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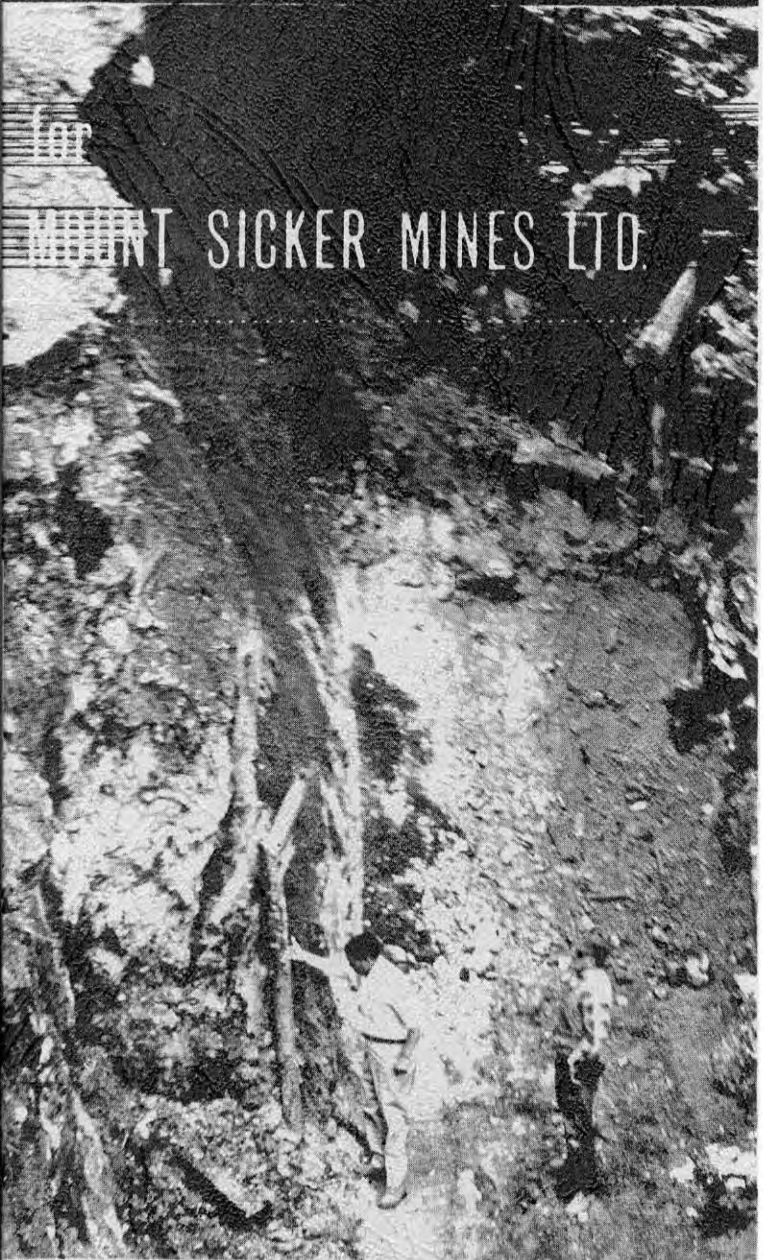
FEASIBILITY STUDY

for

MOUNT SICKER MINES LTD.



Bio-Metals Corporation Limited

PROGRESS REPORT
on
FEASIBILITY STUDY
for
MOUNT SICKER MINES LTD.

Vancouver, B.C.
May 24, 1967.

Late in 1966 an agreement between Bio Metals Corporation Ltd. and Mount Sicker Mines Ltd. was signed and implementation was commenced. As part of the agreement Bio Metals undertook to make a feasibility study to determine whether or not the dumps and metal-bearing zones at the mine near Duncan, B.C. could be leached.

The work has been conducted under the guidance of C. L. Emery, P. Eng. and is supervised directly by Garry M. Hughes, B.Sc. on behalf of Bio Metals Corporation Ltd. Several technicians and one other engineer have participated in the work done to date.

Work done includes:

1. Trips to the property for examination, sampling and design layout.
2. Continuous leaching tests on prepared samples done in Bio Metals laboratory.
3. Chemical tests and assays to determine ore characteristics and values.
4. Engineering estimates and reports. Work began in November 1966 and is currently in progress.

Property

Mount Sicker Mines Ltd. holds 26 Crown Granted claims and an additional 26 claims located on Mount Sicker which lies eight miles north of the city of Duncan, B.C. on Vancouver Island.

The geology of the Mount Sicker Mines (formerly the Twin "J" Mine) has been covered by John S. Stevenson in C.I.M.M. Trans. Vol. XLVIII, 1945, pages 294 to 308. The rocks of the area including the mine are cherty tuffs, graphitic schists, sodic-andesite porphyry, sodic-rhyolite porphyry and sodic diorite. The tuffs and schists form a band about 100 to 150 feet wide and these are visible in the mine workings over a length of at least 2100 feet according to Stevenson. They may extend much farther.

The ore bodies developed to date are found as replacements in the folded cherty tuffs and related schists, and because of faulting occurs as two separate easterly trending bodies running parallel and about 150 feet apart.

The north ore body has a strike of about 1700 feet and a down-dip extent of about 120 feet. It ranged from one to ten feet in thickness as defined by the cut-off grade used at the time of exploration. However, surface indications and other records indicate that if a lower cut-off grade is used much more ore may be available.

The south ore body lies about 150 feet south of the north ore body and has a length of 2100 feet, a depth of 150 feet and a thickness of about 20 feet using old criteria for ore. Present criteria will likely result in a substantial increase in some of these dimensions.

There are two types of ore in each body. An ore type, barite forms a matrix containing some calcite and a fine-grained mixture of pyrite, chalcopyrite, sphalerite and gelena. In the other type barite and enclosing schists are replaced with quartz in long lenticular masses and stringers. This ore is mineralized with pyrite, chalcopyrite, sphalerite and gelena. This last type formed the bulk of the material extracted to date.

Using the stated figures the original North and South ore bodies contained about 1,100,000 tons of ore. The grade was about 4% copper. If a lower grade is used the volume would be substantially increased.

Production History

South Orebody: The south orebody was mined between 1898 and 1909 by three separate companies each of which worked one claim. The production figures compiled by Stevenson are as follows:

Mine	Tons	Au oz/ton	Ag oz/ton	% Cu
Lenora (1898-1907)	78,983	0.132	3.5	3.75
Tyee (1901-1909)	168,290	0.145	2.6	3.8
Richard III (1903-1907)	5,405	0.136	3.1	2.3
	252,678	0.140	2.9	3.76

The main shaft of the Tyee reached a depth of 1400 ft.

North Orebody: Between 1943 and 1947 the north orebody was worked by the Twin "J" Mines Ltd. which was an amalgamation of the three previous companies. A 150 ton per day mill was erected on the property and between July, 1943 and May 1944, 34,893 tons of ore were milled with an average assay of:

Au oz/ton	Ag oz/ton	Cu%	Zn%	Pb%
0.075	2.05	1.32	6.12	0.60

Operations were suspended in 1944 and resumed in 1947 when 8,295 tons of ore were treated with an average assay of:

Au oz/ton	Ag oz/ton	Cu%	Zn%
0.061	1.91	1.05	3.23

The mine was re-opened in 1950 by Vancouver Island Base Metals Ltd. This company treated 9,754 tons with an average assay of:

Au oz/ton	Ag oz/ton	Cu%	Zn%	Cd lbs/ton**
0.032	1.59	0.44	3.66	0.27

In November of 1964 the present owners mined 167 dry short tons from a large stope pillar extending to the surface above the Lenora mine. The material as sent to the Tacoma smelter assayed as follows:

Au oz/ton	Ag oz/ton	Cu%
0.10	2.26	3.04

Zinc was neither assayed for nor recovered by the smelter but according to J. P. Elwell, the consulting engineer at the time the zinc: copper ratio appeared to be around 3:1.

Results of Current Work

Inspection and survey of the property indicate that there are three major mine dumps containing sulphide-bearing material. Sampling of the three dumps was done by driving shallow trenches across their tops with a small bulldozer. It was noted that all of the dumps contained a large percentage of fine material in the regions sampled but that the lower portions of the dumps contained more coarse material. This is normal in a dump built by tipping. The coarse lumps roll down and the fines build higher up. Visual inspection indicated more sulphide in the lumps than in the fines.

The Tyee dump is the largest and, depending on the assumed geometry of the surface on which it sits, this dump may contain from 50,000 to 150,000 tons of material. A good estimate is probably 100,000 tons or more. The material is partially oxidized and averages 0.48% copper in the samples taken. Zinc and lead mineralization appear to be present in about the mine ratio and may amount to 1.5% zinc with smaller amounts of lead. Part of this dump is made up of ore in a barite gangue.

**Cadmium is worth $2.60 per lb. (E & M.J. December 1966)

The upper Lenora dump contains between 20 and 50,000 tons of material, depending on the assumed rock base contours. The samples indicate a grade of 0.22% copper with the usual ratio of lead and zinc. Visual inspection indicates the true content including the coarse material may run as much as double that indicated in the assays.

The material tested was high in fines, all of which are partially leached by weathering.

The lower Lenora dump contains an unknown tonnage used as fill in the mine yard. Its grade is marginal at about 0.17% copper.

From data gathered in publications it is estimated that approximately 750,000 tons of unmined ore grading 3% copper is located in the known ore bodies, principally in the North Orebody. This tonnage will be substantially increased if a lower cut-off point is acceptable. A possible cut-off point might be .2% copper. This will be determined by the operation of the pilot plant.

The best information available indicates that the mine workings are all interconnected from the Lenora adits through the Tyee to the King Richard shaft. The Lenora No. 3 adit is the lowest level although the Tyee shaft is deeper than this level. All existing drainage is through the Lenora adit. The adits are accessible with a minimum amount of repair work.

There are a number of old buildings at the site two of which could be repaired and used.

Assays

A total of 17 samples was taken from the three dumps. Surface oxidation seemed extensive and trenches were dug to depths of from 5 to 10 feet with a small tractor and samples were taken over a cross-section of the trenches.

Visual examination of the trenched material showed considerable green carbonate and other oxidized forms of copper together with sulphides which appeared to be mostly chalcopyrite, zincblende and galena with some pyrite.

The ore in the upper portions of the dump was very fine, probably mostly minus one-inch material. At the toe of the Tyee dump the ore was mostly lumps of six inch or larger. This distribution of sizes is normal in tipped material and is ideal for percolation leaching.

The results of the samples are given below. Samples T1 to T5 are from the upper portion of the Lenora dump at about the elevation of Lenora No. 2 level. Samples T6 to T11 are from the lower Lenora dump at about the same level as Lenora No. 3 adit.

Location	Sample No.	Total Cu %	Acid Sol. Cu %	% of total Cu which is iron sulphide
Lenora	T-1	0.41	0.13	31.7
Upper	T-2	0.19	0.05	26.3
Dump	T-3	0.10	0.03	30.0
	T-4	0.18	0.03	16.6
	T-5	0.20	0.09	45.0
Average		0.22		29.7
Lenora	T-6	0.50	0.01	20.0
Lower	T-7	0.10	0.01	10.0
Dump	T-8	0.07	trace	—
	T-9	0.10	0.02	20.0
	T-10	0.27	0.06	23.2
	T-11	0.11	0.03	27.2
Average		0.17		20.1
Tyee	T-12	0.35	0.03	11.4
Upper	T-13	0.46	0.04	6.5
Dump	T-14	0.66	0.03	16.7
	T-15	0.45	0.11	20.0
Average		0.48		13.7
Barite	T-16	0.21	0.04	19.0
Dump	T-17	0.32	0.03	9.3
Average		0.27		14.2

Chemical Tests

An important test on any ore to be leached is that used to determine how much sulphuric acid is required to satisfy the immediately soluble components of the ore. These components include carbonate minerals such as calcite and malachite as well as oxides of the various metals. Some of the silicates are partially soluble and the acid will dissolve all the soluble components.

As the bacteria oxidize further metals more acid is required. However the extra acid is supplied by the sulphuric acid made from the sulphur associated with the sulphides. If pyrite is present extra acid is automatically available and this may provide part or all of the total acid required by the ore.

In most ores the initial heap will require acid to be added to initial bacterial leaching. The next heap will probably be satisfied at least in part from the excess acid of the first heap after first heap leaching has commenced.

Initial costs of acid are therefore generally higher than ordinary operating costs. Extra acid can be supplied by purchasing concentrated acid or by purchasing sulphur or pyrite concentrates and producing the acid directly from bacterial leaching.

Acid consumption tests are run directly on pulverized samples to obtain ultimate consumption. Other tests are run as leaching progresses to determine the net acid consumption required to start a new leach.

Typical powdered sample tests are as below:

Sample No.		Acid consumption
Lenora	T-1	46 lbs. H_2SO_4 per ton ore
Upper	T-2	46 " " " " "
Dump	T-3	43 " " " " "
	T-4	45 " " " " "
	T-5	38 " " " " "
Lenora		
Lower	T-6	89 " " " " "
Dump	T-7	100 " " " " "

The average total acid consumption for all the samples from the Upper Lenora, Lower Lenora and Tyee dumps is 43, 167 and 114 pounds of H_2SO_4 per tons of material respectively.

Acid consumption in the actual leaching tests are as follows:

Upper Lenora	23 pounds H_2SO_4 per ton
Tyee	35 pounds H_2SO_4 per ton

at these points the acid supplied by the bacteria is sufficient for further leaching purposes.

Net acid consumption is a direct cost but is offset by the value of metals dissolved in the process. In terms of copper this will be a little under one pound per ton from the dumps.

Heap Leaching

Preliminary work indicated the possibility of leaching the Tyee dump without moving it. This contains a large amount of available copper and other materials. At present only the copper will be considered but it is expected that the other metals will eventually be recovered. This heap contains about 9.6 pounds of copper per ton and, depending on the base geometry there are probably 100,000 tons in the heap. About 14% of the copper is acid soluble and this amount would be recoverable independent of bacterial leaching.

The heap is already in an ideally packed condition for direct leaching and is probably large enough to retain heat made in the leaching process during the winter months.

Visual examination of the surrounding rocks indicates that the Tyee dump is enclosed in a natural trough which dips at a pronounced angle down the hillside. However, the underlying rock when exposed in exploration pits

proved to be too porous to permit effective base construction. It seems probable that the dump is actually overlying part of the mine structure. If the dump is leached in place the solution flow will be at least partially directed into the old mine workings and would probably exit from the Lenora No. 3 adit.

For this reason an original plan to leach this heap first has been deferred. The heap can be leached later, possibly in conjunction with in situ leaching of new ore.

The Upper Lenora dump is spread over a considerable length of rock outcrop which includes a cross-section of the main ore body. This dump assayed 0.22% copper in five trench samples in the copper part of the dump. The coarse material may grade higher. At the above rate there are 4.4 pounds of copper per ton of ore and probably 18 pounds of zinc per ton. About 30 percent of this copper is acid soluble and the acid consumption on the Lenora is lower than on the Tyee dump. The ore is more quartz ore.

The dump is easily accessible for loading and placing on a prepared leach pad for which there is a suitable location nearby.

This dump has a great deal of fine material on top but appears much coarser underneath. If the fines persist it will be necessary to add and mix more coarse ore in the heap or else to screen out and remove some fines in order to permit free percolation of leach solutions. This can be determined during the preliminary stages of replacement of the heap.

The Pilot Plant

It is proposed to build a leach base near the site of the fresh water reservoir between the old mill and the Lenora No. 3 adit.

Selection of this site is based on the following considerations:

a. The site is a clay bed already cleared and easily graded.
b. All drainage from mine workings and from the proposed heap can be collected in a prepared storage trough at the lower end of the heap.
c. There is a convenient permanent site for a processing plant between the heap site and the old mill.
d. There is a short truck haul over a road already graded to transport the Lenora Upper dump to the heap site.
e. There is room for expansion of the heap.

It is proposed to use the Lenora Upper Dump for heap material. The dump is conveniently located and its removal will expose a good cross-section of the ore zone with sufficient change in elevation to permit open pit mining of a substantial amount of material if so desired.

The Lenora heap assayed 0.22% copper but probably runs higher as only the weathered surface fines were assayed.

Recovery rates on dump ore according to the laboratory tests should be in excess of 0.125 pounds of copper per ton of ore per week from the sulphides in the heap. The acid soluble copper will dissolve whenever acid is available and will be recovered rapidly.

An economic operation would require a production of about 500 pounds of copper per day. If this is produced from sulphides a heap of 28,000 tons would be required. However, when the acid soluble copper is included a smaller heap will pay its way. It is recommended that the initial heap be 15,000 to 20,000 tons.

An ore containing 4.4 pounds of copper per ton will require about 35 weeks to leach at the rate of .125 pounds per ton of ore per week. Actually the rate will slow down as the sulphide content decreases and we expect about 70% of the copper to be recovered in about 50 weeks.

It is clear that for an economic operation additional ore must be placed under leach at a rate sufficient to maintain any desired production.

A pilot plant serves several purposes. It is a unit capable of culturing large quantities of bacteria under control and such quantities will be required for a large scale leaching operation. The plant will provide necessary field data on which to base the design of a full scale operation. It will also provide solution and processing equipment for a possible in situ leach test.

In Situ Leaching

Removal of the Upper Lenora dump will expose a steep rock face which should be a good cross-section of the copper-bearing formation. This region could be the location of an attempt to leach the ore in place.

There is obviously caved ground running back in from the hillside and there are two adits giving direct access to the region.

If the upper Lenora adit is re-opened sufficiently to permit access to the first caved stope it may be possible to circulate solution

through the existing workings to recover considerable copper. The solution may drain out through the Lower Lenora adit at which it can be directed to a sump for further circulating.

On the other hand the area might be further fractured by blasting or by hydraulic fracture and a leach circuit could be established.

The rock mechanics analysis, a copy of which is attached herewith, indicates several sets of planes of preferred shear. These are the planes on which fracturing will occur most readily. The azimuths and the dips of these planes indicate a block type fracture may be expected although the traces of the planes in the horizontal section are in two groups, approximately 135° and 60°.

The direction of thrust is not determinable in the lab but the line of thrust is about 119° and dipping slightly to the north west. The arrow head for this vector must be determined by field tests.

The inherent strain energies on all planes were substantial although not obviously high. This means that fracturing should be accomplished at relatively low pressures.

There is a remarkable uniformity in the relaxation rates on the three directions of each sample indicating no rock bursting characteristics in the ground sampled.

The preliminary rock mechanics tests should be supplemented by some field work before planning a fracturing programme.

Capital Requirement

The costs listed here cover the following operations:

1. Completion of the existing reservoirs for water supply.
2. Construction of a heap base.
3. Construction of a recovery plant.
4. Construction of a heap.
5. Incidental pumps, and circulating system for the heap.
6. Operating costs for the heap as a pilot plant.

No costs are included for in situ work at this time.

The figures given below are not quotations but are based on experience elsewhere. They are subject to change depending on final specifications and quotations for the work and materials. An iron powder precipitator has been estimated because it is unlikely that the chemical precipitator currently under test will be available at the commencement of the pilot operation.

Water reservoir completion	$ 500
Access roads	500
Heap base, plywood and polyethelene construction on clay base	5,000
Move 20,000 tons dump ore at .30 cents per ton	6,000
Building for the processing plant	6,000
Acid storage	3,500
Pumps	3,000
Piping system	4,000
Precipitator (iron powder)	2,000
Solution storage tanks	3,000
Culture tanks, etc	2,000
Laboratory equipment	2,500
Filters, dryers, handling equipment etc.	21,000
Utility vehicle	4,000
Contingencies	5,000
Labour to production	10,000
Engineering to production	10,000
Total	$89,000

Operating Costs

Operating costs will include the daily requirement of labour, power, supplies, overhead and general expenses of head office, taxes and write-offs.

Monthly operating costs will be about as follows:

Labour - 4 men @ $450/month	$ 1,800
Resident engineer	750
Power (electricity, about 50 HP)	550
Iron powder consumption (500 lbs. per day at 7c)	1,050
Acid (initial costs high pending regeneration)	1,000
Maintenance	500
General engineering	1,000
Overhead at 20% of the above	1,320
Contingencies	1,000
	$ 8,970

This operating cost allows for no additional expansion of the heap. Once the mine is examined and the circulation characteristics are established it should be possible to proceed to leach the Tyee dump and ore in situ. Under these conditions expansion will be simply a matter of increasing capacity.

To achieve a production rate of 5,000 pounds of copper per day it will be necessary to have a body of ore similar to the Tyee dump under leach. This will require ore to be added to the total under leach at the rate of about 200,000 tons per year. A much higher rate would be preferable.

At present the cost of this is unknown. If in situ leaching can be accomplished it should cost not more than 20c per ton. If mining is required the cost could be as high as $3.00 per ton but might be as cheap as 50c per ton. This cost could vary from $3,500 to $35,000 per month.

The operating cost of the plant will commence when the plant goes on line and because of the size of the plant it will be barely economic at the start. However acid costs will decrease rapidly until more ore is placed under leach.

As ore is added the operating costs will increase but the capital costs will not increase much. The plant as designed can handle more than the production rate to be initially established. It is based on a 5,000 pound per day rate except for the precipitator which can be readily increased.

A cash flow chart is included here to show the preparation and commencement of the pilot plant operation and the required expansion to a larger unit based on the use of the Tyee dump ore, in situ leaching, or mining.

In the chart no capital expenditure is shown except that total costs of expansion have been set up at $10,000 per month starting in the third month.

Also it is assumed that excess acid from the first pilot operation will be available for expansion. It is also assumed that chemical precipitation will be available in the fourth month.

Conclusions

The operation of a pilot leach heap of about 20,000 tons will permit a marginal income operation during the first three months of its operation.

Expansion by addition of ore under leach either in situ, in dumps, or by mining and heaping will produce an economically sound operation.

The pilot heap should be commenced according to the design data attached herewith for the base.

Design of the plant and other facilities should be proceeded with at once.

The work of exposing the outcrop at Lenora is equivalent to a stripping operation and all such work should be co-operative between the operation of exploration and leaching.

Respectfully submitted,

C. L. Emery, Ph.D., P. Eng.

Rock Mechanics Analysis, Mount Sicker Samples

Outlined below is a summary of the findings of the analysis of the oriented hand specimens taken from the Mount Sicker property.

Table 1: Directions of Principal Strain— Horizontal Faces

Lenora 1	105°	15°
Lenora 2	135°	45°
Tyee 1	105°	15°
Tyee 2	118°	28°
Averages	115°	25°

Table 2: Preferred Shear Planes

Sample number		Azimuth	Dip
Lenora 1		105°	vertical
Lenora 2	Primary	30°	80° S.E.
	Secondary	125°	85° S.W.
Tyee 1	Primary	165°	70° N.E.
	Secondary	95°	60° S.W.
Tyee 2		66°	10° N.W.

Table 3: Time Dependant Strain Magnitudes

Sample No.	N-S Face		E-W Face		Horizontal Face	
Lenora 1	-732	-772	-614	-614	-755	-738
Lenora 2	-694	-684	-630	-621	-876	-825
Tyee 1	-880	-910	-671	-658	-819	-880
Tyee 2	-718	-725	-866	-822	-525	-536
	-756	-770	-695	-678	-743	-744

Table 4: Thrust Vectors

Sample number	Azimuth	Dip
Lenora 1	111°	28° N.W.
Lenora 2	125°	25° S.E.
Tyee 1	105°	6° S.E.
Tyee 2	135°	24° N.W.
Averages	119°	5° N.W.

Discussion of Results

1. Directions of Principal Strain

The average major principal strain direction on the horizontal faces of the oriented hand specimens has an azimuth of 115°. The direction of the minor principal strain averages 25°. All four samples taken showed principal strain direction quite close to these average values, which indicates that field force in the area examined shows uniform directional characteristics.

The attached map shows the relationship of the properties of the hand samples to the geological structure, and indicates that the major principal strain directions trend generally parallel to the strike of the vein system at Sicker, which is 95° azimuth.

2. Preferred Shear Planes

Samples Lenora (1), (2) and Tyee (1) contain sets of preferred shears that trend parallel to strike of the vein system and all planes are steeply dipping.

Sample Tyee 2 is the only specimen to exhibit sub-horizontal shear planes, namely planes striking 66° and dipping 10° to the N.W. Further, samples Tyee 1 and Lenora 2 contain shears that trend generally at right angles to the vein structure, between the two ore zones indicated, and it is felt these would play an important part in any studies for hydraulic fracturing in situ.

3. Time Dependant Strain Magnitudes

All faces examined exhibited moderate magnitudes of principal strain, and similar amounts of strain were recorded for each set of faces. The higest principal strains were found on the N-S face of sample Tyee 1, while the lowest time dependant strain was measured on the horizontal face of sample Tyee 2.

4. Thrust Vectors

The analysis of the thrust vectors of each sample confirms the information developed from the principal strain directions indicated on the horizontal sample faces. Thus the Thrust force field of the area has an average azimuth of 119°, with a shallow dip.

General

At the time the samples were taken, no geological map of the Sicker area was available, but it is felt that the analysis presented will indicate the prominent features of the vein system satisfactorily for inclusion in the studies of feasibility.

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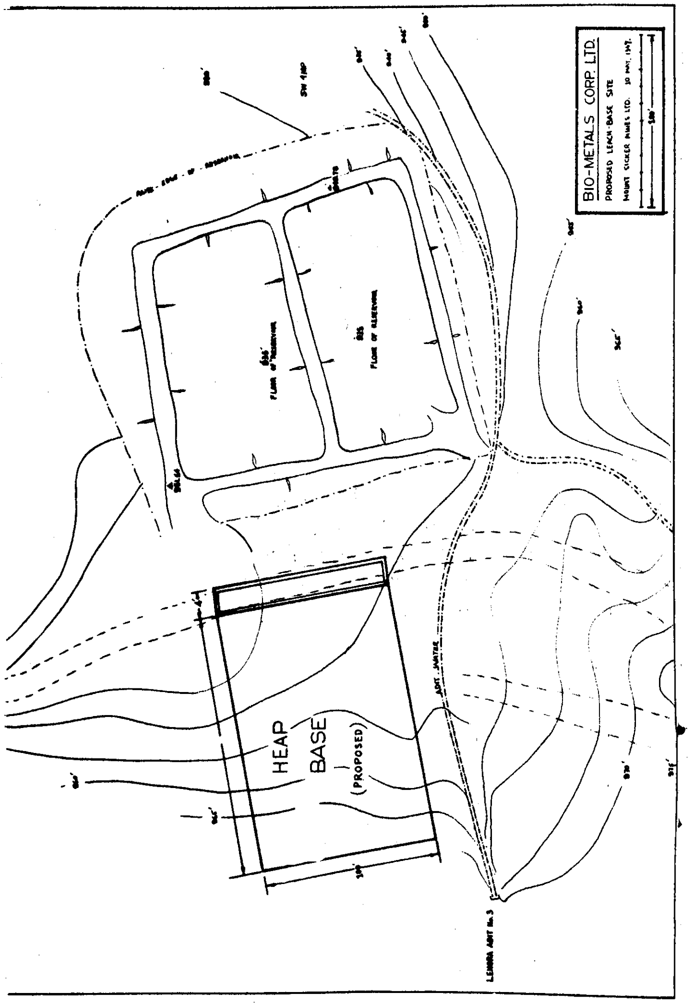
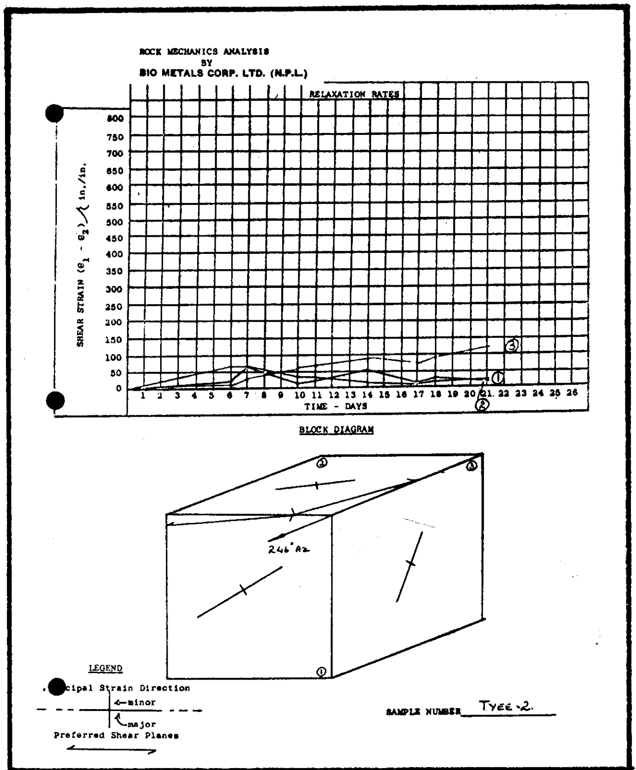
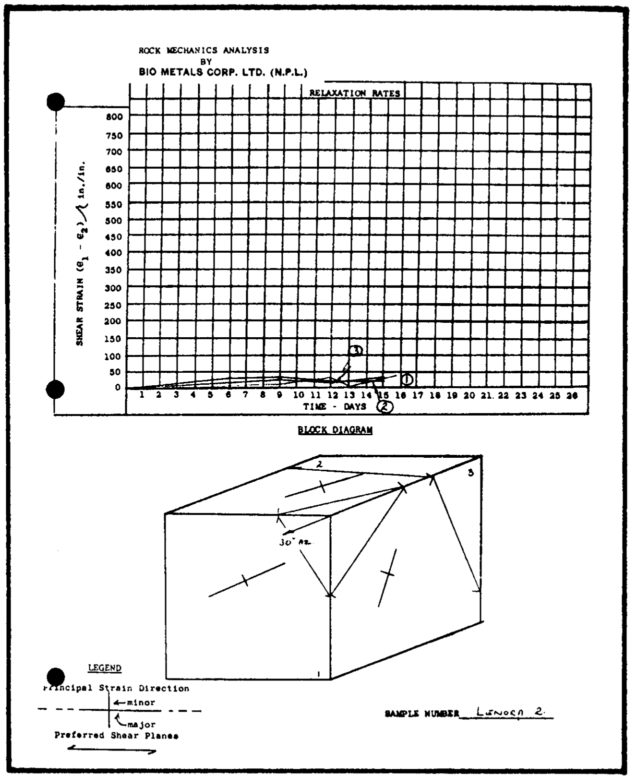


Operating Cash Flow

	Month from start					
	1	2	3	4	5	6
Labour, power, supplies	$ 4,850	$ 4,850	$ 4,850	$ 4,850	$ 6,300	$ 8,300
Engineering	1,750	1,750	1,750	1,750	1,750	1,750
Marketing	—	300	300	1,000	1,000	1,500
Overhead	1,320	1,320	1,320	1,320	1,500	1,500
Expansion	—	—	10,000	10,000	10,000	10,000
Contingencies	1,000	1,000	1,000	1,000	1,000	1,000
Total cost	8,970	9,270	19,270	19,920	21,550	24,050
Cumulative costs	8,970	18,240	37,510	56,780	78,330	102,380
Tons copper produced	—	7.5	7.5	30	45	60
Value @ 45c/lb.	—	6,750	6,750	27,000	40,500	54,000
Cumulative value	—	6,750	13,500	40,500	81,000	135,000
Cumulative gain or loss	—	11,490 (loss)	24,010 (loss)	16,280 (loss)	2,670 (gain)	32,620 (gain)


ROCK MECHANICS ANALYSIS
BY
BIO METALS CORP. LTD. (N.P.L.)
RELAXATION RATES
SHEAR STRAIN (e1 - e2) μ in./in.
TIME - DAYS
BLOCK DIAGRAM
246° Az
LEGEND
Principal Strain Direction
minor
major
Preferred Shear Planes
SAMPLE NUMBER TYEE 2.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BLOCK DIAGRAM
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LEGEND
Principal Strain Direction
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major
Preferred Shear Planes
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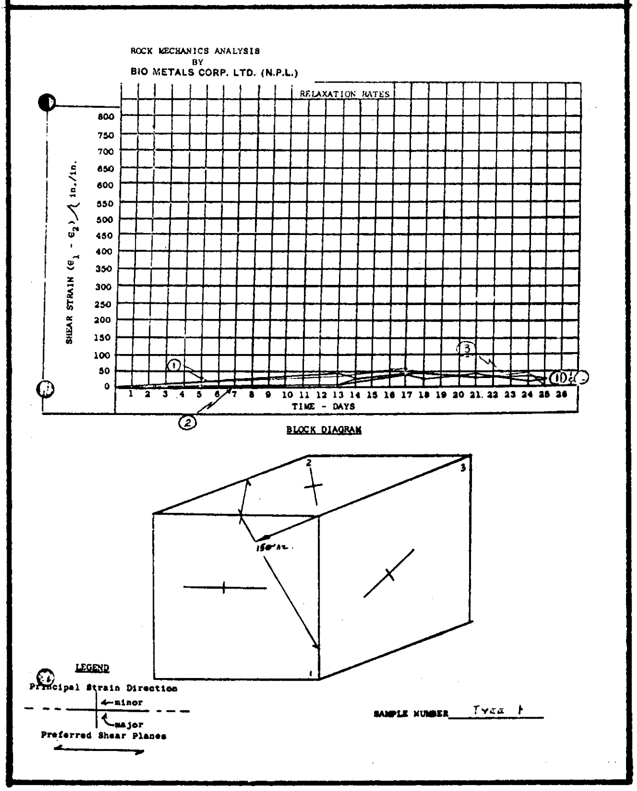
ROCK MECHANICS ANALYSIS
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BIO METALS CORP. LTD. (N.P.L.)
RELAXATION RATES
SHEAR STRAIN (e1 - e2) μ in./in.
TIME - DAYS
BLOCK DIAGRAM
150° Az
LEGEND
Principal Strain Direction
minor
major
Preferred Shear Planes
SAMPLE NUMBER Tyee 1

MOUNT SICKER MINES LTD NPL

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