UNDERGROUND MINING AND PYROCHLORE ORE PROCESSING

AT NIOBEC MINE, QUEBEC, CANADA

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Abstract

The property of the Niobec mine is located in the St-Honor@ carbonatite complex near Chicoutimi; located 525 km north of Montreal, Quebec.

The original discovery of the carbonatite complex was made by the "Société Québéçoise d' Exploration Miniére" (SOQUEM) in 1967 as a result of an airborne radiometric survey in search of uranium deposit. With the exploration of the complex, a rare earth zone was first exposed followed by the discovery of two niobium zones.

The ore concentration process was jointly developed by the Quebec Department of Natural Resources Laboratories and SOQUEM after an intensive bench and pilot plant testing. Even though the deposit contains several useful minerals, only two of them, pyrochlore Na Ca Nb_2O_6 F and columbite (Fe, Mn) (Nb, $Ta)_2O_6$, are recovered at the present time.

The construction of the underground mine and mill complex with an initial milling capacity of 1,360 metric tons/day (1,500 tons/day) was completed in 1976. Since the construction, the original capacity was gradually increased to 1,590 metric tons/day (1,750 tons/day) and then in early 1981 after a major expansion to 2,086 metric tons/day (2,300 tons/day).

Underground Mining at Niobec Mine, Quebec, Canada

Introduction

The property of the Niobec mine is located in the St-Honoré carbonatite complex near Chicoutimi; located 525 km (325 miles) north of Montreal, Quebec. The original discovery of the carbonatite complex was made by the "Société Québécoise d'Exploration Minière" (SOQUEM) in 1967 as a result of an airborne radiometric survey in search of uranium deposit.

With the exploration of the complex, a rare earth zone was first exposed followed by the discovery of two niobium zones. Exploration of these niobium zones was mainly carried out with 30,000 m (100,000 feet) of diamond drilling. In 1974, a joint decision was taken by SOQUEM and Teck Corporation to proceed with the exploitation of the niobium deposit. For this purpose, the company "Niobec Inc." was then formed. Niobec name is an abreviation of niobium of Quebec.

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Geology, Ore Reserve and Ore Control

The St-Honoré carbonatite complex (Figure 1) is of magmatic origin, about 650 million years old and occurs along the Saguenay rift zone which is 240 km (150 miles) long and roughly 25–40 km (15–25 miles) wide. The complex has a nearly circular shape and covers about 25 square km (10 square miles).

The rock formations in the area are typical of this part of the Grenville province and are made up of anorthosites, monzonites, syenites and gneisses. Carbonatite and satellite rocks are however overlaid by a capping of Ordovician Trenton limestone, lying with unconformity over the carbonatite, with a thickness reaching up to 75 m (250 feet). The carbonatite complex (Figure 2) itself is made up of mainly dolomitic sideritic and calcitic carbonatite.

Two major niobium zones covering a 600 m (2,000 feet) by 750 m (2,500 feet) area in the southern sector of the carbonatite core, are composed of lenticular bands of varying lithology with niobium grading 0.2 to 1.00 percent Nb_2O_5. The dip is vertical to subvertical. Zone No. 1 extends some 550 m (1,800 feet) long and is characterized by the presence of columbite and the whole range of sodic to ferrian pyrochlores. This type of mineralization presents many metallurgical complexities. Zone No. 2 presently into production extends over 800 m (2,600 feet) long by 250 m (800 feet) wide and includes ten ore lenses containing more than 0.50 percent Nb_2O_5. The mineralization is mainly a sodic pyrochlore. The ore concentration process description here contained will be limited to this type of pyrochlore.

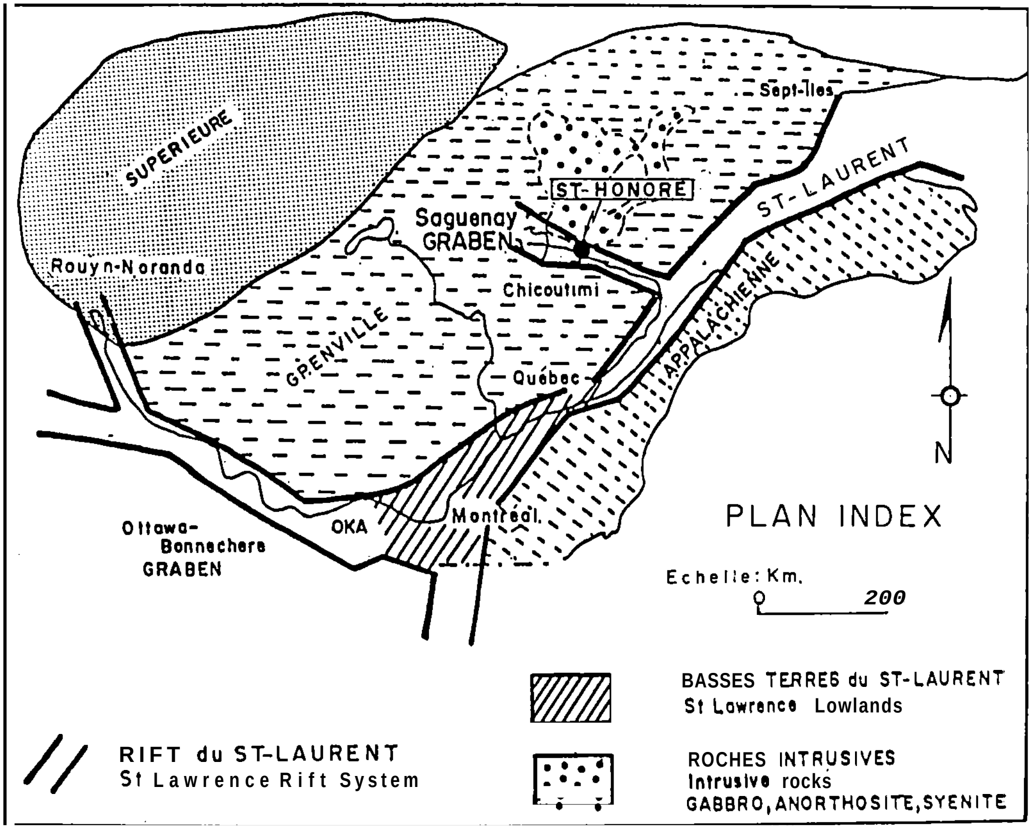

Figure 1 The St. Honoré Carbonatite Complex.

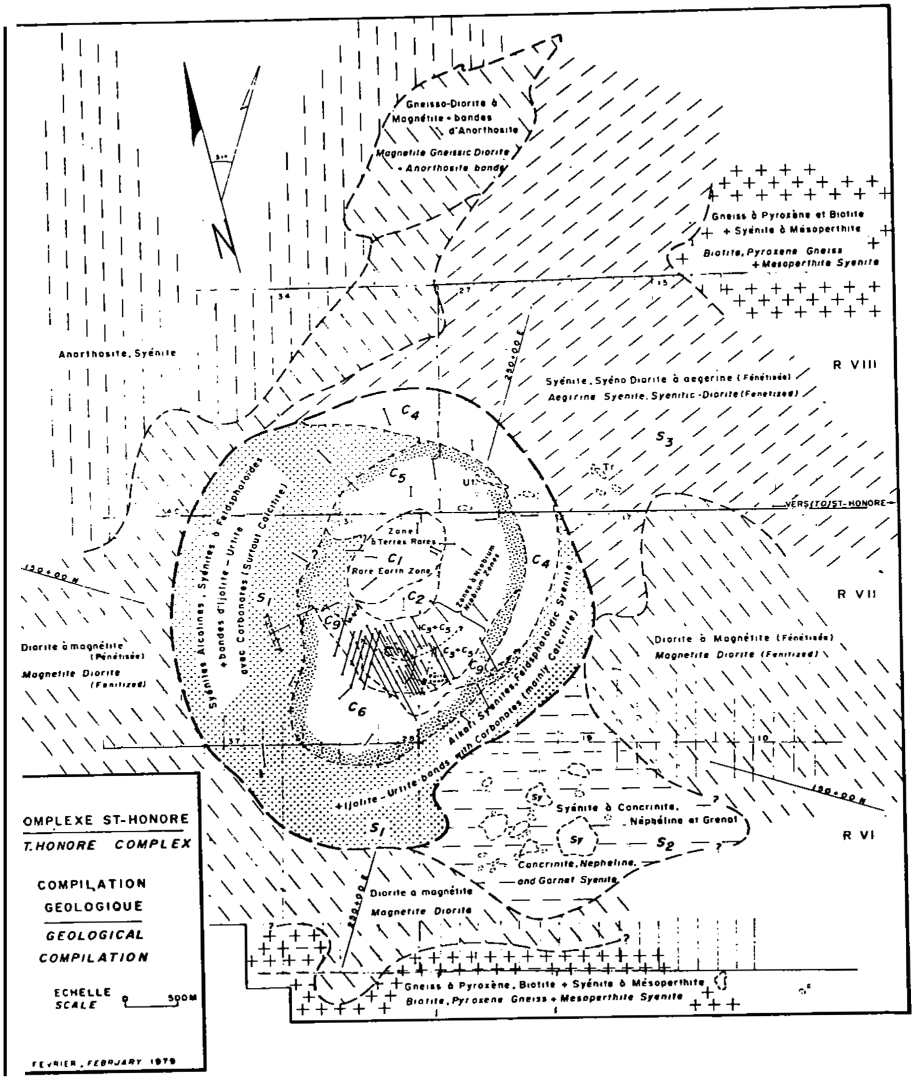

Figure 2. Geology of the St. Honoré Carbonatite Deposit.

The mining reserves are established at 11,000,000 m. tons (10,000,000 tons) grading 0.67 percent Nb_2O_5. Apatite content, estimated at 6-8 percent, offers a potential by-product as phosphate fertilizer currently under evaluation.

Underground definition drilling started during the pre-production phase in 1975. This drilling is done on a regular pattern of 15 m (50 foot) spacing vertical section, normal to ore lenses. Some holes on 7.5 m (25 foot) spacing sections are being drilled at extremities of ore bodies or to check some particular points.

Open stope layouts and quality control (grade + metallurgy testing) are based on this fundamental information since no visual day-to-day identification is possible. This singular situation necessitates a closed cooperation between geological, engineering and production departments to obtain a valuable extraction control. Mapping and muck sampling help in part to get better contours of those complex ore lenses. Muck sampling is carried out from all mine developments, stope sills and draw points as a grade check. Metallurgical bench tests using the underground drill core rejects are pursued to check and optimize the millability of different ore zones to be mined out.

A five year production planning is being used to determine the yearly, quarterly and monthly operating schedule. A pre-determined ore mixture provided from many underground open stopes is used to optimize in the concentration process the quality of the pyrochlore concentrate and the overall mill recovery.

Underground Mine Development

The knowledge of the main ore zones obtained from surface diamond drilling program coupled with the expected behavior of the rock made us adopt the open stope and pillar method with mechanized trackless mining. The rock quality and competence were obtained while driving a decline to obtain a pre-production bulk sample. Access for mining (Figure 3) is through a four compartment shaft ultimately sunk to 404 m (1,325 feet). This shaft is used for production and service with two cages over skip combination operated in balance. Mining levels from the 91 m (300 foot) horizon are established at 46 m (150 foot) intervals interconnected by a series of declines on a -20 per cent grade. That decline breaks through to surface and it is used for both No. 1 fresh air ventilation circuit and for servicing the mine with the diesel operated equipment to all operating levels.

Ore pass and waste pass systems are somewhat parallel and are feeding a loading pocket located at the 381 m (1,250 foot) horizon.

Two mining blocks 91 m (300 feet) in height were established between the 91 m and 182 m (300 and 600 foot) levels and the 213 m and 305 m (700 and 1,000 foot) levels leaving a 30 m (100 foot) ore sill between the 182 m and 213 m (600 and 700 foot) levels.

The sizes of the underground headings are:

a) 3.2 m H X 4.6 m W (10.5 ft H X 15 ft W) for general development.

b) 3.8 m H X 3.9 m W (12.5 ft H X 13 ft W) for stope preparation development.

The drifting is accomplished with the following sequence:

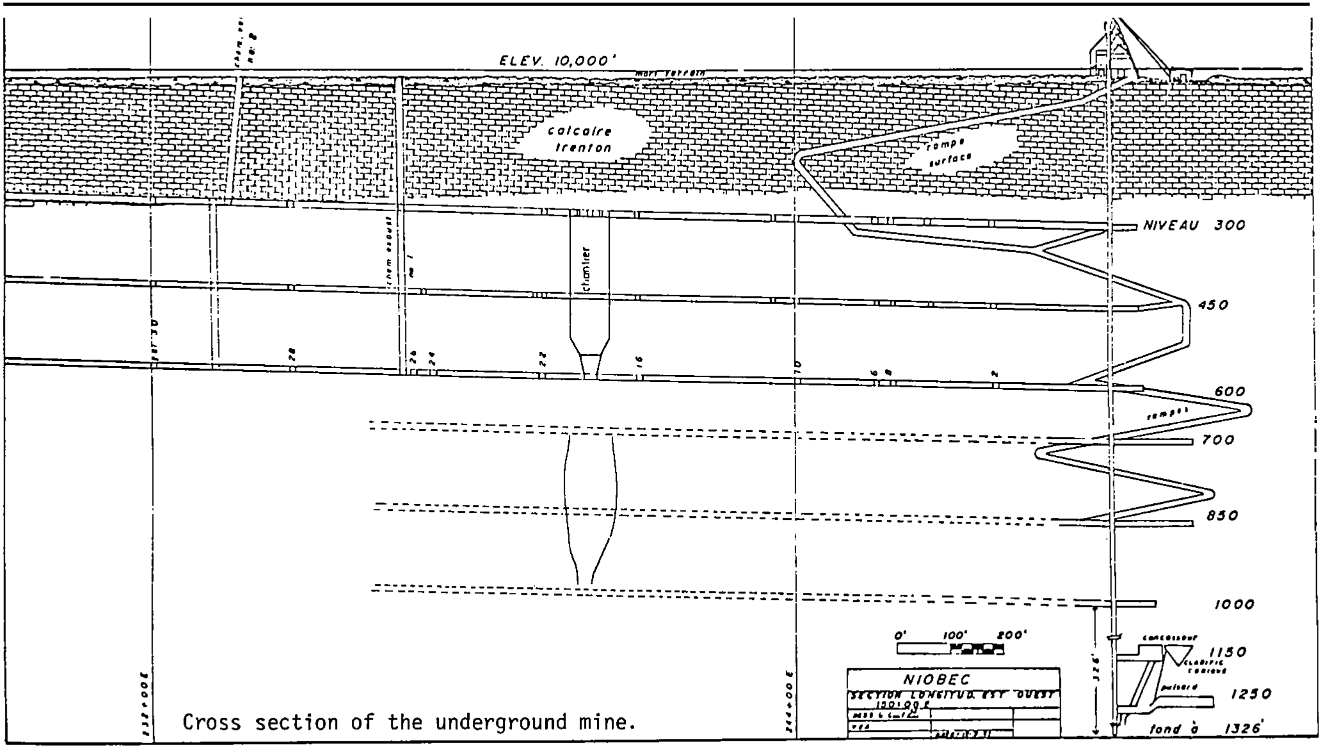

Figure 3. Cross-section of the underground mine.

- A miner utilizing a two boom MJM-21 jumbo drills off 2 - 3.65 m (12 feet) rounds per 8 hours shift.

- With the use of a scissor lift, two miners install the necessary rockbolts for ground support and load and blast two drift rounds per 8 hours shift.

- A three men crew does the scaling and mucks out the rounds with the use of a 912 L.H.D. unit and two Jarvis Clark 12 metric tons (13 tons) trucks.

This method has proven advantageous, safe and very efficient when 12 headings are concurrently driven on different development levels.

Mining Method

The open blast hole stopes and pillars method is used throughout the mine. Once the ore lenses are out lined, the mine planning group determine the number of stopes, pillars and their respective sizes. Some stopes are mined from assay walls with their long axis on strike with the formation. Others are transverse stopes and their width is limited to 24 m (80 feet). Pillars between stopes are also 24 m (80 feet) in width.

The stope preparation of a transverse stope is as follows. Access drifts are driven concurrently on all three levels in the center of the pillars left. On the upper two levels of a stope, drill drifts are driven 3.8 m H X 3.9 m W (12.5 ft H X 13 ft W) and parallel to each other, 6.7 m (22 feet) center to center leaving a 2.7 m (9 feet) rib pillar between drifts. On the bottom level, draw points are open at 60" from the haulage way. The stope sill is in the center of the stope and is driven 2.7 m H X 6.4 m W (9 feet H X 21 feet W) from which 10.6 m (35 feet) uppers inclined at 70° are drilled fanning outward.

The drilling is accomplished with an Atlas Copco ITH unit "Roc 601", fitted with an adaptor to drill 11.4 cm (4 1/2 inches) diameter holes. This type of unit drills the production holes 45 m (150 feet) long on a 3 m X 3.3 m (10 ft X 11 ft) pattern and the 3.6 m X 3.6 m (12 ft X 12 ft) drop raise. The drilling and blasting of the drop raise within the location of the slot complete the stope preparation. The stope blasting is done two or three rows of holes at a time re-treating gradually to the end of the stope until it is mined out.

The drilling performance is as follow:

Drilling performance per bit (av.)	1,525 m	(5,000 feet).
Ratio of drilling per hour	5.7 m	(18.75 feet).
Tons produced per foot of drilling	5.5 m tons	(6 tons).

The blasting agents used for dry ground is AN-FO and water gel in wet holes; both are exploded with a detonating cord and a booster fitted with SP electric blasting caps.

The production loading and transportation is accomplished with three 15 m^3 (19.5 cu yd) trucks and three 915 LHD units. This equipment is utilized to draw from 4 to 5 active stopes.

The haulage way is a 15 cm (6") thick concrete pad and where a lesser amount of tonnage is hauled from a stope, the roadbed is built of crushed rock and is maintained with a Bobcock Allatt Grader.

The loading and hauling equipment availability and utilization is as follow:

		Percentage Availability	Percentage Utilization
Loading	- 912 LHD units	77	88
	- 915 LHD units		
Hauling	- 9.5 cu yd units	70	84
	19.5 cu yd units	81	77

Ventilation

A total of 9,900 m^3/min (350,000 ft^3/min) of fresh air is presently pushed into the mine by three 168 cm (66") Joy Axivane fans two of which are set-up in parallel and produce 6,800 m^3/min (240,000 ft^3/min) of air for the No. I ventilation system which is mainly the decline in the east end of the mine; the other fan unit has blades set to produce 3,100 m^3/min (110,000 ft^3/min) of air for No. II ventilation system which is a 3.3 m X 3.9 m (11' X 13') ventilation raise through to surface at the west end of the mine. When necessary, the No. II system is prepared to install a second fan and possibly to produce up to 7,000 m^3/min (250,000 ft^3/min). Each ventilation system has a 7,000 kw-h (24 million Btu) propane gas heating system. The propane consumption for the 1980–81 winter was 1,000,000 litres. (220,000 gallons) for a total cost of $160,000. There are two viciated air exits, the main shaft and a 3.0 m X 3.6 m (10' X 12') raise.

Auxiliary ventilation is done by fans varying from 48 cm (19") diameter to 96 cm (38") diameter. The tubing is of plastic material and varies from 61 cm (24") diameter to 91 cm (36") diameter. Air operated electrically controlled ventilation steel doors are hinged to a 25 cm (10") thick concrete frame. Most of these main doors must be locked open for large blasts in the mine.

Dewatering

During the first year and half of operation, water flows from various sources in the mine reached a total amount of 6,400 l/min (1,400 g.p.m.). The water table stabilized around 137 m (450 feet) in depth from the shaft collar. Gradually, that volume diminished and during the last two years, the mine water volume kept fluctuating between 2,700 to 3,200 l/min (600 to 700 g.p.m.) with the water table constant in elevation.

Good ditches are maintained to conduct the drainage to the inter levels drain hole system made of 11.4 cm (4 1/2") diameter bore holes down to the 259 m (850') level settling sump which is 3.3 m (11') wide X 4.2 m (14') high X 30 m (100') long; the clear water sump is of same size and the water level is so controlled as to maintain a 422,000 l (93,000 imperial gallons) reservoir capacity in case of a mishap.

From the 259 m (850') level, another drain hole system conducts the water to a conical settling sump on the 350 m (1,150') level and finally to the mine bottom clear water sump on the 381 m (1,250') level.

There are two pumping stations for the mine, one on the 381 m (1,250') level and the other on the 259 m (850') level. The mine water is pumped to surface in stages with two 150 kw (200 HP) Allis Chalmer pumps installed on the 381 m (1,250') level and two 300 kw (400 HP) Mather and Platt pumps on the 259 m (850') level. Pumps are alternately operated in both pumping stations.

On the surface, the mine water is discharged in a series of three reservoirs used to precipitate impurities and finally part of this water may be used for mineral processing as required.

Pyrochlore Ore Processing At the Niobec Concentrator, Quebec, Canada

Mineralogical Composition and Size of Liberation

The whole deposit is highly heterogeneous and contains on average 0.67 percent Nb_2O_5. Important variations of Nb_2O_5 content and the metallurgical composition of ore are very frequent. The typical fresh type carbonate ore is represented by:

		% W
Carbonates:	*dolomite - the most calcite abundant ankerite siderite	64.9
Sulphides:	pyrite pyrrhotite	0.9
Oxides:	pyrochlore columbite	1.1
	magnetite hematite ilmenite	1.7
	apatite	6.8
	zircon	0.2
Silicates:	biotite chlorite Na, K feldspars pyroxenes nepheline	21.1
Others:	barite fluorite hydrocarbons	occassionally

* Dolomite/calcite 5/1

A total of eight different pyrochlore types can be present. Except for coloration, they are almost identical in composition. The most common type, Na-pyrochlore is shown in Figure 4. The ratio between pyrochlore and columbite is about 10/1. Both minerals are liberated after grinding at 95–100 percent - 65 mesh (208 μm).

500 μm

Figure 4. Pyrochlore crystals.

Since the carbonates are the most represented gangue minerals, the process flowsheet is based on their elimination.

Process Flowsheet

Briefly, the process flowsheet (Figure 5) consists of ore reduction to the size of liberation of useful minerals, desliming, carbonate flotation, followed by desliming again, magnetic separation, pyrochlore flotation and by two sections of pyrite flotation with leaching of the final concentrate between them.

Crushing, Grinding and Sizing

The mined ore is crushed underground in a jaw crusher to about 85 percent - 6" (152 mm), hoisted to the surface and accumulated in a shaft bin. Then it is conveyed to a rod deck screen. The oversize is crushed in the Hazemag impact crusher and together with the undersize stocked in two 1,400 ton bins. Four vibrating feeders regulated by a belt-scale supply the feed from the bins to the tertiary crushing arrangement consisting again of a rod screen and an Allis-Chalmers cone crusher. At the discharge from the crushing plant, the ore is reduced to 100 percent - 3/4" (20 mm).

The softness of the ore and the fragility of pyrochlore crystals necessitate a careful grinding and a close sizing to avoid high losses of pyrochlore in slimes. This is accomplished in a circuit of two mills, two series of D.S.M. screens and a screw classifier (Figure 6).

The rod mill operates in an open circuit, the ball mill in a closed circuit with the D.S.M. screens and the screw classifier. The resulting pulp contains 95 percent of particles less than 65 mesh (208 μm).

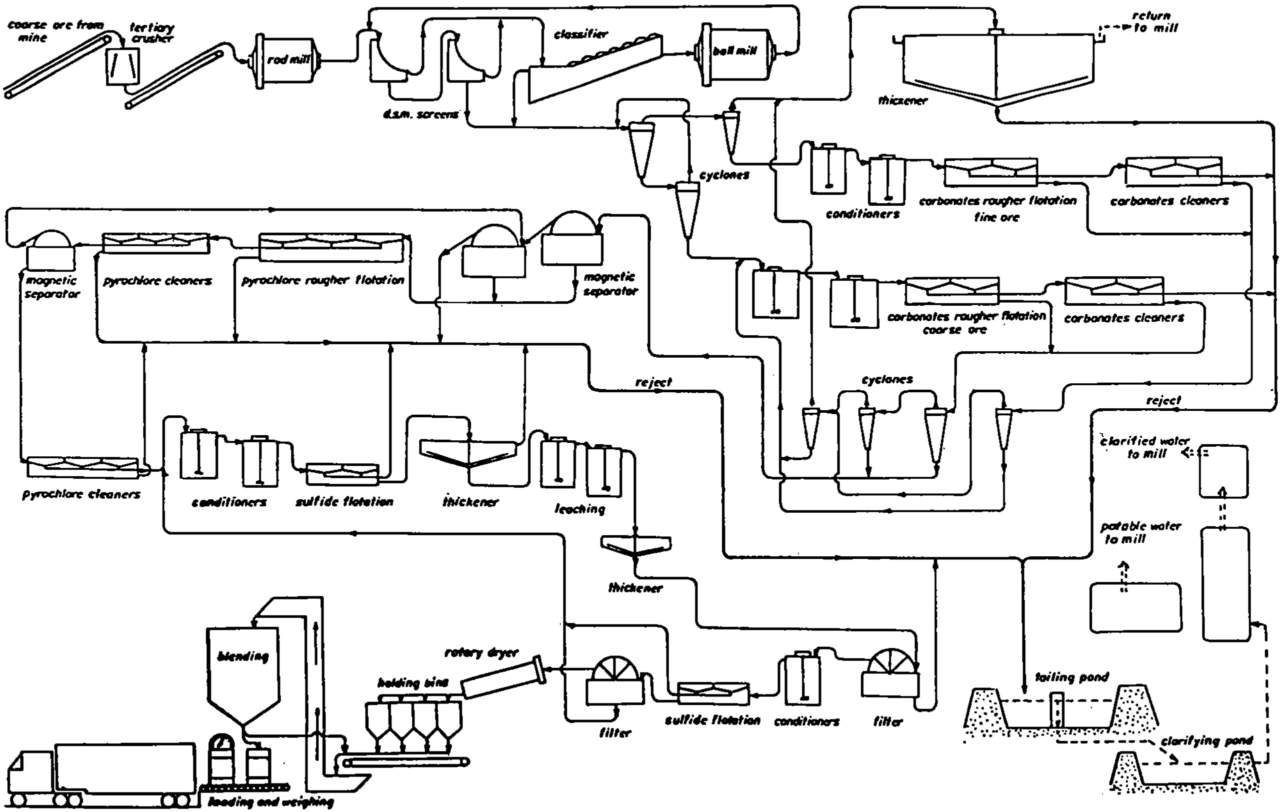

Figure 5. Niobec Pyrochlore benefication plant flow-sheet.

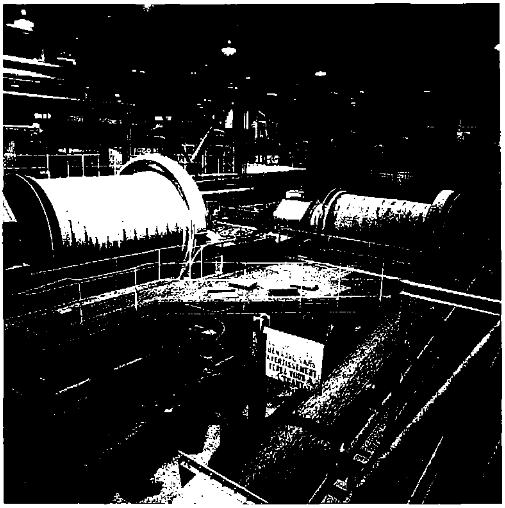

Figure 6. General view of the grinding circuit with the ball mill at the left and the 20d mill to the right.

Desliming

Even a close control of the grinding and the sizing cannot prevent some overgrinding to <10 µm. The <10 µm material being harmful to flotation will have to be removed. This is done in a series of Ø 10" and Ø 4" cyclones *(254* mm and *102* mm) (Figure 7). Beside removing <10 µm material the cyclones split the sand fraction into two parts.

The part *10-40* µm is fed to the fine carbonate flotation, the portion > *40* µm to the coarse carbonate flotation. The cyclone overflows, containing about *10-12* percent of the total pyrochlore, are sent to a thickener where slimes settle down and the recovered water is reused in the grinding and desliming circuits.

Carbonate Flotation

In order to prevent the partial flotation of carbonates in the pyrochlore flotation section itself these are removed ahead of it. Two separate sections, the fine and coarse carbonate flotations are used for this purpose. This arrangement increases the volume of carbonates removed when compared to the flotation in one section only. The flotation is performed at natural pH of around 8 using the emulsified fatty acids as collector, sodium silicate as pyrochlore depressant and water softener.

Table I. Average reagent consumption

	Lb/t	g/metric ton
Fatty acids (Acintol FA-3)	0.45	225
Emulsifier - TO:	0.16	80

About 25-30 percent of the mill feed is removed in the carbonate concentrate and 2-5 percent of the total pyrochlore is lost in it. The carbonate flotation tailings are sent to the water change and the magnetic separation sections.

Water Change and Magnetic Separation

Due to a shortage of potable quality water, the tailing pond reclaimed water is being used in all mentioned process sections. This water, because of a high salt content and residual reagents, is unsuitable to be used in the pyrochlore flotation and will have to be replaced. A Ø 10" and Ø 4" cyclone arrangement, at the discharge from each carbonate flotation section, removes most of $<10\ \mu m$ slimes formed during pulp handling and replaces the hard water by potable quality water. The total salt content is reduced by this about 3.5 fold. Two drum magnetic separators in series (Figure 7) remove a great part of magnetite from the combined cyclone underflows which are then sent to the pyrochlore flotation section.

Pyrochlore Flotation

It consists of a rougher and several cleaning stages (Figure 8) It recovers a bulk of pyrochlore, iron oxides, pyrite and silicate minerals in roughers. The flotation is performed under gradually decreasing pH from very slightly acidic to very acid using fluosilicic acid, oxalic acid or the blend of two of them as pH regulators and a diamine salt as a collector. The collector, even under very acid conditions, is not specific enough to collect pyrochlore only, and sodium silicate in combination with starch have to be used to improve its selectivity.

The careful pH control and the addition of modifiers allow the gradual rejection of carbonates, biotite, iron oxides and feldspars. The average dosage of the collector and the pH regulators in lb/t of the mill feed is:

Table II. Average reagent consumption

	Lb/t	g/metric ton
CES -109 (collector)	0.28	140
H_2SiF_6	1.44	719
$C_2H_2O_4$	0.63	314

The upgraded concentrate, at the discharge from the pyrochlore flotation section, contains up to 40-45 percent Nb_2O_5 but at the same time sulphur in form of pyrite, some carbonates and apatite. The marketable grade of concentrate is obtained only after the pyrite flotation, followed by the concentrate leaching in hydrochloric acid and by the pyrite flotation again.

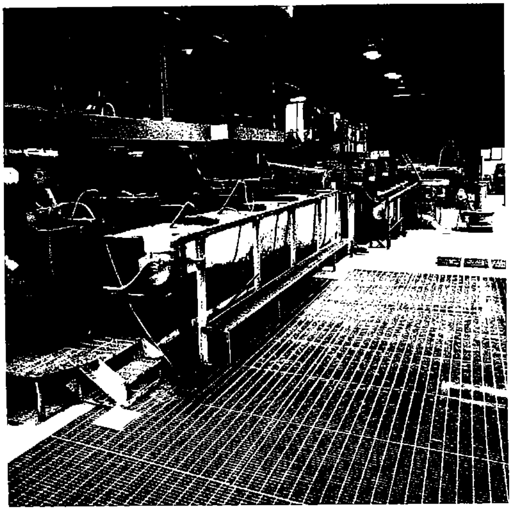

Figure 7. Magnetic separation section with de-sliming.

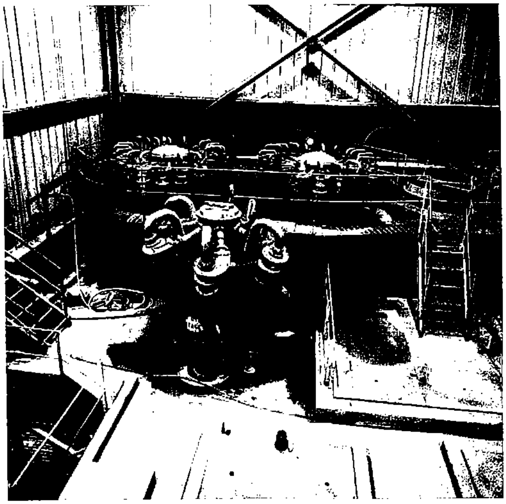

Figure 8. General view of pyrochlore flotation section.

Sulphide Flotation and Leaching

The only sulphides present in Niobec ore are pyrite and occasionally pyrrhotite. They are removed by the reverse flotation with xanthates in alkaline conditions at pH of around 10.5. To obtain a concentrate containing less than 0.10 percent sulphur, this has to be done in two separate sections with a leaching section between them (Figure 9).

The hydrochloric acid leach, performed at ambient temperature, removes residual carbonates, apatite and iron minerals representing about 0.1-0.2 percent of the overall mill feed. The leach liquor is discarded, the concentrate filtered, dried and packed.

Drying, Packing and Shipping

The concentrate cake, at the discharge from the filter, containing around 10 percent of moisture is dried in a rotary drier at 450 F (230° C) to <0.1 percent, then the material is conveyed to storage bins of a total capacity of 150 tons. Depending on its quality, the concentrate is blended to meet the market requirements and packed in barrels. Each barrel contains 320 kg of material. Fifty-five barrels constitute one lot. Lots are shipped by trucks to our clients in the U.S.A. or by boat in containers overseas. A container usually has two lots.

Water Supply and Tailing Disposal

Due to shortage in supply of potable water the tailing pond reclaimed water is used in ore preparation and carbonate flotation. This water represents about 70-75 percent of our total requirements. It has a high mineral salt content and is not suitable for the pyrochlore flotation. The potable water is used in all other sections.

The cycloning method of construction and the natural settling is used for the construction of the tailing pond. A clarifying pond and three retention ponds (Figure 10) are used to improve the water quality prior to its use in the mill.

Quality of Final Concentrate and Its Control

The production process and the quality of the concentrate are controlled by automatic sampling and laboratory assaying (Figure 11).

In addition to this the mill was recently equipped with an on-stream analyzer.

The quality of the produced concentrate varies in considerable proportions.

The typical client requirement may be as follows:

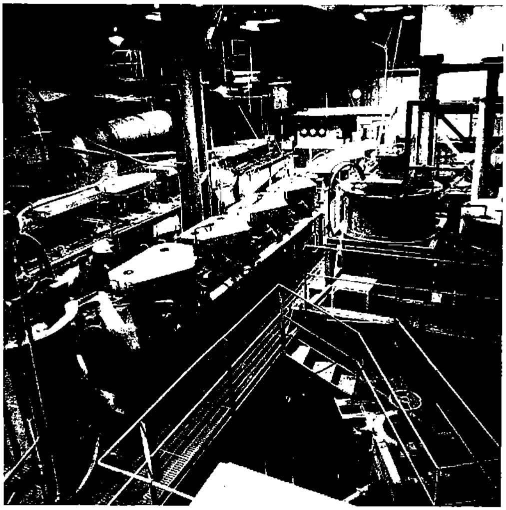

Figure 9. General view of the pyrite flotation (left) and the leaching section (right).

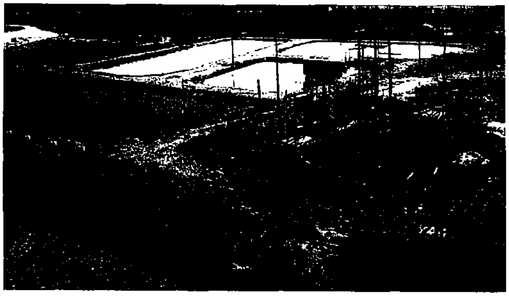

Figure 10. General view of the potable and reclaimed water ponds.

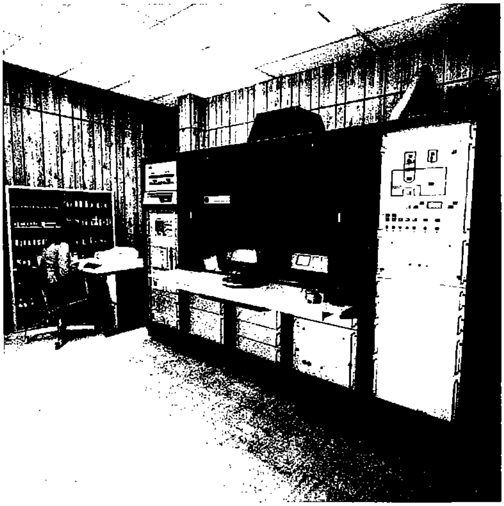

Figure 11. Modern six-element assay X-ray machine.

Table III. Typical quality of pyrochlore concentrate

% Nb_2O_5	+ 54 min.
% SiO_2	- 3.0
% P	- 0.1
% S	- 0.1
% SnO_2	- 0.05
% TiO_2	- 3.0
% Ta_2O_5	- 0.6

Typical Metallurgical Balance

	% W	%Nb_2O_5	Distr. Nb_2O_5
Feed	100.00	0.55	100.00
Slimes	14.48	0.42	11.06
Carbonate Conc.	25.19	0.05	2.29
*Pyrochlore tailings	59.69	0.17	17.37
Final concentrate	0.63	60.10	69.29

*Includes magnetics and pyrite flotation tailings.

Typical Quality of Produced Concentrate (percent)

Nb_2O_5	60.10
SiO_2	2.02
P_2O_5	0.07
S	0.10

Summary of Production Data

	1976*	1977	1978	1979	1980
Short tons of milled ore	250,160	583,889	615,807	627,628	657,074
Mill heads % Nb_2O_5	0.82	0.71	0.70	0.67	0.63
Lb of Nb_2O_5 produced	2,048,588	5,341.940	5,707,340	5,444,826	5,440.159
Overall recovery	49.70	64.27	65.71	64.78	65.78

*From May 1st to September 30, 1976

Summary of Reagent Consumption

	Lt/t	g/metric ton
Fatty acids	0.45	225
Fatty acid emulsifier	0.16	80
Starch	0.02	10
Sodium silicate	0.90	450
Diamine acetate	0.28	140
Fluosilicic acid	1.44	719
Oxalic acid	0.63	314
Sodium hydroxide	0.78	390
Copper sulphate	0.11	55
Potassium amyl xanthate	0.01	5
Polyacrilamide	0.02	10
Hydrochloric acid	4.66	2,330
Frother	0.02	10
Grinding balls	0.84	420
Rinding rods	0.22	110