ARCHER, CATHRO
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CONSULTING GEOLOGICAL ENGINEERS

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020317

SUMMARY REPORT

WILLIAMS CREEK COPPER DEPOSIT

CARMACKS AREA, YUKON TERRITORY

MARCH, 1988

ALAN R. ARCHER, P.Eng.

BOX 4127, 3125 THIRD AVENUE, WHITEHORSE, Y.T. Y1A 3S9
(403) 667-4415

TEXT

FIGURES

INTRODUCTION

The Williams Creek copper deposit was found by Dawson Range Joint Venture in 1970 while exploring claims optioned from prospector G. Wing. The venture was financed by Straus Exploration, Inc., Great Plains Development of Canada Ltd., Trojan Consolidated Minerals Ltd. and Molybdenum Corporation of America through an agreement in which Archer, Cathro & Associates Limited acted as manager earning a small interest in net profits and the right to acquire abandoned properties. By 1982, all of the participants had officially abandoned their interests in Williams Creek and the property has been maintained by Archer, Cathro since then.

The G. Wing option agreement carries a residual 10% interest in net profits from any mining venture conducted on the claims. Wing vended his interest to A. Arsenault in October, 1970 and Archer, Cathro optioned the Arsenault interest in late 1987.

HISTORY

The Williams Creek copper deposit was found by prospecting in August of 1970 and was explored by bulldozer trenching and two x-ray holes later that year. Exploration in 1971 included road construction, grid soil sampling, geophysical surveys, trenching, 14,083 feet of drilling in 15 holes on the main zone plus an additional 4,274 feet of drilling on other mineralized zones on the property and metallurgical testing. Work in 1972, which included 5,022 feet of drilling, was directed to other parts of the property. Except for a legal survey of the claims covering the main zone in 1974, no work has been done on the property except for a few hours of trenching in 1987 for assessment purposes.

In 1971, the Dawson Syndicate (Silver Standard Mines Ltd. and Asarco) and United Keno Exploration (United Keno Hill ML, Canadian Superior Exp. Ltd. and Falconbridge Nickel Mines Ltd.) discovered the Minto copper district some twenty-five miles to the northwest. Subsequent work between 1971 and 1974 located the Minto deposit which is jointly covered by claims held by the two syndicates. This deposit has a drill-indicated reserve of 7.22 million tons grading 1.86% Cu, 0.2 oz/ton Ag and 0.015 oz/ton Au and is open pittable with a five to one stripping ratio. A feasibility study conducted in 1976 indicated that production was uneconomic at that time.

United Keno Hill has conducted exploration in the twenty-five mile belt between the two deposits from time to time since 1971 and, in 1977, staked a new area of mineralization about five miles northwest of the Williams Creek deposit from which drilling in 1980 returned up to 3.51% Cu across 44 feet.

CLAIM STATUS

The Williams Creek property presently consists of a contiguous group of 19 full size and fraction mineral claims recorded in the Whitehorse Mining District as follows:

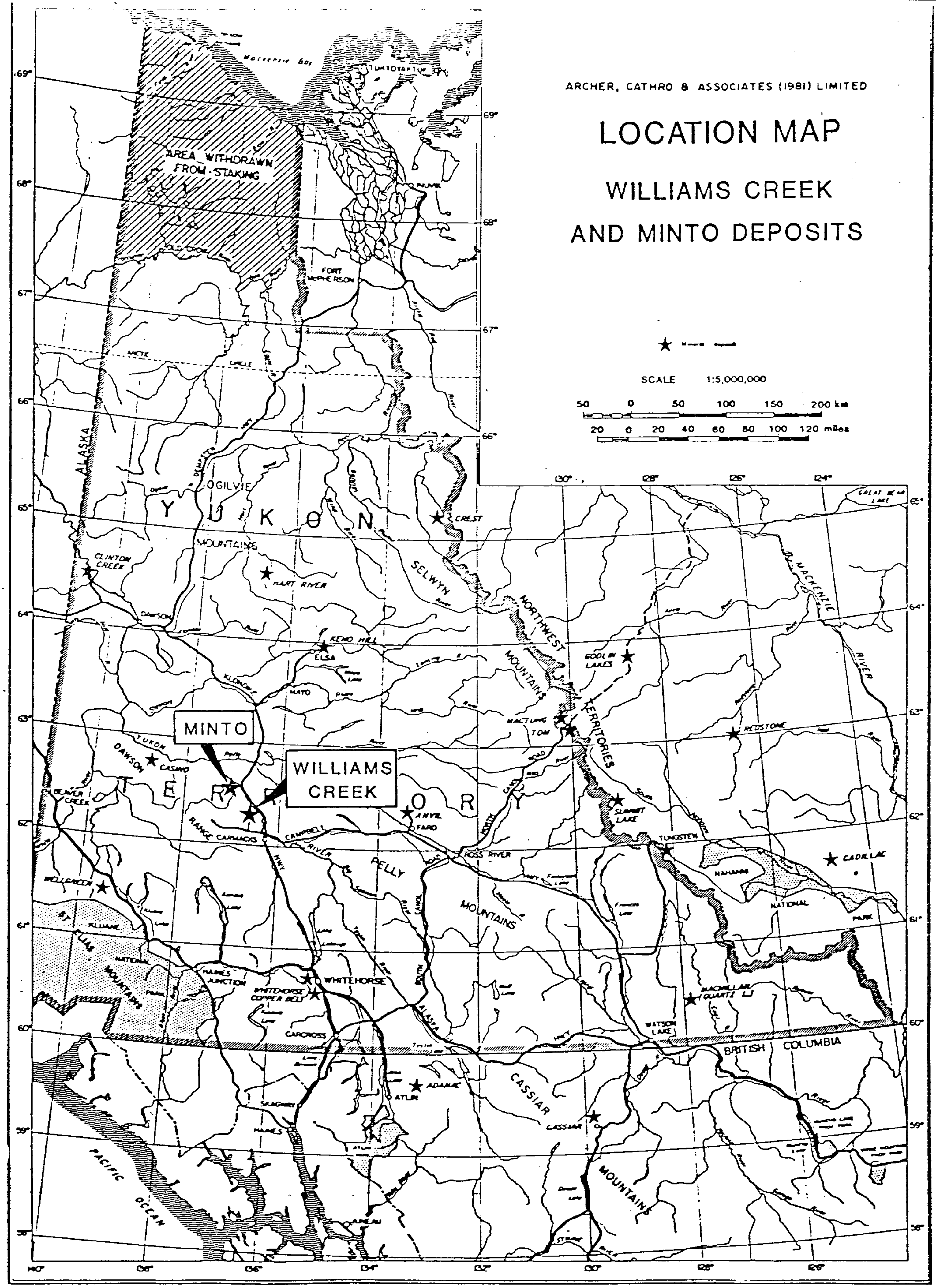
Claim Name	Grant Numbers	Expiry Date
Boy 20	Y5118	9 March/88
Boy 22	Y51120	9 March/88
Boy 24	Y51122	9 March/88
Boy 51-58	Y51149-Y51156	9 March/88
Boy 83	Y51181	9 March/88
Boy 85	Y51183	9 March/88
Dun 1-3F	Y59382-Y59384	9 March/88
War 32	Y59373	21 November/88
AC 2-3F	Y91722-Y91723	21 November/88

Sufficient bulldozer trenching was done in 1987 to bring the claims to a 1990 expiry date.

ACCESSIBILITY

The Williams Creek deposit is accessible by a seven mile long, four-wheel drive summer road north from Mile 22 on the secondary Freegold road that extends west of Carmacks which lies some 110 road miles north of Whitehorse.

The Minto district is accessible by a winter bulldozer road north of Williams Creek or by helicopter from Minto on the Klondike Highway some 12 miles to the east.


ARCHER, CATHRO & ASSOCIATES (1981) LIMITED
LOCATION MAP
WILLIAMS CREEK
AND MINTO DEPOSITS
SCALE 1:5,000,000
50 0 50 100 150 200 km
20 0 20 40 60 80 100 120 miles
AREA WITHDRAWN
FROM STAKING
TUKTOYAKTUK
INUVIK
FORT McPHERSON
ALASKA
OGILVIE
YUKON
MOUNTAINS
TERRITORY
CREST
CLINTON CREEK
HART RIVER
SELWYN
DAWSON
KENO HILL
ELSA
MAYO
NORTHWEST
MOUNTAINS
TERRITORIES
GOOLIN LAKES
MINTO
WILLIAMS
CREEK
MACTUNG
TOM
REDSTONE
CASINO
BEAVER CREEK
ANVIL
FARO
SUMMIT LAKE
TUNGSTEN
CARMACKS
CAMPBELL
PELLY
ROSS RIVER
RIVER
NAHANNI
NATIONAL
PARK
CADILLAC
WELLGREEN
MOUNTAINS
HAINES JUNCTION
WHITEHORSE
WHITEHORSE COPPER BELT
MACMILLAN (QUARTZ LK)
WATSON LAKE
CARCROSS
BRITISH COLUMBIA
CASSIAR
ADANAC
ATLIN
SKAGWAY
HAINES
CASSIAR
MOUNTAINS
PACIFIC
OCEAN
JUNEAU
MACKENZIE
RIVER
GREAT BEAR LAKE
SOUTH
CANOL ROAD
ALASKA
HWY

MINERALIZATION AND ORE RESERVES

The Williams Creek deposit is a steep-dipping (-70° east) tabular zone of weakly schistose rock of diorite composition enclosed by granodiorite. It is believed to be a recrystallized roof pendant of volcanic rocks. The deposit has a sharp footwall contact and a less well defined, somewhat gradational, hanging wall contact. It has a strike length of 1800 feet at surface, reducing to 1100 feet at a depth of 1200 feet below surface and has an average width of 96 feet.

Drilling at approximately 400 foot centres has traced the deposit to 1500 feet below surface and shows that it terminates to the north by absorption into the granodiorite and to the south by an assay cutoff due to increasing pyrite. Mineralization consists of disseminated bornite, chalcopyrite and pyrite with the occasional irregular veinlet of bornite and chalcopyrite and is best developed in the footwall portion of the deposit. Minor gold and silver values are present and appear to be preferentially contained in the bornite. Small quantities of molybdenite are also present.

The schistose host rock has an unusually high porosity and this has resulted in the copper minerals being almost totally oxidized to malachite and azurite to approximately 800 feet below surface. For this reason, reserves have been divided into oxide and sulphide portions. The oxide portion is subdivided into a lower grade near surface portion called "leached oxide" and a deeper higher grade portion called "enriched oxide". Drill-indicated reserves, as outlined on the longitudinal section in pocket, are further divided into footwall reserves and hanging wall reserves under the assumption that underground mining would only be profitable on the better grade footwall mineralization. A summary of drill-indicated reserves follows.

<u>Footwall Reserves</u> (using a 0.6% Cu cutoff)

Mineralized width ranges from 20 to 105 feet and averages 53 feet.

	Million Tons	% Copper	oz/ton Silver	oz/ton Gold
Leached Oxide	1.324	0.79	NA	NA
Enriched Oxide	4.024	1.93	0.228	0.027
Sulphide Ore	4.016	1.35	0.071	0.014
As compared to Minto Deposit @	7.22	1.86	0.200	0.015

<u>Hanging Wall Reserves</u> (using a 0.6% Cu cutoff)

Mineralized width ranges from 17 to 71 feet and averages 43 feet.

Leached Oxide	1.305	0.702	NA	NA
Enriched Oxide	3.041	0.960	0.150	0.020
Sulphide Ore	2.724	0.678	0.084	0.011

Preliminary metallurgical tests on both oxide and sulphide mineralization by Treadwell Corporation of Connecticut in 1971 (see Appendix I) produced unusually favourable results for copper recovery and the Treadwell metallurgist's summary comment was: "In total, I consider this a nearly ideal ore". No test work was done on recovery of precious metals.

Respectfully submitted,

ARCHER, CATHRO & ASSOCIATES (1981) LIMITED

/mc

A.R. Archer, B.A.Sc., P.Eng.

APPENDIX I

RESULTS OF PRELIMINARY METALLURGICAL TESTS

PRELIMINARY TESTS

ON SAMPLES

FROM WILLIAMS CREEK PROJECT

FOR STRAUS EXPLORATION INC.

120 BROADWAY

NEW YORK, NEW YORK

J.W. Goodwin

September 10, 1971

INTRODUCTION

This work was done at the request of Lyman Hart of Straus Exploration Inc.

The work consists of preliminary flotation and acid leaching tests on samples from the Williams Creek Project.

Objectives

There are two objectives for this work. The first is to make an approximation of copper recovery by flotation and leaching.

The second objective is to recover oxide copper by flotation.

Conclusions

1.) The samples are amenible to conventional processing.

2.) Recoveries to solution in excess of 90% of the acid soluble copper are probable with acid consumption of 2.7 lb. per lb. of copper.

3.) Recoveries in excess of 90% of the sulfide copper are probable with a concentrate grade of 50% Cu.

4.) Microscopic examination of the sulfide concentrates indicate that molybdenite may be present in economic amounts.

Summary

Three samples of Williams Creek ore were tested - an oxide ore, a mixed oxide-sulfide ore, and a sulfide ore. Since all three samples responded so well to the first, most obvious treatment methods chosen we did not pursue an oxide copper flotation scheme which we initiated as an alternate to acid leaching.

Oxide Sample

Acid leaching the oxide sample with 10% H_2SO_4 at room temperature for four hours gave us copper extractions of 90.86% and 89.51% with an acid consumption of 100 lb. per ton of ore or 2.5 lb. per lb. of copper. Microscopic examination indicated that sulfide copper accounted for most of the 0.20% remaining in the residue.

Mixed Sample

Acid leaching the mixed ore extracted 29.98% of the copper with a consumption of 2.7 lb. H_2SO_4 per lb. copper. Sulfide flotation recovered an additional 62.66% as a 45.38% Cu concentrate with only one cleaning step. The cleaner tailings accounted for 6.15% of the copper. If we assume that half of

this will eventually be lost to the final tailing, the overall recovery would be 95.71% for the mixed ore.

Sulfide Sample

Rougher flotation of the sulfide sample resulted in 97.30% copper recovery in a concentrate assaying 21.46% Cu. Molybdenite, possibly in economic amount, is present in the sulfide concentrates.

Samples

Three samples were received from the Williams Creek property. The first, which we designated as lot 389, was described as oxide ore. The second, lot 390, was mixed oxide-sulfide ore. And the third, lot 392, was sulfide ore from DD hole No. 7, 970 to 1097 feet.

All samples were crushed in closed circuit with a 10 mesh screen before testing.

Briquetts for microscopic examination were made from all samples and test products, and they are available on your request.
Head analyses follow:

Lot 389 - Oxide	1.82% Cu
Lot 390 - Mixed	3.90% Cu
Lot 392 - Sulfide	1.42% Cu

Williams Creek Project

Screen Analysis

Lot 389 100% Through 10 mesh

			%	Accum. %
+14	- 10	Mesh	8.08	8.08
+ 48	- 14	Mesh	33.93	42.01
+ 65	- 48	Mesh	10.64	52.65
+ 100	- 65	Mesh	9.16	61.81
+ 150	- 100	Mesh	8.32	70.13
+ 200	- 150	Mesh	6.88	77.01
+ 270	- 200	Mesh	3.81	80.82
+ 325	- 270	Mesh	1.81	82.63
	- 325	Mesh	17.37	100.00

Lot 390 Flotation Tailing, 10 min. grind

+ 65		Mesh	.16	.16
+ 100	- 65	Mesh	1.47	1.63
+ 200	- 100	Mesh	26.48	27.95
+ 270	- 200	Mesh	13.74	41.69
+ 325	- 270	Mesh	6.60	48.29
	- 325	Mesh	51.71	100.00

Williams Creek Project

Test 389-2, Oxide Ore

7/21/71

Objective: Determine acid consumption and copper recovery in leach.

Procedure: Leach minus 10 mesh sample in 100 gpl H_2SO_4 at 60% solids. Leach 4 hours at room temperature on rolls.

Data:

Sample No.	Product	Weight GM-L	% or GPL Cu	% or GPL H_2SO_4	Gm Cu	Gm H_2SO_4	% Dist Cu
389-1	Heads	250	1.82		4.55		
389-2-1	Residue	236.82	0.18		0.43		9.14
389-2-2	Preg Soln	.119	27.27	16.04	3.25	1.91	90.86
389-2-3	Wash Soln	1.0	1.02	10.63	1.02	10.63	
389-2-4	Stock Acid	.167		94.10		15.71	
	Calc Head	250	1.87		4.70		

Acid consumption = $\frac{15.71 - 12.54}{250}$ = $\frac{3.17 \text{ gm } H_2SO_4}{250 \text{ gm Ore}}$ or $\frac{25 \text{ lb. } H_2SO_4}{\text{Ton Ore}}$

Note: The acid content of the wash seems high and 6.4 gm H_2SO_4 should have been consumed just to make $CuSO_4$ with 91% of the copper present.

Williams Creek Project

Text 389-3, Oxide Ore

7/21/71

Objective: Determine acid consumption and copper recovery in leach. (Check test 389-2)

Procedure: Leach minus 10 mesh sample in 100 gpl H_2SO_4 at 60% solids. Leach 4 hours at room temperature on rolls.

Data:

Sample No.	Product	Weight GM-L	% or GPL Cu	% or GPL Fe	% or GPL H_2SO_4	Gm Cu	Gm Fe	Gm H_2SO_4	% Distribution Cu
389-3-1	Heads	600	1.88			11.28			
389-3-2	Residue	567.43	0.22			1.25			10.49
389-3-3	Preg Soln	0.212	25.4	2.48	19.16	5.38	0.53	4.06	89.51
389-3-4	Wash Soln	1.0	5.28	0.58	3.97	5.28	0.58	3.97	
389-3-5	Stock Acid	0.400			94.53			37.81	
	Calc Head	600	1.98	0.18		11.91	1.11		

Acid consumption is $\frac{29.87 \text{ 9m } H_2SO_4}{600 \text{ gm Ore}} = \frac{100 \text{ lb } H_2SO_4}{\text{Ton Ore}}$

or $\frac{29.87 \text{ gm } H_2SO_4}{11.91 \text{ gm Cu}} = \frac{2.5 \text{ lb. } H_2SO_4}{\text{lb. Cu}}$

1 - Check 389-1
2 - Check 389-2-4
3 - Acid soluble Fe in heads

Williams Creek Project

Test 390-2, Mixed oxide-sulfide

7/21/71

Objective: Leach oxide fraction and float the remaining sulfide.

Procedure: Leach minus 10 mesh sample in 100 gp. H_2SO_4 at 60% solids. Leach 4 hours at room temperature on rolls.

Grind residue 10 min. in lab rod mill at 60% solids. No reagents.

Float with Z-6 and pine oil.

Data:

Operation	Time	pH	Reagents, lb/ton $Ca(OH)_2$	Z-6	Pine Oil	Remarks
Grind	10'			No Reagents		
Condition	30"	6.5		0.005	.007	
Rougher	5'				.014	Overcollected
Condition	2'		0.8		.007	
Rougher	5'	8.8				Froth better
Condition	2'		1.7			
Rougher	5'	10.0				
Condition	2'					.02 lb/ton fuel oil
Rougher	5'	9.5				F.O. killed froth
Condition Ro Cone	2'	8.0		No Reagents		
Clr. Float	5'					

Sample No.	Product	Weight Gm-L	% or GPL Cu	% or GPL H_2SO_4	Gm Cu	Gm H_2SO_4	Overall % Distribution Cu
390-1	Heads	594.42	3.90		23.18		
390-2-1	Preg Soln	0.221	16.70	49.27	3.69	10.89	29.98
390-2-2	Wash Soln	0.1	3.03	8.68	3.03	8.68	
390-2-7	Stock Acid	0.400		94.53		37.81	
390-2-3	Residue	564.29	2.86		16.14		
	Calc Leach Heads	594.42	3.84		22.86		
Less sample for residue	assay	32.65	2.86		0.93		
	Flotation Heads	531.64	2.86		15.21		
390-2-4	Tailing	446.02	0.06		0.27		1.20
390-2-5	Middling	54.69	2.53		1.38		6.15
390-2-6	Conc	30.93	45.38		14.04		62.66
	Calc Float Heads	531.64	2.95		15.69		
	Calc Overall Heads	594.42	3.92		23.34 (22.41 for % Dist.)		100.00

Acid Consumption is $\frac{18.24 \text{ 9 m } H_2SO_4}{594.42} = \frac{61.37 \text{ lb } H_2SO_4}{\text{Ton Ore}}$

or $\frac{18.24 \text{ 9m } H_2SO_4}{6.72 \text{ 9m Cu}} = \frac{2.7 \text{ lb } H_2SO_4}{\text{lb acid soluble Cu}}$

Williams Creek Project

Test 392-2, Sulfide Ore

8/5/71

Objective: Sulfide recovery by flotation

Procedure: Grind minus 10 mesh ore for 5 minutes in lab rod mill at 60% solids. No reagents. Float with Z6 and pine oil.

Data:

Operation	Time	pH	$Cu(OH)_2$	Z-6	Pine Oil	Remarks
Grind	5'					
Condition	3'	8.6		.005	.007	
Rougher	5'					Good Float
Condition	2'	8.5			.002	
Rougher	Froth is barren - no couc made					
Condition	2'	11.0	2.12	.005		
Rougher	Froth still barren - no couc made					

Sample No.	Product	Weight Gm	Cu %	Cu Gu	Cu % Distribution
392-1	Heads	495.85	1.42	7.04	
392-2-1	Cone	32.0	21.46	6.87	97.36
392-2-2	Tailing	463.85	.041	0.19	2.70
	Calc Head		1.42	7.06	100.0

ORE RESERVES WILLIAMS CREEK PROPERTY No. 1 ZONE - 1971

		FOOTWALL PLUS HANGING WALL						FOOTWALL						HANGING WALL					
CATEGORY	BLOCK No	TONS (000)	TOTAL Cu %	SULFIDE Cu %	OXIDE Cu %	Ag oz/ton	Au oz/ton	TONS (000)	TOTAL Cu %	SULFIDE Cu %	OXIDE Cu %	Ag oz/ton	Au oz/ton	TONS (000)	TOTAL Cu %	SULFIDE Cu %	OXIDE Cu %	Ag oz/ton	Au oz/ton
LEACHED OXIDE (1)*	1	968		NA	NA	NA	NA	496	0.63	NA	NA	NA	NA	472	1.05	NA	NA	NA	NA
	2	291	0.69	NA	NA	NA	NA	153	0.47	NA	NA	NA	NA	138	0.93	NA	NA	NA	NA
	3	1,011	0.86	NA	NA	NA	NA	416	1.20	NA	NA	NA	NA	595	0.62	NA	NA	NA	NA
	4	710	0.54	NA	NA	NA	NA	412	0.78	NA	NA	NA	NA	298	0.22	NA	NA	NA	NA
total no cutoff		2980	0.73					1477	0.785					1503	0.702				
0.6% total Cu cutoff		2529	0.805					1324	0.79					1205	0.823				
0.8% total Cu cutoff		1026	0.10					416	1.20					610	1.02				
ENRICHED OXIDE	5	183	1.56	NA	NA	NA	NA	NIL						183	1.56	NA	NA	NA	NA
	6	314	1.51	0.04	1.47	0.29	0.016	NIL						314	1.51	0.04	1.47	0.29	0.016
	7	308	0.85	0.15	0.70	0.04	0.01	NIL						308	0.85	0.15	0.70	0.04	0.01
	8	1339	0.69	0.13	0.56	0.10	0.02	532	1.27	0.19	1.08	0.10	0.02	807	0.30	0.09	0.21	NA	NA
	9	760	0.69	0.05	0.64	0.138	0.015	~~380~~	0.41	0.07	0.34	0.11	0.01	380	0.97	0.03	0.94	0.165	0.02
	10	1562	1.46	0.06	1.40	0.165	0.008	1304	1.634	0.047	1.587	0.178	0.009	258	0.61	0.11	0.50	0.10	0.003
	11	1717	1.32	0.08	1.24	0.130	0.030	816	1.87	0.09	1.78	0.13	0.03	901	0.82	0.07	0.75	0.13	0.03
	12	2000	1.89	0.30	1.59	0.319	0.034	1.045	2.86	0.44	2.42	0.474	0.05	955	0.82	0.15	0.67	0.15	0.017
	13	1046	0.83	0.34	0.49	0.045	0.012	327	1.44	0.65	0.79	0.1	0.015	719	0.55	0.20	0.35	0.02	0.01
total no cutoff		9229	1.26	0.16	1.10	assumed		4404	1.80	0.21	1.59	assumed		4825	768	0.108	0.66	assumed	
0.6% total Cu cutoff (2) **		7065	1.52	0.17	1.35	not		4024	1.93	0.225	1.705	.228 not .327		3041	960	0.09	0.87	not	
0.8% total Cu cutoff (2) **		4574	1.84	0.15	1.69	recoverable		3697	1.98	0.19	1.79	recoverable		877	1.29	0.03	1.26	recoverable	
SULFIDE	14	3264	1.09	1.09	NIL	.041	0.01	2380	1.24	1.24	NIL	0.02	0.01	884	0.68	0.68	NIL	0.10	0.01
	15	2117	1.25	1.25	NIL	0.16	0.018	1282	1.57	1.57	NIL	0.172	0.021	835	0.77	0.77	NIL	0.144	0.014
	16	1359	0.80	0.80	NIL	0.025	0.01	354	1.35	1.35	NIL	0.04	0.01	1005	0.60	0.60	NIL	0.02	0.01
total no cutoff		6740	1.08	1.08	NIL	0.077	0.0125	4016	1.35	1.35	NIL	0.071	0.0135	2724	0.678	0.678	NIL	0.084	0.0113
0.6% total Cu cutoff		6740	1.08	1.08	NIL	0.077	0.0125	4016	1.35	1.35	NIL	0.071	0.0135	2724	0.678	0.678	NIL	0.084	0.0113
0.8% total Cu cutoff		4016	1.35	1.35	NIL	0.071	0.0135	4016	1.35	1.35	NIL	0.071	0.0135	NIL	NIL	NIL	NIL	NIL	NIL

NOTE (1)* Copper assay of leached oxide zone assumed to be 100% oxide copper
(2)** Cutoff grade was applied to oxide assay rather than total copper assay

TOTAL RESERVES
(using only oxide copper assays for oxide zone)

	FOOTWALL PLUS HANGING WALL		FOOTWALL		HANGING WALL	
CATEGORY	TONS (000)	COPPER %	TONS (000)	COPPER %	TONS (000)	COPPER %
no cutoff	18949	1.04	9887	1.37	9052	0.67
0.6% cutoff	16334	1.15	9364	1.43	6970	0.78
0.8% cutoff	9616	1.485	8129	1.545	1487	1.16

DIAMOND DRILL HOLE ASSAYS

HOLE No.	WALL	TRUE WIDTH (ft)	TOTAL COPPER %	OXIDE COPPER %	SULFIDE COPPER %	SILVER oz/ton	GOLD oz/ton
1 & 2	1970 X-RAY HOLES, ASSAYS NOT RELIABLE						
3	HW	40	1.51	1.47	.04	0.29	0.016
	FW	24	0.20	0.13	.07	NA	NA
4	HW	33	0.97	0.94	03	0.165	0.02
	FW	33	0.41	0.34	07	0.11	0.01
5	HW	64	0.82	0.67	0.15	0.15	0.017
	FW	70	2.86	2.42	0.44	0.474	0.05
6	HW	17	0.61	0.50	0.11	0.10	.003
	FW	86	1.634	1.587	0.047	0.178	.009
7	HW	41	0.77	NIL	0.77	0.144	0.014
	FW	63	1.57	NIL	1.57	0.172	0.021
8	UNMINERALIZED						
9	HW	28	0.85	0.70	0.15	0.04	0.01
	FW	16	0.05	NA	NA	NA	NA
10	HW	57	0.82	0.75	0.07	0.13	0.03
	FW	48	1.87	1.78	0.09	0.13	0.03
11	HW	71	0.60	NIL	0.60	0.02	0.01
	FW	25	1.35	NIL	1.35	0.04	0.01
12	HW	44	0.55	0.35	0.20	0.02	0.01
	FW	20	1.44	0.79	0.65	0.10	0.015
13	HW	89	0.274	NOT ASSAYED			
	FW	36	0.67				
14	HW	TRACES OF MINERALIZATION					
	FW	18	0.41	NOT ASSAYED			
15	HW	39	0.68	NIL	0.68	0.10	0.01
	FW	105	1.24	NIL	1.24	0.02	0.01
16	HW	44	0.30	0.21	0.09	NOT ASSAYED	
	FW	29	1.27	1.08	0.19	0.02	0.01
17	HW	32	0.18	NIL	0.18	NOT ASSAYED	
	FW	TRACES OF MINERALIZATION					

LEGEND

Assays from surface trenching

o Hole location on footwall of zone

NA - Not assayed

HW - Hanging wall

FW - Footwall

NOTE - Base elevation of 2800' obtained by aneroid barometer assuming Yukon River at Carmacks at 1550' above sea level

FIG. W 25

ARCHER, CATHRO & ASSOCIATES LTD.

FOOTWALL LONGITUDINAL SECTION
ORE RESERVES No. 1 ZONE

WILLIAMS CREEK PROPERTY

DAWSON RANGE JOINT VENTURE

SCALE IN FEET (VERTICAL AND HORIZONTAL)

200 0 200 400 800

To accompany report dated Jan. 1972