. REFERENCES

(1) Maconachie, R.J. 1940 - Research on Surf Inlet Tailing

(2) Report of the Minister of Mines, 1918 - F.40

ASSAY REPORT

SAMPLE DESC.	AU OZ/TON	AG OZ/TON
H1A	0.049	0.09
H1B	0.055	0.06
H1C	0.055	0.03
H1D	0.053	0.09
H1E	0.055	0.06
H1F	0.055	0.06
H1G	0.052	0.03
H1H	0.052	0.06
H1I	0.055	0.06
H2A	0.058	0.09
H2B	0.065	0.06
H2C	0.064	0.06
H2D	0.064	0.09
H2E	0.064	0.09
H3A	0.067	0.09
H3B	0.084	0.06
H3C	0.064	0.06
H3D	0.075	0.03
H3E	0.075	0.06
H4	0.046	0.03
	0.060	0.063

76fltsu

CDN RESOURCE LABORATORIES LTD.
#8, 7550 RIVER ROAD, DELTA, B.C. V4G 1C8 / TEL (604) 946-4448

ASSAY REPORT

TO: Gary Hawthorn
3650 Emerald Drive
North Vancouver, B.C.
V7R 3B8

FILE NO.: 86-2

DATE: January 8, 1986

ATTENTION: Gary Hawthorn

PROJECT:

Sample Description	Au oz/ton	Ag oz/ton
H1A	0.049	0.09
H1B	0.055	0.06
H1C	0.055	0.03
H1D	0.053	0.09
H1E	0.055	0.06
H1F	0.055	0.06
H1G	0.052	0.03
H1H	0.052	0.06
H1I	0.055	0.06
H2A	0.058	0.09
H2B	0.065	0.06
H2C	0.064	0.06
H2D	0.064	0.09
H2E	0.064	0.09
H3A	0.067	0.09
H3B	0.084	0.06
H3C	0.064	0.06
H3D	0.075	0.03
H3E	0.075	0.06
H4	0.046	0.03

Rejects retained one month,
pulps one year, unless
specific arrangements made.

Duncan Sanduan
Certified Assayer of British Columbia

CDN RESOURCE LABORATORIES LTD.

#8, 7550 RIVER ROAD, DELTA, B.C. V4G 1C8 / TEL (604) 946-4448

ASSAY REPORT

TO: Gary Hawthorn
3650 Emerald Drive
North Vancouver, B.C.
V7R 3B8

FILE NO.: 85-214

DATE: December 4, 1985

ATTENTION: Gary Hawthorn

PROJECT: TRM - Surf Inlet

Sample Description	Au oz/ton	Ag oz/ton	Cu %	Fe %	S %	Insol. %
10777	.036	.28				
10778	.006	.02				
10779	.006	.02				
10780	.015	.06				
10781	.032	.15				
10782	1.032	1.69		13.7	8.00	64.4
10783	.029	.17				
10784	.034	.05				
10785	.032	.12				
10786	.029	<0.02				
10787	.032	.10				
10789	.031	.16				
10790	.034	.17				
10791	.046	.06				
10792	.052	.51				
ıst Surf	.067	.29				
st Surf	.151	.20				
10793	.108	.94				
10794	1.092	.85				
10795	.047	.03				
10796	3.331	3.34	1.39	25.5	22.44	37.4
10797	.031	.02				

	Au (ppm)	Ag (ppm)
10788	.96	1.3

Rejects retained one month, pulps one year, unless specific arrangements made.

Duncan Sanderson ....
Certified Assayer of British Columbia

APPENDIX II

Harris
EXPLORATION
SERVICES

MINERALOGY AND GEOCHEMISTRY

534 ELLIS STREET, NORTH VANCOUVER, B.C., CANADA V7H 2G6 TELEPHONE (604) 929-5867

Job #85-75

January 21st, 1986

Report for: Murray McClaren,
TRM Engineering,
701-744 West Hastings St.,
VANCOUVER, B.C.
V6C 1A5

Samples:

5 samples of products from metallurgical test work on material from Surf Inlet, B.C., as follows:

Sample	Slide No.	Au assay (oz/ton)
Plant tailings	85-231X	0.061
E. Surf Dump, compo	85-232X	0.067
W. Surf Dump, compo	85-233X	0.151
E. Surf flotation conc.	85-234X (A & B)	1.03
W. Surf flotation conc.	85-235X (A & B)	3.33

Samples were prepared as polished thin sections. Two sections of each of the flotation concentrates were prepared so as to maximize the chances of observing particulate Au.

Descriptions:

1. Plant Tailings

This material consists of angular particles, mainly in the size-range 0.05 - 0.25mm (50 - 250 microns).

The particles consist chiefly of quartz, with minor plagioclase, sericite and carbonate, and occasional hornblende.

Opaques are sparse. They consist dominantly of pyrite, at an estimated concentration of 0.2 - 0.4%. Rare specks of chalcopyrite and iron oxides were also seen.

The sulfides are as liberated grains.

2. E. Surf Dump: composite heads

The material mounted is a -10+20 mesh fraction sieved from the crushed, homogenized head sample. It consists of rock and mineral fragments, mainly in the size range 0.2 - 2.0mm.

The dominant particle type is a more or less altered, fine-grained, intrusivetextured rock of quartz diorite to granodiorite composition. This is composed of quartz, plagioclase (variably replaced by sericite and carbonate), minor K-feldspar and hornblende, and a little biotite. The mafics are partially altered to chlorite and epidote.

A considerable proportion of grains (30 - 40%) consist totally, or largely, of quartz. This is probably indicative of a phase of veining or silicification affecting the intrusive.

The sulfide content is estimated at approximately 0.5% and consists almost entirely of pyrite. The majority of this is in the form of a few relatively coarse, free grains, up to 2.0mm in size. A very minor proportion is as individual smaller grains (0.01 - 0.05mm) or small clumps within silicate rock fragments.

The great majority of fragments (both intrusive and quartz) are devoid of sulfides.

3. <u>W. Surf Dump: composite heads</u>

This material was prepared as for the E. Surf compo and presents a generally similar appearance under the microscope.

The intrusive material appears to be dominantly quartz dioritic in composition (K-feldspar is not seen) and may be slightly more altered (fresh hornblende or biotite are rare, and sericite and carbonate are possibly more abundant than in the E. Surf sample, sometimes making up descrete particles). A high proportion of the grains (perhaps as much as 50%) consist of essentially monomineralic quartz,

The sulfide content, though still low (estimated at about 1%), is noticeably higher than in the E. Surf compo. Again it consists essentially of pyrite, though very minor accessory chalcopyrite and pyrrhotite were also noted.

The pyrite is dominantly in the form of free grains, 0.5 - 2.0mm in size. A small proportion occurs as grains 0.01 - 0.1mm in particles of quartz diorite or quartz. There is no observable tendency for the sulfides to occur preferentially in the free quartz. Most of the quartz particles are quite devoid of sulfides.

4. <u>E. Surf Flotation Concentrate</u>

This material consists of angular grains showing a size range of about 2 - 150 microns.

The estimated mode is as follows:

Quartz) Feldspars)	50
Sericite	8
Carbonate	10
Mafic silicates	10
Pyrite	20
Chalcopyrite	1
Iron oxides) Metallic iron)	1

The sulfide particles show a remarkable degree of liberation, approaching 100%. Pyrite and chalcopyrite also show essentially complete liberation one from the other. The pyrite appears homogenous and free of inclusions of other minerals.

The majority of the sulfide particles (estimated >80%) fall in the upper part of the size range (>40 microns).

The majority of the silicate particles appear free of included or attached sulfides and it appears that considerable upgrading of the concentrate would be possible by reflotation.

5. W. Surf Flotation Concentrate

This material shows a similar size range to the E. Surf concentrate.

Estimated mode

Quartz) Feldspars)	28
Sericite	7
Carbonate	10
Mafic silicates	5
Pyrite	45
Chalcopyrite	4
Iron oxides) Metallic iron)	1

Significant differences from the E. Surf concentrate are notably higher overall sulfide content and a somewhat higher ratio of chalcopyrite to pyrite.

As in the other product, liberation of sulfides from silcates and of chalcopyrite from pyrite appears essentially complete. It therefore appears that differential flotation to produce a separate copper concentrate would be quite feasible if economically justified.

Mode of Occurrence of Gold

Intensive microscopic examination of both the East and West Surf concentrates has failed to provide an explanation for the relatively high assay values (1.0 and 3.3 oz/ton respectively).

The extreme homogeneity of the pyrite in both concentrates and the rarity of any inclusions is a striking feature.

Only one example of native Au was seen (in the W. Surf concentrate). This was in the form of a small cluster of minute grains (1 - 2 microns in size) intergrown with chalcopyrite and an unidentified phase (probably petzite).

Prompted by the apparent lack of Au in the metallic form, and by reports of the occurrence of gold tellurides in a sample from the property (J.A. McLeod, Cominco Ltd, 1981), the two concentrates were submitted for analysis for Te.

Results were as follows:

E. Surf concentrate	Te	14 ppm
W. Surf concentrate	Te	68 ppm.

The presence of Te is thus confirmed. Moreover, the Te contents show a general correlation with the S and Au levels in the respective products.

In light of the above, the products were re-examined under the microscope to make sure that gold in the form of tellurides had not been overlooked. Nothing was found that would account for these Te values in terms of gold tellurides.

A number of examples of optically unidentifiable phases in the slides of the W. Surf concentrate were marked and checked for composition by scanning electron microanalysis. Of 8 such grains examined, 2 proved to be metallic iron (a contaminant from grinding); 3 were too small to be locatable under the SEM (they were inclusions, 1 - 2 microns in size, in pyrite); and 3 proved to be tellurides.

Of the latter, one grain (a 6 micron inclusion in pyrite) is a composite of Pb telluride and Ag telluride and the other two (free grains, 25 and 30 microns in size) are a Ag-Au telluride, probably petzite $(AgAu)_2Te$.

Although the existence of telluride minerals in the Surf Inlet mineralization is thus confirmed, the extreme rarity of the phases fails to account for the analysed Te levels, or for the Au values in terms of previous metals tellurides.

It appears, therefore, that the Au in these samples must be held in submicroscopic form within another mineral - most likely the pyrite. Numerous examples of this mode of occurrence exist in the literature. The fact that exploitation of the Surf Inlet deposit in the past has always been via production and sale of an auriferous sulfide concentrate rather than by gravity and/or cyanidation, adds weight to this possibility, as does the strong correlation of Au values with S contents in the present products:

	S (%)	Au (oz/ton)
E. Surf concentrate	8.3	1.03
W. Surf concentrate	22.5	3.33

As an additional check on this probability, 5 separate analyses for Au on very small portions (0.1g) of the W. Surf concentrate were done. Results are as follows:

	Au (ppb)	Equiv. oz/ton
Portion 1	116,000	3.31
" 2	121,000	3.46
" 3	110,000	3.14
" 4	117,000	3.34
" 5	108,000	3.10

The extremely low variance of these replicate micro-analyses is a striking confirmation of the hypothesis that the Au is present in a homogenously distributed (possibly molecular or solid solution) form in the pyrite.

Agreement of this order between analyses is hard to obtain on most Au-bearing materials even by conventional assays using weights in the order of 20g. To obtain it on replicate portions 1/200th of this size would be essentially inconceivable if any significant proportion of the Au existed in discrete particulate form. It is also significant that every one of the 0.1g portions analysed very close to the official assay of 3.33 oz/ton for this material.

A comparable experiment was then carried out with regard to the Te. Four replicate analyses on 0.2g portions of the W. Surf concentrate gave the following results:

	Te (ppm)
Portion 1	100.0
" 2	97.5
" 3	97.5
" 4	122.5

These data (coupled with the rarity of microscopically visible tellurides) suggest that most of the Te in these materials is also held in solid solution in the sulfides.

The discrepancy between these figures and the original analysis of 68 ppm is considered by the analyst to be a function of superior digestion achieved in the later work.

Conclusions:

1. The sulfide mineralogy at Surf Inlet (insofar as it is represented by the dump material) is very simple, consisting of major pyrite and minor chalcopyrite.

2. Liberation of the sulfides is essentially complete at the grind used for the present tests. The bulk of the sulfides occur in the upper part of the size range, suggesting that adequate liberation may still be achieved at coarser grinds.

3. Only a very minor proportion of the Au occurs in particulate form. The bulk of it is indicated as being in a homogenously dispersed, sub-microscopic form in pyrite.

4. The poor results obtained in cyanidation tests (around 20% recovery) fit the above conclusion. Total decomposition of the sulfides (by chemical, biohydrometallurgical or pyrometallurgical means) would appear necessary to liberate the Au.

5. Te contents of the Surf concentrates are substantial. Only traces of tellurides can be seen, so this element too may exist dominantly in dispersed form in the sulfides. It is therefore unlikely to be a factor in Au recovery.

6. Exploration at Surf Inlet should be geared to locating bodies of rock enriched in pyrite.

J.F. Harris

J.F. Harris Ph.D.

Chemex Labs Ltd.

Analytical Chemists • Geochemists • Registered Assayers

212 Brooksbank Ave.
North Vancouver, B.C.
Canada V7J 2C1

Phone: (604) 984-0221
Telex: 043-52597

CERTIFICATE OF ASSAY

TO : HARRIS EXPLORATION SERVICES

534 ELLIS ST.
NORTH VANCOUVER, B.C.
V7H 2G6

CERT. # : A8610107-001-A
INVOICE # : I8610107
DATE : 20-JAN-86
P.O. # : NONE

Sample description	Prep code	S % (Leco)					
E. SURF CON	214	8.56	--	--	--	--	--
W. SURF CON-1	214	22.70	--	--	--	--	--
W. SURF CON-2	214	--	--	--	--	--	--
W. SURF CON-3	0	--	--	--	--	--	--
W. SURF CON-4	0	--	--	--	--	--	--

VO! rev. 4/85

Registered Assayer, Province of British Columbia

Chemex Labs Ltd.

Analytical Chemists • Geochemists • Registered Assayers

212 Brooksbank Ave.
North Vancouver, B.C.
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CERTIFICATE OF ANALYSIS

TO : HARRIS EXPLORATION SERVICES

534 ELLIS ST.
NORTH VANCOUVER, B.C.
V7H 2G6

CERT. # : A8610107-001-A
INVOICE # : I8610107
DATE : 20-JAN-86
P.O. # : NONE

Sample description	Prep code	Te ppm					
E. SURF CON	214	--	--	--	--	--	--
W. SURF CON-1	214	100.00	--	--	--	--	--
W. SURF CON-2	214	97.50	--	--	--	--	--
W. SURF CON-3	0	97.50	--	--	--	--	--
W. SURF CON-4	0	122.50	--	--	--	--	--

VOl rev. 4/85

Certified by ....

Tellurium ppm:

A 5.0 gram sample is digested with aqua-regia to dryness. The residue is taken up in 25% HCl and the solution adjusted with HBr to 3M Br-. After the reduction of iron with abscorbic acid the tellurium bromide complex is extracted into MIBK, washed and analyzed via A.A., correcting for background absorption.

Detection limit: 0.1 ppm

Gold F.A.-A.A. Combo Method ppb:

For low grade samples and geochemical materials, 10 gram samples are fused in litharge, carbonate and siliceous flux with the addition of 10 mg of Au-free Ag metal and cupelled. The silver bead is parted with dilute HNO_3 and then treated with aqua regia. The salts are dissolved in dilute HCl and analyzed for Au on an atomic absorption spectrophotometer.

Detection limit: 5 ppb

COMINCO LTD.,EXPLOR TION RESEARCH LAB.
1486 East Pender Street
Vancouver,B.C.
Canada V5L 1V8
Telephone (604)254-0881
Telex 04-507730

To : Harris Exploration Services
534 Ellis Street,
North Vancouver,B.C.,V7H 2G6

Job No : X86-002
Date : 08-JAN-86

****
*   Analysis   report   *
****

No. of samples submitted	element analysis	method	lower limit of detection
1	Au	Geochemistry	10 ppb

Sumi Ltd.,Exploration Research Lab. Analysis report ; # X86-0002 Harris Date ; 08-JAN-86 P ; 1

#	sample number	Au ppb
1	W.Surf.Compo.-0.1 sm.	e116000
2	W.Surf.Compo.-0.1 sm.	e121000
3	W.Surf.Compo.-0.1 sm.	e110000
4	W.Surf.Compo.-0.1 sm.	e117000
5	W.Surf.Compo.-0.1 sm.	e108500

NOTE ; "e" - estimated

APPENDIX III

STATEMENTS OF QUALIFICATIONS

STATEMENT OF QUALIFICATIONS

I, Johan T. Shearer of the City of Port Coquitlam, in the Province of British Columbia, do hereby certify:

1. I graduated in Honours Geology (B.Sc. 1973) from the University of British Columbia and the University of London, Imperial College (M.Sc. 1977).

2. I have practiced my profession as an Exploration Geologist continuously since graduation and have been employed by such mining companies as McIntyre Mines Ltd., J.C. Stephen Explorations Ltd. and Carolin Mines Ltd. I am presently employed by TRM Engineering Ltd.

3. I am a Fellow of the Geological Association of Canada. I am also a member of the Canadian Institute of Mining and Metallurgy, the Geological Society of London and the Mineralogical Association of Canada.

4. I have visited the property on June 8, 1985 and between November 6-11, 1985 and examined diamond drill core, underground workings and collected samples. I have also reviewed reports and other documents relating to the property.

Dated at Vancouver, British Columbia, this 15th day of January, 1986.

STATEMENT OF QUALIFICATIONS

I, Gary William Hawthorn, of the District of North Vancouver, Province of British Columbia, hereby certify as follows:

1. I am a Registered Professional Engineer residing at 3650 Emerald Drive, North Vancouver, B.C.

2. I graduated with a Bachelor of Science in Mining Engineering from Queen's University in Kingston, Ontario in 1964.

3. I have practiced my profession continually since graduation.

DATED at Vancouver, British Columbia, this 4th day of February, 1986.

STATEMENT OF QUALIFICATIONS

I, Jeffrey Frederick Harris, of North Vancouver, British Columbia, do hereby certify that:

1. I am a consulting geologist with an office at 354 Ellis Street, North Vancouver, British Columbia.

2. I am a graduate of the Royal School of Mines, London (B.Sc. in Mining Geology, 1956) and of the Australian National University, Canberra (Ph.D., 1965).

3. I have practiced my profession of geologist since 1956, being employed for 6 years by the Geological Survey of Tanganyika, and for 17 years by Cominco Ltd. I have been an independent consultant since 1983.

4. I am a Fellow in good standing of the Geological Association of Canada.

Dated at Vancouver, British Columbia, this 23rd day of September, 1986.

STATEMENT OF QUALIFICATIONS

I, Sharon L. Gardiner, of the District of North Vancouver, in the Province of British Columbia, do hereby certify:

1. I graduated with a Bachelor of Science, Honours Degree in Earth Sciences (cooperative program) from the University of Waterloo in May, 1979.

2. I have practiced my profession continuously since graduation.

3. I am a Fellow of the Geological Association of Canada.

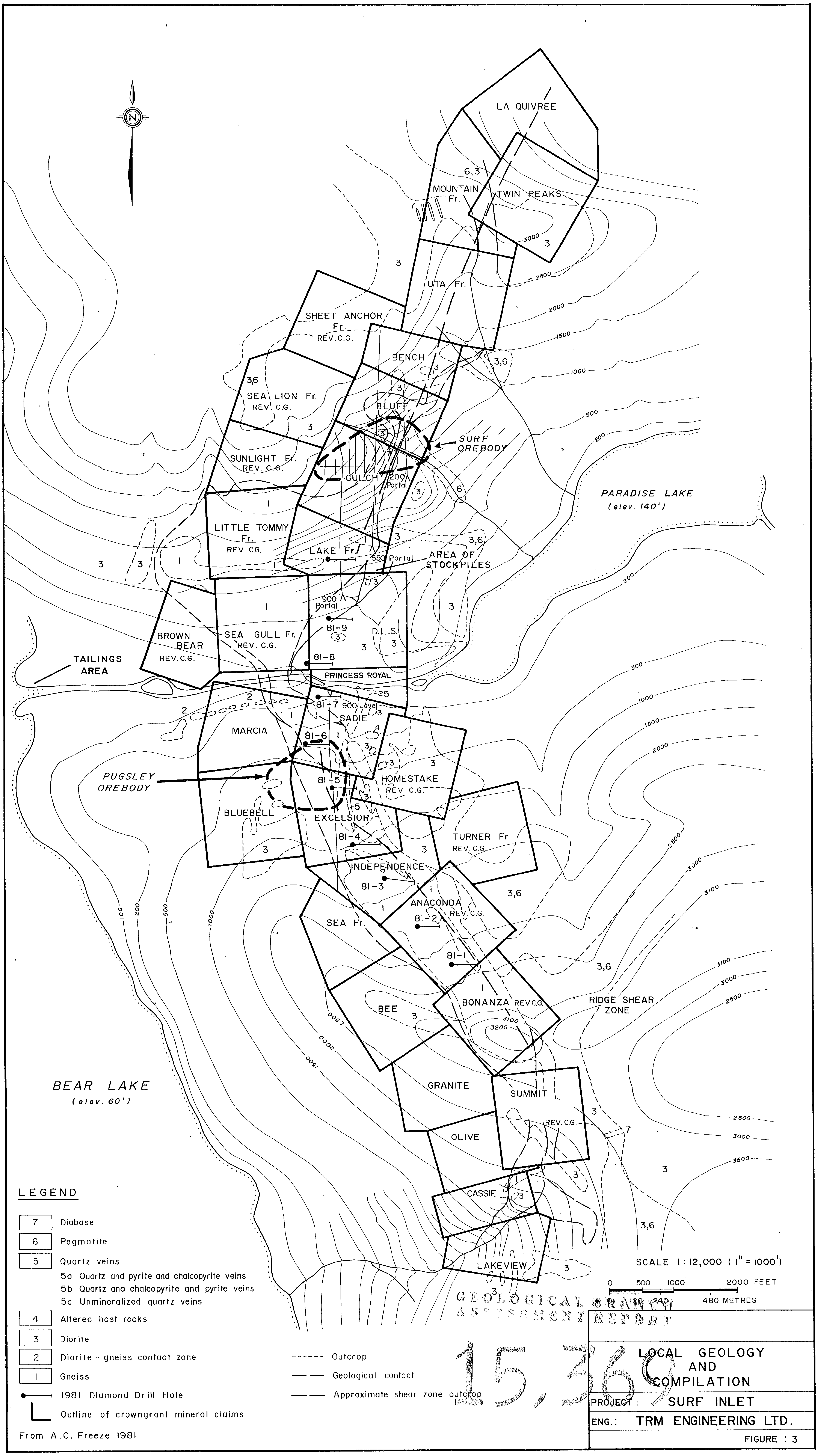
4. I compiled this summary using reports written by J. Shearer, G. Hawthorn and J. Harris.

Dated at Vancouver this 30th day of September, 1986.

APPENDIX IV

COST STATEMENT

FIELDWORK		
Wages		
J. Shearer Nov. 6-11/85, 6 days @ $160/day		$ 960.00
P. Larkin Nov. 6-11/85, 6 days @ 90/day		540.00
C. Bishop Nov. 6-11/85, 6 days @ 115/day		690.00
		$ 2,190.00
Transportation		
Vancouver-Prince Rupert, 2 return airfares		$ 660.00
Prince Rupert to Property, Otter, 665 km @ $3.00/km		2,000.00
		$ 2,660.00
Accommodation		
18 man-days @ $35/man-day		$ 630.00
Field Supplies		200.00
		$ 830.00
	TOTAL FIELDWORK	$ 5,680.00
METALLURGICAL TESTING (Report - Appendix I)		
Wages - G. Hawthorn, 1.5 days @ $400/day		$ 600.00
Analytical Costs, 20 Au, Ag assays @ $10.50/sample		210.00
Courier, Freight		176.85
		$ 986.85
MINERALOGICAL STUDY (Report - Appendix II)		
Wages - J. Harris, 1 day @ $350/day		$ 350.00
Analytical Costs		
Prep 8 Mounting - 7 polished sections @ $20/sample		140.00
6 Te Analyses @ $5.50/sample		33.00
5 Au Analyses @ $4.40/sample		22.00
2 S Analyses @ $6.00/sample		12.00
S.E.M. Microprobe analyses		350.00
		$ 907.00
REPORT		
J. Shearer, Jan/86, 2 days @ $160/day		$ 320.00
S. Gardiner, Sep/86, 2 days @ $115/day		230.00
		$ 550.00
Drafting - 10 hrs @ $15/hr		$ 150.00
Reproduction Costs		200.00
		$ 350.00
	TOTAL TESTING AND REPORT	$ 2,703.85
	TOTAL COST	$ 8,473.85


N
LA QUIVREE
6,3
MOUNTAIN Fr.
TWIN PEAKS
7
3
3000
2500
2000
1500
1000
500
200
UTA Fr.
SHEET ANCHOR Fr.
REV. C.G.
BENCH
3,6
SEA LION Fr.
REV. C.G.
BLUFF
SURF OREBODY
SUNLIGHT Fr.
REV. C.G.
GULCH
200 Portal
6
PARADISE LAKE
(elev. 140')
LITTLE TOMMY Fr.
REV. C.G.
1
LAKE Fr.
550 Portal
AREA OF STOCKPILES
3,6
900 Portal
81-9
D.L.S.
BROWN BEAR
REV. C.G.
SEA GULL Fr.
REV. C.G.
TAILINGS AREA
81-8
PRINCESS ROYAL
2
5
81-7
900 Level
SADIE
4
MARCIA
81-6
PUGSLEY OREBODY
HOMESTAKE
REV. C.G.
81-5
BLUEBELL
EXCELSIOR
TURNER Fr.
REV. C.G.
81-4
INDEPENDENCE
81-3
ANACONDA
REV. C.G.
SEA Fr.
81-2
81-1
3100
BONANZA REV. C.G.
RIDGE SHEAR ZONE
BEE
3200
BEAR LAKE
(elev. 60')
GRANITE
SUMMIT
REV. C.G.
OLIVE
3500
CASSIE
LAKEVIEW
100
SCALE 1:12,000 (1" = 1000')
0 500 1000 2000 FEET
0 120 240 480 METRES
LEGEND
7 Diabase
6 Pegmatite
5 Quartz veins
5a Quartz and pyrite and chalcopyrite veins
5b Quartz and chalcopyrite and pyrite veins
5c Unmineralized quartz veins
4 Altered host rocks
3 Diorite
2 Diorite - gneiss contact zone
1 Gneiss
1981 Diamond Drill Hole
Outline of crowngrant mineral claims
Outcrop
Geological contact
Approximate shear zone outcrop
From A.C. Freeze 1981
GEOLOGICAL BRANCH
ASSESSMENT REPORT
15,369
LOCAL GEOLOGY AND COMPILATION
PROJECT: SURF INLET
ENG.: TRM ENGINEERING LTD.
FIGURE : 3