THE UTILIZATION OF COPPER-NICKEL TAILING

- A Literature Survey -

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Minnesota State Planning Agency
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by

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ABSTRACT

It is estimated that the tailing that would be produced by a Minnesota copper-nickel concentrator of minimal size to sustain a smelter would amount to 20,000,000 tons per year. Such a mass of fine mineral material would present a considerable disposal problem. It might be possible to use some of the tailing to backfill the space in the underground mine vacated by the ore. It might also be possible to rework the tailing for the recovery of by-products and the elimination of asbestiform minerals and residual sulfides. However, the bulk of the tailing must end up in a huge pond or basin. Each of these procedures entail involved ramifications. Each procedure must be closely examined prior to the exploitation of a copper-nickel deposit so that a minimum of damage is done to the environment and the maximum economic value is extracted from the tailing.

The likely mineralogical, chemical, and size characteristics of the potential tailing were garnered and studied in the light of the three types of disposition. The literature on backfilling was perused for examples that would be applicable to a Minnesota underground copper-nickel mine. A possibility of substituting taconite tailings was considered. The economic values intrinsic to the major minerals in the tailing were summarized and possible approaches for their recovery were established: anorthite as a raw material for the production of aluminum and ceramics, ilmenite as a source of titanium oxide, magnetite as an iron-making raw material, graphite for electrical and chemical industries, and olivine for foundry sand and sand blasting applications. A cursory study indicated that the residual sulfide minerals in the flotation tailing might be concentrated with a wet, high-intensity, magnetic separator. Then the sulfide-rich fraction could be treated separately for the recovery of additional copper and nickel or for their disposal in a more environmentally compatible manner. In this way the source of heavy-metal ions in the tailing might be eliminated. Hydrous minerals were also shown to be concentrated in a certain magnetic fraction, which may contain asbestiform particles, and perhaps should be disposed of separately. Recommendations were made for desirable research into their concentration problems. The study and design of a possible tailing disposal area remain undeveloped.

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INTRODUCTION

Though several methods exist for the possible recovery of the copper and nickel in a Duluth gabbro ore, it appears that concentration of the sulfides by flotation followed by smelting offers the most realistic approach. The combination of these processes will of necessity be a large-scale operation. A recent economic evaluation of a hypothetical Minnesota copper-nickel deposit assumed as a starting point a minimum-sized smelter producing 150,000 tons[a] of copper-nickel matte per year.[1] A smelter of that size would require 680,000 tons of flotation concentrate per year. Pilot plant tests on one sample of Duluth gabbro ore[b] have indicated that the weight recovered as concentrate would be about 3 percent of the weight of the ore.[2] Hence, the mill feed required would be about 22,670,000 tons per year. The tailing, by difference, would total about 22,000,000 tons per year.

This amount of tailing will present a considerable disposal problem. It may be possible to use a portion of the tailing to backfill the space in the underground mine vacated by the ore. It may also be possible to rework the tailing for the recovery of by-products and the elimination of asbestiform minerals and residual sulfides. However, the bulk of the tailing must be contained in a huge pond or basin. Each of these methods of tailing treatment must be closely examined so that a minimum of damage is done to the environment and the maximum economic value is extracted from the tailing.

CHARACTERISTICS OF THE TAILING

Before one can discuss the disposal of the tailing, one should describe its essential characteristics. First, it will be quite fine. Any coarse country rock attached to the ore will be removed before the ore gets to the mill. To liberate the copper and nickel sulfides every ton of the feed to the mill will be ground through 65 mesh. Hence, the flotation tailing discharge from the mill will have a size distribution approximating that shown in Table 1.[2]

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[a] A short or net ton of 2000 lb will be used throughout this survey

[b] A 120-ton sample removed in 1974 by the International Nickel Company from a test pit on the Spruce Road about 4 miles east of Minnesota State Highway 1 near the South Kawishiwi River.

TABLE 1. DISTRIBUTION OF SIZES IN COPPER-NICKEL TAILING

Mesh Size	Weight Percent	Cumulative Wt % Finer	Cumulative Wt % Coarser
+65	1	–	1
-65 +100	14	99	15
-100 +150	26	85	41
-150 +200	22	59	63
-200 +270	12	37	75
-270 +325	4	25	79
-325 +400	5	21	84
-400 +500	5	16	89
-500	11	11	–
Total	100		

The known copper-nickel deposits are primarily associated with anorthosite and troctolite. Hence, the dominant minerals in the tailing will be anorthite, olivine, amphibole, chlorite, and pyroxene with minor amounts of biotite and ilmenite (Table 2).[3]

TABLE 2. DISTRIBUTION OF MINERALS IN COPPER-NICKEL TAILING

Mineral	Weight Percent	Millions of Annual Tons
Anorthite	59	13
Olivine	19	4.2
Pyroxene[a]	14	3.1
Biotite	3	0.7
Ilmenite	3	0.7
Pyrrhotite		0.1
Magnetite	2	0.1
Chalcopyrite		0.1
Total	100	22.0

[a]Includes amphibole and chlorite

USING THE TAILING AS BACKFILL

One obvious approach to the disposal problem is to return a portion of the tailing to the mine as backfill. Backfilling may be necessary as a safety

measure and to prevent subsidence if the mine workings are close to the surface. In this event, if the flotation tailing is not used as backfill, some other material must be employed, such as waste rock from the mine and surrounding area or tailing from the nearest taconite operation.

Not only is mill tailing the most widely used material for backfilling, it is also the most effective method of supporting the walls of mined-out underground openings. Once the hydraulic system for handling the fill has been installed, it requires a minimum of time and labor during the mining cycle.[4] Mill tailing has the following advantages as mine filling:

      (a)   It can be rapidly transported by air, water, or both.

      (b)   Places and spaces that would be inaccessible by other means of transport can be filled solidly and completely.

      (c)   When well placed and drained, mill tailing has greater compressive resistance than any other material except concrete.

      (d)   If the mine is adjacent to the mill, an abundance of fill is assured.

      (e)   Ultimately, when well settled and not violently disturbed, mill tailing will stand unsupported in an almost vertical face. If it contains iron sulfides (as would the Minnesota tailing), it may become, through oxidation, almost as firm as rock. At the Horne Mine in Noranda, Quebec, pyrrhotite mill tailing is added to coarser filling for this reason.

      (f)   Solidly placed mill tailing, being almost impervious to air, prevents short-circuiting of ventilating currents through stopes thus filled.[5]

Any slime (material finer than 20 microns) that is in the mill tailing must be removed before the tailing can be used since slime retards drainage through barriers. Desliming may be done by classification at the mill or in cones erected close to the point of discharge into the mine. In former years, the tailing used for backfilling was mainly coarser than 100 mesh, but the world-wide tendency towards finer grinding has diminished the proportion of plus 100-mesh material suitable for filling.

At Matahambre, Cuba, the mill tailing before desliming averages 49 percent minus 200 mesh. After desliming, the tailing is 49 percent coarser than 65 mesh and 97 percent coarser than 200 mesh. The deslimed tailing supplies 60 percent of all the material required for backfilling. The remainder comes about equally from ore sorting and rock development.[6]

For many years cyanide-plant tailing was used at the Homestake Mine in South Dakota only to augment and consolidate coarse filling. More recently it has been used as the sole material for delayed filling of both shrinkage and square-set stopes. The tailing, averaging 50 percent finer than 200 mesh, is sluiced from the cyanide vats to a dewaterer and then is discharged at about 60 percent solids to a header leading underground.[7]

When the copper mines in Butte, Montana, were all underground, thickened mill tailing, 50 percent finer than 200 mesh, in a pulp containing 18 to 30 percent solids, was dropped through 8-inch pipes and distributed laterally in 4-inch pipes. A head of 100 feet propelled the pulp 800 feet horizontally.[8]

Using the tailing from the copper-nickel concentrator as backfill in the mine will, of course, almost certainly preclude the future recovery of any potential resources in the tailing. It might be wiser to use tailing from one of the nearby taconite plants.

Erie, for example, produces 20 to 25 million tons of taconite tailing per year, about 30 percent of which is cobber tailing (minus 10 mesh in size), and about 70 percent is finisher tailing (91 percent minus 325 mesh). At present the cobber tailing goes to a hydroseparator for the removal of coarse materials, which are used for dam and road construction. The hydroseparator overflow, plus the finisher tailings, flows by gravity into several tailing thickeners. The underflow from the hydroseparators is joined by the under-flow from the tailing thickeners and is pumped to two tailing basins north of the concentrator. If Erie would release any or all of the underflow products, it would be possible to pump them to the copper-nickel mine site, where they could be used immediately for backfilling.

REWORKING THE MILL TAILING

Since a large amount of energy will have been expended to grind the ore to the liberation size of the copper and nickel minerals, it would be advantageous to attempt to recover other liberated minerals from the tailing. The copper-nickel mineralization in the Duluth gabbro is so low in degree that the ore has a gross metal value of only $15 to $20 per ton (1975 prices),[1] and it appears that, under present economic conditions, the profitability of the processing will be marginal at best. It is important, therefore, to investigate the possibility of developing a technology that would improve the overall economics by recovering possible by-products. The flotation tailing, since it contains many potentially valuable constituents and since it has already been ground, warrants serious consideration. The most promising minerals in the tailing for by-product recovery are anorthite, olivine, ilmenite, and perhaps graphite. Remote possibilities exist for the exploitation of magnetite.

Anorthite

It has been suggested that the anorthite, the dominant mineral in the copper-nickel tailing, might find application in the aluminum industry, since it contains about 29 percent alumina, Al_2O_3. Although usually termed anorthite, that is, calcium-aluminum silicate, $CaO \cdot Al_2O_3 \cdot 2SiO_2$, it is more properly called anorthosite, since it also contains up to 40 percent albite. Albite is a sodium-aluminum silicate, $Na_2O \cdot Al_2O_3 \cdot 6SiO_2$.

At the present time the production of aluminum is commercially possible only from bauxite, which is a hydrated form of aluminum oxide, with the

approximate formula $Al_2O_3 \cdot 2H_2O$. A purified aluminum oxide is first prepared from the bauxite, then dissolved in molten cryolite (sodium-aluminum fluoride, Na_3AlF_6), and finally electrolyzed to produce metallic aluminum.

In 1974 the United States imported over 14 million tons, or 85 percent of the bauxite used in the domestic production of aluminum.[9] However, the recent drastic increase in bauxite mining taxes by Jamaica and the subsequent formation of the International Bauxite Association point to the need for the development of domestic, nonbauxitic resources. Also, recent increases in the cost of aluminum production have forced the price of aluminum ingots to move much closer to the point where materials other than bauxite can be competitive. If such a point is reached, and as the technology to produce aluminum from nonbauxitic ores is developed and proven, aluminum companies will begin to shift to domestic resources.[10]

According to the U. S. Bureau of Mines,[11] the future aluminum resources of the United States consist mainly of anorthosites. As yet it is not economic to extract aluminum from anorthosites because the intermediate step to produce purified aluminum oxide is complicated and costly. Nevertheless, considerable research by industry and government is being directed toward developing an economic process to produce aluminum from anorthosites.

Anorthosite also has a commercial value when used as a feldspar in the glass and ceramic industries, where it serves as a fluxing agent due to its relatively low melting point. The glass and ceramic industries used about 950,000 tons of feldspar in 1974 and will probably use about 1,000,000 tons in 1980.[12] However, since feldspar is a relatively inexpensive commodity ($17 to $24 per short ton),[13] it is strongly affected by freight rates, with the result that most grades move within fairly short distances, usually under 1000 miles. Since most of the glass and ceramic industrial centers are in the northeast section of the United States, it is doubtful if the Minnesota anorthosite could find a market in any existing plants.

If there were a glass plant in Minnesota, the Minnesota anorthosite could be used in it if the iron content of the anorthosite could be reduced to acceptable limits. For flint glass, these limits are about 0.1 to 0.2 percent Fe_2O_3 and for amber glass about 0.3 to 0.4 percent Fe_2O_3. Tailings recently produced at the Mineral Resources Research Center have contained from 11 to 17 percent Fe_2O_3, depending upon the location of the pit from which the ores were taken. Hence, the anorthosite concentrate would have to achieve a considerable reduction in iron content. For a Minnesota operation to be economically feasible, it would have to produce the high-quality flint glass. A low-quality amber glass could not compete with eastern glasses due to the cost of the transportation to eastern markets. The anorthosite in the copper-nickel tailing may be too fine for use in a glass plant. When anorthosite is used for glass making, manufacturers usually require it to be between 30 mesh and 200 mesh in size. As shown in Table 1, the copper-nickel flotation tailing would be 36 percent finer than 200 mesh.

The anorthosite in the tailing might be used in a Minnesota ceramic plant. When a feldspar is used in ceramics, it is usually pulverized to at least 200 mesh (74 microns). Specifications for porcelain enamel require

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that feldspar be ground finer than 150 mesh (100 microns). The high sodium
content of the Minnesota anorthosite may restrict its use by pottery manu-
facturers who frequently prefer a high potassium content. Ordinarily they
also will not tolerate biotite, since its platy structure impairs the
physical strength.

Olivine

The olivine in the copper-nickel tailing varies in composition between
fayalite (Fe_2SiO_4) and forsterite (Mg_2SiO_4).

Recent observations showed that when olivine was substituted for silica
sand in foundry casting and sand blasting, it contributed greatly to the
reduction of silicosis problems.[14] Since the tailing would probably contain
nearly 20 percent olivine, the economic potential of the olivine in the
tailing could be substantial for casting and sand blasting applications in
northern Minnesota. However, fibrous amphiboles associated with the Minnesota
olivine would have to be removed before it could be used for sand blasting.
Fibrous minerals are suspected to be potential.

Ilmenite

According to the U. S. Bureau of Mines, in 1976 the United States pro-
duced 618,000 tons of ilmenite concentrates and imported another 144,000
tons. Most of the ilmenite, a ferrous titanate with the formula $FeTiO_3$,
was used to make pigments. Titanium dioxide has high opacity and covering
power. The contract price for the ilmenite concentrates, which contained
45 to 65 percent TiO_2, was $55 per long ton on the East Coast.

The United States has substantial reserves of ilmenite and, ordinarily,
the well-known occurrence of ilmenite in the Duluth Complex is not considered
a deposit that can be processed economically.[15] However, since the ilmenite
in the copper-nickel tailing is substantially liberated, the tailing might
be thought of as an ilmenite resource. If the flotation concentrator processes
about 22.67 million tons of copper-nickel ore per year, the ilmenite in the
tailing will amount to about 580,800 tons per year, certainly a quantity
worth considering. However, the problem of producing a marketable ilmenite
concentrate (45% TiO_2) from the tailing (less than 2% TiO_2) will require con-
siderable research. High-intensity magnetic separation in combination with
flotation is a possible process.

Graphite

Although graphite was not listed in Table 2 as one of the components
of the copper-nickel tailing, according to Mainwaring,[16] graphite has been
observed associated with sulfides in a number of drill cores from widely
separated locations at the base of the Duluth Complex. The graphite-sulfide
association may even form somewhat continuous layers. In the southern part
of Section 16, Township 57 North, Range 14 West, the graphite occurs with the
sulfides at depths of 260, 400, and 750 feet. This area is about 35 miles

north of the City of Duluth. Although Amax Exploration did not notice any graphite when they were sinking their shaft, they may have missed it because they were not specifically looking for it. It may have been present, but not in an amount greater than one-half of one percent. However, when Amax tunneled from the shaft, they found graphite in the massive-sulfide regions, some occurrences containing as much as one percent graphite. The graphite became most apparent in the Amax flotation concentrates which sometimes contained as much as 7 percent graphite.

The sulfides in these graphite-bearing rocks are essentially the same as the sulfides occurring elsewhere: mainly pyrrhotite, pentlandite, and cubanite with minor chalcopyrite. There are two striking differences between the graphite-sulfide assemblages and other sulfide-bearing rocks. They are: (1) the intimate association of the graphite with the sulfides and (2) the tendency for oxide rims to occur between the sulfide grains and the graphite grains. The graphite occurs only at the margins of masses of sulfide minerals, rarely enclosed completely within the sulfide. The sulfides are intergrown with filaments and concretions of graphite flakes.

The ore samples in which the graphite is found show evidence of having been contaminated by carbonaceous material from the footwall of the Duluth Complex. Much of the southern part of the Complex is underlain by carbonaceous shales and slates.

Depending on its purity, graphite may be used to manufacture electrodes, crucibles, pencils, lubricants, colloidal graphite, electric brushes, and dry cells. Graphite grades are specified by fixed carbon, volatile matter, ash, and size distribution.

For the past 20 years the demand in the United States for natural graphite has averaged 52,000 short tons annually. Approximately 95 percent of the domestic needs for natural graphite are being met through imports. Depending upon the carbon content, particle size, quantity sold, and packaging, the 1973 price for domestic graphite ranged between $250 and $600 per ton.

Graphite is one of the easiest minerals to segregate into a rough concentrate and one of the hardest to refine. It floats very readily, hence froth flotation is the simplest method for roughing. But cleaning by flotation is substantially ineffective without intervening treatment that liberates, and since such treatment leaves the gangue coated with graphite and hence floatable, subsequent separation is delicate, even if possible. If sulfides are present, they will also float and contaminate the graphite concentrate. Still, a few mills have operated successfully and the production of a graphite concentrate from the Minnesota copper-nickel tailing may be possible.

IMPOUNDING THE MILL TAILING

If the bulk of the flotation tailing is used for backfilling, only a small basin or pond will be required to contain the remainder. However, if

it is decided not to use the tailing for backfill, or even if as much as half of the tailing is recovered as by-products, a huge pond or basin will be needed to contain the 10 million tons of excess material that would be discarded every year (about 1,000 carloads per day). This does not include the millions of tons of water that would be discharged with the tailing.

Measures will have to be taken to limit or prevent the discharge of heavy-metal ions (copper, nickel, zinc, iron, cobalt, etc.) into the environment. Although nearly 90 percent of the copper and 70% of the nickel in the ore will probably be recovered in the flotation concentrate, the mill tailing will still contain about 0.04 percent copper, 0.04 percent nickel, 0.01 percent cobalt, and about 0.1 percent sulfur. Much of the metals will occur as sulfides, but some will be wrapped up as insoluble silicates. The sulfides could be leached, but the solutions must be discarded in an environmentally compatible manner.

The release of heavy-metal ions into the environment through the leaching of the tailing is a problem of potential concern. Recent research at the Mineral Resources Research Center indicates that the dissolution of the heavy-metal ions can be prevented by buffering the pH of the water in the tailing through the addition of crushed limestone.

Any asbestiform minerals that are present in the tailing would be deposited in the basin where they would eventually become dry and windblown. It would be better, if they are not too fine, to remove them from the tailing and dispose of them as backfilling in the mine.

The design of tailing pond dams, the dispersion of the effluent, the control of dust, and the selection of disposal sites are problems beyond the limits of this survey.

APPROPRIATE INVESTIGATIVE WORK

A recent bench-scale test at the Mineral Resources Research Center showed that a high-intensity, magnetic separator could be used to divide the flotation tailing into a number of products of different mineralogical contents depending upon the intensities of the magnetic fields imposed. The magnetic separation process simply consisted of a number of identical steps. In each successive step the nonmagnetic tailing from the preceding step was retreated at a slightly higher magnetic intensity. The magnetic intensity was varied by increasing the imposed current from 0.5 ampere to 5 amperes. The final tailing and each of the concentrates were microscopically examined and grain counts were made of their mineralogical contents.

As shown in Table 3, the magnetic tailings produced at 4 and 5 amperes were virtually pure anorthite. The 4-ampere magnetic concentrate was high in sulfides and ilmenite. The 2-ampere concentrate was high in biotite. The magnetic concentrates produced at 0.5 and 1.0 amperes were high in olivine, pyroxene, amphibole, and opaque minerals (mainly magnetite). Since these experimental separations were not made on completely liberated particles

TABLE 3a.   MINERALOGICAL COMPOSITION OF A FLOTATION TAILING FRACTIONATED BY HIGH-INTENSITY MAGNETIC SEPARATION

Product	% Wt	Anorthite %	Dist	Olivine %	Dist	Pyroxene & Amphibole %	Dist	Biotite %	Dist	Opaques %	Dist
0.5A Conc	14.19	1.5	0.35	61.4	45.04	22.7	22.45	1.4	7.25	13.0	44.1
1.0A Conc	17.49	2.0	0.59	45.2	40.90	38.7	47.21	3.5	22.10	10.5	43.9
2.0A Conc	7.49	16.3	2.05	29.6	11.48	33.5	17.50	16.7	45.29	3.9	7.0(
3.0A Conc	2.97	36.2	1.82	12.9	1.96	30.1	6.21	16.8	18.12	4.0	2.84
4.0A Conc	1.82	65.7	2.02	6.6	0.62	15.0	1.88	8.1	5.43	4.6	2.00
5.0A Conc	5.02	95.8	8.10	-	-	3.3	1.19	0.9	1.81	-	-
5.0A Tail	51.02	99.0	85.07	-	-	1.0	3.56	-	-	-	-
Composite	100.00	59.38	100.00	19.34	100.00	14.34	100.00	2.76	100.00	4.18	100.00

TABLE 3b.   MINERALOGICAL COMPOSITION OF OPAQUE MINERALS IN A FLOTATION TAILING FRACTIONATED BY HIGH-INTENSITY MAGNETIC SEPARATION

Product	Pyrrhotite %	Dist	Chalcopyrite* %	Dist	Ilmenite %	Dist	Magnetite %	Dist
0.5A Conc	1.30	28.36	0.65	20.46	8.45	45.45	2.6	74.00
1.0A Conc	1.58	41.79	1.05	40.91	7.88	52.27	0.53	18.00
2.0A Conc	2.34	25.37	0.98	15.91	0.59	1.52	0.20	4.00
3.0A Conc	0.78	2.99	1.78	11.36	0.39	0.38	0.20	2.00
4.0A Conc	0.46	1.49	2.99	11.36	0.46	0.38	0.23	2.00
5.0A Conc	-	-	-	-	-	-	-	-
5.0A Tail	-	-	-	-	-	-	-	-
Composite	0.67	100.00	0.44	100.00	2.64	100.00	0.50	100.00

*Plus cubanite

of ideal size, fuzzy splits were obtained due to middlings, wide size distributions, and nonuniform magnetic conditions across the processing stream.

Procedures for the recovery of by-products from the flotation tailing would appropriately be investigated by first fractionating a tailing sample with a wet, high-intensity, magnetic separator into crude concentrates of anorthite, ilmenite, and magnetite, and then developing suitable concentration techniques for purifying each fraction.

Anorthite

Recently two processes for producing a high-aluminum, low-iron, anorthite concentrate were investigated at the Mineral Resources Research Center. These processes were the multiple-step, high-intensity, magnetic separation previously described and flotation.

The flotation process consisted of two stages, each stage preceded by a conditioning step. In the first step an amine collector was used to float off and remove the biotite. In the second stage sulfonated oils were used to remove the iron-bearing minerals. Both stages employed acid circuits. The tailing from the second stage, being high in anorthite, was further upgraded by high-intensity magnetic separation. This process has been used successfully for over 37 years on other feldspars.

The results of the investigation of the two processes indicated that wet, high-intensity, magnetic separation was better than flotation for the following reasons:[19]

1. Economics. An anorthite concentrate can be produced by magnetic separation for approximately $2.00 per ton, but a concentrate produced by flotation would cost $5.00 per ton.

2. Equipment. Although the initial capital cost for magnetic separation would be higher than that for flotation, magnetic separation would require less ancillary equipment than flotation and thus would entail less down-time.

3. Environmental. Flotation requires a highly acid circuit and chemical agents that could cause serious problems in the event of accidental spills when being transported to the plant or when being used in the plant. The flotation tailing would contain reagents that might require special handling in the tailing basin.

4. Productivity. The percent weight recovered is higher by magnetic separation than by flotation.

As previously mentioned, the principal raw material at present for aluminum production is bauxite. However, before bauxite can be used, it must be separated from impurities, such as ferric oxide and silica, by being dissolved in a sodium hydroxide solution. The solution thus obtained precipitates hydrous aluminum oxide, in a very pure form, when agitated

with a small portion of previously separated material as seeds. The purified aluminum oxide is dissolved in molten cryolite in an electrolytic furnace. The aluminum that is produced by electrolysis collects at the bottom of the furnace and is tapped out from time to time and cast into bars or ingots.

To obtain pure aluminum oxide from anorthosite is more complicated. Several possible methods have been proposed and studied. In the lime-soda process the anorthosite is sintered with limestone, soda ash, and recycled residue to convert the alumina (Al_2O_3) to sodium aluminate and the silica to dicalcium silicate. The sintered product is then leached with a dilute sodium carbonate and the resulting slurry is processed to separate the residue from the leach liquor. The leach liquor is then treated with lime in autoclaves to precipitate dissolved silica, which is removed by settling and filtration. The desilicated liquor is carbonated to precipitate alumina trihydrate, which is separated from the liquor, washed, and calcined to produce alumina. The alumina is then fed to an electrolytic furnace for the production of aluminum.

Anorthosite can also be leached with caustic soda, but it may not be practical to produce alumina (Al_2O_3) by this method. Besides sodium aluminate ($NaAlO_3$), considerable sodium silicate (Na_2SiO_3) would also be produced and would impose a serious disposal problem. It might be advisable to investigate a more reactive agent, such as chlorine, for this purpose.

A preliminary study of the practicality of chlorinating the anorthite is recommended. Since the anorthite will have been thoroughly deslimed during the flotation and magnetic concentration, the study of its behavior during chlorination might well be carried out in a fluid-bed reactor.

The preparation of aluminum chloride is the basic step in the winning of aluminum metal by the ALCOA, ALCAN, Peacy and Grimshaw, and Toth processes.[20] Aluminum metal is then produced either by fused-salt electrolysis, by disproportionation of aluminum monochloride, or by reduction with metallic manganese.

It is reported that the new ALCOA process prototyped in a 15,000 tpy plant at Palestine, Texas, involves chlorination of pure alumina followed by fused-salt electrolysis. The process is expected to require only 70 percent as much power as today's most efficient plants.[21] A modified ALCOA process, therefore, might be proposed to utilize anorthite as raw material rather than bauxite. Even if the technology did not prove economically competitive with processes using imported bauxites, a technically proven process available in the wings could set a ceiling on bauxite processes.[22]

Olivine

The olivine contents of the magnetic concentrates that were produced with magnetic currents of 0.5 and 1.0 amperes (Table 3) merit some consideration. Currently, olivine is being processed by both dry and wet methods. Where the olivine deposits are sufficiently high in quality so as not to require substantial upgrading, as in Norway and Sweden, dry crushing and simple screening is adequate. If the olivine deposits are somewhat lower

in grade, as in North Carolina, gravity separation in a wet pulp is employed. After initial washing the material passes through hydrocyclones and is then filtered. Olivine produced by the wet process contains less fines than olivine produced by the dry process.

Recent developments seem to indicate that the consuming industries are demanding olivine products with more exacting specifications. The new criteria call for a high MgO content, an absence of accessory minerals, and---for many applications---an absence of fibrous dust. Olivine is very readily altered to serpentine, $Mg_6(OH)_6Si_4O_{11} \cdot H_2O$, a potentially fibrous mineral. Since serpentine yields water when ignited, the new standards for commercial olivine call for a low loss-on-ignition.

Obviously, it is becoming increasingly difficult for processors using only dry screening to achieve and sustain the new and more rigid specifications. At the very minimum, they are being forced to add a final washing to remove the dust fractions. Calcining processes have been utilized with some degree of success where the nature of the ore requires this treatment to obtain an acceptably low loss-on-ignition.

The olivine in the copper-nickel tailing is associated with pyroxenes and amphiboles, which would have to be removed as potential hazards in "asbestosis" problems, especially if the olivine is to be used for foundry casting and sand blasting. Since the tailing will be substantially finer than 65 mesh, flotation is the most likely process. The separation of the pyroxene and amphibole from the olivine may be difficult since the flotation characteristics of the three minerals are quite similar. However, the floatabilities of pyroxene and amphibole with anionic collectors in an alkaline pulp appear to be less than that of olivine.[23]

Ilmenite

The predominance of ilmenite in a wet, high-intensity, magnetic concentrate with sulfide minerals (the four-ampere concentrate in Table 3) makes it particularly attractive for further treatment. Perhaps after the copper and nickel have been recovered by flotation, an ilmenite concentrate could be produced by magnetic separation.

To lessen the dependence of our titanium industry upon foreign ore sources, the direct chlorination of domestic ilmenite ores is being examined closely. The technology to produce the intermediate product titanium tetrachloride from ilmenite has been adequately demonstrated both in the laboratory and in commercial production. Such titanium tetrachloride is sufficiently pure for either metal or pigment production. Iron chloride is also produced from the ilmenite and a separate stream leading to the dechlorination facility should be established to handle this material. A raw materials cost analysis[15] (excluding operating and capital costs) showed a distinct economic advantage for ilmenite chlorination over rutile chlorination, an advantage that would be further enhanced if the chlorine were recovered from the iron chloride for recycling.

Magnetite

With a hypothetical magnitude of operation in excess of 22 million tons per year, a magnetite concentrate amounting to about 80,000 tons annually could be recovered by magnetic separation. However, since the magnetite in the tailing is closely associated with the ilmenite, the TiO_2 content of the magnetite concentrate should first be carefully assessed through electron microprobe analysis and chemical analysis to determine if the titanium is in solid solution with the magnetite or occurs as unliberated ilmenite. Titanium dioxide is a very troublesome impurity in an iron ore. It is partly reduced in a blast furnace and forms compounds of titanium, carbon, and nitrogen that interfere with the topping of the hot metal and slag. The generally accepted tolerance of TiO_2 in an iron ore burden is about 0.1 percent.[24]

Graphite

Since graphite is a naturally floatable mineral, any liberated graphite in the copper-nickel tailing could be readily concentrated by flotation, using kerosene and a frother. However, the presence of talc, another naturally floating mineral, in the Duluth Complex may present a complication. If the graphite is intimately associated with chalcopyrite in some manner, the graphite concentrate may be contaminated by chalcopyrite and the separation of the two minerals may be difficult. The distribution of graphite occurrences in the region, the sizes of the individual graphite grains, and the estimated purity of the final graphite concentrate are topics that merit further examination.

CONCLUSIONS

The tailing that would be produced by a Minnesota copper-nickel concentrator would contain potentially valuable minerals such as anorthite, olivine, and ilmenite. Therefore, before the exploitation of a deposit is underway, methods for the recovery of these minerals as by-products should be investigated. Simultaneously, market surveys to establish the economic potentials of the minerals should be completed. It must not be forgotten that, since these minerals will have been already ground to liberation size, the process of highest cost in their recovery, comminution, will have been eliminated. Although using the tailing as backfill is advantageous for esthetic and environmental reasons, once the tailing has been returned to the mine, any economic minerals in the tailing will have been irretrievably lost. Therefore, it is essential to study thoroughly the economic potential of these minerals before using them as backfill. At the same time the possibility of removing residual sulfides and asbestiform minerals from the tailing should be investigated. An attempt should be made to recover the metals from the residual sulfides by a process such as fine grinding followed by flotation or leaching. It might be possible to obtain taconite tailing for use as backfill. Then, if backfill is needed at the mine, some of the fractions rich in residual sulfides and/or asbestiform minerals could be discarded in an environmentally acceptable manner by using them in addition to the taconite tailing as backfill. The remainder of the copper-nickel tailing could be impounded for possible future reclamanation.

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