

Volume 2-Chapter 3

MINERAL PROCESSING

Minnesota Environmental Quality Board  
Regional Copper-Nickel Study  
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## A NOTE ABOUT UNITS

This report, which in total covers some 36 chapters in 5 volumes, is both international and interdisciplinary in scope. As a result, the problem of an appropriate and consistent choice of units of measure for use throughout the entire report proved insurmountable. Instead, most sections use the system of units judged most common in the science or profession under discussion. However, interdisciplinary tie-ins complicated this simple objective, and resulted in the use of a mix of units in many sections. A few specific comments will hopefully aid the reader in coping with the resulting melange (which is a reflection of the international multiplicity of measurement systems):

- 1) Where reasonable, an effort has been made to use the metric system (meters, kilograms, kilowatt-hours, etc.) of units which is widely used in the physical and biological sciences, and is slowly becoming accepted in the United States.
- 2) In several areas, notably engineering discussions, the use of many English units (feet, pounds, BTU's, etc.) is retained in the belief that this will better serve most readers.
- 3) Notable among the units used to promote the metric system is the metric ton, which consists of 2,205 pounds and is abbreviated as mt. The metric ton (1,000 kilograms) is roughly 10% larger (10.25%) than the common or short ton (st) of 2,000 pounds. The metric ton is quite comparable to the long ton (2,240 pounds) commonly used in the iron ore industry. (Strictly speaking, pounds and kilograms are totally different animals, but since this report is not concerned with mining in outer space away from the earth's surface, the distinction is purely academic and of no practical importance here).

4) The hectare is a unit of area in the metric system which will be encountered throughout this report. It represents the area of a square, 100 meters on a side ( $10,000 \text{ m}^2$ ), and is roughly equivalent to  $2\frac{1}{2}$  acres (actually 2.4710 acres). Thus, one square mile, which consists of 640 acres, contains some 259 hectares.

The following table includes conversion factors for some common units used in this report. Hopefully, with these aids and a bit of patience, the reader will succeed in mastering the transitions between measurement systems that a full reading of this report requires. Be comforted by the fact that measurements of time are the same in all systems, and that all economic units are expressed in terms of United States dollars, eliminating the need to convert from British Pounds, Rands, Yen, Kawachas, Rubles, and so forth!

Conversions for Common Metric Units Used in the Copper-Nickel Reports

1 meter	= 3.28 feet = 1.094 yards
1 centimeter	= 0.3937 inches
1 kilometer	= 0.621 miles
1 hectare	= 10,000 sq. meters = 2.471 acres
1 sq. meter	= 10.764 sq. feet = 1.196 sq. yards
1 sq. kilometer	= 100 hectares = 0.386 sq. miles
1 gram	= 0.037 oz. (avoir.) = 0.0322 Troy oz.
1 kilogram	= 2.205 pounds
1 metric ton	= 1000 kilograms = 0.984 long tons = 1.1025 short tons
1 $\text{m}^3$	= $1.308 \text{ yd}^3 = 35.315 \text{ ft}^3$
1 liter	= 0.264 U.S. gallons
1 liter/minute	= 0.264 U.S. gallons/minute = 0.00117 acre-feet/day
1 kilometer/hour	= 0.621 miles/hour
degrees Celsius	= $(5/9)(\text{degrees Fahrenheit} - 32)$

### 3.1 INTRODUCTION AND SUMMARY OF FINDINGS

The processing chapter of this report deals with the stage of the overall system which converts the low grade ore material of the type expected to result from mining the Duluth Gabbro Complex of northeastern Minnesota into a valuable concentrate, suitable as feed material for a smelter/refinery complex producing finished copper and nickel metal.

Areas of concern include crushing, grinding, flotation concentration, tailing disposal, and total water system management. Although all of these areas are important in the overall picture, the disposal of waste material and the management of the closely associated water system are of major importance to environmental considerations. These areas most directly affect the environment as they occupy the majority of the land devoted to processing (tailing basin); they introduce the main sources of potential pollution resulting from processing (dust, chemicals, and leaching); and they consume the major local resources used in the mineral separation process (ore and water).

Because of the environmental emphasis in this study, the majority of this chapter deals with the tailing basin and associated water system. The document is complete, however, in all of the above areas.

Figure 1 is a sketch of a typical processing facility showing the flow of materials from the mine through the final products, i.e. concentrate to smelter and tailing (waste) to the basin for settling and disposal. Note also the indication of clarified water returned to the plant system for reuse. Each of these phases of the overall processing system will be discussed in detail in this

chapter and summarized into hypothetical models for copper-nickel development in Chapter 5. The reader is urged to consider both chapters when evaluating the processing requirements of Minnesota's potential copper-nickel industry.

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Figure 1

Processing of any sulfide ore material generally follows the flowsheet shown in Figure 2. Each step in sequence can be identified with the facilities shown in Figure 1. These steps or stages consist of:

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Figure 2

- 1) Size reduction of the mined material to a size at which there is optimum liberation of the valuable minerals from the waste rock or gangue material. Generally, the sequence consists of several stages of crushing followed by multiple stages of grinding. The optimum liberation size is set by economic criteria rather than consideration of the absolute liberation size, as grinding costs increase rapidly as the amount of grinding increases and the resulting size or mesh of grind decreases.
  
- 2) Beneficiation of the ground ore to separate a valuable concentrate from the worthless gangue or tailing. Beneficiation or concentration takes many forms, from simple gravity separation suitable for titanium beach sands, to magnetic separation applied to Minnesota's taconite industry, to highly complex sulfide ore flotation systems in which chemical additives are used to adjust solution pH levels and affect mineral surfaces, allowing subsequent separation to occur. Some form of this latter system would be required for treatment of ore from a Minnesota copper-nickel mine.

FIGURE 1

# TYPICAL PROCESSING FACILITY LAYOUT

SCALE: 1" = 100' (VERTICAL) 1" = 100' (HORIZONTAL)

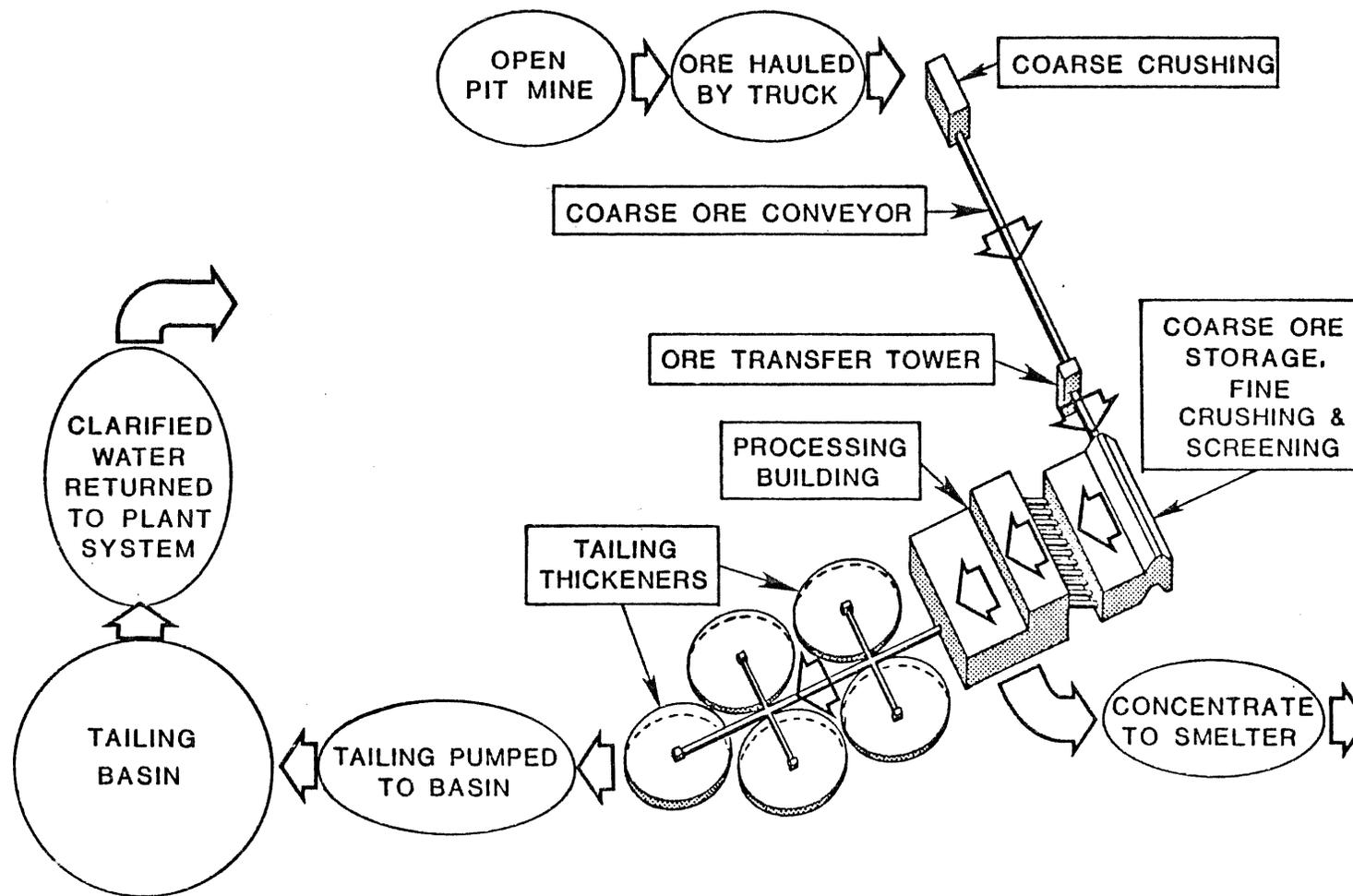
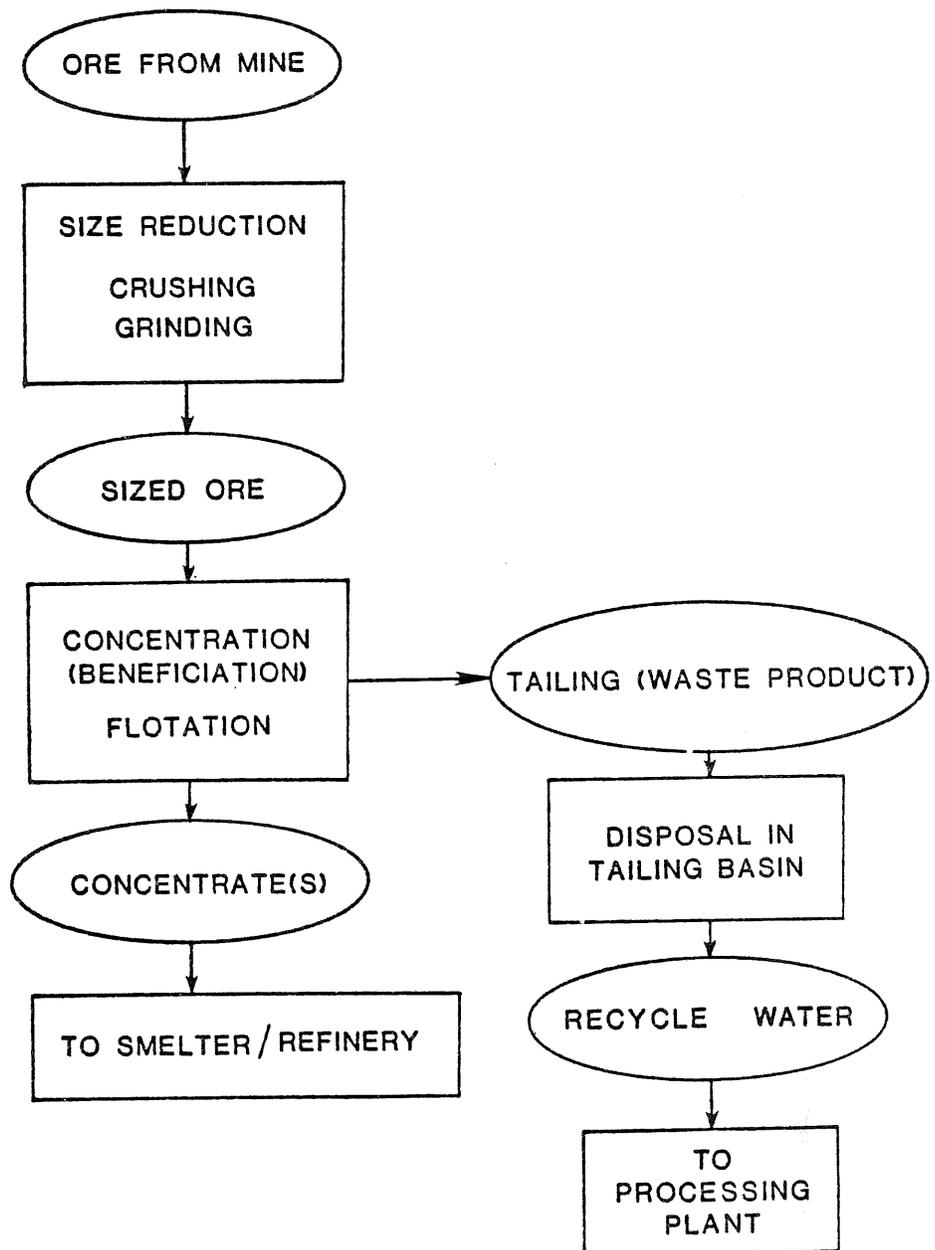


FIGURE 2  
GENERALIZED PROCESSING FLOWSHEET  
MINNESOTA CU-NI ORE



3) Proper disposal of waste material in one or more tailing basins which are structurally and environmentally sound.

4) Transportation of concentrate to a smelter/refinery facility.

Integrated into these stages is a water system which acts as a transport medium for the material being processed and which for modern operations typically provides for maximum reuse of process water and minimum fresh water make-up requirements. Also included are a variety of miscellaneous materials handling systems necessary to insure a smooth operation.

The processing of Minnesota copper-nickel resource material must follow these same definite sequential paths, but within each path are various alternate choices or possible options, each of which may offer advantages or disadvantages to the total system. Some of these choices may not be voluntary, but may in fact be determined as a result of information not available at this time. Others will depend on company preference. One such set of options is the choice of using bulk or differential flotation of the copper and nickel sulfide minerals. Bulk flotation collects all sulfides as one product and separates it from one waste material. The separation of copper from nickel must then be performed in the smelter/refinery complex. Bulk flotation practice is well-proven in the copper industry, and has been adapted for Minnesota copper-nickel ores by both private industry and public research institutions. However, the composition of this concentrate, in terms of its copper and nickel content, would be unique to Minnesota. This means that presently proven and successful smelting techniques in use elsewhere in the world may not be directly applicable to this concentrate. It means that a great deal of research and development may be needed to perfect such a successful treatment process.

Differential, or selective, flotation is the separation of mineral constituents into a simple copper concentrate and a nickel-copper concentrate, both similar to concentrates presently produced elsewhere in the world. This separation occurs in the mill, thereby allowing subsequent smelting and refining stages to treat the separate concentrates by established methods. The advantage, then, of differential over bulk flotation is this production of 2 concentrates, each of which are treatable by well-proven smelting and refining techniques for the recovery of copper and nickel metals. The potential disadvantages include higher total costs, lower overall metal recovery, a greater potential for water quality problems and the need to plan and construct a mill relying on technology and flowsheet designs that have not yet been well-proven on a production scale (see discussion on differential and selective flotation, section 3.3.3 of this chapter).

Mining companies contemplating development in Minnesota have considered both methods of concentrating the copper-nickel gabbro resources. Figures 3 and 4 each idealize one example of the bulk and differential or selective flotation applications and how their valuable products are subsequently treated for metal production. The bulk system (Figure 3) details the stages beyond concentrate production to indicate the complexity involved in separating the copper from the nickel. The differential or selective system (Figure 4) is shown in a more simplified form, as the corresponding stages are less complex with only one metal product of principal interest in each line. With the limited data available at this time, it is not yet clear whether the choice of type of flotation to be employed will be strictly dictated by economics and the mineralogy of the ore, or whether the choice can be made according to the preference of each company involved in any future development of the resource.

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Figures 3 and 4

The following summary of findings includes results shown in Chapter 5 on the integrated models. The data developed there was based on subsequent information in this chapter and is therefore presented here in summary form.

Production capacities of the modeled processing facilities are integrated with the mining and smelter/refinery facilities such that processing will accept all ore material produced at the mine, and will then produce sufficient concentrate to supply the needs of the modeled smelter/refinery complex. The starting point for such an integration was threefold:

- 1) Modeled smelter/refinery production of 100,000 mtpy of combined copper and nickel, with associated losses in the system (see Chapter 4)
- 2) Ore grades assumed from the literature and current data of:

	<u>Underground</u>	<u>Open Pit</u>
% Cu	0.800	0.494
% Ni	0.185	0.114

- 3) Finally, processing metal recoveries based on data supplied by industry and government of:

88.89% Cu recovery

73.75% Ni recovery

Based on these starting points, 5 processing models were developed with yearly capacities of 5.35 to 20.00 X 10<sup>6</sup> mtpy of ore, as follows:

5.35 X 10<sup>6</sup> mtpy    underground ore

11.33 X 10<sup>6</sup> mtpy    open pit ore

FIGURE 3 TYPICAL EXAMPLE OF BULK CU-NI FLOTATION TOTAL SYSTEM FLOWSHEET

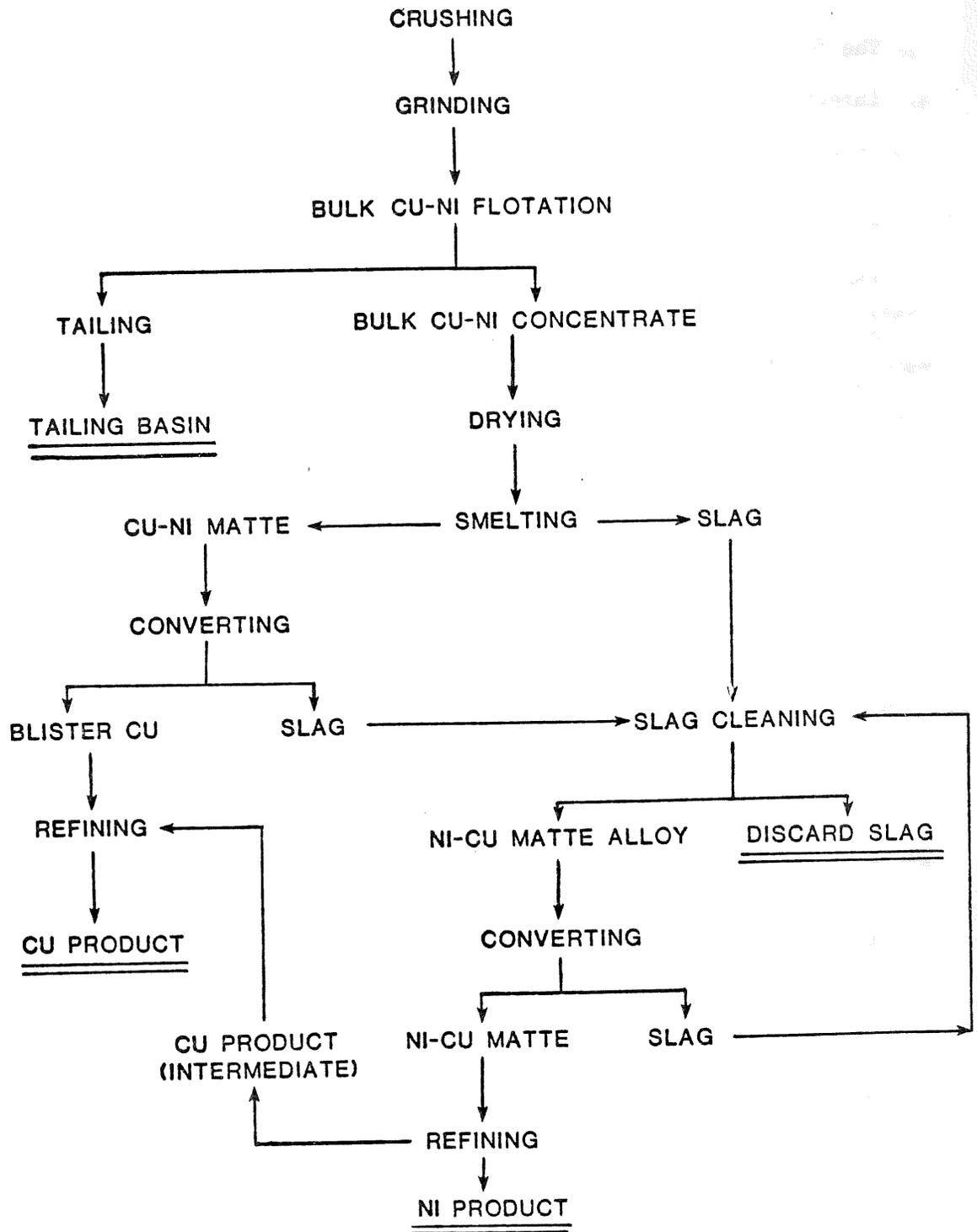
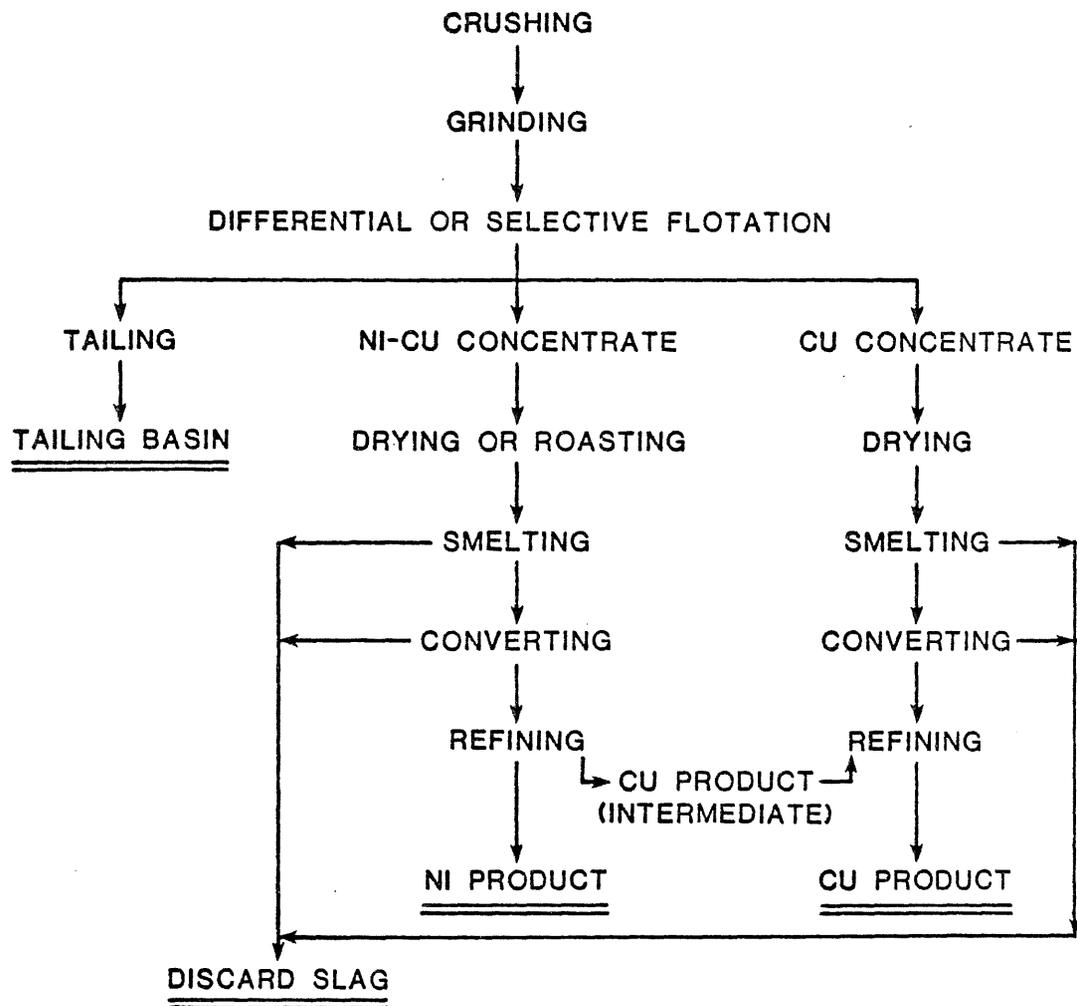


FIGURE 4

TYPICAL EXAMPLE OF DIFFERENTIAL OR SELECTIVE  
CU-NI FLOTATION TOTAL SYSTEM FLOWSHEET



16.68 X 10<sup>6</sup> mtpy combination of underground and open pit ores  
described above

12.35 X 10<sup>6</sup> mtpy underground ore

20.00 X 10<sup>6</sup> mtpy open pit ore

Processing facilities were developed for each of the above cases to cover the expected range of production; however, only the 3 largest operations produce enough concentrate to supply the smelter/refinery complex. Each, in fact, produce the exact amount of bulk concentrate required:

635,259 mtpy concentrate

13.825% Cu

2.547% Ni

As discussed later, differential flotation in the processing system would result in 2 concentrates, one high in copper and the other with roughly equal amounts of both copper and nickel, but the sum of both concentrates would equal the bulk concentrate described above.

A processing facility must be located as close as possible to the corresponding mining facility to minimize transportation costs of the ore to the plant. Location within a mile of the mine boundaries would result in a reasonable transportation cost provided the risk of damage caused by mine blasts would be negligible. In the case of underground mining, the distance could be reduced as such blasts tend to be much smaller than open pit blasts and their effects are greatly absorbed by the volume of material between the blast area and the surface location of the plant.

However, once the initial plant investment has been recovered and the original mine nears exhaustion, the life of the processing plant and subsequent facili-

ties could be extended by transporting ore from more distant mining operations to the existing treatment facilities. Often, economics dictate such an effort if suitable resources and a demand for the product exist. Additionally, processing facilities may be expanded to include treatment of ore materials from isolated mines during the life of the original mine, such as has been done in Minnesota's taconite industry.

Construction of processing facilities, such as discussed here and in Chapter 5, is estimated to require some 2 1/2 yr with a time lag of up to 3 yr after mine construction begins; the final objective being full processing production capacity when the mine is able to supply its full measure of feed material.

Allowing sufficient time after the granting of final permits for the detailed engineering, equipment purchase, and facility construction, 3 1/2 to 5 1/2 yr would probably pass from the decision to proceed point until the processing facility would be in production. Allowing a year for start-up problems, full production could be reached in 4 1/2 to 5 1/2 yr from the starting point.

The Duluth Gabbro material is typically very low in sulfide content (see Volume 3-Chapter 2) and, as discussed earlier, less than one percent of recoverable copper plus nickel metal. The majority of the material passing through a processing facility will, therefore, be discharged as waste or tailing and must be disposed of in an environmentally sound manner. Modeled development alternatives discussed in Chapter 5 show between 94.9 and 96.8% of the feed material being discharged as waste, depending on the combination selected. In the various model operations, these percentages translate into 11.7 to  $19.4 \times 10^6$  mtpy of tailing for disposal in an acceptable manner.

Since actual development may include different technology than that modeled in this report, and site specific material grades could vary considerably from the

modeled grades, it is reasonable to assume the split between concentrate and tailing will vary from the modeled values. However, it is safe to conclude that no more than 10% of any sizable quantity of mineralized material, is expected to be recovered as copper- and nickel-bearing concentrate, leaving at least 90% to be disposed of as tailing.

Later in this chapter a discussion of secondary tailing treatment to recover other mineral values is presented (section 3.2.5). Any additional recovery would, of course, reduce the total tailing to be disposed of, as would back-filling the mine with coarse tailing material (Chapter 2).

Since tailing material will require permanent disposal sites which are environmentally and economically sound, a considerable portion of this chapter is devoted to this subject. Processing by itself is not expected to result in significant environmental problems, but tailing disposal due to the sheer magnitude of the solid and liquid materials involved, and due to the permanency of the resulting site, must be considered in great detail.

Areal requirements for processing facilities are discussed both in this chapter and in Chapter 5. Actual plant requirements (120 to 400 acres) are small compared to the tailing basin needs, modeled as 1,067 to 4,016 acres for disposal of waste material generated from operations ranging from 5.35 to 20.00 X 10<sup>6</sup> mtpy, respectively. All tailing basins were designed to contain the material at an average depth of 70 ft and to be circular in shape. Such a design would be highly unlikely in actual practice as advantage would be taken of existing topography; however, for the purposes of this report, the hypothetical tailing basins as designed are sufficient.

In addition to the actual areas described above, another 40% is assumed necessary as undisturbed area surrounding each facility. This area is included in the property boundaries but is not directly disturbed.

Processing requirements, including energy, manpower, water, chemicals, capital, and operating costs were modeled based on available information. The details are given in a previous processing report (Veith 1978) and additionally developed in this chapter and in Chapter 5. Summary values will be given here, and the reader is referred to the above reports for additional information.

At full rated production, the designed hypothetical processing facilities require energy to operate machinery and to heat the buildings housing the facilities. An average of  $247 \times 10^6$  BTU of equivalent energy is necessary for each metric ton of ore processed, of which almost 98% (about 23 KWH) is electrical energy.

Manpower needs were developed based on existing industry data generated by both private industry and government. An average manpower value of 26 man-years for each million metric tons of annual ore capacity resulted, with a range of 34 down to 21 when increasing the plant size from 5.5 to  $20.00 \times 10^6$  mtpy, respectively. The obvious increase in manpower efficiency when increasing the plant capacity results from the fact that a minimum number of people is necessary regardless of the operation size, and increasing the capacity does not increase the manpower needs proportionately. For example, one person may be necessary to control one grinding mill, but the same person may as well control 3 or 4 mills in the same plant area at the same time.

Water necessary in copper-nickel processing models was estimated on the basis of the amount needed to suspend the solid material in a slurry of about 30% solids.

The result was a total water need of 650 gal/mt of ore for processing alone, of which about 96% would be returned from the tailing basin via a recycle water system and 4% would need to be supplied from an outside water source. (This does not imply that only 4% of the total water used is lost to evaporation, seepage, etc., but when including the inflow from precipitation to the tailing basin, the deficit is only 4% of the total needed.)

Chemical reagents are the heart of the mineral flotation concentration process, as they are necessary to condition sulfide mineral surfaces in a slurry such that the mineral particles will attach themselves to rising air bubbles and thus be removed from the total as a concentrated sulfide product. Flotation of copper-nickel sulfide as a bulk concentrate is detailed in the model development and requires only a simple xanthate collector and an alcohol frother to produce the required concentrate. Quantities of each reagent ranging from 0.1 to 0.2 lb/mt of ore are necessary to effect such a separation, and neither is expected to result in environmental problems, as they either remain with the concentrate and would be destroyed in the smelter or rapidly break down when exposed to air in the tailing basin (Iwasaki et al. 1978).

Differential or selective flotation which results in separate copper and nickel-copper sulfide concentrates requires a more complex reagent suite and may therefore provide a greater potential for environmental impacts. However, no data is available to evaluate such a system at this time.

Capital investment and operating costs for modeled copper-nickel processing facilities vary as do the manpower needs described above. The capital investment range is from \$16.7 to \$11.6 per annual metric ton of ore treatment capacity, when increasing the plant size from 5.35 to 20.00 X 10<sup>6</sup> mtpy, respectively.

Corresponding operating costs vary from \$2.6 to \$2.3 per metric ton of ore, obviously not ranging as much as the capital costs, since many of the component values are derived on a consumption per metric ton basis. For example, increasing the plant size by a factor of 4 would decrease the unit capital cost by 30%, but the corresponding operating cost would decrease by only 12%, reflecting primarily the increased manpower efficiency of the larger operation.

A brief comparison of modeled Minnesota copper-nickel operations to the existing taconite industry is valuable to put copper-nickel into perspective. Taconite processing involves mining and coarse crushing similar to proposed open pit copper-nickel mining, but there the similarity stops. Taconite occurs in bands within the host rock and can, therefore, be initially concentrated in a coarse (+1/4 in) state. The resulting concentrate must then be finely ground (-325M) and subjected to multiple stages of intensive magnetic separation and washing to effect a final concentrate which then must be agglomerated for transportation to and use in the blast furnace.

Kakel<sub>1</sub> (1978) reports a total energy use equivalent of  $5.15 \times 10^6$  BTU per net ton (nt) of iron metal delivered to the blast furnace as Minnesota's Mesabi Range taconite pellets. This value translates to  $1.27 \times 10^6$  BTU/mt of iron ore processed, of which  $0.93 \times 10^6$  BTU/mt ore is for direct energy requirements (electricity, diesel oil, etc.) through all phases of pellet production and excluding only transportation to the blast furnace. Of this amount, about 51% or  $0.48 \times 10^6$  BTU/mt ore can be attributed directly to processing functions. These data generally agree with Energy Agency data (Hirsch 1975), which is not detailed here.

In comparison, the copper-nickel development models require 0.70 to  $0.98 \times 10^6$  BTU/mt ore as direct energy for mining and processing, of which 25 to 30% or

$0.25 \times 10^6$  BTU/mt ore can be attributed to processing alone. Thus, although taconite and copper-nickel are comparable in total energy consumption through pellet production and concentrate production, respectively, the intermediate stages are not comparable. For example, taconite with its lower stripping ratio requires less energy for mining than copper-nickel; and magnetic separation is more energy intensive than flotation, so processing of copper-nickel is less energy consuming than taconite.

Kakela also includes brief employment data for the taconite industry through pellet production. The figure of 4,380 nt pellets/man-yr is given, which translates into 13,350 mt ore/man-yr. On the basis of mining and processing operations only, the range for copper-nickel is 6,500 to 14,500 mt ore/man-yr, thus comparable to taconite.

Taconite expansion in 1973 (Kakela 1978) ranged in capital cost from \$50 to \$75 per long ton (lt) of annual pellet capacity installed, or about \$16 to \$25 per annual metric ton of ore capacity. Comparing again to copper-nickel mining and processing only, the analogous capital investment ranges from \$22 to \$27 per annual metric ton of ore capacity, within the range of taconite development.

Although no operating cost data was available for the taconite industry, it is not expected to differ greatly from that developed for copper-nickel. Modeled totals range from \$6 to \$11/mt ore to produce copper and nickel metal, with 70 to 75% directly attributable to mining and processing.

Finally, water requirements in the taconite industry range from 3,000 to 5,000 gal/mt ore treated, on a once-through flow basis (one case may even approach 7,000 gal/mt ore). Copper-nickel requirements are much lower with a once-through flow requirement of 650 gal/mt ore, due to reasons discussed earlier in this

introduction, i.e. the much greater weight reduction in copper-nickel processing and the lower need of the copper-nickel concentration system, relative to taconite processing. The result is slurry densities in portions of the taconite concentrating operation which are considerably lower than those used in the concentrating of copper-nickel sulfides.

In conclusion, it is seen that for most major parameters (energy, manpower, costs), the open pit mining and processing requirements of copper-nickel and taconite are roughly comparable. This appears to be true since the significantly increased material handling needs of copper-nickel (due to a relatively high stripping ratio) are offset by the need for an agglomeration step in taconite processing which is not needed in copper-nickel processing. The notable exception is in the area of water requirements, where taconite needs may be an order of magnitude greater than those of copper-nickel, as a result of the unique requirements of the magnetic separation process.

## 3.2 PROCESSING FACILITIES

This section will develop all the aspects of the processing of copper-nickel sulfide materials necessary to prepare them for subsequent smelting and refining to pure metal products. In detail, the areas to be discussed include:

- 1) Crushing
- 2) Grinding
- 3) Flotation (concentration)
- 4) Tailing disposal
- 5) Water management
- 6) Material handling
- 7) Pollution control

### 3.2.1 Crushing

Crushing is the first stage in processing mineralized material into a saleable metal product. This is generally a dry operation in which run-of-mine ore is received and mechanically broken into smaller particles suitable for feeding the grinding equipment to be described later. The crushing operation can also be a source of dust and noise which may be released to the environment.

The general approach in the past has been to crush the ore in stages to about 3/4 in. maximum size which is suitable as feed for subsequent conventional rod mill-ball mill grinding. The trend recently has been to crush finer (to about 1/2 in.), bypass the rod mill, and feed directly to a ball mill as the only grinding stage. Apparently crushing is more energy efficient than grinding; however, some control over the grinding system may have to be sacrificed if this approach is used to reduce the operating and capital costs of grinding.

3.2.1.1 Primary Crushing--Primary crushing reduces the run-of-mine ore from a maximum size of 2 to 4 ft down to 8 to 10 in. for subsequent handling and treatment. In underground mines the primary crusher is underground and considered a mining function. In open pit operations the primary crusher is located either in the pit or, more typically, on the surface outside the ultimate pit limit, and is considered a processing function.

Underground primary crushing systems are included in the section on underground mining. Such an underground system consists of an ore dumping system, crusher, feeders, dust control facilities, conveyors, loading pockets, and skip loaders (all of which are considered mining functions and are charged as mining costs). Run-of-mine ore is generally held to a maximum size of 2 ft for an underground mine. Once the ore is raised to the surface and dumped it enters the processing phase.

In contrast, ore from an open pit mine enters the processing phase when it is dumped into the primary crusher on the surface. The primary crushing system here consists of the crusher, feeders, dust control systems, and conveyors to transport crushed ore to the coarse ore storage. Run-of-mine ore from open pit operations is generally held to a maximum size of 4 ft.

Primary crushing is generally found to be more economical if done in one large unit rather than in 2 or more smaller units. First, larger units can accept larger rock and thus primary and secondary blasting costs in the mine are reduced. Secondly, one large unit is generally less expensive in both capital and operating costs than would be the total for several smaller units.

Gyratory crushers are generally the most productive of all primary crushers and are currently the type installed, except where unusual circumstances such as

extremely wet ore dictate another selection. If a gyratory is not selected, a jaw crusher is the next best alternative.

Commercial gyratory primary crushers are available in sizes ranging up to those capable of crushing a piece of ore measuring 60 in. on one side. Such a crusher will have a 1,000 hp drive motor, weigh about 580 mt, and cost about  $\$2.5 \times 10^6$  including spare parts. Figure 5 illustrates such a crusher installation showing the flow of material through the system.

---

Figure 5

3.2.1.2 Secondary Crushing--Secondary crushing facilities are identical in nature for both open pit and underground mining operations. They consist of secondary and tertiary crushers closed-circuited with sizing screens to reduce the primary crusher product to about  $-3/4$  in. for feeding the conventional rod mill-ball mill grinding system. All necessary conveyors, storage bins, and dust collection facilities are included in this operation.

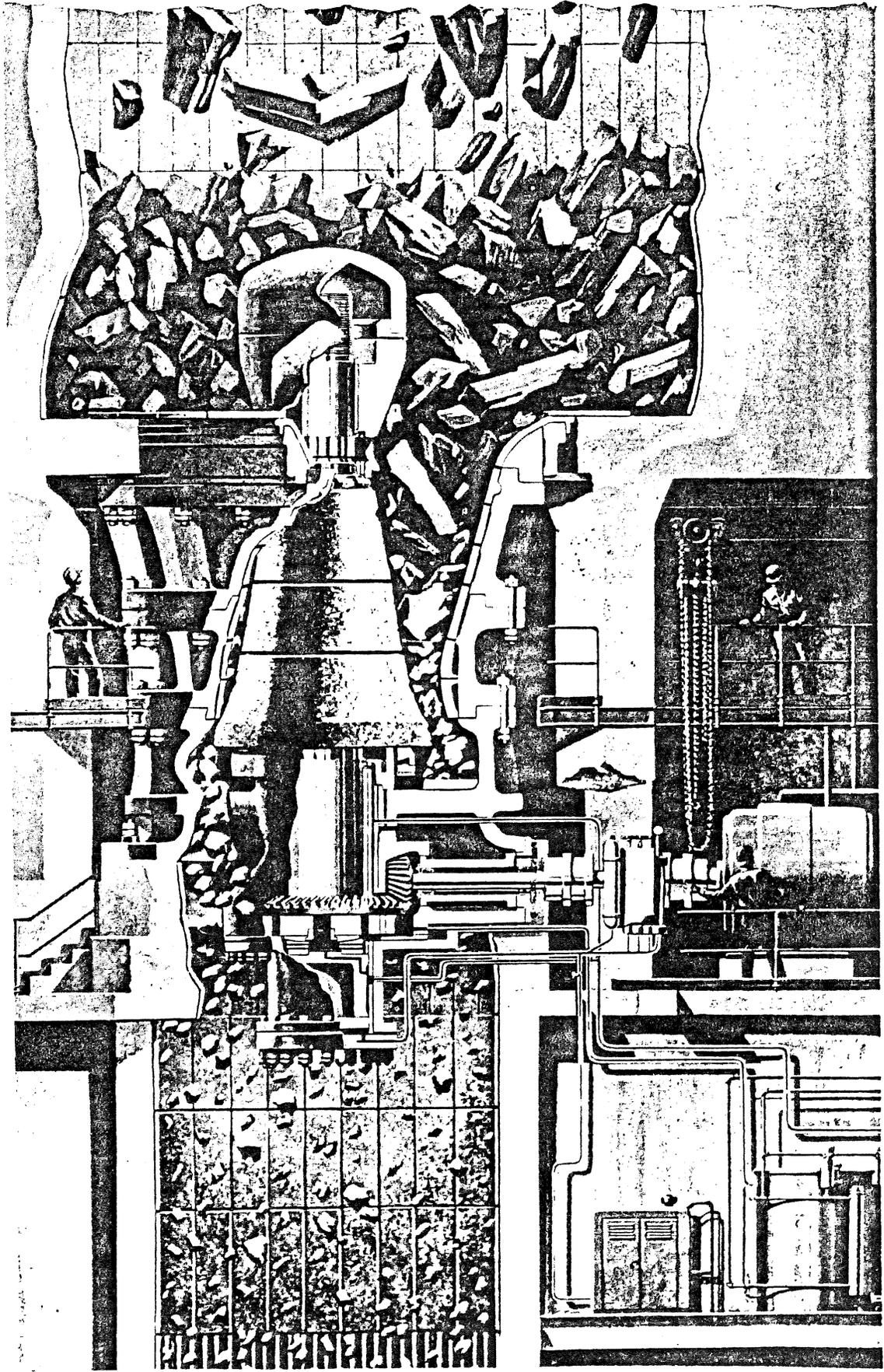
Cone crushers are generally selected for this operation as they provide relatively high productivity and minimum overcrushing of the crude ore. They can also handle wet materials and are relatively maintenance free.

Secondary cone crushers are also available in a range of sizes capable of crushing the primary crusher discharge to (eventually)  $-1/2$  in. Such crushers range in price up to \$300,000 each, weight up to 110 mt, and are driven by motors up to 400 hp. Secondary crushers are similar in design to primary crushers, with somewhat different crushing chamber configurations more suitable to size reduction of closely sized material.

FIGURE 5

# PARTIAL CUTAWAY VIEW OF A TYPICAL PRIMARY GYRATORY CRUSHER INSTALLATION

(SOURCE: METAL MINING & PROCESSING, SEPT. 1964)



Once again, if a rod mill unit is not used, secondary and tertiary crushing reduces the material to about 1/2 in. maximum size for feeding directly to the ball mill. However, 3 stages of crushing are generally used to reduce the mined ore to -5/8 in. for subsequent rod mill-ball mill grinding.

### 3.2.2 Grinding

Grinding is a process in which the crushing product is tumbled in large rotating drums, often in the presence of steel rods or balls. The purpose of the process is to further reduce the size of the ore particles to facilitate and optimize the liberation of the valuable minerals (sulfides here) from the barren waste rock or gangue minerals. This grinding procedure for copper-nickel material just prior to mineral separation requires about the same energy per unit of ore as does taconite grinding, typically 15 to 20 kwh/mt ore. Energy requirements are approximated in the laboratory by measuring the energy needed to reduce the ore from a known coarse size to a known fine size. The Bond Index is the result of one such method which determines the energy requirement to reduce the 80% passing size of the feed to a selected 80% passing size in the product. The Bond Index, or Work Index, is then calculated for the size distribution needed and a properly sized motor is selected. A grinding mill is then fitted to the motor size and processing tonnage rate desired for the system.

Two types of grinding are generally used: a conventional rod and ball mill system, or an autogenous and pebble mill system. Both may be suited to Minnesota copper-nickel material and will be discussed in some detail. The degree to which the grinding of Duluth Gabbro ore succeeds in liberating the sulfide minerals from the silicate wastes affects not only the economic viability of the operation from the viewpoint of the mining company, but also the

extent to which air and water emissions may result from unrecovered sulfur, copper, nickel, and trace metals discharged to the tailing basin.

3.2.2.1 Grinding Parameters--Estimations of power requirements and grinding mill size are based on the Bond Indices as determined by preliminary tests run on mineralized Minnesota Duluth Gabbro samples. The criteria given below are based on the use of an 80% passing value with an initial 150M grind and a final 325M grind in the regrind mills ("M" refers to Tyler mesh size, the number of openings in a square mesh screen per linear inch of screen. A 200M opening is equivalent to 74 microns.).

Later testwork (Iwasaki 1978) at the Minerals Resource Research Center (MRRC) indicated that use of a coarser grind, to a nominal size of about 65M, is beneficial for initial flotation, followed by regrinding the intermediate concentrate nominally to 270M before final separation for maximum economic recovery of copper and nickel. Still more recent testwork indicates initial grinds of 80%-100M and an energy level of 12 to 18 kwh/mt ore is sufficient for liberation of the copper-nickel minerals. If the coarser grind approach is feasible, grinding mill requirements and power consumption could decrease dramatically from that used in this writing.

Typical crushing and grinding parameters as used here for a Minnesota copper-nickel operation are as follows:

Primary crusher discharge - minus 8 in.

Secondary crusher discharge - minus 3/4 in.

Conventional grinding discharge - 80% passing 150M

Rod mill open circuit grinding to 80% passing 35M

Ball mill closed circuit grinding to 80 % passing 150M

Bond Index - 15 kwh/mt of ore

Regrind mill grinding discharge - 80% passing 325M

Assumed Bond Index - 20 kwh/mt of ore

3.2.2.2 Conventional Grinding--Typical conventional systems consist of coarse grinding in rod mills followed by fine grinding in ball mills to liberate the desired minerals. Rod mills grind by the tumbling action of steel rods, pinching and crushing the rock pieces between the rods as they roll on one another. A ball mill grinds primarily by attrition, or the wearing down of particle surfaces caused by the intimate contact of grinding balls with the ore, coupled with the tumbling and cascading action of the balls as the mill rotates.

Rod mills (Figure 6) generally have a length-to-diameter ratio of 1.3 to 1, or less for most efficient operations. A typical rod mill would be 14 X 18 ft, driven by a 1,750 hp motor, and would cost about \$800,000.

Ball mills (Figure 7) have a much more flexible configuration, ranging from length-to-diameter ratios of 1:1 up to 5:1. Typically sized units would be:

9 1/2 X 18 ft, 700 hp motor, \$400,000

11 X 20 ft, 1,000 hp motor, \$500,000

16 1/2 X 28 ft, 4,500 hp motor, \$1,600,000

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Figures 6 and 7

Both rod mills and ball mills are selected on the basis of their respective feed material sizes, discharge material size requirements, feed tonnage rates, hardness of the feed material, etc. Such criteria are used to determine the power necessary to perform the required work, and then grinding mills are selected to fit the needs.

FIGURE 6

## CUTAWAY VIEW OF TYPICAL ROD MILL

(SOURCE: ALLIS CHALMERS BULLETIN 07B 1193)

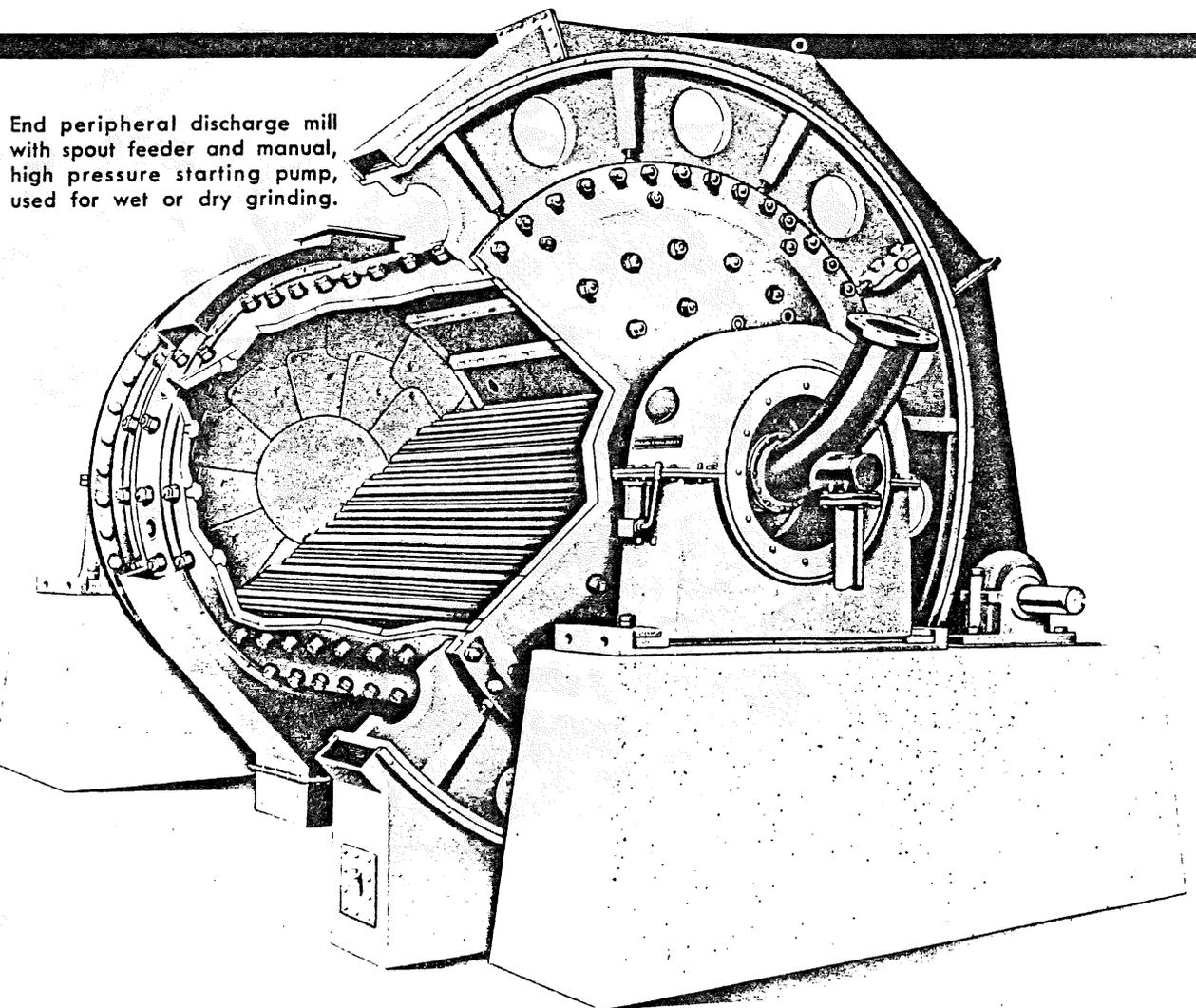


FIGURE 7

## CUTAWAY VIEW OF TYPICAL BALL MILL

(SOURCE: ALLIS CHALMERS BULLETIN 07B 1192)

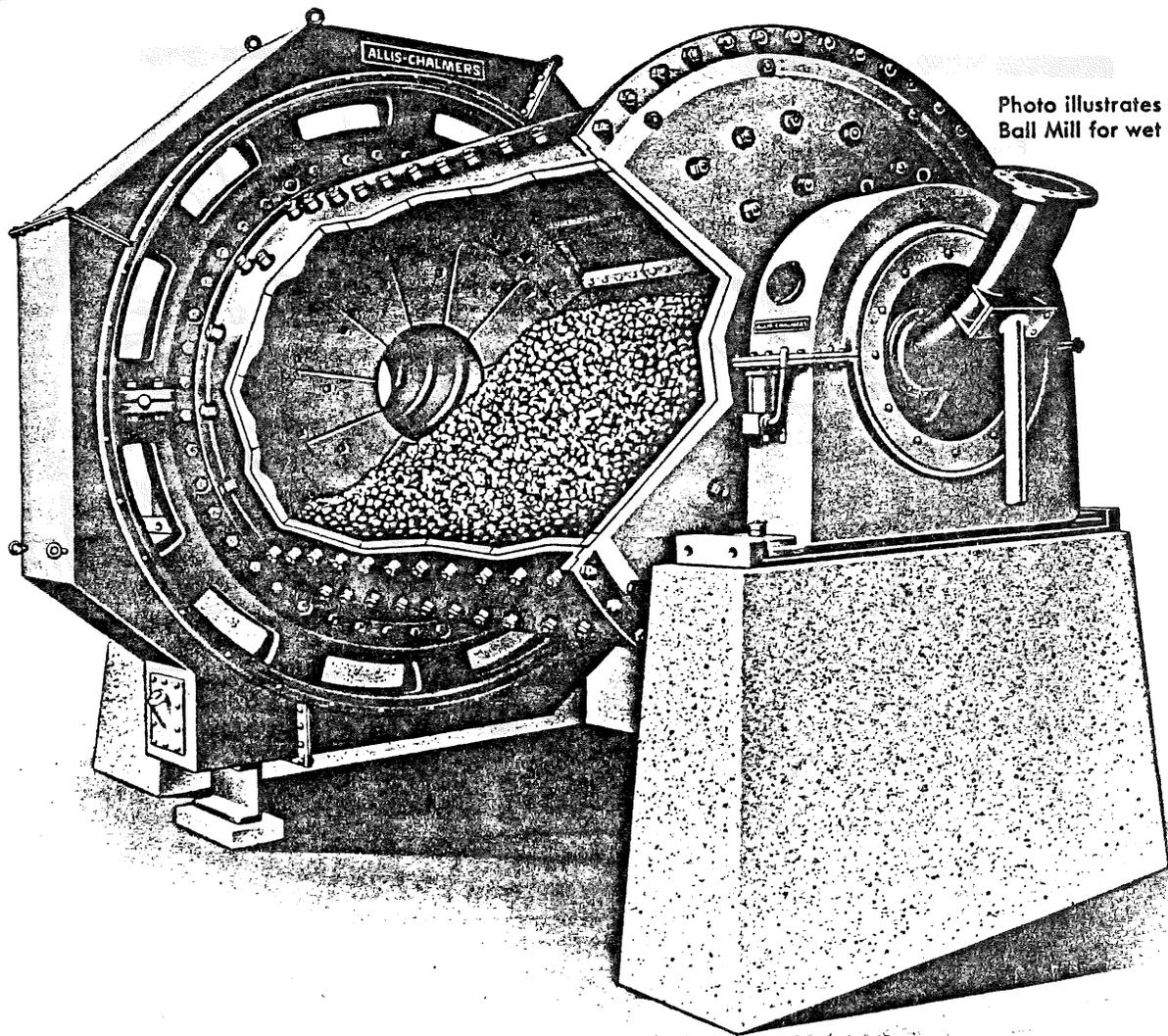


Photo illustrates overflow  
Ball Mill for wet grinding

Rod mills are generally effective in grinding from a feed size of 3/4 in. down to about 35M (420 microns). Beyond that point ball mills are more efficiently used to grind the material to the desired size of liberation. Electrical power consumption and steel wear of grinding media and mill liners increase dramatically with an increased fineness of grind. The best approach, therefore, is to liberate the minerals at as coarse a size as possible.

3.2.2.3 Autogenous Grinding--Autogenous grinding makes use of the ore characteristics to crush and grind itself to an optimum liberation size with a minimum of overgrinding. Theoretically, in autogenous grinding the ore minerals break away from the waste minerals along grain boundaries rather than across grains. Therefore, since less energy is required to separate along grain boundaries than to break grains, the valuable minerals are liberated with less energy consumption and at an overall coarser size which results in less generation of fines (particles too small to be efficiently recovered in the flotation process) than with conventional grinding. This not only saves energy, but the reduction in fines reduces stability problems and suspended sediment loading of water in the tailing basin, decreases the possible environmental impacts of accidental spills, and increases the metal recovery in the overall process.

Secondary crushing is not needed in the autogenous system as primary crusher discharge is fed directly to the mill and the coarser material acts as grinding media, also eliminating the need for rods and balls (most systems are actually semi-autogenous as a relatively small ball charge is used to level out ore variations). The large diameter of the mill allows the coarse material to be carried far up the liner wall before it finally cascades down and crushes other ore particles on impact, or crushes itself against the mill liner.

Coarse grinding is done in the autogenous mill. Ore pebbles (+2 in.) are produced in the mill, stored, and fed as grinding media to pebble mills which complete the grinding process as do the ball mills in the conventional circuit. However, ore pebbles are used in place of grinding balls to grind the finer ore particles to liberation size, consuming themselves in the process.

Typical autogenous mills (Figure 8) have length-to-diameter ratios of about 0.5:1 and range in size up to about 36 ft in diameter. Two examples are a 27 X 10 ft mill driven by a 5,000 hp motor and costing about  $\$1.8 \times 10^6$ , and a 28 X 15 ft mill driven by a 7,000 hp motor and costing about  $\$2.6 \times 10^6$ .

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Figure 8

Pebble mills used in conjunction with autogenous mills have the same characteristics as a ball mill. A typical unit would be 15½ X 31 ft in size, driven by 2,500 hp motor and costing about  $\$1.0 \times 10^6$ .

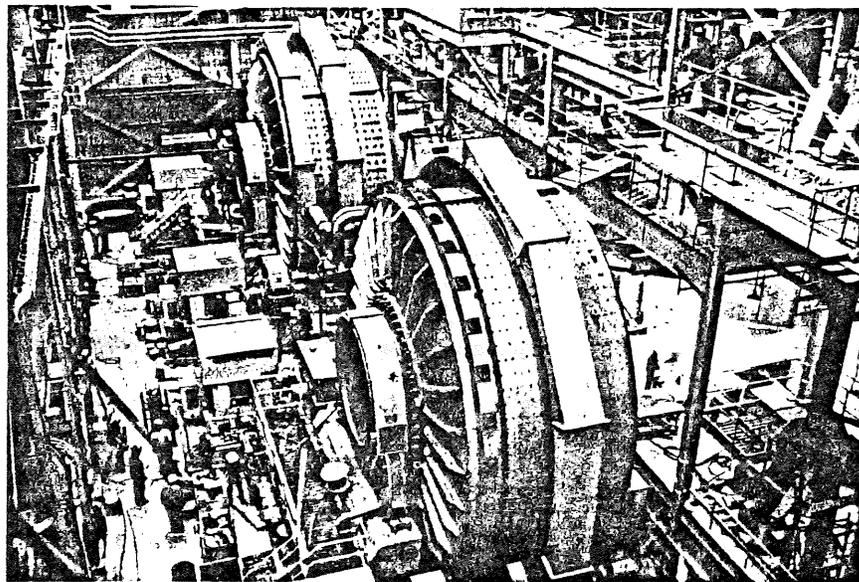
Generally, the cost of grinding mills per installed horsepower decreases with an increase in horsepower. The examples above illustrate this trend ranging from  $\$460\text{--}560/\text{hp}$  for mills with motors from 700 to 1750 hp, and  $\$360\text{--}370/\text{hp}$  for mills with 4,500–7,000 connected horsepower.

Autogenous and pebble mill grinding generally have some advantages over conventional rod and ball mill grinding. First, lower capital investment and fewer operating dollars are needed as secondary crushing is eliminated. Secondly, steel grinding media consumption is greatly reduced as rods are completely eliminated and only a small ball charge may be needed. Thirdly, with the tendency of autogenous grinding to separate the particles at grain boundaries rather than to break across grains, less power is consumed, fewer fines are produced, and

FIGURE 8

## TYPICAL AUTOGENOUS GRINDING MILL INSTALLATION

(SOURCE: MINING ENGINEERING, FEBRUARY, 1973)



flotation reagent consumption should be lower. These potential advantages however, are often offset by variations in ore characteristics which cause wide variations in grinding mill performance. The result may be less control over the system and lower overall recovery resulting in an operation that is less efficient than one using conventional grinding methods.

Typically, a successful, fully autogenous system can save 10-15% in capital costs over the conventional rod mill-ball mill circuit, with 20-25% savings in operating costs. This assumes a system in which the feed material is completely suited to either autogenous or conventional grinding methods and that an acceptable product will result from the application of either method.

Initial investigations indicated autogenous grinding would offer the above advantages of grinding Minnesota copper-nickel material. However, more recent studies indicate that conventional grinding techniques may require lower power and grinding media consumption than previously predicted, thereby greatly reducing the potential advantage of autogenous grinding systems. This report therefore will deal with conventional systems in detail, and with autogenous systems only as a secondary possibility. At this time, the issue of the applicability of autogenous grinding to gabbro material must be considered to be unresolved.

3.2.2.4 Size Classification--Figures 9 and 10 illustrate general size reduction flowsheets, including both conventional rod mill-ball mill grinding and autogenous mill-pebble mill grinding as optional size reduction facilities. Major stages are shown with the resulting products as they are separated out and directed to various crushing, grinding, and sizing stages.

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Figures 9 and 10

FIGURE 9

SIZE REDUCTION WITH CONVENTIONAL GRINDING OPTION

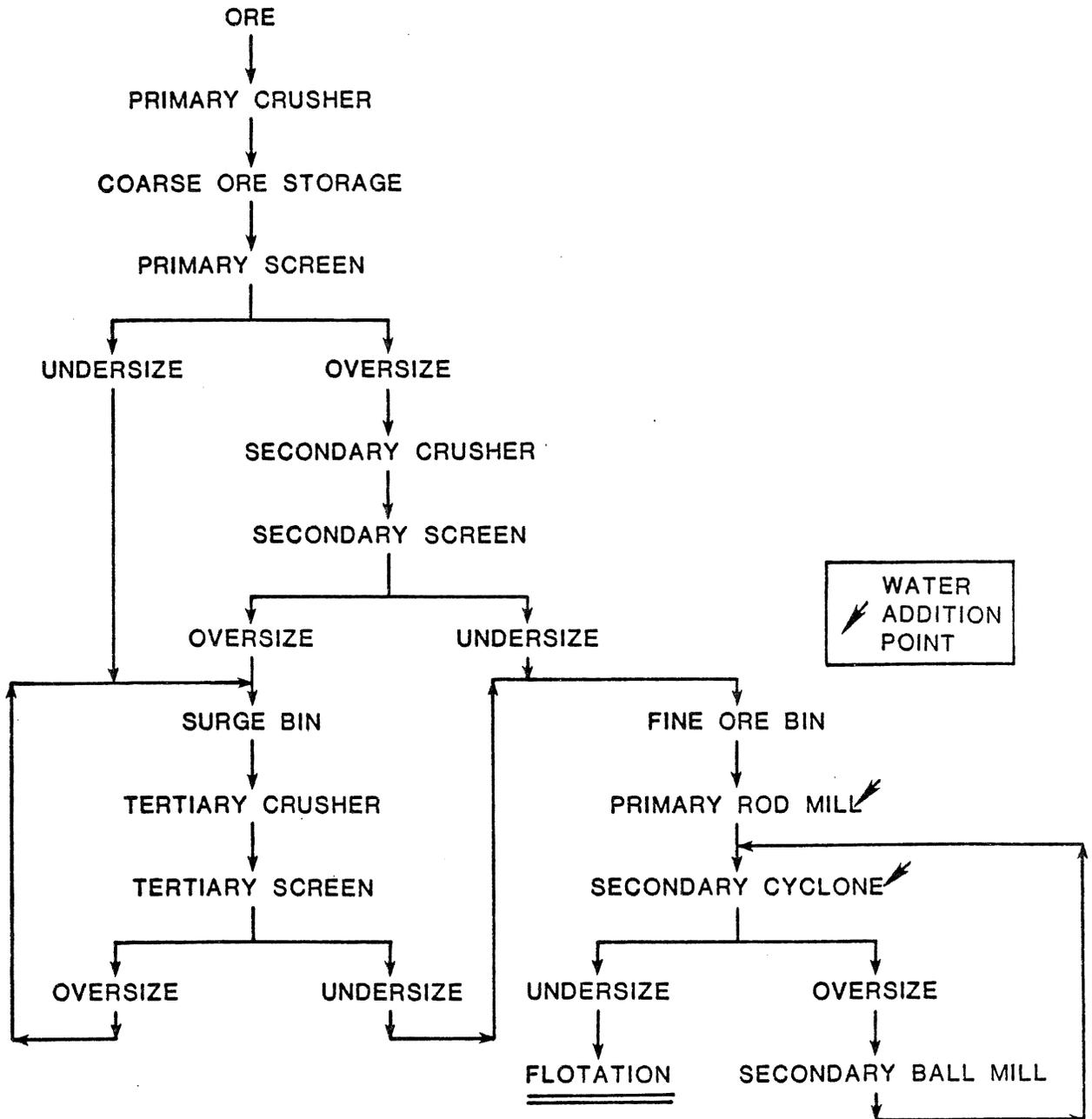
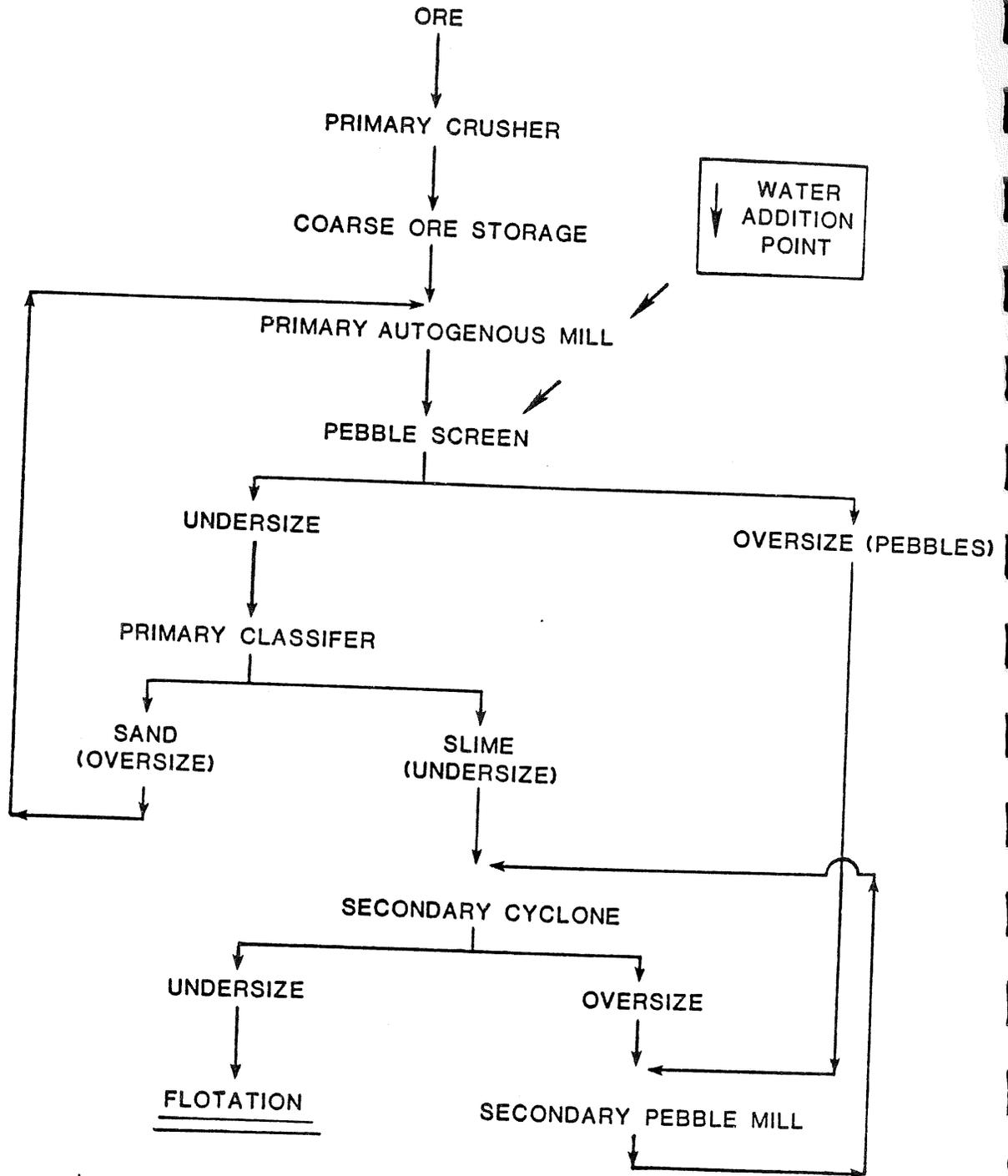


FIGURE 10 SIZE REDUCTION WITH AUTOGENOUS GRINDING OPTION



The total size reduction picture consists of crushing and grinding stages connected by sizing or classification equipment to control the size of the material passing to each succeeding stage. The objective is to reduce the ore material to a given size with the least amount of finer or undersize material generated by over-crushing or over-grinding. This objective is accomplished by vibrating screens in the crushing circuit which remove and recycle the unfinished material for additional crushing. As the particles are made fine enough, they are then passed on to the next stage for further size reduction and finally for separation into concentrate and tailing products.

In the grinding circuits, the same objective is accomplished with cyclone separators which size the particles according to their mass and recycle the oversize for additional grinding while the undersize is passed on to the succeeding system stage.

The autogenous classification system contains a pebble screen removing coarse material to serve as grinding media in a pebble mill, and a classifier which separates particles according to their mass in a pool consisting of a high density ore-water mixture. Finished material overflows the classifier pool and is passed on to the next process stage; oversize material is dewatered and recycled for further grinding.

Thickeners are also used as size classification devices; however, in the example cases they are used only to reduce the water content of the material for subsequent treatment, and to produce clarified overflow water for internal recycle through the mill water system.

Water additions are made as needed to carry the ore through the process at a solids level compatible with the treatment being applied. These addition points are indicated by arrows on the flowsheets.

### 3.2.3 Flotation

Flotation is the heart of the process separating copper-nickel minerals from the Duluth Gabbro rocks. Basically, the system involves treating a ground ore-water slurry with organic chemicals called collectors. These chemicals adhere selectively to specific mineral surfaces, including the copper- and nickel-bearing sulfides, to render them water repellent, while other minerals are not affected and remain easily wetted. The slurry is then agitated in a tank and the introduction of air as fine bubbles causes a froth to develop. The water repellent minerals adhere selectively to the air bubbles which are strengthened and stabilized by the addition of a frothing chemical. The bubbles then rise to the surface where they are skimmed off and form the concentrate. Wetted minerals remain in the pulp and are discharged as tailing, along with the residual process chemicals.

Two general types of flotation can be applied to Minnesota copper-nickel resource material: bulk and differential flotation.

3.2.3.1 Bulk Flotation--Bulk flotation recovers both the copper and nickel values in a single sulfide concentrate which would then necessitate separation of the metals in the smelting operation. It is the simplest of flotation procedures with Minnesota ore, requiring only a common xanthate collector and an alcohol frother.

3.2.3.2 Differential Flotation--Differential, or selective flotation, on the other hand, separates the metal values into a copper concentrate and a nickel-copper concentrate, each of which undergo separate pyrometallurgical treatment to recover the valuable metals. Normally, one metal sulfide is depressed with a reagent combination while the other is floated. Then, with appropriate adjust-

ments to the pulp chemistry, the remaining sulfide is activated and floated to produce the second concentrate. Preliminary laboratory and pilot plant investigations indicate differential flotation may be preferable to bulk flotation (see the following section). It is more difficult than bulk flotation and requires extra equipment and chemical reagents, but if successful it provides a more desirable product for subsequent smelter treatment. The environmental implications include the potential reduction of smelter air pollution at the expense of a possible increase in water quality problems in the processing water system.

### 3.2.3.3 Differential and Selective Flotation vs. Bulk Flotation Considerations--

Early mineral processing testwork was directed toward the production of a bulk copper-nickel concentrate, containing reasonable recoveries of both metals, and assuming the separation of metals could be made in the pyrometallurgical stage. However, with the grades of metals recovered in the concentrate (10-15% Cu and 2-3% Ni), it became apparent that conventional smelter/refinery techniques would have a difficult time separating the concentrate into the metals and a slag. The overall grade of the concentrate and the ratio of copper to nickel precluded the application of proven smelter technology. If a bulk concentrate were the product, new pyrometallurgical facilities would have to be built as no available smelter and refinery complex could handle the resultant feed material.

For this reason, the emphasis was shifted to the production of 2 concentrates: one high in copper and low in nickel, and the other containing the remaining metal values in comparable amounts of both. Such products would hopefully be compatible with existing smelter technology and thus simplify the need for extensive research in that area. They would also offer the possibility of off-site treatment in existing facilities.

Two flotation approaches are available to separate the copper and nickel metal sulfides:

1) Differential flotation in which a bulk sulfide concentrate is first made to recover the majority of the valuable metals and produce a final tailing. In the next flotation step nickel sulfide is depressed in the bulk concentrate and a high-copper concentrate is recovered. The "tailing" from the high-copper flotation step becomes the nickel-copper concentrate.

2) Selective flotation involves depressing the nickel sulfide in the entire ore and then recovering a high-copper concentrate. The nickel sulfides are then reactivated and floated as a nickel-copper concentrate with a final tailing being the remaining material.

In both cases, the copper sulfide step must remove a maximum amount of copper to leave approximately equal amounts of both metals to be recovered in the copper-nickel concentrate. If too little copper is removed in the first concentrate, the remaining copper predominates, nickel recovery falls off, and the proportion of copper to nickel becomes distorted towards copper in the nickel-copper product.

Based on information available to date, the two-product flowsheet appears to be the best way to treat Minnesota copper-nickel material as separation of the bulk of the copper from the nickel is more easily done in the flotation system than in the smelter. The first product should be high in copper (15-25%) and low in nickel (less than 1%) to be treatable with existing copper smelter technology. The second product should contain comparable amounts of both metals (5-8%) in order to be treatable by existing smelter technology.

From the environmental point of view, differential flotation may have the advantage of reduced reagent consumption, particularly in the depressant needed for the nickel sulfide. In selective flotation the nickel depressant is added to the entire ground ore (pulp) and then remains with the tailing. However, with differential flotation the depressant is added to the bulk sulfide concentrate which is only a fraction, typically 5% or less, of the total ore, thus reducing greatly the amount of reagent needed and providing a much smaller pulp volume which need be controlled if the depressant is undesirable in the tailing water.

MRRRC tested the differential flowsheet on the material described in Table 1. Several nickel depressants were used, such as combinations of lime and cyanide, and lime and dextrine (starch). The lime-dextrine mixture is environmentally less objectionable than the lime-cyanide combination, and is in use in Canada and Finland.

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Table 1

The flowsheets shown in Figures 11 and 12 are typical of the differential and the selective methods used to produce the desired concentrates. The range of products resulting from both selective and differential flotation are similar and together should total about the same as the bulk flotation product, in terms of overall copper and nickel content. The difference, of course, is in the distribution of the metals and the amenability to subsequent pyrometallurgical treatment. Table 2 lists the range of expected values from both systems plus a comparison with the bulk flotation concentrate.

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Figures 11 & 12, Table 2

Table 1. MRRC differential flotation ore sample analysis  
(AMAX lot 2 ore).

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0.79% Cu	42.84% SiO <sub>2</sub>
0.179% Ni	19.83% Al <sub>2</sub> O <sub>3</sub>
0.025% Co	4.89% CaO
12.90% Fe	9.21% MgO
1.49% S	1.62% TiO <sub>2</sub>
0.19% C	

FIGURE 11

# GENERALIZED DIFFERENTIAL FLOTATION CU-NI SULFIDE FLOWSHEET

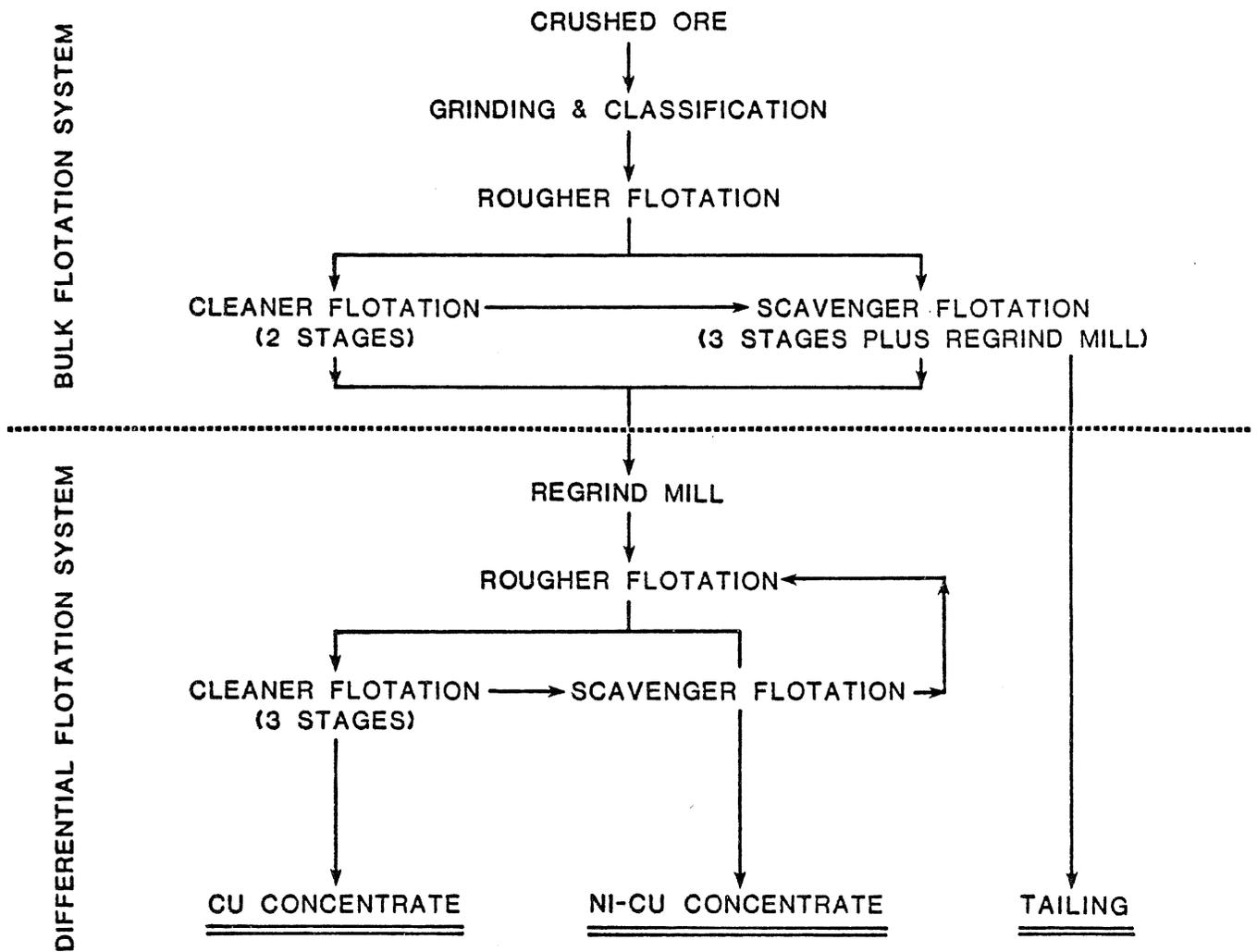


FIGURE 12

GENERALIZED SELECTIVE FLOTATION  
CU-NI SULFIDE FLOWSHEET

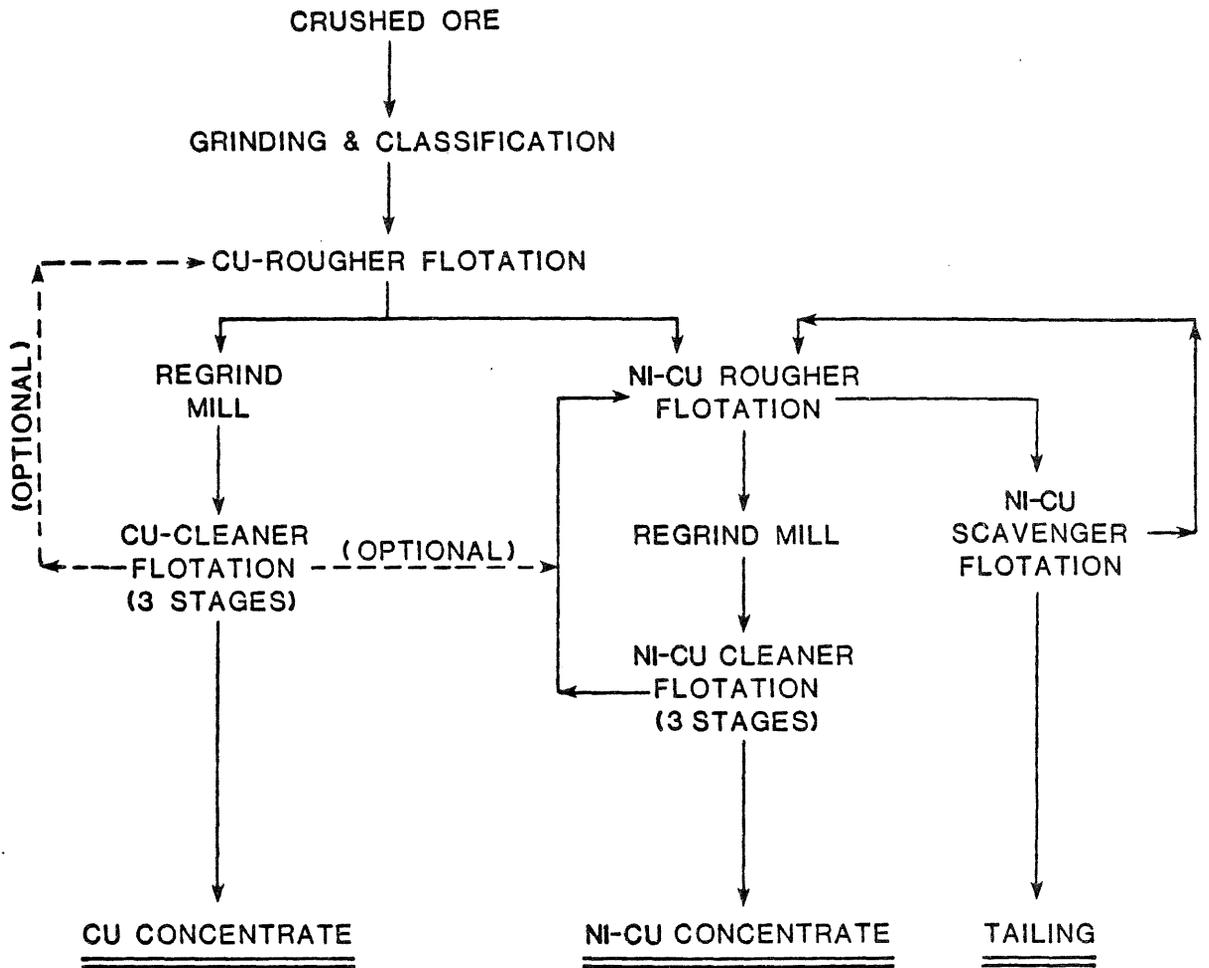


Table 2. Comparison of selective or differential flotation results to bulk flotation results, Minnesota Cu-Ni material.

ANALYSIS	BULK FLOTATION CONCENTRATE	CU CONCENTRATE	NI-CU CONCENTRATE	COMBINED CONCENTRATE	FINAL TAILING
% Cu	10-22	11-24	3-8	---	0.04-0.13
% Cu recovery	85-95	60-88	6-20	84-97	3-16
% Ni	2-3	0.2-0.8	2-10	---	0.04-0.15
% Ni recovery	65-85	4-19	55-72	60-78	22-40
% Co	0.1-0.2	0.04-0.05	0.1-0.4	---	---
% Fe	21-42	36-42	18-30	---	---
% S	19-33	32-34	9-20	---	---
% SiO <sub>2</sub>	3-23	---	---	---	---
% Al <sub>2</sub> O <sub>3</sub>	1-7	---	---	---	---
% CaO	1-3	---	---	---	---

In comparison to bulk flotation, the water system in either differential or selective flotation systems is expected to be more complex due to the reagents necessary for nickel depression and subsequent activation. However, no additional environmental concerns are anticipated at this point. For example, reagent additions (lime) in the differential system can raise the process water pH from the neutral range of 6 to 8 to about 12 to effect nickel depression. This water would then have to be acidified to near-neutral and settled in a basin before reuse or discharge.

One environmental concern is over the high pH water described above and what might happen if this water were accidentally discharged to the environment. First, since this water is an integral part of the process system and would normally pass through a neutralizing step before being discharged to the tailing system, the principal accidental spillage possibility is in the case of total plant failure which then requires emergency dumping of the contained pulps to prevent plugging of the system. Such dumping would be to collection sumps in the lower level of the plant or to tailing thickeners, both of which could contain the water for neutralization before it would be discharged. If for some reason the high pH water did get into the tailing disposal system and then reach the basin, it would be so diluted by the large volume of water in the pond that there should be no noticeable change in the overall basin discharge water.

Testwork with recycle water in the selective flotation system in which  $\text{SO}_2$  as  $\text{H}_2\text{SO}_3$  was used as a nickel sulfide depressant, did allow an overall reduction in reagent addition levels compared to once-through use of water. No further data is available on reagent additions or water quality from this system approach, although no additional problems have been reported or are currently anticipated.

Reagent addition levels used by MRRC are summarized in Table 3 for the differential flotation system.

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Table 3

3.2.3.4 Chemical Reagents--Testwork to date on bulk flotation procedures at the MRRC indicate reagent addition rates of less than 0.1 lb/st of ore for both collector and frother are sufficient for maximum mineral recovery. However, since additional chemicals may be necessary for pH control and flocculation, and differential flotation procedures would require more extensive reagent systems than simple bulk flotation, consideration must be given to more complex reagent systems.

The following discussion on flotation reagents has been extracted from a review of an original paper by Dr. R.D. Crozier, appearing in the April, 1978, issue of Mining Magazine.

Collectors are organic chemicals used to induce the surfaces of sulfide minerals to become water repellent, thereby promoting their selective attachment to air bubbles for transportation to the surface of the flotation cell for removal as a concentrated product. Frothers are chemical reagents which aid in the formation and stabilization of the froth to permit mechanical removal of the sulfide particles in the froth.

Many types of collectors are available, but xanthates are the dominant type accounting for almost 60% of world consumption. Over the past 15 years collector consumption has been relatively constant at 0.08 to 0.09 lb/mt of ore. Since reagent costs have increased as much as 100% in that time period, and ore grades have declined, collector reagent costs per metric ton of metal produced

Table 3. MRRC differential flotation reagent schedule.

REAGENT	ADDITION RATE, lb/mt	PURPOSE
Sodium Aerofloat <sup>1</sup>	0.04-1.66	collector
Z - 200 <sup>2</sup>	0.004	collector
MIBC <sup>3</sup>	0.11	frother
Z - 11 <sup>4</sup>	0.06	collector
CaO (lime slurry)	22.1	depressant
Dextrine <sup>5</sup>	2.2-2.8	depressant

<sup>1</sup>Sodium diethyl dithiophosphate (American Cyanamid Co.)

<sup>2</sup>Isopropyl Thionocarbamate (Dow Chemical Co.)

<sup>3</sup>Methyl isobutyl carbanol (Shell Chemical Co.)

<sup>4</sup>Sodium Isopropyl xanthate (Dow Chemical Co.)

<sup>5</sup>Stadex 140 (Corn Products Co.)

are significantly higher. A comparison of collector consumptions around the world indicates use levels ranging from 0.03 to 0.14 lb/mt ore treated.

The trend of frother usage has changed over recent years from the natural types such as pine oil and cresylic acids, to the synthetic frothers such as polyglycol ethers and alcohol type reagents. Synthetic frothers offer more consistency, reduced collector properties, and some specialty-type properties such as the ability to dissolve non-water soluble collectors--all of which are advantages over natural types of frothers. Latest available data for frother consumption indicates frother usage averages 0.08 lb/mt ore.

Table 4 provides some insight into the economic aspects of flotation chemicals. This 1976 cost data for U.S. production and consumption illustrates the rather dramatic price increases over the past 15 years.

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Table 4

Table 5 lists some of the available xanthate reagents, the most commonly used of all collectors. ( A detailed description of xanthates can be found in S.R. Rao's book, "Xanthates and Related Compounds," Marcel Dekker, Inc., New York, 1971). Generally, they are formed by reacting caustic soda with alcohol and carbon disulphide. Sodium and potassium metal can be substituted for the caustic. Xanthates are readily soluble in water and must be used in alkaline circuits due to their instability in acid environments.

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Table 5

A few commonly available frothers are listed in Table 6. Since frothers are surface active compounds, they are dramatically affected by the recycle water

Table 4. Approximate costs of selected flotation reagents  
 U.S. cents/lb fob U.S. factory (1977).

	1960	1976
<b>Frothers</b>		
MIBC	18.5	35
Pine Oil	14.8	14.8
Dow Froth 250	26.7	50
Cresylic Acid	5.6	40
<b>Collectors</b>		
Xanthate Z-6	33.15	55
Z-11	26.7	49
Sodium Aerofloat	26.7	42
R238	26.7	45
Minerec A	46.0	83
<b>Misc. Chemicals</b>		
Soda Ash	2.75	3.55
Copper Sulphate (hydrate)	12.92	37.0
Sodium Cyanide	21.36	34.0
Zinc Sulphate (monohydrate)	12.54	24.0

Table 5. Commonly available xanthate collectors.

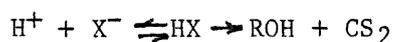
CHEMICAL FORM	TRADE NAMES	
	Dow Chemical	American Cyanamid
K Ethyl	Z-3	R.303
Na Ethyl	Z-4	R.325
K Isopropyl	Z-9	R.322
Na Isopropyl	Z-11	R.343
K Butyl	Z-7	-
Na Isobutyl	Z-14	R.317
K sec Butyl	Z-8	-
Na sec Butyl	Z-12	R.301
K Amyl	Z-6	R.350
K sec Amyl	Z-5	-
K Hexyl	Z-10	-

system. Extensive large-scale testwork is absolutely necessary before final decisions can be made concerning the type and amount of frother to use due to this sensitivity. Additionally, collectors and frothers tend to interact and the combination selected may depend on this interaction to effect an efficient separation in the mill.

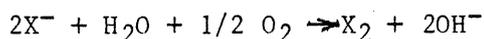
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Table 6

Decomposition of Flotation Reagents--The final report of the University of Minnesota's MRRRC on Minnesota Cu-Ni mineral processing studies (Iwasaki 1978), conducted as part of the work of the Regional Copper-Nickel Study, discusses the decomposition of flotation reagents. Stability (or decomposition) is important not only from the standpoint of the residual build up effect on the flotation system balance, but also from the standpoint of degradation in tailing basins and the potential for release to the environment through seepage and discharge. (The toxicity of flotation reagents to aquatic life is discussed in Volume 4-Chapter 1 of this report.) Xanthates are known to be unstable in aqueous solutions and below pH 8 the rate of their decomposition depends strongly on pH, according to:



In the alkaline range up to pH 12 the rate is known to be virtually independent of pH and the decomposition reaction considered to be most probable is the oxidation of xanthate to dixanthogen:

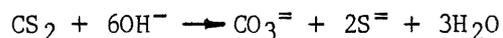


The decomposition rate of xanthates increases rapidly as the solution pH is reduced from neutral, but remains constant as the pH is increased from neutral.

Table 6. Commonly available frothers.

<u>FROTHER TYPE</u>	<u>COMMERCIAL NAME</u>
Alcohols	Cyanamid R.70 MIBC
Aromatic Alcohols	Coal tars-many available
Pine Oil	Hercules "Yarmor"
Polyglycol Ethers	Cyanamid R.65 Dowfroth 250

The eventual fate of the above products of decomposition, particularly in neutral and alkaline ranges, is of interest as this is the pH range of copper-nickel flotation. A mechanism involving the formation of monothiocarbonate as an intermediate product in the decomposition of xanthate in an alkaline medium has been suggested. The thiocarbonate may then split into sulfide and carbon disulfide, which in turn would hydrolyze into carbonate and sulfide, according to:



Two series of tests at 2 addition rates were conducted at the MRRC to determine the decay rate of typical solutions of KAX (potassium amyl xanthate) isolated from air, exposed to air (sitting), and aerated. Over a period of 8 days, KAX concentrations decreased as much as 80%. Although the pH ranged from 7.5 to 8.6, the collector was unstable and those solutions exposed to air or aerated decomposed most rapidly, implying that oxygen plays an important role in the decomposition. Figure 13 indicates the results of the decomposition tests for the 2 different addition rates in the case of the beakers sitting exposed to the air. It is interesting that the decay rates are generally comparable in the 2 cases, with half-lives on the order of 3 to 4 days.

Figure 13

Methyl isobutyl carbinol (MIBC), having the following structural formula,

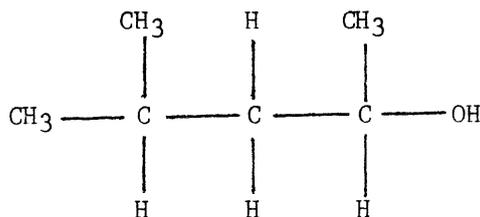
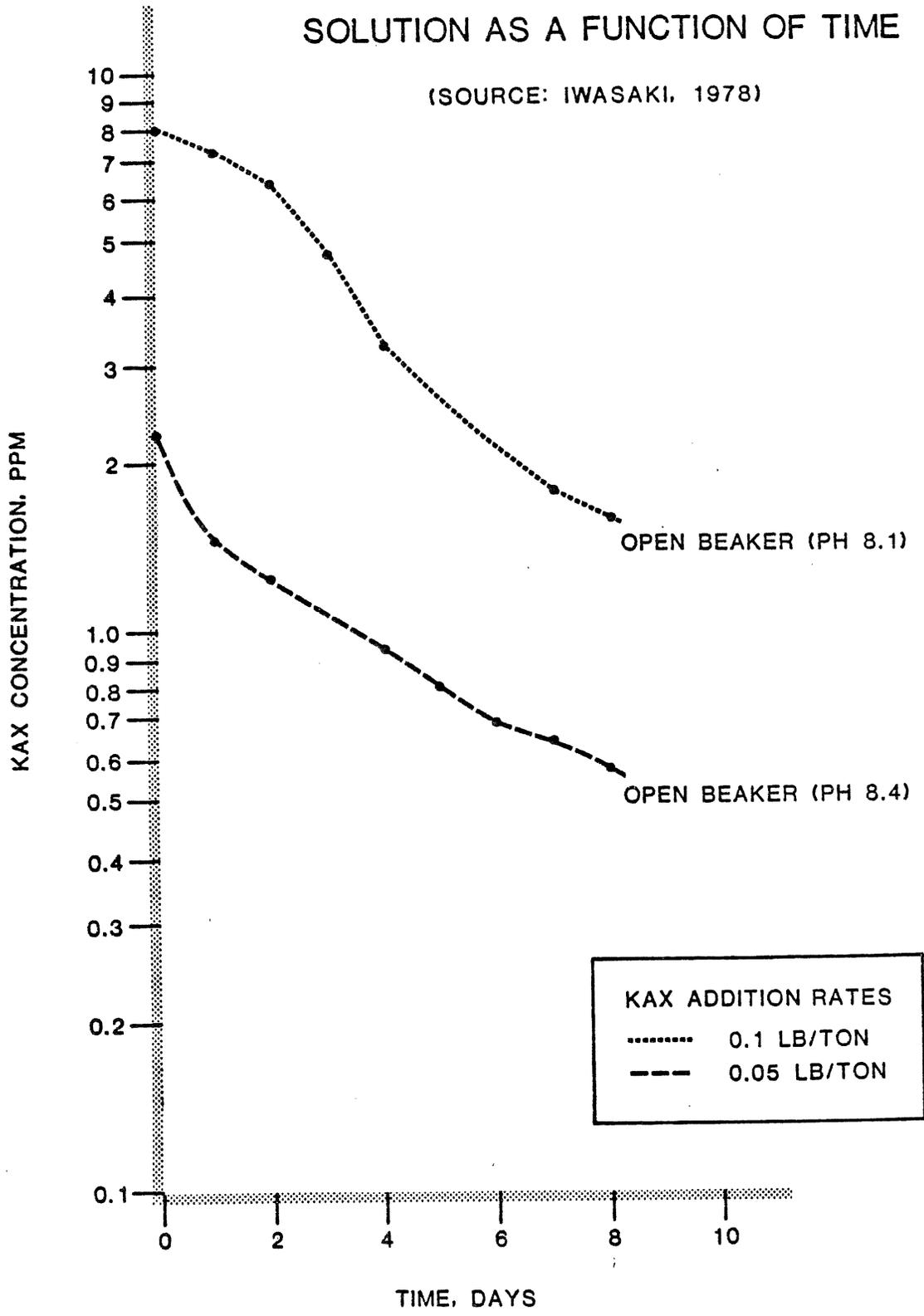


FIGURE 13

DECOMPOSITION OF KAX IN FLOTATION PULP SOLUTION AS A FUNCTION OF TIME

(SOURCE: IWASAKI, 1978)



was used in the investigations and is typical of the type of frother which might be used in a bulk flotation process.

Analytical techniques developed at the University showed that MIBC decomposition could be minimized by keeping the solutions in sealed containers. However, MIBC can decompose measurably in air-exposed tailing slurries within a week, and almost completely in a one-month period. The observed half-lives in bench scale residual tailing water samples typically ranged from one to 5 days.

In bench scale flotation tests conducted at MRRC on mineralized samples from the Duluth Complex the feed slurry pH after conditioning averaged 8.8. Following the flotation process, the residual concentration of potassium amyl xanthate (KAX) in the tailing slurry solutions ranged from 1 to 1.5 ppm. At a constant addition level of 0.05 lb/st ore, the KAX concentration without adsorption is estimated to be 11.5 ppm, assuming that the purity of the collector used was 69%. A residual concentration of 1 to 1.5 ppm would then correspond to 90% abstraction. The term "abstraction" denotes that the consumption of the collector includes not only adsorption by minerals and removal with the concentrate, but also precipitation by heavy-metal ions in solution and decomposition. It was noted in early pilot plant runs that the residual KAX concentrations had to be at least 1 to 2 ppm to insure optimum recoveries of sulfide minerals. The residual KAX in tailing slurries decomposed within a week; however, the exact path of the decomposition of xanthate has not been established, but it is presumed to decompose eventually into  $H_2O$ ,  $CO_2$ , and  $SO_4^{=}$ . More research is needed in this area to clearly delineate the potential water quality impacts which may occur if collector decomposition products should be released to the environment.

The residual concentration of methyl isobutyl carbinol (MIBC) in tailing slurry solutions centered around 7 ppm. The concentration of MIBC without adsorption

is estimated to be 16.7 ppm at an addition rate of 0.05 lb/st ore. Since the residual concentration of 7 ppm included the removal in the froth and the addition of dilution water during flotation, some dilution did occur.

It appeared that there was very little abstraction of the MIBC by the ground ore. The equilibrium concentrations prior to flotation were in the range of 15 to 20 ppm and the residual MIBC in the tailing slurries decomposed within a week.

Final bulk flotation tests performed by the MRRC on Minnesota copper-nickel ore resulted in a somewhat different distribution of reagents. The testwork was continuous with an ore feed rate of 1000 lb/hr (0.45 mtp). Reagent addition levels were:

KAX collector 0.276 lb/mt

MIBC frother 0.083 lb/mt

MG500 flocculant 0.011 lb/mt

The collector and frother were added to the flotation system and the flocculant was used to remove fine tailing material from the recycle water.

The system pH level ranged from 8.4 to 8.9 and total water useage averaged 618 gal/mt of ore. Recycle water averaged 581 gal/mt ore, or 94% of the total system needs.

Residual reagent levels in the recycle water (equivalent to the water transported to the tailing basin in a full-scale operation) averaged:

1.7 ppm KAX

2.3 ppm MIBC

0 ppm MG500

When compared to the addition levels, the residual reagent levels indicate extraction rates by the solid material greater than predicted from bench-scale testwork with lower initial levels. Almost 97% of the KAX was removed by the concentrate material, 87% of the MIBC was extracted, and all of the MG500 was removed with the flocculated fine tailing. Part of the observed extraction rates may actually be due to reagent decomposition prior to recycling.

The literature shows MIBC toxicity levels of 100 to 1000 ppm and xanthate toxicity levels of 0.01 to 0.1 ppm. Sodium ethyl and sodium isopropyl xanthates were used, but similar results would be expected with KAX. Tests specifically run for the Study using aquatic invertebrates (Daphnia pulicaria) and fathead minnows with sodium isopropyl xanthate showed toxicities of 22 and 38 ppm, respectively, considerably less toxic than reported in the literature. See Volume 4-Chapter 1 for details and further discussion of the above findings.

Based on the MRRC information, the literature, and Study bioassay test results, discussed in the above reference, expected levels of MIBC in the tailing water are significantly below toxic levels. Anticipated KAX levels are questionable depending on the guidelines accepted, and could pose a problem. The flocculant appears to be no problem at all.

The trace element analyses of the tailing slurry solutions showed very few unusual elements upon aging. The slurry pulp, or pH showed a tendency to decrease from near 9 during flotation to about 8 after a month. The concentration of copper in solution remained near 10 ppb throughout the period. The concentrations of nickel in solution were essentially below the limit of detection by the analytical method used (10 ppb). All available data on solid tailing samples averaged 0.013 to 0.004% Zn, and the zinc ion concentrations

were seen to increase in many tailing water samples to a few tenths of a part per million in a month. In some of the pulp solutions arsenic was reported to be present in the range of 0.2 to 0.4 ppm. In the pulp solutions of virtually all the samples the presence of chromium in the range of 0.006 to 0.1 ppm, of vanadium in the range of 0.001 to 0.03 ppm, and of molybdenum in the range of 0.03 to 1 ppm was noted. The significance of such observations should be carefully evaluated with further testing.

Trace element analyses of concentrate and tailing water samples taken during a pilot plant test at MRRC on Duluth Gabbro Complex material indicated similarities with the results of earlier bench tests. The lowering of the pulp pH through sulfuric acid addition had the effect of raising the concentrations of various ions.

3.2.3.5 Design--Flotation schemes (Figure 14) are designed in the laboratory, refined in pilot plant testwork, and improved to maximum efficiency during the years of commercial operation which follow in a successful venture. A flotation system is dynamic in nature, changing as new innovations are devised and tested, as the ore character changes, as costs change, and as the goals of the operating company are adjusted to better meet the demands of current and future market situations.

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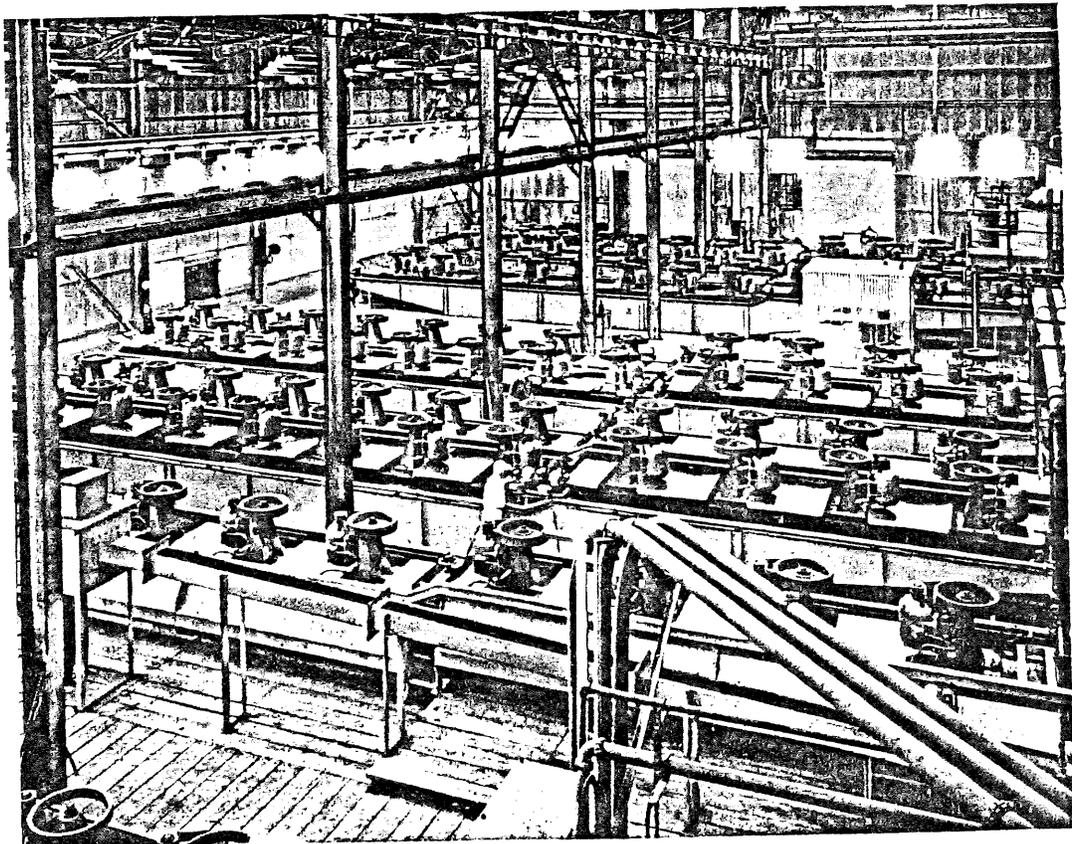
Figure 14

Generally a flotation system consists of well-defined, sequential steps which depend entirely on the liberation characteristics of the ore and on the separation characteristics of the elements of the system. The ground ore is first conditioned with chemical reagents to adjust the pH and prepare the

FIGURE 14

## TYPICAL FLOTATION PLANT INSTALLATION

(SOURCE: MINING ENGINEERING, FEBRUARY, 1972)



mineral surfaces for collection or rejection by the subsequent flotation stages. Flotation then follows in order: rougher, cleaner, recleaner; with scavenger stages applied to the tailing produced by the previous stages to remove the last of the economically recoverable sulfides. Intermediate concentrates or middling products may be reground in ball mills and sized with cyclones to further liberate the valuable minerals for more efficient separation in the following stages.

Thickeners can be used to increase the solids content of any slurry product within the system, or to recover water for direct recycle to the system. Such thickening is usually necessary when a product becomes too dilute for the next processing stage.

Final products are typically a concentrate and a tailing. In the case of a fully integrated, on-site Minnesota copper-nickel operation, concentrate would be pumped to a smelter and tailing would be pumped to the tailing basin.

Flotation units installed in modern plants range up to 500-600 ft<sup>3</sup> in capacity and larger units in excess of 1,000 ft<sup>3</sup> capacity have been designed. The size and number of flotation units generally decrease as tailing material is rejected in the system, to accommodate the decrease in amount of material to be treated.

Cell capacities vary with the particular stage of the system and the retention time necessary in that stage to effect a good separation. Generally the cell capacities range from 65 mtpd for 500 ft<sup>3</sup> cells down to 2 mtpd for 50 ft<sup>3</sup> cells. Typical cell sizes and costs are as follows:

Rougher Flotation, 500 ft<sup>3</sup> cell, \$19,000 each

Cleaner Flotation, 180 ft<sup>3</sup> cell, \$11,000 each

Cleaner Flotation, 100 ft<sup>3</sup> cell, \$ 6,500 each

Cleaner Flotation, 50 ft<sup>3</sup> cell, \$ 3,500 each

In designing a flotation scheme for a commercial plant, particular attention must be paid to the contact time required for all mineral surfaces to be fully affected by the reagents (conditioning time) and to the retention or flotation time required for an efficient separation of valuable minerals in the flotation cell. The conditioning is generally done in the grinding circuit where contact between minerals and chemicals is intense, or in a conditioner just ahead of the flotation where violent agitation results in complete contact. The time necessary to perform this function can be estimated in the laboratory and sufficient time insured in the commercial operation by sizing the equipment properly.

Sizing a flotation unit follows much the same procedure: the flowsheet is designed, the necessary retention times for efficient recovery are measured, and the product weights and pulp densities (percent solids in a pulp stream) are measured. Pulp volumes per unit time are then factored by the effective volumes of flotation cells to determine the number of cells necessary in each flotation stage to give the proper length of contact time and overall plant capacity.

The overall design is based on the number of grinding circuits necessary to grind the required amount of ore. The total number of rougher cells necessary is divided evenly among the grinding circuits and fractions are rounded off (usually upward to build in 10 to 15% excess capacity). Similar distribution is made with the remaining equipment and lines are combined as the amount of material to be treated decreases. In the case of Minnesota copper-nickel, more than 90% of the material will be discarded as tailing so that a considerable equipment reduction is effected as the tailing material is removed from the circuit.

### 3.2.4 Tailing Disposal Systems

3.2.4.1 Introduction--The purpose of a tailing disposal system is to contain the waste material from the processing plant in a manner which is economical and in compliance with environmental, safety, and reclamation requirements. A second equally important purpose is to provide a means by which the large quantity of water needed to carry the tailing to the basin can be clarified and recycled to the processing system for reuse. Since there is no revenue generated from the operation of tailing basins, the cheapest workable method is generally the one used, if it is acceptable on the basis of safety and environmental considerations.

The Minnesota copper-nickel resource is low in metal grade and therefore large quantities of solid waste or tailing will be generated in the processing plant. For every metric ton of ore processed typically, 0.95-0.96 mt of tailing will be produced and must be properly disposed. Because of these large quantities of tailing produced from potential copper-nickel mines (Table 7), tailing basins will be a major land consuming feature of this potential new industry. If the entire  $4 \times 10^9$  mt of copper-nickel resource were eventually exploited, the models used here indicate that up to  $50 \text{ mi}^2$  of land in the Study Area would be covered with tailing material if basins have an average depth of 70 ft.

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#### Table 7

Figure 15 illustrates a generalized tailing system, and Table 8 briefly describes component parts of the system, all of which will be detailed in this section of the report.

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#### Figure 15, Table 8

Table 7. Tailing production from potential Minnesota copper-nickel mines.

OPERATION SIZE 10 <sup>6</sup> mtpy ore	TAILING PRODUCTION	
	10 <sup>6</sup> mtpy	10 <sup>6</sup> mt total <sup>a</sup>
5.35	5.08	117
11.33	10.97	274
12.35	11.72	269
16.68	16.04	391
20.00	19.36	484

<sup>a</sup>Assumes a 30 yr overall life, including development and shutdown periods.

FIGURE 15 GENERALIZED SCHEMATIC OF A TYPICAL DISPOSAL SYSTEM

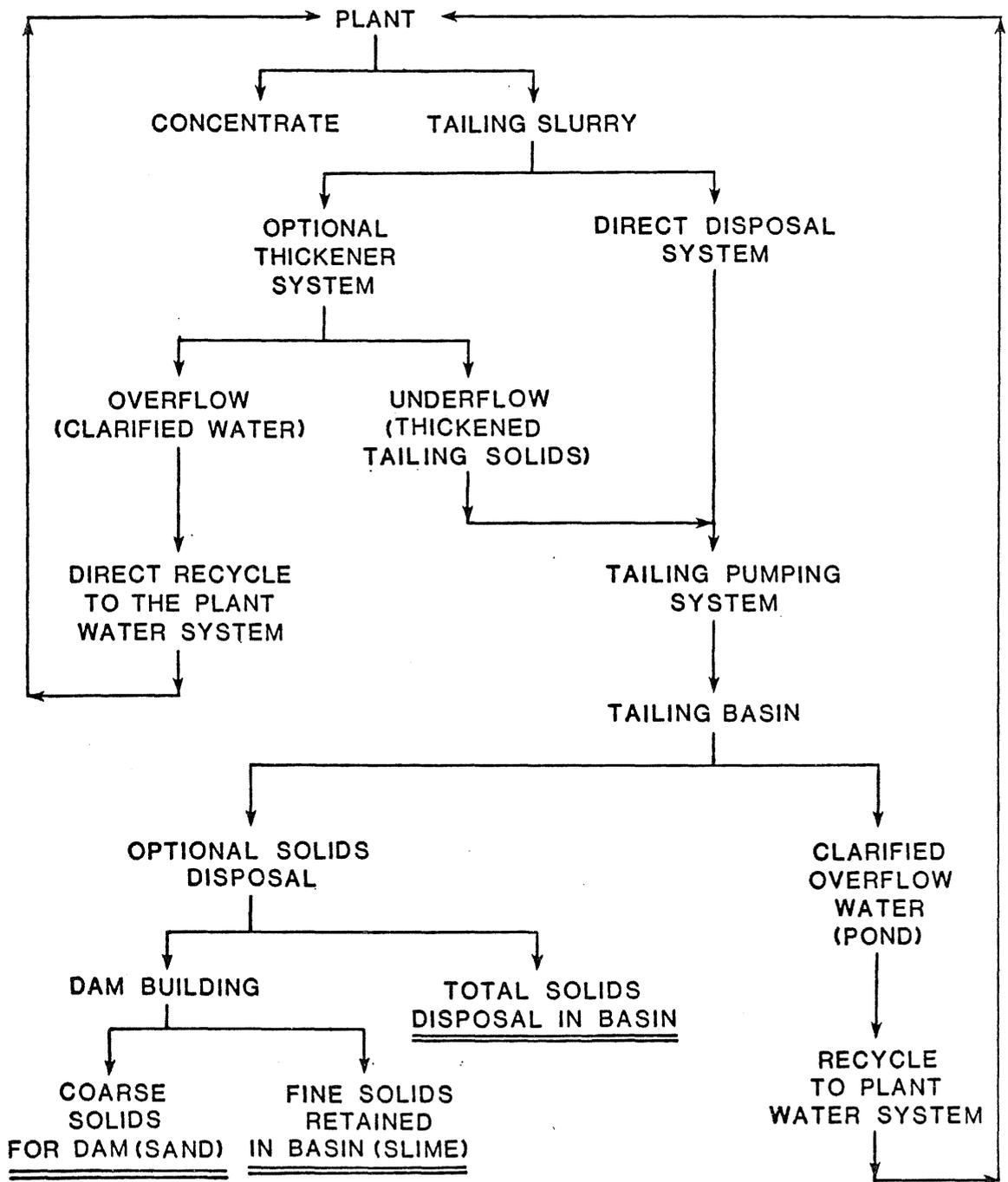


Table 8. Brief description of a typical tailing system.

SYSTEM OR MATERIAL	PROCEDURE OR TREATMENT
Plant tailing	Cyclone to remove coarse material thicken fine material, combine and pump to tailing basin. Thickener overflow returned directly to plant water system. May pump directly without thickening.
Tailing pumping system and pipe lines	Consists of pumps in stages and tailing line suitable to transport material from plant to basin. Generally have 2 lines, one operating and one as a spare with automatic switchover in case of failure of the operating line.
Tailing separation and/or distribution system at the basin	Tailing would be separated into a coarse fraction for dam building and a fine fraction for settling by cyclones set up along the periphery of the basin, or totally fed to the basin for settling through a spigotting system in the same location.
Tailing dam	Construction methods and materials used to accommodate the tailing material according to the established design. Component parts are starter dam, underdrain, and tailing dam, with minor but very costly specialty features.
Water return system	Pump barge or decant structure and pipeline to reclaim clarified pond water and return it to the plant.
Seepage control system	Both dam and basin linings, and water collection systems which return seepage to the basin.

Generally, the tailing system handles solid waste material from the processing operation, disposes of it, and returns the transport water for reuse in the processing system. The tailing can either be thickened at the plant site to remove a large portion of the water for immediate reuse, or pumped directly to a basin for settling and decantation of the total recycle water.

At the basin site, the tailing is either pumped directly into the basin behind previously prepared dams for settling and decantation, or the coarse fraction is removed for dam building and only the fine fraction is discharged behind the dam. In either case, the contained water which is liberated when the solids settle is collected in a pond, decanted, and returned to the processing system.

Not shown in Figure 15 are other sources of water which may be sent to the tailing basin for clarification, water retained in the settled solids, seepage water losses, precipitation gains, and evaporation losses. These are discussed in detail either in this section of the report or in the water balance section of the water resources chapter (Volume 3-Chapter 4) of this report.

This section presents information that is limited to tailing disposal in a basin. Utilization of tailing material as backfill in underground mines is covered in Chapter 2 on mining. Recovery of valuable mineral commodities from the tailing material is also briefly discussed later in this report.

3.2.4.2 Siting Considerations--Suitable tailing disposal areas are usually identified on the basis of topography, surface and subsurface geophysical features, watershed conditions, and distance from the operation facilities. Topography determines requirements for economically containing the total volume of tailing material over the life of the operation. Surface and subsurface geophysics indicate the suitability of local material for construction of the

initial retaining dams and the potential for seepage from the basin into the local groundwater system. Watershed conditions indicate the normal flow of water through the area and the controls necessary to insure against the contamination of surface water.

In addition to the above engineering factors, various land use, land ownership (and control), and environmental factors are also important in the selection of tailing disposal areas. Alternative operating approaches such as a single large basin versus several smaller basins sequentially developed can significantly affect area selection. Once the area is identified and selected, estimates are made of the dam requirements and the pumping systems necessary to transport the tailing to the basin, distribute it within the basin and return the recycle water to the plant operation.

A major economic variable in considering alternative sites is the cost of transporting the tailing material to the site and the transport of recycle water back to the processing plant. In addition to the increased piping cost as the distance between the processing plant and the basin increases, factors such as added pumping head, spill collection basins, and river crossings can also increase costs significantly.

Based solely on economical, hydrological, and geotechnical criteria, the Study Area can be subdivided and ranked for tailing disposal suitability as shown on Figure 16, interpreted from Golder, 1978. In general, a band 28 mi wide was subdivided into 8 areas, of which 6 are suitable for disposal on the basis of the factors considered. Other factors not considered in this analysis also have a significant bearing on basin location and suitability, and are discussed in other areas of this report.

---

Figure 16

Ranking of the areas by Golder and Associates was based on the following reasoning:

Class I Area - meets all criteria of short distance from plant to tailing basin, favorable topographic features, lies within a tributary river watershed, lies within zones of stable foundation soils and easy seepage control, does not overlap iron mining.

Class II Area - fails to meet one of the above criteria for a Class I area.

Class III Area - fails to meet 2 or more of the above criteria for a Class I area.

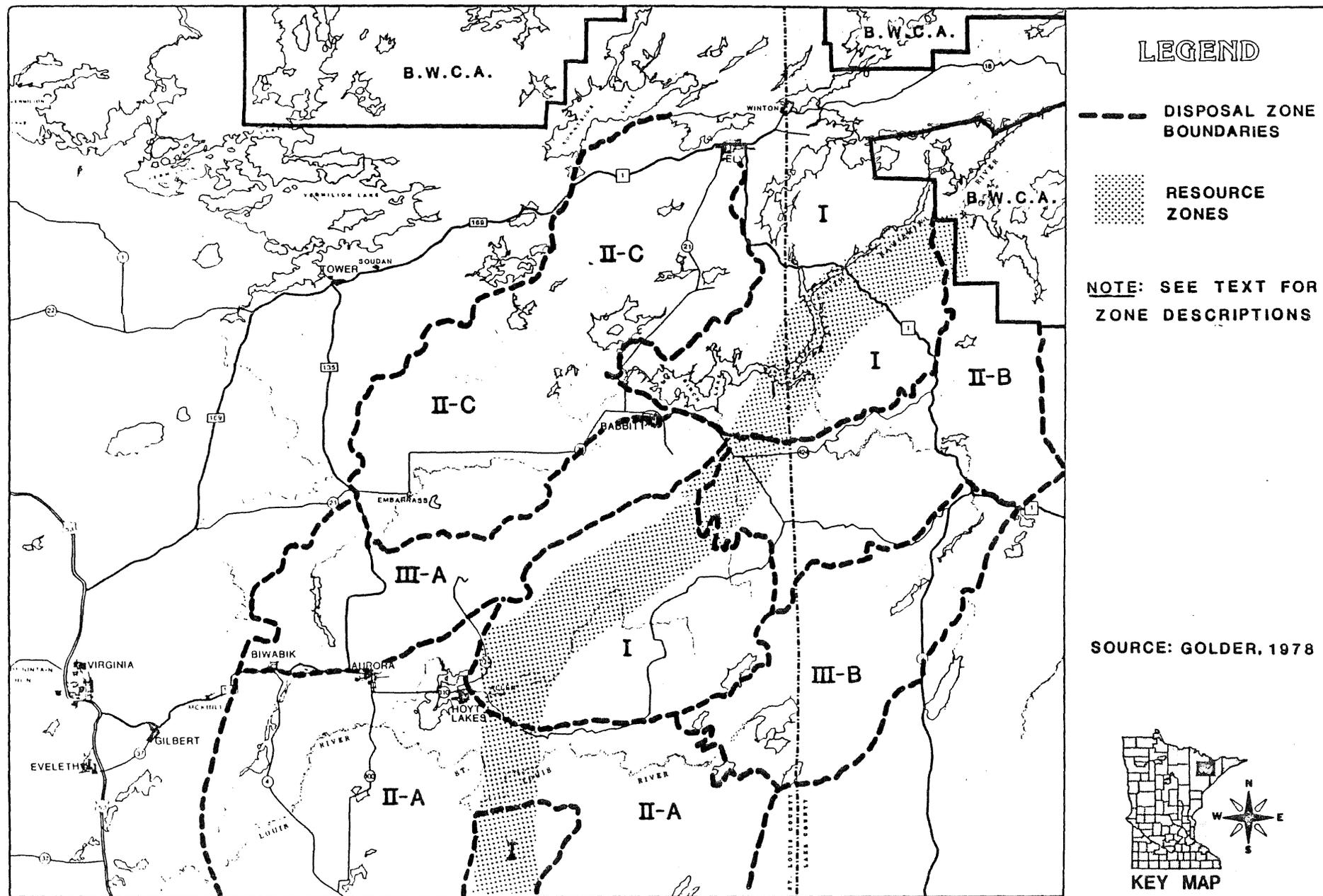
Boundaries for the classifications coincide in part with either watershed boundaries, rivers, BWCA limits, or topographical--surficial geological boundaries. The reasons for the ranking of the areas by Golder are given in Table 9.

---

Table 9

Excluding the areas occupied by Birch and White Iron lakes in the north of the region, the usable Class I area in total is 296 m<sup>2</sup>. More than twice this amount of Class II area is available for more costly tailing disposal systems. The minor amount of Class III area is largely unsuitable for tailing disposal.

3.2.4.3 Overall Basin Considerations--The height of the tailing basin embankment necessary to ensure a safe and efficient operation for a given milling



**FIGURE 16** MEQB REGIONAL COPPER-NICKEL STUDY  
 SUBDIVISION AND CLASSIFICATION OF TAILING DISPOSAL AREAS

Table 9. Ranking of areas for tailing disposal (see map-Figure 16).

AREA	COMMENTS
I	Shallow to deep relatively impervious strata, within short distance of potential mine site, and favorable topography for construction of tailing basins.
II-A	Within major watershed of St. Louis River, or too far from mine site.
II-B	Deep pervious deposits of Dunka Basin or too far from mine site.
II-C	Deep pervious deposits of Embarrass Basin, or too far from mine site.
III-A	Unfavorable topography and overlaps with iron mining.
III-B	Unfavorable lakeland topography, deep pervious deposits, and too far from mine site.

SOURCE: Golder 1978.

capacity is governed to some extent by the size of the grind, but mostly by the terrain within the tailing area. A starter dam constructed from borrow material is a very important part of the entire impoundment. The purpose of this dam is to initially contain the tailing and provide a pond large enough to insure sufficient water supply and clarification at the start of operations. The steeper the terrain within the embankment area, the higher the starter dam must be to supply the storage necessary for the fine tailing and water until the embankment can be raised with the coarse tailing sand. It is far better to make the starter dam a bit higher than required because of unknown factors at startup of an impoundment. These unknowns are: 1) the efficiency of segregation of the sand and slime; 2) the angle of repose of the beach area; and 3) most important, the retention time needed in the pond to obtain clean water for recycle.

Figure 17 shows how the area of a single tailing basin changes with variation in average depth of the tailing solids. The curves indicate the variation in area required of tailing basin depths ranging from 50 to 150 ft for mine sizes of 5 and  $20 \times 10^6$  mtpy assuming a variety of modeling assumptions such as tailing density and weight recovery. This range was selected to typify metal mining operations applicable to Minnesota copper-nickel. Obviously, as the depth decreases below 50 ft, the area needed will increase dramatically for the cases shown. On the other end of the scale, although deep basins require significantly less area to store the same volume, dam stability, visual impacts, fugitive dust, reclamation, and slope erosion become more of a problem.

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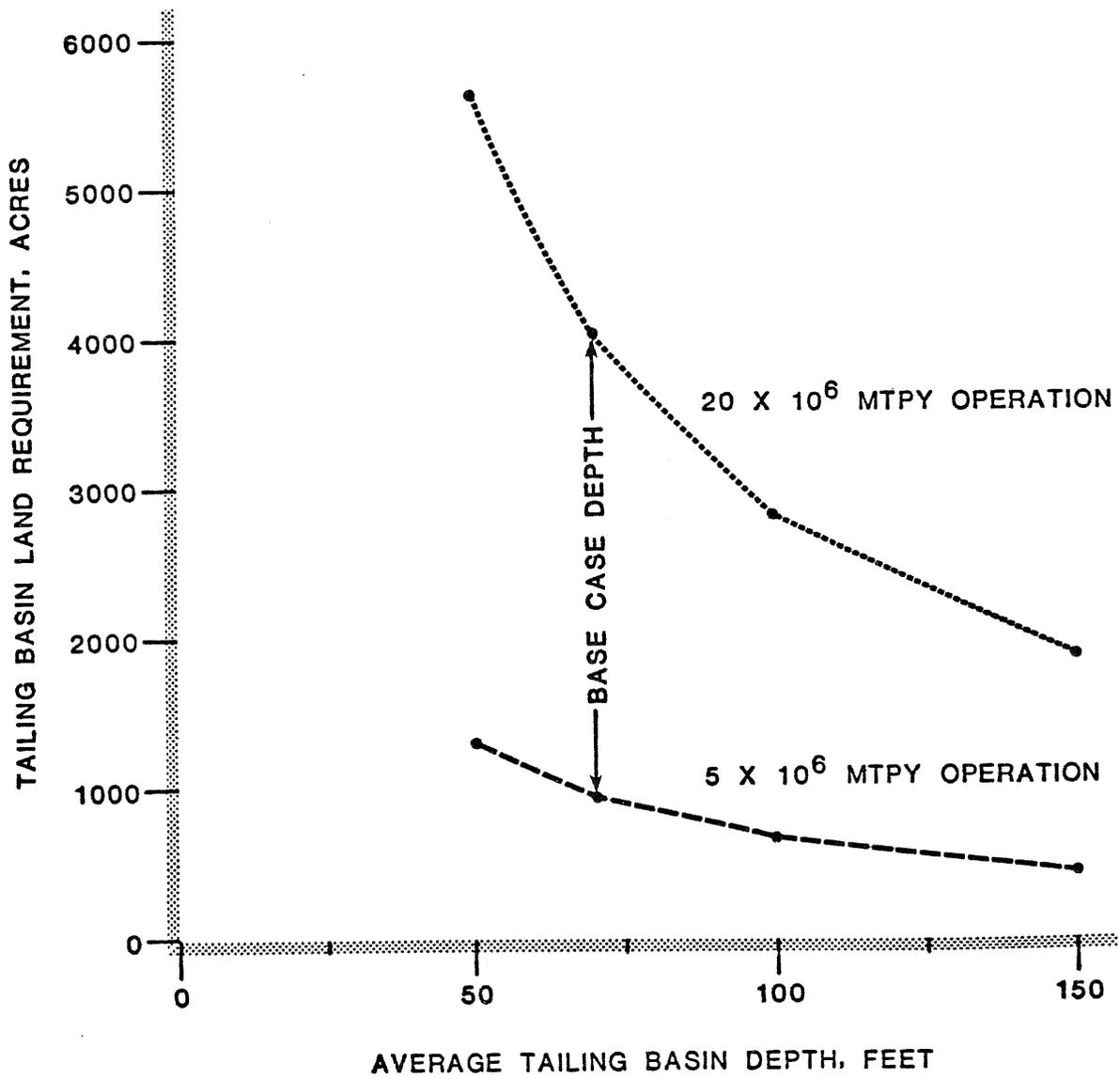
Figure 17

Total tailing production over the life of the mine is another major factor affecting basin size. Figure 18 compares the relationship between storage

FIGURE 17

### TAILING BASIN AREA VARIATION WITH DEPTH OF TAILING

(ASSUMING A TOTAL LIFE OF 30 YEARS;  
SEE TEXT FOR OTHER MODELING ASSUMPTIONS USED)



volume and basin size assuming a circular basin with an average depth of 70 ft (75 ft high dam), and average tailing density of 90 lb/ft<sup>3</sup> (0.84 mt/m<sup>3</sup>). The values shown for the AMAX and INCO operations are for general comparative purposes and are based on available information concerning the general scale and life of operations at these 2 locations. This figure can be used as a general guide in understanding the amount of land required for tailing disposal if different percentages of the total copper-nickel resource are exploited. For example, roughly 30,000 acres would be required if the entire  $4 \times 10^6$  mt of resource were extracted, under the modeling assumptions used here.

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Figure 18

One approach to tailing disposal is the use of multiple smaller basins rather than a single large basin, with the flexibility of being able to place disposal areas in smaller, more desirable locations if a single large location is not available near the processing operation. Such would be the case if the near-plant area includes lands which should be avoided for a variety of reasons (e.g. unique habitat) or if the terrain is not suitable for a large disposal site.

Multiple basins would require multiple transport systems if operated consecutively or moving of transport systems if operated sequentially. However, they do offer the possibility of sequential permanent rehabilitation as the small basins are filled; on the other hand, the costs will likely be higher due to more dam building needs and transportation system requirements.

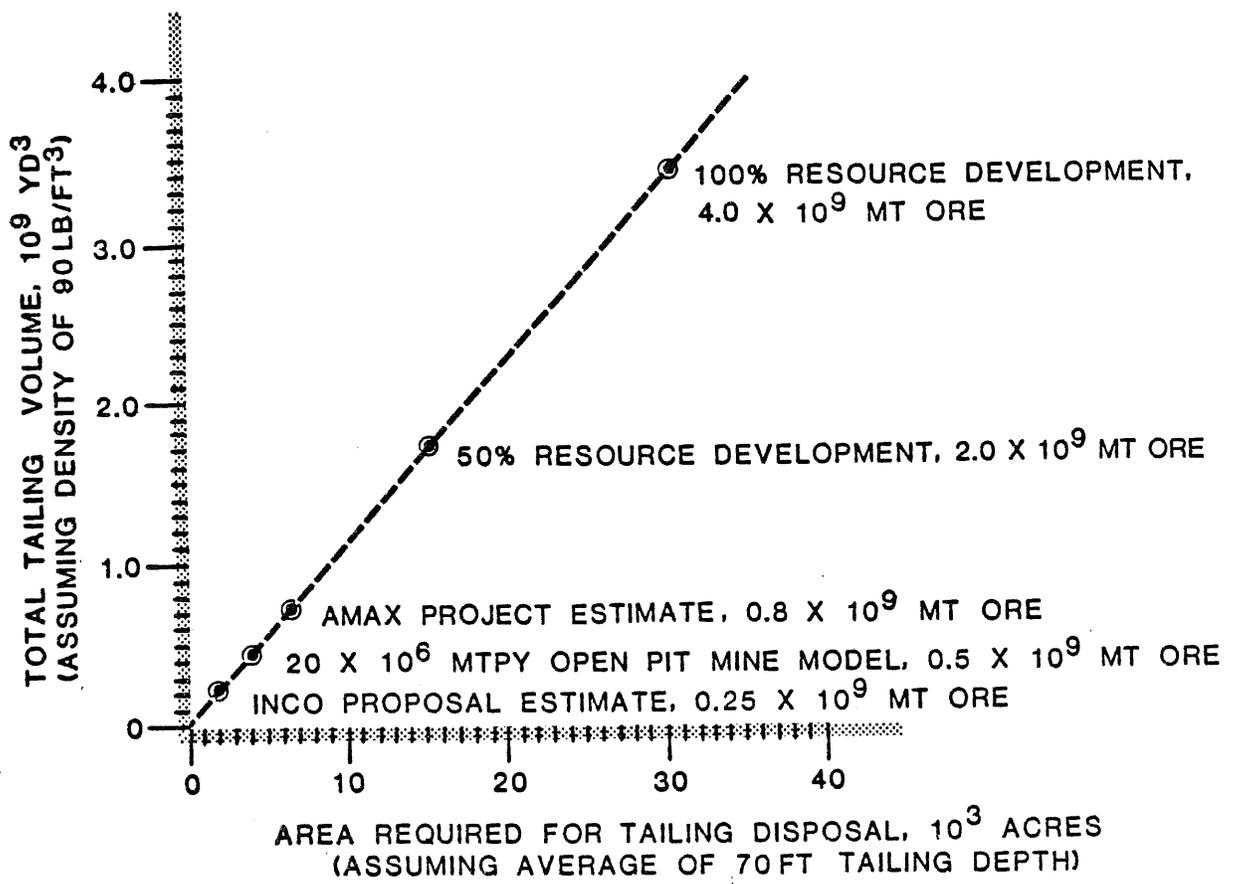
A compromise between the single large basin and multiple small isolated basins is the cellular type of construction, in which sections of the total embankment are built separately, allowing for a smaller active area at any given time.

FIGURE 18

# APPROXIMATE LAND REQUIREMENTS FOR MINNESOTA CU-NI RESOURCE DEVELOPMENT TAILING DISPOSAL

**ASSUMPTIONS:**

- ALL TAILING BASINS CONTAIN MATERIAL AT AVERAGE OF 70FT DEPTH
- ALL TAILING STORED AT AVERAGE DENSITY OF 90 LB/FT<sup>3</sup>



This approach would be more expensive and probably occupy slightly more area than the single basin, but it does offer the potential for reduced dust liftoff as portions of the total can be revegetated permanently before the operation comes to an end. This basic concept is being constructed by Reserve Mining Company at their Silver Bay location.

In order to reduce the total area occupied by a given amount of tailing, it is necessary to increase the embankment height, thereby increasing the rate at which the embankment must be raised each year. There is no established rate at which an embankment can be raised, but for a given material, size gradation, and tailing slurry density there is a definite maximum rate of rise above which stability becomes a problem. If the tailing cannot drain as fast as it is placed in the basin, the phreatic water surface rises and results in a discharge through the dam face above the toe dam. When this occurs, seepage and piping take place, lowering the safety factor toward the danger point. Possible solutions are to allow time for drainage and to place a filter and rock surcharge on the toe. A rapid annual rise is undesirable because the material does not have time to properly drain, consolidate, and stabilize, nor is there time to raise the peripheral dam.

Where the maximum annual rise in basin height is limited to less than 8 ft/yr (generally accepted maximum) to insure stability, the active disposal area must be at least 20 acres per 1,000 mt of daily capacity. Operating at this upper limit of rise per year for continuous operation might be safe, but this depends again on the grind, slurry density, and type of material being impounded. From an operating and safety point of view, a figure of 30 acres per 1,000 mt of daily capacity is much better for the lower limit of a mature basin. For the  $20 \times 10^6$  mtpy mine model, the minimum area required then is 1,700 acres, as

compared to just over 4,000 acres modeled for the total basin over the life of the mine.

The minimum tailing basin requirement of 1,700 acres for the  $20 \times 10^6$  mtpy open pit mine model affects basin design in different ways depending on design goals. If the goal is to minimize land area impacts, then a basin 1,700 acres in size with an average depth of just over 150 ft (see Figure 17) meets this design goal. On the other hand, if the goal is to facilitate early basin reclamation by use of multiple basins operated in sequence and maintaining a maximum average depth of 70 ft, then 2 basins would be required (each 2,000 acres in size). If a 1,700 acre basin is too large based on siting or reclamation goals, then more than one basin will have to be operated at a time.

As mentioned previously, one way to reduce the land area needed for tailing disposal is to increase the thickness of the tailing by increasing the height of the retaining embankments, but this does not necessarily reduce overall disposal costs. For example, the largest area with the lowest embankment will provide the maximum ratio of tailing storage volume to retaining embankment volume, and hence, minimum overall cost, exclusive of land costs, to a mine operator faced with the cost of constructing the embankments.

In general, the height of tailing retaining embankments is not limited by stability requirements that cannot be overcome by engineering design. However, very high embankments (greater than 100 ft) on flat terrain may be objectionable from a visual standpoint. Furthermore, embankments that protrude significantly above natural topographic features could have a marked influence on the storage to embankment ratio, especially if natural hills and ridges are used as part of the retaining embankments. This will become apparent by considering how basins

might be constructed in the Toimi Drumlin area in the southeastern portion of the Study Area, if the drumlin ridges are utilized as segments of the embankments.

Bearing in mind that the maximum local relief is not more than about 100 ft anywhere in the Study Area, other than along the Giants Range, it is reasonable to assume that the average thickness of tailing would be of the order of 70 ft, if a principal design goal is the maximum utilization of natural topographic features.

The area of the tailing pond required for adequate clarification of the water prior to reclaim or discharge to local streams is difficult to determine by theoretical means. Although the settling velocities of various types and grain sizes of solids can be determined theoretically and experimentally, many factors influence the actual effectiveness of the pond. Basically, the problem is to provide sufficient retention time to permit the very fine fractions to settle before the water reaches the point of decantation. Major factors affecting particle settling time are the size of grind, the tendency to slime (clay type minerals), the pH of the water, wave action, and depth of water.

The size of grind required for liberation of the metal is usually sufficiently fine to produce particle sizes whose settling rates are governed by Stokes' Law, with a high percentage under 200M. Particles in the range of 300M or 50 microns with a settling rate of 0.05 in./sec can be affected by wind action, but will settle in a reasonable time. The major problem is caused by the small percentage of particles in the range of 2 microns or less which produce turbidity. These particles have settling rates lower than 0.01 in./sec in still water and, under conditions prevalent in most tailing ponds, require some days to settle below the turbulence caused by wave action.

Various rules for clarification have been accepted as a result of observation in existing ponds. Among these are:

1) The quality requirements of the water returned to the mill or the watershed will determine the retention time for any particular operation. The time required may be as low as 2 days and as high as 10 days, with an average of about 5.

2) The area of the pond should be sized to provide 10 to 25 acres of pond area for each 1,000 mt of tailing solids delivered per day. An average of 15 acres (0.0234 mi<sup>2</sup>) per 1,000 mt is usually considered adequate, unless some unusual conditions are present.

3.2.4.4 Embankment Design and Construction--Several methods of tailing embankment or dam construction are available for consideration, as detailed in the Golder and Associates report. Where a tailing embankment is to be constructed predominantly of sand recovered from the tailing slurry, 3 basic construction methods can be used. These methods, illustrated in Figure 19, are commonly called: upstream, downstream, and centerline.

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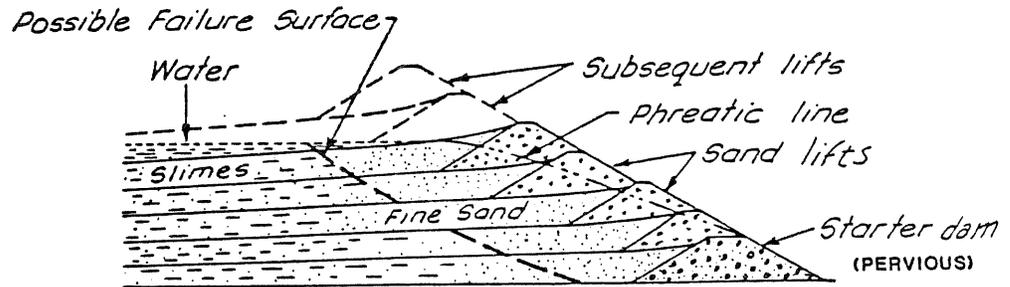
Figure 19

In the upstream method the crest of the embankment is raised by placing tailing sand in successive layers onto the upstream side of an initial starter dam. The initial starter dam forms the downstream toe of the ultimate dam. Materials used in construction of the initial starter dam should be pervious, relative to the tailing, and the gradation of the materials should conform to the requirements for filters with respect to the materials placed on the upstream side of the starter dam. This approach generally requires less dam construction

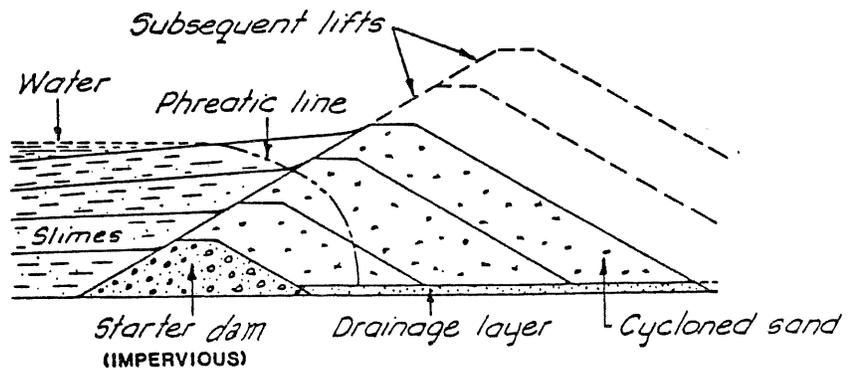
FIGURE 19

# TAILING EMBANKMENT DESIGN

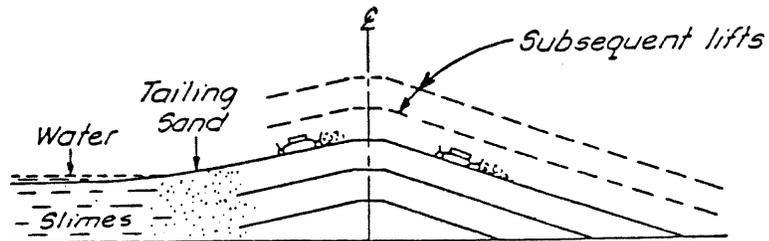
(SOURCE: GOLDER, 1978)



a) UPSTREAM METHOD



b) DOWNSTREAM METHOD



c) CENTER LINE METHOD

material as the dam is progressively built out over slime-sand layers of previously settled tailing. However, a certain amount of instability is present with this method due to the fact that slimes which lie under the dam do not readily compact and may be liquified, which can result in a failure of the dam.

The upstream technique is really only suitable for low tailing embankments located in areas where the consequences of failure are minimized; in general, they do not meet today's standards of safety and pollution control (Golder 1978).

In the downstream method tailing sand is placed on the downstream side of the initial starter dam which forms the upstream toe of the ultimate dam and should be impervious relative to the tailing sand. This approach requires the greatest volume of dam material, but also provides the greatest control and stability as the dam is entirely built of the coarser fraction of the tailing and no slime pockets are intermixed with the coarse.

An alternate to the downstream approach is the centerline method which combines both upstream and downstream designs in construction. No particular advantage over the downstream method is apparent, unless there is a shortage of construction material and a slightly reduced safety factor is acceptable. The crest of the embankment is maintained in approximately the same horizontal position as the embankment is raised to its final height. The elevation of the crest of the dam is raised by placing successive layers of material on the crest and on both the upstream and downstream fill slopes.

After the design is established the general procedure is to first construct a starter dam of local borrow material or waste rock from the mining operation. This dam will retain the initial tailing material and form the basis for subsequent dam construction with coarse tailing material (+200M). A seepage

control system is also installed, consisting of ditching, diking, and pumps to collect water seeping through the dam and return it to the system.

The tailing is piped to the basin, and then distributed along the starter dam, either by spigotting to the upstream side for natural settling and clarification, or separated into coarse and fine fractions by cyclones. The coarse fraction is used to build up the dam, either upstream or downstream, while the slurry containing the fine fraction is discharged behind the dam for settling, permanent storage, and water clarification. As the operation progresses, the dam is raised in stages until the ultimate required height is reached.

As previously discussed, several basins may be necessary to contain all of the tailing material produced, particularly if surface topography will not allow one large basin. Economics, reclamation, operating flexibility, and environmental control are primary considerations in such systems.

Influences of Tailing Characteristics on Design--The 3 basic construction methods described above lead to substantially different embankment cross-sections (Figure 19) and produce different embankment material characteristics. The embankment stability conditions are radically affected by the characteristics of both the material deposited behind the dam and the material used to construct the dam. This section summarizes the design considerations detailed in the 1978 Golder report.

Tailing Material Shear Strength--In the upstream method of construction, the stability of the final embankment is dependent, to a large degree, on the shear strength characteristics of the tailing deposited upstream of the embankment. The shear strength of the sedimented tailing is governed, in part, by the gradation and density of the solids, the consistency of the slurry, and the distribu-

tion of the pore water pressures within the deposit after sedimentation. When initially deposited, the tailing has a very low shear strength. The strength increases with time as drainage and consolidation take place under the weight of overlying materials.

A tailing embankment constructed by the upstream method will generally have a lower factor of safety than the same sized tailing embankment constructed by either the downstream or the centerline method, as the upstream retaining dam is built on layers of slime and sand-slime mixtures. Such an embankment may then be subject to failure by liquefaction and unsuitable for areas subject to seismic activity. For high embankments (in the order of 100 ft or more in height), the method is generally unsuitable even in non-seismic areas such as Minnesota.

In the downstream method, all of the embankment section lies outside the boundaries of the sedimented tailing. Materials incorporated in subsequent stages of the embankments may consist of the coarse fraction of the tailing separated by cycloning or by gravity separation on the beach, coarse mine discard, or rock or soil obtained from nearby borrow pits. The downstream method of construction permits controlled placement and compaction to achieve high shear strength characteristics, and permits the incorporation of drainage facilities to control the position of the phreatic water surface within the embankment. It is, therefore, inherently safer than the upstream method.

In the centerline method, the underflow from cyclones is deposited both upstream and downstream from the initial starter dam. The centerline of the ultimate dam is directly over the center of the initial starter dam. Earthmoving equipment can be used to flatten both slopes, or the slopes can be left at the angle of repose.

Sand Characteristics-The nature of the sand incorporated in the embankment cross-section is one of the most critical factors influencing the design, and the construction, of those embankments which utilize tailing sand as the prime embankment construction material.

Whatever the method of separation, if the sand deposited on the embankment has to be spread and compacted, it must be relatively free-draining. As deposited, it will generally have a water content considerably higher than the optimum range for compaction of 10-20%.

If the sand deposited on the embankment has a relatively high content of clay-like "slime," it will not only be slow in draining to optimum water content, but it will be difficult to handle. If the slime content is low, water will drain rapidly from the sand and spreading and compaction by mechanical equipment can be carried out within a short time after deposition.

Figures 20 and 21 show the nature of the variation in tailing size distribution from spigoted products due to pulp density variation. Clearly from the dam builder's point of view, the 30% pulp density system will make available a superior dam building material as a coarse product nearer the discharge point.

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Figures 20 and 21

As shown in Figure 22, raw copper-nickel tailing solids are likely to contain between 50% and 80% fines (-200M) by weight. Such fine structure will greatly accentuate any possibility of embankment instability and would practically eliminate construction of any type of tailing embankment. Therefore, for placement in the embankment, it is necessary to separate the coarser sand fractions from the raw tailing.

FIGURE 20

## GRADATION OF METAL MINE TAILING COARSE GRIND, LOW PULP DENSITY

(SOURCE: SODERBERG & BUSCH, 1977)

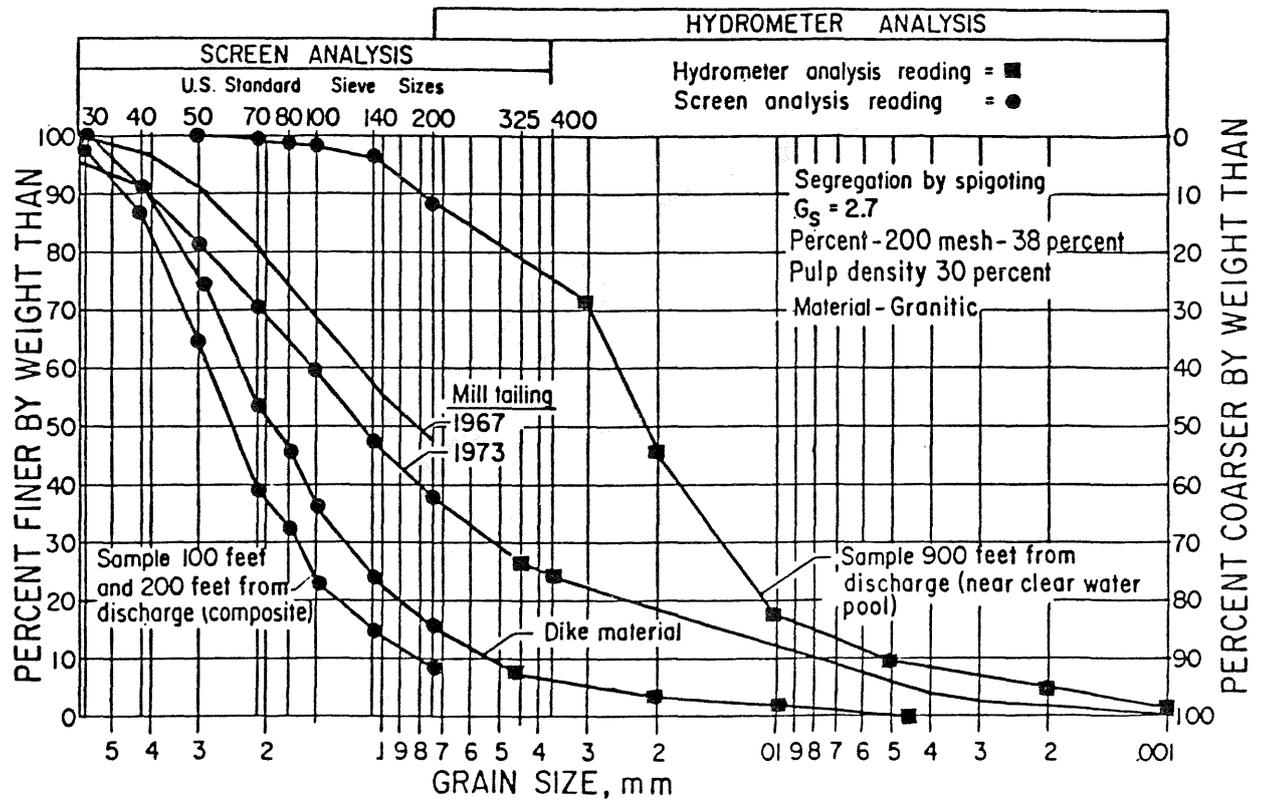
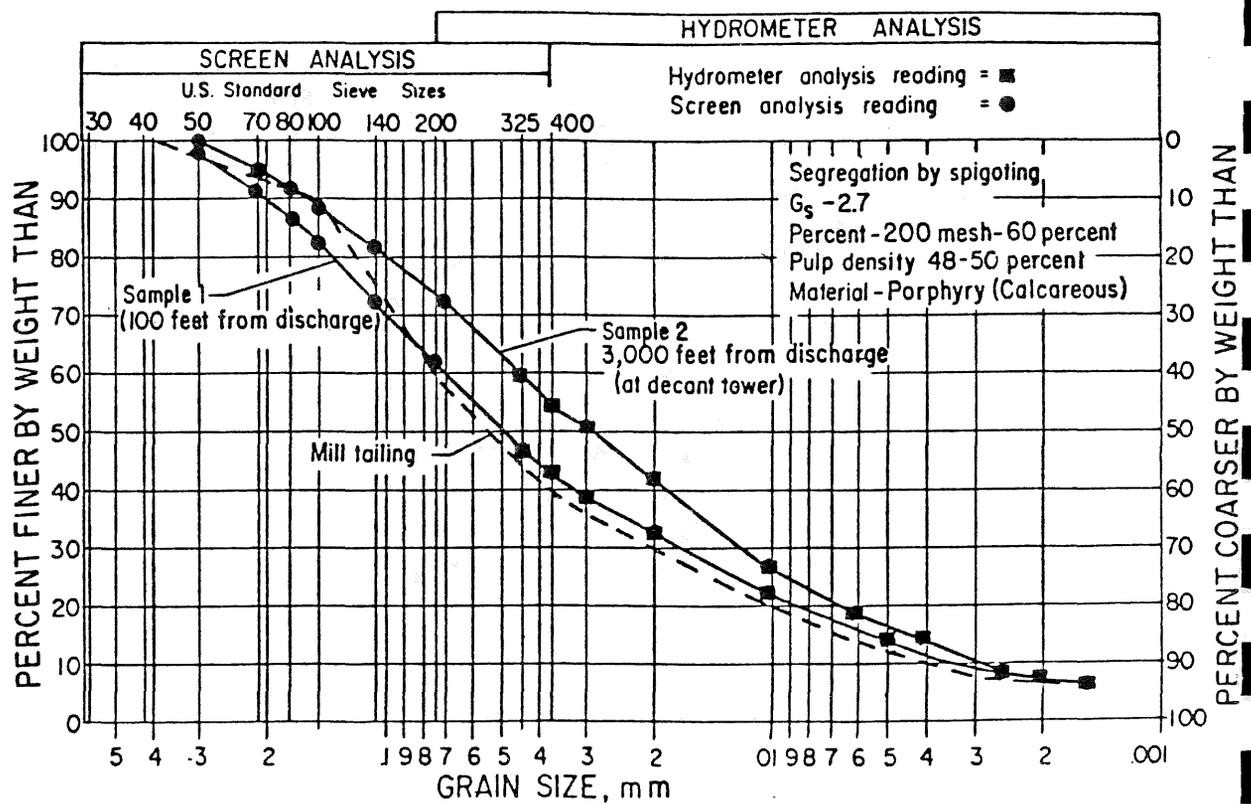


FIGURE 21

## GRADATION OF METAL MINE TAILING FINE GRIND, HIGH PULP DENSITY

(SOURCE: SODERBERG & BUSCH, 1977)



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Figure 22

Sand separation can be accomplished by several methods. With the upstream method of construction, raw tailing is often discharged into the pond through closely spaced spigots, from a header pipe installed on the embankment crest. The coarser sand fraction settles out on the beach formed just upstream of the crest. This method is illustrated on Figure 23. Successive embankment crests are raised by scraping sand from the beach by bulldozer or dragline.

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Figure 23

Spigotting is unlikely to be practicable with the downstream and centerline methods of construction even when the raw tailing is very coarse. Usually, it will be more practicable with these methods to separate the coarser sand by passing the raw slurry through cyclones, or through open-channel or pipe sluices. The cyclones are usually mounted on the crest of the embankment, as shown on Figure 24, but they can also be mounted on an abutment at one side of the embankment. From the cyclones, the fine overflow containing the "slime" is piped to the tailing basin; the underflow containing the coarser sand drops, or is discharged through pipes or launders, onto the embankment.

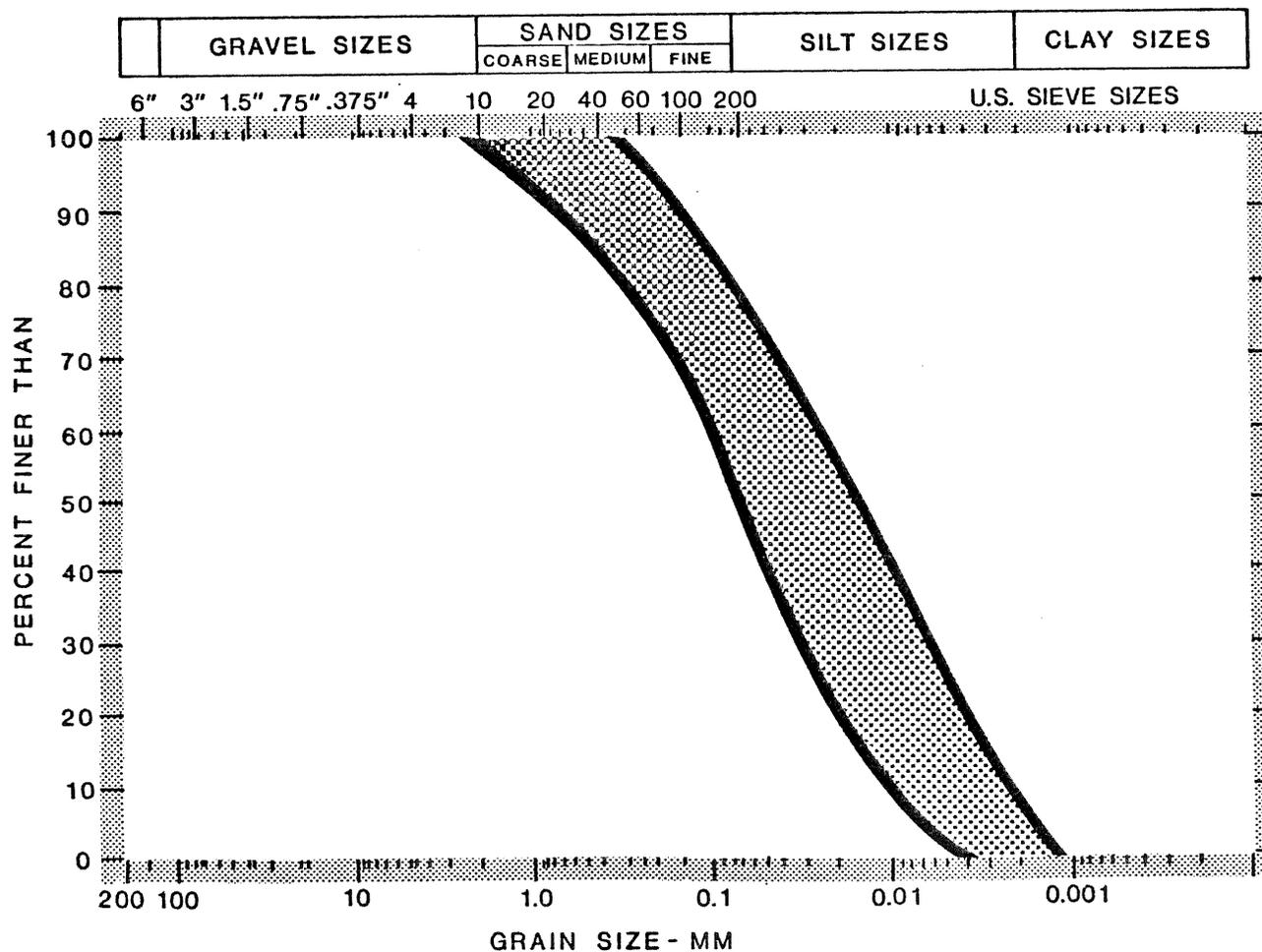
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Figure 24

Variations in sand yield have a dual effect--a decrease in yield will slow the rate of rise of the embankment crest and increase the rate of rise of the tailing in the basin. Particularly with finely ground tailing, the yield may be too low to fill the complete embankment section with sand at a rate sufficient

FIGURE 22

TYPICAL METAL MINE TAILING SIZE DISTRIBUTION RANGE



(SOURCE: MINES BRANCH, 1972)

FIGURE 23

### TYPICAL SPIGOTTING ARRANGEMENT

(SOURCE: MINES BRANCH, 1972)

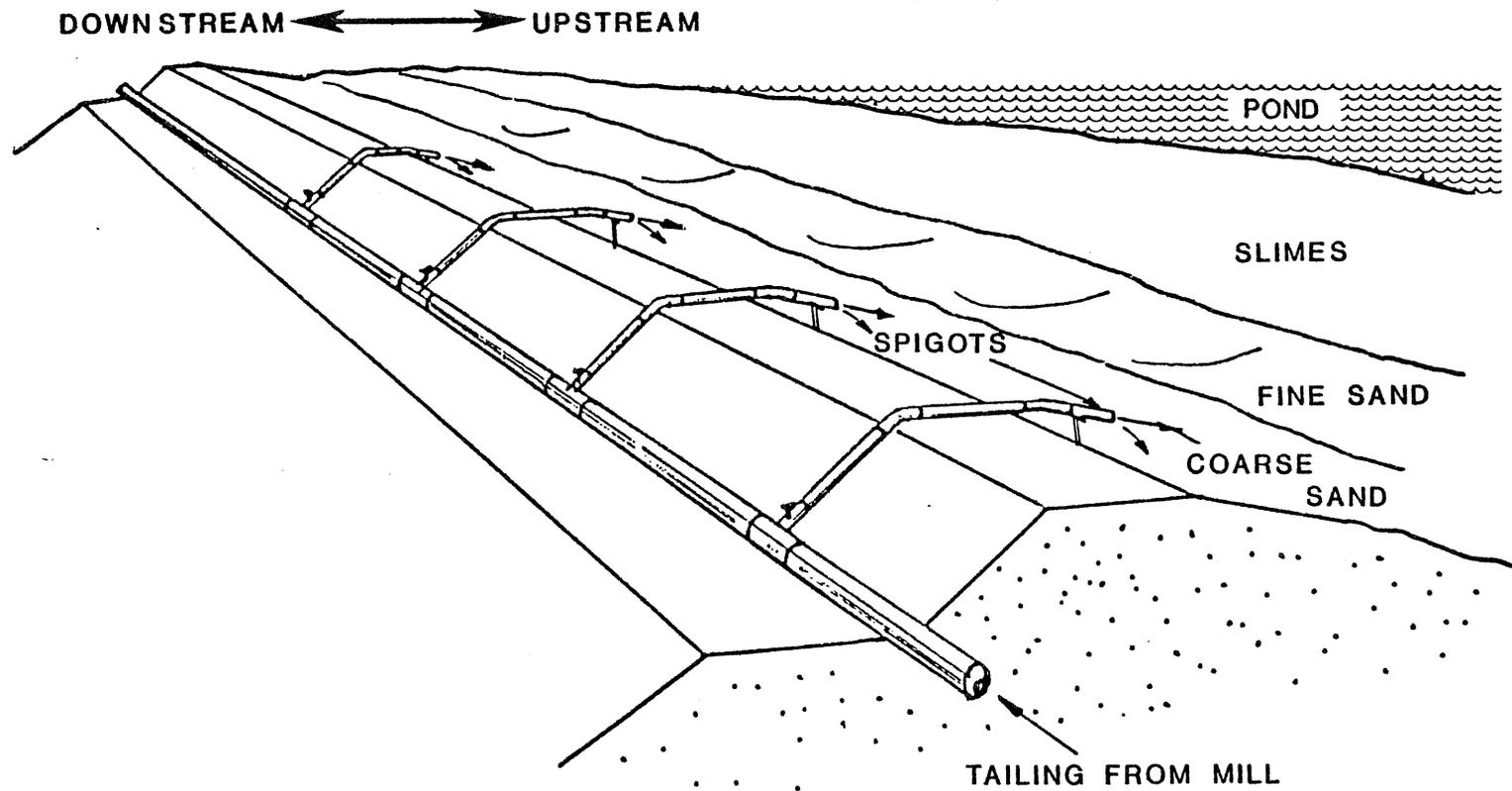
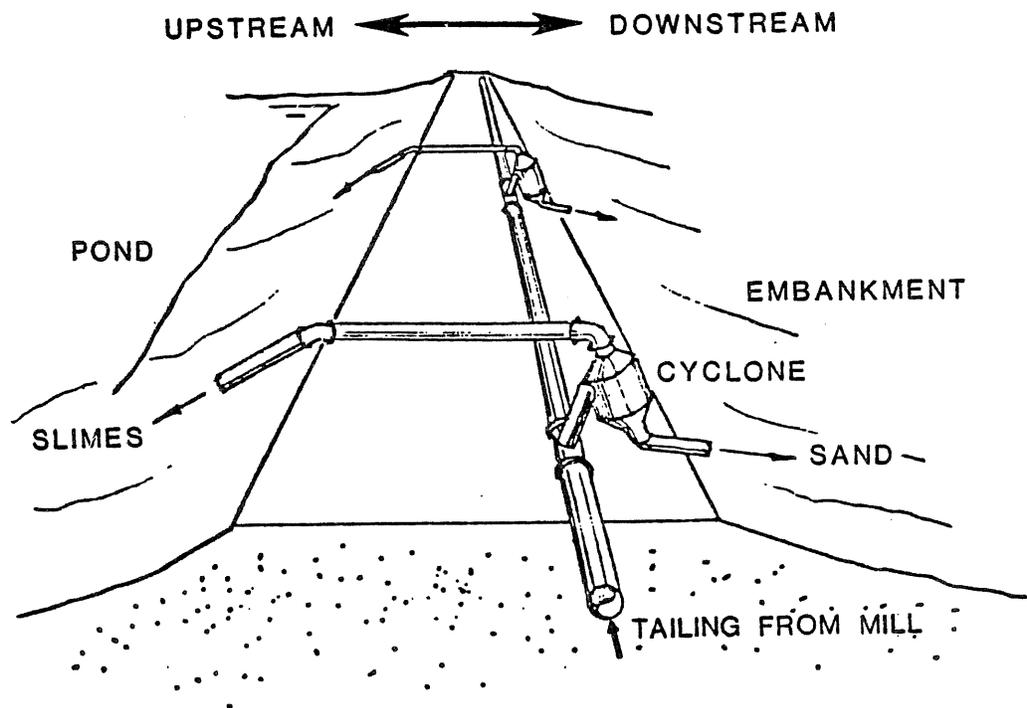


FIGURE 24

# TYPICAL CYCLONING ARRANGEMENT

(SOURCE: MINES BRANCH, 1972)



to keep the crest above the basin surface. This may require augmentation of the sand in the section with borrow or dry waste materials.

The sand yield from cyclones can be computed from information on the size gradation of the raw tailing and the characteristics of the cyclones. For example, in a typical case such as expected in Minnesota operations, approximately 50% of the tailing may be coarser than 200 M and therefore the split of coarse and fine fractions might be 50% of the total to each. This separation could likely be effected with 2-stage cycloning.

Fugitive Dust Control--Fugitive dust control of tailing basins is an important part of the operating and decommissioning procedures of such a facility. The fugitive dust emissions from a tailing basin are small in comparison to the projected emissions from open pit mining operations (see Volume 3-Chapter 3), but are significant in their own right.

In this section, only the subject of mitigating measures will be discussed. The section on air quality impacts in Volume 3-Chapter 4 of this report presents in greater detail the subjects of emission rates and impacts on ambient air quality. Due to the high content of fine particles (generally less than 50 microns in diameter), there is concern that dust from a tailing basin might pose environmental problems. It is known that under dry conditions, such as may exist in Minnesota during the late summer or during the winter in the absence of snow cover, dust lift-off does occur. Available information generally indicates that winds above 10 mph (5 m/sec), which typically blow 25-30% of the time in the region, may be sufficient to cause dust lift-off. If appropriate mitigating measures are not used, the resulting increase in total suspended particulates (TSP) in the atmosphere could pose a series of problems. Air quality standards

set by the state and federal governments may be violated. Visible dust poses aesthetic problems, as well as health and safety hazards by exposing persons to increased intake of particulates, possibly including mineral fibers, in the air they breathe and by impairing visibility at extreme high levels. The resulting dust deposition on vegetation, land, and water surfaces may pose further hazards by introducing elevated levels of metals and other constituents of the dust into these systems. These topics are discussed in the air section noted above, as well as in Volume 3-Chapter 3 and Chapter 2 of this volume in the reclamation section.

A tailing basin is a dynamic operation that is in a constant state of change. As a result, new sections of exposed tailing and dam lifts are constantly exposed to wind erosion and possible fugitive dust generation. Because of the dynamic nature of tailing basins, fugitive dust control methods must be varied and equally dynamic. Exposed tailing in a state of change require temporary control, while other areas can receive permanent control measures.

For modeling purposes it is assumed that 80% of the basin area would be covered by the settling pond. Thus, only 20% of the total area would be subjected to wind erosion, and of that area perhaps half might be managed using temporary measures and half with more permanent measures. This split between temporary and permanent control will vary greatly depending on the basin age, dam construction method, and other operating factors. For example, with upstream dam construction, permanent dust control can be practiced on the dam face (downstream) as it is largely permanent when built. However, with downstream construction, where the downstream face of the dam is constantly being renewed, temporary measures of dust control are required until the final dam height is reached.

Potential dust mitigating measures which can be used include routine watering, chemical stabilization, use of vegetation, or reduction of surface wind speeds across the exposed surfaces by using windbreaks and shelterbelts (Golder 1978). Some discussion of each method is in order here; details can be found in the above referenced sections.

Watering is an effective dust suppressant for time periods ranging from a few hours to several days. The use of watering results in the formation of a thin surface crust, but this crust is easily destroyed by equipment movement over the surface or by abrasion from loose particles blown across the surface.

Therefore, watering must be accomplished frequently to be an effective dust control method. Limitations of the weight of equipment that can be used on the tailing basins generally prevents the use of watering trucks, and elaborate methods such as automatic sprinkling systems or large-wheeled, light-weight application vehicles must be used.

Chemical stabilizers also react with tailing to form a protective crust, but the time span of effectiveness is significantly longer compared to water treatment. The same limitations for the use of heavy equipment also applies to this control method, although use of aircraft is a potential method for application. There are presently about 65 chemicals which can potentially be used.

Fugitive emissions can also be controlled by physical stabilization such as with rock, slag, bark, straw, etc. The practicality of this approach depends mainly on the local availability of these materials and economic considerations of purchase and placement.

Vegetative stabilization is a very effective control of fugitive dust emissions. However, there are problems associated with this method. There may be

resistance to vegetative growth due to excessive salts and heavy metals in the tailing as well as high surface temperatures and lack of water. Also, windblown particles may destroy young plants. These problems can be overcome with a combination chemical-vegetation technique. The chemicals reduce the sandblasting effect and serve to hold water near the surface. Vegetation germination and growth can also be enhanced by use of buried organic layers.

The large size (both area and height) of the tailing basin effectively eliminates the erection or growth of wind barriers from practical consideration. Generally, the sheltered distance downwind from a barrier is only 5 to 10 times the height of the barrier. Reduction in emissions for this application may be a factor of 0.60 to 0.90 or greater providing that these exposure criteria are met. The use of natural topographic features as wind barriers is a possibility in the rolling terrain of the region. But, again, due to the large dimensions of likely tailing basins, this approach would not be an effective control method, although development in its effectiveness may be sufficient if the resource region actually involves several smaller tailing basins. The feasibility and efficiency of using natural wind barriers is highly dependent on local topography and, therefore, must be evaluated on a site specific basis.

An effective, but expensive, method of controlling fugitive dust during the operation would be to construct water-type dams rather than tailing dam which would allow flooding of the basin right to the upstream dam face (allowing sufficient freeboard). Such a dam would probably have to be entirely constructed before tailing was deposited in the basin for settling, and would necessitate greater front-end capital expenditures. Stage construction of the dam (lower front-end capital costs) is perhaps possible, but insuring complete sealing of the basin, such as is typically found in water dams, would be a difficult construction task.

The most effective mitigating measure is the reclamation of the tailing basin by covering with soil and planting with vegetation. Fugitive emissions from such an approach are expected to be negligible. The operating approach for most effective use of reclamation as a dust mitigation method is the concept of multiple tailing basins. This approach allows comparatively rapid filling of a small basin, and subsequent reclamation while another small basin is being filled. Continued throughout the operating life of the property, this sequential approach eliminates the exposed beach area of a single large basin and the need to delay permanent reclamation until the property is exhausted. Based on information presented earlier, the minimum basin area for a model  $20 \times 10^6$  mtpy mine is 1700 acres, of which 80% would be covered by water.

A variation of the multiple tailing basin approach is that of sectionalizing a single large basin into several small basins, and sequentially filling them as in the case of the multiple basin approach. Either method, although probably more expensive than the single basin design, can provide more permanent dust mitigation as the operation proceeds and thus be more acceptable from an environmental standpoint.

A summary of the effectiveness and costs for alternative control methods (Golder, 1978) is presented in Table 10. Detailed evaluations of stabilization methods for tailing are available from the U.S. Bureau of Mines in IC 7896 (Dean et al. 1974), and provided the source of the Golder comparison.

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Table 10

Copper-nickel tailing basins are expected to be similar to Minnesota's taconite tailing basins in many respects significant to dust control and mitigation

Table 10. Summary of alternative mitigating measures, tailing basin dust control.

CONTROL METHOD	CONTROL EFFECTIVENESS	MAINTENANCE	TOTAL CONTROL COST (per acre <sup>a</sup> )
Watering	fair	continual	---
Vegetation (hydroseeding or matting)	65%	minimal	\$200-\$1000
Slag cover (9 in. depth)	good	moderate	\$400-\$1350
Chemical stabilization	good	moderate	\$300-\$1000
Combined chemical-vegetation	excellent	minimal	\$150-\$ 350
4 in. soil cover and vegetation	excellent	minimal	\$400-\$ 850
12 in. soil cover and vegetation	excellent	minimal	\$1000-\$2250

SOURCE: Dean et al. 1974.

<sup>a</sup>1977 cost data, adjusted from 1973 figures.

requirements. Since dust generation from copper-nickel tailing basins is only projected at this time, the state can gain valuable experience in predicting actual emissions and the effectiveness of mitigation by studying existing taconite operations. Only in this manner of study can the most realistic projections be made.

In summary, the Golder report states that the optimum control measure would be reclamation of the tailing basin by use of a soil covering with the planting of vegetation. Soil availability and costs may be a limitation for this approach, but unnatural fugitive emissions would be essentially eliminated and conformance to air quality regulations can be assured. Depending on site specific conditions, a combination of chemical-vegetation mitigating measure plus the use of natural wind barriers (such as terrain features) is an alternative possibility.

Seepage Control--Control of seepage from tailing basins is important for 3 reasons:

- 1) To maintain the egress of polluted water to within acceptable limits.
- 2) To ensure stability of retaining embankments and their foundations.
- 3) To reclaim water for mill processing.

Limited available information on tailing basin water quality indicates the problems which are normally associated with the base metals industry will not be present in the processing of Duluth Complex sulfides (see Volume 3-Chapter 4 for more details). However, this lack of information calls for careful consideration of the potential risks when making decisions concerning sulfide processing in Minnesota, which could increase potential water pollution and the associated control requirements.

The first rule in pollution control and site water management is to direct all uncontaminated water around a tailing basin. This reduces the amount of water potentially requiring treatment, and hence reduces the treatment costs. While large amounts of seepage may be treated at an acceptable cost by collecting, recycling, and neutralizing during the operating life of a mine, after abandonment the cost can be excessive. Whatever the approach, it is necessary first to decide on the degree of control required to meet various water quality goals and then to determine the mitigation measures required to meet these limits. Such an analysis can be found in the water quality and aquatic biology impact sections of this report.

Reclaiming water from tailing basins is important for both pollution control and water conservation. Clearly, the ideal arrangement is a closed system, but very few mines ever achieve it. Studies by the U.S. Bureau of Mines (Golder 1978) indicate that nationwide, copper mines recirculate only up to about 50% of tailing pond water, although in the northern states the quantity is perhaps up to 75%. The problems that arise are build-up of reagents affecting the flotation process, excess precipitation into the pond, and costs of pumping and piping.

Quite naturally, a mine superintendent might like to use only fresh water in the mill and discharge it entirely with the tailing. In theory, however, treatment of reclaimed water for mill usage ought to be no more costly than treating polluted water before it is discharged to the environment.

The water balance for a tailing basin is determined from a knowledge of surface inflow, seepages, precipitation and evaporation, and the quantity of water from the mill (see water balance section of water resources report, Volume 3-Chapter 4). Soil mechanics analysis can be used to determine seepage quantities.

An assessment of seepage through the embankment and foundations is an essential step in the design of any tailing basin. If the seepage is allowed to pass through the body of the embankment, the shear strength of the materials is reduced and a flatter outside slope must be used to achieve sufficient strength. If seepage exits on the face or downstream of the toe dam, and the hydrostatic pressures exceed the weight of the soil above, there is a danger that piping may develop. This can lead to progressive backward erosion and subsequent collapse of the dam. This mechanism is potentially very dangerous and historically has been a major factor contributing to the failure of earth dams. In addition, particular care must be taken to prevent seepage and piping along any culvert that may be located through or under the embankment as these structures are naturally the weakest link in the embankment and will cause failure if not properly installed.

The quantity of seepage that escapes from a tailing basin depends very much on the nature of the foundation soils, on details of embankment construction, and on the method of deposition of the tailing. It is dependent on the permeability of the tailing, the height of water, the thickness of tailing, and the area covered. Clearly, however, if water ponds against a pervious embankment, the water escapes through the embankment, and the rate of escape depends upon the hydraulic gradient and the embankment permeability. Alternatively, if slimes are beached against the embankment and water is ponded against an impervious barrier such as a drumlin of till at the opposite end of the basin, the seepage path is downward through the tailing. In this latter case, if the foundations are a more pervious outwash sand and gravel, the seepage after an initial blinding layer has been placed is dependent on the area of the basin, the hydraulic gradient, and the coefficient of permeability of the slimes.

The time required for water to seep through the tailing depends on the velocity and the length of the drainage path. Seepage is controlled by the rate at which water can flow through the foundation soil, tailing, or the embankment. If, for example, the area underlying the basin is uniformly covered by a thick deposit of either peat, silty clay, or clay till, the seepage out of the basin would be at least an order of magnitude less than with more pervious gravel or outwash material. Because of the extreme variation in permeability of glacial deposits, more precise estimates could only be made on a site-specific basis.

Under Drains-If studies indicate that seepage pressures through the embankment should be reduced to improve stability, it is usual to provide drains beneath the downstream slope. The drainage system may consist of granular blankets, strip drains, or drainage pipes. The type of drains will depend on the availability of suitable drainage materials, potential seepage volume, and foundation conditions. Drains will control piping and seepage from the slope and will allow steeper slopes and less embankment fill. In addition, they will negate the possibility of freezing of the slope, which impedes seepage, and they will minimize the development of ice lenses and subsequent surface sloughing during the spring thaw.

If perforated pipe drains are used, they must be designed to withstand the total vertical load. The perforations should be placed down and the perforation diameter should not exceed 0.5 times the 85% finer size of the surrounding drainage soil. Since pipe drains cannot be repaired, they should not be used where moderate to high settlement could damage them. Also, the material with which the pipes are constructed must preclude any possibility of deterioration by corrosion.

Blanket or strip drains are laid down prior to placement of the embankment. If the volume of seepage is expected to be moderate to heavy, blanket drains are preferable. If the potential seepage is small, strip drains of pervious material may be sufficient. If the foundation is a source of seepage, caused by artesian conditions or consolidation drainage due to increasing the height and weight of the embankment, the drainage layers must be designed to carry this volume of water.

The thicknesses of the drainage layers will be a function of the seepage volume and soil gradation in the embankment and the foundation. They should never be less than 12 in. and preferably should be at least 36 in. thick. The final embankment should incorporate a coarse toe drain to ensure free drainage and to control piping.

Chemical tests should be performed on the embankment and drain materials, and the seepage water to ensure compatibility. For instance, the drains should not be made of carbonate rocks if the seepage water is acidic.

The drainage layers must be carefully designed and constructed if they are to function satisfactorily on a long-term basis. Their capacity should be over-designed in the event that leakage develops. Where the embankment contains zones of material having significantly different gradation, or where the gradations of the foundation and embankment materials differ markedly, the zones of different material must be separated by filter zones to prevent piping and subsequent subsurface erosion. The filter must meet 2 requirements: it must be more permeable than the adjacent finer soil that it is protecting so that it will drain freely, and it must have a gradation designed to prevent passage of the soil particles into the drainage layer. Particular care must be taken that

segregation does not occur during construction. Details of filter design criteria can be found in standard soil mechanics textbooks, or in the CANMET Manual (1977a).

**Impervious Layers**-Where the embankment is constructed using the downstream method, it may be necessary to seal the inside face with an impervious membrane or zone of fill to reduce seepage from the pond through the dam. It is preferable to deposit the slimes on the inside face rather than to deposit them in the upstream area of the pond with the pond water directly against the inside face of the retaining embankment. In the latter case, a flatter inside face is required to allow for the potential of suddenly drawing down the pond water level.

The amount of seepage through the foundations can be reduced by placing an impervious blanket on the inside of the pond. This may consist of slimes or of fine grained overburden. If the pervious soils in the foundation are shallow, a core trench backfilled with impervious soil may be used. Where the dam is located on an impervious foundation or an impervious geologic barrier outcrops downstream, a small collection dam may be constructed downstream and the water that seeps through the embankment may be returned to the tailing pond by pumping.

**Injection and Pumping Wells**-Where the tailing basin is located over thick pervious deposits, positive pollution control can be accomplished by developing a system of injection and pumping wells downstream of the retaining embankment. The injection wells are located downstream of the pumping wells. Uncontaminated water is pumped into the injection wells and the contaminated water is extracted from the pumping wells. By maintaining the piezometric elevations at the injec-

tion wells moderately above those at the pumping wells, a hydraulic barrier will develop which will cut off the escape of tailing pond seepage. Methods of seepage control are summarized in Figure 25.

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Figure 25

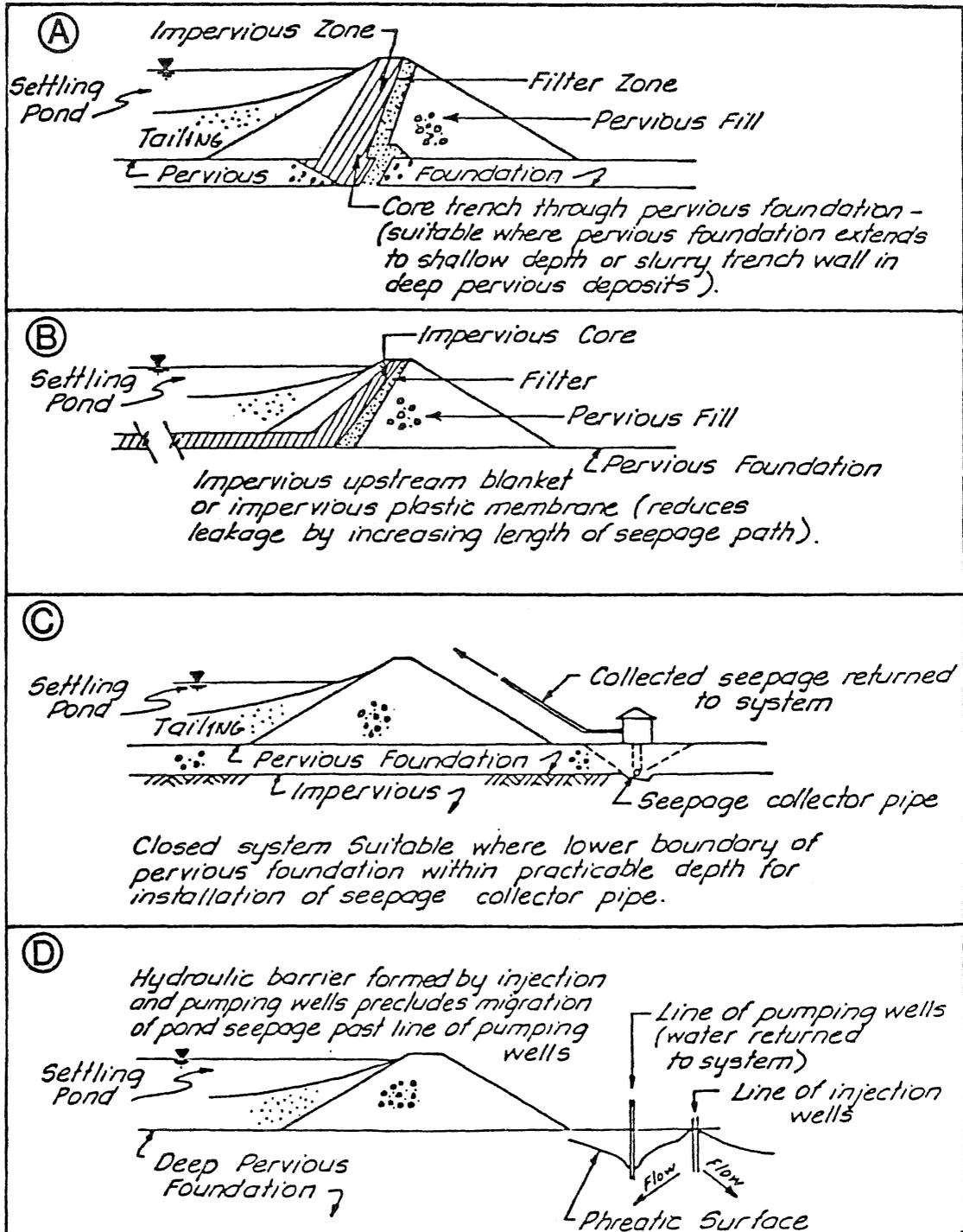
If artesian pressures develop in the foundation or below the toe of the embankment, there is a danger of piping and instability developing. This problem can usually be controlled by the installation of pressure relief wells in the foundation located at the toe of the dam. The spacing, depth, and design of the wells is dictated by the soil stratification, permeability, and water pressures. Experience documented by the U.S. Army Corps of Engineers (1963) for hydro and storage dams is invaluable in determining design and construction requirements. Alternatively, it may be possible to control potential erosion and piping by construction of weighted filters extending outward from the downstream toe.

Cut-Off Trenches-Grout curtains have been used partially to intercept seepage on many earth dams. Because of the variable success and usually high cost, this method of seepage control is not recommended for tailing disposal. Other methods that may warrant consideration for special conditions include sheet pile cut-offs or slurry trenches. The bentonite-slurry trench method of producing an impervious cutoff wall requires excavation of a trench to bedrock or to an underlying impervious soil deposit. The material on each side of the trench is retained in position by stabilizing the excavation with a heavy slurry of bentonite, which is poured into the trench as excavation proceeds. When the trench excavation has progressed sufficiently so that backfilling operations will not interfere with excavation, the bentonite slurry is progressively displaced with impervious fill material to form the cutoff wall. Slurry trench walls are dug

FIGURE 25

# SEEPAGE CONTROLS

SOURCE: GOLDER, 1978



with a large clam shell machine and backfilled with either a soil-bentonite or cement-bentonite mixture. The maximum depth of installation is generally limited to about 90 ft by the capacity of the digging machines. An example of a cement-bentonite slurry trench wall used as a foundation cut-off beneath a tailing embankment was a project at Cleveland Cliffs Iron Company's Tilden Mine near Marquette in the upper peninsula of Michigan. Costs for this wall, 2 ft thick, were quoted as \$3.64/ft<sup>2</sup> for walls up to 40 ft deep and \$4.10/ft<sup>2</sup> for walls up to 80 ft deep.

Plastic Liners-Increasing use is now made of plastic liners in sealing tailing basins to contain toxic substances, particularly if naturally occurring impervious clay soils are not readily available. A useful study on various types of liners including natural and treated soils, asphalt treatments, and plastic materials was carried out by the U.S. Environmental Protection Agency (EPA 1975). The report discussed costs, methods of installation, and deterioration of plastic membranes. If plastic or bituminous liners are designed as stabilizing features in a retaining embankment, great care has to be taken in installation; excessive settlement can stretch and tear a membrane, likewise clumsy handling with machinery can puncture a plastic liner.

An interesting use of a PVC membrane in a tailing disposal system in Minnesota is at the Minorca Taconite mine owned by Inland Steel Mining Company (Gubbe 1977). The use of a plastic membrane in this instance is to limit the amount of seepage.

Basins Overlying Peat--Of special interest in the copper-nickel region of Minnesota is the construction of tailing basins on peat. It is quite feasible to build embankments on peat, but organic soils should not be included in the

body of the embankment. Peat is a highly deformable material even after consolidation under high loads. The permeability of fibrous peat in its natural state can be quite high in an unloaded condition, but permeability decreases dramatically under loads such as that imposed beneath the base of an embankment. In general, conventional methods of seepage control are acceptable but with certain precautions. Because of the high deformability of peat, rigid vertical cut-offs and pressure relief wells can be fractured by load being transferred from the embankment to the cut-off or wells. Also, the use of thin impervious clay zones to control seepage through embankments placed on peat should be avoided unless the clay is well graded, such as a till; if cracking occurs in the impervious zones as a result of excessive settlements of the peat, well-graded material are self-healing and the cracks fill up with soil.

The benefits of permanent seepage control from a water quality standpoint, employing sealed basins that do not allow for drainage and consolidation of the tailing, should be weighed against the loss in land usage resulting from the softness of tailing in the basin. Downward seepage, on the other hand, promotes rapid consolidation of the tailing. Thus, the ground becomes firm and useable. Moreover, in this condition it easily becomes capable of supporting a layer of waste rock, about 3 to 5 ft thick, which is a positive, permanent, and relatively maintenance-free method of fugitive dust control.

Multiple Cell or Basin Approach--To meet water quality standards, the under-seepage can be minimized by using a number of small basins in sequence rather than a single large basin. This would result in a need for extra retaining embankments at an added cost, but it may be cheaper than using elaborate seepage control measures. Once an allowable level of seepage egress has been determined based on water quality standards, the maximum size of basin can be calculated

using an estimate of the likely underseepage. Such an approach, however, is very site-specific and a generalized approach could be totally misleading in practice.

The concept for limiting uncontrolled seepage is that a series of smaller ponds would be operated in sequence, rather than building a single pond of large areal extent. On decommissioning of each small cell the seepage would still continue, although it would tend to decay over a number of years. After decommissioning the basins would be drained and consolidated sufficiently for final reclamation.

Water System Considerations--The tailing basin contains a pond which serves to clarify the water and also acts as a retention or storage area to even out fluctuations in the overall water management system. Generally speaking, the pond should provide 5 days storage capacity and a minimum surface area of 15 acres/1,000 mtpd of tailing for proper clarification. For example, for a  $20 \times 10^6$  mtpy operation, pond water storage should contain a minimum of  $186 \times 10^6$  gal of water (based on a plant water requirement of 650 gal/mt ore treated) and cover an area of at least 830 acres. However, based on usual tailing basin operations, it is estimated that 80% of the basin will be covered by water for dust control and water management, supplying many times the minimum pond area required for proper mill operation and water clarification.

A system is also installed to remove clarified water from the settling pond and return it to the processing system. Several approaches are possible: 1) a decant tower and pumping system; 2) a syphon and pumping system; or 3) a pump barge and pumping system. Each approach has its own advantages, but the best method for reliability and flexibility appears to be the pump barge. This system can be easily moved as the settling pond is shifted, it depends solely on

electrical power for operation, and it can be protected during adverse winter operating conditions. On the other hand, the decant tower and syphon systems do not depend on electricity for operation and are more permanently located within the dam complex. However, they are subject to failure as the decant tower system can be fractured by shifting ground and thus be cut off from the recycle water pumping system, and the syphon system can lose its syphon and must be manually reprimed to return it to normal operation.

Since all 3 methods require electrical power to recirculate the water to the processing plant, the system least subject to other types of failure appears best. This feature, plus the added flexibility of being able to easily relocate the intake, makes the pump barge the most probable method of reclaiming water in a tailing pond system.

3.2.4.5 Tailing Basin Costs--The major factors influencing tailing basin capital and operating costs are:

- 1) Embankment
- 2) Seepage Control
- 3) Fugitive Dust Control
- 4) Reclamation
- 5) Tailing Transport and Water Reclamation

In order to analyze the implications of these factors on the overall tailing disposal cost and processing cost, an idealized tailing basin design had to be used. For example, it was assumed that construction takes place on a flat plane with no natural relief. Thus, to contain a given volume of tailing, dam design dictates a certain height and cross-section, and a given length to provide the necessary volume. Of necessity, calculations on this basis result in maximum

figures as in almost every case an operating company would take advantage of local topography and thereby reduce dam requirements significantly.

Several examples will serve to illustrate variations possible in tailing basin and dam design. For a  $20 \times 10^6$  mtpy open pit copper-nickel operation in Minnesota, with a 30 yr life, a tailing basin which would ultimately have an average depth of 70 ft of tailing material would cover an area of 4,016 acres, assuming a final average tailing density of  $90 \text{ lb/ft}^3$  (solids). For such containment, a dam 75 ft in height would be sufficient to protect the structure from flood conditions and wave action. Such a dam would surround the basin and could vary from almost 9 mi in length for a circular basin design to about 13 mi in length for a rectangular design with the long side 4 times the short side. Irregular basin shapes could require dam lengths somewhere between the 2 examples or, in fact, longer than the 13 mi length given. By using the natural contours of the land, dam designers could provide adequate tailing storage capacity while reducing the dam length by  $1/3$  to  $1/2$ .

Embankment--Engineering construction costs are very difficult to assess on a generalized basis as each installation has costs peculiar to the specific location and techniques adopted. Loading, hauling, and any processing of material required for embankment construction is a significant and sometimes major part of basin costs. Techniques requiring a specialist contractor to perform the work, perhaps using special equipment not available locally, can involve very large fixed mobilization and demobilization expenses; the cost of such work therefore varies widely on a unit basis depending on the total quantity involved.

Very often a mining company can construct an embankment itself at significantly lower cost than by employing an outside contractor. Apart from the contractor's

profit, the company may be able to use equipment that is being under-utilized elsewhere in the mining operation. Also, differences in union agreements dealing with inside and outside construction workers can have a significant influence on overall labor costs; as an inside worker may not be allowed to work outside if the inside work is being held up. Mining companies have more latitude with their union labor and can generally direct them to where the demand is, providing there is no infringement of job classification.

The least expensive approach to embankment building is to use mill tailing sand. At Brenda Copper Mine in British Columbia, for example, the cost of the embankment which will ultimately utilize about  $32 \times 10^6 \text{ yd}^3$  of tailing sand was estimated to be not more than 5 ¢/yd<sup>3</sup>. The tailing flows under gravity from the mill to a cyclone station located on one abutment, and from there is distributed hydraulically to the embankment. Double stage cyclones are used. The first stage works by gravity alone; the second stage consumes some energy. The only real costs are labor, cyclone equipment, power to operate the second stage cyclone, and some dozing of the sand. Compaction is not used, waste rock was used in a starter dam, and some local impervious borrow was used for an upstream blanket and for filters and underdrains. The cost of 5 ¢/yd<sup>3</sup> applies only to the tailing sand, and it is based on 1970 costs of equipment.

Where tailing is used for construction, it is reasonable to assume that the average tailing embankment cost will be about 15-20 ¢/yd<sup>3</sup> including spigoting or cycloning, labor, and dozing equipment (Golder 1978). These costs are exclusive of tailing transportation costs (discussed later in this report), and any measures for foundation seepage cutoffs, dust control, and reclamation.

Tailing material, if properly prepared, is extremely competent as dam building material. Preparation consists mainly of sizing to provide the coarse fraction

relatively free of slimes, physically located near the dam construction area. Coarse tailing is also useful as back fill material in underground mining operations where it is used to fill voids left by ore removal and to provide support for overlying or adjacent geologic structures (see Chapter 2 of this report).

If both back filling and dam building are desired, end uses of the coarse tailing material, it must be determined how much coarse tailing will be available and how much is required in both end uses. If sufficient tailing will be produced to supply both back filling and dam building, there is no problem. If insufficient material will be produced, a trade-off situation exists in which costs of other support systems for the underground mine and dam construction of borrow material must be compared, with economics dictating the final outcome. Other factors such as environmental concerns and secondary mineral recovery from tailing material must also be considered.

In the case of an underground mine resulting in 100,000 mtpy of copper and nickel metal, only 10-15% of the coarse tailing would be needed for dam construction, leaving more than adequate material for back filling mined out areas.

If tailing material is not the main element in the embankment construction, substitute material must be located, prepared, and brought to the site at an additional cost considerably greater than for the tailing material. Waste rock from the mine would be suitable, but costs for additional hauling, processing, and placing may be prohibitive. Borrow material, if located close to the construction site and of the proper characteristics, may be suitable, but will also involve additional mining and processing expenses.

Embankment materials, whatever the source, must be properly contoured and compacted to provide the desired dam configuration. With tailing material this task is relatively easy as it naturally drains and compacts as it is placed on the dam crest. Borrow material, however, will require a great deal more handling, compacting, and contouring to achieve the same result.

The foregoing discussion is presented to show some of the variables that can enter into embankment fill costs. Clearly, only a range of costs can be given in the absence of a specific development. The following costs from Golder are based on 1977 dollars. Items for haulage are broken down so that construction using borrow fills from remote sources can be assessed.

1) Excavation and loading of sand and gravel, or till as borrow material can be estimated to cost approximately  $\$1.30/\text{yd}^3$ , assuming that the material can be dug and loaded by a front-end loader or dragline. If ripping or light blasting is needed to remove hard till, the excavated cost should be taken as  $\$1.80/\text{yd}^3$ .

2) Because waste rock from the mine is a necessary excavation whether the material is used or not, the cost of such excavation charged to tailing disposal should be assumed to be zero, unless processing or special blasting techniques are used to yield a material of a more suitable gradation for embankment construction.

3) Haulage costs from borrow areas to the embankment site, or overhaul of waste rock should be taken at  $\$0.50/\text{yd}^3$  per mile of haul. (Overhaul is the additional distance involved in hauling to the embankment rather than to a disposal area.)

4) Placement and compaction of fill in layers in a tailing embankment should be taken as:

Waste rock -  $\$0.50/\text{yd}^3$  (assumes no compaction needed)

Sand and gravel -  $\$0.30/\text{yd}^3$  (assumes only light compaction by  
dozing and spreading)

Cohesive clay till -  $\$0.80/\text{yd}^3$  (assumes good compaction to produce  
impervious seal)

The above costs are based on the assumption that the work would be of high quality performed by a skilled independent earthwork contractor. Mobilization of equipment, camp costs, and so on, are included, but it is assumed that a sizeable piece of embankment construction not less than about  $1.0 \times 10^6 \text{ yd}^3$  would be undertaken.

For design purposes, the dam cross-section shown in Figure 26 was dimensioned based on typical ratios used in the industry. It must be pointed out that these dimensions have not been verified hydrologically to insure that the water surface will in fact not exceed an acceptable level with respect to the dam. This procedure is complicated and can only be done on a case-by-case basis, and then must be verified by monitoring during operation. Dimensions used here, although typical, are therefore by no means fixed.

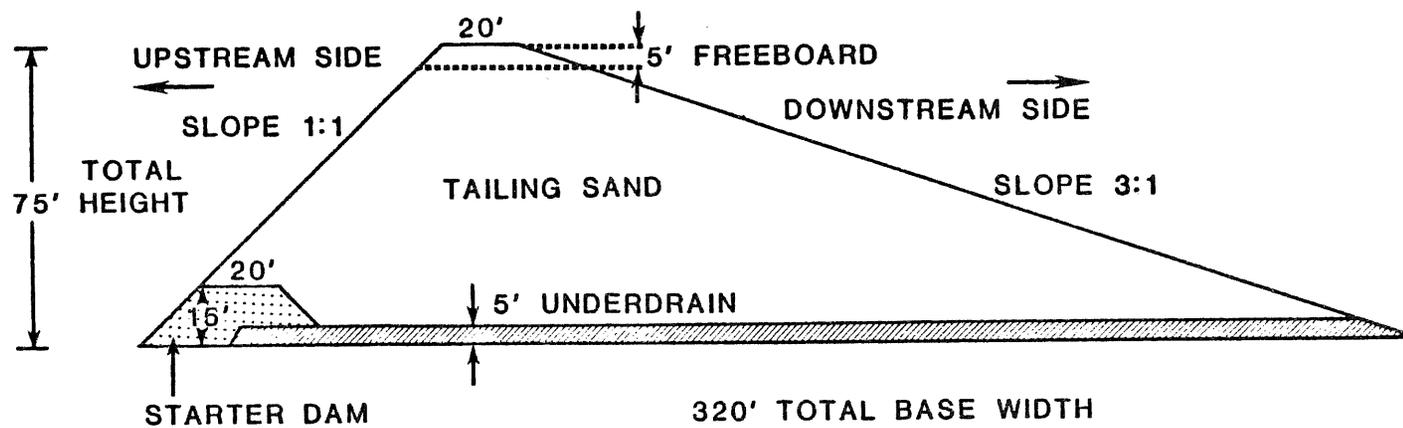
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Figure 26

For the example of a 75 ft tailing dam sufficient to contain an average of 70 ft of tailing material from the  $20 \times 10^6$  mtpy operation mentioned earlier, using downstream construction, dimensions shown on Figure 26 were used. The starter dam contains  $46 \text{ yd}^3$  of material per yard of length, the drain contains 164

FIGURE 26

TYPICAL TAILING DAM CROSS SECTIONAL AREA  
DOWNSTREAM CONSTRUCTION



yd<sup>3</sup> per yard of length, and the tailing dam material itself consists of 1,207 yd<sup>3</sup> per yard of length.

In total then, the entire embankment contains 1,417 yd<sup>3</sup> of material per yard of length, or  $2.5 \times 10^6$  yd<sup>3</sup> per mile of length. Based on a dam length of 8.9 mi for a circular shape, a total of  $22.2 \times 10^6$  yd<sup>3</sup> of material would be needed to construct the embankment, divided as follows:

Starter Dam	$0.7 \times 10^6$ yd <sup>3</sup>	(3%)
Drain	$2.6 \times 10^6$ yd <sup>3</sup>	(12%)
Tailing Sand	<u><math>18.9 \times 10^6</math> yd<sup>3</sup></u>	(85%)
TOTAL	$22.2 \times 10^6$ yd <sup>3</sup>	(100%)

As shown previously, these volumes could vary upwards as much as 40% by changing the configuration of the basin from circular to rectangular.

Golder information discussed previously places the following approximate costs on the above sections of the total dam construction:

Starter Dam	- \$2.60/yd <sup>3</sup> (including excavation, loading, 2-mi haul, placement, and compaction of the sand or gravel)
Drain	- \$1.50/yd <sup>3</sup> (including 2-mi overhaul and placement of the waste rock)
Tailing Sand	- \$0.20/yd <sup>3</sup> (including placement cost only)
Total Average Material	\$0.44/yd <sup>3</sup> for the above design.

From Figure 26 it is obvious that the total volume changes with dam height. Since the starter dam is designed initially to contain a specific amount of tailing regardless of total dam height and the underdrain must transport a specific amount of water through the total dam section, both will increase somewhat

in volume as the total dam height is increased. However, since these volumes are small compared to the tailing sand volume, they can be assumed to be essentially constant for estimating purposes. Therefore, as the dam height is increased (to decrease the total surface area required or to contain more total material) the dam volume per unit length increases and the average unit cost approaches the tailing sand unit cost. In contrast, if the dam height is decreased, requiring a larger basin area, the dam volume per unit length decreases and the unit cost increases due to the larger proportion of the high cost commodities involved.

Figure 27 shows how dam characteristics vary with dam height. Multiplying the unit cost for the 75 ft dam by the volume per mile and the total length required results in the cost for the embankment. Thus, for the  $20 \times 10^6$  mtpy example:

$$(\$0.44/\text{yd}^3)(2.5 \times 10^6 \text{ yd}^3/\text{mi})(8.9 \text{ mi}) = \$9.8 \times 10^6 \text{ total cost}$$

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Figure 27

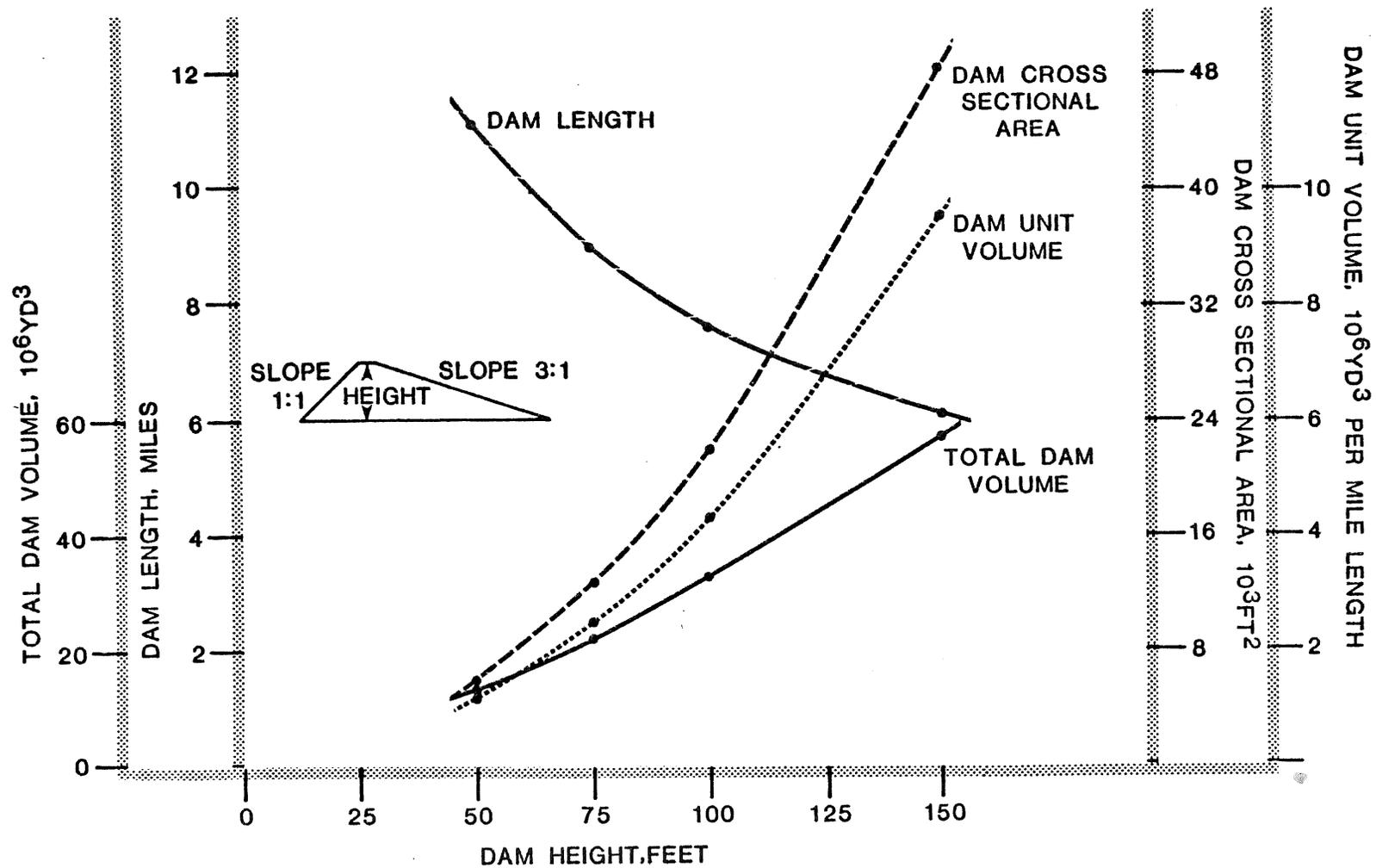
Seepage Control--Cost estimates can be made for various methods of seepage control using the above unit figures, if the control is to be a shallow trench backfilled with compacted till, or if an upstream blanket is employed to collect and direct seepage. A specialized technique such as a bentonite slurry trench cutoff may also be used to prevent seepage through a dam. Such walls can be quite thin, perhaps no thicker than the width of trench in which a digger can operate, say 2 to 3 ft. For planning purposes where deep cutoffs are considered, a cost of \$5 per vertical square foot of wall should be used.

Cutoff trenches key the bedrock to the tailing dam through the overburden, such that the water is contained within the basin. The bentonite slurry forms the

FIGURE 27

VARIATION IN TAILING DAM CHARACTERISTICS WITH DAM HEIGHT

20 X 10<sup>6</sup> MTPY OPEN PIT MINE MODEL



permanent seal, forcing any seepage water over the cutoff, through the dam, and to the surface, allowing for collection with simple surface trenching and ponding.

For the previous example of a  $20 \times 10^6$  mtpy operation, and assuming 30 ft of overburden requiring cutoff trenching, the total length of trench would be 8.9 mi and the vertical square feet of wall would be:

$$30 \text{ ft} \times 8.9 \text{ mi} \times 5,280 \text{ ft/mi} = 1.4 \times 10^6 \text{ ft}^2$$

Using Golder's suggested figure of \$5 per vertical square foot of wall with the deep cutoff of 30 ft, the total cutoff system cost would be:

$$1.4 \times 10^6 \text{ ft}^2 \times \$5.00/\text{ft}^2 = \$7.1 \times 10^6$$

This cost is directly proportional to the thickness or overburden and length of the dam and, as shown, is a very expensive item.

The cost of plastic liners for impervious seals depends upon the type of material used, the thickness of the membrane, and the method of placement. If the membrane is to function as a truly impervious seal, joints have to be sealed and the membrane has to be protected with a 6 to 12 in. layer of fine soil; for proper functioning very careful handling and construction is needed and sufficient money should be allowed for in any contract bid. Normally, a lining would be used only to seal the inside face of the embankment to prevent seepage through the dam, but if the basin is underlain by pervious deposits or designed to remain permanently flooded, the lining might be called for over the entire area.

Useful figures based on 1972-73 costs are given in an EPA report (1975). For planning purposes, a PVC membrane can be assumed to cost  $\$0.010/\text{ft}^2/\text{mil}$

thickness. Thus, if a membrane 20 mils thick is used, the cost of the sheet would be \$1.80/yd<sup>2</sup>. The cost of Hypalon, which is more durable, should be taken at \$0.014/ft<sup>2</sup>/mil thickness. In addition to the cost of the plastic, an installation cost of at least \$0.50/yd<sup>2</sup> should be included.

For any tailing basin application, plastic liners can be used on both the inside face of the dam to seal the embankment, and over the surface of the basin to seal the entire facility. For example, in the 20.00 X 10<sup>6</sup> mtpy operation the dam will have a total inside surface area of 62.2 X 10<sup>3</sup> yd<sup>2</sup> per mile of dam. Using a 20 mil Hypalon liner at \$3.02/yd<sup>2</sup> installed, the total cost of lining the dam would be \$187.8 X 10<sup>3</sup>/mi, or a total of \$1.7 X 10<sup>6</sup>.

For the same example, a total surface pond liner of the same material would cover 4,016 acres and cost \$58.7 X 10<sup>6</sup> installed, or \$14.6 X 10<sup>3</sup>/acre. The less expensive PVC membrane in the same situation would cost \$155.5 X 10<sup>3</sup>/mi or \$1.4 X 10<sup>6</sup> in total for lining the dam, and \$11.1 X 10<sup>3</sup>/acre or \$44.7 X 10<sup>6</sup> in total for lining the basin surface. Obviously, basin lining would be very expensive.

All of the cost data detailed above is summarized in Table 11 and is exclusive of all transportation charges for the tailing material itself. The intent is to show costs attributed to the disposal area alone. Such costs are not necessarily capital alone but a combination of both capital and operating without differentiation in this discussion.

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Table 11

The costs for dust control and reclamation as shown in Table 3 are included in the above data as optional features. Minimum treatment for dust control would

Table 11. Summary of tailing basin and dam costs.<sup>a</sup>

DAM DESIGN ASSUMPTIONS

Downstream construction with 1:1 slope on upstream side and 3:1 slope on downstream side. Total height is 75 ft with 5 ft of freeboard and 70 ft of tailing embankment. Dam is circular in shape and contains a starter dam of local sand or gravel, an underdrain of unmineralized mine waste rock, and the bulk of the dam as tailing sand.

REQUIRED DAM COSTS

Starter dike @ \$2.60/yd <sup>3</sup>	\$0.27 X 10 <sup>6</sup> /mi
Underdrain @ \$1.50/yd <sup>3</sup>	0.42 X 10 <sup>6</sup> /mi
Tailing sand @ \$0.20/yd <sup>3</sup>	0.42 X 10 <sup>6</sup> /mi
Total @ \$0.44/yd <sup>3</sup>	\$1.1 X 10 <sup>6</sup> /mi

OPTIONAL DAM & BASIN FEATURE COSTS

Cut-off trench (30 ft deep)	\$0.79 X 10 <sup>6</sup> /mi of dam length
Dam liner	\$0.19 X 10 <sup>6</sup> /mi of dam length
Basin liner	\$14.62 X 10 <sup>3</sup> /acre of basin
Dust control and reclamation	\$150-\$2250/acre

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<sup>a</sup>See text for a description of each item and assumptions used in cost estimates.

be periodic watering of the exposed surface, but this treatment is applicable only during the active life of the property. More permanent dust control and revegetation, as shown in the table, would range in costs from \$150 to \$2,250 per acre, up to  $\$9 \times 10^6$  for a  $20 \times 10^6$  mtpy operation. Additional discussion of this subject can be found in the reclamation section of Chapter 2.

Tailing and Recycle Water Transportation--Transportation of the tailing material to the basin and return of recycle water to the mill were also subjects detailed by the Golder report. The following discussion summarizes their results.

Tailing material produced in a milling operation must be removed from the plant site and disposed of in an acceptable manner. Likewise, as much of the contained water as possible must be collected and returned to the plant for reuse in the system.

Golder and Associates estimated tailing disposal and recycle water return systems for operations ranging from 12.35 to  $20.00 \times 10^6$  mtpy ore. The capital and operating costs were estimated for distances between the plant and the basin ranging from 1,000 ft to 10 mi. Their analysis found that pumping distances of 1,000 ft and one mile were less expensive when the tailing was transported directly as is. However, with distances greater than one mile, the economics indicated thickening prior to pumping to be less expensive. Both flowsheets are shown in Figure 28.

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Figure 28

Basic design criteria detailed by Golder included:

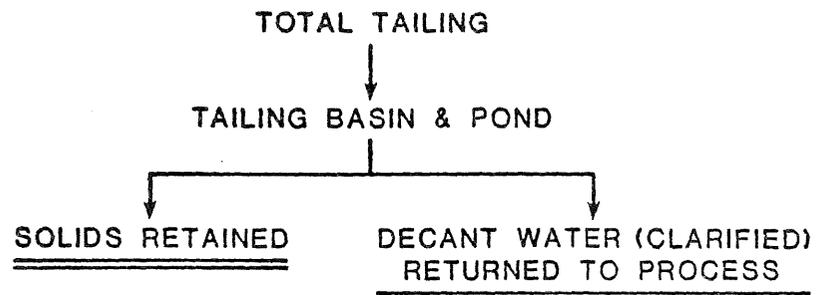
- 1) Metal, fixed speed tailing pumps, staged as necessary.

FIGURE 28

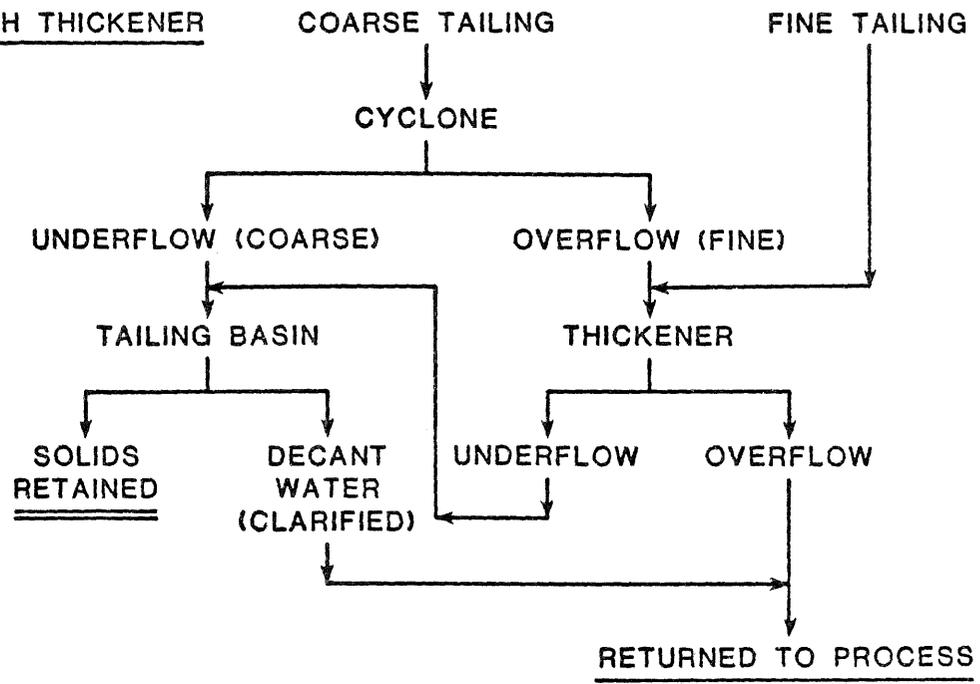
# TAILING DISPOSAL AND WATER RECYCLE SYSTEMS

(SOURCE: GOLDBER, 1978)

## WITHOUT THICKENER



## WITH THICKENER



- 2) One steel tailing line operating and one spare line, both on the surface. Lines one mile and greater are insulated and heat traced.
- 3) Tailing line velocities 4.1-6.4 fps.
- 4) Tailing pumped directly over distances of 1,000 ft and one mile, thickened for pumping distances greater than one mile.
- 5) Reclaim pumps are vertical turbine types, operating from one pumphouse at the basin.
- 6) One polyethylene reclaim pipeline on the surface; lines one mile and greater are insulated and heat traced.
- 7) Reclaim line velocities 4.4-7.8 fps.
- 8) Topography assumed to be flat with 150 ft lift at the dam.

A summary of capital and operating costs generated by Golder appears in Table 12. Analysis showed a cost improvement in thickening the tailing when pumping over one mile and this feature was used in those cases. As can be seen, there is a 12- to 14-fold increase in amortized capital and operating cost when the tailing basin is remotely located 10 mi from the plant instead of 1,000 ft from the processing facility. Refer to Volume 5-Chapter 14 for a discussion of the economic impact on the mining operation of such cost increases.

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Table 12

Overall tailing disposal systems discharging the waste one mile from the modeled processing plants range in capital cost up to  $\$16 \times 10^6$  and operating cost up to  $\$0.56 \times 10^6/\text{yr}$ . Of course, variation in pumping distance causes drastic

Table 12. Tailing transportation and water reclamation cost data.

PLANT SIZE, 10 <sup>6</sup> mtpy ore	PUMPING DISTANCE PLANT TO BASIN	UNIT \$10 <sup>6</sup> /mi	CAPITAL COST		OPERATING COST \$10 <sup>6</sup> /yr	TOTAL AMORTIZED CAPITAL & OPERATING COST, \$10 <sup>6</sup> /yr	TOTAL COST <sup>b</sup> INCREASE FACTOR, WITH PUMPING DISTANCE
			TOTAL \$10 <sup>6</sup>	AMORTIZED <sup>a</sup> \$10 <sup>6</sup> /yr			
12.35	1000 ft	8.41	1.59	0.20	0.27	0.47	---
	1 mi	4.94	4.94	0.62	0.45	1.07	2.3
	5 mi	3.30	16.50	2.07	1.30	3.37	7.2
	10 mi	3.16	31.60	3.97	1.92	5.89	12.5
20.00	1000 ft	11.15	2.11	0.26	0.35	0.61	---
	1 mi	6.53	6.53	0.82	0.56	1.38	2.3
	5 mi	5.26	26.30	3.30	1.87	5.17	8.5
	10 mi	4.49	44.90	5.64	2.76	8.40	13.8

SOURCE: Golder (1978).

<sup>a</sup>Amortized capital over 20 years at 11% of total capital per year.

<sup>b</sup>Amortized capital plus operating cost increase factor, relative to 1,000 ft case.

changes in both capital and operating expense; an increase in capital of 3 to 4 times and an increase in operating of 4 to 5 times when changing the pumping distance from 1 to 10 mi.

For the base case situation of one mile pumping distance and processing plants large enough to feed a smelter/refinery complex producing 100,000 mtpy of copper and nickel metal, tailing disposal represents:

1.5 to 2.1% of total capital costs

6.6 to 6.9% of processing capital costs

0.3 to 0.5% of total operating costs

1.2 to 1.5% of processing operating costs

3.2.4.6 General Practice of Effluent Control in Tailing Basins--Basically, 3 kinds of waste flows may be encountered in mine, mill, and smelter/refinery operations, namely, mine drainage, mill and smelter/refinery process effluents, and contaminated surface drainage, including tailing basin seepage. By far the most common means of treating these wastes is to discharge them into a tailing basin in which the suspended solids (tailing) are settled out, the heavy metal ions in the water may be chemically precipitated and settled out, and the solids are retained in perpetuity. This approach is used in locations where large tonnages of solids are involved, such as near a mill, However, at mines where no mill exists or in the case of an isolated smelter and/or refinery, mine water, plant discharge water, and contaminated surface drainage may be chemically treated in lagoons or handled in a mechanically operated treatment facility.

The mining industry is perhaps unique in the quantity of waste solids generated during the recovery of its products from ores. At many mines, particularly underground mines, the mill tailing wastes form by far the largest portion of

the solid wastes produced. Thus, the disposal of tailing exerts a major influence on the design of a mine-mill complex.

Originally, and until recently, the sole purpose of a tailing impoundment area was to prevent the discharge of substantial amounts of solids into receiving water courses. This, of course, is still a primary requirement of a tailing basin, but over the past few years the situation has changed greatly as a result of environmental regulations and the growing knowledge of the damaging effect on aquatic and human life of various soluble constituents present in mining wastewaters. In many instances, the tailing basin is now also used to provide a large degree of chemical treatment to the contained liquid wastes. A typical basin may be required to perform a large number of functions including:

- 1) removal of tailing solids by sedimentation;
- 2) acid neutralization;
- 3) formation of heavy metal precipitates (hydroxides);
- 4) sedimentation of metal precipitates;
- 5) perpetual retention of settled tailing and precipitates;
- 6) stabilization of oxidizable constituents (e.g. thiosalts and flotation reagent residuals);
- 7) balancing action for fluctuations in influent quality and quantity;
- 8) storm water storage and flow balancing;
- 9) water treatment facility for recycling water to the mill;
- 10) sedimentation and perpetual retention of mineral fibers present in the ore or created in the milling process.

Tailing basins have been virtually universally accepted as the primary means of treatment due to their ability to effectively perform most of the above listed functions at a relatively low cost.

Counteracting the treatment efficiency often achieved in tailing basins is the lack of control that can be exercised over them, particularly in view of conflicting requirements. For example, sedimentation requires quiescent conditions whereas oxidation of soluble constituents is improved by turbulence.

The relative advantages of tailing basins as treatment facilities are:

- 1) perform large number of treatment functions at a low cost;
- 2) can achieve high treatment efficiencies and produce an acceptable effluent quality in many situations;
- 3) are often the only practical means of solids disposal;
- 4) have large retention times producing a balancing effect on effluent quality;
- 5) have large surface areas which aid in oxidation and evaporation;
- 6) can often be constructed using mining equipment and materials;
- 7) require little operating expertise, normally;
- 8) are a commonly used treatment method familiar to the industry.
- 9) cover large surface areas and thus contribute high water volumes from precipitation to the overall water balance. Precipitation will have a diluting effect on the quality of the reclaim water where high concentrations of certain dissolved materials would be detrimental to the process. The large quantity of precipitation will also reduce make-up water requirements.

Disadvantages of tailing basins as treatment facilities are:

- 1) lack responsive means of control, so it is difficult to optimize the large number of processes performed;
- 2) create a potentially severe rehabilitation problem if tailing is acid forming;
- 3) are often difficult to isolate from contributing drainage areas--storm water influences retention time;
- 4) are subject to climatic variations, particularly thermal skimming and seasonal variations in bio-oxidation efficiency;
- 5) are often difficult to operate in a manner which will ensure good flow distribution;
- 6) require careful control of seepage through dams;
- 7) installation is expensive in many situations due to the high cost of retaining structures and the need for good engineering design to prevent problems such as liquefaction and piping, which might be followed by catastrophic failure of the embankment if allowed to occur;
- 8) are often subject to dust problems unless surface stabilization is achieved by revegetation or by the use of chemical binders or rock covers.

#### 3.2.4.7 Innovative Tailing Disposal Methods

Central Discharge System--Conventional tailing disposal is by perimeter spigotting from the retaining embankments. An alternative arrangement, which is amenable to flat lying terrain such as in northeastern Minnesota, is central discharge. This is a relatively new and untried concept, although mines at

Timmins, Ontario, have used it since 1965. Its main proponent is Robinsky (1975), who claims that it is not only less costly to construct and operate than the traditional approaches, but also that it provides better solutions to environmental problems such as seepage control, dam stability, and ease of final reclamation. The method is shown diagrammatically in Figure 29. The tailing is discharged from a central location and allowed to fan out in all directions to form a cone, the discharge point being raised from time to time as the cone builds up. An average slope of the cone of 3 to 4% seems possible by maintaining the water-solids ratio of the tailing at about 2:1 by weight, or by use of a thickener. In reality, the surface of the cone should be slightly convex, since the coarser particles that settle out first will stand at a steeper slope than the fine particles settling out farther away. A low impervious perimeter dam is required to contain the mill water for reclaim. The dam need only be about 10 ft high and thus, the volume of embankment building can be greatly reduced; waste rock with an inside clay seal would be one solution. The method seems relatively easy to operate and regulate, because no mechanical handling of the tailing is involved and only a limited length of header piping has to be periodically raised compared to the traditional method of perimeter discharge. For low perimeter retaining embankments, the volume of tailing contained is obviously very large for an area of, say, 2 mi in diameter. This is because the surface of the tailing rises in the form of a cone above the level of the pond; in the traditional method the tailing is generally below the level of the retaining embankment. With the core projecting above the pond, however, the problems with fugitive dust may be more severe during operation and on abandonment, and the tailing cannot be maintained saturated to prevent this. Experience of winter operation has in some cases been unfavorable. Ice builds up in the cone and the sand sloughs out on melting in the spring, resulting in

flatter angles and reduced storage volumes. The long-term stability of the slopes should be excellent, however.

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Figure 29

Combined Waste Rock-Tailing Basin Storage Concept--Commonly, both waste rock storage piles and tailing basins are revegetated to some extent to reduce dust lift off, water erosion and visual impacts. Such mitigating measures can be carried out during the active life of the operation or at termination, depending on company policy and governmental regulations. Tailing basin material is fine enough in structure to support a surface covering of soil capable of supporting plant life; indeed, with proper additives it may be capable of supporting plant life by itself. On the other hand, waste rock is normally very coarse in size (up to 4 ft) and offers many cavities in a storage pile through which surface coverings would wash and be ineffective in a revegetation program.

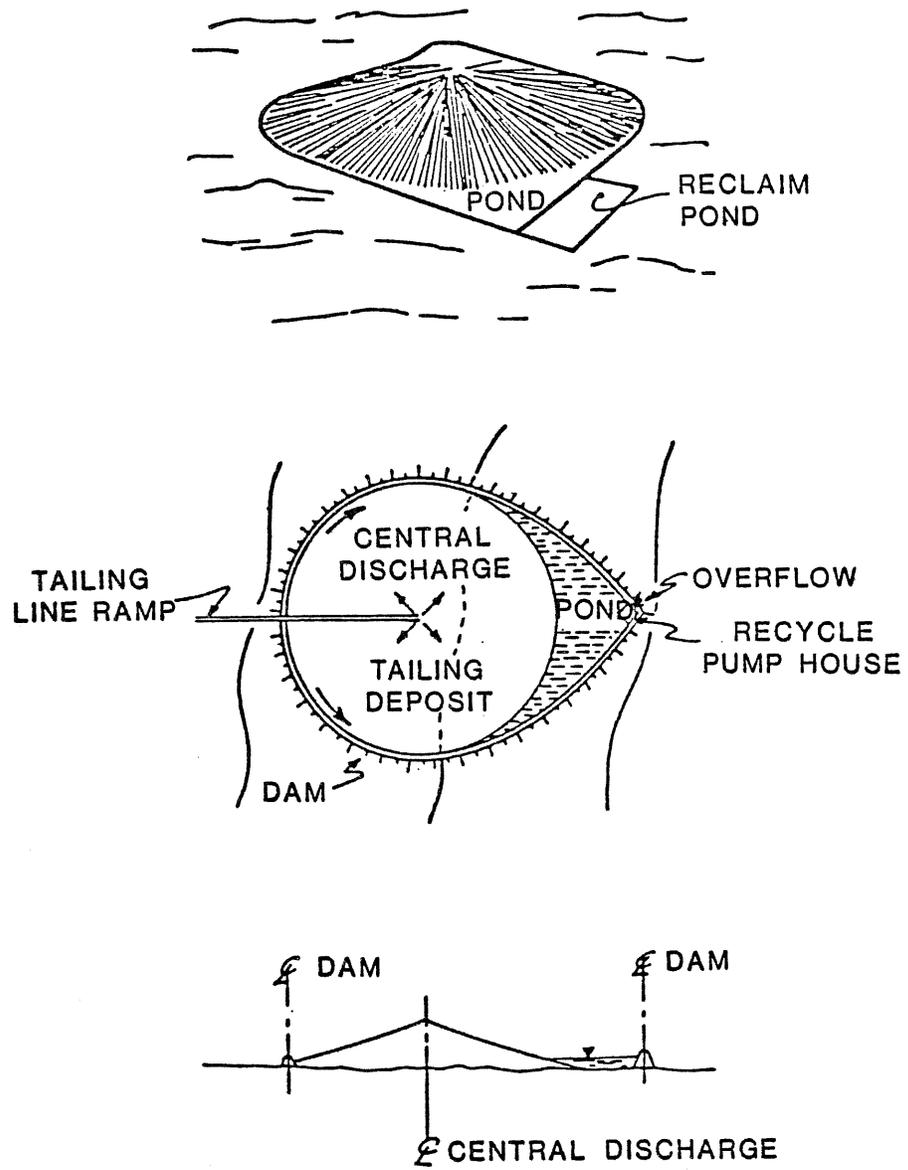
During the Second International Tailing Symposium held in Denver, Colorado, in May, 1978, a subject termed "combined waste rock-tailings storage concept" was discussed by C.D. Brawner of Golder and Associates (Argall 1978). The concept utilizes the void volume of waste rock (up to 40%) to store finer tailing material while building the retaining dam of screened waste. Although the system has not been attempted on a large scale, it offers the potential advantages of:

- 1) less total waste storage volume and area
- 2) more stable waste disposal structure
- 3) waste disposal areas more suitable to rehabilitation
- 4) less exposed material surface area for leaching

FIGURE 29

# CENTRAL DISCHARGE TAILING DISPOSAL

SOURCE: GOLDER, 1978



5) potential buffering capacity of tailing preventing leaching of waste rock heavy metal ions.

If lean ore exists, it would still be stored separately from the waste rock and tailing, as the potential exists of someday processing the material for its contained values.

A disadvantage of combined disposal lies in the potential for future working of tailing material for contained values of other mineral constituents. The addition of waste rock would necessitate additional treatment of size separation and/or reduction before mineral separation could proceed. Also, since the leaching process in the waste rock material is not well understood, it is possible that the resulting release of heavy metals may not be reduced, in which case the control problem might be compounded by the increased volume of material now requiring regulation. Nevertheless, the basic concept is a potentially useful one, worthy of further research.

### 3.2.5 By-Product Recovery from Copper-Nickel Tailing Material

The following discussion on copper-nickel tailing utilization as an economic venture of by-product recovery was summarized from a literature survey prepared for the Regional Copper-Nickel Study by the MRRC (Trethewey 1977). Updated information is included and revisions were made to correspond to the more extensive results of the Study.

Typically, Minnesota Cu-Ni mining and processing operations capable of supporting a smelter-refinery operation producing 100,000 mtpy of metal, would result in 12 to 19 X 10<sup>6</sup> mtpy of tailing material. This material would be ground to at least 65M with a mineralogical composition as listed in Table 13

(see Volume 3-Chapter 2).

---

Table 13

Other than the normal deposition in a tailing basin or partially as a backfill in underground mining operations, the potential exists for economic recovery of mineralogical components of the tailing by standard concentration techniques, such as high intensity, wet magnetic separation. The increased value may pay for the tailing processing only, or if highly successful it may also absorb part of the overall copper-nickel processing costs. In any case, it would have to improve the economic picture of the entire operation to make the additional processing stages an attractive venture.

The potential also exists of concentrating certain undesirable mineralogical components of the tailing, such as residual sulfides, heavy metals, and asbestiform minerals, for separate disposal. Thus, the "harmless" portion of the tailing could be disposed of in the normal manner and the "harmful" portion in an environmentally acceptable manner.

In summary, economic values contained in the tailing minerals which potentially may be concentrated to a salable form are:

- 1) Anorthite as a raw material for the production of aluminum and ceramics.
- 2) Ilmenite as a source of titanium oxide.
- 3) Magnetite as an iron-making raw material.
- 4) Graphite for electrical and chemical industries.
- 5) Olivine for foundry sand and sand blasting.
- 6) Copper-nickel sulfides as additional values to the total system.

Table 13. Mineralogical composition of Minnesota copper-nickel flotation tailing.

<u>MINERAL</u>	<u>APPROXIMATE PERCENTAGE</u>
Anorthite	60
Olivine	19
Pyroxene and Amphibole	14
Biotite	3
Pyrrhotite	1
Chalcopyrite	1
Ilmenite	3
Magnetite	<u>1</u>
TOTAL	100

Other than cursory testwork at the MRRC and by private industry, significant detailed studies have not been conducted on the recovery of these mineral fractions. In many cases, the required purity of the final product is such that normal inexpensive concentration techniques are not effective enough to result in an acceptable product. Even if acceptable products could be economically produced, nearby markets must be available as freight rates to distant outlets could reverse the economic advantage and turn a profitable operation into a losing one.

The most probable mineral fraction recoverable from copper-nickel tailing is its most common constituent, anorthite. Although usually termed anorthite, calcium-aluminum silicate ( $\text{CaO} \cdot \text{Al}_2\text{O}_3 \cdot 2\text{SiO}_2$ ), it is more properly called anorthosite as it contains up to 40% albite ( $\text{Na}_2\text{O} \cdot \text{Al}_2\text{O}_3 \cdot 6\text{SiO}_2$ ). Anorthite contains up to 29% alumina ( $\text{Al}_2\text{O}_3$ ) and therefore is a potential source of aluminum. At present, commercial production of aluminum is possible only from bauxite ( $\text{Al}_2\text{O}_3 \cdot 2\text{H}_2\text{O}$ , approximately), and the U.S. imports the vast majority of the bauxite used in its aluminum industry. The U.S. Bureau of Mines and the aluminum industry are both working on the recovery of aluminum from anorthosite. If the technical and economic problems can be solved, Minnesota's copper-nickel tailing material could become a potential source of aluminum.

The magnetite content of the flotation tailing is less than 1% of the total (actually about 0.5%), which if separable to a high enough purity represents an annual production of 80,000 to 100,000 mt of magnetite ( $\text{Fe}_3\text{O}_4$ ). However, since the magnetite is closely associated with ilmenite and  $\text{TiO}_2$  is a very troublesome impurity in an iron ore burden, it must be determined if the  $\text{TiO}_2$  is in solid solution with the  $\text{Fe}_3\text{O}_4$  (inseparable), or as unliberated ilmenite

grains. Titanium dioxide interferes with the metal-slag separation in the blast furnace production of iron ore and should generally be less than 0.1% of any iron ore fed to the furnace.

High intensity wet magnetic separation could possibly be used to concentrate specific mineral fractions for either economic recovery or acceptable environmental disposal. Testwork at the MRRC indicated possible application of this separation method, and the required magnetic intensity to separate various Duluth Gabbro mineral fractions given in Table 14 also illustrates this possible application.

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Table 14

### 3.2.6 Water System

The water system is perhaps the most critical of the nonseparation phases of the processing operation. Water is used to transport material from one point to another, to suspend material in a container for further treatment, and to transmit chemical reagents to the solids in the system. Sanitary water is necessary to supply fire protection and potable water. Fresh (or make up) water is necessary to replace that lost in the process. A constant supply of water must be assured or the processing system will suffer. Therefore, many dollars are spent on water management areas, control, and water reclamation systems, and on designing the overall process such that it will reflect realistic water requirements.

3.2.6.1 Potable Water--Potable water, for human use and other high quality demands, is necessary for all sanitary facilities in a total system and is generally supplied from wells or local municipal water supplies. Depending on

Table 14. Magnetic intensity required to extract minerals contained in Minnesota Duluth Gabbro rock, using a high intensity wet magnetic separator.

MINERAL	MAGNETIC INTENSITY 10 <sup>3</sup> Gauss
Apatite	14-18
Biotite	10-18
Epidote	14-20
Hornblende	16-20
Ilmenite	8-16
Magnetite	-1
Olivine (fayalite)	11-15
Pyrrhotite <sup>a</sup>	1-4
Serpentine	3.5-18
Titaniferous-Magnetite	0.5-3

SOURCE: Israelson, A.F. 1978. Magnetic separation of minerals. Mining Magazine.

<sup>a</sup>Both monoclinic and hexagonal pyrrhotite is present with the monoclinic form being more magnetic than the hexagonal form.

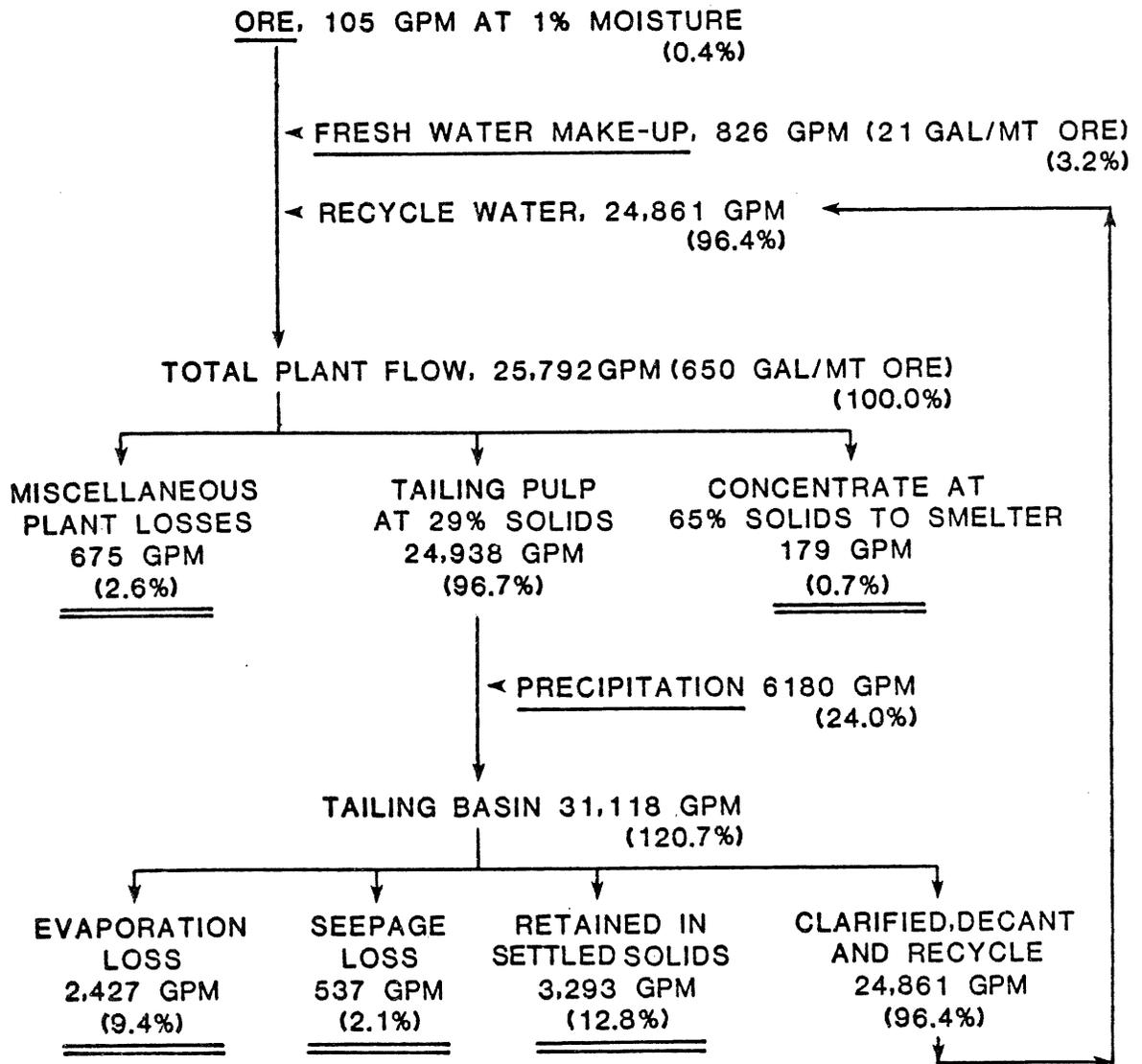
the extent of the facilities, total potable water needs for a fully integrated operation will range from 5 to 15 gpm for each  $10^6$  mtpy ore capacity. For example, a  $20 \times 10^6$  mtpy operation consisting of mine, mill, smelter, and refinery facilities would require a potable water system capable of delivering 100 to 300 gpm, or 50 to  $150 \times 10^6$  gpy. Generally speaking, this demand is composed of comparable requirements from each of the three principal operation components; the mine, the processing plant, and the smelter/refinery complex.

3.2.6.2 Process Water--Process water system components consist of fresh or make-up water supply, water reclamation facilities, and the handling and distribution system necessary to insure a sufficient supply wherever and whenever needed.

Processing water requirements are estimated at 650 gal/mt of ore, to insure sufficient water to obtain proper pulp densities and transport solids throughout the system. A typical water balance is modeled in Figure 30 for a  $20.00 \times 10^6$  mtpy plant, during average year precipitation and evaporation, with the tailing basin located on a semi-permeable base. For details on this balance, see Volume 3-Chapter 4 of the Physical Sciences report. The ore itself contains only about 1% moisture and therefore the bulk of the plant requirement must be made up of recycled water from the tailing basin pond and fresh (or make-up) water from a variety of sources. For the example shown, miscellaneous losses in the plant due to spills and washdown, plus water used to transport the concentrate to the smelter and subsequently to be evaporated in the drying stage, result in the loss of slightly over 3% of the water flow in the plant. The remaining losses are due to pond evaporation, seepage in the tailing basin, and water retention in the settled tailing solids. These amount to more than 24% of the plant flow in this model. Total losses thus exceed 27% and since precipita-

**TYPICAL PROCESSING WATER  
BALANCE MODEL-20X10<sup>6</sup> MTPY  
ORE OPERATION**

(ASSUMING AVERAGE YEAR PRECIPITATION AND EVAPORATION  
AND SEMI-PERMEABLE TAILING BASIN BASE)



tion onto the basin itself replaces almost 24% of the need, fresh water must make up only 3% of the total plant requirement.

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Figure 30

Again, in the Physical Sciences report, Volume 3-Chapter 4, there is a discussion of alternatives involved in a water budget for all systems, under a range of weather conditions such as wet, dry or average periods, and basin locations such as on permeable, semi-permeable, or impermeable bases. Average conditions were modeled here as being representative, in order to compare the relative requirements for each operation.

### 3.2.7 Materials Handling Systems

Materials handling options in ore processing systems are limited due to the nature of the individual products being transported. Ore to the total processing system would be transported to the primary crushing facility either by conveyor or by truck. Another system approach is by rail, but this is not feasible unless the ore is from a distant mine site.

Ore material transport through the crushing and screening systems would be by conveyor, as would the transport to the grinding facilities. Up to this point the material is typically coarser than 0.5in. and as dry as possible. Thus, no other transport system is applicable except in the case of an isolated mine at which the ore would be crushed and transported to the processing facility by conveyor, truck, or rail.

After grinding the material is mixed with water and transported through the system as a slurry either by gravity flow or by pumping. Obviously, no other

system is applicable until some dewatering facility is included. Of the products resulting from a processing operation, only the concentrate would be a candidate for dewatering to the point where a transport system other than gravity flow or pumping could be applied. Concentrate transportation to a nearby smelter would most easily be accomplished by pumping; however, if the smelter is not on site the concentrate would probably be filtered and dried before being transported by truck, rail, and/or possibly barge. Long distance pumping of concentrate is definitely a possibility, but the subject has not been addressed in this report.

In the case of concentrate pumping, the material could range in pulp densities from 30 to 65% solids depending on the system selected. Again, depending on the pumping distance (see tailing pumping discussion), the concentrate line may have to be heat traced or otherwise protected from freezing weather. Accidental spillage would also be of concern so provision would be made to control any such occurrence in a manner facilitating recovery of the spilled material.

A detailed description of the concentrate appears in Volume 3-Chapter 2 of this report. Of importance in materials handling of the concentrate is the size consist (80 to 90% -270 M, or -53 microns) and the chemical composition (Cu, Ni, Fe, S, and trace elements) which could cause potential environmental impacts if spilled and not controlled or recovered.

### 3.2.8 Additional Facilities Required With an Off-Site Smelter/Refinery Complex

In addition to the processing facilities as described for an on-site smelter/refinery operation filtration and drying of the concentrate would be necessary if the subsequent stages are remotely located from the processing plant. In the models considered later in this report, concentrate is pumped

directly to the smelter at 65% solids, and spray drying is then a part of the smelter operation. With a remotely located smelter, filtration and drying of the concentrate at the processing plant would be necessary prior to transporting it to the smelter, particularly during winter conditions when freezing of water pulps in pipelines or rail cars is a distinct possibility.

As an example of the costs involved, filtration and drying facilities for a model  $20 \times 10^6$  mtpy operation will range around  $\$6 \times 10^6$  capital cost and 4-5¢/mt ore total operating cost. As mentioned above, these facilities are included in the smelter model and need not be added to the processing unless the smelter is remote and this cost would then be subtracted from the existing smelter and added to the processing plant. This would also occur in the case where no smelter is included and the concentrate is to be sold as is.

Transportation of dried concentrate between the facilities is an operation whose cost cannot be reasonably estimated without knowing the distance, mode of transportation, and ownership of transporting facilities. This variable is considered further in the smelter/refinery section (see Chapter 4 of this report). At this point, no further discussion will take place other than to say such transportation costs could vary between \$0.02 and \$1.00 per mt concentrate for each kilometer of travel distance depending on the mode of travel. For short distances, conveyors appear to be cheapest and rail the most expensive, with trucks in the middle. Long distance hauling would most likely be done by rail.

### 3.3 PROCESSING FACILITY SITE LAYOUT AND AREA REQUIREMENTS

Site selection and layout for a Minnesota copper-nickel processing facility will depend on many factors. Of major concern from the developer's point of view is the effect of site selection on the efficiency of the processing facility and the overall efficiency of the entire integrated operation. Ideally, a processing facility would be located close to the mine and the smelter/refinery complex to minimize transportation requirements between the integrated stages. Also, since access, power, water, and waste disposal are of primary importance to an efficient operation, the plant should be as close as possible to sources of these needs.

Often, however, a trade-off situation occurs and requires careful cost analysis during site selection, as seldom are all these requirements ideally available at one site. Power lines may have to be run a greater distance, as it may be less costly than pumping tailing that same distance to a suitable tailing basin site.

Economics also would dictate that processing plants have solid foundations and be located on a hillside to take advantage of gravity flow in the system. Good examples of both site requirements are Reserve Mining Company's taconite plant at Silver Bay, Minnesota, and Cities Service Company's Miami Operations plant in Miami, Arizona. Both operations are built on hillsides and take advantage of gravity flow in the processing system. Reserve is a classic example of a site cut out of solid rock to provide both a good foundation and gravity flow through the system.

Area requirements of processing facilities will vary with the design and degree of integration of the total operation facilities, but processing plants and

associated facilities themselves will occupy areas typically ranging from 120 to 400 acres for plants varying in ore capacity from 5.35 to 20.00 X 10<sup>6</sup> mtpy (see Chapter 5 of this volume for model descriptions). Additionally, making certain modeling assumptions (Chapter 5), it is found that tailing basin areas of 1,067 to 4,016 acres are needed for those operations. In total, 1,187 to 4,416 actual acres are needed for the modeled processing operations ranging from 5.35 to 20.00 X 10<sup>6</sup> mtpy in size, plus an additional 40% in associated undisturbed watershed area.

Ideally, the processing plant would be located very near the mining operation (less than one mile) to minimize ore transportation costs to the concentration facility. Then, the tailing disposal area should be as close to the plant as possible (less than a few miles) to hold down the transportation cost of this material. Both the ore and the tailing are large volume, low (or no) value materials and require low unit transportation costs. The concentrate, however, is a low volume, high value product and can typically be economically transported long distances, often hundreds or even thousands of miles.

Figure 31 is a more detailed sketch of Figure 1, a hypothetical processing facility layout sized to process 20.00 X 10<sup>6</sup> mtpy of ore. Facilities are drawn to approximate scale. The dimensions listed are typical only, to illustrate potential visual impacts of such an operation.

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Figure 31

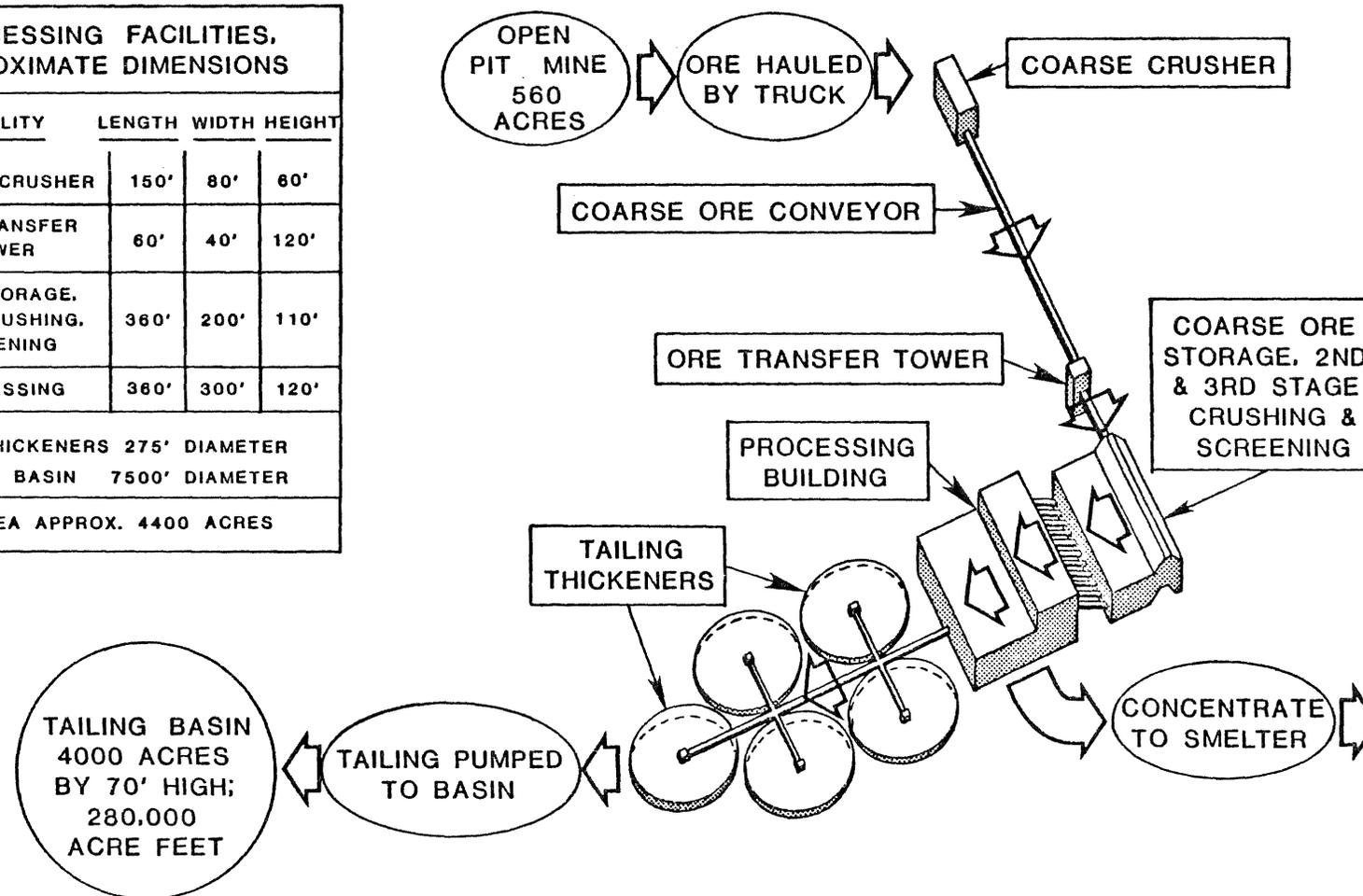
The visual impact of a processing facility is not expected to be significant compared to waste rock storage piles and the tailing basin area or areas. The buildings are generally less than 150 ft in height, and unless they are

FIGURE 31

TYPICAL PROCESSING FACILITY LAYOUT FOR A MODEL  $20 \times 10^6$  MTPY OPERATION

SCALE OF STRUCTURES: 1" = 475'

PROCESSING FACILITIES, APPROXIMATE DIMENSIONS			
FACILITY	LENGTH	WIDTH	HEIGHT
COARSE CRUSHER	150'	80'	60'
ORE TRANSFER TOWER	60'	40'	120'
ORE STORAGE, FINE CRUSHING, SCREENING	360'	200'	110'
PROCESSING	360'	300'	120'
TAILING THICKENERS 275' DIAMETER TAILING BASIN 7500' DIAMETER			
TOTAL AREA APPROX. 4400 ACRES			



constructed on a hillside, would not stand out to a ground level observer several miles away. Tailing basins in the models discussed here assume a 75 ft high dam with waste rock piles 200 ft high. Both would be expected to overshadow the visual impact of the buildings, which in any case are temporary, unlike the basin and rock piles.

#### 3.4 GENERAL INDUSTRY AND PROCESSING COST CONSIDERATIONS

A good idea of probable cost trends in the recovery of copper and nickel in Minnesota can be obtained by examining the trends in the U.S. copper industry in the recent past. Generally speaking (Lewis et al. 1978), the cost of producing copper between 1972 and 1976 increased 85%, an unprecedented rate for the entire industry, mainly due to labor increases of more than 50% and energy increases exceeding 200%. During the same time period, copper producers have had to reduce the price of copper from 70¢ to 60¢/lb, due to market conditions, causing slowdowns or shutdowns in many operations.

Table 15 lists the cost increases over the 1972-1976 time period for a hypothetical Arizona copper operation, by operation. These cost increases can reasonably be applied over the entire copper industry and to hypothetical Minnesota copper-nickel operations as well. Table 16 lists the distribution of costs for the same facility for the years 1972 and 1976. With a 1972 cost index of 100, the 1976 cost index is 185, or 85% higher.

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#### Tables 15 & 16

In the face of rising costs and decreasing ore grades, the mining industry in general has strived for greater productivity and efficiency of operation through technological improvements. Larger and more efficient equipment has reduced the

Table 15. Cost comparison over years 1972-1976 for the copper industry.

<u>OPERATION</u>	<u>INCREASE IN COSTS</u>
Mining	72%
Processing	98%
Smelting	64%
Refining	109%
General & Administrative	<u>87%</u>
Total	85%

SOURCE: Lewis, et al. 1978.

Table 16. Cost distribution change, 1972-1976, to produce copper from a hypothetical Arizona operation.

	COST DISTRIBUTION % of 1972 cost	
	1972	1976
Salaries & labor	19	28
Energy (diesel & electric)	9	25
Consumable materials	13	24
Maintenance supplies	8	14
Smelting	18	31
Refining	12	24
General & Administration	<u>21</u>	<u>39</u>
Total	100	185

SOURCE: Lewis, et al. 1978.

costs of obtaining and treating ore materials. Improved methods have allowed economical treatment of lower quality materials to recover valuable metals.

With all the improvements that have been made to lower overall unit costs, increased requirements for environmental concerns have placed additional cost burdens on the industry and increased demands on the technologies of each mineral system.

The bottom line tells the story. If the costs of obtaining the end product within all industry and environmental demands exceeds the value of the product on the open market, the operation will close down or not be undertaken in the first place. Only when a reasonable profit can be made and there is a demand for the product, will an operation continue.

To facilitate analysis of the mining industry, a set of hypothetical mine models was created, and cost estimates were made for each model size. These are discussed in Chapter 5 of this report. Major category capital and operating costs were determined for models producing 100,000 mtpy of copper and nickel metal as follows:

<u>Category</u>	<u>Capital Cost</u> <u>% Distribution</u>	<u>Operating Cost</u> <u>% Distribution</u>
Exploration & Mining	28	44
Processing	27	30
Smelting & Refining	<u>45</u>	<u>26</u>
Total	100	100

The above distributions are averages for open pit and underground operations ranging from 12.35 to 20.00 X 10<sup>6</sup> mtpy ore, each producing 100,000 mtpy of

metal. Capital costs vary from \$656 to \$761 X 10<sup>6</sup> and operating costs vary from \$120 to \$137 X 10<sup>6</sup> per year. Table 17 lists these cost summaries, and complete details are found in Chapter 5 of this report.

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Table 17

Of interest in this section are the capital and operating cost areas for processing copper-nickel ore. Table 18 breaks down the costs for a 20 X 10<sup>6</sup> mtpy processing plant as defined in the model discussions. Major capital cost areas are crushing, grinding, and flotation which comprise over 70% of the capital investment. Operating costs are primarily supplies and labor (79%) with the remainder primarily power costs.

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Table 18

Potential areas for cost improvement in processing have been identified and are being investigated. Examples of such areas are:

- 1) Size reduction--autogenous grinding vs. conventional grinding; more effective size reduction in crushing operation.
- 2) Concentration--differential or selective flotation vs. bulk flotation; more efficient flotation to improve metal recovery; variations in reagent suite for improved efficiency.

### 3.5 POLLUTION CONTROL TECHNOLOGY

As applied to mineral processing, pollution control technology incorporates well known and well defined techniques and equipment. Examples would be in dust control, water chemistry, and material containment during transportation. Each

Table 17. Model cost summaries.

COST COMPARISON	OPERATION CAPACITY, 10 <sup>6</sup> mtpy <sup>a</sup>				
	5.35	11.33	12.35	16.68	20.00
<u>Capital Cost, % Distribution</u>					
Exploration & mining	57	42	27	31	27
Processing	43	58	24	27	30
Smelting & refining	<u>--</u>	<u>--</u>	<u>49</u>	<u>42</u>	<u>43</u>
Total - %	100	100	100	100	100
- \$10 <sup>6</sup>	206	286	656	765	761
<u>Operating Cost, % Distribution</u>					
Exploration & mining	70	46	54	43	34
Processing	30	54	22	31	38
Smelting & refining	<u>--</u>	<u>--</u>	<u>24</u>	<u>26</u>	<u>28</u>
Total	100	100	100	100	100
- \$10 <sup>6</sup> /yr	46	51	137	126	120

<sup>a</sup>5.35 and 11.33 X 10<sup>6</sup> mtpy operations include exploration, mining, and processing only. The 5.35 and 12.35 X 10<sup>6</sup> mtpy operations are underground, the 11.33 and 20.00 X 10<sup>6</sup> mtpy operations are open pit, and the 16.68 X 10<sup>6</sup> mtpy operation is a combination of the 2 smaller operations.

Table 18. 20 X 10<sup>6</sup> mtpy model processing cost breakdown.

<u>CAPITAL COST AREA</u>	<u>PERCENT</u>
Primary crushing	6
Coarse ore storage	3
Secondary crushing	14
Grinding & classification	36
Flotation	22
Tailing thickening	4
Tailing disposal system	9
Reagent handling system	1
General & administration	<u>5</u>
Total	100
 <u>OPERATING COST AREA</u>	
Personnel	23
Operating supplies	42
Maintenance supplies	14
Power	20
General	<u>1</u>
Total	100

important potential pollution area is listed in Table 19 along with mitigating measures applicable to that particular instance. The effectiveness of such mitigation is also shown in qualitative terms.

---

Table 19

The most serious potential pollution problems associated with the processing operations are:

- 1) the possibility for contamination of ground and surface waters by dissolved salts, heavy metals and processing chemicals as a result of seepage or discharges from the tailing basin; and
- 2) the possibility for a decrease in air quality resulting from the release of dust and mineral fibers from the dry surface of the tailing basin.

In principle, the technology exists to prevent or control these problems both during the operation and following abandonment. In practice, control methods with a high degree of reliability may be prohibitively expensive on a full-scale basis, and in fact may prove to be unnecessary. The potential release mechanisms which could cause these problems to occur are quite complex. When the magnitude of a full scale processing operation is also considered, it is apparent that accurate prediction of the need to plan, finance, and implement a particular level of control to maintain a desired level of environmental quality is highly difficult, if not impossible. As a result, to maintain environmental quality it is necessary in practice to closely monitor the air and water quality around a tailing basin, and to be prepared to deal with problems as they develop. Correspondingly, the effectiveness of controls instituted under these conditions cannot be confidently predicted. This introduces a clear element of

Table 19. Processing pollution potentials and mitigating measures.

PROCESSING OPERATION	POTENTIAL POLLUTION	MITIGATION MEASURE	EFFECTIVENESS OF MITIGATION MEASURES <sup>a</sup>
Crushing & Screening	Noise	Ear protection	1
	Dust & fibers generated in each step & released to air at transfer points & storage areas	Proper design & operation of the facility, enclose crushers.	1
		Install hoods over equipment & transfer points & dust collection systems to collect fugitive dust	2
Grinding	Noise	Ear protection	1
		Proper design, placement, and operation of equipment	1
Flotation	Chemical reagents build up in the water system, spillage, human contact	Control build up with sufficient tailing pond retention, protect against spillage & human contact with intensive safety program	2
Tailing disposal & reclaim water systems	Dust & fibers from dried tailing in basin	Keep covered with water	1
		Chemical stabilization	2
		Revegetation	2
	Seepage from basin	Collect & recycle seepage water	2
		Prevent seepage from both dam & basin	2
		Install collection system along tailing line	2
		Purify water in a treatment plant prior to release or allow adequate excess storage capacity	1
General	Oil & grease spills	Oil & grease traps Proper maintenance	1
	Dust lift-off from plant area and roads	Revegetate, water, & treat roads to prevent dust	1
	Water runoff from plant area	Provide drainage collection, contour to dissipate water	1
	Sanitary facilities	Provide sewage collection and treatment	1

<sup>a</sup>1: Good-proven effective control technology exists and is known to operate reliably in many installations.

2: Fair-proven effective control technology exists or is in the demonstration stage at one or more installations.

3: Poor-proven effective control technology has not been developed or is not yet ready for full-scale operation.

risk to the environment in the areas of both air and water quality as a result of the establishment and operation of a copper-nickel tailing basin. Further, the risk is considerably greater following shutdown than during the operating life when a variety of active control methods can be used. Considerable uncertainty is attached to the predictions of long-term basin reclamation and pollution control.

### 3.6 SITE DECOMMISSIONING AND RECLAMATION

A processing facility may be closed down for one of several reasons:

- 1) temporary closure due to market conditions
- 2) temporary closure due to construction or reconstruction of some portion of the entire facility which requires complete shutdown
- 3) permanent closure of the entire facility due to exhaustion of the resource

The first 2 examples are temporary and would follow routine procedures to continue the facility in some state of rediness, fully expecting to reopen the plant at a future date. The last reason for closure is more permanent and would generally require decommissioning of the site once any hope of reopening has been exhausted. Such a condition will be discussed in this section.

Total decommissioning of a processing facility site would normally involve complete removal of all structures on the property, filling or sealing-off all excavations, and preparing the area for subsequent reclamation, restoration, or revegetation. Reclamation includes all after uses of minelands, restoration indicates attempts to return the land to its previous ecological condition, and revegetation merely provides plant cover.

Since a good portion (40%) of the total land need is present as undisturbed watershed, every effort must be made to blend the disturbed and undisturbed areas in a way consistent with the final planned use of the total area. The mining chapter (Chapter 2, section 2.9) discusses such rehabilitation procedures in detail. Since site decommissioning and subsequent treatment is a definite part of the total mine package, it must be planned for and designed into the overall operation from the beginning. This means that money must ideally be provided for these activities from the start, so that there is no possibility that funds for adequate decommissioning and reclamation will not be available when operations cease, even under adverse economic conditions.

### 3.7 REFERENCES

#### 3.7.1 Cited Processing References

Hirsch, H. 1975. Energy requirements in Minnesota iron ore and taconite mining 1953-2000. Minnesota Energy Agency.

Iwasaki, I., A.S. Malicsi, R.J. Lipp. 1978. Final report to Minnesota Environmental Quality Board, Regional Copper-Nickel Study, mineral processing studies-flotation tests. Mineral Resources Research Center, University of Minnesota.

Kakela, P.J. 1975. Iron ore: energy, labor, and capital changes with technology. Science, Vol. 202.

Lewis, F.M., C.K. Chase, R.B. Bhappu. 1978. Copper production costs update. Mining Engineering Magazine.

Mining Magazine. 1978. Processing of copper sulphide ores: froth flotation reagents-a review.

Veith, D.L. 1978. Minnesota copper-nickel resource processing model, draft report. Regional Copper-Nickel Study, Minnesota Environmental Quality Board.

### 3.7.2 General Processing References

- AMAX. 1977a. Minnamax project, comparative estimates, copper-nickel concentrators. Stearns-Roger, Inc., C-19494.
- AMAX. 1977b. AMAX planning review, Minnamax project. Prepared for Minnesota Copper-Nickel Study.
- AMAX. 1977c. Personal communication. D.B. Smith (AMAX) from C.A. Roloff (Allis-Chalmers).
- AMAX. 1977d. Personal communication. L.A. Darling (AMAX) to D.L. Veith (Cu-Ni).
- Hickok, E.A., and Associates. 1977. Water resources investigation for the possible Minnamax mining facility.
- International Nickel Company. 1975. Description of operating concepts required to establish preoperational monitoring for INCO's proposed spruce road project.
- Lawver, J.E., R.L. Wiegel, N.F. Schulz. 1975. Mineral beneficiation studies and an economic evaluation of Minnesota copper-nickel deposit from Duluth Gabbro. MRRC, Univ. of Minnesota. Prepared for USBM under Grant No. G0144109.
- MDNR. 1975. Mineral resources of a portion of the Duluth Complex and adjacent rocks in St. Louis and Lake counties, northeastern Minnesota. MDNR, Division of Minerals, Minerals Exploration Section, Report 93, Hibbing, MN, 1977.
- MRRC. 1978. Interim Reports, Pilot Plant Copper-Nickel Studies by the MRRC for the LCMR Project A-bulk flotation process, May 12, 1978; Project B-differential flotation process, September 29, 1978
- Nebeker, J.S. 1973. Water use in Arizona's copper industry. Presented to Arizona Water Pollution Control Association, Tucson, AZ.
- Parkinson, E.A., A.L. Mular. 1972. Mineral processing equipment costs and preliminary capital cost estimations. CIMM (13).
- Schulter, R.B., A.B. Landstrom. 1976. Continuous pilot plant testing confirms flotability of Duluth Complex sulfides. Engineering and Mining Journal, April. Also presented at the 37th annual AIME mining symposium, Duluth, MN.

### 3.7.3 Cited Tailing References

Argall, G.O. Jr. 1978. Second International Tailing Symposium World Mining, September, 1978.

CANMET. 1977a. Pit slope manual--Chapter 9: waste embankments. Canada Centre for Mineral and Energy Technology.

Dean, K.C., R. Havens, M.W. Glantz. 1974. Methods and costs for stabilizing fine-sized mineral wastes. U.S. Bureau of Mines, I.C. 7896.

Environmental Protection Agency. 1975. Liners for land disposal sites. Rep. SW-137.

Golder, Brawner and Associates, Ltd. 1978. Report to State of Minnesota Environmental Quality Board on Copper-Nickel Project--Engineering aspects of tailing disposal. Vancouver, B.C. V57 IN5 Canada.

Gubbe, L.W. 1977. Granular-fill dam protected by PVC membrane. Civil Engineering-ASCE.

Israelson, A.F. 1978. Magnetic separation of Minerals Mining Magazine.

Robinsky, E.I. 1975. Thickened discharge-a new approach to tailings disposal. CIM Bulletin.

Soderberg, R.L. and R.A. Busch. 1977. Design guide for metal and non-metal tailings disposal. U.S. Bureau of Mines, I.C. 8755.

Trethewey, W.D. 1977. The utilization of copper-nickel tailing, Mineral Resource Research Center, University of Minnesota.

#### 3.7.4 General Tailing References

CANMET. 1977b. Pit slope manual--Chapter 10: environmental planning. Canada Centre for Mineral and Energy Technology.

Environment Canada. 1975. Mine and mill wastewater treatment. Economic and Technical Review Report, EPS 3-WP-75-5. Water Pollution Control Directorate.

Environmental Protection Agency. 1973. Processing, procedures, and methods to control pollution from mining activities. U.S. Environmental Protection Agency, Washington, D.C. EPA-430/9-73-011.

Ryan, W.A. 1978. Tailing basin design, draft report. Regional Copper-Nickel Study, Minnesota Environmental Quality Board.

SME mining engineering handbook, volume 1. Seeley W. Mudd Memorial Fund of AIME. Society of Mining Engineers of AIME. U.S. Bureau of Mines, Department of Interior.

Mines Branch. 1972. Tentative design guide for mine waste embankments in Canada. Prepared for the Mines Branch--Mining Research Centre, Dept. of Energy, Mines and Resources, Ottawa, TD 145.

Wahler, W.A. and Associates. 1974. Evaluation of mill tailings disposal practices and potential dam stability problems in southwestern United States. General Report, U.S. Department of Interior, Bureau of Mines, Washington, D.C. U.S.B.M. Contract No. 50110520. Volumes 1 to 5.