GM 32215

PRELIMINARY EVALUATION, QUECO PROPERTY

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PRELIMINARY EVALUATION

QUECO PROPERTY

QUEBEC COBALT & EXPLORATION LIMITED
QUEBEC, CANADA

FOR

REPUBLIC STEEL CORPORATION

March 1976

H. E. NEAL & ASSOCIATES LTD.
MINERAL CONSULTANTS
TORONTO - CANADA

Ministère des Richesses Naturelles, Québec
SERVICE DE LA
DOCUMENTATION TECHNIQUE
Date: 25 OCT 1976
No GM: 32215

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QUECO PROPERTY

Specularite Ore in Foreground Looking South-West

to Mt. Wright and Hesse Lake 6 miles in Background.



View of Terrain Queco Property Area



Quebec Cartier Mining - Initial Mining in Foreground, Mill Under Construction in Middle and Queco Property in Background.

1.0 SUMMARY

1.1 This Preliminary Evaluation Of The Queco Property has indicated the existence of reserves of about 700 million tons of magnetite-specularite in the Bloom Lake deposit containing 30% SolFe sufficient to produce 5,000,000 tons per year of concentrate containing 67% SolFe for 50 years at a projected concentration ratio of 2.78.

1.2 Three additional smaller deposits occur within 6 miles of the Bloom Lake deposit.

1.3 A preliminary pit design has outlined 30 years of reserves and metallurgical testwork using over 5,500 feet of drill core has confirmed that saleable gravity concentrates can be produced at -35 mesh, supplemented by magnetite recovery from the gravity tailings.

1.4 Super concentrates can also be produced containing less than 1.5% Silica.

1.5 This study includes the general layout of facilities for a mine and concentrator located 6 miles northeast of the Mt. Wright operations of Quebec Cartier Mining Company.

1.6 The development of the Queco Property is dependent on the use of existing facilities such as an extension of the Fermont townsite, the Quebec Cartier Mining Company's railway to Port Cartier and the highway from Mt. Wright to Labrador City. Consideration of these facilities are outside the scope of this study.

1.7 It is assumed that a pellet plant would be located in the Port Cartier area with concentrates being shipped from the property via conventional ore cars.

1.8 Ore Reserves

	Crude Grade % SolFe	Crude Tons	Waste Tons	Stripping Ratio
Bloom Lake				
Main Pit Zone	30.1	435	288	0.66
Western Extension	30.1	+280	Uncertain	
3 other deposits	24.8	69	52.6	0.76

1.9 A mine design is presented using a conventional open pit operation with trucks, shovels and rotary drills, mining to a maximum depth of 1,000 feet below the present surface. This preliminary design was prepared by Dames & Moore - See Appendix A.

1.10 Metallurgical testwork conducted by Jones & Laughlin, Colorado School of Mines and more extensively by Lakefield Research provide similar and confirmatory results that the Bloom Lake deposit is liberated at -35 mesh to produce a gravity concentrate containing 67% SolFe.

1.11 Projected Concentrate Analyses - from Lakefield Research testwork on 21 composites.

Element	Gravity Concentrate	Super Concentrate A	Super Concentrate B	Super Concentrate C
% SolFe	67.4	71.7	69.1	69.8
% SiO_2	3.57	1.20	1.46	1.39
% Al_2O_3	0.20	0.10	0.21	0.18
% CaO	0.25			
% MgO	0.29			
% TiO_2	0.11			
% Mn	0.051			
% P	0.015			
% S	0.013			
% Na_2O	0.013			
% K_2O	0.007			

Gravity Concentrate reground to 78% -325 Mesh to produce the Super-Concentrate.

A = low intensity magnetic separation

B = high intensity magnetic separation (Jones)

C = A + B

1.12 Metallurgical Results

Projected Balance:	Crude	30.1% SolFe
	Concentrate	67% SolFe
	% SolFe Recovery	80%
	Concentration Ratio	2.78

a) Tabling and Magnetic Separation Results

(Based on 5503 feet of Drill Core)

Description	Head		Table + Magnetic Concentrate			
	% SolFe	% MagFe	Conc. Ratio	% Wt	% SolFe	% Dist SolFe
Main Pit Zone	30.3	13.0	2.4	41.0	66.6	90.3
Western Extension	31.4	0.7	2.6	38.1	67.9	82.6
				(No Mag. Separation)		
Total						
Bloom Deposit	30.8	7.4	2.5	39.7	67.2	86.7
Magnetite-Silicate Waste	17.2	13.5	4.3	23.3	67.5	91.4

The Magnetite-Silicate waste was represented by four composites of total footage of 826 feet which are outside of the reserve estimate and now might be partially considered as potential mill feed.

b) Magnetic cobbing of the gravity tailings recovered additional iron units included in the above table. The cobber concentrate requires grinding to 90 to 97% -400 mesh, producing a magnetic concentrate

b) (continued)

with 69% Fe. Magnetic cobbing on the average added 3% weight and 7% Fe Recovery to the gravity concentrate based on the drill core tested. Some material contains a negligible amount of magnetite while other portions contain up to 35% Magnetic Iron.

c) Wet High Intensity Magnetic Separation of the low intensity tailings produced low-grade concentrates at -100 Mesh and only 60% Fe at -200 Mesh. More work will be required in a pilot plant program before additional iron recovery can be projected. The actual plant tailings may be higher in iron which would make this recovery more practical.

1.13 General Facilities

The initial mining is projected for the Main Pit Zone - the eastern portion of the Bloom Lake Deposit with the crude ore hauled to a plant site towards the north along the northwest side of Golden Lake

A railway spur of about 7 miles in length would connect from the Quebec Cartier Railway to the concentrator site crossing ground Withdrawn From Staking and Queco claims. This area is relatively flat with adequate construction material.

Tailings would initially be stored in the Carrot Lake basin behind the causeway formed by the railway. Future tailings area lies between Golden Lake and Boulder Lake on Queco claims.

An access road of 5.5 miles would connect the Mt. Wright - Labrador City highway with the Concentrator area.

1.14 Additional Work Required Prior To Feasibility Study include the following:

a) Assessment work on Roach Hill Extension - $65,000 for 2000 to 2500 feet of drilling.

b) Drilling - Main Zone: 20,000 feet to verify and outline ore limits

Dump Areas: 5,000 feet

Western Extension: 12,500 feet

c) Bulk Sampling of 300 tons from Main Pit Zone.

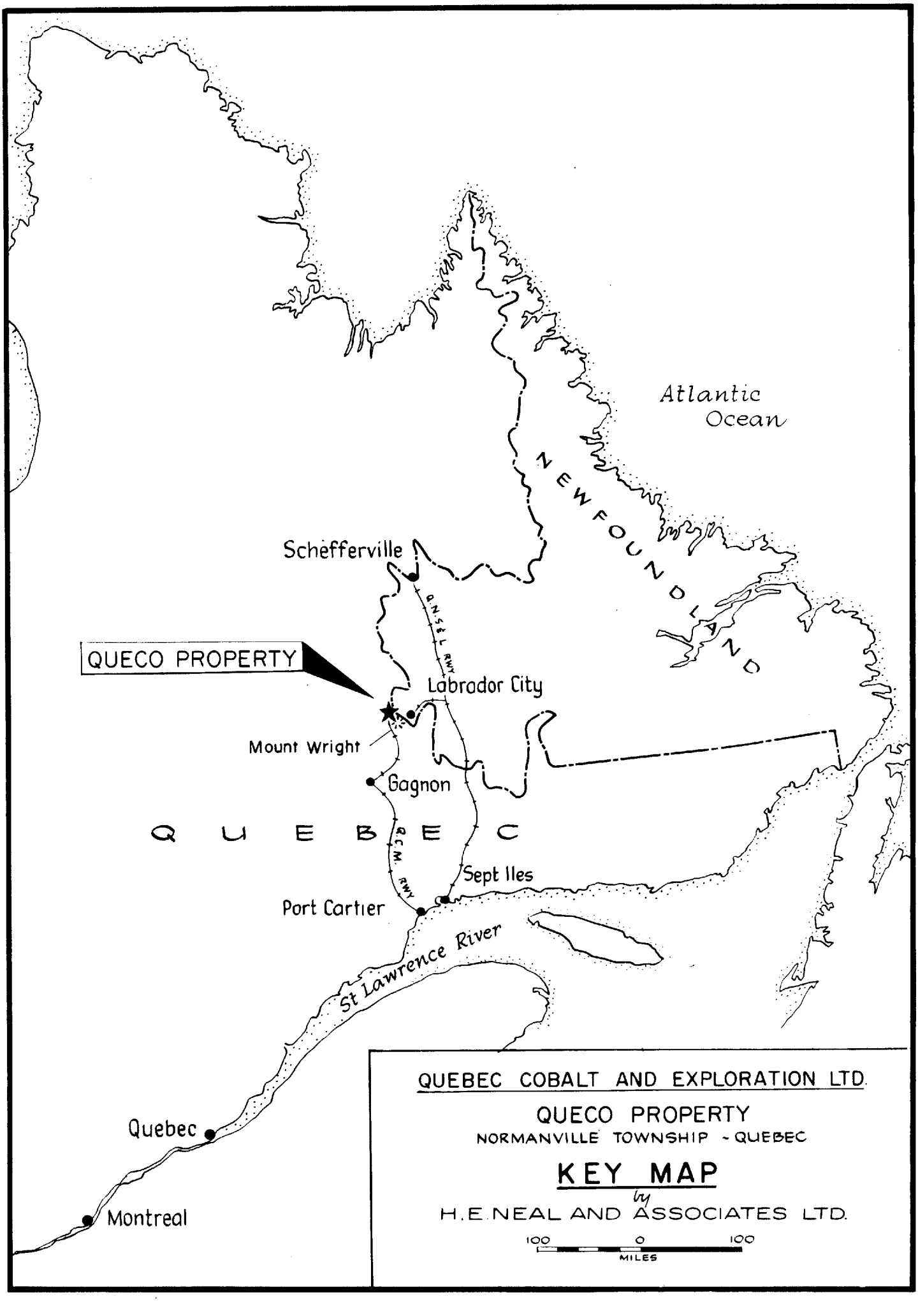
d) Pilot Plant Testing of 300 ton Bulk Sample.

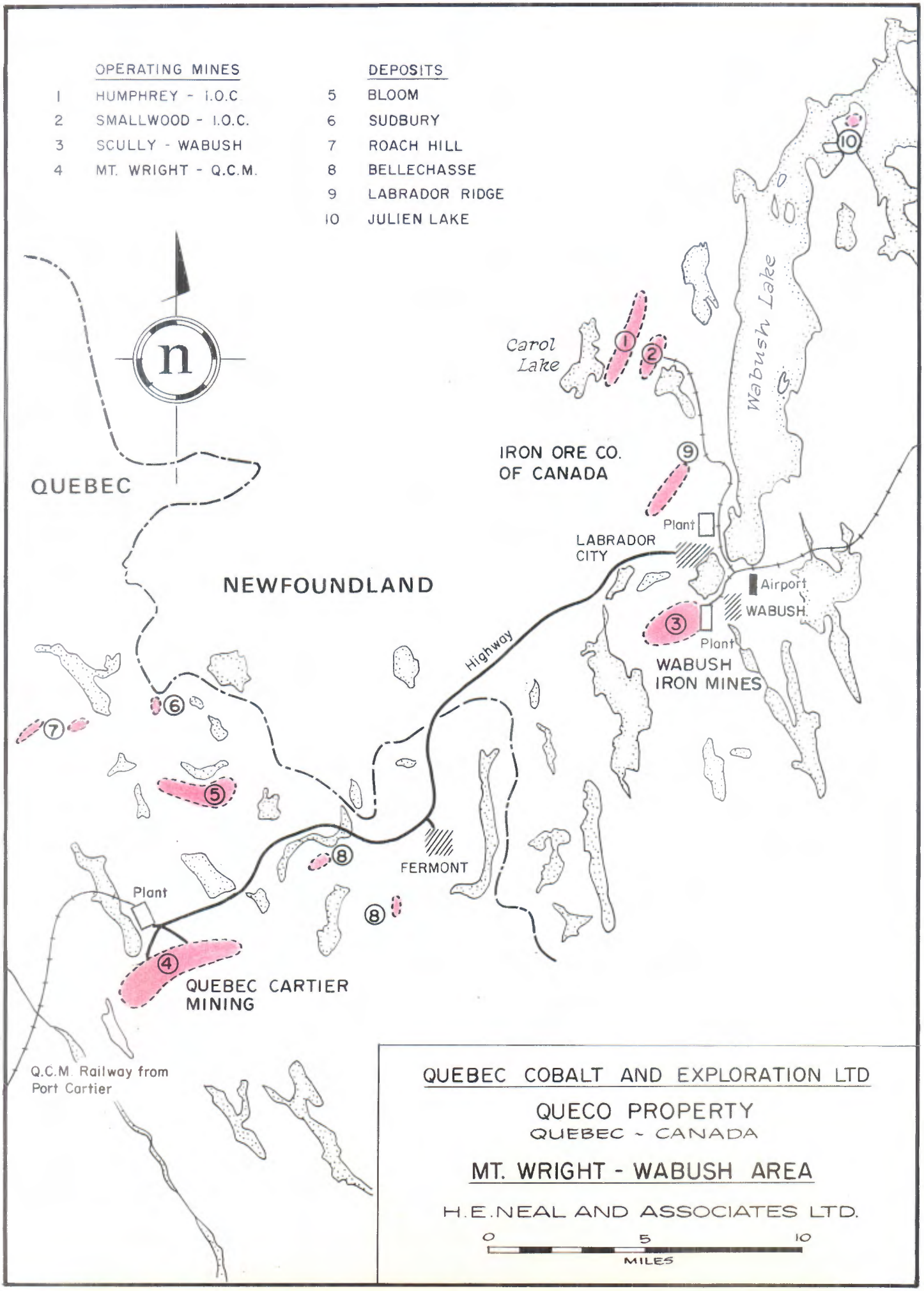
e) Environmental baseline studies.

f) Hydrological studies - water flows, depth of lakes, ground water, stream diversions, etc.

g) Road Construction to Crescent Lake camp prior to initial drill program.

The following maps provide reference for Property Location, Mt.Wright-Wabush Area and Existing and Projected Regional Facilities.


Atlantic Ocean
NEWFOUNDLAND
Schefferville
Q.N.S.&L. RWY.
QUECO PROPERTY
Labrador City
Mount Wright
Gagnon
QUEBEC
P.C.N. RWY.
Sept Iles
Port Cartier
St Lawrence River
Quebec
Montreal
QUEBEC COBALT AND EXPLORATION LTD.
QUECO PROPERTY
NORMANVILLE TOWNSHIP - QUEBEC
KEY MAP
by
H.E. NEAL AND ASSOCIATES LTD.
100 0 100
MILES


OPERATING MINES
1 HUMPHREY - I.O.C
2 SMALLWOOD - I.O.C.
3 SCULLY - WABUSH
4 MT. WRIGHT - Q.C.M.
DEPOSITS
5 BLOOM
6 SUDBURY
7 ROACH HILL
8 BELLECHASSE
9 LABRADOR RIDGE
10 JULIEN LAKE
n
QUEBEC
NEWFOUNDLAND
Carol Lake
Wabush Lake
IRON ORE CO. OF CANADA
Plant
LABRADOR CITY
Airport
WABUSH
Plant
WABUSH IRON MINES
Highway
FERMONT
Plant
QUEBEC CARTIER MINING
Q.C.M. Railway from Port Cartier
QUEBEC COBALT AND EXPLORATION LTD
QUECO PROPERTY
QUEBEC - CANADA
MT. WRIGHT - WABUSH AREA
H.E.NEAL AND ASSOCIATES LTD.
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2.0 INTRODUCTION

H.E. Neal & Associates were requested to prepare a Preliminary Evaluation of the Queco Property by Republic Steel Corporation. This evaluation was to include:

1) Preparation of new geological maps using the 1974 revised topography prepared from new aerial photography and typographic mapping at a scale of 1" = 400 ft. with 10 ft. contours.
2) Revision of the geological cross section to conform to the new contoured maps.
3) Estimation of ore reserves and grade of the first 30 years supply and plant feed.
4) Preliminary mine design including the pit outline, dump area, access roads and railway spur from Mt. Wright and a possible plant site.
5) Tailings disposal areas and Environmental considerations.
6) Metallurgical testing of drill core samples from the 1971-1972 drilling program.
7) Outline of Future Work Requirements.
8) Summary of work conducted on the Queco Property.

H.E. Neal & Associates retained the services of Dames and Moore to prepare the mine design and reserve estimate. All the necessary

geological maps and typographical maps and sections were provided by H.E. Neal & Associates. Close liaison was maintained with Dames and Moore who were assisted by Mr. F.J. Alexander, a senior geologist with H.E. Neal & Associates.

Lakefield Research of Canada conducted all batch metallurgical testwork consisting of heavy liquid analyses, chemical analyses, table tests, low intensity and high intensity magnetic separation testwork. Twenty-one composites were tested representing 5503 feet of drill core. This testwork was outlined and supervised by H.E. Neal & Associates.

Appendix A - contains the Dames & Moore Report "Preliminary Open Pit Design of Quebec Cobalt and Explorations Limited's Bloom Lake Property".

Appendix B - contains the Lakefield Research Report - Progress Report No. 1 of February 10, 1976.

3.0 THE PROPERTY

The property of Quebec Cobalt & Exploration Limited is located about 580 miles northeast of Montreal, Quebec at the southwestern corner of the iron range of the Labrador Trough. See Kep Map. Labrador City with close to 15,000 people is 16 miles northeast and Fermont with 5,000 people is 8 miles east of the Queco property. A highway connecting the Quebec Cartier Mining operations and Labrador City runs 3 miles south of the Bloom Lake deposit. The Map of the Mt. Wright-Wabush Area shows the relation of the 3 mining operations, the three towns and other deposits in the area.

The accompanying Map of Existing and Projected Required Facilities illustrates the relation of the required roads, railway, open pit operation and plant site to the available services. The Area Withdrawn From Staking illustrates the protection provided the Quebec Cartier Mining operations and the town of Fermont by the Province of Quebec.

Quebec Cobalt and Exploration Limited property consists of 271 claims in seven groups of claims as follows:

Bloom Lake	97 claims
Roach Hill	22 claims
Sudbury Hill	6 claims
Labrador Hill	6 claims
Golden Lake	129 claims
South Golden Lake	10 claims
Mogridge Lake	1 claim
	271 claims

The accompanying Map of Claims shows the various groups and a second Map of Claims includes the actual claim number of each of the claims.

These claims are in Normanville and Number 2855 Townships in the County of Saguenay, Quebec.

The status of these claims was checked in the Mining Division of the Department of Natural Resources in Quebec City in February 1976. All claims are currently in good standing until the period June-July 1976. All groups except Labrador Hill are in good standing with sufficient work credits until June-July 1977. The Bloom Lake Group has sufficient credits for more than 15 years although further work will be required on the Roach, Sudbury, Golden and Mogridge Groups to keep them in the future.

4.0 HISTORICAL BACKGROUND

Quebec Cobalt and Exploration Limited was incorporated in February 1952 to acquire claims held by Sursho Mining Corporation. In 1951 the Walsh brothers staked claims on a cobalt showing for Mr. Bill Crawford of Sursho. Quebec Cobalt and Exploration Limited initally optioned one million shares of Sursho and gradually accumulated more shares until full control was obtained in February 1956. The following money was advanced in 1952 to Quebec Cobalt for their field program:

$10,200 by Thompson Landmark Gold Mines Limited.

$28,000 by Arnold Hoffman

$56,000 by Thayer Lindsley.

In 1952 Quebec Cobalt acquired 50 claims from Sursho covering the area from Bloom Lake to Crescent Lake and north to Confusion Lake. Twenty-four claims were staked by Quebec Cobalt during the spring of 1952 covering additional ground west of Crescent Lake and south around Fold Lake. Wilton et Al are indicated as staking 19 claims south and east of Bloom Lake in 1952.

In 1952, a crew of six prospectors including James and Michael Walsh started field work on April 22 under the supervision of Mr. K.M. Brown. The purpose of the program was to sample the cobalt showing and to prospect in the Mt. Wright - Carheil Lake area. In June 1952,

Mr. R. Cunningham, a mining geologist with Quebec Metallurgical Industries, arrived at Bloom Lake to map the various discoveries. An X-Ray drill was sent to the property to assist in the bulk sampling of the cobalt showing. The results of the cobalt sampling were disappointing. Several zones of magnetite-hematite iron formation were outlined and sampled in 1952 in the area between Bloom Lake and Crescent Lake. Mr. R.D. Hoffman and Mr. F.W. Thompson actively participated and encouraged the acquiring of additional claims covering the iron formation.

Mr. Cunningham returned in 1953 but little is known of the work during this season except it was disrupted by the Walsh brothers attempting to travel by the Moisie River to Sept Iles when a supply plane was overdue after break-up. They were subsequently found in the late summer but they suffered from lack of food and were unable to travel the Moisie River. Additional new deposits were outlined in 1953.

In 1954, Mr. R. Cunningham spent the period of June 15 to September 15 in the Bloom Lake area with a crew of 5 men. They mapped the Bloom Lake area, cut picket lines and made a dip-needle survey of the main iron showings. They moved to Boulder Lake and did work on a group of claims previously staked. Additional claims were staked west of the main group towards Carrot Lake. R.D. Hoffman spent the period of

August 14 to September 15, 1954 on the property accompanied by his long-time prospecting associate, Fred W. Thompson.

In 1955 Jones and Laughlin started the first of several field programs after obtaining a lease from Quebec Cobalt and Exploration Limited. Cleveland Cliffs joined Jones and Laughlin on an equal participation. This lease was terminated in 1968.

In 1971 and 1972, Quebec Cobalt conducted a drilling program, mapping, gravity and magnetometer survey. In 1973, 139 additional claims were staked between the Bloom, Boulder and Sudbury groups. In 1974, assessment work was conducted on the 139 newly staked claims as well as the Roach, Labrador and Sudbury groups in order to keep the ground in good standing.

In 1973, Republic Steel Corporation obtained an option on the properties held by Quebec Cobalt and Exploration Limited.

5.0 SUMMARY OF WORK ON THE PROPERTY

5.1 Bloom Lake Deposit

Diamond Drilling

Year	Footage Standard		X-Ray		
1972	12,580	(15 holes)			wire-line and mud drilling
1971	5,562	(9 holes)			wire-line drilling
1966			1,767.5	(26 holes)	
1957	12,322	(23 holes)	1,243	(10 holes)	
1956	3,323	(7 holes)	220	(1 hole)	
TOTAL	33,787ft	54 holes	3,230.5ft	37 holes	

Geological Surveys

1975 - Replotting of all geological data on new base maps and revision of cross sections.

1974 - Golden Lake and Golden Lake South Areas - Republic

1972 - East Central Area - Queco

1971 - Triangle - Carrot Lake Area - Queco

1956 - R.F. Mueller - Goalpost Hill. Detailed Geological Mapping for Jones and Laughlin.

1952 - 1954 - Triangle - Bloom Lake Area - Cunningham - Queco.

Magnetometer Surveys

1974 - Golden Lake and Golden Lake South Areas - 49.75 Line Miles

1972 - Bloom Lake Group - 14.81 Line Miles

1971 - Bloom Lake Group - 9.70 Line Miles

1965-66 Triangle - Crescent Lake Area - 41.6 Line Miles (J & L)

1953-54 Triangle - Bloom Lake Area (dip needle) - 67.94 Line Miles (Queco)

5.1 (continued)

Gravity Surveys and Level Surveys

1972 - Bloom Lake Group - 16.97 Line Miles

1971 - Bloom Lake Group - 11.12 Line Miles

Topographic Survey

1972 - Bloom Lake Group - 8.62 Line Miles
for location of cross section and drill holes.

Linecutting

1974 - Golden Lake and Golden Lake South Areas - 49.75 Line Miles - Republic

1972 - Bloom Lake Group - 16.02 Line Miles - Queco

1971 - Bloom Lake Group - 17.71 Line Miles - Queco

1952-54 - Triangle - Bloom Lake - 67.94 Line Miles - Queco

Aerial Photography and Topographic Mapping

1974 - Topographic Mapping - 1" = 400' scale 10' contours providing new base maps for 11,340 acres.

1973 - Aerial Photography and Aerotrianguration - 32 square miles.

Metallurgic Testwork

1956-58 - Jones and Laughlin.
Bulk Samples 1-7, 10 Zone A Goldpost Hill
Bulk Samples 8, 9 Zone B Drill Core Holes QC- 1-9, 11, 15, 16, 17, 18.
Liberation, magnetic tests, roasting, spirals, flotation and high tension separation testwork.

1972-73 - Colorado School of Mines Research Institute.
Characterization tests, largely consisting of Heavy Liquid tests on 7 samples from 5 drill core saves, collected by C.E. Harpur from J & L.

1975-76 - Lakefield Research of Canada.
Liberation, table tests, magnetic testing and high intensity tests of table tailings and 21 composite samples from 1971-1972 drilling.

5.2 Roach Hill and Roach Hill Extension

Diamond Drilling

1960	4503 feet	(10 holes Roach Hill Extension, 2 holes Roach Hill)

Magnetometer Surveys

1974 - West End of Extension	- 3.64 Line Miles	- Republic
1972 - Roach Hill and Roach Hill Extension East End and Central Sayers and Wingfield	- 14.7 Line Miles	- Queco
1960 - Roach Hill and Roach Hill Extension Superdip - Boulder	- 6.0 Line Miles	
1958 - Roach Hill Extension - W. Kangas Dip Needle		- J & L
1954 - Roach Hill - Dip Needle	- 25.08 Line Miles	- Queco

Geological Surveys

1974 - Roach Hill and Roach Hill Extension H.E. Neal & Associates Ltd for	- Republic
1954 - Roach Hill and Roach Hill Extension	- Queco

Linecutting

1974 - West End of Extension	- 4.02 Line Miles	- Republic
1972 - Flagged	- 14.7 Line Miles	- Queco

Sampling

1974 - 10 samples Roach Hill 17 samples Roach Hill Extension	- Republic

Metallurgical Testwork

1957 - Magnetic Separation tests on 2 small bulk samples from outcrops on Roach Hill Extension. Head grades 32-36% Fe as compared to average from drill holes of 24.0% Fe.

5.3 Sudbury Hill

Diamond Drilling

1960 930 feet in 4 holes - Jones and Laughlin

Magnetometer Surveys

1972 - Seyers and Wingfield - 1.9 Line Miles - Queco
Dip Needle Survey
L.K. Oliver of Jones and Laughlin.

Geological Surveys

1974 - F.J. Alexander - Republic

1957 - L.K. Oliver of Jones and Laughlin

Sampling

1974 - 16 samples - Republic

Metallurgical Testwork

None reported

5.4 Labrador Hill

Magnetometer Surveys

1974 - - 3.8 Line Miles - Republic

1960 - Dip Needle - Jones and Laughlin

Geological Surveys

1974 - F.J. Alexander - Republic

1957 - J.J. Drury and R.D. Hoffman

Sampling

1974 - 10 grab samples - Republic

Metallurgical Testwork

None reported.

6.0 GEOLOGY

6.1 General Geology

The Bloom Lake specularite iron formation forms part of a highly folded and metamorphosed southwestern branch of the Labrador Trough. It is part of an extension of the Carol Lake - Wabush Lake iron range which runs southwesterly to the Mt. Wright area. Further southwest the metamorphosed sediments become less continuous and contorted remnants in the older granite greiss are found for at least 135 miles to Lac Pletipi.

Four deposits occur within the Queco Property consisting of the Bloom Lake deposit which is divided into an eastern Main Pit Zone and a Western Extension, the Sudbury Hill deposit, 3 miles to the north, the Roach Hill deposit, 3 miles to the northwest and the Roach Hill Extension located 4 miles to the northwest.

The iron formation in this area has been metamorphosed and recrystallized to form coarse-grained quartz specularite and quartz iron silicate members. The magnetite content of the Bloom Lake deposit is normally low, averaging just over 10% magnetic iron. The central and western parts contain large tonnage with less than 1% magnetic iron but the content increases in the eastern part of the Main Pit Zone. The Grenville Front forms a major fault zone immediately north of the property and extends for many miles in a northeast-southwest direction.

6.1 (continued)

North of Wabush Lake, this front divides the metamorphosed iron formation to the south from the less altered cherty iron formation extending northward into the Knob Lake area.

Two distinct iron formation bands occur within the Bloom Lake deposit and a third band may be present in the Triangle Lake area as a ferruginous band in the quartzite. In the Western Extension, the iron formation forms a broad doubly plunging syncline with a capping of a gabbro sill. An anticlinal fold in the central part of the deposit forms a shallower arch. The Main Pit Zone is a broad syncline plunging to the east with some crumpling to form an anticlinal fold north of Crescent Lake which brings the iron formation to the surface as an island in the gabbro. This syncline is capped in the central part by gabbro which thickens to the east. To the east the specularite-magnetite facies become thinner and they are interfingered with a quartz-magnetite silicate iron formation. Two north-south faults occur in the eastern section with an associated series of tight folding.

6.2 Stratigraphic Column

The 1971 and 1972 geological surveys on the Bloom Lake deposit along with the drilling provided the following stratigraphic sequence:

6.2. (continued)

Glacial Deposits and Unconsolidated Surface Material

Gabbro, Gabbro-Amphibolite

(Iron Formation)

Quartzite

Biotite Schist

(Quartzite)

Gabbro-Amphibolite

Upper Iron Formation

(Quartzite)

Biotite Schist

Gabbro-Amphibolite

Lower Iron Formation

Basal Schist and Gneiss

( ) denotes discontinuous

6.3 Economic Geology

The oxide iron formation is the main economic unit. The specularite (specular hematite) is similar to that being mined at Mt. Wright by the Quebec Cartier Mining Company at a designed capacity of 16,000,000 tons per year of spiral concentrate.

The specularite is liberated at 35 mesh on the average, with coarser

sections liberated at 10 mesh as shown from heavy liquid separation testwork. The magnetite content is variable from less than 1% to 35%. The higher magnetite material with over 10% magnetic iron is called magnetite-specularite. The magnetite recovery by gravity separation as demonstrated by the Wilfley shaking table is only slightly lower than the specularite. The remaining magnetite reporting in the gravity tailings occurs as finely disseminated grains in larger quartz grains. Cobbing of the magnetic portion from the tailings requires grinding to 90-95% -400 mesh in order to produce magnetic concentrate with 67-70% Fe. The fine hematite middlings are still unliberated at -200 mesh as determined by wet high intensity magnetic separation tests producing a concentrate with 60% Fe. The magnetic content increases towards the eastern part of the Main Pit Zone and in some cases with depth where the rock becomes harder and slightly more fine grained in texture.

The oxide facies is associated with a silicate iron formation which increases towards the eastern part of the Main Pit Zone. The silicate formation consists primarily of well crystallized grunerite-cummingtonite, actinolite, tremolite assemblance. In the eastern part, well crystallized magnetite occurs in the silicate and may form a significant part of the rock.

Recent testwork has shown that the majority of the magnetite can be recovered by gravity separation producing high-grade table concentrates

at -35 mesh. Soluble iron recoveries of 60 - 88% are obtained by gravity with an additional 15 - 20% soluble iron recovery by regrinding of the magnetic tailings.

The silicate iron formation which was considered to be waste should now be re-assessed as potential plant feed. Four Composites, 11, 16, 17 and 18 were tested with the crude soluble iron in the range of 12% to 22% and produced combined table + magnetic concentrates with 66 - 70% soluble iron with weight recoveries from 13.6 to 31%. Some of the higher iron content silicate iron formation removed as stripping or interbedded with the oxide facies can be considered as mill feed thus reducing the stripping ratio and increasing the ore tonnage.

6.3 The accompanying Geological Cross Sections at a scale of 1" = 200 feet, illustrate the outline of the deposit in the Main Pit Zone and the need for more drilling to determine the ore limits. Sections 56+00W, 74+00W and 10+00 SW follows. Pit outlines are included on 4 Cross Sections reproduced by Dames & Moore and located in the pocket at the end of the report. These Sections are 66+00W, 0+00, QC-15 and a Longitudinal Section east of 90+00W.

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7.0 ORE RESERVES

A recalculation of the crude ore reserves for the Bloom Lake, Roach Hill, Roach Hill Extension and Sudbury Hill was prepared by Dames and Moore.

The level of drilling and geological data are quite variable within these deposits. The area east of Section 90+00W of the Bloom Lake deposit provides the most reliable basis for tonnage and grade estimation although the drill sections are up to 1000 feet apart with incomplete drilling. This area was designated as Main Pit Zone representing the most reliable and accessible tonnage for at least 30 years of operation at the rate of 5,000,000 tons per year of concentrate or 14 million tons of mill feed per year.

Mineable Reserves - Crude Ore

In Millions Of Long Tons

Deposit	Crude Grade % Sol Fe	Crude Tons	Waste Tons	Stripping Ratio Waste : Ore
Bloom Lake				
Main Pit Zone	30.1	435	288	0.66
Western Extension	30.1	+280	uncertain	
Sudbury Hill	25.7	4.5	0	0
Roach Hill	28.1	3.3	0	0
Roach Hill Extension	24.5	61	52.6	1.1

The Stripping Ratio in the Main Pit Zone is projected to be about 0.1 for years 1 to 5 due to initially mining of Goalpost Hill - the high part of the deposit.

The Stripping Ratio of the Western Extension will be higher than for the Main Pit Zone due to gabbro capping in the Central area.

7.1 Grade Estimation

The Average Grade was calculated by plotting the Soluble Iron distribution of over 1400 drill hole samples. This average grade was determined to be 28.4% Soluble Iron using all samples from 0% Fe. Of these samples 1300 were over 15% Sol Fe giving an average of 30.1% Soluble Fe. The 15% Fe cut-off was selected arbitrarily and had been used in the 1972 Tonnage Estimate by H.E. Neal. The plus 15% Fe represented 91.5% of the samples which was regarded as suitable plant feed. The remaining 8.5% was included in the waste stripping. The accompanying histogram and smooth curve diagrams illustrate these data.

Soluble Iron analyses have always been used since they represent the iron content of the oxide minerals (magnetite-hematite) which are recoverable and does not include the silicate minerals which contain considerable iron but are not considered to be recoverable in a conventional plant.

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7.2 Tonnage Estimation

Estimates were based on the following:

a) Cut-off grade of 15% soluble Fe.

b) Tonnage Factors: Plant Feed: 11 cubic feet/LT

Waste: 12.5 cubic feet/LT

c) The Main Pit Zone from Sections 90 to 61 West were given an influence of halfway between sections.

d) Iron analyses were used on the histogram plot rather than averaging each section due to widely scattered drill holes.

e) East of Section 61W, the tonnage was calculated using level plans at 200 foot vertical intervals. This was necessary due to the irregular outline of the ore body and the non parallel cross sections.

f) Additional tonnage is readily available by extending the pit about 1000 feet west on what was the former B. Zone.

7.3 Previous Tonnage Estimate

The Dames & Moore grade and tonnage estimate compares favourably to the 1972 estimates by H.E. Neal.and Associates with 502 million LT for the A, B, C Zones covering slightly more area than the Main Pit Zone of Dames & Moore. The grade calculated for this tonnage was 29.5% Soluble Iron using cross-sections in the west half and level plans in the eastern half. This grade is considered to be statistically similar to the 30.1% Sol Fe. calculated by Dames & Moore.

7.3 (continued)

The previous tonnage estimate used a lower stripping ratio and for the Western Extension a separate pit was used at the Carrot Lake area whereas Dames & Moore's estimate included one large pit across the entire length of the Bloom Lake dropout which requires additional stripping after 30 years of operation. This approach is worthwhile since at that time equipment designs may radically lower the mining costs.

The Carrot Lake Zone was estimated to contain an additional 184 million tons of probable and possible mill feed at an average grade of 30.5% SolFe based on widely spaced cross sections. This tonnage was estimated for a separate pit covering the western end of the Bloom Lake deposit and had a Stripping Ratio of 0.55.

8.0 MINING

8.1 General Statement

A review of the various deposits indicates that the most favourable open pit for the initial operation should be the eastern half of the Bloom Lake deposit. A preliminary pit has been designed to include reserves of 436 million long tons of plant feed at a grade of 30.1% Soluble Iron. This pit will be about 5000 feet long in an east-west direction and 6000 feet wide at the eastern end. Mining is proposed to a depth of 1000 feet below the present surface.

These reserves are sufficient to produce 5 million tons per year of concentrate at a concentration ratio of 2.78 based on 80% soluble iron recovery and a 67% Fe concentrate. Additional tonnage could readily be incorporated into this open pit by extending the pit limits about 1500 feet west of section 90+00 W to incorporate most of the original B Zone with only a modest amount of stripping. The present pit design is not intended to be an ultimate design which can only be made after more development drilling has been completed.

The remainder of the Bloom Lake deposit with an additional 25 years of reserves in the Western Extension did not have sufficient evidence for a detailed pit design due to the wide spacing of the drilling. This extension will have a higher stripping ratio if the Main Pit Zone is extended to the western extremity. This will necessitate the removal of an extensive capping of gabbro.

The Stripping Ratio for the Main Pit Zone is 0.66 tons of waste per ton of plant feed. The first five years of operation will have a stripping ratio of about 0.1.

Mining of this deposit is projected to be conventional truck and shovel operation using large diameter rotary drills with 40 - 50 foot benches.

The addendum in the Dames & Moore report contains further data on the location, tonnage and grade of the outlying deposits.

8.2 Mine Design

The design by Dames & Moore is detailed in Appendix A of this report.

The design is based on -

a) Maximum pit slope of 50°.

b) Minimum pit bottom of 200 feet.

c) Mining only the magnetite and specularite iron formation with a cut-off grade of 15% soluble iron.

d) Cut off stripping ratio of 4.5 tons of waste for plant feed at the bottom of the pit.

e) Maximum grade of ultimate pit roads of 8%.

The cut-off stripping ratio was based on Direct Operating Costs of $6.55/ton of crude plant feed, without taxes, townsite or amortization.

ASSUMED DIRECT OPERATING COSTS

FOR STRIPPING RATIO CALCULATIONS

Concentration Ratio of 2.78

	$/ton Crude	$/ton Product
Mining	0.90	2.50
Concentration	1.00	2.78
Pelletizing	1.19	3.30
Overhead	.50	1.39
Handling Charge at Port	.18	.50
Transportation by Rail (assumed)	1.80	5.00
Assumed Royalty	.27	.75
	$ 5.84	$ 16.22
12% Contingency	.70	1.95
Cost Per Ton Crude	$ 6.54	$ 18.17

Value of Crude Plant Feed:

Assumed Pellet Grade: 65.5 % Fe

Price per iron Unit: $ .45 F.O.B. Port Cartier

Gross Value of Pellet: $ 29.48/LT

With Concentration Ratio of 2.78

Value of Plant Feed in place: $ 10.60/LT

Stripping Cost: $ .90/LT

$$\text{Cut-Off Stripping Ratio} = \frac{\text{Value in place/ton - Direct Cost/ton}}{\text{Stripping cost/ton}}$$

$$= \frac{\$10.60 - \$6.55}{\$.90} = 4.5$$

Many different designs are possible with different reserves and stripping ratios depending on the variation of the above factors.

See Pocket At End Of Report for:

Plate 1	Bloom Lake - Ultimate Pit	1" = 400'
1A	Cross Section Along Baseline	1" = 200'
1B	Cross Section 66+00W	1" = 200'
1C	Cross Section QC-15	1" = 200'
1D	Cross Section 0+00NE	1" = 200'
Plate 2	Bloom Lake Deposit Preliminary Pit and Western Extension Showing Surface Facilities and Roads	1" = 800'

9.0 METALLURGY

9.1 General Statement

Metallurgical testwork has been conducted on a batch scale on drill core and 500 pound bulk samples from the Bloom Lake deposit during the period 1956-1975. No metallurgical work has been reported on samples from Roach Hill, Sudbury Hill or Labrador Hill. Two small bulk samples from Roach Hill Extension were tested by Jones & Laughlin for liberation by Davis Tube.

Jones and Laughlin conducted primarily batch tests during the period of 1956-1958 at their Negaunee Laboratory, Michigan. Testwork was based chiefly on the testing of the bulk samples consisting of liberation tests, magnetic tests with minor testwork on the use of spirals, flotation and magnetic roasting. High tension testwork was conducted at Negaunee and by Carpco. This illustrated that high tension separation showed merit in the +325 mesh size of the crude samples. Heavy liquid and Davis Tube tests were conducted on 14 drill core composites.

Mr. C.E. Harpur selected 7 samples from drill core saves from 5 holes drilled in the 1955-1956 period and stored at Negaunee. These samples were tested at the Colorado School of Mines Research Institute in the period of 1972-1973. The work consisted primarily of liberation and Davis Tube testing. These tests essentially confirmed the grade

9.1 (continued)

and liberation of these samples as determined by Jones and Laughlin.

In 1975 a program of laboratory testing was designed and supervised by H.E. Neal & Associates at Lakefield Research of Canada. Tests were conducted on 21 drill core composites representing 5503 feet of core obtained in the 1971-1972 drilling conducted by Quebec Cobalt and Exploration Limited on the Bloom Lake Deposit. Composites were prepared by drill holes using up to 350 feet of length and by rock types such as high specularite, specularite-magnetite and magnetite silicate. Testwork consisted of heavy liquid tests for liberation, table tests to demonstrate the grade of gravity concentrate, low and high intensity magnetic separation of the crude and tailings. Complete chemical analyses were made of normal gravity concentrates and of a super concentrate.

9.2 Jones and Laughlin Testwork

BULK SAMPLES

Samples of approximately 500 pounds each were taken in 1956; 8 from Zone A (Goldpost Hill); 2 from Zone B and 2 smaller samples from Roach Hill Extension.

Analyses of Bulk Samples

Bulk Sample	% Sol.Fe.	% Total Fe.	% Mag.Fe.	Liberation Size in Mesh based on H.L. at 3.5 Sp.Gr.
"A" ZONE				
1	31.60	32.67	3.0	14
2	31.75	33.13	11.71	14-20
3	30.22	31.29	7.52	14
4	18.18	19.33	4.90	14
5 Mag.Silicate	27.77	30.37	26.09	65-100
6	25.85	26.70	12.13	14-20
7	18.33	19.02	1.07	14
10				14
Comp. 1 - 7	27.92	29.30	9.73	
Comp. 1 - 7 & 10	28.45	30.14	9.75	14
"B" ZONE				
8	34.75	35.21	0.21	14
9	36.59	36.97	5.08	14-20
Roach Hill Extension				
1	31.84*		26.04	150 (a)
2	36.22*		34.02	100-150 (a)

* No designation of either % Sol.Fe. or % Total Fe. but % Fe.

(a) Davis Tube tests only.

The average grade of Bulk Samples 1-7 at 27.92 % Sol.Fe. is 2% below the grade of the East Pit (Zones A, B in part and C).

1. Heavy Liquid Separation, Centrifuge at S.G.3.5 on -14^{m-} and -65^{m-}

Sample Number	-14 Mesh							-65 Mesh						
	Sink					Float	Calc. Head	Sink					Float	Calc. Head
	% Wt.	% Fe	%SiO_2	%Fe.Rec.	C.R.	% Fe	% Fe	% Wt.	% Fe	%SiO_2	%Fe.Rec.	C.R.	% Fe	% Fe
"A" Zone														
BS 1	40.8	65.0	6.1	86.2	2.5	7.2	30.8	43.1	68.4	2.2	96.6	2.3	1.8	30.6
2	47.5	64.7	7.7	91.0	2.1	5.8	33.8	50.0	69.5	1.7	97.4	2.0	1.8	35.7
3	45.6	66.7	4.4	91.3	2.2	5.4	33.4	44.9	69.4	1.5	97.5	2.2	1.4	31.9
4	28.6	65.7	5.0	87.7	3.5	3.7	21.4	29.4	68.0	2.5	92.7	3.4	2.2	21.5
5	37.0	57.8	16.6	77.5	2.7	9.8	27.6	35.8	66.6	6.1	90.0	2.8	4.1	26.5
6	38.8	65.0	6.1	91.5	2.6	3.8	27.6	35.8	68.5	2.3	95.4	2.8	1.8	25.7
7	26.5	66.3	4.1	87.1	3.8	3.5	20.2	25.2	68.4	2.1	92.6	4.0	1.8	18.6
10	51.21	68.13	2.76	94.70	1.95	4.00	36.84							
1 - 7	39.19	64.82	6.68	89.77	2.55	4.76	28.30	37.02	68.97	2.04	95.66	2.70	1.84	26.69
	*33.20	66.59	4.66	80.87	3.01	7.83	27.34							
1-7&10	39.34	67.29	4.14	90.73	2.54	4.46	29.18							
"B" Zone														
BS 8	51.4	64.0	4.5	88.9	1.9	8.4	37.0	49.1	65.5	1.8	97.2	2.0	1.8	33.1
9	51.4	66.3	4.8	90.9	1.0	7.1	37.5	50.4	67.0	2.7	97.6	2.0	1.7	35.1

* Sp. Gravity 4.0

Davis Tube Tests By Screen Fractions of Composite B.S. 1 - 7

Mesh	Fraction % Wt.	% Fe.	Recovery on Mesh	Conc. FeDist.
10	66.81	37.02	81.3	9.2
14	57.87	34.10	69.4	9.6
20	52.67	36.86	66.2	9.4
28	38.37	38.55	57.0	9.5
35	27.95	40.70	47.5	12.2
48	18.62	44.93	30.8	15.5
65	14.70	50.23	23.3	9.3
100	13.45	56.83	21.2	8.5
150	15.15	59.90	24.0	5.8
200	17.50	61.75	29.4	3.3
325	19.83	64.97	37.4	3.4
-325	23.83	69.66	58.1	4.3

Heavy Liquid Separation (S.G.3.5) By Screen Fractions - BS 1 - 7

Mesh	% Wt	% Fe	% SiO_2	Recovery on Mesh
- ¼ + 0	36.54	57.29	13.68	80.82
- 6 + 0	30.91	59.75	12.16	76.33
-14 + 0	39.19	64.82	6.68	89.77
-28 + 0	38.52	66.20	5.10	90.90
-48 + 0	39.38	67.89	3.16	94.41
-65 + 0	37.02	68.97	2.07	95.66

Liberation for 66-67% Fe Concentrates is in the range of 28 to 48 Mesh.

2. Spiral Concentration on -14 Mesh Samples

Sample	Concentrate					Tails	Calc Head
	% Wt.	% Fe	% SiO_2	% Fe.Rec.	C.R.	% Fe	% Fe
Composite 1 - 7	34.3	61.77	10.23	77.1	2.91	9.58	27.48

Magnetic Cobbing of the Spiral Tails plus Regrinding to -200M produced an additional 11% Fe Recovery in a Concentrate of 66.7% Fe; 6% SiO_2

3. Spiral and Heavy Liquid Separation (S.G.3.5) -14 Mesh Samples

Composite 1 - 7	33.6	64.40	7.73	78.7	2.98	8.83	27.5

Combined products of spiral cleaning concentrate and H.L. (3.5) sinks (on both -14m +150m and -150m - spiral middlings and tailings)

4. Spiral and Magnetic Separation on -14 Mesh Samples

Composite 1 - 7	(a) 36.3	64.01	-	85.3	2.75	6.3	27.52
Composite 1 - 7 & 10	(b) 37.5	64.07	8.11	83.9	2.67	7.38	28.61

(a) Spiral and D.T. (2 stages) on spiral tails

(b) Spiral and W.M.S. on spiral tails.

5. Magnetic Separation and Heavy Liquid Separation (S.G.3.5.)

Composite 1 - 7	35.0	67.83	4.06	87.3	2.86	5.31	27.15

Magnetic cobbing and H.L. (S.G.3.5) on -14 mesh samples and magnetic cleaning on -200 mesh cobbing concentrates.

6. Carpco High Tension Separation

Sample	Concentrate					Tails	Calc Head
	% Wt.	% Fe	% SiO_2	% Fe.Rec.	C.R.	% Fe	% Fe
Composite 1 - 7	(a) 33.98	68.59	-	87.1	2.94	5.23	26.76
	(b) 38.29	67.26	4.76	86.71	2.61	5.80	26.38
	(c) 37.71	68.54	2.81	86.19	2.65	6.55	29.56
Composite 1 - 7 & 10	(d) 34.5	65.07	-	82.9	2.9	7.07	26.46
	(e) 36.0	64.64	-	84.5	2.78	4.44	27.58

(a) Tested on -14 mesh sample
(b) Tested on -35 mesh sample
(c) Tested on -65 mesh sample
(d) Tested on -35 mesh sample (without desliming)
(e) Tested on -35 mesh sample (with desliming)

6a. Carpco High Tension Separation with -65m +200m deslimed

Sample	Concentrate					Tails	Calc Head*
	% Wt.	% Fe	% SiO_2	%Fe.Rec.*	C.R.	% Fe	% Fe
Composite 1 - 7 & 10	(a) 49.29	68.88	2.32	90.0	2.16	6.92	37.47
	(b) 46.7	68.26	2.72	87.3	2.14	8.70	37.48
	46.7	71.14		91.0	2.14	5.82	37.51

Weight of Deslimed material not known.

(a) Carpco H.T.S. pilot plant tests in 2 stages (rougher and cleaning) with recirculation of cleaning tailing to rougher feed.

(b) J & L Laboratory H.T.S. test at the same conditions as that of (a).

* % Fe recovery and calculated head grade of deslimed materials.

Note: Difference in Head Grade from Section 6 (not deslimed).

7. Carpco High Tension Separation and Magnetic Separation

Sample	Concentrate					Tails	Calc Head *
	% Wt.	% Fe	% SiO_2	%Fe.Rec.*	C.R.	% Fe	% Fe
Composite 1 - 7 & 10	(a) 41.4	65.63		88.3	2.42	6.25	31.30
	(b) 37.4	64.81		88.0	2.67	5.28	27.54

(a) Carpco H.T.S. (rougher, cleaning and scavenging) on -14 + 35m and -35+325m, and magnetic separation on -325 mesh samples.

(b) Three stages of H.T.S. (rougher, cleaning and recleaning) on deslimed samples (-35 +400 mesh) and one stage of magnetic separation on slimes (-400 mesh).

8. Flotation with desliming and Carpco High Tension Separation on -65m

Composite 1 - 7	37.29	69.73	1.12	91.35	2.68	4.02	29.14

Fatty acid flotation on deslimed sample and Carpco H.T. rougher on both +200 and -200 flotation concentrates and one stage of scavenging on Carpco H.T. Ro. tails.

9. Roasting and Magnetic Separation (Davis Tube Tests) on -65 mesh Samples

Sample Number		Conc.					Tails	Calc Head
		% Wt.	% Fe	$\%SiO_2$	%Fe.Rec.	C.R.	% Fe	% Fe
BS.	1	38.31	69.58	4.20	96.1	2.61	1.77	27.80
	2	47.17	72.35	0.62	98.1	2.12	1.23	34.71
	3	40.98	71.58	1.20	97.6	2.44	1.23	30.11
	4	26.88	71.27	1.80	96.0	3.72	1.08	19.97
	5	43.10	60.44	14.96	96.1	2.32	1.84	27.11
	6	38.76	71.12	1.70	97.2	2.58	1.31	28.42
	7	26.60	71.73	1.54	96.3	3.76	1.00	19.81
	10							
Composite 1 - 7		23.47	68.97	4.29	97.7	4.26	1.15	29.11
Composite 1 - 7 & 10								
BS.	8 (a)	52.91	69.8	4.00	97.8	1.89	1.76	37.81
	(b)	51.28	72.1	0.34	97.8	1.95	1.69	37.81
	9	52.63	71.88	1.28	98.8	1.90	1.00	38.25

(a) The ground sample (-65 mesh) was preheated for 5 minutes in the laboratory pipe furnace at 600°C and roasted for 20 minutes with reducing gas and quenched in water.

(b) Tested on 100% -200 mesh sample.

Roasting at -14 mesh followed by Davis Tube tests produced Concentrates with over 10% Silica.

DIAMOND DRILL CORE - BLOOM DEPOSIT

D.D. Hole Number	Footage, Feet	D.D.H. Calc Head		Heavy Liquid Separation S.G. 3.5 -14 mesh							Heavy Liquid Separation, S.G. 3.5 -65 mesh						
				Sink					Float	Calc Head	Sink					Float	Calc Head
		% Fe	%MagFe	% Wt	% Fe	$\%SiO_2$	%FeRec	C.R.	% Fe	% Fe	% Wt.	% Fe	$\%SiO_2$	%FeRec	C.R.	%Fe	%Fe
DDH.QC- 1	5-814	21.53	4.77	36.23	66.95	3.90	89.24	2.76	4.59	27.18	36.3	68.80	1.57	95.05	2.75	2.04	26.28
2	7-297	27.40	2.88	41.06	66.46	4.25	92.38	2.44	4.17	29.54	38.49	67.53	3.01	96.41	2.60	1.57	26.96
3	118-789	25.06	8.04	38.54	63.77	6.03	90.57	2.59	4.16	27.14	34.61	67.54	7.37	95.29	2.89	1.78	24.53
4	7-197	21.01	6.38	30.64	64.95	5.73	90.65	3.26	2.96	21.96	18.95	68.95	1.75	95.51	5.28	1.31	13.68
5	102-306	24.39	8.61	34.99	62.40	8.37	86.77	2.86	5.12	25.17	34.08	65.94	4.21	93.64	2.93	2.31	24.00
6	54-231	27.99	25.64	a) 57.36	48.61	19.60	84.40	1.74	12.09	33.04	a) 55.61	55.12	12.74	92.42	1.80	5.66	33.16
7	37-563			30.08	65.47	4.70	80.11	4.98	4.08	16.41							
8	5-362	17.74	6.90	27.02	62.52	8.46	83.69	3.70	4.51	20.17							
9	6-327	14.11	7.08	b) 24.45	61.56	8.37	77.94	4.09	5.64	19.31							
11	10-420 430-690	27.37	14.68	31.72	62.15	10.00	79.37	3.15	7.50	24.83							
15	238-690	26.76	13.62	c) 30.70	63.89	8.95	77.61	3.26	8.17	25.28							
16	92-460	23.79	12.14	d) 29.05	61.63	9.58	80.07	3.44	6.28	22.36							
17	24-310 320-528	15.17	1.80	e) 37.51	65.24	5.49	84.07	2.67	7.41	29.11							
18	138-600 610-800	28.96	7.34	31.68	63.29	9.07	82.14	3.16	6.38	24.11							

a) Samples for the tests were taken from the D.D.H. footage 80-231
b) " " " " 120-327
c) " " " " 250-670
d) " " " " 92-450
e) " " " " 24-200

ROACH HILL EXTENSION

Davis Tube Tests: Two small bulk samples from two outcrops were tested at grinds of -35 M to - 150 M for Sample # 1 and -35 M to -100M for Sample # 2.

	Davis Tube Concentrate				Tails	Cals Head	Grind
	% Wt.	% Fe	% SiO_2	% Fe.Rec.	% Fe	% Fe	
Sample 1	41.9	68.4	5.0	85.6	8.3	33.5	- 150 M
Sample 2	54.7	66.8	7.1	96.5	2.9	37.8	- 100 M

Head Analyses

	% Fe	% Magnetic Fe
# 1	31.8	26.0
# 2	36.2	34.0

Drilling at a later date gave an average Crude Analysis of 24.0% Fe with lower Magnetic Fe.

9.2 (continued)

Grindability Determination

These results were reported by Jones & Laughlin although no specific procedure was given.

Bulk Sample Composite 1 - 7
(Goalpost Hill)

Mesh	Wi	W
65	8.6	4.3
100	11.5	7.7
150	13.5	11.2
200	15.0	17.1

Wi = KWH/Short Ton for 80% passing 100 Microns

W = Gross KWH/Short Ton

9.3 Colorado School of Mines Research Institute Testwork

The following data are a summary of the report of "A Characterization Of A Canadian Iron Ore" of April 4, 1973.

The seven samples were prepared by taking samples of laboratory rejects from the 1955-1956 drill core collected by Jones and Laughlin. This sampling was conducted by C.E. Harpur to verify the iron and magnetic analyses of ore samples from the Bloom Lake deposit. The samples were also to be used to determine the grade of concentrate obtainable from this deposit.

Testwork on the seven samples gave the following results:

a) Duplicate sampling verified analyses by Jones and Laughlin within sampling and analytical limits.

b) The magnetic content is variable.

c) Heavy Liquid at a Specific Gravity of 3.3 to simulate gravity separation by Humphrey Spirals demonstrated that concentrate could be produced in the range of 66.0 to 68.9% T Fe with T Fe Recoveries in the range of 87 to 91%.

d) Testwork was conducted on -14 Mesh material, although test results showed that -14 + 28 Mesh sink fraction was not completely liberated. In 5 of the 7 samples this fraction contained 58 to 63% T Fe.

e) Davis Tube tests on -200 Mesh float (tailing) produced concentrates with less than 60% T Fe. Indicating that finer grinding was required for liberation.

9.3 f) Davis Tube tests on -200 Mesh sink (Concentrate) showed that magnetic separation produced a super concentrate with 69 - 72% T Fe but represented less than 5% of the crude feed.

g) Metallurgical balances were based on Total Iron Analyses which include the variable amount of iron present in several silicate minerals (grunerite, actinolite, tremolite) found associated with the ore.

Crude Sample Analyses

CSMRI Sample No.	Drill Hole No.	% Total Fe.	% Soluble Fe.	% Magnetic Fe. (-150 M)
1	QC-21 (530-760 ft.)	30.5	27.7	2.25
2	QC-21 (370-530 ft.)	30.1	28.3	24.92
3	QC-14 (1-160 ft.)	30.8	29.6	0.32
4	QC-17 (24-200 ft.)	29.9	29.1	1.16
5	QC- 1 (5-290 ft.)	26.9	26.9	2.27
6	QC-11 (590-690 ft.)	33.8	32.7	9.37
7	QC-11 (210-350 ft.)	30.3	29.1	3.22

Heavy Liquid Sink Concentrate Analyses
- 14 Mesh

CSMRI Sample No.	% Weight	% Total Fe.	% Total Fe. Recovery
1	43.9	66.3	91.6
2	39.8	66.1	87.3
3	41.6	68.0	90.5
4	36.3	67.5	90.1
5	36.3	68.9	89.0
6	45.7	66.5	86.8
7	40.5	66.0	87.8

Observations: a) The crude iron analyses of the samples are close to the average for the deposit.

b) The Heavy Liquid sink results are optimum grades and recoveries which are not normally attainable in an operating plant.

Magnetic Content of -200 Mesh Float

CSRRI Sample No.	-200 Mesh Float		Davis Tube Concentrate			
	% Wt.	% T Fe(Calc)	Overall Ind. % Wt.		% T Fe	Overall Ind. T Fe Rec'y
2	7.2	10.1	0.8	10.5	59.7	62.4
6	8.3	13.6	0.7	8.8	58.9	38.2

Observations: a) Finer grinding than -200 Mesh is required for liberation of the magnetite in the tailings.

9.4 1975 Lakefield Research of Canada Testwork

A systematic metallurgical testing program was conducted using available save samples from the half core of the 1971 and 1972 drilling on the Bloom Lake deposit. Twenty-one Composites were prepared from 11 holes representing a range of friable Specularite, Magnetite-Specularite and Magnetite Silicate samples. Four samples of Magnetite Silicate were outside of the ore reserves and have been considered waste material.

The purpose of the testwork was as follows:

a) Determine the range of liberation within the deposit using Heavy Liquid Separation at Sp.Gr. of 3.3.

b) Conduct two stage table tests on 25 pound samples to more closely simulate Humphrey spiral results and establish whether concentrates with at least 66% Fe could be consistently produced.

c) Determine the amount of magnetite recoverable from tailings and the grind required for a saleable product.

d) Determine the grade and recovery of hematite from the tailings recoverable by WHIMS (wet high intensity magnetic separation).

e) Determine the analyses of concentrates produced from the testwork as suitable material for Oxide Pellets and Super Pellets.

The Lakefield Report of February 10, 1976 is contained in Appendix B.

SUMMARY OF RESULTS

1. Concentrate Analyses

Element	Gravity Concentrate	Super Concentrate A	Super Concentrate B	Super Concentrate C
% Sol Fe	67.4	71.7	69.1	69.8
% SiO_2	3.57	1.20	1.46	1.39
% Al_2O_3	0.20	0.10	0.21	0.18
% CaO	0.25			
% MgO	0.29			
% TiO_2	0.11			
% Mn	0.051			
% P	0.015			
% S	0.013			
% Na_2O	0.013			
% K_2O	0.007			

Super Concentrate was produced by regrinding to 78% -325M followed by

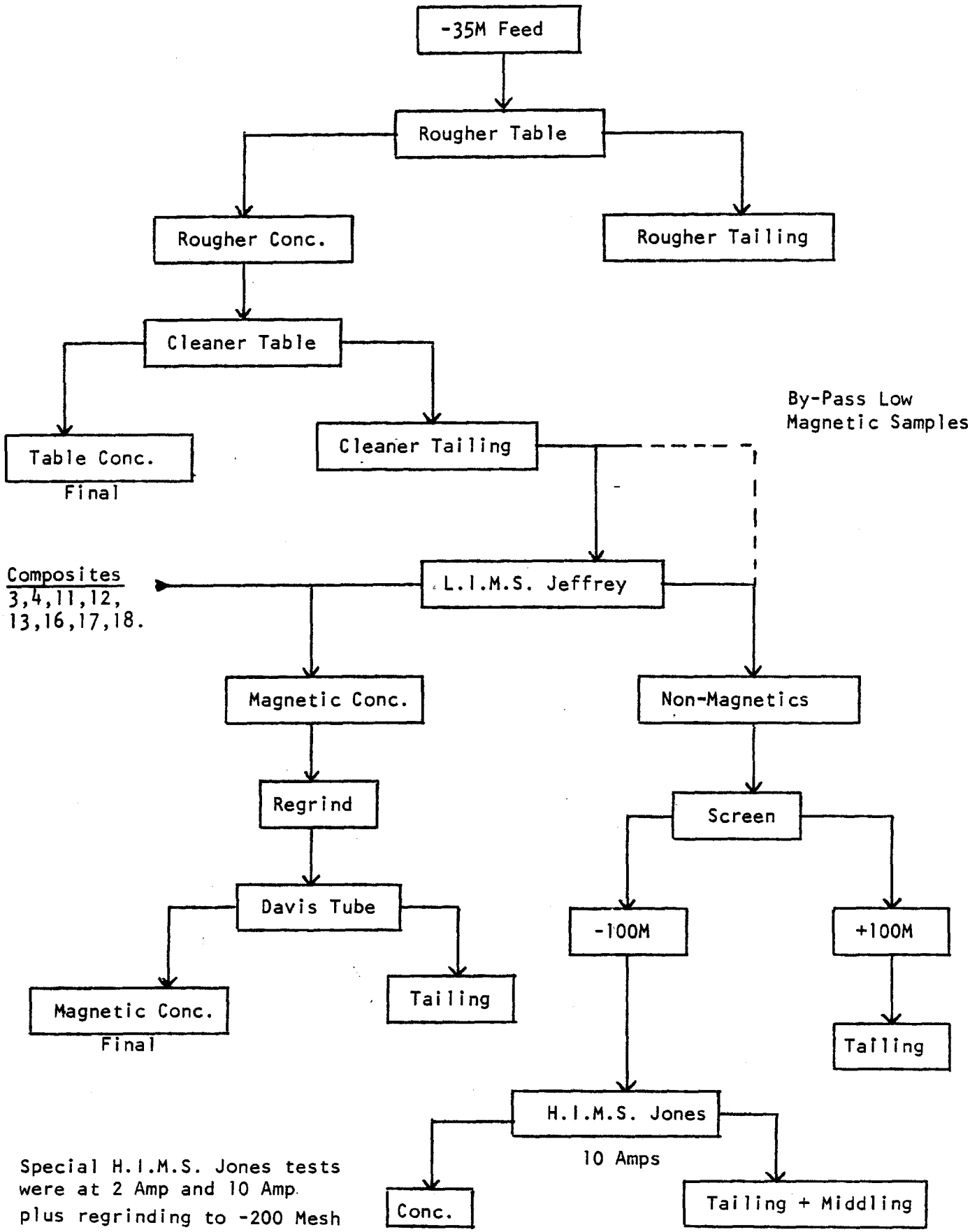
A - low intensity magnetic separation

B - high intensity magnetic separation

C - A + B

A semi-quantitative Spectrographic Analysis was made of the table concentrate with no harmful elements detected.

STANDARD FLOWSHEET



Special H.I.M.S. Jones tests were at 2 Amp and 10 Amp. plus regrinding to -200 Mesh

2. Table Tests With Low Intensity Magnetic Separation Of The Table Tails

All Composite samples were stage crushed to -35 Mesh for table tests. Tabling was made in two stages with the middlings continuously recirculated by a pump. The first stage produced a final rougher tailing and a rougher concentrate which was retreated to produce a cleaner concentrate and a cleaner tailing. The cleaner and rougher tailing were recombined for magnetite recovery by low intensity magnetic separation.

The table test results are grouped according to the zones in the Bloom Lake Deposit.

Main Pit Area:

Composites East from Section 66W or

Composites: 1, 2, 3, 4, 5, 12, 13, 14, 15.

Western Extension:

Composites West from Section 150W (Carrot Lake Zone)

Composites: 6, 7, 8, 9, 10, 19, 20, 21.

Magnetite-Silicate Waste

Four composites were prepared from material outside of the Ore Reserves in order to determine whether it could be treated to produce acceptable concentrates. These composites are from the eastern edge of the Main Pit Area.

Composites: 11, 16, 17, 18.

Tabling and Magnetic Separation

Description	Footage	Head		Table Conc.			Table + Mag.Conc.		
		% SolFe	% MagFe	Wt. %	% SolFe	% Dist. SolFe	Wt. %	% SolFe	% Dist. SolFe
1. Main Pit Zone	2518	30.26	13.02	38.07	66.27	83.38	41.01	66.64	90.32
2. West Extension	2159	31.36	0.74	38.14	67.88	82.56	(No Magnetic Sep.)		
3. Total Bloom Lake Deposit	4677	30.77	7.35	38.10	67.01	82.99	39.68	67.19	86.66
4. Mag. Spec. * Main Pit Zone	1176	33.56	23.53	41.90	65.75	82.09	48.20	66.48	95.48
5. Magnetite-Silicate Waste	826	17.18	13.53	19.72	67.11	77.05	23.26	67.46	91.36

* Composites 3, 4, 12 and 13 which are Magnetite-Specularite containing more than 10% Mag.Fe. in the crude.

The average grade of the Composites tested from the Main Pit Area of 30.26% Fe is close to the average grade of this area of 30.1% Fe.

See Lakefield Report of February 10, 1976 for individual table tests.

Table Tests At -35 Mesh

Composite No.	Hole No.	Cross Section	Head % SolFe	Head % MagFe	Table Concentrate % Wt.	Table Concentrate % SolFe	Table Concentrate % Dist SolFe
MAIN PIT ZONE							
1	71-1	66+00W	28.2	2.60	33.1	68.2	80.0
2	71-1	66+00W	27.2	2.61	31.9	67.3	79.1
3	71-2	56+00W	36.3	27.9	47.2	65.5	84.6
4	71-2	56+00W	26.3	14.1	29.9	67.7	77.0
5	71-2	56+00W	26.2	4.62	35.6	66.5	90.5
12	72-2	10+SW	34.6	19.4	47.7	64.2	88.5
13	72-2	10+SW	38.8	34.9	45.2	66.0	76.9
14	72-2	"	29.5	5.69	40.3	65.4	89.4
15	72-2	"	25.9	4.44	34.7	66.2	88.8
	Average		30.26	13.02	38.1	66.3	83.4
WEST EXTENSION (Carrot Lake Zone)							
6	71-3	164+00W	32.4	0.44	37.8	67.6	78.9
7	71-6	150+00W	30.8	0.25	36.0	68.3	79.8
8	71-8	150+00W	31.0	2.42	37.9	67.7	82.8
9	71-8	150+00W	31.2	0.26	40.1	67.7	87.0
10	71-9	150+00W	27.0	0.24	34.5	67.6	86.3
19	72-9	155+00W	32.9	0.64	39.8	68.1	82.4
20	72-9	155+00W	31.8	0.62	37.9	67.9	80.8
21	72-10	172+00W	35.0	0.59	41.8	68.3	81.7
	Average		31.4	0.74	38.14	67.9	82.6
MAGNETITE-SILICATE WASTE (East of Main Pit Zone)							
11	72-2	10+00SW	11.8	9.43	10.0	69.8	59.5
16	72-3	30+00W	13.1	8.98	16.1	66.6	81.7
17	72-3	30+00W	17.1	11.1	22.4	69.6	88.5
18	72-5	40+00W	22.5	20.4	24.5	65.2	71.0
	Average		17.2	13.5	19.7	67.1	77.1

Magnetic Recovery From Table Tailings

Table tailings from composites with more than 10% Magnetic Iron in the crude were treated on a Jeffrey low intensity magnetic separator to recover the magnetide. The Jeffrey concentrate was low grade and required regrinding to produce an acceptable concentrate. The Davis Tube was used on the reground samples for magnetite recovery.

Observations:

a) Jeffrey concentrates from table tailing contained 24 to 37% soluble Iron.

b) Regrinding to 90 to 97% -400 Mesh is required to produce a magnetic concentrate in the range of 67 to 71% Sol Fe.

c) About 12% additional Fe Recovery can be obtained from Crude Magnetite Specularite by retreating the tailings by low intensity magnetic separation.

d) Magnetite-Silicate Waste produces a high-grade concentrate.

Davis Tube Concentrates
From Reground Jeffrey Concentrates Of Table Tailings

Comp. No.	Hole No.	Jeffrey Conc.		Overall D.T. Concentrate		
		% SolFe	% MagFe	% Wt	% SolFe	%SolFe Rec'y
3	71-2	24.4	22.3	5.9	68.5	11.0
4	71-2	33.2	28.7	4.8	71.2	13.0
12	72-2	29.9	28.2	4.0	69.8	7.0
13	72-2	31.6	29.6	11.4	69.3	20.4
Silicate						
11	72-2	27.9	25.5	3.6	70.5	21.6
16	72-3	21.4	19.8	1.8	71.3	9.8
17	72-3	30.3	27.6	1.1	72.3	4.6
18	72-5	37.4	34.0	6.4	70.6	20.1

Davis Tube Results On Reground

Jeffrey Concentrates (Table Tailings)

Composite No.	Hole No.	Jeffrey Conc. % SolFe	Jeffrey Conc. % MagFe	D.T. Concentrate % Wt.	D.T. Concentrate % SolFe	D.T. Concentrate % FeRec'y	Grind % -400M
3	71-2	24.4	22.6	51.5	43.9	92.7	36.4
				39.5	57.3	92.8	55.4
				35.1	64.4	92.6	84.7
				33.8	66.9	92.7	92.2
				32.9	68.5	92.4	97.6
4	71-2	33.2	32.0	44.9	71.7	97.0	90.8
				44.5	71.2	95.4	96.6
12	72-2	29.9	27.8	39.5	69.8	92.2	90.4
				39.9	69.8	93.1	95.2
13	72-2	32.5	31.2	45.6	68.7	96.4	94.2
				45.0	69.3	96.0	97.2
18	72-5 (Waste)	37.4	35.6	50.7	70.6	95.7	91.6
				50.1	70.6	94.6	97.0

The finest grind was used for combining the magnetite concentrate with the table concentrate.

Wet High Intensity Magnetic Separation

-100 Mesh Tests

Jones High Intensity tests were conducted on the table tailings and Jeffrey separator tailings except for C ompites 11, 16, 17 (Magnetite Silicate)

Samples for these tests were screened at 100 Mesh with the +100 Mesh containing too little iron for further treatment (1.5 to 5.8% Fe.) The -100 Mesh fraction generally produced low-grade concentrates in the range of 6 to 60% Sol.Fe. containing from 2% to 12% of the Sol.Fe. of the samples. The table tailings were low-grade assaying from 2.4 to 11.0% Sol.Fe. These are lower than normal plant tailings in the Wabush area which contain about 15 to 20% Sol.Fe.

Special Tests At -100 and -200 Mesh

Composites 1 and 2 were combined and retabled to produce a higher Fe in the tailings for high intensity testwork.

The table tailings were scavenged with a low intensity Jeffrey separator producing a feed to high intensity separation of 11.4% Sol.Fe.

a) -100 Mesh

The +100 Mesh reject assayed 10.1% Sol.Fe. with the -100 Mesh containing 14.3.

WHIMS Test at 10 and 2 Amperes.

	% Wt.	% SolFe	Fe Dist.
2 Amp. Conc	20.3	39.4	56.0
10 Amp. Conc.	20.6	22.1	31.9
10 Amp. Midds.	19.4	6.5	8.8
10 Amp. Non.Mag.	39.7	1.2	3.3
	100.0	14.3	100.0

-100 Mesh material was not liberated and contained magnetite middlings.

b) -200 Mesh

The table tailings were scavenged with a Jeffrey separator and the Jeffrey tailings containing 11.4% Sol.Fe. were ground to -200 Mesh and treated by the Jones Separator at 2 Amps., followed by 10 Amps. on the Jones first stage tailings and 5 Amps. on the Jones second stage tailings.

b) (continued)

The best concentrate was the 5 Amp. retreatment of the 10 Amps. rougher concentrate and contained 60% Sol.Fe. recovering 28% of the Sol.Fe. on the Jeffrey tailings, equivalent to:

3.6% Wt. of table feed.

60.5% Sol.Fe.

7.8% Sol.Fe. Distribution of the Table Feed.

Comparison Of Results

a) Tabling only to produce high iron recovery.

b) Tabling with high tails followed by wet low and high intensity magnetic separation.

	Combined Concentrate			Tailing		
	% Wt	% SolFe	Fe.Dist	% Wt	% SolFe	Fe.Dist.
a) Tabling Only	32.5	67.8	79.5	67.5	8.39	20.5
b) Tabling + Low and High Int. Mag. Sep at -200 Mesh	34.8	65.1	80.1			
Head	100.0	27.7	100.0			

Note: In conc (a) the Jeffrey Concentrate was not reground and retreated which would have increased the grade with only a slight loss of recovery.

Conclusions:

1. Efficient gravity separation can produce higher grade concentrate with equivalent recoveries and a simpler flowsheet.

2. Jeffrey magnetic treatment of conc (a) would increase the grade and recovery slightly.

3. Samples containing considerable magnetite in the crude (Magnetite-Specularite) produced low-grade wet high intensity magnetic concentrates due to the magnetic middlings which were finer than 200 Mesh.

4. Grinding to -325 might have improved the grade of the WHIMS concentrate but it was not part of the program.

TYPICAL SCREEN ANALYSES

OF TABLE CONCENTRATES

Product	% Wt.	Assay % SolFe	Assay % MagFe	% Distribution SolFe	% Distribution MagFe
Composite 1 - Hole 71-1 Main Pit Zone - Specularite					
- 35 + 48M	5.8	67.7	2.77	5.7	3.4
- 48 + 65M	25.4	68.0	3.21	25.2	17.0
- 65 + 100M	30.7	68.4	4.14	30.7	26.6
-100 + 150M	17.4	68.3	5.18	17.4	18.9
-150 + 200M	12.2	69.3	6.68	12.4	17.0
-200M	8.5	69.4	9.63	8.6	17.1
	100.0	68.4	4.78	100.0	100.0
Composite 3 - Hole 71-2 Main Pit Zone - Magnetite Specularite					
- 35 + 48M	6.1	70.3	49.2	6.5	6.3
- 48 + 65M	13.0	70.2	47.1	13.9	12.9
- 65 + 100M	22.0	68.8	47.1	23.1	21.9
-100 + 150M	15.3	64.8	44.7	15.1	14.4
-150 + 200M	16.7	59.1	42.6	15.0	15.1
-200M	26.9	64.3	51.8	26.4	29.4
	100.0	65.6	47.4	100.0	100.0
Composite 19 - Hole 72-9 Western Extension - Specularite					
- 35 + 48M	7.3	68.7	0.73	7.42	5.92
- 48 + 65M	26.3	69.0	0.73	26.86	21.33
- 65 + 100M	25.4	68.5	0.83	25.75	23.41
-100 + 150M	17.2	65.7	0.90	16.73	17.19
-150 + 200M	11.3	64.5	1.09	10.79	13.68
-200M	12.5	67.3	1.33	12.45	18.46
	100.0	67.6	0.90	100.0	100.0

10.0 WASTE AND TAILING DISPOSAL

Provision is required for the handling of the following tonnages of rock stripping and tailings based on 5,000,000 T.P. year of concentrate production.

	Tons/Year	Tons For 30 Years
Crude Plant Feed	13.9	417
Concentrate	5.0	150
Rock Stripping Waste	9.2	276
Plant Tailings	8.9	267

Plate 2 at the end of this report illustrates the location of two waste dumps for the rock stripping adjacent to the Main Pit. Provision for initial tailings disposal is in the Carrot Lake area where up to 5 years of tailings could be placed up to an elevation of 2200 feet. Further study is required for tailings disposal subject to preliminary indications of what areas would be acceptable. A large area towards Boulder Lake and downstream from the plant site is reserved for tailings disposal.

11.0 ENVIRONMENTAL CONSIDERATIONS

Future mine development will require a thorough understanding of the existing baseline environmental conditions and the impact of such a development. Authorization for any mine development will be required from several agencies within the Government of Quebec including the following departments: Environment, Natural Resources (mining, water resources and forestry subdivisions). The regulations of several Acts must also be followed including: The Environmental Quality Act, The Canada Water Act, The Fisheries Act and the Migratory Birds Convention Act.

At the present time the Environment Quality Act appears to exclude Asbestos and Metal Mines but careful planning should be followed.

The Chief areas of concern regarding this development are:

11.1 Contamination of Surface and Ground Water.

a) The Bloom Lake deposit is adjacent to three watersheds with water flowing to the Wabush-Hamilton River systems, the Pekans-Moise Rivers to the St. Lawrence and northward through Boulder Lake to the Kaniapiskau River - Ungava Bay system.

b) Tailings ponds will require sufficient area for settling and decantation basins to provide clear water overflow and water reclamation for the plant.

11.1 c) Fresh water reservoirs are required up-stream from the plant in such lakes as Bloom Lake and Confusion Lake. Additional water could be pumped to these lakes from adjacent lakes.

d) Crescent and Triangle Lakes will require draining and the streams flowing north from these lakes will require diversion and pumping of this water source from the pit area.

11.2 Dump design and reclamation

Plans and designs of dumps will require careful siting and planning to reduce any adverse appearance. Greater freedom is possible here than in more highly urbanized areas.

11.3 An ecological inventory will be required as part of an initial baseline study before the area is disturbed. The study could be started at the time of the first drilling phase as part of a preliminary study which would also include hydrological study of stream flows in the vicinity of the mine and plant sites. Seasonal variations in water flow will also require measurement over all seasons. An allowance of $25,000 is included in the next proposed drill program for a biologist and a hydrology assistant to work from the drill camp.

APPENDIX A

PRELIMINARY OPEN PIT MINE DESIGN

OF

QUEBEC COBALT & EXPLORATION LIMITED'S

BLOOM LAKE PROPERTY

FOR

REPUBLIC STEEL CORPORATION

TO ACCOMPANY A REPORT BY

H. E. NEAL & ASSOCIATES LIMITED

Ministère des Richesses Naturelles, Québec
SERVICE DE LA
DOCUMENTATION TECHNIQUE
Date: 6 DEC 1976
No GM: 32215

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1. MINE DESIGN - INTRODUCTION

Dames & Moore's involvement in design work for the Bloom Lake deposit was outlined in a proposal to H. E. Neal & Associates Limited of June 27, 1975. The work was to consist of the design of an open pit suitable for delivery of some 14 million long tons of raw iron ore to a concentration plant. From this input, an estimated 5 million long tons of feed to the pellet plant was to be produced annually.

The work was required in order to investigate, in a preliminary but comprehensive manner, such items as recoverable mining reserves, waste stripping requirements, and mine infrastructure (dumps, power distribution and roads).

In order to expedite the design, work was based upon:

a) currently available geological information

b) ancillary data and discussions with H. E. Neal & Associates Limited

c) reasonable cost and production assumptions.

Dames & Moore used the geological plans and sections provided, without further confirmatory drilling for reserves or geotechnical purposes, nor were field visits by mining engineers deemed useful at this preliminary stage.

2. METHOD OF MINING

2.1 General Description

The geological evidence available on sections indicates that mining should commence at the Eastern end of the orebody where some 435.9 million tons of crude iron-ore is available, and associated with favourable amounts of waste material. This would be sufficient to supply a pellet plant with 5 million tons of concentrate per annum for over 30 years.

In addition to the eastern pit, more than 20 years of additional reserves are indicated by initial analysis as being available in a Western Extension of the operations should future drilling confirm early geological interpretation.

The Bloom Lake deposit is seen as being most efficiently and effectively mined by conventional truck and shovel equipment. The production and waste handling requirements are such that large sized operating equipment with significant proven field experience is regarded as being the optimum choice at this time.

During mining, conventional large diameter rotary drilling and blasting of iron ore and waste rock is envisaged. This procedure with 40-50ft bench heights is universally accepted as giving satisfactory operating conditions and costs. Choosing this type of operating policy has the benefit of reducing overall manpower training and requirements for an area where skilled workers are at a premium.

2.2 Mine Design

Working from geological sections, each section along the whole strike length of the orebody was examined for mineable ore using:

a) only the oxide facies of iron formation

b) a minimum pit bottom of 200 ft

c) slopes angles of 50^{o}

d) economic criteria of 4.5 tons of waste per ton of crude iron ore, cut-off stripping ratio (C.O.S.R.)

e) critical examination of each section where an overall ratio of greater than 1:1 resulted from application of the criteria a) to d) above.

The minimum operating width of 200 ft was determined by reference to the size of the equipment envisaged.

In determining the overall slope for the open pit the waste rock material was assumed capable of providing a stable slope at up to 50^{o}. Selected core samples were visually examined in the course of the study and confirmed the use of 50^{o} as a conservative figure in this initial design work. Further work will be necessary to provide definitive values for each individual slope. Subsequent geotechnical work may increase the slope angles proposed in any future design, but only after a program of field testing and analysis has been successfully completed.

The cut-off stripping ratio indicates the amount of waste stripping which can be paid for by a given unit of desired mineral. In the case of Bloom Lake,

it demonstrates how many tons of waste can be economically removed from the pit in order to mine one ton of raw iron ore. Mining at the cut-off stripping ratio results in a break-even operation, showing neither profit nor loss.

The cut-off stripping ratio (C.O.S.R.) is the primary tool used to determine the economic pit limits, leading to the design of the Ultimate Pit; as such it is susceptible to changes in operating cost, product price, processing and transportation costs, and waste stripping costs.

The following calculations indicate how the ultimate C.O.S.R. was derived:

TABLE A

ASSUMED PRODUCTION COSTS PER TON CRUDE IRON ORE
FOR STRIPPING RATIO CALCULATION

	conc. ratio 2.78 /ton crude	/ton concentrate	extended
Mining	0.90¢		0.90¢
Concentrating	1.00¢		1.00¢
Pelletizing		3.30¢	1.19¢
Overheads	0.50¢		0.50¢
Handling Charge at Port		0.50¢	0.18¢
Transportation (rail)		5.00¢	1.80¢
Assumed Royalty		0.75¢	0.27¢
			$5.84
Contingency (12% costs)			0.71¢/ton crude
Total cost per ton of crude iron ore			$6.55

These unit costs were then used in the C.O.S.R. calculation outlined in Table B.

TABLE B

BLOOM LAKE - C.O.S.R.

$$\text{C.O.S.R.} = \frac{\text{Value of in-situ iron/ton - Production costs/ton}}{\text{Average stripping cost/ton waste}}$$

Value of in-situ iron per ton of crude

Pellet grade	65.5%	
Price per iron unit	$0.45	
Revenue per ton pellets		$29.48/ton
Since concentration ratio		2.78
Then value of ore to crushers		$10.60/ton

Production costs per ton		$5.84/ton
Contingency		$0.71
	TOTAL	$6.55

Stripping costs per ton waste 90¢

$$\text{C.O.S.R.} = \frac{10.60 - 6.55}{0.90} = 4.5$$

Therefore the cut-off point occurs when 4.5 tons of waste has to be moved for 1 ton of crude iron ore to be released.

The actual unit mining cost is seen to be fairly insensitive when considered in the light of some of the assumptions made. i.e. the optimum slope angles in the pit (discussed later), the selling price of pellets, transportation costs. However, it should be appreciated that the application of any revised

C.O.S.R. will change the size and shape of the ultimate pit somewhat and might change the overall stripping ratio. The design process is an iterative one and should be refined during any subsequent pre-feasibility studies.

The procedure of reviewing each section for apparent overall ratio enables a realistic overview to be maintained throughout, when identifying potential ore reserves.

Initially, a large orebody was identified as being worthy of economic consideration. Shown in Plate 2, the whole of the strike length between 17,000w and 4,100w (some 12,900 ft) was evaluated from an ore reserve standpoint.

Examination of this primary design suggested that, while the approach is satisfactory, several observations may be made:

a) the western reserves included a significant tonnage which was only geologically inferred and it should not be included in any definitive mine design

b) by concentrating activities in either the eastern or western ends of the pit, significant waste removal could be avoided or deferred for many years

c) the overall pit was extremely large and would last for some 57 years.

The scope of the study was narrowed down to identify the most attractive tonnage, for an assumed project life of 30 years. Such tonnage allows the property to be evaluated in realistic fiscal terms, whilst acknowledging the ultimate use of the remaining reserves once the initial capital expenditure

has been paid off. It also allows concentration of future exploration expenditure and avoids the possibility of wasted money through a dilution of effort and drilling. The eastern end of the pit provided a ready starting point since east of 90.00w over a strike length of 5,000 ft, some 436 million tons were initially identified.

This eastern portion of the orebody was selected for more detailed analysis.

A haul road was incorporated into the initial design and a workable pit design generated.

The detailed calculation of ore reserves is discussed under a separate section.

The siting of the potential main access routes and haulage roads into the pit need to be co-ordinated with the selection of the concentrator plant site. Since the plant site is initially proposed as being north of the pit, a convenient point of egress was determined to be in the north west corner of the pit. Here the shortest route to the plant coincides with one of the lowest topographical levels around the pit, and a road at 5% has been selected to run to a crusher station, since it will be used to haul downhill for many years.

Whilst this siting would fulfil ultimate pit haulage requirements, in the early years of the pit, an alternate best route appears to be from the north east corner of the pit. This temporary road would come from the mine on a level with operations and then descend to the plant at 8%. The ultimate haulage road in the pit is designed to be 80 feet wide, which includes an

allowance for a drainage ditch on the inside edge. The road grades at -8% from the pit entrance, remaining in waste material where possible and on the southern and western edges of the pit down to the pit bottom. The siting of the ultimate road on these walls is considered to best suit the overall pit shape, since it allows for a minimum number of turns and switch backs. Further it reduces the overall exposure of the road to the maximum effects of the freeze thaw cycle of the winter sun.

The pit plan as presented in Plate 1, encroaches into Crescent Lake to the south. The lake discharges across the proposed pit area into Golden Lake. Crescent Lake would need to be drained. Early analysis suggests diversion of this lake into an adjacent watershed, draining the lake, and backfilling the depression. Since the Bloom Lake deposit is right at the height of land where three separate river systems originate, the diversion involves minimal excavation work and capital cost.

Operational experience suggests that the main power line should loop the pit in order to provide a measure of safety in ensuring supply of power. The design incorporates such a loop.

3. ORE RESERVES AND WASTE REMOVAL

The calculation of mineable crude iron ore reserves for Bloom Lake were based on the following:

a) west of 61-00w every section was given an influence halfway to the next section line

b) east of 61-00w, plans were drawn up at 200 ft intervals, vertically

c) densities of:- (i) iron ore 11 cu.ft.per ton
(ii) waste 12.5 cu.ft.per ton
were assumed

d) a cut-off grade of 15% soluble Fe.

Only the iron oxide mineralization was regarded as potential ore. The data on soluble iron grades varies over the length of the orebody, while the mineral type remains constant.

One acceptable method of grading the orebody was therefore to formulate a mathematical model of the grade distribution. The value of soluble iron - Sol % Fe - was taken from the log of each hole on the property, for each 5 ft composite, and tabulated by section.

Over fourteen hundred (1,400) samples were analysed and their average grade was determined to be 28.4% Sol Fe. Of these samples 1,300 were over 15% Sol Fe and averaged 30.1% Sol Fe. Thus 91.5% of in-situ reserves were regarded as being suitable for plant feed. The histogram and smoothed curve of this data are shown on Plate 3.

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The reasoning behind the use of this system of grading is:

a) the approach has mathematical validity, each sample has equal weighting

b) insufficient data exists, for each separate section, over the whole strike length, to provide reliable grading results. (Some sections have no drill holes - only surface showings)

c) it simplifies treatment of internal waste in that 8.5% of the ore zone will be assumed to be waste. i.e. less than 15% Sol Fe.

d) the results obtained compare favourably with an earlier analysis which indicated 29.6% Sol Fe with a 15% Sol Fe cut-off. This difference is not statistically significant.

TABLE C

BLOOM LAKE - MINEABLE ORE RESERVES

(10^6 long tons)

Section	In-situ Iron Ore	Associated Waste
East of 61.00w	296.56	168.34
66.00w	94.73	39.55
74.00w	44.03	18.17
81.00w	35.42	19.37
90.00w	5.69	1.74
Total for Pit Design from Sections	476.43 at 28.4% Fe	247.17
Crude Iron Ore available at 15% Fe cut off	435.93 at 30.1% Fe	287.67

Overall stripping ratio of 0.66 tons waste per ton of crude iron ore includes material less than 15% Fe.as waste.

4. DISPOSAL OF WASTE MATERIAL

Over the first 30 years life of the mine, there will be a requirement to dispose of some 300 million tons of run-of-mine waste and approximately 270 million tons of plant reject. Clearly, the effective handling of such volumes of material will have an important bearing on the overall project profitability.

4.1 Dump Design

The critical factors in the design of dump areas are:

- minimum length of haulage
- layout of dump areas
- timing in the construction and use of particular areas.

These principles are constrained by the items noted below:

a) the dump limits should be 200 feet from the ultimate pit crest to allow for the pit power loop and access thereto, and to avoid slope surcharge problems

b) the dump toes should be some 200 feet from significant streams and lakes as noted on the available plans

c) the dumps should be located on ground currently held by the owners

d) the dumps must not be placed so as to sterilize potential future ore reserves amenable to open-pit mining

e) a swell factor of 1.3 has been assumed for run-of-mine waste.

f) the slope angle has been selected as 35^{o} for initial design purposes which is slightly below the natural angle of repose (36^{o}-38^{o})

g) the dumps will be accessed by haul roads of not steeper than 8%, and will be located compatibly with the main haul road

h) a minimum working width of 200 feet has been provided at the dump top, and the maximum elevation of the dump is not higher than 400 feet above the local pit crest.

i) a number of geotechnical aspects are also important: prime concerns are (i) constituents of the base material
(ii) its attitude with respect to bedrock

j) limits may be imposed upon planned dump heights by bulk cohesion and friction angles anticipated in the waste products

k) hydrology is important in that dumps in the past have been undercut and eroded through the action of flowing water

l) environmental factors influence (i) the base preparation prior to dumping
(ii) the method of dump construction with respect to the timing of reclamation
(iii) treatment of fines run-off contaminating surface water.

Two potential dump areas have been selected. (See Plate 2)

- Dump 1 to the east of the pit should have a capacity of some 216 million tons
- Dump 2 to the north of the pit should have a capacity of some 35 million tons
- Total storage capacity of some 251 million tons.

Additional dump space will be available when Crescent Lake to the south of the pit has been drained. Waste material will also be needed to form the roads and causeways to the plant. Therefore, initial calculations suggest that adequate dump space exists adjacent to the open-pit area.

However, it is prudent to highlight at this time two factors:

a) Mine operations should commence with the removal of any trees and vegetation from the mine and dump areas and all soil encountered should be separately stockpiled for subsequent reclamation activity.

b) Dumps constructed in lifts of between 50'-100' are inherently more stable than those created by tipping over an increasingly high bank.

4.2 Tailings Disposal

The various options available for tailings disposal have not been exhaustively examined. However, the tailings areas need to be on company owned land, in close proximity to the plant area on a site which can be readily prepared to meet structural and environmental requirements.

Part of the selection criteria applied in the choice of plant location to the north of Golden Lake was the availability of land suitable for tailings.

The overall requirement for tailings areas has been calculated from:

- crude ore to plant 13.9 million tons per annum
- concentrate ex plant 5.0 million tons per annum
- tailings 8.9 million tons per annum

- bulk characteristics assumed - 1 ton of tailings occupies 20 cu.ft. (i.e. 112 lbs per cu.ft.)
- each year 8.9 x 20 million cu.ft. of tailings area will be required. (178 million ft^3)

In the early years of the mine, it is proposed that Carrot Lake, to the west of the mine area, provides a suitable disposal area. The railway extension would provide a retaining embankment and add considerably to the lake's storage capacity. This initial tailings area is shown in Plate 2 and has an approximate capacity of 45 million tons, to an elevation of 2200. This is sufficient for the first 5 year life of the mine.

Alternate lakes exist to the north west of the proposed railway. Ideally, these should be used as decant lakes in the early years and tailing areas in latter years. The water complex from Carrot Lake down to Boulder Lake provides the most favourable tailings area complex, since numerous natural dam sites appear to be available, and it is downstream of the pit and plant areas.

5. PLANT LOCATION

In assessing the relative merits of potential sites for the concentration plant, the following criteria were applied:

- the plant needs to be as close as possible to the centre of mass of the total mineable ore reserves
- the site should be sufficiently well away from the open pit area to avoid damage from blasting, estimated at over 2,000 ft.
- the area should be relatively flat and accessible by rail and road
- there should be the facility to easily dispose of the large volume of tailings into suitably designed settling areas
- fresh water should also be available for delivery to the plant
- the site must be on land that the company has leased
- adequate land needs to be available for stockpiling of concentrated material and for blending purposes
- provision needs to be made for loadout facilities including a rail loop and by-pass siding. Unit train operations are envisaged with maintenance and siding facilities located at Port Cartier.

Inspection of geological and topographical data leads to selection of a flat area to the north of Golden Lake (shown in Plates 1 and 2). This site fits all the criteria and has adequate space for a warehouse, shop and office complex at a suitably adjacent site, yet is sufficiently removed from the pit and plant.

Fresh intake water for the plant facilities readily exists to the north east and south east, up stream of the plant area in the Kaniapiscau River

watershed, also in the adjacent Hamilton River watershed. Bloom Lake to the south east potentially offers one of the most favourable sources of fresh water. Tailings would be expelled downstream to the west.

Water should be reclaimed downstream of Boulder Lake and contiguous downstream lakes for make-up to the plant. This would reduce the fresh water requirements.

Access to the plant from the open pit would be by way of a haulage road from the ultimate pit road across a causeway over Golden Lake to the south of the plant site where a crusher is proposed. However, the alternative exists of having the crusher site adjacent to the open pit exit and conveying material from an ex-crusher stockpile across the lake to the plant site. The relative merits and costs of these alternatives have not been exhaustively examined at this time.

The foundation conditions existing in this area are as yet undetermined. Significant preparatory work may be required in order to provide a suitable site with good foundation conditions and proper drainage.

Furthermore, the plant would have to meet the detailed environmental impact guidelines currently being introduced in the Province.

6. EQUIPMENT SELECTION

As previously mentioned, the Bloom Lake deposit is seen as being mined by conventional truck and shovel operations.

Annual production of 5 million long tons of concentrate is equivalent to a crude iron ore extraction of 13.9 million long tons assuming a plant concentration of 2.78.

The instantaneous stripping ratio will vary during the life of the mine from approximately 0.1 tons of waste per 1 ton of crude iron ore in the initial five years up to 0.9 tons of waste per 1 ton of crude iron ore in years 11 onwards.

Hence, the total annual tonnage to be moved will vary from 15.3 million tons per annum up to 26 million tons per annum.

With this size of operation, trucks with a capacity of 120-150 tons teamed to a 15 cubic yard shovel offer a combination proven in similar conditions. Consequently, this configuration has been used as a base point in developing equipment requirements. Using average mining tonnages to be moved in any one year, the approximate number of major items has been calculated and summarized in Table D - Major Mine Equipment Summary. (The back-up calculations are included under "Supporting Data".)

A capital cost summary for the open pit operations has been prepared as a preliminary guide - Table E - using current Canadian Dollar prices with an

amount for spares inventory. Lead times for delivery have not been examined in detail, but a recent estimate is 10-18 months for trucks, loaders, dozers and drills with 18-30 months for shovel units.

TABLE D

MAJOR MINE EQUIPMENT SUMMARY

ITEM	REQUIREMENT			OPERATING LIFE (YEARS)	AVAILABILITY	CAPITAL COST/UNIT CANADIAN $ INCL. 5% SPARES
YEARS	0-5	6-10	11-30			
Drills	2	2	3	20	.8	396,000
Shovels	3	4	5	20	.8	1,627,000
Trucks	9	11	13	8-10	.8	462,000
Front end loaders (10-12 cu.yd)	2	2	3	8	.7	368,000
Dozers	4	5	6	10	.75	226,000
Ancillary equipment						
Grader	2	2	3	10	.8	153,000
Air track	2	2	2	20	.85	55,000
Supply truck	3	3	3	10	.85	45,000
Water truck	1	1	1	10	.85	45,000
Bulk explosives	Normally supplied under contract for bulk explosives					
Flatbed	1	1	1	10	.85	70,000

TABLE E

INITIAL CAPITAL COST SUMMARY

Major Mine Equipment at Start-up

ITEM	NUMBER	TOTAL COST ($ x 1,000)
Drill	2	792
Shovel	3	4,881
Truck	9	4,158
Loader	2	736
Dozer	4	904
Ancillary	-	666
Communications	-	50
Roads	-	120
Power Network	-	395
	TOTAL	12,702

7. ENVIRONMENTAL FACTORS

The need for environmental input related to major mine development in Quebec, has stemmed from past experience and a recognition of environmental fragility. Environmental concerns are increasingly linked to legislation at both the federal and provincial levels, and in the latter case it is known that a number of acts and their regulations are currently being formulated and reviewed. It can be anticipated that future mine development will only be permitted with understanding of the relevant baseline conditions, and of the environmental impact of the proposed project. The most significant acts which will affect potential mine development at Bloom Lake are:

a) The Environment Quality Act

b) The Canada Water Act

c) The Fisheries Act

d) The Migratory Birds Convention Act

The Environment Quality Act will probably be of greatest importance, although at the present time it appears that the regulations of this act specifically exclude both Asbestos and Metal Mines. It can be expected that applicants for a Certificate of Authorization to proceed with major mine developments will be required to submit a land reclamation plan that would identify native soils and vegetation, the timing and duration of any disturbances, and details of a reclamation program.

A review of the proposed mine development suggests that the areas of concern that can be immediately identified include:

a) Surficial and groundwater regime contamination

(i) The proposed development straddles the headwaters of three drainage systems

(ii) Run-off from waste dumps can be expected to add to the suspended solid content, turbidity and total dissolved solids of the local streams and lakes

(iii) Run-off and seepage from the tailings ponds can be expected to infiltrate both surficial and groundwater regimes.

b) Dump aesthetics and reclamation

(i) There is an increasing tendency among provincial legislators to require dump treatment (contouring, revegetation) contemporaneous with mining, which has profound implications for dump design and scheduling

(ii) The nature of the local climate and native species indicates that any form of revegetation may be difficult.

An ecological inventory of the project area should be completed shortly after any decision to proceed with a complex of the magnitude noted in the text. Timing of special studies is important with respect to such items as the growing season, migratory patterns and spawning. Seasonal variations in flow rates will influence the concerns expressed in a) above, and a systematic survey of local vegetation would be necessary to determine nutrient budgets, the microclimatic regime and to indicate forest stability and productivity.

ADDENDUM

OUTLYING DEPOSITS

Sudbury Hill - Roach Hill - Roach Hill Extension

INTRODUCTION

Preliminary geological data exists concerning three iron ore showings to the north west of the mining complex discussed in the foregoing text, and their locations are shown on Plate 4. The approximate distances to the plant location are noted below.

DEPOSIT	DISTANCE TO PLANT (miles)
Sudbury Hill	2.4
Roach Hill	4.8
Roach Hill Extension	5.9

These distances are those that may be assumed to apply upon completion of the tailings complex of the Bloom Lake reserves, rather than the immediate plan distance.

Initial commentary concerning the economic potential of these showings, with respect to the proposed mine is required to complement the main study.

METHODOLOGY

The data base for the preliminary evaluation of these deposits is contained in the following plans and sections, of Quebec Cobalt and Exploration Limited, prepared by H. E. Neal and Associates.

a) Sudbury Hill Property, Geological Plan; scale 1"=200'

b) Sudbury Hill, Sections 13, 14; 15, 16; scale 1"=50'

c) Roach Hill Property and Extension, Geological Plan; scale 1"=400'

d) Roach Hill, Sections RHE 1; 2, 4; 3, 5; 6, 8, 10; 7; 9; and RH 11, 12; scale 1"=50'

The interpretations presented on these drawings may be regarded as quite preliminary in nature, and a number of assumptions are required in order to allow calculations of potential reserves, thus;

a) Roach Hill has been described as a "synclinal remnant", and its volume estimated using the formula:- area of surface expression times one third of the indicated depth of deposit

b) Roach Hill Extension has reserves compiled by assigning areas of influence to its component sections from 700'-2900' apart. Further, the oxide facies have been assumed to persist to a depth of 100' below the lowest point indicated by diamond drilling, on a given section

c) Ore grades reported are simply the averages of those recorded on the individual sections.

In the case of the Sudbury and Roach Hill deposits only minimal amounts of stripping would be required, and no problems could be anticipated from slope stability given the configuration and depth of any final pit applicable here. The slope angles selected for a potential Roach Hill Extension pit is 50° for the waste wall, and an angle conformable to the ore footwall for the other main slope.

Some immediate operating and capital cost differentials may be identified, based primarily on an added haulage cost and assumed road and power requirements.

These considerations are summarized in the table below:

COST INCREMENTS

	OPERATING	CAPITAL*	
	Haulage @ 8¢ ton mile	Powerline @ $10/ft	Road @ $40/ft
DEPOSIT	¢/ton	$	$
Sudbury Hill	19	127,000	507,000
Roach Hill	38	254,000	1,014,000
Roach Hill Extension	47	312,000	1,246,000

*Estimates to service each deposit as discrete entity, common usage has not been assumed.

The effect of these amounts on the C.O.S.R. for the Roach Hill Extension will result in the approximation

$$\text{C.O.S.R.} = \frac{10.60 - (6.55 + 0.47)}{0.90} = 4.0$$

since a distribution of the capital amounts has a negligible influence, and the difference in concentration ratio with respect to the Bloom Lake reserves has been neglected. The reserves indicated for the Roach Hill Extension are all contained within pit limits established by this cut-off of 4.0 tons of waste per ton of ore.

RESERVES AVAILABLE

The results of the deliberations described above are tabulated below:

Deposit	Indicated Ore Tons x 10^6	Grade %Fe	STRIPPING Overburden Tons	Waste Rock Tons	O.S.R. Total waste/Ore
Sudbury Hill	4.5	25.7	0	0	0
Roach Hill	3.3	28.1	0	0	0
Roach Hill Extension	61.0	24.5	12.4	52.6	1.1

DISCUSSION OF FINDINGS

Without full access to the corporate strategy of a potential owner of the reserves in question, it is inappropriate to offer firm conclusions regarding the results of this study. However, a number of observations may be made, based upon two key concerns.

a) How can the outlying deposits complement the main Bloom Lake reserves?

b) What would be the significance of releasing the claims involved?

The main report shows that in addition to the reserves available in the proposed pit, over 350 million tons of iron ore are indicated immediately to the west, the grade of which is higher than that of the outlying deposits. Thus, should access to additional tonnage be required during or after the 30 year life of the main zone, then such access could readily be made available from this source. At the sort of production rate postulated, the reserves of Bloom Lake apparently exceed 50 years supply, which is a period greater than that normally considered for strategic planning purposes.

There is no evidence that the outlying deposits offer any advantage from a blending point of view, and, it is suggested that the reserves available in Sudbury Hill and Roach Hill are too small to warrant further analysis in

conjunction with a project of this scope.

Significant quantities of overburden, albeit of unknown quality, overlie the Roach Hill Extension, and some of this material may be of use for tailings impoundment construction. Such features may be required within two miles of this resource.

A prospective operator could be vulnerable to opportunities seized by outside parties, should a decision be made to release the claims covering, in particular, the Roach Hill Extension. Thus, while the tonnage and grade are not particularly attractive to a party holding the main Bloom Lake reserves, a case could be made for a small scale contracted operation that would aim to feed any local large scale producer with supplementary ore. Such an operation could have a high nuisance value both in its conceptual/ permit stage and in actual operation.

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An Investigation of

THE RECOVERY OF IRON

from samples

submitted by

QUEBEC COBALT AND EXPLORATION LIMITED

Progress Report No. 1

(per H.E. Neal and Associates)

Project No. L.R. 1873

Ministère des Richesses Naturelles, Québec
SERVICE DE LA
DOCUMENTATION TECHNIQUE
6 DEC 1976
Date:
No GM: 32215

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
February 10, 1976

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INTRODUCTION

Instructions to proceed with the preparation of Quebec Cobalt and Exploration Company Limited Composite samples from drill core rejects stored at Lakefield, were contained in a letter dated September 8, 1975 from Mr. H.E. Neal.

In discussions following the initiation of the test program, a detailed flowsheet for the testing of all composites was prepared. Exceptions to the flowsheet procedure were made for samples low in magnetite.

Phase I of the testwork consisted of heavy liquid separation at 6 mesh size to determine the liberation size. Phase II consisted of tabling at -35 mesh, low-intensity magnetic separation in a Jeffrey separator, and high-intensity magnetic separation in a Jones separator.

LAKEFIELD RESEARCH OF CANADA LIMITED

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

D.M. Wyslouzil, P. Eng.,
Chief Metallurgist

Investigation by: R.G. Williamson
J.S. Mason

S U M M A R Y

1. Tabling and Magnetic Separation

Head analyses and metallurgical results obtained on 21 composite samples are shown in Table No. 1.

Table No. 1 - Metallurgical Results

Composite QC-1	Head		Table Concentrate			Table + Mag. Conc. (1)		
	% Sol. Fe	% Mag. Fe	Weight %	Assay % Sol. Fe	% Dist. Sol. Fe	Weight %	Assay % Sol. Fe	% Dist. Sol. Fe
1	28.2	2.60	33.1	68.2	80.0	-	-	-
2	27.2	2.61	31.9	67.3	79.1	-	-	-
3	36.3	27.9	47.2	65.5	84.6	53.1	65.8	95.6
4	26.3	14.1	29.9	67.7	77.0	34.7	68.2	90.0
5	26.2	4.62	35.6	66.5	90.5	-	-	-
6	32.4	0.44	37.8	67.6	78.9	-	-	-
7	30.8	0.25	36.0	68.3	79.8	-	-	-
8	31.0	2.42	37.9	67.7	82.8	-	-	-
9	31.2	0.26	40.1	67.7	87.0	-	-	-
10	27.0	0.24	34.5	67.6	86.3	-	-	-
11	11.8	9.43	10.0	69.8	59.5	13.6	70.0	81.1
12	34.6	19.4	47.7	64.2	88.5	51.7	64.6	95.5
13	38.8	34.9	45.2	66.0	76.9	56.6	67.7	97.3
14	29.5	5.69	40.3	65.4	89.4	-	-	-
15	25.9	4.44	34.7	66.2	88.8	-	-	-
16	13.1	8.98	16.1	66.6	81.7	17.9	67.1	91.5
17	17.1	11.1	22.4	69.6	88.5	23.5	69.7	93.1
18	22.5	20.4	24.5	65.2	71.0	31.1	65.9	91.1
19	32.9	0.64	39.8	68.1	82.4	-	-	-
20	31.8	0.62	37.9	67.9	80.8	-	-	-
21	35.0	0.59	41.8	68.4	81.7	-	-	-

(1) D.T. concentrate after regrinding of Jeffrey concentrate.

Summary - Continued

2. Analysis of Table Concentrates

A composite was prepared from equal weights of QC-75-1 to 10, 12 to 15 and 18 to 21. At the time the composite was prepared, concentrates from composites 11, 16 and 17 were not available.

Element		Content
Soluble Iron	(Fe)	67.4 %
Silica	(SiO_2)	3.57 %
Alumina	(Al_2O_3)	0.20 %
Lime	(CaO)	0.25 %
Magnesia	(MgO)	0.29 %
Titania	(TiO_2)	0.11 %
Manganese	(Mn)	0.051 %
Phosphorus	(P)	0.015 %
Sulphur	(S)	0.013 %
Soda	(Na_2O)	0.013 %
Potash	(K_2O)	0.007 %

Semi-quantitative Spectrographic Analysis

Aluminum (Al_2O_3)	0.3 %	Manganese	0.01 %
Antimony	-	Magnesium (MgO)	0.05 %
Arsenic	-	Molybdenum	0.005 %
Barium	-	Neodymium (Nd_2O_3)	-
Beryllium (BeO)	-	Nickel	< 0.002 %
Bismuth	-	Phosphorus	-
Boron	-	Silver	-
Calcium (CaO)	< 0.005 %	Silicon (SiO_2)	5 %
Cadmium	-	Sodium (Na_2O)	-
Cerium (CeO_2)	-	Strontium	-
Chromium	< 0.01 %	Tantalum (Ta_2O_5)	-
Cobalt	-	Thorium (ThO_2)	-
Columbium (Cb_2O_5)	-	Tin	-
Copper	0.002 %	Titanium	-
Gallium	-	Tungsten	-
Germanium	-	Uranium (U_3O_8)	-
Iron (Fe)	H	Vanadium	-
Lanthanum (La_2O_3)	-	Yttrium (Y_2O_3)	-
Lead	< 0.005 %	Zinc	-
Lithium (Li_2O)	-	Zirconium (ZrO_2)	-

Figures are approximate:

CODE H - high - 10 - 100 % approx.
M - medium - 1 - 10 % approx.
L - low - .1 - 1 % approx.

- Not Detected-Elements looked for but not found
X Not looked for
< Less Than

Summary - Continued

3. Special Tests

A composite sample was prepared from QC-75-1 and 2 for tabling and HIMS tests using a Jones separator. Objective of the table test was to simulate a spiral test to reject more iron to the tailing.

The table concentrate assayed 68.1 % Fe and contained 63 % of the iron. By low-intensity magnetic separation of the table tailing a concentrate assaying 52.6 % Fe was recovered, representing 9.3 % of the total iron. These two products combined would result in concentrate assaying 65.6 % Fe with a recovery of 72.3 % Fe.

The tailing from the magnetic separation then was treated in the Jones HIMS at 2 amperes and at 10 amperes, after screening and after regrinding, respectively. Only low-grade products were produced, between 39 % and 60 % Fe. The low grade was due to the abundance of gangue, containing minute grains of magnetite or hematite, that was recovered at the high magnetic flux density. There were some indications that by the careful selection of the variables, such as pulp density, length of feed cycle, water pressures and flux densities, better results could be obtained even without regrinding.

4. Jones Separation Tests

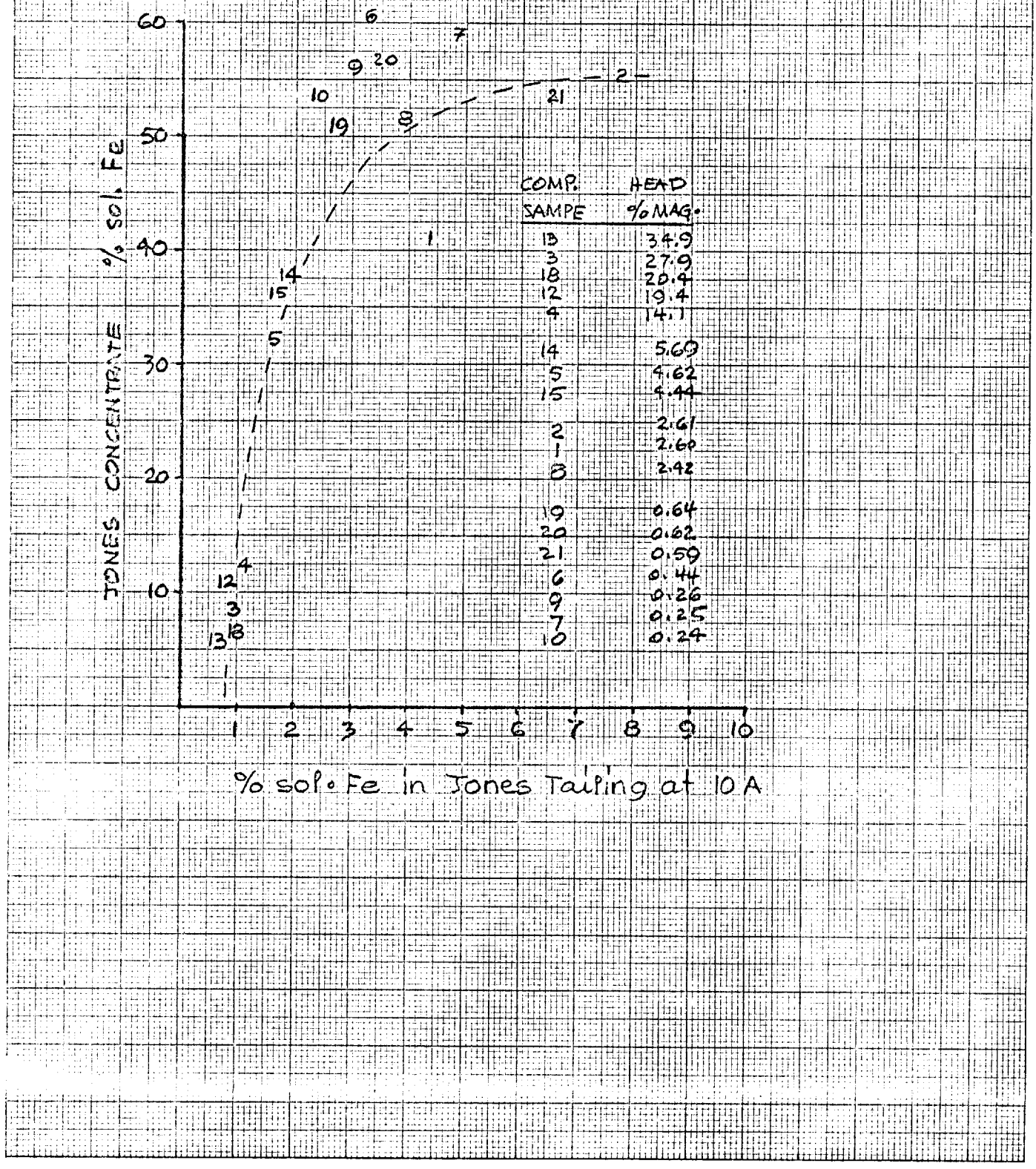
Standard separation tests at 10 amperes were performed on all composite samples, except No. 11, 16 and 17. The results indicated that from some samples concentrates assaying over 50 % iron could be obtained in a single pass, but from other samples the concentrate assayed only about 10 % Fe or 35 % Fe. By plotting the concentrate grades against the tailings assays, three groups could be isolated, as shown in Figure 1.

Summary - Continued

4. Jones Separation Tests - Continued

The lowest-grade Jones concentrates were produced with samples high in magnetite. Because of the fine dispersion of magnetite in quartz, the Jones concentrate consisted primarily of quartz-magnetite middling particles.

FIG. 1 JONES CONC-GRADE — MAG. CONTENT IN HEAD



COMP. SAMPE	HEAD % MAG.
13	34.9
3	27.9
18	20.4
12	19.4
4	14.1
14	5.69
5	4.62
15	4.44
2	2.61
1	2.60
8	2.42
19	0.64
20	0.62
21	0.59
6	0.44
9	0.26
7	0.25
10	0.24

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Summary - Continued

5. Production of a Super-Concentrate

A composite sample of table concentrates was ground in a ball mill to 78 % minus 325 mesh, and then was passed through a Jeffrey drum separator. The concentrate was collected and the tailing was passed through the Jones separator at 10 amperes. Concentrate, middling and non-magnetic tailings were collected separately for weight and assay.

Table No. 2 - Metallurgical Results

Product	Weight	Assays, %			% Distribution		
	%	Fe (Sol.)	SiO_2	Al_2O_3	Fe (Sol.)	SiO_2	Al_2O_3
Jeffrey Conc.	22.6	71.7	1.20	0.10	23.8	8.4	12.8
Jones Conc.	62.3	69.1	1.46	0.21	63.1	28.0	74.3
Middling	8.0	61.4	8.80	0.15	7.2	21.7	6.8
Tailing	7.4	57.1	19.2	0.15	5.9	41.9	6.1
Head (Calc.)	100.0	68.2	3.25	0.18	100.0	100.0	100.0
Combined Conc.	84.9	69.8	1.39	0.18	86.9	36.4	87.1

6. Heavy Liquid Separations

Separation tests on individual size fractions were performed with a liquid having a specific gravity of 3.3.

Samples containing predominantly hematite had good liberation at 35 mesh, whereas samples containing mainly magnetite did not yield high-grade sink products at any particle size.

SAMPLE PREPARATION

Composite samples were prepared from the minus 10 mesh rejects stored at Lakefield from the 1971-72 drilling program. The minus 10 mesh portion represented a volumetric aliquot of the samples received.

One-half of each reject was removed by riffling and composited as requested. From the final composites 10 kg charges were prepared for testwork, and size fractions were prepared for heavy-liquid separation tests.

Composites: 1971-72 Drilling

Composite QC-75	Hole No.	Cross Section	Interial Feet From	To	Sample Length feet	Head Assays (Calc.) % Sol. Fe	% Mag. Fe	Head Assay (from tests) % Sol. Fe	% Mag. Fe	Rock Type *
1	71-1	66+00W	367	700	333	27.29	2.49	28.2	2.60	Sp.
2	71-1	66+00W	700	1000	300	27.95	2.52	27.2	2.61	Sp.
3	71-2	56+00W	14	360	346	36.43	27.97	36.3	27.9	Mag. Sp.
4	71-2	56+00W	360	700	340	25.82	13.75	26.3	14.1	Mag. Sp.
5	71-2	56+00W	700	1000	300	26.46	4.12	26.2	4.62	Sp.
6	71-3	164+00W	13	290	278	31.04	0.36	32.4	0.44	Fr. Sp.
7	71-6	150+00W	20	300	280	29.63	0.43	30.8	0.25	Fr. Sp.
8	71-8	150+00W	50.5	400	349.5	30.44	2.74	31.0	2.42	Sp.
9	71-8	150+00W	400	750	350	31.70	0.37	31.2	0.26	Sp.
10	71-9	150+00W	20	264.1	244.1	26.34	0.34	27.0	0.24	Sp.
11	72-2	10+00SW	275.3	395	119.7	10.21	8.55	11.8	9.43	Sil.
12	72-2	10+00SW	395	645	250	33.33	20.43	34.6	19.4	Mag. Sp.
13	72-2	10+00SW	645	885	240	37.35	34.18	38.8	34.9	Mag.
14	72-2	10+00SW	885	1085	200	27.25	4.89	29.5	5.69	Sp.
15	72-2	10+00SW	1085	1294	209	25.68	5.28	25.9	4.44	Sp.
16	72-3	30+00W	15	225	210	7.43	5.18	13.1	8.98	Sil.
17	72-3	30+00W	225	435.7	210.7	16.25	10.27	17.7	11.1	Sil.
18	72-5	40+00W	52.8	338	285.2	21.49	19.75	22.5	20.4	Sil.
19	72-9	155+00W	480	720	240	30.11	0.49	32.9	0.64	Sp.
20	72-9	155+00W	720	960	240	29.23	0.39	31.8	0.62	Sp.
21	72-10	175+00W	28	208	177	30.58	0.44	35.0	0.59	Fr. Sp.

* Sp. - Specularite
Mag. - Magnetite
Fr. - Friable
Sil. - Silicate

Sample Preparation - Continued

Composite No. QC-75-1

Hole No.	Sample No.	Sample Interval	Sample Length
71-1	Q10201	367 - 380	13
	Q10202	380 - 390	10
	Q10203	390 - 400	10
	Q10204	400 - 410	10
	Q10205	410 - 420	10
	Q10206	420 - 430	10
	Q10207	430 - 440	10
	Q10208	440 - 450	10
	Q10209	450 - 460	10
	Q10210	460 - 470	10
	Q10211	470 - 480	10
	Q10212	480 - 490	10
	Q10213	490 - 500	10
	Q10214	500 - 505	5
	Q10215	530 - 540	10
	Q10216	540 - 550	10
	Q10217	550 - 560	10
	Q10218	560 - 570	10
	Q10219	570 - 580	10
	Q10220	580 - 590	10
	Q10221	590 - 600	10
	Q10222	600 - 610	10
	Q10223	610 - 620	10
	Q10224	620 - 630	10
	Q10225	630 - 640	10
	Q10226	640 - 650	10
	Q10227	650 - 660	10
	Q10228	660 - 670	10
	Q10229	670 - 680	10
	Q10230	680 - 690	10
	Q10231	690 - 700	10

Total Length = 308 feet

Sample Preparation - Continued

Composite No. QC-75-2

Hole No.	Sample No.	Sample Interval	Sample Length
71-1	Q10232	700 - 710	10
	Q10233	710 - 720	10
	Q10234	720 - 730	10
	Q10235	730 - 740	10
	Q10236	740 - 750	10
	Q10237	750 - 760	10
	Q10238	760 - 770	10
	Q10239	770 - 780	10
	Q10240	780 - 790	10
	Q10241	790 - 800	10
	Q10242	800 - 810	10
	Q10243	810 - 820	10
	Q10244	820 - 830	10
	Q10245	830 - 840	10
	Q10246	840 - 850	10
	Q10247	850 - 860	10
	Q10248	860 - 870	10
	Q10249	870 - 880	10
	Q10250	880 - 890	10
	Q10251	890 - 900	10
	Q10252	900 - 910	10
	Q10253	910 - 920	10
	Q10254	920 - 930	10
	Q10255	930 - 940	10
	Q10256	940 - 950	10
	Q10257	950 - 960	10
	Q10258	960 - 970	10
	Q10259	970 - 980	10
	Q10260	980 - 990	10
	Q10261	990 - 1000	10

Total Length = 300 feet

Sample Preparation - Continued

Composite No. QC-75-3

Hole No.	Sample No.	Sample Interval	Sample Length
71-2	Q10101	14 - 20	6
	Q10102	20 - 30	10
	Q10103	30 - 40	10
	Q10104	40 - 50	10
	Q10105	50 - 60	10
	Q10106	60 - 70	10
	Q10107	70 - 80	10
	Q10108	80 - 90	10
	Q10109	90 - 100	10
	Q10110	100 - 110	10
	Q10111	110 - 120	10
	Q10112	120 - 130	10
	Q10113	130 - 140	10
	Q10114	140 - 150	10
	Q10115	150 - 160	10
	Q10116	160 - 170	10
	Q10117	170 - 180	10
	Q10118	180 - 190	10
	Q10119	190 - 200	10
	Q10120	200 - 210	10
	Q10121	210 - 220	10
	Q10122	220 - 230	10
	Q10123	230 - 240	10
	Q10124	240 - 250	10
	Q10125	250 - 260	10
	Q10126	260 - 270	10
	Q10127	270 - 280	10
	Q10128	280 - 290	10
	Q10129	290 - 300	10
	Q10130	300 - 310	10
	Q10131	310 - 320	10
	Q10132	320 - 330	10
	Q10133	330 - 340	10
	Q10134	340 - 350	10
	Q10135	350 - 360	10

Total Length = 346 feet

Sample Preparation - Continued

Composite No. QC-75-4

Hole No.	Sample No.	Sample Interval	Sample Length
71-2	Q10136	360 - 370	10
	Q10137	370 - 380	10
	Q10138	380 - 390	10
	Q10139	390 - 400	10
	Q10140	400 - 410	10
	Q10141	410 - 420	10
	Q10142	420 - 430	10
	Q10143	430 - 440	10
	Q10144	440 - 450	10
	Q10145	450 - 460	10
	Q10146	460 - 470	10
	Q10147	470 - 480	10
	Q10148	480 - 490	10
	Q10149	490 - 500	10
	Q10150	500 - 510	10
	Q10151	510 - 520	10
	Q10152	520 - 530	10
	Q10153	530 - 540	10
	Q10154	540 - 550	10
	Q10155	550 - 560	10
	Q10156	560 - 570	10
	Q10157	570 - 580	10
	Q10158	580 - 590	10
	Q10159	590 - 600	10
	Q10160	600 - 610	10
	Q10161	610 - 620	10
	Q10162	620 - 630	10
	Q10163	630 - 640	10
	Q10164	640 - 650	10
	Q10165	650 - 660	10
	Q10166	660 - 670	10
	Q10167	670 - 680	10
	Q10168	680 - 690	10
	Q10169	690 - 700	10

Total Length = 340 feet

Sample Preparation - Continued

Composite No. QC-75-5

Hole No.	Sample No.	Sample Interval	Sample Length
71-2	Q10170	700 - 710	10
	Q10171	710 - 720	10
	Q10172	720 - 730	10
	Q10173	730 - 740	10
	Q10174	740 - 750	10
	Q10175	750 - 760	10
	Q10176	760 - 770	10
	Q10177	770 - 780	10
	Q10178	780 - 790	10
	Q10179	790 - 800	10
	Q10180	800 - 810	10
	Q10181	810 - 820	10
	Q10182	820 - 830	10
	Q10183	830 - 840	10
	Q10184	840 - 850	10
	Q10185	850 - 860	10
	Q10186	860 - 870	10
	Q10187	870 - 880	10
	Q10188	880 - 890	10
	Q10189	890 - 900	10
	Q10190	900 - 910	10
	Q10191	910 - 920	10
	Q10192	920 - 930	10
	Q10193	930 - 940	10
	Q10194	940 - 950	10
	Q10195	950 - 960	10
	Q10196	960 - 970	10
	Q10197	970 - 980	10
	Q10198	980 - 990	10
	Q10199	990 - 1000	10

Total Length = 300 feet

Sample Preparation - Continued

Composite No. QC-75-6

Hole No.	Sample No.	Sample Interval	Sample Length
71-3	Q 1	13 - 20	7
	Q 2	20 - 30	10
	Q 3	30 - 40	10
	Q 4	40 - 50	10
	Q 5	50 - 57	7
	Q 6	60 - 70	10
	Q 7	70 - 80	10
	Q 8	80 - 90	10
	Q 9	90 - 100	10
	Q10	100 - 110	10
	Q11	110 - 120	10
	Q12	120 - 130	10
	Q13	130 - 140	10
	Q14	140 - 150	10
	Q15	150 - 160	10
	Q16	160 - 170	10
	Q17	170 - 180	10
	Q18	180 - 190	10
	Q19	190 - 200	10
	Q20	200 - 210	10
	Q21	210 - 220	10
	Q22	220 - 230	10
	Q23	230 - 240	10
	Q24	240 - 250	10
	Q25	250 - 260	10
	Q26	260 - 270	10
	Q27	270 - 280	10
	Q28	280 - 290	10

Total Length = 274 feet

Sample Preparation - Continued

Composite No. QC-75-7

Hole No.	Sample No.	Sample Interval	Sample Length
71-6	Q35	20 - 30	10
	Q36	30 - 40	10
	Q37	40 - 50	10
	Q38	50 - 60	10
	Q39	60 - 70	10
	Q40	70 - 80	10
	Q41	80 - 90	10
	Q42	90 - 100	10
	Q43	100 - 110	10
	Q44	110 - 120	10
	Q45	120 - 130	10
	Q46	130 - 140	10
	Q47	140 - 150	10
	Q48	150 - 160	10
	Q49	160 - 170	10
	Q50	170 - 180	10
	Q51	180 - 190	10
	Q52	190 - 200	10
	Q53	200 - 210	10
	Q54	210 - 220	10
	Q55	220 - 230	10
	Q56	230 - 240	10
	Q57	240 - 250	10
	Q58	250 - 260	10
	Q59	260 - 270	10
	Q60	270 - 280	10
	Q61	280 - 290	10
	Q62	290 - 300	10

Total Length = 280 feet

Sample Preparation - Continued

Composite No. QC-75-8

Hole No.	Sample No.	Sample Interval	Sample Length
71-8	Q77	50.5 - 60	9.5
	Q78	60 - 70	10
	Q79	70 - 80	10
	Q80	80 - 92	12
	Q81	94.3 - 101.6	7.3
	Q82	104.8 - 115	10.2
	Q83	115 - 125	10
	Q84	125 - 135	10
	Q85	135 - 145	10
	Q86	145 - 155	10
	Q87	155 - 165	10
	Q88	165 - 175	10
	Q89	175 - 185	10
	Q90	185 - 195	10
	Q91	195 - 205	10
	Q92	205 - 214.7	9.7
	Q93	214.7 - 221.3	6.6
	Q94	221.3 - 230	8.7
	Q95	230 - 240	10
	Q96	240 - 250	10
	Q97	250 - 260	10
	Q98	260 - 270	10
	Q99	270 - 280	10
	Q100	280 - 290	10
	Q101	290 - 300	10
	Q102	300 - 310	10
	Q103	310 - 320	10
	Q104	320 - 330	10
	Q105	330 - 344	14
	Q106	344 - 350	6.0
	Q107	350 - 360	10
	Q108	360 - 370	10
	Q109	370 - 383	13
	Q110	383 - 390	7.0
	Q111	390 - 400	10

Total Length = 344 feet

Sample Preparation - Continued

Composite No. QC-75-9

Hole No.	Sample No.	Sample Interval	Sample Length
71-8	Q112	400 - 410	10
	Q113	410 - 420	10
	Q114	420 - 430	10
	Q115	430 - 440	10
	Q116	440 - 450	10
	Q117	450 - 460	10
	Q118	460 - 470	10
	Q119	470 - 480	10
	Q120	480 - 490	10
	Q121	490 - 500	10
	Q122	500 - 510	10
	Q123	510 - 520	10
	Q124	520 - 530	10
	Q125	530 - 540	10
	Q126	540 - 550	10
	Q127	550 - 560	10
	Q128	560 - 570	10
	Q129	570 - 580	10
	Q130	580 - 590	10
	Q131	590 - 600	10
	Q132	600 - 610	10
	Q133	610 - 620	10
	Q134	620 - 630	10
	Q135	630 - 640	10
	Q136	640 - 650	10
	Q137	650 - 660	10
	Q138	660 - 670	10
	Q139	670 - 680	10
	Q140	680 - 690	10
	Q141	690 - 700	10
	Q142	700 - 710	10
	Q143	710 - 720	10
	Q144	720 - 730	10
	Q145	730 - 740	10
	Q146	740 - 750	10

Total Length = 350 feet

Sample Preparation - Continued

Composite No. QC-75-10

Hole No.	Sample No.	Sample Interval	Sample Length
71-9	Q147	20 - 27.5	7.5
	Q148	27.5 - 35	8.5
	Q149	35 - 45	10
	Q150	45 - 56	11
	Q151	56 - 65	9
	Q152	65 - 75	10
	Q153	75 - 88.5	13.5
	Q154	88.5 - 95	6.5
	Q155	95 - 105	10
	Q156	105 - 115	10
	Q157	115 - 125	10
	Q158	125 - 135	10
	Q159	135 - 145	10
	Q160	145 - 155	10
	Q161	155 - 166.5	11.5
	Q162	166.5 - 175	8.5
	Q163	175 - 186.7	11.7
	Q164	186.7 - 200	13.3
	Q165	200 - 215	15
	Q166	215 - 225	10
	Q167	225 - 240	15
	Q168	240 - 250	10
	Q169	250 - 264.1	14.1

Total Length = 244.1 feet

Sample Preparation - Continued

Composite No. QC-75-11

Hole No.	Sample No.	Sample Interval	Sample Length
72-2	QC 1	275.3 - 285	9.7
	QC 2	285 - 295	10
	QC 3	295 - 305	10
	QC 4	305 - 315	10
	QC 5	315 - 325	10
	QC 6	325 - 335	10
	QC 7	335 - 345	10
	QC 8	345 - 355	10
	QC 9	355 - 365	10
	QC 10	365 - 375	10
	QC 11	375 - 385	10
	QC 12	385 - 395	10

Total Length = 119.7 feet

Sample Preparation - Continued

Composite No. QC-75-12

Hole No.	Sample No.	Sample Interval	Sample Length
72-2	QC 13	395 - 405	10
	QC 14	405 - 415	10
	QC 15	415 - 425	10
	QC 16	425 - 435	10
	QC 17	435 - 445	10
	QC 18	445 - 455	10
	QC 19	455 - 465	10
	QC 20	465 - 475	10
	QC 21	475 - 485	10
	QC 22	485 - 495	10
	QC 23	495 - 505	10
	QC 24	505 - 515	10
	QC 25	515 - 525	10
	QC 26	525 - 535	10
	QC 27	535 - 545	10
	QC 28	545 - 555	10
	QC 29	555 - 565	10
	QC 30	565 - 575	10
	QC 31	575 - 585	10
	QC 32	585 - 595	10
	QC 33	595 - 605	10
	QC 34	605 - 615	10
	QC 35	615 - 625	10
	QC 36	625 - 635	10
	QC 37	635 - 645	10

Total Length = 250 feet

Sample Preparation - Continued

Composite No. QC-75-13

Hole No.	Sample No.	Sample Interval	Sample Length
72-2	QC 38	645 - 655	10
	QC 39	655 - 665	10
	QC 40	665 - 675	10
	QC 41	675 - 685	10
	QC 42	685 - 695	10
	QC 43	695 - 705	10
	QC 44	705 - 715	10
	QC 45	715 - 725	10
	QC 46	725 - 735	10
	QC 47	735 - 745	10
	QC 48	745 - 755	10
	QC 49	755 - 765	10
	QC 50	765 - 775	10
	QC 51	775 - 785	10
	QC 52	785 - 795	10
	QC 53	795 - 805	10
	QC 54	805 - 815	10
	QC 55	815 - 825	10
	QC 56	825 - 835	10
	QC 57	835 - 845	10
	QC 58	845 - 855	10
	QC 59	855 - 865	10
	QC 60	865 - 875	10
	QC 61	875 - 885	10

Total Length = 240 feet

Sample Preparation - Continued

Composite No. QC-75-14

Hole No.	Sample No.	Sample Interval	Sample Length
72-2	QC 62	885 - 895	10
	QC 63	895 - 905	10
	QC 64	905 - 915	10
	QC 65	915 - 925	10
	QC 66	925 - 935	10
	QC 67	935 - 945	10
	QC 68	945 - 955	10
	QC 69	955 - 965	10
	QC 70	965 - 975	10
	QC 71	975 - 985	10
	QC 72	985 - 995	10
	QC 73	995 - 1005	10
	QC 74	1005 - 1015	10
	QC 75	1015 - 1025	10
	QC 76	1025 - 1035	10
	QC 77	1035 - 1045	10
	QC 78	1045 - 1055	10
	QC 79	1055 - 1065	10
	QC 80	1065 - 1075	10
	QC 81	1075 - 1085	10

Total Length = 200 feet

Sample Preparation - Continued

Composite No. QC-75-15

Hole No.	Sample No.	Sample Interval	Sample Length
72-2	QC 82	1085 - 1095	10
	QC 83	1095 - 1105	10
	QC 84	1105 - 1115	10
	QC 85	1115 - 1125	10
	QC 86	1125 - 1135	10
	QC 87	1135 - 1145	10
	QC 88	1145 - 1155	10
	QC 89	1155 - 1165	10
	QC 90	1165 - 1175	10
	QC 91	1175 - 1185	10
	QC 92	1185 - 1195	10
	QC 93	1195 - 1205	10
	QC 94	1205 - 1215	10
	QC 95	1215 - 1225	10
	QC 96	1225 - 1235	10
	QC 97	1235 - 1245	10
	QC 98	1245 - 1255	10
	QC 99	1255 - 1265	10
	QC 100	1265 - 1275	10
	QC 101	1275 - 1285	10
	QC 102	1285 - 1294	9

Total Length = 209 feet

Sample Preparation - Continued

Composite No. QC-75-16

Hole No.	Sample No.	Sample Interval	Sample Length
72-3	QC 103	15 - 25	10
	QC 104	45 - 55	10
	QC 105	117 - 125	8
	QC 106	125 - 135	10
	QC 107	135 - 145	10
	QC 108	145 - 155	10
	QC 109	155 - 165	10
	QC 110	165 - 175	10
	QC 111	175 - 185	10
	QC 112	185 - 195	10
	QC 113	195 - 205	10
	QC 114	205 - 215	10
	QC 115	215 - 225	10

Total Length = 128 feet

Sample Preparation - Continued

Composite No. QC-75-17

Hole No.	Sample No.	Sample Interval	Sample Length
72-3	QC 116	225 - 235	10
	QC 117	235 - 245	10
	QC 118	245 - 255	10
	QC 119	255 - 265	10
	QC 120	265 - 275	10
	QC 121	275 - 285	10
	QC 122	285 - 295	10
	QC 123	295 - 305	10
	QC 124	305 - 315	10
	QC 125	315 - 325	10
	QC 126	325 - 335	10
	QC 127	335 - 345	10
	QC 128	345 - 355	10
	QC 129	355 - 365	10
	QC 130	365 - 375	10
	QC 131	375 - 385	10
	QC 132	385 - 395	10
	QC 133	395 - 405	10
	QC 134	405 - 415	10
	QC 135	415 - 425	10
	QC 136	425 - 435.7	10.7

Total Length = 210.7 feet

Sample Preparation - Continued

Composite No. QC-75-18

Hole No.	Sample No.	Sample Interval	Sample Length
72-5	QC 148	52.8 - 60	7.2
	QC 149	60 - 70	10
	QC 150	70 - 80	10
	QC 151	80 - 90	10
	QC 152	90 - 100	10
	QC 153	100 - 110	10
	QC 154	110 - 120	10
	QC 155	120 - 130	10
	QC 156	130 - 140	10
	QC 157	140 - 150	10
	QC 158	150 - 160	10
	QC 159	160 - 170	10
	QC 160	170 - 180	10
	QC 161	180 - 190	10
	QC 162	190 - 200	10
	QC 163	200 - 210	10
	QC 164	210 - 220	10
	QC 165	220 - 230	10
	QC 166	230 - 240	10
	QC 167	240 - 250	10
	QC 168	250 - 260	10
	QC 169	260 - 270	10
	QC 170	270 - 280	10
	QC 171	280 - 290	10
	QC 172	290 - 300	10
	QC 173	300 - 310	10
	QC 174	310 - 320	10
	QC 175	320 - 330	10
	QC 176	330 - 338	8

Total Length = 285.2 feet

Sample Preparation - Continued

Composite No. QC-75-19

Hole No.	Sample No.	Sample Interval	Sample Length
72-9	QC 197	480 - 490	10
	QC 198	490 - 500	10
	QC 199	500 - 510	10
	QC 200	510 - 520	10
	QC 201	520 - 530	10
	QC 202	530 - 540	10
	QC 203	540 - 550	10
	QC 204	550 - 560	10
	QC 205	560 - 570	10
	QC 206	570 - 580	10
	QC 207	580 - 590	10
	QC 208	590 - 600	10
	QC 209	600 - 610	10
	QC 210	610 - 620	10
	QC 211	620 - 630	10
	QC 212	630 - 640	10
	QC 213	640 - 650	10
	QC 214	650 - 660	10
	QC 215	660 - 670	10
	QC 216	670 - 680	10
	QC 217	680 - 690	10
	QC 218	690 - 700	10
	QC 219	700 - 710	10
	QC 220	710 - 720	10

Total Length = 240 feet

Sample Preparation - Continued

Composite No. QC-75-20

Hole No.	Sample No.	Sample Interval	Sample Length
72-9	QC 221	720 - 730	10
	QC 222	730 - 740	10
	QC 223	740 - 750	10
	QC 224	750 - 760	10
	QC 225	760 - 770	10
	QC 226	770 - 780	10
	QC 227	780 - 790	10
	QC 228	790 - 800	10
	QC 229	800 - 810	10
	QC 230	810 - 820	10
	QC 231	820 - 830	10
	QC 232	830 - 840	10
	QC 233	840 - 850	10
	QC 234	850 - 860	10
	QC 235	860 - 870	10
	QC 236	870 - 880	10
	QC 237	880 - 890	10
	QC 238	890 - 900	10
	QC 239	900 - 910	10
	QC 240	910 - 920	10
	QC 241	920 - 930	10
	QC 242	930 - 940	10
	QC 243	940 - 950	10
	QC 244	950 - 960	10

Total Length = 240 feet

Sample Preparation - Continued

Composite No. QC-75-21

Hole No.	Sample No.	Sample Interval	Sample Length
72-10	Core		
	QC 268	28 - 32	4
	QC 269	54 - 58	14
	QC 270	135 - 139	4
	QC 271	140 - 148	8
	QC 272	148 - 157	9
	QC 273	176 - 186	10
	QC 274	186 - 192	6
	Sludge		
	QC 261	35 - 43	8
	QC 263	43 - 53	10
	QC 265	53 - 63	10
	QC 262	63 - 68	5
	QC 258	68 - 80	12
	QC 264	80 - 90	10
	QC 259	90 - 100	10
	QC 260	100 - 110	10
	QC 276	110 - 120	10
	QC 279	110 - 120	10
	QC 284	120 - 130	10
	QC 277	130 - 140	10
	QC 287	140 - 148	8
	QC 275	148 - 157	9
	QC 297	157 - 167	10
	QC 288	167 - 178	11
	QC 296	178 - 188	10
	QC 278	188 - 198	10
	QC 280	198 - 208	10

Total Length : Core 55 feet
Sludge 183 feet

1. Tabling Tests & Low Intensity Magnetic Separation

Method of Testing

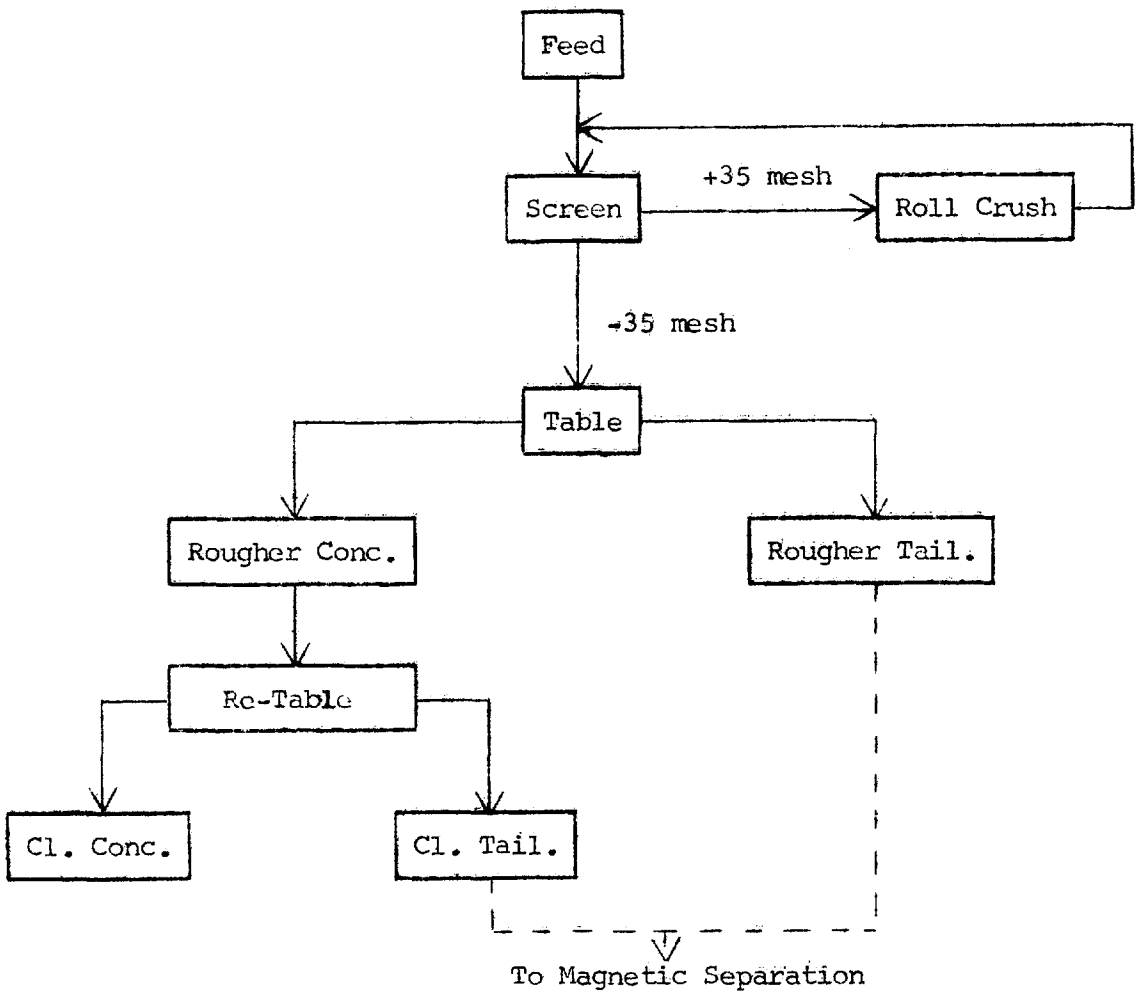
Test charges weighing 10 kg or 5 kg, respectively, were treated on a Wilfley table with a 50" by 24" sand deck. The middling fraction was continuously recirculated. The rougher concentrate was collected and repassed again. All products were filtered, dried and assayed.

Table rougher and cleaner tailings were then combined in the proper proportion and treated in the Jeffrey low-intensity magnetic separator, or the Jones high-intensity magnetic separator.

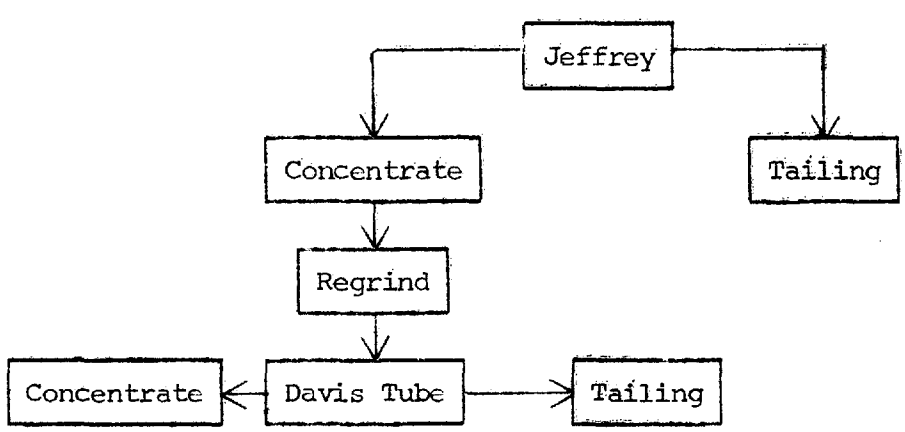
Davis tube tests were performed on the Jeffrey concentrates after regrinding.

Tabling Tests & Low-Intensity Magnetic Separation - Continued

Basic Gravity Flowsheet



Basic Magnetic Separation Flowsheet



Composite QC-75-1

Metallurgical Results

Product	Weight %	Assays, %		% Distribution	
		Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
Table Cleaner Conc.	33.1	68.2	5.00	80.0	63.7
Table Cleaner Tail.	5.8	48.8	5.97	10.0	13.3
Table Rougher Tail.	61.1	4.60	0.98	10.0	23.0
Head (Calculated)	100.0	28.2	2.60	100.0	100.0

Calculated Grades and Recoveries

Table Rougher Conc.	38.9	65.3	5.14	90.0	77.0
Comb ned Table Tail.	66.9	8.43	1.41	20.0	36.3

Metallurgical Results - Table Cleaner Conc. from Test 24

+48 mesh	5.8	67.7	2.77	5.7	3.4
-48 + 65 mesh	25.4	68.0	3.21	25.2	17.0
-65 +100 mesh	30.7	68.4	4.14	30.7	26.6
-100 +150 mesh	17.4	68.3	5.18	17.4	18.9
-150 +200 mesh	12.2	69.3	6.68	12.4	17.0
-200 mesh	8.5	69.4	9.63	8.6	17.1
Head (Calculated)	100.0	68.4	4.78	100.0	100.0

Composite QC-75-1 - Cont.

Metallurgical Results - Table Feed to Test No. 24

Product	Weight	Assays, %		% Distribution	
	%	Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
+48 mesh	6.0	29.3	2.01	6.3	4.5
-48 +65 mesh	23.5	29.1	2.01	24.7	17.8
-65 +100 mesh	26.6	27.5	2.25	26.4	22.6
-100 +150 mesh	14.7	26.5	2.69	14.0	14.9
-150 +200 mesh	11.8	26.4	3.31	11.2	14.7
-200 mesh	17.4	27.7	3.89	17.4	25.5
Head (Calculated)	100.0	27.7	2.65	100.0	100.0

Test No. 24 - Rougher Tailing

+ 100 mesh	55.4 g	57.2 %
- 100 mesh	41.5 g	42.8 %
	96.9 g	100.0 %

Composite QC-75-2

Metallurgical Results

Product	Weight	Assays, %		% Distribution	
	%	Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
Table Cleaner Conc.	31.9	67.3	5.74	79.1	70.1
Table Cleaner Tail.	7.2	46.4	4.89	12.3	13.4
Table Rougher Tail.	60.9	3.85	0.70	8.6	16.5
Head (Calculated)	100.0	27.2	2.61	100.0	100.0

Calculated Grades and Recoveries

Table Rougher Conc.	39.1	63.5	5.58	91.4	83.5
Combined Table Tail.	68.1	8.34	1.15	20.9	29.9

Composite QC-75-3

Metallurgical Results

Product	Weight %	Assays, %		% Distribution	
		Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
1. Table Cleaner Conc.	47.2	65.5	48.90	84.6	82.6
2. Table Cleaner Tail.	2.8	31.1	27.00	2.4	2.7
3. Table Rougher Tail.	50.0	9.53	8.18	13.0	14.7
4. Head (Calculated)	100.0	36.6	27.9	100.0	100.0
5. Jeffrey Conc.	17.9	24.4	22.3	12.0	14.3
6. Jeffrey Tailing	34.9	3.64	2.46	3.4	3.1
7. Feed (Products 2+3)	52.8	10.7	9.19	15.4	17.4
8. DT Conc.	5.9	68.5	-	11.0	-
9. DT Tailing	12.0	2.75	-	1.0	-
10. Feed (Product 5)	17.9	24.4	-	12.0	-

Calculated Grades and Recoveries

Ro. Table Conc.: (Products 1 and 2)	50.0	63.6	47.7	87.0	85.3

Composite QC-75-3 - Continued

Metallurgical Results - Table Feed to Test No. 26

Product	Weight %	Assays, %		% Distribution	
		Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
+48 mesh	7.4	36.1	27.5	7.4	7.2
-48 +65 mesh	18.1	34.6	26.4	17.4	17.0
-65 +100 mesh	22.6	34.7	26.4	21.8	21.3
-100 + 150 mesh	13.7	34.8	25.8	13.3	12.6
-150 +200 mesh	12.2	36.1	26.9	12.2	11.7
-200 mesh	26.0	38.6	32.6	27.9	30.2
Head (Calculated)	100.0	36.0	28.1	100.0	100.0

Metallurgical Results - Table Concentrate from Test No. 26

1. +48 mesh	6.1	70.3	49.2	6.5	6.3
2. -48 +65 mesh	13.0	70.2	47.1	13.9	12.9
3. -65 +100 mesh	22.0	68.8	47.1	23.1	21.9
4. -100 +150 mesh	15.3	64.8	44.7	15.1	14.4
5. -150 +200 mesh	16.7	59.1	42.6	15.0	15.1
6. -200 mesh	26.9	64.3	51.8	26.4	29.4
Head (Calculated)	100.0	65.6	47.4	100.0	100.0

Calculated Grades and Recoveries

Products 1 and 2	19.1	70.3	47.8	20.4	19.2
Products 1 to 3	41.1	69.5	47.4	43.5	41.1
Products 1 to 4	56.4	68.2	46.7	58.6	55.5
Products 1 to 5	73.1	66.1	45.7	73.6	70.6

Composite No. QC-75-4

Metallurgical Results

Product	Weight	Assays, %		% Distribution	
	%	Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
1. Table Cl. Conc.	29.9	67.7	34.2	77.0	72.8
2. Table Cl. Tail.	3.9	50.7	29.0	7.5	8.0
3. Table Ro. Tail.	66.2	6.17	4.06	15.5	19.2
4. Head (Calc.)	100.0	26.3	14.1	100.0	100.0
5. Jeffrey Conc.	10.7	33.2	28.7	13.5	21.9
6. Jeffrey Tailing	59.4	4.22	1.25	9.5	5.3
7. Feed (Prod. 2+3)	70.1	8.64	5.45	23.0	27.2
8. Davis Tube Conc.	4.8	71.2	-	13.0	-
9. Davis Tube Tail.	5.9	2.20	-	0.5	-
10. Feed (Prod. 5)	10.7	33.2	-	13.5	-

Calculated Grades and Recoveries

Table Rougher Conc.	33.8	65.7	33.6	84.5	80.8

Composite No. QC-75-5

Metallurgical Results

Product	Weight %	Assays, %		% Distribution	
		Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
Table Cleaner Conc.	35.6	66.5	10.9	90.5	84.0
Table Cleaner Tail.	2.6	22.6	5.09	2.2	2.8
Table Rougher Tail.	61.8	3.07	0.98	7.3	13.2
Head (Calculated)	100.0	26.2	4.62	100.0	100.0

Calculated Grades and Recoveries

Table Rougher Conc.	38.2	63.5	10.5	92.7	86.8
Combined Table Tail.	64.4	3.87	1.15	9.5	16.0

Composite No. QC-75-6

Metallurgical Results

Table Cleaner Conc.	37.8	67.6	0.46	78.9	39.8
Table Cleaner Tail.	3.5	50.8	0.64	5.5	5.0
Table Rougher Tail.	58.7	8.58	0.41	15.6	55.2
Head (Calculated)	100.0	32.4	0.44	100.0	100.0

Calculated Grades and Recoveries

Table Rougher Conc.	41.3	66.2	0.47	84.4	44.8
Combined Table Tail.	62.2	11.0	0.42	21.1	60.2

Composite No. QC-75-5

Metallurgical Results

Product	Weight	Assays, %		% Distribution	
	%	Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
Table Cleaner Conc.	36.0	68.3	0.27	79.8	39.3
Table Cleaner Tail.	8.8	24.1	0.26	6.9	9.3
Table Rougher Tail.	55.2	7.48	0.23	13.3	51.4
Head (Calculated)	100.0	30.8	0.25	100.0	100.0

Calculated Grades and Recoveries

Table Rougher Conc.	44.8	59.6	0.27	86.7	48.6
Combined Table Tail.	64.0	9.75	0.23	20.2	60.7

Composite No. QC-75-8

Metallurgical Results

Table Cleaner Conc.	37.9	67.7	4.79	82.8	75.2
Table Cleaner Tail.	12.1	22.3	2.55	8.7	12.8
Table Rougher Tail.	50.0	5.24	0.57	8.5	12.0
Head (Calculated)	100.0	31.0	2.42	100.0	100.0

Calculated Grades and Recoveries

Table Rougher Conc.	50.0	56.7	4.26	91.5	88.0
Combined Table Tail.	62.1	8.57	0.97	17.2	24.8

Composite No. QC-75-9

Metallurgical Results

Product	Weight %	Assays, %		% Distribution	
		Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
Table Cleaner Conc.	40.1	67.7	0.32	87.0	48.7
Table Cleaner Tail.	12.8	10.5	0.24	4.3	11.8
Table Rougher Tail.	47.1	5.77	0.22	8.7	39.5
Head (Calculated)	100.0	31.2	0.26	100.0	100.0

Calculated Grades and Recoveries

Table Rougher Conc.	52.9	53.8	0.30	91.3	60.5
Combined Table Tail.	59.9	6.78	0.23	13.0	51.3

Composite No. QC-75-10

Metallurgical Results

Table Cleaner Conc.	34.5	67.6	0.28	86.3	40.1
Table Cleaner Tail.	12.5	9.86	0.22	4.5	11.6
Table Rougher Tail.	53.0	4.68	0.22	9.2	48.3
Head (Calculated)	100.0	27.0	0.24	100.0	100.0

Calculated Grades and Recoveries

Table Rougher Conc.	47.0	52.2	0.26	90.8	51.7
Combined Table Tail.	65.5	5.66	0.22	13.7	59.9

Composite No. QC-75-11

Metallurgical Results

Product	Weight %	Assays, %		% Distribution	
		Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
Table Cleaner Conc.	10.0	69.8	64.0	59.5	67.9
Jeffrey magnetics	9.9	27.9	25.5	23.7	26.7
Jeffrey Tailings	80.1	2.45	0.64	16.8	5.4
Head (Calculated)	100.0	11.8	9.43	100.0	100.0
Davis Tube Conc.	3.6	70.5	-	21.6	-
Davis Tube Tail.	6.3	3.94	-	2.1	-
Feed (Mag. Conc.)	9.9	27.9	-	23.7	-
Table Concentrate + Davis Tube Conc.	13.6	70.0	-	81.1	-

Composite No. QC-75-12

Metallurgical Results

Product	Weight %	Assays, %		% Distribution	
		Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
1. Table Cl. Conc.	47.7	64.2	34.6	88.5	84.9
2. Table Cl. Tail.	10.7	11.4	8.54	3.5	4.7
3. Table Ro. Tail.	41.6	6.61	4.86	8.0	10.4
4. Head (Calc.)	100.0	34.6	19.4	100.0	100.0
5. Jeffrey Conc.	9.9	29.9	28.2	7.5	9.5
6. Jeffrey Tailing	42.4	2.38	0.33	4.0	5.6
7. Feed (Prod. 2+3)	52.3	7.59	5.60	11.5	15.1
8. Davis Tube Conc.	4.0	69.8	-	7.0	-
9. Davis Tube Tail.	5.9	3.38	-	0.5	-
10. Feed (Prod. 5)	9.9	29.9	-	7.5	-

Calculated Grades and Recoveries

Table Rougher Conc.	58.4	54.5	29.8	92.0	89.6

Composite No. QC-75-13

Metallurgical Results

Product	Weight %	Assays, %		% Distribution	
		Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
1. Table Cl. Conc.	45.2	66.0	59.2	76.9	76.6
2. Table Cl. Tail.	5.1	42.1	39.3	5.5	5.7
3. Table Ro. Tail.	49.7	13.7	12.4	17.6	17.7
4. Head (Calc.)	100.0	38.8	34.9	100.0	100.0
5. Jeffrey Conc.	26.1	31.6	29.6	21.2	22.4
6. Jeffrey Tailing	28.7	2.57	1.50	1.9	1.0
7. Feed (Prod. 2+3)	54.8	16.4	14.9	23.1	23.4
8. Davis Tube Conc.	11.4	69.3	-	20.4	-
9. Davis Tube Tail.	14.7	2.2	-	0.8	-
10. Feed (Prod. 5)	26.1	31.6	-	21.2	-

Calculated Grades and Recoveries

Table Rougher Conc.	50.3	63.6	57.2	82.4	82.3

Composite No. QC-75-14

Metallurgical Results

Product	Weight %	Assays, %		% Distribution	
		Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
Table Cleaner Conc.	40.3	65.4	11.4	89.4	80.7
Table Cleaner Tail.	7.7	9.19	3.65	2.4	4.9
Table Rougher Tail.	52.0	4.63	1.57	8.2	14.4
Head (Calculated)	100.0	29.5	5.69	100.0	100.0

Calculated Grades and Recoveries

Table Rougher Conc.	48.0	56.4	10.1	91.8	85.6
Combined Table Tail.	59.7	5.23	1.84	10.6	19.3

Metallurgical Results - Feed to Test No. 36

+48 mesh	6.0	28.5	3.23	6.1	4.1
-48 +65 mesh	22.3	28.6	3.65	22.7	17.4
-65 +100 mesh	25.1	27.9	4.21	24.9	22.7
-100 +150 mesh	14.3	28.2	4.92	14.3	15.0
-150 +200 mesh	12.0	27.9	5.61	11.9	14.4
-200 mesh	20.3	27.9	6.07	20.1	26.4
Head (Calculated)	100.0	28.1	4.66	100.0	100.0

Composite No. QC-75-14 - Continued

Metallurgical Results - Table Concentrate from Test No. 36

Product	Weight %	Assays, %		% Distribution	
		Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
1. +48 mesh	4.6	68.8	6.48	4.82	2.49
2. -48 +65 mesh	21.5	69.0	7.52	22.60	13.53
3. -65 +100 mesh	25.5	68.3	10.06	26.53	21.46
4. -100 +150 mesh	18.2	64.0	12.59	17.74	19.17
5. -150 +200 mesh	13.6	58.6	14.3	12.14	16.27
6. -200 mesh	16.6	63.9	19.5	16.16	27.08
Head (Calculated)	100.0	65.6	12.0	100.00	100.00

Calculated Grades and Recoveries

Products 1 and 2	26.1	69.0	7.34	27.42	16.02
Products 1 to 3	51.6	68.6	8.68	53.95	37.48
Products 1 to 4	69.8	67.4	9.70	71.70	56.65
Products 1 to 5	83.4	66.0	10.45	83.80	72.92

Composite No. QC-75-15

Metallurgical Results

Product	Weight %	Assays, %		% Distribution	
		Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
Table Cleaner Conc.	34.7	66.2	10.4	88.8	81.3
Table Cleaner Tail.	5.1	13.9	3.06	2.7	3.6
Table Rougher Tail.	60.2	3.64	1.11	8.5	15.1
Head (Calculated)	100.0	25.9	4.44	100.0	100.0

Calculated Grades and Recoveries

Table Rougher Conc.	39.8	59.5	9.5	91.5	84.9
Combined Table Tail.	65.3	4.44	1.27	11.2	18.7

Composite No. QC-75-16

Metallurgical Results

Table Cleaner Conc.	16.1	66.6	45.2	81.7	81.0
Jeffrey Magnetics	6.6	21.4	19.8	10.7	14.6
Jeffrey Tailing	77.3	1.29	0.51	7.6	4.4
Head (Calculated)	100.0	13.1	8.98	100.0	100.0

Davis Tube Conc.	1.8	71.3	-	9.8	-
Davis Tube Tailing	4.8	2.38	-	0.9	-
Feed (Mag. Conc.)	6.6	21.4	-	10.7	-

Table Conc. + Davis Tube Concentrate	17.9	67.1	-	91.5	-

Composite No. QC-75-17

Metallurgical Results

Product	Weight %	Assays, %		% Distribution	
		Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
Table Cleaner Conc.	22.4	69.6	43.2	88.5	87.6
Jeffrey Magnetics	2.9	30.3	27.6	5.0	7.2
Jeffrey Tailings	74.7	1.55	0.77	6.5	5.2
Head (Calculated)	100.0	17.7	11.1	100.0	100.0
Davis Tube Conc.	1.1	72.3	-	4.6	-
Davis Tube Tailing	1.8	3.9	-	0.4	-
Feed (Mag. Conc.)	2.9	30.3	-	5.0	-
Table Conc. + Davis Tube Concentrate	23.5	69.7	-	93.1	-

Composite No. QC-75-18

Metallurgical Results

Product	Weight %	Assays, %		% Distribution	
		Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
1. Table Cl. Conc.	24.5	65.2	62.1	71.0	74.6
2. Table Cl. Tail.	8.3	28.4	24.4	10.5	10.0
3. Table Ro. Tail.	67.2	6.21	4.72	18.5	15.4
4. Head (Calc.)	100.0	22.5	20.4	100.0	100.0
5. Jeffrey Conc.	12.9	37.4	34.0	21.4	21.5
6. Jeffrey Tailing	62.6	2.73	1.28	7.6	3.9
7. Feed (Prod. 2+3)	75.5	8.65	6.87	29.0	25.4
8. Davis Tube Conc.	6.4	70.6	-	20.1	-
9. Davis Tube Tail.	6.5	4.77	-	1.3	-
10. Feed (Prod. 5)	12.9	37.4	-	21.4	-

Calculated Grades and Recoveries

Table Rougher Conc.	32.8	55.9	52.5	81.5	84.6

Composite No. QC-75-19-1

Metallurgical Results

Product	Weight %	Assays, %		% Distribution	
		Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
Table Concentrate	39.8	68.1	1.11	82.4	69.0
Table Middling	12.0	29.1	0.42	10.6	7.8
Table Tailing	48.2	4.79	0.31	7.0	23.2
Head (Calculated)	100.0	32.9	0.64	100.0	100.0

Calculated Grades and Recoveries

Table Rougher Conc.	51.8	61.0	0.95	93.0	76.8
Combined Table Tail.	60.2	9.63	0.33	17.6	31.0

Composite No. QC-75-19-2

Metallurgical Results

1. Table Cl. Conc.	46.4	65.9	1.08	90.5	75.7
2. Table Cl. Tail.	8.9	12.4	0.40	3.2	5.4
3. Table Ro. Tail.	44.7	4.74	0.28	6.3	18.9
4. Head (Calc.)	100.0	33.8	0.66	100.0	100.0
5. Jeffrey Conc. *	0.01	47.7	-	0.2	-
6. Jeffrey Tailing	53.59	6.00	-	9.3	-
7. Feed (Prod. 2+3)	53.60	6.00	0.30	9.5	24.3

Calculated Grades and Recoveries

Table Rougher Conc.	55.3	57.3	0.97	93.7	81.1

*No Davis Tube tests

Composite No. QC-75-19-3

Metallurgical Results

Product	Weight	Assays, %		% Distribution	
	%	Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
Table Cleaner Conc.	44.8	67.2	0.85	86.5	38.7
Table Cleaner Tail.	7.0	22.4	0.91	4.5	6.5
Table Rougher Tail.	48.2	6.49	1.12	9.0	54.8
Head (Calculated)	100.0	34.8	0.99	100.0	100.0

Calculated Grades and Recoveries

Table Rougher Conc.	51.8	61.1	0.85	91.0	45.2
Combined Table Tail.	55.2	8.51	1.09	13.5	61.3

Metallurgical Results - Feed to Test No. 39

+48 mesh	10.8	33.2	0.45	11.3	9.7
-48 +65 mesh	25.7	32.9	0.41	24.7	20.8
-65 +100 mesh	25.1	33.6	0.41	24.6	20.4
-100 +150 mesh	12.7	34.5	0.46	12.8	11.5
-150 +200 mesh	9.9	35.3	0.49	10.2	9.7
-200 mesh	15.8	35.8	0.89	16.4	27.9
Head (Calculated)	100.0	34.3	0.51	100.0	100.0

Composite No. QC-75-19-3 - Continued

Metallurgical Results - Table Concentrate from Test No. 39

Product	Weight	Assays, %		% Distribution	
	%	Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
1. +48 mesh	7.3	68.7	0.73	7.42	5.92
2. -48 +65 mesh	26.3	69.0	0.73	26.86	21.33
3. -65 +100 mesh	25.4	68.5	0.83	25.75	23.41
4. -100 +150 mesh	17.2	65.7	0.90	16.73	17.19
5. -150 +200 mesh	11.3	64.5	1.09	10.79	13.68
6. -200 mesh	12.5	67.3	1.33	12.45	18.46
Head (Calculated)	100.0	67.6	0.90	100.00	100.00

Calculated Grades and Recoveries

Products 1 and 2	33.6	68.9	0.73	34.28	27.25
Products 1 to 3	59.0	68.7	0.77	60.03	50.66
Products 1 to 4	76.2	68.1	0.80	76.76	67.85
Products 1 to 5	87.5	67.6	0.84	87.55	81.54

Composite No. QC-75-20

Metallurgical Results

Product	Weight %	Assays, %		% Distribution	
		Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
Table Cleaner Conc.	37.9	67.9	0.59	80.8	36.2
Table Cleaner Tail.	8.8	32.8	0.73	9.1	10.4
Table Rougher Tail.	53.3	6.04	0.62	10.1	53.4
Head (Calculated)	100.0	31.8	0.62	100.0	100.0

Calculated Grades and Recoveries

Table Rougher Conc.	46.7	61.3	0.61	89.9	46.6
Combined Table Tail.	62.1	9.84	0.63	19.2	63.8

Composite No. QC-75-21 (Sludge)

Metallurgical Results

Table Cleaner Conc.	41.8	68.4	0.58	81.7	35.2
Table Cleaner Tail.	5.8	12.6	0.63	2.1	5.4
Table Rougher Tail.	52.4	10.8	0.78	16.2	59.4
Head (Calculated)	100.0	35.0	0.59	100.0	100.0

Calculated Grades and Recoveries

Table Rougher Conc.	47.6	61.6	0.58	83.8	40.6
Combined Table Tail.	58.2	11.0	0.77	18.3	64.8

Davis Tube Tests on Reground Jeffrey Magnetic Conc.

Metallurgical Results

Composite QC-75	Pebble Mill Time per 100 g ore	Head Assays % Sol. Fe	Head Assays % Mag. Fe	Concentrate Wt. %	Concentrate Assay % Sol. Fe	Concentrate % Rec'y Sol. Fe	-400 Mesh %	Tailing Assay % Sol. Fe
3	4	24.4	22.6	51.5	43.9	92.7	36.4	3.7
	8			39.5	57.3	92.8	55.4	2.9
	16			35.1	64.4	92.6	84.7	2.8
	24			33.8	66.9	92.7	92.2	2.7
	32			32.9	68.5	92.4	97.6	2.8
4	24	33.2	32.0	44.9	71.7	97.0	90.8	1.8
	32			44.5	71.2	95.4	96.6	2.7
12	24	29.9	27.8	39.5	69.8	92.2	90.4	3.8
	32			39.9	69.8	93.1	95.2	3.4
13	24	32.5	31.2	45.6	68.7	96.4	94.2	2.2
	32			45.0	69.3	96.0	97.2	2.4
18	24	37.4	35.6	50.7	70.6	95.7	91.6	3.3
	32			50.1	70.6	94.6	97.0	4.1

Method of Testing

100 gram samples of Jeffrey concentrates were ground in a Pebble mill with 10 pounds flint pebbles for the required time. A head sample was removed from the ground concentrates for chemical analysis and for size analysis.

Davis tube conditions were as follows:

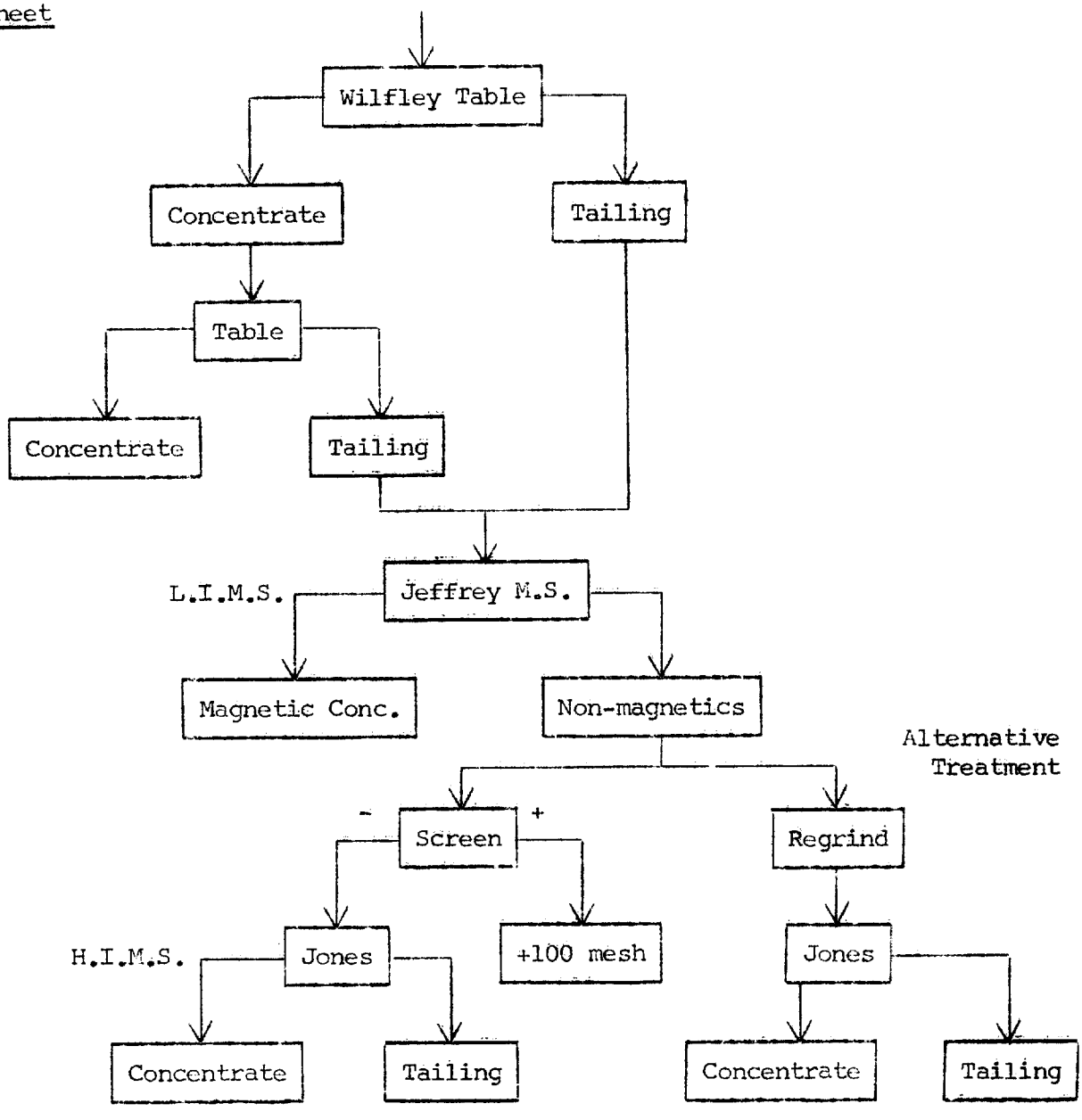
Feed Weight	10 grams
Water Flow	400 ml minute
Oscillations	100 strokes minute
Tube Angle	45°
Current to Coils	2 amperes
Time	4 minutes

The concentrate was collected, dewatered, dried, weighed and assayed. The tailing sample was discarded.

2. Special Tests

Composite 1 + 2

Flowsheet



FLOWSHEET A

FLOWSHEET B

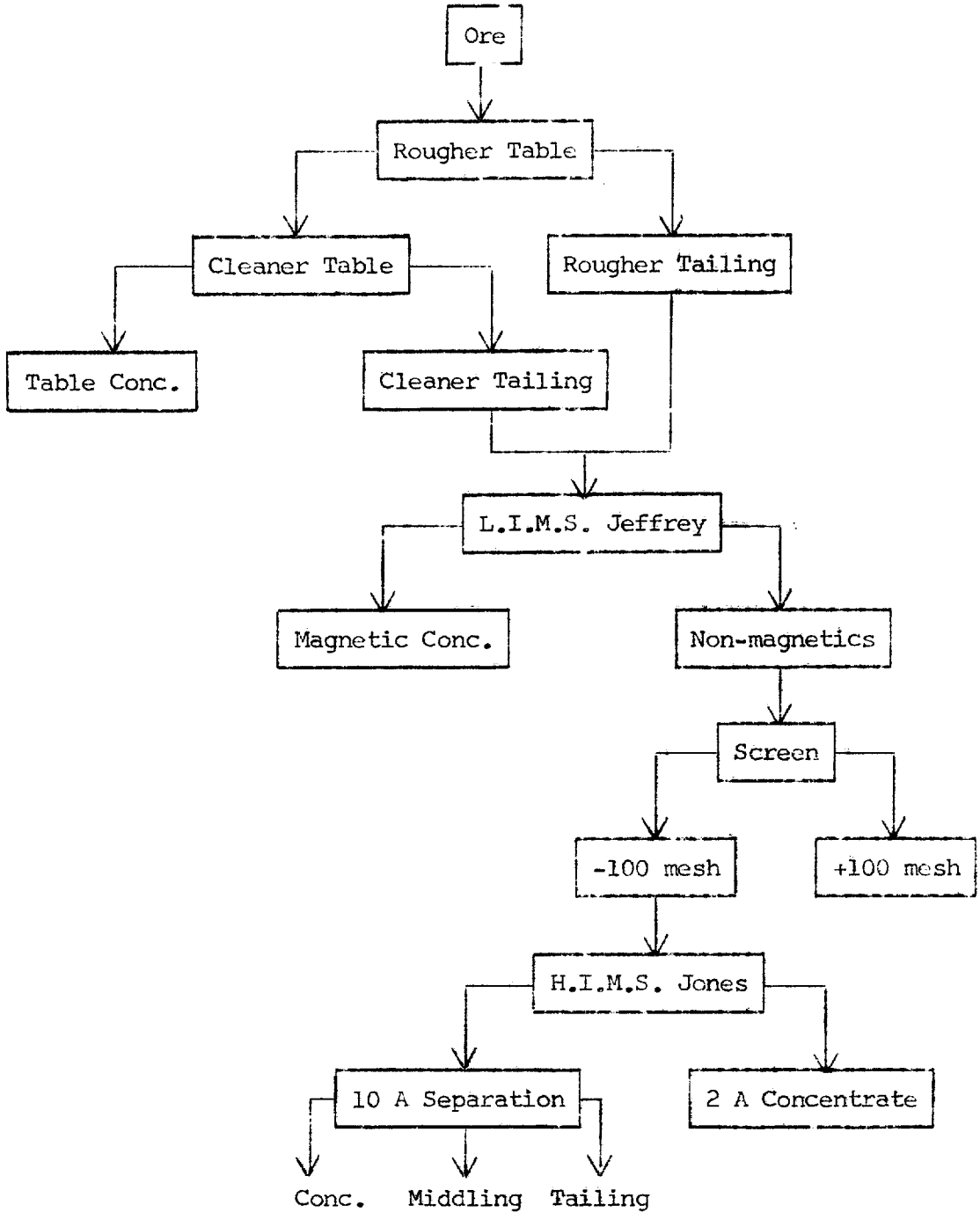
2. Special Tests - Continued

Method of Testing

The purpose of this test was to reject more iron into the table tailing, simulating a spiral operation, and to recover this iron by magnetic separation.

After removing the magnetic portion by low-intensity magnetic separation (Jeffrey Separator) the tailing from this operation was split into two halves. One-half was screened to remove the +100 mesh material (reject), and the -100 mesh portion was treated in the Jones separator at 10 amperes. The second half was reground in stages to minus 200 mesh before magnetic separation. All products were collected, dried and assayed.

FLOWSHEET A


Ore
Rougher Table
Cleaner Table
Rougher Tailing
Table Conc.
Cleaner Tailing
L.I.M.S. Jeffrey
Magnetic Conc.
Non-magnetics
Screen
-100 mesh
+100 mesh
H.I.M.S. Jones
10 A Separation
2 A Concentrate
Conc.
Middling
Tailing

Flowsheet A - Continued

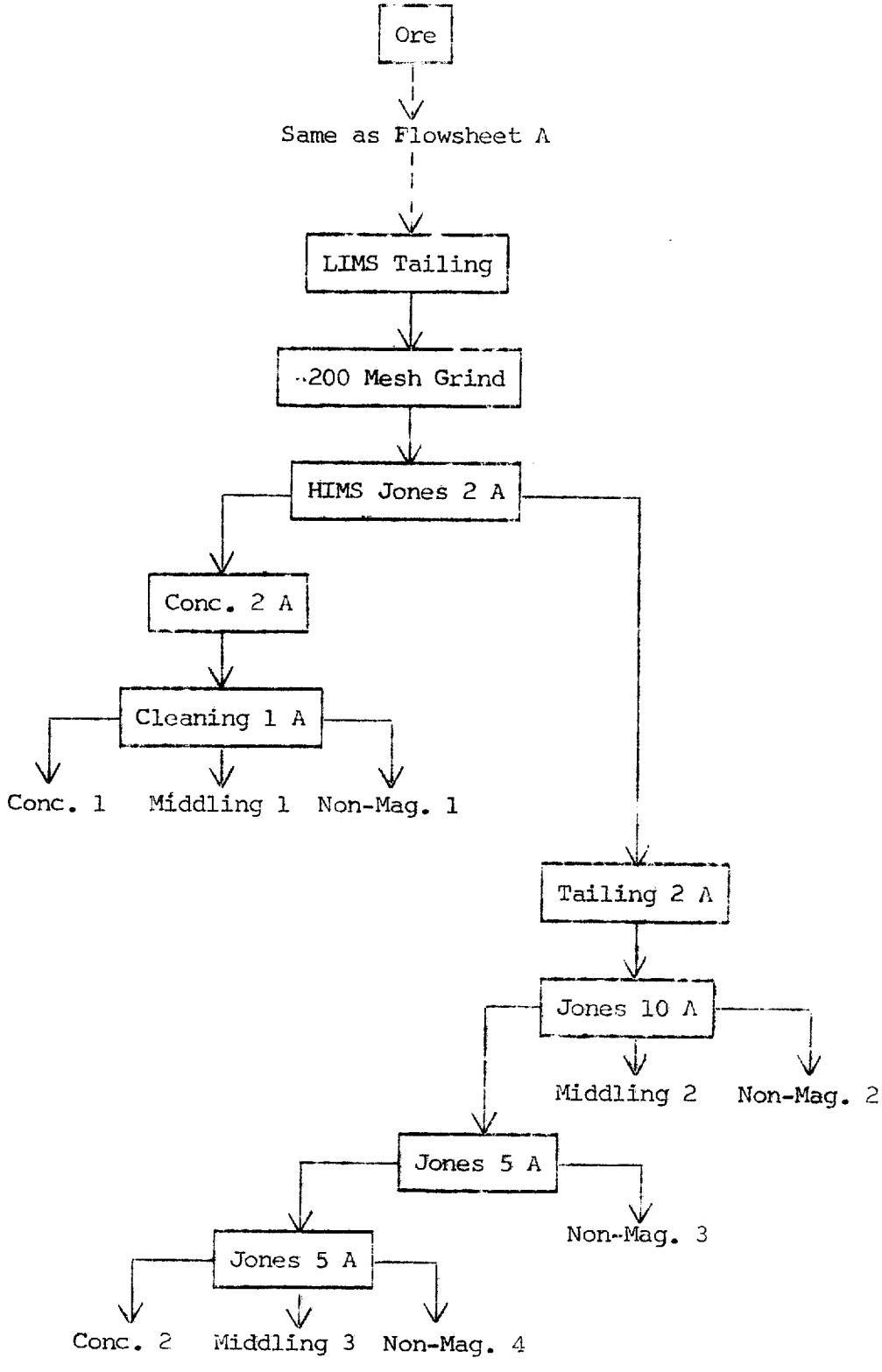
Overall Metallurgical Balance

Product	Weight %	Assays, % Sol. Fe	% Distribution Sol. Fe
Table Conc.	26.2	68.1	63.0
LIMS Conc.	5.0	52.6	9.3
HIMS Conc. 2 A	4.3	39.4	6.0
Conc. 10 A	4.4	22.1	3.4
Middling 10 A	4.1	6.49	1.0
Tailing 10 A	8.5	1.17	0.4
Plus 100 mesh reject	47.5	10.1	16.9
Head (Calculated)	100.0	28.3	100.0

Individual Metallurgical Balances

Stage	Products	Weight % Ind.	Weight % Overall	Assay % Sol. Fe	% Sol. Fe Dist. Ind.	% Sol. Fe Dist. Overall
Tabling	Cleaner Concentrate	26.2	26.2	68.1	63.0	63.0
	Cleaner Tailing	10.9	10.9	62.6	24.1	24.1
	Cleaner Middling	1.0	1.0	49.1	1.8	1.8
	Rougher Tailing	61.9	61.9	5.07	11.1	11.1
	Head (Calculated)	100.0	100.0	28.3	100.0	100.0
L.I.M.S.	Concentrate	6.8	5.0	52.6	25.2	9.3
	Tailing	93.2	68.8	11.4	74.8	27.7
	Feed to Jeffrey	100.0	73.8	14.2	100.0	37.0
Screening	Plus 100 mesh	69.0	47.5	10.1	61.0	16.9
	Minus 100 mesh	31.0	21.3	14.3	39.0	10.8
	Feed (Jeffrey Tail.)	100.0	68.8	11.4	100.0	27.7
H.I.M.S.	Magnetic Conc. 2 A	20.3	4.3	39.4	56.0	6.0
	Magnetic Conc. 10 A	20.6	4.4	22.1	31.9	3.4
	Middling 10 A	19.4	4.1	6.49	8.8	1.0
	Non-magnetics 10 A	39.7	8.5	1.17	3.3	0.4
	Feed (-100 mesh)	100.0	21.3	14.3	100.0	10.8

FLOWSHEET B


Ore
Same as Flowsheet A
LIMS Tailing
-200 Mesh Grind
HIMS Jones 2 A
Conc. 2 A
Cleaning 1 A
Conc. 1
Middling 1
Non-Mag. 1
Tailing 2 A
Jones 10 A
Middling 2
Non-Mag. 2
Jones 5 A
Non-Mag. 3
Jones 5 A
Conc. 2
Middling 3
Non-Mag. 4

Flowsheet B - Continued

Overall Material Balance

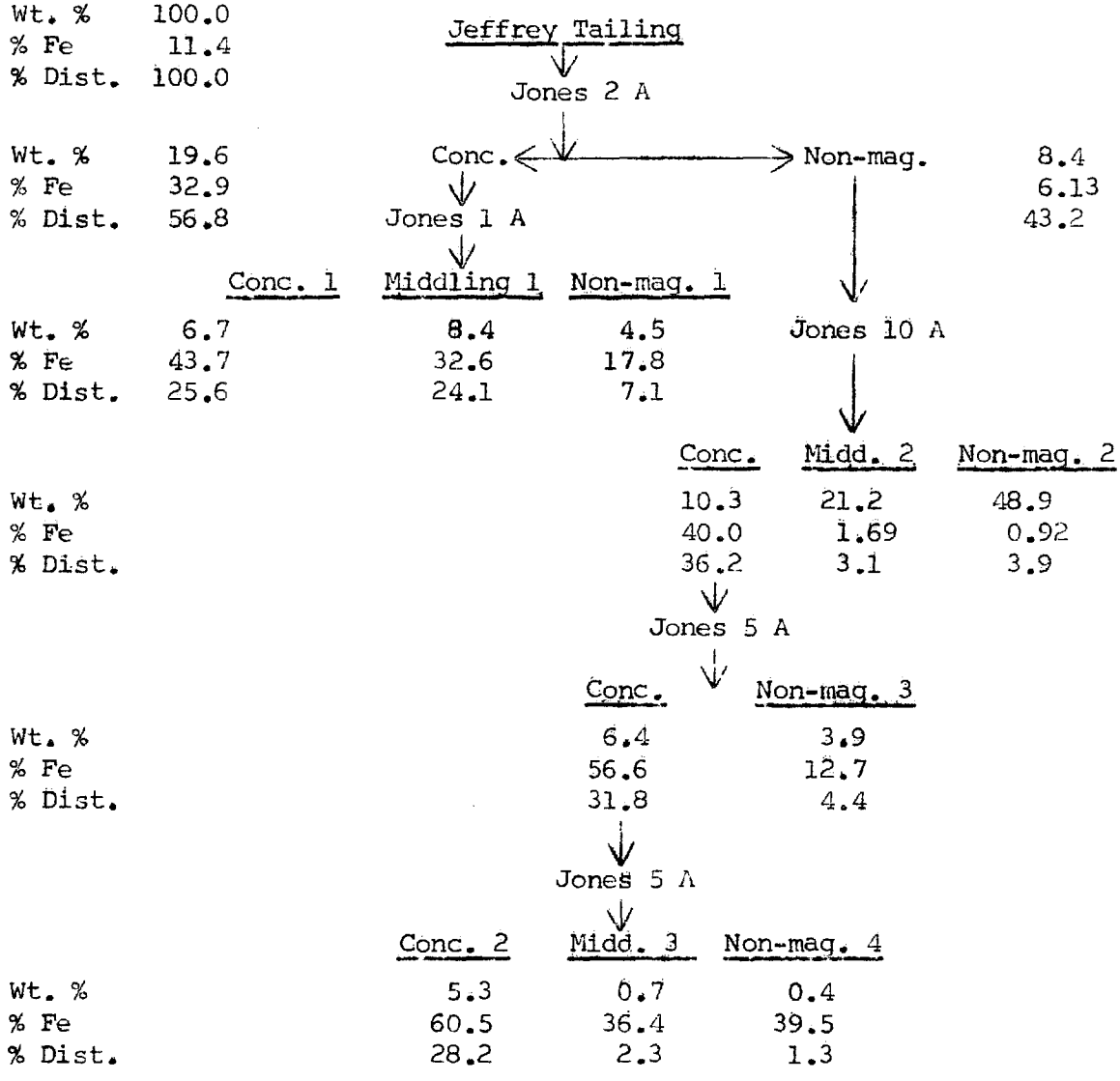
Products	Weight %	Assays, % Sol. Fe	% Distribution Sol. Fe
Table Concentrate	26.2	68.1	63.0
LIMS Concentrate	5.0	52.6	9.3
HIMS Concentrate 2	3.6	60.5	7.8
Middling 3	0.5	36.4	0.6
Non-mag. 4	0.2	39.5	0.4
Non-mag. 3	2.7	12.7	1.2
Middling 2	14.6	1.69	0.9
Non-mag. 2	33.7	0.92	1.0
Concentrate 1	4.6	43.7	7.1
Middling 1	5.8	32.6	6.7
Non-mag. 1	3.1	17.8	2.0
Head (Calc.)	100.0	28.3	100.0

Individual Material Balances

Tabling and LIMS same as for Flowsheet A.

The Jones separation is shown in Flowsheet C.

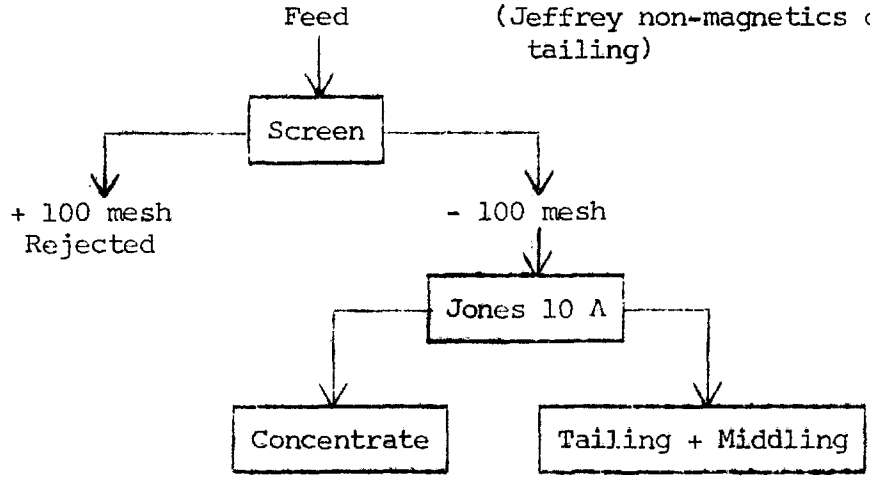
FLOWSHEET C JONES SEPARATION



Wt. %	100.0							
% Fe	11.4							
% Dist.	100.0							
Wt. %	19.6							8.4
% Fe	32.9							6.13
% Dist.	56.8							43.2
		Conc. 1	Middling 1	Non-mag. 1				
Wt. %		6.7	8.4	4.5				
% Fe		43.7	32.6	17.8				
% Dist.		25.6	24.1	7.1				
					Conc.	Midd. 2	Non-mag. 2	
Wt. %					10.3	21.2	48.9	
% Fe					40.0	1.69	0.92	
% Dist.					36.2	3.1	3.9	
						Conc.	Non-mag. 3	
Wt. %						6.4	3.9	
% Fe						56.6	12.7	
% Dist.						31.8	4.4	
			Conc. 2	Midd. 3	Non-mag. 4			
Wt. %			5.3	0.7	0.4			
% Fe			60.5	36.4	39.5			
% Dist.			28.2	2.3	1.3			

3. Jones High Intensity Magnetic Separation

Basic Flowsheet



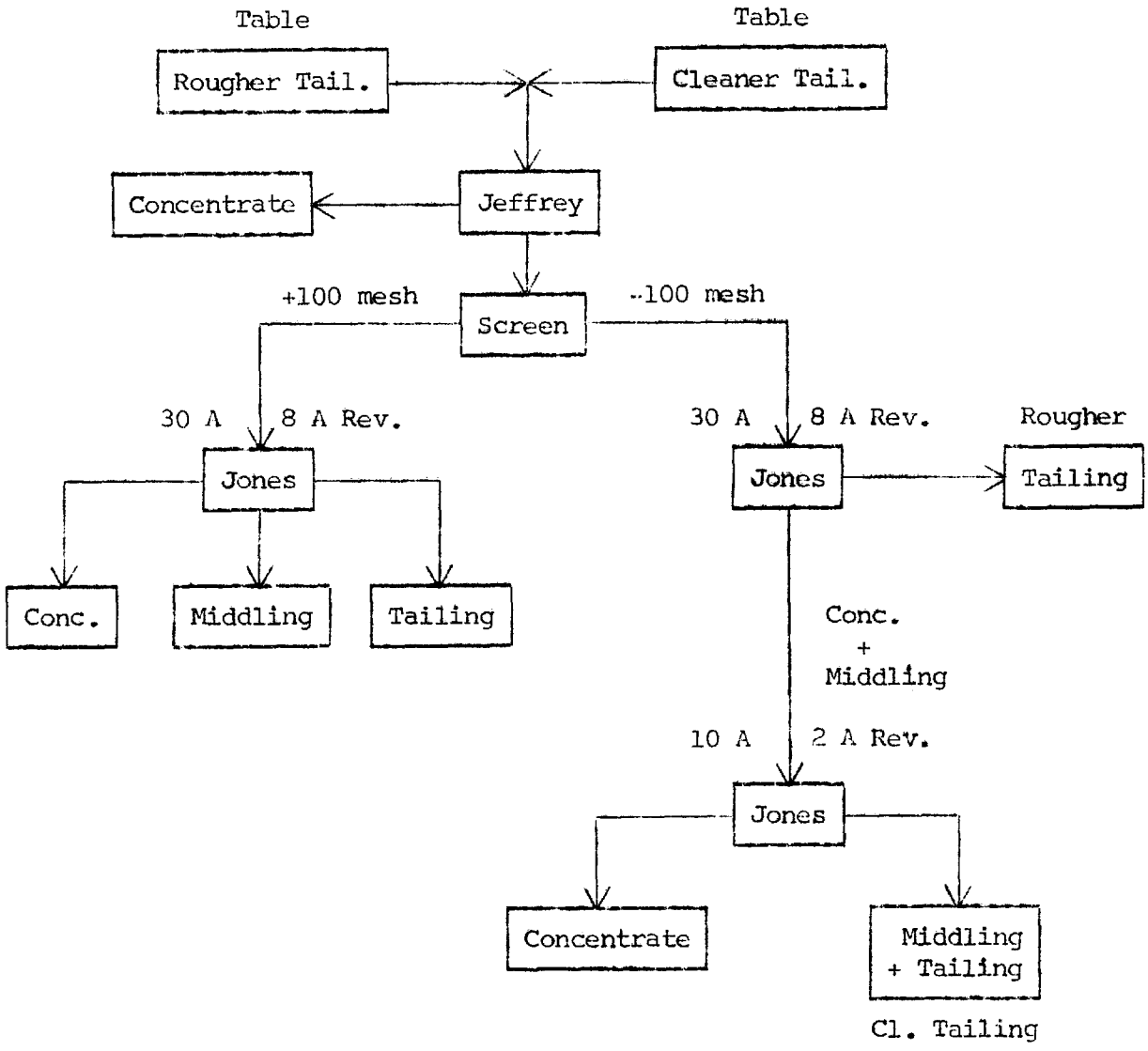
Method of Testing

The feed to the Jones high-intensity magnetic separation consisted either of the combined table tailings, or the non-magnetic products from the Jeffrey low-intensity magnetic separation stage. In most tests a single separation at 10 amperes (2 A reverse current) was performed, and the two products were assayed. In some tests the middling product was not combined with the final non-magnetic fraction, but was assayed separately. Composite 19 was treated following a more complex flowsheet. The details were shown with the test.

3. Jones HIMS - Continued

Flowsheet — Composite QC-75-19

Exploratory Test



Note: Less than 0.01 % of the ore sample was recovered as a magnetic fraction in the Jeffrey separator. The plus 100 mesh fraction was treated separately, but yielded only low-grade products. On the basis of the results from this test the standard flowsheet was developed.

3. Jones HIMS - Continued

Test Results - Composite No. QC-75-19

Product	Weight %	Assays, %		% Distribution	
		Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
Table Concentrate	46.4	65.9	1.08	90.5	75.7
Combined Tailing	53.6	6.0	0.30	9.5	24.3
Head (Calculated*)	100.0	33.8	0.66	100.0	100.0
Jeffrey Concentrate	0.01	47.7	-	0.2	-
Jeffrey Tailing	53.59	6.00	-	9.3	-
Feed (Combined Tail.)	53.6	6.00	0.30	9.5	24.3
-100 mesh Jones Conc.	2.10	39.6	1.46	2.44	-
Jones Middling	2.34	4.80	0.16	0.33	-
Jones Tailing	6.66	1.11	0.11	0.22	-
+100 mesh Jones Conc.	7.88	10.0	0.77	2.32	-
Jones Middling	20.04	4.78	0.31	2.83	-
Jones Tailing	14.57	3.09	0.18	1.34	-
Feed (Jeffrey Tailing)	53.59	6.00	0.30	9.48	-

*Test listed as 19-2 under Table Tests.

3. Jones HIMS - Continued

Standard Tests - Metallurgical Results

Composite QC-75	Products	Weight % Ind.	Weight % Overall	Assays % Sol. Fe	Assays % Mag. Fe	% Sol. Fe Dist. Ind.	% Sol. Fe Dist. Overall
1	Concentrate	11.4	7.6	42.0	5.09	56.8	11.4
	Tailing	19.7	13.2	4.37	0.26	10.2	2.0
	+100 mesh	68.9	46.1	4.04	1.13	33.0	6.6
	Feed (1) (Calc.)	100.0	66.9	8.43	1.41	100.0	20.0
2	Concentrate	5.6	3.8	55.5	6.56	37.5	7.8
	Tailing	22.3	15.2	7.76	0.32	20.8	4.3
	+100 mesh	72.1	49.1	4.82	0.98	41.7	8.8
	Feed (1) (Calc.)	100.0	68.1	8.34	1.15	100.0	20.9
3	Concentrate	20.8	7.3	8.39	4.66	47.9	1.6
	Tailing	29.1	10.2	0.97	0.35	7.8	0.3
	+100 mesh	50.1	17.4	3.22	2.13	44.3	1.5
	Feed (2) (Calc.)	100.0	34.9	3.64	2.46	100.0	3.4
4	Concentrate	14.4	8.6	12.4	3.14	42.3	4.0
	Tailing	28.1	16.7	1.17	0.30	7.9	0.8
	+100 mesh	57.5	34.1	3.65	1.24	49.8	4.7
	Feed (2) (Calc.)	100.0	59.4	4.22	1.25	100.0	9.5
5	Concentrate	4.5	2.9	32.2	9.26	37.7	3.6
	Tailing	28.7	18.5	1.58	0.28	11.7	1.1
	+100 mesh	66.8	43.0	2.93	0.97	50.6	4.8
	Feed (1) (Calc.)	100.0	64.4	3.87	1.15	100.0	9.5

(1) Combined table tailings
(2) Jeffrey non-magnetics

Standard Tests - Metallurgical Results - Continued

Composite QC-75	Products	Weight % Ind.	Weight % Overall	Assays % Sol. Fe	Assays % Mag. Fe	% Sol. Fe Dist. Ind.	% Sol. Fe Dist. Overall
6	Concentrate	11.2	7.0	60.5	1.15	61.8	13.0
	Tailing	38.0	23.6	3.30	0.31	11.4	2.4
	+100 mesh	50.8	31.6	5.80	0.26	26.8	5.7
	Feed (1) (Calc.)	100.0	62.2	11.0	0.42	100.0	21.1
7	Concentrate	10.1	6.5	59.0	0.57	61.6	12.4
	Tailing	26.2	16.8	4.87	0.20	13.1	2.6
	+100 mesh	63.7	40.7	3.87	0.19	25.3	5.2
	Feed (1) (Calc.)	100.0	64.0	9.75	0.23	100.0	20.2
8	Concentrate	8.6	5.3	51.6	4.80	51.9	9.0
	Tailing	24.5	15.2	3.98	0.36	11.4	2.0
	+100 mesh	66.9	41.6	4.70	0.71	36.7	6.2
	Feed (1) (Calc.)	100.0	62.1	8.57	0.97	100.0	17.2
9	Concentrate	6.0	3.6	56.2	0.95	49.4	6.4
	Tailing	22.3	13.4	3.01	0.35	9.9	1.3
	+100 mesh	71.7	42.9	3.84	0.17	40.7	5.3
	Feed (1) (Calc.)	100.0	59.9	6.78	0.23	100.0	13.0
10	Concentrate	6.1	4.0	53.8	0.57	57.6	7.9
	Tailing	28.1	18.4	2.38	0.29	11.8	1.6
	+100 mesh	65.8	43.1	2.63	0.13	30.6	4.2
	Feed (1) (Calc.)	100.0	65.5	5.66	0.22	100.0	13.7
12	Concentrate	11.1	4.7	10.8	1.19	50.2	2.0
	Tailing	24.6	10.4	0.89	0.27	9.2	0.4
	+100 mesh	64.3	27.3	1.50	0.21	40.6	1.6
	Feed (2) (Calc.)	100.0	42.4	2.38	0.33	100.0	4.0

(1) Combined Table Tailings
(2) Jeffrey non-magnetics

Standard Tests - Metallurgical Results - Continued

Composite QC-75	Products	Weight % Ind.	Weight % Overall	Assays, % Sol. Fe	Assays, % Mag. Fe	% Sol. Fe Dist. Ind.	% Sol. Fe Dist. Overall
13	Concentrate	24.2	6.3	5.79	3.59	54.6	11.6
	Tailing	25.3	6.6	0.71	0.28	7.0	1.5
	+100 mesh	50.5	13.2	1.78	1.11	38.4	8.1
	Feed (2) (Calc.)	100.0	26.1	2.57	1.50	100.0	21.2
14	Concentrate	6.5	3.9	37.7	8.39	46.8	5.0
	Tailing	22.6	13.5	1.88	0.23	8.1	0.9
	+100 mesh	70.9	42.3	3.33	1.75	45.1	4.7
	Feed (1) (Calc.)	100.0	59.7	5.23	1.84	100.0	10.6
15	Concentrate	5.7	3.7	36.4	6.80	46.5	8.7
	Tailing	22.8	14.9	1.73	0.31	8.9	1.7
	+100 mesh	71.5	46.7	2.76	1.13	44.6	8.3
	Feed (1) (Calc.)	100.0	65.3	4.44	1.27	100.0	18.7
18	Concentrate	11.2	7.0	6.11	1.67	25.1	1.9
	Tailing	23.0	14.3	0.96	0.34	8.1	0.6
	+100 mesh	65.8	40.9	2.77	1.54	66.8	5.1
	Feed (2) (Calc.)	100.0	62.2	2.73	1.28	100.0	7.6
19-3	Concentrate	8.3	4.6	52.0	2.6	50.7	6.8
	Tailing	21.8	12.0	2.74	0.39	7.0	1.0
	+100 mesh	69.9	38.6	5.16	1.13	42.4	5.7
	Feed (1) (Calc.)	100.0	55.2	8.51	1.09	100.0	13.5
20	Concentrate	10.5	6.5	56.5	1.01	62.7	12.0
	Tailing	24.4	15.2	3.57	0.43	9.2	1.8
	+100 mesh	65.1	40.4	4.09	0.64	28.1	5.4
	Feed (1) (Calc.)	100.0	62.1	9.48	0.63	100.0	19.2

(1) Combined table tailings
(2) Jeffrey non-magnetics

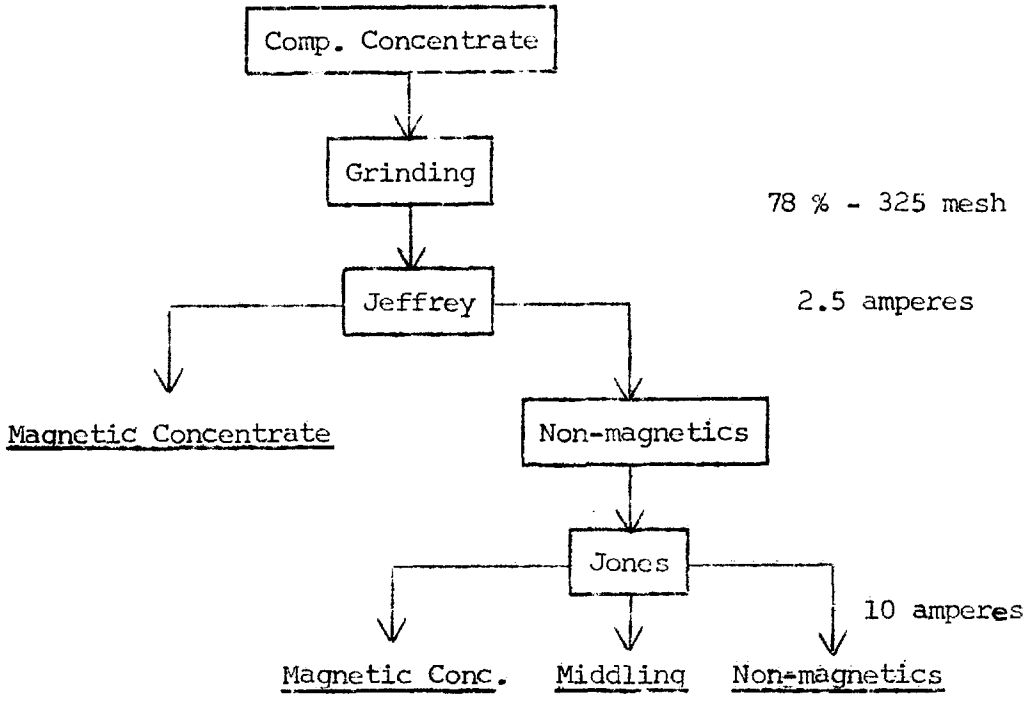
Standard Tests - Metallurgical Results - Continued

Composite QC-75	Products	Weight %		Assays %		% Sol. Fe Dist.	
		Ind.	Overall	Sol. Fe	Mag. Fe	Ind.	Overall
21	Concentrate	11.6	6.8	53.5	1.12	56.5	10.3
	Tailing	41.4	24.1	6.59	0.47	24.8	4.5
	+100 mesh	47.0	27.3	4.38	0.95	18.7	3.5
	Feed (1)(Calc.)	100.0	58.2	11.0	0.77	100.0	18.3

(1) Combined table tailings
(2) Jeffrey non-magnetics

4. Production of Super Concentrate

Flowsheet



Test Procedure

Grinding: 1460 grams of combined table concentrate at minus 35 mesh was ground for 60 minutes in a laboratory ball mill at 65 % solids. A sample of the pulp was screened on a 325 mesh Tyler sieve, and the products were dried and weighed. It was found that 78 % of the material was finer than 325 mesh. Estimated power input was 25.5 kwh/ton (2000 lb.).

Test Procedure - Continued

Jeffrey Separation: The ground pulp was treated under the following conditions:

Current to Coils	:	2.5 amperes
Surface Flux Density	:	750 gauss
Drum Speed	:	60 rpm

Magnetics and non-magnetics were separately collected. After decantation the non-magnetic fraction was treated in the Jones separator.

Jones Separation: Conditions: 10 amperes main current
2 amperes reverse current
30 psig water pressure

The Jeffrey non-magnetics were treated in a single pass, and the three products were collected, dried, weighed and assayed.

Metallurgical Results

Product	Weight	Assays, %			% Distribution		
	%	Sol. Fe	SiO_2	Al_2O_3	Sol. Fe	SiO_2	Al_2O_3
Jeffrey Magnetic Conc.	22.6	71.7	1.20	0.10	23.8	8.4	12.8
Jones Magnetic Conc.	62.3	69.1	1.46	0.21	63.1	28.0	74.3
Middling	8.0	61.4	8.80	0.15	7.2	21.7	6.8
Non-magnetics	7.1	57.1	19.2	0.15	5.9	41.9	6.1
Head (Calculated)	100.0	68.2	3.25	0.18	100.0	100.0	100.0
Concentrates Combined	84.9	69.8	1.39	0.18	86.9	36.4	87.1

5. Heavy Liquid Separation

Method of Testing

Samples of 1000 grams minus 10 mesh ore were screened to give the following size fractions: +20, -20 +35, -35 +65, -65 +100, -100 +200, and -200 mesh.

Each fraction was treated in a methylene iodide solution with a specific gravity of 3.3. The plus 20 mesh ore was treated in an open beaker, the minus 200 mesh in a centrifuge, and all other fractions in a separatory funnel. All products were recovered, washed, dried and prepared for soluble iron analysis.

Metallurgical Results

Composite No.	Head Assay Sol. Fe (Calc.)	Weight %	Total Sink Assay % Sol. Fe	% Dist. Sol. Fe	Estimated Liberation Size, mesh*
QC-75-1	29.0	39.1	63.8	86.0	35
2	27.4	37.1	64.4	87.2	35
3	36.9	52.3	61.6	87.3	65
4	25.0	38.8	53.7	83.5	65
5	25.4	33.6	66.0	87.5	35
6	30.4	41.8	64.7	88.9	20
7	28.0	45.0	61.7	91.3	20
8	30.5	41.8	64.7	88.7	35
9	30.2	44.7	60.4	89.4	65
10	27.0	35.6	67.6	89.1	20
11	9.4	37.5	20.0	80.1	-
12	34.4	52.3	59.9	91.1	65
13	39.1	61.0	56.7	88.5	65
14	28.6	40.6	63.6	90.3	35
15	27.8	38.5	66.3	91.9	35
16	12.5	20.5	45.8	74.9	-
17	15.9	20.1	63.4	80.2	65
18	22.0	41.8	46.8	88.8	-
19	34.5	49.6	64.0	91.9	35
20	29.6	41.0	63.7	88.2	35
21	32.5	45.2	62.5	86.9	20

*Size fraction with iron assay of over 60 %

Composite No. QC-75-1

Metallurgical Results

Product Mesh Size	Fraction	Weight % Ind.	Weight % Overall	Assay % Sol. Fe	% Dist. Sol. Fe Ind.	% Dist. Sol. Fe Overall
+20	Float	73.7	12.7	10.2	35.6	4.5
	Sink	26.3	4.5	51.8	64.4	8.1
-20	Float	56.2	13.9	7.37	12.4	3.5
+35	Sink	43.8	10.9	62.5	87.6	23.5
-35	Float	54.0	16.4	4.11	6.8	2.3
+65	Sink	46.0	14.0	65.9	93.2	31.8
-65	Float	60.0	6.5	2.66	5.5	0.6
+100	Sink	40.0	4.4	68.8	94.5	10.4
-100	Float	62.2	6.8	1.66	3.8	0.4
+200	Sink	37.8	4.2	69.0	96.2	10.0
-200	Float	80.9	4.6	16.8	55.1	2.7
	Sink	19.1	1.1	58.0	44.9	2.2
Head (Calc.)		-	100.0	29.0	-	100.0
Total	Float	-	60.9	6.6	-	14.0
	Sink	-	39.1	63.8	-	86.0

Composite No. QC-75-2

Metallurgical Results

Product		Weight %		Assay %	% Dist. Sol. Fe	
Mesh Size	Fraction	Ind.	Overall	Sol. Fe	Ind.	Overall
+20	Float	76.6	11.6	10.0	38.8	4.2
	Sink	23.4	3.6	51.7	61.2	6.8
-20	Float	62.2	15.9	6.77	15.1	3.9
+35	Sink	37.8	9.6	62.6	84.9	21.9
-35	Float	57.6	18.6	3.46	6.6	2.3
+65	Sink	42.4	13.7	66.9	93.4	33.4
-65	Float	56.8	6.8	1.56	2.9	0.4
+100	Sink	43.2	5.1	68.1	97.1	12.7
-100	Float	59.4	6.3	0.733	1.6	0.2
+200	Sink	40.6	4.3	68.5	98.4	10.7
-200	Float	81.3	3.7	13.6	51.1	1.8
	Sink	18.7	0.8	56.6	48.9	1.7
Head (Calc.)		-	100.0	27.4	-	100.0
Total	Float	-	62.9	5.6	-	12.8
	Sink	-	37.1	64.4	-	87.2

Composite No. QC-75-3

Metallurgical Results

Mesh Size	Product Fraction	Weight % Ind.	Weight % Overall	Assay % Sol. Fe	% Dist. Sol. Fe Ind.	% Dist. Sol. Fe Overall
+20	Float	44.4	7.1	14.2	15.8	2.7
	Sink	65.6	8.9	51.1	84.2	12.3
-20	Float	41.5	9.0	11.7	12.3	2.9
+35	Sink	58.5	12.7	59.5	87.7	20.5
-35	Float	50.0	14.4	6.78	9.4	2.6
+65	Sink	50.0	14.5	65.2	90.6	25.6
-65	Float	53.2	5.4	5.09	7.9	0.7
+100	Sink	46.8	4.7	67.8	92.1	8.6
-100	Float	49.2	7.0	4.17	5.6	0.8
+200	Sink	50.8	7.1	68.0	94.4	13.1
-200	Float	52.4	4.8	22.9	29.5	3.0
	Sink	47.6	4.4	60.1	70.5	7.2
Head (Calc.)		-	100.0	36.9	-	100.0
Total	Float	-	47.7	9.9	-	12.7
	Sink	-	52.3	61.6	-	87.3

Composite No. QC-75-4

Metallurgical Results

Product Mesh Size	Fraction	Weight % Ind.	Weight % Overall	Assay % Sol. Fe	% Dist. Sol. Fe Ind.	% Dist. Sol. Fe Overall
+20	Float	66.2	14.4	8.94	29.7	5.2
	Sink	33.8	7.4	41.5	70.3	12.3
-20	Float	63.4	14.7	7.36	20.3	4.3
+35	Sink	36.6	8.5	49.9	79.7	17.0
-35	Float	62.9	15.4	5.52	13.3	3.4
+65	Sink	37.1	9.2	61.0	86.7	22.4
-65	Float	55.9	5.9	3.50	6.9	0.8
+100	Sink	44.1	4.7	60.1	93.1	11.3
-100	Float	51.3	6.3	2.09	3.5	0.5
+200	Sink	48.7	6.0	60.2	96.5	14.5
-200	Float	60.7	4.5	12.9	27.9	2.3
	Sink	39.3	3.0	50.4	72.1	6.0
Head (Calculated)		-	100.0	25.0	-	100.0
Total	Float	-	61.2	6.8	-	16.5
	Sink	-	38.8	53.7	-	83.5

Composite No. QC-75-5

Metallurgical Results

Product Mesh Size	Product Fraction	Weight % Ind.	Weight % Overall	Assay % Sol. Fe	% Dist. Sol. Fe Ind.	% Dist. Sol. Fe Overall
+20	Float	75.4	11.8	7.03	27.4	3.3
	Sink	24.6	3.8	57.1	72.6	8.6
-20	Float	66.9	18.9	5.15	13.8	3.8
+35	Sink	33.1	9.3	65.2	86.2	23.9
-35	Float	61.9	18.9	3.05	6.8	2.3
+65	Sink	38.1	11.6	68.2	93.2	31.1
-65	Float	60.2	6.8	1.91	4.1	0.5
+100	Sink	39.8	4.5	68.1	95.9	12.1
-100	Float	60.7	6.1	0.84	1.9	0.2
+200	Sink	39.3	4.0	68.3	98.1	10.8
-200	Float	89.5	3.9	15.9	69.2	2.4
	Sink	10.5	0.4	60.4	30.8	1.0
Head (Calculated)		-	100.0	25.4	-	100.0
Total	Float	-	66.4	4.8	-	12.5
	Sink	-	33.6	66.0	-	87.5

Composite No. QC-75-6

Metallurgical Results

Product Mesh Size	Product Fraction	Weight % Ind.	Weight % Overall	Assay % Sol. Fe	% Dist. Sol. Fe Ind.	% Dist. Sol. Fe Overall
+20	Float	58.3	4.5	10.2	18.7	1.5
	Sink	41.7	3.3	62.2	81.3	6.7
-20	Float	52.7	15.5	7.22	11.0	3.7
+35	Sink	47.3	13.8	64.8	89.0	29.4
-35	Float	61.7	22.0	4.24	9.5	3.1
+65	Sink	38.3	13.6	65.4	90.5	29.2
-65	Float	58.4	7.3	1.63	3.4	0.4
+100	Sink	41.6	5.2	65.6	96.6	11.2
-100	Float	52.8	5.5	0.65	1.1	0.1
+200	Sink	47.2	4.9	64.3	98.9	10.4
-200	Float	76.7	3.4	20.7	53.0	2.3
	Sink	23.3	1.0	60.4	47.0	2.0
Head (Calculated)		-	100.0	30.4	-	100.0
Total	Float	-	58.2	5.8	-	11.1
	Sink	-	41.8	64.7	-	88.9

Composite No. QC-75-7

Metallurgical Results

Mesh Size	Product Fraction	Weight % Ind.	Weight % Overall	Assays % Sol. Fe	% Dist. Sol. Fe Ind.	% Dist. Sol. Fe Overall
+20	Float	61.6	6.4	6.52	14.3	1.5
	Sink	38.4	4.0	62.8	85.7	9.0
-20	Float	57.1	18.7	4.75	9.0	3.2
+35	Sink	42.9	14.1	63.7	91.0	32.0
-35	Float	62.1	21.5	3.50	8.6	2.7
+65	Sink	37.9	13.1	60.7	91.4	28.3
-65	Float	55.9	6.0	1.70	3.5	0.4
+100	Sink	44.1	4.8	58.9	96.5	10.1
-100	Float	47.9	3.9	0.83	1.2	0.1
+200	Sink	52.1	4.2	61.1	98.8	9.2
-200	Float	60.8	2.0	11.7	23.8	0.8
	Sink	39.2	1.3	58.1	76.2	2.7
Head (Calculated)		-	100.0	28.0	-	100.0
Total	Float	-	58.5	4.1	-	8.7
	Sink	-	41.5	61.7	-	91.3

Composite No. QC-75-8

Metallurgical Results

Product		Weight %		Assay %	% Dist. Sol. Fe	
Mesh Size	Fraction	Ind.	Overall	Sol. Fe	Ind.	Overall
+20	Float	56.0	7.0	10.7	18.9	2.5
	Sink	44.0	5.5	58.4	81.1	10.5
-20	Float	56.0	15.3	6.13	11.0	3.1
+35	Sink	44.0	12.0	63.3	89.0	24.9
-35	Float	60.4	18.8	4.34	9.1	2.7
+65	Sink	39.6	12.4	66.2	90.9	26.9
-65	Float	59.0	7.1	2.70	5.4	0.6
+100	Sink	41.0	5.0	68.2	94.6	11.2
-100	Float	56.4	6.3	2.08	3.7	0.4
+200	Sink	43.6	4.9	69.4	96.3	11.1
-200	Float	65.9	3.7	16.9	34.6	2.0
	Sink	34.1	2.0	61.8	65.4	4.1
Head (Calculated)		-	100.0	30.5	-	100.0
Total	Float	-	58.2	5.9	-	11.3
	Sink	-	41.8	64.7	-	88.7

Composite No. QC-75-9

Metallurgical Results

Product		Weight %		Assay %	% Dist. Sol. Fe	
Mesh Size	Fraction	Ind.	Overall	Sol. Fe	Ind.	Overall
+20	Float	55.7	3.5	10.5	19.4	1.2
	Sink	44.3	2.7	55.0	80.6	4.9
-20	Float	48.8	11.3	7.71	11.4	2.9
+35	Sink	51.2	11.9	57.0	88.6	22.4
-35	Float	57.3	20.7	4.45	8.6	3.0
+65	Sink	42.7	15.4	63.7	91.4	32.6
-65	Float	55.6	8.2	2.24	4.0	0.6
+100	Sink	44.4	6.5	67.9	96.0	14.6
-100	Float	53.1	6.9	1.24	2.4	0.3
+200	Sink	46.9	6.1	57.9	97.6	11.7
-200	Float	69.2	4.7	16.7	44.6	2.6
	Sink	30.8	2.1	46.6	55.4	3.2
Head (Calculated)		-	100.0	30.2	-	100.0
Total	Float	-	55.3	5.8	-	10.6
	Sink	-	44.7	60.4	-	89.4

Composite No. QC-75-10

Metallurgical Results

Product Mesh Size	Fraction	Weight % Ind.	Weight % Overall	Assay % Sol. Fe	% Dist. Sol. Fe Ind.	% Dist. Sol. Fe Overall
+20	Float	66.5	6.9	7.73	19.5	2.0
	Sink	33.5	3.4	63.5	80.5	8.0
-20	Float	61.6	15.7	5.30	11.2	3.1
+35	Sink	38.4	9.8	67.1	88.8	24.3
-35	Float	64.6	19.9	3.23	7.9	2.4
+65	Sink	35.4	10.8	69.2	92.1	27.7
-65	Float	63.0	8.2	1.79	4.2	0.5
+100	Sink	37.0	4.8	69.7	95.8	12.4
-100	Float	61.8	7.9	1.08	2.4	0.3
+200	Sink	38.2	4.9	70.2	97.6	12.7
-200	Float	75.5	5.8	12.3	39.9	2.6
	Sink	24.5	1.9	57.1	60.1	4.0
Head (Calculated)		-	100.0	27.0	-	100.0
Total	Float	-	64.4	4.6	-	10.9
	Sink	-	35.6	67.6	-	89.1

Composite No. QC-75-11

Metallurgical Results

Mesh Size	Product Fraction	Weight % Ind.	Weight % Overall	Assay % Sol. Fe	% Dist. Sol. Fe Ind.	% Dist. Sol. Fe Overall
+20	Float	60.5	13.7	3.57	22.8	5.2
	Sink	39.5	8.9	18.5	77.2	17.6
-20	Float	65.3	20.4	2.29	22.7	5.0
+35	Sink	34.7	10.9	14.7	77.3	17.1
-35	Float	67.5	13.9	2.26	19.9	3.3
+65	Sink	32.5	6.7	17.8	80.1	12.7
-65	Float	56.5	4.6	2.45	12.2	1.2
+100	Sink	43.5	3.6	22.8	87.8	8.8
-100	Float	47.6	4.6	1.89	12.8	0.9
+200	Sink	52.4	5.1	28.4	87.2	15.4
-200	Float	68.8	5.3	7.45	32.0	4.2
	Sink	31.2	2.3	35.0	68.0	8.6
Head (Calculated)		-	100.0	9.4	-	100.0
Total	Float	-	62.5	3.0	-	19.9
	Sink	-	37.5	20.0	-	80.1

Composite No. QC-75-12

Metallurgical Results

Product Mesh Size	Fraction	Weight % Ind.	Weight % Overall	Assay % Sol. Fe	% Dist. Sol. Fe Ind.	% Dist. Sol. Fe Overall
+20	Float	43.4	5.6	8.96	11.7	1.5
	Sink	56.6	7.2	50.4	88.3	10.5
-20	Float	54.3	14.1	5.36	10.1	2.2
+35	Sink	45.7	11.8	56.8	89.9	19.5
-35	Float	47.8	14.4	3.74	5.2	1.6
+65	Sink	52.2	15.8	62.9	94.8	28.9
-65	Float	40.5	4.9	2.84	2.9	0.4
+100	Sink	59.5	7.3	63.5	97.1	13.5
-100	Float	39.0	4.8	2.26	2.2	0.3
+200	Sink	61.0	7.5	64.0	97.8	14.0
-200	Float	58.6	3.9	25.8	37.9	2.9
	Sink	41.4	2.7	59.9	62.1	4.7
Head (Calculated)		-	100.0	34.4	-	100.0
Total	Float	-	47.7	6.4	-	8.9
	Sink	-	52.3	59.9	-	91.1

Composite No. QC-75-13

Metallurgical Results

Product		Weight %		Assay %	% Dist. Sol. Fe	
Mesh Size	Fraction	Ind.	Overall	Sol. Fe	Ind.	Overall
+20	Float	29.5	5.3	16.4	12.8	2.2
	Sink	70.5	12.7	46.6	87.2	15.2
-20	Float	32.0	7.3	13.2	10.6	2.5
+35	Sink	68.0	15.5	52.5	89.4	20.8
-35	Float	40.8	9.2	8.19	8.3	1.9
+65	Sink	59.2	13.4	62.4	91.7	21.4
-65	Float	44.2	4.6	6.07	5.8	0.7
+100	Sink	65.8	5.7	65.1	94.3	9.5
-100	Float	46.9	6.3	5.45	6.1	0.9
+200	Sink	59.1	7.2	67.0	93.9	12.3
-200	Float	49.5	6.3	20.3	26.3	3.3
	Sink	50.5	6.5	55.8	73.7	9.3
Head (Calculated)		-	100.0	39.1	-	100.0
Total	Float	-	39.0	11.5	-	11.5
	Sink	-	61.0	56.7	-	88.5

Composite No. QC-75-14

Metallurgical Results

Mesh Size	Product Fraction	Weight % Ind.	Weight % Overall	Assay % Sol. Fe	% Dist. Sol. Fe Ind.	% Dist. Sol. Fe Overall
+20	Float	74.3	10.7	8.05	32.2	3.0
	Sink	25.7	3.7	49.8	67.8	6.4
-20	Float	61.7	16.2	5.14	11.8	2.9
+35	Sink	38.3	10.0	61.5	88.2	21.5
-35	Float	52.5	15.9	3.03	4.5	1.7
+65	Sink	47.5	14.4	66.6	95.5	33.5
-65	Float	52.8	6.0	1.68	2.7	0.4
+100	Sink	47.2	5.4	67.7	97.3	12.8
-100	Float	55.6	6.3	1.05	1.9	0.2
+200	Sink	44.4	5.0	68.4	98.1	12.0
-200	Float	66.8	4.3	10.2	26.7	1.5
	Sink	33.2	2.1	56.2	73.3	4.1
Head (Calculated)		-	100.0	28.6	-	100.0
Total	Float	-	59.4	4.7	-	9.7
	Sink	-	40.6	63.6	-	90.3

Composite No. QC-75-15

Metallurgical Results

Mesh Size	Product Fraction	Weight % Ind.	Weight % Overall	Assay % Sol. Fe	% Dist. Sol. Fe Ind.	% Dist. Sol. Fe Overall
+20	Float	74.1	10.0	5.14	32.5	1.3
	Sink	25.9	3.5	59.5	67.5	7.5
-20	Float	60.5	17.0	3.53	7.6	2.2
+35	Sink	39.5	11.0	66.3	92.4	26.3
-35	Float	54.5	16.2	2.39	4.1	1.4
+65	Sink	45.5	13.5	67.5	95.9	32.8
-65	Float	55.7	6.2	1.45	2.7	0.3
+100	Sink	44.3	5.0	67.2	97.3	12.1
-100	Float	61.1	6.8	0.82	1.9	0.2
+200	Sink	38.9	4.4	67.4	98.1	10.7
-200	Float	83.6	5.3	11.5	47.8	2.2
	Sink	16.4	1.1	64.0	52.2	2.5
Head (Calculated)		-	100.0	27.8	-	100.0
Total	Float	-	61.5	3.7	-	8.1
	Sink	-	38.5	66.3	-	91.9

Composite No. QC-75-16

Metallurgical Results

Product		Weight %		Assay %	% Dist. Sol. Fe	
Mesh Size	Fraction	Ind.	Overall	Sol. Fe	Ind.	Overall
+20	Float	81.0	14.4	5.92	45.6	6.8
	Sink	19.0	3.4	32.7	56.4	8.9
-20	Float	81.3	21.1	4.19	32.2	7.1
+35	Sink	18.7	4.9	38.4	67.8	15.0
-35	Float	79.8	19.7	2.43	16.6	3.8
+65	Sink	20.2	5.1	48.3	83.4	19.6
-65	Float	76.5	8.0	1.63	9.1	1.0
+100	Sink	23.5	2.5	52.8	90.9	10.5
-100	Float	74.0	9.1	1.09	5.1	0.8
+200	Sink	26.0	3.2	58.1	94.9	14.8
-200	Float	83.1	7.2	9.78	47.0	5.6
	Sink	16.9	1.4	54.2	53.0	6.1
Head (Calculated)		-	100.0	12.5	-	100.0
Total	Float	-	79.5	4.0	-	25.1
	Sink	-	20.5	45.8	-	74.9

Composite No. QC-75-17

Metallurgical Results

Mesh Size	Product Fraction	Weight % Ind.	Weight % Overall	Assay % Sol. Fe	% Dist. Sol. Fe Ind.	% Dist. Sol. Fe Overall
+20	Float	88.9	16.7	6.38	53.1	6.7
	Sink	11.1	2.1	45.2	46.9	6.0
-20	Float	83.9	20.3	4.28	32.6	5.4
+35	Sink	16.1	3.9	59.1	67.4	14.5
-35	Float	72.7	17.5	2.23	8.1	2.5
+65	Sink	27.3	6.6	67.5	91.9	28.0
-65	Float	84.8	9.3	1.33	9.7	0.8
+100	Sink	15.2	1.7	69.5	90.3	7.4
-100	Float	69.1	9.1	0.96	3.0	0.5
+200	Sink	30.9	4.1	69.6	97.0	18.0
-200	Float	79.8	7.0	8.78	37.0	3.9
	Sink	20.2	1.7	59.0	63.0	6.3
Head (Calculated)		-	100.0	15.9	-	100.0
Total	Float	-	79.9	3.9	-	19.8
	Sink	-	20.1	63.4	-	80.2

Composite No. QC-75-18

Metallurgical Results

Product Mesh Size	Product Fraction	Weight % Ind.	Weight % Overall	Assay % Sol. Fe	% Dist. Sol. Fe Ind.	% Dist. Sol. Fe Overall
+20	Float	52.8	10.8	5.20	11.4	2.6
	Sink	47.2	9.7	45.3	88.6	20.0
-20	Float	60.7	16.5	3.49	10.8	1.7
+35	Sink	39.3	10.7	44.7	89.2	21.7
-35	Float	66.4	15.1	2.57	9.5	1.8
+65	Sink	33.6	7.6	48.9	90.5	16.9
-65	Float	57.5	5.2	2.48	6.5	0.6
+100	Sink	42.5	3.8	48.9	93.5	8.4
-100	Float	50.5	5.6	2.38	4.7	0.6
+200	Sink	49.5	5.5	50.0	95.3	12.5
-200	Float	53.0	5.0	17.4	30.2	3.9
	Sink	47.0	4.5	45.4	69.8	9.3
Head (Calculated)		-	100.0	22.0	-	100.0
Total	Float	-	58.2	4.2	-	11.2
	Sink	-	41.8	46.8	-	88.8

Composite No. QC-75-19

Metallurgical Results

Product		Weight %		Assay %	% Dist. Sol. Fe	
Mesh Size	Fraction	Ind.	Overall	Sol. Fe	Ind.	Overall
+20	Float	50.3	2.9	8.95	13.3	0.8
	Sink	49.7	2.9	59.4	86.7	5.0
-20	Float	39.5	8.3	6.38	6.4	1.5
+35	Sink	60.5	12.6	61.8	93.6	22.5
-35	Float	50.9	19.2	4.27	6.3	2.4
+65	Sink	49.1	18.5	66.1	93.7	35.4
-65	Float	53.3	7.7	2.64	4.4	0.6
+100	Sink	46.7	6.8	66.3	95.6	13.1
-100	Float	51.7	7.3	1.10	1.9	0.2
+200	Sink	48.3	6.9	62.8	98.1	12.6
-200	Float	72.0	5.0	18.2	42.3	2.6
	Sink	28.0	1.9	63.8	57.7	3.3
Head (Calculated)		-	100.0	34.5	-	100.0
Total	Float	-	50.4	5.6	-	8.1
	Sink	-	49.6	64.0	-	91.9

Composite No. QC-75-20

Metallurgical Results

Product Mesh Size	Fraction	Weight % Ind.	Weight % Overall	Assay % Sol. Fe	% Dist. Sol. Fe Ind.	% Dist. Sol. Fe Overall
+20	Float	69.6	6.5	8.65	26.1	1.9
	Sink	30.4	2.9	56.0	73.9	5.5
-20	Float	57.9	14.8	6.91	13.3	3.5
+35	Sink	42.1	10.8	62.0	86.7	22.6
-35	Float	59.6	21.1	4.50	9.2	3.2
+65	Sink	40.4	14.2	65.6	90.8	31.6
-65	Float	55.5	7.1	2.22	4.0	0.5
+100	Sink	44.5	5.7	66.7	96.0	12.8
-100	Float	49.7	5.9	1.06	1.6	0.2
+200	Sink	50.3	6.0	63.4	98.4	12.8
-200	Float	71.3	3.6	20.8	45.3	2.5
	Sink	28.7	1.4	62.3	54.7	2.9
Head (Calculated)		-	100.0	29.6	-	100.0
Total	Float	-	59.0	5.9	-	11.8
	Sink	-	41.0	63.7	-	88.2

Composite No. QC-75-21

Metallurgical Results

Product		Weight %		Assay %	% Dist. Sol. Fe	
Mesh Size	Fraction	Ind.	Overall	Sol. Fe	Ind.	Overall
+20	Float	52.5	1.8	11.6	16.5	0.6
	Sink	47.5	1.7	64.9	83.5	3.4
-20	Float	41.0	6.2	7.17	7.2	1.4
+35	Sink	59.0	9.0	64.4	92.8	17.8
-35	Float	52.8	17.2	4.03	6.7	2.1
+65	Sink	47.2	15.4	63.3	93.3	30.0
-65	Float	57.1	9.2	2.72	5.3	0.8
+100	Sink	42.9	7.0	64.5	94.7	13.9
-100	Float	55.5	8.9	2.87	5.8	0.8
+200	Sink	44.5	7.1	58.1	94.2	12.7
-200	Float	69.5	11.5	20.8	44.5	7.4
	Sink	30.5	5.0	59.2	55.5	9.1
Head (Calculated)		-	100.0	32.5	-	100.0
Total	Float	-	54.8	7.7	-	13.1
	Sink	-	45.2	62.5	-	86.9

LAKEFIELD RESEARCH OF CANADA LIMITED
Lakefield, Ontario
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