DENISON MINES LIMITED

RECOVERY OF MAGNETITE FROM WHITEHORSE COPPER TAILINGS

SUMMARY REPORT

NOVEMBER 1984

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ILLUSTRATIONS

1.0 INTRODUCTION

Magnetite is used in the heavy media circuit of many coal cleaning plants. The consumption of magnetite is largely related to the quantity of material processed by the heavy media plant. Quintette Coal Limited is presently consuming about 10 000 tonnes per year, Luscar Ltd., near Hinton, Alberta consume about 17 000 tonnes per year.

At the present time there is no producer of magnetite in Western Canada. All the magnetite consumed in B.C. and Alberta comes from a stockpile owned by Craigmont Mines Limited of Merritt, B.C. The stockpiled material is of inferior quality and is being consumed quite rapidly. Accurate data is not available but at present sales rates the stockpile should be exhausted by 1990.

For the foregoing reasons Denison Mines Limited have a continuing interest in securing an alternative supply of magnetite. It has been noted by Mr. V.V. Jutronich that the tailings which were produced by Whitehorse Copper Mines Ltd. in the Yukon Territory contained significant quantities of magnetite. A preliminary investigation including sampling, metallurgical testwork, capital and operating cost estimates was completed in October 1984 with encouraging results. Work has continued since the initial report was issued and has resulted in refinements to the capital and operating cost estimates. This report summarizes the October study as modified by more recent findings.

2.0 SUMMARY

The tailings dams at Whitehorse Copper Mines in the Yukon Territory contain approximately 1 800 000 tonnes of magnetite and could potentially be a secure, long term source of heavy media grade magnetite for Quintette and other coal cleaning plants.

This report presents data which show that the tailings can be processed to recover a high grade magnetite using portions of the existing processing plant complemented by some new equipment. The proposed project is operated on day shift, five days per week, seven months per year to produce 40 000 tonnes per year of material. The production of additional magnetite may be justified by market conditions and could be readily accommodated.

Capital costs, excluding acquisition costs and operating costs are summarized in the tables which follow.

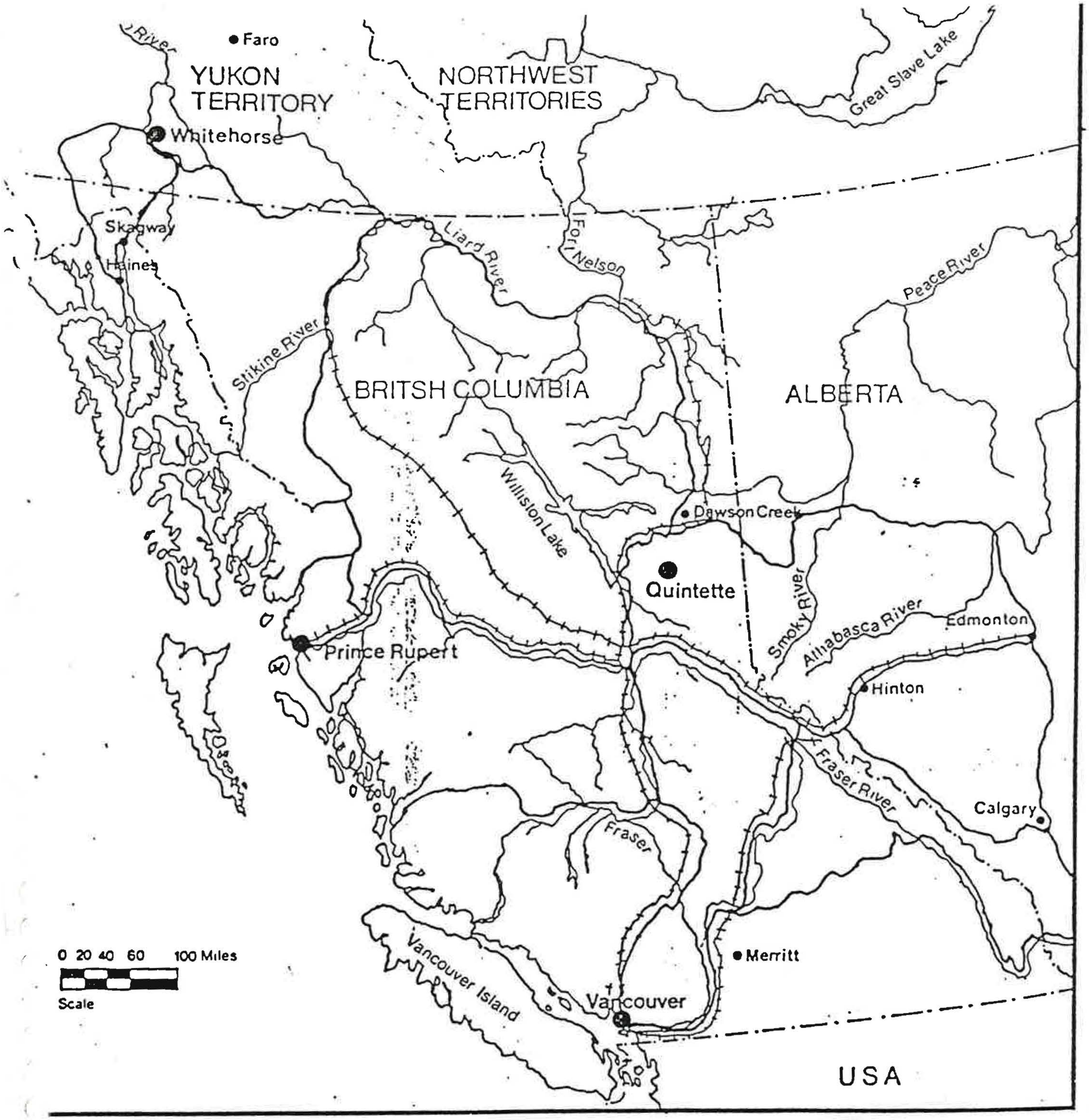
TABLE 2-1

CAPITAL COST SUMMARY

Item	C$ 1984
General Mill Building	$ 111 000
Process Equipment	652 000
Sampling, Testwork, Data	20 000
Engineering	70 000
Indirect Construction Costs	90 000
Pre-Startup Operating	137 000
Working Capital	559 000
Contingency	147 000
Grand Total	$ 1 786 000

TABLE 2-2
OPERATING COST SUMMARY

Item	C$ 1984/a
Administration	$ 142 000
Reclamation	104 000
Processing	365 000
Packaging	88 000
Transportation	2 800 000
Totals	$ 3 499 000
	$ 87.48/t product



Denison Mines Limited

Magnetite Project

Location Map

3.0 PROJECT DESIGN CRITERIA

3.1 SUMMARY

Preliminary evaluation of the Whitehorse Copper Mines tailings dams indicate that the 10 000 000 tonnes of tailings contain about 19% magnetite representing a reserve of 1 800 000 tonnes. Higher grades can be obtained by selective reclamation.

Initial testwork shows that a heavy media grade magnetite can be recovered using simple magnetic separation techniques and parts of the existing Whitehorse Copper Mines milling facilities.

Handling and transportation studies to date indicate that the plant product can be stored and transported in 2 tonne capacity reinforced plastic bags. Transportation is economically realized by back-hauling in food and beverage vans.

3.2 TAILINGS RESERVES

A detailed survey of the existing tailings dams has yet to be performed. Data provided in Table 3-1 is based on relative surface areas of tailings deposits, published dam ages and samples recently taken from the dams by Denison and Kilborn personnel.

TABLE 3-1

APPROXIMATE MAGNETITE RESERVES, WHITEHORSE COPPER TAILINGS

Area Designation	Estimated Tailings-tonnes	Magnetite Content % Range	Magnetite Content % Average	Magnetite Reserve-tonnes
1	1 670 000	15 - 34	23	384 000
3	1 160 000	15 - 27	20	232 000
4	1 480 000	16 - 28	20	296 000
A	3 340 000	8 - 26	17	568 000
B	2 050 000	15 - 21	18	369 000
Totals	9 700 000	8 - 34	19	1 849 000

3.3 MAGNETITE RECOVERY PLANT

The design criteria which follow are based on a projected market demand, an operating strategy which recognizes climatic conditions in the Yukon, preliminary metallurgical tests and background data from other relevant operations.

TABLE 3-2
PROJECT DESIGN CRITERIA

Production Criteria		
Annual Magnetite Production	t/a	40 000
Operating Time - Process Plant	h/a	1 600
	h/day	11
	d/week	5
	weeks/a	29
Production Rate	t/h	25.0
Product Grade	% magnetite	92.0
Plant Feed Grade	% magnetite	20.0
Overall Recovery	%	92.0
Plant Feed Rate	t/h	125.0
Annual Tailings Reclaim Rate	t/a	200 000
Tailings Reclaim		
Method	Wheel Loader and Reslurry	
In-Situ Density	t/m³	1.9
In-Situ Average Moisture	%	10.0
Reclaim Reslurry Solids	%	50
Reslurry Surge Time	m	10

Conditioning and Primary Separation

Equipment for Conditioning		Existing Rod Mill
Primary Separator Type		Single Drum, Concurrent
Primary Separator Feed Solids	%	33.33
Primary Separator Drum Diameter	mm	1 220
Primary Separator Feed Rate	m³/slurry/h/m	105
Primary Separator Magnetics Recovery	%	92
Primary Concentrate Percentage Solids	%	66.67
Primary Concentrate Magnetite Content	%	65.0

Regrinding and Secondary Separation

Coarse Feed Size Analysis	% -400 mesh	32
	80% past microns	100
Finished Product Size Analysis	% - 325 mesh	93
	% - 400 mesh	82
	80% past microns	35
Work Index (metric)	kWh/t	32
Secondary Separator Type	Double Drum, Countercurrent	
Secondary Separator Feed Solids	%	20.0
Secondary Separator Drum Diameter	mm	1 220
Secondary Separator Feed Rate	m³ slurry/h/m	175
Secondary Separator Magnetics Recovery	%	96
Secondary Concentrate Percentage Solids	%	66.67
Secondary Concentrate Magnetite Content	%	92.0

Product Filtration and Packaging

Feed Rate	t/h	25
Filtration Rate	t/h/m²	0.3
Filter Aid Dose	kg/t product	0.1
Filter Cake Moisture	%	8.0
Product Package		Bag
Bag Capacity - Rating	kg	2 000
Bag Capacity - Dry Mass Packed	kg	1 750

3.4 MAGNETITE SPECIFICATIONS

Magnetite for use in heavy media plants is usually purchased as 'B' grade i.e. about 90% -325 mesh. The chemical composition and detailed screen analysis of magnetite varies from supplier to supplier. Typical analyses are provided below in Table 3-3.

TABLE 3-3

TYPICAL HEAVY MEDIA GRADE MAGNETITE SPECIFICATIONS

	Producer						
Parameter	Foote	Craigmont Typical	Craigmont Quintette	Noracor	Pitkin	Am. Verm.	Whitehorse Copper
Analyses-%							
Fe	62.2	65-68	60.7	67	-	64	68.6
Magnetite	91-94	+90	83.9	93	85	88	94.8
SiO_2	1.5	3	9.26	5.4	-	1.0	0.96
Al_2O_3	3.5	4	1.68	-	-	1.0	0.18
S	0.15	-	-	-	-	-	0.04
P	0.012	-	0.018	-	-	0.8	0.017
H_2O	-	<10	-	7-8	2-3	-	8
Density - g/ml	4.7-4.9	-	4.60	4.9	-	-	4.85
Size - Mass %							
+100	0.1	-	0.3	0.4	7.4	-	0
+200	1.0	-	3.0	2.6	18.8	-	1.2
+325	7.6	-	10.7	16	12.2	-	7.8
-325	91.4	80-93	86.0	81	61.6	96	91.0

4.0 SUMMARY OF CAPITAL DETAILS

4.1 SUMMARY

Pre-production capital costs are summarized in Table 4-1 and discussed overleaf.

TABLE 4-1

CAPITAL COST ESTIMATE

Area	Cost C$ New	Cost C$ Other	Cost C$ Total
General Mill Building			
Structural, New or Modified	-	-	81 000
Building Services Modification	-	-	5 000
New Process Piping Material			10 000
New Electrical Material			10 000
New Instrumentation			5 000
Sub Total - General Mill Building			111 000
Process Equipment			
Tailings Reclaim	164 000	7 000	171 000
Condition and Primary Sep.	69 000	2 000	71 000
Regrind and Secondary Sep.	144 000	7 000	151 000
Filter and Packaging	73 000	5 000	78 000
Tailings Disposal	-	6 000	6 000
Bags (5 000 x 2 t)	175 000	-	175 000
Sub Total - Process Equipment	625 000	27 000	652 000
Additional Sampling, Testwork and Data			20 000
Engineering Services			70 000
Indirect Construction Costs			90 000
Pre-Startup Operating Cost			137 000
Working Capital			559 000
Contingency (25% of $588 000)			147 000
Grand Total			$1 786 000

Notes:

1. Costs are given in last quarter 1984 Canadian dollars.
2. Aquisition costs are excluded.

4.2 GENERAL MILL BUILDING

Structural costs mainly comprise costs for providing a new timber and concrete truck loading ramp adjacent to the mill building. The other items included in the general cost category are largely self explanatory.

Costs reflect the availability of materials in the Whitehorse Copper Mines warehouse. Most of the work considered here is to be performed by the five plant operating personnel prior to startup. Labour costs are included with the pre-startup operating cost category.

4.3 PROCESS EQUIPMENT

The 'New' costs noted in this area include for the purchase and installation of new equipment. Major items of equipment, cost sources and costs are as follows:

Cyclone Cluster	Technequip Limited	$ 52 000
Magnetic Separators	Eriez of Canada Limited	$155 000
Bags and Bagging Equipment	Bonar and Bemis Ltd.	$210 000
Reclaim Hopper/Conveyor/Screen	Rexnord	$ 78 000
Fork Lift	Wajax Industries Limited	$ 20 000

4.4 ADDITIONAL SAMPLING, TESTWORK AND DATA

The required program of sampling, testwork and data aquisition is expected to cost a total of $20 000. This sum covers the cost to obtain and distribute samples for further analysis, equipment sizing testwork, and costs for travel to Whitehorse to confirm engineering and other data.

4.5 ENGINEERING SERVICES

A budget allowance of $70 000 is proposed to cover engineering costs. It is envisioned that much of the normal engineering costs can be avoided by field routing pipelines and electrical cables and by locating equipment in the field. Some engineering assistance will be required in particular regarding pipe line selection, structural steel and concrete design, mechanical design, purchasing assistance and start-up help.

4.6 INDIRECT CONSTRUCTION COSTS

This cost allowance includes for construction equipment rental, power and fuel consumed during the construction period and other indirect costs.

4.7 PRE-STARTUP OPERATING COST

This cost covers payroll and administration department operating costs from the date of aquisition (assumed to be January 1, 1985) to plant startup, July 1, 1985.

4.8 WORKING CAPITAL

Working capital is made up as follows:

Payroll and administration costs for two months - $46 000.

Process costs less bag costs for two months operation (11 000 t) - $93 000.

Transportation costs for one month (6 000 t) - $420 000.

4.9 CONTINGENCY

A contingency allowance is made equal to 25% of the general mill building and process equipment costs (excluding the bags). This allowance is intended to cover the costs of items which have not been specifically itemized in the capital cost estimate.

5.0 SUMMARY OF OPERATING COSTS

5.1 SUMMARY

Project operating costs are summarized below in Table 5-1. Explanatory notes are provided overleaf.

TABLE 5-1
OPERATING COST SUMMARY

Cost Area	Annual Cost - $1000/a Labour	Supplies	Power	Fuel	Other	Total	Production Cost $/t Product
Administration	58	5	-	2	77	142	3.55
Reclamation	33	45	6	20	-	104	2.60
Processing	99	133	130	3	-	365	9.13
Packaging	-	88	-	-	-	88	2.20
Transportation	-	-	-	-	2800	2800	70.00
Totals	190	271	136	25	2877	3499	87.48

NOTES:

1. Production cost based on 40 000 tonnes of dry product per year.
2. Labour cost includes 30% payroll burden.
3. Maintenance supply costs included under supplies.
4. All costs are given in last quarter, 1984 Canadian dollars.

5.2 ADMINISTRATION OPERATING COSTS

The proposed project will employ a single project manager/ administrator who will also control payroll and accounting. Annual administration payroll costs are calculated to be $58 000 comprising $45 000 direct remuneration and $13 000 payroll burden (30%).

Administration supplies are expected to be minimal and an allowance of $5 000 only is proposed.

It is assumed that the administrator is provided with a rental vehicle and fuel. Fuel costs are taken as $2 000 per annum. The rental vehicle will also serve as the site service vehicle.

The following miscellaneous cost allowances are also made:

TABLE 5-2
MISCELLANEOUS ADMINISTRATION COSTS

Item	Cost $/Year
Vehicle Rental	4 000
Communications (telephone, telex, courier, etc.)	5 000
Insurance	10 000
Travel	5 000
Consultants and Custom Analyses	5 000
Legal and Audit Costs	10 000
Municipal Taxes	38 000
TOTAL	$ 77 000

5.3 RECLAMATION

Mining or reclamation of the tailings will employ the existing Caterpillar 966D wheel loader followed by reslurrying of the tailings and pumping to the mill.

Estimates indicate that the existing wheel loader with a 4 cubic yard bucket will be able to recover the desired 125 t/h at travel distances up to 200 m (700 ft).

A single reclaim operator will comfortably handle the required work load. It is assumed that he will receive remuneration of $25 000/year and that payroll burdens amount to $8 000 for a total labour cost of $33 000/year.

Supply and fuel costs for the wheel loader are generated in Table 5-3.

The cost of supplies for maintaining the reslurry and pumping system is taken as $6 000/a or 5% of the cost of maintainable equipment.

The power required to reslurry the tailings and pump the slurry to the mill from the upper tailings areas is estimated to be 75 HP or 56 kW. Annual costs at an incremental rate of $0.063/kWh are $6 000/a.

TABLE 5-3

WHEEL LOADER FUEL AND SUPPLY COSTS

Item	Units	Data
Operating Hours	h/a	1 600
Number of Units		1
Fuel		
Fuel Consumption	L/h/unit	32
Annual Fuel Consumption	L/a	51 200
Fuel Cost	$/L	0.40
Annual Fuel Cost	$/a	20 480
Supplies		
Oil, Lube and Filters	% fuel cost	30
Oil, etc. Cost	$/h/unit	3.84
Base Price of Unit	$	193 600
Maintenance Cost Factor		0.8
Maintenance Base Hours	h	10 000
Maintenance Parts Costs	$/h/unit	15.49
Tyre Costs	$/tyre	1 600
Number of Tyres	tyres/unit	4
Average Tyre Life	h	1 400
Tyre Cost Factor		1.15
Tyre Costs	$/h/unit	5.26
Total Supply Costs	$/h/unit	24.59
	$/year	39 344

5.4 PROCESSING COSTS

Reclaimed tailings are conditioned in the existing rod mill. Magnetite is then separated, reground, cleaned and filtered. Moist filter cake is then packed in 2 t bags at a production rate of 25 t/h.

Two men are required to operate and maintain the processing plant. A third man is required to attend to the packaging of product and the movement of loaded bags to the immediate storage area. Labour costs are taken to be $25 000 per man plus payroll burden of $8 000 per man for costs of $33 000 per man or $99 000/year.

Processing supplies are estimated for a production rate of 40 000 t/a as follows:

TABLE 5-4
PROCESSING CONSUMABLE SUPPLY COSTS

Item	Unit	Consumption per unit	Consumption per year	Unit Cost $	Annual Cost $/a
Grinding Balls	kg	0.44/t feed	88 000	0.900	79 200
Filter Cloths	cloth	0.005/t product	200	20	4 000
Filter Aid	kg	0.1/t product	4 000	2.400	9 600
Totals	-	-	-	-	$92 800

Notes:
1. Unit costs are delivered.
2. Ball consumption based on 0.07 kg/kWh, 22 kWh/t rougher concentrate and 0.3 t conc/t feed.

Maintenance parts cost, including screen cloth, cyclone liners, pump parts and the like are expected to cost an additional $40 000/a for a total of $132 800/a.

Installed and operating electrical power data are summarized below in Table 5-5.

TABLE 5-5
PROCESS POWER CONSUMPTION

Item		Inst. HP	Draw HP
Rod Mill		450	100
Primary Separator Feed Pump		20	10
Primary Separator Drive	(48" x 120")	7.5	5
Demag Coil		1	1
Mill Pumps (2 inst. 1 op)		50	36
Ball Mill		900	900
Tailings Pump		100	92
Reclaim Pump		100	25
Denver Conditioner		7.5	5
Conc. Transfer Pump		5	2.5
Secondary Separator Feed		15	10
Secondary Separator Drive	(48" x 72")	5	3
Dorrco Agitator		5	3
Vacuum Pump	(CL2002)	100	70
Filtrate Pump		2	0.5
Disc Drive		5	3
No. 13 Conveyor	20' long x 0' lift	5	2
Drier Drive		7.5	5
Bagger Feed Conveyor		2	1
Bagger System		1	1
Water Pumps		100	25
Compressor	2-one running 375 cfm	100	60
Lights/Sump Pumps/Cranes etc.		50	10
Totals		2039	1370

The average operating horsepower equates to an average demand of 1022 kW or 8.2 kWh/tonne feed (40.9 kWh/tonne product). The Yukon Electrical Company Limited rate schedule Y-31 calls for a demand charge of $26 700/year and an energy charge of $0.063/kWh. Applying these data yield an annual power cost of $129 700.

Fuel will be consumed in the processing plant during operation of the product handling forklift. An overall annual allowance of $3 000 is proposed.

Since the plant is operated during the warmer 7 months of the year fuel is not required for comfort or process heating.

5.5 PACKAGING

Product is moved to market in re-fillable 2000 kg capacity bulk bags containing 1750 kg of dry product. For this study it is assumed that an average of 9.1 trips are possible before bags are discarded. On this basis one bag will move 16 t of product to market. Since a single bag costs $35, annual costs are $87 500 or $2.20 per ton.

5.6 TRANSPORTATION

As discussed elsewhere there are several ways in which product might be moved to market. A preliminary proposal has been received from Yukon Freight Lines of $57.50/s.ton ($63.25/tonne) delivered to Northern B.C./Alberta. The product will be shipped with approximately 8% moisture. The freight rate on a dry tonnage basis therefore becomes $68.75/dry tonne product or $2 750 000/a.

Empty bags must be returned to Whitehorse for refilling. Approximately 1600 bags can be loaded into a standard van for a return cost of $2.20/bag or $1.25/t dry product. Total transportation cost therefore becomes $70/dry tonne of product or $2 800 000/a.

6.2 PRESENT TAILINGS DAMS

As shown on the drawing overleaf, tailings are located in two general areas viz adjacent to the mill (Areas 1-4) and in valleys North of the mill (A and B valleys). Areas 1-4 were used until 1975 and the North valleys thereafter.

In the absence of detailed records or surveys the mass distribution of tailings between the areas alongside the mill and the valleys has been estimated from historical data. Detailed distribution within the two general areas has been calculated on the basis of surface areas.

The magnetite content of each dam has been evaluated on the basis of preliminary samples taken in early October 1984. Further information concerning the sampling and subsequent analyses are recorded in the preliminary Kilborn report and the Lakefield report.

The inventory of magnetite in the tailings is recorded in Table 6-1 below. Tailings moisture levels in early October ranged from 3 to 20% in the tailings areas adjacent to the mill and from 6 to 27% in the valley dams. Size analyses show that the Area 1 tailings are about 54% -200 mesh. The other tailings areas are somewhat finer in size.

TABLE 6-1

APPROXIMATE MAGNETITE RESERVES, WHITEHORSE COPPER TAILINGS

Area	Estimated	Magnetite Content %		Magnetite
Designation	Tailings-tonnes	Range	Average	Reserve-tonnes
1	1 670 000	15 - 34	23	384 000
3	1 160 000	15 - 27	20	232 000
4	1 480 000	16 - 28	20	296 000
A	3 340 000	8 - 26	17	568 000
B	2 050 000	15 - 21	18	369 000
Totals	9 700 000	8 - 34	19	1 849 000

When examined in October, all tailings areas were found to be competent and could be worked by machines without difficulty. A possible exception is the decant area of dam A which presently contains a small pond of water. However, this is of little concern since the A area would not be reclaimed for many years at which time the pond, if present, could be drained.

Access to all tailings areas is by way of gravel roads which are presently in good condition.

6.3 TAILINGS RECLAMATION SYSTEMS

Several reclamation schemes have been studied including:

- Hydraulic monitoring and pumping to the process plant.

- Reclamation with an elevating scraper and dumping into a new hopper or a pile and bulldozing into the existing hopper.

- Reclamation by front-end loader and rear-dump truck transport to existing truck dump hopper.

- Front-end loader reclaim into a re-slurry tank located on the tailings and pumping to the process plant.

The latter alternative was selected for the following reason:

- A Caterpillar 966 wheel loader is on site and would be aquired with the property.

- The selected technique offers low operating and capital costs and allows use of existing materials.

In the proposed system the wheel loader digs tailings and transport them to a reslurry plant. The reslurry plant incorporates a hopper with belt feeder, an elevating conveyor, a trash removal screen, an agitated reslurry tank and a return pump. During operation the system meters tailings over the screen and into the reslurry tank. Water is added to the screen and tank at a rate sufficient to give a 50% solids slurry.

As reclamation proceeds the travel distance for the loader increases and costs will rise. When the distance is about 200 m, the reslurry system and associated power and pipe lines must be relocated to a point closer to the reclaim face. This movement would be required perhaps once or twice per operating season.

6.4 MAGNETITE RECOVERY

Reclaimed tailings pumped from the dam area are delivered to the existing rod mill. This unit is operated with a low load of rods to condition the plant feed.

Slurry from the rod mill is pumped to a new 48 in. x 120 in. magnetic separator located on a new, high-level floor in the grinding area of the mill. Primary separator tailings are sampled and then gravitated to the tailings pump box. The latter is the original pump box relocated into the grinding bay.

The tailings pump discharges into a 10 in. diameter line which conducts tailings to the old Little Chief open pit mine. Tailings slurry settles in the pit and clear water is reclaimed by a submersible pump/syphon system for re-use in the process.

The primary magnetic concentrate is reground to 93% -325 mesh in the existing ball mill. A new cyclone cluster, comprising ten 6 in. diameter cyclones and ancillary launders, is used to effect the required classification.

The ground concentrate is run through existing pipes into the existing Denver conditioner tank which is used to provide surge capacity ahead of the secondary magnetic separator.

The secondary separator is a double drum, Steffenson type cleaner with internal, intermediate repulping. Tailings from this unit are returned to the primary separator feed tank to ensure recovery of any misplaced magnetite.

The cleaner concentrate is filtered with the use of a dewatering aid to a cake moisture level of about 8%. It is then transferred through the existing dryer to new conveying and semi-automatic bagging equipment. It can be noted that the drier could be used to further reduce moisture levels if required.

The final product is packed in bulk bags holding 1 750 kg of product (dry basis). Bags are provided with a top filling spout and a bottom dump spout equipped with a draw string closure. Loops are provided to permit fork-lift movement of loaded bags.

The proposed bags are re-useable and an average of 9 trips before retirement is anticipated. Other methods of transportation and storage are possible and are discussed in the following section.

The only reagent used in the process is the dewatering aid. The proposed material is non-toxic and bio-degradable. At the anticipated circuit operating pH level of about 8, the decomposition time is 6 days. Since the new tailings pit will provide a far longer residence time, problems are not anticipated.

6.5 TRANSPORTATION TO MARKET AND STORAGE

The 40 000 dry tonnes of product require transportation from the Yukon to markets in Northern British Columbia and Alberta. A number of alternatives have been investigated including:

1. Road or rail to Skagway or Haines, Alaska, barging to Ridley Island (Prince Rupert) and road/rail transport inland.

2. As above but landing at Vancouver and inland transportation by road/rail.

3. Road transportation on the Alaskan Highway to Dawson Creek and thence by provincial highways into B.C. and Alberta.

Recent correspondence with Yukon Freight Lines Ltd. produced an estimate of $57.50 per short ton for back-haul trucking of material. This equates to $68.75/dry tonne of product. Occasionally a van will be required to transport empty bags back to the plant. This will increase costs to $70/t product.

In the proposed back-haul transportation system, empty food and beverage vans will stop at the plant site and be loaded with 20 to 30 t of product prior to returning to B.C. At the present time Yukon Freight Lines Ltd. have approximately 1200 trucks per year return empty from the Yukon.

Loading of vans is achieved using a new timber ramp equipped with a dock leveller. A heavy duty fork-lift truck is used for loading operation. Offloading operations at customers plants will similarly require the use of a dock and fork-lift truck.

Magnetite is only produced during the summer but is consumed by customers throughout the year. Magnetite consumers generally have bulk storage facilities equal to three months of consumption. At the end of a production season, the supply/storage situation will be as noted overleaf in Table 6-2.

TABLE 6-2
MAGNETITE DISPOSITION AT END OF SEASON

Item		Tonnes Dry Product
Magnetite produced over season		40 000
Magnetite consumed by users		22 500
Magnetite in storage	- total	17 500
	- users	10 000
	- project	7 500

In order to match the available back-haul rate, the magnetite stored by the project will have to be kept on the property at Whitehorse. The quantity to be stored is most conveniently held in shipping bags since covered bulk storage space on the property is limited and bagged product may be kept outside.

The area adjacent to the bagging equipment can accommodate about 500 tonnes of dry product in bags. The remaining 7 000 t of storage would be located outside in the general area of the new loading ramp and in the shops. The storage factor is 3 t/m² therefore about 2 300 m² (25 000 ft² or 160 x 160 ft) is required at site. The average reclaim rate during the winter season will be 70 tonnes or about 40 bags per week day and require the full time attention of one day-shift operator.

6.6 WASTE AND EFFLUENT DISPOSAL

The Little Chief open pit produced in excess of 1 100 000 tonnes of ore and at least 1 300 000 tonnes of waste. With an in-situ density of 2.8 tonnes/m³ the pit volume is about 850 000 m³. Assuming a deposited tailings density of 1.62 dry tonnes/m³ (101 lb/ft³) the pit can store 1 400 000 t of tailings corresponding to approximately 9 years of production.

A number of options are available for storing new tailings after the Little Chief open pit is full. These include:

- Use of the underground workings, from which 2 600 000 tonnes of ore alone were removed - enough volume for at least 9 years magnetite recovery operation.

- Use of Area 1 tailings dam which will be empty after about 8 years of operation.

- Use of other open pits in the area, e.g. the War Eagle (899 000 t ore), Arctic Chief (202 000 t ore) and Black Cub South (167 000 t ore). Cumulatively these three pits would be adequate for about 10 years operation if strip ratios were similar to that at Little Chief.

As noted previously the process only employs a bio-degradable dewatering aid and no other reagent. Furthermore, the project will be a net consumer of water; a discharge stream will not be produced by the process. The actual rate of fresh water consumption is minor, amounting to about 17.3 m^3/h (76 USGPM) or 0.14 m^3 water/t tailings handled (33 US gallons/ton of tailings). This water could be obtained from the Yukon river using the original water pumps. Alternatively fresh water could be obtained from Copper Lake or from one of the un-named lakes on the ridge which is South of the plant site.

7.0 ABANDONMENT REQUIREMENTS

The Yukon Territory Water Board has reviewed an abandonment plan submitted to the Board by Whitehorse Copper Mines Ltd. Kilborn have not yet reviewed Whitehorse Copper's plan but have received a copy of the Board's response to the plan. Denison and Kilborn representatives have met with officials of the Board.

The Board's main concern appears to be the migration of ground water to the Yukon river and resulting bank-slides. It is not clear if the Whitehorse Copper tailings actually contribute to slides or whether there is simply a local perception of the connection between the tailings and the slides. The Board felt that diversion of ground water flow would be necessary if the tailings remain in their present location. When the Board was told of the possible relocation of the tailings they expressed confidence that the perceived problem would be solved.

Other concerns or interests expressed by the Board include:

- The effect of pits and waste dumps on water quality.
- Proposals regarding revegetation of tailings.
- Quality of water leaving tailings areas.
- Hydrology of proposed spillways etc.
- The long term maintenance of spillways, dams, etc.

In discussing the proposed development with the Board it was apparent that the proposal would receive their endorsement for the following reasons:

1. The board would have an operating, locally based company with which to deal.
2. The development would fill at least the Little Chief pit and possibly others and thereby reduce concerns.
3. The Board perceived that the project would remove tailings from present areas and relocate them in more suitable, controlled areas.

8.0 PROJECT SCHEDULE

A preliminary project summary schedule is provided overleaf. The schedule assumes the use of Sala magnetic separators for which a six month delivery has been promised. Eriez have offered better delivery times for separators but this selection would not necessarily improve the start-up date.

The attached schedule shows a start-up in late July 1985 provided that the property is expeditiously aquired and magnetic separators are ordered early in December 1984. All other equipment would require ordering by the end of December 1984.

Production during 1985 should equate to about 20 000 dry tonnes of product.

9.0 MANPOWER

.1 CONSTRUCTION

Approximately 4 000 manhours of non-contracted installation labour are required spread over 6 months. This equates to a construction crew of 4 men. The operating crew comprises a working manager and 4 men. Because of the nature of the construction tasks it is proposed that construction be executed by the plant operators supplemented by specialized or general contractors as required to complete the job.

Three employees of Hudson Bay Mining and Smelting are presently performing clean-up, maintenance and security duties at Whitehorse Copper. It is proposed that these men be retained for their local knowledge and relevant skills.

All construction will be executed by local personnel living at home in Whitehorse or the immediate area. Construction camps, travel allowance, subsistence allowance and most other factors which contribute to high construction overhead (indirect) costs will be avoided.

.2 OPERATING

The proposed project must operate with the minimum of staff. Specialists cannot be supported; all personnel must be versatile and capable of operating, performing maintenance and carrying out repair work.

A project manager is required to liaise with local government agencies, control payroll and accounting, report to head office and otherwise superintend the operation.

The processing plant requires three operators. Any one man must be able to execute any one of the several general tasks covered by processing. These include process operator, maintenance man and bagger operator. The latter job is virtually full time; 160 bags per day have to be placed into the bagger, removed by fork-lift and placed into vans. It is likely that the job of bagging will be done in short spells by each of the operators.

The task of reclaiming the tailings with the wheel loader is performed by a single operator working a 12 hour shift during the production season. Reclamation operating will be alternated with process operating by the employees.

During the operating season of 29 weeks each of the 5 project employees works 12 hours per day, five days a week for a total of 1 740 h/man. During the remaining 23 weeks of the year the only tasks are as follows:

- Security control

- Maintenance

- Loading into vans of 7 500 tonnes (4 300 bags) of magnetite on site at end of production season. This task involves the loading of about 300 vans or about 3 vans every week day.

The forementioned tasks will be performed over the 23 week (115 working days) non-production season by the five project employees, each of whom will work 25 eight hour days to give an annual time of 1940 hours.

TABLE 9-1
OPERATING MANPOWER

Job Title	Number	Annual Costs $/year — Salary per man	Payroll Burden per man	Total Cost
Project Manager	1	45 000	13 000	58 000
Process Operator	3	25 000	8 000	99 000
Reclaim Operator	1	25 000	8 000	33 000
TOTALS	5	$145 000	$45 000	$190 000

10.0 ADDITIONAL DATA REQUIREMENTS

The following data should be obtained or confirmed prior to a final commitment to the project.

Additional Sampling

Areas 1, 3 and 4 should be re-sampled to greater depths.

Additional Testwork

Additional composite samples have been made up and shipped to Sala and Eriez for confirmatory test work. The work should be completed by mid-December and should refine or confirm present estimates of rougher recovery and grade, cleaner recovery and grade, rougher concentrate grinding work index and magnetic separator requirements.

Product Evaluation

The test program will produce several kilograms of typical product which should then be made available to potential customers.

Product Transportation

The transportation alternatives should be further examined and best attainable shipping rates negotiated and confirmed.

Process Plant Equipment

The condition of all equipment necessary for the proposed plant must be carefully ascertained through field studies. The availability of spares in the warehouse should also be verified.

DMW #1

An Investigation of

THE RECOVERY OF MAGNETITE

from tailing samples

submitted by

KILBORN LIMITED

Progress Report No.1

Project No. L.R. 2898

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.

LAKEFIELD RESEARCH
A Division of Falconbridge Limited
Lakefield, Ontario
November 21, 1984

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INTRODUCTION

In a letter dated October 9, 1984, Mr. J.R. Goode of Kilborn Limited requested that we conduct analytical work and bench scale testwork on tailing samples to investigate the recovery of magnetite. The results and direction of the testwork were frequently discussed in telephone conversations with Mr. Goode.

LAKEFIELD RESEARCH

D. M. Wyslouzil

D.M. Wyslouzil, P. Eng.,

Manager.

K.W. Sarbutt.

K.W. Sarbutt,

Chief Project Engineer.

R.W. Deane.

R.W. Deane,

Chief Mineralogist.

Experimental Work by: L. Paquette

O.F.C. Cook

R.G. Irwin

ANALYTICAL WORK

1. Samples 1 to 74

The moisture content and magnetic Fe content for samples 1 to 74 was determined.

Analytical Work - Cont'd

1. Samples 1 to 74 - Cont'd

Sample No.	% Moisture	Dry Weight, g	% Mag. Fe	Sample No.	% Moisture	Dry Weight, g	% Mag. Fe
1	4.0	122.5	23.2	38	6.3	124.7	19.0
2	6.2	122.4	17.5	39	14.5	102.6	14.4
3	8.1	132.9	20.1	40	11.8	102.3	16.0
4	8.6	134.3	14.9	41	17.7	108.4	12.8
5	15.0	125.9	11.9	42	26.6	166.7	11.3
6	4.4	116.5	24.3	43	22.0	105.1	12.2
7	4.0	80.7	14.0	44	16.6	102.3	11.3
8	4.1	122.7	20.3	45	13.6	82.5	10.5
9	8.7	124.5	18.0	46	13.5	118.1	13.3
10	6.6	125.8	13.2	47	14.3	104.6	11.9
11	7.1	100.0	12.5	48	13.9	105.9	12.6
12	5.9	105.5	12.5	49	6.9	111.3	14.9
13	9.8	123.1	10.6	50	9.3	110.3	14.6
14	6.7	114.1	14.6	51	12.9	119.5	10.7
15	8.5	116.9	15.1	52	10.0	132.9	15.3
16	12.0	117.4	15.4	53	11.4	130.2	16.1
17	11.0	121.7	12.8	54	10.0	129.8	19.5
18	11.3	127.8	11.4	55	9.3	136.6	13.6
19	5.9	100.9	18.8	56	6.4	121.4	14.4
20	6.5	126.5	21.7	57	4.3	103.0	16.9
21	11.6	117.7	13.6	58	7.4	120.9	16.9
22	9.3	120.8	16.0	59	3.4	120.6	12.5
23	8.9	114.8	16.4	60	5.8	111.5	14.7
24	7.0	124.1	13.5	61	9.0	116.6	13.8
25	18.4	115.9	8.69	62	4.0	113.6	15.6
26	16.7	110.6	7.82	63	14.3	128.2	12.8
27	6.2	116.6	8.54	64	13.3	125.7	18.0
28	8.4	113.8	14.3	65	20.2	121.0	13.9
29	15.7	88.8	7.60	66	13.0	106.8	11.8
30	24.2	58.5	6.68	67	5.6	117.0	13.0
31	25.9	71.9	6.12	68	9.4	128.8	19.6
32	26.7	73.3	5.50	69	5.6	129.0	20.3
33	15.9	73.3	9.92	70	8.4	118.1	17.3
34	14.9	97.6	10.5	71	9.4	126.5	11.9
35	12.4	107.8	8.4	72	9.5	122.6	17.5
36	5.6	127.2	17.2	73	10.3	131.9	17.7
37	7.9	102.9	12.6	74	14.6	123.9	12.5

Analytical Work - Cont'd

Additional analyses were conducted on samples 8 to 18.

Sample No.	Assays, %			Assays, g/t	
	Fe (Total)	Cu	Ga	Au	Pt
8	23.6	0.13	0.0006	0.18	<0.10
9	21.1	0.14	0.0006	0.13	<0.10
10	16.2	0.12	0.0002	0.11	<0.10
11	16.1	0.12	0.0002	0.16	<0.10
12	15.4	0.11	0.0004	0.13	<0.10
13	14.3	0.12	0.0002	0.10	<0.10
14	17.4	0.16	0.0003	0.24	<0.10
15	18.6	0.14	0.0003	0.13	<0.10
16	18.1	0.16	0.0004	0.09	<0.10
17	16.0	0.12	0.0002	0.07	<0.10
18	14.5	0.11	0.0002	0.10	<0.10

Size analyses were conducted on samples 1, 8-18, 23, 25, 27-34, 41-48, 50-54, 60, 63-66 and 72.

Size Analyses

Sample No. 1

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 35	1.1	1.1	98.9
48	5.8	6.9	93.1
65	13.1	20.0	80.0
100	17.1	37.1	62.9
150	17.8	54.9	45.1
200	18.2	73.1	26.9
270	16.4	89.5	10.5
400	5.8	95.3	4.7
- 400	4.7	100.0	-
Total	100.0	-	-

Analytical Work - Cont'd

Screen Analyses - Cont'd

Sample No. 8

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	8.6	8.6	91.44
100	10.6	19.2	80.8
150	15.9	35.1	64.9
200	15.9	51.0	49.0
270	16.7	67.7	32.3
400	10.6	78.3	21.7
- 400	21.7	100.0	-
Total	100.0	-	-

Sample No. 9

+ 65	5.4	5.4	94.6
100	7.9	13.3	86.7
150	12.9	26.2	73.8
200	14.1	40.3	59.7
270	15.8	56.1	43.9
400	11.2	67.3	32.7
- 400	32.7	100.0	-
Total	100.0	-	-

Sample No. 10

+ 65	10.4	10.4	89.6
100	13.7	24.1	75.9
150	16.5	40.6	59.4
200	15.3	55.9	44.1
270	11.6	67.5	32.5
400	8.8	76.3	23.7
- 400	23.7	100.0	-
Total	100.0	-	-

Analytical Work - Cont'd

Screen Analyses - Cont'd

Sample No. 11

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	3.3	3.3	96.7
100	7.7	11.0	89.0
150	12.5	23.5	76.5
200	16.2	39.7	60.3
270	17.6	57.3	42.7
400	12.9	70.2	29.8
- 400	29.8	100.0	-
Total	100.0	-	-

Sample No. 12

+ 65	6.1	6.1	93.9
100	11.2	17.3	82.7
[illegible]	[illegible]	[illegible]	[illegible]
400	10.2	76.4	23.6
- 400	23.6	100.0	-
Total	100.0	-	-

Sample No. 13

+ 65	3.6	3.6	96.4
100	7.7	11.3	88.7
150	13.4	24.7	75.3
200	13.4	38.1	61.9
270	14.6	52.7	47.3
400	11.7	64.4	35.6
- 400	35.6	100.0	-
Total	100.0	-	-

Analytical Work - Cont'd

Screen Analyses - Cont'd

Sample No. 14

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	13.5	13.5	86.5
100	14.4	27.9	72.1
150	15.3	43.2	56.8
200	14.0	57.2	42.8
270	10.4	67.6	32.4
400	8.1	75.7	24.3
- 400	24.3	100.0	-
Total	100.0	-	-

Sample No. 15

+ 65	4.9	4.9	95.1
100	7.1	12.0	88.0
150	10.7	22.7	77.3
200	15.2	37.9	62.1
270	15.2	53.1	46.9
400	11.2	64.3	35.7
- 400	35.7	100.0	-
Total	100.0	-	-

Sample No. 16

+ 65	3.6	3.6	96.4
100	5.8	9.4	90.6
150	9.0	18.4	81.6
200	10.8	29.2	70.8
270	14.3	43.5	56.5
400	11.7	55.2	44.8
- 400	44.8	100.0	-
Total	100.0	-	-

Analytical Work - Cont'd

Screen Analyses - Cont'd

Sample No. 17

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	3.2	3.2	96.8
100	6.5	9.7	90.3
150	11.3	21.0	79.0
200	13.0	34.0	66.0
270	13.0	47.0	53.0
400	12.9	59.9	40.1
- 400	40.1	100.0	-
Total	100.0	-	-

Sample No. 18

+ 65	3.9	3.9	96.1
100	7.8	11.7	88.3
150	13.3	25.0	75.0
200	12.5	37.5	62.5
270	13.7	51.2	48.8
400	10.9	62.1	37.9
- 400	37.9	100.0	- -
Total	100.0	-	-

Sample No. 23

+ 65	3.5	3.5	96.5
100	5.3	8.8	91.2
150	9.9	18.7	81.3
200	14.2	32.9	67.1
270	14.9	47.8	52.2
400	11.0	58.8	41.2
- 400	41.2	100.0	-
Total	100.0	-	-

Analytical Work - Cont'd

Screen Analyses - Cont'd

Sample No. 25

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 100	1.1	1.1	98.9
150	5.2	6.3	93.7
200	8.9	15.2	84.8
270	13.8	29.0	71.0
400	12.6	41.6	58.4
- 400	58.4	100.0	-
Total	100.0	-	-

Sample No. 27

+ 65	3.2	3.2	96.8
100	10.9	14.1	85.9
150	16.8	30.9	69.1
200	16.4	47.3	52.7
270	13.2	60.5	39.5
400	10.5	71.0	29.0
- 400	29.0	100.0	-
Total	100.0	-	-

Sample No. 28

+ 65	5.7	5.7	94.3
100	8.9	14.6	85.4
150	14.9	29.5	70.5
200	16.4	45.9	54.1
270	15.3	61.2	38.8
400	9.3	70.5	29.5
- 400	29.5	100.0	-
Total	100.0	-	-

Analytical Work - Cont'd

Screen Analyses - Cont'd

Sample No. 29

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	2.7	2.7	97.3
100	5.4	8.1	91.9
150	9.4	17.5	82.5
200	10.3	27.8	72.2
270	11.6	39.4	60.6
400	8.5	47.9	52.1
- 400	52.1	100.0	-
Total	100.0	-	-

Sample No. 30

+ 65	1.0	1.0	99.0
100	2.4	3.4	96.6
150	4.2	7.6	92.4
200	5.6	13.2	86.8
270	6.3	19.5	80.5
400	5.9	25.4	74.6
- 400	74.6	100.0	-
Total	100.0	-	-

Sample No. 31

+ 65	1.8	1.8	98.2
100	2.6	4.4	95.6
150	3.1	7.5	92.5
200	2.6	10.1	89.9
270	3.5	13.6	86.4
400	4.0	17.6	82.4
- 400	82.4	100.0	-
Total	100.0	-	-

Analytical Work - Cont'd

Screen Analyses - Cont'd

Sample No. 32

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	0.8	0.8	99.2
100	1.7	2.5	97.5
150	2.1	4.6	95.4
200	2.1	6.7	93.3
270	2.5	9.2	90.8
400	3.4	12.6	87.4
- 400	87.4	100.0	-
Total	100.0	-	-

Sample No. 33

+ 65	2.9	2.9	97.1
100	6.6	9.5	90.5
150	9.5	19.0	81.0
200	10.2	29.2	70.8
270	11.3	40.5	59.5
400	8.8	49.3	50.7
- 400	50.7	100.0	-
Total	100.0	-	-

Sample No. 34

+ 65	7.8	7.8	92.2
100	9.8	17.6	82.4
150	12.3	29.9	70.1
200	10.7	40.6	59.4
270	11.1	51.7	48.3
400	7.8	59.5	40.5
- 400	40.5	100.0	-
Total	100.0	-	-

Analytical Work - Cont'd

Screen Analyses - Cont'd

Sample No. 41

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	0.4	0.4	99.6
100	1.1	1.5	98.5
150	3.0	4.5	95.5
200	4.5	9.0	91.0
270	7.1	16.1	83.9
400	9.0	25.1	74.9
- 400	74.9	100.0	-
Total	100.0	-	-

Sample No. 42

+ 100	0.4	0.4	99.6
150	1.1	1.5	98.5
200	1.9	3.4	96.6
270	3.3	6.7	93.3
400	5.6	12.3	87.7
- 400	87.7	100.0	-
Total	100.0	-	-

Sample No. 43

+ 65	1.1	1.1	98.9
100	3.4	4.5	95.5
150	5.7	10.2	89.8
200	8.4	18.6	81.4
270	8.4	27.0	73.0
400	6.5	33.5	66.5
- 400	66.5	100.0	-
Total	100.0	-	-

Analytical Work - Cont'd

Screen Analyses - Cont'd

Sample No. 44

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	6.8	6.8	93.2
100	8.4	15.2	84.8
150	10.4	25.6	74.4
200	8.4	34.0	66.0
270	8.4	42.4	57.6
400	6.8	49.2	50.8
- 400	50.8	100.0	-
Total	100.0	-	-

Sample No. 45

+ 48	5.7	5.7	94.3
65	10.6	16.3	83.7
100	10.6	26.9	73.1
150	11.4	38.3	61.7
200	11.0	49.3	50.7
270	10.6	59.9	40.1
400	7.6	67.5	32.5
- 400	32.5	100.0	-
Total	100.0	-	-

Sample No. 46

+ 65	3.1	3.1	96.9
100	7.7	10.8	89.2
150	14.0	24.8	75.2
200	14.9	39.7	60.3
270	13.6	53.3	46.7
400	11.3	64.6	35.4
- 400	35.4	100.0	-
Total	100.0	-	-

Analytical Work - Cont'd

Screen Analyses - Cont'd

Sample No. 47

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	9.4	9.4	90.6
100	12.1	21.5	78.5
150	14.1	35.6	64.4
200	11.3	46.9	53.1
270	10.5	57.4	42.6
400	7.0	64.4	35.6
- 400	35.6	100.0	-
Total	100.0	-	-

Sample No. 48

+ 65	6.4	6.4	93.6
100	12.1	18.5	81.5
150	14.3	32.8	67.2
200	14.3	47.1	52.9
270	12.8	59.9	40.1
400	7.9	67.8	32.2
- 400	32.2	100.0	-
Total	100.0	-	-

Sample No. 50

+ 65	4.9	4.9	95.1
100	7.8	12.7	87.3
150	12.1	24.8	75.2
200	14.2	39.0	61.0
270	16.3	55.3	44.7
400	11.7	67.0	33.0
- 400	33.0	100.0	-
Total	100.0	-	-

Analytical Work - Cont'd

Screen Analyses - Cont'd

Sample No. 51

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	2.2	2.2	97.8
100	5.8	8.0	92.0
150	10.3	18.3	81.7
200	10.8	29.1	70.9
270	11.2	40.3	59.7
400	11.2	51.5	48.5
- 400	48.4	100.0	-
Total	100.0	-	-

Sample No. 52

+ 65	4.0	4.0	96.0
100	6.8	10.8	89.2
150	11.6	22.4	77.6
200	12.4	34.8	65.2
270	14.3	49.1	50.9
400	11.6	60.7	39.3
- 400	39.3	100.0	-
Total	100.0	-	-

Sample No. 53

+ 65	2.2	2.2	97.8
100	4.0	6.2	93.8
150	8.9	15.1	84.9
200	13.3	28.4	71.6
270	15.1	43.5	56.5
400	11.6	55.1	44.9
- 400	44.9	100.0	-
Total	100.0	-	-

Analytical Work - Cont'd

Screen Analyses - Cont'd

Sample No. 54

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	1.3	1.3	98.7
100	3.4	4.7	95.3
150	8.4	13.1	86.9
200	11.3	24.4	75.6
270	16.4	40.8	59.2
400	14.7	55.5	44.5
- 400	44.5	100.0	-
Total	100.0	-	-

Sample No. 60

+ 48	5.9	5.9	94.1
65	9.5	15.4	84.6
100	17.0	32.4	67.6
150	18.2	50.6	49.4
200	15.4	66.0	34.0
270	10.7	76.7	23.3
400	6.7	83.4	16.6
- 400	16.6	100.0	-
Total	100.0	-	-

Sample No. 63

+ 65	0.4	0.4	99.6
100	1.3	1.7	98.3
150	5.3	7.0	93.0
200	7.6	14.6	85.4
270	11.6	26.2	73.8
400	12.4	38.6	61.4
- 400	61.4	100.0	-
Total	100.0	-	-

Analytical Work - Cont'd

Screen Analyses - Cont'd

Sample No. 64

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	0.8	0.8	99.2
100	2.5	3.3	96.7
150	7.1	10.4	89.6
200	9.6	20.0	80.0
270	13.4	33.4	66.6
400	13.4	46.8	53.2
- 400	53.2	100.0	-
Total	100.0	-	-

Sample No. 65

+ 65	0.4	0.4	99.6
100	1.1	1.5	98.5
150	3.0	4.5	95.5
200	4.9	9.4	90.6
270	7.8	17.2	82.8
400	9.3	26.5	73.5
- 400	73.5	100.0	-
Total	100.0	-	-

Sample No. 66

+ 65	2.6	2.6	97.4
100	7.8	10.4	89.6
150	13.3	23.7	76.3
200	13.3	37.0	63.0
270	11.5	48.5	51.5
400	10.4	58.9	41.1
- 400	41.1	100.0	-
Total	100.0	-	-

Analytical Work - Cont'd

Screen Analyses - Cont'd

Sample No. 72

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 100	1.3	1.3	98.7
150	6.9	8.2	91.8
200	11.3	19.5	80.5
270	15.2	34.7	65.3
400	15.2	49.9	50.1
- 400	50.1	100.0	-
Total	100.0	-	-

Analytical Work - Cont'd

2. Samples 75 to 88

Samples 75-88 were analyzed for Mag. Fe, total Fe, Cu, Ga, Au and Pt.

Sample No.	Assay, %				Assay, g/t	
	Total Fe	Mag. Fe	Cu	Ga	Au	Pt
75	57.3	53.8	0.16	0.0011	0.14	<0.10
76	3.57	0.93	0.16	0.0004	1.49	<0.20
77	2.90	0.41	0.13	0.0002	0.21	<0.10
78	57.6	54.6	0.15	0.0012	0.12	<0.10
79	3.76	1.01	0.14	<0.0001	1.29	<0.10
80	3.13	0.41	0.14	<0.0001	0.14	<0.10
81	50.5	45.6	0.19	0.0011	0.48	<0.10
82	5.60	2.90	0.23	0.0011	2.76	<0.30
83	3.26	0.77	0.19	<0.0001	0.09	<0.10
84	49.9	43.6	0.21	0.0009	0.10	<0.10
85	3.26	0.93	0.18	<0.0001	0.14	<0.10
86	2.85	0.46	0.14	<0.0001	0.13	<0.10
87	2.93	0.33	0.13	<0.0001	0.12	<0.10
88	13.7	10.4	0.14	0.0002	0.79	<0.10

The gold assays on some of the samples were repeated:

Sample No.	Original Assay		Repeat Assay	
	Sample Wt. g	Assay, Au, g/t	Sample Wt. g	Assay, Au, g/t
76	5	1.49	1.7	0.18
79	15	1.29	3.7	0.14
81	15	0.48	2.4	0.16
82	3	2.76	1.4	1.19
88	15	0.79	15	0.13

Analytical Work - Cont'd

2. Samples 75 to 88

Samples 75-88 were analyzed for Mag. Fe, total Fe, Cu, Ga, Au and Pt.

Sample No.	Assay, %				Assay, g/t	
	Total Fe	Mag. Fe	Cu	Ga	Au	Pt
75	57.3	53.8	0.16	0.0011	0.14	<0.10
76	3.57	0.93	0.16	0.0004	1.49	<0.20
77	2.90	0.41	0.13	0.0002	0.21	<0.10
78	57.6	54.6	0.15	0.0012	0.12	<0.10
79	3.76	1.01	0.14	<0.0001	1.29	<0.10
80	3.13	0.41	0.14	<0.0001	0.14	<0.10
81	50.5	45.6	0.19	0.0011	0.48	<0.10
82	5.60	2.90	0.23	0.0011	2.76	<0.30
83	3.26	0.77	0.19	<0.0001	0.09	<0.10
84	49.9	43.6	0.21	0.0009	0.10	<0.10
85	3.26	0.93	0.18	<0.0001	0.14	<0.10
86	2.85	0.46	0.14	<0.0001	0.13	<0.10
87	2.93	0.33	0.13	<0.0001	0.12	<0.10
88	13.7	10.4	0.14	0.0002	0.79	<0.10

The gold assays on some of the samples were repeated:

Sample No.	Original Assay		Repeat Assay	
	Sample Wt. g	Assay, Au, g/t	Sample Wt. g	Assay, Au, g/t
76	5	1.49	1.7	0.18
79	15	1.29	3.7	0.14
81	15	0.48	2.4	0.16
82	3	2.76	1.4	1.19
88	15	0.79	15	0.13

Analytical Work - Continued

Semi-quantitative XRF scans were conducted on samples 84 to 88.

Element	Sample 84	Sample 85	Sample 86	Sample 87	Sample 88
Titanium	ND	ND	ND	ND	ND
Chromium	ND	ND	ND	ND	ND
Manganese	TL	T	T	T	T
Iron	H	LM	LM	LM	MH
Cobalt	ND	ND	ND	ND	ND
Nickel	ND	ND	ND	ND	ND
Copper	TL	TL	TL	TL	TL
Zinc	T	T	T	T	FT
Arsenic	ND	ND	ND	ND	ND
Bismuth	ND	ND	ND	ND	ND
Lead	ND	ND	ND	ND	ND
Uranium	ND	ND	ND	ND	ND
Thorium	ND	ND	ND	ND	ND
Yttrium	ND	ND	ND	ND	ND
Columbium	ND	ND	ND	ND	ND
Molybdenum	ND	ND	ND	ND	ND
Silver	ND	ND	ND	ND	ND
Cadmium	ND	ND	ND	ND	ND
Tin	ND	ND	ND	ND	ND
Antimony	ND	ND	ND	ND	ND

CODE:
H - 10 % plus
MH - 5 - 15 %
M - 1 - 10 %
LM - .5 - 5 %
L - .1 - 1 %
TL - .05 - .5 %
T - .01 - .1 %
FT - Less than .01 %.
ND - Not Detected

Analytical Work - Cont'd

Screen analyses were conducted on samples 75, 77, 78, 80 and 83-88.

Screen Analyses

Sample No. 75

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	1.0	1.0	99.0
100	2.0	3.0	97.0
150	5.1	8.1	91.9
200	18.8	26.9	73.1
270	21.8	48.7	51.3
400	12.2	60.9	39.1
- 400	39.1	100.0	-
Total	100.0	-	-

Sample No. 77

+ 48	2.0	2.0	98.0
65	8.8	10.8	89.2
100	12.0	22.8	77.2
150	18.1	40.9	59.1
200	16.9	57.8	42.2
270	12.4	70.2	29.8
400	8.8	79.0	21.0
- 400	21.0	100.0	-
Total	100.0	-	-

Sample No. 78

+ 65	2.3	2.3	97.7
100	3.8	6.1	93.9
150	8.0	14.1	85.9
200	16.3	30.4	69.6
270	23.5	53.9	46.1
400	14.8	68.7	31.3
- 400	31.3	100.0	-
Total	100.0	-	-

Analytical Work - Cont'd

Screen Analyses - Cont'd

Sample No. 80

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	11.5	11.5	88.5
100	14.1	25.6	74.4
150	18.4	44.0	56.0
200	14.5	58.5	41.5
270	12.8	71.3	28.7
400	8.9	80.2	19.8
- 400	19.8	100.0	-
Total	100.0	-	-

Sample No. 83

+ 65	6.5	6.5	93.5
100	10.7	17.2	82.8
150	15.0	32.2	67.8
200	15.9	48.1	51.9
270	12.1	60.2	39.8
400	7.5	67.7	32.3
- 400	32.3	100.0	-
Total	100.0	-	-

Sample No. 84

+ 48	4.3	4.3	95.7
65	9.8	14.1	85.9
100	11.8	25.9	74.1
150	13.3	39.2	60.8
200	20.0	59.2	40.8
270	20.4	79.6	20.4
400	11.0	90.6	9.4
- 400	9.4	100.0	-
Total	100.0	-	-

Analytical Work - Cont'd

Screen Analyses - Cont'd

Sample No. 85

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 35	2.8	2.8	97.2
48	10.4	13.2	86.8
65	18.9	32.1	67.9
100	27.7	59.8	40.2
150	21.7	81.5	18.5
200	8.8	90.3	9.7
270	4.0	94.3	5.7
400	1.6	95.9	4.1
- 400	4.1	100.0	-
Total	100.0	-	-

Sample No. 86

+ 35	2.5	2.5	97.5
48	8.4	10.9	89.1
65	17.1	28.0	72.0
100	25.2	53.2	46.8
150	24.3	77.5	22.5
200	11.2	88.7	11.3
270	4.7	93.4	6.6
400	1.9	95.3	4.7
- 400	4.7	100.0	-
Total	100.0	-	-

Sample No. 87

+ 35	2.2	2.2	97.8
48	8.8	11.0	89.0
65	15.9	26.9	73.1
100	24.7	51.6	48.4
150	21.1	72.7	27.3
200	14.1	86.8	13.2
270	5.3	92.1	7.9
400	1.8	93.9	6.1
- 400	6.1	100.0	-
Total	100.0	-	-

Analytical Work - Cont'd

Screen Analyses - Cont'd

Sample No. 88

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 48	8.1	8.1	91.9
65	14.4	22.5	77.5
100	20.0	42.5	57.5
150	20.7	63.2	36.8
200	14.8	78.0	22.0
270	9.3	87.3	12.7
400	4.4	91.7	8.3
- 400	8.3	100.0	-
Total	100.0	-	-

BENCH SCALE TESTWORK

1. Composite Preparation

5 composite samples were prepared from the remaining unpulverized material for testwork.

	Sample No.	Assay, % Sol. Fe	Assay, % Mag. Fe
Composite 1	1 - 24	18.4	16.4
Composite 2	25 - 40	13.6	11.7
Composite 3	41 - 50	14.8	13.0
Composite 4	51 - 62	16.8	14.9
Composite 5	63 - 74	16.9	15.1

Testwork was also conducted on a sample labelled "Grab Bulk Sample".

The size analysis of the composites were:

Composite 1

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	7.9	7.9	92.1
100	9.9	17.8	82.2
150	14.3	32.1	67.9
200	13.9	46.0	54.0
270	14.7	60.7	39.3
400	10.7	71.4	28.6
- 400	28.6	100.0	-
Total	100.0	-	-

Composite 2

+ 65	5.2	5.2	94.8
100	8.2	13.4	86.6
150	11.6	25.0	75.0
200	12.9	37.9	62.1
270	11.6	49.5	50.5
400	9.8	59.3	40.7
- 400	40.7	100.0	-
Total	100.0	-	-

Bench Scale Testwork - Cont'd

Screen Analyses - Cont'd

Composite 3

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	5.2	5.2	94.8
100	7.4	12.6	87.4
150	9.3	21.9	78.1
200	13.0	34.9	65.1
270	11.5	46.4	53.6
400	7.0	53.4	46.6
- 400	46.6	100.0	-
Total	100.0	-	-

Composite 4

+ 65	8.4	8.4	91.6
100	10.5	18.9	81.1
150	13.8	32.7	67.3
200	13.0	45.7	54.3
270	13.0	58.7	41.3
400	9.2	67.9	32.1
- 400	32.1	100.0	-
Total	100.0	-	-

Composite 5

+ 65	0.7	0.7	99.3
100	2.6	3.3	96.7
150	7.9	11.2	88.8
200	11.6	22.8	77.2
270	12.3	35.1	64.9
400	13.6	48.7	51.3
- 400	51.3	100.0	-
Total	100.0	-	-

Bench Scale Testwork - Cont'd

2. Rougher Separation

Each composite was passed once through a 305 mm x 150 mm Jeffrey single drum magnetic separator at 2.0 amperes (700-800 gauss). The results are shown in Table 1.

Table 1 - Rougher Separation Tests

Comp.	Product	Weight		Assays, %, g/tonne					% Distribution				
		g	%	Mag. Fe	Sol. Fe	Cu	S	Au	Mag. Fe	Sol. Fe	Cu	S	Au
1	Magnetics	796.5	34.3	46.6	50.6	0.13	0.07	0.12	96.9	91.4	31.1	37.8	23.9
	Non-Magnetics	1526.0	65.7	0.79	2.50	0.15	0.06	0.20	3.1	8.6	68.9	62.2	76.1
	Head (calc.)	2322.5	100.0	16.5	19.0	0.14	0.06	0.17	100.0	100.0	100.0	100.0	100.0
	(direct)	-	-	16.4	18.4	-	-	-	-	-	-	-	-
2	Magnetics	300.3	22.9	50.6	51.6	0.16	0.13	0.08	95.2	88.2	24.0	22.9	25.3
	Non-Magnetics	1009.6	77.1	0.78	2.09	0.15	0.13	0.07	4.8	11.8	76.0	77.1	74.7
	Head (calc.)	1309.9	100.0	12.2	13.4	0.15	0.13	0.07	100.0	100.0	100.0	100.0	100.0
	(direct)	-	-	11.7	13.6	-	-	-	-	-	-	-	-
3	Magnetics	209.6	25.3	48.1	52.9	0.16	0.11	0.06	94.2	87.5	22.2	18.9	13.5
	Non-Magnetics	619.6	74.7	1.01	2.57	0.19	0.16	0.13	5.8	12.5	77.8	81.1	86.5
	Head (calc.)	829.0	100.0	13.0	15.3	0.18	0.15	0.11	100.0	100.0	100.0	100.0	100.0
	(direct)	-	-	13.0	14.8	-	-	-	-	-	-	-	-
4	Magnetics	346.6	27.8	48.6	53.7	0.15	0.10	0.25	94.4	88.3	25.3	36.0	25.7
	Non-Magnetics	898.1	72.2	1.11	2.74	0.17	0.07	0.28	5.6	11.7	74.7	64.0	74.3
	Head (calc.)	1244.7	100.0	14.3	16.9	0.16	0.08	0.27	100.0	100.0	100.0	100.0	100.0
	(direct)	-	-	14.9	16.8	-	-	-	-	-	-	-	-
5	Magnetics	343.8	29.8	45.4	49.4	0.12	0.08	0.11	93.6	87.1	22.1	29.8	36.9
	Non-Magnetics	809.2	70.2	1.32	3.04	0.18	0.08	0.08	6.4	12.9	77.9	70.2	63.1
	Head (calc.)	1153.0	100.0	14.5	16.9	0.16	0.08	0.09	100.0	100.0	100.0	100.0	100.0
	(direct)	-	-	15.1	16.9	-	-	-	-	-	-	-	-

Bench Scale Testwork - Cont'd

3. Cleaner Separation

3.1. Individual Composites

The rougher magnetic concentrates were reground in 100 g batches in a pebble mill and 10 g samples were treated in a Davis tube. The Davis tube conditions were:

Tube Oscillation:	100 strokes/minute
Current to Poles:	2 amperes
Retention Time:	5 minutes
Water:	1 L/minute

The grinding and Separation results are summarized in Table 2.

Table 2 - Davis Tube Test Results

Comp.	Rougher Concentrate					Regrind					DT Conc.		
No.	Weight %	Assay % Sol. Fe	% Dist. Sol. Fe	%-400 Mesh	K_{80} µm	Time Min.	kWh/t (ro. conc.)	Product K_{80}	Product %-400	W.I. (m)	Weight* %	Assay % Sol. Fe	% Dist. Sol. Fe
1	34.3	50.6	91.4	31.8	100	10 20	18.1 36.2	42 24	73.4 89.6	33.3 34.8	23.8 23.7	70.1 68.6	89.9 89.7
2	22.9	51.6	88.2	37.3	90	7.5 15	13.6 27.2	45 28	70.4 85.6	31.1 32.5	17.0 17.0	68.6 68.7	87.3 86.9
3	25.3	52.9	87.5	45.8	78	7.5 15	13.6 27.2	42 25	76.4 89.2	33.1 31.3	18.8 18.7	68.1 89.0	86.2 86.3
4	27.8	53.7	88.3	30.9	102	7.5 15	13.6 27.2	49 29	66.4 85.2	31.0 31.4	21.7 21.5	67.3 68.4	87.2 87.1
5	29.8	49.4	87.1	62.8	54	7.5 15	13.6 27.2	32 22	83.2 91.6	33.4 35.3	21.4 21.2	68.0 67.9	85.6 85.6

* Overall

Bench Scale Testwork - Cont'd

3. Cleaner Separation - Cont'd

Additional analyses were conducted on the concentrates from selected tests.

Table 3 - Concentrate Analyses

Sample	Assays, %										S.G.
	Sol. Fe	Mag. Fe	Cu	SiO_2	Al_2O_3	P	S	CO_2	CaO	MgO	
Comp. 1 20 min DT Conc.	68.6	66.3	0.055	0.96	0.18	0.017	0.04	0.13	0.080	1.67	4.85
Comp. 2 15 min. DT Conc.	68.7	67.0	0.076	0.92	0.19	0.017	0.04	0.13	0.094	1.64	4.83
Comp. 3 7.5 min. DT Conc.	68.1	66.8	0.081	1.24	0.22	0.017	0.06	0.09	0.10	1.83	4.83
Comp. 4 15 min. DT Conc.	68.4	66.2	0.066	1.18	0.16	0.017	0.04	0.11	0.077	1.65	4.85
Comp. 5 15 min. DT Conc.	67.9	65.8	0.060	1.14	0.19	0.016	0.05	0.04	0.090	1.95	4.84

The major diluent in the concentrates was diopside, a Ca-Mg silicate.

3.2. Composite Sample

A composite was then prepared of the remaining reground rougher concentrates and a cleaner test on the Jeffrey magnetic separator conducted to compare the separation results to the Davis tube results.

The results are compared in Table 4.

Table 4 - Comparison of Davis Tube and Jeffrey Separator

Product		Weight % Overall	Assay, %		% O'all Dist.	
			Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
Comp. M1-M5	Davis Tube Conc.	20.5	68.5	66.4	87.2	94.3
	Davis Tube Tail.	7.5	2.91	1.19	1.3	0.6
Comp. M1-M5	Jeffrey Conc.	19.9	67.8	65.6	82.2	89.0
	Jeffrey Tail.	8.1	12.8	10.8	6.3	5.9

Bench Scale Testwork - Cont'd

3. Cleaner Separation - Cont'd

Concentrate grade was similar but the recovery was substantially less from the Jeffrey separator.

The Jeffrey concentrate was treated by elutriation to attempt to further upgrade it.

Table 5 - Elutriation

Product	Weight, % Overall	Assay, %		% Overall Dist.	
		Sol. Fe	Mag. Fe	Sol. Fe	Mag. Fe
Magnetic Concentrate	19.7	68.1	65.8	81.6	88.4
Slimes	0.2	46.3	43.6	0.6	0.6
Jeffrey Conc.	19.9	67.8	65.6	82.2	89.0

The concentrate was upgraded slightly by desliming. The products from the cleaner separation and elutriation were also assayed for Au and Cu and an overall balance for Au and Cu including the rougher separation stage calculated.

Table 6 - Comp. M1-M5 Overall Balance

Product	Weight %	Assays, %, g/t				% Distribution			
		Sol. Fe	Mag. Fe	Cu	Au	Sol. Fe	Mag. Fe	Cu	Au
Magnetic Conc.	19.7	68.1	65.3	0.086	0.17	81.6	88.4	10.1	20.2
Slimes	0.2	46.3	43.6	0.20	2.30	0.6	0.6	0.2	2.8
Cl. Non-Magnetics	8.1	12.8	10.8	0.34	0.24	6.3	5.9	16.5	11.7
Ro. Non-Magnetics	72.0	2.59	1.0	0.17	0.15	11.5	5.1	73.2	65.3
Comp. M1-M5 (Calc.)	100.00	16.4	14.6	0.17	0.16	100.0	100.0	100.0	100.0

Bench Scale Testwork - Cont'd

3. Cleaner Separation - Cont'd

The magnetic concentrate further assayed:

SiO_2	-	1.49 %
Al_2O_3	-	0.26 %
CaO	-	0.15 %
MgO	-	2.09 %
S	-	0.10 %

3.3. Quintette Magnetite Analysis:

A sample of Quintette magnetite (ex Craigmont) was received from Kilborn and comparative analyses and size analyses conducted. The results are compared to those of the M1 to M5 concentrate.

	Quintette Magnetite	M1-M5 Concentrate
Sol. Fe, %	60.7	68.1
Nag, Fe (Satmagan), %	57.6	65.8
Cu, %	0.058	0.086
SiO_2, %	9.26	1.49
Al_2O_3, %	1.68	0.26
P, %	0.018	0.017
S, %	0.06	0.10
CO_2, %	0.66	0.23
CaO, %	1.20	0.15
MgO, %	0.67	2.09
Specific Gravity	4.60	4.82

Mesh Size (Tyler)	Quintette Magnetite			M1-M5 Concentrate		
	% Retained		% Pass.	% Retained		% Pass.
	Ind.	Cum.	Cum.	Ind.	Cum.	Cum.
+ 65	0.1	0.1	99.9	-	-	-
100	0.2	0.3	99.7	-	-	-
150	0.4	0.7	99.3	0.2	0.2	99.8
200	2.6	3.3	96.7	1.5	1.7	98.3
270	4.6	7.9	92.1	8.1	9.8	90.2
400	19.8	17.7	82.3	14.2	24.0	76.0
- 400	82.3	100.0	-	76.0	100.0	-
Total	100.0	-	-	100.0	-	-

Bench Scale Testwork - Cont'd

4. Testwork on "Grab Bulk Sample"

A rougher separation test was conducted with the Jeffrey separator.

Table 7 - Rougher Separation

Product	Weight %	Assay, %		% Distribution	
		Mag. Fe	Sol. Fe	Mag. Fe	Sol. Fe
Rougher Magnetics	18.5	49.4	52.0	94.7	85.9
Rougher Non-Magnetics	81.5	0.63	1.94	5.3	14.1
Head (calculated)	100.00	9.7	11.2	100.0	100.0

A portion of the rougher magnetics was reground and a Davis tube test conducted. The remainder of the magnetic concentrate was split in half and two different grinds conducted. The ground material was then cleaned on the Jeffrey separator. The results of the separation are shown in Table 8.

Table 8 - Cleaner Separation

Magnetic Separator	Grind K_{80} µm	Product	Weight %	Assays, %		% Distribution	
				Mag. Fe	Sol. Fe	Mag. Fe	Sol. Fe
Davis Tube	42	Magnetics	13.9	66.7	69.1	93.9	84.4
		Non-Magnetics	4.6	1.74	3.78	0.8	1.5
		Ro. Magnetics (calc.)	18.5	50.5	52.9	94.7	85.9
Jeffrey Separator	26	Magnetics	14.5	64.6	65.6	91.4	82.3
		Non-Magnetics	4.0	8.54	10.3	3.3	3.6
		Ro. Magnetics (calc.)	18.5	52.5	53.6	94.7	85.9
Jeffrey Separator	18	Magnetics	14.6	62.5	65.3	89.9	81.0
		Non-Magnetics	3.9	12.7	14.9	4.8	4.9
		Ro. Magnetics (calc.)	18.5	52.0	54.7	94.7	85.9

Bench Scale Testwork - Cont'd

4. Treatment of "Grab Bulk Sample" - Cont'd

Concentrate grade and recovery was lower from the Jeffrey separator and this may have been because of the very fine grinds. The combined magnetics from the Jeffrey separator tests were repassed twice to upgrade them.

Table 9 - Recleaner Separation

Product	Weight %	Assay, %		% Distribution	
		Mag. Fe	Sol. Fe	Mag. Fe	Sol. Fe
3rd Cleaner Magnetics	13.3	66.1	69.0	86.2	77.6
3rd Cleaner Non-Magnetics	0.2	51.8	54.7	1.0	0.9
2nd Cleaner Non-Magnetics	1.0	36.4	38.4	3.5	3.3
Cleaner Magnetics	14.5	63.9	66.7	90.7	81.8

Two more passes upgraded the concentrate to the same grade as the Davis tube separation.

Filtration tests were conducted on the final concentrate.

Table 10 - Filtration Tests

Feed % Solids	Filter Time, Seconds			Filter Cake Data			Filtrate Rate **
	Form	Dry	Cycle	Thickness,mm	% Moist.	Rate*	
60	30	90	180	15	8.4	717	390
60	45	90	202.5	15	9.9	614	368

* Dry kg per hour per square metre

** Litres per hour per square metre

Bench Scale Testwork - Cont'd

5. Flotation

Two flotation tests were conducted on a sample of M1-M5 rougher non-magnetics to investigate the recovery of a sulphide concentrate. The tests were conducted with and without regrinding.

Table 11 - Flotation Results

Test No.	Grind %-200 Mesh	Product	Weight %	Assays,%, g/t			% Distribution		
				Au	Cu	S	Au	Cu	S
1	57	Rougher Conc.	11.95	0.63	0.57	0.40	50.6	36.8	38.6
		Rougher Tailing	88.05	0.08	0.13	0.086	49.4	63.2	61.4
		Head (calc.)	100.00	0.15	0.19	0.12	100.0	100.0	100.0
2	90	Rougher Conc.	19.00	0.70	0.47	0.39	65.5	48.2	55.5
		Rougher Tailing	81.00	0.09	0.12	0.073	34.5	57.8	44.5
		Head (calc.)	100.00	0.20	0.19	0.13	100.0	100.0	100.0

There was some concentration of Au and Cu but weight recoveries in the flotation stage were high.

Bench Scale Testwork - Cont'd

6. Mineralogy

Samples of magnetic concentrate and rougher magnetic tailings were submitted for mineralogical examination.

Both samples contained very fine-grained particles which made examination difficult.

The magnetic concentrate contained magnetite in excess of 95 percent, ilmenite, hematite, chalcopyrite, bornite, chalcocite/covellite, pyrite and a non-opaque mineral identified as diopside $CaMg(Si_2O_6)$. The hematite and ilmenite occurred with magnetite as attachments and forming mixed grains with almost equal amounts of magnetite and other mineral. The copper sulphides were present as inclusions in magnetite and in isolated instances attached to magnetite. Pyrite was present as free, fine-grained particles. Diopside occurred with magnetite as attached particles believed to be remnants of a large diopside particle in which the magnetite had been an inclusion. The hematite plus ilmenite together represented at best, approximately 1 percent of the sample. The sulphide minerals accounted for less than 0.1 percent of the total weight.

The Jeffrey Tailings consisted of silicates, sulphides and iron oxide. The major silicate minerals were in decreasing order of abundance: diopside, $CaMg(Si_2O_6)$, plagioclase feldspar $Ab_{90}An_{10}$-$Ab_{70}An_{30}$ chlorite, $(Mg,Fe,Al)_3(Si_4O_{10})(OH)_2$ quartz, SiO_2, phlogopite $KMg_3(AlSi_3O_{10})(OH)_2$, orthopyroxene $(Mg,Fe)_2Si_2O_6$, secondary silicates, garnet, serpentine, carbonate and amphibole.

Bench Scale Testwork - Cont'd

6. Mineralogy - Cont'd

The estimated composition by mineral species was:

Mineral	Weight, %
Diopside	50 - 60
Plagioclase	< 5
Quartz	15 - 20
Phlogopite	< 5
Orthopyroxene	~ 10
Opaques	2 - 4
Other	1 - 3
Total	100

The sulphide minerals were in decreasing order of abundance:

bornite	Cu_5FeS_4	Chalcopyrite	$CuFeS_2$
covellite/digenite	CuS	Chalcocite	Cu_2S
vallerite	$Cu_3Fe_4S_7$	Molybdenite	MoS_2
pyrite	FeS_2	Galena	PbS

The oxide minerals were in decreasing order of abundance:

magnetite	$FeO.Fe_2O_3$	Ilmenite	$FeTiO_3$
hematite	Fe_2O_3		

Neither dolomite nor wollastonite were not identified in the x-ray powder diffractograms.

The oxide minerals were present generally as fine-grained (15 to < 2 micrometres) inclusions in silicate and were most commonly seen in diopside, chlorite and phlogopite. The copper sulphide minerals were present as mixed grains of chalcopyrite and bornite; bornite and chalcocite; chalcocite and digenite or chalcocite and covellite. In each of these pairs the second-named mineral occurred as a product of alteration of the first named mineral. The copper sulphides were present as occasional discrete grains, attached to silicate grains, and most commonly

Bench Scale Testwork - Cont'd

6. Mineralogy - Cont'd

as inclusions in silicate. Trace amounts of vallerite, pyrite and molybdenite were identified also as inclusions in silicate grains although rarely, if at all, in diopside.

The sample assayed:

47.1 %	SiO_2
7.20 %	Al_2O_3
14.8 %	CaO
16.3 %	MgO
0.97 %	Na_2O
0.91 %	K_2O
2.56 %	CO_2

Bench Scale Testwork - Cont'd

7. Additional Test Details

7.1. Davis Tube Tests

Composite No.	Grind Min.	Head Assay,% Calc.		Concentrate			Tailing Assay,% Sol. Fe
		Sol. Fe	Mag. Fe	Weight %	Assay, % Sol.Fe	% Rec'y Sol.Fe	
M1	10	49.5	48.7	69.5	70.1	98.4	2.63
Mag.Conc.			46.6*		66.8*		1.06*
M1	20	48.2	47.3	69.0	68.6	98.1	2.89
Mag.Conc.			46.6*		66.3*		1.09*
M2	7.5	51.5	51.0	74.3	68.6	99.0	2.07
Mag.Conc.			50.6*		66.6*		1.25*
M2	15	51.7	50.9	74.1	68.7	98.5	3.07
Mag.Conc.			50.6*		67.0*		1.30*
M3	7.5	51.5	50.7	74.5	68.1	98.5	3.02
Mag.Conc.			48.4*		66.8*		1.15*
M3	15		48.0	74.0	69.0	98.6	2.85
Mag,Conc.			48.4*		66.4*		1.09*
M4	7.5	53.2	52.5	78.0	67.3	98.7	3.09
Mag.Conc.			48.6*		65.8*		1.10*
M4	15	53.6	52.9	77.3	68.4	98.6	3.27
Mag.Conc.			48.6*		66.2*		1.13*
M5	7.5	49.8	48.9	71.9	68.0	98.3	3.08
Mag.Conc.			45.4*		66.5*		1.39*
M5	15	49.2	48.3	71.2	67.9	98.3	3.08
Mag.Conc.			45.4*		65.8*		1.35*

* Satmagan

Bench Scale Testwork - Cont'd

7.1. Davis Tube Tests - Cont'd

Screen Analyses

Magnetic Concentrate M1

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	3.7	3.7	96.3
100	4.8	8.5	91.5
150	8.9	17.4	82.6
200	15.2	32.6	67.4
270	20.4	63.0	47.0
400	15.2	68.2	31.8
- 400	31.8	100.0	-
Total	100.0	-	-

Magnetic Concentrate M2

+ 65	1.4	1.4	98.6
100	3.5	4.9	95.1
150	7.3	12.2	87.8
200	16.7	28.9	71.1
270	18.8	47.7	52.3
400	15.0	62.7	37.3
- 400	37.3	100.0	-
Total	100.0	-	-

Magnetic Concentrate M3

+ 65	1.6	1.6	98.4
100	2.9	4.5	95.5
150	6.1	10.6	89.4
200	11.9	22.5	77.5
270	17.3	39.8	60.2
400	14.4	54.2	45.8
- 400	45.8	100.0	-
Total	100.0	-	-

Bench Scale Testwork - Cont'd

7.1. Davis Tube Tests - Cont'd

Magnetic Concentrate M4

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	4.1	4.1	95.9
100	6.1	10.2	89.8
150	8.5	18.7	81.3
200	15.9	34.6	65.4
270	20.7	55.3	44.7
400	13.8	69.1	30.9
- 400	30.9	100.0	-
Total	100.0	-	-

Magnetic Concentrate M5

+ 65	0.3	0.3	99.7
100	1.0	1.3	98.7
150	3.1	4.4	95.6
200	5.6	10.0	90.0
270	10.1	20.1	79.9
400	17.1	37.2	62.8
- 400	62.8	100.0	-
Total	100.0	-	-

Comp. M1 - Jeffrey Mags., 10' P.M. Grind

+ 150	0.2	0.2	99.8
200	1.8	2.0	98.0
270	9.8	11.8	88.2
400	14.8	26.6	73.4
- 400	73.4	100.0	-
Total	100.0	-	-

Comp. M1 - Jeffrey Mags., 20' P.M. Grind

+ 200	0.4	0.4	99.6
270	2.4	2.8	97.2
400	7.6	10.4	89.6
- 400	89.6	100.0	-
Total	100.0	-	-

Bench Scale Testwork - Cont'd

7.1. Davis Tube Tests - Cont'd

Comp. M2 - Jeffrey Mags, 7.5' Grind

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 100	0.2	0.2	99.8
150	0.2	0.4	99.6
200	2.2	2.6	97.4
270	11.2	13.8	86.2
400	15.8	29.6	70.4
- 400	70.4	100.0	-
Total	100.0	-	-

Comp. M2 - Jeffrey Mags., 15' Grind

+ 150	0.2	0.2	99.8
200	0.4	0.6	99.4
270	4.0	4.6	95.4
400	9.8	14.4	85.6
- 400	85.6	100.0	-
Total	100.0	-	-

Comp. M3 - Jeffrey Conc., 7.5' Grind

+ 100	0.2	0.2	99.8
150	0.2	0.4	99.6
200	1.8	2.2	97.8
270	8.2	10.4	89.6
400	13.2	23.6	76.4
- 400	76.4	100.0	-
Total	100.0	-	-

Comp. M3 - Jeffrey Conc., 15' Grind

+ 200	0.4	0.4	99.6
270	2.4	2.8	97.2
400	8.0	10.8	89.2
- 400	89.2	100.0	-
Total	100.0	-	-

Bench Scale Testwork - Cont'd

7.1. Davis Tube Tests - Cont'd

Comp. M4 - Jeffrey Conc., 7.5' Grind

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 150	0.4	0.4	99.6
200	3.4	3.8	96.2
270	12.8	16.6	83.4
400	17.0	33.6	66.4
- 400	66.4	100.0	-
Total	100.0	-	-

Comp. M4 - Jeffrey Conc., 15' Grind

+ 200	0.6	0.6	99.4
270	4.6	5.2	94.8
400	9.6	14.8	85.2
- 400	85.2	100.0	-
Total	100.0	-	-

Comp. M5 - Jeffrey Conc., 7.5' Grind

+ 150	0.2	0.2	99.8
200	0.8	1.0	99.0
270	3.6	4.6	95.4
400	12.2	16.8	83.2
- 400	83.2	100.0	-
Total	100.0	-	-

Comp. M5 - Jeffrey Conc., 10' Grind

+ 200	0.2	0.2	99.8
270	1.4	1.6	98.4
400	6.8	8.4	91.6
- 400	91.6	100.0	-
Total	100.0	-	-

Bench Scale Testwork - Cont'd

7.1. Davis Tube Tests - Cont'd

Grab Bulk Mag. Conc., 15' P.M. Grind

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 150	0.2	0.2	99.8
200	1.0	1.2	98.8
270	8.2	9.4	90.6
400	16.2	25.6	74.4
- 400	74.4	100.0	-
Total	100.0	-	-

Size Analyses

Bulk Grab Mag. Conc., 25' Grind

Particle Size	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 200 mesh	0.4	0.4	99.6
270	1.0	1.4	98.6
26.6 μm	19.2	20.6	79.4
20.6	32.2	52.8	47.2
14.4	16.3	69.1	30.9
9.9	10.4	79.5	20.5
7.7	5.6	85.1	14.9
- 7.7 μm	14.9	100.0	-
Total	100.0	-	-

Specific Gravity = 4.79

Bulk Grab Mag. Conc., 50' Grind

+ 26.6 μm	3.6	3.6	96.4
20.6	9.4	13.0	87.0
14.4	20.8	33.8	66.2
9.9	18.6	52.4	47.6
7.7	11.9	64.3	35.7
- 7.7 μm	35.7	100.0	-
Total	100.0	-	-

Specific Gravity = 4.77

Bench Scale Testwork - Cont'd

7.2. Comp. M1 - M5

Purpose: To upgrade the magnetic concentrate.

Procedure: The reground magnetics of tests M1 to M5 were combined and repassed through the Jeffrey at 2.5 amperes to further upgrade the product.
The magnetics produced were elutriated in a 2 litre separatory funnel to remove slimes.

Feed: The combined reground magnetic concentrates of samplle M1 to M5.

Observations: The concentrate contained large gangue particles with small inclusions of magnetite as the main contaminant.

Metallurgical Results

Products	Weight			Assays, %		% Distribution			
	g	% Ind.	% O'all	Sol. Fe	Mag. Fe	Sol. Fe Ind.	Sol. Fe O'all	Mag. Fe Ind.	Mag. Fe O'all
1. Mag. Cl. Conc.	250.0	70.40	19.7	68.1	65.8	92.2	81.6	93.1	88.4
2. Slimes	3.0	0.85	0.2	46.3	43.6	0.7	0.6	0.6	0.6
3. Non-Magnetics	102.1	28.75	8.1	12.8	10.8	7.1	6.3	6.3	5.9
Head (calculated)	355.1	100.00	28.0	52.0	49.7	100.0	88.5	100.0	94.9

Calculated Grades and Recoveries

Products 1 plus 2	-	-	19.9	67.8	65.6	92.9	82.2	93.7	89.0

Screen Analysis

M1-M5 Cleaner Magnetics

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 150	0.2	0.2	99.8
200	1.5	1.7	98.3
270	8.1	9.8	90.2
400	14.2	24.0	76.0
- 400	76.0	100.0	-
Total	100.0	-	-

Bench Scale Testwork - Cont'd

7.3. Filtration

Purpose: To conduct filtration tests on the magnetic concentrate.

Procedure: Pour-on filtration tests were conducted using a 8 cm filter media surface. Form time was time to surface dryness. 100 g/t Aerodic 100 was added to the second test.

Feed: 185.3 g 2nd pass magnetic concentrate.

Cloth Specifications:

Media Number:	POPR 858
Material	Polypropylene
Air-Flow (m^3/m^2)	4.27
Yarn:	Spun Staple
Weave	70 x 32
Weight (g/m^2)	434
Thread Count:	70 x 32
Finish	Griege

Test Conditions

Test No.	Filter Cloth	Slurry		Filter Vac. (cm Hg)		Filter Time - Sec.		
		% Solids	Temp. °C	Form	Dry	Form	Dry	Cycle
F1	POPR858	60	R.T.	63.5	63.5	30	90	180
F2	"	60	R.T.	63.5	63.5	45	90	202.5

* 100 g/t Aerodic 100 added

Test Results

Test No.	Filter Cake Data						Filtrate Data		
	Cracks No.	Thick. mm	Weight - Grams Wet	Dry	% Moist.	Rate *	Volume mL	Clarity	Rate **
F1	2	15	196.3	179.9	8.4	717	98	Good	390
F2	0	15	192.6	173.5	9.9	614	104	Good	368

* Dry kg per hour per square metre
** Litres per hour per square metre

Bench Scale Testwork - Cont'd

7.4. Flotation

Float 1

Purpose: To float a sulphide concentrate from the non-magnetic rougher product.

Procedure: As below.

Feed: 1 kg combined non-magnetics from Tests M1 to M5.

Grind: nil

Conditions:

Stage	Reagents Added, grams/tonne			Time, minutes		pH
	$CuSO_4$	A350	MIBC	Cond.	Froth	
Rougher Float	-	20	10	1	4	8.0
	-	20	10	1	4	-
Scav. Cond.	250	-	-	3	-	7.8
	-	50	10	1	5	-

Metallurgical Results

Product	Weight %	Assays,%,g/t			% Distribution		
		Au	Cu	S	Au	Cu	S
1. Rougher Concentrate	11.95	0.63	0.57	0.40	50.6	36.8	38.6
2. Scav. Concentrate	4.40	0.15	0.38	0.21	4.4	9.0	7.4
3. Scav. Tailing	83.65	0.08	0.12	0.08	45.0	54.2	54.0
Head (calculated)	100.00	0.15	0.19	0.12	100.0	100.0	100.0

Calculated Grades and Recoveries

Products 2 plus 3	88.05	0.083	0.13	0.086	49.4	63.2	61.4

Bench Scale Testwork - Cont'd

Float 1 - Cont'd

Screen Analysis

Combined Products

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	7.3	7.3	92.7
100	9.1	16.4	83.6
150	13.9	30.3	69.7
200	12.8	43.1	56.9
270	12.5	55.6	44.4
400	8.8	64.4	36.6
- 400	36.6	100.0	-
Total	100.0	-	-

Bench Scale Testwork - Cont'd

Float 2

Purpose: To float a sulphide concentrate from the reground non-magnetic rougher product.

Procedure: As per Float 1.

Feed: 1 kg combined non-magnetics tests M1 to M5.

Grind: 15 minutes at 50% solids in the lab ball mill.

Conditions:

Stage	Reagents Added, grams/tonne			Time, minutes		pH
	$CuSO_4$	A350	MIBC	Cond.	Froth	
Rougher Float	-	20	10	1	4	-
	-	20	10	1	4	-
Scav. Cond.	250	-	-	3	-	
	-	50	10	1	5	-

Metallurgical Results

Products	Weight %	Assays,%,g/t			% Distribution		
		Au	Cu	S	Au	Cu	S
1. Rougher Concentrate	19.00	0.70	0.47	0.39	65.5	48.2	55.5
2. Scavenger Conc.	6.80	0.16	0.32	0.22	5.4	11.8	11.2
3. Scavenger Tailing	74.20	0.08	0.10	0.06	29.1	40.0	33.3
Head (calculated)	100.00	0.20	0.19	0.13	100.0	100.0	100.0

Calculated Grades and Recoveries

Products 2 plus 3	81.00	0.087	0.12	0.073	34.5	51.8	44.5

Bench Scale Testwork - Cont'd

Float 2 - Cont'd

Screen Analysis

Combined Products

Mesh Size (Tyler)	% Retained Individual	% Retained Cumulative	% Passing Cumulative
+ 65	0.2	0.2	99.8
100	0.6	0.8	99.2
150	2.7	3.5	96.5
200	6.7	10.2	89.8
270	13.0	23.2	76.8
400	14.2	37.4	62.6
- 400	62.6	100.0	-
Total	100.0	-	-

LAKEFIELD RESEARCH
Lakefield, Ontario
November 21, 1984 / tmg

NATIVE GOLD RECOVERY AT WHITEHORSE COPPER MINES

Whitehorse Copper Mines operates a 2,600 ton per day mine and mill complex on the outskirts of Whitehorse, Yukon Territory, approximately twelve kilometers from the city center. Production at this property began in May of 1967 under the name, New Imperial Mines, from a number of open pit deposits along the Whitehorse Copper Belt.

Lower grade ore and falling copper prices resulted in the stoppage of mining and milling operations in June of 1971. During an eighteen month shutdown period the company name was changed to Whitehorse Copper Mines Limited, and the Little Chief underground ore body was developed. In December of 1972 the mill was restarted and the underground mine began supplying ore. At present, the Little Chief deposit is still active, and current ore reserves will support approximately three more years of operation.

I am sure the initial thought to many people here is "What is a copper mine doing presenting a paper at a Placer Forum?". The following statistics should answer this question.

Total metal production from the underground orebody through the period 1973 to 1979 was 151,782,542 lbs. of copper, 1,533,104 oz. of silver and 128,168 oz. of gold. The gold statistic is of particular interest, because it is equal to the gold content of reported placer production from all placer mining operations in the Yukon through the same period.

During the pre-production feasibility study on the underground ore body, little emphasis was placed on the gold value. Drill core assays indicated an average grade of 0.025 ounces of gold per ton or ore; and gold recovered with the concentrate was considered a minor by-product. In recent years, gold is analyzed in parts per billion when establishing ore cut-off grades. At current metal prices, precious metals represent 45% of Whitehorse Copper's total revenue. We are quickly approaching the status of Whitehorse Gold Mines.

Every spring when truck loads of equipment pass through Whitehorse on their annual migration to the Dawson area gold creeks, we smile a bit. We like to think we are mining the mother lode: 1200 feet below the surface and within the city limits of Whitehorse.

The milling circuit employs conventional crushing, grinding and froth flotation to recover a copper sulfide concentrate, assaying 45% copper, 8 ½ oz./ton silver and 3/4 of an ounces per ton gold. The gold is all in * a very fine native form. It responds well to Xanthate collector and recovery in flotation is in the 85 - 90% range. For the first three years of underground production all of the gold value reported to the copper concentrate. When mining activity reached a certain level in the ore body, coarse native gold began to appear in the mill circuit. It occurs as flat nuggets , pounded in the mills; with some pieces measuring up to ½ inch in diameter. The fineness is 890 to 900. It concentrates in the grinding circuit wherever the slurry flow is interrupted.

It was at this time that we realized gold recovery techniques were almost a lost technology. Gold mining had been relatively dormant since the Second World War and there was a thirty year gap where the old timers had passed on, and metallurgists had directed their attention to large tonnage copper-molydenum or lead-zind ore dressing. Gold and silver were being recovered mainly as by-products. Techniques such as jigs, shaking tables, amalgam drums and cyanidation were not in common use.

Our initial move was to call on the local placer miners to suggest recovery methods. Sluice box style riffles were installed in the sloping rodmill and ballmill discharge launders.

The riffles collect a very complex heavy media product that includes: magnetite, tramp steel, copper from the blasting wire, tungsten carbide from drill bits, brass: in fact just about anything with a high specific gravity that has been introduced during mining and the crushing state. Although this product is recovered readily by basic sluicing, a 'clean-up' method to extract the flat gold particles was a very difficult project.

Some recovery techniques experimented with are as follows:

1. Additional concentration by sluicing was ineffective. The product is all heavy media to start with and it quickly blinds the riffles allowing flat gold to plane in the water over top.

2. Cyanidation did not work well because a good portion of the gold was too coarse. Also, the high copper concentration yielded an extremely high cyanide consumption.

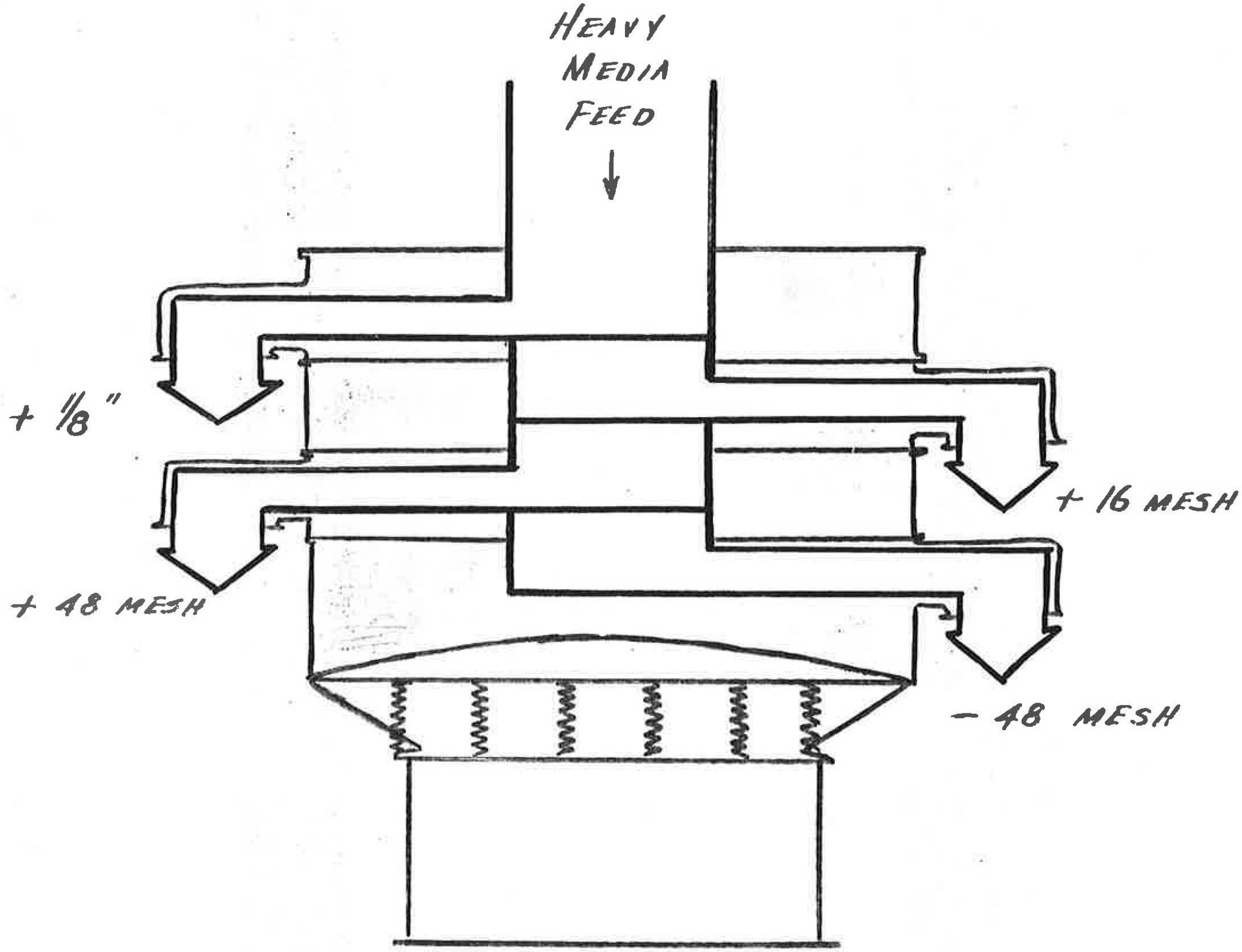
3. Mercury amalgamation or quick silver techniques were unsuccessful. Magnetite and gangue meterial is pounded in the malleable gold surface during grinding and this inhibits the infinity for mercury. The hazardous nature of mercury was also a consideration in rejecting its use.

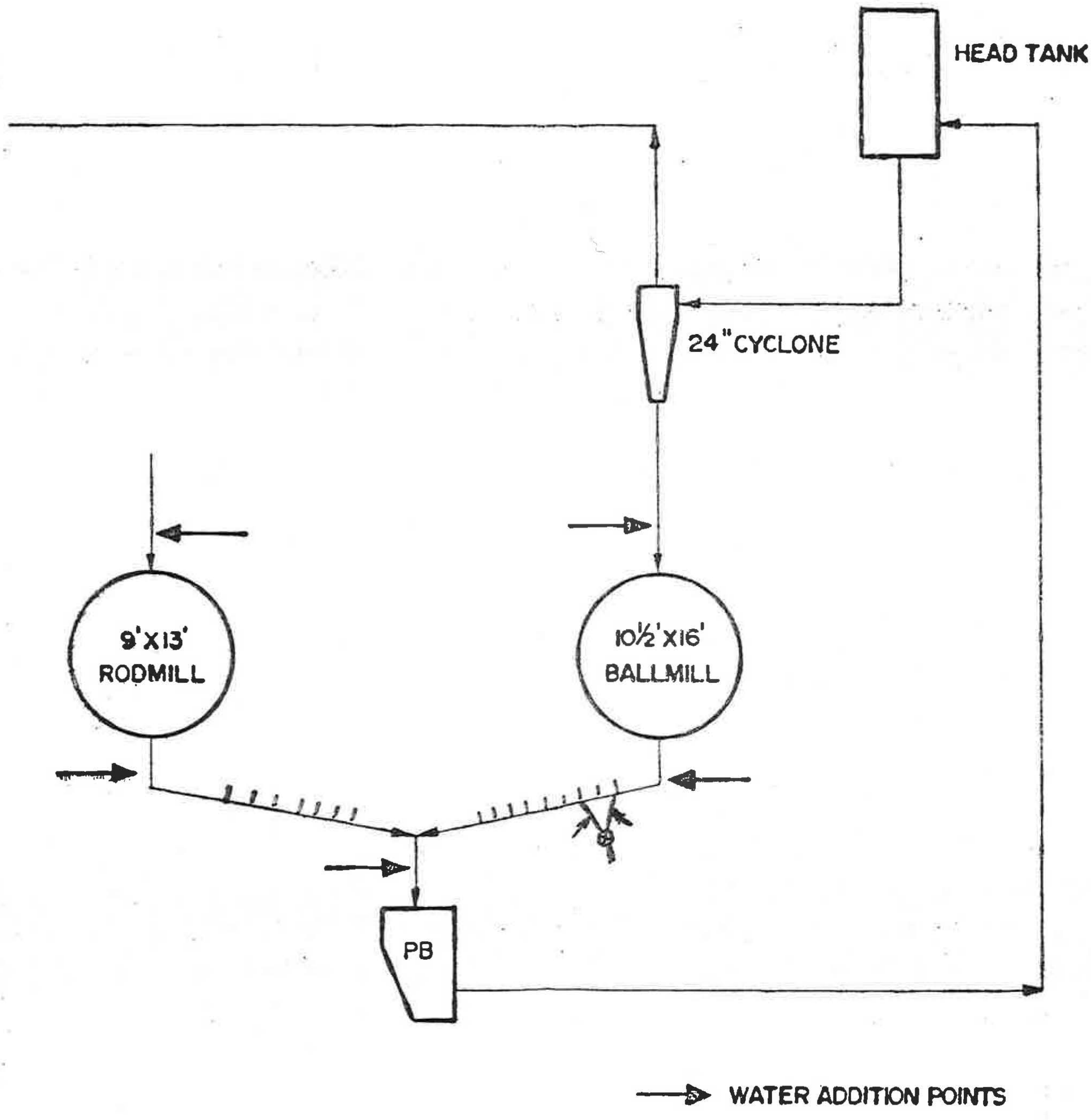
Testwork ruled out the aforementioned techniques, but we have utilized a number of other methods to clean up our heavy media material. A readily saleable gold product is being extracted.

The most important parameter to efficient separation of gold from the other heavy materials is particle sizing. Similar sized and shaped materials will respond better to flowing film or gravity classification.

We have set up our recovery procedure based on four size fractions:

+ 1/8 "
+ 16 mesh
+ 48 mesh
\- 48 mesh


HEAVY MEDIA FEED
+ 1/8"
+ 16 MESH
+ 48 MESH
- 48 MESH



WHITEHORSE COPPER MINES LTD.

GRINDING CIRCUIT

METAL PRODUCTION - WHITEHORSE COPPER MINES

YEAR	COPPER (lbs)	SILVER (oz troy)	GOLD (oz troy)	YUKON PLACER PRODUCTION (oz troy)
1973	21,563,064	250,627	13,888	8,509
1974	21,035,329	209,512	17,731	12,136
1975	20,062,121	217,397	18,630	19,378
1976	24,364,262	241,159	18,550	21,312
1977	26,340,682	249,672	24,058	25,692
1978	20,923,048	195,765	19,443	25,780
1979	17,494,036	168,972	15,868	35,074
TOTAL	151,782,542	1,533,104	128,168	147,881
1980 (estimated)			20,600	

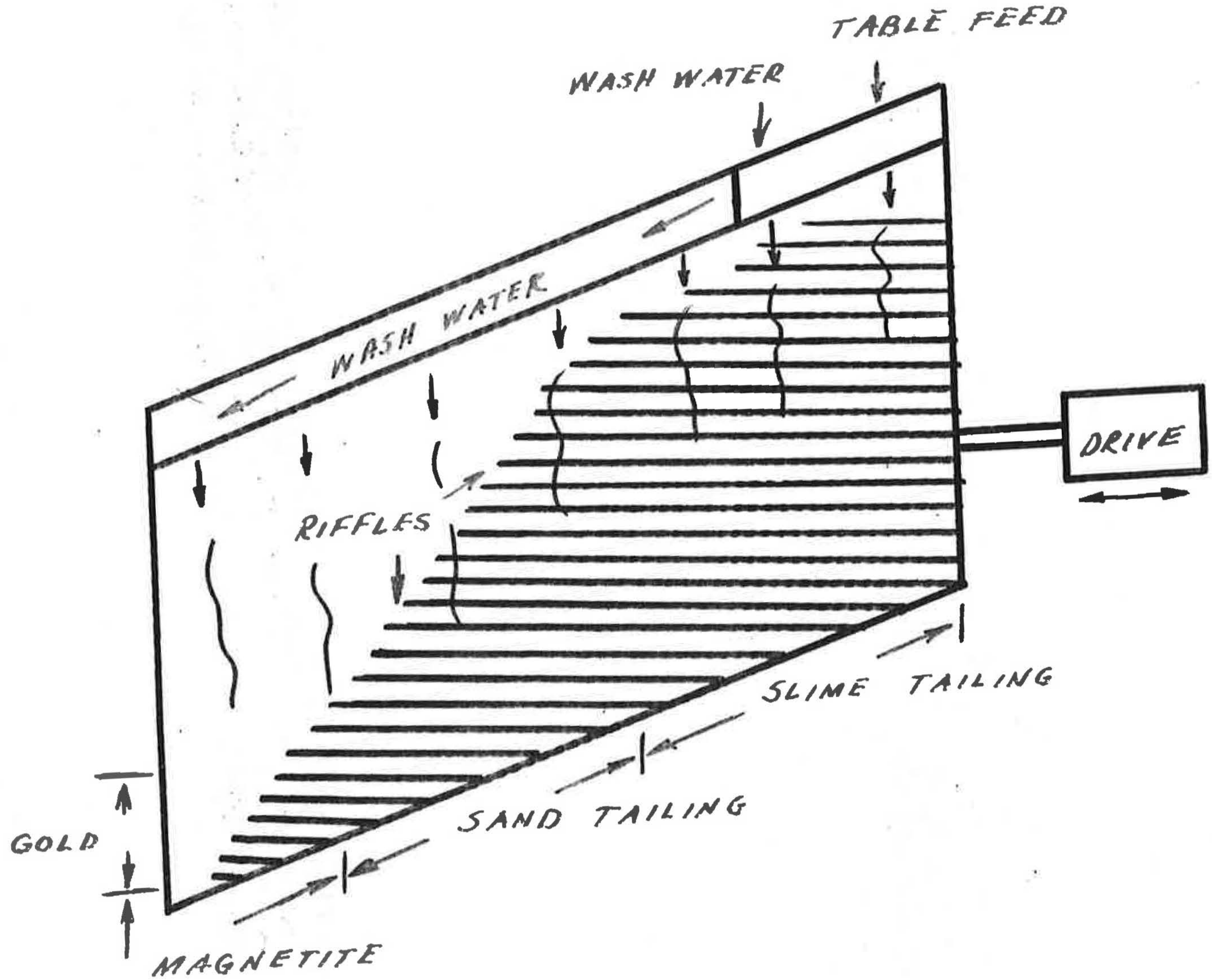


TABLE FEED
WASH WATER
WASH WATER
RIFFLES
DRIVE
SLIME TAILING
SAND TAILING
GOLD
MAGNETITE

SCREEN SIZES

+ 1/8 INCH

+ 16 MESH

+ 48 MESH

− 48 MESH

+ 1/8 INCH

↓

MAGNETIC SEPARATION

↓

NITRIC ACID LEACH

↓

HAND PICK

+ 16 ¢ + 48 MESH

↓

MAGNETIC SEPARATION

↓

TABLE

↓

RE-TABLE MIDDLINGS

- 48 MESH

↓

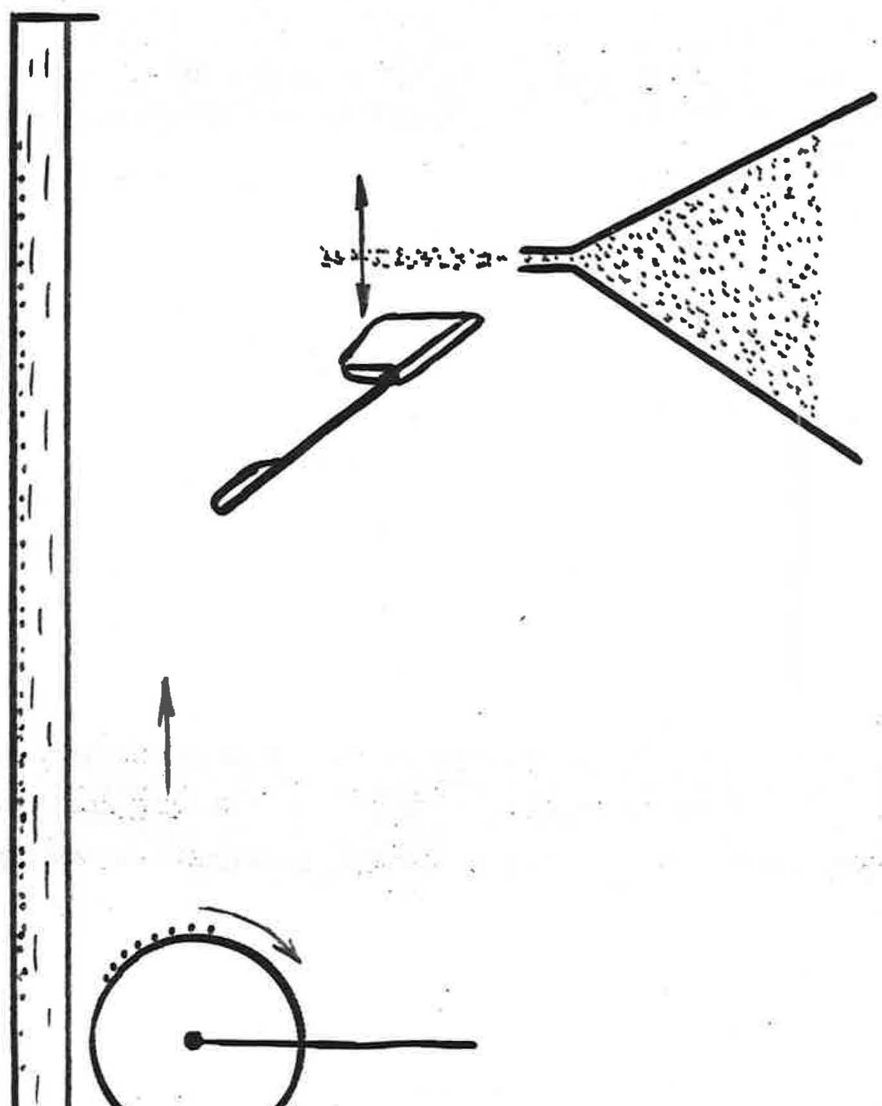
TABLE

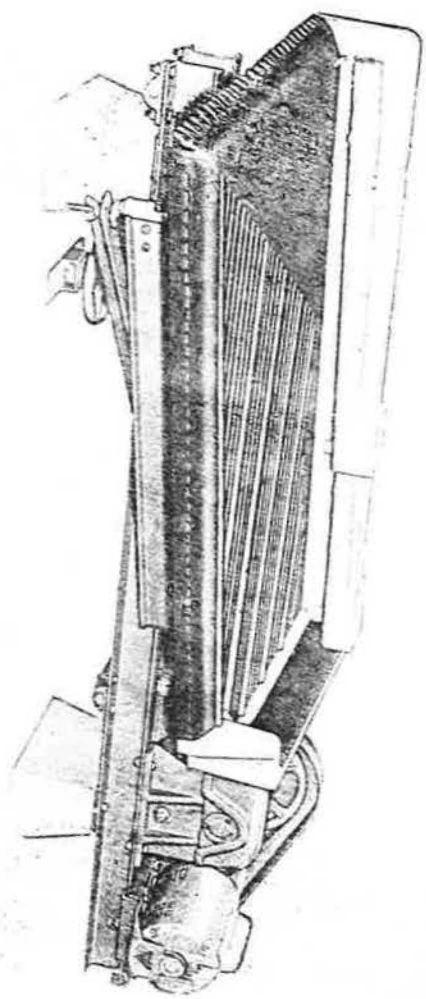
↓

DRY

↓

SAMPLE





Gold in Skarns of the Whitehorse Copper Belt, Yukon Territory, Canada

by: Lawrence D. Meinert
Associate Professor
Washington State University
Pullman, WA 99164-2812

Introduction

Skarns in the Whitehorse Copper Belt have produced 134,000 tons of copper, 97,000 kg of silver, and 7,700 kg of gold from 11 million tons of ore between 1967 and 1982 (Tenney, 1981; Watson, 1984). By the end of mining activities in 1982, gold and silver contributed a significant proportion of the ore value, thus sparking a renewed interest in the occurrence of gold in the Whitehorse skarns. A calculated gold grade (assuming 100% recovery) of 0.7 g Au/ton for the Whitehorse deposits is probably too low but neither exploration nor production drill holes were routinely assayed for gold. Hurreau (1985) estimated that the 10 million tons of tailings from the Whitehorse deposits contain about 1,500 kg of gold and Tenney (1981) reported that tailings assays ranged up to 250 g Au/ton near the tailings decant. It appears likely that not all the gold was recovered from Whitehorse skarn ore, that the calculated average gold grade for the deposit is too low, and that further gold-silver potential in the district may exist.

Worldwide, about 1.5×10^9 grams of gold have been produced from skarn deposits. The majority of this production has come as a by-product of large scale mining of skarns associated with porphyry copper deposits (Meinert, 1986). Most skarns in porphyry copper districts contain less than 0.5 g Au/ton although some deposits (e.g. Bingham, Utah, Cameron and Garmoe, 1983) contain local zones with

WHITEHORSE COPPER BELT PRODUCTION 1967 - 1982

YEAR	TONS MILLED	HEADS % Cu	RECOVERY % Cu	PRODUCTION Cu (lbs)	PRODUCTION Au (oz.)	PRODUCTION Ag (oz.)	DEPOSIT
1967	453,046	1.16	69.76	7,322,180	8,752	89,786	Little Chief
1968	732,095	1.03	80.38	12,153,016	16,595	148,188	L. Chief, Arctic Chief
1969	805,519	1.09	86.84	15,192,201	14,560	196,469	Arctic Chief, War Eagle
1970	852,461	1.03	91.97	16,113,635	5,579	211,614	War Eagle
1971	337,758	1.02	75.90	5,236,738	3,646	53,273	B.Cub South, Keewenaw
	3,180,879	1.06	82.95	56,017,770	49,132	699,330	TOTAL OPEN PITS
1973	700,053	1.83	84.33	21,563,064	13,888	250,627	Little Chief
1974	626,541	1.86	90.22	21,035,329	17,731	209,512	Little Chief
1975	738,062	1.52	89.36	20,062,121	18,630	217,397	Little Chief
1976	800,836	1.69	89.74	24,364,262	18,550	241,159	Little Chief
1977	901,459	1.65	88.35	26,340,602	24,058	249,672	Little Chief
1978	863,092	1.40	86.34	20,923,048	19,443	195,765	Little Chief
1979	914,060	1.12	85.66	17,494,036	15,868	168,972	Little Chief
1980	854,306	1.58	87.61	23,651,040	22,102	240,273	Little Chief
1981	800,378	1.42	87.97	19,994,225	15,727	194,363	Little Chief
1982	897,624	1.39	84.16	20,989,608			Little Chief
	8,096,412	1.53	87.40	216,417,410	(165,997)	(1,967,740)	TOTAL UNDERGROUND
	11,277,291	1.40	86.45	272,435,180	(215,129)	(2,667,970)	TOTAL PRODUCTION

Recovery Grades = 1.21% Cu, 0.021 oz./ton Au, 0.257 oz/ton Ag

(604) 669-8959

SILVERTIP MINING CORPORATION

December 16, 1998

Mr. Brian Scott
PO Box 66
Tagish, YT
Y0B 1T0
Tel: (867) 393-1308
Fax: (867) 399-3660

Dear Brian:

Thank you for your letter dated August 28, 1998 regarding magnetite potential from reprocessed tailings at Whitehorse Copper.

Silvertip would be very interested in pursuing the potentials to develop a useable dense media product. Table 3-3, which demonstrates the typical heavy media grade magnetite specifications of Whitehorse Copper, indicates that all of the criteria for our DMS circuit will be met or exceeded by the final product; however additional testing will be necessary before it can be determined if a useable product can be produced from your property.

I will call you the next time I am in the Yukon, please let me know then if it is convenient for us to meet.

Sincerely,

SILVERTIP MINING CORPORATION

Stephen Robertson

Stephen Robertson, P.Geo.
Project Manager

:cyf

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Craigmont Mines

Raw Ore Feed or Old Mine Tailings from the former Craigmont Mine Tailings stockpile is fed through a central loading point that places the material onto a series of conveyor belts. This material then passes from the conveyor belts through a trommel screen, which removes any large unwanted material. The material that passes through the trommel screen is then transformed into slurry prior to entering the process plant.

Once the slurry enters the plant, it is fed through a sequence of magnetic separators, which separate the magnetic material from the non-magnetic material. The magnetic material passes through several 'cleaner' magnetic circuits in a pre-determined sequence as it is upgraded to meet customer quality specifications. The non-magnetic material is then returned to the tailings pond.

In addition to the magnetic separation the ore material also passes through a grinding circuit where it is reduced in size to meet customer specifications.

Once the magnetite is of appropriate quality, it is dried to between 6% and 8% moisture prior to shipping.

[**History** | **Products** | **Magnetite** | **Hematite** | **Production** | Sales | **Contacts**]