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THE RUFFNER MINE

ATLIN, BRITISH COLUMBIA

A

PRODUCTION FEASIBILITY STUDY

BY:

JAMES C. SNELL
MINING GEOLOGIST

FOR:

MR. C.W. DANSEY
DR. L. ROSS
MR. J. WHIST
ATLIN SILVER CORPORATION

APRIL 1975.

See 77

C. W. Dansey
Atlin Silver Corp.
Box # 10
Savona, B.C.
May 4th/75

Deputy Chief Gold Commissioner
Dept. of Mines - Victoria - B.C.

Dear Mr. Rutherford
Further to the application for a production permit submitted on Feb. 7th/75, enclosed please find a up-dated feasibility, maps & mill design etc. - As I find further financing & planning difficult until I have assurance from your Dept. that Atlin Silver

will receive a Production Permit your earliest attention would be appreciated.

During the early days of Highland Valley I worked for a mining engineer Rutherford. If you are the same man; greetings from Ransey & Burdett.

Yours very truly,

A. W. Ransey.

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DEPT. OF MINES
AND PETROLEUM RESOURCES

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ILLUSTRATIONS

April 4th, 1975.

Directors,
Atlin Silver Corporation,
Kamloops, B.C.

Gentlemen:

RE: THE RUFFNER MINE, ATLIN, B.C.

Please find enclosed as per your request a production feasibility study including concentrator design and plans pertaining to the Ruffner Mine, located near Atlin, B.C.

This report, termed, "The Ruffner Mine, Atlin, British Columbia - A Production Feasibility Study" is a compilation and evaluation of all available past information concerning the mineral property and its development and exploration.

The writer has conducted extensive personal investigation of the property and organized the direction of all recent development and exploration work on the property.

Included in this study are conclusions and recommendations pertaining to production on the property. The recommendations have been reached through careful independent consideration of all facts and data accumulated from past work and evaluated at present metal prices.

The study includes recommended exploration programming, preproduction and production schedules with budgets as well as a complete concentrator design with plans.

The study has been conducted and completed following a private request from the officers of Atlin Silver Corporation, because of the author's exposure to the property.

Sincerely,

James C. Snell

James C. Snell, B.Sc.
Mining Geologist.

DECLARATION OF AUTHORSHIP

I, James C. Snell with a business and residential address in British Columbia, do hereby certify that: -

1. I am a mining geologist.
2. I am a graduate of the University of Alaska, School of Earth Science and Mineral Industry.
3. I am a graduate of the Provincial Institute of Mining Haileybury, Ontario.
4. I have received a Bachelor of Science Degree in the Geological Sciences.
5. I have supervised and directed exploration and development programs and conducted field investigations between 1966 and 1974 on the mine property described in this study.
6. That I have compiled the information and developed the conclusions and recommendations reached in this report.
7. I am enrolled as an Engineer in Training with the Association of Professional Engineers of the Province of British Columbia since 1967.

Dated at Kamloops, British Columbia, this fourth day of April, 1975.

James Snell

3. SUMMARY

Atlin Silver Corporation holds by option 27 Crown Granted mineral claims and two mineral leases on Mount Vaughn, 12 miles northwest of Atlin, British Columbia, The claims have been well developed beginning in 1900 to 1969 by several operators. Considerable underground work has been completed on the property outlining available, proven, probable and possible ore.

At the Ruffner Mine two principal ore bearing dyke systems have been explored. The veins are located within the fault-dyke systems and mineralization has been exposed up to 20 feet in width consisting of quartz, calcite, galena, sphalerite, pyrrhotite, arsenopyrite, chalcopyrite, pyrite and various silver minerals including proustite and native silver. The fault systems strike northeast and dip to the northwest at 65° to 70°. The host rock is lamprophyre dyke in a country rock of Jurassic granite and granodiorite. The mineralized veins are confined to the main faults and to some extent cross faults and the mineralization is intimately associated with lamprophyre dykes.

Ore Reserves developed at this time on the No. 2 and No. 4 dyke systems are: -

Proven - 24,636 tons @ 32.7 oz. Ag. per ton
Probable - 26,500 tons @ 27.3 oz. Ag. per ton

Total Corrected Reserves are 63,920 tons grading 18.63 oz. Ag. per ton and 5% combined Pb-Zn.

Possible Reserves envelope each ore body and the many less well defined mineral showings on the property. Potential Reserves are estimated at 20% of the dyke system.

Metallurgical Tests indicate that total net smelter returns will be $78.13 per ton. Calculations presented in this study show a cost to the concentrate bin of $43.70 per ton providing a net cash flow of $34.43 per ton.

A 50 ton per day mill has been recommended for the property. On the basis of the total and potential ore reserves a larger installation is justified, however, the erratic and discontinuous nature of the ore bodies will hinder proper extraction of ore at a greater rate.

The production rate initially recommended is 25 tons per day and this can be expanded following successful start up and adequate delivery of ore from the mine.

The value of products at 25 tons per day will be about 400 oz. of silver per day at 85% silver recovery and 5% combined lead-zinc.

The capital cost of bringing the mine into production is calculated at $487,500.00.

The net cash flow of Atlin operations is calculated at $860.75 per day at $4.25 per oz. for silver and 25¢ for combined lead-zinc at a production rate of 25 tons per day.

From the results of the investigation summarized above the writer concludes that the Ruffner Mine of Atlin Silver Corporation can be a profitable small producer.

4. INTRODUCTION

In 1974 Mr. C.W. Dansey of Kamloops, B.C. acquired an option on a group of Crown Granted mineral claims on the west slope of Mount Vaughn near Atlin, B.C.; from a Vancouver Company, I.P. Metals Ltd. The option was subsequently transferred to a company incorporated by Mr. Dansey, Dr. Ross and Mr. Whist, called Atlin Silver Corporation. At the request of these three officers of the Company, the writer is to provide a feasibility study and a concentrator design, based upon accumulated information from the past history of exploration and development on the Property. The writer as President of Interprovincial Silver Mines Ltd., of which I.P. Metals Ltd. is a subsidiary company, supervised and programmed exploration and development of the property between 1966 and 1969, during which time most of the information incorporated into this feasibility study was accumulated.

Silver-lead-zinc mineralization was discovered in 1901 and considerable development was completed between 1921 and 1933. The property was again under development in 1951, 1952, 1965 and between 1966 and 1969, over one million dollars was spent investigating the various workings and developing reserves.

The two main ore bearing systems have been opened up by 8 adits, 2 shafts and 4 raises for a total length of underground openings in excess of 10,000 feet. All workings have encountered silver-lead-zinc mineralization to a greater or lesser degree. Several hundred tons of hand sorted ore have been shipped during exploration and development beginning in 1901.

The work completed by Interprovincial Silver Mines Ltd., included road access and construction, portal rehabilitation, underground rehabilitation, surface and underground diamond drilling, surface trenching, drifting, raising, mapping, sampling, metallurgical and ore reserve studies and production studies. This information and that gathered from older records has been made available to Atlin Silver Corporation and has been used as source material for this study.

5. GENERAL INFORMATION

5.1 The property is known historically as the Ruffner Mine, a name derived from the original developer. There does exist a number of related mineral occurences in addition to the mine property. The entire region has been termed by the writer as "The Atlin Silver District".

The Atlin Silver District includes the following known mineral occurances: -

1. The Ruffner Mine
2. The Big Canyon
3. The Vulcan
4. No. 5
5. Adanac

Some smaller mineral occurences are known but have not been named.

5.2 Claims and Ownership

The following mineral claims are held under option by Atlin Silver Corporation from I.P. Metals Ltd.

Crown Granted Mineral Claims: -

1.	L4636	-	Duck Pond
2.	L4643	-	Pine Fr.
3.	L4635	-	4th of July
4.	L4633	-	Blacksmith
5.	L1175	-	The Hurrah
6.	L4634	-	Cabin
7.	L4647	-	Rainbow Fr.
8.	L4642	-	Frontal Fr.

9.	L1172	-	Nellie
10.	L1173	-	Barber
11.	L4651	-	Apachee
12.	L6103	-	Twin Moose
13.	L6104	-	Blackie
14.	L6102	-	Grandview
15.	L6101	-	Jim
16.	L4650	-	Cherokee
17.	L4649	-	Ptarmigan
18.	L6100	-	Miriam Fr.
19.	L4646	-	Mountain Hobo
20.	L4645	-	Ted Fr.
21.	L4638	-	Portal
22.	L4637	-	Cranberry
23.	L4644	-	Willow Fr.
24.	L4646	-	Horseshoe
25.	L4641	-	Commanche
26.	L4648	-	Silver Wedge Fr.
27.	L1174	-	Tom

Mineral Leases:

28.	M20 -	L1170
29	M21 -	L1171

The above listed claims and leases are known collectively as the Ruffner Mine.

5.3 Location - (59^{o}43'N - 133^{o}30' W)

The Atlin Silver District is located in the extreme north central part of British Columbia, eight miles north-east of the town of Atlin and 25 miles south of the Yukon boundary.

The properties and mineral occurences are situated between 3900 and 7000 foot elevation on the west slope of Mount Vaughn in the valley of Vulcan and Crater Creeks and on the northwest slope of Mount Leonard.

Mineralization occurs within an area two miles by three miles from Fourth of July Creek on the north to the headwaters of Vulcan Creek on the south.

5.4 Access

The Atlin District of northern B.C. is served by well maintained all weather gravel roads which connect the town to Carcross, Yukon, 100 miles distant, Whitehorse, Yukon, 110 miles distant and Dawson Creek via the Alaska Highway, 900 miles south.

The Ruffner Mine is serviced by a fifteen mile all weather gravel road from the town of Atlin, B.C.

Production from the mine will be trucked 110 miles to Whitehorse and transported south via White Pass and Yukon Route by rail and ship or by alternate route via truck back haul on the Alaska Highway.

Transfer of materials to the mine will be the reverse.

5.5 Facilities

Facilities and services at the Atlin townsite are limited to:

1. Post Office
2. R.C.M.P.
3. Magistrate
4. Mining Recorder
5. 2 grocery stores
6. 1 general store
7. 2 churches
8. 2 service stations
9. 1 hotel with dining
10. Dept. of Highways
11. House keeping cabins
12. B.C. Hydro & Power.

There are no water or sewage facilities in the town. B.C. Hydro operates two diesel electric units for local home power supplies.

The D.O.T. maintains a 2500 ft. gravel flight strip for light aircraft and float equipped aircraft operate from Atlin Lake during the summer months.

5.6 Climate & Vegetation

The Atlin area lies within a sub-arctic continental northern coniferous environment. Local variations within this environment are a result of elevation differences from 2300 feet above sea level at Atlin townsite to 7000 feet at Mount Vaughn.

The summers are generally short and warm, with an average temperature in June, the warmest month, of $51^{O}F$.

The winters are long and cold. The average temperature in January, the coldest month, is $2^{O}F$.

The average annual precipitation is 11 inches.

Higher elevations receive greater precipitation, and are somewhat cooler in summer and warmer in winter.

The valleys are generally broad and terraced, and muskeg occurs in poorly drained basins.

The glacial terraces are covered by grass and jack-pine parkland. Sufficient timber exists locally for mine support purposes. Tree line lies at about 4000 feet and wide, gentle sloping open grassy alpine valleys and plateaus occur at upper elevations.

5.7 History

Fritz Miller and Ken McLaren reached Atlin and discovered gold in Pine Creek in the Summer of 1898. Three thousand persons were working and living in the district by the end of 1898. In 1899 the population of the immediate placer camp had reached 5000 persons. The Silver-lead deposits were discovered during the course of gold prospecting, and in 1918, Mr. M.J. Ruffner optioned the Big Canyon claims and staked the Atlin Ruffner property to the north and east.

Initial underground work was commenced by Mr. Ruffner on the extensive vein zones in 1921 at the 4975 foot level of the No. 4 Vein.

In 1923 Ruffner incorporated Atlin Silver Lead Mines Ltd., and financed modest underground development over the next few years.

In 1925 the 4300 tunnel was driven into the 43-2 ore body, exposing high grade silver ore, some of which was shipped and marketed. Approximately 200 tons was cobbed and shipped, and averaged about 200 oz. Ag. per ton.

M. J. Ruffner died in 1929, and the properties were optioned to C.V. Bob of Toronto, who continued development on the No. 2 and No. 4 systems. In 1934 Bobjo Mines Ltd., acquired control of the properties, and extended the 5150-4 and 4300-2 levels.

In 1951 Atlin Ruffner Mines B.C. Ltd., shipped 44 tons of ore, grossing 7 oz. Au, 5343 oz. Ag., 36,197 lbs. lead, and 5,824 lbs. zinc. In 1952 the same Company drilled 20 exploration drill holes for a total length of 4,000 feet on the Vulcan and Big Canyon properties.

Armore Mines Limited of Toronto acquired control of the properties in 1965, and conducted preliminary exploration programs until 1966.

Between 1921 and 1966 the No. 2 system has been explored by six adits, one shaft and numerous trenches for a total length of 5,500 feet over a vertical interval of 1980 feet from elevation 3900 feet to 5820 feet. Mineralization has been encountered throughout this dimension.

The No. 4 system has been explored by three adits and numerous open cuts for a horizontal length of 5,500 feet, and a vertical interval of 1,500 feet.

There has been no significant production from any of these workings. B.C. Minister of Mines Reports indicate that a total of 264 tons of cobbed high grade material has been shipped.

5.8 Exploration by Interprovincial Silver Mines Ltd.

Interprovincial Silver Mines Ltd. optioned the Big Canyon Property, purchased the Vulcan property, and staked all significant open ground in 1966. The Company completed 9,120 feet of AX drilling to delineate mineralized veins in the Big Canyon and on the Vulcan vein. The Company constructed a modern exploration camp to service the program, and ten miles of access roads to various old workings during the summer of 1967. The Ruffner Mine was optioned by the Company in 1967, and a program of underground rehabilitation was scheduled.

In 1968 the Company gained access to five adit levels at the Ruffner Mine. The program included slashing and re-timbering, sampling and mapping and installation of diamond drill cross-cuts. Each exposed mineral block was investigated above and below the adit level by means of diamond drilling.

Results of the preliminary exploration program completed in 1968 indicated several locations containing sufficient grade and quantities of ore to support a modest underground mining operation. A program of mine preparation was undertaken in 1969 on the 43-2 ore body.

6. GEOLOGY

6.1 Regional Geology

The Atlin District lies just to the east of the main mass of the Coast Range batholith and is centred in the Atlin Horst. The Silver District lies within a stock of zoned intrusives which lies east of the north end of Atlin Lake and which extends eastward for 50 miles. The eastern geological boundary is denoted by the Teslin Lake lineament. A significant fault complex penetrates the district from Atlin Lake south through O'Keefe Mountain and easterly into the Cassiar District. This extensive zone of weakness is identified by a belt of Permian ultra-basic intrusions termed the Atlin Intrusions.

The mineral bearing stock east of Atlin Lake has been mapped as Jurassic granite and granodiorite with a central core of Cretaceous alaskite and monzonite. The silver-lead-zinc deposits described in this report lie within the Jurassic granites, one to five miles west of the alaskite contact. In the vicinity of these deposits the granite is coarse grained to porphyritic. The contact between the acid core of the stock and the more basic periphery is denoted by an alteration halo and a remenant belt of Palaeozoic formations 2 - 5 miles wide trending northerly into which has been intruded ultra-basic rocks of the Atlin Intrusions in Permian time.

The silver-lead-zinc deposits are marked by wide spaced strong fault-shear zones which strike northeast and dip northwest at 65° to 70°. In the vicinity of Mount Vaughn the fault structures are parallel and are spaced several hundred feet apart. The systems can be traced individually several thousand feet, the No. 2 system can be traced over a mile by surface trenching and underground workings.

The faults have been intruded by lamprophyre dykes ranging in width from one to fifty feet. The dykes are discontinuous but recur for thousands of feet along strike.

Economic mineralization is always associated with the lamprophyre which is characteristically a dark gray-green, aphanitic to fine crystalline, biotite feldspar rock, which weathers to a black appearance and is oxidized and rusty red near mineralization.

6.2 Economic Geology

The silver-lead-zinc mineral deposits occur either as fissure filling in granite along branch faults, or more commonly as fissure filling through the lamprophyre dyke system or as replacement and breccia ore associated with the latter. The deposits are steeply dipping usually occurring along the major shear zones and generally localized at points of cross and branch fracturing. The deposits are erratic and discontinuous and may grade abruptly into fault gouge.

Several overlapping mineral zones may occur; generally, however, the zone is confined to a single vein or a double vein with a lower grade envelope and centre. Fractured wall rock is common along the hanging wall and footwall accompanied by minor chloritization of the dyke rock.

Both arsenopyrite and pyrrhotite are common generally in the fringe areas of an ore body and these ores are notably deficient in silver, but may carry some gold. Some very high silver values occur at various locations on the property and even in narrow widths this ore can be mined profitably. The high silver values are attributed to wire silver and proustite.

The principal mineral bearing structures in the district are the: -

No. 1	-	Silver Fox (Unexplored).
No. 2	-	Ruffner explored laterally for 5500 feet and vertically for 2000 feet by surface trenching and underground workings.
No. 3	-	Ruffner (Unexplored) Possibly en echelon sections of No. 4.
No. 4	-	Ruffner explored both on and off the Company property for 5,000 feet and vertically for 1500 feet by surface trenches and underground workings.
No. 6	-	Vulcan - Big Canyon explored laterally for 6,000 feet and 2,000 feet vertically. (The No. 6 system is not controlled by the Company.)

Several additional mineral occurences are known to exist in the vicinity of Mount Leonard and Mount Vaughn not the least of these being Ruby Creek Molybdenum (Adanac).

The No. 2 system has been explored by 6 adits and numerous trenches and is consistently mineralized. Where parallel ore bearing veins merge, usually associated with breccia replacement ore; lenses of good width occur and will accommodate mining widths of 4 to 8 feet.

No. 4 system is very similar to the No. 2 and has been explored on the Company's property, by 3 adits and numerous trenches. The ore in this system usually contains greater amounts of galena and less sphalerite than the No. 2, however, this may be a result of vertical zoning. The No. 4 fault does not appear to be quite as strong as the No. 2 fault where the actual zone of weakness can range up to 5 and 6 feet in width. The No. 2 system including the dyke ranges up to 50 feet in width. Observation of the No. 4 system indicates an aggregate width generally not exceeding 20 feet.

6.3 Mineralogy

The principal sulphide minerals present in the ore are as follows - in order of probable formation during a single stage of epithermal mineralization.

1. Galena - PbS

   Occurs massive in fissure filling to sparsely disseminated in a replacement gangue of lampropyre, calcite and quartz.

2. Sphalerite - ZnS

   Occurs similar to galena and according to (Chamberlain) sphalerite and galena show a mutual boundary relationship under microscopic examination.

3. Arsenopyrite - FeAsS

   Occurs usually finely disseminated in association with galena and sphalerite, is rarely massive and is not always present.

Microscopic examination details arsenopyrite in euhedral crystals rimming sphalerite and galena.

4. Pyrite - FeS_2

   Is present locally as coarse well crystallized aggregates, which may be replaced by marcasite.

5. Chalcopyrite - $CuFeS_2$

   Occurs locally and is usually associated with heavy to massive sphalerite mineralization. Microscopic examination reveals that chalcopyrite occurs along grain boundaries between arsenopyrite and galena and also as blebs in sphalerite.

6. Pyrrhotite - Fe_1xS

   Occurs quite liberally in association with other sulphide minerals and appears to become quite prevalent near the extremities of an ore zone.

7. Tetrahedrite - $(Cu,Fe,Zn,Ag)_{12}\ Sb_4S_{13}$

   has been observed microscopically in sphalerite.

8. Enargite - $Cu_3A\overset{s}{/}_5S_4$

   Has been observed microscopically.

9. Ruby Silver (Proustite) Ag_3SbS_3

   Occurs locally in considerable quantities. It is bright blood red in colour on freshly exposed surfaces, and is observed as blobs up to two inches in diameter within a sulphide mass. Veins and veinlets of massive proustite have also been observed up to one inch in thickness. This mineral is possibly of secondary origin, but substantially up-grades ore shoots wherever it occurs.

10. Native Silver

Occurs rarely but has been observed to occupy vugs in massive sulphide bodies, in the wire silver form. It is usually associated with Proustite and may be of secondary origin.

7. ORE RESERVES

7.1 Classifications and Definitions

Sampling practices used by Interprovincial Silver Mines Ltd., personnel during the 1967 - 1969 program, were designed to represent as accurately as possible the true value of the erratic ore bodies. All sampling was completed with care and a pneumatic chipper was used wherever possible. Prior to sampling in most of the old workings fresh surfaces were exposed by slashing of drift backs. The fresh exposures were mapped and channel sampled on measured intervals, usually 5 foot intervals. Sample widths across the vein were determined by vein character and degree of mineralization such as massive, sparse, etc.

Ore Reserves have been classified into two categories only defined as follows:

1. Proven Ore -
   Ore exposed in mine workings with a limit of projection to 10 feet per foot of average true vein width. Ore blocked out on 4 sides. Ore blocked on 2 and 3 sides, the block being intersected by diamond drill holes on the same structure. 5 foot projections per foot of vein width in all directions from a diamond drill hole intersection.

2. Probable Ore -
   Projections beyond proven ore with a limit of projection to 10 feet per foot of average true vein width. All projections are limited to defined geological structure.

Possible ore classifications have not been used in this study, due to the erratic nature of the ore bodies both laterally and vertically. Considerable possible ore envelopes each ore body and possible ore exists surrounding poorly defined mineral occurances on the property. Due to the type of deposit a possible ore classification would not only be ambiguous but would undoubtedly be misleading.

Considerable potential for ore occurs both vertically and laterally throughout the extensive dyke systems on the property. Systematic underground development of these systems will eventually prove up the merit of the property.

Most of the present ore reserves and areas of known mineralization can be explored, developed and made available for mining by short adits below existing workings. The exploration potential of the district is not limited to the No. 2 and No. 4 dyke systems presently controlled by the Company. An effort should be made to acquire other surrounding properties of merit following commencement of production. The distribution of ore shoots within the system is excellent and is estimated by the writer to include 15% to 25% of the structure.

It is evident from the ore reserves developed on the property in past years that a profitable mining operation is feasible at present high metal prices. It is felt that costs during the recent period of inflation have not kept pace with metal values.

7.2 TABLE 1

Summary of all Ore Reserves

Vein	Mine Level	Proven	Ag(oz/T)	Probable	Ag(oz/T)
No. 2	3900-2	possible reserves			
No. 2	4100-2	possible reserves			
No. 2	4300-2	21,936	31.7	16,180	24.6
No. 2	5000-2	possible reserves			
No. 2	5600-2			2,520	32.8
No. 4	5150-4	900	28.2	3,730	34.3
No. 4	5300-4			2,090	19.5
No. 4	5700-4	1,800	47.8	1,980	25.0
TOTAL		24,636	32.7	26,500	27.3

Total Proven and Probable	=	51,136 tons (not corrected)
Average Grade	=	29.1 oz. Ag. per ton
Sampling error correction factor	=	less 15% grade
Average Grade	=	24.83 oz. Ag. per ton
Dilution Factor	=	25%
TOTAL PROVEN AND PROBABLE RESERVES	=	63,920 tons (corrected)
Average Grade	=	18.63 oz. Ag. per ton

Assays for lead and zinc have not been included in the above calculations, however, combined lead and zinc will range from 5% to 10%. The head sample used for metallurgical studies is 7%. The corrected value for dilution and sampling error is expected to be 5%.

ORE VALUE

Silver Value per ton =	18.63 oz.	x $4.25	=	$79.18
Base Metal Value /ton =	100 lb.	x $ .25	=	25.00
Total Ore Value per Ton =				$ 104.18

7.3 Ore Reserve Analysis

The 5600-2 level near the summit of Mount Vaughn on the eastern end of the No. 2 system was investigated in March 1968. The adit was originally driven in the 1920's to 150 feet. The adit was extended 300 feet in 1968 at which point mineralization was encountered. Heavy ground conditions at this point necessitated driving parallel to the vein for approximately 100 feet before swing back to the structure. Drifting was continued for another 75 feet and minor mineralization was encountered. An inclined raise at 55^{o} east was driven on the vein at this point beneath the 2C (5750-2) shaft workings. The raise was continued for 125 feet.

Ore Reserves: are calculated at 2620 tons of probable ore grading 32.8 oz. Ag.per ton. Samples taken by J.M. Ruffner in the 2C shaft and workings have been used for calculations of reserves in the workings. No reserves have been calculated in the 5600-2 drift.

The 5000-2 level was not opened up nor investigated in 1968. Possible ore for this adit is based upon assays taken by J.M. Ruffner and upon information gathered by Dolmage.

The 4900 elevation trench below the 5000-2 level reveals an interesting mineral zone of about 8 feet in width. A grab sample of galena from the oxidized zone graded 28.74 oz. Ag. per ton. No ore reserves have been allocated to this zone, however, possible ore exists and should be investigated at a later date.

Rehabilitation of the 4300-2 level began in May, 1968. The workings were retimbered and slashed for 600 feet to permit geological mapping and sampling. The main ore zone is exposed for 100 feet east and west of the 43-2 RSE and winze and consists of a well defined vein with a hanging wall zone and a foot wall zone, with a lower grade centre. The ore zone was mapped and sampled in vertical section in the winze to 60 feet below the level and in the raise to 75 feet above the level. Mineralization consits of massive and disseminated galena in a breccia gangue with associated sphalerite, pyrrhotite, pyrite, chalcopyrite, arsenopyrite and some light ruby silver prousite. The ore body is faulted off at the end of the 43-2 raise.

A total of 20 B.Q.W. drill holes were completed above and below the level from two separate drill cross cuts.

The results of the drilling and sampling program are detailed on maps and sections illustrated in this study.

Ore Reserves: calculated ore reserves for the 43-2 orebody are proven 21,936 tons grading 31.7 oz. Ag. per ton, probable 16,180 tons grading 24.6 oz. Ag. per ton.

The 4100-2 level was completely rehabilitated to the bottom of the winze in the summer of 1969. The drift beyond the first cross cut is timbered and old workings outside the rehabilitated drift are not readily accessible. Heavy ground conditions in the vicinity of the winze necessitated by-passing the winze to break through into the old drift on the other side. One raise round has been driven from the by-pass drift with the intention of breaking into the winze about 24 feet above the level. Minor scattered mineralization was encountered on this level and may be classified as possible ore. This mineralization is probably the bottom of the 43-2 ore body.

The mining plan for the 43-2 orebody will require rehabilitation of the winze which is to be used as an ore pass and for stope access below the 4300 level. The winze is completely timbered between the two levels a distance of 240 feet. The winze is inclined at 70^{o} for 140 feet above the 4100 level and flattens to 55^{o} for the remaining 100 feet. The muck compartment is full for 180 feet above the 4100 level representing 600 tons of muck.

A sub-level from the winze has been driven on ore for a short distance 50 feet below the 4300-2 level and is accessible to the west only.

The 3900-2 level is the main low level adit into the mine. The cross-cut has been driven in granite and is in good condition for 2600 feet. A mineral zone was encountered at the intersection of the No. 2 system and has been drifted on in both directions. Heavy ground conditions have prevented proper evaluation of this zone. Possible ore occurs here and this will be an important exploration target.

The 3900-2 cross-cut did not reach the downdip extention of the No. 4 dyke system.

The 5150-4 level is the western most development on the No. 4 system on the property. Other workings exist several thousand feet to the west on the No. 4 system on the Big Canyon claim not owned by the Company. The adit was rehabilitated and slashed in 1968 to permit mapping and sampling. The vein on this level is quite erratic in nature and the only ore shoot on this level lies at the east end of the adit along a cross fracture in granite. The vein material is primarily massive galena and sphalerite with minor arsenopyrite. Silver values in short sections range to 200 oz. Ag. per ton.

Ore Reserves: calculated ore reserves for the 5150-4 level are proven ore 900 tons grading 28.2 oz. Ag. per ton and proabable ore 3730 tons grading 34.3 oz. Ag. per ton.

The 5300-4 level portal was cleared and access was gained for 75 feet at which point the adit was completely caved. Good sample and assay records were available from previous operators, Atlin Ruffner Mines Ltd., and ore reserve calculations are based on this information.

Ore Reserves: calculated ore reserves for the 5300-4 level are probable ore 2090 tons grading 19.5 oz. Ag. per ton.

The 5700-4 level was rehabilitated and extended 200 feet in 1968. The main ore zone occurs near the west end (front) of the adit and shows impressive widths and grade for 85 feet. Mineralization consits of massive galena with chalcopyrite and sphalerite over widths up to 3 feet. The silver grade ranges up to 80 oz. per ton. This level is on the edge of the permafrost zone and normal mining and drilling problems were encountered under these conditions.

Ore Reserves: calculated ore reserves for the 5700-4 level are proven ore 1800 tons grading 47.8 oz. Ag. per ton and probable ore 1,980 tons grading 25.0 oz. Ag. per ton.

8. EXPLORATION AND DEVELOPMENT

8.1 Introduction

The objective of any exploration program is the development of ore reserves at minimum cost in order to perpetuate profitable daily production and extend the mine life as long as practical. In this regard continuous exploration followed by development of newly exposed mineral occurances is mandatory.

A general long range exploration plan should be devised in order that systematic exploration and development can proceed according to a predetermined general mine plan. The degree of exploration and development should be directly proportionate to a percentage of the net cash flow. These funds should be expended on a continuing basis according to the general development plan which can be modified according to results. The original mine plan if proven satisfactory should not be altered.

The life expectancy of the Ruffner Mine will be directly proportionate to underground drift advance along the vein dyke systems at 100 foot level intervals. The locations containing possible ore such as the 3900-2 level should be investigated first. These possible ore targets will be given priority according to cost of access and development.

Climatic conditions at the mine site will effect exploration programming from season to season during the year, however, effects of climate should not effect long range objectives. These objectives must be tailored to climate. Summer work must be directed to surface ore bodies and mineral showings at higher elevations. Winter work programs should include investigation at lower elevations, of possible ore within the mine and systematic development of the dyke system.

The 3900 low level cross-cut should be a focus in the general mine plan. The cross-cut has been driven for 2600 feet in granite at right angles to strike and has intersected the approximate centre of the No. 2 dyke system. This significant intersection has provided close to 2000 feet of backs on the No. 2 vein system. The cross-cut should eventually be extended an additional 500 feet to intersect the down dip extension of the No. 4 system.

The location of the mill at the 3900 portal is essential for mill water supply. It is also essential that this level be used for low level main haulage from the mine. Ore can therefore be delivered to the coarse ore bin directly from mine cars.

It is logical therefore that exploration expand from this centre (3900-2 cross-cut) in both directions along the No. 2 system. Investigation of possible reserves on this level would be the initial winter work program. Preference would be given to extending the west drift to beneath the 4100-2 level a distance of 700 feet. A raise between 3900-2 and 4100-2 level would provide underground transfer of ore from the 43-2 ore body directly to the coarse ore bin.

Considerable underground ore storage would be provided in the stopes (shrinkage), in the 43-2 winze and in the 3900-4100 raise. This underground storage capacity would prevent the need for surface stock piles of mill feed which would be subject to freezing.

In summary long range development will centre on the 3900 cross-cut, all mill feed would be delivered through this level and systematic level development on both the No. 2 and No. 4 would commence on this elevation.

Known ore bodies at or near the surface would be fully developed and mined, the ore being delivered to the mill by dump truck. The near surface ore bodies would of course be expanded to the maximum and exploration at these locations would be given priority during production and according to the season.

The development method recommended, would be a line drive in the granite footwall of the dyke system with cross-cuts through the system at regular intervals of about 50 feet. The geology of the entire system would be projected between cross-cuts following mapping and sampling of the cross-cuts. Good grade mineralization over mining widths would be drifted on between cross-cuts parallel to the main haulage level. Stoped ore can be extracted through raises driven from the main haulage drift into the vein. Diamond drilling to block up dip ore can be accomplished from the main haulage drift.

The above described system of exploration and development will have several distinct advantages: -

1. A complete investigation of the vein-dyke fault system through its entire width, at regular 50 ft. intervals along strike. Accurate projection of geology, including structure and mineralization.

2. Elimination of expensive timbering in stope preparation. No timber and maintenance required along haulage levels below old stopes. Ore between the raise and the level can be extracted by open stopping methods and mucked off the tracks in the parallel sub drift.

3. Easily maintained and safe haulage levels which will become extensive as systematic exploration continues over the known strike lengths on 100 foot levels

4. Ore can be extracted as it is encountered in exploration without detriment to the extraction of ore developed at a distance further from the 3900 cross-cut.

5. The rate of drift advance will be much greater in the granite footwall than drifting within the faulted dyke system. Total mine access costs may be reduced by as much as 50%.

6. Further exploration and stoping in previously mined areas at a later date will be possible.

8.2 Exploration Methods

Air photos have been used successfully on the property by the writer to locate mineralized fault zone extensions. Longitudinal topographic depressions occur along the fault systems and are visible in air photos. The writer has used a helicopter to locate these depressions visible on the photos and fault locations are marked with flagging after landing.

The depressions identifying the fault zone become stream courses during spring thaw, resulting in visible patters of bare ground in a snow covered landscape.

Percolating ground water generally seeps from the faults during summer months resulting in a biological identification.

The most effective method of surface exploration following the location of the zones as previously described is of course by bulldozer trenching across the structures at intervals. A hydraulic ripper attachment is essential due to the existence of late spring icing conditions.

Surface diamond drilling has been used with limited success on the property in the past due to generally poor core recovery in galena and loss of all economic values in the faulted ground. Of course faulted ground is essential to the formation of ore bodies. If diamond drilling on the property is contemplated in the future it is suggested that short holes only beneath trenches be drilled using BQ or HQ equipment.

The development of a surface ore shoot determined by dozer trenching can best be accomplished by a short adit and raise. Provided that ore is encountered it can be shipped to the mill in order that part of the exploration costs can be recovered immediately. An exposed surface ore shoot that does not meet the standards of continuity and grade might best be tested by drilling.

Topographic conditions that exist on the property are excellent for adit development of the mine. The parallel veins strike from lower elevations through the top of Mount Vaughn providing access at numerous elevations along the entire length. Surface ore bodies can be developed and mined by short adits using mobile equipment during the summer months. Road access to all parts of the property is quite good. Minor repairs must be contemplated each spring.

The correlation of surface and underground surveys on detailed plans and sections, followed by systematic projection of the known to the unknown of all pertinent geological information will be essential to proper investigation on the property.

A minimum of diamond drilling underground should be contemplated as a prelude to a major raise program and to develop reserves below a level.

Where possible a mine preparation crew should follow the exploration crew to increase the percentage of available ore that can be blended and shipped for concentration.

8.3 Targets

Exploration targets in this feasibility will be kept to the minimum and scheduling will be for one year only. The Budget for exploration during construction and the first year of production will be kept to a minimum as several years mill feed is presently available.

During the extraction of known ore bodies it is quite probable that additional possible ore will be projected that must be explored immediately. For this reason any long range exploration scheduling at this time will be impractical.

The primary summer exploration target will be the 5700-4 level where several thousand tons of proven and probable ore of good grade are known to exist below the level. This block of ore can be reached by a short adit 100 feet below the 5700-4 level. It is proposed that the exploration crew using mobile equipment first extract the 900 tons of ore above the level and ship this as mill feed to cover exploration costs. During the time that this ore is being extracted, portal site preparation on the 5600-4 level can be completed in order that the adit can be collared and timbered immediately following mining on 5700-4 level.

Climate conditions limit economic operations at these higher elevations from May 15th to September 15th and time is the essence of this program. Ore encountered on 5600-4 level development should immediately be stock piled at the mill and extraction of the ore block should be completed by September 15th if possible.

Several good winter exploration programs are envisaged by the writer, however, the program recommended is one which will more than likely provide additional mill feed immediately at minimum cost.

The 3900-2 level is the lowest access to the mine and exploration and development on this level should be contemplated with a view to low level main haulage via tram and ore cars to the coarse ore bin at the mill.

The 3900-2 level has penetrated the mountain for about 2600 feet and fairly extensive mineralization has been encountered along the 39-2 drift both east and west of the cross-cut. Due to very bad ground conditions and decomposed timber this mineral zone has not been investigated to the point where it could be included in ore reserves. It is suggested that access be gained along the No. 2 zone on the 3900-2 level to determine the existence, extent and grade of this mineralization. Extraction of existing ore grade material can be immediate.

Preliminary to development of the 3900-2 level will be re-routing of mill water supplies contained in this level and development of new storage facilities for this water. A short adit and raise into the 3900-2 level from below or an adequate sump must be constructed prior to development on this level.

8.4 TABLE 2

Schedule Exploration and Development

Location	Program	Schedule	Personnel
1. 5700-4	Portal Rehabilitation	Apr.15-21/76	2 men
2. 5600-4	Portal Construction	Apr.21-May15/76	2 men
3. 5600-4	500 ft. drift	May15-July1/76	4 men
4. 5600-4	100 ft. raise	July 1-15/76	4 men
5. 4300-2	Sub-level development	July 15-Sept.15/76	2 men
6. 43-2 STP	Stope preparation	Sept.15-Oct.15/76	2 men
7. 3900-2	Water reservoir const.	Oct.15-Dec.1/76	2 men
8. 3900-2	Adit rehabilitation	Mar.1-May15/77	2 men

A management decision will be required by May 15th, 1977 as to exploration and development priorities in regards personnel allocation for the summer of 1977.

The 2 man exploration and development crew will be allocated to pre-production, ore pass preparation, stope preparation, and mining (stoping) beginning June 1, 1975 to April 15th, 1976 at which time the above schedule begins.

8.5 Personnel

Initially exploration and development personnel will be maintained at 1 - two man crew, at a payroll cost of about $150.00 per day. The crew will be directed to mine preparation and pre-production stope preparation and then to stoping in order to develop a mill feed stock pile during the first few months of production.

Should the milling rate be increased to 50 T.P.D. it will be advisable to increase the number of men by one additional 2 man crew in order to maintain the proper ore reserve ratio.

The priority of the exploration and development crew will be:

1. Stope preparation.
2. Development and preparation of possible and probable ore.
3. New exploration headings to locate potential ore.

8.6 Exploration and Development Costs

Payroll costs for a two man exploration and development crew will be about $150.00 per day for 20 days per month. Payroll costs per month will be $3,000.00. The crew will require support in the form of transport, air and maintenance. The degree of support will be dependent on the working location.

Exploration and Development Cost estimating will therefore be dealt with on an individual job basis incorporating payroll and support costs into the budget.

TABLE 3

Budget

Schedule	Requirements	Cost
April 15-21 1976	Portal rehabilitation	
	Payroll	$ 1000.00
	Vehicle - 6 days @ $10.	60.00
	Front End Loader - 3 days @ $125.	375.00
	Dozer - 10 hrs. @ $75.	750.00
	Total	$ 2185.00
April 21 - May 15/76	Portal Construction	
	Payroll	$ 1500.00
	Vehicle - 10 days @ $10	100.00
	Dozer - 20 hrs. @ $75.	1500.00
	Front End Loader 3 days @ $125.	375.00
	Timber	1500.00
	Total	$ 4975.00
May 15 - July 1/76	Drifting with timber 10 ft./day of 7x8 with 2 crews	
	Payroll - per day	$ 300.00
	Timber, air, misc. support	300.00
	Cost per 10 feet	600.00
	Cost per foot	60.00
	Cost per 500 feet - 500 x 60	$30000.00

Schedule	Requirements	Cost
July 1 - July 15/76	Raise with timber 8 ft/day of 3 x 4 with 2 crews.	
	Payroll- per day with bonus	$ 400.00
	Timber, air, misc. support powder	400.00
	Cost per 8 feet	800.00
	Cost per foot	100.00
	Cost per 100 feet	$10000.00
July 15 - Sept. 15/76	Sub-level 8 ft/day of 6' x 6' with 1 crew	
	Payroll per day	$ 150.00
	Support per day	150.00
	Cost per 8 ft.	300.00
	Cost per foot	37.50
	Cost per 150 feet	$ 5625.00
Sept. 15 - Oct. 15/76	Stope preparation includes raises 2 men per day.	
	2 men for 20 days 150x20	$ 3000.00
	Support - 150 x 20	3000.00
	Total	6000.00
Oct. 15 - Dec. 1/76	Water Reservoir	
	Portal	$ 3000.00
	Drifting in granite $40. per foot.	
	Drifting 50 feet 50 x 40	2000.00
	Raise in granite $75. per foot	
	Raise 20 feet - 75 x 20	1500.00
	Total	$ 6500.00

TOTAL ESTIMATE EXPLORATION & DEVELOPMENT

COST 1976 = $ 65,285.00

Exploration and Development Costs per Ton - 1976

Tons/day	Days	Tons/mo.	Months	Tons/year
25	25	625	11	6,875

1976 Budget = $ 65,285.00

Cost per Ton of Ore = $\frac{65285}{6875}$ = $ 9.50

9. PRODUCTION

9.1 Schedule Underground Pre-Production

TABLE 4

Location	Program	Schedule	Personnel
4300-2	Access road rehab.	June 1 - 2	
4300-2	Portal rehab. de-ice	June 2 - 7	2 men
4100-2	Portal rehab. de-ice	June 2 - 7	2 men
4100-4300	Restore Services	June 2 -15	4 men
43-2 Winze	By-pass raise to winze	June 15-21	4 men
43-2 Winze	Pull muck from winze 600 tons	June 21-30	4 men
43-2 Winze	Re-timber 240 feet	July 1-31	2 men
4100-2	Surface ore dump and chute	July 1-31	2 men
4300-2	43-2 Stope preparation and commence stoping, ore to ore pass stock pile for mill start up	Aug.1-31	4 men

The objectives of the underground pre-production program beginning June 1 and ending August 31 will be to prepare the mine for continuous production at a rate of 25 tons per day and to develop a small stock pile of development muck for concentrator run in.

Ore delivery to the mill will begin in the 43-2 stope through the ore pass to the 4100-2 level, from the ore pass to the surface dump by tram on the 4100-2 level and from the surface dump to the mill coarse ore bin by dump truck.

Delivery of ore to the mill via the 4300-2 level is prevented by surface topographic conditions which will prevent safe transfer of ore to the mill from the 4300-2 level by dump truck.

9.2 Surface and Concentrator Schedule

TABLE 5

Location	Program	Schedule	Personnel
Mine Mill	Access road construction and rehabilitation	May 1-7	Supervision
Camp	Office renovation	May 1 - 7	2 men
Mill Site	Excavation	May 7 -21	Supervision
Tailings	Excavation	May 21-31	Supervision
3900-2	Portal Rehabilitation water storage dam	May 21 - June 15	4 men
Mill Site	Retaining walls - concrete	May 15 - June 15	Supervision
Mill Site	Generator & compressor pad, generator and compressor installation	May 21 - June 1 - June 15	Contractor
Mill Site	Crushing and concentrator equipment installation	June 15 - June 30	
	Ore bins	June 15-30	4 men
Mill Site	Building Construction	June 21- July 15	4 men
Mill Site	Plumbing-Electrics Conveyors	July 1 - July 31	4 men
Mill Site	Misc.	Aug. 1-15	4 men
	Mill Start Up	August 15	

9.3 Underground Pre-Production Budget

TABLE 6

Schedule	Requirements	Costs
June 1 - 2	Access Roads D8 - 20 hrs. @ $ 75.	$ 1500.00
June 2 - 7	Portal Rehabilitation	2000.00
June 2 - 15	Restore services 4 men @ $ 200. per day 10 days, plus Misc.	2500.00
June 15 - 21	Raise 25 ft. @ $ 100./ ft.	2500.00
June 21 - 30	Muck out winze 4 men @ $300. per day plus services @ $300. /day for 8 days.	4800.00
July 1 - 31	Winze timbering 240 ft. @ $75./ ft.	18000.00
July 1 - 31	Surface dump and chute 2 men - 20 days @ $ 50. per man day $ 2000. material $ 2000.	4000.00
Aug. 1 - 31	Stope preparation 4 men @ 20 days @ $300. per day $ 6000. Service & timber 4000.	10000.00
Total Underground Pre-Production Costs -		$ 45300.00
Plus Contingencies	TOTAL -	$ 50000.00

9.4 Surface Pre-Production Budget

See Section 10:3-6 for details of concentrator and concentrator construction and installation costs.

Concentrator - Capital Cost (12:6)	$ 57000.00
Concentrator installation & Construction (12.3)	94875.00
TOTAL -	$ 151875.00

Additional surface pre-production costs as per schedule are: -

TABLE 7

Schedule	Requirements	Costs
May 1 - 7	Access roads 40 hrs.- D8 @ $75.	$ 3000.00
May 1 - 7	Office renovation	1000.00
May 7 -21	Powder magazine and shop construction	5000.00
May 21-31	Tailing disposal 24 hrs. D8 @ $75.00	1800.00
May 21 - June 15	3900-2	4000.00
May 21 - June 1	Generator & Compressor pad installation and buildings	5000.00
	Total	$ 19800.00

9.5 Summary Pre-Production Costs

1.	Underground - Table 6	$ 50,000.00
2.	Surface - Table 7	19,800.00
3.	Concentrator Capital Cost (12.6)	57,000.00
4.	Concentrator Installation & Construction (12.3)	94,875.00
5.	Engineering, Supervision, Administration Transportation, Room and Board	25,000.00
	TOTAL	246,675.00
	TOTAL - Including contingencies	$ 275,000.00

10. PRODUCTION - MINING

10.1 Mining Methods

The actual physical characteristics of the developed ore bodies and ground conditions surrounding the ore bodies will determine the mining methods to be used and will result in modification of the mining methods during production.

At the Ruffner Mine the ore bodies characteristically are small and irregular. The average vein widths will be 1 to 4 feet and the average mining width will be a minimum of four feet. The average vein dip will be between 60° and 70°. Faulting along the footwall will result in a weak footwall and post mineral faulting will off-set ore bodies along strike and dip.

The ore bodies themselves are irregularly distributed along persistent structures and exploration will be confined to the main vein-dyke systems.

The sulphide ore both massive and disseminated is usually competent and the hanging wall is expected to stand well although slabbing might be a problem in stopes.

Small, irregular ore bodies usually call for some form of selective, high cost mining which should result in maximum extraction of higher grade ore.

Stoping methods can be adjusted to suit the characteristics of each ore body. The following schedule of methods has been outlined as a guide to stoping methods under variable underground conditions.

The constant conditions of narrow vein and steep dip are common at the Ruffner Mine.

Schedule of Application of Underground Mining Methods

TABLE 8

Type of Ore Body	Dip	Strength of Ore	Strength Of Walls	Methods
Very narrow veins	Steep	Strong or Weak	Strong or Weak	Resuing
Narrow vein	Steep	Strong	Strong	i. open under hand stopes ii. open over hand stopes iii. shrinkage stopes iv. filled flat back stopes v. filled rill stopes
Narrow vein	Steep	Strong	Weak	i. filled flat back stopes ii. filled rill stopes iii. square set stopes
Narrow vein	Steep	Weak	Strong	i. open underhand stopes ii. square set stopes
Narrow veins	Steep	Weak	Weak	i. square set stopes ii. top slicing iii. cross cut methods

The initial production location will be the 43-2 stope above the 4300-2 level at the winze. The second production location will be the 43-1A stope above the 4300-2 level near the portal. Recommended mining procedure for the 43-2 stope is a flat back shrinkage stoping method with stull support or rock bolts where hanging wall tends to slab. Recommended mining procedure for the 43-1A stope will be open, overhand, flat back resuing methods. Access will be by timbered man way of lagged stulls. Waste will be knocked to the level mucked with a muck machine, trammed to the 4300-2 portal and dumped. Ore will be blasted following removal of the waste, mucked from the level into cars and trammed to the ore pass. Should the ore continue up dip for any great distance, the drift back will have to be timbered and chutes installed. In this case, by carrying the mill hole with the stope back, the resuing method can be continued using waste for back fill and slushing ore into the mill hole. Recommended mining procedures for stope 43-1B below the 4300-2 level at this location will be by underhand methods. Should the ore continue down dip for any great distance, further stoping must be carried out from the sub level extended from the 43-2 winze. Using the under hand method, stope benches will be cut below the level. An inclined timber chute can be constructed to facilitate scrapping the ore up to the level. The ore will be mucked off the level with a muck machine and trammed to the ore pass.

10.2 Stoping Procedures

Stope preparation for the 43-2 stope will include drifting to the limits of ore on the level to the east and west of the winze. Cut out the first back stope and enlarge the drift to a height of 15 feet above the drift. Include both the hanging wall, foot wall and low grade centre of the vein in the cutting out stope as the entire width will grade ore in this stope. Remove broken ore with a muck machine down the ore pass. In order to establish a haulage way the drift back will be timbered with post and caps and lagged for the length of the ore shoot. Centre posts at 6 foot intervals and build chutes in alternate sets on 12 foot centres. Carry timber manways and tugger chutes at each end of the stope. Manways can be two lines of lagged stulls for vein widths under 8 feet. Widths in excess of 10 feet are not anticipated in which case cribbed manways should be built. The stope should be flat backed with breasts 8 feet high and holes flat 6 feet deep. Shallow holes will be required to decrease over break as the ore body rolls Sufficient muck will be drawn from the stope to allow miners to work off the muck pile. This will require about 40% draw of muck following each blast, at the appropriate chute. Ore can be transferred from the chute to the ore pass by hand trammed ore car as the distance is short.

Approximate ore broken per 2 man crew per shift across 6 foot width in the stope will be: $\frac{8 \times 6 \times 6}{10}$ = 28 to 30 tons.

At least 50% of the broken ore must be left in the stope for ground support and in order to work at the ore face. When the limits of the ore body has been mined and stoping is terminated the stope can be drawn empty. A considerable mill feed reserve will be created using this mining method. A capital investment will also be required with a delayed return.

Between 10 and 15 tons of mill feed will be available per shift from this stope which will not be sufficient to meet the demand at a 25 ton per day mill rate. A cross shift will therefore be required for stoping until a stock pile has been established. It is proposed that the exploration and development crew be cross shift to the production crew until such time as the stock pile is sufficient to maintain the mill rate.

10.3 Personnel & Costs

Personnel requirements during the first few months of production will be 2 - 2 man crews in the stope and a tram crew of two men who will tram ore to the surface on the 4100-2 level, fill the dump truck and deliver ore to the mill. A total of six men will therefore be required to mine and deliver the ore to the concentrator. As the stock pile builds the cross shift can be pulled from the stope and placed back on development until the stock pile is drawn to a point where they are again required for production.

Mining Costs

TABLE 9

2 miners @ $ 75.00 per day each		$150.00
Plus bonus of $25. per day ea.		50.00
2 helpers @ $ 50.00 per day each		100.00
Plus bonus $10. per day each		20.00
2 labourers @ $ 40.00 per day each		80.00
Plus bonus $10. per day each		20.00
Total payroll cost per day		420.00
Payroll cost per ton of ore $\frac{420}{60}$	=	7.00
Mine services and support costs per ton of ore	=	5.00
Total mining cost per ton of ore	=	$ 12.00

10.4 Production Equipment Requirements and Costs

TABLE 10

1.	Mancha trammer, spare batteries and charger	$ 6,000.00
2.	6 mine cars, 3 flat cars (new)	10,000.00
3.	Slusher	3,000.00
4.	Tugger	2,000.00
5.	4 drills, steel, parts, hose accessories	8,000.00
6.	2 pumps (pneumatic)	2,000.00
7.	1 Generator 5 Kw, 3 phase	2,500.00
8.	1 Compressor - 600 CFM (Used)	20,000.00
9.	1 Generator, Caterpillar, D348	37,000.00
10.	Powder	5,000.00
11.	Tools, Welder	5,000.00
12.	Ventilation Pipe	1,000.00
13.	2 Fans	2,000.00
14.	Victaulic pipe & couplings 4"x2"	5,000.00
15.	Crawler Tractor (Front End Loader) Used	15,000.00
16.	1 Dump Truck 5 - 10 ton (used)	10,000.00
17.	2 3/4 ton 4 x 4 trucks	14,000.00
18.	Wash trailer	5,000.00
19.	Miscellaneous	10,000.00
TOTAL		$ 162,500.00

11. METALLURGY

11.1 Summary

Considerable testing of ores from the Ruffner Mine has been conducted in the past by Britton Research Limited for Interprovincial Silver Mines Ltd., and results have been made available to Atlin Sliver Corporation. Additional testing work has also been conducted by Mr. H.E. Pawson. All details of tests are presently on file with Atlin Silver Corporation, and provide complete information on flowsheets, grinding requirements, the chemistry for floatation and other pertinent data. This report will include a review of this information, conclusions and concentrator design based on the conclusions.

Due to high shipping costs from Atlin, B.C. to any smelter either in southern B.C. or the U.S., shipping low grade zinc concentrate this distance might not be profitable and will certainly be dependent upon metal prices. Shipping costs, smelter costs and penalties may make this product uneconomical. Bulk shipments of zinc concentrate by container to Japan should be investigated.

Provided that the zinc concentrate is to be stockpiled at the mill site, this product should contain minimum silver values, the bulk of the silver being shipped with the lead concentrate. The lead concentrate must however be of good grade with a minimum waste content to provide low shipping and smelter costs per ton of ore, and above all mill recoveries must be adequate to prevent excessive metal loss to tails.

A review of the metallurgical balance of the three tests conducted by Britton Research Limited, 11.2 indicates that test 2 provides the optimum in reduced shipping and smelting costs for the silver - lead concentrate and provides for 84.2% recover of silver and 95.5% recovery of lead, with the lead content close to 50% and a concentrate percentage weight of ore of 9%.

Should the corresponding zinc concentrate be stockpiled, this product would contain only 2.5% of the recoverable silver, but would also contain 72.6% of the recoverable zinc.

Should metal prices provide for the shipment of both a silver-lead concentrate and a zinc concentrate, consideration should be given only to a high grade lead concentrate and zinc concentrate with a combined minimum shipping weight providing that metal loss to tails is reasonable. Test 3 (Britton 11.2) gives a total concentrate percentage weight of ore of 11.81% a total silver recovery of 83.7% (similar to Test 2) a total lead recovery of 95.9%and a total zinc recovery of 86% in the lead and zinc concentrate.

The metal price of zinc less smelter costs and penalties, less increased shipping costs for zinc concentrate 3.81% by weight of ore would determine the economy of processing zinc concentrate.

Concentrate drying has not been included in the mill flow sheet for reasons of initial capital cost. In order to reduce shipping costs such a modification is of immediate concern as cash flow develops and funds are available.

For purposes of valuation and smelter return calculations gold and cadmium content will not be considered but will act as a contingency bonus. For purposes of calculating net cash flow and smelter returns criteria for Test 3 (Britton 11:2)) will be used. This presupposes that a zinc concentrate will be shipped.

The following metal values are used for cash flow purposes: -

Metal	Recovery	Value
1. Silver - Ag	83.7%	$ 4.25 per oz.
2. Lead - Pb	95.9%	.20 per lb.
3. Zinc - Zn	86.0%	.30 per lb.

11.2 Tests

Four samples of ore and one sample of waste from the Ruffner Mine were used by Bitton Research Limited to make up a composite head sample for test purposes, which had the following assay.

Gold	- Au	0.02 oz. per ton
Silver	- Ag	23.8 oz. per ton
Lead	- Pb	4.60%
Zinc	- Zn	2.44%
Cadmium	- Cd	0.02%
Arsenic	- As	1.44%
Sulphur	- S	6.62%

Specific Gravity - 3.11
10.3 ft^3 per short ton.

Three complete flotation tests were carried out under identical conditions except that sodium cyanide added, varied from nil to 0.05 to 0.1 pounds per ton of ore.

Results showed that the grade of lead concentrate increased with increasing cyanide additions but silver recovery in the lead concentrate dropped. Part of the silver lost from the lead concentrate by increasing additions of cyanide was recovered later in the zinc concentrate. There was no change in the grade of zinc in the zinc concentrate.

The over all recovery of silver would be increased by 4% by omitting cynide from the circuit, but the lead assay of the lead concentrate would be reduced from 55% to 37% and the concentrate percentage weight of ore would be increased from 8.0% to 11.63% thereby increasing the shipping and smelting costs by almost 50%.

In each test the lead rougher concentrate was cleaned once and the zinc rougher concentrate was cleaned twice. The grade of zinc concentrate dropped with second recleaning.

The mill flow sheet is an economic compromise between market schedule of concentrate, cost of concentration and concentrate transportation costs. The lithologic character of the ore has controlled the method of concentration, recovery and cost. Reduced tonnage transported is the reward for increase in concentrate grade but results in higher metal loss to tails.

Metallurgical Balance from Composite Test Sample

Test Results determined by Variable additions of NaCN

TABLE 11

1 - Lead Cleaner concentrate assaying 179.9 oz. Ag. per ton, no NaCN additions.

Product	Weight %	Assays Au	Ag	Pb	Zn	Fe	% Recoveries Au	Ag	Pb	Zn
Lead Cleaner Concentrate	11.63	.14	179.9	37.48	4.01	20.70	74.9	88.8	94.1	19.1
Zinc Recleaner Concentrate	3.16	.03	13.6	.91	49.77	12.00	3.5	1.8	.6	64.3
Head	100.	.02	23.8	4.60	2.44		100.	100.	100.	100.
Combined Conc.	14.79						68.4	89.06	94.7	83.4

2 - Lead Cleaner assaying 219.3 oz. Ag. per ton, .05 lb. per ton NaCN addition

Product	Weight %	Assays Au	Ag	Pb	Zn	Fe	% Recoveries Au	Ag	Pb	Zn
Lead Cleaner Concentrate	9.00	.18	219.3	49.63	3.45	13.84	74.7	84.2	95.5	13.0
Zinc Recleaner Concentrate	3.51	.02	18.5	1.11	49.32	11.91	3.3	2.7	0.8	72.6
Head	100.	.02	23.8	4.60	2.44		100	100.	100.	100.
Combined Conc.	12.51						78.0	86.9	96.3	85.6

3. - Lead Cleaner concentrate assaying 233.7 oz. Ag. per ton, 0.1 lb. per ton NaCN add.

Product	Weight %	Assays Au	Ag	Pb	Zn	Fe	% Recoveries Au	Ag	Pb	Zn
Lead Cleaner Concentrate	8.00	.19	233.7	55.07	3.3	10.31	67.3	80.0	94.9	10.7
Zinc recleaner Concentrate	3.81	.04	23.1	1.28	48.93	12.07	7.1	3.7	1.0	75.3
Head	100.0	.02	23.8	4.60	2.44		100.	100.	100.	100.
Combined Conc.	11.81						74.4	83.7	95.9	86.0

According to Pawson, a natural circuit is indicated for silver-lead flotation. Alkalinity provided by lime will have an adverse effect on some of the silver minerals. Pyrite depression with lime during lead flotation will depress the silver minerals, prousite, pyrargyrite and stephanite. Iron will have to be controlled with NaCN during cleaning.

11.3 Smelter Returns

As per test No. 3 (Britton) assuming 100 tons milled assaying 23.8 oz. Ag. per ton, 4.6% Pb and 2.44% Zn per ton, no values considered for cadmium and gold, 11.81 tons of combined concentrates are produced equalling 8.00 tons of silver-lead concentrate and 3.81 tons of zinc concentrate representing 83.7% of the silver content, 95.9% of the lead content and 86.0% of the zinc content recovered.

Total Metal per 100 tons of ore	=	2380 oz. Ag.
		9200 lbs. Pb
		4880 lbs. Zn

8 tons of silver-lead concentrate produced equals

Ag	2380 oz.	x 80.0% rec.	x 95% payment	x $4.25	=	$7687.40
Pb	9200 lb.	x 94.9% rec.	x 95% payment	x .20	=	1658.85
Zn	4880 lb.	x 10.7% rec.	x 85% payment	x .30	=	133.15
			GROSS			$ 9479.40

3.81 tons of zinc concentrate produced equals

Ag	2380 oz.	x 3.7% rec.	x 95% payment	x $ 4.25	=	355.54
Pb	9200 lb.	x 1.0% rec.	x 95% payment	x .20	=	17.48
Zn	4880 lb.	x 75.3% rec.	x 85% payment	x .30	=	937.03
			GROSS			$ 1310.05

Gross Smelter Returns = $10,789.45 per 100 tons

Gross Smelter Returns per ton = $107.89

Gross Smelter Returns - Silver Lead Concentrate	$ 9479.40
LESS: Shipping, smelting, marketing $80. x 8	640.00
Dec 22/75 revision 77.00 21.00 10.00 108.00 x 8	836.00
Net Smelter Return - Silver-lead concentrate	$ 8839.40

Dec 22/73 Revision

Gross Smelter Return - Zinc concentrate	$ 1310.05
LESS: 12% gross value of concentrate	
Shipping, smelting, marketing $80.x3.81 (50.00; 77.00; 21.00; 10.00; 108.00)	304.80
Net Smelter Return Zinc Concentrate	$ 1005.25
Total Net Smelter Return	$ 9844.65
Net Smelter Return per ton of ore	98.45
Total Shipping, smelting, marketing costs per 100 tons	$ 944.80
Cost per Ton of Ore	$ 9.45

Run of the mine ore with an average grade of 23.8 oz. Ag. per ton, 4.6% Pb per ton, 2.44% zinc per ton, would provide the following returns: -

Ore value per ton	=	$ 134.19
Net Return	=	$ 98.45
Milling losses, smelting, shipping, marketing costs	=	$ 35.74
Percentage cost and mill loss factor	=	26.5%
Ore grade net cash flow factor	=	73.5%

11.4 Flowsheet

The type of flowsheet used depends upon the characteristic of the crude ore and decisions to be made as to location, capacity and grade of product. The essential elements of the flow sheet must be such as to take advantage of the variations in physical and chemical properties of the mineral constituents of the crude ore; the details will vary; in general high recovery of a low grade product and low recovery of a high grade product are equally easy and cheap to obtain.

Character of the finished product is determined by consideration of the character of the crude ore and of the concentration process available and variations of marketability with quality. If metalliferous products are to be shipped long distances at high freight rates (Ruffner Mine); ore dressing must aim at maximum elimination of every waste substance physically rejectable including water.

Maximum profit is the aim of a concentrating operation and this is usually attained by some combination of recovery and grade of concentrate that does not include the maximum of either.

The quantity and quality of ore available determines not only whether a mill should be built, but dictates capacity and type of construction. The mineralogical character of the ore is controlling as to the treatment scheme. Capacity is determined by potential mine capacity, cost of construction, available capital and demand for product.

The mineralogical character of the ore is important from the standpoint of distribution of different minerals and relative quantities as well as chemical nature and physical characteristics. It determines and limits the number and kinds of processes available and at what grain size they may be applied.

Rich ores require and can stand the cost of more elaborate treatment than poor ores, large ore bodies justify larger and more elaborate mills than small. When freight rates are high or smelter penalties for impurities great, extensive treatment designed to raise the grade of concentrate is justified or high grade concentrate may be made at the expense of low recovery.

Tonnage is an important factor in flow sheet design. If tonnage is small, the simplest type of flow sheet only should be considered, even when a low ration of concentration might indicate an elaborate mill.

In general small mills with a capacity of 25 T.P.D. to 75 T.P.D. require a coarse ore bin with a capacity of about 24 hours, a grizzly should be installed with openings smaller than the jaw crusher opening. Generally only a single toggle type jaw crusher is used for crushing to as small a size as possible in one pass and for simplicity and economics no secondary crusher is used. The Atlin Silver Corp. mill has been designed to include a secondary crusher. This will facilitate and allow crushing to a fine size - 3/8" ball mill feed in order to increase grinding capacity to a fine mesh 75% - 200 as required by metallurgical tests, rather than a standard grind for flotation purposes of 48 mesh.

Additional enclosed ore storage is obtained with the inclusion of a fine ore bin having a capacity of 2 days mill feed. This will smooth out the flow of dry pulp to the grinding circuit. Discharge from the fine ore bin will be through a simple pipe type belt feeder.

Valuable metallics including sulphides which free coarse and fine metallic silver will be taken out of the pulp stream as early as possible in order to reduce concentrating costs and to prevent losses due to sliming. This will be accomplished by means of a unit cell at the ball mill discharge. If appreciable amounts of coarse silver are encountered in the ore, it will have to be removed with a gravity type machine probably a mineral jig. Gravity concentration has not been included in the flow sheet as no coarse native silver has been observed in the ore.

A spiral classifier has been included in closed circuit with the ball mill and unit cell in order that control of the fine grind 75% - 200 mesh can be maintained and the recommended pulp size can be delivered to flotation.

Subaeration type flotation machines are recommended and have been purchased. One multicell unit is to serve for rougher and cleaner purposes for each concentrate. The subaeration type machine will give the best service in a small mill since it posesses maximum operating flexibility and adaptability to changing ore conditions common in the type ore body at Ruffner Mine.

In small mills the concentrate is usually run directly to the filter or to a surge tank or sump and then to the filter. In large mills a thickner is used ahead of the filter. A thickner has been included in the flowsheet for lead concentrate only, due to appreciable amounts of lead contained in the ore, to give flexibility to increase rate of production if required and due to the small filter size (purchased). A surge tank may be required ahead of the zinc filter.

A disc type vacuum filter is recommended for small mill service and this is what will be used at the mill.

Drying is the process of removing water from solid or semi-fluid materials by vaporization and is employed in mill operations to save freight charges and to facilitate concentrate handling. No dryer unit has been included in the flow sheet for reasons of capital cost. It is mandatory that a drying system be installed as an initial mill modification in the near future. The design of the concentrate bins will allow for drainage of water from the concentrate.

Flow Sheet recommendations (Britton)

The following flow sheet has been recommended by Britton Research Limited to the Interprovincial Silver Mines Ltd., for Ruffner Mine ore and has been used as a basis for mill design for Atlin Silver Corporation.

1. Crushing to - 1/2"
2. Grinding to - 75% - 200 mesh in closed circuit with classifier and mineral jig.

3. Silver-Lead rougher flotation, with two stage cleaning, recleaner tailings returned to head of cleaner cells; cleaner tailing returned to ball mill discharge.
4. Conditioning of lead rougher tailings.
5. Zinc rougher flotation, three stage cleaning, cleaner tailings recirculated to head of previous stage.
6. Thickening, filtering, drying.

Pawson (1969) recommends a flotation reagent choice of potassium isopropyl xanthate for a lead silver collector, aerofloat 31 as a promoter, cresylic acid as a frother and Z-200 as a zinc collector. Adjustments in additions of sodium cyanide can be made according to Britton.

Reagent details are available in Britton Research Limited - Progress Report No. 1, Project B206 - January 13th, 1969.

12. CONCENTRATOR

12.1 Concentrator Requirements

The writer located for C.W. Dansey of Atlin Silver Corporation a small used crushing plant and concentrator at Silverton, B.C. It was determined that this plant would approximate flow sheet requirements (Britton) with some minor equipment additions and with some modifications. The plant was inspected by Mr. Dansey and a decision was made to purchase the mill in the fall of 1974. The purchase price was $ 26,000.00.

Subsequently, the plant was removed from its Silverton location at a cost of $5,000.00 and has been rehabilitated by Nelson Machinery Limited of Vancouver during the winter of 1974-75. Cost of rehabilitation and the purchase of a 6 cell flotation unit and a unit cell is $20,000.00. All electric motors have been inspected and new bearings have been installed by Thompson Valley Rewind, Kamloops, B.C. at a cost of $1,000.00.

The plant is to be shipped to Atlin B.C., in April, 1975 at an expected cost of $5,000.00.

No engineering diagrams, mill plans or equipment specifications were available from the previous owner. All of this information has been gathered, researched and drawn by the writer and is felt to be correct.

The following is a list of the main equipment purchased with an estimate of specifications (Taggart)- Pumps, motors and other miscellaneous equipment is now shown.

Equipment & Specifications

Jaw Crusher - Type - Blake   Make - Pacific
Model - 10" x 20"

Recommended R.P.M. range - 250 - 300

Recommended H.P. - 20 - 25

Full through machine feet - 2.5

Capacity - moderately hard hs - 12.5 tons per hr. @ 1.5" set.

Specific gravity ore 3.11 (Britton)

Specific gravity of calcite 2.7

Estimate equivalent toonage capacity of ore - 13 tons/hr.

No adjustment has been made for hardness.

Life of jaw plates varies according to the material used and service required.

Three to six months is an average life for manganese steel plates in ordinary service at 24 hours per day.

Reduction Crusher - Type - Hydrocone
Make - Alice Chalmers R322.

Speed - Gyrations per minute - 300 - 360

Recommended H.P. - 30

Gape - 4"

Short throw type size 3/8" = 14 tons per hour.

No adjustment for specific gravity.

Wearing parts, life (days) at 24 hours estimated.

Eccentric - 75 - 100 days

Manganese mantle - 300 days

Gears - 200 days

Manganese concaves - 150 days.

Ball Mill - Type - Conical   Make - Harding
Model 22" x 6'

Weight - 15,300 lbs.

Liners - 10,000 lbs.

Ball Change - 5.25 - 6.75 tons - 35 to 50% of mill volume

Speed - R.P.M. - 21.2 - 27.7

Motor - 50 - 60 H.P. recommended

Power consumption - 52 - 60

Estimate capacity - Tons per hour = 3/8 " feed to
200 mesh of grind = 2.2 tons
Tons per day - 52.8 average ore

Steel consumption est. - lbs. per ton of new feed
Bulls - 1.25
Liners - 1.0

Estimated liner life - 250 days.

Classifier - Type - Spiral Make - Akins
Model - 30" x 14'9"

Capacity - Overflow tons per 24 hours -
100 mesh @ 20% solids = 75 tons
150 mesh @ 18% solids = 50 tons
200 mesh @ 15% solids = 35 tons

Flotation Cells - Type - Sub aeration
Make - Denver No. 21
Model No. 21 - 6 cell units.

Dimensions - Length 38" Width 38" Depth 39"

Cell Volume - 40 $ft.^3$

Motor - 7.5 H.P.

Maximum recommended flow, tons of soilds per 24 hours per 6 cell unit.

15% solids - 230 Tons (less 1/3 - 2 cleaner cells)
= 150 tons - 50 tons per 8 hours.

20% solids = 320 tons (less 1/3 - 2 cleaner cells)
= 210 tons = 70 tons per 8 hours.

<u>Filter</u> - Type - American Disc   Make - Eimco
Model - 4' x 4'

Layout for vacuum system - See Taggart 16-06 - Fig. 6

Speed - 4 to 5 R.P.M.

Feed - Pb

Size - 70% - 200 mesh

Solids - 75%

Tons of solids per hour	- 3
Tons of solids per day	- 72
Coke thickness	- 3/4"
Moisture	- 7.5% at 45% solids in feed
Filter Bag life est.	- 25 days (cloth)

Moisture - Lead concentrate moisture content can be lowered by diminishing thickness of cake
1/4" cake moisture content = 5%
Zinc concentrate moisture content can be lowered by increasing thickness of cake.
1 1/4" cake moisture content = 6.5%

The required capacity of a mineral dressing plant depends upon the potential yield of the source of supply, the rate of yield depends upon the geometry of the source, market demand and available capital. The selection of a mill size for the Ruffner Mine was based primarily upon the geometry of the ore body. Available and potential reserves would dictate a larger installation. New ore development will be directly preportionate to available capital for drifting along the strike of the known vein system at regular intervals. Frequency of ore bodies is unknown along the system and therefore the timing of discovery is unknown. Present reserves that are known should provide for a constant supply of mill feed for three to five years, at 50 T.P.D. during which time the discovery of new ore can be expected. The design of the 50 T.P.D. plant is such that with slight modification a milling rate of 60 T.P.D. might be attained. A milling rate of less than 50 T.P.D. can quite easily be arranged by a cut back in shifts. Considerable flexibility is therefore provided in milling capacity which will be necessary with this type of ore body where variation in grade and tonnage mined can be significant.

A greater degree of enclosed storage capacity than normal for this size installation has been included in the design to ensure that irregularities in mine production rates are smoothed out in the mill. This will be essential especially during summer months when ore supplies are expected to be provided from various small workings at different locations and at higher elevations on the property where several miles of truck haulage is anticipated.

Crusher over capacity with respect to concentrator requirements and several days supply of mill feed in the fine ore bin will allow the crushing plant to operate independently of the mill.

Transport of dry pulp is by two conveyors at the horizontal and on modest incline. Elimination of excessive dry pulp transport was a major consideration in design concept. Proper adjustment of site elevations and the secondary crusher installation over the fine ore bin has minimized costly pulp transport systems to the essential. Wet pulp transport will consist of pumping through plastic pipe which will minimize initial high cost plumbing requirements.

For reasons of simplicity and reduced capital and operating cost, the entire concentration process will be by flotation. Consideration will be given later to gravity concentrate as mentioned.

Water Requirements

Water requirements for milling purposes are expected to be 3 tons of water per ton of ore treated or 150 tons of water per 24 hours at 50 T.P.D.

150 tons x 200 gal = 30,000 gal. of water per day.
Comparison of water requirements to flow in a 1" pipe.

1" pipe @ 1.86 ft. per sec. = 5 gal. per minute or
7,200 gal U.S. per 24 hrs.
1" pipe @ 7.44 ft. per sec. = 20 gal. per minute or
28,800 gal. U.S. per 24 hrs.

Comparison of water requirements to flow in a 2" pipe.

2" pipe @ 2.04 ft. per sec. = 20 gal per minute or
28,800 gal. per 24 hrs.

Approximate water requirements for the concentrator will be 20 gallons per minute.

The entire source of mill water will be from underground workings 3900 level and 4100 level and will flow by gravity to the mill. A final study of water flow from these sources will be taken prior to mill construction in view of storage dam and sump requirements underground.

Power Requirements

Total installed horsepower crushing plant and mill	=	200
50% of largest motor, ball mill drive 50 H.P. (Surge)	=	25
Misc. blowers, heaters, lights 10%	=	20

Total Requirements = 245 h.p.

Requirements per 24 hr. ton = 4.9 H.P.

The mill requirements outlined as far as possible, satisfy the following demands.

1. Transfer of feed and products by the shortest and most direct route utilizing gravity flow so far as economically practical.

2. Convenience of superintendance due to there being only two main working levels with inside access to all working areas.

3. Possibilities for modification and enlargement.

4. Dust making operations confined to one building.

12.2 Concentrator Location

The location of the concentrator at the Ruffner Mine will be close to the main developed ore body and below the three main lower level adits. The dump ramp for the coarse ore bin will be slightly lower in elevation than the lowest mine access, 3900 level and will be located at that portal. The source of mill feed for the next few years will be the 43-2 ore body and the main ore haulage levels will be 3900 and 4100, located above the mill. Consideration will be given to a slusher chute from 4100 level dump down to the coarse ore bin at a later date, initial mill feed delivery will be by dump truck. At a later stage in mine development it is proposed that the 3900 level accommodate ore haulage at which time ore will be trammed by underground tram directly to the coarse ore bin. This program will require raise access between 3900 and 4100 levels. The occurance of mineral is quite extensive laterally on the property and it will be necessary during summer months to truck ore from high levels of the mine.

The decision on the type of mill (terraced or one-level) and site are based on comparison and consideration of the following points: -

1. Elevation of ore.
2. Elevation of water.
3. Elevation of tailings
4. Final elevation of concentrate.
5. Capital cost of terrraced mill
6. Topography and climate.
7. Labour and supervision terraced versus level mill.

The slope of ground selected for the mill site is a determining factor in the cost of construction and in cost of operation. Steep slopes of 25^{o} to 30^{o} or more should be avoided, moderate slopes of 5^{o} to 15^{o} are preferable. The expensive many terraced mill should be avoided.

In modern mills crushing is done in a building separate and independent of the concentrator, inclined conveyors distribute the crushed product to ore storage bins.

The mill design for Atlin Silver Corporation is a compromise between a terraced mill and a one level mill. The concept is essentially 2 - one level plants one above the other. The crushing unit is a separate one level unit located directly above a one level concentrator. By proceeding in this manner expensive conveyor systems for elevation of crushed product have all but been eliminated, thus costly conveyor housing and heating do not exist. Essentially ore flow will be by gravity from the upper level at the secondary crusher to the short horizontal ball mill feed conveyor.

One elevation difference exists between cone crusher and ball mill facilitating supervision, employee arrangement and most important simplifying material handling.

To simplify design and construction of the concentrator this plant has been designed as a one level unit, various processes and equipment adjusted to required height within the unit by means of timbered or concrete piers. The concentrator site will be level for additional simplification and accumulating water on the floor in the wet part of the mill will be drained via concrete conduits one foot wide sloping at 1/2" to the foot from front to back of the mill and from sides to centre. Normal procedures for disposal of water in larger more elaborate mills is a sloping floor.

Concentrate will not exceed 15% of the weight of the original ore and an elevation drop to permit gravity disposal of concentrate is expensive. Therefore a gravity disposal and concentrate storage system within the main building has been conceived.

The height of vertical members has been reduced to a few dimensions and wherever possible wall footings and roof have been brought to a common level.

The location of the mill below the 3900 level means that all ore developed or to be developed in the foreseable future will be located above the level of the coarse ore bin and as such can be transported by mine car, truck or chute to the coarse ore bin assisted by gravity.

Water supply to the mill will flow by gravity from underground workings at 3900 and 4100 elevations.

The concentrator will be located for tailings disposal by gravity to a previously designated disposal site on a crown granted mineral claim at a considerable distance from surface drainage. The tailings launder will be either rectangular plank construction or more likely plastic piping enclosed and insulated. The velocity of flow required for gravity transportation of tailings in rectangular flumes depends upon the size of the grains, their specific gravity and on the dilution of the pulp. Generally mill tailings is a siliceous gangue (S.G. 2.6) and contains 25% to 30% solids. A tailing launder 24" wide and 6" deep has a mean radius of .33 feet. Tailing grain size of 100 mesh requires a velocity of 3.8 feet per second. Slope requirements for gravity disposal under the above described conditions requires a drop of 1 inch per 8 feet.

Tailings will be impounded at the disposal site by excavations in the earth on a gentle sloping hillside. Several excavations, one below the other, will be required to prevent the escape of toxic waters. Dried tailings will be utilized later for road construction purposes. No consideration has been given to water recovery and all water is expected to be dispersed by ground perculation into glacial boulder till. Should winter storage prove a problem a peripheral dam about 3 feet high will be constructed in the fall around the downslope perimeter of the tailing pond. Should overflow result from prolonged cold weather, spill will be retained in the lower excavation.

Concentrate will be pumped to the filter elevation within the mill building and will be delivered by gravity to storage through chutes to 45 gal. drums or similar containers situated on mine flat cars vertically below storage. The loaded flatcars will be trammed by hand to outside storage for shipment from the property. The concentrate storage site will be located adjacent to the main access road from the property. Minor storage site preparation and road access construction will be required for concentrate handling.

The topography at the mill site will be moderately sloping to the north at about 15°. Excavation required for the two main levels will be completed with a D-8 bulldozer. The two main levels are: -

1. Crusher level
2. Concentrator and storage level.

Due to severe winter climatic conditions the entire plant with the exception of concentrate container storage will be housed in three insulated, heated buildings. These buildings will house coarse and fine ore bins, concentrate bins and all conveyor systems. Access to all phases of the operation will not require operators or supervision to leave the buildings.

12.3 Installation and Costs

The following installation procedures are recommended with cost estimates.

1.	Three elevations terraced by bulldozer levelled for correct elevation with access road construction to each elevation - D8 - 50 hrs. @ $75./hr.	$ 3,750.00
2.	Location and construction of 2 main retaining walls with ore bin supports followed by drain tile and back fill. 250 cu. yds. concrete @ $ 75. per yd.	18,750.00
3.	Location and construction of crusher mounts, ball mill mounts concentrate storage bin support mounts, minor retaining walls - 70 cu. yds. concrete @ $75.00 per yard	5,250.00

The concrete for retaining walls and machinery support mounts should be designed to obtain maximum strength through a combination of high grade concrete (3000 lb/in^2) and adequate reinforcing steel, suggest using old mine rail located on the property.

There will be no bedrock available within the designated mill site area and therefore considerable concrete will be required as noted above.

4.	Installation of machinery and construction of ore bins, timber walkways and timber stairways			$ 10,000.00
5.	Building construction, suggest permanent pole type construction, with steel roof and siding sheets, insulated roof and walls, 2" x 6" wall studs cut locally			
	Buildings delivered	=	$ 15,000.	
	Local cut timber	=	2,500.	
	Erection	=	5,000.	
	Concrete footings	=	1,000.	23,500.00
6.	Plumbing			10,000.00
7.	Wiring (All electric panels were salvaged from the Silverton, B.C. location			15,000.00
				$ 86,250.00
8.	Miscellaneous and contingencies includes water and tailings - 10%			8,625.00
	TOTAL			$ 94,875.00

12.4 Operation

It will be necessary that the services of a qualified practical mill operator be enlisted prior to and during construction, to review the mill plans and budget and to modify the concentrator design during construction to provide optimum operating conditions. This mill operator should later supervise milling on the property during production.

It is suggested that the mill be run in on lower grade ores and that the grade of ore be increased with proficiency in recovery.

An initial minimum production rate of 25 T.P.D. should be adequate to provide a profit during initial milling on the property. The crushing plant will operate for 2 hours per day or 4 hours every 2 days. The ball mill capacity on 3/8" feed to 200 mesh is 2.2 tons per hour. It is probable that by decreasing feed size to 1/4" and grinding to 75% - 200 mesh that ball mill capacity can be increased to 3 tons plus per hour, sufficient to grind 25 tons in an 8 hour shift.

Coarse ore will be stock piled during mill construction and will be adequate to supply the mill without feed fluctuations and on a continuing basis following start up. The main ore storage will be underground during the first three years of production and during cold winter months it may be necessary to maximize underground storage. Delivery of ore to the mill from the mine should be scheduled to meet mill demand without interruption.

12.5 Personnel and Costs

25 T.P.D. - Rate of production

1. Mill Operator	- Operator and Supervision per month	$ 1,500.00
2. Mill Assistant	- Crusher operator, concentrate handling, general assistant - per month	800.00
TOTAL - Per Month		$ 2,300.00

Operate 25 days per month at 25 T.P.D.
= 625 tons per month.

Personnel cost per ton of ore $ 3.68

50 T.P.D. - Rate of Production

1.	Mill Operator - Operator & Supervisor	$ 1,500.00
2.	Mill Assistant - Crusher Operator and Concentrate handling	800.00
3.	Mill Assistant (2) - Mill Assistant x Shift	800.00
	TOTAL - Per Month	$ 3,100.00

Operate 25 days per month at 50 T.P.D.
= 1,250 tons per month.

Personnel cost per ton of ore = $ 2.48

12.6 Budget and Operating Cost Estimate

1. Capital Cost Mill

a.	Purchase Price	$ 26,000.00
b.	Removal from original site	5,000.00
c.	Rehabilitation, additional equip.	20,000.00
d.	Rehabilitation, electric motors	1,000.00
e.	Shipping Vancouver - Atlin	5,000.00
		57,000.00
f.	Construction, installation, buildings (12.3)	94,875.00
	TOTAL ESTIMATED CAPITAL COSTS	$ 151,875.00
	Cost per ton treatment capacity	$ 3,000.00

2. Mill Operating Costs

a.	Personnel requirement 25 T.P.D. (12.5) - Per ton	=	$ 3.68
b.	(Personnel requirement (50 T.P.D. (12.5) - Per ton	=	$ 2.48))
c.	Power costs, deisel generator servicing, maintenance $2.25 per hour, or $ 18.00 per shift 25 gal. diesel fuel per 8 hour shift @ $ .60 = $ 15.00 General contingencies $4.00 per 8 hour shift. Total cost per shift per 25 tons = $ 35.00	=	$ 1.40 per ton
d.	Balls, chemicals, crusher plates, filter bags, general supplies	=	$ 1.50 per ton
e.	Maintenance, mechanic, parts, grease	=	$ 2.00 per ton
f.	Contingencies and misc.	=	$ .42 per ton

Estimated Total Milling Cost @ 25 T.P.D. = $9.00 per ton.

13. ADMINISTRATION

13.1 Head Office - (Month)

1.	Management, Engineering, Accounting	$ 3,500.00
2.	Telephone, postage	250.00
3.	Rent and Secretarial	750.00
4.	Transportation and accommodation	500.00
	Total - Per Month	$ 5,000.00

13.2 Mine Office - (Month)

1.	Mine Manager	$ 1,500.00
2.	Contingencies	$ 500.00
	Total - Per Month	$ 2,000.00

Administration Costs per month - Total - $ 7,000.00

Administration Costs per ton of ore: -

$$\frac{7000}{625} = \$\ 11.20$$

14. MARKETING

Calculations in Sec. 11.3 shows mill losses, smelter costs, shipping costs and marketing costs as a factor of 26.5% of the value of ore worth $134.19. This percentage factor in terms of dollars is $35.74 per ton. These costs are expected to drop as less concentrate will be produced at the calculated ore grade of the mine. However, due to a lower grade, for average mine grade ore the cost and loss factor will remain about the same. The cost and loss factor used in net cash flow calculations will be 25%.

It can be assumed from the calculations that shipping, smelting and marketing costs per ton of ore are $9.45 ? excluding mill losses. These calculations are based on the test head sample.

Marketing procedures are under consideration at the present time. It is probable that the lead-silver concentrate will be sold to either Cominco at Trail, British Columbia, or will be delivered to East Helena, Montana.

Zinc concentrate will probably be marketed at Kelloge, Idaho.

15. COST SUMMARY

15.1 Capital Costs

1.	Pre-production Cost (9.5) (including concentrator and installation)	$ 275,000.00
2.	Production Equipment (10.4)	162,500.00
3.	Working Capital	50,000.00
	Total Capital Requirements	$ 487,500.00

15.2 Operating Costs Per Ton of Ore @ 25 T.P.D.

1.	Exploration and Development Costs 1976 (8.6)	$ 9.50
2.	Mining and Transporation Costs to Mill (10.3)	12.00
3.	Milling Costs (12.6)	9.00
4.	Administration includes mine office and engineering (13.2)	11.20
5.	Climatic Factor, snow removal, heat.	2.00
	Total Operating Costs Per Ton of Ore to Concentrate bin.	$ 43.70

16. CONCLUSIONS

16.1 Cash Flow (Test Ore) - 25 T.P.D.

A review of the three metallurgical tests conducted by Britton Research Limited on an average grade test sample indicates that providing a zinc concentrate is to be marketed along with silver-lead concentrate Test 3 produced a preferable product for shipping and marketing purposes.

At a production rate of 25 tons per day; 2 tons of silver-lead concentrate would be produced and .95 tons of zinc concentrate would be produced. Using a test head grade of 23.8 oz. Ag. per ton and 7% combined lead-zinc per ton:

Net Smelter Returns, (Test ore)	$ 2,461.16
Operating Costs per ton (15.2) = $ 43.70	
Operating Costs Per Day = 25 x $43.70	$ 1,092.50
Net Cash Flow on Test Ore	$ 1,368.66

16.2 Net Cash Flow (Average Grade Ore) - 25 T.P.D.

Average Ore Grade	=	18.63 oz. Ag. per ton and 5% Pb-Zn per ton (25¢)
Value of Ore per ton	=	$104.18
Percentage Cost and Mill Loss Factor 25% (11.3)	=	$ 26.05
Net Smelter Returns per ton of ore	=	$ 78.13
Operating Costs per ton of ore (15.2)	=	$ 43.70
Net Cash Flow per ton	=	$ 34.43
Net Cash Flow per day (34.43 x 25)	=	$860.75

16.3 Revenue

Net Cash Flow per day	$ 860.75
Operating days per month	25
Net Cash Flow per month	$ 21,518.75
Operating months per year	11
Net Cash Flow per year	$ 236,706.25
Ore requirements per year - in tons	6,875
Proven & Probable Resersves- in tons	63,920
Mine Life at 25 T.P.D. expected on present reserves - in years	9
Total Net Cash Flow	$ 2,130,356.25
Captial Costs	$ 487,500.00
Profit before taxes, royalties amortization	$ 1,642,856.25

17. RECOMMENDATIONS

Careful examination of the details in this feasibility study indicates that the Ruffner Mine can be placed into production on a small scale. Through careful mining practices with particular attention being paid to overbreak and dilution in stopes, ore grades exist at present metal prices to cover costs and provide a profit on the operation, at a minimum production rate of 25 tons per day. The concentrator has been designed with a maximum milling capacity of about 60 tons per day. The rate of production is therefore quite flexible should larger widths of lower grade ore be encountered.

A production rate in excess of 60 tons per day if necessary would require installation of a larger ball mill and filter.

Production at the minimum rate of 25 tons per day should continue uninterupted with the exception of winter down time for overhaul and maintenance. A minimum of 6 years production at 25 tons per day and a maximum of 9 years production is anticipated in order to extract and process the present reserves. Providing metal prices stabilize at present world market levels a profit would be realized on the mine.

Special emphasis should be placed on cost control and budget both during production and during pre-production. The production rate per day is not sufficient to absorb excessive operating expenses. Careful mining practice will be required as dilution will quickly turn the expected net cash flow into a net cash outflow.

The writer has carefully examined all details of the information included in this feasibility and recommendations have been made after careful consideration of these details.

This study is herewith, respectfully submitted.

James C. Snell, B. Sc.
Mining Geologist.

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