92F/106 Texada Prescott.

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Preliminary Report on

TEXADA MINE

(Nanaimo Mining Division)

by

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A. Sutherland Brown, Ph.D., P.Eng.

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# PRELIMINARY REPORT OF TEXADA MINE

## INTRODUCTION:

The mine management of Texada Mines Limited at a meeting with Mr. J. E. McMynn, Deputy Minister of the Department of Mines and Petroleum Resources suggested that an effect of the Mineral Land Tax Act would be to reduce the life of the mine from 3 years to 20 months. The writers were asked to visit the mine to assess the situation in regard to mine life and the impact of changing costs. To make a preliminary appraisal, the writers visited the mine on May 2 and 3, had extensive discussions with the staff, examined mine records and geological plans and sections, and spent the morning of May 3 underground visiting stope areas with which we were not very familiar.

## LOCATION:

The Texada Mines Limited's iron and copper mine is situated on the west coast of Texada Island, north of Gillies Bay (Latitude 49 degrees 43 minutes; Longitude 124 degrees 34 minutes). All of the orebodies and mine working are within a mile of the loading dock on the Strait of Georgia.

## HISTORY:

The Texada Mine probably has the longest history of any lode mine in British Columbia, and the longest record of shipment of any iron mine. The property was acquired by the Puget Sound Iron Company in the 1870's and between 1885 to 1903 and in 1908 shipped 28,898 tons of magnetite ore to the company blast furnace at Irondale, Washington. Texada Mines Limited was formed as a private company in May 29, 1951, purchased the Puget Sound holdings on Texada, and proceeded to develop and explore the reserves then thought to be of the order of one million tons.

Production of lumpy iron began in the spring of 1952 and four open pits were eventually developed - 92F106 Prescott, 92F107 Paxton, 92F259 Lake, and 92F258 Yellow Kid. The original cobbing plant has been repeatedly elaborated and modified so that it is now a complex mill producing iron and copper concentrates. Exploration showed mineralization extended below levels mineable by open pit methods and the decision was taken to obtain a long term contract and develop an underground mine. Production from underground began in 1964 and the open pit mining was phased out in 1966. Soon after production began from underground, it was realized that the relatively new trackless mining offered greater efficiency because of the geological nature of the orebodies. Production by trackless methods was begun in 1966 at the relatively shallow dipping Lake orebody as a trial. Thereafter, the whole mine was gradually converted to trackless mining. To the end of 1973, the mine has produced:

| | | |
|---|---|---|
| 1885 - 1919 | 28,898 tons | Lumpy magnetite ore |
| 1952 - 1956 | 1,239,825 tons | Lumpy magnetite ore |
| 1956 - 1973 | 7,785,715 tons | Iron concentrates |
| | | Copper concentrates |

PROPERTY:

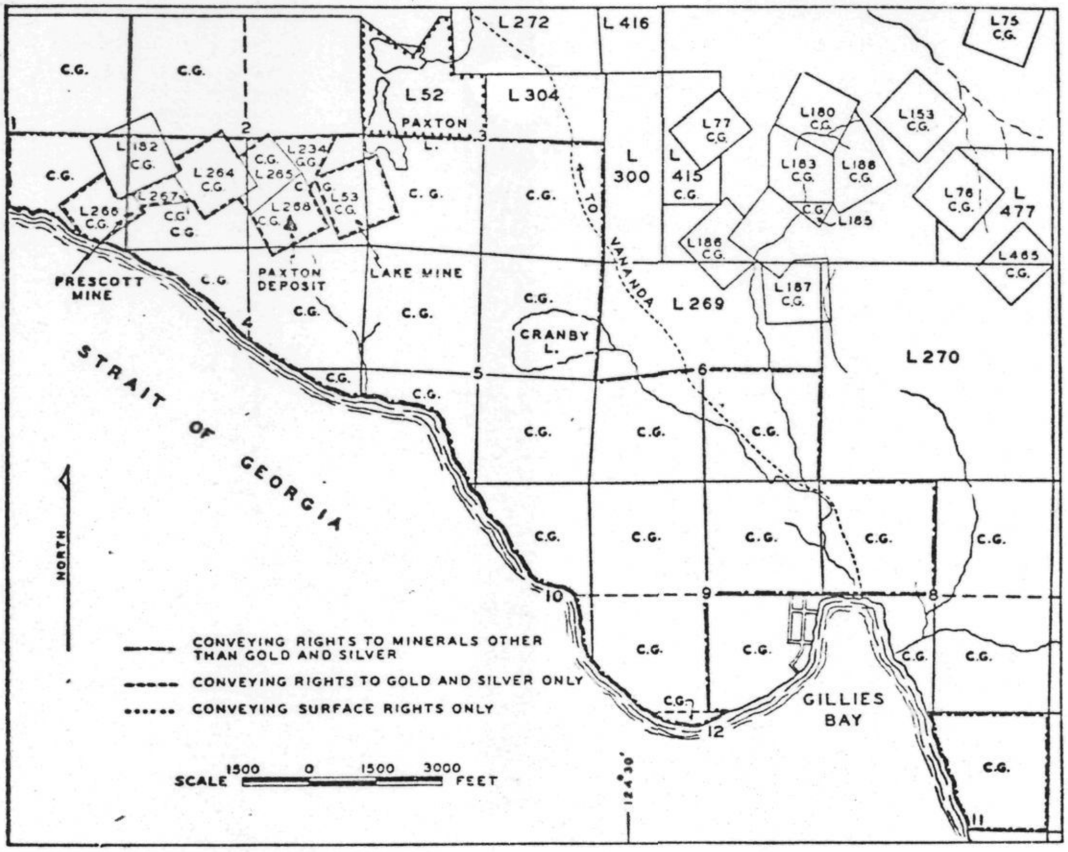


**Figure 1** Holdings of Texada Mines Limited.

Texada Mines Limited purchased the holdings of Puget Sound Iron Company on Texada Island and these included real estate with rights to minerals other than gold and silver, and crown granted mining claims conveying rights to gold and silver and/or to all metals. Figure 1 shows these holdings and the following list shows the status of land in the vicinity of the mine according to Mining Titles Division.

GEOLOGY:

The geology of the Texada deposit has been studied by many geologists of which the first of any importance was McConnell in 1914. It was not until the underground exposures were studied that a really clear conception of the geology was possible (Sutherland Brown, 1964). Appendix 1 includes a recent summary of the geology from the International Geological Congress Guidebook (Sutherland Brown, 1972). The salient features only will be outlined here, and on Figures 2 to 6, regional, mine surfaces, underground, and sections.

The Texada Mine is a skarn deposit developed at the interface of a thick basalt pile with an overlying limestone near the contact of an intrusive quartz diorite. It is characteristic of such deposits in the Insular Belt of British Columbia. The orebodies consist of massive to fairly massive magnetite with variable amounts of iron and copper sulphides and lime-silicate (skarn) minerals generally enveloped in a larger body of skarn. The ore and skarn minerals have replaced the pre-existing rocks by substitution in the solid state. These replaced rocks include, in order of importance, limestone, basalt, and minor quartz diorite intrusive. The ratio of skarn to ore is highest in the basalt and lowest in the uppermost fingers of ore in limestone. The distribution of skarn and ore is controlled by the interplay of the following factors; basalt-limestone contact, proximity to the intrusive body, and particularly to overhanging contacts, pre-ore faults, and breccia pipes. The mine is at the southern terminus of a broad syncline of limestone overlying the basalt and this interface meets the intrusive at about 300 feet below the surface. The limestone volcanic contact dips gently northward, but is involved in a few sharp folds and jostled by many small pre-ore faults. The warped and broken surface of this contact in conjunction with the intrusive is a most important locus of ore. At the very least a thin

# STATUS

## TEXADA ISLAND LAND DISTRICT

M/C

| | |
|---|---|
| Lot 182 | - conveyed all minerals, precious and base |
| Lot 53 | - conveyed gold and silver ore |
| Lot 234 | - " " " " " |
| Lot 264 | - " " " " " |
| Lot 265 | - " " " " " |
| Lot 266 | - " " " " " |
| Lot 267 | - " " " " " |
| Lot 268 | - " " " " " |

SURFACE:

Section 1 - S.W. 1/4 - reserved gold and silver 1/3 interest reverted - second crown grant reserved all minerals precious and base including coal, petroleum, and natural gas.
S.E. 1/4 - reserved gold and silver
N.W. 1/4 - reverted 1919
N.E. 1/4 - reverted 1919

Section 2 - N.W. 1/4 - reserved gold and silver reverted 1919
N.E. 1/4 - crown
S 1/2 - reserved gold and silver

Section 3 - N 1/2 - crown
S 1/2 - reserved gold and silver

Section 4 - N.E. 1/4 - crown
S.E. 1/4 - reserved all minerals precious and base including coal and petroleum.
S.W. 1/4 - crown
N.W. 1/4 - reserved gold and silver

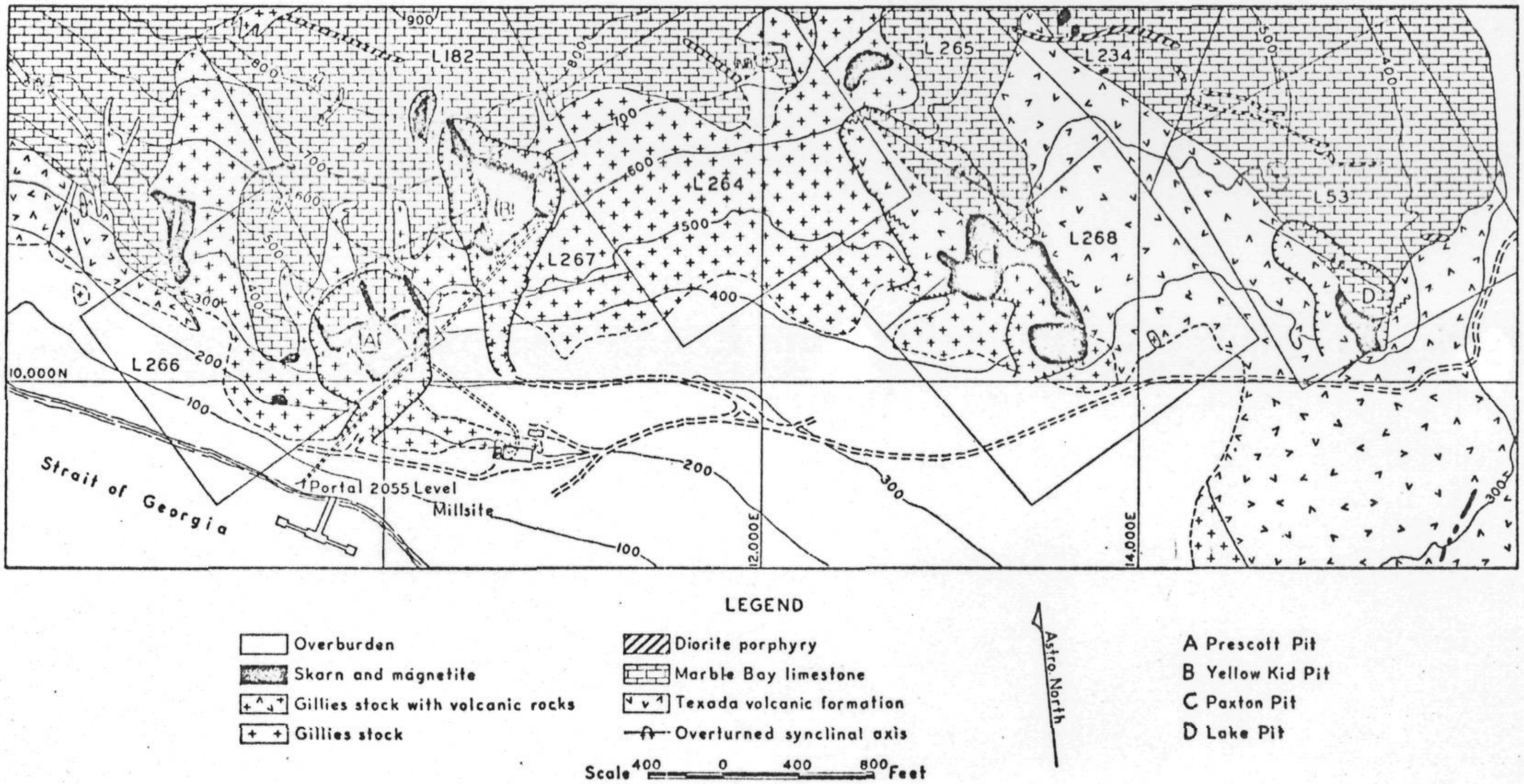


Figure 3. Texada Mines Ltd. Geology of mine area.

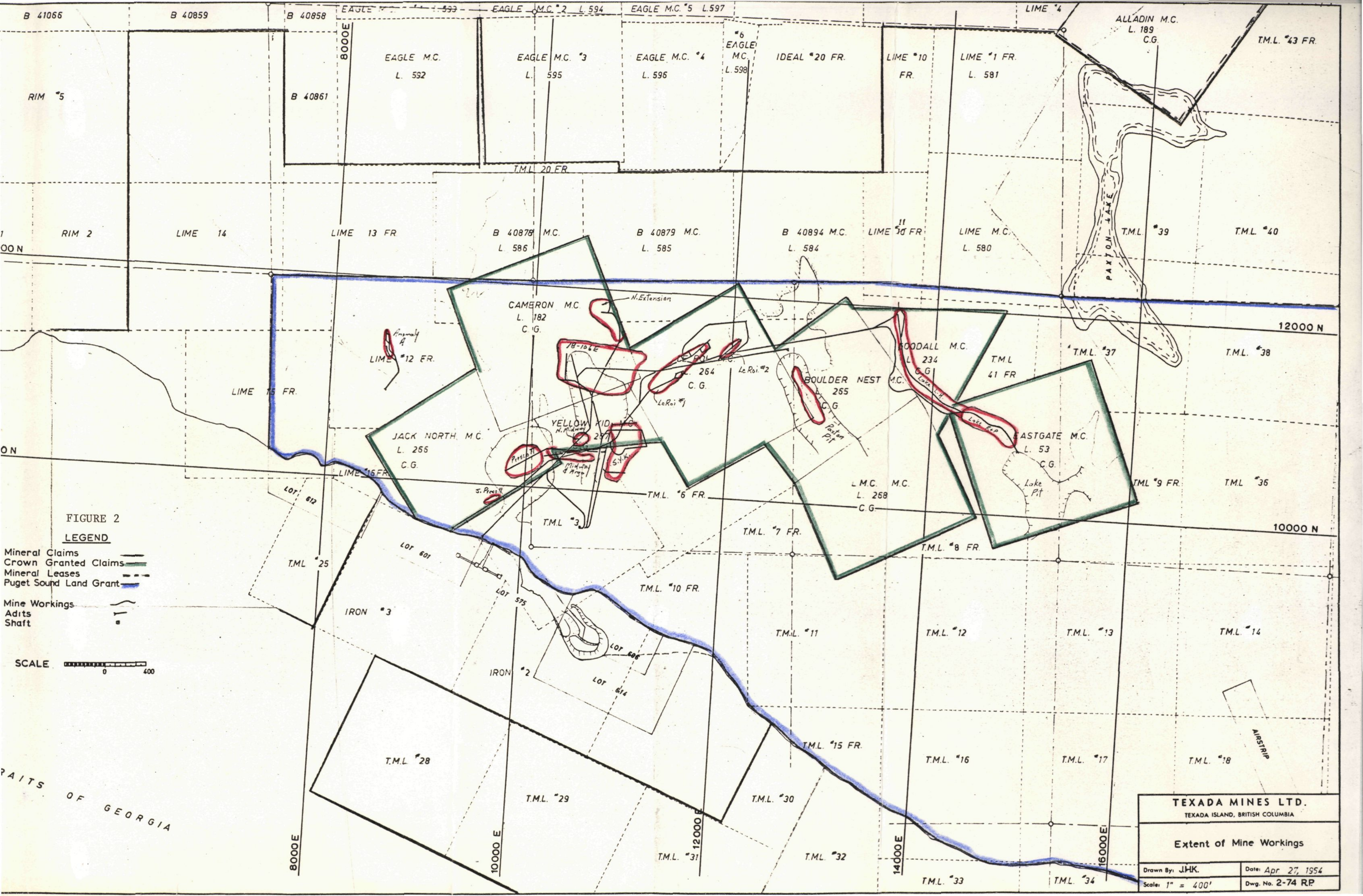


FIGURE 2

LEGEND

Mineral Claims
Crown Granted Claims
Mineral Leases
Puget Sound Land Grant
Mine Workings
Adits
Shaft

SCALE 0 400

**TEXADA MINES LTD.**
TEXADA ISLAND, BRITISH COLUMBIA

Extent of Mine Workings

| | |
|---|---|
| Drawn By: J.H.K. | Date: Apr 27, 1954 |
| Scale: 1" = 400' | Dwg. No. 2-74 R.P. |

layer of ore and skarn occurs at the contact and as distance increases from the intrusive, the thickness generally decreases. At sharply overturned keels of minor folds, thick hinges of iron are common (Paxton and Lake orebodies). In the western part of this property (Prescott, Midway, and Yellow Kid), a system of continuously connected bodies exist near the intrusive contact that bear a resemblance to an espalier tree. These bodies extend upward from a narrow trunk at the 1,455 level (see Figure 4), branch broadly out at the limestone contact and extend upward through the limestone as a diffusing group of lesser branches. In general, magnetite ore is zoned innermost along the conduits in the system, with skarn enveloping it and outward a zone of minor alteration. The result of these various controls is that magnetite ore occurs as a series of pods along the intersection of two warped surfaces (limestone/volcanic contact and intrusive/stratified rock contact), with other bodies related to ramifying pre-ore faults and breccia pipes.

Exploration of the property has been conducted in a systematic and thorough manner over the latter history of the property. The distribution of the favourable sites, as they are now known, are fairly predictable and have been explored. To aid exploratory drilling, a new ground magnetometer survey was recently carried out and any anomalies have been drilled as have extensions of known ore. In the latter case, ore found is invariably costly to recover because it is further down dip, deeper, and further from haulage and ventilation. In addition, ore as it feathers out is normally thinner and more iron sulphide rich, so generally less rewarding.

RESERVES:

The reserves established as of 31 March 1974 are as follows:

| | DLT | Fe | Cu |
|---|---|---|---|
| Total mineable | 2,876,000 | 37.36 | 0.39 |
| Reserve dumps | 61,000 | | |
| Surge | 21,000 | | |
| Total iron | 2,958,000 | | |
| Total copper | 844,000 | 27.43 | 0.90 |

This tonnage at present production should last 3 years.

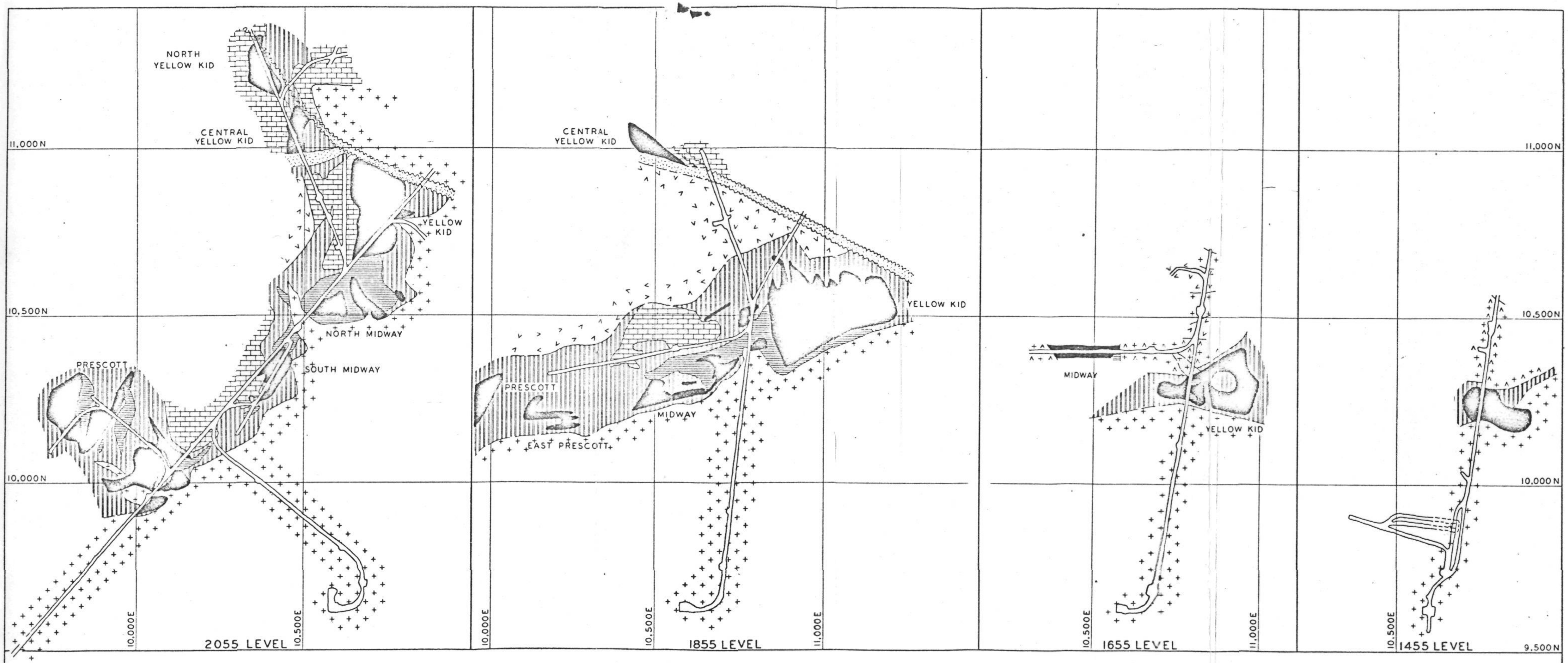


Figure 4
SIMPLIFIED GEOLOGY OF MAIN LEVELS
TEXADA MINES LTD.

Scale 0 — 500 Feet

- Post-ore porphyry
- Magnetite and sulphide orebodies
- Skarn
- Quartz diorite and volcanic rocks
- Gillies stock, quartz diorite-granodiorite
- Diorite porphyry
- Marble Bay limestone
- Texada volcanic formation

Appendix 2 includes a complete statement of reserves as of 31 March 1974.

The company has estimated that certain of the higher cost and/or lower grade stopes of this reserve will have to be withdrawn as a result of the Mineral Land Tax Act and that others, because of scheduling and access, will not be recoverable prior to shut down. These are shown in Appendix 3.

This is a very complicated subject in which there are many unknowns, a condition in itself that make engineering calculations invalid. Factors that are difficult to assess are:- 1) true impact of Mineral Land Tax; 2) whether or not this will be regarded as deductible from federal taxes; 3) effect and continuity of elevated prices for copper; and 4) availability and cost of labour.

A geological factor of considerable importance is that skarn ores have little or no gradient from ore to waste, so that stopes are located in well-mineralized rock but the costs of mining, exploration, development, and haulage for a particularly stope signify whether or not it is ore. Although the mine has been very cost-conscious and has exercised considerable efficiency in mining and milling, it has not kept costs on every individual stope so that an appraisal of high cost stopes is not possible. Some attempt will be made to evaluate these factors later.

MINING:

Mining by Texada Mines Limited began in 1952, using open pit mining methods. Underground mining commenced in 1965 and by 1966 all ore was obtained by underground mining methods. Initially, underground mining was done by longhole open stoping methods, using track haulage. During 1966, conversion to trackless mining was initiated due to the flexibility of this method in dealing with orebodies of differing shapes and scattered distribution. Longhole stoping was continued although room-and-pillar mining methods are being used for two rather flat-lying orebodies. Due to the geological nature of the ore, certain orebodies and parts of orebodies are rich in iron and copper sulphides, while other areas are magnetite rich and sulphide poor. These two ore types are mined and processed separately. Prior to delivery to the mill ore, other than production from the Lake room-and-pillar stope, is crushed underground to minus four and one-half inches. A paper presented by Mr. A. M. Walker, Vice-President of Texada Mines Limited (copy attached) describes underground mining of the Texada iron mine (mining costs presented in this paper refer to costs prevailing during 1972).

MILLING:

Concentrating of the ore at Texada Mine is done in "batches" related to the selective mining process. Two distinct methods are used, depending on whether the ore being treated contains mostly iron with a relatively small percentage of copper or whether it contains an appreciable quantity of copper as well as iron.

When the mill is treating "iron ore", the mill capacity is between 4,200 and 4,400 tons per day. In this process, "cobbing" of the ore is done prior to grinding. Magnetic cobbing removes approximately 1,000 tons of non-magnetic material. In this process about 13 per cent of iron is lost (largely non-magnetic iron) and about 15 per cent of the copper (presumably most copper that is not closely associated with magnetite). An analysis of grab samples taken by the writers from recent coarse reject material is shown in the appendix. Total iron is 17 per cent, soluble iron mostly sulphide 11.8 per cent, and copper 0.2 per cent. Recovery of the remaining iron, by wet magnetic separation after grinding, is reported to be about 95 per cent. Recovery of the copper, by flotation, is only about 65 per cent, due mainly to the rather coarse grind.

When the mill is treating copper ore, the mill capacity is about 2,000 tons per day. In this process no cobbing is done and the mill circuit is adjusted to provide finer grinding. This results in a copper recovery of about 80 per cent. It is understood that the iron recovery remains about the same (95 per cent) but in this process a greater proportion of fine iron concentrate is produced. Fine iron concentrate is not as desirable as coarse iron concentrate (there are contract limitations in this regard) and it is necessary to blend the fine iron concentrate with the coarser iron concentrate produced when the mill is treating iron ore. It is unfortunate that the ore grade hoisted is unknown except by application of calculation of the reserves from a known source, allowing for a standard 10 per cent dilution. Likewise the coarse waste is not sampled regularly so control is not complete.

It has been indicated that during the last six month period, the mill treated iron ore during 87 days and copper ore during 56 days.

The accompanying paper, presented by Mr. L.D.H. Smillie, Mill Superintendent, describes the milling process at Texada Mine in considerable detail.

EVALUATION:

The mine management has indicated that a number of orebodies could not be mined profitably under proposed legislation. This would reduce the life of the mine and in so doing, would also mean that other orebodies could then not be mined because of necessary scheduling. Two of the orebodies considered as possibly uneconomic are the Lake room-and-pillar stope, and the Lake 2070 stope. We have taken the latter two stopes to analyse because they represent a significant part of the ore withdrawn. Certain assumptions have been made:- gold is assumed to have a constant value of $170.00 per ounce and silver $4.50 per ounce, but copper was calculated at three prices - $0.80, $1.00, and $1.20. The Mineral Land Tax would apply to the Texada Mine but, for simplicity, we have used the provisions of the Mineral Royalties Act which is supposed to have a similar effect. The value used for 120 per cent of the gross value of the designated value of copper was $0.70, as recommended by J. S. Poyen. No calculation of royalties on iron or precious metals were carried out because of uncertainty of designated values. It will be noted that metal values are calculated in U.S. dollars while operating expenses are in Canadian dollars.

Calculations for the value of copper and iron concentrate and the contracts involved form Appendix 6 and 7.

The Lake room-and-pillar and 2070 stope are adjacent orebodies at the eastern part of the mine. The Lake room-and-pillar is a shallow north dipping orebody that continues from the Lake open pit at the thickened keel of an overturned syncline (see Figure 5 ). Down dip the 2070 orebody is a continuation but its thickness is such that it can be mined by longhole methods. The Lake room-and-pillar ore is hauled upslope out of the portal of the Lake pit. The 2070 ore is hauled underground and because of incomplete development must be handled three times. Cost studies show that this is cheaper than haulage upslope out of the Lake portal.

2070 Lake Orebody:

Reserves on March 31, 1974 -

397,000 long tons at 44.14 per cent iron and 0.15 per cent copper

$(397{,}000 \times \frac{2240}{2204.6} = 403{,}374$ metric tons)

An estimated 10 per cent of the ore is not recoverable. However, dilution is estimated at 10 per cent. Thus, the reserve tonnage figure should be mined but the grade will be reduced by 10 per cent.

| | | | |
|---|---|---|---|
| Iron | 44.14 | Copper | 0.15 |
| | 4.41 | | 0.01 |
| | 39.73 per cent iron | | 0.14 per cent copper |

Value of iron in one metric ton of ore -

Recovery from cobbing 87 per cent (13 per cent is non-magnetic)

Recovery of magnetic iron 95 per cent

Overall iron recovery is .95 x .87 = 82.7 per cent

Iron value of one metric ton of ore is:

$.827 \times \frac{39.73}{64.00} \times \$10.17 = \$5.22$ U.S.

Value of copper in one metric ton -

Recovery from cobbing 85 per cent (15 per of copper is lost in cobbing)

Recovery from flotation 65 per cent

Overall copper recovery is .65 x .85 = 55.3 per cent (when milling combined iron and copper ore)

While the value of iron in the orebody is fixed by the terms of the contract the value of copper will vary with the price on the London Metal Exchange. This is shown by calculating the copper content values based on prices of \$0.80, \$1.00, and \$1.20 U.S. per pound of copper.

Copper at \$0.80 per pound U.S. -

Value of copper in one metric ton of ore -

$.553 \times \frac{0.14}{22.31} \times 243.61 = \$0.84$ U.S.

Copper at \$1.00 per pound U.S. -

Value of copper in one metric ton of ore -

$$.553 \times \frac{0.14}{22.31} \times 309.14 = \$1.07 \text{ U.S.}$$

Copper at $1.20 per pound U.S. -

Value of copper in one metric ton of ore -

$$.553 \times \frac{0.14}{22.31} \times 374.60 = \$1.30 \text{ U.S.}$$

Payment is received for both gold and silver, which is recovered in the copper concentrate. By using average values supplied for one short ton of copper concentrate, the following estimated value for gold and silver in one metric ton of ore has been obtained.

| | | |
|---|---|---|
| Gold | $0.21 | U.S. (gold at $170.00 U.S. per ounce) |
| Silver | $0.17 | U.S. (silver at $4.50 U.S. per ounce) |
| | $0.38 | U.S. |

From the foregoing the following values are obtained for one metric ton of ore based on prices of $0.80, $1.00, and $1.20 U.S. per pound of copper.

Copper at $0.80 per pound U.S. -

Value of one metric ton of ore -

$5.22 + $0.84 + $0.38 = $6.44 U.S.

Estimated Mineral Land Tax -

1974 - $0.016

1975 - $0.031 $6.44 - .03 = $6.41 U.S.

Copper at $1.00 per pound U.S. -

Value of one metric ton of ore -

$5.22 + $1.07 + $0.38 = $6.67 U.S.

Estimated Mineral Land Tax -

1974 - $0.37

1975 - $0.44 $6.67 - .44 = $6.23 U.S.

Copper at $1.20 per pound U.S. -

Value of one metric ton of ore -

$5.22 + $1.30 + $0.38 = $6.90 U.S.

Estimated Mineral Land Tax -

1974 - $0.70

1975 - $0.77 $6.90 - .77 = $6.13 U.S.

It has been reported that the mining cost for the 2070 Lake orebody is $2.94 Canadian per long ton. (This figure is comprised of $1.85 direct cost and $1.09 indirect cost - average mining cost through mine, including development and production, is $2.84 per long ton). The total costs are as follows:

| | |
|---|---|
| Mining | $2.94 per long ton |
| Milling | $1.05 per long ton |
| General Administration | $0.60 per long ton |
| Miscellaneous | $0.37 per long ton |
| Depreciation | $0.77 per long ton |
| Total | $5.73 per long ton |

Total cost per metric ton -

$$\frac{2204.6}{2240} \times 5.73 = \$5.65 \text{ Canadian}$$

A new wage contract is due in June, 1974. If overall costs are inflated 10 per cent over the next year, the total costs would increase by about 50 cents per metric ton (assuming that depreciation should not rise with inflation). However, the copper contract is in the process of being renegotiated (contract to September 30, 1974). Thus, it is assumed that current negotiations should obtain a greater return from copper values.

Lake Room-and-Pillar Stope:

Reserves on March 31, 1974 -

302,000 long tons at 46.95 per cent iron and 0.15 per cent copper

$$(302{,}000 \times \frac{2240}{2204.6} = 306{,}849 \text{ metric tons})$$

The estimated recovery from this "room-and-pillar" stope is 65 per cent. However, as there will be an estimated 10 per cent dilution, the overall tonnage that can be mined is -

$$.75 \times 306{,}849 = 230{,}137 \text{ metric tons}$$

At 10 per cent dilution, the grade will be reduced to 42.26 per cent iron and 0.14 per cent copper.

The iron content of this ore has a recoverable value of $5.55 U.S. per metric ton while the values of the copper, gold, and silver content will be the same per ton as the ore in the 2070 Lake orebody. Thus, the following values are obtained for one metric ton of ore based on prices of \$0.80, \$1.00, and \$1.20 per pound of copper.

Copper at \$0.80 per pound U.S. -

Value of one metric ton of ore -

\$5.55 + \$0.84 + \$0.38 = \$6.77 U.S.

Estimated Mineral Land Tax -

1974 - \$0.016

1975 - \$0.031 \$6.77 - .03 = \$6.74 U.S.

Copper at \$1.00 per pound U.S. -

Value of one metric ton of ore -

\$5.55 + \$1.07 + \$0.38 = \$7.00 U.S.

Estimated Mineral Land Tax -

1974 - \$0.37

1975 - \$0.44 \$7.00 - .44 = \$6.56 U.S.

Copper at \$1.20 per pound U.S. -

Value of one metric ton of ore -

\$5.55 + \$1.30 + \$0.38 = \$7.23 U.S.

Estimated Mineral Land Tax -

1974 - \$0.70

1975 - \$0.77 \$7.23 - .77 = \$6.46 U.S.

It has been reported that the mining costs for the Lake room-and-pillar orebody is \$3.17 Canadian per long ton. The total costs are as follows:

| | |
|---|---|
| Mining | \$3.17 per long ton |
| Milling | \$1.05 per long ton |
| General Administration | \$0.60 per long ton |
| Miscellaneous | \$0.37 per long ton |
| Depreciation | \$0.77 per long ton |
| Total | \$5.96 per long ton |

Total cost per metric ton -

$$\frac{2204.6}{2240} \times \$5.96 = \$5.87 \text{ Canadian}$$

It will be noted from the foregoing that when the proposed Mineral Land Tax is deducted, the value of one metric ton of ore decreases as the copper price rises. This is due, in part, to the "participation" clause in the copper selling contract. The company represents a very small supplier of copper and iron concentrates in Japan. The period during which negotiations for a contract was conducted was one of low metal prices. A world over-supply of iron ore has come into being since Texada Mines Limited started production so that while costs have risen by an order of magnitude, the price of iron concentrates has declined. Copper has become increasingly important for the mine in production and revenue. The contract was likewise negotiated during a period of low prices and cannot be said to be favourable to the company. The iron contract is firm to the end of mining, but perhaps it could be adjusted. The copper contract is adjusted every second year.

CONCLUSIONS:

We consider the operations of Texada Mines Limited are efficient and show little room for significant improvements. One possible exception is related to the lack of knowledge of the grade of ore hoisted or of the coarse waste rejects from iron ore. A very cursory sampling of recent coarse rejects (Sample 2) and fine material collected beneath the conveyor belt idlers (Sample 1) showed the coarse ore pile contains about 0.2 per cent copper. Most of the soluble iron shown in these samples were pyrrhotite by observation. The mine management is not unaware of this weakness and tried to introduce a programme of sampling and analysis by atomic absorption rather than their normal colorimetric methods. They said that difficulties of interference made them abandon the method but considerable advances have occurred in these fields in recent years and it might still be worth the cost involved.

COMMENTS:

1) The efficiency of the operation in our opinion is generally very good and little room for improvement is apparent.
2) The Mineral Land Tax Act reduces the value of the ore in the stopes appraised by approximately 11 per cent when the London Metal Exchange price for copper is $1.20 per pound U.S. This means some of the stopes are marginal, allowing for depreciation and little else.

RECOMMENDATIONS:

1) The geological potential for finding significant economic orebodies is very low and so we recommend that no programme of geological or exploration help be mounted by this Department.
2) The mine should reconsider the desirability of sampling hoisted ore and coarse rejects for more effective control.
3) The Department or Provincial Government should seriously consider a mechanism for participation in contract negotiations.

REFERENCES:

Bacon, W. R. (1952): Prescott, Paxton, Lake (Texada Mines Ltd.), Minister of Mines, B.C., Ann. Rept., 1952, pp. 217-221.

Janes, T. H. (1958): Iron Ore and Other Raw Material Sources for a Primary Iron and Steel Industry in Western Canada, C.I.M., Trans., Vol. LXI, pp. 314-323.

McConnell, R. G. (1914): Texada Island, B.C., Geol. Surv., Canada, Mem. 58.

Sangster, D. F. (1964): Contact Metasomatic Magnetite Deposits of Southwestern British Columbia, Ph.D. Thesis (unpublished), U.B.C.

Sangster, D. F. (1969): The Contact Metasomatic Magnetite Deposits of Southwestern B.C., Geol. Surv., Canada, Bull. 172.

Smillie, L.D.H. (1973): Sea Water Flotation, unpublished report, Texada Mines Ltd., Nov. 29, 1973.

Sutherland Brown, A. (1969): Mineralization in British Columbia and the Copper and Molybdenum Deposits, C.I.M., Bull., Vol. 62, pp. 26-40.

Sutherland Brown, A. (1972): Texada, XXIV I.G.C., Guidebook, Field Excursion A09-C09, pp. 15-20.

Sutherland Brown, A. (1964): Texada Mines Ltd., Minister of Mines, B.C. Ann. Rept., 1964, pp. 146-151.

Swanson, C. O. (1925): The Genesis of the Texada Island Magnetite Deposits, Geol. Surv., Canada, Sum. Rept., 1924, Pt. A, pp. 106-144.

Walker, A. M. (1974): The Transition to Trackless Mining at Texada, C.I.M., Bull., Vol. 67, pp. 38-43.

## DAY 2

**TEXADA**: By A. Sutherland Brown.

**LOCATION**: Lat. 49° 43′ — Long. 124° 34′ — Gillies Bay on the west coast of Texada Island.

**OWNERSHIP**: Texada Mines Ltd. (a subsidiary of Kaiser Aluminum & Chemical Corp.).

Texada is a copper-iron skarn deposit typical in most respects of the class as developed in the Insular Belt. It has the longest history of production of this group of deposits with very minor production at intervals from 1885 and steady production since 1952. From this year to the end of 1970, 16.5 million tons have been mined yielding approximately 8.8 million tons of iron concentrate grading about 65 per cent iron with 16,500 tons of copper, 20,000 ounces of gold, and 450,000 ounces of silver.

Production from 1952 until 1964 was from four separate open pits (*see* Fig. 3) and since then entirely from underground. Originally access underground was by a shaft and four main levels at 200-foot (61 metres) intervals shown on Figure 7, but trackless mining now utilizes surface trucks with a new system of inclined large diameter tunnels.

### Regional Setting

Texada Island, although close to the mainland coast, is formed of the typical Insular Belt stratigraphy, hence is underlain mostly by the very thick Triassic oceanic basalt (Karmutsen Formation, 4,500 metres). This is overlain by a massive Upper Triassic limestone (Quatsino Formation, 600 metres) which outcrops in a belt extending from the Texada mine northward to the northeast point of the island. A number of small plutons have intruded the stratified section. One emplaced at the southern termination of the limestone belt (Gillies stock) is mainly responsible for both the structure and metasomatism of the basalt and limestone, near their common boundaries. It has a potassium-argon age of 120 million years (Lower Cretaceous).

The stratified rocks of Texada Island generally occur in tilted panels or gentle folds cut by block faults. The limestone belt appears to be synclinal with some minor sharp folds near intrusive plutons and faults.

### Local Geology

The Karmutsen basalts of northwestern Texada Island are well pillowed but in the vicinity of the Gillies stock are variably metamorphosed, most generally to a chloritized and epidotized greenstone. The Quatsino limestone is predominantly a massive grey microcrystalline rock but in the vicinity of the mine it is bleached white and coarsely recrystallized. The rocks of the Gillies stock are slightly variable, but the commonest phase is a grey equigranular, medium-grained, mafic-rich augite-bearing granodiorite to quartz diorite which contains occasional large pyroxene crystals. Related pre-ore feldspar porphyries are variable appearing rocks with plagioclase and hornblende phenocrysts and a stony-looking, dark grey-green, fine-grained matrix; in hand specimen the feldspars appear to have vague gradational boundaries with the matrix. Garnet - pyroxene - epidote - actinolite skarn and magnetite-sulphide bodies may replace basalt, limestone, Gillies stock, or diorite porphyry with textures commonly diagnostic of each. Post-ore rocks in the vicinity are limited to large tabular dykes of grey diorite porphyry that has fewer phenocrysts than the pre-ore feldspar porphyries, late large tabular dykes of green porphyritic andesite and related grey-green andesite dykes with rare hornblende phenocrysts.

### *Orebodies and Their Structural Setting*

The orebodies are clustered around a salient at the north end of the Gillies stock. On the surface the four orebodies that outcrop were developed by separate open pits called Prescott, Yellow Kid, Paxton, and Lake (*see* Fig 3). The structure of the eastern orebodies (Paxton and Lake) differs from the western ones. The Lake and Paxton orebodies replace limestones, basalts, and minor amounts of quartz diorite at the keels of synclines which plunge gently *westward* and are overturned toward the *northeast*. The position, orientation, and rarity of these overturned folds in the whole area of Texada Island indicate that they may have been produced by lateral thrusting accompanying emplacement of the stock. Figure 6 shows a section through the Paxton orebody which occurs at the synclinal keel of the limestone, within an envelope of skarn that replaces limestone and greenstone and some diorite which intrudes the volcanic rocks. Post-ore porphyry dykes are prominent, and pre-ore dykes are not definitely recognized. Whether or not the upper limbs of these folds are thrusts, the keels are loci of small steep faults, some of which may be pre-ore.

The western orebodies, of which only the Prescott and Yellow Kid reach the surface, form a ramifying, upward-branching system that in three dimensions crudely resembles a tree, the thick stem of which is found at the lowest levels (1455-1655) at or near the east-west contact of the stock with the volcanics. At upper levels the contact of the stock warps to the east and overhangs the older rocks on the lower levels. The limestone and skarn developed from it generally dip at moderate attitudes southward, and are in contact with the stock down to the 1655 level. Below this level there are only volcanic and plutonic rocks. The orebodies branch and blossom out upon reaching the "limestone" and follow the warped contact of diorite, limestone, and volcanics in the upper levels.

The distribution of feldspar porphyry is important in the western orebodies. These porphyries are rarely seen in the open pits or much above the 2055 level and are absent from the Paxton and Lake orebodies. On the 2055 and 1855 levels they are prominent, but only in the ore zones; and on the 1655 and 1455 levels they occur to a minor degree in the ore zone. The porphyry masses are irregular and discontinuous, because of their original form and because they have been replaced by skarn and truncated by the main granodiorite.

Evidence of pre-ore brecciation is also important in the western orebodies. Much of the upper orebodies have textures that resemble breccia textures, with "fragments" of magnetite and filling of coarse calcite or skarn. In orebodies below the limestone, breccia textures are more clearly revealed, both in unreplaced and in replaced mimetic form. Breccias of volcanic fragments in diorite are common throughout the contact area. In addition, in some ore zones there is indication of a later brecciation, with quartz and sulphides filling interstices and with quartz crystal faces common.

### *Mineralogy and Zoning*

The skarn deposits of Texada Island show a marked zoning with the oxide orebodies surrounded by a virtually complete envelope of calc-silicate minerals and in turn surrounded by a zone of alteration with new silicate minerals and minor sulphides in rock that is clearly recognizable as to origin. The orebodies are composed principally of low-titanium magnetite with a variable but small amount of calc-silicate minerals or calcite and 1 to 3 per cent sulphides, chiefly chalcopyrite and pyrite, but with traces of pyrrhotite, arsenopyrite, and rare sphalerite. The skarn envelope is formed principally of andradite garnet with variable but lesser amounts of epidote, hedenbergite-diopside, and actinolite with minor calcite, magnetite, and pyrite. The outer altered zone varies markedly with the nature of the original rock but commonly has patches of calc-silicate minerals and sulphides in the host rock. The igneous rocks normally are intensely chloritized and the limestone coarsely recrystallized. In general the skarn envelope is three or four times the volume of the orebody. The altered zone passes gradationally outward to fresh rock and is several times larger than the skarn zone.

## Conclusions

Conclusive evidence of metamsomatic replacement of basalt, limestone, and plutonic rocks by skarn and ore includes the following:

(1) Kernels of less altered rocks in skarn and ore.

(2) Textures of skarn and ore that are mimetic of the host.

(3) Projection of the contacts of basalt, limestone, and pluton from areas of unaltered and altered zones into the skarn and ore zones.

(4) Differing character of skarn and ore where replacing different hosts. In general skarns replacing limestone are garnet-rich, those replacing volcanic rocks epidote-rich, and ore replacing limestone commonly is sulphide-rich.

The orebodies are arranged around the pluton with conduits, breccia zones or faults, apparently leading in toward it. Those of the west at least form an upwardly branching system that generally follows the contact zone and in its lower part seems spatially related to a pipe-like breccia. Where the conduit system, pipes or flat faults, reach the limestone the main ore and skarn bodies blossom out.

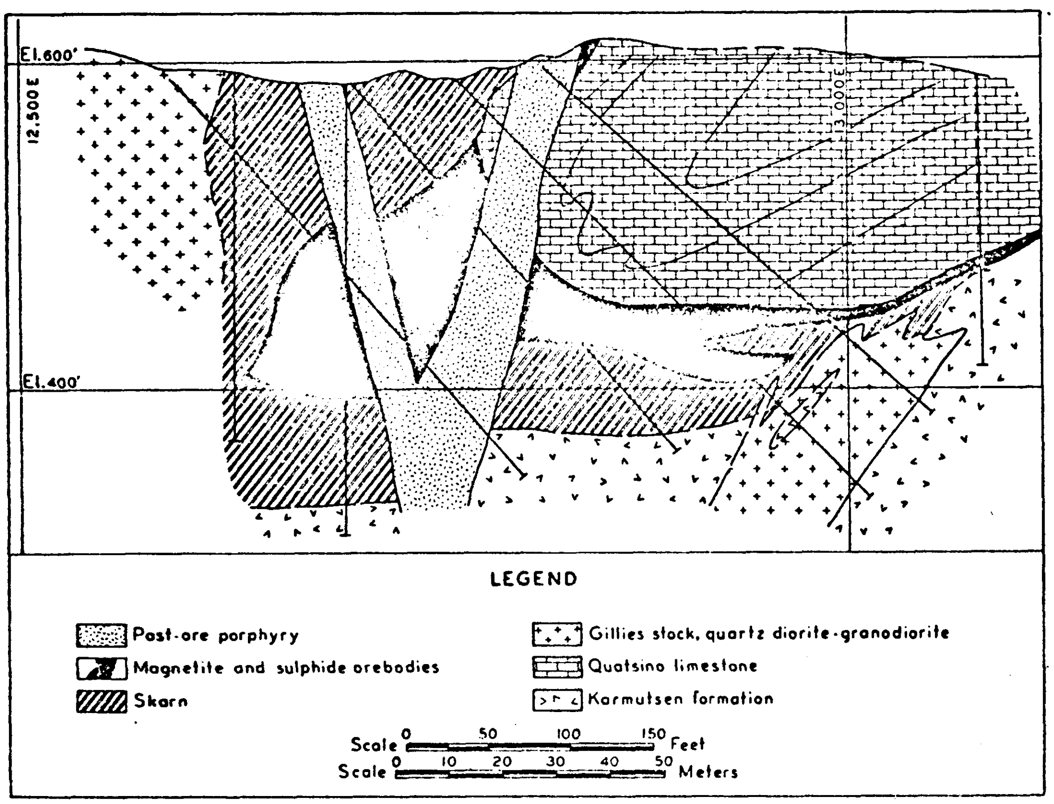


**Fig. 6.** Cross Section of Paxton Pit.

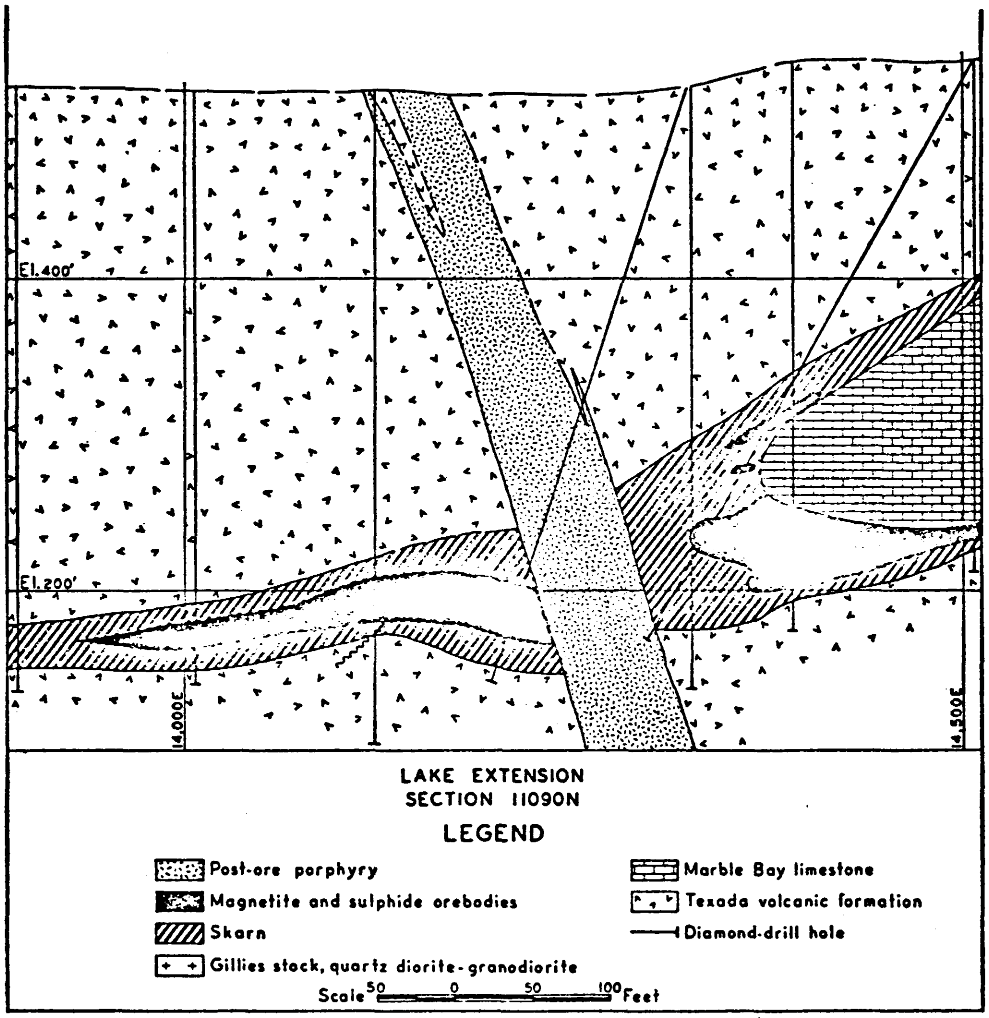


Figure 5. Texada Mines Ltd. Cross-sections of Paxton and Lake orebodies.

TEXADA MINES LTD.

NOTES ON ORE RESERVES TO
MARCH 31, 1974

1) No dilution factor has been included in the calculations.

2) Total ore reserves decreased by 171,000 L.T. during the first quarter of 1974. Production during this period was 246,000 L. T.

3) The average iron grade decreased from 38.19% to 38.07%. The average copper grade increased from 0.35 to 0.40%.

4) The minable copper reserves increased from 26 to 29% of the total minable reserves.

5) During the period the overdraw was 41,000 L.T.

| Stope | Overdraw |
|---|---|
| 1550 Prescott | 33,000 L.T. |
| 1750 Lower S.Y.K. | 8,000 L.T. |

6) Stope limits were revised in the 1750 Lower S.Y.K. reclassing 7,000 L.T. as Pillar instead of Positive.

7) Revisions were made in the following stopes:

| | | |
|---|---|---|
| 1455 S.Y.K. | + 5,000 L.T. | -a correction. |
| 1550 Midway I | -49,000 L.T. | -revised to blasthole ring tonnage. |

8) The accompanying tables summarize the changes in the ore reserves from December 31, 1973 to March 31, 1974.

9) All of the 18-106 L.H. Stope reserves are considered as copper ore. As a result figures from December 31, 1973 are not directly comparable.

R. G. PATERSON
Chief Geologist

Appendix 2

TEXADA MINES LTD.

Changes in Ore Reserves from December 31, 1973 to March 31, 1974

Tonnages in 1000's of long tons.

| | STOPE | DECEMBER 31, 1973 | | | MARCH 31, 1974 | | | CHANGE | CAUSE OF CHANGE |
|---|---|---|---|---|---|---|---|---|---|
| | | LT x 1000 | Fe | Cu | LT x 1000 | Fe | Cu | | |
| 1) | Le Roi 2A | 48 | 40.07 | 0.13 | 40 | 40.78 | 0.13 | - 8 | Production. |
| 2) | Le Roi 2B | 72 | 22.41 | 1.18 | 59 | 23.08 | 1.16 | - 13 | Production. |
| 3) | Upper S.Y.K. | 40 | 44.36 | 0.28 | 38 | 44.38 | 0.28 | - 2 | Production. |
| 4) | 18-106 L.H. Stopes | 337 | 27.30 | 0.63 | 286 | 26.56 | 0.64 | - 51 | Production. |
| 5) | 18-106 Room & Pillar Stopes | 96 | 30.55 | 0.96 | 93 | 30.84 | 0.99 | - 3 | Production. |
| 6) | North Extension | 129 | 43.40 | 0.85 | 201 | 36.95 | 1.18 | + 72 | -6, Production, exploratio |
| 7) | 1455 S.Y.K. | 73 | 45.04 | 0.13 | 26 | 43.66 | 0.16 | - 47 | Revised & Production (-52) |
| 8) | 1550 Midway I | 128 | 42.32 | 0.12 | 79 | 41.90 | 0.13 | - 49 | Revised. |
| 9) | 1455 Midway III | 57 | 39.84 | 0.21 | 50 | 39.84 | 0.21 | - 7 | Production. |
| 10) | 1550 Midway II Cu Zone | 24 | Neg. | 0.90 | 9 | Neg. | 0.90 | - 15 | Production. |
| 11) | 1550 Prescott | 65 | 35.24 | 0.06 | 46 | 34.00 | 0.07 | - 19 | Production. |
| 12) | Anomaly A. | 69 | 49.22 | 0.17 | 67 | 49.22 | 0.17 | - 2 | Production. |
| 13) | Lake Room & Pillar | 307 | 46.94 | 0.15 | 302 | 46.95 | 0.15 | - 5 | Production. |
| 14) | 2070 Lake | 419 | 44.11 | 0.15 | 397 | 44.14 | 0.15 | - 22 | Production. |
| | | | | | | | TOTAL | -171 | |
| | Total Minable Reserves | 3089 | 37.55 | 0.34 | 2876 | 37.36 | 0.39 | -213 | |
| | Minable Copper Reserves | 794 | 28.01 | 0.85 | 844 | 27.43 | 0.90 | + 50 | |
| | Total Reserves | 3752 | 38.19 | 0.35 | 3581 | 38.07 | 0.40 | -171 | |

# TEXADA MINES LTD.
## ORE RESERVES
PERIOD ENDING March 31, 1974

BY: R. G. Paterson

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ESERVES IN 1000's OF L.TONS

| L CK or STOPE | POSITIVE | | | PROBABLE | | | RECOVERABLE PILLAR | | | NONRECOVERABLE PILLAR | | | BROKEN | | | TOTAL | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| ove 2055 Level | | | | | | | | | | | | | | | | | | |
| Roi 2A | 3 | 36.49 | 0.10 | | | | 7 | 37.78 | 0.09 | 21 | 44.25 | 0.20 | 9 | 36.49 | 0.10 | 40 | 40.78 | 0.13 |
| Roi 2B | 24 | 19.33 | 1.27 | | | | | | | 27 | 27.56 | 1.06 | 8 | 19.33 | 1.27 | 59 | 23.08 | 1.16 |
| UBTOTAL | 27 | 21.24 | 1.14 | | | | 7 | 37.78 | 0.07 | 48 | 34.86 | 0.68 | 17 | 32.52 | 0.65 | 99 | 30.23 | 0.74 |
| 055 - 1855 Levels | | | | | | | | | | | | | | | | | | |
| orth Midway | 9 | 34.78 | 0.24 | | | | 18 | 49.38 | 0.11 | 23 | 50.09 | 0.45 | 4 | 34.78 | 0.24 | 54 | 43.73 | 0.27 |
| pper S.Y.K. | | - Nil - | | | | | | | | 11 | 45.44 | 0.31 | 27 | 43.97 | 0.27 | 38 | 44.38 | 0.28 |
| Roi #1 Ore Body | | | | | | | | | | | | | | | | | | |
| ope #1A | | - Nil - | | | | | 109 | 42.44 | 0.10 | | | | | | | 109 | 42.44 | 0.10 |
| illar #1A | | | | | | | | | | 188 | 37.34 | 0.36 | | | | 188 | 37.34 | 0.36 |
| tope #1B | 246 | 33.00 | 0.27 | | | | | | | 19 | 44.09 | 0.56 | | | | 265 | 33.80 | 0.29 |
| illar #1B | | | | | | | 30 | 39.80 | 0.35 | 23 | 39.80 | 0.35 | | | | 53 | 39.80 | 0.35 |
| #1C | | - Nil - | | | | | 60 | 41.54 | 0.18 | 13 | 31.99 | 0.40 | | | | 73 | 39.83 | 0.20 |
| UBTOTAL Le Roi #1 | (246 | 33.00 | 0.27) | | | | (199 | 41.47 | 0.16) | (243 | 37.81 | 0.38) | | | | (688 | 37.23 | 0.27) |
| idway Cu Zone | | | | 41 | 21.26 | 1.86 | | | | | | | | | | 41 | 21.26 | 1.86 |
| outh Prescott Zone | | | | 56 | 32.64 | 0.13 | 122 | 38.58 | 0.27 | | | | | | | 178 | 36.71 | 0.23 |
| 3-106 L.H. Stopes | 218 | 23.03 | 0.62 | | | | 36 | 43.05 | 0.54 | 14 | 41.59 | 1.33 | 18 | 24.65 | 0.61 | 286 | 26.56 | 0.64 |
| 3-106 R & P Stopes | 27 | 30.83 | 1.07 | | | | 12 | 38.47 | 0.74 | 54 | 29.14 | 1.00 | | | | 93 | 30.84 | 0.99 |
| orth Extension | 87 | 36.96 | 1.07 | 46 | 36.93 | 1.40 | 34 | 36.94 | 1.18 | 34 | 36.94 | 1.18 | | | | 201 | 36.95 | 1.18 |
| UBTOTAL | 587 | 29.81 | 0.55 | 143 | 30.76 | 1.03 | 421 | 40.65 | 0.32 | 379 | 37.60 | 0.58 | 49 | 36.12 | 0.39 | 1579 | 34.85 | 0.53 |

# TEXADA MINES LTD.
## ORE RESERVES

SERVES IN 1000's OF L.TONS

PERIOD ENDING March 31, 1974

BY: R. G. Paterson

Page 2

| K or STOPE | POSITIVE | | | PROBABLE | RECOVERABLE PILLAR | | | NONRECOVERABLE PILLAR | | | BROKEN | | | TOTAL | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 55 - 1655 Levels | | | | | | | | | | | | | | | | |
| 50 Lower S.Y.K. | 23 | 43.60 | 0.35 | | | | | 51 | 52.77 | 0.46 | | | | 74 | 49.92 | 0.43 |
| 00 S. Yellow Kid | 93 | 47.04 | 0.46 | | 36 | 50.12 | 0.34 | | | | | | | 129 | 47.89 | 0.42 |
| Y.K. 106 N. Pillar | | | | | | | | 62 | 43.97 | 0.13 | | | | 62 | 43.97 | 0.13 |
| 55 S.Y.K. | 10 | 43.99 | 0.23 | | | | | | | | | | | 10 | 43.99 | 0.23 |
| go #1 | 35 | 29.27 | 0.97 | | 33 | 13.67 | 1.13 | | | | | | | 68 | 22.02 | 1.06 |
| BTOTAL | 161 | 42.49 | 0.54 | | 69 | 32.69 | 0.72 | 113 | 47.94 | 0.27 | | | | 343 | 42.31 | 0.49 |
| 55 - 1455 Levels | | | | | | | | | | | | | | | | |
| 55 S.Y.K. | | - Nil - | | | | | | | | | 26 | 43.66 | 0.16 | 26 | 43.66 | 0.16 |
| 50 Midway I | 10 | 42.98 | 0.13 | | 58 | 41.74 | 0.11 | 11 | 41.95 | 0.22 | | - Nil - | | 79 | 41.90 | 0.13 |
| 50 Midway II | 7 | 39.74 | 0.25 | | | | | 16 | 35.19 | 0.70 | | - Nil - | | 23 | 36.57 | 0.56 |
| 55 Midway III | 45 | 39.84 | 0.21 | | | | | | | | 5 | 39.84 | 0.21 | 50 | 39.84 | 0.21 |
| 50 Midway II Cu Zone | | - Nil - | | | | | | | | | 9 | Neg. | 0.90 | 9 | Neg. | 0.90 |
| 5 Midway I | 93 | 38.99 | 0.17 | | 32 | 39.67 | 0.20 | | | | | | | 125 | 39.16 | 0.18 |
| 50 Prescott | 23 | 36.37 | 0.05 | | 23 | 31.62 | 0.08 | | | | | - Nil - | | 46 | 34.00 | 0.07 |
| BTOTAL | 178 | 39.12 | 0.17 | | 113 | 39.09 | 0.13 | 27 | 37.94 | 0.50 | 40 | 33.36 | 0.33 | 358 | 38.38 | 0.20 |
| low 1455 Level | | | | | | | | | | | | | | | | |
| 50 Midway | 155 | 42.60 | 0.08 | | | | | | | | | | | 155 | 42.60 | 0.08 |
| 50 S. Yellow Kid | 65 | 43.00 | 0.05 | | | | | | | | | | | 65 | 43.00 | 0.05 |
| BTOTAL | 220 | 42.72 | 0.07 | | | | | | | | | | | 220 | 42.72 | 0.07 |

# TEXADA MINES LTD.

## ORE RESERVES

ESERVES IN 1000's OF L.TONS

PERIOD ENDING March 31, 1974

BY: R. G. Paterson



| L CK or STOPE | POSITIVE | | | PROBABLE | | | RECOVERABLE PILLAR | | | NONRECOVERABLE PILLAR | | | BROKEN | | | TOTAL | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| scellaneous | | | | | | | | | | | | | | | | | | |
| omaly A | 66 | 49.22 | 0.17 | | | | | | | | | | 1 | 49.22 | 0.17 | 67 | 49.22 | 0.17 |
| xton Pit | | | | 216 | 25.93 | 0.76 | | | | | | | | | | 216 | 25.93 | 0.76 |
| ke Extension | | | | | | | | | | | | | | | | | | |
| ke R & P | 157 | 46.62 | 0.15 | | | | 72 | 47.30 | 0.15 | 72 | 47.30 | 0.15 | 1 | 46.62 | 0.15 | 302 | 46.95 | 0.15 |
| 70 Lake Stope | 301 | 43.54 | 0.15 | | | | 2 | 41.69 | 0.15 | 66 | 47.22 | 0.15 | 28 | 43.54 | 0.15 | 397 | 44.14 | 0.15 |
| btotal Miscellaneous | 524 | 45.18 | 0.15 | 216 | 25.93 | 0.76 | 74 | 47.15 | 0.15 | 138 | 47.26 | 0.16 | 30 | 43.86 | 0.15 | 982 | 41.35 | 0.29 |
| TAL | 1697 | 38.27 | 0.33 | 359 | 27.85 | 0.87 | 684 | 40.26 | 0.31 | 705 | 40.97 | 0.45 | 136 | 36.57 | 0.35 | 35.81 | 38.07 | 0.40 |

| Total Minable Reserves | | | | Minable Copper Reserves | | |
|---|---|---|---|---|---|---|
| Positive | 1697 | 38.27 | 0.33 | 391 | 27.00 | 0.82 |
| Probable | 359 | 27.85 | 0.87 | 303 | 26.97 | 1.01 |
| Broken | 136 | 36.57 | 0.35 | 35 | 20.11 | 0.84 |
| Recoverable Pillar | 684 | 40.26 | 0.31 | 115 | 32.33 | 0.92 |
| TOTAL | 2876 | 37.36 | 0.39 | 844 | 27.43 | 0.90 |

OTNOTES: Minable copper reserves are 29% of total minable reserves.

TM

**TEXADA MINES LTD.**
GILLIES BAY.
BRITISH COLUMBIA

TELEPHONE: MUTUAL 2-1010
(VANCOUVER)
486-7411
(GILLIES BAY)

Appendix 3

To: A. M. Walker April 29, 1974

From: R. G. Paterson

Subject: Minable Ore Reserves & New Taxation

1) As of March 31, 1974 the minable reserves and stockpile total 2,958,000 L.T.'s.

| | |
|---|---|
| Minable Reserves | 2,876,000 L.T.'s |
| Reserve Dumps | 61,000 L.T.'s |
| Surge | 21,000 L.T.'s |
| TOTAL | 2,958,000 L.T.'s |

This tonnage would last approximately 3 years.

2) Under the proposed Bill 31 and the Mineral Land Tax, the following stopes would appear to be uneconomical.

| Stope | Tonnage (L.T.) |
|---|---|
| S. Prescott | 178,000 |
| 1350 Midway | 155,000 |
| 1350 S. Yellow Kid | 65,000 |
| Paxton Pit | 215,000 |
| Lake Room & Pillar | 217,000 |
| 2070 Lake Stope | 303,000 |
| 402 Stope | 60,000 |
| TOTAL | 1,193,000 |

This tonnage is equivalent to 14 months production.

3) A schedule was drawn up eliminating the above reserves. According to this schedule the revised reserves will last until the end of November, 1975. There is insufficient time to fully extract the stopes listed below.

| | |
|---|---|
| 18-106 Room & Pillar | 27,000 L.T. |
| North Extension | 88,000 L.T. |
| TOTAL | 115,000 L.T. |

The total amount of ore that it may be necessary to abandon is 1,308,000 L.T. reducing the life of the mine by 16 months.

 

Appendix 4

# The Transition to Trackless Mining at Texada

A. M. Walker, Vice-President,
Texada Mines Ltd.,
Gillies Bay, B.C.

## Abstract

*Texada Mines Ltd. has been a producer of magnetite and copper concentrates from a typical west-coast metasomatic skarn zone since 1952 and during the past 20 years has progressed from an open-pit iron mining operation to conventional underground longhole open-stope mining, using 125-hp slushers and rail haulage, to completely trackless mining. More than 17 million long tons of ore have been extracted and milled, with approximately 8 million long tons originating from underground.*

*Underground development consisted of an 800-foot, vertical, four-compartment shaft, with 200-foot level intervals, equipped with a 13.5-ton counterbalanced skip and a 16-man counterbalanced cage serviced by two appropriate Koepe automatic hoists.*

*Longhole open stoping proved very successful for the first three years, when a decision was made to make the transition to trackless mining. This plan involved the driving of 9,287 feet of 14- by 11-ft decline at grades of -10% to -16%, at a cost of $55.97 per foot, to reach the lowest level in the mine, providing access and ventilation to the known ore reserves.*

*The mine is producing 90,000 long tons per month on a five-day week. Approximately 10% comes from room-and-pillar stopes, at an over-all average of 66 tons per man-shift and costs of $2.56 per long ton hoisted.*

## Introduction

THE PROPERTY of Texada Mines Ltd. is located on the west coast of Texada Island (Fig. 1) about 100 miles northwest of Vancouver. A 30-car ferry operates between Powell River and Texada Island, making access possible from the mainland or via connecting ferries from Vancouver Island. A scheduled air service operates from Vancouver to a company-maintained airstrip near the mine.

The climate is moderate, with about 30 inches of rain during the winter months and relatively dry summers.

The open-pit operation began in 1952 on a series of short-term ore contracts. Subsequent exploration programs indicated sufficient reserves for an underground operation and, following the negotiation of a ten-year contract with Japanese steel mills, the underground development work started in 1962. Early in 1964 the first ore was hoisted from underground; surface mining was phased out by 1966.

To date, more than 17 million long tons of ore have been milled, with approximately 8 million tons coming from underground operations. Shipments of iron concentrates total almost 9 million long tons. Copper concentrates are produced as a fringe benefit from mill heads averaging less than 0.3% Cu.

## Geology

The mineral occurrences are classified as metasomatic in origin. The orebodies consist of magnetite, chalcopyrite, pyrite and pyrrhotite contained in an envelope of garnet-epidote-actinolite skarn. The skarn zone occurs along the contact of the dioritic Gillies Bay stock and also extends out along much of the contact between the Marble Bay limestone and the older basalts of the Texada Formation.

Eight major orebodies have been outlined within an area of roughly 2000 by 5000 feet. All are extremely irregular in shape, varying from steeply dipping tabular or stock-like bodies to relatively thin flat-lying lenses. In every case, ore continuity is disrupted by extensive faulting. The size of the orebodies varies from one hundred thousand to almost three million long tons.

The mineralogy of the ore, a mixture of skarn and magnetite, makes it abrasive, hard and heavy, with a specific volume of 10 cubic feet per long ton. In general, ground conditions are good, permitting large openings of up to 50 by 100 feet, with only an occasional need for roof support.

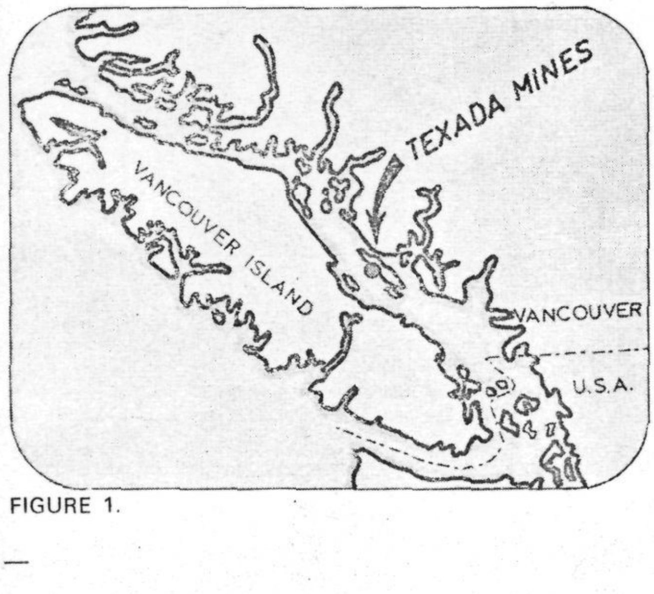

FIGURE 1.

---

**Arnold M. Walker**, P.Eng., graduated from Queen's University in 1940 and was employed by Sladen Malartic Gold Mines as a shiftboss. He spent 4 years as a Lieutenant with the R.C.E. during the war, returning to Barnat Mines in Malartic as mine captain and chief engineer. Following a year prospecting in B.C. in 1951, he moved to Campbell Chibougamau Mines Ltd., where he introduced the first trackless C & F stoping techniques as general superintendent. He later became manager of the company, and in 1963 spent a year as manager of the Whalesback mine in Newfoundland. He has been manager and vice-president of Texada Mines Ltd. since 1964, where he introduced trackless mining following the termination of open-pit operations.

**Paper Presented:** at the 75th Annual General Meeting of the CIM, Vancouver, April 1973.

**Keywords:** LHD mining, Underground mining, Trackless mining, Texada Mines Ltd., Magnetite, Copper concentrates, Skarn zones, Ramps, Stopes, Room-and-pillar mining, Longhole stoping, Alimak raising, Drop raising, Drifting, Transloaders.

**CIM Transactions:** Vol. LXXVII, pp. 8-13, 1974.

## Pre-Trackless Mining

The underground mine was developed from an 800-foot shaft, with an adit connecting with the shaft and main levels at 200-foot intervals. A crusher station below the lowest level is connected with the shaft by 800 feet of conveyorway inclined at 15 degrees.

Longhole open stoping methods, utilizing 125-hp slusher hoists with 72-inch scrapers in scrams ranging from 100 to 250 feet in length, produced about 1000 long tons per slusher shift. In most stopes direct scraping to mill holes was possible, but where tramming was necessary 130-cu. ft Granby cars were filled from chutes or by direct scraping. Haulage distances were short and productivity was very satisfactory.

## Trackless Planning

As early as 1965, detailed mine planning clearly showed that a large expenditure would be necessary in the following two or three years to provide ore-handling facilities and assure uninterrupted production. The following factors were involved:

(1) stopes scheduled for production were beyond the limits of the main ore-pass system;
(2) development costs would be high for some of the extremely irregular orebodies unless a flexible method of mining was employed;
(3) a number of small orebodies could be mined simultaneously if the equipment had a high degree of mobility;
(4) two gently dipping orebodies, averaging about 30 feet in thickness, were amenable to room-and-pillar mining.

The decision to gradually convert to trackless mining was based on an examination of trackless operations and a careful study of the comparative costs of conventional and trackless mining. In addition, improvements in safety, accident prevention and supervision, expected as a result of the change, would be of increasing benefit. The experience of mine personnel and the company's mining consultant favoured the use of Joy Transloaders for both production and stope preparation. The decision was influenced by the lower cost and also the ready availability of Transloaders at that time. However, one of the room-and-pillar zones was to be opened by a decline from surface, and this necessitated the purchase of a front-end loader capable of loading trucks. A Wagner ST5A Scooptram was selected for this role.

In November 1966, a decline from surface to the Lake Room and Pillar Zone was started using the Scooptram in conjunction with a three-boom jumbo constructed on the chassis of a surplus open-pit ore truck.

Two 2-wheel-drive Transloaders were taken underground through the shaft to the 1855 level (145 feet below sea level) and assembled in a previously prepared maintenance garage. These were followed by a 4-wheel-drive Transloader, which was put into service on the 1655 level.

## Access Ramps

While trackless development work was underway on the 1855 level, a service and access ramp was started from all four available locations. The ramp was driven at -10% from surface, elevation 2070, to the 1855 level and then steepened to -16% down to the 1655 level. The decline from surface was started with the Scooptram and an Eimco 916 L.H.D. The addition of the Eimco 916 enabled the ramp and the room-and-pillar mining to proceed simultaneously using the same drill jumbo. In the other ramp headings rubber-tired drill trucks with jacklegs, as well as a three-boom jumbo, were utilized.

In total, 3,468 feet of ramp was driven to provide access from surface down to the 1655 Level. A further 1,754 feet of -15% decline has since been driven to connect with the 1455, the lowest level.

The East Ramp, 2,575 feet in length, was completed within the past three years and connects the 1855 level with the Lake Room and Pillar area. The ramps and main haulageways are driven at 14 by 11 ft.

This system of ramps, in addition to providing access, also serves as the main ventilation airway. It is supplemented by several exhaust airways, including a 48-in.-diameter borehole drilled 400 feet from surface to the 2055 level. A summary of the costs of these ramps is shown in Table 1.

TABLE 1 — Summary of Ramp Costs
ramp size 14 ft by 11 ft)
Footage 9,287)

| | Cost/Foot |
|---|---|
| Supervision & Labour | $21.68 |
| Explosives | 7.89 |
| (Steel & Bits | 3.54 |
| (Miscellaneous | 3.81 |
| Repair & Maintenance | 3.36 |
| Timber & Supports | 0.57 |
| Load-Haul-Dump | 15.06 |
| TOTAL DIRECT COST | $55.97 |

## Stope Preparation

Longhole stopes are commonly developed with the haulage and undercut drifts along or just below the footwall of the ore zone and with sub-levels at 100-foot intervals. Sub-levels are driven with trackless equipment whenever possible, with access from the ramps. In some instances, where the stope tonnage is insufficient to justify lengthy drives in waste, drill sub-drifts are extended from track drifts or from raises within the stope limits. These 8- by 8-ft sub-drifts are driven with jacklegs and small air or electric slusher hoists. Trackless sub-drifts and undercuts are kept as small as the equipment permits, generally 12 by 10 ft, with a minimum size of 10 by 9 ft. Figures 2, 3, 4, 5 and 6 illustrate a typical longhole stope in plan and section.

Scram drifts are 14 by 11 ft, with drawpoints spaced at 40- to 70-ft centers along one or both sides of the scram. Draw points and undercuts are 12 by 10 ft and the average drawpoint is 30 feet in length.

Raises between 100 and 250 feet in length are driven with an Alimak raise climber. Slot and drill raises are 6 by 8 ft or 8 by 8 ft; service and ventilation raises are 6 by 6 ft. Short raises up to 50 feet in length, often required for undercut slots, are driven as open raises using stopers and standard staging methods.

An increasing number of raises up to 85 feet in length are being drilled and blasted using longhole drills and drop raising techniques.

The average advance in Alimak raises is 3 feet per man-shift, at a cost of \$50.00/foot. Drop raising costs for 400 feet of completed raise have averaged \$30.00/foot. A breakdown of these costs is shown in Tables 2 and 3.

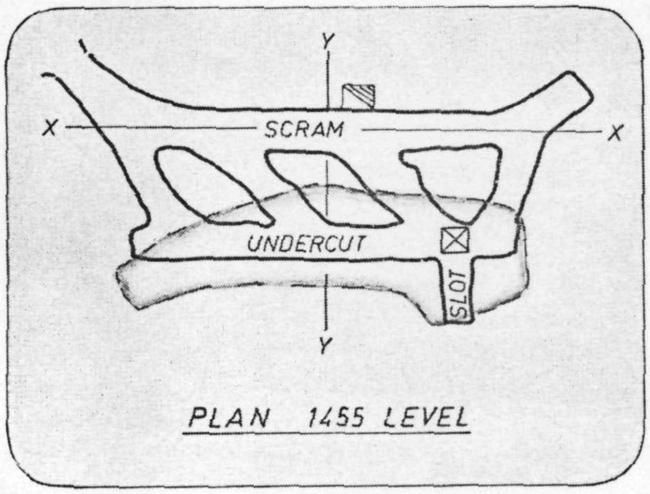


FIGURE 2.

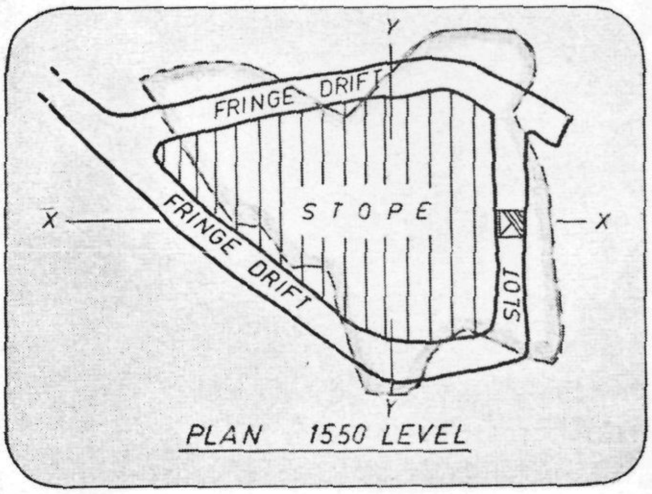


FIGURE 3.

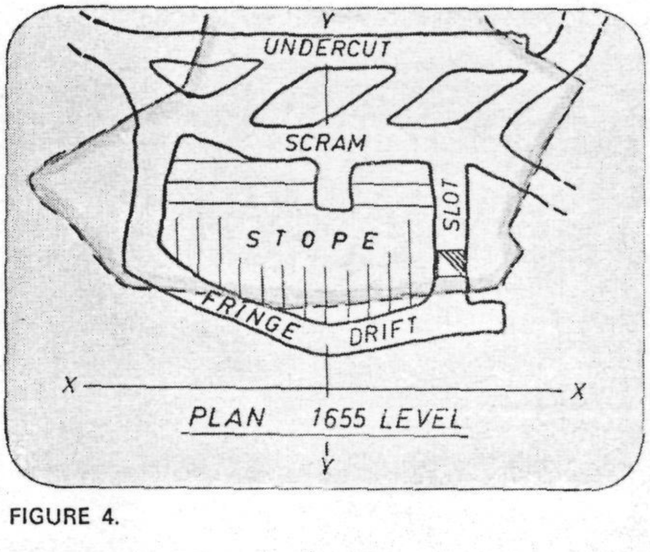


FIGURE 4.

Drifting performance using trackless equipment has been steadily improving, currently averaging 5 feet per man-shift, with costs of about \$45.00/foot. Costs are shown in Table 4.

Wherever possible, multiple headings are made available. Drilling is done by a one-man crew using a three-boom drill jumbo, and mucking is done between drilling shifts. Blasting is done by the miner, using ammonium nitrate explosives or 80% Forcite in 1½- by 8-in. cartridges. Cilgel "B" in 1- by 8-in. cartridges is used in raising and to a minor extent in small subdrifts.

## Room-and-Pillar Mining

Both of the zones amenable to room-and-pillar mining are continuous over a considerable horizontal extent, but are quite irregular in thickness and attitude. Iron ore grades usually show good continuity, in con-

TABLE 2 — Alimak Raising

(footage 468 ft; size 7 by 8 ft)

| | Cost/Foot |
|---|---|
| Supervision & Labour | \$31.31 |
| Explosives | 5.95 |
| Steel & Bits | 4.37 |
| Load & Haul | 2.77 |
| Repair & Maintenance | 6.35 |
| Miscellaneous | 0.72 |
| Timber & Supports | 0.71 |
| TOTAL DIRECT COST | \$52.18 |

TABLE 3 — Drop Raising

(footage 364 ft; size 4 by 5 ft)

| | Cost/Foot |
|---|---|
| Supervision & Labour | \$16.89 |
| Explosives | 4.88 |
| Steel & Bits | 7.13 |
| Load & Haul | 1.66 |
| TOTAL DIRECT COST | \$30.56 |

TABLE 4 — Trackless Drifting*

(footage 6,404.5 ft; size 14 by 11 ft)**

| | Cost/Foot |
|---|---|
| Supervision & Labour | \$17.30 |
| Explosives | 6.33 |
| Steel & Bits | 3.96 |
| Equip. Rep. & Maint. | 2.80 |
| Load-Haul-Dump | 11.34 |
| Miscellaneous | 1.55 |
| Trucking | 0.66 |
| Timber & Supports | 1.83 |
| TOTAL DIRECT COSTS | \$45.77 |

*Includes minor raising footage.
**Average size of openings.

trast to the erratic copper mineralization. Diamond drilling on sections 50 feet apart is used for the preliminary layout. Initially, one stope is advanced and percussion test holes are drilled on the intermediate 25-foot sections to delimit the ore for the adjacent 100 feet on both sides of the stope.

Stopes are usually 40 feet in width, separated by 20-ft-wide pillars. Mining heights range from 12 to 40 feet, with thicknesses of up to 18 feet being mined in one pass. Thicker zones are mined by taking a top cut by drifting and slashing, securing the back and then benching the remaining ore.

Ore extraction is maintained with two- and three-boom Joy Drillmobiles, CF93 drills and 10-foot steel. Extension steel is used for slashing and benching when conditions are suitable.

Productivity in the room-and-pillar areas has varied from 90 to 110 tons per man-shift. Mining costs for these two areas are shown in Tables 5 and 6.

## Longhole Stoping

Blast-hole drilling is done with DH-123 drills, using 4-foot rods with 2-inch tungsten carbide bits. Drills are mounted on horizontal or vertical bars, stabilized by additional bars due to the size of the drifts.

Slot blast holes are spaced 4 feet apart in parallel rows with a 3-foot burden. Parallel fans of holes are drilled from the undercut drift with a burden of 5 feet and a toe spacing of 5 to 6 ft.

The main rings, generally vertical fans of up and down holes, are drilled with a burden of 5 or 6 feet and 8-foot toe spacing. This drilling is done from fringe drifts or a centrally located drill drift. Figures 1, 2, 3, 4 and 5 illustrate typical longhole drilling patterns.

Average performance is close to 200 feet per man-shift, at a total cost of 80 cents per foot.

Loading and blasting is done by separate crews under close supervision. The explosives used are 80% Forcite, in 1½-in. and 1¾-in. cartridges, and Amex. Recently, a metallized Amex (Anfomet) has been used with good results. Local conditions dictate the type of explosive and the size of the blast. Generally, large blasts are preferred because of the improved fragmentation. Once the slot has been opened, the blasting face of the stope is usually kept nearly vertical for the full height of the stope in order to reduce caving of drilled ground.

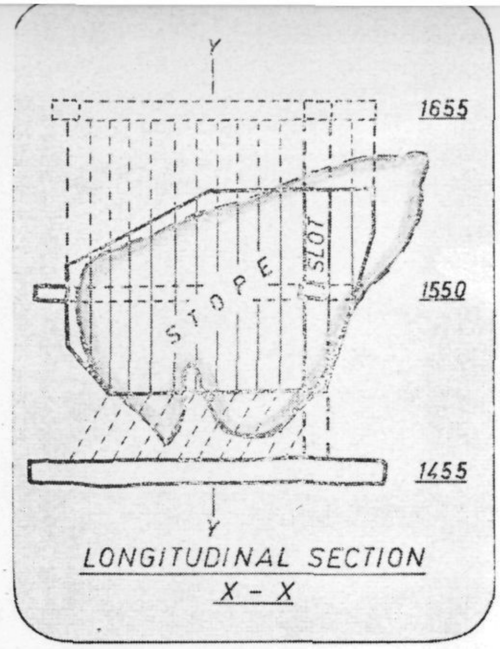


FIGURE 5.

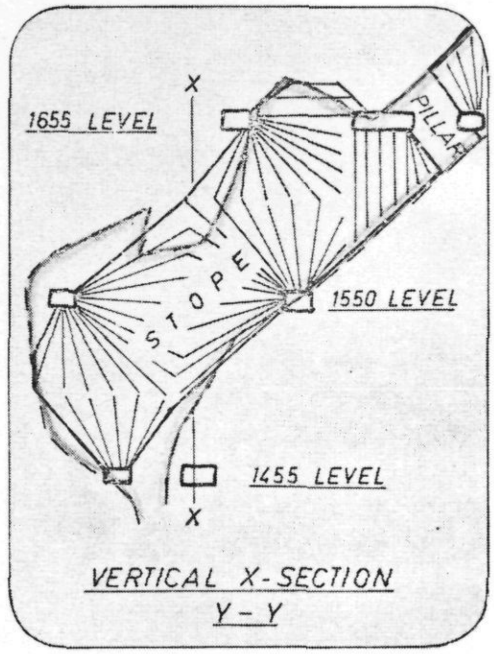


FIGURE 6.

TABLE 5 — 18-106E Room-and-Pillar Costs

(Production 82,928 l. tons)

| | Cost/Ton |
|---|---|
| Supervision & Labour | $ 0.60 |
| Explosives | 0.29 |
| Test-Hole Drilling | 0.07 |
| Steel & Bits | 0.12 |
| Trucking | 0.22 |
| Equipment R & M | 0.09 |
| *Miscellaneous* | 0.04 |
| Timber & Supports | 0.09 |
| Load-Haul-Dump | 0.62 |
| TOTAL DIRECT COSTS | $ 2.14 |

TABLE 6 — Lake Room-and-Pillar Costs

(Production 81,311 l. tons)

| | Cost/Ton |
|---|---|
| Supervision & Labour | $ 0.48 |
| Explosives | 0.21 |
| Steel & Bits | 0.18 |
| Equipment R & M | 0.18 |
| Trucking | 0.11 |
| *Miscellaneous* | 0.08 |
| Timber & Supports | 0.01 |
| Load-Haul-Dump | 0.47 |
| TOTAL DIRECT COSTS | $ 1.72 |

All stope blasts are initiated electrically and both 220- and 440-volt systems are available.

Powder factors vary considerably and a rough average would be 0.35 lb per long ton. Secondary powder consumption, ranging from nearly zero in some small stopes to 0.3 lb/long ton, is directly related to the primary factor, although minor sloughing of waste from stope walls does occur. Considerable effort is being applied to determining the right balance among ring and toe spacing, type of explosive, method of priming and delay pattern which will result in improved total costs.

## Production

Based on a monthly production of 90,000 long tons, about 10% is produced from room-and-pillar stopes, up to 5% from development headings and the remainder from longhole stopes. This tonnage is handled by eight Transloaders, of which four are 4-wheel-drive models, and one ST5A Scooptram hauling to the main ore passes, transfer raises or secondary haulage units. A particularly difficult ore-handling problem in the 2070 Lake stope has been solved by combining the loaders and trucks. Here, Transloaders muck to a loading chute and two Wagner MTT 423 telescopic shuttle cars haul 2,400 feet to a transfer raise, where the ore is picked up 100 feet below by Transloaders and hauled 800 feet to the ore pass. Table 7 shows the production costs for this stope.

TABLE 7 — 2,070 Lake Production Costs

(Production 22,544 l. tons)

| | Cost/Ton |
|---|---|
| Supervision & Labour | $ 0.03 |
| Explosives | 0.33 |
| Blasting Labour | 0.19 |
| Longhole Drilling | 0.41 |
| Trucking | 0.39 |
| Timber & Supports | 0.01 |
| Load-Haul-Dump* | 0.55 |
| Miscellaneous | 0.01 |
| TOTAL DIRECT COSTS | $ 1.92 |

*Double handling by Transloader.

Room-and-pillar production in the 18-106 Zone is handled by 4-wheel-drive Transloaders directly to the main ore-pass system. In the Lake room-and-pillar stopes, the Scooptram loads directly into a converted 5-ton open-pit truck. The truck hauls to surface stockpiles or directly to the mill.

About 55,000 long tons per month are hauled directly to the main ore passes over an average haulage distance of 300 feet.

## Roads

As an important factor in efficient transportation, roads are well maintained on ramps, haulageways and as far as practicable in development headings. Two Allis Chalmers Model "D" graders, with 12-foot blades, operate at least 50 hours per week. In addition, one Transloader is used almost exclusively for ballasting roads with minus-1-inch crushed rock. Ditches are maintained in most haulageways, with drainage to small sumps either directly or via drain holes drilled for that purpose. The cost of road maintenance is charged to production.

## TRACKLESS EQUIPMENT

During the first year of trackless mining, tire costs were considered to be excessive. The following policy was adopted after a study of the factors affecting these costs:

(1) ramps are kept to minimum practicable grades, generally less than 15% — only four-wheel-drive L.H.D. equipment is used to advance inclined headings;

(2) roads are maintained by grading and ballasting;

(3) several brands of tires with variations in tread design and construction were tested on different loaders under a variety of conditions and assessed with respect to initial cost, performance and final cost per ton;

(4) bucket modifications, including the addition of teeth, were made to reduce the loading effort and tire slippage;

(5) L.H.D. operators were given additional instruction designed to reduce tire damage.

Along with the production and road maintenance equipment, four converted jeeps are used for nipping and servicing. Three diesel Lobos are used by supervision and three recently acquired Wagner personnel carriers are involved primarily with supply handling. The Lobo is a light four-wheel-drive vehicle suitable for nipping and use by supervision. It is powered by a single cylinder 11.5-hp Mag diesel. The Wagner personnel carriers are much larger; these particular units are capable of carrying over one ton of supplies or up to ten persons. They are powered by Deutz diesels, ranging from 44 to 66 horsepower.

## Trackless Mining Costs

Table 8 shows the combined operating costs of the Joy Transloaders. It should be noted that these costs include operating labour averaging $9.40 per hour. This cost is made up of wage and fringe benefits, contract earnings, an allowance for supervision and secondary blasting, and all road maintenance labour charges.

TABLE 8 — Transloader Costs (Dev. & Prod.)

June 1/71 to May 31/72
(Operating hours 19,879)

| | Cost/Hour |
|---|---|
| Operating Labour | $ 9.40 |
| Rep. & Maint. Labour | 4.29 |
| Fuel & Lube | 1.30 |
| Rep. Parts & Supplies | 5.31 |
| Shop Overhead | 2.76 |
| Tires | 1.57 |
| TOTAL | $24.63 |

Repair parts and repair labour costs have been separated, and a proportion of indirect repair costs is shown separately as "shop overhead".

Tire costs have been reduced sharply from $3.78 per operating hour during the first two years of operation to $1.57 per hour currently. The cost figures given in the table are averages for June 1971 to May 1972.

Table 9 shows the same costs expressed in cents per ton hauled, with an over-all performance of 65.2 long tons per hour. The hours used in these figures are taken from the engine running time and therefore include time not spent directly on production work. They also include the cost of development muck handling.

TABLE 9 — Transloader Costs (Dev. & Prod.)

June 1/71 to May 31/72
Long tons 1,296,720
Performance 65.2 l.t./hour

| | Cost/Long Ton |
|---|---|
| Operating Labour | 14.4c |
| Rep. & Maint. Labour | 6.6 |
| Fuel & Lube | 2.0 |
| Rep. Parts & Misc. | 8.2 |
| Shop Overhead | 4.2 |
| Tires | 2.4 |
| TOTAL | 37.8c |

More than five years of experience with L.H.D. equipment indicates the relative advantage of Transloaders over other types for production mucking at this operation. The Transloader is limited in some uses due to its inability to load directly into ore carriers or backfill stopes without expensive chute preparation and, for this reason, a combination of Transloader and Scooptram types seems preferable.

Compared with front-end loaders, the better load distribution in the Transloader results in lower repair costs and smoother travelling characteristics, reducing operator fatigue. With the exception of the power and drive train, rebuilding or converting Transloaders from two- to four-wheel drive is quite within the capabilities of a well-equipped machine shop.

Several modifications have been made as a result of maintenance experience over a period of time. A gradual replacement of the original engines with the six-cylinder Deutz has resulted in improved operating performance. Due to the heavy and abrasive characteristics of the magnetite ore, plate thickness of the bucket assembly has been increased by 50% using CHT 360 steel (similar to T1) and the bale arm has been extended to compensate for the additional weight. Heavier transmissions and improved planetary assemblies have been necessary because of the increased weight of the front ends. The original drive chains on the four-wheel-drive machines are being replaced by a heavier type. Other changes will undoubtedly be made as new weak points are corrected.

Tire costs are considerably lower with the Transloader, partly because of their smaller size, but also due to the loading characteristics of the machine. Another feature of the Transloader, the cable lift on the bucket, is a safety feature when mucking drawpoints. Large rocks falling from the drawpoint and striking the bucket lip will break the cable, reducing the possibility of damage to the machine and injury to the operator. The same occurrence with a hydraulically operated front-end bucket actually resulted in serious damage to the machine, but the operator fortunately escaped injury. An additional advantage is the self-draining feature of the Transloader bucket after loading in wet drawpoints.

## Mining Costs

In trying to compare the present trackless operation with track and slusher operations of seven years ago, especially on a unit cost basis, several factors must be evaluated or at least recognized.

The first of these is the continued increase in both labour (70%) and material (10%) costs since 1965.

The second factor is that the larger and less irregular orebodies were developed and mined first. Obviously, these orebodies governed the location of the crusher and ore-pass system. As a result, mining at Texada has seen a progressive increase in haulage distance, along with the increased costs of exploring and developing smaller and more erratic orebodies.

One positive factor has been the decreasing turnover of personnel in recent years. Although difficult to evaluate in monetary terms, the effect on operating and maintenance costs is no doubt significant.

The trend to mechanization has increased over-all mine performance from 44 long tons per man-shift in 1965-1966 to 66 long tons per man-shift in 1972. Table 10 illustrates the reduction in total employees, especially underground, and the decreasing rate of turnover since 1965.

TABLE 10 — Employee Statistics

| Year | Total U/G | Total Shops | Surface, Mill, Staff, Misc. | Total Mine | % Turn-Over |
|---|---|---|---|---|---|
| 1965 | 106 | 46 | 110 | 262 | 120 |
| 1966 | 118 | 47 | 103 | 268 | 129 |
| 1967 | 113 | 49 | 122 | 284 | 65 |
| 1968 | 90 | 44 | 96 | 230 | 35 |
| 1969 | 81 | 40 | 111 | 232 | 26 |
| 1970 | 76 | 42 | 112 | 230 | 20 |
| 1971 | 73 | 41 | 106 | 220 | 19 |
| 1972 | 70 | 39 | 105 | 214 | 28 |

In the final analysis, it is the cost per ton that determines the success of most mining operations. The 1966 cost per ton hoisted at Texada was $2.60. The present cost of $2.56 per ton indicates that mechanization has just compensated for normal inflationary trends and increasingly adverse mining conditions. Continued effort must therefore be directed to finding new ways to improve over-all performance so that maximum extraction of the ore reserves can be accomplished.

*(Reprinted from The Canadian Mining and Metallurgical Bulletin, January, 1974)*
**Printed in Canada**

## ABSTRACT

The Texada Mill is a unique iron operation producing High Grade sinter iron concentrate and a by-product copper, gold, silver concentrate. The plant is complex and extremely flexible to handle the many varieties of ore. Two iron concentrates are made and blended. A coarse iron concentrate from primary grinding and a fine concentrate after rougher flotation to remove all the sulphides.

All the tailings from iron processing are collected, dewatered, reground and floated for additional copper recovery.

The entire milling operation has been carried out in all sea water for seven years and mixtures of sea, fresh, and reclaimed water for ten years prior to that.

A very stable float results from the use of sea water at the natural pH8. Reagent Consumption is lower and Metallic Corrosion of mill liners and grinding media is actually less than local fresh water or mixtures of fresh and sea water.

Marine growth in pipe lines can be periodically removed by violently bubbling high pressure air up through the water filled mains.

# "SEA WATER FLOTATION"

TEXADA MINES LTD.
GILLIES BAY, B. C.

B Y

L. D. HAIG - SMILLIE

November 29, 1973

LOCATION:

Texada Mines is located on tide water on the west side of Texada Island 100 miles northwest of Vancouver, B. C.

The Island is served by a 30 car ferry, tug and barge and a scheduled air service from Vancouver to a company maintained air strip near the mine.

The climate is moderate with about 30 inches of rain during the winter and relatively dry summers.

HISTORY:

The property was developed for high grade Iron between 1883 and 1908. Texada Mines was incorporated as a private company in 1951 to mine and process lump ore for shipment to Japan. After the lump ore was exhausted a satisfactory contract for sinter concentrate was obtained and a plant incorporating grinding, flotation, and wet magnetic separation was built. The designed capacity was 1000 long tons of concentrate per day.

The old dry separation plant was converted to crushing with dry cobbing of the coarse pit run ore.

Provision was made to recover some of the process water.

MINEROLOGY:

The mineral occurrences are classified as metasomatic in origin. The ore bodies consist of magnetite, chalcopyrite, pyrite, and pyrrhotite in a gangue of garnet - epidote - actinolite skarn.

The ore is hard, and extremely abrasive. The specific volume is 10 cubic feet per long ton. The pulp is resistant to flow and exhibits rapid settling, below 100 Mesh abrasion is minimal and it acts as a heavy media at 200 Mesh.

There are three major types of ore:

1) Massive magnetite with little or no sulphides.
2) Massive find grained magnetite with chalcopyrite, pyrite, and pyrrhotite.
3) Pyritic copper ore: Sparse magnetite containing massive blobs and or disseminated sulphides, striated cubes, octahedrons, and pyritohedrons. This mineral under the electron microprobe analyser shows that the chalcopyrite occurs as partial rimmings or coatings of the pyrite particles. Masses of chalcopyrite are found wholly inside pyrite crystals from 1 to 50 microns in size. Fractures 1 to 1/10 micron in width are commonly found filled with chalcopyrite. It is therefore necessary to float some of the pyrite with the copper

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MILL PRODUCTS:

TABLE I shows the Basic Iron Concentrate specifications:

TABLE I

| Mineral | No Penalty | Desired Grade | Rejection |
|---|---|---|---|
| Contained Iron | 62.00% | 64.5+% | 62.00% |
| Copper | 0.12% | 0.08% | 0.15% |
| Sulphur | 3.00% | 0.25% | 1.00% |
| Phosphorus | 0.10% | 0.02% | 0.15% |
| Moisture | 7.00% | 5.75% | 7.00% |
| 100% Minus 10 Mesh | | 40% Minus 100 Mesh | |

A premium is applicable in respect to copper content below 0.12%.

The mill operates five days a week producing 1900 long tons of premium sinter iron concentrate per day assaying 65% Iron, 0.055% Copper, 0.40% Sulphur, and 42% - 100 Mesh. This is made by blending two iron concentrates:

1) A coarse iron concentrate made from ball mill circulating load assaying 64.5% Iron.
2) A fine iron concentrate made from rougher flotation tailing 66+%.

The proportions depend on the ore being treated. To yield the highest net cash return it is necessary to produce the maximum amount of coarse iron that can be blended with the fine iron concentrates.

Over grinding is costly and undesirable. The primary grind is kept as coarse as possible. Tailing is collected, dewatered, reground and floated for additional copper recovery.

A by-product copper concentrate, containing gold and silver, high in pyrite, makes the operation economic.

Coarse waste rock is sold for riprap or crushed and sold as aggregate.

MILLING PROCESS:

CRUSHING:

The crushing plant has a 30,000 ton mine run ore and a 2,000 ton surface ore surge pile. The dry cobbing separators were retained to remove as much waste as possible prior to second and third stage crushing. The plant is completely flexible for any type of ore, or fur producing crushed rock or heavy aggregate.

........................Cont'd....

Table II shows the mine run ore before and after magnetic cobbing.

TABLE II

| Mineral | Mine Run Ore | Rod Mill Feed |
|---|---|---|
| Soluble Iron | 35.4 % | 43.8 % |
| Magnetics | 43.3 % | 54.9 % |
| Copper | 0.287% | 0.341% |
| Sulphur | 1.86 % | 2.04 % |
| Oxide Copper | | 0.025% |
| Weight Recovery (Concentrate) | 50.4 % | 67. % |

MILLING:

The milling of iron ore or of copper ore alone is not economic at the available production tonnage. Copper bearing high grade magnetite is essential to the process. Selective mining is practiced to segregate ore in which the copper and magnetite are mixed from those containing unassiciated copper mineral.

This intimately mixed ore is magnetically cobbed during crushing to upgrade the iron content without serious loss of copper mineral. Ore which contains uncombined copper mineral can not be upgraded magnetically during crushing without serious loss of copper mineral and the total volume of ore must be milled.

For this reason both the crusher and the mill are extremely flexible and complex. To meet our contract specifications processing is done in three distinct circuits:

1) Coarse Iron Concentrate:

A portion of the cyclone underflow in each primary ball mill is cut out and sent to six stage magnetic separation. The concentrate is dewatered in a vibrating dewaterizer and the concentrates go to the drying kiln collecting conveyor. (Flow Sheet #1).

2) Fine Iron Concentrate:

Primary cycline overflow goes to a conditioner then to flotation. The tailings go to 4 stage magnetic separation. Magnetic concentrate goes to a thickener which also acts as a hydro separator. The underflow is filtered and is conveyed to the 9' x 60' drying kiln where the moisture is reduced from 10.5 to 5%. The concentrate is weighed and stacked in an open stock pile. (Flow Sheet #2).

....................Cont'd......

3) Regrinding and Secondary Flotation: (Flow Sheet #3)

All the tailings are collected, dewatered and ground in two parallel regrind curcuits.

The coarse iron tailing is divided between two 36" classifiers. Sands from the first classifier go to a 8' x 10' ball mill then an 8 cell Agitair flotation machine, followed by single stage magnetic separation. Separator tailing goes to the fine iron tailing sump.

First copper cleaner tailings are retreated in this circuit.

Sands from the second classifier go to an 8' x 12' Marcy regrind ball mill, then to an 8 cell Abitair flotation machine. Tailings go to a double drum magnetic separator and then to waste.

The fine iron tailings are cycloned to remove the sands which are reground in the 8' x 12' Marcy mill.

The magnetic separator tailing is very dilute. Cycloning then is essentially a dewatering operation. This overflow water plus excess classifier weir overflow and the recleaner tailing is floated in a 6 cell DR#30 flotation machine. Tailings from the DR go to a double drum separator then to waste.

FLOTATION HISTORY:

Fresh and reclaimed water supplies were not adequate. In order to maintain the operation we found it necessary to supplement the water supply with sea water.

It was recognized from the start that Texada ore pulps were subject to a buffering action. Lime additions from two to five pounds per ton were required to change the alkalinity from the normal pH8. (Graph #1).

It was also noted that increasing the lime increased the grade of concentrate but very substantially reduced the recovery. (Graph #2).

Microscopic examination revealed a slime and a carbonate coating on the sulphide mineral. Frequently numerous clean chalcopyrite grains could be found in the tailing.

Mill tests conducted in water not from the island invariably gave better results.

Test data and smelter schedules indicated that 21 - 22% copper was our most economic grade of concentrate. (Graph #3).

In 1966 tailings disposal problems made it necessary to add an additional 500 G.P.M. of sea water to our tail race. Much to our embarrassment a good strong copper froth began to appear on the lower reaches of the tailings launder. As a result we recognized the problems with the Texada flotation circuit...Cont'd

1) Lime being a flocculant was favoring slime coatings on the mineral surface.
2) The calcium ion and or the OH ion was excluding or displacing the xanthate from adsorbing onto the mineral surface.
3) The calcium coating deposited could be dissolved with excess dilution.
4) Sea water had no detrimental effect on flotation.

All the lime was removed from the circuit and just enough soda ash added to ensure an active pyrite float. Recoveries increased enormously.

In 1966 the mine went to trackless mining, underground water supplies were unusable due to colloidal slimes and oil. The unusually dry season created a critical water shortage. The mill went to 100% sea water.

ROUGHER FLOTATION:

Reagents are added to the rod mill as this has been found to be the most satisfactory point of addition.

The reagents added are 343 at 0.015 lbs., 238 at 0.01 lbs., Aerofloat 33 and Z-200 at 0.0005 lbs., and Soda Ash at 0.048 lbs. per long ton.

Primary cyclone overflow at pH8, 42% solids and 40% - 100 Mesh goes to a collecting conditioner, then to an 8 cell DR#30 flotation machine. Aerofroth 65 at 0.008 lbs. per ton is fed to the feed well. Booster amounts of reagent 343 and reagent 242 at 0.0001 lbs. per ton are added to the tail box and the pulp goes to an 8 cell Agitair flotation machine. Four of these cells are equipped with DR mechanism kits. Both banks of cells have automatic level control.

The coarse grind requires a tough small bubble, short lived froth. The slime levels are shallow grading from 5" to 1". The pull is hard and all possible sulphides and sulphide-magnetite locked grains are removed as concentrate to a 6' x 6' conditioner used as a pump sump.

Average recovery of copper to this point is 82%.

When the host rock is limestone the flotation develop a large bubbly dirty grey froth. Concentrate grade is usually high but recovery falls off.

REGRIND FLOTATION:

The 8' x 10' regrind circuit feed is a clean coarse sand higher in copper than normal beads after the magnetite is removed.

Small booster additions of reagents 343 and 238 are added to the mill feed spout. Minor amounts of 242 and Aerofroth 65 are added to the feed well of the first Agitair cell. First cleaner tailing, high in magnetics is retreated in this flotation circuit.

Froth levels are 10", the froth is black and buoyant; possibly a little tight, but excellent for the purpose.

.................Cont'd...

The 8' x 12' regrind circuit grinds one half of the coarse tailing, plus all the cycloned sands from rougher flotation, and from the 8' x 10' circuit, and Pyrite sands from the 6' x 6' rougher concentrate sump.

Booster amounts of reagent 343 and 238 are added to the ball mill feed spout. Reagent 242 is fed to the sump ahead of cycloning.

This circuit is extremely high in pyrite. The froth is small bubble and very heavily laden with mineral. The pulp level starts at 12 inches, the pH is 8, the pulp is 67% - 100 Mesh and cell residence time is 5-7 minutes. Tailings go to a double drum magnetic separator then to waste.

We find it advantageous to pass the dilute cyclone overflow together with surplus classifier overflow and the recleaner tails through a 6 cell DR#30 flotation machine to recover slimed sulphides.

This cell has automatic level control set at a 15 inches froth level. No additional reagent is required. A very steady float recovers a concentrate assaying 9 to 12% copper.

Tailings go to a double drum magnetic separator, then to waste.

CLEANING:

All the rougher flotation concentrates go to the 6' x 6' conditioner. Very coarse particles settle out. These particles are pumped from the bottom of the tank to a 10" cyclone. The overflow returns to the sump. The underflow which is 80 + 48 Mesh, and 1.5% copper is reground in the 8' x 12' regrind ball mill.

Rougher concentrates are low in insoluble material but contain locked magnetite - sulphide grains floated in primary roughing. This material is removed in first stage cleaning and returned for grinding via the 8' x 10' regrind circuit.

First stage cleaning is a 6 cell DR#24 flotation machine with automatic level control. The first two cells can be sent directly to filtration, if desired, or all six cells go to the 4 cell Denver #21 recleaners.

The four cell recleaners have two standard mechanisms and two DR units. The concentrate enters the #1 cell. The first three cells are pulled for concentrate, the fourth cell goes to #2 cell for retreatment.

Second Stage cleaning is essentially a pyrite - pyrite with chalcopyrite inclusions step. This separation is accomplished by a very judicious addition of Cyanide and lime at 0.002 and 0.066 pounds per ton. No excess xanthate is carried over into the cleaning circuit.

................Cont'd...

Traces of free cyanide ion will easily depress iron minerals. The Cyanide ion in our case is used to react with the cations of the heavy metals.forming insoluble salts.

At the low pH 8:0 - 9:1 it is unlikely that the OH-ion strips much xanthate from the pyrite. This would suggest that the Ca+ ion is selectively removing enough xanthate from the clean pyrite, and not from the chalcopyrite inclusions, to allow this very delicate separation to proceed. Copper concentrate contains large amounts of pyrite with chalcopyrite inclusions. Cleaner tailing is predominately clean, bright pyrite with very little copper.

It is ironic that we have resorted to the use of the very process that plagued the operation for so long to make this very successful cleaning separation.

TABLE III

Reagent Consumption per long ton
On a Yearly Basis for Three Typical Periods

| Reagent | 1973 | 1967 | 1965 |
|---|---|---|---|
| Sodium Isopropyl Xanthate | 0.024 | 0.115 | 0.058 |
| Reagent 238 | 0.010 | 0.007 | 0.013 |
| Reagent 242 | 0.001 | 0.001 | ----- |
| Aerofloat 33 | 0.005 | 0.0016 | 0.001 |
| Aerofroth 65 | 0.016 | 0.023 | 0.025 |
| Z-200 | 0.005 | 0.004 | ----- |
| Soda Ash | 0.048 | ----- | ----- |
| Sodium Cyanide | 0.002 | 0.0002 | 0.0002 |
| Lime | 0.066 | 0.45 | 0.71 |

Reagent Consumption has been reduced in full use of sea water.

## COPPER FILTRATION:

The cupriferous pyrite rougher concentrate is flakey material and resistant to flow. It plugs up the thickener underflow line as the fluid is drawn through the platey material.

Copper concentrates are pumped to a cyclone at the copper filter. The underflow goes directly into the filter bath. The overflow to the 10' x 30' copper thickener for thickening.

Filter cake is 24% copper, 40% + 325 Mesh, 0.12% Chlorides, 9% Moisture. Thickener overflow water returns to the rougher concentrate sump, filtrate goes to waste.

.................Cont'd.....

SOLUBILITY:

Plant scale tests were run to determine the solubility of our copper in sea water. The maximum copper detected was 0.25 ppm. The average 0.004 ppm. Copper values while milling in fresh water were so similar we concluded there was no appreciable solution of copper during processing. This is not true for several mines in B. C.

CORROSION:

Corrosion is severe because there is no stifling by corrosion products, but it is less severe in sea water than in fresh or fresh-sea mixtures.

CORROSION GRINDING MEDIA:

We anticipate a sharp increase in the grinding media and liner steel consumption with sea water in contact with fresh abraded iron and steel.

TABLE IV

Ball and Rod Consumption

| Year | Total Tons | Milled Tons | Conc. Tons | Lbs. Balls | Rods |
|---|---|---|---|---|---|
| 1958 | 623,402 | 493,727 | 337,945 | 0.990 | ----- |
| 1959 | 873,204 | 502,125 | 348,615 | 0.877 | ----- |
| 1962 | 992,312 | 821,913 | 585,525 | 0.545 | 0.612 |
| 1963 | 855,675 | 750,704 | 435,771 | 0.524 | 0.654 |
| 1965 | 1,150,653 | 845,805 | 562,368 | 0.694 | 0.510 |
| 1966 | 1,145,543 | 840,257 | 594,951 | 0.574 | 0.637 |
| 1969 | 1,222,176 | 858,396 | 530,895 | 0.502 | 0.710 |
| 1971 | 1,075,517 | 753,155 | 473,291 | 0.880 | 0.741 |
| 1972 | 1,047,605 | 782,677 | 490,666 | 0.784 | 0.702 |

In 1970 we installed another regrind ball mill which wholly accounts for the increased steel consumption.

CORROSION ROD MILL:

The 9' x 14' rod mill is a low overflow type. End liners are manganese, shell plates chrome molly, wedge bars now one piece manganese steel.

The last two discharge end liner changes were made prematurely to prevent wear on the millhead from racing. A set of ni hard liners of new design were installed 12 months ago and show very little wear. We anticipate 4,000,000 tons from these liners.

Shell plates have been changed three times in 12 1/2 years. The last set 2/3 worn due to loose wedge bars.

.....................Cont'd......

Wedge bar life has been uniform. The change from fresh to sea water had no influence on liner consumption.

TABLE V

Rod Mill Liner Life

| Months Service | | Steel | Long Tons Milled |
|---|---|---|---|
| Feed End Liners | 34 | Manganese | 2,248,570 |
| | 60 | Manganese | 4,430,305 |
| | 54 | Manganese | 3,541,866 |
| Discharge End | 24 | Manganese | 1,551,160 |
| | 55 | Manganese | 3,835,378 |
| | 28 | Manganese | 1,817,334 |
| (In Service) | (12) | Ni Hard | (738,333) |
| Shell Plates | 57 | Manganese Steel | 3,783,618 |
| | 42 | Manganese Steel | 2,984,839 |
| | 36 | Manganese Steel | 2,112,033 |

TABLE VI

Rod Mill Wedge Bars

| | | | |
|---|---|---|---|
| Lifter (Wedge) Bars | 12 | Caps. Mang. Steel | 919,639 |
| | 15 | | 988,132 |
| | 22 | | 1,052,632 |
| | 26 | 1 Piece Lifter Disch. End | 1,322,833 |
| | 18 | 1 Piece Lifters | 1,238,854 |
| | 22 | | 1,015,016 |
| | 21 | | 1,365,962 |

CORROSION GRATE BALL MILLS:

No sea water entered the grinding circuit until 1960. During the first years of operation grate and pan liner wear was excessive. It was obvious from the worn parts that electrolytic corrosion and not abrasion was the cause of the failure. We increased the nickel content of our liners and placed 3/8" stainless steel shims under the grates. The problem disappeared. We worried a little about the possible new locatiion of the anode.

Feed end liners were manganese steel and lasted 20 months, milling 384,000 tons; 453,400 tons; and 480,300 tons, respectively.

Number one mill was rubber lined in 1968.

Inner grates are 18-CW. Their life depends on the condition of the face wedge bars. The following table will indicate the effect of sea water on the grates:

......................Cont'd....

TABLE VII

Grate Ball Mill Wedge Bars

| Year | Tons Milled | Sea Water Ratio |
|---|---|---|
| Nov. 1959-Dec. 1960 | 320,254 | 30% Sea Water |
| 1960-Mar. 1961 | 166,219 | (Grates too hard) |
| 1961-Nov. 1962 | 626,038 | 50% Sea Water |
| 1962-Sept. 1964 | 735,912 | 50% Sea Water |
| 1964-Aug. 1966 | 852,593 | 60% Sea Water |
| 1966-Feb. 1968 | 692,193 | 70% Sea Water |
| 1968-Sept. 1970 | 518,105 | 100% Sea Water |
| 1970-July 1971 | 625,946 | 100% Sea Water |
| 1971-Dec. 1973 | 917,414 | 100% Sea Water |

Closed circuit fine grinding for copper ore has been required for 15% of the tons milled during the past two years. Liner steel consumption is down in primary grinding with the full use of sea water.

REGRIND BALL MILL 8' x 12':

We now know that corrosion is very severe with large amounts of pyrite in aerated sea water. Mixtures of sea and fresh water are more corrosive than all sea water.

This regrind mill was designed to be operated in sea water, grinding the highly pyritic tailing and the platey cupriferous pyrite from the rougher concentrate.

Liner wear was normal but bolt maintenance was so excessive that the mill was converted to rubber in 1966 using standard profile Skega. Shell Plates are still in place.

REGRIND MILL 8' x 10' CANADIAN ALLIS-CHALMERS:

This mill was installed with a half worn set of Ni Hard liners. These liners lasted 20 months and were changed only because it was not practical to change single worn liners. The service life was far in excess of what we anticipated. The mill was rubber lined.

CORROSION - FILTERS:

The original Eimco 6' x 3 ring Taconite iron filter required a new tank every 1 1/3 years. The bath completely corroded away. The valve was changed every two years.

This filter was overhauled and used for a copper concentrate filter in 1961. The tank was changed once in 1967, no further changes are expected.

A 6' x 6 ring Eimco Taconite filter with snap blow was installed in 1961. The tank front was changed after 5 years. No sign of wear has occurred during the past 7 years.

................Cont'd...

Filter valves have been refaced three times in 12 1/2 years. Service life was 29 months, 76 months, the present heads have been on for 45 months. We do not anticipate any repair in the immediate future.

FLOTATION TANKS:

Flotation tanks using fresh water holed through after 8 months operation. The bottom 8" of the tanks were lined with planking and the steel brushed with concrete. No further problems have developed.

MARINE LIFE:

Marine life is always a problem when using sea water. The foot valves, intake, delivery lines, and the holding tanks rapidly build up with barnacles, mussels, oysters, clams, starfish, sea urchins, and sea worms up to 18" long and 1/2" or more in thickness.

This marine life can easily be destroyed but most of it remains fastened to the place of growth for long periods of time making an excellent site for new growth. When these crustations do come loose they drift into pipe lines and cut off the flow of water. They are particularly troublesome in magnetic separator spiggots.

When the mill was operating 5 days a week the tanks and lines were back filled with fresh water during shut down. This was quite effective until the foot valves began to wear out very rapidly. Replacement was a major operation.

We have four 500 G/M Johnson deep well pumps on mill water supply and one for fire protection. Two pumps are connected to each of the two 8" delivery lines to the 50,000 gallon storage tank 185' above sea level and 20 feet above the top level of the mill. This tank is inter-connected, top and bottom with a 100,000 gallon tank formerly used for fresh water.

The intake half of the sea water tank is screened off to a depth of 10 feet to prevent transient shells from going directly into the discharge pipes. A door is fitted into this half of the tank, close to the bottom, for periodic clean out.

The mill is supplied by two 8", three 6", and one 4" water line. Each pipe system is run full size to the lowest work area in its' respective location. Each line terminates with a quick opening valve or pipe cap. A 2" nipple with a plug valve and air line fitting is welded into the line.

When a line begins to slow up cn water delivery the section is shut down, and a high pressure air hose connected to the nipple at the bottom of the line.

.................Cont'd...

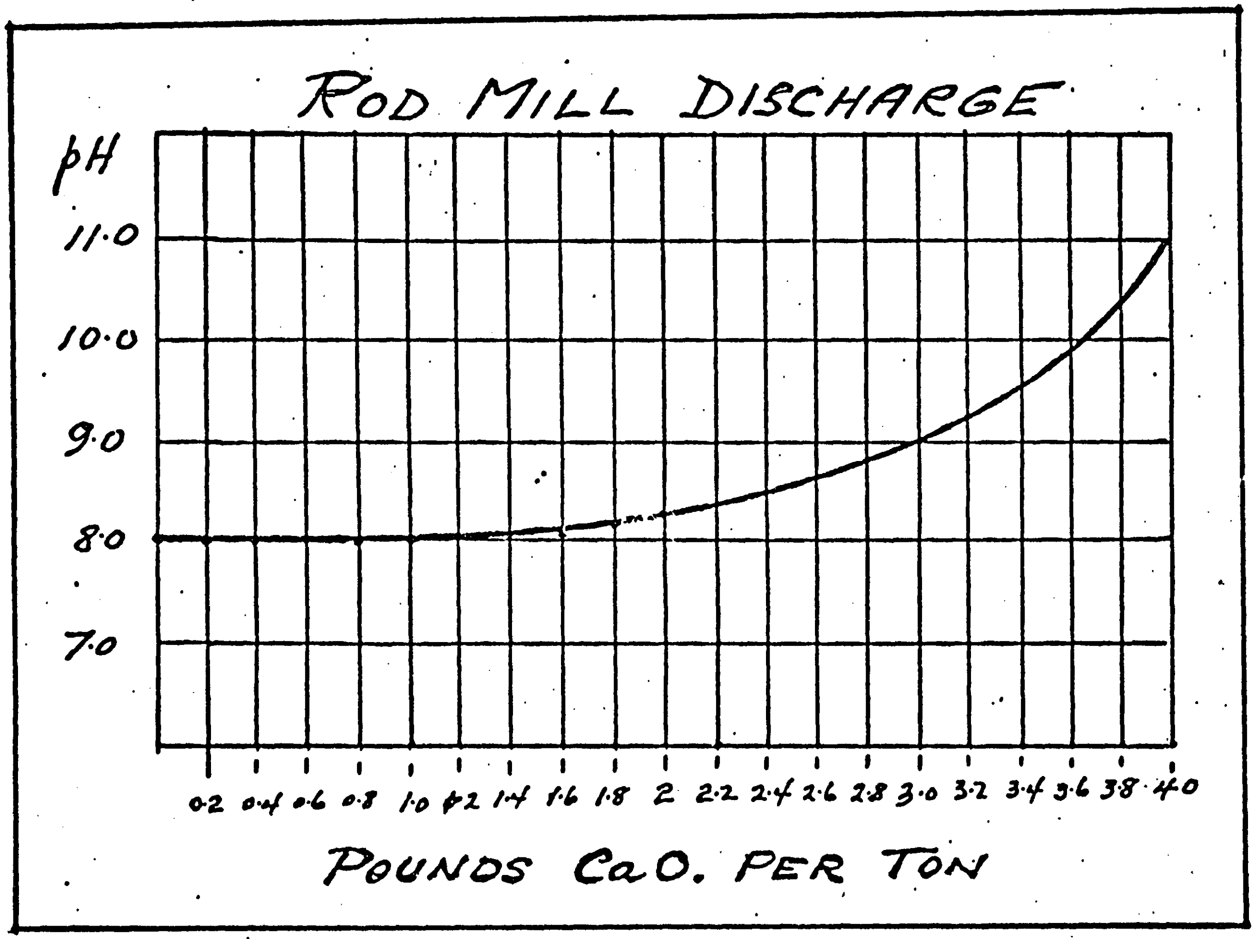
Several quick hard air blows are made into the water filled pipe back to the tanks then the bottom drain is quickly opened. If there is any plug up, the process is repeated. Shell removal is excellent and complete.

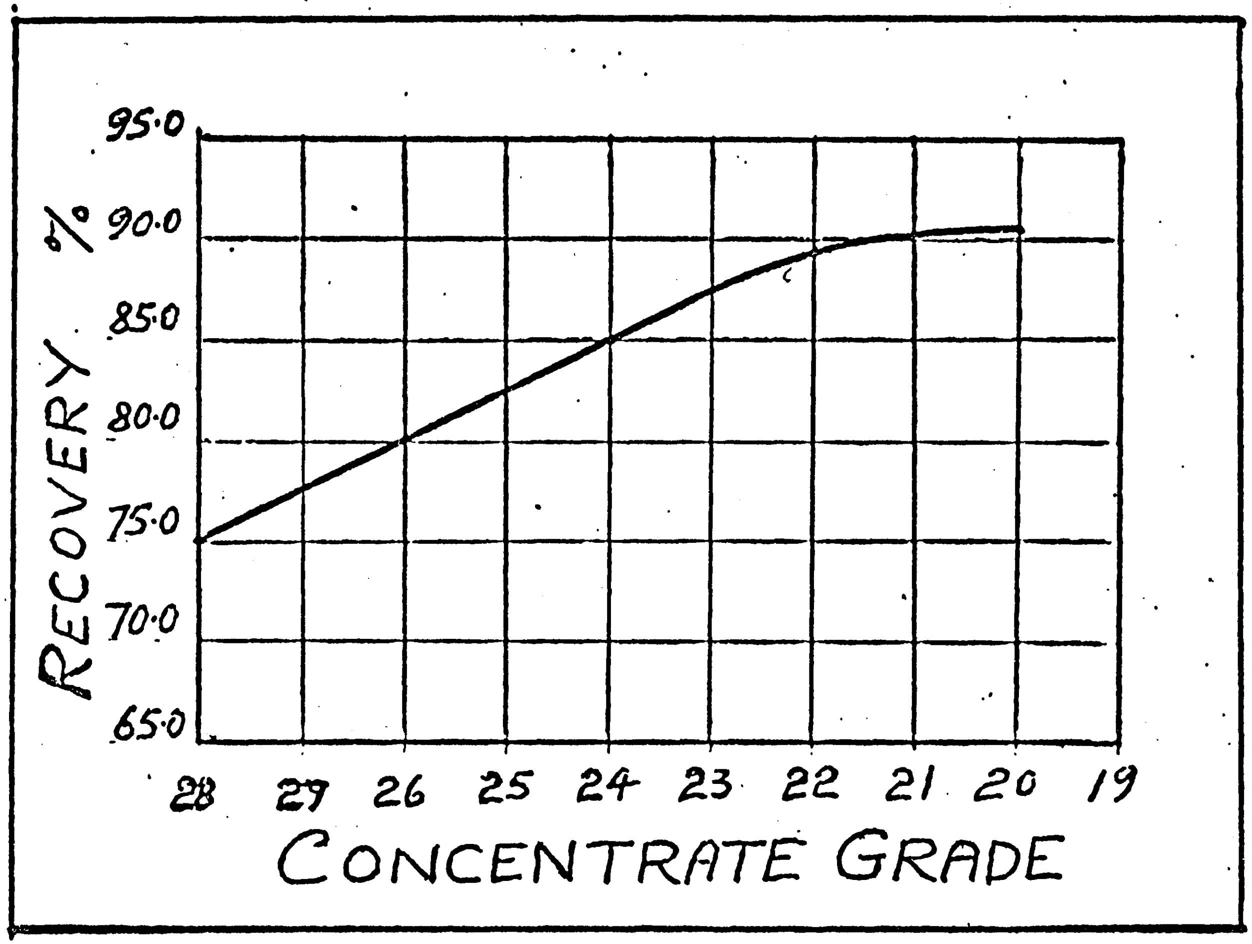
We suggest the shells be removed from the area immediately as Aerofloat smells like a rose compared to a pile of rapidly decomposing marine life.

Summing up the 17 years of operation we find that the full use of sea water gives us a very stable easily controlled float. Reagent consumption is greatly reduced and reagent feeding has been simplified. The natural pH of the sea water is the most satisfactory level for Texada ores. It is extremely important to float certain types of pyrite. This is best done at low pH.

After making allowance for fine grinding, steel and liner consumption is less in full sea water than it was in fresh and mixtures of sea and fresh water. This is probably due to increased ionization of the diluted sea water.

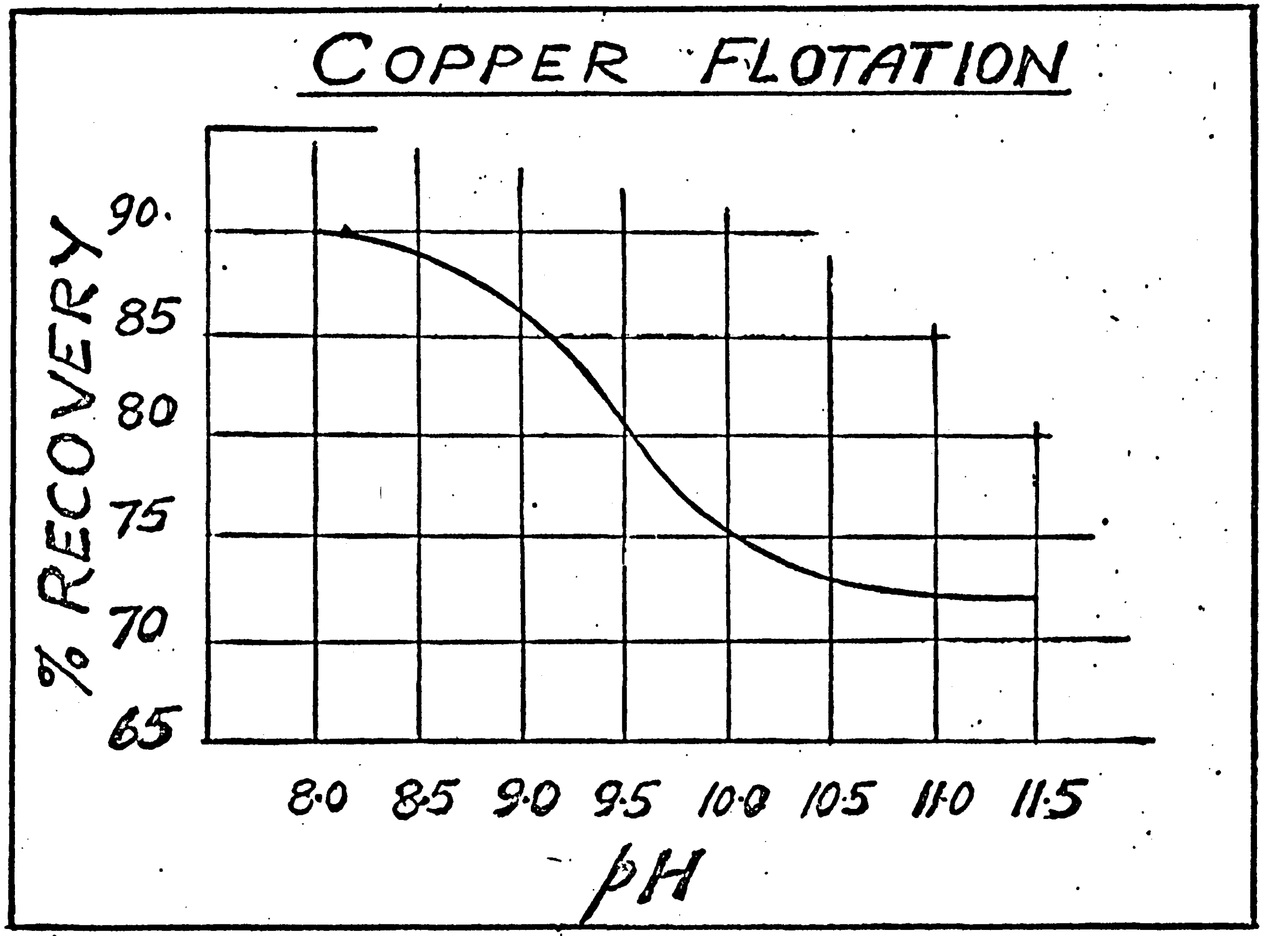
Atmospheric corrosion is severe and causes more problems than the sea water in the processing. This is a location problem and not a function of the use of sea water.


ROD MILL DISCHARGE
pH
11.0
10.0
9.0
8.0
7.0
0.2 0.4 0.6 0.8 1.0 1.2 1.4 1.6 1.8 2 2.2 2.4 2.6 2.8 3.0 3.2 3.4 3.6 3.8 4.0
POUNDS CaO. PER TON


RECOVERY %
95.0
90.0
85.0
80.0
75.0
70.0
65.0
28
27
26
25
24
23
22
21
20
19
CONCENTRATE GRADE

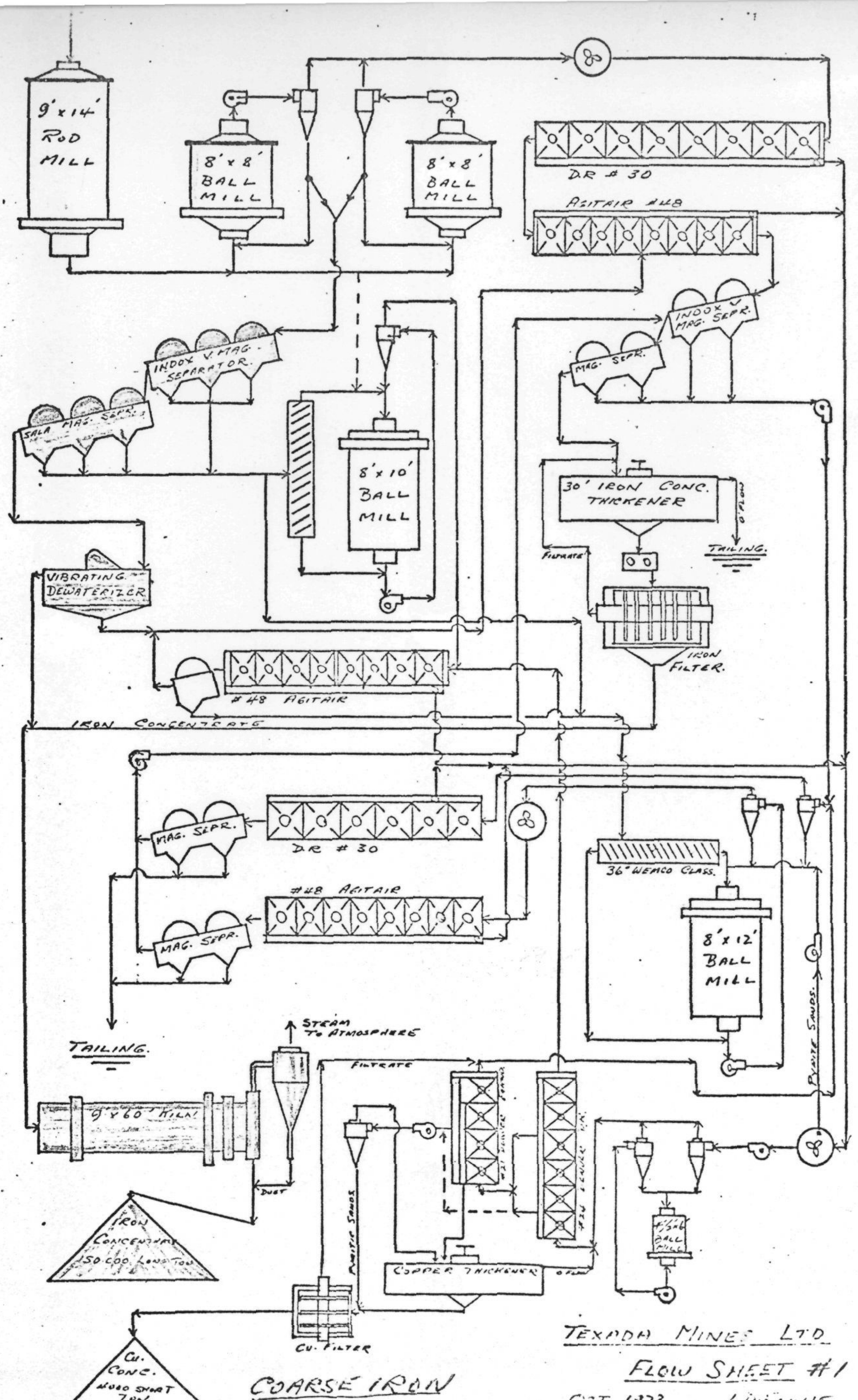
GRAPH #2

5.16


COPPER FLOTATION
% RECOVERY
90.
85
80
75
70
65
8·0
8·5
9·0
9·5
10·0
10·5
11·0
11·5
pH

GRAPH #3

5.17



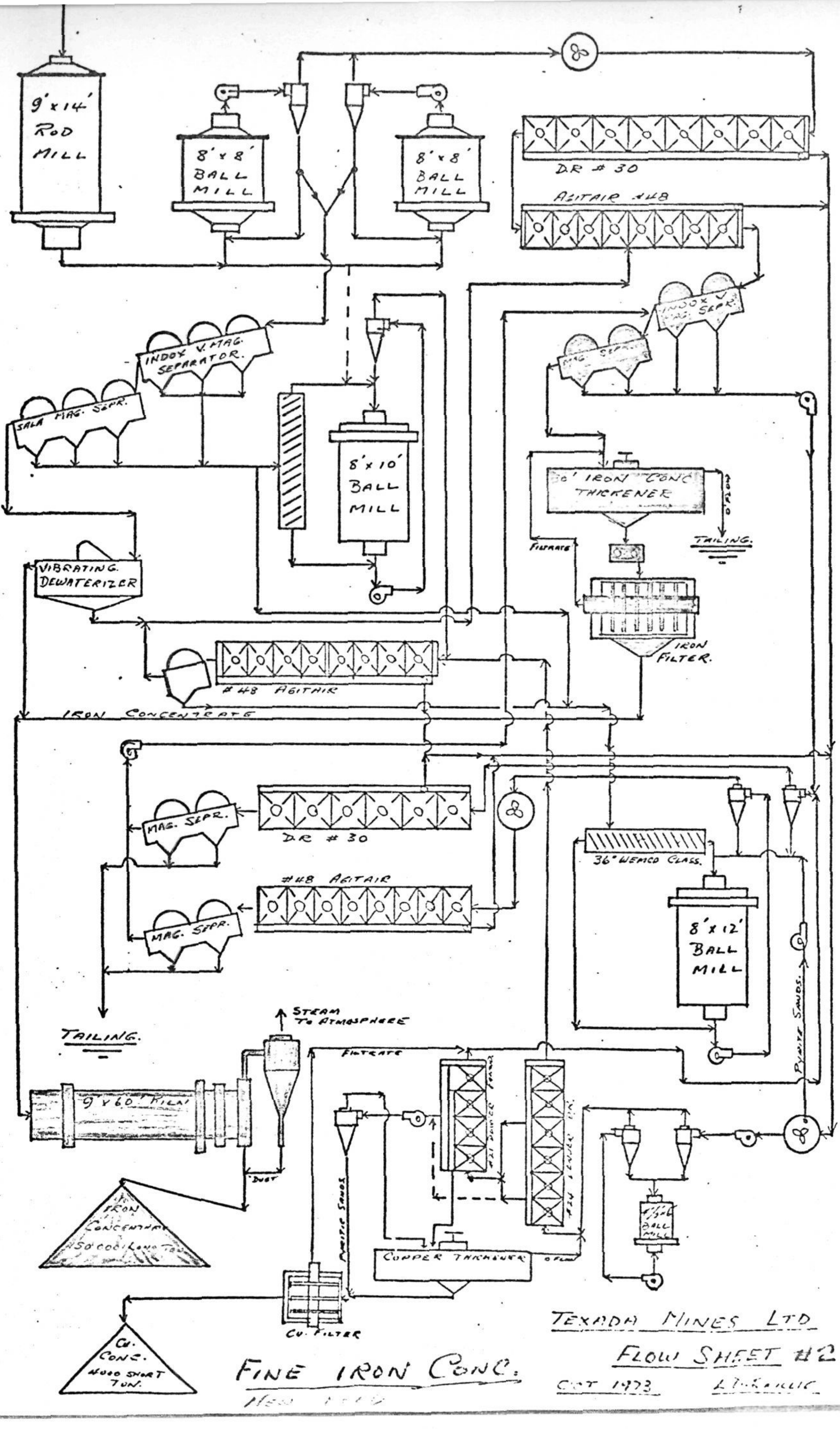

COARSE IRON

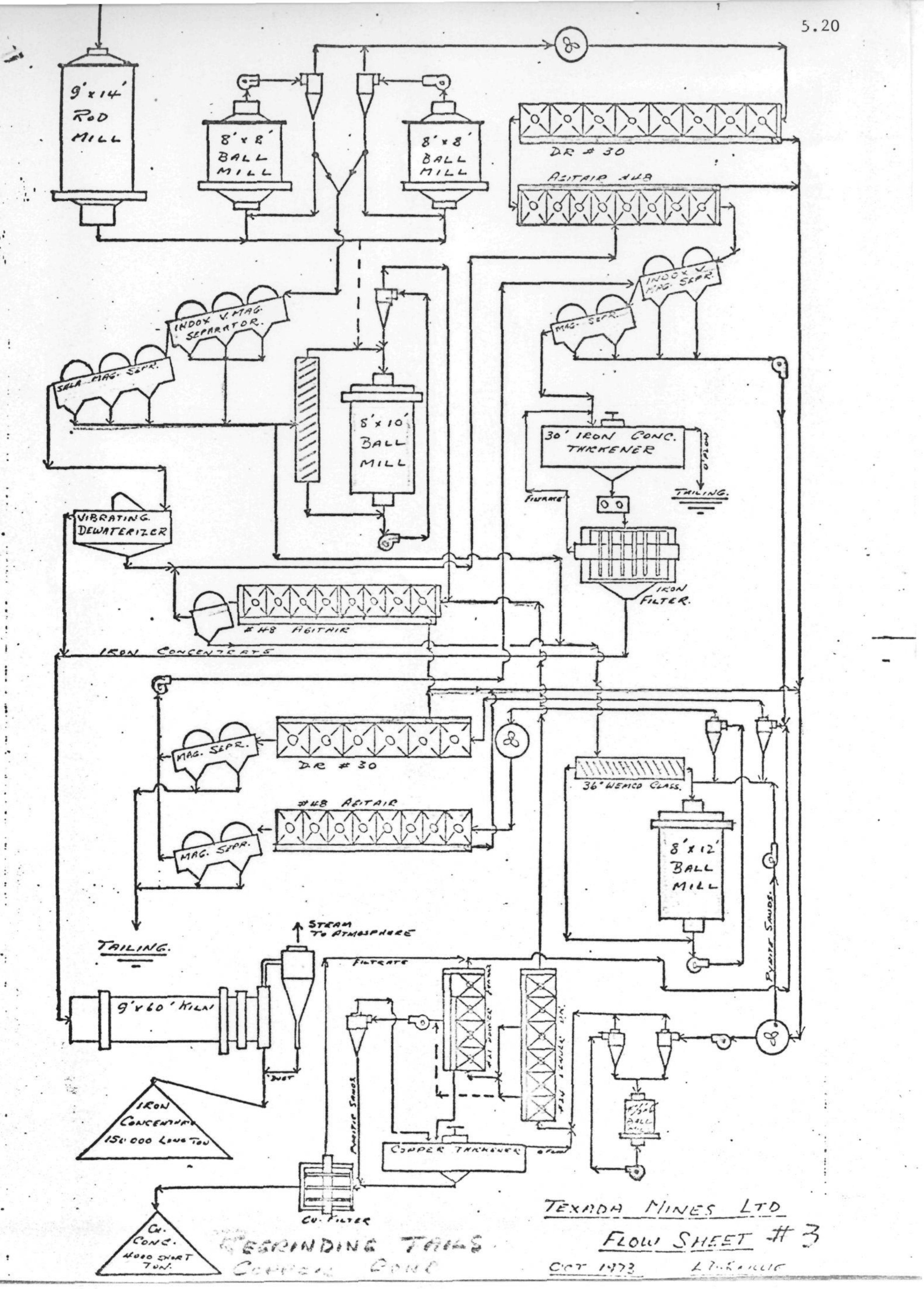
NEW FEED

TEXADA MINES LTD

FLOW SHEET #1

OCT 1973


9'x14' ROD MILL
8'x8' BALL MILL
8'x8' BALL MILL
DR # 30
AGITAIR #48
INDOX V. MAG. SEPARATOR.
SALA MAG. SEPR.
INDOX V. MAG. SEPR.
MAG. SEPR.
8'x10' BALL MILL
VIBRATING DEWATERIZER
36' IRON CONC THICKENER
O'FLOW
TAILING.
FILTRATE
IRON FILTER.
#48 AGITAIR
IRON CONCENTRATE
MAG. SEPR.
DR # 30
#48 AGITAIR
MAG. SEPR.
36" WEMCO CLASS.
8'x12' BALL MILL
PYRITE SANDS.
TAILING.
STEAM TO ATMOSPHERE
FILTRATE
9'x60' KILN
DUST
IRON CONCENTRATE 50,000 LONG TON
PYRITIC SANDS
LOWER PR.
24 UPPER PR.
BALL MILL
COPPER THICKENER
O'FLOW
CU. FILTER
CU. CONC. 4000 SHORT TON.
TEXADA MINES LTD
FLOW SHEET #2
OCT 1973
FINE IRON CONC.


9'x14' ROD MILL
8'x8' BALL MILL
8'x8' BALL MILL
DR # 30
AGITAIR #48
INDOX V. MAG. SEPR.
MAG. SEPR.
INDOX V. MAG. SEPARATOR.
SALA MAG. SEPR.
8'x10' BALL MILL
30' IRON CONC. THICKENER
O'FLOW
FILTRATE
TAILING.
VIBRATING DEWATERIZER
IRON FILTER.
#48 AGITAIR
IRON CONCENTRATE
MAG. SEPR.
DR # 30
#48 AGITAIR
MAG. SEPR.
36" WEMCO CLASS.
8'x12' BALL MILL
TAILING.
STEAM TO ATMOSPHERE
FILTRATE
PYRITE SANDS.
9'x60' KILN
DUST
IRON CONCENTRATE 150,000 LONG TON
PYRITE SANDS
COPPER THICKENER
O'FLOW
CU. FILTER
CU. CONC. 4000 SHORT TON.
REGRINDING TAILS
COPPER CONC
TEXADA MINES LTD
FLOW SHEET #3
OCT 1973

Average grades for Copper and Iron concentrates shipped from September, 1973 to February, 1974 inclusive:-

| | Copper conc. | Iron conc. |
|---|---|---|
| September, 1973 | 25.06 % Cu Dry | 64.44 % Fe Dry |
| October | 21.65 | 65.24 |
| November | 20.94 | 65.20 |
| December | 21.44 | 64.21 |
| January, 1974 | 22.44 | 64.09 |
| February | 22.35 | 63.06 |
| | 6 ) 133.88 | 6 ) 386.24 |
| | 22.31 % Cu Dry | 64.37 % Fe Dry |

## IRON CONCENTRATE

Value of iron concentrate is $9.12 (U.S.) per dry metric ton F. O. B. Texada Island based on the ore containing sixty-two per cent 62% iron and twelve-one hundredths of one per cent copper.

For each additional percentage point above 62% iron there is a bonus or premium of 40 cents.

For each one-hundredth of a percentage below 0.12% contained copper a premium of 5 cents is paid.

The average grade of iron concentrate shipped contains approximately 64% Fe and 0.07% Cu

Therefore, the value of 1 metric ton of iron concentrate shipped is:-

| | | | | |
|---|---|---|---|---|
| | | | | $9.12 |
| plus 2 | x | 40 cents | (iron bonus) | .80 |
| plus 5 | x | 5 cents | (copper bonus) | .25 |
| | | | | $10.17 U. S. |

Copper L. M. E. = $0.80 /lb

Grade of Copper concentrate 22.31% Cu Dry

| | | | | |
|---|---|---|---|---|
| 2204.6 (D.M.T.) | x (22.31% - 1.10%) = | | | |
| 2204.6 | x | .2121 | = 467.6 lb. of copper | |
| 467.6 | x | (80 - 4.5 - (30 x .3) = | | |
| 467.6 | x | (80 - 13.5) = | | |
| 467.6 | x | 66.5 cents = | | $310.89 |

Smelting charge $28.00 D.M.T. $28.00

Freight $35.00 W.L.T. (10% moisture)

$$\frac{35.00}{.9} = \$38.89 \text{ D.L.T.}$$

$$38.89 \times \frac{2204.6}{2240} = \$38.28$$

| | | |
|---|---|---|
| Insurance | 1.00 | |
| | $67.28 | 67.28 |
| | | $243.61 |

Value of 1 dry metric ton of copper concentrate (22.31% Cu) when considering L.M.E. price of $0.80 per pound for copper = $243.61.

Copper L. M. E. $1.00 /lb

Grade of Copper concentrate 22.31% Cu Dry

| | | | | |
|---|---|---|---|---|
| 2204.6 (D.M.T.) | x | (22.31% - 1.10%) | = | |
| 2204.6 | x | .2121 | = 467.6 lb of copper | |
| 467.6 | x | (100 - 4.5 - (50 x .3)) | = | |
| 467.6 | x | (100 - 19.5) | = | |
| 467.6 | x | 80.5 cents | = | $376.42 |

| | | | |
|---|---|---|---|
| Smelting charge | $28.00 D.M.T. | $28.00 | |
| Freight $35.00 W.L.T. (10% moisture) | | | |
| | $\frac{35.00}{.9}$ = $38.89 D.M.T. | | |
| | $38.89 \times \frac{2204.6}{2240}$ = | $38.28 | |
| Insurance | | 1.00 | |
| | | $67.28 | 67.28 |
| | | | $309.14 |

Value of 1 dry metric ton of copper concentrate (22.31%) Cu when considering L.M.E. price of $1.00 per pound for copper = $309.14.

Copper L. M. E. $1.20

Grade of Copper concentrate 22.31% Cu Dry

2204.6 (D.M.T.) x (22.31% - 1.10%) =

2204.6 x .2121 = 467.6 lb of copper

467.6 x (120 - 4.5 - (70 x .3)) =

467.6 x (120 - 25.5) =

467.6 x 94.5 cents = $441.88

Smelting charge $28.00 D.M.T. $28.00

Freight $35.00 W.L.T. (10% moisture)

$$\frac{35.00}{.9} = 38.89 \text{ D.L.T.}$$

$$38.89 \times \frac{2204.6}{2240} = \$38.28$$

Insurance 1.00

$67.28 67.28

$374.60

Value of 1 dry metric ton of copper concentrate (22.31% Cu) when considering L.M.E. price of $1.20 per pound for copper = $374.60.

This AGREEMENT entered into as of September 30, 1972

Appendix 7

| | |
|---|---|
| between (Seller) | TEXADA MINES LTD.<br>Texada Island, British Columbia<br>Canada |
| and (Buyer) | MITSUBISHI CORPORATION<br>No. 6-3, Marunouchi 2-chome, Chiyoda-ku, Tokyo<br>Japan |

WHEREBY Seller agrees to sell and Buyer agrees to buy all copper concentrates produced by Texada Mines Ltd. at Texada Island from October, 1972, through September 30, 1976, delivered CIF-FO Japanese port, upon the terms and conditions hereinafter set forth.

1) DEFINITIONS:

As used herein, unless otherwise expressly specified, the word

a) "Ton" shall mean a metric ton of 1,000 kilograms or 2,204.6 pounds avoirdupois.

b) "Ounce" shall mean a troy ounce or 31.1035 grams.

c) "Unit" shall mean one hundreth (1/100) of one dry ton of 1,000 kgs.

d) "Dollar" and "Cents" shall mean the United States of American dollar.

e) "Year" shall mean the contract year October 1 through September 30.

f) CIF-FO shall mean delivery cost, insurance, freight according to "Incoterms 1953" and FO meaning Free Out (free of costs of unloading).

2) MATERIAL ASSAY:

Approximate typical assay:

| | | |
|---|---|---|
| Cu | : | 23% (Min. 20%) |
| Au | : | 3-7 g per ton |
| Ag | : | 150-225 g per ton |
| S | : | 35% |

3) QUANTITY:

The total production of Texada copper concentrates from October 1972 through to the end of September 1976. The present estimated quantity is 7,000 - 12,000 tons per year. Copper production quantities vary and are subject to the iron ore production.

Seller shall regularly advise Buyer of its production schedule.

4) SHIPMENT:

There shall be two to four shipments per year in a lot of 2,500 - 3,500 tons per vessel at Seller's option. Seller shall advise Buyer of the definite schedule.

5) DELIVERY:

Delivery shall be made in bulk CIF-FO Onahama or Kawasaki (Toyo Wharf) or Naoshima at Buyer's option. Buyer shall declare to Seller discharge port of each shipment within 10 days from the receipt of Seller's shipping advice.

Discharge rate shall be 1,500 wet long tons per weather working day, Sundays and holidays excepted unless used. If used, actual time used to count.

The rate of dispatch and demurrage shall be mutually agreed upon. Stowage in deep tanks shall not be accepted by Buyer.

6) PRICE:

The price for copper concentrates to be shipped from October 1, 1972, through to September 30, 1974, shall be as follows:

Copper: Full content less one point one (1.1) units of copper per ton of concentrate shall be paid for at the London Metal Exchange Official (morning session) price-per-pound quotation for Electrolytic Copper Wirebars Settlement less refining charge of four point five (4.5) cents per pound of payable copper.

**Price Participation:** In case the aforementioned London Metal Exchange Settlement quotation exceeds fifty (50) cents per pound , then before deducting Refining Charge, zero point three (0.3) cents per each one (1) cent in excess of fifty (50) cents per pound shall be added to the Refining Charge (R/C) of four point five (4.5) cents per pound.

**Gold:** No payment shall be made if the gold contents are 1 gram per ton or less. If over 1 gram per ton, the gold shall be paid for at the following rate at the means of the monthly averages of initial and final London prices as published in the "Metals Week" for the quotational period, less $1.00 per troy ounce of payable gold.

If over 1 gram per ton, 90% of the gold contents.
If over 2 grams per ton, 94% of the gold contents.
If over 3 grams per ton, 95% of the gold contents.

**Silver:** No payment shall be made if the silver contents are 30 grams per ton or less. If over 30 grams per ton, 90% of the full silver contents shall be paid for at the Handy & Harman New York price for unrefined silver-bearing materials which is at a discount, currently one cent from the published prices as quoted in "Metals Week" for the quotational period.

**Treatment Charge:** $28.00 per DMT CIF-FO

7) NEW PRICE:

Within not less than six months prior to September 30, 1974, Buyer and Seller shall commence negotiations on new prices and terms for the two remaining contractual years (October 1, 1974, to September 30, 1976), subject to the iron ore production.

8) QUOTATIONAL PERIOD:

All quotations for copper, gold and silver used in final settlement shall be the arithmetical average prices for the calendar month in which falls the date of the arrival of carrying vessel at the port of discharge at which the material shipped under this contract is discharged. The date of arrival shall be the date as mentioned in the customs declaration.

All sterling quotations shall be converted into dollars at the sterling exchange rate as published in the "Metals Week", averaged over the quotational period.

9) CURRENCY ADJUSTMENT:

In case of an official change of official standard rate between Yen and US dollar (currently ¥308/$1.00) during the life of this contract, the Treatment Charge (T/C) and Refining Charge (R/C) for copper shall be adjusted in proportion to such change.

The adjustment for T/C and R/C shall be applied to all unshipped quantity of concentrates at the time of the official standard rate change.

10) PAYMENT:

Ninety percent (90%) of the provisional invoice value based on Bill of Lading weights and assays shall be made available to Seller 60 days after the vessel's arrival at the port of discharge.

Soon after the shipment is made, Seller shall present to Buyer full set of clean on board Ocean Bills of Lading marked freight prepaid together with Insurance Policy, Seller's provisional invoice, assay certificate and weight certificate.

Balance shall be paid full against Seller's final invoice soon after determination of settlement prices, weights, and assays.

If any provisional payment exceeds the amount of such final invoice, then Seller shall refund promptly the difference to Buyer.

11) <u>WEIGHING, SAMPLING, AND DETERMINATION OF MOISTURE</u>:

At shipping port, the material shall be weighed and a determination of moisture be made in order to establish the net dry shipped weight hereunder. The net wet shipped weight shall be established by <u>draft survey conducted by Surveyors</u>, and the moisture determination shall be made by Seller under the supervision of Surveyors and using the sample to be taken at the time of loading the materials into ocean-going steamer.

All necessary expenses for such weighing and moisture determination are for Seller's account. Buyer shall have the right to be represented at these operations for his account. Seller shall furnish Buyer with Surveyors' shipped weight and moisture certificates at Seller's expense.

The material shall then be weighed, sampled, and determination of moisture be made at the receiving smelters in Japan in the usual technical manner at Buyer's expense. The weight and moisture figures thus established shall govern as final.

Seller may appoint Overseas Merchandise Inspection Co., Ltd., as Seller's representative, at Seller's expense, at the weighing, sampling, and moisture determination carried out at receiving smelters. Buyer shall take into consideration any of the reasonable suggestions by Overseas Merchandise Inspection Co., Ltd., in carrying out such operations.

12) <u>ANALYSIS:</u>

For copper, gold, and silver shall be made independently by Seller's representative and the receiving smelter, and results shall be exchanged promptly in the customary manner.

Should the difference between Seller's and Buyer's result of analysis be not more than:

| | | |
|---|---|---|
| Cu | : | 0.3% |
| Au | : | 0.5 grams per dry ton |
| Ag | : | 30.0 grams per dry ton |

the exact mean of the two results (for each element) shall be taken as the agreed result for settlement. In the case of any greater difference, the umpire sample which shall have been sealed by representatives of both parties at the time of sampling may be referred, at the request of either party, to:

Ledoux & Company
359 Alfred Avenue, Teaneck, New Jersey

If the umpire's result comes between the other two or coincides with either it shall be taken as the agreed result; otherwise, the agreed result for the purpose of final settlement shall be the middle of the three.

The cost of the umpire analysis shall be borne by the party whose result is further from the umpire's result, but should the umpire's result be the exact mean of that of the Buyer and that of the Seller, then the cost of the umpire's analysis shall be borne equally by the both parties.

13) DISABILITY:

In the event of any strike, act of God, war, force majeure, lockout, combination of workmen, interference of Trade Unions, act of Government or Government appointed agents, suspension of labor, fire, accident, lack of railroad or seaborne freight facilities or delays en route, refusal of import or export licenses, or of any other cause whatsoever beyond the control of Seller or of Buyer whether of the foregoing nature or not, preventing or hindering them from giving delivery or receiving or smelting respectively, then delivery under this agreement shall be suspended during such time, provided always that written notice shall be given of any inability by either of the parties contracting hereto.

This clause shall not relieve Buyer of his obligation to receive delivery in the case of such material as is actually on board ocean-going vessel at the time of giving or receiving notice of force majeure. However, should such event that prevents, delays or interrupts the performance of either party continue more than six months, both parties may negotiate and mutually decide to terminate this contract or a part thereof or vary the terms of this contract.

14) INSURANCE:

Seller shall procure at Seller's expense and/or self-insure against All Risks of carriage involved in the contract, including heating and spontaneous combustion, war risk, strikes, riots, and civil commotions from F.O.B. ocean-going vessel at shipping port upto the receiving smelters' yards in Japan for a value calculated on the provisional invoice value plus 10% which, however, shall be adjusted later so as to cover 110% of the settlement amount calculated in accordance with the provisions of the "settlement in case of loss" clause.

15) SETTLEMENT IN CASE OF LOSS:

In the event of total loss payment shall be made for 100% of the value based on the arithmetical average process as quoted in the "Metals Week" for the calendar month in which falls the date of the ocean Bill of Lading and on the basis of Bill of Lading weights and analysis of the Seller's provisional invoice.

In the event of partial loss, payment shall be made on basis of the weights as per Seller's provisional invoice, less normal transit loss determined by the insurance underwriters, the analysis of that part of the material which has been safely delivered at receiving smelters and the metal prices of the quotational period.

Seller shall render every assistance to Buyer in obtaining settlement from underwriter.

16) TAXES AND DUTIES:

All taxes and duties whether existing or new on copper concentrates and contained metals or commercial documents relating thereto shall be borne by Buyer in the event of their existence in any country in which this material is discharged. Any export tax or freight tax or duty whether existing or new levied in the country of origin shall be borne by Seller.

17) ARBITRATION:

A dispute which may arise under this contract and which cannot be disposed of by the mutual agreement, shall be decided by arbitration in Japan. Arbitrators may be initiated by either party giving thirty days' notice in writing to the other of commencement of arbitration proceedings. Thereupon, a board of three (3) arbitrators shall be appointed, of whom one shall be chosen by Buyer, one by Seller, and third by two so chosen.

If either Buyer, or Seller fails to choose an arbitrator within fourteen (14) days after receiving notice of commencement of arbitration proceedings or if two of the arbitrators selected cannot agree upon a third arbitrator within fourteen days after they have been chosen a representative of the International Chamber of Commerce in Japan will, upon request of either party, appoint the arbitrator or arbitrators required to complete the board. The decisions of the majority of the arbitrators shall be final and binding upon both parties. Expense of arbitration shall be borne by losing party.

18) LICENSE:

This contract is subject to Canadian Export License and Japanese Import License.

19) SUCCESSION:

This agreement shall bind and inure to the benefit of the parties, their legal representatives, successors, and assigns.

IN WITNESS HEREOF the parties hereto have caused these presents to be signed on their respective behalves effective the Day and Year first written above.

Seller: TEXADA MINES LTD. Buyer: MITSUBISHI CORPORATION

Edward Vickers [signature] [signature]

7.10

# MITSUBISHI CORPORATION
(MITSUBISHI SHOJI KAISHA, LTD.)

TELEX: Nos. TK2222 to TK2228 ANSWERBACK CODE MITUBISI TK

6-3 MARUNOUCHI 2-CHOME CHIYODA-KU, TOKYO, JAPAN

MAIL ADDRESS C.P.O. BOX 22, TOKYO 100-91 JAPAN

PHONES: TOKYO (210) 2121

CABLE ADDRESS FOR GENERAL USE: MITSUBISHICORP TOKYO

CABLE ADDRESS FOR THIS BUSINESS:

YOUR REF. IN REPLYING PLEASE QUOTE OUR REF. TOK/HH-D536 TOKYO Sept. 25, 1972

Kaiser Aluminum & Chemical Corporation

Messrs. ~~Kaiser Shoji Kabushiki Kaisha~~

Asano Daiichi Bldg.

5-1, 2-chome Ginza

Chuo-ku, Tokyo

Attn: Mr. T. Morita

Dear Sirs,

Re: Texada Copper Concentrates

Enclosed please find the original contract sheet concerning the captioned material duly countersigned on our part.

In this regard we would like to advise you that we are returning this contract sheet under the following understandings concerning the insurance clause.

A) Texada's underwriters shall agree to pay the full amount of loss determined to be due under the insured terms and conditions and shall make payment direct to us (cargo owners) on demand.

B) Texada will arrange with the Underwriters to cover H.S.C. for all shipments.

C) In case a General Average is declared, the Underwriters shall issue their Letter of Guarantee to the shipowners instead of a cash deposit.

D) Insured Amount for each shipment shall be adjusted so as to cover 110% of the final invoice amount.

If the above coincides with your understandings please countersign and return us one copy of this letter thereby signifying your agreement.

Yours truly,

Agreed:

KAISER ALUMINUM & CHEMICAL CORPORATION

~~KAISER SHOJI KABUSHIKI KAISHA~~

Director of Insurance

MITSUBISHI CORPORATION

Manager of Copper Ore Sect.
Non-Ferrous Metal Dept. A

CC: MIC-SFC
MCL-VCR

THIS AGREEMENT made and entered into as of March 20, 1973, by and between NIPPON STEEL CORPORATION (Yawata Iron & Steel Co., Ltd. and Fuji Iron & Steel Co., Ltd. were consolidated into Nippon Steel Corporation on April 1, 1970) and NIPPON KOKAN KABUSHIKI KAISHA (hereinafter collectively called "Buyer") and TEXADA MINES, LTD. (hereinafter called "Seller") with respect to the amendment of the contract for the supply of Texada iron ore (hereinafter called "Ore") made and entered into October 1, 1966, by and between both parties (hereinafter called the "C & F Contract"),

WITNESSETH THAT:

Both of the parties hereto have mutually agreed to amend the C & F Contract as follows:

1. Tonnages due for delivery during the 7th delivery period (1 October 1972 - 30 September 1973) under the C & F Contract which have not been delivered on or before 30 September 1973 shall be cancelled.

2. The following price and other terms and conditions shall be applicable to tonnages delivered during the 8th, 9th and 10th delivery periods under the C & F Contract:

(a) Delivery shall be made F.O.B. Texada island.

(b) Price shall be Nine Dollars and Twelve Cents ($9.12) United States Currency per dry metric ton, delivered sixty-two percent (62%) iron, three percent (3%) sulfur and twelve-hundredths of one percent (0.120%) copper on a dry basis of 105 degrees centigrade (105°C.).

(c) The quantities to be delivered and accepted shall be as follows:

(i) For the 8th delivery period:
(1 October 1973 - 30 September 1974) 400,000 dry MT

(ii) For the 9th delivery period:
(1 October 1974 - 30 September 1975) 400,000 dry MT

(iii) For the 10th delivery period:
(1 October 1975 - 30 September 1976) 400,000 dry MT
plus or minus thirty percent (30%) at Seller's option to be exercised no later than 3 months prior to the start of the 10th delivery period.

For the 8th and 9th delivery periods, 312,000 dry MT shall be delivered to Nippon Steel Corporation and the balance (88,000 dry MT) shall be delivered to Nippon Kokan Kabushiki Kaisha. For the 10th delivery period, seventy-eight percent (78%) of the tonnage delivered shall be delivered to Nippon Steel Corporation and twenty-two percent (22%) shall be delivered to Nippon Kokan Kabushiki Kaisha.

Notwithstanding the provisions of the C & F Contract, the first shipment of ore for the 8th delivery period shall be made during September 1973.

(d) Loading rate, despatch and demurrage rate at loading port shall be as follows:

| Vessel Type | Loading Rate (Long Ton) | Despatch Money | Demurrage |
|---|---|---|---|
| 1) Vessel below 17,000 dead-weight tons | 5,000 | $ 200 | $ 400 |

| | | | |
|---|---|---|---|
| 2) Vessel of 17,000 to 22,000 dead-weight tons. | 6,500 | $ 500 | $1,000 |
| 3) Vessel of 22,000 to 30,000 dead-weight tons. | 8,000 | $ 650 | $1,300 |
| 4) Vessel of 30,000 to 45,000 dead-weight tons. | 20,000 | $ 900 | $1,800 |
| 5) Vessel above 45,000 dead-weight tons | 21,000 | $1,200 | $2,400 |

7.13

3. All other terms and conditions of the C & F Contract and the "Basic Contract for Sale and Purchase of Texada Iron Ore" made and entered into October 1, 1963, as amended, shall remain in effect

IN WITNESS WHEREOF, the parties hereto have executed this Agreement as of the day and year first above written.

BUYER:

NIPPON STEEL CORPORATION

By Takashi Omori
for G. CHIBA, GENERAL MANAGER
IRON ORE DEPARTMENT

NIPPON KOKAN KABUSHIKI KAISHA

By T. Nemoto
T. NEMOTO, GENERAL MANAGER
RAW MATERIAL PURCHASE DEPT.

BUYER'S AGENT:

MITSUBISHI CORPORATION

By
E. HASEO, DIRECTOR, GENERAL MANAGER
OF FERROUS RAW MATERIALS DEPT.

SELLER:

TEXADA MINES, LTD.

By
E. H. SANGWINE, PRESIDENT

SELLER'S AGENT:

KAISER SHOJI K.K.

By
S. E. ERICKSON
REPRESENTATIVE DIRECTOR

041973

OFFICE OF THE CHIEF ANALYST AND ASSAYER

Appendix 8

THE GOVERNMENT OF THE PROVINCE OF BRITISH COLUMBIA

# DEPARTMENT OF MINES AND PETROLEUM RESOURCES

VICTORIA

SAMPLE RECEIVED FROM DR. A. SUTHERLAND BROWN, Deputy Chief

ADDRESS Geological Division

| LABORATORY No. | SUBMITTER'S MARK | LABORATORY REPORT |
|---|---|---|
| | | Instrumental Results |
| 13946M | #1 | Total Fe 48.0 ± 2% |
| | | Soluble Fe 41.2 ± 2% (aqua regia 20 min. 80°C) |
| | | Au<0.5 ppm |
| | | Ag < 5 ppm |
| | | Cu 0.265 ± 0.01 |
| 13947M | #2 | Total Fe 17.0 ± 0.8% |
| | | Soluble Fe 11.8 ± 0.6% (aqua regia 20 min. 80C) |
| | | Au<0.5 ppm |
| | | Ag < 5 ppm |
| | | Cu 0.20 ± 0.01 |



DATE May 7, 1974

Paul F. Ralph
for CHIEF ANALYST AND ASSAYER.