EVALUATION REPORT

UTICA MINES LTD.

HORN SILVER MINE 672517

OSOYOOS MINING DIVISION, B.C.

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EXHIBITS

Wright Engineers Limited

I

INTRODUCTION

The firm of Chapman, Wood & Griswold Ltd. was requested by Mr. Isaac Shulman of Utica Mines Ltd. to undertake an economic evaluation of the Horn Silver underground mine near Keremeos, B.C.

In order to assist the mine management in developing and assembling geologic and assay data, Mr. John F Fairley, geological engineer on the staff of Chapman, Wood & Griswold Ltd., spent approximately two months during March and April, 1966 mapping underground geology and otherwise collaborating in assembling ore reserve data with Mr. E. Livgard, geologist and S. Radvak, mine manager, for Utica.

J.A. Wood and J.F. Fairley examined the workings and held discussions with Messrs. Radvak and Livgard on May 10.

Mr. Livgard spent 3 days in the latter part of June reviewing ore reserve data with Mr. Fairley.

Mr. J.Black of Utica submitted financial and property data incorporated into this evaluation.

II

SUMMARY OF FINDINGS

A. At a milling rate of 7,200 tons per month (86,400 tons per year) there is an indicated net operating profit from the Horn Silver mine of $10.43 per ton equal to $75,000 per month or $900,000 per year.

B. Combined Measured and Indicated reserves are estimated at 183,500 tons of which 85% are extractable.

C. It is estimated that funds required subsequent to May 1, 1966 to establish production at the above rate would be $1,264,000. If this amount is borrowed, cash flow should permit repayment of the loan plus interest at 6 3/4% at the end of 22 months operation at which time Measured and Indicated reserves would be exhausted.

D. If Inferred reserves are accepted, the total projected reserves are 507,500 tons, 85% extractable.

E. Net cash flow from extraction of Measured plus Indicated plus Inferred reserves is estimated at $4,495,000 over a period of 5 years. This amount would permit repayment of the above loan plus interest, repayment of expenditures prior to May 1, 1966 in the amount of $1,492,000 and leave $618,000 as available for distribution.

F. Since the Inferred class of ore reserves is to be regarded with less confidence than Measured and Inferred ore, it is obvious that although an acceptable rate of profitability per ton is indicated, there is no assurance of continued operating life when the first two classes of ore are exhausted.

G. It is concluded that although there is justification for putting the properties into production at a nominal rate of 7,200 tons per month, a certain degree of risk is involved and mine development must be designed to augment Measured and Indicated classes of ore as rapidly as possible.

Respectfully submitted,

CHAPMAN, WOOD & GRISWOLD LTD.

E. P. Chapman Jr.

E. P. Chapman, Jr.

John A. Wood

John A. Wood

J. F. Fairley

July 11, 1966

III

TERMS OF REFERENCE & CONTROLS

A. Classification of ore reserves is based on definition of the terms Measured, Indicated and Inferred which represent probability of occurrence in descending order.

B. Metal prices are based on current quotations in effect at COMINCO smelter at Trail, B.C. and are expressed in Canadian dollars.

C. Net mine returns are based on shipment of jig and flotation concentrates to the Trail smelter.

IV

LOCATION AND ACCESS

The Horn Silver Mine is in the Osoyoos Mining Division 18 miles south of Keremeos, B. C., 1300 feet above and approximately one mile west of the Similkameen River. A 2.3 mile, 11% grade, tote-road connects the main highway to the mine buildings at 2622 elevation portal.

This road is well constructed and all-weather, but is single lane with sharp bends. It is proposed to construct another road segment of approximately 1500 feet from the 2422 portal to join the existing tote road, thus cutting off approximately one third of the present distance.

Latitude is approximately 49°03', longitude, 119°42'.

Terrain is a rugged version of the Interior Plateau bordering the Okanagan Mountain Range to the west. Rock outcrop prevails from the fluvioglacial terraces of the Similkameen Valley at 1300 elevation to the rolling Interior Plateau surface around 5000 elevation.

The Great Northern Railway serves Keremeos and also the village of Cawston, which is 13.6 miles from the mine, and 11.3 miles from a general millsite area.

Truck haulage distance from millsite to COMINCO smelter at Trail is 154 miles.

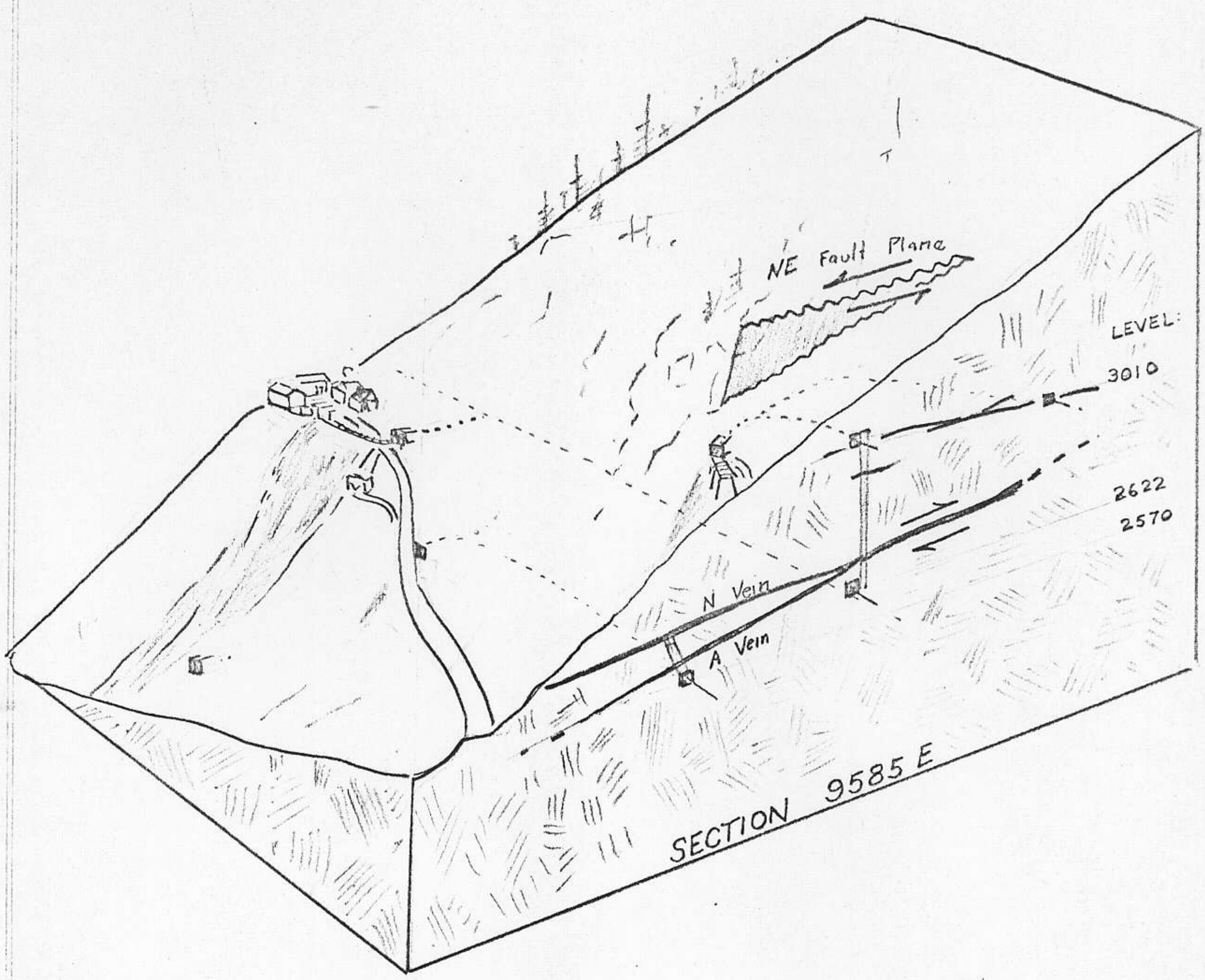

Fig. 1. Idealized Descriptive Block Sketch of Horn Silver Mine

V

PROPERTIES

Ninety-eight mineral claims including fractions are presently controlled by Utica Mines Ltd.* Six are held under option agreement, with 5000 dollars outstanding. Some of the claims listed here are not shown on the map, due to their recent staking date. No investigation of claim titles was made by Chapman, Wood & Griswold Ltd.

Claim No.		Record No. or Receipt No.	Expiry Date
Fully Owned:			
Silver Bell Nos. 1-8		6960N-6966N	Oct. 11, 1967
Crown grant Horn Silver		Lot 1928	
Crown grant Silver Bell		Lot 23935	
Ruby Silver	#1 Fr.	9158	Feb. 28, 1967
	#5 Fr.	9159	" " "
	#6, 7, 8	9160, 1, 2	" " "
	#9 Fr.	9163	" " "
	#10, 11	9164, 5	" " "
	#12 Fr.	9166	" " "
	#13, 14, 15	9167, 8, 9	" " "
	#16, 17, 18	9170, 1, 2	" " "
	#19 Fr.	9173	" " "
Sterling Silver Nos. 1-8		9123 - 9220	May 29, 1967
Sterling Silver Fr. #1		9221	" " "
Silver Plate		5516	Oct. 2, 1967
Silver Dart		5155	May 30, 1967
Golden Horn		5870	July 19, 1967
Kit Nos. 1-6		9972-9977	Apr. 22, 1967
Nick Nos. 1-6		9198-9203	" " "

* Information supplied by J. C. L. Black, July 5, 1966

Claim No.	Record No. or Receipt No.	Expiry Date
Fully Owned (cont'd)		
Key Nos. 1-8	12956-12963	Mar. 8, 1967
11-20	12964-12973	" " "
Ben Nos. 2-19	Recorded May 3/66 Receipt No. 28442 J	May 3, 1967
Frank 9, 10	Receipt No. 83502 D	Mar. 8, 1967
Frank 24, 25	Receipt No. 83502 D	" " "
British 23 Fr.	Receipt No. 83504	Mar. 28, 1967
Key Fr. 25, 26	Receipt No. 28441 J	Apr. 26, 1967
Under Option:		
Horn Silver Nos. 2-6	8125-8128	Apr. 24, 1967
Crown Grant British	Lot 3064 S	

THE HORN SILVER PROPERTY (outlined portion)
From Mineral Claim Map 82E/4 (E(M)) as with the Vancouver Mining Recorder, June 22, 1966

The forfeited group on the south has been restaked as the Ben Group.

VI

CLIMATE

The Interior Plateau of B. C. is typically dry, with a high temperature variance. Precipitation in the region of the mine would be expected to total approximately 8 inches annually. Light snow and below-freezing conditions prevail for January and February.

Vegetation is sparse up to about 4500 elevation, principally pine, grass, and sagebrush, above which open forests predominate.

VII

WATER AND POWER

Water

The proposed millsite is located on fluvio-glacial pediment about 1/4 mile east of the Similkameen River at elevation 2600' ASL and 1200' below the mine portals.

A well has been drilled near the millsite entrance and is considered to be a suitable source of water for milling purposes. Available data are:

depth	-	105 ft.
casing diameter	-	10 in.

one week test indicates no draw-down at
150 gpm, but will not hold at 185 gpm.

Water requirement for a 300 tons per day mill is estimated at not over 150,000 gallons per day or 104 gpm.

There is also a possibility that some water for milling purposes may be reclaimed from tailings, but this factor cannot be determined until it has been established that build-up of reagents would not adversely affect metal recoveries.

Water for mining purposes is provided by sump pumps collecting and distributing underground seepage.

Power

Presently the mine is generating 600 HP on site at a cost of approximately 1¢ /KWH.

Although a contract has not been negotiated, it has been reported that total requirements can be obtained from West Kootenay Power & Light transmission line, which is under construction and which will pass immediately adjacent to the millsite, at an approximate cost of 5-6 mills per KWH.

Assuming this source will be utilized, a line should be run to the mine for normal usage, and present generating equipment would be retained as stand-by mine power source.

Total power requirements are estimated as follows:

Use	Days/Wk.	HP	Hrs./Day	% Load
Mine	6	750	16	70
Primary crushing	6	60	16	80
Secondary crushing	6	75	16	95
Mill	6	325	24	100
Misc.	7	40	24	50

Power Consumption - Estimated

Bhp - Hrs./Wk.

Mine	6 days x 750 hp x 16 hrs./day x 70% load =	50,400
Pr. crushing	6 days x 60 hp x 16 hrs./day x 80% load =	4,600
Sec. crushing	6 days x 75 hp x 16 hrs./day x 95% load =	6,800
Mill	6 days x 325 hp x 24 hrs./day x 100% load =	46,800
Misc.	7 days x 40 hp x 24 hrs./day x 50% load =	3,400
		112,000 Bhp-Hrs./Wk.

$$\frac{112,000 \text{ Bhp - Hrs./Wk.}}{1.341 \text{ Bhp/KW} \times .95 \text{ motor eff.}} = 87,900 \text{ KWH/Wk.}$$

$$87,900 \text{ KWH/Wk.} \times 52 \text{ wks.} = 4,570,800 \text{ KWH/Yr.}$$

or

$$\frac{4,570,800 \text{ KWH/Yr.}}{12 \text{ mo./yr.}} = 380,900 \text{ KWH/Mo.}$$

At $0.006/KWH = $2,285/Mo.

Power Cost Distribution - Estimated

Mine	45%	=	$1,028/Mo.
Mill	52%	=	$1,188/Mo.
Other	3%	=	$ 69/Mo.

VIII

HISTORY

Chronological Events

1901	The Discovery claims were staked by J. Hunter
1909	These were crown granted
1914 to 1925	Small amounts of ore were shipped
1925 to 1926	The Horn Silver Corp. operated a concentrator at 22 t.p.d., which ceased due to high mining cost, little ore and milling difficulties
1927 to 1930	Horn Silver Corp. concentrated on development work
1958	Canada Radium Corp. optioned the property, did more development work, made some small shipments
1959	Santos Silver Mining Co. optioned the property and did some diamond drilling and sampling
1964	Utica Mines Ltd. optioned the property and staked some additional mineral claims

Utica Mines Ltd. has, since undertaking the development, spent about 1.2 million dollars, chiefly on development headings, sampling, surveying and diamond drilling.

IX

GEOLOGY AND MINERALOGY

The Horn Silver deposit consists of a series of narrow, gently-dipping, quartz-filled, shear zones which carry significant quantities of silver and gold, and lesser concentrations of lead, zinc and copper. They have been explored underground over an area of approximately 500 ft. by 2000 ft. Post mineral shearing and profuse faulting give poor local continuity to the veins, but apparently do not affect the general structure. The host rock is monzonite. See Fig. 1.

Regional Considerations

Host rock so the Horn Silver shear veins is a monzonite phase of the Kruger Syenite[1], a member of the broadly determined Nelson plutonic rocks of probable Jurassic age. Approximately one half mile north of the mine lies a steep contact with the Carboniferous Kobau Group quartzites, schists, and greenstones. Also to the north, one mile distant, lies the roughly vertical plug of Mesozoic Richter Mountain hornblendite (older than the Kruger Syenite). One half mile south there is a steep contact with younger Mesozoic granodiorites. This contact exhibits some half-mile order displacements on left-hand-separation faults. These N.E. trending regional vertical faults parallel a few seen in the underground workings.

Detailed examination of air photos BC 5 123, 023 and 024 indicates the presence of one well developed NNW linear dipping vertically or steeply west in the mine area. This feature is believed to correlate with a prominent fault mapped underground in 9305 Drift, and which does not

(1) GSC Map 341A

displace the vein. Various prominent regional faults designated in GSC mapping were also readily identified by photosurvey, but none are in close proximity to the mine. Hence it is concluded that extension of the quartz veins in the mine area will not be terminated by faulting, but simply by exhaustion of injected silica. This criterion is important when considering the probability of ore extensions in the Inferred category. Refer to Fig. 2.

Structure

Jointing and faulting are intense, but vein displacement is not great. The controlling structure to the presently important veins is a series of shear zones the strike of which may vary from N70W to East-West to N85E respectively for West, Central and East portions of the mine. The dip is variable from 30° to 10°, with steeper dips showing more intense parallel jointing. Many of the vein shear surfaces have bonded. Other vein shears of varying strikes and generally low dips occur, but extent is limited, and metal content is generally low. In the vicinity of 9700E, 26 level, a vein trending approximately N40-50E and with variable dip E is becoming apparent as development progresses. 5898 sub, 24 level, follows a relatively strong vein dipping 45°S which is also in evidence on 26 level, NW portion. 2675 level poses another exception, dipping around 30° north.

Numerous steeply dipping post mineral faults with random orientations from NW to NE, and random degree of movement, displace the veins up to two feet, but the average net effect is that of no displacement.

The monzonite mass is apparently rather competent in spite of the joints, slickensides, breccia and gouge. Some old stopes spanning over 50 feet have stood without caving for more than 20 years. Perhaps the effect of concentrated faulting is a controlled slow collapse.

A limited number of the steep NE faults (approx. 25°/60°W) within the mine show large lateral displacements of undetermined magnitude, but vertical movement only on the order of tens of feet. The larger regional faults roughly parallel the system. These faults are the only ones to pose a definite structural boundary in the mine area. All of the faulting, shearing and mineralization can be found to transgress supposedly later faults. In other words, the stages of tectonics and mineralization are largely all coincident. This also has the result that configurations and attitudes of the open cavities are rarely similar.

Mineralogy

The most abundant vein minerals are quartz and pyrite. The economic minerals in probable order of abundance are galena, sphalerite, tetrahedrite, chalcopyrite, argentite, and native silver. Cerargyrite has been reported. Variable quantities of calcite, K-feldspar, sericite, chlorite, and hematite also occur. Molybdenite has been found in one shear vein outcropping lower on the hillside.

Erratic and exceptional high silver values (e.g. above 150 oz./T Ag) usually occur in open structures probably caused by movement during and after mineralization. Thus, secondary native silver causes the upgrading. The silver/lead ratio is therefore not constant, as can be seen from the assays.

Silicification and chloritization may carry beyond the shear surface for some two or three feet into the wall rock, and higher silver grades than normal in the monzonite occur in these areas.

Gold values have been observed to vary according to pyrite content.

Mineralization is banded or in "patches" and is not always visibly apparent. Post-quartz and mineralization movement frequently leaves open cavities, ground "sugar" quartz, and monzonite fragments and sheets in the vein. Apparently economic mineral emplacement is both by fissure filling and replacement (of monzonite inclusions in the quartz matrix, and of silicified wall rock).

There are apparently no critical variations in mineral assemblages in the mine.

The monzonite host rock is medium grained, roughly 40 to 60 percent mafics which generally exhibit chloritic alteration; and contains prominent pink feldspar crystals. Frequent orange-pink coloured K-feldspar crowded veins cut the monzonite with random orientation and occasionally share with and bear some relation to the quartz filling of the vein shears. Dykes composed mostly of pyroxene and hornblende also cut the monzonite. Possibly the K-feldspar veins are related to the granodiorite intrusion, and the ultramafic veins to the Richter Mt. hornblendite. The latter does not apparently bear any relationship to the mineralization.

Recoverable metal values are principally silver and gold, while minor amounts of lead, zinc and copper are also generally present in the veins.

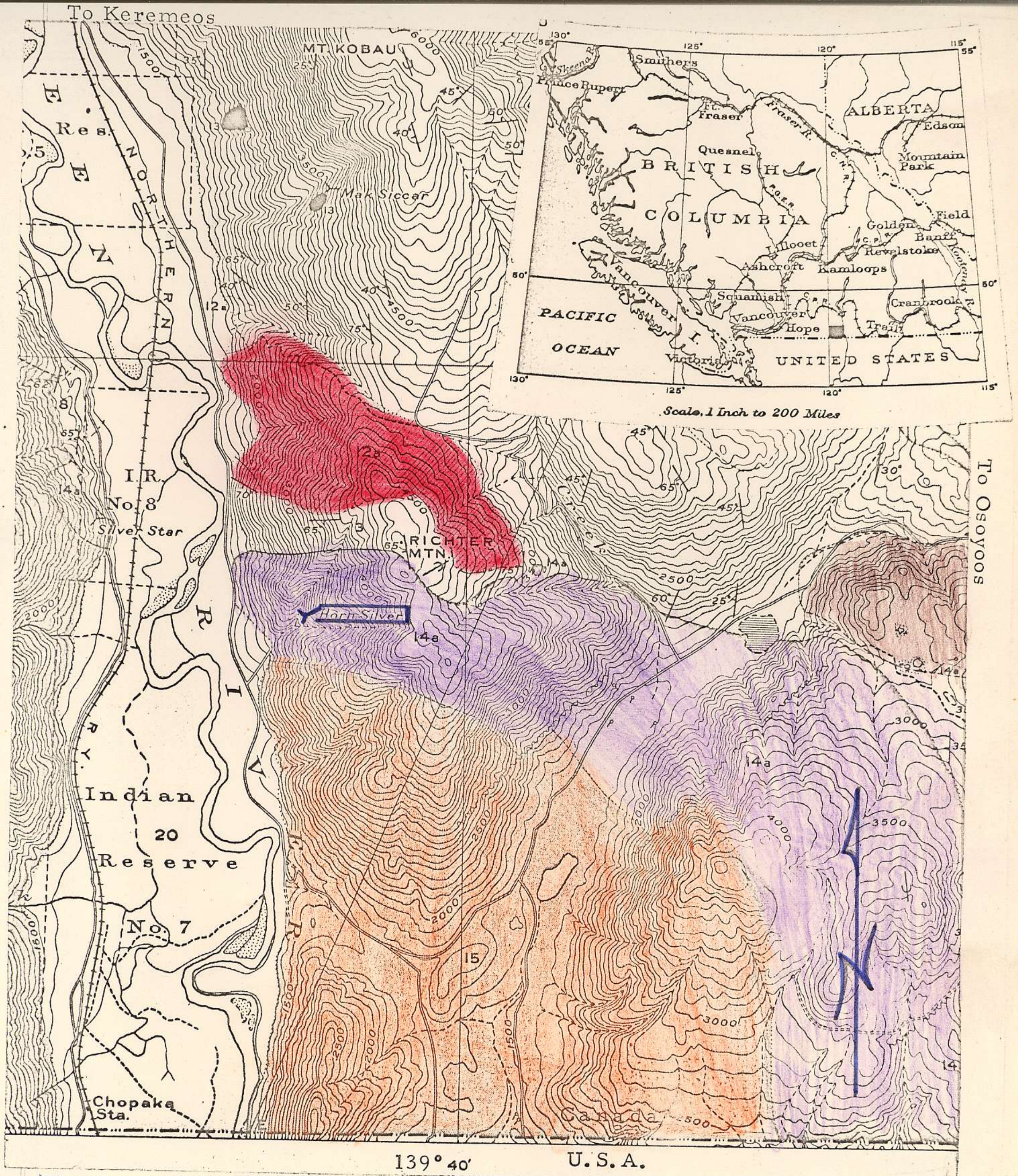

Fig. 2. Location and Surface Geology, HORN SILVER MINE
From G.S.C. Map 341A, Keremeos; geology: H. S. Bostock, 1929, 30

Scale: 1 in. = 1 mi.

Contour Interval: 100 ft.

15	Granodiorite	
14a	Kruger Syenite	
12a	Richter Mt. Hornblendite	Jurassic?
3	Kobau Gp., quartzite, schist, greenstone	Carboniferous?

CHAPMAN WOOD & GRISWOLD LTD.

X

SAMPLING & ASSAYING

The characteristics of the deposit with respect to metal distribution are: an extremely high variance (ie. spotty silver values and erratic vein widths.

Available and credible assay data from approximately 3,000 samples from four sources were made available: Santos Silver assay sheets (few statistics), Utica Mines Ltd. by Richards (majority of statistics) and Utica Mines Ltd. by present management, E. Livgard, (large number of statistics) and Canadian Exploration Ltd. check sampling (few statistics). Agreement between sources was good on an average basis. No check sampling was done or considered necessary by Chapman, Wood & Griswold Ltd.

Core recovery from diamond drill holes has been consistently very poor and only a very few assays have been considered accurate enough to include in the analysis.

The data consist principally of vein samples across a measured width, with assays for Au and Ag. Some wall rock samples were taken prior to present management. Several composite samples from the pulps have been assayed for Pb., Zn., and Cu. The sample lateral intervals were irregular, being variously 5,6,10 and 12 feet.

The majority of present samples are being assayed by J.O.Dolphin, Osoyoos, B.C

XI

ORE RESERVES

Estimation of ore reserves at the Horn Silver mine involves numerous variables and uncertain factors which ordinarily would not be encountered in evaluating either a bulk type deposit or one confined to a single well defined vein. The multiplicity of narrow veins, the complex structure, and irregularity of distribution of gold-silver values combine to make establishment of uniformity of sampling intervals difficult if not altogether impossible. While the present management has improved the sampling and mine development methods, the succession of prior operators bequeathed an assortment of assay data which, though useful, is not readily assimilated. For example, the several stages of development were variously sampled at 5, 6, 10 and 12 foot linear intervals; and same assays represented only actual vein widths, while others correlated with assumed mining widths.

The present management has been laboriously sorting and compiling all available mine sample and assay data in an effort to provide an acceptable basis for a proper evaluation of the mine. Mr. Fairley, during his assignment to Utica, assisted in this compilation. In addition, Mr. Livgaard, Utica Geologist, spent three days in Chapman, Wood & Griswold Ltd. office during the latter part of June assisting in determination of polygonal ore block dimensions and values. It must be stated that general, but not complete, agreement on block limits was reached; and that those limits as indicated on our Drwg. No. 514 appended are those which, in our opinion, conform to acceptable critical guidelines.

Two separate methods of ore reserve estimation were attempted:

A. Polygonal Blocks

Constructed in a horizontal plane, weighting assays individually against lateral interval of influence along mine openings within block limits. Sample thickness less than 5.0 feet was corrected to 5 feet by diluting with zero grade wall rock. No correction was made for the lateral extension which might occur by rotating the horizontal plane dimensions to a low angle inclination since it is estimated that such a correction would only approximate the tonnage represented by the volume of present openings. Hence the present opening tonnage is left in block reserves.

B. Computerized Statistical Analysis

A statistical ore reserve and trend surface analysis was undertaken, utilizing equipment and personnel at the University of California at Berkeley. Because of the complex nature of the Horn Silver deposit, the programming has also proved complicated and some further checking of data is indicated before the results can be accepted. Time will not permit full resolution of this evaluation prior to submission of this report, but it is hoped that the analysis will be available in the near future.

DESIGNATION OF ORE PROBABILITY

In estimating the value of the Horn Silver deposit we have chosen to apply the following definitions to ore classification in order of decreasing probability.

A. Measured Ore

Ore for which tonnage has been computed from dimensions revealed in workings and for which grade has been computed from the results of detailed sampling. The sites for inspection, sampling and measurement are so closely spaced and the geological character is so well defined that the size, shape and mineral content are well established. The computed tonnage and grade are judged to be accurate within observable limits on either 3 or 4 sides of a block. There are no terminal projections beyond the face of an opening.

B. Indicated Ore

Ore for which tonnage and grade are computed partly from specific measurements, samples, or production data; and partly from projection for a reasonable distance on geologic evidence. The sites available for inspection are too widely or otherwise inappropriately spaced to outline the ore completely or to establish its grade throughout. Terminal projections beyond the face of an opening and also lateral extensions beyond assay walls are not more than 50 feet.

C. Inferred Ore

Ore for which quantitative estimates are based largely on broad knowledge of the geologic character of the deposit and for which there are few, if any, samples or measurements. In essence the lateral limits form a halo around the Measured plus Indicated blocks, the periphery being a general extension corresponding to approximately one half the lateral dimension of the area controlled by Measured plus Indicated classes in the East and Central zones of the deposit. No Inferred reserves are estimated for the West zone since the veins are exceptionally narrow and grade is generally marginal.

Areas of constructed blocks were determined by planimeter and total block volume based on 5 foot thickness for each principal vein system. Tonnages of Measured plus Indicated blocks were then subtracted. Fifty percent of the remainder was then designated Inferred ore (67% of existing openings are in ore).

DENSITY FACTOR

Specific gravity of ore in Britton metallurgical test F2 was reported as 2.78.

$$(2.78)\ (62.4\ \text{lb.}/\text{ft}^3) = 173.47\ \text{pcf}$$

$$\frac{2000\ \text{lbs.}}{173.47} = 11.53\ \text{cubic feet per ton}$$

Add allowance for opening caused by fractures and replacement voids .47

= 12.00 cubic feet per ton

12.0 is the density factor used in all tonnage calculations in this report.

SUMMARY OF RESERVES - POLYGONAL METHOD

Class	Tons	Au oz./Ton	Ag oz./Ton
Measured	40,000	0.04	20.0
Indicated	143,500	0.04	17.9
	183,500	0.04	18.3
Inferred	324,000	assumed comparable grade	

DETAILED CALCULATIONS - TABLE I

TABLE I

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SUMMARY OF ORE RESERVES

CALCULATION BY POLYGONAL BLOCKS

Ore Class	Zone	Vein	Block	Tons	Au oz./T	Ag oz./T
Measured	East	(A) 1	1	1,100	0.06	21.6
			2	12,500	0.06	21.2
			3	630	0.02	29.7
			4	4,400	0.06	24.9
		Sub Total		18,630	0.06	22.4
		(N) 3	1	1,270	0.01	12.8
			2	1,360	0.01	15.1
			3	4,240	0.04	19.2
			4	9,220	0.04	20.9
			5	2,180	0.02	11.1
		Sub Total		18,270	0.03	18.3
	Central	(A) 4	1	3,100	0.04	15.6
TOTAL MEASURED				40,000	0.04	20.0

cont'd.

Table I cont'd

Ore Class	Zone	Vein	Block	Tons	Au oz./T	Ag oz./T
Indicated	East	(A) 1	5	6,080	0.06	17.8
			6	6,270	0.04	18.1
			7	5,010	0.09	29.7
			8	5,490	0.02	29.7
		Sub Total		22,850	0.05	23.3
		(A) 2	1	2,990	0.04	19.6
		(N) 3	6	18,320	0.01	11.3
			7	4,850	0.01	10.4
			8	20,140	0.06	33.0
			9	6,050	0.02	9.4
			10	4,070	0.02	22.4
		Sub Total		53,430	0.03	20.0
	Central	(A) 4	2	15,220	0.03	14.3
			3	6,570	0.12	12.5
		Sub Total		21,790	0.06	13.8
		(B) 5	1	3,850	0.06	14.9
			2	6,610	0.03	13.9
			3	180	0.02	11.8
			4	3,630	0.04	13.2
		Sub Total		14,270	0.04	14.0

cont'd.

Table I cont'd

Ore Class	Zone	Vein	Block	Tons	Au oz./T	Ag oz. T
	West	(A) 6	1	4,700	Nil	14.0
			2	3,230	0.03	10.0
			3	2,140	0.04	13.9
			4	3,510	0.02	10.9
			5	3,550	0.01	11.4
			6	6,600	0.07	17.2
			7	1,930	Nil	10.6
		Sub Total		25,660	0.03	13.3
	*Dump Stockpile			2,500	0.02	20.5
TOTAL INDICATED				143,490	0.039	17.79
TOTAL MEASURED PLUS INDICATED				183,490	0.04	18.28

* Skin sampled and tonnage estimated by CW&G Ltd.

cont'd

Table I cont'd

Ore Class	Zone	Vein	Block	Tons	Minus Measured Indicated	Waste Allowance 50%	Net Est. Reserve/T
Inferred	East	(A) 1	9	248,500	204,000	102,000	102,000
		(N) 3	11	345,300	273,600	136, 800	136,800
		(A) 7	1	37,500	37,500	18,750	18,750
				631,300	515,100	257,550	257,550
	Central	(A) 4	4	157,987	133,100	66,550	66,550
TOTAL INFERRED							324,100

XII

MINING

Design of a complete mining plan is not practicable at the present stage of mine development. A generalized system of room-and-pillar extraction with ultimate caving retreat has been proposed by the mine management, and the method appears to be suitable.

Development of ore blocks is proceeding according to a general pattern of advancing inclined drifts in panels on approximately 100' or 200' centers and at 90° to established track haulage drifts. Laterals at fairly wide spacing are being driven normal to the drifts, and eventually will conform to a fairly close order pattern, say 32' - 35' centers, in preparation for advancing short stubs to connect the laterals in such a manner as to leave adequate pillar support until pillar recovery commences.

Wire rope operated scraper haulers (slushers) are utilized to transfer broken rock to loading pockets serving battery powered locomotive trains. It is planned that trains will unload into an intermediate ore pass serving a main haulage level on which trains will transfer ore to a surface bin of 50 tons capacity. From the surface bin trucks will haul by contract 2.25 miles on downhill grade to coarse ore storage at the millsite.

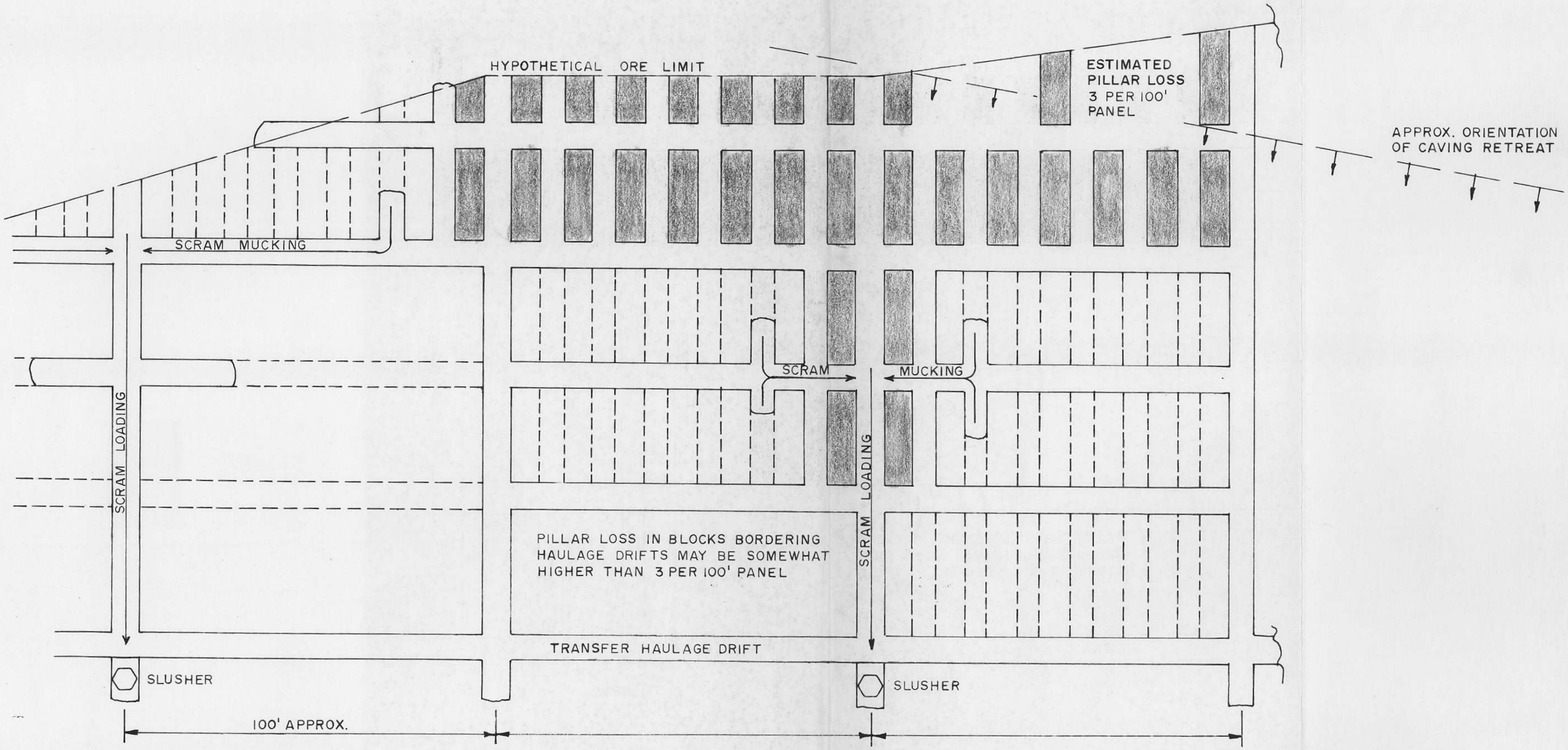
Normally, where actual vein widths are less than 5 feet, rock will be broken to a 5 foot minimum stoping width in the laterals, stubs, and pillars. Panel drifts and track haulageways, where driven in the vein (or veins), however, will have to carry a 6 or 6 1/2 foot back; and hence ore and waste may have to be broken separately in many instances..

Lateral ore limits have not as yet been determined, and will need to be established before a program of pillar salvage can be implemented. Hence it is evident that "primary extraction" (production from drifts, laterals and stubs) will be the principal source of ore supply for the early stages of operation.

Since pillar recovery from flat, narrow veins is almost never complete, it becomes necessary to estimate a percentage extraction factor that can be anticipated with a reasonable degree of confidence. For the purposes of the present evaluation an idealized mining plan has been devised, incorporating the basic ideas proposed by management (with which we concur). From analysis of this plan we have derived estimates of overall extraction factors which, in our opinion, should be applied to calculation of recoverable reserves. These factors are:

Volume of ore removed from primary openings - drifts, laterals, stubs		56%
Volume of pillars	44%	
Pillar loss	1/3	
Pillar recovery, total reserves		29%
Extraction factor, total reserves		85%

To estimate the optimum production potential of the mine a trial extraction analysis was made, using personnel requirement and classification proposed by mine management. This analysis indicates that in either the primary extraction stage or the pillar salvage phases it should be possible to maintain a normal daily production rate of 275-325 tons of ore from two shift operation. Hence a monthly rate of 7200 tons should be attainable with a 6 day work week (possibly ultimately 5 day week).



UTICA MINES LTD
HORNSILVER MINE

OSOYOOS MINING DIVISION B.C.

IDEALIZED PLAN
ROOM & PILLAR MINING SYSTEM WITH CAVING RETREAT

0 30 60 90 120 ft.

CW & G Ltd. Drwg. No.:- 515 DATE:- JULY 7/66

XIII

METALLURGY

A series of bench scale metallurgical tests was conducted by Britton Research Laboratories of Vancouver on various samples of ore submitted by Utica Mines Ltd. from the Horn Silver property.

These samples ranged in grade from 5.9 oz. Ag and 0.02 oz. Au to 137 oz. Ag and 0.14 oz. Au per ton.

Most of the testwork was confined to sample G which is believed to generally approximate probable high average mining grade. It assayed:

Au	-	0.045	oz./ton
Ag	-	27.1	oz./ton
Pb	-	0.38	%
Zn	-	0.62	%
Cu	-	0.07	%
Fe	-	6.36	%
S	-	3.15	%
SiO_2	-	65.02	%

Four different types of tests were conducted:

1. Jigging, bulk sulfide flotation and cleaning
2. Jigging, bulk sulfide flotation, cleaning and recleaning, followed by separation of galena and pyrite
3. Cyanidation of flotation tailing
4. Jigging and cyanidation.

The results indicate that recovery of gold and silver in excess of 90%, and of lead and zinc in the range of 90% may be achieved by methods 1 and 2; and that cyanidation of flotation tailings will yield additional recovery of gold and silver to the extent of approximately $1.50 per ton of tailing (or about $1.35 per ton of ore). However in method 4 the effect of build-up of copper and arsenic in a cyanidation circuit would probably lead to a loss of efficiency in the process. In our opinion, therefore, the suitability of method 4 has not been clearly demonstrated.

Since there is some arsenic in the ore, arsenic content of bulk sulfide flotation concentrate is sufficiently high to incur a smelter penalty. Attempts to lower the arsenic content by depressing pyrite from galena were partially successful, and possibly require further investigation.

Further improvement in metallurgical recoveries can probably be demonstrated by further testwork and actual operating practice. However, for the purpose of the present evaluation, we have chosen to utilize the conditions and results set forth in Britton test No. F 2. Mr. Britton indicates that results to be anticipated from full-scale milling of ore having approximately the composition of sample G would, by utilizing a combination of jigging, bulk sulfide flotation, and cleaning, be as follows: (see Appendix for details)

<u>Jig Concentrate</u>

Assay	-	1300 oz. silver per ton
Silver recovery	-	25%
Gold recovery	-	15%
Lead recovery	-	2%
*Zinc recovery	-	1%

* COMINCO smelter will not pay for this amount of zinc in low lead concentrate.

Flotation Concentrate

Assay	-	250 oz. silver per ton
Silver recovery	-	69%
Gold recovery	-	77%
Lead recovery	-	88%
**Zinc recovery	-	89%

Total Concentrate

Silver recovery	-	94%
Gold recovery	-	92%
Lead recovery	-	90%
**Zinc recovery	-	90%

Product Assays, Test F 2

Jig Concentrate

Ag	-	1321.9 oz./ton
Au	-	1.47 oz./ton
Pb	-	1.75 %
Zn	-	1.37 %

Flotation Recleaner Concentrate

Ag	-	253.2 oz./ton
Au	-	0.56 oz./ton
Pb	-	4.29 %
Zn	-	6.48 %
Cu	-	not assayed
As	-	2.24 %
Sb	-	0.09 %
Bi	-	tr.
Fe	-	32.62 %
S	-	38.46 %
SiO_2	-	6.66%
CaO	-	1.24 %

The bench scale tests indicate a work index of 14 KWH per ton and grinding energy requirement of approximately 13 KWH per ton.

Basic operating data for a milling plant of 250-300 tons per day (7,200 tons per month) capacity are estimated herewith:

Power required -

Item	Connected HP	Load Factor %
Crushing	135	80
Milling	325	100
Other	40	50
	500	91

** COMINCO will pay only for excess above 70 lbs. zinc per ton of concentrate. Hence this recovery figure is modified in Profit and Loss calculations.

CHAPMAN, WOOD & GRISWOLD LTD.

Water required -

Makeup 1.7 tons per ton of ore

= (1.7) (300)

= 510 tons/day

$$= \frac{1,020,000 \text{ lbs.}}{6.948}$$

= 146,800 Imp. gallons per day

= 102 gallons per minute.

The amount of water which might be relcaimed from tailings has not been determined.

Apparent optimum grind - 63% -200 mesh

Moisture content of concentrates - approx. 10%

Concentration ratio - approx. 15 to 1 at 18-20 oz./ton silver content in heads

Concentrate production, per month

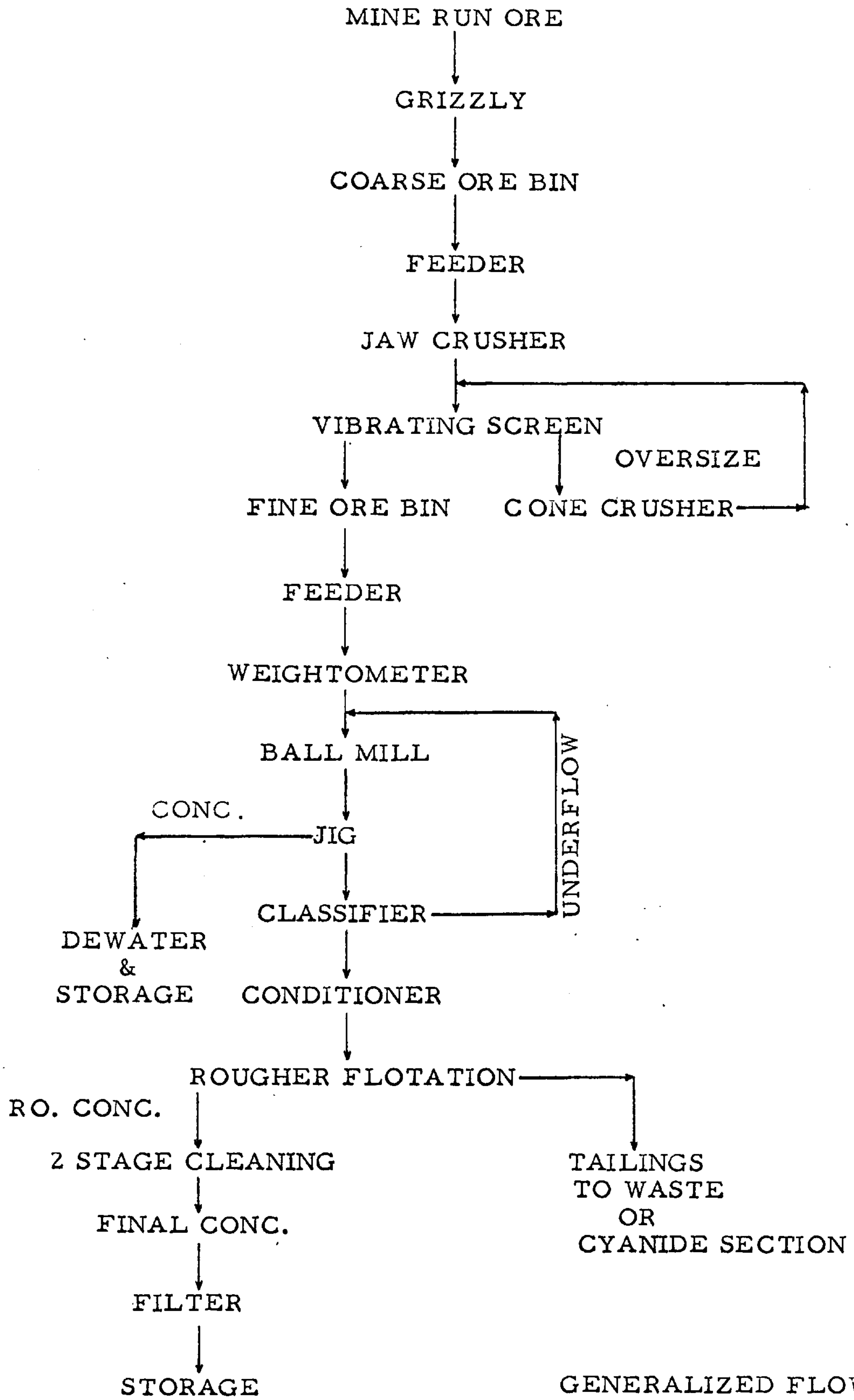
- Jig conc. - 24-30 tons
  Flot conc. - 358-370 tons

Reagent costs -

Copper sulfate	(0.7)	(.218)	=	\$0.153
Sod. sulfide	(0.2)	(.20)		0.040
xanthate	(0.38)	(.338)		0.128
aerofloat 25	(0.096)	(.33)		0.032
pine oil	(0.045)	(.15)		0.007
minerec B. C.	(0.015)	(.71)		0.011
other				0.050
				\$0.421

Grinding media -

est. consumption 2 lbs./ton @ \$0.11 = \$0.220.



GENERALIZED FLOW DIAGRAM
FOR
UTICA MINES LTD.
HORN SILVER MINE
300 TPD CONCENTRATOR

CW&G LTD. DRWG. NO. 411
JULY, 1966

XIV

OPERATING COSTS

In preparing estimates of operating costs, allowance has been made for approximately 28 percent escalation in labor costs, as well as 26 percent overhead on all wages and salaries to cover payroll taxes, insurance, and fringe items.

No provision has been made for escalation of materials costs, as it is assumed that increased operating efficiency over a period of time will compensate therefor.

A. SUMMARY

Basis - 7200 tons/mo.

86,400 tons/yr.	Cost per Mo.	Cost per Ton Milled
Mining	$53,136	$ 7.38
Milling	21,952	3.05
Administration	2,592	0.36
Haulage	9,413	1.31
Miscellaneous	669	0.09
Total	$87,762	$12.19

B. DETAIL

Operating Costs - 7200 Tons per Month

1. Mining

Category

Mining Labor		Cost Per Month	Cost Per Ton
Miner	31 @ 553.60	$17,162	
Timberman-trackman	2 @ 553.60	1,107	
Trammer	4 @ 488.00	1,952	
Mechanic 1st cl.	1 @ 604.80	605	
Mechanic 2nd cl.	1 @ 553.60	554	
Warehouseman	1 @ 535.00	535	
Overtime allowance		1,000	
	40	$22,915	$ 3.18

		Cost Per Month	Cost Per Ton
Mine Staff			
Geologist	1 @ 976.00	$ 976	
Foreman	1 @ 787.00	787	
Shift Boss	2 @ 630.00	1,260	
Accountant-Clerk	1 @ 598.00	598	
Sampler-Surveyor	1 @ 472.00	472	
	6	4,093	0.57
Explosives		3,600	0.50
Supplies			
Bits, steel, etc.		3,500	
Pipe and rail		1,500	
Rock bolts		1,000	
Timber		1,000	
Shop		1,000	
Misc. parts		2,000	
Office		200	
		10,200	1.42
Power			
Using 6 mill rate		1,028	0.14
Mine Plant Replacement		4,000	0.55
Assaying			
50 mine samples/day	@ 2.00	2,500	0.35
Contingencies	10%	4,800	0.67
TOTAL MINING		$53,136	$ 7.38

2. Milling

Category

Milling Labor		Cost Per Month	Cost Per Ton
Mechanic	1 @ 604.80	$ 605	
Operator	3 @ 604.80	1,814	
Swingman	1@ 584.00	584	
Crusher operator	2 @ 553.60	1,107	
Helper	3 @ 488.00	1,464	
Overtime allowance		500	
	10	6,074	$ 0.84
Mill Staff			
Supt.	1 @ 1,039	1,039	
Electrician	1 @ 787	787	
Assayer	1 @ 756	756	
	3	2,582	0.36
Supplies			
Reagents		3,024	
Grinding media		1,584	
Replacement parts		1,500	
Shop		750	
Laboratory		750	
		7,608	1.06
Power			
Using 6 mill rate		1,188	0.16
Plant replacement		2,500	0.35
Contingency - 10%		2,000	0.28
TOTAL MILLING		$21,952	$ 3.05

	Cost Per Month	Cost Per Ton
3. Administration		
Mine Manager	$ 1,292	
Vancouver office	1,000	
Travel	300	
	2,592	$0.36
4. Haulage		
Ore, mine to mill, contract @ 0.75/T	5,400	
Concentrates, Keremeos to Tadanac, B.C. 380 tons + 10% moisture = (418 T)(.06)(160 mi.)	4,013	
	9,413	1.31
5. Miscellaneous		
Power, Gen. Surface	69	
Warehouse control	300	
Audits	100	
Metal sales expense	200	
	669	0.09
TOTAL estimated operating costs	$87,762	$12.19

XV

CAPITAL COSTS

An estimate is presented herewith of the total capital requirement for bringing the Horn Silver Mine into production at a nominal milling rate of 250 to 300 tons per day, equivalent to 7200 tons per month or 86,400 tons per year.

Included in the estimate are data furnished by Utica on expenditures recorded as at May 1, 1966.

It is assumed that during the period required to design and erect a milling plant, mine development would continue at a rate somewhat less than half that projected after start of production.

Chapman, Wood & Griswold Ltd. has not made a detailed estimate of the cost of a 300 ton mill, but has used basic guideline data derived from established installations.

<u>Total Investment to Start of Production</u>

Pre-production Expense:	
Mine development to May 1, 1966	$1,211,349
Est. development May-Dec., 1966	160,000
Property acquisition to May 1, 1966	138,000
Property acquisition, contingent	16,000
	$1,525,349
Plant:	
Mine to May 1, 1966	$ 191,000
Mine, further requirement	136,000
	$ 327,000
Mill - 300 Tons x $2,000	600,000
Total plant	$ 927,000

Operating Reserve:

4 months x $88,000	$ 352,000
TOTAL CAPITAL REQUIREMENT	$ 2,804,349
Monies required from May 1, 1966	$ 1,264,000

XVI

PROFIT AND LOSS PROJECTIONS

The following projections are based on two separate objectives. The first is to determine estimated rate of return on capital required from May 1, 1966 to establishment of production, namely $1,264,000. In this case the ore reserve base is confined to the total of Measured plus Indicated tonnage.

The second projection analyzes profit and loss for total capital expenditure of $2,804,000. For this evaluation the ore reserve base includes tonnage of all three classes, Measured, Indicated and Inferred.

Metal sales estimates are based on shipping both jig and flotation concentrates to the COMINCO smelter at Trail, B.C.

COMINCO indicated verbally to CW&G Ltd. that it would accept these concentrates, but would penalize for high arsenic content.

Bunker Hill Company smelter at Kellogg, Idaho advised by letter it would not accept either the jig or flot concentrate because of the high arsenic-low lead ratio.

PROJECTION A:

<u>Basic Assumptions</u>

1. Capital to be recovered: $1,264,000

Pre-production	$160,000
Property acquisition	16,000
Mine plant	136,000
Milling plant	600,000
Operating reserve	352,000

2. Milling reserve:

183,500 T @ 85% extraction = 156,000 Tons

3. Ore grade:

Au	-	0.04 oz./T
Ag	-	18.3 oz./T
Pb	-	0.35 %
Zn	-	0.50 %

4. Production rate:

250-300 tons per day
7,200 tons per month
86,400 tons per year

5. Milling recoveries:

		Au	Ag	Pb	Zn
Jig conc.	%	15	25	2	--
Flot. conc.	%	77	69	88	89
Total conc.	%	92	94	90	89

Tons per month Jig 24-30 Flot 358-370

6. Composition of Concentrates:

Marketable Metals	Au oz.	Ag oz.	Pb %	Zn %
Jig	1.4	1300	1.7	1.3
Flot	0.6	250	4.2	6.4

Other Constituents %	Cu	As	Sb	Bi	Fe	S
Jig	1.5	na	na	na	na	na
Flot	0.7	2.2	.09	tr.	33.	38.

	CaO	SiO_2	H_2O
Flot	1.2	7.0	10.0

7. Calculated Tadanac (Trail) Realized Metal Prices:

Au/oz.	$37.60	-	1.25	=	$36.35
Ag/oz.	1.398	-	0.02	=	1.378
Pb/lb.	0.134	-	0.006	=	0.128
Zn/lb.	0.137	-	0.055	=	0.082

8. Calculated Returns per Ton of Concentrate at Tadanac:

Jig Conc.

Au	-	(1.4 oz.) (.95 content) ($36.35)	=	$ 48.34
Ag	-	(1300 oz.)(.95 cont.) (1.378)	=	1,701.83
Pb	-	(34 lb.) (.925 cont.) (0.128)	=	4.02
Zn	-	26 lb. no payment		--
				$1,754.19
deduct - treatment charge, incl. As penalty and Pb deficiency				-20.00
moisture penalty				- 1.00
Net return per ton of conc.				$1,733.19
Net return per month @ 24 tons				$41,596.00
Net return per ton of ore milled $\frac{41,596}{7200}$				$ 5.78

Flot Conc.

Au	-	(0.6 oz.) (.95 cont.) ($36.35)	=	$ 20.72
Ag	-	(250 oz.) (.95 cont.) (1.378)	=	327.27
Pb.	-	(84 lb.) (.925 cont.) (0.128)	=	9.94
Zn	-	(128 lb.-70) (0.082)	=	4.76
				$ 362.69
deduct - treatment charge, incl. As penalty and Pb deficiency				-20.00
moisture penalty				- 1.00
Net return per ton of conc.				$ 341.69
Net return per month @ 356 tons				$121,642.00
Net return per ton of ore milled $\frac{121,642}{7200}$				$ 16.89

9. Calculated Net Mine Return Per Ton of Ore Milled:

Metal	Content	Mill Recovery	Smelter Content	Realized Met. Price	Net Return
Au	0.04 oz.	.92	.95	36.35/oz.	$ 1.27
Ag	18.3 oz.	.94	.95	1.309/oz.	21.39
Pb	7.0 lb.	.90	.925	0.128/lb.	0.74
Zn	10.0 lb.	.89	.45	0.082	0.33

Total return before smelter treatment $23.73

Est. deduction for smelter treatment, arsenic and moisture penalties

$$\frac{(24 + 356 \text{ tons})(\$21.00)}{7200 \text{ tons/mo.}} = -1.11$$

Net mine return per ton of ore milled $22.62

(Compared with calculation of net returns from concentrates shipped) $22.67

10. Net Operating Profit:

Net mine return per ton milled	$22.62
Less operating costs	-12.19
Net operating profit per ton	$10.43
Net operating profit per month	$75,096.00
Net operating profit per year	$901,152.00

11. Cash Flow:

Expenditure from May 1, 1966 to establish production is $1,264,000.00

If this amount is borrowed at 6 3/4%, it is returned with interest at end of 22 months operation, leaving $278,000 to add to operating reserve.

The operation will continue if sufficient additional reserves of similar grade have been developed.

PROJECTION B:

Basic Assumptions

1. Total capital expenditure $2,804,000

Pre-production	$1,371,000
Property acquisition	154,000
Mine plant	327,000
Milling plant	600,000
Operating reserve	352,000

2. Milling reserve:

507,500 T @ 85% extraction = 431,000 tons

Items 3 through 10 are the same as in Projection A.

11. Cash Flow

Total capital expenditure	$2,804,000
Assumed amount of borrowing	1,351,000
Interest on borrowing @ 6.75 percent	

Borrowed capital is returned in 2 years of operation from 80% of net cash flow.

The remainder of total expenditure ($1,492,000) is returned in 5 years operation.

$618,000 is available for distribution.

UTICA MINES LTD.
HORN SILVER MINE
PROFIT AND CASH FLOW PROJECTION A
MILLING RATE 7,200 TONS/MO.
86,400 TONS/YR.

Assumptions

Capital to be recovered	$1,264,000	(Expenditure from May 1, 1966)
Pre-production	160,000	
Properties	16,000	
Mine Plant	136,000	
Milling Plant	600,000	
Operating Reserve	352,000	

Milling Reserves	156,000 Tons
Net mine return/ton milled	$22.62
Operating cost	12.19
Net operating profit/ton	10.43

	YEARS 1 (8 mos.)	2 (12 mos.)	3 (10 mos.)	Total
Direct Operating Profit	Pre-Prod.	901	726	1,627
Taxable Profit		nil	nil	nil
Add: allowances claimed		901	726	1,627
Less: debt retirement @ 80% of gross cash flow		721	543	1,264
interest on borrowing		48	37	85
Net cash to operating reserve		132	146	278

Capital of $1,264,000 recovered in 1 yr. and 10 mos. of operation.
Continuance of operation contingent on development of additional reserves.

Tabulated figures are thousands.

UTICA MINES LTD.
HORN SILVER MINE
PROFIT AND CASH FLOW PROJECTION B
MILLING RATE 7,200 TONS/MO.
86,400 TONS/YR.

Assumptions

Tot. Capital Expenditure	$2,804,000
Pre-production	1,371,000
Properties	154,000
Mine Plant	327,000
Milling Plant	600,000
Operating Reserve	352,000

Milling Reserves	431,000 Tons
Net mine return/ton milled	$22.62
Operating cost	12.19
Net operating profit/ton	10.43

	1	2	3	4	5	Total
Direct Operating Profit	Pre-prod.	901	901	901	891	4,495
Less: deprec. (30% declining)		t a x	f r e e	p e r i o d	689	689
pre-prod. exp.					202	202
Taxable profit		nil	nil	nil	nil	
Net profit		901	901	901	-	
Plus: deprec.					689	
pre-prod. exp.					202	
Net cash flow		901	901	901	891	4,495
Less: loan retirement (1,351)		721	630			1,351
interest on borrowing		91	42			133
Net cash available		89	229	901	891	2,110
*P.V. factor @ 12% compound interest		.79719	.71178	.63552	.56743	
P.V. of net cash available		71	163	573	505	1,312

Borrowed capital of $1,351,000 returned in 2 years of operation.
Remaining amount of capital expenditure ($1,492,000) can be returned in 5 years operation and leave $618,000 for distribution.

Tabulated figures are in thousands

* Parks, Table 2

CERTIFICATION

I, E.P.Chapman Jr., of Vancouver, B.C. do hereby certify:

1. That I am a Mining and Geological Engineer, residing at 2135 Argyle Street, West Vancouver, B.C.

2. That I am President of Chapman, Wood & Griswold Ltd., Consulting Mining Engineers and Geologists, with offices at 133 East 14th Street, North Vancouver, B.C.

3. That I am a registered Professional Engineer in the Province of British Columbia and a member of the Consulting Engineers Division of the Association of Professional Engineers of British Columbia.

4. That I have practiced my profession for 31 years.

5. That neither I nor any employee of my company has any direct or indirect financial interest in Utica Mines Ltd. or in any property owned or controlled by it.

E.P. Chapman Jr.

E.P.Chapman, Jr., P. Eng.

July 11, 1966

CERTIFICATION

I, John A. Wood of West Vancouver, B C do hereby certify:

1 That I am a Geologist, residing at 5019 Howe Sound Lane, West Vancouver, B.C

2. That I am Vice-President of Chapman, Wood & Griswold Ltd., Consulting Mining Engineers and Geologists, 133 East 14th Street, North Vancouver, B.C.

3. That I have practiced my profession for 33 years.

4. That I have no direct or indirect financial interest in Utica Mines Ltd., or in any property owned or controlled by it.

John A. Wood

John A. Wood

July 11, 1966

CERTIFICATION

I, John F. Fairley, of Vancouver, B.C. do hereby certify:

1. That I am a Geologist, residing at 3704 McKechnie Avenue, Vancouver, B.C.

2. That I am employed by Chapman, Wood & Griswold Ltd., Consulting Mining Engineers and Geologists, 133 East 14th Street, North Vancouver, B.C.

3. That I am a registered Professional Engineer in the Province of British Columbia.

4. That I have practiced my profession for three years.

5. That I have no direct or indirect financial interest in Utica Mines Ltd., or in any property owned or controlled by it.

John F. Fairley

July 11, 1966

EXHIBITS

CONCENTRATION TESTS ON SAMPLES OF ORE

FROM THE HORNE SILVER PROPERTY

submitted by

UTICA MINES LIMITED

Progress Report No.1

(Informal)

Project No.: B102

Date: May 18, 1966

Investigation by: John W. Britton, P.Eng.,
Consulting Metallurgist.

BRITTON RESEARCH LABORATORIES
755 BEATTY STREET
VANCOUVER 3, B.C.

PHONE 681-6032

BRITTON RESEARCH LABORATORIES
755 BEATTY STREET
VANCOUVER 3, B.C.

JOHN W. BRITTON, A.R.S.M., B.Sc., P.Eng.

May 18, 1966

Mr J. Black,
Utica Mines Limited,
904 - 510 West Hastings Street,
Vancouver 2, B.C.

Dear Mr Black,

Re: Concentration tests on Horne Silver ore

We summarise below the results of the work completed to date on the various samples of ore from your Horne Silver property.

1. Assays of samples:

		Sample No.							
		A	B	C	D	E	F	G	H
Gold	Oz/ton	0.12	0.14	0.05	0.02	0.003	0.04	0.045	0.13
Silver	Oz/ton	78.1	137.0	26.1	5.9	0.28	28.1	27.1	107.6
Lead	%	0.67	1.09	0.49			0.26	0.38	0.88
Zinc	%	1.38	1.86	0.66			0.58	0.62	1.62
Copper	%	0.18	0.23	0.08			0.05	0.07	0.21
Iron	%	3.60	10.13	7.49			5.27	6.36	6.87
Sulphur	%	3.54	9.73	3.46			2.83	3.15	6.64
Silica	%	77.70	36.98	48.12			65.02	56.57	57.34
Specific gravity:		2.75	2.86					2.78	2.81
Flotation test #:								F2	F1

Sample identification:

Our No.	Description	Date received
A	9555E	March 10, 1966
B	9600W vein	" " "
C	#1 East Drift/and wall rock	April 7, 1966
D	#1 West Drift	" 7, "
E	Wall rock	" 7, "
F	1 part B + 5 parts D	
G	6 parts C + 1 part B + 5 parts D (= 1 part C + 1 part F)	
H	1 part A + 1 part B	

cont.

Mr J. Black (cont.)

2. Work Index of ore: 14 K.W.H. per ton (2000 pounds).
3. Grinding energy required (approx.): 13 K.W.H. per ton.
4. Anticipated results for full-scale milling (based on results of test F2):

Jig concentrate:	Assay: 1300 oz silver per ton	
	Gold recovery	15%
	Silver recovery	25%
	Lead recovery	2%
	Zinc recovery	1%
Flotation concentrate:	Assay: 250 oz silver per ton	
	Gold recovery	77%
	Silver recovery	69%
	Lead recovery	88%
	Zinc recovery	89%
Total concentrates:	Gold recovery	92%
	Silver recovery	94%
	Lead recovery	90%
	Zinc recovery	90%

Note: In addition to the assays shown in the table of results for test F2, the flotation concentrate is being assayed for As, Sb, Bi, Fe, S, SiO_2 and CaO.

Yours very truly,

BRITTON RESEARCH LABORATORIES

John W. Britton, P.Eng.

John W. Britton, P.Eng.
Consulting Metallurgist

p/copy Chapman, Wood and
Griswold ✓

JWB/t

BRITTON RESEARCH LABORATORIES
755 BEATTY STREET
VANCOUVER 3, B.C.

Utica "H" – Test F1 results

#	Product	Weight %	Assays Au oz/ton	Assays Ag oz/ton	Assays Pb %	Assays Zn %	Assays Cu %	Assays Fe %	Distribution % Au	Distribution % Ag	Distribution % Pb	Distribution % Zn	Distribution % Cu	Distribution % Fe
1	Jig concentrate	0.87	2.30	3095.6	3.83	2.99	0.38	24.92	15.84	25.69	3.6	1.6	1.6	3.2
2	Cleaned 1st flotation conc.	14.81	0.68	502.4	5.77	10.17	1.05	30.72	79.73	70.95	91.7	93.0	74.0	66.2
3	Cleaner tailing	3.41	0.06	26.2					1.62	0.86				
4	2nd concentrate	0.62	0.08	59.1					0.40	0.35				
5	3rd concentrate	0.84	0.08	27.0	0.32	0.31*			0.53	0.21	3.1	1.7*		
6	4th concentrate	0.42	0.04	17.4					0.13	0.07				
7	5th concentrate	1.16	0.04	30.2					0.37	0.34				
8	Slime†	2.38	0.01	10.6					0.19	0.24				
9	Rougher tailing	75.49	0.002	1.8	0.02	0.08			1.19	1.29	1.6	3.7		
10	Head (calculated)	100.00	0.13	104.9	0.93				100.00	100.00	100.0			
10	Head (direct assays)		0.13	107.6	0.88	1.62	0.21	6.87				100.0	100.0	100.0

† From thickening of jig tailing. * Calculated.

Cumulative results:

#	Product	Weight %	Au oz/ton	Ag oz/ton	Pb %	Zn %	Cu %	Fe %	Dist. Au	Dist. Ag	Dist. Pb	Dist. Zn	Dist. Cu	Dist. Fe
1+2	Jig + 1st flotation concs.	15.68	0.77	646.3	5.66	9.77	1.01	30.40	95.57	96.64	95.3	94.6	75.6	69.4
1 to 3	Jig + 1st rougher concs.	19.09	0.64	535.5					97.19	97.50				
1 to 4	Jig + 1st + 2nd concs.	19.71	0.63	520.5					97.59	97.85				
1 to 5	Jig + 1st + 2nd + 3rd concs.	20.55	0.60	500.4					98.12	98.06				
1 to 6	Jig + 1st + 2nd + 3rd + 4th concs.	20.97	0.59	490.7					98.25	98.13				
1 to 7	Jig + 1st + 2nd + 3rd + 4th + 5th concs	22.13	0.56	466.5					98.62	98.47				
1 to 8	Jig + 1st + 2nd + 3rd + 4th + 5th concs + Slime	24.51	0.51	422.3	3.44	6.36*			98.81	98.71	98.4	96.3		
4 to 7	2nd + 3rd + 4th + 5th concs.	3.04	0.06	33.4					1.43	0.97				

UTICA — Test F1 conditions.

Reagent	1	2	3	4	5	6	7	8	9	Total
$CuSO_4 \cdot 5H_2O$	—	—	0.5	—	—	0.2	—	—	—	0.7
Na_2S (60/62 %)	—	—	—	—	—	—	—	0.2	—	0.2
Aero 404	—	—	—	—	—	—	0.05	0.05	—	0.10
CX 51 (1)	—	—	—	0.20	0.05	0.05	—	—	0.05	0.35
Aerofloat 25	—	—	—	0.048	0.024	0.024	0.024	0.024	—	0.144
Pine oil	—	—	—	0.018	—	—	—	—	0.018	0.036
M.I.B.C.	—	—	—	0.015	—	—	—	—	—	0.015
Pulp volume - Ml (2)	—	—	4800	4800	4800	4800	4800	4800	2600	—
% solids.	65	—	33	33						—
Time - Minutes	30	—	5	12	4	4	4	4	6	—
pH	—	—	—	7.7	7.6	7.5	7.5	7.7	7.7	—
Temperature - °C	—	—	—	20	21	21	21	22	18	—

Notes: (1) Potassium amyl xanthate.
(2) Per 2000 grams of original ore.

Stages:
1. Grinding (63% - 200 mesh)
2. Jigging
3. Conditioning.
4. Rougher flotation - 1st concentrate
5. Rougher flotation - 2nd concentrate
6. Rougher flotation - 3rd concentrate
7. Rougher flotation - 4th concentrate
8. Rougher flotation - 5th concentrate.
9. Cleaning of 1st concentrate

Utica "G" — Test F2 results.
(Bulk sulphide flotation)

#	Product	Weight %	Assays Au oz/ton	Ag oz/ton	Pb %	Zn %	Units Au	Ag	Pb	Zn	Distribution % Au	Ag	Pb	Zn
1	Jig concentrate	0.50	1.47	1321.9	1.75	1.37	0.735	661.0	0.88	0.69	14.3	25.1	2.5	1.3
2	Recleaner concentrate	6.99	0.56	253.2	4.29	6.48	3.914	1769.9	29.99	45.30	76.0	67.2	85.6	87.2
3	Recleaner tailing	0.31	0.06	23.2	0.44	0.45	0.019	7.2	1.47	1.50	0.4	0.3	4.2	2.9
4	Cleaner tailing	3.02	0.04	23.9			0.121	72.2			2.4	2.7		
5	Rougher tailing	89.18	0.004	1.4	0.03	0.05	0.357	124.9	2.68	4.46	6.9	4.7	7.7	8.6
6	Head (calculated)	100.00	0.051	26.4	0.35	0.52	5.146	2635.2	35.02	51.95	100.0	100.0	100.0	100.0
6	Head (direct assays)		0.045	27.1	0.38	0.62								

Cumulative results:

1	Jig concentrate	0.50	1.47	1321.9	1.75	1.37	0.735	661.0	0.88	0.69	14.3	25.1	2.5	1.3
1+2	Jig + recleaner concs.	7.49	0.62	324.6	4.12	6.14	4.649	2430.9	30.87	45.99	90.3	92.3	88.1	88.5
1 to 3	Jig + cleaner concs.	7.80	0.60	312.6	—	—	4.668	2438.1	—	—	90.7	92.6	—	—
1 to 4	Jig + rougher concs.	10.82	0.44	226.5	2.99	4.39	4.789	2510.3	32.34	47.49	93.1	95.7	92.3	91.4

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UTICA – Test F2 conditions
(Bulk sulphide flotation)

	STAGE						
Reagent	1	2	3	4	5	6	Total
$CuSO_4 \cdot 5H_2O$	—	—	0.5	0.2	—	—	0.7
Na_2S (60/62%)	—	—	—	0.2	—	—	0.2
CX 51 (1)	—	—	—	0.30	0.05	0.03	0.38
Aerofloat 25	—	—	—	0.096	—	—	0.096
Pine oil	—	—	—	0.018	0.018	0.009	0.045
M.I.B.C	—	—	—	0.015	—	—	0.015
Pulp volume – Ml (2)	—	—	4800	4800	2600	1200	—
% solids	65	—	33	33	7	11	—
Time – Minutes	30	—	5	18	6	6	—
pH	—	—	—	7.5	7.7	7.5	—
Temperature – °C	—	—	—	21	20	20	—

Notes: (1) Potassium amyl xanthate
(2) Per 2000 grams of original ore.

Stages: 1. Grinding (63% -200 mesh)
2. Jigging
3. Conditioning
4. Rougher flotation
5. Cleaning
6. Recleaning

<u>CONCENTRATION TESTS ON SAMPLES OF ORE</u>

<u>FROM THE HORNE SILVER PROPERTY</u>

submitted by

<u>UTICA MINES LIMITED</u>

<u>Progress Report No.2</u>

(Informal)

<u>Project No.</u> B102

<u>Date:</u> June 27, 1966

<u>Investigation by:</u> John W. Britton, P. Eng.,
Consulting Metallurgist.

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JOHN W. BRITTON, A.R.S.M., B.Sc., P.Eng.

June 27, 1966

Mr J. Black,
Utica Mines Ltd.,
904 - 510 West Hastings Street,
Vancouver 2, B.C.

Dear Mr Black,

Re: Concentration tests on Horne Silver ore

Following are the results of our latest test-work on composite sample "G" (for head assays and results of previous tests see Progress Report No.1, dated May 18, 1966).

Test F3 Jigging, bulk sulphide flotation and cleaning

This test was carried out under conditions similar to those of test F2 but the grinding time was increased from 30 to 40 minutes. The ground ore contained 73% minus 200 mesh, as compared with 63% in test F2. Test conditions and results are shown in tables 2 and 3.

Comments: The gold recovered in jigging and rougher flotation increased from 93.1% in test F2 to 96.8% in test F3, but the silver recovery fell from 95.7% to 94.5%. After cleaning the flotation concentrate twice, the gold recovered by jigging and flotation was almost identical with that obtained in test F2 (90.3%) but the silver recovery (90.8%) was 1.5% lower than in the previous test. The higher loss of silver was probably due to increased sliming of the silver minerals. Finer grinding therefore does not appear to be advantageous.

Test F4 Jigging, bulk sulphide flotation, cleaning and recleaning, followed by separation of galena and pyrite.

In this test, grinding, jigging and flotation were carried out as in test F2 but the recleaned concentrate was treated in order to depress the pyrite, with a view to reducing the weight of concentrate to be shipped and smelted and also to reduce the arsenic content of the silver-lead concentrate.

cont.

Mr J. Black (cont.)

Test results and conditions are shown in tables 3 and 4.

Comments: Although three stages of separation were carried out, almost one-third of the pyrite still remained in the silver-lead concentrate. The final silver-lead concentrate assayed 1.24 oz/ton gold, 635.2 oz/ton silver. 12.38% lead, 12.59% zinc, 26.50% iron and 1.63% arsenic; the combined pyrite concentrates assayed 0.22 oz/ton gold, 62.5 oz/ton silver, 0.79% lead, 3.37% zinc, 33.74% iron aod 2.69% arsenic. 13.4% of the total gold and 12.5% of the total silver passed into the pyrite concentrate, together with 10.8% of the lead and 29.6% of the zinc. This concentrate obviously could not be discarded, but it should be possible to recover most of its gold and silver content by cyanidation.

Mainly due to a higher head assay caused by the erratic distribution of free gold, the gold recovery in the recleaned bulk concentrate and jig concentrate was higher than in the previous test (94.6% as compared with 90.3% in test F2), using the same conditions. The silver recovery in the recleaned bulk concentrate and jig concentrate, 91.8%, was similar to that obtained in test F2 (92.3%). The lead and zinc recoveries were fairly similar (5% higher and 5% lower respectively). The results therefore essentially confirmed those of the earlier test.

Test C1 Cyanidation of test F2 tailing

A sample of test F2 rougher tailing was cyanided for 24 hours at 40% solids with a 0.1% solution of sodium cyanide, using an open bottle on rollers. The pulp was filtered and the residue was retreated under the same conditions with a fresh cyanide solution. The filtrates were assayed for gold and silver. Results are shown in table 5.

cont.

Mr J. Black (cont.)

Cyanide and lime consumptions were as follows:

	NaCN consumed Lb/ton of tailing	CaO consumed Lb/ton of tailing
First cyanidation	0.41	2.38
Second cyanidation	0.03	1.44
Total	0.44	3.82

Comments: 40.9% of the gold present in the tailing was recovered in the first cyanidation and 9.1% in the second cyanidation, giving a final residue assaying 0.002 oz/ton. 73.2% of the silver was recovered in the first cyanidation and 5.3% in the second, giving a final residue assaying only 0.30 oz/ton. The overall gold and silver recoveries obtained by jigging, bulk rougher flotation and cyanidation of the tailing were 96.5% and 99.0% respectively. The value of the additional gold and silver recovered in the first 24 hours cyanidation was $1.50 (Can.) per ton of tailing.

Test C2 Jigging and cyanidation of ore

A 1000 gram sample of ore was ground to 63% minus 200 mesh and jigged. The jig tailing was cyanided for 72 hours at 40% solids with a 0.2% sodium cyanide solution, filtered, washed and re-cyanided for 18 hours with a 0.1% cyanide solution. The residue was dried and one-half was ground to 90% minus 200 mesh, 71% minus 325 mesh. This was cyanided for 18 hours at 40% solids with a 0.1% cyanide solution, filtered and washed. Metallurgical results are shown in table 6. Cyanide and lime consumptions were as follows:

	NaCN consumed Lb/ton of ore	CaO consumed Lb/ton of ore
First cyanidation	2.77	3.80
Second cyanidation	0.17	1.05
Third cyanidation	0.80	3.13
Total	3.74	7.98

cont.

Mr J. Black (cont.)

Comments: 38.3% of the gold was recovered in jigging and 50.5% in the first cyanidation, with only 0.6% in the second cyanidation. Regrinding released a further 3.2% of the gold, giving an overall recovery of 92.6%. 27.6% of the silver was recovered in the jigging operation, 64.9% in the first cyanidation, 1.1% in the second and 1.7% in the third cyanidation, giving an overall recovery of 95.3%. The overall gold and silver recoveries were similar to those obtained by jigging and rougher flotation in test F2; jigging and a single cyanidation stage gave results similar to those obtained in the jig concentrate and recleaned flotation concentrate of test F2. The possibility of treating the ore by cyanidation instead of flotation should therefore be considered. It should be pointed out that, if part of the silver is present in a mineral such as galena and is not recoverable by cyanidation, any increase in the lead content of the ore is likely to be accompanied by a reduction in the silver recovered by cyaniding. To overcome this, it would be necessary to install a flotation plant in order to remove the silver-bearing lead minerals before, or possibly after, cyanidation.

In addition to gold and silver, the cyanidation solution contained small amounts of copper and arsenic, both of which are undesirable.

Yours very truly,

BRITTON RESEARCH LABORATORIES

John W. Britton, P.Eng.

John W. Britton, P.Eng.

Consulting Metallurgist

Enclosures (6)

Copies to: Chapman, Wood and Griswold

Mr E. H. Bronson

JWB/t

Utica "G" – Test F3 results.

Table 1.

#	Product	Weight %	Assays Au oz/ton	Assays Ag oz/ton	Units Au	Units Ag	Distribution % Au	Distribution % Ag
1	Jig concentrate	0.20	7.53	2230.3	1.506	446.1	26.7	17.3
2	Recleaner concentrate	6.90	0.52	275.8	3.588	1903.0	63.7	73.5
3	Recleaner tailing	0.30	0.05	35.0	0.015	10.5	0.3	0.4
4	Cleaner tailing.	2.89	0.12	28.9	0.347	83.5	6.1	3.3
5	Slime*	3.05 } 89.71	0.002	1.6	0.179	143.5	3.2	5.5
6	Rougher tailing	86.66 }						
7	Head (calculated)	100.00	0.056	25.9	5.635	2586.6	100.0	100.0
7	Head (direct assays)		0.045	27.1				

* From thickening of jig tailing.

Cumulative results:

#	Product	Weight %	Au oz/ton	Ag oz/ton	Units Au	Units Ag	Distribution Au	Distribution Ag
1	Jig concentrate	0.20	7.53	2230.3	1.506	446.1	26.7	17.3
1+2	Jig + recleaner concs.	7.10	0.72	330.9	5.094	2349.1	90.4	90.8
1 to 3	Jig + cleaner concs.	7.40	0.69	318.9	5.109	2359.6	90.7	91.2
1 to 4	Jig + rougher concs.	10.29	0.53	237.4	5.456	2443.1	96.8	94.5

Utica "G" — Test F3 flotation conditions.

Table 2

Reagent	STAGE 1	2	3	4	5	6	Total
$CuSO_4 \cdot 5H_2O$	—	—	0.5	0.2	—	—	0.7
Na_2S (60/62 %)	—	—	—	0.2	—	—	0.2
Z X 51 (1)	—	—	—	0.30	0.05	0.03	0.38
Aerofloat 25	—	—	—	0.096	—	—	0.096
Pine oil	—	—	—	0.018	0.018	0.009	0.045
M.I.B.C.	—	—	—	0.015	—	—	0.015
Pulp volume – Ml (2)	—	—	4800	4800	2600	1200	—
% solids	65	—	33	33			—
Time – Minutes	40	—	5	18	6	6	—
pH	—	—	—	7.6	7.7	7.6	—
Temperature – °C.	—	—	—	22	20	19	—

Notes: (1) Potassium amyl xanthate
(2) Per 2000 grams of original ore.

Stages: 1. Grinding (73% -200 mesh)
2. Jigging *
3. Conditioning
4. Rougher flotation.
5. Cleaning.
6. Recleaning.

* Jig tailing thickened. Thickener overflow flocculated and filtered (slime). Underflow floated.

Utica "G" — Test F4 results.

Table 3

#	Product	Weight %	Assays oz/ton Au	Assays oz/ton Ag	Assays % Pb	Assays % Zn	Assays % Fe	Units Au	Units Ag	Units Pb	Units Zn	Units Fe	Distribution % Au	Distribution % Ag	Distribution % Pb	Distribution % Zn	Distribution % Fe
1	Jig concentrate	0.10	35.42	4190.8	2.00*	1.00*		3.54	419.8	0.20	0.10		44.7	17.5	0.6	0.2	
2	Pb concentrate	2.33	1.24	635.2	12.38	12.59	26.50	2.89	1480.0	28.85	29.33	61.75	36.5	61.8	82.0	53.7	9.7
3	3rd Pyrite concentrate	2.17															
4	2nd " "	2.24 } 4.80	0.22	62.5	0.79	3.37	33.74	1.06	300.0	3.79	16.18	161.95	13.4	12.5	10.8	29.6	25.5
5	1st " "	0.39															
6	Recleaner tailing	0.26 } 3.39	0.02	27.5	0.43	0.54		0.07	93.2	1.46	1.83		0.9	3.9	4.1	3.4	
7	Cleaner tailing.	3.13															
8	Rougher tailing.	89.38	0.004	1.16	0.01	0.08		0.36	103.7	0.89	7.15		4.5	4.3	2.5	13.1	
9	Head (calculated)	100.00	0.079	24.0	0.35	0.55		7.92	2396.7	35.19	54.59		100.0	100.0	100.0	100.0	
9	" (direct assays)		0.045	27.1	0.38	0.62	6.36					636.00					100.0

*Assumed — not enough sample for assay.

Additional assays: Pb concentrate 1.63% As.
Pyrite concentrate 2.69% As.

Cumulative results:

#	Product	Weight %	Assays oz/ton Au	Assays oz/ton Ag	Assays % Pb	Assays % Zn	Assays % Fe	Units Au	Units Ag	Units Pb	Units Zn	Units Fe	Distribution % Au	Distribution % Ag	Distribution % Pb	Distribution % Zn	Distribution % Fe
1	Jig concentrate	0.10	35.42	4190.8	2.00	1.00		3.54	419.8	0.20	0.10		44.7	17.5	0.6	0.2	
1+2	Jig + Pb recleaner concs.	2.43	2.65	781.8	11.95	12.11		6.43	1899.8	29.05	29.43		81.2	79.3	82.6	53.9	
1to5	Jig + Bulk recleaner concs.	7.23	1.04	304.3	4.54	6.31		7.49	2199.8	32.84	45.61		94.6	91.8	93.4	83.5	
1to7	Jig + Bulk rougher concs.	10.62	0.71	215.9	3.22	4.47		7.56	2293.0	34.20	47.44		95.5	95.7	97.5	86.9	
2to5	Concentrate before separation.	7.13	0.55	249.6	4.58	6.38	31.37	3.95	1780.0	32.64	45.51	223.70	49.9	74.3	92.8	83.3	35.2

Utica "G" — Test F4 conditions.

Table 4

	STAGE										
Reagent	1	2	3	4	5	6	7	8	9	10	Total
$Ca(OH)_2$	—	—	—	—	—	—	1.0	—	0.8	0.6	2.4
$CuSO_4 \cdot 5H_2O$	—	—	0.5	0.2	—	—	—	—	—	—	0.7
$NaCN$	—	—	—	—	—	—	—	—	0.01	0.02	0.03
Na_2S (60/62%)	—	—	—	0.2	—	—	—	—	—	—	0.2
CX 51 (1)	—	—	—	0.30	0.05	0.03	—	0.005	—	—	0.45
Aerofloat 25	—	—	—	0.096	—	—	—	—	—	—	0.096
Pine oil	—	—	—	0.018	0.018	0.009	—	—	—	—	0.045
M. I. B. C.	—	—	—	0.015	—	—	—	0.004	—	0.002	0.021
Pulp volume – Ml(2)	—	—	4800	4800	2600	1200	1200	1200	1200	1200	—
% solids	65	—	33	33							—
Time – Minutes	30	—	5	18	6	6	5	5	3	3	—
pH	—	—	—	7.7	7.9	7.9	10.8	10.9	11.1	11.3	—
Temperature – °C.	—	—	—	23	19	20	20	20	20	20	—

Notes: (1) Potassium amyl xanthate.
(2) Per 2000 grams of original ore.

Stages:
1. Grinding (63% -200 mesh)
2. Jigging
3. Bulk conditioning.
4. Bulk rougher flotation.
5. Bulk cleaning.
6. Bulk recleaning.
7. Pb conditioning.
8. Pb rougher flotation (Tailing = 1st Pyrite conc.)
9. Pb cleaning (Tailing = 2nd Pyrite conc.)
10. Pb recleaning. (Tailing = 3rd Pyrite conc.)

* Jig tailing thickened before flotation (only trace of slime in overflow)

Utica "G" — Test C1. Cyanidation of test F2 tailing.

Table 5

#	Product	Cyan. period. hours.	Assay		Total mg		Oz/ton tailing		Distribution %			
			Au oz/ton	Ag oz/ton	Au	Ag	Au	Ag	Based on tailing Au	Based on tailing Ag	Based on orig. ore Au	Based on orig. ore Ag
1	1st Cyan. filtrate	24	—	—	—	—	0.0016	1.03	40.9	73.2	2.8	3.4
2	2nd " "	24	—	—	—	—	0.0004	0.07	9.1	5.3	0.6	0.3
3	Residue (480.4g)	—	0.002	0.30	0.033	4.94	0.0020	0.30	50.0	21.5	3.5	1.0
4	Head (480.0g)	—	0.004	1.4	0.066	23.04	0.004	1.40	100.0	100.0	6.9	4.7

1+2	Combined filtrates	48	—	—	0.033	18.10	0.002	1.10	50.0	78.5	3.4	3.7

Utica "6" — Test C2 results
(Jigging and cyanidation)

Table 6

#	Product	Cyan. period. hours	Amount	Assays Au	Assays Ag	Total mg Au	Total mg Ag	Distribution % Au	Distribution % Ag
1	Jig concentrate	—	2.00 g	10.52 oz/ton	3670.2 oz/ton	0.72	251.67	38.3	27.6
2	1st Cyan. filtrate	72	—	—	—	0.95	581.75	50.5	64.9
3	2nd " "	18	—	—	—	0.01	9.90	0.6	1.1
4	3rd " "	18	—	—	—	0.06	15.39	3.2	1.7
5	Final residue	—	990.6 g	0.004 oz/ton	1.23 oz/ton	0.14	42.34	7.4	4.7
6	Head (calculated)	—	1000.0 g	0.055 oz/ton	26.3 oz/ton	1.88	901.05	100.0	100.0

Assays of first cyanidation filtrate (before dilution):

Au 0.63 mg/liter; Ag 387.8 mg/liter;
Cu 98 mg/liter; As 17 mg/liter.

Cumulative results:

1	Jig concentrate	—	2.00	10.52 oz/ton	3670.2 oz/ton	0.72	251.67	38.3	27.6
1+2	Jig conc. + 1st filtrate	72	—	—	—	1.67	833.42	88.8	92.5
1 to 3	Jig conc. + 1st + 2nd filtrates	90	—	—	—	1.68	843.32	89.4	93.6
1 to 4	Jig conc. + 1st + 2nd + 3rd filtrates	108	—	—	—	1.74	858.71	92.6	95.3

Residue from 2nd Cyanidation reground, before 3rd Cyanidation, 90% − 200 mesh, 71% − 325 mesh.


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7000N
6500N
6000N
8000E
8500E
9000E
9500E
10,000E
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2425 PORTAL
3010 PORTAL
2570 DRIFT EAST
No.1 EAST DRIFT
SHURSON ADIT
EL. 3933
4-4
1-9
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6-4
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inches
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