Texada Mine

92F106-07

PROPERTY FILE

007115

W.A. No.

NAME Geol Rpt

SUBJECT

M28-1506

A Brief Description of the Mineral Deposits occurring on Mineral Properties owned by:

Texada Mines Ltd. - Texada Island, B. C.

The mineral occurrences are classified as metasomatic in origin. The ore bodies 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 ore bodies have been outlined within an area 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 ore bodies 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 feet by 100 feet with only an occasional need for roof support.

TEXADA MINES LTD. - PROPERTY LIST

...rcel ...mber	Name	Description	Area	Aquired by:	Date	Taxes	Remarks
	SE 1/4 Section 1	Crown-granted Land	128	Purchase.	May 29, 1951		Conveys all mineral rights, with the exception of Gold and Silver.
	SW 1/4 Section 2	Crown-granted Land	160	Purchase from P.S.I. Co.	May 29, 1951		"
	SE 1/4 Section 2	Crown-granted Land	160	"	May 29, 1951		" (& water).
	SW 1/4 Section 3	Crown-granted Land	147	"	May 29, 1951		", water, & coal at 5¢/Ton Royalty, excluding the bed of Paxton Lake.
	SE 1/4 Section 3	Crown-granted Land	160	"	May 29, 1951		", and right to gravel, stone for P. Works.
	NW 1/4 Section 4	Crown-granted Land	57	"	May 29, 1951		", Water.
	Fr. NE 1/4 Section 4	Crown-granted Land	150	"	May 29, 1951		", Water.
	Fr. SE 1/4 Section 4	Crown-granted Land	19	"	May 29, 1951		Reserves on all minerals, precious or base, Coal, Petroleum, & Water.
	NW 1/4 Section 5	Crown-granted Land	160	"	May 29, 1951		Conveys all mineral rights, with the exception of Gold, Silver, Coal Reserved, & Water.
	NE 1/4 Section 5	Crown-granted Land	160	"	May 29, 1951		Conveys all mineral rights, with the exception of Gold and Silver, & includes the bed of Cranby Lake.
	SW 1/4 Section 5	Crown-granted Land	80	"	May 29, 1951		Conveys all mineral rights, with the exception of Gold, Silver, Coal @ 5¢/Ton Royalty, and Water Reserves.
	SE 1/4 Section 5	Crown-granted Land	160	"	May 29, 1951		"
	SW 1/4 Section 6	Crown-granted Land	160	"	May 29, 1951		"
	SE 1/4-Section 6	Crown-granted Land	160	"	May 29, 1951		"
	NW 1/4 Section 8	Crown-granted Land	160	"	May 29, 1951		Conveys all mineral rights, with the exception of Gold, Silver, & Water.
	NW 1/4 Section 9	Crown-granted Land	160	"	May 29, 1951		", Coal Royalty, & Timber.
	NE 1/4 Section 9	Crown-granted Land	160	"	May 29, 1951		" " " " "
	Fr. SW 1/4 Section 9	Crown-granted Land	144	"	May 29, 1951		" " " " "
	Fr. N 1/2 Section 10	Crown-granted Land	92	"	May 29, 1951		" " " " "
	Fr. SE 1/4 Section 10	Crown-granted Land	7	"	May 29, 1951		" " " " "
	N 1/2 Section 11	Crown-granted Land	140	"	May 29, 1951		" " " " "
	Fr. N 1/2 Section 12- -Fr. NE 1/4 sold to Dr. Sanderson. -Fr. NW 1/4 Retained	Crown-granted Land	6	"	May 29, 1951		", East 1/2 of this section, exchanged.
	Lot 52 on the NW 1/4 Section 3.	Crown granted Land		"	May 29, 1951		Conveys Surface Rights only & excludes bottom of Paxton Lake. (Crown reserves minerals, precious or base, coal & petroleum).

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al er	Name	Description	Area	Aquired by:-	Date	Taxes	Remarks
	Lot 53 East Gate	C.G. Min. Claim	51.56 Acres	Purchased from P.S.I. Co.	May 29, 1951		Gold and Silver only.
	Lot 234 Goodall	C.G. Fr. M.C.	20.16 Acres	" "	" "		" "
	Lot 264 LeRoi	*C.G. M.C.	46.28 Acres	" "	" "		" "
	Lot 265 Boulder Nest	*C.G. M.C.	31.37 Acres	" "	" "		" "
	Lot 266 Jack North	C.G. M.C.	36.34 Acres	" "	" "		" "
	Lot 267 Yellow Kid	C.G. M.C.	12.03 Acres	" "	" "		" "
	Lot 268 L.M.C.	C.G. M.C.	41.08 Acres	" "	" "		" "
	Lot 182 Camoron	*C.G. M.C.	51.65 Acres	(W. Estate of ?Wylie)	Late 1952	Paid to 13/7/75	
	Lot 580 Lime	*M.C. 13933H	49¼ Acres	Staked by Texada Mines Ltd. by J.K. Halley	June 27, 1956	Recorded 13/7/56	Tag number 234948
	Lot 581 Lime #1 Fr.	*Fr. M.C. 13934H	42½ Acres	" "	" "	Paid to 13/7/75	Tag number 234949
	Lot 584 B-40894	*M.C. 13322G	38 Acres	Aquired from Ideal Cement Co.	June 12, 1958	Paid to 24/6/75	Staked June 24, 1955.
	Lot 585 B-40879	*M.C. 13298G	41 Acres	" "	" "	Paid to 17/6/75	Staked June 17, 1955.
	Lot 586 B-40878	*M.C. 13297G	32 Acres	" "	" "	Paid to 17/6/75	Staked June 17, 1955.
	Lime No.10 Fr.M.C.	*Fr. M.C. 14518G	16 Acres	Staked by W.Miller	June 4, 1958	Paid to 13/6/75	Recorded 13/6/58. Recovers part area Lime No.3. Tag 258573
	Lime No.11 Fr.M.C.	*Fr. M.C. 14519G	16.7 Acres	" "	June 4, 1958	Paid to 13/6/75	Recorded 13/6/58. Recovers part area Lime NO.2. Tag 258569
	Alladin M.C. Lot 189	Reverted Crown grant Min. Claim Lease	51.65 Acres	Texada Mines Ltd.	Jan. 18,1963	As of 1974 $104/yr	M.L.#10 March 12, 1963 for 21 yea
	T.M.L. No.3 Fr.	Fractional M.C. 14306(E)	34.1 Acres	Staked by Texada Mines Ltd.(G.McKee)	May 7, 1957	Paid to 15/5/75	Covers Mill, office etc. and part of Prescott Pit. Recorded 15/5/ Tag number 258563
	Lime No.12 Fr.M.C.	*Fractional M.C. 14524H	44 Acres	Staked W.Miller	July 10, 1958	Paid to 14/7/75	Covers Anomaly "A". Recorded 14 Bill or Sale 17/1/67 Millers n Tag number 258570
	Lime No.13 Fr.M.C.	*Fractional M.C. 14585P	40 Acres	Staked G.McKee	Nov. 10, 1958	Paid to 24/11/75	Covers Limestone west of Lot 586 Tag number 258575
	Lime No.14 M.C.	*Mineral Claim 14586P	51 Acres	Staked G. McKee	Nov. 10, 1958	Paid to 24/11/75	Covers Limestone and Magnetite Northwest corner of property Tag number 258576
	Lime No.15 M.C.	*Mineral Cliam 14587P	36.2 Acres	Staked G. McKee	Nov. 10, 1958	Paid to 24/11/75	Dump room, West of Y.K. & Prescott Tag number 258577
	Lime No.16 Fr.M.C.	*Fractional M.C. 14588P	6.4 Acres	Staked G. McKee	Nov. 14, 1958	Paid to 24/11/75	Dump room, West of Y.K. & Prescott. Tag number 258578

-aims marked thus grouped June 12, 1961
-nder the name "Texada Mines Ltd."

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.1 r	Name	Description	Area	Aquired by:-	Date	Taxes	Remarks
	T.M.L. No.6 Fr.	Fractional M.C.	29.2 Acres	Staked by W.Miller Agent forT.M.L.	April 14, 1961	Paid to 17/4/75	Covers T.M.L. 4&5 above, East of T.M.L. #3. Tag 303213. Rec.#153:
	T.M.L. No.7 Fr.M.C.	Fractional M.C.	11 Acres	" "	Jan. 10, 1962	Paid to 17/1/75	Fr. between T.M.L. #6 and L.M.C. Explosives Magazine and temporary shops. Tag 303217. Recorded 17/1/62 #15596A.
	T.M.L. No.8 Fr.M.C.	Fractional M.C.	10 Acres	" "	Jan. 9, 1962	Paid to 1975	Fr. between L.M.C. and Eastgate, Swamp and dumps. Tag 303223 Recorded 17/1/62 #15597A
	T.M.L. No.9 Fr.M.C.	Fractional M.C.	30 Acres	" "	Jan. 9, 1962	Paid to 1975	Fr. South & East of Eastgate, Du and Paxton Power L.M.C. Tag 30322: Recorded Jan 17, 1962 #15598A
	T.M.L. No.10 Fr.M.C.	Fractional M.C.	23.3 Acres	" "	Jan.10, 1962	Paid to 1975	Fr. mill and slide area down to sc Recorded 17/1/62 #15599A Tag 303214
	T.M.L. No.11 M.C.	Mineral Claim	51.65 Acres	" "	Jan.10, 1962	Paid to 1975	Slide area and East, below explosi Magazine. Tag 303215 Recorded 17/1/62 #15600A
	T.M.L. No.12 Fr. M.C.	Fractional M.C.	43 Acres	" "	Jan. 9, 1962	Paid to 1975	Swamp area and Y.K. dump extensio: Tag 303221 Recorded 17/1/62 #156
	T.M.L. No.13 M.C.	Mineral Claim	51.65 Acres	" "	Jan. 9, 1962	Paid to 1975	Gravel Pit area. Tag 303220 Recorded 17/1/62 #15602A
	T.M.L. No.14 M.C.	Mineral Claim	51.65 Acres	" "	Jan.11,1962	Paid to 17/1/75	Gravel Pit - Lumber yard area. Tag 303224 Recorded 17/1/62 #156
	T.M.L. No.15 Fr.M.C.	Fractional M.C.	29.3 Acres	" "	Jan.10,1962	Paid to 1975	Explosives Magazine hillside to se Tag 303216 Recorded 17/1/62 #1
	T.M.L. No.16 Fr.M.C.	Fractional M.C.	43 Acres	" "	Jan. 9, 1962	Paid to 1975	Gravel pit to Cox's Lagoon, West Tag 303219 Recorded 17/1/62 #156
	T.M.L. No.17 M.C.	**Mineral Claim	51.65 Acres	" "	Jan. 9, 1962	Paid to 1975	Gravel Pit to East side Cox's Lag Tag 303218 Recorded 17/1/62 #15
	T.M.L. No.18 M.C.	**Mineral Claim	51.65 Acres	" "	Jan.11, 1962	Paid to 1975	N.W. end of Airstrip. Tag 303225 Recorded 17/1/62 #15607A
	T.M.L. No.19 M.C.	**Mineral Claim	43 Acres ?	" "	Jan 22, 1962	Paid to 1975	Southeast end of Airstrip,Tag 277: Staked 11/1/62 as Fractional M.C. replaced 22/1/62 as Full M.C. Recorded 25/1/62 #15608A
	T.M.L. No.20 Fr.M.C.	*Fractional M.C.	9.6 Acres	" "	March 29/62	Paid to 4/75	Fraction against Eagle Claims, alc our North Bdry. Tag 417653 Recorded 4/62 #156880 Grouped 20/4/65

ims marked thus grouped April 20, 1965
er the name "Texada Mines Ltd."
ginal grouping January 1963.

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...el ...er	Name	Description	Area	Aquired by:-	Date	Taxes	Remarks Tag No.	Record No.	
	T.M.L. No.25 M.C.	Mineral Claim	Partly covered by Hretchka staking	Staked by W.Miller Agent for T.M.L.	Sep. 6, 1962	Paid to 1975	450617	15917 M	
	T.M.L. No.26 M.C.	Mineral Claim	Completely covered by Hretchka staking	" "	Sep. 6, 1962	Paid to 1975	450618	15918 M	
	T.M.L. No.27 M.C.	Mineral Claim	Completely covered by Hretchka staking	" "	Sep. 6, 1962	Paid to 1975	450619	15919 M	
	T.M.L. No.28 M.C.	Mineral Claim		" "	Sep. 6, 1962	Paid to 1975	450620	15920 M	
	T.M.L. No.29 M.C.	Mineral Claim		" "	Sep. 6, 1962	Paid to 1975	450621	15921 M	
	T.M.L. No.30 M.C.	Mineral Claim	Partly covered by Hretchka staking	" "	Sep. 6, 1962	Paid to 1975	450622	15922 M	Texada Island foreshore fro... Jack North M.... and Airport.
	T.M.L. No.31 M.C.	Mineral Claim		" "	Sep. 6, 1962	Paid to 1975	450623	15923 M	
	T.M.L. No.32 M.C.	Mineral Claim		" "	Oct. 3, 1962	Paid to 1975	450624	16005 N	
	T.M.L. No.33 M.C.	Mineral Claim		" "	Sep.20, 1962	Paid to 1975	450633	15997 N	
	T.M.L. No.34 M.C.	Mineral Claim		" "	Sep.20, 1962	Paid to 1975	450632	15998 N	
	T.M.L. No. 35 M.C.	Mineral Claim		" "	Oct. 3, 1962	Paid to 1975	450625	16006 N	
	T.M.L. No.36 M.C.	Mineral Claim		" "	Nov.28, 1962	Paid to 1975	450626	16124 R	East of Paxton Lake and sout... of Alladin M....
	T.M.L. No.37 M.C.	Mineral Claim		" "	Nov.28, 1962	Paid to 1975	450627	16125 R	
	T.M.L. No.38 M.C.	Mineral Claim		" "	Nov.28, 1962	Paid to 1975	450628	16126 R	
	T.M.L. No.39 M.C.	Mineral Claim		" "	Nov.28, 1962	Paid to 1975	450629	16127 R	
	T.M.L. No.40 M.C.	Mineral Claim		" "	Nov.28, 1962	Paid to 1975	450630	16128 R	
	T.M.L. No.41 Fr.M.C.	Fractional M.C.		" "	Nov.29, 1962	Paid to 1975	450631	16129 R	
	T.M.L. No.42 Fr.M.C.	Fractional M.C.		" "	Dec. 3, 1962	Paid to 1975	450616	16130 R	
	T.M.L. No.43 Fr.M.C.	Fractional M.C.		" "	Dec. 3, 1962	Paid to 1975	450615	16131 R	
	Lime No.17	*W. end of Property beyond Lime #19		" "	May 1, 1964	Paid to 1975	417661	17283 E (7/5/64)	
	Lime No.18	*S.W. of Lime 14 and West of Lime #15		" "	May 1, 1964	Paid to 1975	417660	17284 E (7/5/64)	
	Lime No.19	*W. of Welcome Bay and beyond T.M.L.#25		" "	May 1, 1964	Paid to 1975	417662	17285 E (7/5/64)	
	Lime No.20	*Mineral Claim	51 Acres	Staked by G.McKee Agent for T.M.L.	May 8, 1964	Paid to 1975	417663	17286 E(14/5/64)	West of Eagle Cla...

...ouped with "Texada Mines Ltd.
...pril 20th, 1965.

92F/10E
TE:ADA MINE
92F-106,107,257,258,259

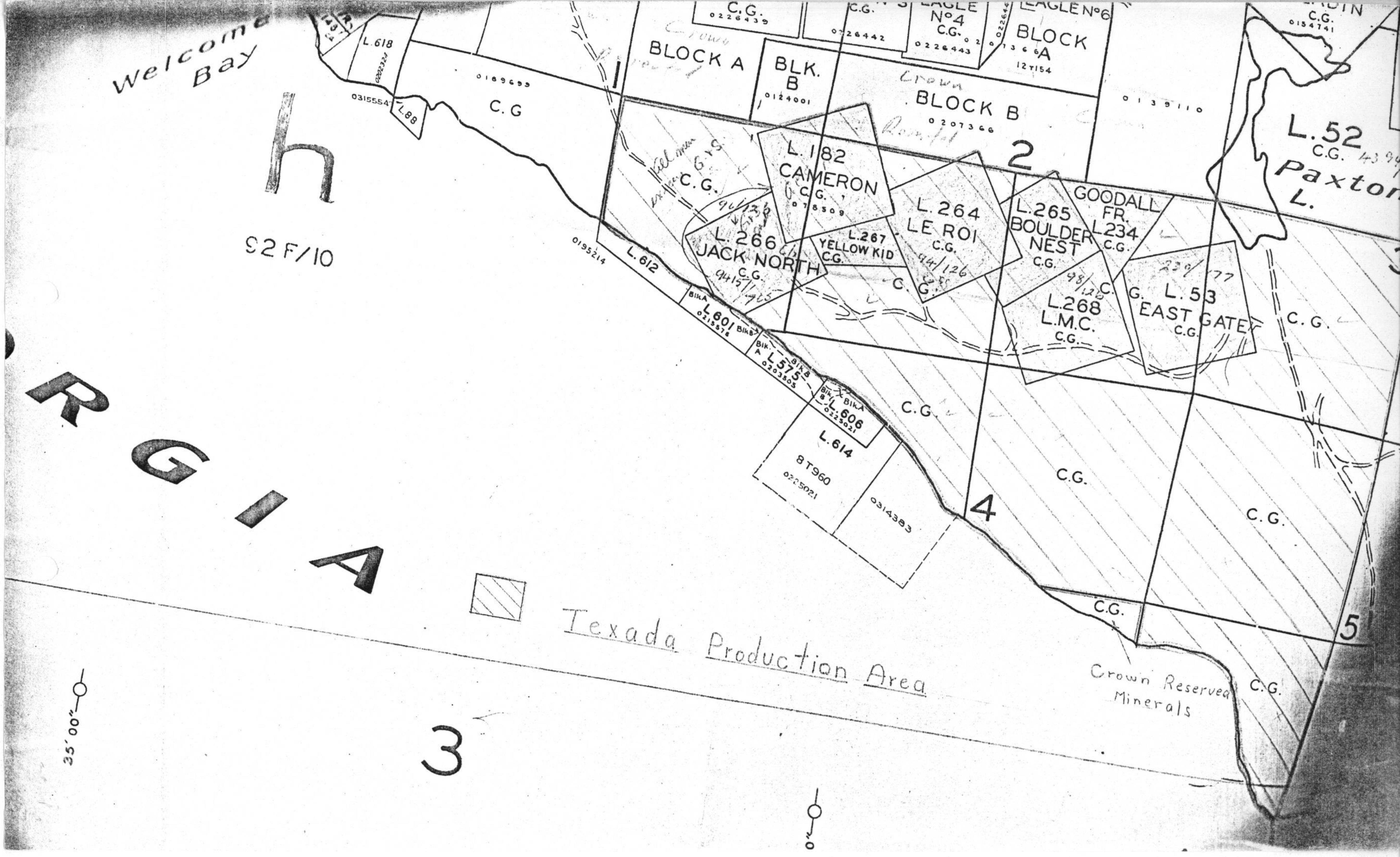
February 18, 1974

TEXA' MINES LIMITED
(Nana..mo Mining Div.)

LOT	CLAIM NAME	ACREAGE
53	Eastgate	51.56
234	Goodall	20.16
264	LeRoi	46.84
265	Boulder Nest	31.37
266	Jack North	36.34
267	Yellow Kid	12.03
268	L.M.C.	41.08
182	Cameron	58.30
	TOTAL =	297.68

NOTE: Crown granted land plus portions of Lot 264 (.56 acre) and Lot 182 (6.65 acre) north of C.G.L. Boundary. 2725.71 ac.

L. 25
L. 20
L. 593
L. 594
L. 597
L. 423
TML 42F
L. 189
L. 276
L. 272
TML 43F
L. 416
L. 592
L. 595
L. 596
L. 598
IDEAL 20 F
LIME 10F
L. 581 LIME 1 F.
LIME 20
L. 52
TML. 39
TML. 40
L. 304
LIME L. 580
TML 20F
B 40878 L. 586
B 40879 L. 585
B 40884 L. 584
LIME 11F
LIME 14
LIME. 13F
1
2
3
PAXTON LAKE
L. 300
WELCOME BAY
L. 182
265
L. 234
TML 41
TML. 38
TML 37
LIME 12F
LIME 15 F
LIME 18
L. 267
L. 266
L. 268
TML. 36
TML. 9F
TML. 6F
TML. 7F
TML. 8F
TML. 3
L. 612
LIME 16F
TML. 25
L. 601
L. 575
LIME 17
LIME 19
TML. 10 F
TML. 11
TML. 12
TML. 13
TML. 14
IRON. 3
IRON. 1
L. 614
TML. 15
4
TML. 26
CRANBY LAKE
TML. 16
TML. 17
TML. 18
5
IRON. 2
TML. 30
TML. 28
TML. 19
TML. 29
TML. 32
TML. 33
TML. 34
TML. 35
TML. 31
O F


Welcome Bay
h
92 F/10
RGIA
BLOCK A
BLK. B
0124001
BLOCK B
0207366
BLOCK A
127154
EAGLE No4
C.G.
0226443
0226442
C.G.
0226439
0189699
0139110
L.618
0315554
L.88
C.G.
L.52
C.G.
Paxton L.
2
L.182
CAMERON
C.G.
078509
L.264
LE ROI
C.G.
L.265
BOULDER NEST
C.G.
GOODALL FR.
L.234
C.G.
L.266
JACK NORTH
C.G.
L.267
YELLOW KID
C.G.
L.268
L.M.C.
C.G.
L.53
EAST GATE
C.G.
0195214
L.612
L.601
0215574
L.575
0203505
L.606
0225021
L.614
8T960
0225021
0314383
4
5
3
Texada Production Area
Crown Reserved Minerals
35' 00"

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.

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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).

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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 régrind 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. Cyclôning 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 on 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...

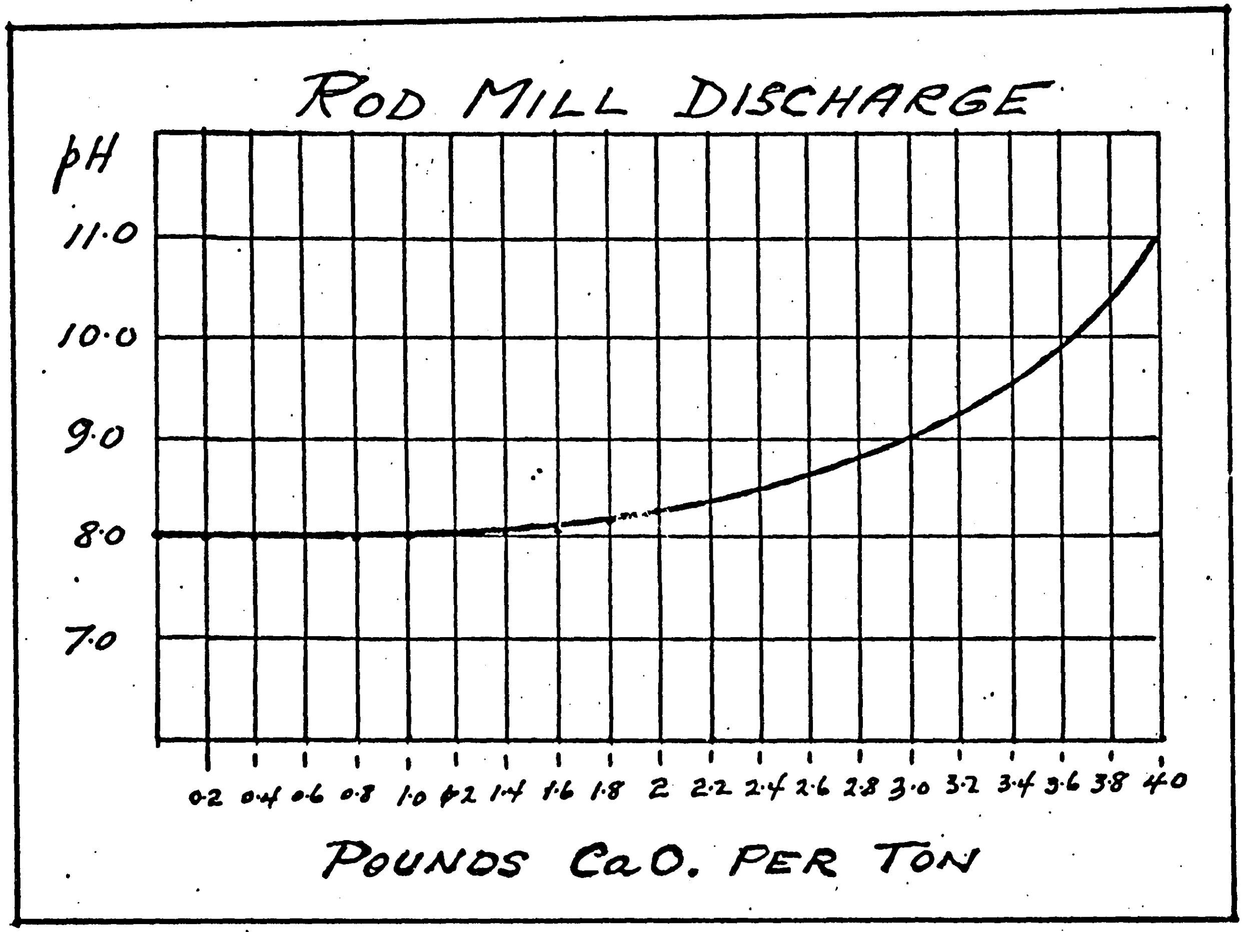
Several quick hard air blows are made into the water filled pipe back to the tanks then the bottom drain is quickly opened. If there is any plug up, the process is repeated. Shell removal is excellent and complete.

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.

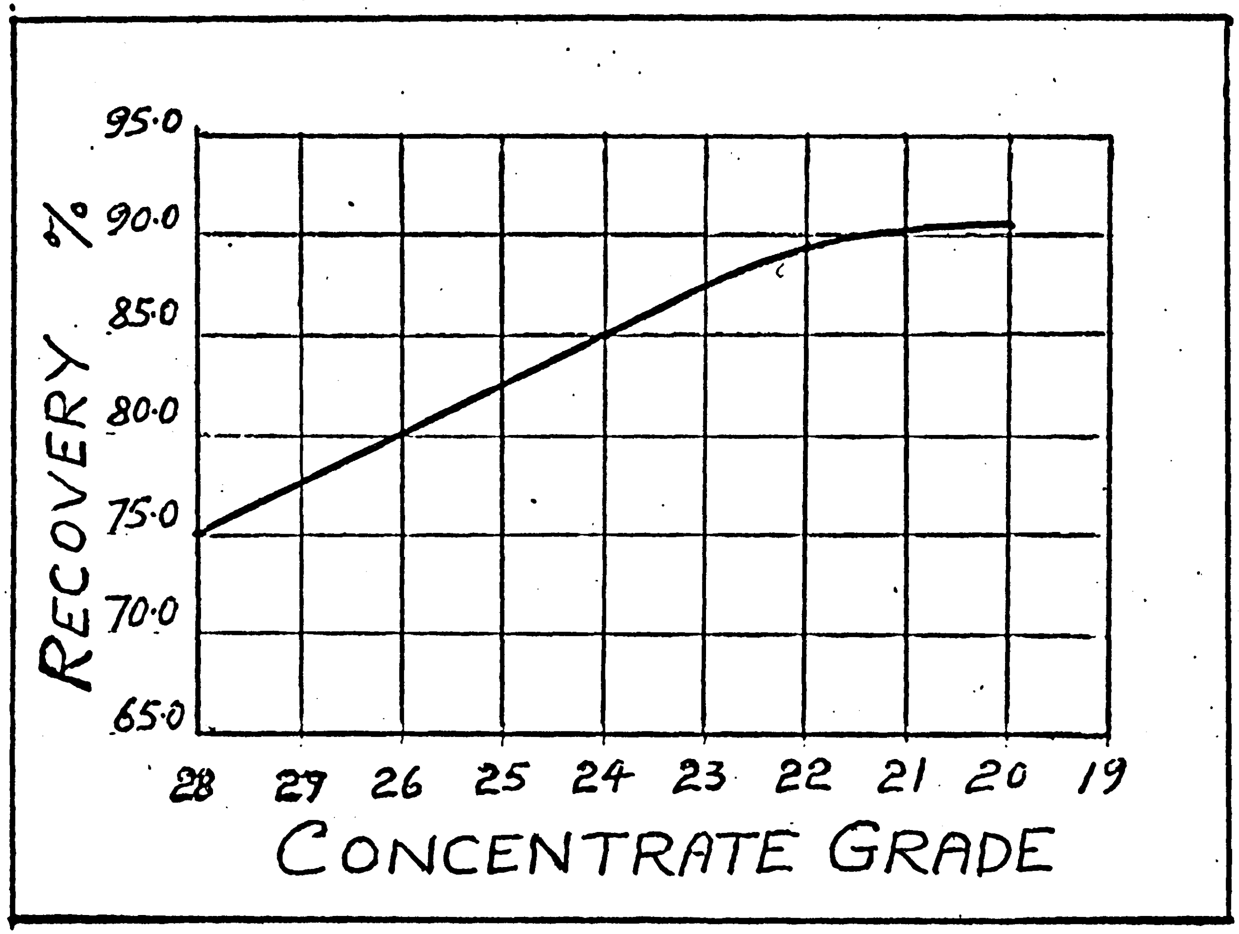
Summing up the 17 years of operation we find that the full use of sea water gives us a very stable easily controlled float. Reagent consumption is greatly reduced and reagent feeding has been simplified. The natural pH of the sea water is the most satisfactory level for Texada ores. It is extremely important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

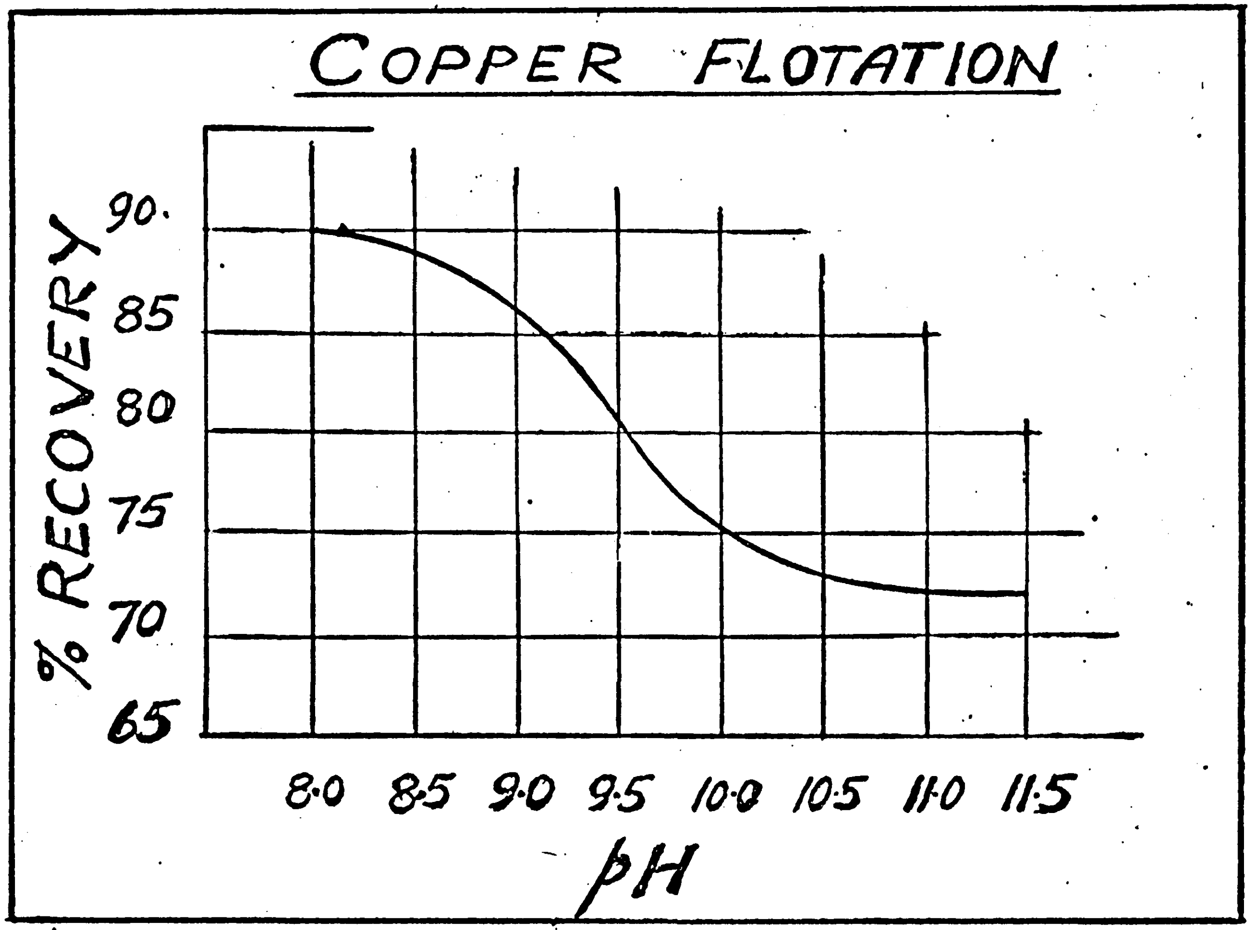
Atmospheric corrosion is severe and causes more problems than the sea water in the processing. This is a location problem and not a function of the use of sea water.


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

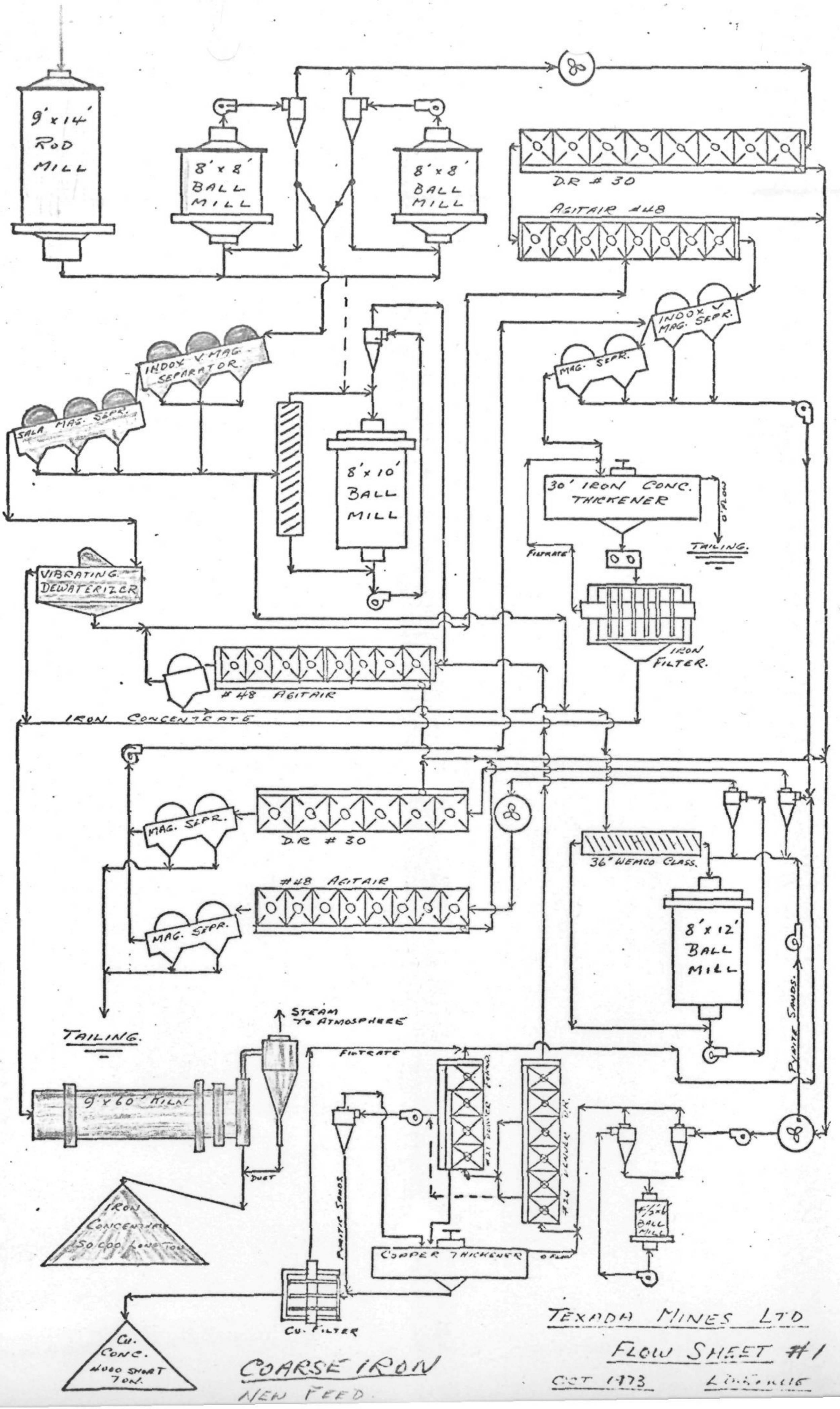
GRAPH #1


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


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



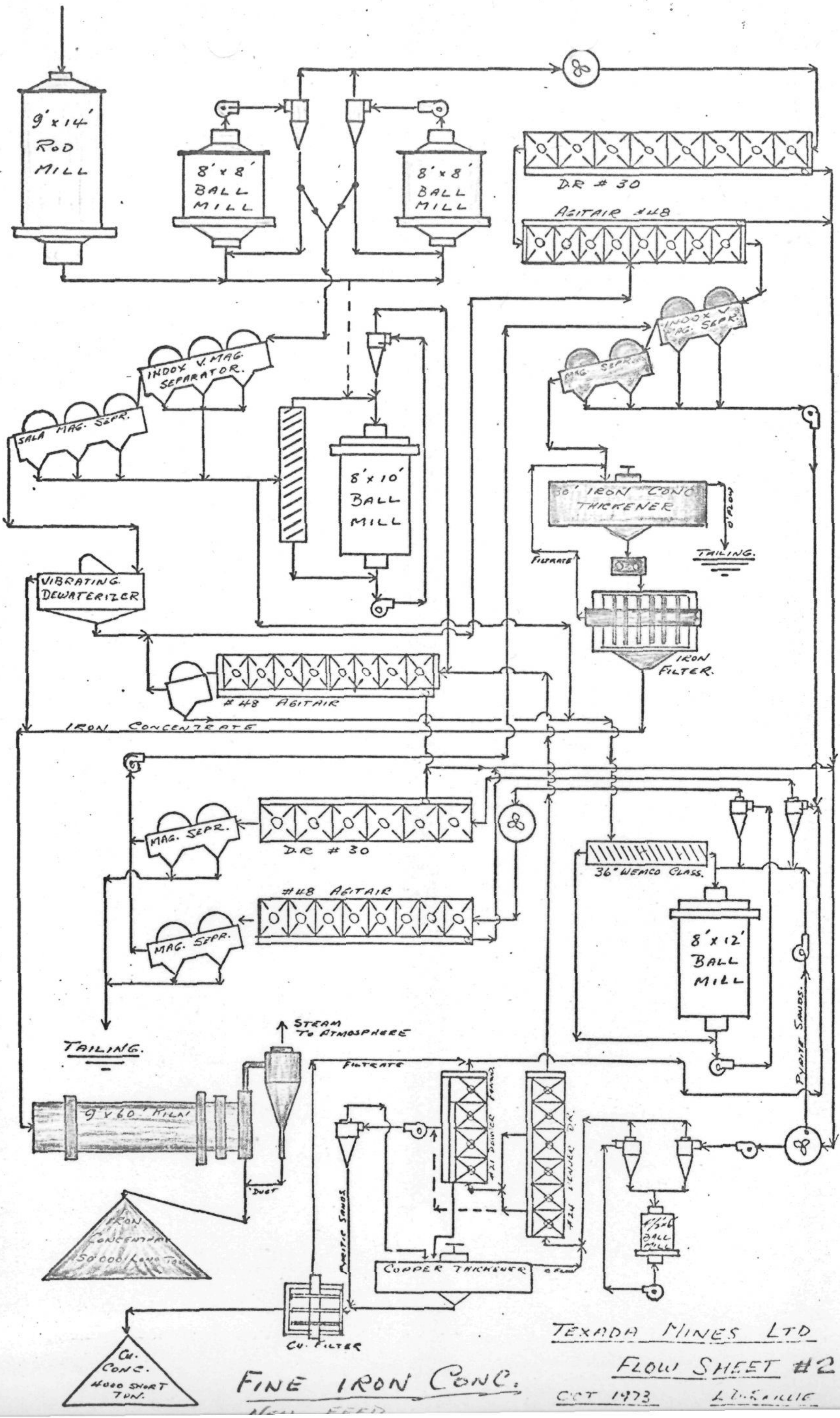
TEXADA MINES LTD

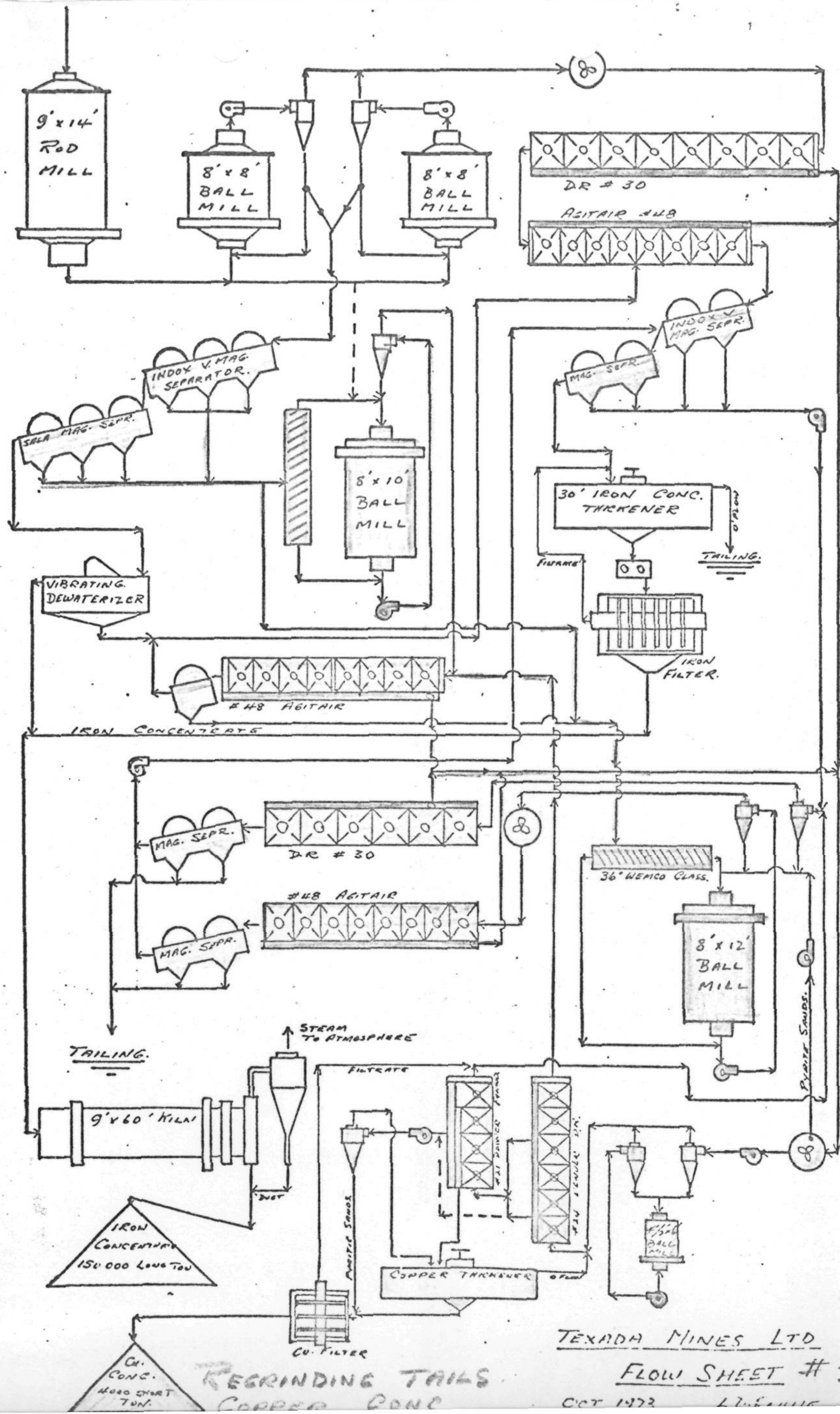
FLOW SHEET #1

COARSE IRON

NEW FEED

OCT 1973


9' x 14' ROD MILL
8' x 8' BALL MILL
8' x 8' BALL MILL
D.R # 30
AGITAIR #48
INDOX V. MAG. SEPARATOR.
SALA MAG. SEPR.
INDOX V. MAG. SEPR.
MAG. SEPR.
8' x 10' BALL MILL
30' IRON CONC THICKENER
O'FLOW
TAILING.
FILTRATE
IRON FILTER.
VIBRATING DEWATERIZER
#48 AGITAIR
IRON CONCENTRATE
MAG. SEPR.
D.R # 30
36" WEMCO CLASS.
#48 AGITAIR
MAG. SEPR.
8' x 12' BALL MILL
PYRITE SANDS.
TAILING.
STEAM TO ATMOSPHERE
FILTRATE
9' x 60' KILN
DUST
IRON CONCENTRATE 50 000 LONG TON
PYRITIC SANDS
COPPER THICKENER
O FLOW
CU. FILTER
CU. CONC. 4000 SHORT TON.
TEXADA MINES LTD
FLOW SHEET #2
FINE IRON CONC.
OCT 1973



TEXADA MINES LTD

FLOW SHEET #3

OCT 1973

Appendix 6

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 D.L.T.

38.89 x $\frac{2204.6}{2240}$ = $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 x $\frac{2204.6}{2240}$ = $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.

Preliminary Report on TEXADA MINE

(Nanaimo Mining Division)

By W. C. Robinson

A. Sutherland Brown

Geology Division
Department of Mines and Petroleum Resources
Victoria, British Columbia

May 9, 1974

CONTENTS

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 - Prescott, Paxton, Lake, and 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:

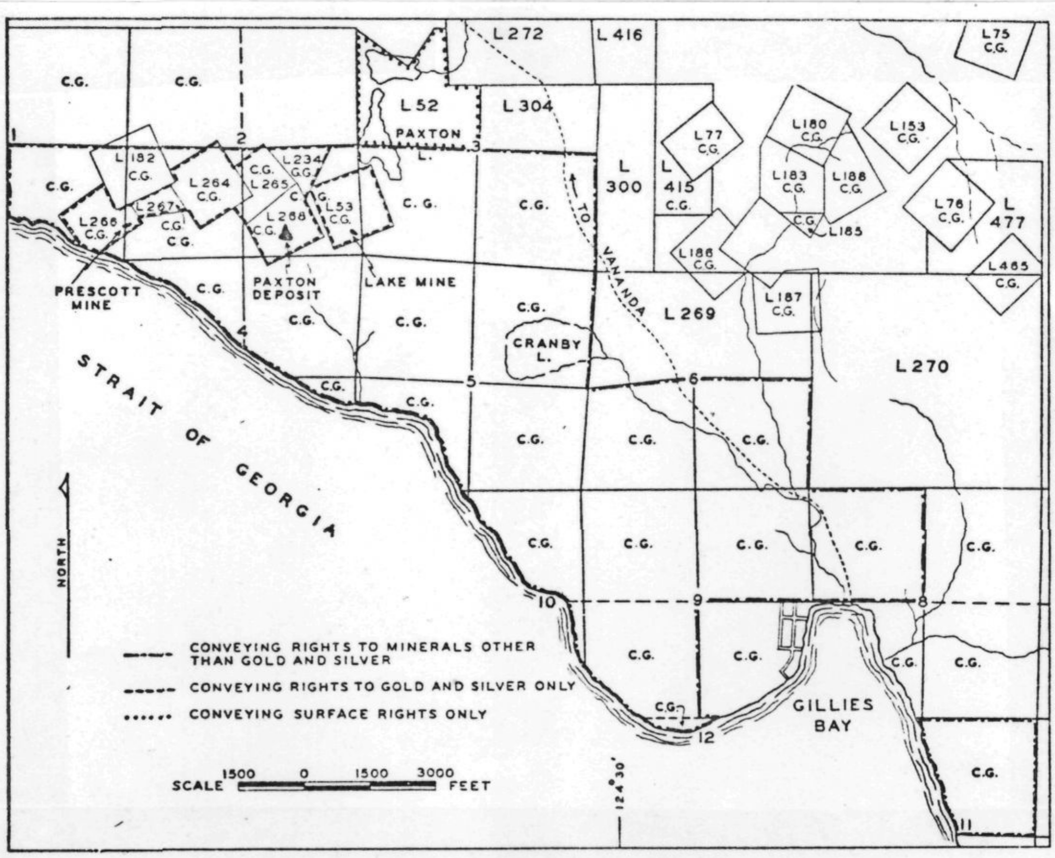


Figure 1 Holdings of Texada Mines Limited.

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

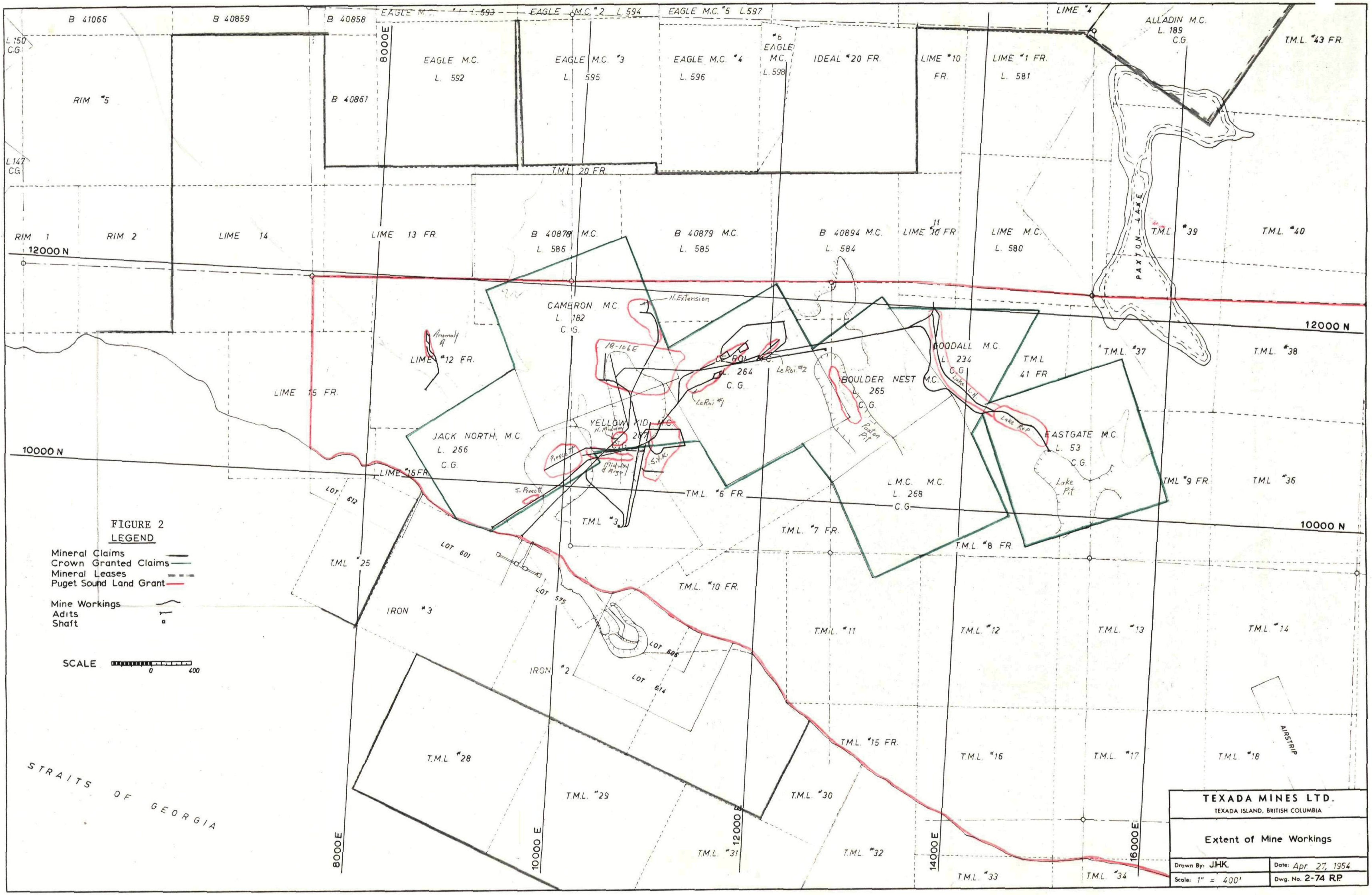
N.W. 1/4 - reserved gold and silver

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


B 41066
B 40859
B 40858
EAGLE M.C. #1 L. 593
EAGLE M.C. #2 L. 594
EAGLE M.C. #5 L. 597
LIME #4
ALLADIN M.C. L. 189 C.G.
T.M.L. #43 FR.
L.150 C.G.
8000E
EAGLE M.C. L. 592
EAGLE M.C. #3 L. 595
EAGLE M.C. #4 L. 596
#6 EAGLE M.C. L. 598
IDEAL #20 FR.
LIME #10 FR.
LIME #1 FR. L. 581
RIM #5
B 40861
L.147 C.G.
T.M.L. 20 FR.
RIM 1
RIM 2
LIME 14
LIME 13 FR.
B 40878 M.C. L. 586
B 40879 M.C. L. 585
B 40894 M.C. L. 584
LIME #11 FR.
LIME M.C. L. 580
PAXTON LAKE
T.M.L. #39
T.M.L. #40
12000 N
CAMERON M.C. L. 182 C.G.
N. Extension
Anomaly A
LIME #12 FR.
18-106E
LE ROI M.C. L. 264 C.G.
Le Roi #2
Le Roi #1
BOULDER NEST M.C. L. 265 C.G.
Paxton Pit
GOODALL M.C. L. 234 C.G.
Lake L.H.
Lake Ret.
T.M.L 41 FR
T.M.L. #37
T.M.L. #38
LIME 15 FR.
JACK NORTH M.C. L. 266 C.G.
YELLOW KID M.C. L. 267
N. Midway
Prescott
Midway & Argo
S.Y.K.
EASTGATE M.C. L. 53 C.G.
Lake Pit
10000 N
LIME #16 FR.
LOT 612
S. Prescott
T.M.L. #6 FR.
L.M.C. M.C. L. 268 C.G.
T.M.L #9 FR.
T.M.L #36
T.M.L. #3
T.M.L. #7 FR.
T.M.L. #8 FR.
FIGURE 2
LEGEND
Mineral Claims
Crown Granted Claims
Mineral Leases
Puget Sound Land Grant
Mine Workings
Adits
Shaft
T.M.L #25
LOT 601
LOT 575
T.M.L. #10 FR.
IRON #3
T.M.L. #11
T.M.L. #12
T.M.L. #13
T.M.L. #14
SCALE 0 400
LOT 606
IRON #2
LOT 614
AIRSTRIP
T.M.L. #15 FR.
T.M.L. #16
T.M.L. #17
T.M.L. #18
STRAITS OF GEORGIA
T.M.L. #28
T.M.L. #29
T.M.L. #30
8000E
10000 E
12000 E
14000 E
16000 E
T.M.L. #31
T.M.L. #32
T.M.L. #33
T.M.L. #34
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.

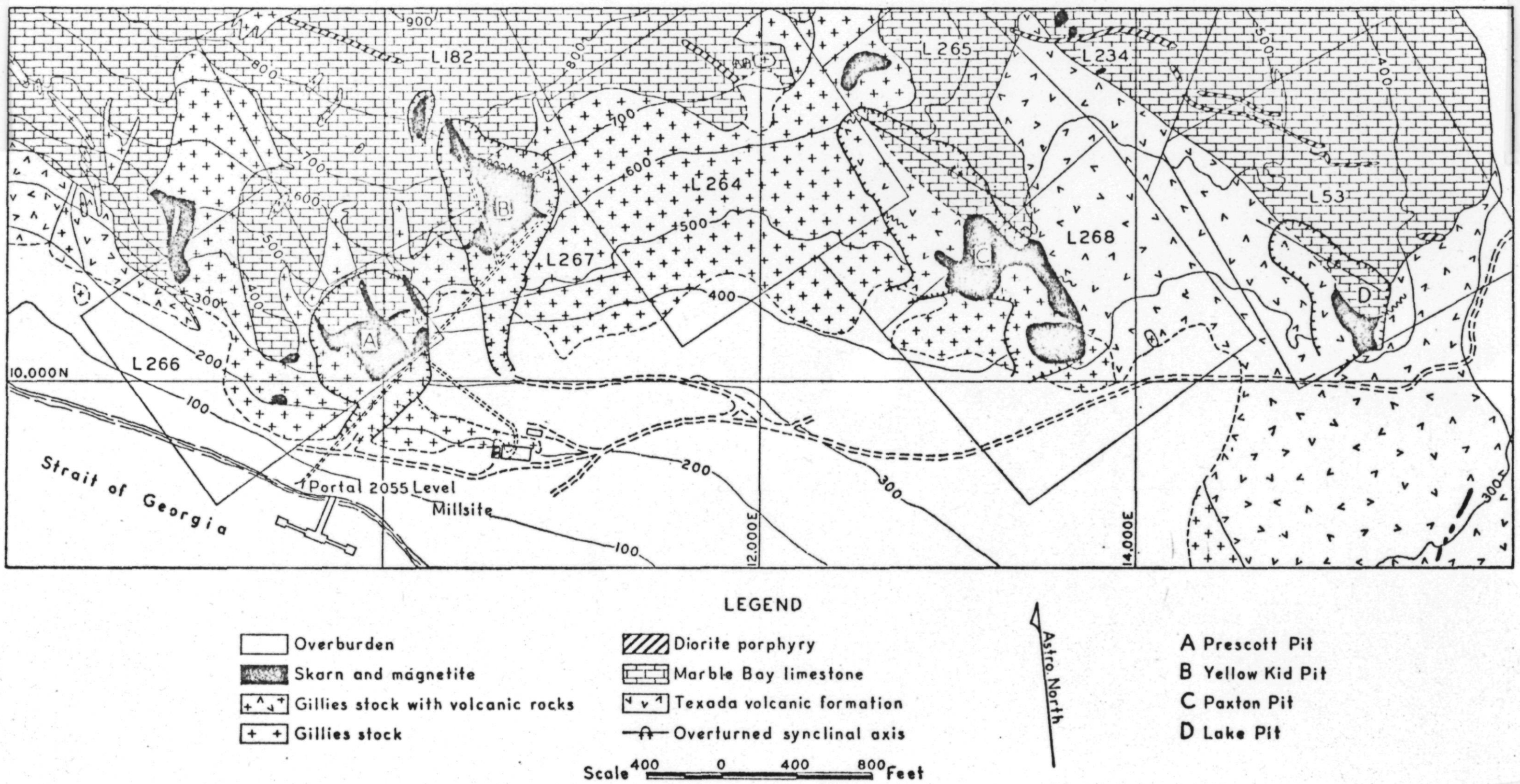


Figure 3. Texada Mines Ltd. Geology of mine area.

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.

<u>RESERVES</u>:

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.

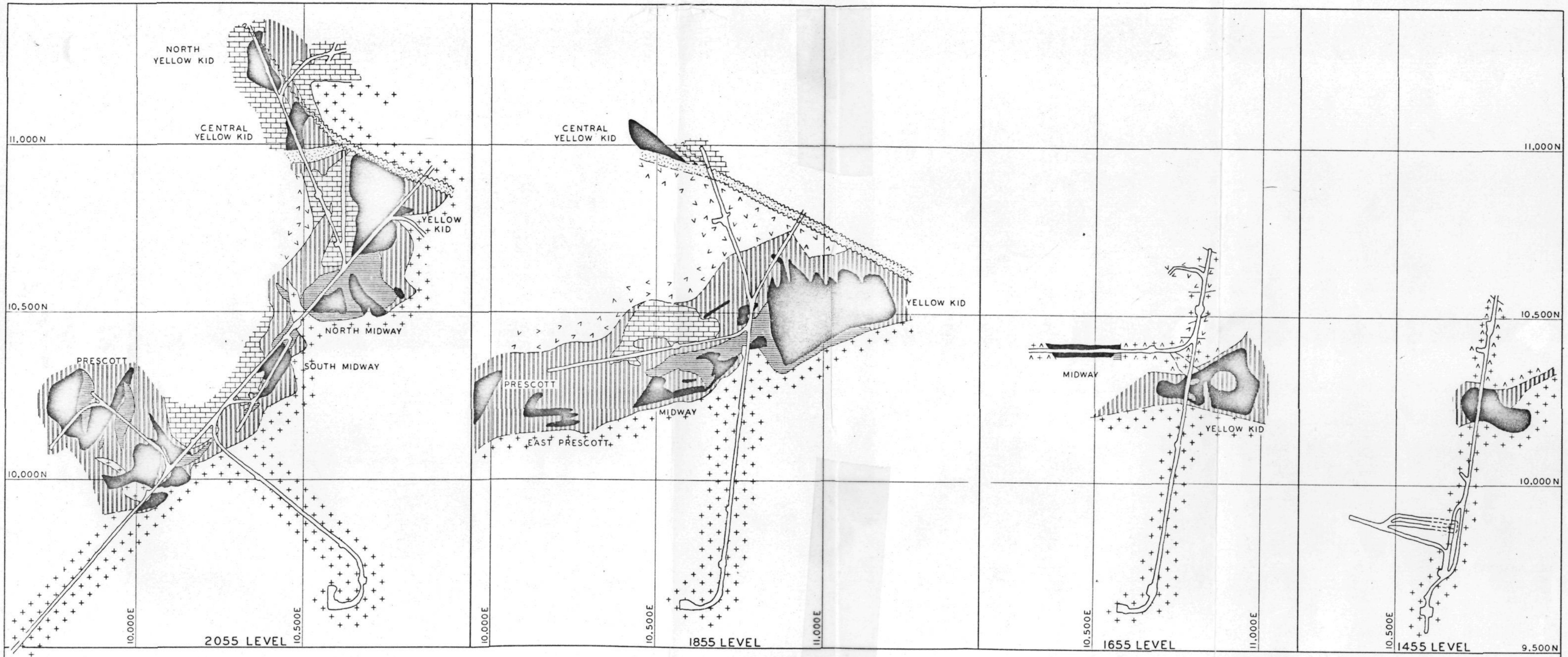


Figure 4

SIMPLIFIED GEOLOGY OF MAIN LEVELS

TEXADA MINES LTD.

Post-ore porphyry

Magnetite and sulphide orebodies

Skarn

Quartz diorite and volcanic rocks

Scale 0 — 500 Feet

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 $.95 \times .87 = 82.7$ per cent

Iron value of one metric ton of ore is:

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

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 $.65 \times .85 = 55.3$ 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 -

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

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.

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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.

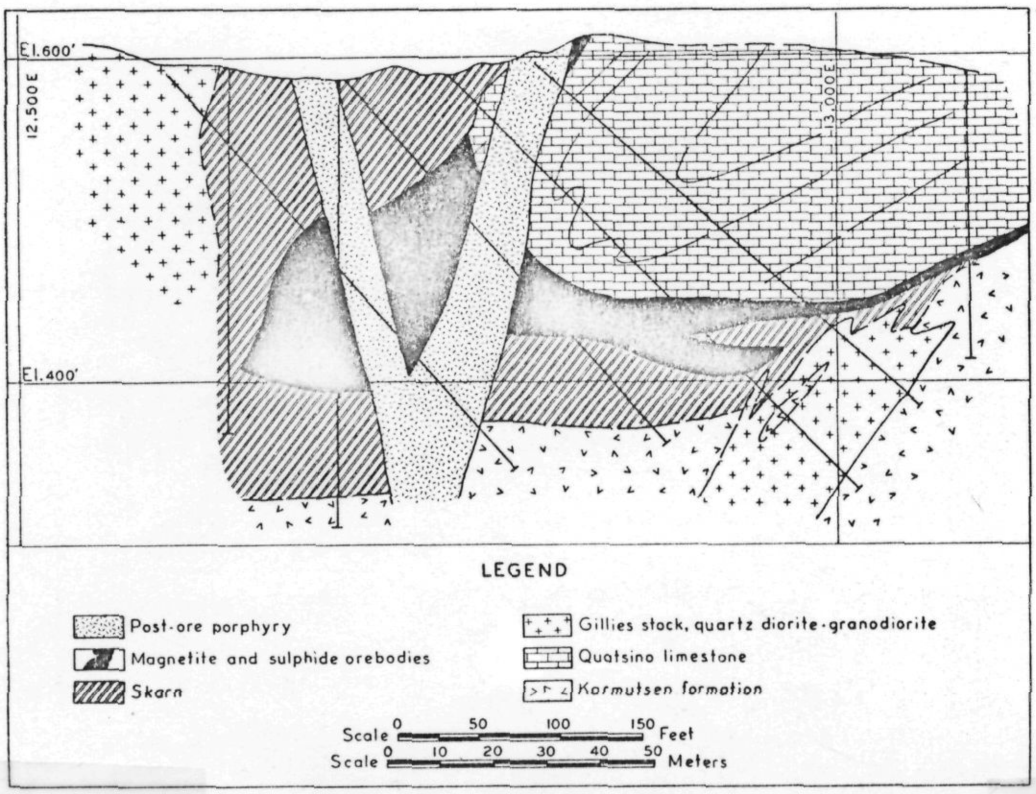

Fig. 6. Cross Section of Paxton Pit.

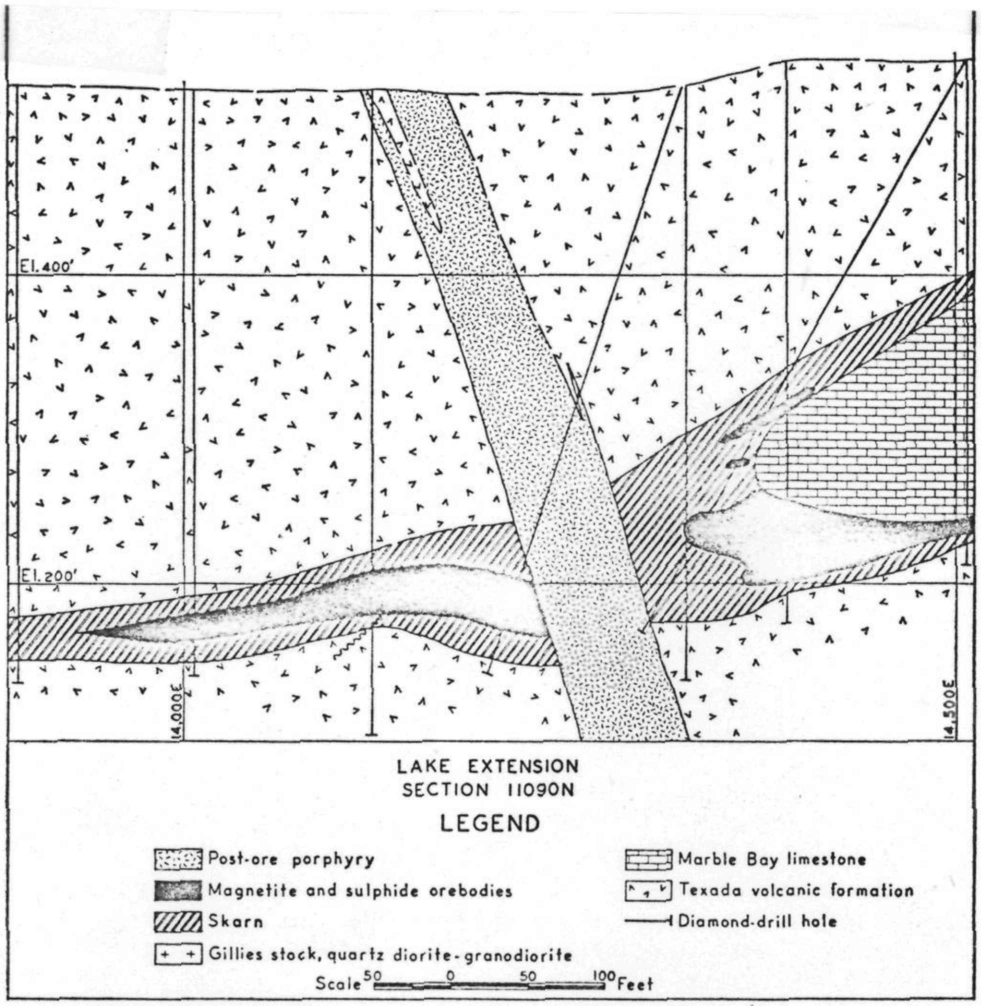

Figure 5. Texada Mines Ltd. Cross-sections of Paxton and Lake orebodies.

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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

TEXADA MINES LTD.

ORE RESERVES - SUPPLEMENTARY NOTES

A. GENERAL NOTES

1. Definition of terms

   (a) Positive - reserves drilled on sections no further than fifty feet apart. In the majority of cases calculations are based on a mining layout.

   (b) Probable - reserves drilled off but sections further than fifty feet apart; or some uncertainty exists in the mining economics. Usually classed as geological reserves.

   (c) Recoverable Pillar - positive ore contained within the limits of a temporary pillar. Calculations based on a mining layout.

   (d) Nonrecoverable Pillar - positive ore within the limits of a permanent pillar. Listed on ore reserves as part of general ore inventory ~~and occasionally~~ because there is some possibility of mining part of the pillar.

   (e) Broken - blasted ore remaining in stope.

2. Stopes classed as high copper ore have had the iron grade adjusted to an estimated recoverable iron grade.

3. Broken but nonrecoverable ore is approximately equal to the average amount of dilution due to overbreak. Neither factor is included in the calculations.

4. Waste within the stope limits is considered as zero grade iron.

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O R E   R E S E R V E S

B. DESCRIPTION OF STOPES AND ORE BODIES

Ore reserves as of June 30th, 1973. Schedule was drawn up in January 1973.

Above 2055 Level:

Le Roi #2A Stope

1. Location - approximately midway between the main mine area and the Lake Ore Body. Access by inclined ramp via the Le Roi #1C Scram.

2. Geology - extremely irregular lenses of magnetite, strike trend is north east and plunging south west. Separated from the Le Roi #1 Ore Body by a diorite porphyry dyke.

3. Mill Tests - sampled but tests not run. Ore is expected to cob to a below average, but acceptable mill feed due to the mixed structure of magnetite and skarn. Probably harder than average grindability.

4. Reserves -

Long-hole Stope	35,000 L.T.	at 36.49% Fe,	0.10% Cu
Recoverable Pillar	7,000 L.T.	at 37.78% Fe,	0.07% Cu

Available from salvage drawpoint in the room and pillar stope and from a salvage drawpoint in the scram.

Pillar	21,000 L.T.	at 44.25% Fe,	0.20% Cu

Undercut pillar, pillar between longhole and room and pillar stopes along with ore beyond the stope limits.

5. Schedule - production starting in September 1973 at 10,000 long tons per month. - Must precede Le Roi #1B and Le Roi #1C because of access.

Le Roi #2B Stope

1. Location - at east end of Le Roi #2A stope but at a higher elevation. Access through the Le Roi #2 Scram.

2. Geology - a zonal relationship of magnetite-chalcopyrite and chalcopyrite-skarn. Classed as high copper ore.

3. Mill Tests - requires fine grinding and is also a very hard ore. Flotation followed by a magnetic separation recommended to control sulphur content in iron concentrates.
Fe concentrates at 48 Mesh 66.37% Fe, 0.36% Cu, 2.10% Sulphur. Work index 20.9 KWH/L.T.

Le Roi #2B Stope Cont'd.

4. Reserves -

Longhole stope	54,000 L.T. at 19.33% Fe,	1.27% Cu
Nonrecoverable pillar	27,000 L.T. at 27.56% Fe,	1.06% Cu

The pillar is largely the vertical extension of the ore body which is uneconomic due to the amount of development work required.

5. Schedule - presently not scheduled. Will probably replace the North Extension ore in part. Must be mined before April 1974 when production starts from the Le Roi #1B Stope according to the present schedule.

Between 2055 & 1855 Levels:

North Midway

1. Location - cuts through the 2055 Level about half way between the Main Midway and Main Yellow Kid ore bodies. Scram located on 1950.

2. Geology - stope almost finished. Ore body was stock-like with about 100 feet of vertical extent. Two offshoots from the main ore body are classed nonrecoverable. One lies to the west of the 2055 Adit and contains 9,000 long tons and the other lies to the east of the stope with reserves of 14,000 long tons of magnetite with chalcopyrite. Both pinch out just below the 2055 Level.

3. Mill Tests - good milling qualities, but grade may now be affected by dilution.

4. Reserves -

Longhole Stope	9,000 L.T. at 34.78% Fe,	0.24% Cu
Drilled, not blasted.		
Recoverable Pillar	18,000 L.T. at 49.38% Fe,	0.11% Cu
Complete recovery would cut the 2055 Adit.		
Nonrecoverable Pillar		
East of stope	14,000 L.T. at 49.80% Fe,	0.68% Cu
West of 2055 Adit	9,000 L.T. at 50.54% Fe,	0.10% Cu
Broken	10,000 L.T. - Grade uncertain.	

Another 50,000 long tons averaging 27.70% Fe and 0.14% Cu lying directly above the stope ~~are~~ is not included in the reserves.

5. Schedule - scheduled to be finished in April 1973 except for Pillar recovery which is not scheduled.

18 Central Yellow Kid

1. Location - situated to north west of Yellow Kid Ore Body with the scram on the 1855 Level. Salvage stope will remove part of the undercut pillar on 1855 Level.

18 Central Yellow Kid Cont'd.

2. Geology - Central Yellow Kid is the fault offset extension of the Main Yellow Kid Ore Body. The ore body is bounded on both sides by faults which converge and cut off the ore body a short distance below the 1855.

- the ore is magnetite with fairly good copper value. Numerous calcite and hematite fracture fillings are a common feature.

3. Mill Tests - stope nearly finsihed. Broken reserves probably diluted by cabing which is expected to exceed the average dilution factor of 10%.

- high grade iron concentrates sometimes difficult to obtain due to fine-grained calcite.

4. Reserves -

Nonrecoverable Pillar along Yellow Kid Dyke	12,000 L.T. at 48.00% Fe,	0.28% Cu
In 2055 U.C. Pillar	15,000 L.T. at 47.80% Fe,	0.11% Cu
Broken	6,000 L.T. at 43.24% Fe,	0.31% Cu

5. Schedule - Possibly some recovery of broken ore.

Upper South Yellow Kid Stope

1. Location - above 1855 Level between the Lower Main and Lower South Yellow Kid Stopes.

2. Geology - only broken reserves left, grade fairly good.

3. Mill Tests - cobbing should upgrade ore to satisfactory mill feed.

4. Reserves -

Nonrecoverable 11,000 L.T. at 45.44% Fe, 0.31% Cu
Along Yellow Kid Dyke for support and also in undercut pillar.
Broken 34,000 L.T. at 43.97% Fe, 0.27% Cu

5. Schedule - delayed until August 1975. Haulageway for Le Roi and Lake Stopes uses the 18-110E Scram and the broken ore has been left as a safety precaution

Le Roi #1 Ore Body

1. Location - immediately north east of Yellow Kid Ore Body at the start of the East Ramp. Stopes and pillars are numbered from West to East.

2. Geology - the Le Roi #1 Ore Body strikes N 45° E, dips steeply south east and plunges south west. The ore body is a thick tabular zone of irregular lenses of magnetite with chalcopyrite separated by bands of skarn.

Le Roi #1 Ore Body Cont'd.

3. Mill Tests - ore upgrades easily with cobbing. A finer than average grind is necessary to keep copper values down in iron concentrate. No other problems.

4. Reserves - Le Roi #1A Stope

Main Stope - finished.
Recoverable Pillar 109,000 L.T. at 42.44% Fe and 0.10% Cu.
Salvage of west and downward extension and part of undercut pillar. Salvage stope will cut haulageway between the 1855 Le Roi #1 and #2 Scrams.

- Le Roi #1 A Pillar

Between #1A and #1B Stopes 188,000 L.T. at 37.34% Fe and 0.36% Cu. Presently classed as nonrecoverable but it appears possible to recover about 50% of the ore.

- Le Roi #1B ~~Stope~~ ~~Pillar~~ Stope

Stope 246,000 L.T. at 33.00% Fe and 0.27% Cu
Nonrecoverable 19,000 L.T. at 44.09% Fe and 0.56% Cu
Outside economical stope limits.

- Le Roi #1 B ~~Stope~~ Pillar

Recoverable 30,000 L.T. at 39.80% Fe and 0.35% Cu
Recoverable from #1B Scram and Le Roi #1C Salvage.
Nonrecoverable 23,000 L.T. at 39.80% Fe and 0.35% Cu
Beyond economical stope limits.

- Le Roi #1C Stope

Stope - finished.
Salvage 60,000 L.T. at 41.54% Fe and 0.18% Cu
Salvage of ore in undercut pillar and below 1980 Scram from a scram coming off the East Ramp.
Nonrecoverable 13,000 L.T. at 31.99% Fe and 0.40% Cu
Ore on footwall of stope and in undercut pillar.

5. Schedule -

(a) Le Roi #1A Salvage - production in July 1973, finish in June 1974. Behind schedule.

(b) Le Roi #1B Stope - production starts in April 1974 and continues to April 1975. Must be delayed until Le Roi #2 is finished.

(c) Le Roi #1B Pillar - production starts in May 1975.

(d) Le Roi #1C Salvage - production starts in May 1975.

Midway Copper Zone

1. Location - in the Midway Zone between the elevations of 1800 and 1950.

2. Geology - the skarn zone close to the limestone contains veins and coarsely disseminated chalcopyrite. The copper mineralization is typically quite irregular. Minor magnetite mineralization is associated with the copper ore.

3. Mill Tests - fine grinding is necessary but in spite of low iron content, the iron recovery is good. A good separation of pyrite and chalcopyrite was obtained in the mill test. Gold values are below average and silver values are slightly higher than average.

4. Reserves -

   Probable 41,000 L.T. at 21.26% Fe and 1.86% Cu
   Classed as probable because of the irregular nature of the ore and further exploration is needed. At present there is no definite mining layout.

5. Schedule - production is due to start in April 1975.

South Prescott Zone

1. Location - two small ore bodies located on both sides of the West Ramp at about the 1950 elevation.

2. Geology - irregular lenses of magnetite separated by skarn.

3. Mill Tests - tests have not been made but no problems are expected. The ore is very similar to the nearby Prescott.

4. Reserves -

   Block A - Recoverable pillar 93,000 L.T. at 39.78% Fe, 0.32% Cu
   Geological reserves. Located to east of Ramp. Partially recoverable when West ramp is no longer required.

   Block B - Probable - 56,000 L.T. at 32.64% Fe, 0.13% Cu. Block B cuts across the West ramp from the west side. Classed as probable because of indefinite mining plans.

   Recoverable Pillar - 29,000 L.T. at 34.77% Fe, 0.15% Cu. Pillar lies across the West Ramp.

5. Schedule - Production was originally scheduled to start in July 1974, It would be advisable to delay the mining as long as possible because of the location of the ore zone.

18-106 Zone

1. Location - North of the Yellow Kid Dyke. Ore occurs in a skarn zone between limestone and a footwall of volcanics or diorite. Thick or steeply dipping portions are mined by longhole stoping and the remainder by room and pillar methods.

2. Geology - Relatively flat lying mineralization consisting of magnetite, chalcopyrite, pyrite and pyrrhotite. Magnetite commonly contains good grade disseminated chalcopyrite. Zonal banding of ore minerals is a common feature.
   - the zone is still open to the north and will be drilled as soon as drill locations are available in the North Extension.

3. Mill Tests - A mill test on a composite sample of the entire 18-106 zone was done in 1966. Fine grinding is necessary to lower Cu values in the iron concentrate. Following flotation tests and magnetic separation the Fe concentrates ran 1.819% S. Gold and silver values are slightly above average.

4. Reserves -

(a) Longhole stopes

402 Stope - minable reserves 313,000 L.T. at 23.03% Fe, 0.62% Cu
Selective mining of iron ore is possible. Iron zones should yield a concentrate at 40% minus 100M of 65% Fe, 0.07% Cu, 0.45% S. Work index 12.5 KWH/L.T. Production just starting.

433 Stope - recoverable pillar 24,000 L.T. at 47.89% Fe, 0.54% Cu. At south end of stope and extending down to East ramp. Possibly recoverable when East ramp is not needed.

(b) Room & Pillar stopes

401 - 4,000 L.T. at 29.80% Fe, 0.51% Cu - Recoverable pillar.
403 - 8,000 L.T. at 37.65% Fe, 1.17% Cu
6 405 - 7,000 L.T. at 43.69% Fe, 0.60% Cu - Temporary pillar.
9? 406 -18,000 L.T. at 16.33% Fe, 1.53% Cu
407 - 4,000 L.T. at 41.77% Fe, 0.94% Cu
409 - 2,000 L.T. at 28.40% Fe, 1.01% Cu
411 - 4,000 L.T. at 28.65% Fe, 0.78% Cu - 3,000 Broken
5 435 -29,000 L.T. at 38.24% Fe, 1.01% Cu

(c) North Extension

135,000 L.T. at 43.40% Fe, 0.85% Cu. Area North of Goodall Dyke. Zone still open to the North.
Mill Test - Fine grinding necessary and Sulphur content high.
Fe concentrate at all -65 Mesh, 67% Fe, 0.06% Cu, 1.05% Sulphur.
Schedule - one stope is advancing for exploration.

1750 Lower SYK Stope

1. Location - South of pillar on 10600 North, slusher scrams at 1750 elevation.

2. Geology - Irregular lenses of magnetite in skarn.

3. Mill Tests - Good milling ore. Low sulphur content in Fe concentrates. Gold and silver values are below average.

4. Reserves -

   30,000 L.T. at 38.31% Fe, 0.28% Cu. Recoverable from eastern part of 106 N. Pillar.

   44,000 L.T. at 54.23% Fe, 0.48% Cu. Nonrecoverable Pillar located above and to West of stope.

5. Schedule - Originally scheduled for early 1973. Requires 20 ft. of drifting and 16,000 feet of blasthole drilling.

18 S.Y.K. Stope

1. Location - On the Northwest corner of the Lower S.Y.K. Stope. Scram at about the 1800 elevation.

2. Geology - High grade magnetite ore locally with good copper values.

3. Mill Tests - Tested as part of Lower S.Y.K. Expected to be good milling ore.

4. Reserves -

   South Block - 61,000 L.T. at 48.15% Fe, 0.50% Cu

   North Block - 32,000 L.T. at 44.94% Fe, 0.41% Cu

   Ore Pass Pillar - 36,000 L.T. at 50.12% Fe, 0.34% Cu. Required to protect #2 Ore Pass. Can be recovered from North Block Scram.

5. Schedule - The Blocks will be mined in the order listed above. Production is scheduled to start in December 1974, and continue through 1975. Scheduled late to protect #2 Ore Pass.

S.Y.K. 106 N Pillar — Nonrecoverable

16 SYK Stope

1. Location - The south and downward extension of the Yellow Kid ore body between the 1750 Lower South Yellow Kid Stope and the scram on the 1655 Level.

2. Geology - Good grade magnetite with lenses of skarn.

3. Mill Tests - Good milling ore.

4. Reserves - 22,000 Long tons at 43.99% Fe, 0.23% Cu located on the west side of the stope near the 1855 level, immediately south of 18 SYK stope. Ore is drilled but not blasted.

1655

S.Y.K. ~~106 N Pillar~~ Cont'd/

Stope has been overdrawn by 35,000 L.T.

5. Schedule - Stope was scheduled to be finished. Remainder of ore will be left until nearby ramps are not required, probably late in 1974. 16 S.Y.K. undercut pillar is recovered from 14 S.Y.K.

Argo #1 Stope

1. Location - between the Midway and Yellow Kid zones. Ore body lies to the North of the Midway ore bodies and immediately below the West Ramp. Ore extends from 1750 to 1855 elevations.

2. Geology - Irregular lenses of magnetite with chalcopyrite, or skarn with chalcopyrite. Strike approximately E-W with the footwall dipping South at angles as low as 50°.

3. Mill Tests - Mill tests presently being done. No problems expected. Ore is classed as copper mill feeds but should produce a fair tonnage of iron concentrates.

4. Reserves - to be revised on completion of exploration drilling.

35,000 L.T. at 29.27% Fe, 0.97% Cy - Stope
33,000 L.T. at 13.67% Fe, 1.13% Cu - Recoverable. Pillar lies under West Ramp.

5. Schedule - Development ahead of schedule and some test hole drilling is required. Production scheduled to start in September 1974. Recoverable pillar not scheduled.

14 South Yellow Kid

1. Location - Scram on the 1455 Level. Stope includes all of the S.Y.K. ore body from 1655 to 1455 Levels.

2. Geology - Thick tabular body trending E-W and dipping south. Magnetite lenses in skarn with a low chalcopyrite content.

3. Mill Tests - Not tested. No problems expected. One should upgrade easily and yield concentrates with a low sulphur content.

4. Reserves -
59,000 L.T. at 43.66% Fe, 0.16% Cu - Stope
39,000 L.T. at 43.66% Fe, 0.16% Cu - Broken
26,000 L.T. at 50.58% Fe, 0.08% Cu - Recovery of Undercut, pillar from the 16 SYK Stope.

5. Schedule - Production is 48,000 L.T. behind schedule, but delay is not critical. Scheduled to be finished in May 1975.

1550 Midway I

1. Location - At the west end of the Midway Zone, with only the 1550 Prescott to the west. Stope extends from 1550 to 1855 elevations.

2. Geology - Irregular lenses of magnetite separated by skarn, sometimes joining up to form a single thick ore body. Trend is E-W and dip is practically vertical.

3. Mill Tests - A composite sample from the Midway I & II was tested and found to yield a concentrate of 65.00% Fe, 0.045% Cu, 0.40% S, at 40% minus 100 mesh. Work index was average at 12.5 to 13 KWH/L.T.

4. Reserves -

59,000 L.T. at 42.98% Fe, 0.13% Cu - Stope
58,000 L.T. at 41.74% Fe, 0.11% Cu - Recoverable pillar protecting west ramp. Located between Midway I & II Stopes. Extracted from 1455 level.
7,000 L.T. at 42.98% Fe, 0.13% Cu - Broken reserves.
11,000 L.T. at 41.95% Fe, 0.22% Cu - Nonrecoverable pillar. Undercut pillar and outside of economic stope limits on the south side near the top of the stope.

5. Schedule - Presently in production along with 1550 Midway II and 14 Midway III. Production is approximately a month ahead of schedle.
   - Pillar recovery is scheduled for the last five months of 1975.

1550 Midway II

1. Location - Scram on 1550 sublevel. Separated from Midway I by a temporary pillar on the west end. Stope extends from 1550 to about the 1720 elevation.

2. Geology - Ore body is located at the north side of the Midway Zone. Strike is E-W and dip is vertical. The ore is high grade magnetite with average copper values. The irregular shape of the mineralization results in dilution with skarn and a lowering of the iron grade.

3. Mill Tests - Readily cobs to a good grade mill feed. See mill tests for 1550 Midway I.

4. Reserves -

43,000 L.T. at 39.74% Fe, 0.25% Cu - Stope
16,000 L.T. at 39.74% Fe, 0.25% Cu - Broken
16,000 L.T. at 35.19% Fe, 0.70% Cu - Nonrecoverable. Includes 10,000 in U.C. pillar and 6,000 long tons on the North side of the Copper Zone, outside of the stope limits.

5. Schedule - see 1550 Midway I.

1455 Midway III

1. Location - Downward extension of the 1655 Main Midway Stope. Located at the east end of the Midway Zone.

2. Geology - Irregular lenses of magnetite with minor chalcopyrite located along the Sulphur or diorite side of the Midway Zone.

3. Mill Tests - Very good milling ore. Work index 12 KWH/L.T.

4. Reserves - 83,000 L.T. at 39.84% Fe, 0.21% Cu - Stope
2,000 L.T. at 39.84% Fe, 0.21% Cu - Broken

5. Schedule - Presently in production along with 1550 Midway I & II. Part of this stope must be delayed until production is finished from the 1550 Midway.

1550 Midway II Copper Zone

1. Location - Immediately above 1550 Midway II Stope and extending upwards to the 1800 elevation.

2. Geology - A zone of irregular disseminations of chalcopyrite in skarn. Gangue consists of garnet, actinolite, calcite and some epidote. Negligible magnetic iron.

3. Mill Tests - Fine grinding is necessary liberate the chalcopyrite. Copper concentrate contained .097 oz/ton Au and 8.9 oz/ton Ag.
- A good recovery of magnetite is possible in spite of the small amount available. Sulphur content is high in iron concentrates.

4. Reserves - 32,000 L.T. at negligble iron, 0.90% Cu - Stope.

5. Schedule - Production in June 1974.

1455 Midway I

1. Location - at the West end of the Midway Zone directly below 1550 Midway I Stope.

2. Geology - Very irregular lenses of magnetite with minor chalcopyrite. Strike trend E-W, dip variable from vertical to 65^{o} North.

3. Mill Tests - Not tested but believed to be similar in quality to 1455 Midway III.

4. Reserves -

93,000 L.T. at 38.99% Fe, 0.17% Cu - Stope
32,000 L.T. at 39.67% Fe, 0.20% Cu - Recoverable.
Pillar along 10300 E below the West Ramp.

5. Schedule - Must be delayed until 1550 sublevel is no longer needed. Scheduled for January 1975. Pillar recovery is scheduled for last half of 1975.

1550 Prescott Stope

1. Location - The downward extension of the 1655 Prescott Stope. Located to West of the West Ramp on 1550.

2. Geology - Good grade lenses of magnetite separated by skarn and feldspar porphyry. General trend E-W with ore lenses dipping N at $60^{o}+$.

3. Mill Tests - Not tested. Should be similar to 1655 Prescott which was good milling ore.

4. Reserves - 68,000 L.T. at 36.37% Fe, 0.15% Cu - Stope.
   23,000 L.T. at 31.62% Fe, 0.08% Cu - Recoverable pillar on footwall of stope.

5. Schedule - Production scheduled for January 1974. Must precede 1455 Midway I in order to have access to ore pass.

1350 Midway

1. Location - Extension of Midway ore bodies from 1455 to 1350 elevations.

2. Geology - Veins and lenses of magnetite in skarn with very little copper mineralization. Ore extends below 1350 elevation which is considered the economic limit of mining.

3. Mill Tests - Cobbing produces an acceptable mill feed yielding a high grade concentrate with a low sulphur content. Work index 12 KWH/L.T.

4. Reserves - Geological reserves since no mining layout has been made. 155,000 L.T. at 42.60% Fe, 0.08% Cu.

5. Schedule - Not Scheduled.

1350 South Yellow Kid

1. Location - Below and to the South of the 1455 SYK Stope

2. Geology - Lenses of high grade magnetite with very little chalcopyrite. Mineralization extends to depth but 1350 elevation is the economic cut-off.

3. Mill Tests - Results very similar to 1350 Midway. Ore slightly softer.

4. Reserves - Geological reserves, no mining layout. 65,000 L.T. at 43.00% Fe 0.05% Cu.

5. Schedule - Not included in schedule.

Anomaly "A"

1. Location - Located to West of Prescott Pit. Serviced by separate adit.

2. Geology - A relatively thin magnetite ore body with some chalcopyrite and pyrite occurring along the skarn zone between a diorite dyke and limestone. Zone strikes North and dips are usually quite steep.

3. Mill Tests - Mill tests indicated that fine grinding would be necessary. However, this was not found to be true in actual milling of the ore.
   - A sulphur content of 0.57% was attained in mill tests.
   - Work index was 11.5 KWH/L.T.

4. Reserves - 68,000 L.T. at 49.22% Fe, 0.17% Cu - Stope

5. Schedule - Initial stope finished, development work and longhole drilling required before final stope can be mined. Originally production from the North block was scheduled for August 1973, but plans now call for a later production date and stockpiling of the ore for milling during summer shutdowns. At least 20,000 L.T. should be mined by July 1974.

Paxton Pit

1. Location - ore is located at the north end of the North Paxton Extension.

2. Geology - Copper mineralization with some magnetite occurs in the floor of the pit. Mineralization strikes N15^{o}W and dips at low angles (15^{o}) to the S.W. Very little stripping is required.

   Ore contains a high percentage of pyrite and pyrrhotite.

3. Mill Tests - Classed as copper mill feed. Tests indicate iron concentrates would run from 1.4 to 1.5% Sulphur. Copper recovery acceptable with copper concentrates running over 6 oz/ton Ag and around 0.3 oz/ton Au.

4. Reserves

   216,000 L.T. at 25.93% Fe (estimated magnetic Fe) & 0.76% Cu

5. Schedule - Scheduled for last six months of 1975 for a total production of 60,000 L.T.
   - Ore lies below parts of the last three benches and very little stripping is required.
   - Test holes have been drilled on 30 foot centres to check mineralization.
   - Pit walls and roads would have to be put in condition before production could start.

Lake Room & Pillar Stopes

1. Location - between the end of the Lake Pit and extending N.W. to the 2070 Lake Stope

2. Geology - The ore zone varies from 16 to 45 feet thick. The strike attitude is NW and dips are at low angles to the SW, with a plunge of approximately N 15°W at 10°. Mineralization consists of magnetite, relatively minor chalcoyprite and varying amounts of pyrite and pyrrhotite.

3. Mill Tests - Mill tests indicate a generally hard grinding ore producing a high iron concentrate with sulphur values in excess of 1%. An acceptable but above average amount of copper is retained in the iron concentrates and copper concentrates average about 21% Cu.
 - Room and Pillar mining produces a high grade mill feed with very little waste dilution and good fragmentation.

4. Reserves - A detailed stope inventory is being prepared and at June 30th, 1973, the reserves were:

 182,000 Long tons at 46.62% Fe and 0.15% Cu - Stopes.

 Actual minable tonnage will be less than this since the mining layout has been revised and some high pyrrhotite areas abandoned.

 72,000 Long tons at 47.30% Fe and 0.15% Cu - Recoverable.

 Pillar tonnage was originally estimated at 50% of the total pillar reserves. It is doubtful if this percentage can be achieved.

 72,000 Long tons at 47.30% Fe and 0.15% Cu - Unrecoverable.

 2,000 Long tons at 46.62% Fe and 0.15% Cu - Broken.

5. Schedule - Originally scheduled at 8,000 L.T. per month but 6,000 L.T. is more realistic. Ore will be stockpiled for summer shutdowns and blended with Anomaly "A" production.

2070 Lake Stope

1. Location - North end of the Lake Ore Body.

2. Geology - Thickness and dip are suitable for longhole mining. Attitude is NW by steep angles to SW. One is magnetite with a high percentage of pyrite and pyrrhotite particularly close to the margins of the ore body.

3. Mill Tests - The 2070 Lake Stope is the source of the high sulphur iron concentrates. The remaining Sough Block is presently being tested and is apparently even worse than the North Block.

4. Reserves -

 342,000 L.T. at 43.54% Fe and 0.15% Cu - Stope
 40,000 L.T. at 43.54% Fe and 0.15% Cu - Broken
 2,000 L.T. at 41.69% Fe and 0.15% Cu - Recoverable Pillar
 66,000 L.T. at 47.22% Fe and 0.18% Cu - Nonrecoverable
 Pillar beyond limits of draw.

2070 Lake Stope Cont'd.

5. Schedule - Production scheduled at 15,000 long tons per month through to August 1975.

R. G. PATERSON,
Chief Geologist.

RGP:vlm
July 23, 1973.

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

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

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).

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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.

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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.

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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 on 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...

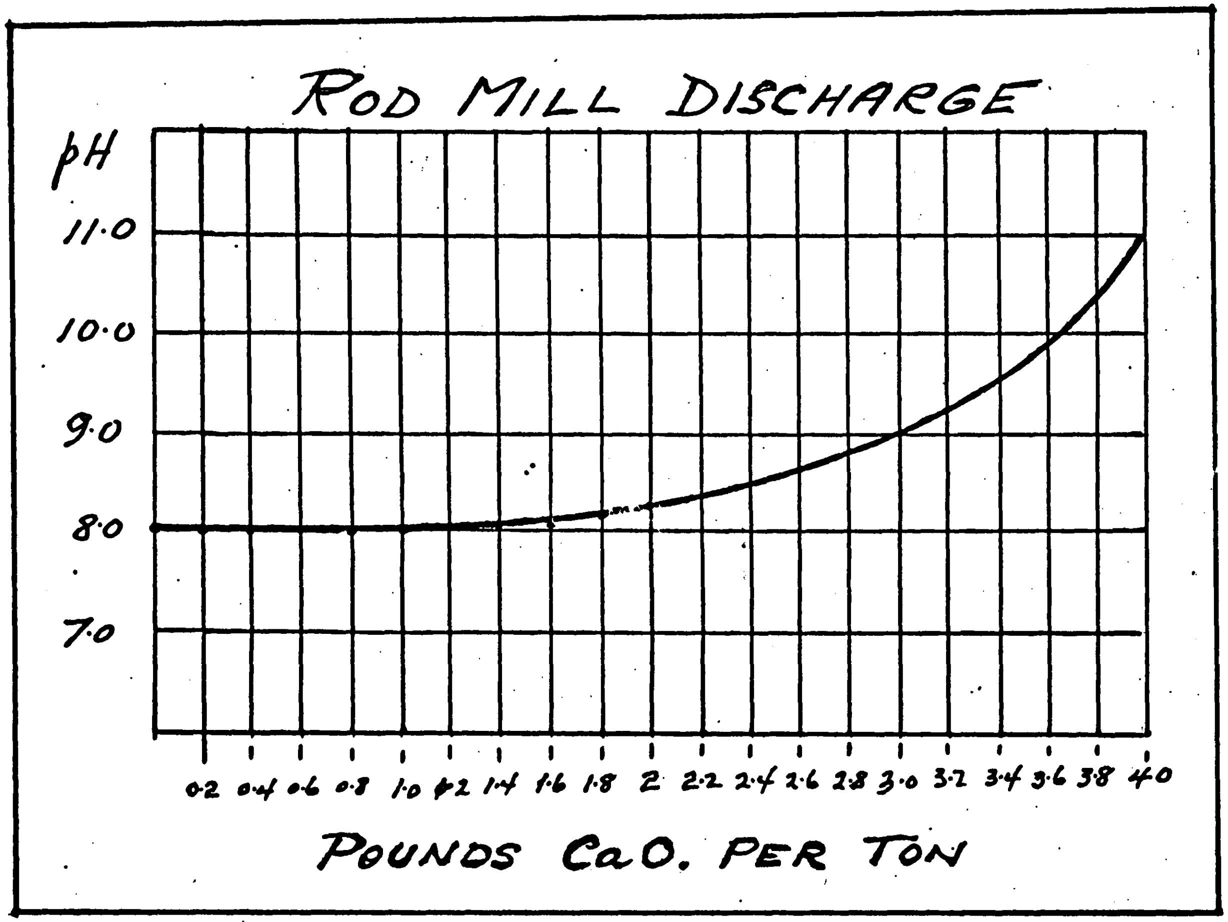
Several quick hard air blows are made into the water filled pipe back to the tanks then the bottom drain is quickly opened. If there is any plug up, the process is repeated. Shell removal is excellent and complete.

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.

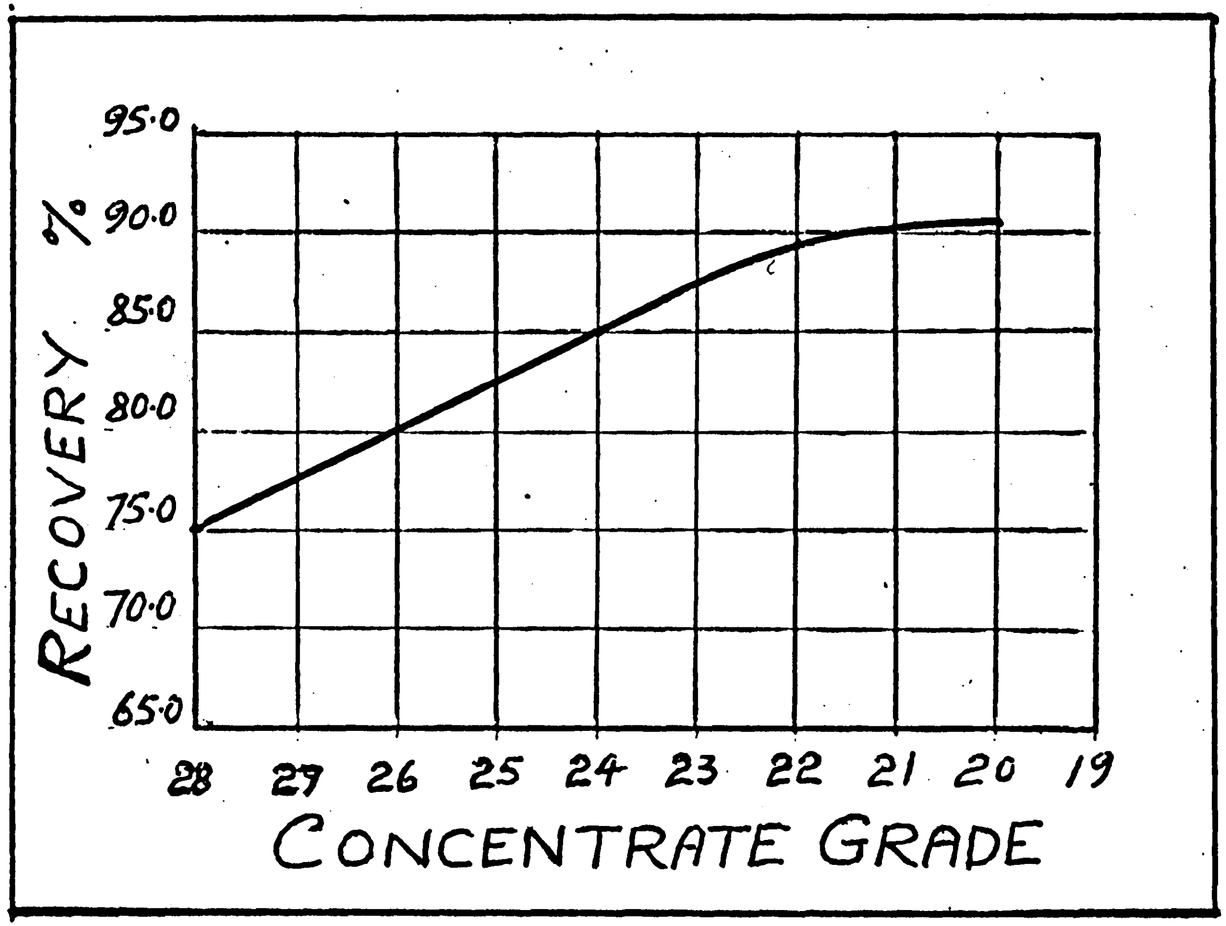
Summing up the 17 years of operation we find that the full use of sea water gives us a very stable easily controlled float. Reagent consumption is greatly reduced and reagent feeding has been simplified. The natural pH of the sea water is the most satisfactory level for Texada ores. It is extremely important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

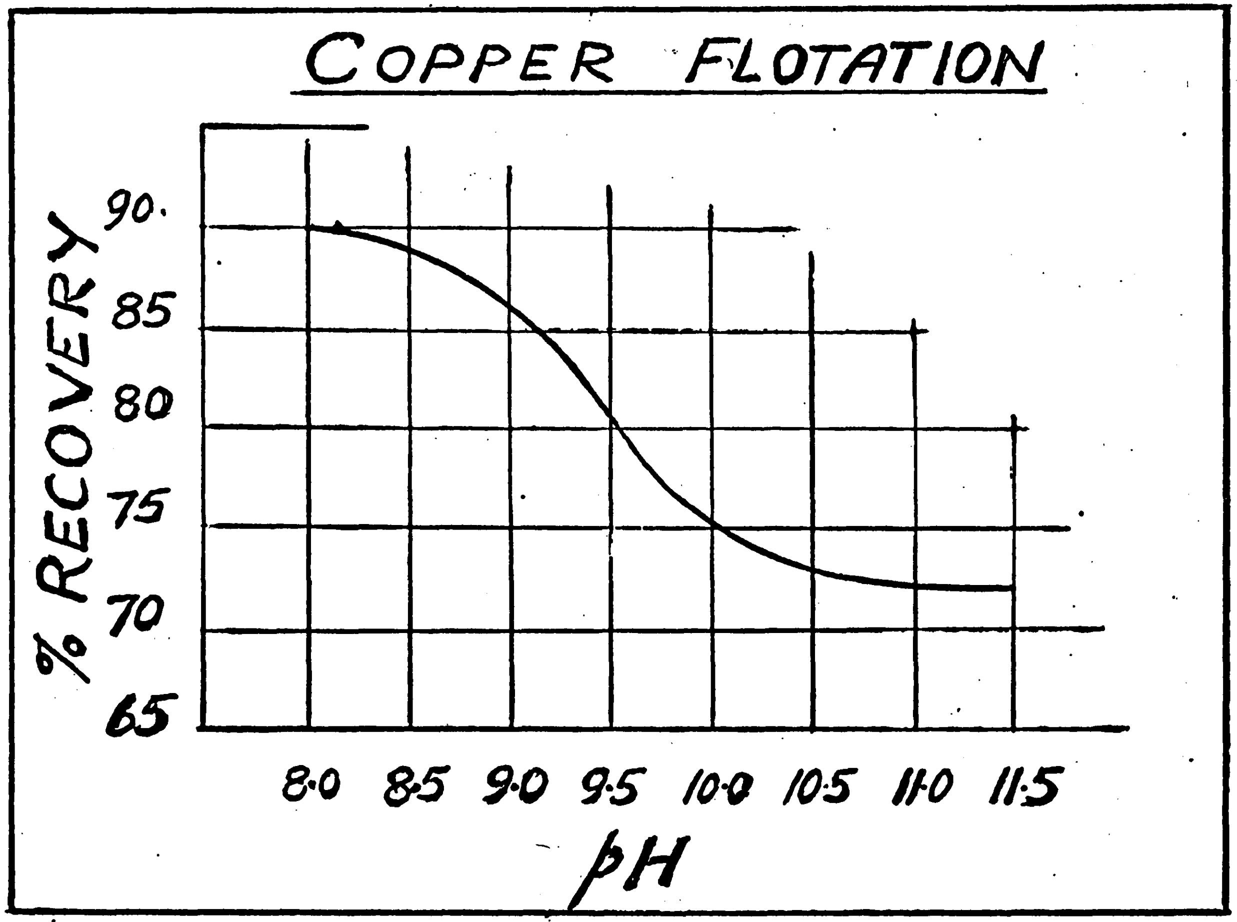
Atmospheric corrosion is severe and causes more problems than the sea water in the processing. This is a location problem and not a function of the use of sea water.


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

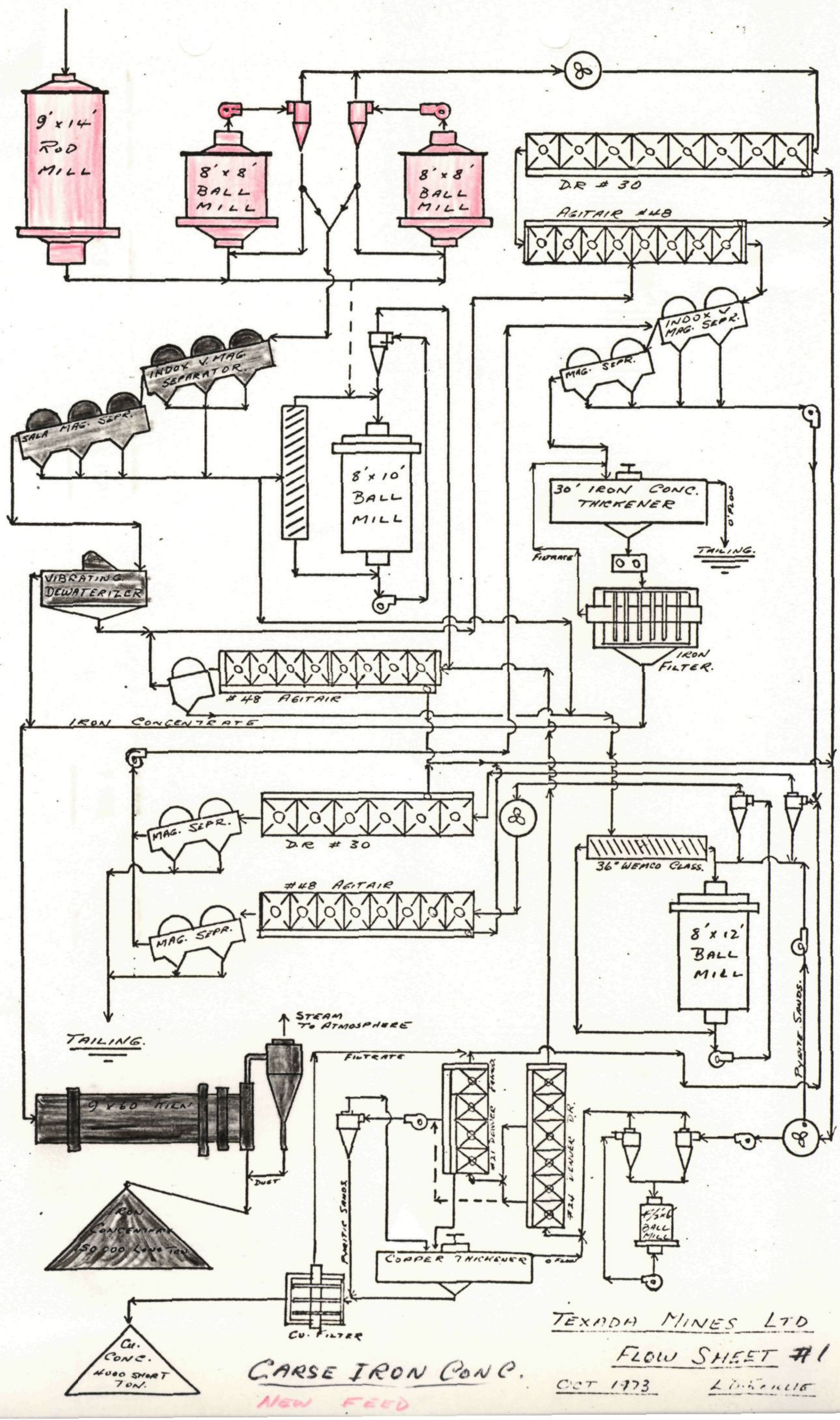
GRAPH #1


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


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



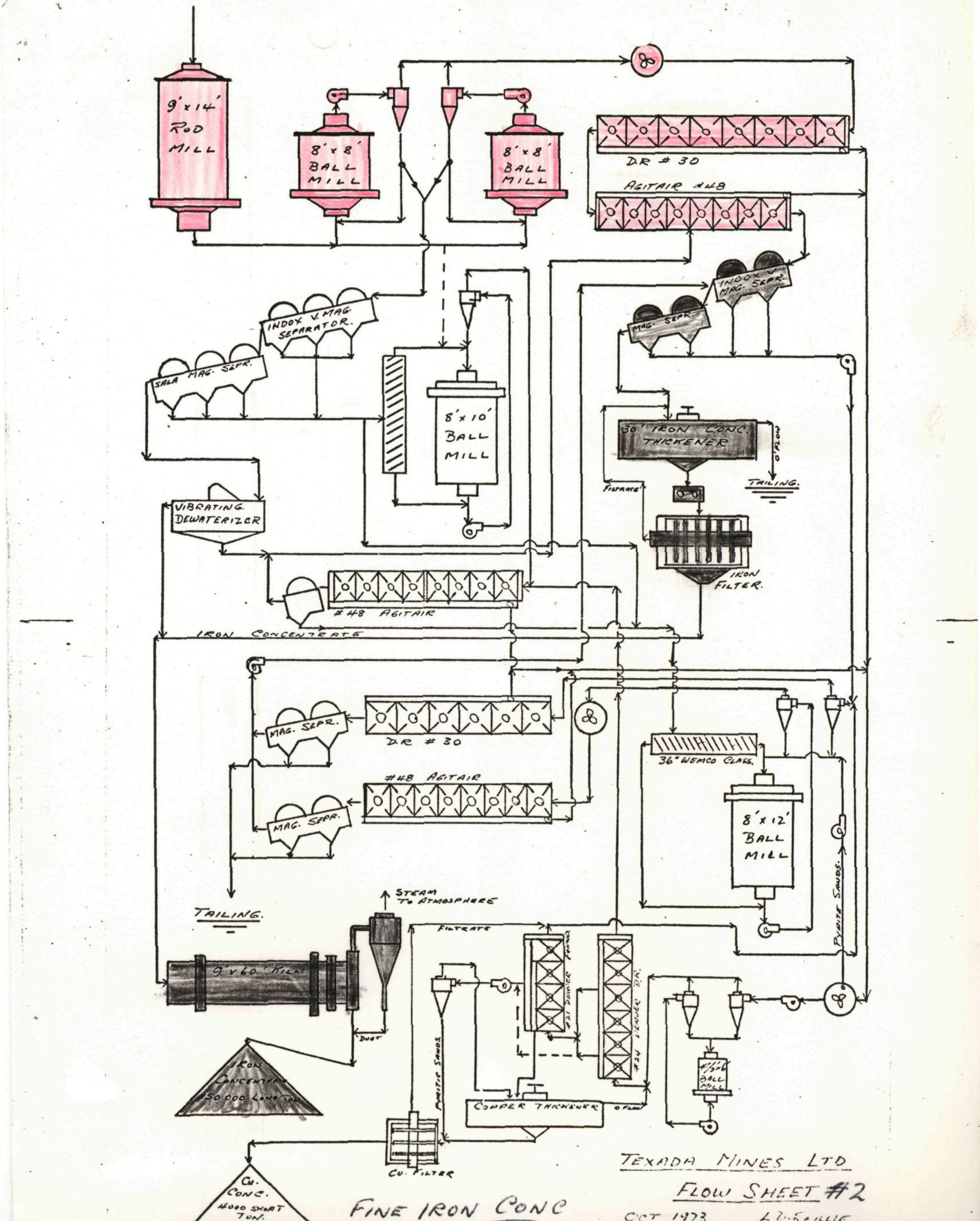
TEXADA MINES LTD

FLOW SHEET #1

CARSE IRON CONC.

NEW FEED

OCT 1973



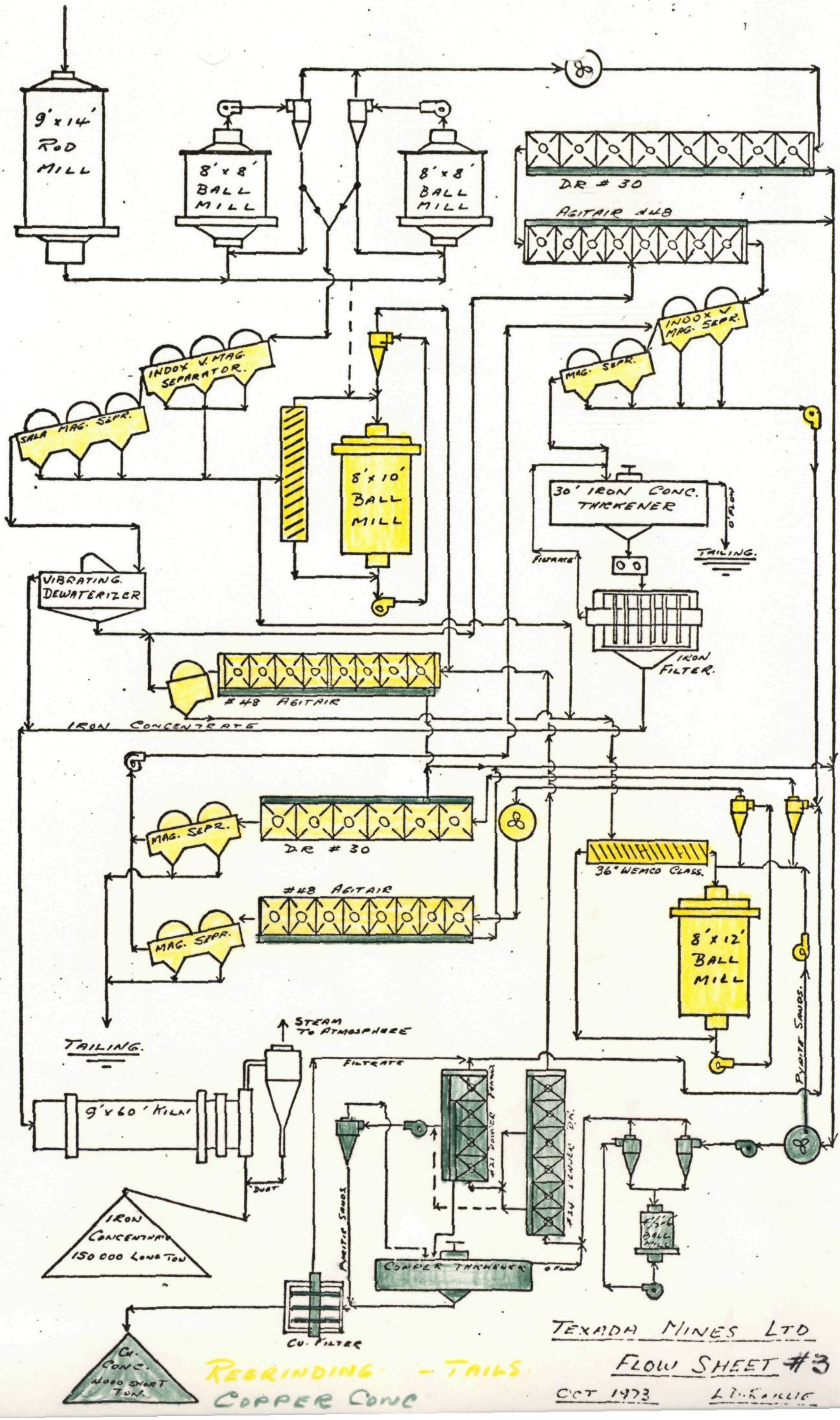
TEXADA MINES LTD

FLOW SHEET #2

OCT 1973

FINE IRON CONC

NEW FEED



REGRINDING - TAILS

COPPER CONC

TEXADA MINES LTD

FLOW SHEET #3

OCT 1973

L.T. Kallie

Texada Mines Ltd. - Production Statistics 1973

R. Paterson

Month	Hoisted to: Fe Surge	Cu Surge	Cu R.D.	Ore Hauled to Mill from: Reclaim	Cu R.D.	Fe R.D.	Lake SP	Anomaly A	Lake R&P to Lake S.P.	To A.A. S.P.	Waste to Aggregate Prod.	Lake W.D.	UG - W.D.	Misc Hauling
in 658	79134	5524	6367	1375	4668	—	—	—	6044	—	2439	48	—	4137 - CWD +
eb 653	75251	6402	6182	5525	6777	—	—	—	6048	—	5586	48	—	4464 - CWD; 429 - Agg to
March 5	83182	6083	6105	5400	5836	—	—	—	4224	—	—	—	—	760 - Agg to; 735 - Waste to
pr. 130	67793	9437	230	5325	4857	—	—	—	4296	168	—	—	—	—
May 3045	69432	13613	8574	2275	1925	—	—	—	1251	6507	—	—	—	—
une 7605	68315	9290	12654	850	3325	—	—	—	—	5484	—	—	—	—
uly 5186	35186	—	897	1400	8625	—	11583	13959	192	—	—	—	—	—
ug. 7988	64848	13140	11040	—	3025	—	—	—	4968	—	—	—	—	—
ept. 5696	56088	11608	3564	Fe Conc 400	2352	—	—	—	4387	—	1062 967	—	—	—
Oct. 2280	48408	22872	1164	—	—	—	—	—	4488	—	2536	—	360	2636 WD to
Nov. 2760	82760	—	3324	1550	—	—	—	—	3078	—	—	—	—	242 - Agg to
Dec. 1616	40588	11028	—	800	—	—	—	—	3684	—	—	—	240	—
otal	~~770988~~ 770985	114197	60101	24900	41390	0	11583	13959	42660	12159	11528	96	600	10767; ~~6661~~ 1237 CWD - P.; 1431 Agg - UG; 735 Waste - H

Month	Ore Milled	Ore Rod Mill	Waste Cobbed	Conc. Prod. Fe (L.T.)	Conc. Prod. Cu (S.T.)	Conc. Shipped Fe (L.T.)	Conc. Shipped Cu (S.T.)	Hrs.
in	86393	71,117	15276	42688	788.0	66062	3551.5	598/616
eb	95947	69597	26350	44925	866.2	—	—	567/576
March	109,678	77711	31967	51406	1204.6	66132	—	612/624
pr.	89944	66651	23293	41988	1062.0	53763	—	581/592
May	82218	60958	21260	48246	676.1	67258	3713.9	559/568
une	81802	57959	23843	36138	701.6	—	—	536/600
uly	62933	50648	12285	36769	532.2	58580	—	471/496
ug.	82634	62485	20149	37948	838.3	51715	—	566/600
ept.	71303	53459	17844	31042	777.5	—	—	484/496
Oct.	69227	52877	16350	28151	872.7	69395	3439.4	509/520
Nov.	88220	61789	26431	36488	629.1	—	—	487/504
Dec.	52112	39999	12113	20078	476.8	52600	—	349/360
otal	972,411 (~~97~~)	725,250	247,161	455,867	9,425.1	485,505	10,704.8	6319/6552

Production Statistics For 1973 – Summary Fe Shipments

Net. Wt.	Moisture	Dry Wt.	Fe Grade	Eu Grade	S	P	Mn	SiO_2	Al_2O_3	CaO	MgO
66062	6.3	61900.09	64.76	0.069	0.821	0.010	0.14	4.56	0.61	2.33	0.49
66132	5.7	62362.18	64.89	0.074	0.589	0.010	—	4.04	0.75	—	—
53763	5.2	50967.32	64.68	0.052	1.190	0.010	0.14	4.66	0.78	2.39	0.54
67258	5.0	63895.10	64.78	0.048	0.780	0.009	0.16	4.58	0.78	2.39	0.52
58580	6.2	54948.04	63.34	0.072	1.570	0.010	—	—	—	—	—
51715	4.7	49284.40	63.19	0.052	0.797*	0.010	0.13	5.02	0.61	2.61	0.51
69395	5.2	65786.46	63.92	0.072	0.993	0.010	0.16	5.06	0.77	2.42	0.53
52600	6.0	49440.00	63.94	0.048	0.865	0.010	NA	N.A.	N.A.	N.A.	N.A.
485,505	5.538	458,583.89									
	0%		64.22	0.062	0.941	0.010	0.146	4.653	0.717	2.428	0.518
Nat. = $\frac{LWT - DLT}{LWT}(100)$ =		5.545	60.66	0.059	0.889	0.009	0.138	4.395	0.677	2.293	0.489

543,765.56 S.W.T. 493,300.91 W.M.T.
513,613.96 S.D.T. 465,947.52 D.M.T.

Milled – 972,411 L.W.T. @ 2.5% H_2O = 948,101 D.L.T.
Production – 455,867 L.W.T. @ 5.538 % H_2O = 430,621 D.L.T. @ 64.22 % Fe
∴ Tails @ 0% H_2O = 517,480 D.L.T @ 12.0 % Fe

Fe Heads = [(430,621)(64.22) + (517,480)(12.0)] ÷ 948,101 = 35.72 % Fe (dry)

or $\frac{(430621)(64.22) + (517480)(12)}{972411}$ = 34.83 % Fe natural @ 2.5% H_2O

Copper Shipments - 1973

Shipment	S.W.T.	Moisture	S.D.T.	Cu %	lbs. of Cu	Au oz/T	Oz. of Gold	Ag oz/T	Oz. of Ag
#41	3551.52	8.906	3235.44	22.465	1,453,683.192	.188	608.263	5.56	17,989.046
#42	3713.92	9.358	3364.81	22.655	1,524,595.411	.210	706.610	5.47	18,405.511
#43	3439.43	9.478	3113.38	23.595	1,469,204.022	.175	544.842	6.78	21,108.716
	10,704.87	9.247	9713.63		4,447,482.625		1,859.715		57,503.273
Wet Assays (Natural)				20.773		.174		5.37	
Dry Assays				22.893		.191		5.92	

Copper Production – 1973

Production 9,425.1 S.W.T.

Moisture 9.247 %

Conc. prod. dry = 8,553.6 S.D.T.

Cu content dry @ 22.893 % Cu = 1958.18 S.T. Cu

Au @ .191 oz/T = 1,633.738 oz Au

Ag @ 5.92 oz/T = 50,637.312 oz Ag

Conc. Ratio

Input – S.T. milled (dry, assuming 2.5 % H_2O) = (948,101 DLT)(1.12) = 1,061,873.1 S.D.T

Output 8,553.6 S.D.T.

$$\text{Conc. Ratio} = \frac{1{,}061{,}873.1}{8553.6} = \frac{124}{1}$$

$$\text{Probable input assay} = \frac{22.893}{124} = 0.185\ \%$$ assuming 100% recovery

$$\text{Probable mill feed assay}\ \frac{0.185}{.60} = 0.308\ \%\ Cu$$

Production Statistics - 1973

Ground	725,250
Cobbed	247,161
Milled	972411
to Mill	
Fe	770985
Cu.	114,197
Reclaim	24900
Cu Res	41390
ke S.P	11583
A S.P	13959
Total	977014 ~~976,997~~
Dec 31/73	28018
Dec 31/72	25615 - 3000 (Adj. Jan/73)
ff	+ 5103
e Bin	
Dec 31/73	2800
Dec 31/72	3600
Diff	-800
Change	+ 4603
Milled	972411 - Checks

Reserve Dump Total Dec 31/72	19655 L.T.
Del. to Dump 1973 from	
Shaft-Cu RD	60101
Lake	42660
AA	12159
Total	114,920
From Res Dumps	
Cu RD	41390
Lake SP	11583
AA S.P.	13959
Total	66932
Balance	67643 — Book 67295 - Adj. unnecessary

Fe Conc Stockpile Dec 31/72	75083
Prod 1973	455867
Total	530950
Fe shipments	485505
Fe Conc Stockpile Dec 31/73	45445 Checks
Cu Conc Stockpile Dec 31/72	3235.1
Prod. 1973	9425.1
Total	12660.2
Cu Shipments	10704.8
Cu Conc S.P. Dec 31/73	1955.4 Checks

OFFICE OF THE CHIEF ANALYST AND ASSAYER

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

CS REJECT ALS FINES TEXADA MINE CS.

SEMI QUANTITATIVE SPECTROGRAPHIC ANALYSIS

Laboratory No.	13946M	13947M					
Submitter's No.	#1	#2					
Si	<10.0	>10.0					
Mn	0.25	0.3					
Al	7.0	7.0					
Mg	0.75	0.75					
Pb	T↓	—					
Ca	10+	>15.0					
Fe	>20.0	10.0					
V	T	T					
Cu	0.5	0.35					
Ag	T+	T					
Zn	0.025	—					
Na	0.35	0.45					
K	—	—					
Ti	0.07	0.1					
Zr	T	T					
Ni	T	T					
Co	T	T					
Sr	0.02	0.04					
Cr	T	T					
Ba	T	T					
Traces:	Au, Bi, Ga, Mo, W.	Mo, Bi, Ga, Au					

DATE May 7, 1974.

PROPERTY FILE

Paul F. Ralph
for. CHIEF ANALYST AND ASSAYER.

TEXADA MINES LTD.
GILLIES BAY,
BRITISH COLUMBIA

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

To: A. M. Walker April 29, 1974

From: R. G. Paterson

Subject: Minable Ore Reserves & New Taxation

65

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.

TONNAGE: 609,050

Stope	Tonnage (L.T.)	
S. Prescott	178,000	NO COSTS. (MUST ELIMINATE RAMP.
1350 Midway	155,000	LACK OF DEVELOPMENT
1350 S. Yellow Kid	65,000	NO DEVELOPMENT.)
Paxton Pit	215,000	← ACCESS ROAD – ALL CU ORE
Lake Room & Pillar	217,000	– COST ↗ THAN AV.
2070 Lake Stope	303,000	– TRIPLE HANDLING. –
402 Stope	60,000	– GRADE ↘
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.	– ACCESS.
North Extension	88,000 L.T.	ACCESS & VENTILATION.
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.

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.

PROPERTY FILE

R. G. Paterson

R. G. PATERSON
Chief Geologist

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
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 FLAT ZONE	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, exp
7) 1455 S.Y.K.	73	45.04	0.13	26	43.66	0.16	- 47	Revised & Productio
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

ESERVES IN 1000's OF L.TONS

PERIOD ENDING March 31, 1974

BY: R. G. Paterson

LOCK 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
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			
Positive	1697	38.27	0.33
Probable	359	27.85	0.87
Broken	136	36.57	0.35
Recoverable Pillar	684	40.26	0.31
TOTAL	2876	37.36	0.39

Minable Copper Reserves		
391	27.00	0.82
303	26.97	1.01
35	20.11	0.84
115	32.33	0.92
844	27.43	0.90

OTNOTES:

Minable copper reserves are 29% of total minable reserves.

TEXADA MINES LTD.

ORE RESERVES

SERVES IN 1000's OF L.TONS

PERIOD ENDING March 31, 1974

BY: R. G. Paterson

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

TEXADA MINES LTD.
ORE RESERVES

ESERVES IN 1000's OF L.TONS

PERIOD ENDING March 31, 1974

BY: R. G. Paterson

LOCK 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.
Roi 2B	24	19.33	1.27							27	27.56	1.06	8	19.33	1.27	59	23.08	1.
BTOTAL	27	21.24	1.14				7	37.78	0.07	48	34.86	0.68	17	32.52	0.65	99	30.23	0.
55 - 1855 Levels																		
rth 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.
per S.Y.K.		- Nil -								11	45.44	0.31	27	43.97	0.27	38	44.38	0.
Roi #1 Ore Body																		
tope #1A		- Nil -					109	42.44	0.10							109	42.44	0.
llar #1A										188	37.34	0.36				188	37.34	0.
tope #1B	246	33.00	0.27							19	44.09	0.56				265	33.80	[illegible]
llar #1B							30	39.80	0.35	23	39.80	0.35				53	39.80	0.
tope #1C		- Nil -					60	41.54	0.18	13	31.99	0.40				73	39.83	0.
BTOTAL Le Roi #1	(246	33.00	0.27)				(199	41.47	0.16)	(243	37.81	0.38)				(688	37.23	0.
idway Cu Zone				41	21.26	1.86										41	21.26	1.
outh Prescott Zone				56	32.64	0.13	122	38.58	0.27							178	36.71	0.
8-106 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.
8-106 R & P Stopes	27	30.83	1.07				12	38.47	0.74	54	29.14	1.00				93	30.84	0.
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.
BTOTAL	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.