VENUS PLACER GEOLOGICAL REPORT

DATA COMPILATION EVALUATION

CONRAD GOLD DISTRICT - YUKON

WINDY ARM AREA OF TAGISH LAKE

ON

YUKON PLACER CLAIM

SANDPIPER-2, P32656

LATITUDE 60°02.5' AND LONGTITUDE 134°35.4'

NTS SHEET 105-D-2

WHITEHORSE MINING DISTRICT

YUKON TERRITORY

FOR

R.G.HILKER: CALGARY, ALBERTA

ASSESSMENT WORK REPORT

SEPT. 03, 1991 TO SEPT. 03, 1992

BY

R.G.HILKER, P.ENG.

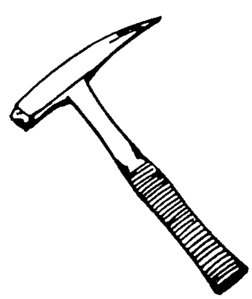
TRON DUIK CONSULTANTS LTD.

CALGARY, ALBERTA

EFFECTIVE DATE SEPTEMBER 3, 1992

"CONFIDENTIAL REPORT"

WHITEHORSE MINING DISTRICT



This report has been examined by
the Geological Evaluation Unit under
Section 41 Yukon Placer Mining Act
and is recommended as allowable
representation work in the amount
of $ 1,000.00 .

Robert Deklerk

for. Chief Geologist, Exploration and
Geological Services Division, Northern
Affairs Program for Commissioner of
Yukon Territory.

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TABLE OF CONTENTS

INTRODUCTION

History of Venus Tailings Deposit

Mining first started in the Conrad Gold District - Yukon during the 1920 era at the "Venus Mine". During mid-1960 or 1965 further exploration was conducted on the ore reserves of the Venus Mine. A 300 ton (272 tonne) per day capacity mill was constructed and ore treated from the Venus Mine from September 1970 until June 1971. The mill tailings were contained in a pond near the mill site that is located 13.9 miles (22.4 km) from Carcross, Yukon on the Skagway Highway. The Skagway road connects Carcross, Yukon with Skagway, Alaska. The abandoned Venus tailings deposit is located 58.6 miles (94.4 km) by road southwest of Whitehorse, Yukon.

The Venus Mine property was re-examined in 1979 by United Keno Hill Mines Whitehorse exploration department. The Venus Mine site is located near Pooly Creek, in the Conrad District, 14.9 miles (24 km) from Carcross on the Skagway road and 59.6 miles (96 km) from Whitehorse. Early in 1980 United Keno Hill Mines Ltd. (UKHM) announced plans to reopen the Venus Mine by June 1981. Consequently, applications to the British Columbia Waste Management Branch were submitted for construction and operation of a mill in British Columbia and to the Yukon Territorial Water Board for a water use license at the Yukon Venus Mine site. The Canadian Environmental Protection Service provided a "Baseline Study of The Watershed Near Venus Mine, Yukon and proposed Venus Mill, British Columbia" (October 1981).

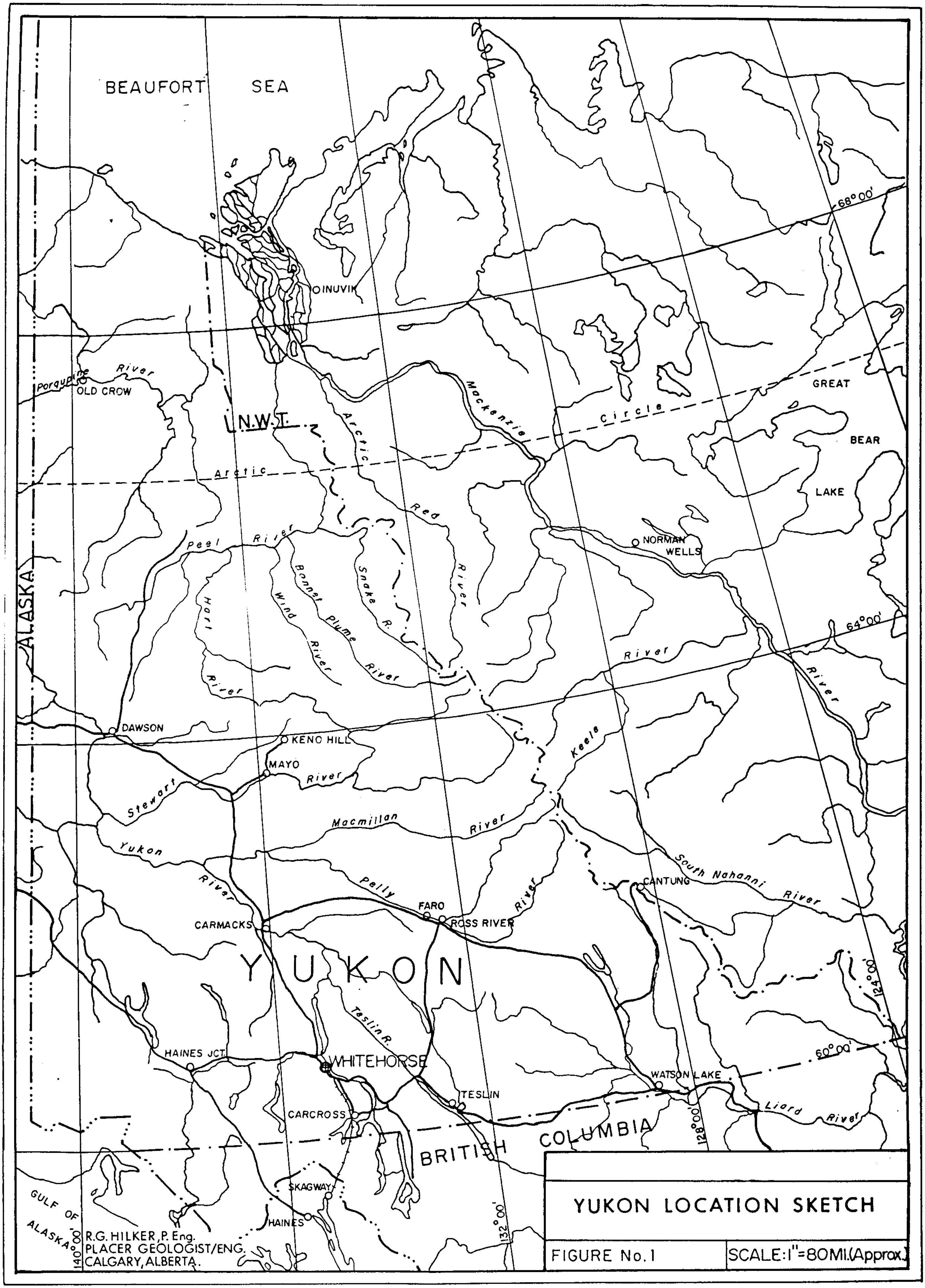
In March 1980 UKHM exploration department conducted exploration on the abandoned (1970-71) tailings pond deposit. The UKHM plans were to remill the abandoned tailings contained in the 1970-71 pond at the new Venus mill site located 6.9 miles (11.2 km) southwest on the Skagway road, from the 1970-71 tailings pond. When the Venus Mine and mill operated from September 1970 until June 1971, two types of concentrate were produced.

- silver/zinc/cadmium that contained 1 oz. Au and 30 oz. Ag per ton (31.25 grams gold and 937.5 grams silver per metric ton).
- lead/silver/gold concentrate that contained 7 oz. Au and 270 oz. Ag per ton (218.8 grams Au and 8437.6 grams Ag per metric ton).
- host rock was andesite breccia with crystalline quartz veins. Quartz Veins - carbonate, pyrite and arsenopyrite.

The abandoned Venus tailings pond 1980 exploration delineated a volume and grade of the tailings on data dated March 17, 1980, UKHM exploration department Whitehorse (see table and drill figure).

- Volume of tailings 1,025,484 cubic feet 29,029 cubic meters.
- Drilled Grade:
  Gold (Au) - 0.09 oz/ton or 2.80 gm/tonne
  Silver (Ag) - 1.3. oz/ton or 40.74 gm/tonne
  Zinc (Zn) - 0.37%
  Sulfides of pyrite and arsenopyrite

UKHM Ltd. reports in 1980 that at the Venus, the country rock in the mine workings consists primarily of pale green to green, competent, cherty andesite breccia, alternating with dark green colored andesite flows and possibly minor tuff. The Venus Mine property was re-examined in 1979 and earlier ore estimates were confirmed. It was decided in April 1980 to bring the Venus Mine into production. Ore reserves were calculated at 120,000 tons (109,000 tonnes) containing a grade of 0.22 oz/ton (8.84 gm/t) gold, 6.60 oz/ton (205.3 gm/t) silver, 1.89% lead, and 1.37% zinc. The ore is associated with 8% iron pyrite, 10% arsenopyrite and quartz. The surface facilities for the mine or the new Venus mill site is located on the eastern side of the Skagway road southwest of the mine in British Columbia. The Venus mill site was built in British Columbia during 1980-1981, 20.9 miles (33.6 km) from Carcross, Yukon on the Skagway road. An electric power line was constructed from Carcross adjacent to the Skagway road to service the new Venus mill in British Columbia. The Venus electric power line passes through the site of the 1970-71 mill and abandoned tailings deposit. During 1981 approximately 15,000 tons of tailings (1970-71) were removed from the pond and hauled to the new UKHM Ltd. Venus Mine mill in British Columbia.



YUKON LOCATION SKETCH

FIGURE No. 1

SCALE: 1" = 80 MI. (Approx.)

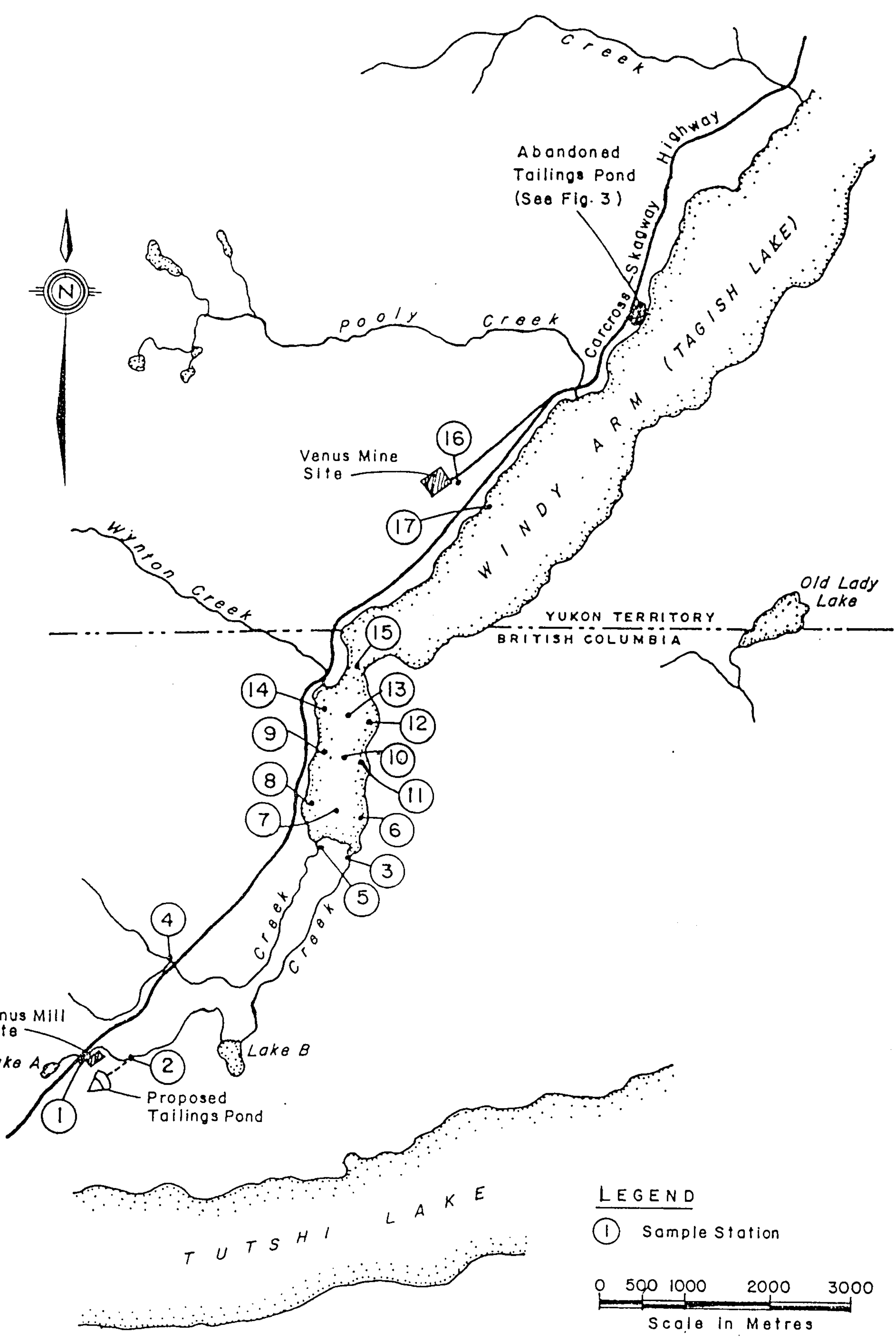


FIGURE 2 SAMPLE STATION LOCATIONS IN STUDY AREA
(See Figure 3 for detail)

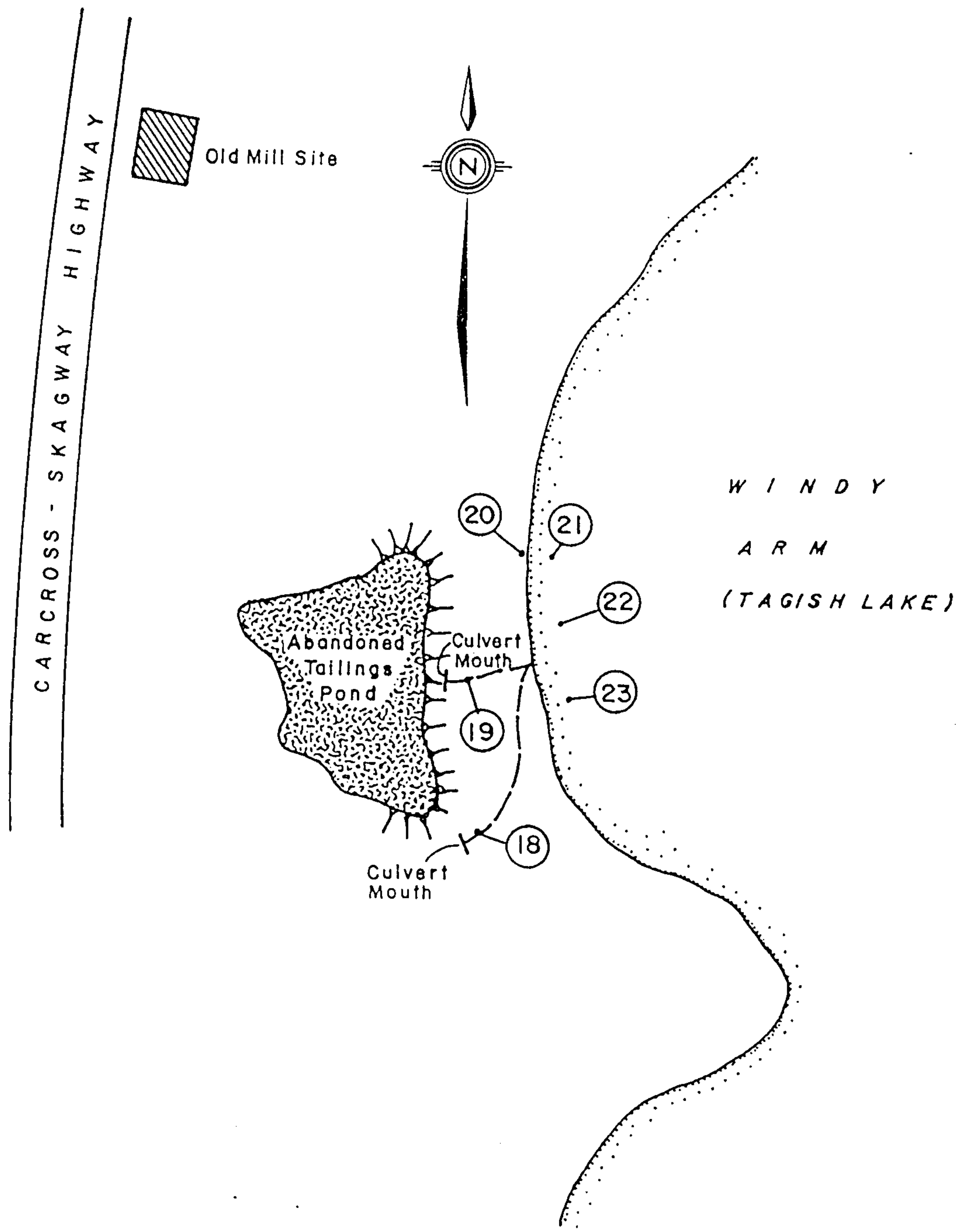


FIGURE 3 DETAIL SKETCH OF ABANDONED TAILINGS POND SHOWING LOCATIONS OF STATIONS 18 TO 23 (Not to Scale)

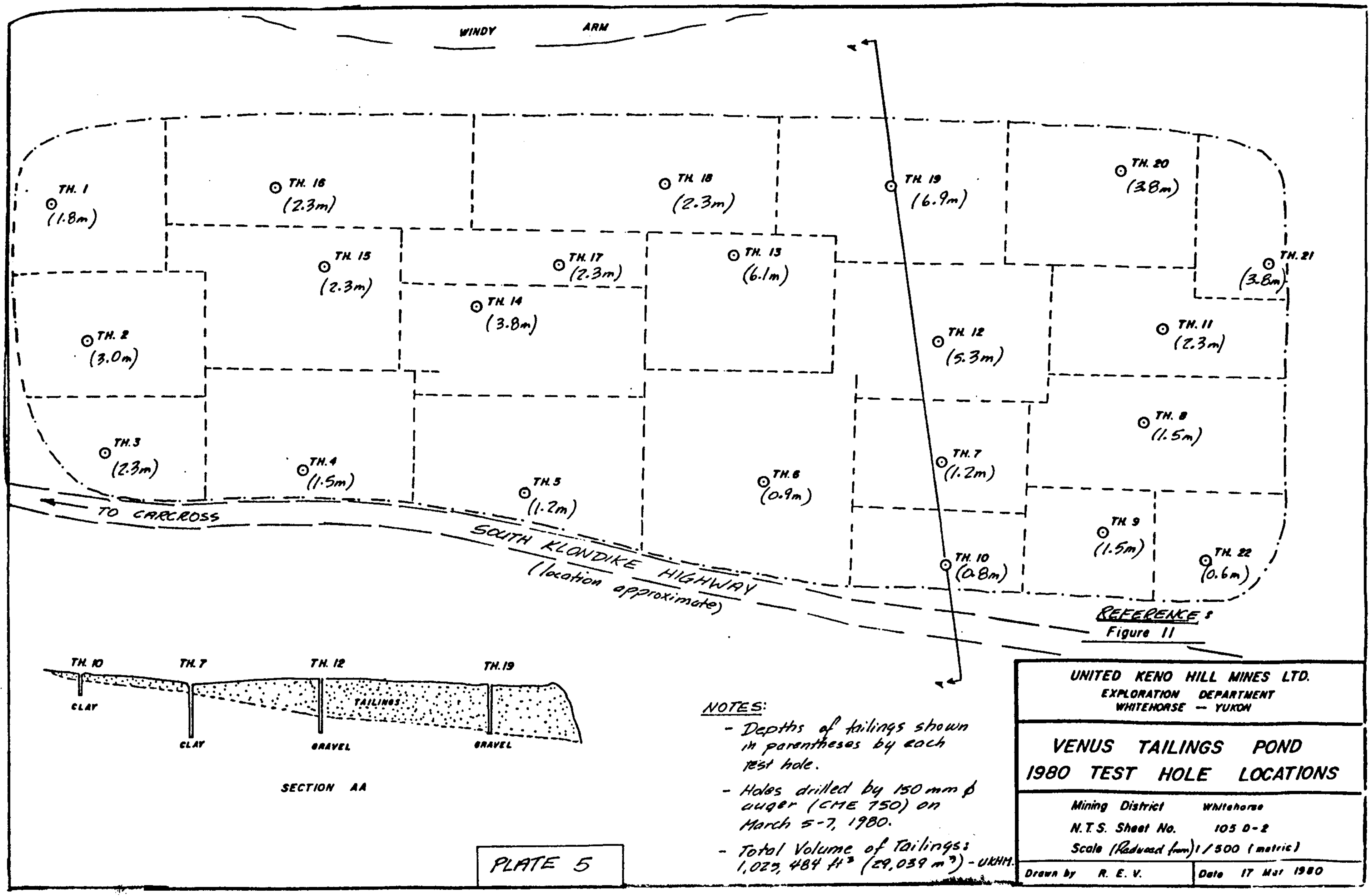
VENUS TAILINGS POND - YUKON
U.K.H.M. 1980 TEST HOLE ASSAYS

GRADES, DEPTH and TONNAGE

HOLE NO.	Au oz/ton	Ag oz/ton	Pb %	Zn %	Depth	Cu.Ft. Volume	20 cu.ft./ton Volume
1	.05	1.38	0.40	0.36	6.0	32,613	1,631
2	.05	1.66	0.43	0.41	10.0	41,775	2,089
3	.09	1.99	0.49	0.41	7.5	23,911	1,196
4	.11	1.85	0.36	0.24	5.0	28,863	1,443
5	.06	1.30	0.30	0.27	4.0	26,349	1,317
6	.02	1.23	0.30	0.23	3.0	28.074	1,403
7	.10	1.31	0.32	0.24	4.0	16,884	844
8	.13	1.08	0.30	0.23	5.0	31,343	1,567
9	.13	1.19	0.53	0.36	5.0	14,539	727
10	.11	1.44	0.29	0.22	2.5	7,256	363
11	.02	1.13	0.30	0.23	7.5	35,307	1,765
12	.13	1.49	0.40	0.27	17.5	105,600	5,280
13	.05	1.45	0.42	0.27	20.0	113,408	5,670
14	.07	0.82	0.26	0.34	12.5	71,863	3,593
15	.08	1.68	0.48	0.47	7.5	47,525	2,376
16	.11	1.41	0.42	0.33	7.5	52,066	2,603
17	.15	1.37	0.36	0.54	7.5	21,181	1,059
18	.09	0.95	0.29	0.31	7.5	56,188	2,809
19	.13	1.20	0.39	0.34	22.5	149,708	7,485
20	.09	1.14	0.31	0.30	12.5	69,520	3,476
21	.12	1.00	0.27	0.35	12.5	45,910	2,296
22	.17	1.28	0.31	0.21	2.0	5,601	280
						TOTAL	51,272

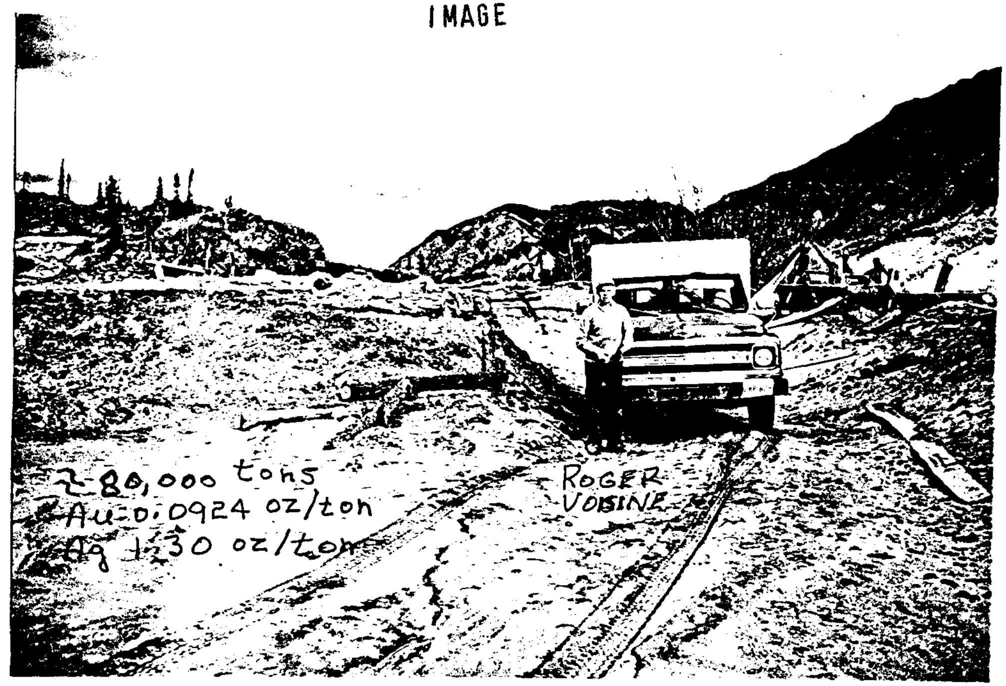
Note: The UKHM 1980 volume calculations were based on an "estimated" 20 cubic feet/ton tonnage factor. The tonnage factor should be 10.88 cubic feet/ton based on the specific gravity of 2.94 determined on the tailings concentrate.

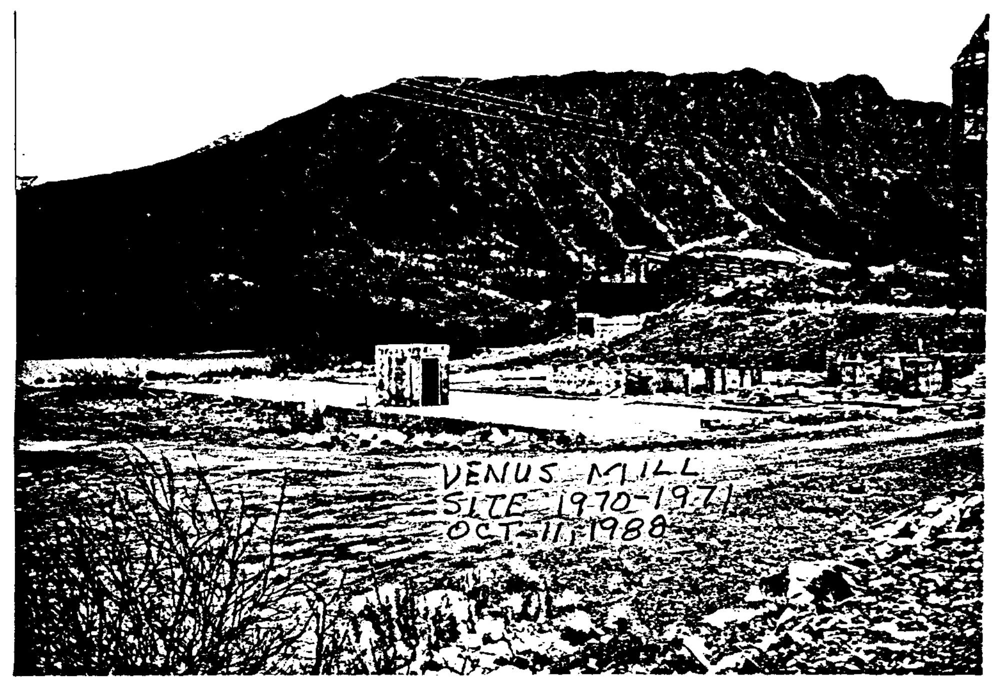
/5



WINDY ARM
TH. 1 (1.8m)
TH. 16 (2.3m)
TH. 18 (2.3m)
TH. 19 (6.9m)
TH. 20 (3.8m)
TH. 15 (2.3m)
TH. 17 (2.3m)
TH. 13 (6.1m)
TH. 21 (3.8m)
TH. 2 (3.0m)
TH. 14 (3.8m)
TH. 12 (5.3m)
TH. 11 (2.3m)
TH. 8 (1.5m)
TH. 3 (2.3m)
TH. 4 (1.5m)
TH. 5 (1.2m)
TH. 6 (0.9m)
TH. 7 (1.2m)
TH. 9 (1.5m)
TH. 10 (0.8m)
TH. 22 (0.6m)
TO CARCROSS
SOUTH KLONDIKE HIGHWAY
(location approximate)
REFERENCE:
Figure 11
TH. 10
TH. 7
TH. 12
TH. 19
TAILINGS
CLAY
CLAY
GRAVEL
GRAVEL
SECTION AA
NOTES:
- Depths of tailings shown in parentheses by each test hole.
- Holes drilled by 150 mm ϕ auger (CME 750) on March 5-7, 1980.
- Total Volume of Tailings: 1,025,484 ft³ (29,039 m³) - UKHM.
PLATE 5
UNITED KENO HILL MINES LTD.
EXPLORATION DEPARTMENT
WHITEHORSE — YUKON
VENUS TAILINGS POND
1980 TEST HOLE LOCATIONS
Mining District Whitehorse
N.T.S. Sheet No. 105 D-2
Scale (Reduced from) 1/500 (metric)
Drawn by R. E. V.
Date 17 Mar 1980

BEST ATTAINABLE IMAGE





LOCATION AND ACCESS

The Venus tailings pond concentrate was deposited during September 1970 until June 1971, from a mill that was located 500 feet northeast of the pond. The tailings pond is situated on the east side of the Skagway road and the shoreline of Windy Arm - Tagish Lake. The Skagway road is a paved highway between Carcross, Yukon and Skagway, Alaska, that is an ocean port on tidewater. The Venus tailings are 13.9 miles (22.4 km) from Carcross by the Skagway road. The new United Keno Hill Mines mill site is situated 20.0 miles (33.6 km) from Carcross, Yukon, and is situated in British Columbia. The abandoned tailings deposit is 58.6 miles (94.4 km) from Whitehorse, Yukon via Carcross by the Skagway highway.

The Venus tailings are located on the Carcross topography sheet, scale 1:50,000 NTS sheet 105-D-2 at approximately latitude 60°02.5' and longitude 134°35.4'. The Sandpiper 2 Yukon Placer claim is located on NTS Sheet 105-D-2, scale 1" = 1/2 mile, within the Whitehorse Mining District.

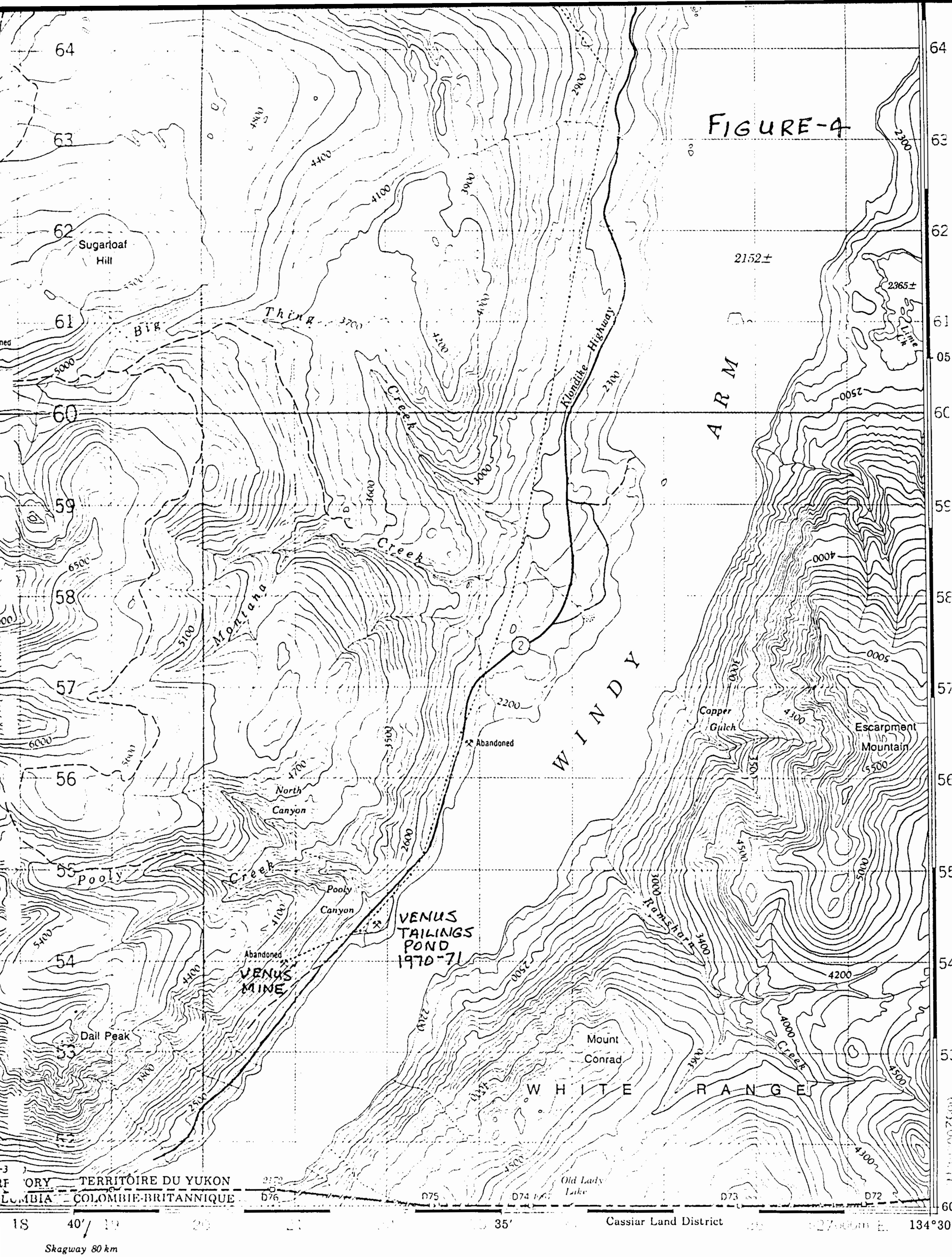
YUKON PLACER AND QUARTZ CLAIMS

The abandoned Venus tailings deposit is overlayed by the Sandpiper-2 Yukon Placer claim and the Raindrop 1-2 grant #YB06940-YB06941, Yukon Quartz Mining claim. The placer and two quartz claims are located in the Whitehorse Mining District, Yukon and plotted on NTS sheet 105-D-2.

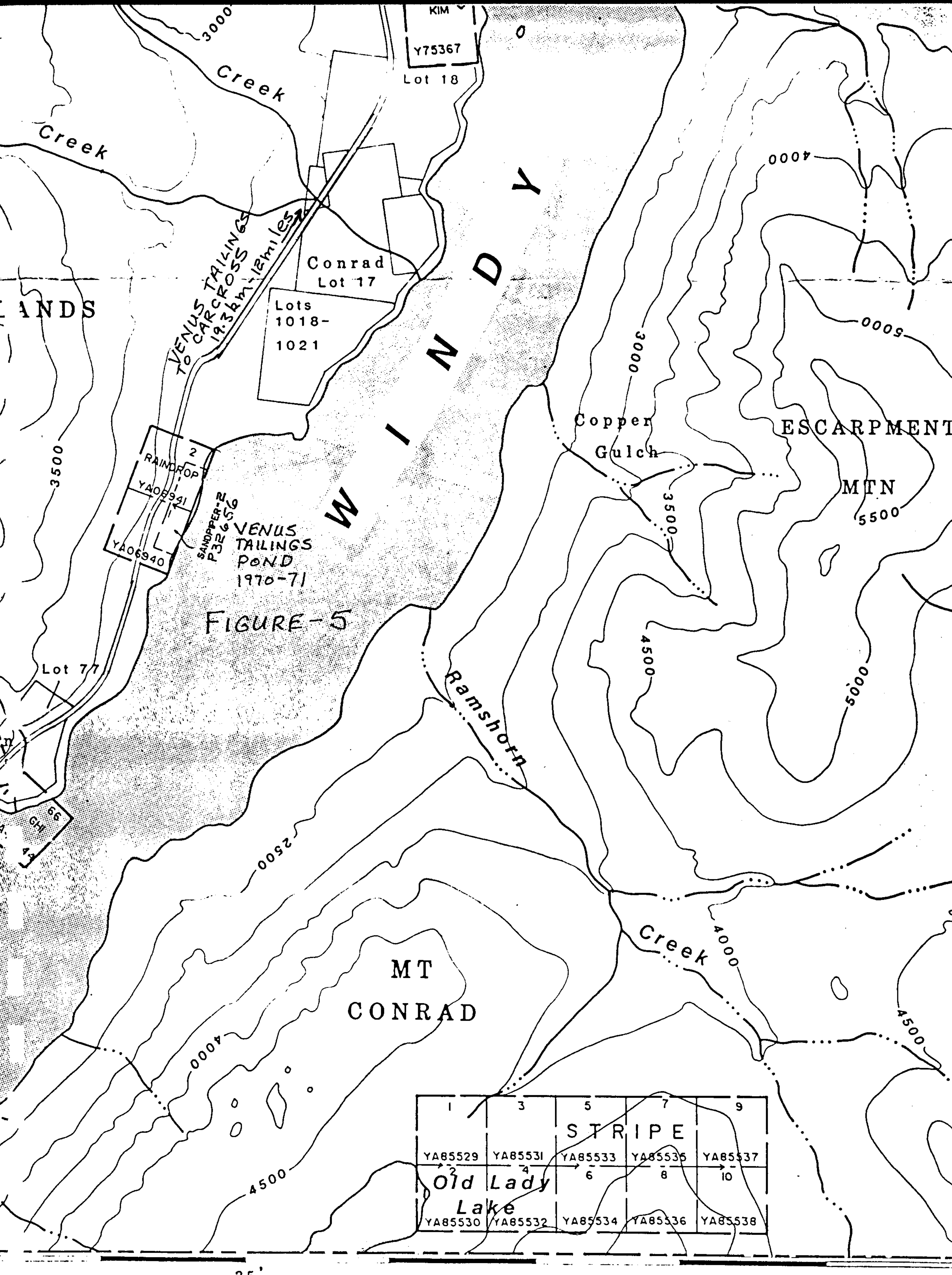
The Whitehorse Mining Recorder has declared that the abandoned Venus tailings concentrate to be a placer type of deposit and that all rights to the concentrate are included in the Sandpiper-2 (P32656) Yukon Placer claim and the Yukon Placer Mining Act.

Claim	Grant #	Anniversary Date	Reg. Owner
Sandpiper-2	P32656	3 Sept 1992	R.G. Hilker
Sandpiper-2	P32656	3 Sept. 1997	

This placer geological report is submitted for expenditures of $1,000.00 to cover a 5 year period of work commitments on the Sandpiper 2 Yukon Placer claim (3 Sept. 1992 to 3 Sept. 1997).



Établie par la DIRECTION DES LEVÉS ET DE LA CARTOG



KIM
Y75367
Lot 18
3000
Creek
Creek
4000
VENUS TAILINGS
To CARCROSS
19.3km-12miles
Conrad
Lot 17
LANDS
Lots
1018-
1021
WINDY
5000
3000
Copper
Gulch
ESCARPMENT
MTN
5500
3500
2
RAINDROP
YA06941
YA06940
SANDPIPER-2
P32656
VENUS
TAILINGS
POND
1970-71
3500
FIGURE-5
4500
5000
Lot 77
Ramshorn
66
GHI
2500
Creek
4000
MT
CONRAD
4500
4000
4500
STRIPE
1
3
5
7
9
YA85529
YA85531
YA85533
YA85535
YA85537
2
4
6
8
10
Old Lady
Lake
YA85530
YA85532
YA85534
YA85536
YA85538
35'

METALLURGICAL TREATMENT REFRACTORY GOLD ORES

1) Vancouver Petrographics Ltd. - Petrographic Study Venus Gold Mode Occurrence 1989.

Vancouver Petrographics Ltd.

JAMES VINNELL, Manager
JOHN G. PAYNE, Ph.D. Geologist
CRAIG LEITCH, Ph.D. Geologist
JEFF HARRIS, Ph.D. Geologist
KEN E. NORTHCOTE, Ph.D. Geologist

P.O. BOX 39
8080 GLOVER ROAD,
FORT LANGLEY, B.C.
V0X 1J0
PHONE (604) 888-1323
FAX. (604) 888-3642

Report for: R.G. Hilker,
Tron Duik Consultants Ltd.,
324 Silver Valley Rise, N.W.,
CALGARY, Alberta, T3B 4B2

Invoice 8193
May 1989

Samples: 7611 (#1 Head), 7612 (#2 Head), 7613 (#3 Head); 7617 (#1 Con), 7618 (#2 Con), 7619 (#3 Con)

Summary:

Four polished sections were examined for each sample of heads, and two were examined for each sample of concentrates.

Native gold occurs in Sample 7613 as a train of four grains in one fragment of arsenopyrite. It occurs in Sample 7617 as several free grains, in part associated with minor arsenopyrite. No native gold was seen in other samples.

Photographs were taken of native gold grains and of a galena grain containing grains tentatively identified as Pb-sulfosalt and grains of an unknown mineral, possibly a Pb-telluride.

Features common to all samples include the following:

Sulfides are dominated by arsenopyrite and less pyrite, with the ratio being in the range of 2/1 to 3/2. Minor sulfides include sphalerite and galena. Trace minerals include chalcopyrite, a Pb-sulfosalt, native gold, and an unknown mineral, possibly a Pb-telluride.

Sulfide fragments are mainly of single grains. Common aggregates include blebby inclusions of galena in pyrite, and intergrowths of sphalerite and less galena. Galena inclusions in pyrite average 0.01-0.015 mm in size, with a few up to 0.08 mm across. Galena commonly contains scattered inclusions averaging 0.005-0.01 mm in size of Pb-sulfosalt; one galena grain also contains inclusions from 0.003-0.005 mm size of an unknown mineral, probably a lead telluride. Sphalerite is medium to dark orangish brown in color. A few grains contain a grain or two of chalcopyrite averaging 0.03-0.05 mm across. Rare or unusual aggregates of base-metal sulfides are listed under individual samples.

Pyrite and arsenopyrite grains commonly contain vague fractures, and a few grains, mainly of pyrite, are granulated intensely.

Pyrite is altered to hematite in a few grains; alteration is concentrated on grain borders and along coarse fractures. Galena commonly is altered irregularly along grain borders to anglesite(?).

LIST OF PHOTOGRAPHS

Numbers of photographs refer to numbers on negative film and on back of prints. All photos were taken in reflected light with a blue filter, and using a 40X objective lens, which gives the prints a magnification of 480X.

Number	Sample	Description
S, 1	7617	three grains of native gold surrounded by secondary As(?) minerals containing minor relic arsenopyrite; other fragments are of arsenopyrite
2	7617	three grains of native gold; other fragments are of arsenopyrite
3	7617	elongate grain of native gold containing minor patches of arsenopyrite
4 to 7	7613	train of grains of native gold in arsenopyrite; grains outlined in print of Photo 4
8, 9	7618	galena surrounding pyrite grain; galena contains a few inclusions of Pb-sulfosalt (A) and Pb-telluride(?) (B); grains identified in print of Photo 8

John G. Payne

John G. Payne
604-986-2928

Sample 7611 (Head #1)

No native gold was seen.

A few pyrite grains contain intergrowths of angular grains of galena up to 0.03 mm across. One pyrite grain contains a few inclusions up to 0.008 mm in size of chalcopyrite and up to 0.01 mm in size of sphalerite. One pyrite grain contains moderately abundant blebby inclusions of galena averaging 0.01-0.02 mm across and flakes of sericite/chlorite up to 0.02 mm long.

One arsenopyrite grain is cut by a veinlet of galena up to 0.02 mm wide.

One grain of galena 0.15 mm across contains a lens 0.03 mm long of a slightly darker grey mineral with moderate reflectivity and moderate anisotropism. It probably is a sulfo-salt of Pb and As/Sb.

Sample 7612 (#2 Head)

No native gold was seen.

One pyrite grain contains a blebby inclusion 0.015 mm across dominated by chalcopyrite with less pyrrhotite and much less galena.

One arsenopyrite grain bordering a galena grain contains a subhedral inclusion 0.03 mm in size of pyrite. A few arsenopyrite grains contain a few blebby inclusions of sphalerite/galena averaging 0.01 mm in size. One of these also contains a blebby inclusion of chalcopyrite 0.01 mm long.

Sample 7613 (#3 Head)

Native gold (light yellow) occurs in Section 7613-2 in an irregular veinlet up to 0.012 mm wide and 0.08 mm long cutting an arsenopyrite grain. The veinlet is made up of four equant grains averaging 0.01-0.02 mm in size connected by narrow bridges (Photos 4, 5, 6, and 7).

Chalcopyrite forms an elongate, angular fragment 0.1 mm long. It forms an equant grain 0.035 mm across in quartz.

This sample contains less galena than the other two heads samples.

. . ./10

Sample 7617 (#1 Con)

Native gold occurs in 7 grains. One fragment Ø.1 mm across is dominated by three equant grains of native gold Ø.Ø3-Ø.Ø6 mm in size, surrounded by secondary As-minerals(?) with minor inclusions of primary arsenopyrite (Photos S, 1). One cluster contains three slightly elongate fragments of bright yellow native gold up to Ø.Ø8 mm in size (Photo 2). One fragment Ø.14 mm long is dominated by bright yellow, slightly tarnished native gold (Photo 3); along its borders are a few equant grains averaging Ø.Ø1 mm in size of arsenopyrite.

A few grains consists of intimate intergrowths of sphalerite and arsenopyrite. A few pyrite grains contains several inclusions of sphalerite averaging Ø.Ø2-Ø.Ø3 mm in size.

Sample 7618 (#2 Con)

No grains of native gold were seen.

One fragment Ø.2 mm across contains an equant grain of pyrite Ø.14 mm across partly surrounded by galena (Photos 8, 9). Galena contains a few inclusions averaging Ø.ØØ5-Ø.Ø15 mm in size of the Pb-As/Sb sulfosalt as in Sample 7611, and two equant grains averaging Ø.ØØ3-Ø.ØØ5 mm in size of a white mineral which is slightly more reflective than galena. It may be a Pb-telluride, or less probably a telluride of Au.

One sphalerite grain contains an inclusion of chalcopyrite Ø.Ø3 mm across.

One chalcopyrite grain Ø.Ø5 mm across contains patches of galena along its margin.

One pyrite grain contains a few blebby inclusions of chalcopyrite up to Ø.Ø15 mm in size.

Sample 7619 (#3 Con)

No grains of native gold were seen.

One large fragment of sphalerite contains several, closely packed, subhedral, cubic grains of pyrite averaging Ø.Ø3-Ø.Ø8 mm in size.

Chalcopyrite forms one strongly fractured grain Ø.15 mm long.

A few pyrite grains are strongly granulated and a few are altered strongly to hematite along grain borders and fractures.

TERRAMIN RESEARCH LABS LTD.

ANALYTICAL REPORT

Tron Duik Consultants Ltd.

Bob Hilker

Date: June 1, 1989

Job No: 89-108

Project: "VENUS"

P.O. No:

9 Sands

Signed: [signature]

. . ./12

14-2235 30th Avenue N.E., Calgary, Alberta,T2E 7C7
Phone (403) 250-9460 Fax (403) 291-7064

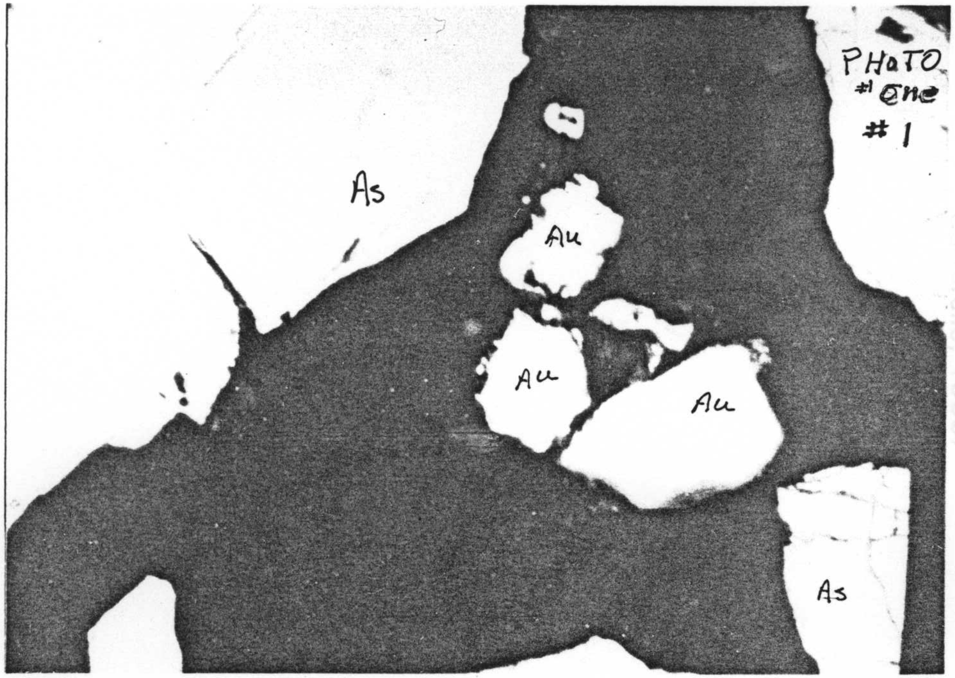
Job#: 89-108

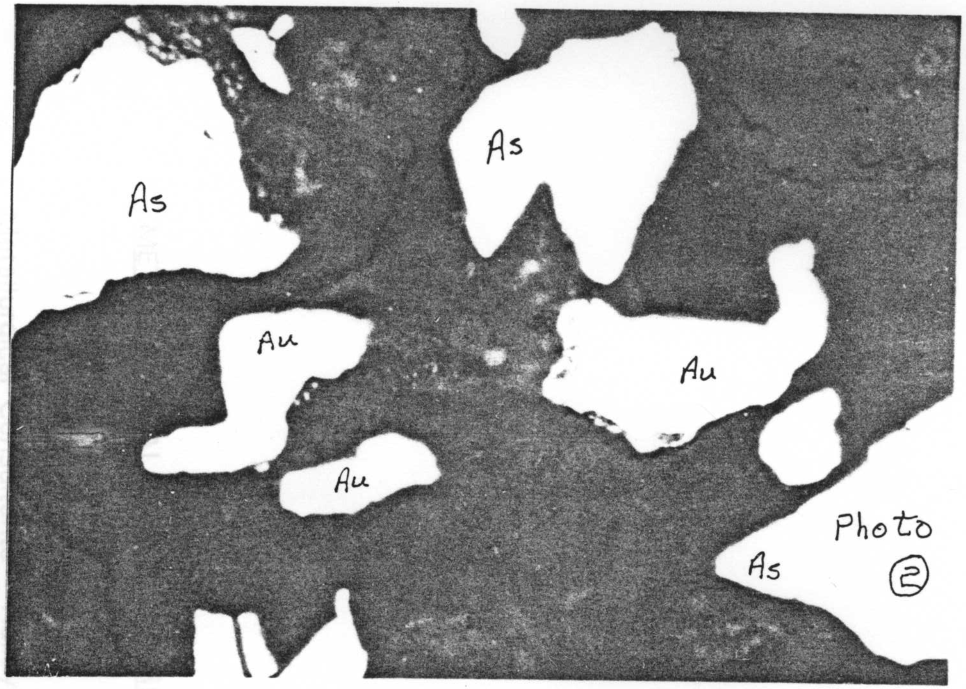
Project: "VENUS"

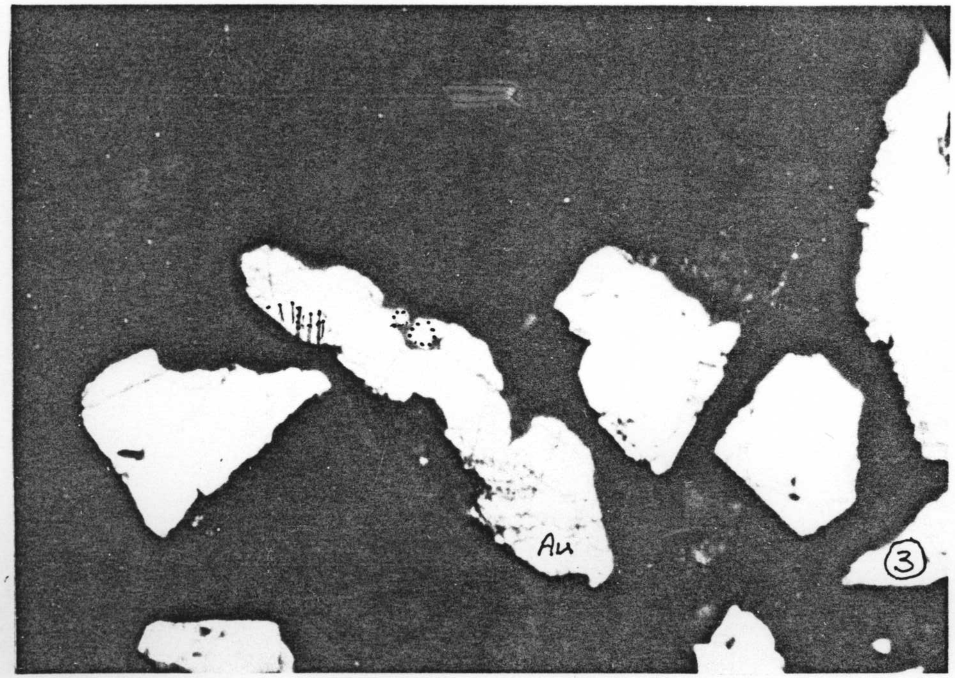
	Sample Number	Au ppb	Au oz/ton	Ag ppm	Ag oz/ton	As %	Spec Grav
Heads	7611	4560	0.133	61.6	1.80	6.1	2.94
	7612	6600	0.193	75.6	2.21	6.3	
	7613	5040	0.147	64.1	1.87	6.2	
Tails	7614	5800	0.169	55.8	1.62		
	7615	7140	0.208	80.2	2.34		
	7616	4620	0.135	54.5	1.59		
Conc's	7617	68000	1.99	187.0	5.46		
	7618	54200	1.58	164.0	4.79		
	7619	31200	0.911	138.5	4.04		

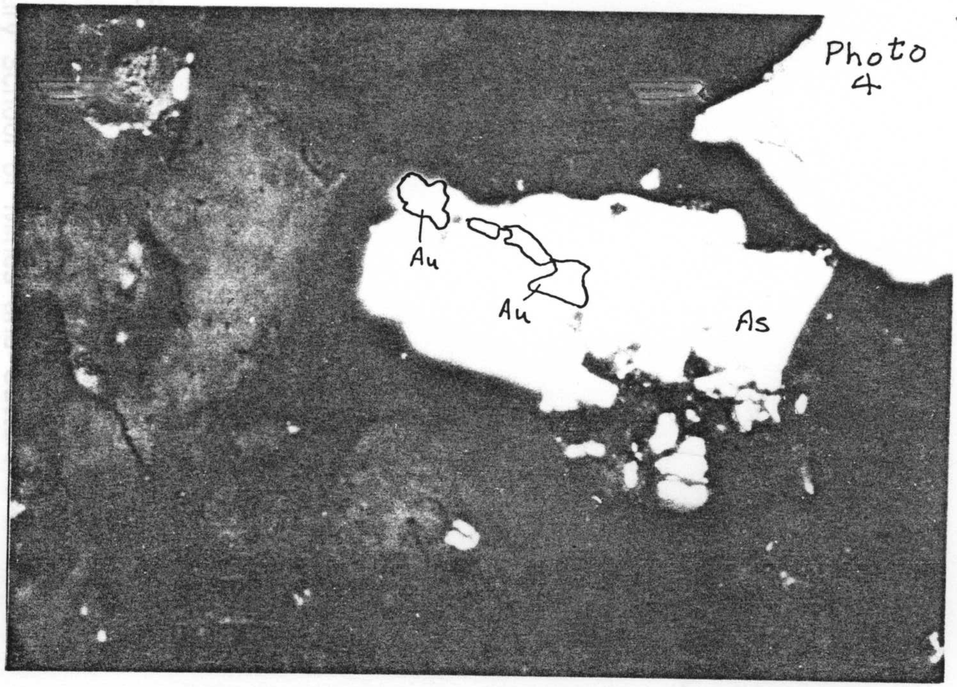
. . ./13

BEST ATTAINABLE
IMAGE









METALLURGICAL TREATMENT REFRACTORY GOLD ORES

2. Metallurgical Consultants - Haileybury School of Mines and Cattarello & Cattarello, 1988.

Cattarello & Cattarello
Metallurgical Consultants
P.O. Box 330 Haileybury, Ontario P0J 1K0 Phone 672-5045

May 2, 1988

Dear Ted & Bob:

We have now completed numerous tests on the Venus tailings which contains approximately 50,000 Tons of tailings assaying 0.09 oz/ton GOLD and 1.3 oz's/ton Silver. In a number of tests we have made a flotation concentrate assaying between 1.0-2.0 oz/ton Au and 2.0-4.0 oz/ton Ag. The Gold recovery is in the 80.0 to 85.0% range while the Silver recovery is in the 50 to 60% range.

Direct cyanidation of the flotation concentrate reduces gold and silver recoveries into the 50% range. Cyanidation of the flotation tails will recover approximately 45% of the gold and silver in the tailings. In running 300 T.P.D. or 9,000 tons per month the total costs would run $18-$20/ton. This includes Labour, Loading into the Mill, Reagents, Power and Effluent Treatment. This does not include shipping and smelting costs.

It strongly indicates that to make this project feasible the tailings would have to be concentrated at the site. In this case it is important to be open to any processes which could make a high grade product at this site.

As soon as further sample arrives, another series of new tests will be started, with aims to produce a higher grade flotation concentrate and then to cyanide both the concentrate and tails.

All the best,

Yours truly,

Carlo Cattarello

Carlo Cattarello

CC/jm

. . . /14

BELL - WHITE ANALYTICAL LABORATORIES LTD.

P.O. BOX 187. HAILEYBURY, ONTARIO TEL: 672-3107

Certificate of Analysis

NO. 0178 **DATE:** January 12, 1988

SAMPLE(S) OF: Fines (31) **RECEIVED:** January 1988

SAMPLE(S) FROM: Mr. Carlo Cattarello, Haileybury School of Mines

Sample No.	Oz. Gold	Oz. Silver
33001	0.146	3.87
2	0.720	12.80
3	0.084	1.27
4	0.066	1.87
5	0.131	1.68
6	0.548	3.32
7	0.082	1.11
8	0.068	1.21
9	0.694	3.78
33010	0.022	0.65
1	0.076	1.96
33051	0.336	4.12
2	0.030	0.53
3	0.656	2.12
4	0.068	0.99
5	0.108	1.57
6	0.022	0.52
7	0.428	3.31
144469	0.536	3.46
144470	0.026	0.44
1	2.048	4.24
2	0.172	1.12
148251	1.060	3.34
2	0.612	3.37
3	0.288	2.36
4	1.880	2.37
5	0.062	0.62
6	0.020	0.56
7	0.090	1.20
8	0.416	1.84
9	0.020	0.55

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N ACCORDANCE WITH LONG-ESTABLISHED NORTH
A RICAN CUSTOM, UNLESS IT IS SPECIFICALLY STATED
O ERWISE GOLD AND SILVER VALUES REPORTED ON
T SE SHEETS HAVE NOT BEEN ADJUSTED TO COMPEN-
S...E FOR LOSSES AND GAINS INHERENT IN THE FIRE
ASSAY PROCESS.

BELL-WHITE ANALYTICAL LABORATORIES LTD.

Per

ANALYTICAL REPORT

Tron Duik Consulting

Date : 87/09/22

Job #: 87-375

Project: Venus Tailings

No. of Samples: 5 Tailings

Signed: ____ypzzbl____________

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14 - 2235 30th Ave. N.E., Calgary, Alberta, T2E 7C7
(403) 250-9460

Venus Tailings Project

Sample Number	Au ppb	Ag ppm	Au oz/ton	Ag oz/ton
3687	1780	59.0	0.052	1.72
3688	1980	4.30	0.058	0.125
3689	2880	47.0	0.084	1.37
3690	1900	25.0	0.055	0.73
3691	6160	42.0	0.180	1.23

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14 - 2235 30th Ave. N.E., Calgary, Alberta, T2E 7C7
(403) 250-9460

TERRAMIN RESEARCH LABS LTD.

METALLURGICAL TREATMENT REFRACTORY GOLD ORES

3. Mountain States R & D International Inc., 1991 - Treatment of Refractory Ores and Concentrates, A Simplified Process (lime-oxygen process that utilizes low temperatures and pressure).

13801 E. BENSON HIGHWAY
P.O. BOX 310
VAIL, ARIZONA 85641

TEL. (602) 762-5364
TUCSON ONLY 624-7990
TELEX 9102502482 MSRD
TELEFAX 602-762-5717

Mountain States
R & D International, Inc.
A World Leader in Mineral Technology

728 SPICE ISLANDS DRIVE
SPARKS, NEVADA 89431

TEL. (702) 356-7572
TELEFAX 702-356-7791

March 26, 1991

Tron Duik Consultants Ltd.
324 Silver Valley Rise N.W.
Calgary, Alberta T3B 4B2

Attention: R. C. Hilker, P.E.

Subject: Simplified Treatment of Refractory Gold Ores

Reference: MSRDI Proposal No. 91-037

Gentlemen:

In response to our recent telephone conversation concerning the above simplified process for refractory gold ores, Mountain States R & D International, Inc. (MSRDI) is pleased to forward you the attached technical paper on the subject.

In regard to your tailings situation, it would be beneficial to treat your tailings or better still a gravity or flotation concentrate using our technique for the recovery of precious metals, and the containment of the arsenic.

MSRDI will be pleased to undertake a preliminary (scoping) test program to determine the practical viability of our technique under Phase I.

Should the results of our scoping tests be favorable, we would undertake a more detailed testing program to develop and confirm the flow sheet, and to obtain operating and design parameters under Phase II.

Finally, we would recommend a Feasibility Study under Phase III to determine the capital and operating costs as well as financial analysis.

COST PROPOSAL

Phase I -- Scoping Tests
Two - Autoclave tests on tailings
Two - Flotation tests to preconcentrated precious metals in a flotation concentrate (about 10 weight percent)
Two - Autoclave tests on concentrates

Estimated Costs: Phase I US & 5,000.00

Tailings Sample Requirements: 50 Pounds Composite Sample.

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The costs for Phase II, and Phase III will be provided and will be contingent on successful completion of Phase I work.

Should our proposal be acceptable, please sign the attached copy of the MSRDI Project Compliance Agreement and return it to us with an advance payment of $3,000.00. We are prepared to initiate the project immediately on receipt of the Agreement and the required sample.

We take this opportunity to thank Tron Duik Consultants Ltd., for the confidence you have shown in our Company and we assure you of our continual interest and cooperation in this most interesting and timely project.

With warmest regards,

Roshan B. Bhappu
President
RBB/rrj

Enclosures: MSRDI Project Compliance Agreement
Condition of Terms (Attachment 1)
Labor Rate Schedule 100190M (Attachment 2)
Waste Disposal (Attachment 3)
Analytical Price List (Attachment 4)

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Treatment of refractory ores: A simplified process

L. Lichty, G. Ramadorai, R. Bhappu and R. Roman

Treatment of refractory gold ores containing arsenic has always presented a major problem in their processing to recover the contained precious metal values. The major difficulties are concerned with lower recoveries and production of hazardous fumes, effluents and residues.

Due to these arsenic related problems, many otherwise economically attractive precious metal deposits and dumps all over the world are not exploited.

In recent years, treatment processes for refractory gold ores and concentrates have been explored. A few commercial operations employing roasting or pressure leaching have resulted.

L. Lichty, G. Ramadorai, R. Bhappu and R. Roman, members SME, are metallurgist, senior project engineer, president and manager, respectively, with Mountain State R&D International Inc., Box 310, Vail, AZ 85641. This article is excerpted from the authors' presentation at the SME GOLDTech 4 symposium *Advances in Gold and Silver Processing*, in Reno, NV, in September.

Roasting has limitations due to environmental constraints related to off gases. Pressure leaching involving high temperatures (180° to 200° C or 356° to 392° F) and pressures (2.75 MPa or 400 psi) is accompanied by high capital and operating costs.

An alternate route to roasting for treating pyrite and arsenopyrite ores and concentrates is the lime-oxygen hydromet process. This is carried out at a moderate temperature (100° C or 212° F) and pressure (689 kPa or 100 psi).

The authors presented a simplified process for treating such arsenic containing ores and concentrates using low temperature and pressure oxidation. This low-cost treatment with compressed air and lime renders arsenic as an inert component in the residue, meeting all the requirements of the EPA toxicity test. The precious metal values are then extracted from the autoclaved residue by standard cyanidation.

Results of the authors' study make it evident that the proposed lime-oxygen process appears to be an attractive alternative for treating arsenic-containing ores, concentrates and arsenic contaminated products. The inherent advantages of this process are:

- The process is relatively simple and uses low temperature (about 100° C or 212° F) and pressures (345 to 482 kPa or 50 to 70 psig).
- Due to the low pressures involved, the process can be carried out with compressed air rather than using an expensive oxygen plant.
- The reaction is completed in a relatively short period, about 30 minutes to one hour.
- The arsenic is fixed in an insoluble complex in the residue. And this residue meets the EPA toxic test for arsenic.
- Due to the simplicity and effectiveness of the process, the required capital and operating costs would be much lower than for alternative processes, such as conventional autoclaving and roasting.
- Besides treatment of arsenic-containing ores and concentrates, the process is applicable to the treatment of arsenic contaminated dross, speiss, smelter acid sludges and industrial waste products.

It is hoped that this proposed lime-oxygen process, with its technical and economic advantages, will provide the mining industry a simplified process for treating arsenic containing mineral products and industrial wastes. ♦

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Chapter 19

TREATMENT OF REFRACTORY ORES AND CONCENTRATES A SIMPLIFIED PROCESS

by

L. Lichty, G. Ramadorai, R. Bhappu, and R. Roman

Mountain States R & D International, Inc. (MSRDI)

ABSTRACT

Treatment of refractory gold ores containing arsenic have always presented a major problem in their processing to recover the contained precious metal values. The major difficulties are concerned with lower recoveries, production of hazardous fumes, effluents and residues. Because of these arsenic related problems, many otherwise economically attractive precious metal deposits and dumps all over the world are not exploited.

In recent years, treatment processes for refractory gold ores and concentrates have been explored and a few commercial operations employing roasting or pressure leaching have resulted. Roasting has limitations due to environmental constraints related to off gases. Pressure leaching involving high temperatures (180°C to 200°C) and pressures (400 psi) is accompanied by high capital and operating costs.

An alternate route to roasting for treating pyrite and arsenopyrite ores and concentrates is the "lime-oxygen hydromet" process carried out at a moderate temperature (100°C) and pressure (100 psi). A simplified process for treating such arsenic containing ores and concentrates utilizing low temperature and pressure oxidation is presented. This low cost treatment with compressed air and lime renders arsenic as an inert component in the residue, meeting all the requirements of the EPA Toxicity Test. The precious metal values are then extracted from the autoclaved residue by standard cyanidation. This paper discusses this new development.

TREATMENT OF REFRACTORY GOLD ORES A SIMPLIFIED PROCESS

Refractory ores are characterized by poor precious metal extractions upon direct, CIL or CIP cyanidation. Their refractory nature may be attributed to the presence of carbon, locking of gold and silver in sulfide minerals, silica lockup, etc. These sulfide minerals could be sulfosalts, arsenopyrite, pyrrhotite, enargite, cinnabar, etc. Gangue minerals typically consist of clays, carbonate minerals, silica, and others.

Arsenic sulfide ores and concentrates have always presented a major problem in their treatment to recover the contained metal values since the available processes produce hazardous fumes, effluents, and residues. On one hand, the existing

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pyrometallurgical treatment, involving a roasting step, results in the emission of toxic fumes while, on the other hand, in the hydrometallurgical techniques arsenic goes into solution and is then precipitated using expensive processes involving high temperatures and pressures.

Similarly, the stockpiled or dumped flue dusts, resulting from past smelting practices and often containing arsenic, are considered hazardous and pose an environmental problem. These toxic dumps need to be disposed of in an environmentally acceptable form. Moreover, most of these hazardous wastes contain precious, base and rarer metals (cadmium, indium, cobalt, etc.) which cannot be recovered since suitable methods are not available to treat these refractory and hazardous products.

Thus, arsenic is a problem to the mining industry worldwide and several potentially lucrative mines, and flue dust dumps in the United States, Canada, South America, Australia, Africa, Europe, and Asia are not worked because of the inherent arsenic problem. These mines and dumps contain hundreds of millions of dollar value in form of precious metals such as gold, silver, platinum, and rarer metals such as gallium, indium, germanium, and the like.

Because of the potential economic attractiveness and current environmental concern, considerable research and development efforts have been expanded in recent years to develop economically attractive and environmentally safe processes. Although some partially successful processing techniques have been developed, a truly fool-proof process has not been presented for universal use. MSRDI hopes to fill this gap. A brief review of the roasting treatment method and the Cashman and USBM autoclave processes are presented and compared with the MSRD developed Lime-Oxygen Hydrometallurgical process.

Roasting

While roasting of pyritic feed material is performed primarily to eliminate sulfur, the resulting calcine quality, porosity, and degree of sulfation are also important. Porosity of the calcine must be maintained so that the gold/silver extractions are maximized. Sulfur dioxide gases produced in roasting are converted to sulfuric acid or scrubbed with alkali and the sludge is disposed.

Roasting of carbonaceous ore is executed to eliminate preg robbing carbon. The carbonaceous ore can be present as carbonaceous sediments, humic acid, and graphite. Preg robbing is associated with only the first two types listed above. Carbon is oxidized to carbon dioxide by ensuring an excess of air.

With arsenopyrite ores and concentrates, two-stage roasting is practiced commercially. During first-stage roasting at 500°C, arsenic elimination is on the order of 65 to 80 percent and pyrite conversion is between 20 and 40 percent. Arsenic is collected as the volatile arsenic trioxide. Air ratios are 65 to 80 percent of stoichiometric needs to ensure the formation of As_2O_3.

The Outokumpu process is an intriguing means for the treatment of arseniferous concentrates. In this process, arsenical copper ores are treated with sulfur gas to remove arsenic as arsenic sulfide and leave behind essentially arsenic-free copper minerals. The application of this process to treat complex gold/silver pyrite concentrates with arsenic, antimony, bismuth, mercury, selenium, and tellurium has been patented by Outokumpu. Presumably this will also be applicable to arsenopyrite feed material.

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The lime pellet process published in the literature for copper ores fixes sulfur values as gypsum by roasting feed material as pellets containing lime. The application of this process to gold/silver ores could be interesting. However, sulfur levels must be low; otherwise, dilution of resulting calcine with attendant lower PM grades results. The porosity of calcine may also be poor, resulting in low gold/silver extractions. Soda ash could be used in place of lime to improve calcine porosity if the economics are justified.

Roasting is usually performed in a single-stage or in two-stage fluid-bed roasting followed by gas cleaning and acid production or gas scrubbing with alkali to neutralize SO_2 gases. The addition of a post-oxidation chamber to complete the oxidation of residual organics will minimize black acid -- sulfuric acid contaminated with organic matter.

Another matter of concern would be the SO_3 levels in the roaster off-gases. If these are high, then two-stage roasting may be required in order to minimize SO_3 formation; otherwise, single-stage roasting could be satisfactory.

The sulfur dioxide in the cleaned gases can be converted to sulfuric acid or scrubbed and the resulting gypsum sludge disposed. The SO_2 in the cleaned gases can be converted to sulfuric acid in a double contact-double absorption plant or a single contact-single absorption plant, depending on the environmental needs for SO_2 discharge.

Flue gas desulfurization (FGD) can be achieved by scrubbing with alkalis; for example, limestone, lime, soda ash, etc.

The advantages and disadvantages of roasting are as follows:

<u>Advantages</u>:

- O Can handle ores and concentrates
- O Eliminate preg robbing carbonaceous material
- O Low operating costs
- O Tolerates a wider variety of feeds
- O Good temperature control can be maintained with a fluid bed roaster

<u>Disadvantages</u>:

- O Toxic fumes are produced
- O Control of calcine porosity can be a problem
- O Extractions tend to be lower than hydromet processes
- O Sulfur dioxide capture and conversion to sulfuric acid or gypsum can be expensive
- O Environmental mitigation costs high

The advantages and disadvantages of high temperature/pressure autoclave process are as follows:

<u>Advantages</u>:

- O No toxic fumes produced. Waste products fixed in the leach residue or sludge after treatment of leach liquor
- O Higher gold extractions compared to roasting processes
- O Less objectionable to the public since visual fumes are not produced
- O Produces a more stable leach product with respect to iron, arsenic, etc.

<u>Disadvantages</u>:

- O High capital and operating costs
- O High cost alloys needed for application
- O More susceptible to sulfur variation in feed materials
- O Usually does not deactivate preg robbing carbon

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USBM Chloride-Oxygen Process

This proposed process is based on the initial studies by United States Bureau of Mines laboratories at Reno, Nevada. This effort reflects the Bureau's goal to maximize minerals and metals recovery from domestic resources and to reduce environmental conflicts and impacts.

The generalized leaching and precipitation reactions involved in the above process are as follows:

$$MS + 2HCl + 1/2O_2 \longrightarrow MCl_2 + S + H_2O \quad (1)$$

where MS denotes metal sulfides and MCl_2 represents metal chlorides (Cu, Pb, Zn, Fe, Ni, Co, etc.).

When the final pH is 1.4 to 2.0, the dissolved iron is oxidized and precipitated as goethite, FeO(OH), as follows:

$$3/2\ H_2O + FeCl_2 + 1/4\ O_2 \longrightarrow FeO(OH) + 2HCl \quad (2)$$

The HCl generated above is once again available for leaching more sulfides.

Some of the sulfur produced in reaction (1) is oxidized to H_2SO_4, thus

$$H_2O + S + 3/2\ O_2 \longrightarrow H_2SO_4 \quad (3)$$

Therefore, $CaCl_2$ is added to remove sulfate as gypsum and regenerate HCl as follows:

$$H_2SO_4 + CaCl_2 \longrightarrow 2HCl + CaSO_4 \quad (4)$$

HCl generated in this reaction is again available for dissolving additional base-metal sulfides.

The above equations clearly show that if the total amount of chloride in the reactor is sufficient to provide counter ions for all of the elements except iron, an almost iron-free base-metal pregnant solution can be obtained.

It should be noted that the newly developed Cashman Process, promoted by Artech Recovery Systems, is based on the above USBM process.

In the above processes, the reaction for arsenopyrites oxidation is as follows:

$$2FeAsS_2 + 7O_2 + 2CaCl_2 + 2HO_2O \longrightarrow 2FeAsO_4 + 2CaSO_4 + 4HCl \quad (5)$$

Besides the above reaction, calcium can combine with arsenic to form insoluble calcium arsenate, $Ca_3(AsO_4)_2$.

Moreover, in the presence of iron, which is usually contained in these feed sources, the iron is oxidized as an insoluble iron oxide (goethite) or complexed with arsenic to form iron arsenate ($FeASO_4$). Thus, the combined arsenic-iron-calcium residue containing their complexes [$FeAsO_4$ + FeO(OH) + $CaSO_4$ + $Ca_3(AsO_4)_2$] are tied together in a multiple molecular mixture which not only meets, but exceeds, the standard of the EPA solubility test for disposal of waste. Accordingly, the residue can be disposed of in an environmentally safe way in ponds without any danger of contaminating existing groundwater tables. It should be noted that the calcium and iron arsenate formed under auto-claving conditions tend to be more stable than the same products formed under ambient conditions.

Any chloride based process has two inherent problems which will make these processes unattractive. First is the corrosion problem in chloride media. The second is the continued build-up of chloride in solution if gypsum is the final sulfur product. This would require a bleed stream to remove chlorides from the leach circuit.

Lime-oxygen Hydrometallurgical Process

As pointed out earlier, the USBM Process was initially developed for the treatment of arsenic containing base-metal ores and concentrates as well as for the flue dusts. In these cases, the utilization of the chloride system was necessary for solubilizing and extracting base metals such as copper, lead, and zinc. On the other hand, for the treatment of arsenopyrite/pyrite ores and concentrates, the utilization of the chloride system is not necessary. For such types of feed materials, the "lime-oxygen" system cab be the preferred alternative.

In this "lime-oxygen hydromet" process, the arsenopyrite ores or concentrates are subjected to low temperature (80-120°C) and pressures (50 to 125 psig) using compressed air and/or oxygen for 0.5 to 3 hours.

After depressurization, the pulp is transferred to an agitator, to which lime hydrate is added. Agitation is continued at a temperature of 90°C, for a period of 0.5 to 2 hours, during which time the soluble sulfur and arsenic is "fixed" as insoluble calcium sulfate and calcium arsenate. The amount of lime hydrate used depends on the sulfur and arsenic content of the feed but is usually about 15 percent weight of the feed. This treatment appears to reduce the solubility of arsenic in the residue to below the permissible 5 ppm.

Moreover, the residues resulting from the reaction are readily cyanidable under standard conditions with little or no settling and filtration problems.

MSRDI has tested several arseno-pyrite concentrates using the above "lime-oxygen" technique and in most of the cases, gold recoveries in excess of 94 percent were achieved as shown in the Table I.

In general, it is economically feasible to treat the flotation concentrates rather than the ores. However, in some cases, it is not possible to achieve high recoveries of gold/silver by flotation. In such instances, it would be necessary to treat the ore. Since the ore may not contain sufficient sulfide sulfur to form the insoluble calcium iron-sulfide-arsenic compounds, it may be necessary to add H_2SO_4 to the autoclave. MSRDI is currently investigating the treatment of arsenic ores using this "lime-oxygen" system, and the results of these tests will be reported in the near future.

It should be noted that the "lime-oxygen" technique is applicable to carbonaceous ores and concentrates containing arsenopyrite. However, in this case, it is necessary to have activated carbon present in the cyanidation step to prevent reabsorption of gold into the natural carbon. On the other hand, the treatment of graphite ores in which some of the gold values are intimately associated with graphite, the "lime-oxygen" technique did not respond as well as in the previous cases. In such cases, it would be beneficial to float the non-arsenic containing graphite after the treatment and treat it by roasting to recover the residual gold values. It is doubtful if even higher tempera-tures and pressures would be effective for decomposing this graphitic component. MSRDI plans to investigate this possibility in the near future.

The "lime-oxygen" process is also applicable to other metallurgical operations where arsenic is a problem. These include treatment of dross and speiss (in lead smelting operations), acid sludge from smelters, and flue dusts.

Plant Design Considerations

Since the temperatures and pressures in the "lime-oxygen" technique are comparatively lower than in conventional acid autoclaving systems, the materials of construc-tion for the lower temperature/ pressure autoclaves would be

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comparatively less stringent. In this case, a stainless steel shell with fiberglass liner, or a polymer liner (polypropylene), may be sufficient to provide the required corrosion and erosion protection. Costlier steel shells, brick lining, glass liners, or titanium construction are not necessary for the proposed "lime-oxygen" system. However, a titanium impeller may be necessary.

Based on the above requirements, the "lime-oxygen" system should be less capital and operating cost intensive for a given retention time. Preliminary estimates indicate costs which are about one-half to one-third the cost of conventional autoclaving. An oxygen plant which is a must for conventional autoclave plants is not necessary and aeration with an air compressor is sufficient. This is an additional economic advantage of considerable cost importance.

Based on the above considerations, it is estimated that a 300 tpd plant treating arsenopyrite concentrates by the "lime-oxygen" process would cost around \$9 to \$12 million with an operating cost around \$35 to \$45 per ton of concentrate treated.

Conclusions

Based on the above discussion, it is evident that the proposed "lime-oxygen" process appears to be a very attractive alternative for treating arsenic containing ores, concentrates, and arsenic contaminated products. The inherent advantages of this process are:

- The process is relatively simple and utilizes low temperature (about 100°C) and pressures (50 to 70 psig).
- Because of the low pressures involved, the process can be carried out with compressed air rather than using an expensive oxygen plant.
- The reaction is completed in a relatively short period: 30 minutes to 1 hour.
- The arsenic is fixed in an insoluble complex in the residue and that this residue meets the EPA toxic test for arsenic.
- Because of the above simplicity and effectiveness of the process, the required capital and operating costs would be much lower than for other alternative processes such as conventional autoclaving, roasting, etc.
- Besides treatment of arsenic containing ores and concentrates, the process is applicable to the treatment of arsenic contaminated dross, speiss, smelter acid sludges, and industrial waste products.
- It is hoped that this proposed "lime-oxygen" process, with its technical and economic advantages, will provide the mining industry worldwide a simplified process for treating arsenic containing mineral products and industrial wastes.

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Table I

Results of Lime-Oxygen Hydromet Process

Product	Origin	Metal Content: Percent Arsenic	Metal Content: Assay (g/mt) Gold	Metal Content: Assay (g/mt) Silver	% Recovered By Cyanidation After Pressure Oxidation: Gold	% Recovered By Cyanidation After Pressure Oxidation: Silver	Analysis of Soluble Metal in Cyanide Residue (in p.p.m.): Arsenic	Analysis of Soluble Metal in Cyanide Residue (in p.p.m.): Mercury
Flotation Concentrate	Greece	10.3	23.6	24.0	99.2	91.4	1.0	--
Flotation Concentrate	Canada	10.1	28.8	8.2	96.6	52.2	2.8	--
Flotation Concentrate	U.S.A.	0.04	291.8	240.7	99.3	69.2	ND	--
Flotation Concentrate	Australia	19.6	92.6	18.9	97.3	80.0	0.6	ND
Flotation Concentrate	U.S.A.	7.4	55.5	36.0	98.5	79.6	1.8	--

ND = None Detectable

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1. Haileybury School Of Mines - Catterello & Cattarello: A flotation study of the Venus tailings concentrate January-May 1988, Haileybury, Ontario.

2. A Study Into The Feasibility of Small Scale Custom/Portable Milling in the Yukon by UMA Engineering Ltd., R.W.Kenway, P.Eng. and G.W.Hawthorn, P.Eng., mineral processing engineer - April 1987 for Economic Development Mines and Small Business, Energy Mines Branch, Government of Yukon.

3. Mountain States R & D International Inc. - Simplified Treatment of Refractory Gold Ores and Concentrates by L. Lichty, G. Ramadoral, R. Bhappu and R. Roman, March 1991, Vail-Arizona USA.

4. Vancouver Petrographics Ltd. - Petrographic report of four polished thin sections by John G. Payne, May 1989, Fort Langley, British Columbia.

5. United Keno Hill Mines Ltd. - Geological evaluation of the 1970-71 abandoned tailings pond deposit. 1980 Venus tailings pond test drill hole data and related assays of grade by UKHM Exploration Department, Whitehorse, Yukon.

6. Baseline Study of The Watershed Near Venus Mine Yukon and Venus Mill British Columbia by Department of Environment Environmental Protection Service - Yukon Branch by Mary Ellen Jack, October, 1981, Whitehorse, Yukon.

7. The Effect of The Abandoned Venus Mine Tailings Pond on the Aquatic Environment of Windy Arm, Tagish Lake, Yukon Territory by W. Robson and K. Weagle, May 1978, Fisheries and Environment Canada, Yukon Branch, Whitehorse, Yukon.

8. Terramin Research Labs Ltd., Analytical Reports 1988, 1989, 1990, 1992, Calgary, Alberta.

ECONOMIC PRELIMINARY FEASIBILITY

1. Venus Tailings Volume Calculations 1992 - Analytical Report, Specific Gravity 2.94

   Venus Tailings Weighted Average Grade Calculations - Analytical Report - Gold Silver Values 1992.

   UKHM 1980 Venus Tailings Drill Hole Locations

VENUS TAILINGS - WINDY ARM, Y.T.
VOLUME CALCULATIONS 1980 DATA

Hole	Au oz/t	Ag oz/t	Depth ft	Volume cf	Volume tons
1	.05	1.38	6.0	32,613	3,000
2	.05	1.66	10.0	41,775	3,840
3	.09	1.99	7.5	23,911	2,200
4	.11	1.85	5.0	28,863	2,650
5	.06	1.30	4.0	26,346	2,400
6	.02	1.23	3.0	28,074	2,580
7	.10	1.31	4.0	16,884	1,550
8	.13	1.08	5.0	31,343	2,880
9	.13	1.19	5.0	14,539	1,330
10	.11	1.44	2.5	7,256	670
11	.02	1.13	7.5	35,307	3,240
12	.13	1.49	17.5	105,600	9,700
13	.05	1.45	20.0	113,408	10,420
14	.07	0.82	12.5	71,863	6,600
15	.08	1.68	7.5	47,525	4,360
16	.11	1.41	7.5	52,066	4,780
17	.15	1.37	7.5	21,181	1,940
18	.09	0.95	7.5	56,188	5,160
19	.13	1.20	22.5	149,708	13,760
20	.09	1.14	12.5	69,520	6,690
21	.12	1.00	12.5	45,910	4,220
22	.17	1.28	2.0	5,610	510
				1,025,484 cf	94,480 tons

1) Specific gravity tailings 2.94 and tonnage factor 10.88 cf/ton.

R.G.Hilker, P.Eng. volume calculations August 24, 1992.

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TERRAMIN RESEARCH LABS LTD.

ANALYTICAL REPORT

Tron Duik Consultants Ltd.

Bob Hilker

Date: June 1, 1989

Job No: 89-108

Project: "VENUS"

P.O. No:

9 Sands

Signed: ______________

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14-2235 30th Avenue N.E., Calgary, Alberta, T2E 7C7
Phone (403) 250-9460   Fax (403) 291-7064

Job#: 89-108

Project: "VENUS"

	Sample Number	Au ppb	Au oz/ton	Ag ppm	Ag oz/ton	As %	Spec Grav
Heads	7611	4560	0.133	61.6	1.80	6.1	2.94
	7612	6600	0.193	75.6	2.21	6.3	
	7613	5040	0.147	64.1	1.87	6.2	
Tails	7614	5800	0.169	55.8	1.62		
	7615	7140	0.208	80.2	2.34		
	7616	4620	0.135	54.5	1.59		
Conc's	7617	68000	1.99	187.0	5.46		
	7618	54200	1.58	164.0	4.79		
	7619	31200	0.911	138.5	4.04		

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VENUS TAILINGS - WINDY ARM, Y.T.
WEIGHTED AVERAGE CALCULATIONS 1980 DATA

Hole	Volume	Depth	Au	Ag	fAu	fAg
1	3,000	6.0	0.05	1.38	150	4,140
2	3,840	10.0	0.05	1.66	192	6,374
3	2,200	7.5	0.09	1.99	198	4,378
4	2,650	5.0	0.11	1.85	292	4,902
5	2,400	4.0	0.06	1.30	144	3,120
6	2,580	3.0	0.02	1.23	52	3,173
7	1,550	4.0	0.10	1.31	155	2,030
8	2,880	5.0	0.13	1.08	374	3,110
9	1,330	5.0	0.13	1.19	173	1,583
10	670	2.5	0.11	1.44	74	965
11	3,240	7.5	0.02	1.13	65	3,661
12	9,700	17.5	0.13	1.49	1261	14,453
13	10,420	20.0	0.05	1.45	521	15,109
14	6,600	12.5	0.07	0.82	462	5,412
15	4,360	7.5	0.08	1.68	349	7,325
16	4,780	7.5	0.11	1.41	526	6,740
17	1,940	7.5	0.15	1.37	291	2,658
18	5,160	7.5	0.09	0.95	464	4,902
19	13,760	22.5	0.13	1.20	1789	16,512
20	6,690	12.5	0.09	1.14	602	7,626
21	4,220	12.5	0.12	1.00	506	4,220
22	510	2.0	0.17	1.28	87	653
	94,480				8727	123,046

1) Wt.Ave Au = $\frac{8,727}{94,480}$ = 0.0924 oz/ton gold

2) Wt. Ave Ag = $\frac{123,046}{94,480}$ = 1.30 oz/ton silver

R.G.Hilker, P.Eng. calculations August 24, 1992.

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TERRAMIN RESEARCH LABS LTD.

ANALYTICAL REPORT

Tron Duik Consultants Ltd.

Bob Hilker

Date: August 28, 1992

Job No: 92-222

Project: Venus Tailings

P.O. No:

7 Composites

Signed: [signature]

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14-2235 30th Avenue N.E., Calgary, Alberta,T2E 7C7
Phone (403) 250-9460 Fax (403) 291-7064

Job#: 92-222

Project: Venus Tailings

Sample Number	Au ppb	Au oz/ton	Ag ppm	Ag oz/ton	S.G.
4501	20380	0.593	87	2.54	
4502	4760	0.139	42	1.22	
4503	3820	0.111	38	1.11	
4504	3460	0.101	47	1.37	3.03
4505	5520	0.161	45	1.31	
4506	3520	0.102	47	1.37	
4507	5260	0.153	49	1.43	3.03

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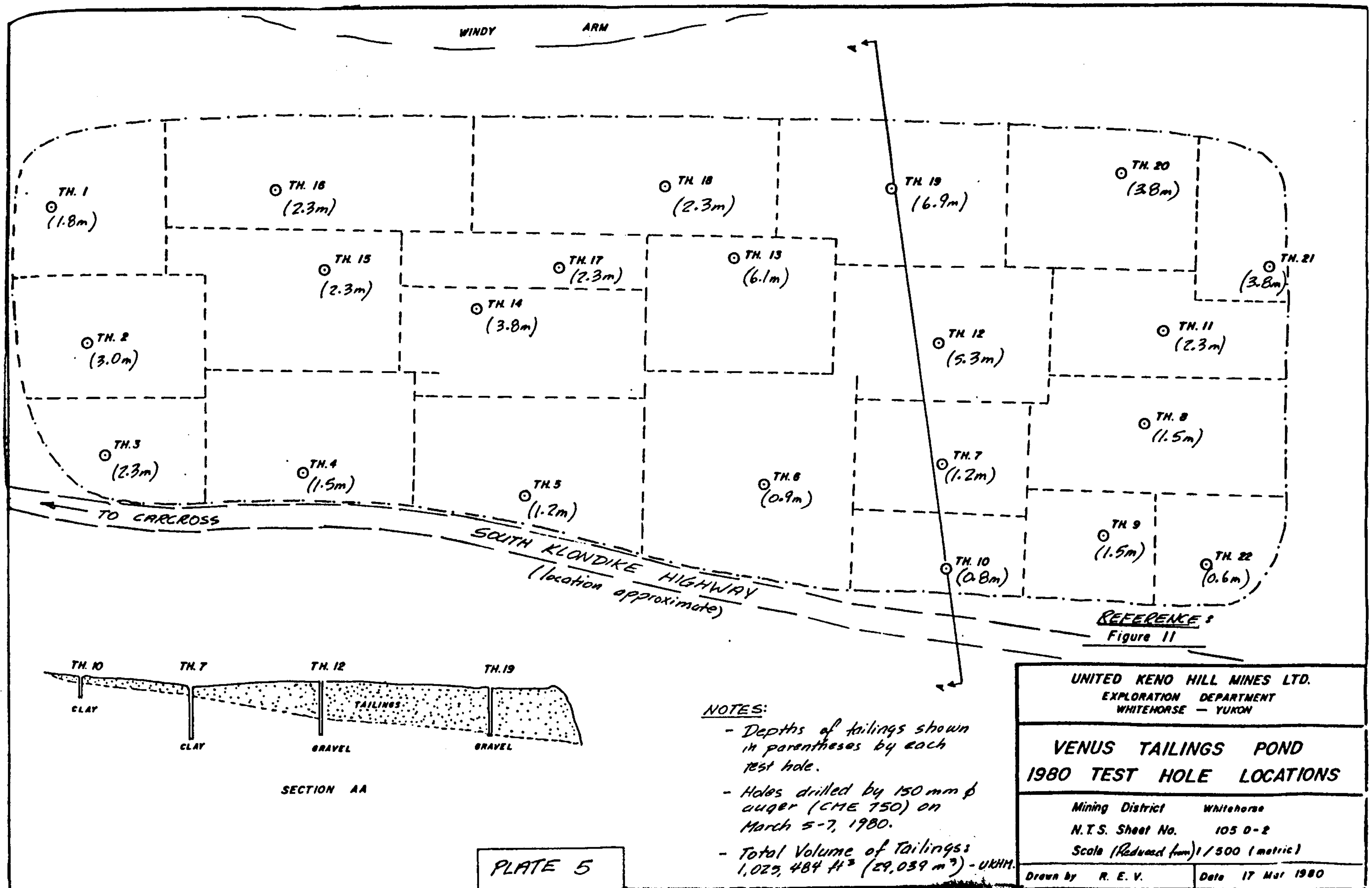
WINDY ARM
TH. 1 (1.8m)
TH. 16 (2.3m)
TH. 18 (2.3m)
TH. 19 (6.9m)
TH. 20 (3.8m)
TH. 21 (3.8m)
TH. 15 (2.3m)
TH. 17 (2.3m)
TH. 13 (6.1m)
TH. 14 (3.8m)
TH. 2 (3.0m)
TH. 12 (5.3m)
TH. 11 (2.3m)
TH. 8 (1.5m)
TH. 3 (2.3m)
TH. 4 (1.5m)
TH. 5 (1.2m)
TH. 6 (0.9m)
TH. 7 (1.2m)
TH. 9 (1.5m)
TH. 10 (0.8m)
TH. 22 (0.6m)
TO CARCROSS
SOUTH KLONDIKE HIGHWAY
(location approximate)
REFERENCE: Figure 11
TH. 10
TH. 7
TH. 12
TH. 19
TAILINGS
CLAY
CLAY
GRAVEL
GRAVEL
SECTION AA
NOTES:
- Depths of tailings shown in parentheses by each test hole.
- Holes drilled by 150 mm ϕ auger (CME 750) on March 5-7, 1980.
- Total Volume of Tailings: 1,025,484 ft³ (29,039 m³) - UKHM.
UNITED KENO HILL MINES LTD.
EXPLORATION DEPARTMENT
WHITEHORSE — YUKON
VENUS TAILINGS POND
1980 TEST HOLE LOCATIONS
Mining District Whitehorse
N.T.S. Sheet No. 105 D-2
Scale (Reduced from) 1/500 (metric)
Drawn by R. E. V.
Date 17 Mar 1980
PLATE 5

Projected Possible Secondary Flotation Concentrate

1. Drilled Volume and Weight Average Grade

- Volume Tailings (Hilker 1992)............. 94,480 tons
- Volume Tailings Removed (1981)............ 14,480 tons
- Volume Venus Tailings Deposit .... 80,000 tons

- Weighted Average Grade Gold............... 0.0924 oz/ton
- Weighted Average Grade Silver............. 1.3000 oz/ton

2. Haileybury School of Mines - Carlo Cattarello re-flotation tests of Venus tailings 1988 indicated a recovery of 80-85% gold or 83% Au and a recovery of 50-60% silver or maximum of 50% Ag.

   - 1988 "test flotation concentrate" assayed
     Gold - 1.0 - 2.0 oz/ton
     Silver - 2.0 - 4.0 oz/ton

3. Therefore: The Venus tailings testing indicates that by re-flotation the deposit could have a precious mineral inventory in a concentrate.

   - Gold 80,000 tons x 0.0924 oz/t x 0.83 = 6,135 oz Au.
   - Silver 80,000 tons x 1.3 oz/t x 0.50 = 52,000 oz Ag.

   - Potential Value Gold: 6,135 oz x $400/oz= $2,452,000
   - Potent.Value Silver: 52,000 oz x 4.00 oz= $208,000
   - Potential Value Precious Metals........... $2,660,000

Potential Volume Secondary Flotation Concentrate

1. The Haileybury 1988 testing indicated that a flotation concentrate potentially contains 1.0-2.0 oz/ton gold.

   - Therefore concentrate 1.5 oz/ton Au ÷ wt. ave. gold in tailings 0.0924 oz/t is a "volume reduction" of f 16.23.

   - Volume Flotation Concentrate 80,000 tons ÷ f16.23 = 4,900 tons of concentrate that potentially contains 1.5 oz/t gold and 2.0 oz/t silver plus pyrite, arsenopyrite and quartz. Potential gold recovery in 4,900 tons of concentrate that contains 1.5 oz/t Au is 7,350 oz of gold.

   - The cost of running 300 TPD or 9,000 tons per month in a portable mill is estimated to be $18-$20 per ton and includes labour, loading into mill, flotation, reagents, power and effluent treatment.

   - Therefore the cost of re-flotation of the Venus tailings possibly costs 80,000 tons x $19 per ton = $1,520,000.

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2. Potential Value Precious Minerals in Concentrate

   - Potential gold recovery of 7,350 oz/ton gold and 52,000 oz/ton silver contained in 4,900 tons of re-flotation concentrate.

   - Potential Value Gold 7,350 oz x $400/oz = $2,940,000
   - Potential Value Silver 52,000 oz x $4.00/oz = 208,000

   - Potential Value Precious Metals $3,148,000

Note: Further grinding and flotation testing is required to determine and confirm the preliminary flotation test work conducted in 1988.

Projected Economic Autoclave Recovery

The Mountain States R & D International Inc. method of autoclave tests on a flotation concentrate is a potential treatment of the refractory gold tailings. The estimated cost of Phase I testing is U.S. $5,000.00 and further lime-oxygen temperature and pressure testing may be required.

1. For the purpose of this report and preliminary feasibility economics of the Venus tailings, the writer prognosticates that 4,900 tons of re-flotation concentrate could be treated by Mountain States recover system at an estimated cost of $100 per ton from a rented portable tank system.

   - Therefore, projected possible cost of treating 4,900 tons of concentrate at $100/ton = $500,000.

Potential Economic Summary Venus Project

1. Drilled Volume Tailings.............. 80,000 tons
   Weighted Average Gold................. 0.0924 oz/ton
   Weighted Average Silver .... 1.3 oz/ton

2. 1988 Re-flotation Tests Indicate:
   - Recovery 80-85% gold or 83% in tailings
   - Recovery 50-60% silver or 50% in tailings

   1988 Test Flotation Concentrate Assayed
   - Gold 1.0-2.0 oz/ton or average 1.5 oz/t
   - Silver 2.0-4.0 oz/ton

3. Method One - Calculation of Precious Metals
   - Gold 80,000 tons x 0.0924 oz/t x 83% = 6,135 oz
   - Silver 80,000 tons x 1.3 oz/t x 50% = 52,000 oz

   - Value Gold 6,135 oz x $400/oz = $2,452,000
   - Value Silver 52,000 oz x $4.00/oz = 208,000

   - Potential Value Precious Metals = $2,660,000

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4. Potential Volume Flotation Concentrate - 4,900 tons

   - Potential 1.5 oz/t gold per ton concentrate x 4,900 tons = 7,350 oz Au.
   - Cost of re-flotation 80,000 tons x $19/ton - $1,520,000.

5. <u>Method Two</u> - Calculations of Precious Metals

- Potential gold recovery 7,350 oz x $400/oz =	$2,940,000
- Potential silver recovery 52,000 oz x $4.00/oz =	208,000
- Potential Value Precious Metals	$3,148,000

6. Potential Recovery Costs of Precious Metals

- Re-flotation 80,000 tons x $19/ton -	$1,520,000
- Lime-oxygen temperature pressure method	500,000
Potential Total Cost of Recovery -	$2,020,000

<u>Potential Net Profit From Venus Tailings</u>

1. <u>Method One</u>: Precious Metal Recovery

- Potential gold recovery 6,135 oz Au =	$2,452,000
- Potential silver recovery 52,000 oz Ag -	208,000
- Potential Value Precious Metals...........	$2,660,000
- Potential Cost of Recovery................	2,020,000
- Potential Net Profit......................	640,000

2. <u>Method Two</u> - Precious Metal Recovery

- Potential gold recovery 7,350 oz Au -	$2,940,000
- Potential silver recovery 52,000 Ag -	208,000
- Potential Value Precious Metals...........	$3,140,000
- Potential Cost of Recovery	2,020,000
- Potential Net Profit	$1,120,000

<u>RECOMMENDATIONS</u>

1. Further flotation testing be conducted on samples collected from the Venus tailings deposit. Estimated cost $10,000.00

2. Testing of Venus tailings by Mountain States R & D International Inc. Estimated costs Phase I - U.S. $5,000 and possible Phase II - U.S. $5,000.

R. G. HILKER
GEOLOGICAL
ENGINEER

R.G.Hilker, P.Eng.
Calgary, Alberta
February 27, 1993.

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CERTIFICATION

I, ROBERT G. HILKER, of 324 Silver Valley Rise N.W., in the City of Calgary, in the Province of Alberta, Canada, DO HEREBY CERTIFY:

1. THAT I am a Consulting Geological Engineer with an office located at 324 Silver Valley Rise N.W., in the City of Calgary, in the Province of Alberta, T3B 4B2.

2. THAT I am a graduate of Michigan Technological University located at Houghton, Michigan, U.S.A., where I obtained a Bachelor of Science Degree in Geological Engineering (Exploration Option) in 1962.

3. THAT I am a registered Professional Engineer (Geological); in the Association of Professional Engineeers, Geologists and Geophysicists of Alberta-#38356; The Association of Professional Engineers of the Yukon Territory-#98.

4. THAT I have practised my profession as an engineer and geologist for the past thirty years.

5. THAT I have personally examined the Venus tailings pond deposit and sampled the concentrate since October 1988 and recently examined and sampled the Venus tailings on September 28, 1991.

6. THAT I have personally prepared the Venus Placer Geological Report and Data compilation Evaluation: Effective Date September 3, 1992, located on the Sandpiper 2-P32656 Yukon Placer claim, Whitehorse Mining District, NTS Sheet 105-D-2. The writer prepared documented report data by field examination, analytical reports of samples collected from the tailings pond and researching metallurgical literature for precious metal recovery of refractory ores and concentrates.

7. THAT I have a direct and contingent interest in the Sandpiper-2 Yukon Placer claim that overlays the Venus tailings pond gold deposit.

Dated this 27th day of February 1993 and Effective Date of Report September 3rd, 1992.

Association of Professional Engineers of the Yukon Territory
R. G. HILKER
GEOLOGICAL
ENGINEER

R.G. Hilker

R.G. Hilker, P.Eng.

TRON DUIK CONSULTANTS LTD.
324 Silver Valley Rise N.W.
Calgary, Alberta T3B 4B2

Report Expenditures
Venus Placer Geological Report
Conrad Gold District, Yukon
Windy Arm Area of Tagish Lake

Effective Date September 3, 1992.

INVOICE

Field travel in Yukon September 28, 1991, analytical reports, compilation of technical data and professional fees

1.	Yukon Field Travel Sept. 28, 1991.............		$450.00
	- Field Expenses & Vehicle.........	$150.	
	- Professional Fees................	300.	
2.	Report Preparation Expenses..................		1,900.00
	- Office expenses & duplications -	$300.	
	- Professional fees 4 days x $400 -	1,600.	
	TOTAL REPORT COSTS.................		$2,350.00

Note: R.G. Hilker certifies that a minimum of $1,100 expenditures was made on the Sandpiper-2 Placer Claim for 5-1/4 years assessment work.

Association of Professional Engineers of the Yukon Territory
R. G. HILKER
GEOLOGICAL
ENGINEER

R.G.Hilker, P.Eng.

VENUS TAILINGS

ENVIRONMENTAL SECTION

from

Baseline Study of the Watershed Near Venus Mine,

Yukon and Venus Mill, B.C.;

Regional Program Report No. 81-18 by Mary Ellen Jack,

October 1981 (Venus Tailings quoted sections).

R.G.Hilker, P.Eng.

Report September 3, 1992

TABLE-1
STATION LOCATION AND DESCRIPTION
"BASELINE STUDY VENUS TAILINGS-REPORT 81-18, M.E.JACK"

STATION NUMBER	LOCATION AND DESCRIPTION
16	Station 16 was at the 792m adit of Venus mine which is located 9.6 km north of the new mill site. The sampling location was at a site where adit water flowed down beneath mining waste rock and surfaced. This was approximately 30m below the main adit water pipe and 120m above the Skagway road.
17	Station 17 was located in Windy Arm approximately 20m offshore from the Venus mine site.
18	Station 18 was located on a creek that discharges from a metal culvert. The culvert is situated beside the abandoned Venus Mine tailings pond. Station 18 was at the discharge end of the culvert and before the tailings drainage joined up with the creek.
19	Station 19 was located at the discharge end of an asbestos culvert draining the tailings pond. This culvert is about 6m below the tailings pond (vertical measure) and approximately 40m away (horizontal measure).
20	Station 20 was located at the base of a rock knoll (5-7m high), which borders a corner of the abandoned Venus Mine tailings pond. Seepage is indicated by discoloration of the rock face and surrounding ground. The lake surface is 2-3m (horizontal measure) away.
21-23	Stations 21, 22 and 23 were located in an arc 20-30m offshore from the abandoned Venus Mine tailings pond. See Figure 3 for exact locations.

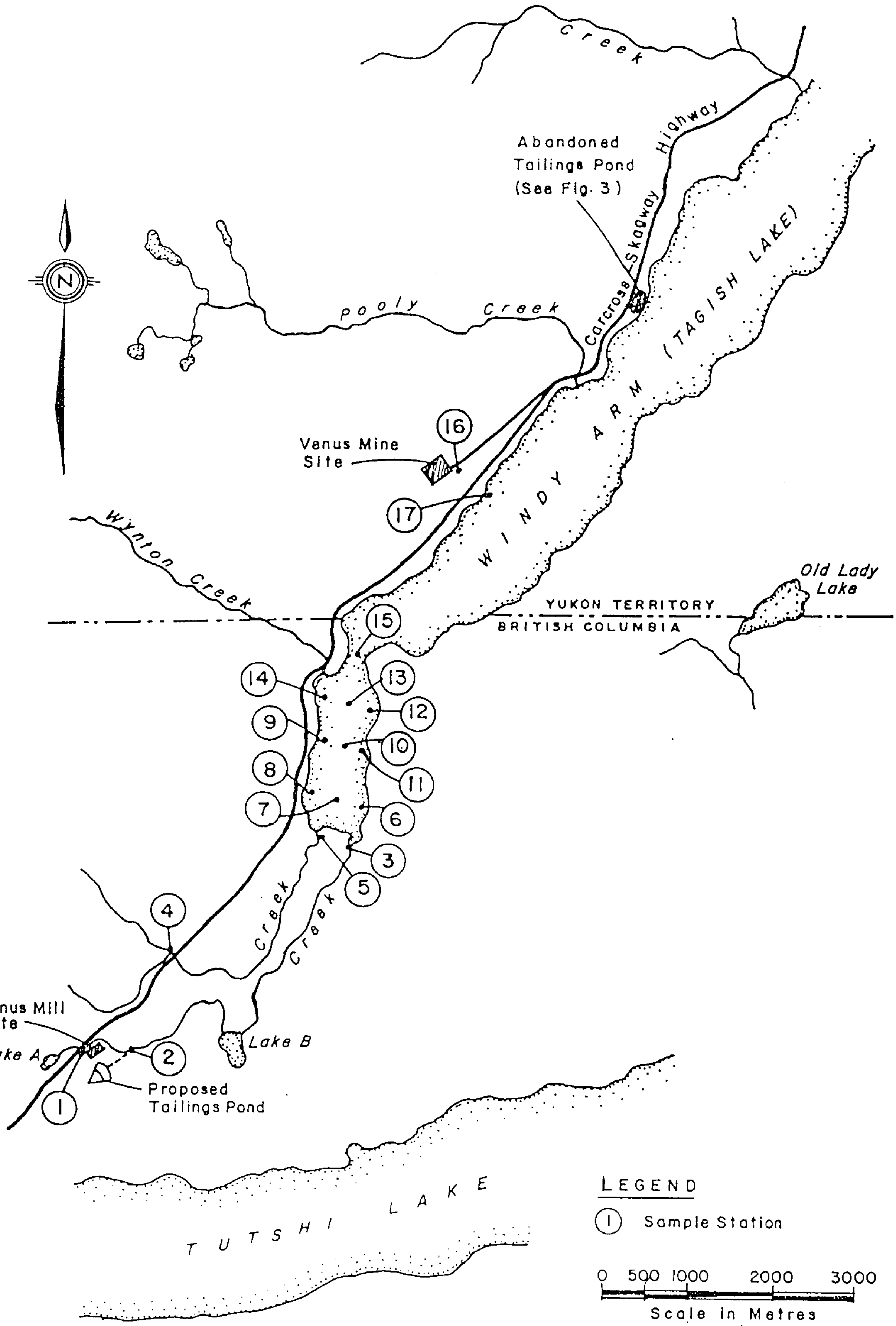

FIGURE 2 SAMPLE STATION LOCATIONS IN STUDY AREA
(See Figure 3 for detail)

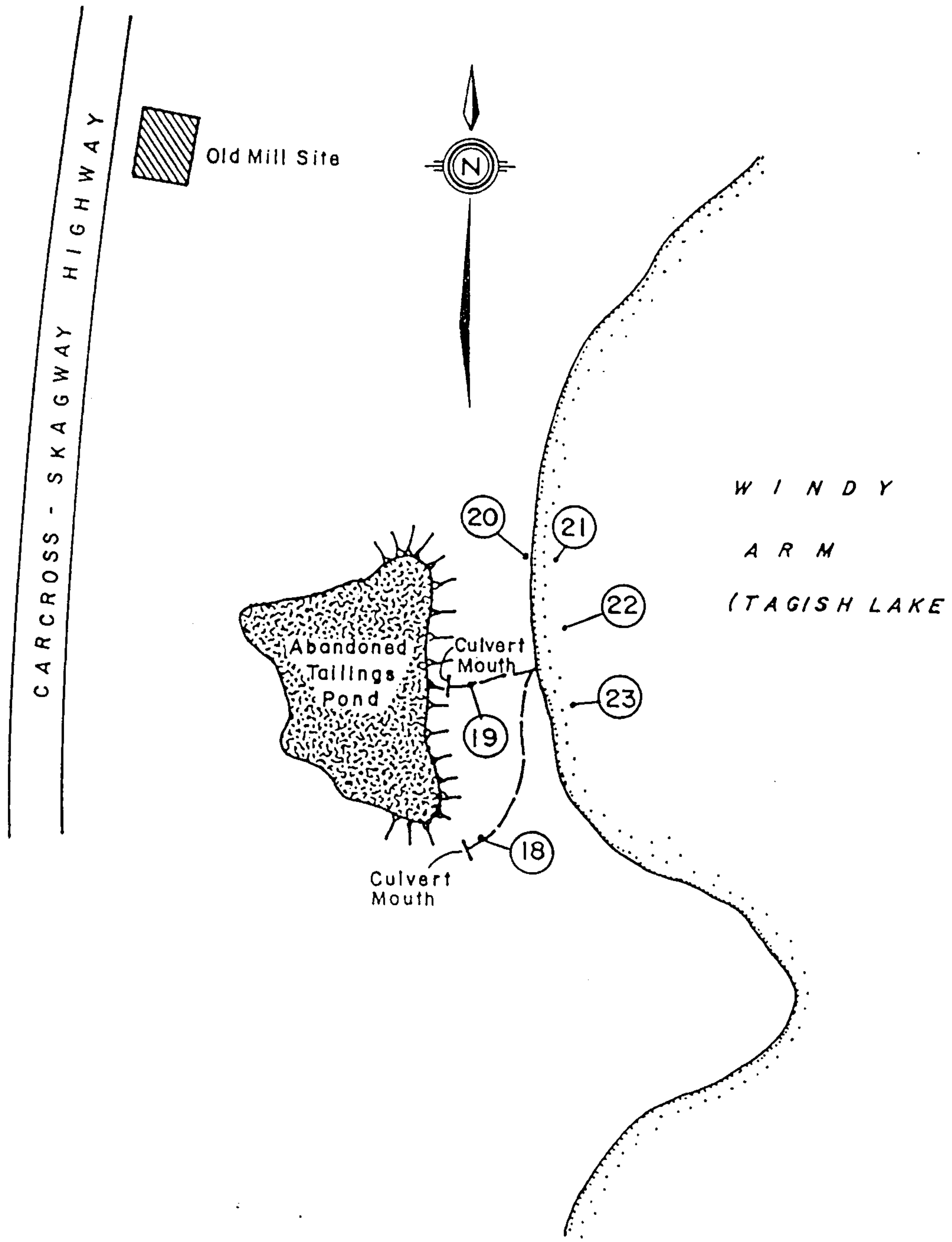

FIGURE 3 DETAIL SKETCH OF ABANDONED TAILINGS POND SHOWING LOCATIONS OF STATIONS 18 TO 23 (Not to Scale)

Subsection 3.1.2.3

Stations affected by past mining and miling operations: 16-23.

The mine water (Station 16), the abandoned tailings pond drainage (Station 19) and the abandoned tailings pond seepage (Station 20) had higher concentrations of some metals than the creek and lake stations that were not affected by past mining activities. Concentrations of arsenic (As), Calcium (Ca), iron (Fe), magnesium (Mg), sodium (Na), silicon (Si), strontium (Sr) and zinc (Zn) were higher. Only the concentration of barium (Ba) was lower than at unaffected creek and lake stations. Only iron concentrations were higher than unaffected lake stations in the lake stations offshore from the abandoned tailing pond. It is possible that arsenic, lead and zinc were higher too but the detection limits of the analytical methods used for these metals were not low enough to show this (N.B. See section 3.1.2.4).

Arsenic concentrations exceeded the upper limit for drinking water and the limit for healthy aquatic life in the minewater (Station n16), the abandoned tailings pond drainage (Station 19) and the abandoned tailings pond seepage (Station 20). The source of this is the 10% arsenopyrite in Venus Mine ore and the previously processed arsenopyrite in the abandoned tailings.

The iron concentration exceeded the upper drinking water limit of 0.3 mg/l Fe but was less than the limit for healthy aquatic life of 1.0 mg/l Fe in the abandoned tailings pond drainage (Station 19). The source of this iron is believed to be iron pyrite in Venus Mine ore and in the abandoned tailings.

The manganese concentration was slightly higher than the upper drinking water limit of 0.05 mg/l Mn but lower than the limit for healthy aquatic life of 1.0 mg/l Mn in the drainage from the abandoned tailings pond (Station 19).

The lead concentration was three times the upper drinking water limit of 0.05 mg/l Pb and fifteen times the limit for healthy aquatic life of 0.03 mg/l Zn in the drainage from the abandoned tailings pond (Station 19).

The mercury concentration exceeded the upper limit for healthy aquatic life of 0.0001 mg/l Hg at all stations affected by past mining operations but only to the same extent as other samples taken from the unaffected creeks and lake in this study.

Although both silver and cadmium concentrations were under the upper limit for drinking water for these stations affected by past mining operations, the detection limit of the analytical methods used were not low enough to show if their concentrations exceeded the upper limits for healthy aquatic life. Both metals are present in the abandoned tailings. Silver is a major constituent of the ore and cadmium occurs as a trace element with its concentration linked to the concentration of zinc which is also a major constituent of the ore. Both metals are considered to be very toxic to aquatic life and an upper limit of 0.001 mg/l Ag and 0.0002 mg/l Cd have been recommended.

The drainage from the abandoned tailings pond (Station 19) was compared to the 1975 metals data for this station as reported in Robson et al (1978). The concentrations of arsenic, cadmium, iron, lead and zinc have stayed approximately the same. The concentration of barium appears to have decreased by a factor of ten and the mercury *concentration appears to have increased by a factor of ten. Even allowing for likely differences in analytical techniques the drainage from the abandoned tailings pond has maintained similar metal concentrations for five years and been a continuing source of metals to Windy Arm.*

Subsection 3.1.2.4

General comments.

The detection limits of the analyses used were not low enough to determine whether arsenic, lead and selenium concentrations exceeded the raw drinking water criteria or if arsenic, cadmium, lead, nickel, selenium, and silver concentrations exceeded healthy aquatic life limits. For example, the detection limits for arsenic, lead and selenium were 0.15 mg/l As, 0.080 mg/l Pb, and 0.080 mg/l Se and the drinking water limits were 0.05 mg/l As, 0.05 mg/l Pb and 0.01 mg/l Se. All lake and unaffected creek samples had concentrations below detection limit for these metals. In addition the analyses detection limits for cadmium, nickel and silver were 0.010 mg/l Cd, 0.080 mg/l Ni and 0.030 mg/l Ag and the healthy aquatic life upper concentrations of arsenic, cadmium, silver, nickel, selenium and silver were 0.05 mg/l As, 0.0002 mg/l Cd, 0.005 mg/l Pb, 0.025 mg/l Ni, 0.01 mg/l Se and 0.0001 mg/l Ag. Since concentrations of arsenic, lead and selenium in the unaffected watershed were low, these metals are not a health problem. However, since this was a baseline study, documentation of arsenic, cadmium, lead, nickel, selenium and silver concentrations would have aided in the evaluation of this watershed and provided better baseline information for the purpose of identifying future changes in concentrations.

Subsection 3.2.3

Stations Affected by Past Mining and Milling Operations: 16 to 23.

The numbers of bottom fauna in mine water (Station 16) and in the sample from a stream draining the area beside the abandoned tailings pond (Station 18) were low. Mine water contained 0.3 mg/l As which exceeds the upper limit for healthy aquatic life and this probably caused the low number of bottom fauna. The data was not conclusive however because lower numbers of bottom fauna were found in an unaffected creek (Station 2). The low numbers of bottom fauna at Station 18 were probably caused by high concentrations of cadmium, lead, and zinc and the presence of cyanide in the sediment at this station. The lowest diversity index was also found at Station 18. This reflects the fact that there were a large proportion of the Psectrocladius genus of family Chironomidae at Station 18.

The lake stations offshore from the abandoned tailings pond (Stations 21, 22, 23) were at 5 meters depth and the diversity indices at stations 22 and 23 were much lower than those found by Baker (1979a) at similar depths in Tagish Lake. This is probably the result of proximity to the abandoned tailings pond and the high concentrations of cadmium, lead and zinc in the sediments at these statons.

In conclusion, the bottom fauna data show some effects of the water quality in mine water and in water draining the area of the abandoned tailings pond. As well, the bottom fauna at lake stations offshore from the abandoned tailings pond show the effects of the high metal content in the sediment.

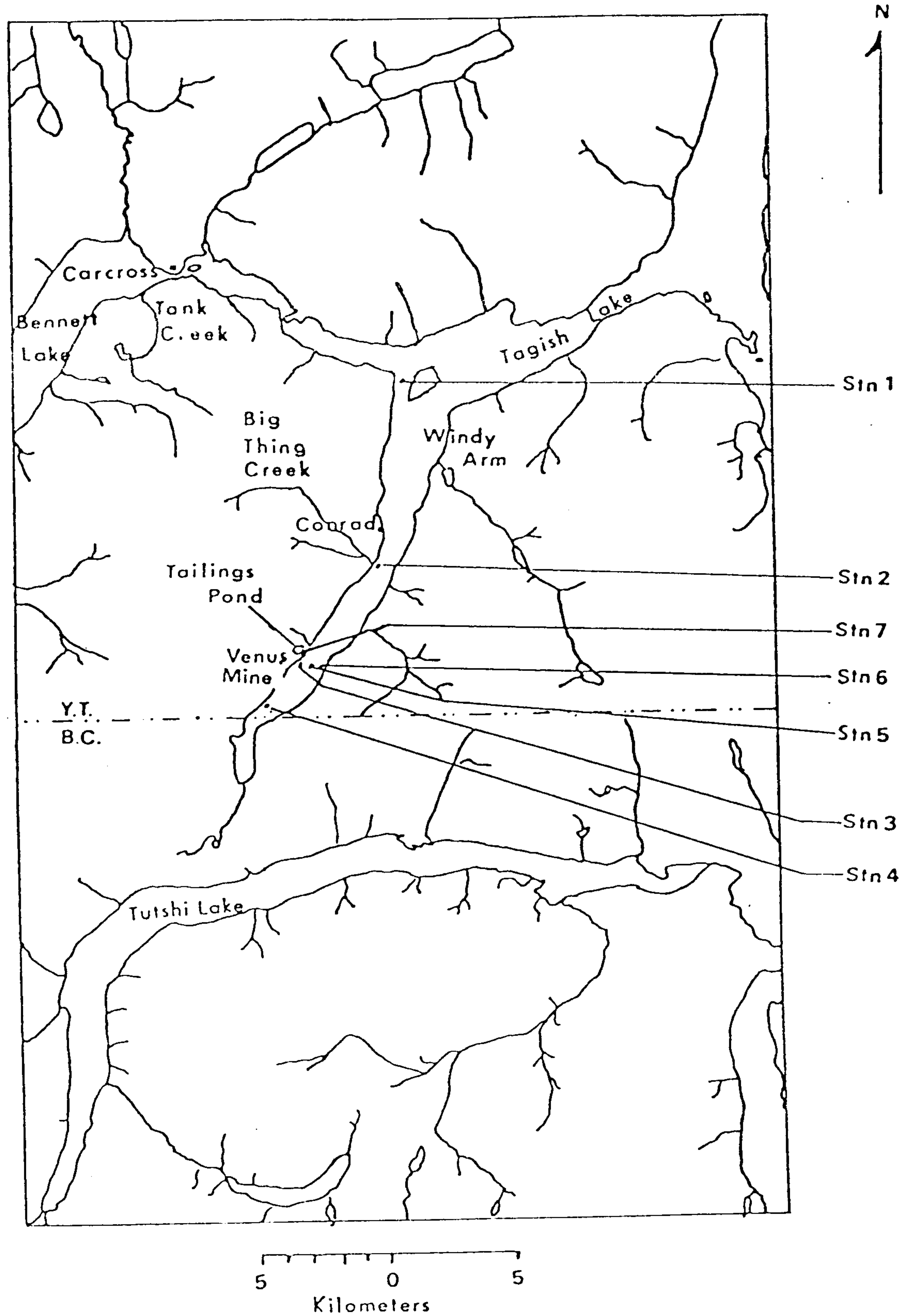

FIGURE 2: MAP OF THE STUDY AREA SHOWING THE SAMPLING STATIONS AND THE LOCATION OF THE ABANDONED TAILINGS POND.

CONCLUSIONS AND RECOMMENDATIONS

Subsection 4.1

Baseline Environmental Characteristics of Predevelopment Creek and Lake Stations

The water quality, bottom fauna, and sediment data from the southern end of Windy Arm (Tagish Lake) and the two creeks that drain into it were similar to other unaltered lakes and creeks in the Yukon. It was noted that creek samples contained more dissolved salts and had higher total hardness than the lake samples. The low dissolved salts and soft water in the lake are characteristic of a low productivity or oligotrophic lake. Since some metals such as lead, nickel and zinc are more toxic to aquatic life in soft water the lake would be particularly sensitive to the addition of metals to an extent which would increase the present concentrations.

There were only two areas of concern:
- *There were two iron concentrations slightly elevated over drinking water criteria. These are not of serious concern and serve as baseline information.*
- *The mercury concentrations in water were all higher than the upper limit of 0.0001 mg/l Hg recommended to protect consumers of fish and mercury concentrations in both water and fish appear to have risen since 1975 and 1976 respectively.*

Subsection 4.2

Environmental Characteristics of Stations Affected by Past Mining and Milling Operations

The continuing effects of past mining and milling operations were shown in water quality, bottom fauna and sediment data of stations close to these locations. The arsenic and iron concentrations in water were elevated in mine water (Station 16) and arsenic, iron, and lead concentrations in water were elevated in the abandoned tailings pond drainage (Station 19). Compared to other lake and stream sediment, significantly higher cadmium, lead and zinc concentrations occurred in the streambed near the abandoned tailings pond (Station 18), in the sediment of the drainage from the abandoned tailings pond (Station 19), and in lake sediments at two stations offshore from the abandoned tailings pond (Stations 22 and 23). Significant concentrations of cyanide were also found in sediment from the streambed near the abandoned tailings pond (Station 18), in the sediment of the drainage from the abandoned tailings pond (Station 19), and in lake sediment from the three stations offshore from the abandoned tailings pond (Stations 21, 22, 23). The bottom fauna from mine water (Station 16) were low in number, probably because of the high concentration of arsenic in the water. However, the data were not conclusive because low numbers of bottom fauna were also found in an unaffected creek (Station 2). The bottom fauna in the stream originating near the abandoned tailing pond (Station 18) had a low diversity index probably because of the presence of cyanide and the high concentrations of cadmium, mercury, lead and zinc in the sediment at this station. There were two main areas of concern:

- *Arsenic concentrations were elevated in the mine water (Station 16) and the drainage and seepage from the abandoned tailings pond (Stations 19 and 20 respectively). Treatment of mine water to remove arsenic during operation is planned by the company when the mine re-opens. The company also plans to recover the tailings from the abandoned tailings pond for reprocessing at their new mill site. Since the arsenic concentration was higher in the abandoned tailings pond drainage* *than it was in the mine water and the tailings are saturated with water, it may also be necessary to treat tailings water to remove arsenic.*

- *Cyanide was found in the sediments of the stream originating near the abandoned tailings pond (Station 18), the drainage from the abandoned tailings pond (Station 19) and the lake stations offshore from the abandoned tailings pond (Stations 21, 22, 23). It is likely that cyanide is stil present in the tailings even though drainage from the abandoned tailings (Station 19) contained no detectable cyanide. Consideration must be given in removing tailings so that cyanide does not become resolubilized into the water from the tailings.*

<u>*Subsection 4.3*</u>

<u>*Recommendations*</u>

1. *That water quality, bottom fauna and the concentration of cyanide and metals in sediment be monitored for changes during and after Venus mill and mine operation in the watershed they affected,*

2. *That water samples at Stations 1 and 15 be collected in 1981 and analyzed by methods having detection limits at or below 0.01 mg/l for arsenic, 0.05 mg/l for manganese, 0.05 mg/l for lead and 0.01 mg/l for selenium since these are the recommended upper limits for drinking water (Anon. 1977). If possible, these water samples should also be analyzed by methods having detection limits at or below 0.0001 mg/l for silver, 0.0002 mg/l for cadmium, 0.0001 mg/l for mercury and 0.005 mg/l for lead, since these are recommended upper limits for healthy aquatic life (Taylor 1980, Reeder 1979a, 1979b, Demayo 1980).*

3. *That concentrations of mercury in fish be routinely monitored in Windy Arm since mercury concentrations have possibly increased over those found in 1976 (Robson 1978) and if these increases show a continuing trend they may exceed the Health and Welfare recommended guideline of 0.5 mg/kg Hg wet weight.*

Source: Environment Canada, Environmental Protection Service Pacific Region, Yukon Branch: Regional Program Report No. 81-18; Baseline Study of The Watershed Near Venus Mine, Yukon and Venus Mill, British Columbia, by Mary Ellen Jack, October 1981 (p8, 16-18, 21, 24-26).