GM 26835

PRELIMINARY CAPITAL AND OPERATING COSTS ESTIMATE FOR THE PRODUCTION OF IRON ORE CONCENTRATES

PRELIMINARY CAPITAL AND OPERATING COST ESTIMATE

6

for the

PRODUCTION OF IRON ORE CONCENTRATES

from the
property of

E X P O   I R O N   L I M I T E D

Toronto, Ontario, Canada
August 1, 1970

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1. SUMMARY

Expo Iron Limited holds title to 30 claims in Houdet Township, Quebec, within which is found a potential open pit iron deposit containing an estimated drill-proven reserve of 11,905,200 tons* with an average grade of 16.91% Soluble Iron. Additional geological inferred reserves are estimated to be 4,000,000 tons of equivalent grade material.

The present study demonstrates that an annual production of 208,600 tons of dry concentrate containing 69% iron can be produced from the Expo Iron deposit. Quantitatively this would represent sufficient concentrates to supply an iron metallizing plant capable of producing 150,000 tons per year of pellets containing 95% iron. The mine life of the drill-proven reserves at this production rate would be 10.2 years.

To place the property in production it is estimated that an overall capital investment of $6,257,200 will be required, exclusive of all interest and financing charges. Estimated direct operating cost will be $10.29 per ton of concentrate produced. No allowance has been made for interest, amortization or depreciation; nor has there been an inclusion of Quebec Mining or Federal Corporation taxes.

* Calculations throughout this report are based on a short ton of 2000 lbs.

2. INTRODUCTION

The present report contains an outline of the geology and iron reserves potential of the Expo Iron property together with an operating plan and cost estimate for the production of iron ore concentrates. Reports on analytical work and ore dressing tests conducted by Lakefield Research of Canada append this report together with a comprehensive study of the test results, mill design and flowsheet prepared by H. E. Neal of F. E. Neal & Associates.

Terms of reference for operating plans, plant layout, and capital and operating cost estimates relate to the production of sufficient iron ore concentrates, (69% Fe) to be used in the production of 150,000 tons of metallized pellets (95% Fe) annually.

While it is recognized that concentrates produced from the Expo Iron deposit by commercial magnetic separation means will be high in sulphur, the terms of reference of the present study precluded any investigation of the means and costs of lowering the sulphur content.

Tests are currently being conducted for the extraction of sulphur-bearing minerals in the concentrates and preliminary physical results have proven successful. Further tests on the lowering of sulphur will have to be carried out to supply data for a cost analysis of pertinent flowsheet changes should this prove desirable.

In general the handling of sulphur in the concentrates is beyond the scope of the present report.

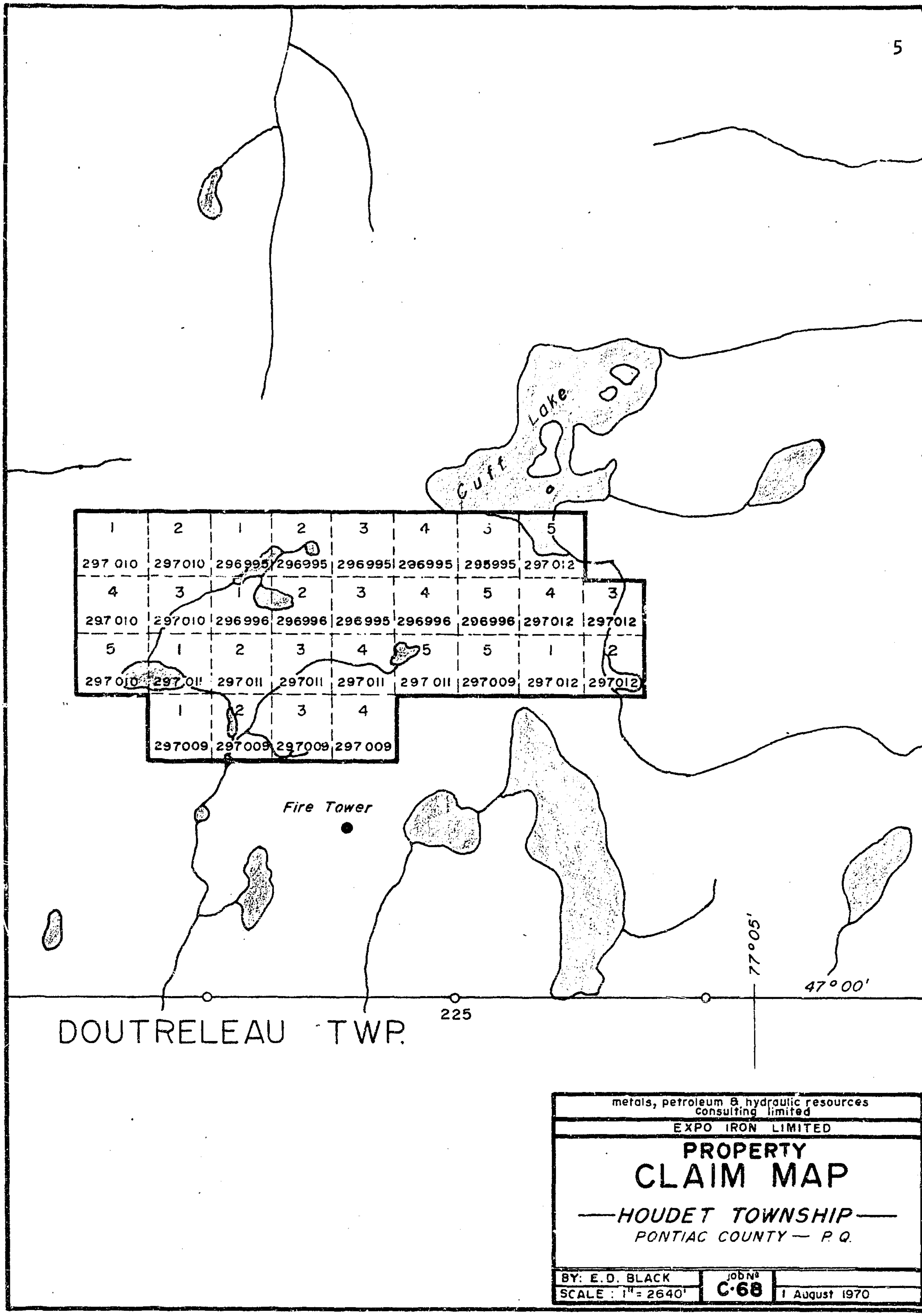
3. P R O P E R T Y

The Expo Iron property consists of 30 contiguous 40-acre mining claims, each situated in Houdet Township, Pontiac County, Province of Quebec.

Records at the Quebec Department of Natural Resources in Quebec City show that the mining claims are in good standing and registered in the name of Expo Iron Limited. Following are the pertinent claims, licence numbers and renewal dates involved:

Licence No.	Claim Numbers	Renewal Dates
297009	Claims 1 - 5 inclusive	May 3, 1971
297010	Claims 1 - 5 inclusive	May 1, 1971
297011	Claims 1 - 5 inclusive	May 3, 1971
297012	Claims 1 - 5 inclusive	May 2, 1971
296995	Claims 1 - 5 inclusive	Apr.26,1971
296996	Claims 1 - 5 inclusive	Apr.27,1971

Expo Iron Limited is a public company incorporated under the Quebec Mining Companies Act.



Cuff Lake
1 297 010
2 297010
1 296995
2 296995
3 296995
4 296995
5 296995
5 297012
4 297 010
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1 296996
2 296996
3 296995
4 296996
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4 297012
3 297012
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1 297011
2 297011
3 297011
4 297011
5 297 011
5 297009
1 297 012
2 297012
1 297009
2 297009
3 297009
4 297 009
Fire Tower
77° 05'
47° 00'
225
DOUTRELEAU TWP.
metals, petroleum & hydraulic resources consulting limited
EXPO IRON LIMITED
PROPERTY
CLAIM MAP
HOUDET TOWNSHIP
PONTIAC COUNTY — P.Q.
BY: E.D. BLACK
SCALE : 1" = 2640'
Job No C-68
1 August 1970

4. LOCATION AND ACCESS

The property is situated 87 miles north of Mont Laurier in the south-central part of Houdet Township. Access is by way of a 29-mile gravel truck-road leading west from the Mont Laurier-Val d'Or Highway at Mile Post 87. This truck-road is owned and maintained by the local lumbering companies. A two-mile branch road passes directly through the middle of the Expo claim group one half mile to the west of Cuff Lake.

The road distance from Montreal to the property is 261 miles. Combined road and rail distance to Montreal is approximately the same.

The property is also accessible by way of Highway No. 11 from Hull and Ottawa, a total distance of 198 miles. Locally a new all-weather gravel road is being built which will pass four miles to the south of the property and will connect with Highway No. 58 at Mile Post 59.

5. HISTORY AND EXPLORATION RECORD

The property forms part of a group of claims originally held by O'Leary Malartic Mines Limited in 1957-58. This company carried out prospecting, trenching, sampling and a dip needle survey.

In 1957, the property was optioned to Holannah Mines Ltd. which organization carried out a limited diamond drilling programme. Holannah did not exercise its option and O'Leary Malartic allowed the claims to lapse in 1959.

Claims covering the iron-bearing zones were subsequently re-acquired by Trans-Nation Minerals Ltd. and a detailed ground magnetometer and outcrop mapping operation was completed in the summer of 1960. No further work followed and the ground was once again allowed to revert to the Crown.

In 1963 the property was restaked on behalf of American Chibougamau Mines Ltd. This company carried out approximately 6500 feet of diamond drilling in 19 holes.

According to some reports, concentration tests were carried out in some of the early stages; however, information regarding the type of test work and results is lacking and it is assumed that this metallurgical work was done by Holannah Mines Limited.

American Chibougamau Mines Ltd. constructed a bush road to the main showing and bulldozer-stripped approximately 2000 feet of strike-length along the mineralized zone.

American Chibougamau subsequently dropped the property and the present group of 30 claims was staked in April of 1969 by Mr. C. E. Gauthier on behalf of Mr. C. Charlebois. The latter transferred the present claim group to Expo Iron Limited.

In the fall of 1969, on behalf of Expo Iron Limited, Metals, Petroleum & Hydraulic Resources Consulting Limited managed a programme of systematic geological mapping, ground magnetometer work and close spaced diamond drilling. Drilling amounted to 4207 feet in 15 holes. The results of this exploration work, subsequent laboratory analyses and ore concentration tests, carried out on the drill core by Lakefield Research of Canada Ltd., proved encouraging.

Additional diamond drilling amounting to 2759.5 feet in 18 holes was completed in June of 1970. This work substantiated earlier geological interpretation and provided limits for reserve and grade calculations.

Additional analyses and ore dressing tests were conducted in early 1970.

Together the results of the 1969 and 1970 exploration and laboratory test work form the technical backup for the present report.

6. TOPOGRAPHY AND DRAINAGE

The mean elevation of south-central Houdet Township is approximately 1200 feet above sea level. The general Expo Iron property area is gently rolling with a local relief of slightly in excess of 100 feet.

The Iron deposits lie along the crest of a 50 to 100 foot ridge between Cuff Lake and Kearney Lake. This ridge splits the drainage. Cuff Lake and its tributary streams flow northward and Kearney Lake and its outlets flow southward. Both drainage systems eventually find their way into the Coulonge River which flows south into the Ottawa-St. Lawrence system at Fort Coulonge.

The local topography and drainage are well suited for the supply of fresh water and disposal of waste water from mining operations. No water shortage, contamination problems or water pollution hazards are envisioned.

7. CLIMATE AND VEGETATION

Winters in the property area are cold and snowy compared with more southerly populated areas such as Montreal and Ottawa. Summers are generally warm and wet and the Spring and Fall periods are more-or-less wet and cool. Combined mean annual rain and snowfall amounts to 30 - 35 inches of precipitation.

Ordinarily the lakes begin to freeze in late November and breakup commonly occurs toward the end of April.

The local bush is thick with a mature cover of mixed spruce, fir, tamarack, white pine, birch and poplar. The underbrush is not generally considered heavy.

8. GEOLOGY

8.1 Regional Setting - All of the consolidated rocks in the area are of Precambrian Age.

Most of the region is underlain by orthogneiss, paragneiss and injection gneiss. Pink granite and pegmatite, both relatively scarce, constitute the youngest intrusive rocks.

Localized iron formation forms part of the paragneiss series lying at or near the contact between the orthogneiss and injection gneiss. Biotite and garnetiferous biotite gneiss are intimately related with the iron formation. Pegmatites intrude and transect the iron formation at several points, however, these injected rocks do not represent a volumetrically large portion of the outcroppings.

From a structural point of view, regional mapping illustrates a pronounced east-west trend, within dominantly northeastward to northward dipping foliated rocks. Folding is pronounced and best illustrated in the banded iron formation where surface trends can be interpolated by magnetic survey data. Drag folding is also abundantly evident. Faulting is minor both in extent and displacement and apparently does not influence the structure of the local iron formation.

Metamorphism is of the garnet-amphibolic facies and all of the rocks are of medium to coarse crystallinity.

The region has been glaciated and a generous thickness of till overlies most of the ledge-rock; consequently, outcrops are scarce. Depressed areas are lake, swamp or muskeg filled.

8.2 Local Geology - Within the Expo claim group combined outcrop and magnetic survey data have outlined a complexly folded iron-bearing zone. Broad dimensions of this zone are approximately 14,000 feet in length by 100 to 500 feet in width.

Within this general iron-bearing zone iron formation occurs as a layered, tabular body, locally folded into a complex of anticlinal and synclinal structures. These structures are east-west trending, northeasterly plunging and generally overturned toward the south.

The iron formation lies between biotite granite gneiss on the hanging wall and carbonate-bearing biotite and granite gneiss on the footwall. At least three separate iron formation units are recognized. The principal iron members are separated by low-iron-bearing horizons more-or-less equivalent in thickness to the iron formation units themselves. The low-iron-bearing unit lying between the two principal iron members could possibly be selectively mined.

Texturally and compositionally the iron formation can best be described and identified as a low-grade, laminated, granular, quartz-ferro-magnesium-silicate-magnetite, metataconite. Garnetiferous-biotite sub-units are intimately interstratified with quartz-amphibolite, both with or without appreciable magnetite content.

Narrow layers of garnetiferous-amphibolite are also interlayered with the iron-bearing members. Pyrite and pyrrhotite occur as secondary iron minerals usually in large disseminated grains surrounding or within the magnetite aggregates. In general these sulphides are more abundant within the so-called "internal waste" bands than within the principal iron-formation units.

Systematic mapping, ground magnetic traverses and diamond drilling carried out as a forerunner to the present evaluation were largely confined to an outcrop zone on the main fold structure. An area of approximately 3000 feet by 100-500 feet has been drill tested to an average depth of 200 feet. Total drilling completed, amounts to 6966.5 in 33 holes (See Table 1).

A pertinent geological plan at a scale of 100 feet to the inch and 15 drill sections at 50 feet to the inch append this report.

8.3 Reserves Potential - It is estimated that a "drill-proven reserve" of approximately 12,000,000 short tons of open pit minable material, grading 16.9% Soluble Iron, exists in the area of detailed mapping and diamond drilling. (See also "Tonnage and Grade" - Mining Section following).

Additionally, it is estimated that "geologically inferred reserves" of approximately 4,000,000 of similar grade material lie within zones of possible open pit extension. Approximately 5000 feet of additional diamond drilling and accompanying geological and geophysical mapping will be required to upgrade this potential to the category of "drill-proven reserves".

TABLE I

EXPO IRON LIMITED
DIAMOND DRILLING RECORD

Hole No.	Section	Location re Baseline	Angle	Footage From		Footage To	Footage Total
69 - 1	2 + 50 W	0 + 50 N	45° 30' S	0	-	251	251
69 - 2	4 + 50 W	1 + 00 N	43° 00' S	0	-	300	300
69 - 3	6 + 50 W	1 + 25 N	44° 00' S	0	-	273	273
69 - 4	8 + 50 W	2 + 00 N	44° 00' S	0	-	340	340
69 - 5 - 1	10 + 50 W	2 + 50 S	43° 00' N	0	-	330	330
69 - 5	10 + 50 W	2 + 25 N	43° 00' S	0	-	259	259
69 - 6	12 + 50 W	2 + 00 N	43° 00' S	0	-	250	250
69 - 7	14 + 50 W	1 + 50 N	44° 00' S	0	-	270	270
69 - 8	16 + 50 W	2 + 75 N	44° 00' S	0	-	318	318
69 - 9	18 + 50 W	2 + 00 N	44° 00' S	0	-	279	279
69 - 10	20 + 50 W	1 + 75 N	45° 00' S	0	-	258	258
69 - 11 - 1	22 + 50 W	2 + 65 N	45° 00' S	0	-	260	260
69 - 12	26 + 50 W	2 + 00 N	45° 00' S	0	-	279	279
69 - 13	30 + 50 W	1 + 25 N	46° 00' S	0	-	265	265
69 - 14	24 + 50 W	1 + 75 N	45° 00' S	0	-	275	275
70 - 1	30 + 50 W	2 + 25 N	46° 00' S	0	-	181	181
70 - 2	30 + 50 W	0 + 25 N	45° 00' S	0	-	126.5	126.5
70 - 3	28 + 50 W	1 + 75 N	45° 00' S	0	-	250	250
70 - 4	28 + 50 W	0 + 25 N	44° 00' S	0	-	121	121
70 - 5	26 + 50 W	0 + 50 N	42° 00' S	0	-	146	146
70 - 6	24 + 50 W	0 + 45 N	43° 00' S	0	-	139.5	139.5
70 - 7	22 + 50 W	1 + 00 N	44° 00' S	0	-	158	158
70 - 8	20 + 50 W	0 + 25 N	44° 00' S	0	-	119	119
70 - 9	18 + 50 W	0 + 50 N	46° 00' S	0	-	160	160
70 - 10	16 + 50 W	1 + 25 N	45° 00' S	0	-	180.5	180.5
70 - 11	16 + 50 W	0 + 25 S	45° 30' S	0	-	158	158
70 - 12	14 + 50 W	0 + 25 N	44° 00' S	0	-	150	150
70 - 13	12 + 50 W	1 + 00 N	45° 00' S	0	-	171	171
70 - 14	12 + 50 W	0 + 00	44° 00' S	0	-	150	150
70 - 15	10 + 50 W	1 + 00 N	44° 30' S	0	-	174	174
70 - 16	8 + 50 W	1 + 00 N	43° 00' S	0	-	175	175
70 - 17	6 + 50 W	0 + 25 N	44° 00' S	0	-	100	100
70 - 18	4 + 50 W	0 + 00	44° 00' S	0	-	100	100
TOTAL		33 holes					6966.5 ft.

9. MINING

9.1 SUMMARY

The reserves at the Expo deposit are estimated to be 11,905,200 tons with an average grade of 16.91% Soluble Iron. At an annual production rate of 1,157,730 tons the mine life will be 10.2 years.

The deposit can be mined by open cut methods from a pit which will ultimately be 3000 feet in length, 600 feet in width and 200 feet in depth. The following production layout is proposed:

Access from the open pit to the concentrator and rock dumps will be at the south west end of the pit. Extraction rates will be 4630 tons of crude and 1230 tons per day of waste rock. The mineralized material and waste will be drilled, blasted, removed by front-end loaders and hauled in 35-ton trucks. The mining operations will be conducted on a two-shift per day, 5-day per week basis. A total of 49 men will be required to mine the deposit.

Based on this production arrangement, the total direct cost of operating is estimated to be \$0.60/ton of all material or \$0.75 per ton of crude.

The initial capital cost for the mine equipment will be $\underline{\underline{\$1,080,000}}$

9.2 TONNAGE AND GRADE

Tonnage and grade calculations for the Expo Iron orebody provide the following drill proven reserves data:

Tonnage	-	11,905,200 Tons
Average Grade	-	16.91% Soluble Iron
Waste Rock	-	3,158,650 Tons
Stripping Ratio		0.265:1
Overburden	-	795,160 cu yds

Factors used in these calculations include:

Tonnage factors	Iron Formation	9.4 cu ft/ton
	Waste Rock	11.6 cu ft/ton

The tonnage and grade calculations are based on the inclusion of all internal waste bands less than 40 ft in width regardless of their content of iron-bearing minerals. If criteria for the inclusion of internal waste bands is reduced to a width of 20' then the reserves data are adjusted to the following:

Tonnage	-	10,889,200 Tons
Average Grade	-	17.64% Soluble Iron
Waste Rock	-	3,979,290 Tons
Stripping Ratio		0.365:1
Overburden	-	795,160 cu yds

EXPO IRON RESERVES - BY SECTION

	ORE				WASTE		OVERBURDEN	
Cross Section	Area Sq. In.	Interval Ft.	Short Tons 9.4 cu.ft./ST	Grade % Sol Fe	Area Sq. In.	Short Tons 11.6 cu.ft./ST	Area Sq. In.	Cu. Yds.
4 + 50	7.66	150	306,400	15.23	8.86	286,240	2.88	40,000
6 + 50	11.10	200	592,000	17.41	8.88	382,520	2.20	40,740
8 + 50	17.00	200	906,670	19.05	10.00	430,780	3.12	57,780
10 + 50	1.26							
	19.40							
	2.52							
	23.18	200	1,236,270	16.58	8.83	380,370	2.24	41,480
12 + 50	8.22							
	9.03							
	17.25	200	920,000	18.22	4.76	205,050	1.83	33,890
14 + 50	18.55	200	989,340	17.74	3.48	149,910	2.36	43.700
16 + 50	24.90	200	1,328,000	16.98	2.28	98,220	5.78	107,040
18 + 50	19.74	200	1,052,800	17.71	3.60	155,080	3.54	65,550
20 + 50	17.05	200	909,330	16.53	5.70	245,540	3.54	65,550
22 + 50	6.83							
	6.98							
	4.45							
	18.26	200	973,860	15.51	2.19	94,340	5.91	109,440
24 + 50	3.83							
	8.52							
	4.69							
	17.04	200	908,800	14.90	3.56	153,360	3.19	59,070
26 + 50	5.44							
	3.78							
	4.01							
	13.23	200	705,600	15.17	5.08	218,830	1.74	32,220
28 + 50	6.30							
	3.56							
	3.38							
	13.24	200	706,130	17.06	3.52	151,630	2.58	47,780
30 + 50	0.99							
	8.26							
	9.25	150	370,000	17.81	4.80	206,780	2.75	50,920
TOTAL			11,905,200	16.91		3,158,650		795,160

The practicability of selectively mining and extracting the internal waste bands from the principal mineralized zones can only be demonstrated when the mine is placed in operation.

No dilution factor has been applied to the reserves calculations because the separation of mineralized material from waste rock would be accomplished during operations, employing the mining method described in subsequent paragraphs.

9.3 METHOD OF RESERVES CALCULATION

The basic units employed in the reserves calculation are the areas enclosed between interpreted contacts of iron formation and waste rock. These lithologic units are illustrated on the attached cross sections. The grade of Soluble Iron applicable to each significant cross section has been determined by weight averaging the iron assays for corresponding drill hole intersections, including the assays of related internal waste bands.

The section interval employed in the reserve calculations is equal to the sum of half the distance to the two adjacent sections. At the extremities, the two final section intervals are reduced by 25 percent. The volume of material between each cross sectional area is calculated by multiplying the individual section areas by their corresponding section interval.

The tonnage of material between each section is determined by dividing the individual volumes by the applicable tonnage factor. The average grade for the deposit is determined by weight averaging the tonnage and grades calculated for each section.

The tonnage factors employed in the calculations are as follows and are based on densities of similar types of iron-bearing material:

Iron Formation	-	9.4 cu ft per ton
Waste Rock	-	11.6 cu ft per ton

9.4 OPEN PIT DESIGN CRITERIA

The following criteria have been employed in the design of the open pit for the Expo deposit.

9.4.1 Maximum Road Grade - The maximum mine haulage road grade is ten percent. Diesel-powered trucks are available to operate efficiently on ten percent grades.

9.4.2 Maximum Road Width - The maximum road width is thirty-five feet, including ditches. This width is satisfactory for the operation of 35-ton capacity haulage trucks, assuming proper maintenance of the full width of the road and traffic is one-way only. Turnouts on the main haul road will be required at 1000' intervals.

9.4.3 Minimum Turning Radius - The minimum turning radius of the haulage roads is 50 feet. This turning radius is satisfactory for the operation of haulage trucks with capacities of 35 tons.

9.4.4 Minimum Working Width - The minimum working width for the loading units and trucks is 50 feet. This working width is satisfatory for the operation of a 6 cu yd front-end loader and 35-ton truck.

9.4.5 Maximum Overall Pit Slope Angle - The overall pit slope angle on the walls of the proposed pit is 55 degrees. The expected competency of the wall rock and the life of the pit justifies selection of this overall slope angle.

9.4.6 Ultimate Pit Depth and Length - The open pit is designed to extract all of the material between sections 450 west and 3050 west. The pit is 3000 feet in length, 600 feet in width, and 200 feet in depth.

9.4.7 Waste Disposal - The waste rock from the pit is to be transported over the haulage road to dumps to the north of the pit. Separate dumps will be established for low-grade material and waste rock.

9.5 MINE DEVELOPMENT AND OPERATING SCHEDULES

A permanent access road from the pit will be constructed from Section 450 to the primary crusher 500 ft south of the mine. The road and the crusher ramp will be built with the preproduction overburden stripping. A branch road will also be constructed to the waste disposal area.

Bench development will commence after completion of the road and partial stripping of the deposit at or near sections 1050 west where the iron formation is exposed at surface. The excess overburden will be disposed of to the north of the mine and will form the foundation for the waste rock dumps.

The mining operations will be conducted on a two-shift per day, 5 days per week basis. The following table outlines the production schedule for the mine:

MINE PRODUCTION SCHEDULE
(SHORT TONS)

Product	Year	Months(12)	Weeks(50)	Days(250)	Shift(2)
Crude	1,157,730	96,480	23,150	4,630	2,315
Waste	306,800	25,520	6,140	1,230	615
TOTAL	1,464,530	122,000	29,290	5,860	2,930

The mine life will be 10.2 years based on a total reserve of 11,905,200 tons. Complete stripping of the overburden at the mine is not necessary before commencement of production, however, the decision as to whether to defer some overburden stripping must be made in the light of the present three-year tax holiday granted to new mines and the possible changes in the present tax structure.

9.6 MINING EQUIPMENT

The equipment to drill, blast, load and haul mineralized material and waste will be of conventional design. All equipment for the mining operation will be diesel-powered except for the mine de-watering pumps. No auxilliary lighting will be required for the mining operation except at the primary crusher. The size and quantity of equipment selected is based on the yearly production rate of 1,464,530 short tons of material but it is strongly influenced by the need to minimize the number of employees and thereby lower the operating cost. The following equipment will be required:

3 only 5½ cu yd front-end, rubber-tired Loaders

1 only 3 cu yd front-end, rubber-tired Loader

5 only 35-ton rear dump Haulage Trucks

3 only 4"-diameter Percussion Drill with crawler attachment

3 only 900 - CFM Mobile Compressor

2 only 180-HP Crawler Tractor with bulldozer attachment

1 only 115-HP Road Grader

2 only 4-ton flat-bed Service Truck

4 only 3/4-ton Pickup Trucks

2 only 200-GPM gasoline-powered Pumps

1 only 500-GPM electric-powered Pump

9.6.1 Loading Units - Front-end loaders rated for 5½ cu yd capacity are considered best suited to Expo operation. The estimated loading rate for a machine of this size is between 200 and 500 tons per hour, depending on the density of the material to be loaded and the degree of fragmentation resulting from the blasting. The planned production rate is 419 tons per hour. Two 5½ cu yd front-end loaders can handle this output. It is recommended that three identical front-end loaders be purchased to guarantee continuous mine production, care for the rock dumps and perform other tractor jobs in the mine site area. An additional 3 cu yd front-end loader will be required at the mill for loading concentrate into the highway haulage trucks.

9.6.2 Haulage Units - A total of five 35-ton diesel-powered rear-dump haulage trucks are recommended for use at the Expo operation. Four 35-ton trucks working an effective seven-hour shift will haul to the crusher 2650 tons of ore and will also haul 700 tons of waste rock to the dumps. A fifth unit will be required to allow for scheduled maintenance and repair of the truck fleet and to supplement the four regular haulage units during peak production periods.

A round trip cycle for a truck from the loader to the primary crusher (or rock dump) will be seventeen minutes. Based on this cycle and a seven-hour effective shift the production for each truck will be 900 tons per shift.

9.6.3 Drilling and Blasting - Three conventional percussion drills on a crawler-type track mounting are recommended for the drilling operation. Drilling pattern will be four-inch diameter holes on a 10 ft by 10 ft spacing. Bench height will be 25 feet. This type of percussion drill will penetrate at a rate of 175 ft per shift. No drill penetration tests have been conducted on the Expo material, however, a penetration rate of 175 feet per shift can safely be predicted for drilling the type of material found on the Expo property.

A 900-CFM Compressor will be required to operate each production drill. A jackleg drill will be required for secondary blasting.

A high consumption of explosives is anticipated to assure good fragmentation. Well broken material will reduce loading time and maintenance costs on the front-end loaders and also assure maximum production from the primary crusher. The use of 0.90 pounds of slurry-type explosive for each ton of material blasted will assure suitable fragmentation. High explosives will be used for secondary blasting.

9.6.4 Tractors and Graders - Two 180-HP, crawler-type tractors with bulldozer attachment, will be needed to supplement the loaders and to perform general earth moving, stripping and road work that the loaders are not capable of doing. During the preproduction stage, two additional machines will be required - these can be rented on a temporary basis.

A 115-HP road grader will be needed for maintenance of the ore haulage roads, plant-site area, and the access road to the minesite. Snow removal during the winter months will be an important function of this machine.

9.6.5 Service Equipment - Two 4-ton flat-bed trucks will be needed at the property to haul supplies and service both the mill and mine. Four 3/4 ton pickup trucks will be required for the transportation of personnel and supplies in the mine area.

Two 200-GPM gasoline-powered pumps will be needed for dewatering isolated parts of the pit. As the pit deepens, it will be necessary to purchase a 500-GPM electric-powered pump for installation in pit-bottom sumps. The capital cost for this piece of equipment is not considered in the initial equipment cost calculations because the pump will be part of the capital expenditures at some later period in the mine life.

A crane will be required periodically at the mine. The purchase of a crane is not justified because most of the lifting work will be handled by cranes installed in the maintenance shop. For general purpose lifting a hydraulic boom will be purchased for installation on the 4-ton service truck. High lifts will require the rental of a machine from Maniwaki or Val d'Or.

A maintenance building will be needed for the mine equipment, and for this a three-bay garage with attached welding and component repair shop will suffice. To insure that equipment will start on cold winter days a heated storage shed will be attached to the garage to house the mine equipment. Because of the idle time involved in a ten-shift per week mining operation, proper heated storage facilities will be needed for the diesel-powered equipment. This will require a total floor area of 5,000 sq ft.

10. IRON BENEFICIATION TESTWORK

10.1 SUMMARY

A total of six magnetic concentration tests have been conducted on composite drill core samples from the Expo property. Additionally, an autogenous grinding test was carried out on a 4000 lb bulk sample taken from outcrop. The results of these tests indicate that iron concentrate can be made from the Expo material suitable for the production of green iron pellets. The sulphur content of the concentrates is high but can be reduced by flotation or oxidation roast. Test on sulphur reduction remains to be carried out.

The test work was conducted at Lakefield Research of Canada Ltd., Aerofall Mills Ltd. and the Ontario Research Foundation under the direction and supervision of Mr. H. E. Neal.

10.2 MINERALOGY AND ORE ANALYSIS

The mineralized material to be mined and concentrated at the Expo property is comprised mainly of magnetite in an iron-magnesium silicate gneiss. There is a relatively large amount of pyrrhotite and pyrite as secondary minerals. The average grade of the deposit as determined in the reserve calculation is 16.91% Soluble Iron. From the results of ore beneficiation test work conducted at Lakefield Research of Canada Ltd., 11% to 11.5% of the Soluble Iron occurs as magnetite and pyrrhotite, the balance is likely in the form of non-magnetic pyrite.

10.3 CONCENTRATION TESTS

Initially, a composite sample of drill core was made to test the amendability of the Expo material to magnetic separation. This composite was screened into three different sizes; + 1/4 inch, - 1/4 inch + 10 mesh, and - 10 mesh. Each size fraction was treated on a dry magnetic drum separator. The results of this work indicated that 65.3% of the material could be discarded at the - 10 mesh size and that 99.1% of the magnetic iron could be retained. Details of the test are as follows:

Size Fraction	Wt.%	Assay % Sol Fe	Assay % Mag Fe	Distribution Sol Fe	Distribution Mag Fe
+ 1/4 inch	49.5	22.82	19.1	71.7	85.4
- 1/4 inch + 10 mesh	49.8	24.32	20.0	76.7	90.0
- 10 mesh	34.7	42.60	41.0	78.9	99.1

Each dry concentrate produced was tested with a Davis Tube. The results of the test indicated that the highest grade could be made from the - 10 mesh material. Test results were as follows:

Product	Wt.%	Assay % Sol Fe	Distribution Sol Fe
Concentrate	59.4	68.94	96.1
Tailings (Calc.)	40.6	4.06	3.9
Head	100.0	42.60	100.0

Assays of the - 10 mesh Davis Tube concentrate revealed the following chemical analysis:

SiO_2	-	1.87%
Mn	-	0.62%
TiO_2	-	0.18%
P	-	0.007%
S	-	3.74%

The sulphur content is high compared to Lake Erie concentrate standards.

A semi-quantitative spectrographic analysis of this concentrate gave no indication of detrimental quantities of tramp metals and examples are as follows:

Barium	-	0.005%
Chromium	-	0.05%
Cobalt	-	0.02%
Copper	-	0.007%
Molybdenum	-	0.01%
Nickel	-	0.01%
Vanadium	-	0.01%

The success of the initial concentration tests encouraged expansion of the test programme at Lakefield Research and Aerofall Mills.

A new composite head sample was made up from the balance of the drill core used in the initial tests. From this composite two samples were created, one of - 10 mesh material and the other - 20 mesh material. Each sample was sieved with a 150 mesh screen. The oversize was treated on a dry magnetic separator and the undersize on a wet magnetic separator. The results detailed on the following tabulation indicated that better concentrate grades (46.4% Sol Fe) and magnetic iron recovery (98.8%) could be obtained with - 20 mesh material.

TABLE II

METALLURGICAL RESULTS

Test No. 1A Minus 10 Mesh Ore

Product	Wt.%	Assay % Sol Fe	Assay % Mag Fe	% Distribution Sol Fe	% Distribution Mag Fe
+ 150 mesh Sala magnetics	24.5	38.9	-	61.1	-
+ 150 mesh Sala non-magnetics	56.9	5.2	0.4	18.8	-
- 150 mesh Jeffrey magnetics	3.9	64.5	-	16.4	-
- 150 mesh Jeffrey non-magnetics	14.7	3.9	0.1	3.7	-
Head (Calc.)	100.0	15.6	11.6	100.0	100.0
Combined magnetics	28.4	42.5	39.8	77.5	97.9
Combined non-magnetics	71.6	4.9	0.3	22.5	2.1

Test No. 1B Minus 20 Mesh Ore

Product	Wt.%	Assay % Sol Fe	Assay % Mag Fe	% Distribution Sol Fe	% Distribution Mag Fe
+ 150 mesh Sala magnetics	20.7	45.0	-	58.4	-
+ 150 mesh Sala non-magnetics	54.8	5.0	0.2	17.2	-
- 150 mesh Jeffrey magnetics	4.9	63.6	-	19.7	-
- 150 mesh Jeffrey non-magnetics	19.6	3.9	0.1	4.7	-
Head (Calc.)	100.0	15.9	11.9	100.0	100.0
Combined magnetics	25.6	48.6	46.4	78.1	98.8
Combined non-magnetics	74.4	4.7	0.2	21.9	1.2

The combined + 150 and - 150 mesh wet and dry magnetic separator concentrates from the - 10 and - 20 mesh material were each separately ground to three different levels of fineness. The products of these grinds were then Davis Tube concentrated. The results which follow demonstrate that an 81.9% - 325 mesh grind of the - 20 mesh sample gave concentrates with the highest Soluble Iron assay, i.e. 69.42% Sol Fe.

- 10 mesh Combined Magnetic Products

Davis Tube Concentrate - Grind 77% - 325 mesh

Assay		% Recovery Sol Fe	
% Wt.	% Sol Fe	Individual	Overall
57.6	69.00	93.7	72.6

- 20 mesh Combined Magnetic Products

Davis Tube Concentrate - Grind 82% - 325 mesh

Assay		% Recovery Sol Fe	
% Wt.	% Sol Fe	Individual	Overall
66.8	69.42	95.5	74.5

To test the commercial feasibility of making a high grade iron concentrate employing industrial type equipment, a composite sample of the + 150 and - 150 mesh wet and dry magnetic separator concentrates was ground to approximately 79% - 325 mesh and passed over a wet magnetic separator three times. The results obtained indicate 17.1% weight recovery, in concentrates grading 68.7% Sol Fe and representing 73.8% overall Soluble Iron recovery. The overall recovery of magnetic iron was 92.8%. Detailed results are as follows:

Product	Wt. % Ind.	Wt. % O'all	Assay % Sol Fe	Assay % Mag Fe	Assay % TiO_2	Assay % S	% Distribution Sol Fe Ind.	% Distribution Sol Fe O'all
Concentrate	63.2	17.1	68.7	-	0.154	4.34	94.8	73.8
Comb.Tailing	36.8	9.9	6.5	1.8			5.2	4.0
HEAD	100.0	27.0	45.8				100.0	77.8

The results of the latter test have been utilized in part for the mill flow sheet design.

Preliminary flotation tests to reduce the sulphur content of the concentrates have been initiated at Lakefield. Some success has been achieved but more investigation and experimentation is required.

A 4000 lb bulk sample was delivered to Aerofall Mills Ltd. for grinding tests in an 18" by 10" mill. A 1000 lb sample of the bulk material products were collected and air classified into three different fractions, coarse, fine and dust. The coarse and fine products together with their middling fractions were then treated over a dry magnetic separator and combined to give the following results:

% Wt. Recovery	% Sol Fe	% Distribution
30.84	55.7	85.9

The results of this test verified the bench scale concentration tests carried out at Lakefield and indicated that the Aerofall Mill could do an efficient job of grinding the Expo material. Power consumption of the autogenous mill was measured at 2.0 KWH/ton while regrind power was estimated to be 3.5 KWH/ton.

Unfortunately, the bulk sample collected from the property contained an abnormally low percentage of sulphur, suggesting that it was not truly representative of unweathered mineralized materials. For this reason further test work on the bulk sample was stopped.

A belt cobbing test indicated that it would not be feasible to use such a cobbing step as a preconcentrating process on the Expo material.

11. CONCENTRATOR DESIGN

11.1 SUMMARY

Autogenous grinding followed by dry and wet magnetic separation has been selected as the most practical and economical means of concentrating the Expo Iron "ore".

The concentrating plant equipment will include a 36" x 48" Jaw Crusher, 17-foot dry autogenous Grinding Mill, 10' x 17½' Ball Mill, and commercial dry and wet magnetic Separators. Mill throughput will be 3500 tons per day, on a 3-shift/7-day per week/330 days per year basis.

Yearly crude ore milled will amount to 1,157,730 tons and 208,600 tons of dry concentrate will be produced. Daily output will be six hundred and thirty tons of dry iron concentrate grading 69% Soluble Iron. A total of 40 men will be employed in the mill. The total direct cost of operating will be $0.85 ton milled or $4.70/ton of concentrate produced. The initial capital cost for the concentrator excluding the tailing disposal system will be $2,738,700.

11.2 FLOW SHEET CRITERIA

The mill flow sheet is based on a study by H. E. Neal of results of the test work conducted by Lakefield Research (see Appendix). The head grade of the Expo reserves is approximately 1% lower than average grade of the sample material used in the testwork. It has been necessary to factor the test results and thereby equate them to the reserves. The criteria employed to determine the material and water balances are as follows:

Magnetic Concentrate Grade	-	69% Sol Fe
Soluble Iron Recovery	-	73.6%
Crude Grade (Reserves)	-	16.9% Sol Fe
Calculated Weight Recovery	-	18.0%
Concentration Ratio	-	5.56:1
Daily Concentrate Production		630 Dry Tons
		700 Wet Tons

11.3 FLOW SHEET DESIGN

Production from the open pit will be discharged on a pan feeder which will transfer the mine muck to a stationary grizzly with 7" square openings. The oversize from the grizzly will be crushed in a 36" x 48" jaw crusher and the combined product conveyed to a 7000 ton stockpile. The material from the crusher stockpile will be fed to a

17-ft dry autogenous grinding mill with an oil fired dryer to remove the natural moisture. Mill products will include; vertical classifier, cyclone and filter. Of these the coarse vertical classifier and cyclone products will be treated separately in two stages on dry magnetic separators and the concentrate therefrom will then be reground in a conventional 10' diameter by 17½' long wet grinding ball mill. The ball mill product will be passed through two triple drum finisher wet magnetic separators and the final concentrate will be pumped to a thickener. The thickener underflow will be filtered by a 6'9" disc filter producing a finished filter cake containing 9.5 to 10% moisture. The filter cake will be stored in a shed in preparation for shipment and the tailings from both the dry and wet separators will be pumped to the tailing ponds.

Pertinent production rates are as follows:

	STPD	% Wt.	% Sol Fe	% Distribution Sol Fe
Crude	3500	100	16.9	100.0
Dry Drum Separator Conc.	825	24	55.1	77.0
Wet Magnetic Concentrate	700	20	62.1	73.6
Dry Magnetic Concentrate	630	18	69.0	73.6
Non-Magnetic Tails	2800	82	5.5	27.4

Water requirements for the mill operation will be 1470 GPM of which 1170 GPM will be reclaimed from the tailing ponds after decantation and 300 GPM from the fresh water pumped from Kearney Lake to the main mine fresh water storage tank.

12. P R E P R O D U C T I O N

Exclusive of the concentrator construction, the following preproduction work will be required. It is estimated that total preproduction cost will amount to $1,324,500 including the tailing disposal system.

12.1 ROAD CONSTRUCTION

Four miles of access road from the C.I.P. (Antostogan) haulage road to the southeast of the property will have to be reconstructed to a standard suitable for heavy trucks. Ditching and culverts will be the main requirement, plus additional top dressing.

In the mine area roads will be needed leading from the mine to the concentrator, from the mine to the explosive magazine and from the concentrator to the townsite. A total of 5 miles of road will require construction, employing the overburden and rock stripping from the mine for fill. No rock excavation is contemplated.

12.2 PLANT AND TOWNSITE GRADING

Two areas will require levelling and grading, one for the plant area and the other for the townsite area. The total graded area for both facilities will amount to approximately 3 acres. The overburden and rock stripping from the mine will be used for this work. No blasting and rock excavation is contemplated.

12.3 OVERBURDEN STRIPPING

A total of 795,160 cu yds of overburden stripping will be required to completely prepare the mine for operation. It may be possible to defer some of this work and reduce the initial capital expenditure at the expense of reduced earnings in the future. A minimum stripping programme is necessary however to expose enough material for uninterrupted mining during the first three years of tax free operation.

It is estimated that a minimum requirement for the overburden stripping programme will be 200,000 cu yds but for the purposes of this study the full stripping programme is to be completed before the commencement of production.

The stripping of the overburden at the mine will be performed by the equipment to be employed in the mine operation at a later date. Excavation of the 795,160 cu yds of material will require a period of one year during which time the future mining crews may be fully trained in preparation for the mine production.

12.4 MINE DEVELOPMENT

The minimum amount of preproduction rock excavation and stockpiling will be undertaken before commencement of production. The first material to be delivered to the crusher lies directly at surface and only small shallow benches will be developed in preparation for mining with the loaders and trucks. A total excavation of 30,000 tons of material should adequately prepare the mine and allow for tune-up of the concentrating equipment.

12.5 EXPLOSIVE MAGAZINE

A 20' x 20' explosive magazine will be required for storage of both slurry and high explosives. This building will be located at least one mile from the mine.

12.6 ANCILLARY BUILDINGS

Attached as an integral part of the concentrator building will be a service complex comprising a three-bay garage, an engine repair shop, a welding and bit grinding shop, a machine shop, an electrical repair shop, a warehouse, an equipment storage shed, and a boiler room. The garage, shop, boiler room, and warehouse will all require adequate floor area and equipment to maintain the mine and mill equipment with a minimum amount of labour.

12.7 WATER SUPPLY AND SEWAGE DISPOSAL SYSTEMS

The make-up fresh water requirements for the concentrator are estimated to be 300 GPM. The water requirements for a population of 115 people will be comparatively small. Allowing for concentrator changes or expansions and additional residents in the townsite, the fresh water installation will be designed to pump 500 GPM. The water will be pumped from Kearney Lake to a head tank located in the concentrator and will be distributed from there to the townsite and the process.

The sewage disposal system for the townsite will be through a central septic tank. An adequate tile bed will be laid to satisfactorily dissipate the effluent.

12.8 COMMUNICATION

With the assistance of the Bell Canada system a radio-telephone will be installed during the early stage of construction. Ultimately it will undoubtedly be possible to hook, into the Bell microwave system. There is only a minor charge for this installation followed by a charge for usage of the system.

12.9 OIL STORAGE

The oil requirements for diesel equipment, power plant, and heating in the plant and townsite are estimated to be 720,000 GPY. Gasoline requirements will be 4000 GPY. The supplying oil company normally takes the responsibility of installing the storage tanks for the two products with assistance in excavation and erection from the consuming company. A two-week supply of oil will require a storage capacity of 30,000 gallons.

12.10 TOWNSITE

A total of 115 men will be employed at the mine and it is planned to house all of these men in bunk house facilities except for 8 senior employees who will be supplied with married quarters. The bunk house facilities will be composed of mobile trailer-type buildings supplemented with adequate washing and toilet facilities and a complete fresh water supply and a sewage disposal system. The kitchen facilities will be a mobile type structure to be used for both the construction crews and the permanent mine employees.

The senior staff will be housed in mobile homes supplied by the Company and fully serviced by electrical power, fresh water and an adequate sewage disposal system.

The fresh water and sewage system will be designed to serve additional mobile homes if certain employees wish to bring their own personal mobile home to the townsite.

12.11 ELECTRIC POWER GENERATION

The total connected electric load at the mine will be as follows:

Concentrator	-	2500 HP
Shops and Office	-	200 HP
Mine	-	100 HP
Townsite	-	100 HP
		2900 HP

Three 800-KW diesel powered electric generators are recommended to supply this power and will be housed in a separate building close to the concentrator. Proper water cooling facilities will be required for the diesel power units plus an exhaust muffling system.

Power distribution to the concentrator will be by underground cable and to the other facilities by high line. Power is to be generated at 4160 volts and stepped down by transformers to 550 and 110 volts for the ancillary requirements.

12.12 TAILING DISPOSAL

Tailings from the concentrating operation will be pumped at a rate of 1100 GPM through a double 12" diameter pipe system for a distance of 800 ft to the tailing disposal area located to the south west of the mill building.

The first tailing dams to enclose the pond will be constructed of overburden stripped from the deposit. Clearing of the area will be necessary before construction of the dams and before discharge of the tailings commence.

Tailings discharged into the tailing pond will be from spigots located around the perimeter of the ponds. Coarse tailings will be employed to raise the dams when required. The tailings will settle within the pond and the clear effluent will be decanted through a weir to a clear water pond from which water will be reclaimed for use in the concentrator. Only minor quantities of effluent from the mill will be discharged into the Coulonge River system.

13. P E R S O N N E L

13.1 SUMMARY

The Expo operation will require 115 men. Single men will be accomodated in bunk houses at the minesite. Eight single dwelling mobile trailers, to be used by senior staff and their families, will also be required.

The total estimated annual payroll including the fringe benefits will be $838,300 of which $102,500 will represent the salary overhead cost. These wage rates are based on the present Maniwaki wage scale for similar work.

13.2 PERSONNEL REQUIREMENTS AND PAYROLL ESTIMATES

All employees will work a forty-hour week and a 17% fringe factor will be added to the base wage of hourly and salaried employees, to cover both compulsory government contributions and benefits granted by the Company.

Following is a tabulation detailing the personnel requirements, appropriate wage scales and total estimated payroll costs:

Classification	Number		Wages $/Month	Yearly Payroll	
Salaried					
Manager	1		$ 1,500	$ 18,000	
Mine Superintendent	1		1,000	12,000	
Mill Superintendent	1		1,000	12,000	
Mechanical Superintendent	1		1,000	12,000	
Engineer	1		1,000	12,000	
Mill Maintenance Foreman	1		850	10,200	
Mine Shift Foreman	3		800	28,800	
Mill Shift Foreman	4		800	38,400	
Accountant	2		700	16,800	
Assayer	1		700	8,400	
Personnel Safety Supervisor	1		700	8,400	
Assay Technician	1		600	7,200	
Office Clerk	2		500	12,000	
Mill Clerk	1		500	6,000	
Stenographer	2	23	350	8,400	$210,600

Classification	Number		Wages $/Hour	Yearly Payroll	
Hourly Rated					
Mine - Operating					
Loader Operator	5		$ 2.85	$ 28,500	
Haulage Truck Driver	8		2.70	43,200	
Tractor-Grader Operator	3		2.65	11,900	
Drill Operator	5		2.65	26,500	
Drill Helper	4		2.55	20,400	
Blaster	1		2.60	5,200	
Blaster Helper	2		2.50	10,000	
Service Truck Driver	2		2.55	10,200	
General Labour	1		2.45	4,900	
Mine - Maintenance					
Mechanic	5		3.00	30,000	
Mechanic Helper	3		2.70	16,200	
Welder	2		3.00	12,000	
Tireman	1		2.75	5,500	
General Labour	3	45	2.45	14,700	$239,200

Classification	Number		Wages $/Hour	Yearly Payroll	
Mill - Operating					
Crusher Operator	2		$ 2.85	$11,400	
Crusher Helper	2		2.65	10,600	
Grinding Operator	5		2.85	28,500	
Separator Operator	4		2.85	22,800	
Mill Helper	4		2.65	21,200	
Regrind and Filter Op.	4		2.85	22,800	
Mill Labour	4		2.45	19,600	
Mill - Maintenance					
Mechanic	3		3.00	18,000	
Electrician	2		3.00	12,000	
Welder	1	31	3.00	6,000	$172,900
Central Maintenance					
Mechanic	2		3.00	12,000	
Electrician	1		3.00	6,000	
Welder	1		3.00	6,000	
Machinist	2		3.00	12,000	
Carpenter	2		3.00	12,000	
Painter	1		3.00	6,000	
Pipefitter	1		3.00	6,000	
General Labour	2		2.45	9,800	
Power Room Attendan	4	16	3.00	24,000	$ 93,800
SUB TOTAL		115			$716,500
+ 17% Fringe					$121,800
TOTAL					$838,300

14. OPERATING COST ESTIMATE

14.1 SUMMARY

It is estimated that the operating costs at the Expo Iron property will be \$10.293/ton of concentrates produced. The details of operating costs are as follows:

	\$/Year	\$/Ton Conc. (208,600)	\$/Ton Milled (1,157,730)	All Material
Mine	870,400	4.173	0.752	0.594
Concentrator	980,500	4.700	0.847	
Administration	296,200	1.420	0.256	
	\$2,147,100	\$10.293	\$1.855	

14.2 MINE OPERATING COST ESTIMATE

The mine operating costs will be as follows:

	$/Year	$/Ton All Material (1,464,530) T.	$/Ton Concentrate (208,600 Tons)
Operating Labour			
Hourly	188,100	0.129	0.902
Salary	36,500	0.025	0.175
Maintenance Labour			
Hourly	98,800	0.067	0.474
Salary	11,200	0.008	0.054
Operating Supplies	410,800	0.280	1.969
Maintenance Supplies	125,000	0.085	0.599
	$870,400	$0.594	$4.173

14.2.1 Mine Labour Costs - The labour cost for the personnel required for the mining operation is developed employing a 40-hour week for all employees and includes a fringe benefit portion of 17% to cover the cost of unemployment insurance, workman's compensation, vacation pay, overtime and other benefits to be supplied to the employee by the Company.

a) Operating Labour

5	only	Loader Operator (Mine and Mill)	$ 28,500	
8	only	Truck Operator	43,200	
3	only	Tractor Grader Operator	11,900	
5	only	Drill Operator	26,500	
4	only	Drill Helper	20,400	
1	only	Blaster	5,200	
2	only	Blaster Helper	10,000	
2	only	Service Truck Driver	10,200	
1	only	General Labour	4,900	
			$160,800	
31		+ 17% Fringe	27,300	$188,100
1	only	Mine Superintendent	$ 12,000	
2	only	Shift Foreman	19,200	
			$ 31,200	
3		+ 17% Fringe	5,300	$ 36,500
Maintenance Labour				
Directly Assigned -				
5	only	Maintenance Mechanic	$ 30,000	
3	only	Mechanic Helper	16,200	
2	only	Welder	12,000	
1	only	Tireman	5,500	
3	only	General Labour	14,700	
From Central Maintenance Pool -				
½	only	Machinist	3,000	
¼	only	Electrician	1,500	
¼	only	Pipefitter	1,500	
			$ 84,400	
15		+ 17% Fringe	14,400	$ 98,800
1	only	Shift Foreman	$ 9,600	
		+ 17% Fringe	1,600	$ 11,200
GRAND TOTAL - LABOUR				$334,600

14.2.2 Operating Supplies

a) Fuel Oil

	Imp.Gal/Yr		
Loaders	56,000		
Trucks	112,000		
Tractor-Grader	28,000		
Compressors	65,000		
	261,000		
261,000 @ $0.22/Imp.Gal.[2]		$ 57,400	
Gasoline		2,000	
		$ 59,400	
+ 10% Contingency		6,000	$ 65,400

b) Explosives

1,500,000 long tons/year at 0.9 lbs.Exp/Ton = 1,350,000 lbs at $0.19/lbs.	$256,500	
Accessories - 10%	25,700	$282,200

c) Tires (Tire Life to Average 2500 Hours)

Yearly Operating Hours - Loaders	7,000		
Trucks	14,000		
Grader	2,000		
	23,000		

$\frac{23,000}{2,500}$ = 9 tires/year at $2,500 each $ 22,500

Misc. Small Tires	2,000	$ 24,500

d) Drill Bits and Steel

Drill Hole Per Year 220,000'

220,000 at $0.085/ft (bits and rods) $ 18,700

e) Miscellaneous

Cable, Bucket Teeth, Cutting Edges, etc. $ 20,000

GRAND TOTAL OPERATING SUPPLIES $410,800

14.2.3 Maintenance Supplies - The cost of maintenance supplies is estimated to be $125,000 per year. This sum covers items such as new spare parts for the mine equipment and also new additions and renovations to the existing equipment.

14.3 CONCENTRATOR OPERATING COST ESTIMATE (See H.E.Neal Report)

The mill operating costs will be as follows:

	$/Year	$/Ton Milled (1,157,730)	$/Ton Concentrate (208,600)
Operating Labour			
Hourly	160,200	0.138	0.768
Salary	84,200	0.073	0.404
Maintenance Labour			
Hourly (including 3 from Central Maintenance)	63,200	0.055	0.303
Salary	11,900	0.010	0.057
Operating Supplies	501,200	0.433	2.403
Electric Power	159,800	0.138	0.765
	$980,500	$0.847	$4.700

14.4 OVERHEAD

The following personnel will be required for administration of the Mine, Mill and the Townsite:

			$/Year	
1	only	Manager	$18,000	
1	only	Master Mechanic	12,000	
1	only	Engineer	12,000	
1	only	Safety and Personnel	8,400	
2	only	Accountant	16,800	
2	only	Clerk	12,000	
2	only	Stenographer	8,400	
			$87,600	
		+ 17% Fringe	14,900	$102,500

In addition, the following hourly rated personnel, unassigned to Mine or Mill will be required. The salaries of these men does not appear in the previous calculation of operating costs:

6	only	Tradesman at $3.00/hour	$36,000	
4	only	Power Room Attendants at $3/hour	24,000	
2	only	Labour at $2.45/hour	9,800	
			$69,800	
		+ 17% Fringe	11,900	$ 81,700

The cost of administrative service and supply will be as follows:

Roads and Yard Maintenance	$10,000	
Insurance	20,000	
Communications	14,000	
Fire Protection and Safety	7,000	
Heat and Power (exclusive of Mine and Mill)	5,000	
Legal and Engineering	20,000	
Head Office	30,000	
Miscellaneous	6,000	$112,000
TOTAL OVERHEAD		$296,200

$1.420/Ton Concentrate

15. CAPITAL COST ESTIMATE

15.1 SUMMARY

To place the property in production, it is estimated that the Expo Iron Project will require a capital expenditure of $6,257,200, excluding preproduction interest charges and all financing costs. All estimated costs are based on the purchase of new equipment. The utilization of second hand equipment may reduce the initial capital cost by as much as 25%. The used equipment market should therefore be investigated.

Preproduction		
Roads and Grading	$ 37,000	
Overburden Stripping and Mine Development	228,800	
Ancilliary Buildings & Facilities	249,800	
Townsite	276,900	
Power Generation	440,000	
Tailing Disposal	92,000	$1,324,500
Mine Equipment		$1,080,000
Mill (see H.E.Neal Report)		
Crushing Plant	$215,000	
Concentrator	2,304,000	
Engineering and Construction Management	219,700	$2,738,700
Sub Total		$5,143,200
Contingency 10%		514,000
Warehouse Inventory (10% of Equipment Value)		250,000
orking Capital Requirement (3 months)		350,000
TOTAL		$6,257,200

15.2 PREPRODUCTION COST ESTIMATE

The cost of the preproduction work is as follows:

15.2.1	Road Construction			
	4 miles at $3,000/mile		$ 12,000	
	5 miles at $2,000/mile		10,000	$ 22,000
15.2.2	Plant and Townsite Grading			
	3 acres at $5,000/acre			$ 15,000
15.2.3	Overburden Stripping			
	795,160 cu yds at $0.25/cu yd			$ 198,800
15.2.4	Mine Development			
	30,000 tons at $1.00/ton			$ 30,000
15.2.5	Explosive Magazine			$ 15,000
15.2.6	Ancilliary Buildings with Equipment			
	Shops	2500 sq ft @ $20	$ 50,000	
	Garage	3000 sq ft @ $13	39,000	
	Equipment Storage	5000 sq ft @ $ 9	45,000	
	Boiler Room	800 sq ft @ $16	12,800	
	Warehouse	4000 sq ft @ $11	44,000	
	Office Building		12,000	$ 202,800
15.2.7	Water Supply and Sewage Disposal System			
	Pumps and Water Lines		$ 12,000	
	Cuff Lake Intake		5,000	
	Sewage System		5,000	$ 22,000
15.2.8	Communication System			Ø
15.2.9	Oil Storage			$ 10,000
15.2.10	Townsite			
	Kitchen (125)		$ 34,400	
	Bunkhouse (108)		108,000	
	Mobile Homes (8)		104,000	
	Recreation Building		10,500	
	Water and Sewage System		20,000	$ 276,900
15.2.11	Electric Power Generation			
	Generator and Electrics- 3-800 KW Units @ $125,000 with Switch Gear		$400,000	
	Building		15,000	
	Distribution System		25,000	$ 440,000

15.2.12	<u>Tailing Disposal</u>		
	Tailing Pipe - 7500/ @ $ 8/ft	$ 60,000	
	Elbows, Couplings, Spigots	12,000	
	Decant Weir	10,000	
	Fresh Water Return System with Piping	10,000	$ 92,000
15.2.13	TOTAL PREPRODUCTION		$1,324,500

15.3 MINE EQUIPMENT COST ESTIMATE

The following is the cost of the mining equipment:

3	only	5½ cu yd front end Loader	$ 315,000	
		3 at $105,000		
1	only	1 cu yd front end Loader	60,000	
5	only	35-ton rear dump Haulage Truck	375,000	
		5 at $ 75,000		
3	only	4" diameter Percussion Drills with crawler attachment with 3 only 900 CFM Compressors		
		3 at $ 60,000	180,000	
2	only	180 HP Crawler Tractor with bulldozer attachment		
		2 at $ 45,000	90,000	
1	only	115 HP Road Grader		
		1 at $ 32,000	32,000	
2	only	4-ton flatbed Truck with hydraulic boom (Mine and Mill)	11,000	
4	only	3/4 ton Pickup Trucks (Mine, Mill, Administration)	14,000	
		4 at $ 3,500		
2	only	200 GPM gasoline powered Pumps	2,000	
		2 at $ 1,000		
1	only	jackleg Percussion Drill	1,000	$1,080,000
		1 at $ 1,000		

15.4 USED EQUIPMENT

Usually it is possible to purchase used mining equipment and thereby reduce the initial capital cost estimate for such equipment by 25 percent

The use of a Mining Contractor to perform the mining operation is not recommended - furthermore it would be difficult to attract a Mining Contractor to this type of operation.

The use of leased equipment is possible, although the cost of such rental is generally high.

Trucking Contractors are available in the Maniwaki area. It may therefore be possible to save the cost involved in purchasing new trucks by securing a local contractor for the ore haulage from the pit to the crusher.

15.5 CONCENTRATOR COST ESTIMATE

The following capital costs for the concentrator were developed by H. E. Neal whose full report is found as an appendix to the present study.

15.5.1	Crushing Plant		
	Equipment	$130,000	
	Installation	35,000	
	Building and Services	50,000	$ 215,000
15.5.2	Concentrator		
	Equipment and Installation	$1,450,000	
	Electrics	200,000	
	Building	529,000	$2,179,000
15.5.3	Water Storage and Fire Protection		$ 25,000
15.5.4	Freight		$ 100,000
15.5.5	TOTAL		$2,519,000

16. CONCLUSIONS

The present study demonstrates that the Expo Iron Deposit has sufficient drill-proven reserves of magnetically concentratable iron formation to sustain an open pit mining operation and concentrating plant for a minimum of 10 years, with an annual output of 208,600 tons of concentrates per year, grading approximately 69% Soluble Iron.

Detrimental metals such as titanium and phosphorus are below tolerable limits in the concentrates, but sulphur is generally high i.e. approximately 4% S. The sulphur content can be reduced by flotation or pre-metallizing roasting techniques. A complete investigation of the methods and costs of lowering the sulphur have yet to be completed.

No deliterious amounts of tramp metals are in evidence. Possibly some financial benefit can be gained from the manganese content of the concentrates which has been found to be approximately 0.60% Mn.

To place the property in production present cost studies indicate that the required Total Capital Investment, excluding financing charges and preproduction interest, will be $6,257,000; comprising, Preproduction Expenditures of $1,324,500, Mine Equipment Purchase Cost of $1,080,000, Concentrator cost of $2,738,700, Warehouse Inventory cost of $250,000, Working Capital of $350,000 and a Contingency of $514,000.

Based on the output of 208,600 tons of concentrates per year, Total Direct Operating Cost will be $10.29 per ton, excluding amortization, interest, depreciation and taxes.

Respectfully submitted,

E. D. Black, P.Eng.

W. J. Riddell, P.Eng.

Toronto, Ontario, Canada
August 1, 1970

CERTIFICATE

I, ERNEST DAVID BLACK, Consulting Geologist of 31 Windridge Drive, Markham, Ontario, certify that:

1. I am a graduate of McGill University in Montreal, Quebec, and hold a degree of Master of Science in Geology.

2. I am a fellow of the Geological Association of Canada and a Registered Professional Engineer of the Province of Ontario. I have practiced my profession for thirteen years.

3. I have personally examined the Expo Iron Property on several occasions and have based my conclusions on observations and studies of data derived from field and laboratory programmes which I have personally supervised.

4. I have not directly or indirectly received, nor do I expect to receive, any interest, direct or indirect in the property of Expo Iron Limited or any affiliated company or other company named in the foregoing report. I do not now beneficially own any interest or securities in any of the reference companies.

REGISTERED PROFESSIONAL ENGINEER
E. D. BLACK
PROVINCE OF ONTARIO

E. D. Black, P.Eng.

Toronto, Ontario, Canada
August 1, 1970

CERTIFICATE

I, WILLIAM JOHN RIDDELL, Consulting Mining Engineer, 64 Munro Blvd., Willowdale, Ontario, certify that:

1. I am a graduate of Queen's University in Kingston, Ontario, and hold a degree of Bachelor of Science in Mining Engineering.

2. I am a member of the Association of Professional Engineers of Ontario and have practiced my profession for twenty years.

3. I have personally visited the Expo Iron Property and studied in detail the mining aspects discussed in this report.

4. I have not, directly or indirectly, received, nor do I expect to receive any interest, direct or indirect, in the property of Expo Iron Limited or any affiliated company, and I do not beneficially own, directly or indirectly, any securities of Expo Iron Limited or any affiliated company.

Toronto, Ontario, Canada
August 1, 1970

W. J. Riddell, P.Eng.

REGISTERED PROFESSIONAL ENGINEER
W. J. Riddell
PROVINCE OF ONTARIO

A P P E N D I X I

APPENDIX II

METALLURGY

AND

PRELIMINARY DESIGN

EXPO IRON LIMITED

by

H. E. NEAL, P.ENG.

Toronto, Ontario, Canada
August 1, 1970

H. E. Neai & Associates Ltd.

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7. APPENDIX

1. Progress Report No. 1 - Lakefield Research of Canada Limited

2. Progress Report No. 2 - Lakefield Research of Canada Limited

3. Report by Aerofall Mills Ltd.

4. Estimate from Mr. J. Carr, Carr & Donald & Associates re Mill Building

5. Certificate of Chemical and Spectrographic Analyses - Lakefield Research of Canada

1. SUMMARY

1. Estimated Operating Cost 3500 S.T.P.D. — \$0.877 per short ton milled; \$4.876 per short ton concentrate

2. Estimated Capital Cost of Crushing Plant and Mill, Complete with New Equipment — \$2,738,700

3. Projected Grade of Magnetic Concentrate

69.0% Soluble Iron	0.007% P
0.154% TiO_2	0.62% Mn
4.34% Sulphur	1.8% SiO_2

4. Basis for Metallurgical Calculations Crude Mill Feed — 16.9% Soluble Iron; 12.9% Magnetic Iron; 1.3% Sulphur

Soluble Iron Recovery	73.6%
Concentrate Weight Recovery	18.0%
Concentration Ratio	5.56:1
Grind of Final Concentrate	75 to 80% - 325 Mesh

5. The proposed flowsheet consists of primary dry autogenous grinding of - 8" mill feed, air classification, and dry magnetic separation of the mill products. The dry preconcentrate will be reground to 75 to 80% - 325 Mesh, treated on 3-drum finisher wet magnetic separators and filtered to produce a filter cake with 9.5% to 10.0% moisture.

6. Covered storage is provided for 7 days' production of filter cake, with provision for loading the product into trucks by a front-end loader.

7. Testwork was conducted at Lakefield on drill core obtained in 1969 and formed the basis for the regrinding and final concentrate results.

8. Laboratory autogenous grinding and coarse cobbing tests were conducted at Aerofall Mills Ltd., using a surface bulk sample.

9. Tests indicated that the ore responds well to dry autogenous grinding followed by 2-stage dry magnetic separation of the mill products.

10. More complete chemical analyses of the final magnetic concentrate are required.

11. This report is based on only magnetic separation which produced a high-sulphur magnetite concentrate. The sulphur in the concentrate is almost entirely present as pyrrhotite which is largely liberated.

2. INTRODUCTION

This report consists of a review of metallurgical testwork on composite samples of 1969 drill core and a 4000-pound surface bulk sample collected in 1969. The metallurgical data for this report are based on tests using only magnetic separation whereby a concentrate containing 69% Soluble Iron and 4.0 to 4.3% Sulphur was produced at a suitable grind for pelletizing. The projected metallurgical balance was calculated from a combination of the results of testwork at Lakefield Research of Canada Limited and Aerofall Mills Limited.

Sulphide flotation testwork was recommended and has been conducted in order to reduce the Sulphur to below 0.7%. Lurgi of Canada stated that if the concentrate contained less than 0.7% Sulphur then a pre-roasting grate zone would not be required in their metallizing process.

For this study, the client requested that the mill flowsheet and cost estimates be based on producing a high-sulphur magnetic concentrate. The removal of the Sulphur by roasting prior to metallizing was to be investigated by the client.

The mill flowsheet incorporates conventional grinding and processing equipment currently being used successfully in low-grade magnetic taconite operations. The practice of early elimination of final tailings at a coarse size was followed. The preconcentrate from this initial

3. METALLURGICAL TESTWORK

3.1 GENERAL STATEMENT

The metallurgical results are based on testwork conducted on drill core from the 1969 drilling and a 4000-pound surface bulk sample collected in 1969. Instructions for compositing the drill core for testwork were given to Lakefield Research of Canada Limited by Mr. E. D. Black, whereby 238 samples of the total shipment of 308 drill core samples were used.

The initial series of tests at Lakefield Research used the coarse rejects from the assay portions of the drill core. These tests were to show the effect of coarse dry separation on a Sala-Mortsell Separator in the size range of - 1/2" + 1/4"; - 1/4" + 10 M and - 10 M. These results are reported in Progress Report No. 1 of March 3, 1970. (See Appendix 1).

The second phase of testwork at Lakefield Research was conducted on a 350-pound composite sample of - 10 mesh rejects from the same 238 drill core samples. These tests showed the amount of coarse silica rejection by dry magnetic separation at 10 mesh and - 20 mesh along with the relation of grind to final concentrate grade using the preconcentrate.

Some preliminary flotation tests were conducted to attempt to reduce the Sulphur in the final concentrate from 4.3% S to about 0.7% S whereby roasting prior to metallizing might be eliminated. These results are reported in Progress Rep^ t No. 2 of June 25, 1970 by Lakefield Research. (See Appendix 2).

Aerofall Mills Ltd. conducted testwork on a 4000-pound surface bulk sample. One quarter of the sample was used for grinding tests in an 18" x 10" model of an Aerofall Mill to illustrate the response of the sample to autogenous grinding and dry magnetic separation of the products. This flowsheet was based on the successful operation of two taconite plants in Minnesota where Aerofall provided the autogenous mills. This phase of the testwork served as the basis for selecting the mill and motor sizes for the plant design. Three quarter of the bulk sample was used for cobbing tests. The sample was crushed to various sizes to establish whether coarse magnetic cobbing could reject an appreciable amount of final tailings in a conventional two-stage crushing plant. Regrinding of the cobbed concentrates to final concentrate grade was not tried since assays of the Aerofall products showed that the Sulphur head analysis was 0.23% in the bulk sample as compared to 1.3% Sulphur in the drill core which was considered to be more representative.

The second phase of the Lakefield testwork and the tests at Aerofall Mills were outlined and supervised by H. E. Neal.

Chemical analyses on the Aerofall products were made at Lakefield on the fine products from the 18" mill and at Ontario Research Foundation on the coarse cobbing products.

3.2 SUMMARY OF METALLURGICAL TESTWORK

a) Drill core samples ground to 75% to 80% - 325 mesh produced magnetic concentrates in the range of 69% Sol Fe with 4.0 to 4.3% Sulphur.

b) Test No. 2 is used to illustrate the Sulphur and Titania content in the final magnetic concentrate grade of:

68.7%	Sol Fe
0.154%	TiO_2
4.34%	Sulphur

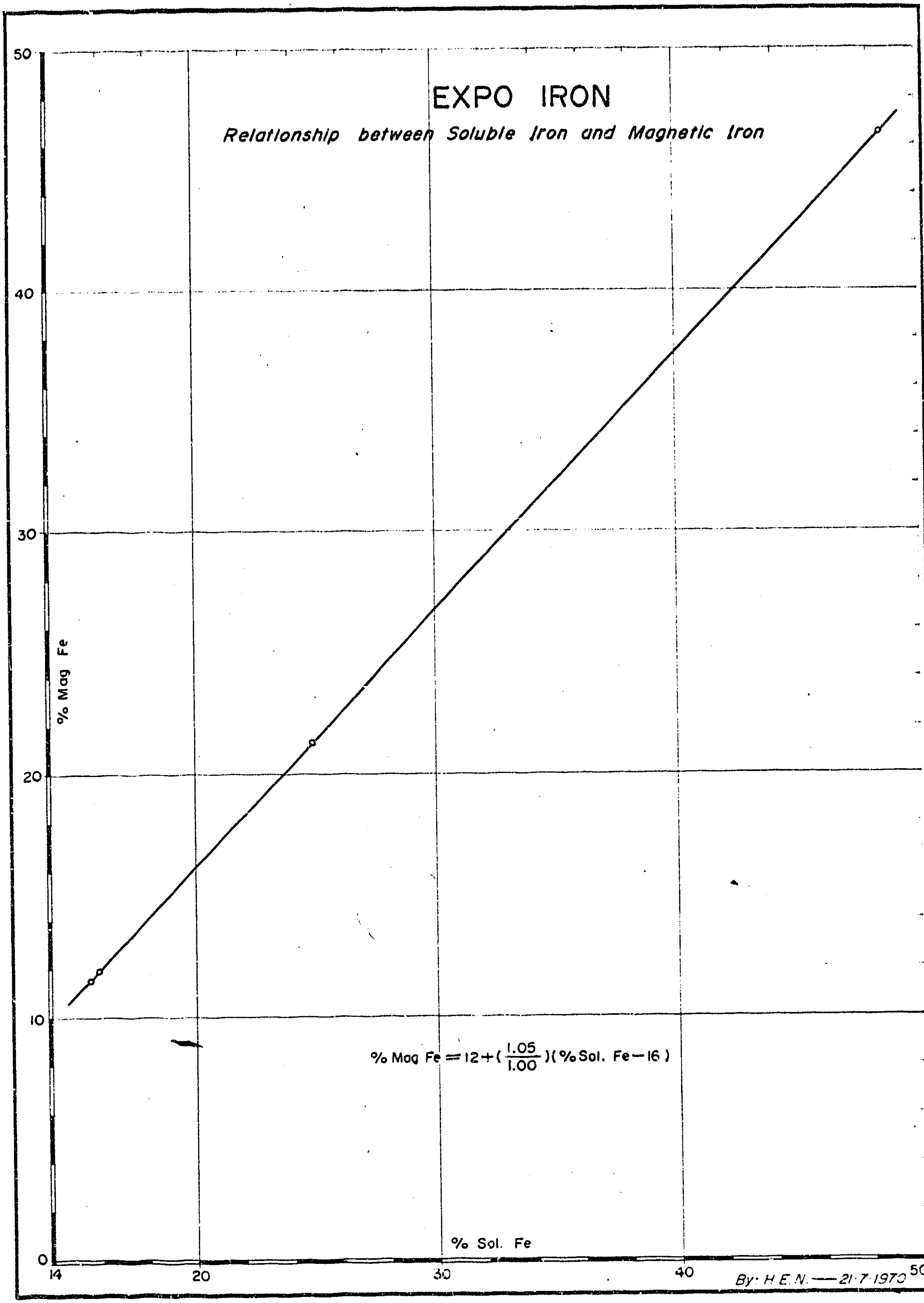
c) Dry grinding and classification on a bulk sample containing 19.97% Sol Fe followed by dry magnetic separation rejected 69% of the weight as tails to produce a preconcentrate containing 55.7% Sol Fe.

d) Coarse magnetic cobbing showed a much lower rejection of waste accompanied by high iron losses.

e) The drill core tested had a crude grade of:

15.6 to 15.9%	Sol Fe
11.6 to 11.9%	Magnetic Fe
1.3%	Sulphur

Sulphur is present in pyrrhotite and pyrite, with the pyrrhotite reporting in the magnetic concentrates. The Soluble Iron analysis reported by Lakefield Research includes the iron contained in the



EXPO IRON
Relationship between Soluble Iron and Magnetic Iron
% Mag Fe
% Sol. Fe
50
40
30
20
10
0
14
20
30
40
50
% Mag Fe = 12 + (1.05 / 1.00)(% Sol. Fe − 16)
By: H.E.N. — 21·7·1970

pyrrhotite, as well as the iron in the magnetite, and other oxides or carbonate, if present. The iron in the pyrite does not report in the Soluble Iron analysis.

The bulk sample had a crude grade of:

19.97%	Sol Fe
0.23%	Sulphur

3.3 LAKEFIELD TESTWORK

3.3.1 Initial Testwork - 1969 Drill Core

The composite drill core sample had the following Head Grade:

16.15%	Sol Fe
11.14%	Magnetic Fe (D.T. Tests)

The drill core was crushed to - 1/2"; screened at 1" and 10 M.

Each fraction was magnetically cobbed to determine the amount of waste rejection and iron losses by cobbing.

Overall Results

		Assay		Distribution	
	% Wt.	% Sol Fe	% Mag Fe	Sol Fe	Mag Fe
Combined Concentrates	47.5	24.87	21.3	73.3	88.1
Combined Tails	52.5	8.20	2.6	26.7	11.9
Head Grade (Calc)	100.0	16.15	11.4	100.0	100.0

Individual Concentrate Grades and Recoveries by Size Fractions

		Assay		Distribution	
Size Fraction	% Wt.	% Sol Fe	% Mag Fe	Sol Fe	Mag Fe
- 1/2" + 1/4"	49.5	22.82	19.1	71.7	85.4
- 1/4" + 10 Mesh	49.8	24.32	20.0	76.7	90.0
- 10 Mesh	34.7	42.60	41.0	78.9	99.1

Observation: Excessive Sol Fe and Magnetic Fe losses occur in the + 1/4" fraction with the concentrate grade only raised from 15.75% Sol Fe to 22.82% Sol Fe for this size. Finer grinding is required for more complete liberation before magnetic separation.

No Sulphur analyses were completed in this series of tests. (See Progress Report No. 1, March 3, 1970).

Appendix 5 contains more complete chemical and spectrographic analyses in Certificate of Analyses by Lakefield Research of Canada.

3.3.2 Second Phase Testwork - 1969 Drill Core

A second composite of the 1969 drill core was prepared from the - 10 Mesh rejects of the original assay samples.

Tests were conducted on material crushed to - 10 Mesh and - 20 Mesh.

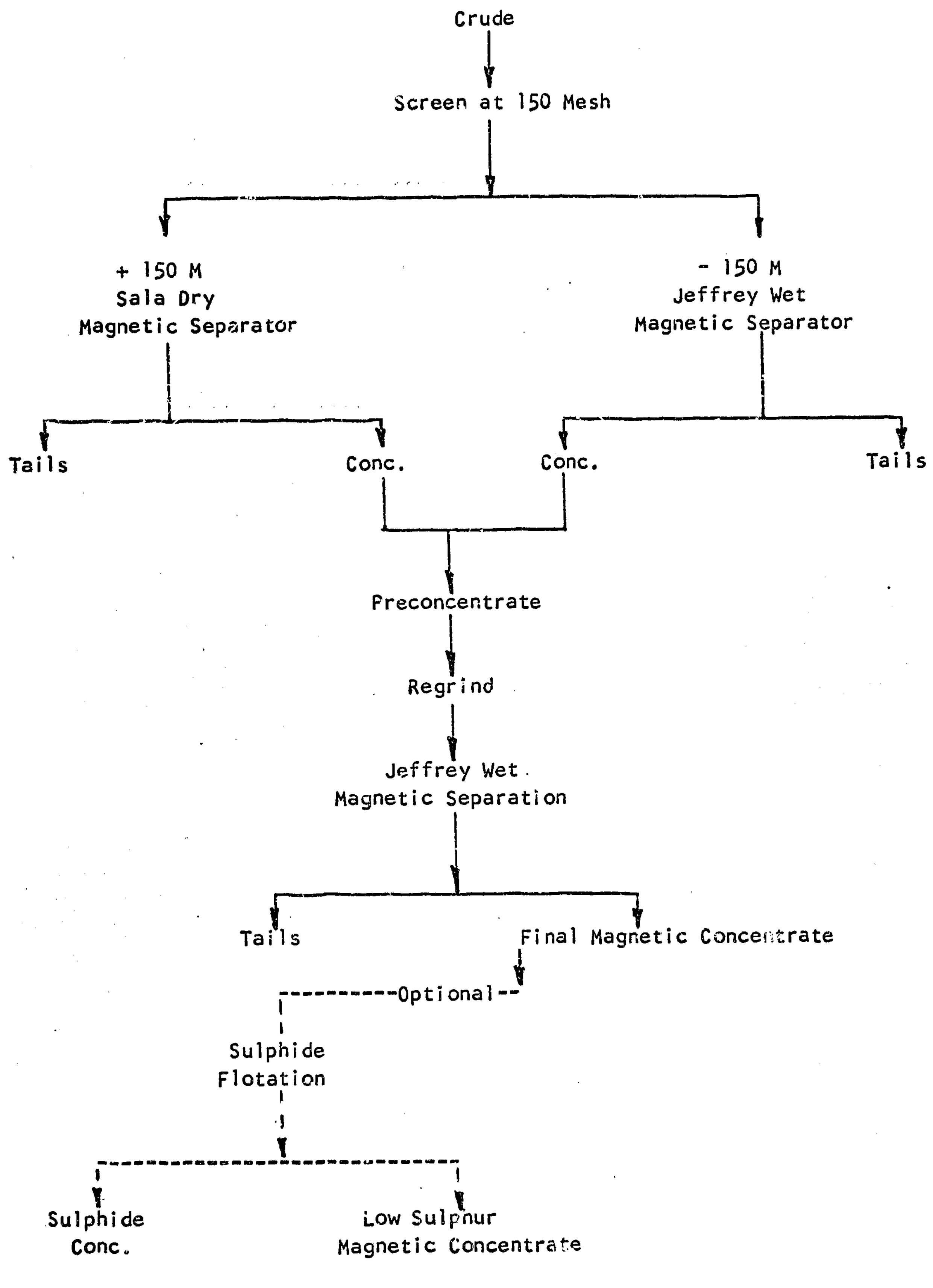
Crude Grade:

	% Sol Fe	% Magnetic Fe
- 10 Mesh sample	15.6	11.6
- 20 Mesh sample	15.9	11.9

Each sample was screened at 150 Mesh; the - 10 Mesh or - 20 Mesh + 150 Mesh fraction was tested on a Sala-Mortsell dry magnetic separator. The - 150 Mesh portion was treated by a wet magnetic separator. The combined dry and wet concentrates were reground to produce a final concentrate. A third series of tests were conducted using combined - 10 and - 20 mesh material.

FLOWSHEET FOR LAKEFIELD TESTWORK

\- 10 Mesh - 20 Mesh



Only preliminary flotation testwork was conducted and is considered outside the scope of this report.

Preconcentration

	Assay			Distribution	
	% Wt.	% Sol Fe	Mag Fe	Sol Fe	Mag Fe
- 10 Mesh					
Combined Magnetics	28.4	42.5	39.8	77.5	97.9
Combined Non-Magnetics	71.6	4.9	0.3	22.5	2.1
- 20 Mesh					
Combined Magnetics	25.6	48.6	46.4	78.1	98.8
Combined Non-Magnetics	74.4	4.7	0.2	21.9	1.2

		Assays			Distribution		
	% Wt.	% Sol Fe	% Mag Fe	% S	Sol Fe	Mag Fe	S
Magnetic Concentrate	17.1	69.1		4.05	74.0		53.0
Jeffrey Tails from Pre-concentrate	9.9	6.5	1.8	-	4.0	1.5	-
Jeffrey Tails from - 150 Mesh	17.1	3.9	0.1	-	4.2	0.1	-
Sala Tailing	55.9	5.1	0.3	-	17.8	1.4	-
Calculated Head	100.0	15.9	11.8	1.30	100.0	100.0	100.0

Effect of Grind on Grade and Recovery of Final Magnetic Concentrate

Sample	Grinding Time	Grind % -325M	% Wt.	Davis Tube Concentrate Assay % Sol Fe	Sol Fe Individual	Distribution Overall
- 10 M	20 min.	55.5	59.2	68.36	95.4	73.9
- 20 M	20 min.	58.7	68.4	68.43	96.3	75.2
- 10 M	30 min.	70.3	58.6	68.42	94.5	73.2
- 20 M	30 min.	73.6	67.6	68.91	95.9	74.9
- 10 M	40 min.	77.2	57.6	69.00	93.7	72.6
- 20 M	40 min.	81.9	66.8	69.42	95.5	74.5

Observation: A grind of 75 to 80% - 325 Mesh is required to produce a concentrate with + 69% Sol Fe. This grind is in the required range for pelletizing.

3.4 AEROFALL TESTWORK

3.4.1 18 Inch Mill Test on Bulk Sample

An 18-inch diameter by 10-inch long mill was used to grind 1000 pounds of bulk sample crushed to - $1\frac{1}{2}$ inches. The feed to this mill was automatically controlled, and the power drawn was continuously measured. Steel balls formed 6.5% of the mill volume. The ground product was removed by an air stream and sized by air classification into a coarse, fine and dust product.

The coarse and fine products were individually fed to two stages of dry magnetic separation on a 24-inch diameter Sala-Mortsell Separator. The speed of the Sala Separator was set at 60 rpm for the initial separation in order to produce a final low-grade tailing. This concentrate was repassed on the Separator at a speed of 180 rpm and produced a concentrate (for regrinding) and a middling. In practice this middling would be recirculated to the Aerofall Mill. The following table contains the results of the testwork:

18-Inch Aerofall Products Treated on Two-Stage Sala-Mortsell Magnetic Circuit

Product	% Wt.	% Sol Fe	% S	Distribution Sol Fe	Distribution S
Vertical Classifier Concentrate (Coarse Product)	10.52	66.00	0.03	34.8	1.5
Cyclone Concentrate (Fine Product)	15.10	66.45	0.09	48.7	5.9
Vertical Classifier Middlings	2.49	11.06	0.22	1.3	2.4
Cyclone Middlings	2.73	8.24	0.20	1.1	2.3
Calculated Combined Concentrate and Middlings	30.84	55.7	0.09	85.9	12.1
Vertical Classifier Tails	24.50	3.41	0.22	4.2	23.3
Cyclone Tails	44.06	4.16	0.34	9.3	64.5
Filter Product	0.60	20.62	0.58	0.6	0.1
Combined Tails	69.16	4.07	.29	14.1	87.9
TOTAL (Calculated Head)	100.00	19.97	0.23	100.0	100.0

Percent Weight Distribution

Product	Vertical Classifier	Cyclone	Filter	Total
Concentrate	10.52	15.10	-	25.62
Middling	2.49	2.73	-	5.22
Tailing	24.50	44.06	0.60	69.16
TOTAL	37.51	61.89	0.60	100.00

3.4.2 Cobbing Tests

About 3000 pounds of the bulk sample were crushed initially to minus 3 inches. Visual observation showed that the - 3" + 2" tailings from cobbing contained magnetite in the host rock with very little true non-magnetic tailings. Cobbed tailings in this size range were tested with a hand magnetic and indicated that too much magnetite was present for cobbing. The sample was then crushed to - 2" to establish the amount of non-magnetic tailings that might be produced by cobbing. The sample was screened at 1" and 6 Mesh. The - 2" + 1" and - 1" + 6 Mesh were treated on a 36 inch Sala dry belt cobber running at 40 rpm. The - 6 Mesh fraction was treated on two Sala-Mortsell magnetic separators; the first to produce a tails and the second time to produce a magnetic concentrate and a middling. The products were sent to the Ontario Research Foundation for chemical analyses.

Metallurgical Balance - Cobbing

Product	% Wt.	% Sol Fe	Distribution Sol Fe
- 2" + 1" Concentrate	20.97	23.7	24.0
- 1" + 6M Concentrate	26.48	22.8	29.2
- 6 M Concentrate	10.81	59.5	31.1
- 6 M Middlings	6.14	14.9	4.4
Combined Concentrate and Middlings	64.40	28.6	88.7
- 2" + 1" Tailing	5.51	10.2	2.7
- 1" + 6M Tailing	5.30	7.0	1.8
- 6" Tailing	24.79	5.7	6.8
Combined Tailing	35.60	6.6	11.3
TOTAL (Calculated Head)	100.00	20.72	100.0

	Percent Weight Distribution			
Product	Concentrate	Middling	Tailing	Total
- 2" + 1"	20.97	-	5.51	26.48
- 1" + 6 Mesh	26.48	-	5.30	31.78
- 6 Mesh	10.81	6.14	24.79	41.74
	52.26	6.14	35.60	100.00

Observation: The above results show that cobbing of - 2 inch material raised the grade of the crude material from 20.72% Sol Fe to 28.6% Sol Fe in its cobbed concentrate. The tailings rejection was 35.6% weight of the crude, which is considerably less than at a finer size.

4. MILL DESIGN

4.1 DESIGN BASIS

4.1.1 Crude Grade

The crude grade to the mill has been calculated by Mr. W. J. Riddell to be 16.9% Soluble Iron. Based on testwork of the drill core composite, this material is expected to contain 1.3% Sulphur, although no Sulphur analyses were made of individual drill core intersections. Pyrrhotite contained in the magnetite-bearing material is largely dissolved during the iron analysis and the iron in the pyrrhotite reports as Soluble Iron.

A 12.95% Magnetic Iron content is equivalent to a crude grade of 16.9% Soluble Iron. (See Graph showing relationship between Soluble Iron and Magnetic Iron following Page 6).

4.1.2 Soluble Iron Recovery

A Soluble Iron Recovery of 73.6% is used for this study. This is based on an average Soluble Iron recovery for tests 1 and 2 by Lakefield on the drill core composite.

Based on:

Magnetic Concentrate Grade	69% Sol Fe
Soluble Iron Recovery	73.6%
Crude Grade	16.9% Sol Fe

The Calculated Weight Recovery is 18.0%

4.1.3 Grinding Power Requirements

The power for primary autogenous grinding was based on testwork by Aerofall Mills in the 18" diameter laboratory mill. The power for regrinding is based on similar experience in the absence of a Bond Work Index.

Aerofall grinding power	2.0 KWH/T Crude
regrind power	3.5 KWH/T Crude
or	
	14.9 KWH/T mill feed (34 TPH)

4.2 OPERATING DATA

4.2.1 Projected Mill Metallurgical Balance

The following metallurgical balance combines the dry grinding, classification and dry magnetic separation testwork by Aerofall and the regrinding testwork by Lakefield Research. The distribution of coarse and fine products is based on the Aerofall results. The grade of the dry magnetic concentrate was assumed to be a combination of the concentrate and middlings, whereas in practice the middlings will be recirculated to the autogenous mill. It also assumes that the filter product would be sent directly to tailings due to its fineness.

Projected Metallurgical Balance

Product	% Wt.	% Sol Fe	% S	Distribution Sol Fe	S
Coarse Concentrate	11.8	55.0	-	38.5	
Dry Cyclone Concentrate	11.9	55.2	-	38.6	
Feed to Regrind	23.7	55.1		77.1	
Final Products					
Coarse Tails	29.2	6.7	-	11.4	
Dry Cyclone Tails	46.1	3.7	-	10.1	
Filter (to waste)	1.0	20.0	-	1.3	
Wet Regrind Tails	5.7	11.2	-	3.6	
TOTAL TAILS	82.0	5.5		27.4	
Final Concentrate	18.0	69.0	4.30	73.6	59.5
Crude Feed	100.0	16.9	1.30	100.0	100.0

Concentration Ratio: 5.56 crude: 1.0 concentrate

5.56 short tons of crude mill feed are required to produce 1 short ton of dry magnetic concentrate.

4.2.2 Water and Material Balance

Description	% Wt.	S.T.P.H. Solids	S.T.P.H. Water	S.T.P.H. Pulp	% Solids	Solids S.G.	Water USGPM	Pulp USGPM	Pulp S.G.
Crude Ore	100.0	145	-	-	100.0				
Dry Tails	76.3	110.5							
Dry Concentrate	23.7	34.5	-	34.5	100.0	4.6			
Cyclone U'Flow	59.5	86.5	37.1	123.6	70.0	4.6	148	223.4	2.55
Make-Up Water		-	14.8	14.8	-	-	59.2	59.2	1.0
Feed to Regrind	83.2	121.0	51.9	172.9	70.0	4.6	207.2	312.7	2.21
Ball Mill Discharge	83.2	121.0	51.9	172.9	70.0	4.6	207.2	312.7	2.21
Make-Up Water		-	123.1	123.1	-	-	494.5	494.5	1.0
Cyclone Feed	83.2	121.0	175.0	296.0	45.9	4.6	701.7	807.2	1.47
Cyclone U'Flow	59.5	86.5	37.1	123.6	70.0	4.6	148	223.4	2.55
Cyclone O'Flow	23.7	34.5	137.9	172.4	20.0	4.6	553.7	583.8	1.19
Wash Water		-	217.1	217.1	-	-	869.8	869.8	1.0
Mag. Separators <u>Final Concentrate</u>	18.0	26.1	17.4	43.5	60	5.0	69.7	90.6	1.92
Final Tails	5.7	8.4	337.6	346.0	2.5	2.8	1353.8	1363.0	1.02
Thickener Feed	18.0	26.1	17.4	43.5	60	5.0	69.7	90.6	1.92
O'Flow	-	-	6.2	6.2	-	-	24.8	24.8	1.0
U'Flow	18.0	26.1	11.2	37.3	70	5.0	44.9	65.8	2.31
<u>Dry Filter Cake</u>		26.1	2.9	29.0	90	5.0	11.6	-	3.57
Filtrate		-	8.3	8.3	-	-	33.3	33.3	1.0
<u>Wet Filter Cake</u>	20.0	29.0							

4.2.3 Mill Water Balance - In U.S.G.P.M.

Location	Water In	Water Out	Reclaim	Make-Up
Regrind Feed	59			
Cyclone Feed Sump	495			
Finisher Magnetic Separators	870			
Finisher Tails		1354		
Tailings Pond (80% of Water)			1081	
Concentrate Thickener O'Flow		25	25	
Filtrate		33	25	
Concentrate		12		
Gland Water	60		60	
Clean Up etc.	40	40	32	
TOTAL	1524	1464	1223	301

Reclaim from Tailing Pond	1223
Fresh Water Make-Up	300 U.S.G.P.M.

4.2.4 Production Rates

	Short Tons
Daily Mill Feed	3500 Tons
Weekly Tonnage	23150 Tons
Yearly Tonnage	1,157,730 Tons
Primary Crushing Rate (2 shifts/day;5 days/week)	290 Tons/Hour
Mill Feed Rate - Based on	330 days per year
Feed Rate per Shift	1165 Tons
Feed Rate per Hour	145 Tons

4.2.5 Mill Balance

	Short Tons Per Day	% Wt.	% Sol Fe	Distribution Sol Fe
Crude	3500	100.0	16.9	100.0
Magnetic Conc. Dry	630	18.0	69.0	73.6
Magnetic Conc. - Natural 10% Moisture in Filter Cake	693	19.8	62.1	73.6
Combined Non-Magnetic Tails (Dry Weight)	2870	82.0	5.5	26.4

4.3 PLANT AND FLOWSHEET DESCRIPTION

4.3.1 Description of Major Equipment

The attached drawings - E-1, E-2, and E-3 illustrate diagrammatically the proposed mill flowsheet, plan and longitudinal mill sections. The following major equipment was selected:

a) Pan Feeder - 42" wide by 10 ft long

b) Jaw Crusher - 36" x 48" with 150 HP motor

c) 17' Diameter Aerofall Mill - complete with preheating hood, air classification, dust collection system and exhaust fan - 600 HP mill, 400 HP fan.

d) 1 - Derrick Screen - 5' x 14' high speed screen

e) 8 - Dry Magnetic Separators - 36" diameter x 8 ft long Indianna General or Sala-Mortsell

f) 2 - 3 drum Finisher Wet Magnetic Separators - 36" diameter x 8 ft long

g) 1 - Regrind Ball Mill - 10' diameter x 17½ ft long (700 HP)

h) 1 - 20 ft diameter Thickener

i) 1 - 8 disc - 6'9" Eimco Filter complete with vacuum pump, filtrate pump and receiver

j) Dry Preconcentrate Storage Bin ahead of the Regrind Ball - (26' diameter x 20' high)

4.3.2 Primary Crusher

Crude ore from the pit will be dumped by 35-ton trucks into a timber crib hopper, lined with steel. The base of the hopper will contain a pan feed measuring 42" wide by 10 feet long and capable of receiving ore up to 3 feet in diameter. Normally ore will be left on the feeder to protect it from severe impact. A 36" x 48" Jaw Crusher will be fed directly by the pan feeder or a grizzly ahead of the crusher could remove the minus 7 inch material with only the + 7" being fed to the crusher. The crusher will operate at a closed size opening of 8 inches with a nominal capacity of 350 to 400 TPH. The crusher will discharge onto a 42" wide conveyor to the coarse ore stockpile.

The hopper and crusher will be enclosed in an insulated steel sheeted building. A crusher operator will control the feeder and the rate of dumping. A helper is provided to assist the operator on lubrication, cleanup and patrol of the coarse ore stockpile conveyor since these are not part of the mill building.

4.3.3 Coarse Ore Stockpile

This stockpile should hold at least 7000 tons of crushed ore to act as a 48 hour surge ahead of the mill in case of interruptions in the pit operation or major maintenance on the jaw crusher. The pile will have a base measuring between 50 and 75 feet in diameter, with a height of about 50 feet. Some bulldozing will be required to widen the pile with the central area providing the active storage. No provision has been made for covering this pile although it would be desirable to eliminate as much of the snow as possible since the ore must be dried in the Aerofall Mill.

Two reciprocating feeders will be located under the pile by which the feed for the Aerofall Mill will be withdrawn on demand by the automatic controls on the Mill. The ore from the feeders will travel by a conveyor to the feed hopper on the Mill.

4.3.4 Aerofall Mill System

The mill selected is a dry autogenous Aerofall Mill, 17 feet in diameter driven by a 600 HP motor. The dry grinding system was selected primarily to allow for dry magnetic separation which has shown better rejection of tailings due to a combination of centrifugal force and the magnetic field. The only autogenous testwork on the Expo ore was conducted in a model

Aerofall Mill which provided the capacity and power data for this study. Wet autogenous grinding is expected to show similar results in grinding but poorer results in magnetic separation, especially due to the lack of circulation of the middlings to the mill.

The Mill will be lined with manganese steel and chrome molybdenum case liners which will allow for the use of 5" steel balls if required for higher capacity.

The Mill is provided with an oil-fired heater to remove the moisture to allow for dry air classification.

Ground ore is withdrawn through the discharge trunion by negative pressure from the main fan with a 400 HP motor. An air classification system consists of a vertical classifier, a cyclone classifier and cyclones, each receiving the finer fractions in succession. A bag filter sys;em is provided to remove the dust.

The vertical classifier product discharges onto a screen with a 14 or 20 mesh screen. The screen oversize is returned to the Mill on the crude ore feed conveyor.

Part of the moisture ladened air is bled off and exhausted to the atmosphere.

4.3.5 Dry Magnetic Separation

The 3-sized products from the classification system are treated separately by two stages of dry magnetic separators. (See Drawing E-1).

The screen undersize of the vertical classifier is conveyed the length of the building and transferred to a second conveyor scissoring back to provide sufficient elevation to feed the dry separator. The feed from the cyclone classifier and possibly 2 cyclones will feed directly to their respective separators.

The two-pass separation consists of feeding an upper drum operated at relatively low speed to produce a final tailing and a rougher concentrate. The second separator operates at high speed and treats the rougher concentrate to produce a concentrate and a middling. The middling is recirculated by conveyor back to the Aerofall Mill feed conveyor. The concentrate is collected under the four lower separators and transferred by conveyor to a bucket elevator beside the storage bin ahead of the regrind mill.

4.3.6 Preconcentrate Storage Bin

The bucket elevator lifts the dry concentrate to the top of the bin onto a transfer conveyor to the centre of the bin. The bin will provide for 24-hour storage or 800 tons of regrind mill feed in a bin 26 ft in diameter by 20 feet high plus the conical base. A belt feeder at the bottom of the bin will regulate the feed to the regrind ball mill.

4.3.7 Regrind Ball Mill

This mill was selected as 10 ft diameter by 17½ ft long with a 700 HP motor based on experience with similar regrinding in the absence of a work index. The regrind mill will operate in closed circuit with cyclones; the underflow returning to the mill, the overflow going to the wet magnetic separators. The grind of the cyclone overflow will be in the range of 75 to 80% - 325 Mesh.

4.3.8 Wet Magnetic Separation

The cyclone overflow will be distributed to 2 - 3-drum finisher wet magnetic separators. Each drum will be 36" in diameter by 8 feet long. The final concentrate from the separators will be pumped to the concentrate thickener.

4.3.9 Dewatering of Final Concentrate

The magnetic concentrate will be thickened to 70% solids and pumped to a 6'9" disc filter with 8 discs. The discharge moisture will be in the range of 9.5% or 10%. The filter cake will be conveyed to a covered storage building. The filtrate from the dewatering will be pumped to the concentrate thickener.

The overflow from the concentrate thickener will be used for make-up water in the cyclone feed sump.

4.3.10 Filter Cake Storage

A building has been provided to hold 5000 tons, equivalent to about 7 days' production. The building will measure 50 feet long, 40 feet deep and 30 feet high. The front will be open to allow easy access for loading by a front end loader into a truck for haulage to the metallizing plant.

4.3.11 Tailings Disposal

Dry tailings from the dry separators will be conveyed to a pulping tank into which the finisher tailings will flow from the wet magnetic separators.

	S.T.P.H.					
	Solids	Water	Pulp	% Solids	Pulp Sp.Gr.	USGPM Pulp
Dry Tails	110.5	-	-	100		-
Wet Tails	8.4	338	346	2.5	1.02	1363
Combined Tails to Pond	118.9	338	457	26	1.20	1520

The combined tails at 26% solids will be pumped to a tailings pond. The water balance was based on reclaiming 80% of the water going to the tailings pond for re-use in the Mill.

5. MILL OPERATING COST ESTIMATE

5.1 SUMMARY OF OPERATING COSTS

Based on 3500 short tons per day -

	Cost Per Short Ton	
	Mill Feed	Concentrate
1) Staff	$ 0.083	$ 0.461
2) Operating Labour	0.152	0.845
3) Maintenance Labour	0.071	0.395
4) Operating Supplies	0.433	2.408
5) Power - (excluding reclaim and fresh water) 11.5 KWH/T at 1.2 cents/KWH	0.138	0.767
TOTAL	$ 0.877	$ 4.876

5.2 BASIS FOR OPERATING COST ESTIMATE

5.2.1 Fringe Benefits

The 17% added to all wages and salaries is intended to cover holiday pay, Workmen's Compensation, insurance and the Canada Pension Fund contributions by the Company.

5.2.2 Wage Rates

Wage rates used for this study are current rates being paid for mill operation in the general area of the Expo Iron property.

5.2.3 Power Costs

This cost of 1.2 cents/KWH is based on generating the power at the mine site by diesel generators using fuel oil at 17.4 cents per gallon delivered.

5.2.4 Concentration Ratio: 5.56:1

5.3 ESTIMATED MILL OPERATING LABOUR COST

	No. of Men	Rate	Monthly Cost
Staff			
Mill Superintendent	1	$1000/month	$ 1,000
Shift Foremen	4	$ 800/month	3,200
Assayer	1	$ 700/month	700
Technician	1	$ 600/month	600
Clerk-Typist	1	$ 500/month	500
Maintenance Foreman	1	$ 850/month	850
		Sub Total	$ 6,850
		17% Fringe	1,165
		TOTAL	$ 8,015

	No. of Men	Rate	Monthly Cost
Mill Operation			
Crusher Operator	2	$ 2.85/hour	$ 1,094
Crusher Helper	2	$ 2.65/hour	1,018
Grinding Operators	3	$ 2.85/hour	2,050
Separator Operators	3	$ 2.85/hour	2,050
Helpers	3	$ 2.65/hour	1,908
Mill Labourers	4	$ 2.45/hour	2,352
Regrind and Filter Operators	3	$ 2.85/hour	2,050
		Sub Total	$ 12,522
		17% Fringe	2,129
		TOTAL	$ 14,651

Mill Maintenance

Unscheduled	No. of Men	Rate	Monthly Cost
Mechanic - Welders - Electricians	6	$3.00/hour	$ 4,320
Scheduled and Shop			
120 hours/week		$3.00/hour	1,548
		Sub Total	$ 5,868
		17% Fringe	998
		TOTAL	$ 6,866

Monthly Feed Rate - 96,480 Short Tons

	Cost Per Ton Mill Feed
Staff	$ 0.083
Mill Operation	0.152
Maintenance	0.071
TOTAL LABOUR	$ .306

Personnel Requirements

Based on 40-hours per week operation, 3 shifts per day, 7 days per week for the mill; 2 shifts per day, 5 days per week for the crusher:

Staff	9
Mill Operation	25
Mill Maintenance (unscheduled)	6
Scheduled	3
TOTAL	43

5.4 OPERATING SUPPLIES

	$/Ton Mill Feed
Aerofall Mill Preheating	$ 0.058
2% moisture removal 60,000 BTU at 17.4 cents/gallon	
Steel Balls	
Aerofall 0.5 lb/ton at 12 cents/lb	0.060
Ball Mill 0.75 lb/ton at 12 cents/lb	0.090
Liners	
Aerofall 0.20 lb/ton at 40 cents/lb	0.080
Ball Mill - rubber 2 years at $25,000/set	0.045
Operating Supplies - including Jaw Crusher, Conveying, Filtering, Pumping, etc.	0.100
Power	
Connected: 2500 HP 90% connected: 2250 HP (1680 KW) 11.5 KWH/ton at 1.2 cents	0.138
TOTAL	$ 0.571

6. CAPITAL COST ESTIMATE

6.1 SUMMARY OF CAPITAL COST

Crushing Plant	$ 215,000
Mill	2,179,000
Water Storage and Fire Protection	25,000
Freight - Mill Equipment	100,000
Engineering	119,700
Construction Supervision, Purchasing and Expediting	100,000
TOTAL	$2,738,700

6.2 BASIS FOR CAPITAL COST ESTIMATE

The Capital Cost Estimate is an order-of-magnitude estimate without the benefit of detailed layout or design. Experience has been drawn from costs of similar installations. The major equipment as listed in Section 5 was priced. The motors and electrical installations are based on $80 per connected HP. The costs of the mill and storage building were estimated by Mr. J. Carr of Carr & Donald & Associates. The sites for the Jaw Crusher, Stockpile area and Mill were assumed to be a level area.

6.3 CAPITAL COSTS

6.3.1 Crushing Plant

Equipment	$ 130,000	
Installation	35,000	
Building, Hopper and Services	50,000	$ 215,000

6.3.2 Mill, Coarse Ore Storage and Concentrate Storage

Equipment and Installation	$1,450,000	
Electrical Motors, Installation and Distribution	200,000	
Buildings - Mill - 100 x 90 x 78' Concentrate Storage 50 x 40 x 30' (including excavation, foundations, structural steel, roofing and cladding	529,000	$2,179,000

6.3.3	Water Storage and Fire Protection	$ 25,000
6.3.4	Freight - 1000 tons of mill equipment at $100/ton	$ 100,000
6.3.5	Engineering - 5% of Crushing Plant and Mill	$ 119,700
6.3.6	Construction Supervision, Purchasing and Expediting (X)	$ 100,000
	TOTAL	$2,738,700

(X) Assuming that this work will be contracted to a small efficient organization.

H.E.Neal

H.E.Neal, P.Eng.

H.E.N.

An Investigation of

THE RECOVERY OF IRON

from samples of Drill Core

submitted by

EXPO IRON LIMITED

Progress Report No. 1

Project No. L.R. 1319

NOTE:

This report refers to the samples as received.

The practice of this Company in issuing reports of this nature is to require the recipient not to publish the report or any part thereof without the written consent of Lakefield Research of Canada Limited.

LAKEFIELD RESEARCH OF CANADA LIMITED
Lakefield, Ontario.
March 3, 1970

INTRODUCTION

In a letter dated February 5, 1970, Mr. E.D. Black of Metals, Petroleum and Hydraulic Resources Consulting Limited, requested us to proceed with a program of cobbing tests on a composite of diamond drill core samples received from Expo Iron Limited during November and December 1969. These shipments consisted of a total of 308 separate samples of drill core, each of which was sampled and assayed for soluble iron.

The program outlined by Mr. Black consisted of the following steps:

(1) Preparation of a composite from the coarse rejects of 238 samples.

(2) Screening into three separate size fractions

(3) Cobbing of each of the three size fractions

(4) Davis Tube tests on all products at minus 200 mesh

S U M M A R Y

1. Head Analysis

No direct analysis was performed on the composite sample. The calculated head assay from the test products was 16.15 % soluble Fe. The Davis Tube tests indicated a magnetic iron content of 11.4 %

2. Coarse Cobbing

Magnetic separation tests on the screened size fractions showed that 50 % of the ore could be rejected at plus ¼ inch and at plus ¼ inch minus 10 mesh, and that 65 % of the ore could be rejected at minus 10 mesh.

Table 1. Weights and Assays of Iron Concentrates

Size Fraction	Weight %	Assay %		Distribution %	
		Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
+ ¼ inch	49.5	22.82	19.1	71.7	85.4
10 mesh	49.8	24.32	20.0	76.7	90.0
- 10 mesh	34.7	42.60	41.0	78.9	99.1

The loss of magnetic iron in the + ¼ inch fraction was about 15 % and in the + 10 mesh fraction about 9 %. Less than 1 % of the magnetic iron was lost in the finest size fraction.

3. Davis Tube Results

The Davis Tube results from the magnetic cobber concentrate showed that there was further rejection of material assaying on average 5 % soluble iron, thus lowering the recovery of soluble iron to about

SUMMARY - Continued

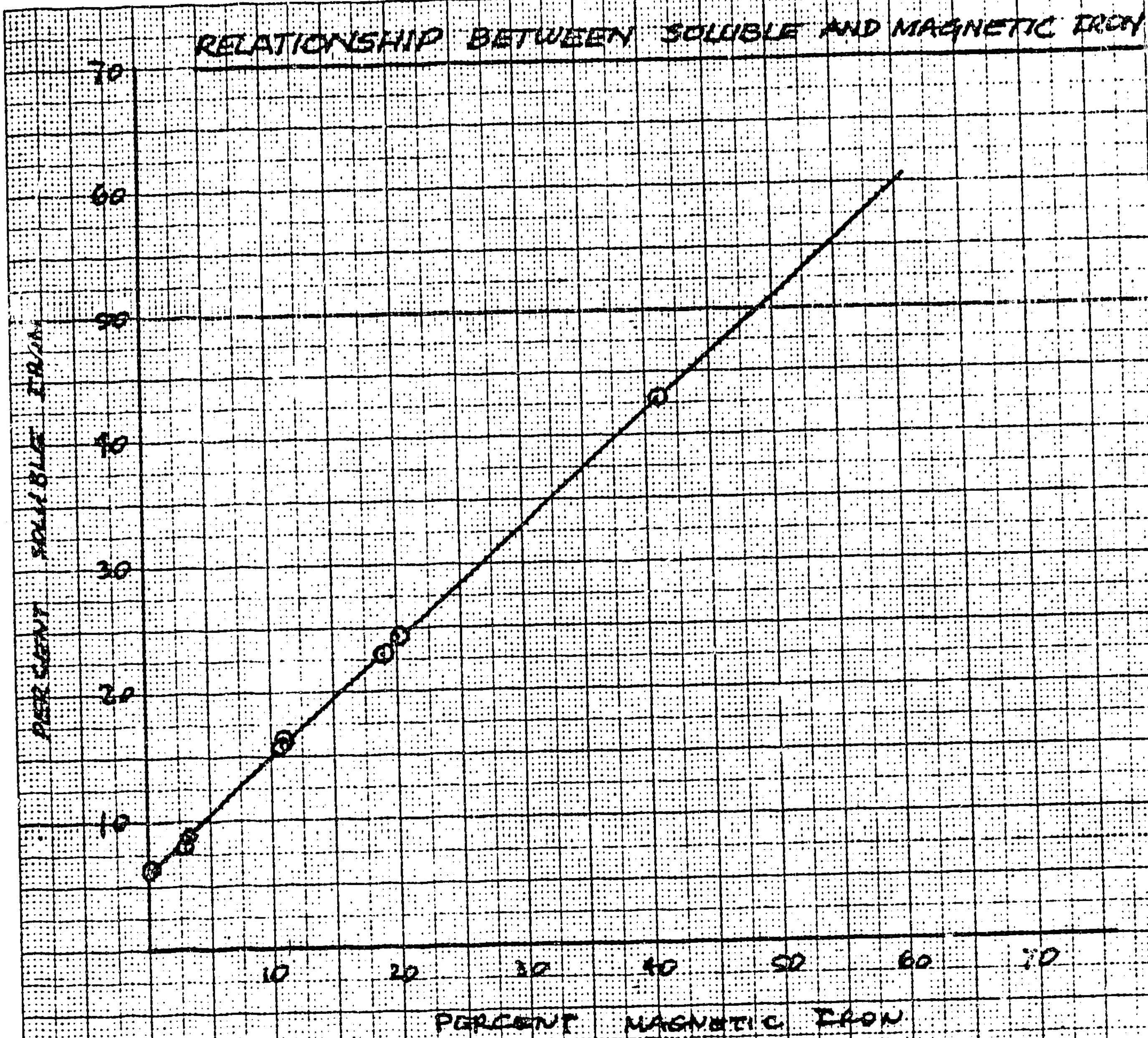
60 % (+ ¼" fraction) and 76 % (-10 mesh fraction), respectively. Similar tests on the non-magnetic fractions showed that the magnetic iron content of the respective size fractions decreased with fineness from 3.2 % and 2.0 % to 0.2 %.

4. General

Fine grinding and magnetic separation tests should be carried out to determine the final concentrate grade and the inherent impurities, the ratio of concentration and the grinding power requirements.

LAKEFIELD RESEARCH OF CANADA LIMITED

A.G. Scobie, P. Eng.,
Manager.


RELATIONSHIP BETWEEN SOLUBLE AND MAGNETIC IRON
70
60
50
40
30
20
10
PERCENT SOLUBLE IRON
10
20
30
40
50
60
70
PERCENT MAGNETIC IRON

SAMPLE PREPARATION

The composite was prepared by riffling a weighted portion (50 grams of sample per foot) from the jaw-crushed reject (approx. - ½ inch) from each of 238 samples listed in a letter from Mr. E.D. Black dated January 30, 1970. The entire resulting composite, about 40 kilograms in weight, was used for the testwork.

DETAILS OF TESTS

1. Screen Sizing

The composite was screened on ¼ inch and 10 mesh screens giving the following three size fractions:

Size Distribution

Size Fraction	Weight %	Assay %		Distribution %	
		Sol. Fe	Mag. Fe(1)	Sol. Fe	Mag. Fe
+ ¼ inch	75.6	15.75	11.1	73.7	72.7
- ¼ inch + 10 mesh	11.3	15.79	11.0	11.0	10.8
- 10 mesh	13.1	18.74	14.4	15.3	16.5
Head (Calc.)	100.0	16.15	11.4	100.0	100.0

(1) Calculated from Davis Tube results

2. Coarse Cobbing

Magnetic separation tests were performed on each of the above size fraction using a Sala Mortsel drum separator, 400 mm diameter and at a peripheral speed of 100 metres per minute. Both the magnetic and non-magnetic fractions were weighed and assayed for soluble iron.

2. Coarse Cobbing - Continued

Metallurgical Results

- ½ + ¼ inch

Product	Weight %	Assay %		Distribution %	
		Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
Concentrate	49.5	22.82	19.1	71.7	85.4
Tailing	50.5	8.81	3.2	28.3	14.6
Head (Calc.)	100.0	15.75	11.1	100.0	100.0

- ¼ inch + 10 mesh

Concentrate	49.8	24.32	20.0	76.7	90.9
Tailing	50.2	7.32	2.0	23.3	9.1
Head (Calc.)	100.0	15.79	11.0	100.0	100.0

- 10 mesh

Concentrate	34.7	42.60	41.0	78.9	99.1
Tailing	65.3	6.06	0.2	21.1	0.9
Head (Ca.c,)	100.0	18.74	14.4	100.0	100.0

Calculated Overall Results

Combined Concs.	47.5	24.87	21.3	73.3	88.1
Combined Tails	52.5	8.20	2.6	26.7	11.9
Head (Calc.)	100.0	16.15	11.4	100.0	100.0

3. Davis Tube Testing

The method of testing in the Davis tube consisted of pulverizing approximately 100 grams from each of the previous products by grinding for 3 minutes in a centrifugal disk-type pulverizer. A sample of the pulverized material was then treated in the Davis tube separator under fixed conditions, collecting and weighing the concentrate and assaying the concentrate for soluble iron.

Davis Tube Conditions

Current to coils:	2.0 amperes
Flux Density:	6500 gauss between poles at centre of tube
Water Flow:	950 ml per minute
Oscillations:	100 strokes per minute
Tube angle:	45 degrees
Retention time:	4 minutes

3. Davis Tube Testing - Continued

Metallurgical Results

+ ½ inch Magnetics

Product	Weight %	Assay %	% Distribution
		Sol. Fe	Sol. Fe
Concentrate	28.2	67.78	83.8
Tailing (Calc.)	71.8	5.16	16.2
Head	100.0	22.82	100.0

- ½ inch + 10 mesh Magnetics

Concentrate	29.6	67.62	82.3
Tailing (Calc.)	70.4	6.12	17.7
Head	100.0	24.32	100.0

- 10 mesh Magnetics

Concentrate	59.4	68.94	96.1
Tailing (Calc.)	40.6	4.06	3.9
Head	100.0	42.60	100.0

+ ½ inch Non-magnetics

Concentrate	5.1	62.42	36.1
Tailing (Calc.)	94.9	5.[illegible]	63.9
Head	100.0	8.81	100.0

- ½ inch + 10 mesh Non-magnetics

Concentrate	3.1	63.37	26.8
Tailing (Calc.)	96.9	5.53	73.2
Head	100.0	7.32	100.0

3. Davis Tube Testing - Continued

Metallurgical Results

- 10 mesh Non-magnetics

Product	Weight	Assay %	Distribution %
		Sol. Fe	Sol. Fe
Concentrate	0.4	40.05	2.7
Tailing (Calc.)	99.6	5.92	97.3
Head	100.0	6.06	100.0

Investigation by : C.W. Payne
O.F.C. Cook

Lakefield Research of Canada Limited,
Lakefield, Ontario
March 3, 1970 / si

An Investigation of

THE RECOVERY OF IRON

from samples of Drill Core

submitted by

EXPO IRON LIMITED

Progress Report No. 2

LAKEF ELD RESEARCH OF CANADA LIMITED
LAKEFIELD, ONTARIO
CANADA

An Investigation of

THE RECOVERY OF IRON

from samples of Drill Core

submitted by

EXPO IRON LIMITED

Progress Report No. 2

Project No. L.R. 1319

NOTE:

This report refers to the samples as received.

LAKEFIELD RESEARCH OF CANADA LIMITED
Lakefield, Ontario.
June 25, 1970

I N T R O D U C T I O N

In a memorandum dated April 9, 1970, Mr. H.E. Neal requested, on behalf of Expo Iron Limited, that we should continue work on the iron ore sample with the object of determining the minimum grind required to produce a Davis Tube concentrate assaying 68 to 70 % acid soluble Fe.

A further memorandum requested the treatment of the magnetic concentrate by flotation with the purpose of reducing the sulphur level to 0.5 - 0.7 %.

S U M M A R Y

Head Assays

Acid soluble Fe ☆	15.76 %
Magnetic Fe ☆	11.8 %
Sulphur (S)	1.30 %

☆ Calculated from test products

Magnetic Separation

Head samples of minus 10 mesh and of minus 20 mesh were treated in parallel by screening, magnetic separation, grinding and further magnetic separation. Initially, three grinds of 20, 30 and 40 minutes, prior to the final Davis Tube separation, were investigated. From the minus 10 mesh head sample the grade of the David Tube concentrate varied from 68.4 % to 69.0 % acid soluble Fe and the recovery from 73.9 % to 72.6 %. From the minus 20 mesh sample these results were 68.4 % to 69.4 % Fe grade and 75.2 % to 74.5 % recovery.

To determine the reproducibility of these results on a larger scale, using the Jeffrey wet magnetic separator instead of the Davis Tube, a composite sample of primary magnetic concentrates from the minus 10 and minus 20 mesh tests, was ground for 35 minutes, upgraded by a three-stage magnetic separation to give a final concentrate of 68.7 % soluble Fe at 73.8 % recovery. The TiO_2 assay of this concentrate was 0.154 %, the sulphur assay 4.34 % .

Sulphide Flotation

Flotation tests were performed to reduce the sulphur content of this final magnetic concentrate. Mineralogical examination indicated that the sulphide, identified as pyrrhotite, was fully liberated at this grind. The lowest sulphur content achieved in the tests to date was 0.96 % S. This was obtained by floating

Summary - Continued

at pH 5.0 with amyl xanthate as collector. Flotation at a natural pH of 7.4, with $CuSO_4$ activation, only removed 9.4 % of the sulphur. By grinding the concentrate from 89.4 % to 98.5 % minus 200 mesh the recovery was increased to 62.6 % leaving a magnetic concentrate assaying 2.21 % S.

Further flotation testwork is recommended to reduce this sulphur content to below 0.7 %.

LAKEFIELD RESEARCH OF CANADA LIMITED

A.G. Scobie, P. Eng.,
Manager.

SAMPLE PREPARATION

A composite of the samples was prepared by combining the minus 10 mesh rejects from 238 core samples received late in 1969. This composite, weighing about 350 pounds, was mixed and three-quarters rejected by riffling and stored. The remaining quarter was again riffled into two halves. One original eighth was retained at 10 mesh, while the other eighth was screened on 20 mesh and the oversize roll-crushed to all minus 20 mesh.

MAGNETIC SEPARATION

Test No. 1

Each of the minus 10 mesh and the minus 20 mesh samples were screened on a 150 mesh screen and the size fractions were weighed, giving the following size distributions;

(a) Minus 10 Mesh Ore Sample

Mesh Size	Weight %	Assays, %		% Distribution
		Sol. Fe	Mag. Fe	Sol. Fe
- 10 + 150 mesh	81.4	15.3	-	79.9
Minus 150 mesh	18.6	16.8	-	20.1
Total	100.0	15.6	11.6	100.0

(b) Minus 20 Mesh Ore Sample

- 20 + 150 mesh	75.5	15.9	-	75.5
Minus 150 mesh	24.5	15.9	-	24.5
Total	100.0	15.9	11.9	100.0

The plus 150 mesh fractions from each of the minus 10 and the minus 20 mesh head samples were magnetically separated by passing over a Sala Mortsel drum separator 400 mm diameter, at a peripheral speed of 400 feet per minute.

Results of Sala Separations

(a) Minus 10 plus 150 mesh

Product	Weight %	Assay, %		% Distribution
		Sol. Fe	Mag. Fe	Sol. Fe
Magnetics	30.0	38.9	-	76.4
Non-magnetics	70.0	5.2	0.4	23.6
Head	100.0	15.3	-	100.0

(b) Minus 20 plus 150 mesh

Magnetics	27.4	45.0	-	77.3
Non-magnetics	72.6	5.0	0.2	22.7
Head	100.0	15.9	-	100.0

The mi[illegible] 150 mesh fractions from each of the minus 10 and the minus 20 mesh head samples were magnetically separated by passing through the Jeffrey wet magnetic separator. Both magnetic and non-magnetic products were collected, filtered, dried and assayed.

Results of Jeffrey Separations

(a) Minus 150 mesh (minus 10 mesh head)

Magnetics	21.3	64.5	-	81.9
Non-magnetics	78.7	3.9	0.1	18.1
Head	100.0	16.8	-	100.0

(b) Minus 150 mesh (minus 20 mesh head)

Magnetics	20.1	63.6	-	80.6
Non-magnetics	79.9	3.8	0.1	19.4

METALLURGICAL RESULTS

Test No. 1A Minus 10 Mesh Ore

Product	Weight %	Assay, %		% Distribution	
		Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
+ 150 mesh Sala magnetics	24.5	38.9	-	61.1	-
+ 150 mesh Sala non-magnetics	56.9	5.2	0.4	18.8	-
- 150 mesh Jeffrey magnetics	3.9	64.5	-	16.4	-
- 150 mesh Jeffrey non-magnetics	14.7	3.9	0.1	3.7	-
Head (Calc.)	100.0	15.6	11.6	100.0	100.0
Combined magnetics	28.4	42.5	39.8	77.5	97.9
Combined non-magnetics	71.6	4.9	0.3	22.5	2.1

Test No. 1B Minus 20 Mesh Ore

+ 150 mesh Sala magnetics	20.7	45.0	-	58.4	-
+ 150 mesh Sala non-magnetics	54.8	5.0	0.2	17.2	-
- 150 mesh Jeffrey magnetics	4.9	63.6	-	19.7	-
- 150 mesh Jeffrey non-magnetics	19.6	3.9	0.1	4.7	-
Head (Calc.)	100.0	15.9	11.9	100.0	100.0
Combined magnetics	25.6	48.6	46.4	78.1	98.8
Combined non-magnetics	74.4	4.7	0.2	21.9	1.2

Test 1C

The combined magnetic products from the minus 10 mesh ore were mixed and riffled into 1000 gram samples. Three of these samples were ground in a Denver laboratory ball mill with 28 pounds of steel balls for 20, 30 and 40 minutes respectively. The ground pulps were filtered and dried. A sample was taken from each of the ground samples for screen analysis and Davis tube testing.

Results of Regrinding and Davis Tube Tests

Grinding Time	Davis Tube Concentrate				D.T. Tailing Assay, % Sol. Fe
	Weight %	Assay, % Sol. Fe	% Recovery Sol. Fe		
			Individual	Overall	
20	59.2	68.36	95.4	73.9	4.83
30	58.6	68.42	94.5	73.2	5.67
40	57.6	69.00	93.7	72.6	6.36

Screen Analysis

Mesh Size (Tyler)	20 Minute Grind		30 Minute Grind		40 Minute Grind	
	% Retained Individual	% Passing Cumulative	% Retained Individual	% Passing Cumulative	% Retained Individual	% Passing Cumulative
+ 65	0.2	99.8	0.1	99.9	0.1	99.9
100	0.7	99.1	0.2	99.7	0.2	99.7
150	3.4	95.7	1.1	98.6	0.5	99.2
200	13.4	82.3	6.4	92.2	3.8	95.4
270	14.6	67.7	10.4	81.8	7.2	88.2
325	12.2	55.5	11.5	70.3	11.0	77.2
- 325	55.5	-	70.3	-	77.2	-
Total	100.0	-	100.0	-	100.0	-

40 Minute Grind: Specific Gravity: 4.08
Surface Area: 1895 cm^2/gm

Similar treatment of the combined magnetic products from the minus 20 mesh ore sample gave the following results:

- 20 Mesh Sample

Grinding Time	Davis Tube Concentrate				D.T. Tailing Assay, % Sol. Fe
	Weight %	Assay, % Sol. Fe	% Recovery Sol. Fe		
			Individual	Overall	
20	68.4	68.43	96.3	75.2	5.61
30	67.6	68.91	95.9	74.9	6.16
40	66.8	69.42	95.5	74.5	6.65

Screen Analysis

Mesh Size (Tyler)	20 Minute Grind		30 Minute Grind		40 Minute Grind	
	% Retained Individual	% Passing Cumulative	% Retained Individual	% Passing Cumulative	% Retained Individual	% Passing Cumulative
+ 65	0.1	99.9	-	-	-	-
100	0.6	99.3	0.2	99.8	0.1	99.9
150	2.7	96.6	0.8	99.0	0.4	99.5
200	11.7	84.9	5.4	93.6	2.7	96.8
270	14.0	70.[illegible]	9.4	84.2	5.9	90.9
325	12.2	58.7	10.6	73.6	9.0	81.9
- 325	58.7	-	73.6	-	81.9	-
Total	100.0	-	100.0	-	100.0	-

40 Minute Grind: Specific Gravity: 4.27
Surface Area: 1998 cm^2/gm

Test No. 2

A composite sample of the Sala and Jeffrey magnetic concentrates from tests 1A and 1B was prepared by combining the concentrates from each size fraction in relation to the respective weight recoveries.

A 1000 gram charge of this composite (for reference purposes called the primary concentrate) was then ground for 35 minutes before passing three times over the Jeffrey wet magnetic separator. The rougher and cleaner non-magnetics were combined for assay.

Metallurgical Results

Product	Weight, %		Assays, %				% Distr. Sol. Fe	
	Ind.	O'all	Sol. Fe	Mag. Fe	TiO_2	S	Ind.	O'all
Concentrate	63.2	17.1	68.7	-	0.154	4.34	94.8	73.8
Comb. Tailing	36.8	9.9	6.5	1.8	-	-	5.2	4.0
Head	100.0	27.0	45.8	11.7	-	-	100.0	77.8

The overall magnetic iron recovery was 92.8 %.

Screen Analysis - Feed to Magnetic Separator.

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	0.1	0.1	99.9
100	0.1	0.2	99.8
150	0.5	0.7	99.3
200	3.3	4.0	96.0
270	6.6	10.6	89.4
325	10.1	20.7	79.3
- 325	79.3	100.0	-
Total	100.0	-	-

2. FLOTATION OF SULPHIDES

A microscopic examination of the magnetic cleaner concentrate indicated that the sulphides, identified as pyrrhotite, were fully liberated. Two tests were performed on the Jeffrey concentrate from test No. 2, one test was performed on freshly ground primary concentrate, before magnetic separation, and the last test was carried out on specially prepared concentrate, omitting the drying stage.

Test No. SFT-1

Purpose: To float sulphide minerals from the final Jeffrey magnetic concentrate from test No. 2.

Feed: 500 grams of final Jeffrey magnetic concentrate.

Conditions:

Stage	Reagents Added, lbs/ton			Time, minutes		
	$CuSO_4$	Z - 6	MIBC	Cond.	Froth	pH
Ro. Conditioning	0.5	-	-	2	-	-
Ro. Con.. No. 1 (1)	-	0.005	0.02	1	5	5.9
(2)	-	0.005	0.02	1	5	-
Ro. Conc. No. 2	-	0.005	0.01	1	6	-

Stage Roughing
Flotation Cell 250 g D-1
Speed:r.p.m. 1000

Metallurgical Results

Product	Weight %	Assays, %		% Distribution	
		Sol. Fe	S	Sol. Fe	S
1. Sulphide Conc. 1	2.2	64.5	8.97	2.0	4.5
2. Sulphide Conc. 2	2.2	66.2	9.32	2.1	4.9
3. Tails	95.6	69.4	4.03	95.9	90.6
Head (Calc.)	100.0	69.2	4.25	100.0	100.0

Test No. SFT-2

Purpose: To remove sulphide minerals from the magnetic concentrate after regrinding to 80 % minus 27 microns.

Feed: 500 grams of final Jeffrey magnetic concentrate from test 2.

Grind: 15 minutes at 50 % solids in laboratory ball mill with 28 lbs of steel balls.

Conditions:

Stage	Reagents Added, lbs/ton			Time, minutes			
	Z - 6	MIBC	$CuSO_4$	Grind	Cond.	Froth	pH
Grind	-	-	-	15	-	-	-
Ro. Conc. (1)	0.05	0.02	-	-	2	1½	7.4
(2)	-	0.02	-	-	½	1½	-
(3)	-	-	0.50	-	3	-	-
(4)	0.05	0.02	-	-	1	5	-
(5)	0.05	0.01	-	-	1	3	-
(6)	-	-	0.50	-	3	-	-
(7)	0.05	0.01	-	-	1	3	-
(8)	0.05	0.01	-	-	1	5	-
Cleaner (1)	-	0.02	-	-	1	4	-

Stage	Flotation
Flotation Cell	250 g Agitair
Speed:r.p.m.	1000

Test No. SFT-2 - Continued

Metallurgical Results

Product	Weight %	Assays, %		% Distribution	
		Sol. Fe	S	Sol. Fe	S
1. Sulphide Cl. Conc.	4.4	64.7	13.9	4.1	14.5
2. Sulphide Cl. Tails	24.1	67.3	8.45	23.5	48.1
3. Sulphide Ro. Tails	71.5	69.8	2.21	72.4	37.4
Head (Calc.)	100.0	69.0	4.23	100.0	100.0

Screen Analysis

Particle Size	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 270 m	1.5	1.5	98.5
26.7 µ	18.2	19.7	80.3
20.7 µ	18.9	38.6	61.4
14.4 µ	16.9	55.5	44.5
9.9 µ	15.7	71.2	28.8
7.7 µ	8.0	79.2	20.8
- 7.7 µ	20.8	100.0	-
Total	100.0	-	-

Test No. SFT-3

Purpose: To float the sulphide minerals from the primary concentrate prior to final magnetic separation.

Procedure: After grinding, float a rougher concentrate and clean once. Maintain a pH of 5 throughout flotation.

Feed: 1000 grams of primary concentrate

Grind: 35 minutes at 65 % solids in the laboratory ball mill.

Conditions:

Stage	Reagents Added, lbs/ton				Time, minutes			pH
	Z-6	MIBC	$CuSO_4$	H_2SO_4	Grind	Cond.	Froth	
Grind	-	-	-	-	35	-	-	8.4
Ro. Float (1)	0.05	0.04	-	13.0	-	2	5½	5.0
(2)	-	-	0.50	-	-	3	-	5.2
(3)	0.05	0.02	-	-	-	1	5	-
Cleaner (1)	-	-	-	-	-	-	2	5.4
(2)	-	0.02	-	-	-	½	4	-

Stage	Rougher	Cleaner
Flotation Cell	500 G D-1	250 g D-1
Speed:r.p.m.	1300	900

Metallurgical Results

Product	Weight %	Assays, %		% Distribution	
		Sol. Fe	S	Sol. Fe	S
1. Sulphide Cl. Conc.	7.0	57.6	27.5	8.7	60.8
2. Sulphide Cl. Tails	3.7	42.4	10.3	3.4	12.0
3. Sulphide Ro. Tails	89.3	45.4	0.96	87.9	27.2
Head (Calc.)	100.0	46.2	3.15	100.0	100.0

Calculated Grades and Recoveries

Products 1 and 2	10.7	52.36	21.6	12.1	72.8

Test No. SFT-4

Purpose: To remove the sulphide minerals immediately after final magnetic separation.

Procedure: Grind, make a magnetic separation and clean the rougher magnetic concentrate twice. Perform a flotation test similar to the preceeding test.

Feed: 500 grams of primary concentrate.

Grind: 17½ minutes at 65 % solids in the laboratory ball mill.

Conditions:

Stage	Reagents Added, pounds per ton					Time, minutes			
	Z - 6	MIBC	$CuSO_4$	H_2SO_4	Pine Oil	Grind	Cond.	Froth	pH
Grind	-	-	-	-	-	17½	-	-	-
Magnetic Separation									7.2
Ro. Float (1)	0.05	0.04	-	5.0	-	-	2½	3½	6.0
(2)	0.05	-	-	-	-	-	1	2	-
(3)	0.05	-	0.50	-	-	-	1	4½	-
Demagnetize cell product and remove one more product.									
(4)	0.05	0.01	-	-	-	-	1	4	-
Demagnetize rougher concentrate before cleaning									
Cleaning	0.05	0.02	-	-	0.08	-	1	5	-

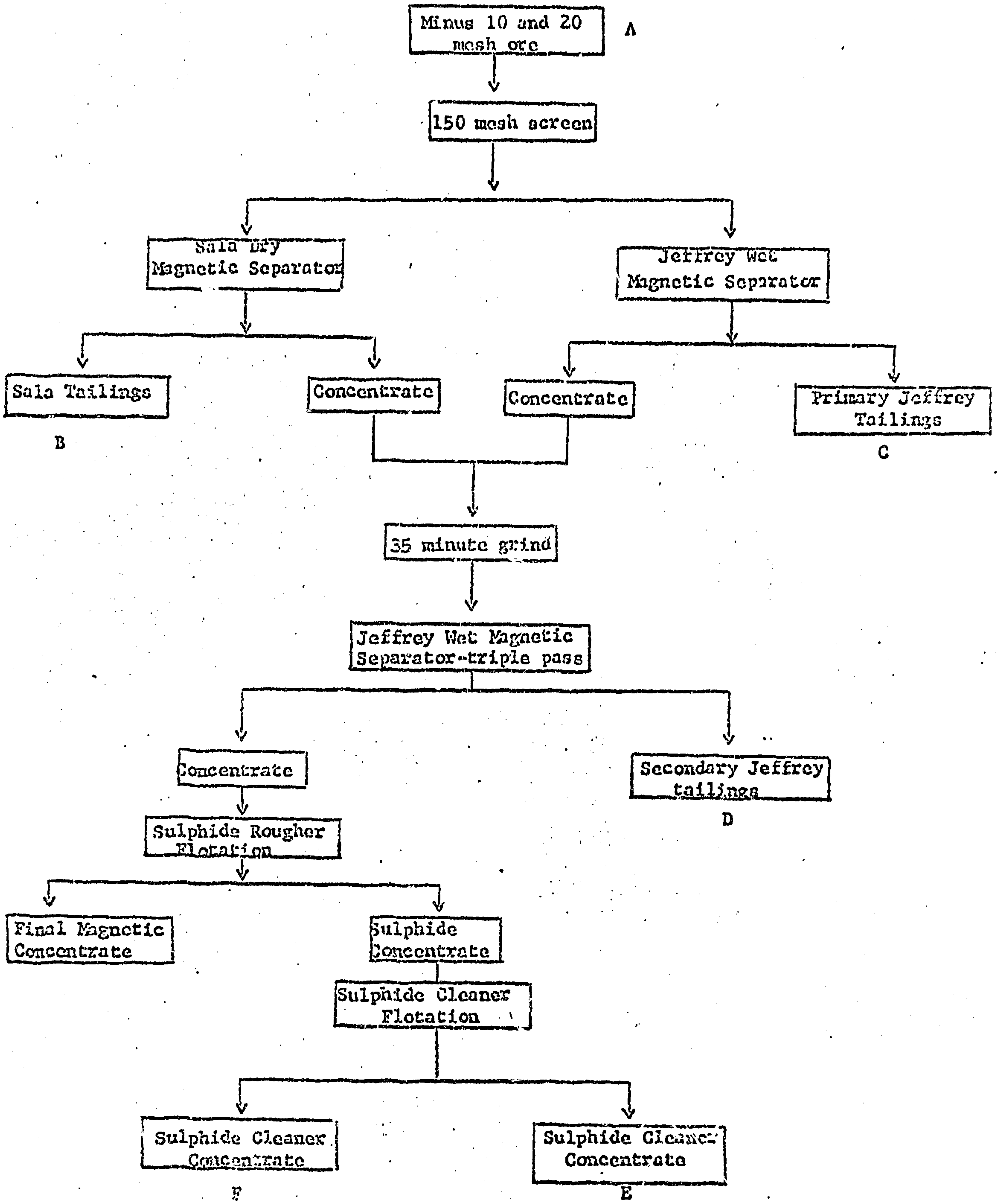
Stage	Rougher and Cleaner
Flotation Cell	250 g D-1
Speed:r.p.m.	900

Test No. SFT-4 - Continued

Metallurgical Results

Product	Weight %	Assays, %		% Distribution	
		Sol. Fe	S	Sol. Fe	S
Combined non-magnetics	39.7	10.0	1.42	8.7	18.8
Sulphide Cl. Conc.	7.0	66.1	21.4	10.2	50.3
Sulphide Cl. Tailing	4.9	69.4	9.28	7.4	15.1
Iron Concentrate	48.4	69.4	0.98	73.7	15.8
Head (Calc.)	100.0	45.6	2.99	100.0	100.0
Comb. Magnetics	60.3	69.0	4.03	91.3	81.2
Sulphide Ro. Conc.	11.9	67.4	16.5	17.6	65.4

FLOWSHEET FOR OVERALL BALANCE



MATERIAL BALANCE

	Product	Weight %	Assays, %			% Distribution		
			Sol. Fe	Mag. Fe	S	Sol. Fe	Mag. Fe	S
A	Head	100.0	15.9	11.8	1.30	100.0	100.0	100.0
B	Sala Tailing	55.9	5.1	0.3	-	17.8	1.4	-
C	Jeffrey Tailing	17.1	3.9	0.1	-	4.2	0.1	-
D	Jeffrey Tailing*	9.9	6.5	1.8	-	4.0	1.5	-
E	Sulphide conc.	2.0	66.1	-	21.4	8.3	-	32.8
F	Sulphide Tailing	1.4	69.4	-	9.28	6.0	-	9.9
G	Iron concentrate	13.7	69.4	-	0.98	59.7	-	10.3

The combined non-magnetic tailings A, B and C represented 82.9 % of the total weight. Composition and distributions were as follows:

	Assays, %			% Distribution		
	Sol. Fe	Mag. Fe	S	Sol. Fe	Mag. Fe	S
Combined tailing	5.0	0.4	0.74	26.0	3.0	47.0

A head assay of 1.30 % S is equivalent to 1.98 % Fe assuming all the sulphide to be present as pyrrhotites. This constitutes 12.6 % of the iron in the feed. Similar conversions on other products from the overall balance gives the following additional calculated distribution:

Product	Weight %	Assay, % Fe		% Distribution of Fe	
		Sulphide	Non-sulphide	Sulphide	Non-sulphide
Comb. non-mag. tail.	82.9	1.12	3.87	47.0	23.0
Sulphide Cl. Conc.	2.0	32.5	33.6	32.8	4.8
Sulphide Cl. Tail.	1.4	14.1	55.3	9.9	5.5
Final iron conc.	13.7	1.49	67.9	10.3	66.7
Head	100.0	1.98	13.96	100.0	100.0

Investigation by: A.G.T. Bigg
C.W. Payne
J.A. Lauder

Lakefield Research of Canada Limited,
Lakefield, Ontario,
June 25, 1970 / si

AEROFALL MILLS LIMITED

Report Number 7005

REPORT OF TESTS

CONDUCTED IN THE 18-INCH AEROFALL MILL

ON

MAGNETIC IRON ORE

FOR

EXPO IRON LTD.

MAY, 1970

Aerofall Mills Limited,
2640 South Sheridan Way,
Clarkson, Ontario, Canada

(416) 822-4950

Report of Tests
Conducted in the 18-inch Aerofall Mill
on
Magnetic Iron Ore
for
Expo Iron Ltd.

May, 1970

INTRODUCTION

During the period from May 7 to May 20, 1970, a sample of Magnetic Iron Ore from Expo Mines Limited was tested in the 18-inch Aerofall mill at the laboratory of Aerofall Mills Limited, Clarkson, Ontario. Arrangements for the work were made by Mr. H.E. Neal, Consulting Engineer for that Company.

I - AEROFALL MILL TESTING

Method:

The ore received was crushed to minus 1½-inch, as feed for the 18-inch mill. The Aerofall mill system comprised an 18-inch diameter by 10-inch Aerofall mill fed by a Syntron feeder. The feed to the mill was controlled automatically by a Milltronics Ltd. feed control system maintaining a pre-set mill sound level as determined by a microphone located below the mill shell. The mill rotates at a speed of 52 rpm or approximately 83% of critical speed. A ball charge of 32 lb. of minus 2 1/4-inch balls was used in the mill amounting to 6.5 percent of the mill volume. Mill power drawn was approximately 0.110 net kilowatts.

The ground product was removed from the mill by an air stream provided by a draft fan. Air flow used was in the order of 120 cfm.

The products were collected in three classified fractions. A coarse fraction was collected in a vertical classifier, and screened on a 10 mesh screen. The screen oversize was returned to the mill as a circulating load. An intermediate product was collected in a 9-inch diameter cyclone; the fine product in a bag filter.

I - AEROFALL MILL TESTING - cont'd.

The mill was run for a total of 25.0 operating hours, the final 20.167 hours giving a capacity of 55.54 lb. per hour, and the following product distribution:

Product	% Weight
Vertical Classifier	39.14
Cyclone	60.17
Filter	0.69
Total	100.00

A screen analysis was made on the mill products over the first three hours of the test period, as follows:

Expo Iron - Test No. I

Product	Screen Oversize	Screen Undersize	Vertical Classifier	Cyclones	Filter	Overall
% Wt.	(3.37)*	43.24	--	56.26	0.50	100.00
+ 4						
6						
8						
10						
14		0.04		0.04		0.04
20		1.59		0.12		0.76
28		5.26		0.58		2.60
35		9.05		2.31		5.21
48		18.82		7.41	0.73	12.31
65		20.18		11.45	0.64	15.17
100		19.39		17.17	0.91	18.05
150		12.46		15.35	1.37	14.04
200		7.51		10.26	0.27	9.02
270		2.39		9.56	2.74	6.42
+325		1.59		5.40	1.46	3.73
-325		1.72		20.35	91.88	12.65
TOTAL		100.00		100.00	100.00	100.00

* Circulating Load

I - AEROFALL MILL TESTING - cont'd.

All products from the mill were then treated on a two-stage Sala-Mortsell 24-inch magnetic separator circuit, the first drum being run at 60 rpm to produce a coarse tailing, the second at 180 rpm to give a magnetic iron concentrate and a middling. In commercial practice this middling would be returned to the mill as a circulating load. Product distribution obtained was as follows:

	Percent Weight			
Product	Vertical Classifier	Cyclone	Filter	Total
Concentrate	10.52	15.10	--	25.62
Middling	2.49	2.73	--	5.22
Tailing	24.50	44.06	0.60	69.16
Total	37.51	61.89	0.60	100.00

The products obtained were shipped to Lakefield Research in Lakefield, Ontario, for assays and regrind tests.

II - COBBING TESTS

Method:

A sample of crude ore was crushed to give a minus two inch product which was screened at one inch and at six mesh. The plus one inch and plus 6 mesh fractions were treated on a 36-inch Sala belt cobber running at 40 rpm to produce a concentrate and a tailing; the minus 6 mesh fraction was treated in the two-stage Sala-Mortsell circuit as used in the Aerofall tests to give a concentrate, a middling and a tailing as follows:

	Percent Weight			
Product	Conc.	Midd.	Tail.	Total
-2 +1 inch	20.97	--	5.51	26.48
-1 inch +6 mesh	26.48	--	5.30	31.78
-6 mesh	10.81	6.14	24.79	41.74
Total	58.26	6.14	35.60	100.00

All products were shipped to Lakefield Research for analysis.

COMMENTS

1. The ore from Expo Mines Limited was found to be ideally suited to treatment in the Aerofall mill giving capacity rate appreciably higher than average ores.

2. From the data obtained it is estimated that an 18-foot diameter Aerofall mill would be required to produce the required tonnage rate of 160 to 170 long tons per hour.

3. An 18-foot diameter Aerofall mill would be driven by a 600 hp motor; the draft fan would be in the order of 400 hp.

4. The relatively large ball charge used in the 18-inch mill is characteristic of that equipment. In a commerical mill, with its higher impact forces, a smaller ball charge would be used.

Carr & Donald & Associates Ltd.

CONSULTING ENGINEERING SERVICES

REF. NO. ________ OWNER H.E. NEAL & ASSOCIATES LTD.

PROJECT NEAR OTTAWA AT "BUILDING AND LEAN TO"

PREPARED BY JC DEPT. CIVIL REF. DWG. --- DATE JULY 21, 1970.

ESTIMATE SHEET

ITEM	DESCRIPTION	UNIT	QUANT.	UNIT RATE	SUBTOTAL	TOTAL
	BROUGHT FORWARD					
1.	EXCAVATION FOR FOUNDATIONS					$ 11,000
2.	FOUNDATION CONCRETE IN PLACE					113,000
3.	STRUCTURAL STEEL IN PLACE					362,000
4.	CLADDING & ROOFING (2" INSULATION)					43,000
					TOTAL:	$ 529,000.00

11319 C. & R.

SHEET ______ OF ______

CARRIED FORWARD

PHONE: 652-3341 P. O. BOX 430

LAKEFIELD RESEARCH OF CANADA LIMITED

LAKEFIELD, ONTARIO

CANADA

APR 3- 1970

Certificate of Analysis

Semiquantitative Spectrographic

Date: April 2, 1970

Received:

From: Expo Iron Limited,
Suite 1510-100 Adelaide Street West,
Toronto 1, Ontario.

Our Reference No. L.R. 1319

Invoice No.

Samples submitted to us show results as follows:

	Sample L.R. 1319	Sample	Sample		Sample L.R. 1319	Sample	Sample
Antimony	-			Phosphorus	X		
Arsenic	-			Platinum	X		
Barium	.005 %			Rhenium	X		
Beryllium (BeO)	-			Rhodium	X		
Bismuth	-			Rubidium	X		
Boron	X			Ruthenium	X		
Cadmium	-			Silver	-		
Cerium (CeO_2)	-			Strontium	X		
Caesium	X			Tantalum (Ta_2O_5)	-		
Chromium	.05 %			Tellurium	X		
Cobalt	.02 %			Thallium	X		
Columbium (Cb_2O_5)	-			Thorium (ThO_2)	-		
Copper	.007 %			Tin	-		
Gallium	-			Titanium	.05 %		
Germanium	-			Tungsten	-		
Gold	X			Uranium (U_3O_8)	-		
Hafnium	X			Vanadium	.01 %		
Indium	-			Yttrium (Y_2O_3)	X		
Iridium	X			Zinc	-		
Lanthanum (La_2O_3)	X			Zirconium (ZrO_2)	-		
Lead	-			ROCK FORMING METALS			
Lithium (Li_2O)	-			Aluminum (Al_2O_3)	X		
Manganese	.5 %			Calcium (CaO)	X		
Mercury	-			Iron (Fe)	H		
Molybdenum	.01 %			Magnesium (MgO)	X		
Neodymium (Nd_2O_3)	X			Silica (SiO_2)	X		
Nickel	.01 %			Sodium (Na_2O)	X		
Palladium	X			Potassium (K_2O)	X		

Figures are approximate percentages:

CODE

H — High	— 10 — 100% approx.	LM — Low Medium	— .5 — 5% approx.
MH — Medium High	— 5 — 50% approx.	L — Low	— .1 — 1% approx.
M — Medium	— 1 — 10% approx.	TL — Trace Low	— .05 — .5% approx.
		T — Trace	— .01 — .1% approx.

FT — Faint Trace — approx. less than .01%.
PT — Possible Trace — Presence not certain.
ND — Not Detected — Elements looked for but not found.
X — Not looked for

SIGNED A G Scobie

A.G. Scobie, P. Eng.

Analysis and Assaying - Mineral Processing Research - Pilot Plant Investigations

G. & T. LTD. H-113588

PHONE: 652-3341

P. O. BOX 430

LAKEFIELD RESEARCH OF CANADA LIMITED
LAKEFIELD, ONTARIO
CANADA

Certificate of Analysis

Date: April 2, 1970

Received:

From: Expo Iron Limited,
Suite 1510-100 Adelaide St. W.,
Toronto 1, Ontario.

Our Reference No. L.R. 1319

Invoice No.

Samples submitted to us show results as follows:

	% Mn	% SiO_2	Total % Fe	% TiO_2	% P
D.T. Conc. Composite	0.62	1.87	68.32	0.18	0.007

N.B. Total number of assays invoiced on No. 8647 is correct.

To: Expo Iron Limited

SIGNED A G Scobie
MANAGER
A.G.Scobie, P. Eng.

Analysis and Assaying - Mineral Processing Research - Pilot Plant Investigations

C E R T I F I C A T E

I, HAROLD EGERTON NEAL, of the Municipality of Toronto, in the Province of Ontario HEREBY CERTIFY:

1. THAT I am a registered Professional Engineer with the Association of Professional Engineers of the Province of Ontario.

2. THAT I am a graduate geologist from the University of Toronto with a degree of Bachelor of Arts (1948) and a degree of Master of Arts (1949).

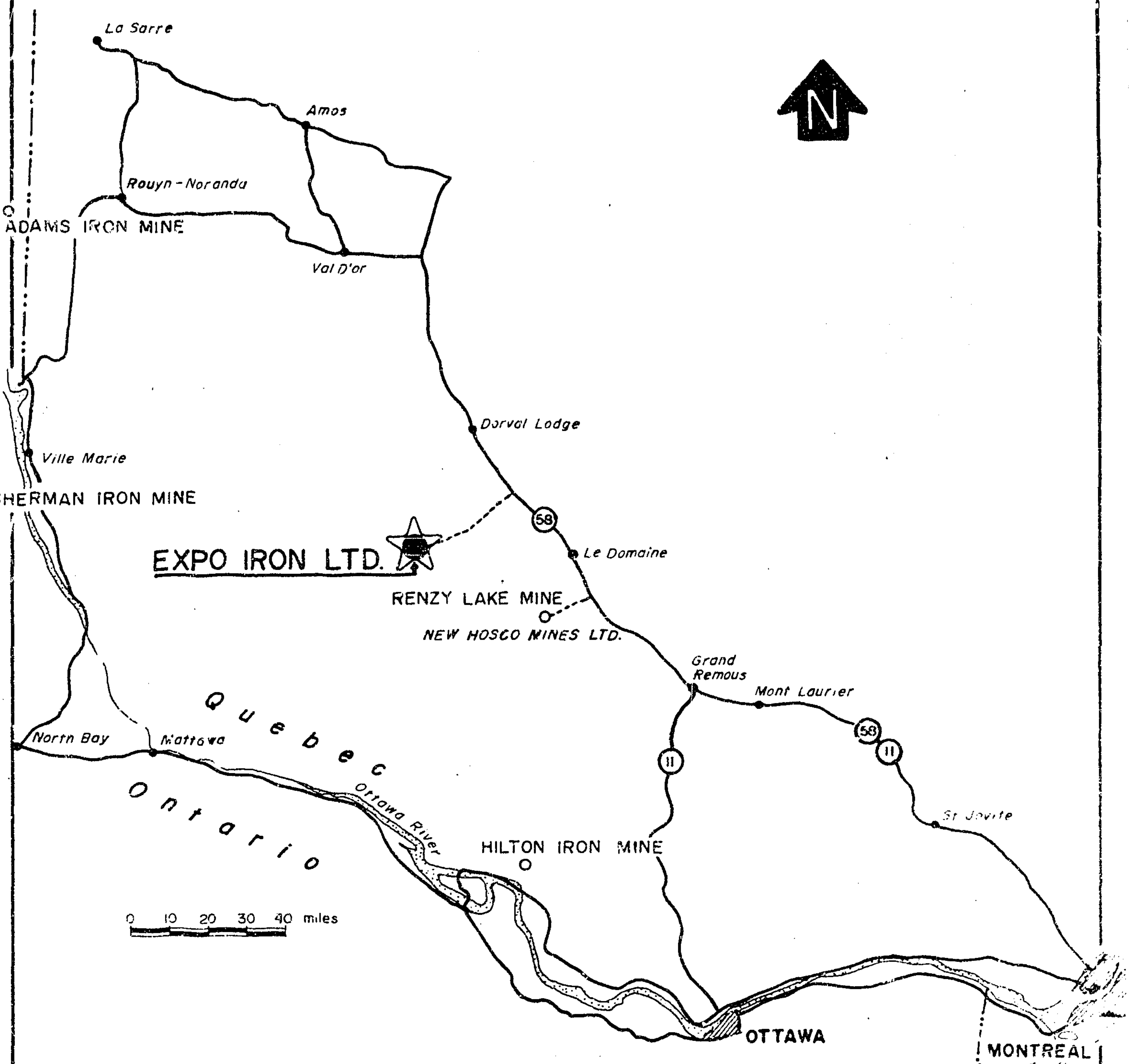
3. THAT I am a consulting Engineer residing at 124 Roxborough Drive, Toronto 5, Ontario.

4. THAT I have practiced as a Consulting Engineer for 8 years and formerly worked for the Iron Ore Company of Canada as Director of Research for 8 years and as a geologist for 7 years.

5. THAT I have no interest, direct or indirect nor do I expect to have any interest in Expo Iron Limited.

Dated at Toronto this 30th day of July, 1970.

H.E.Neal, P.Eng.



LEGEND

- Highways
- River
- Province boundary
- Bush road (gravel)

Ministère des Richesses Naturelles, Québec
27 MAI 1971
SERVICE DES GITES MINÉRAUX
No GM- 26835

metals, petroleum & hydraulic resources consulting limited

EXPO IRON LIMITED

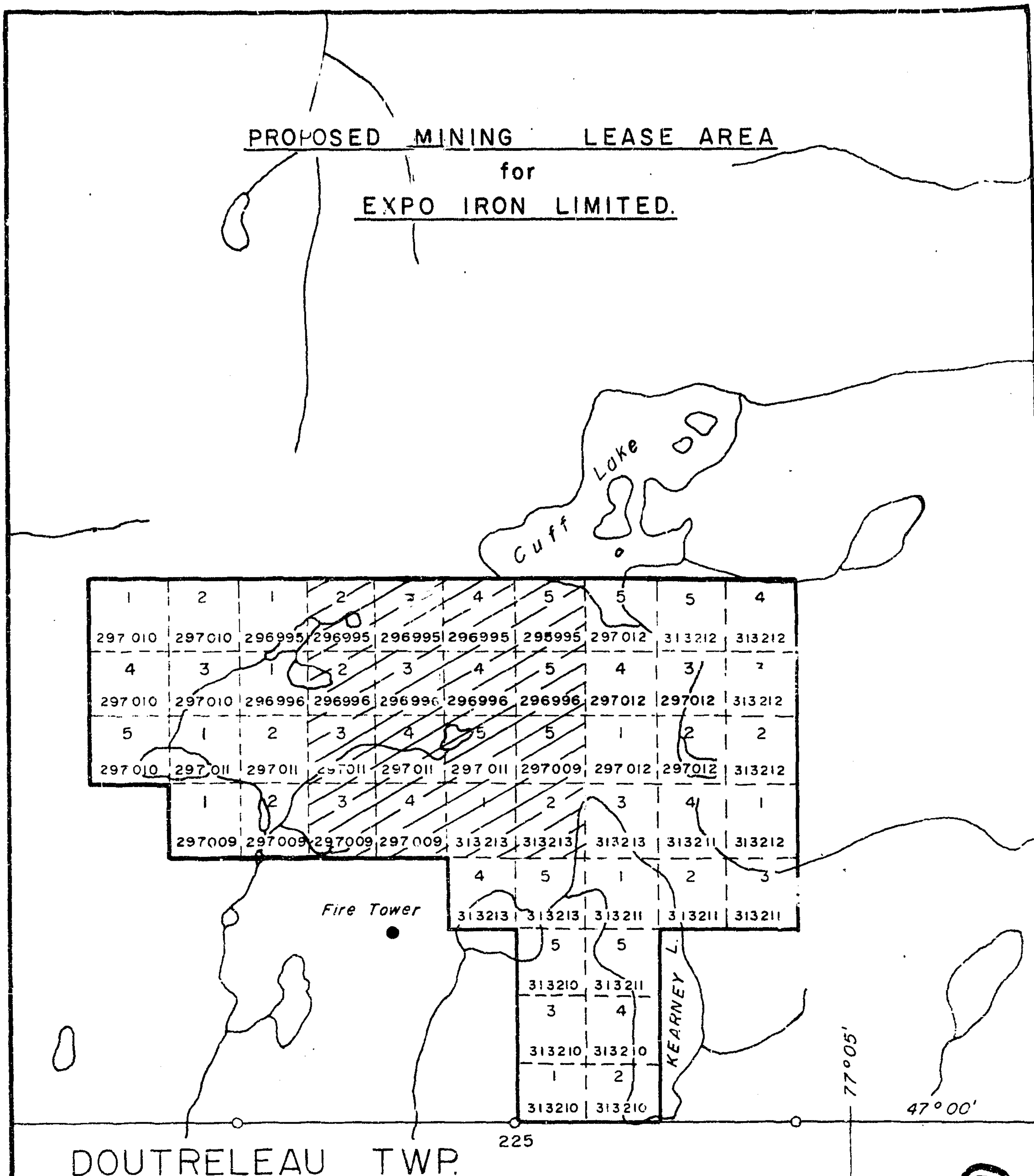
LOCATION MAP of EXPO IRON DEPOSIT

—HOUDET TOWNSHIP—
PONTIAC COUNTY, P.Q.

BY: E.D. BLACK | JOB No C-68 | 1 March 1971

1

PROPOSED MINING LEASE AREA
for
EXPO IRON LIMITED.



③

Orebody, 8 claims = – – – – – – 320 acres.

Plant, waste & tailings, 8 claims = 320 acres.

640 acres.

LEASE AREA

metals, petroleum & hydraulic resources consulting limited

EXPO IRON LIMITED

PROPERTY CLAIM MAP

—HOUDET TOWNSHIP—

PONTIAC COUNTY — P.Q.

BY: E.D. BLACK	JOB Nº	
SCALE : 1" = 2640'	C-68	MARCH 29/71