CN 159

MINNESOTA COPPER-NICKEL RESOURCE

PROCESSING MODEL

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EXECUTIVE SUMMARY

The Minnesota Department of Natural Resources (MDNR) estimates that a potential copper-nickel sulfide resource totaling 4.4 X 10^9 metric tons (mt) exists in northeastern Minnesota. The resource is contained in the basal contact of the Duluth Complex with the underlying geological formations, which extends in a semi-circle outcrop from Duluth through Hoyt Lakes, Ely, and Grand Marais. The identified potential resource averages 0.66% Cu and 0.20% Ni and is located along the contact area from the western boarder of the Boundary Waters Canoe Area (BWCA) southward through Hoyt Lakes and dips an average of 25^o to the southeast.

Research on this copper-nickel sulfide material has been conducted for some years by both private industry and governmental agencies. AMAX and International Nickel Company (INCO) have been the main private companies involved, while the U.S. Bureau of Mines (USBM) and the University of Minnesota's Minerals Resource Research Center (MRRC) have conducted the government investigations. AMAX is currently the only private company actively planning copper-nickel mining operations in Minnesota, at their Minnamax site near Babbitt. The Bureau is planning research on AMAX material in 1978 and the MRRC is currently involved in a pilot plant and bench scale test program of several samples sponsored by the Legislative Commission on Minnesota Resources (LCMR) and the MEQB Copper-Nickel Study.

For the purpose of this study, the term "processing" includes all phases of the overall production of copper-nickel metal involved in upgrading the crude material as it occurs in the ground to a feed material suitable for smelting and refining. These phases consist of crushing, grinding, flotation, tailing disposal and water system technology as related to hypothetical copper-nickel development in northeastern Minnesota.

The Technical Assessment group of the Regional Study is responsible for this processing study and all related technical efforts concerned with exploration, mining, smelting, and refining. The purpose of such studies is to provide information on potential impacts related to copper-nickel processing for evaluation in the socio-economic, biological, and physical science areas.

Because of the low grade nature of the copper-nickel sulfide material, a considerable amount of processing must be done in preparation for eventual smelting and refining recovery of the contained metal values. Processing information was gathered from all available sources, evaluated, and combined for use in generating hypothetical processing models suited to the Study's needs. The processing models had to be based on factual data and yet remain hypothetical such that they could be applied throughout the Study area wherever a potential resource exists. In addition, the processing models must tie closely with the other technical models dealing with mining, smelting, and refining operations.

Processing model variations were designed to encompass a crude ore tonnage range of 4 to 20 X 10^6 mtpy with specific examples of 7.94, 11.33, and 20.00 X 10^6 mtpy, to depict both underground and open pit operations. Underground operations would probably span a production range of 4 to 12 X 10^6 mtpy crude ore averaging 0.80% Cu and 0.20% Ni. Open pit operations would have a 8 to 20 X 10^6 mtpy crude ore production range with average grades of 0.45% Cu and 0.11% Ni.

Bulk concentrates from the example crude ores would analyze 12% to 14% Cu, 2% to 3% Ni, 20% to 35% Fe, and 20% to 30% S, and would vary only in weight and metal recovery depending on their respective crude ore sources. Anticipated weight recoveries range from 3.0% to 5.5% with metal recoveries varying from 80% to 96% Cu and from 55% to 83% Ni.

Processing model variations include primary and secondary crushing, ore storage, grinding, and flotation to produce a concentrate, tailing disposal, recycle and fresh water systems, and all ancillary facilities required for complete operation. Ten variations are offered to account for: 1) capacity variations of 7.94, 11.33, and 20.00 X 10^6 mtpy crude ore; 2) conventional grinding and autogenous grinding options; and 3) underground versus open pit mining primary crushing responsibility.

Flotation involves the chemical and physical separation of the desired minerals as a concentrate from the undesired minerals as a gaugue or tailing. Ground crude ore is suspended in a water pulp, treated with chemical reagents and mechanically separated into a concentrate product and a tailing waste product. The concentrate particles adhere to air bubbles which rise to the surface of the flotation machine and are removed. The tailing particles remain wetted and sink to the bottom of the flotation unit for disposal.

Bulk flotation removes the desired copper and nickel sulfides as a combined concentrate product for subsequent bulk treatment in a smelter-refinery complex which then makes a separation of copper and nickel metals pyrometallurgically.

Differential flotation takes advantage the varying surface properties of the copper and nickel sulfides and separates them in the flotation processing stage. Separate copper concentrates and nickel concentrates are then subjected to individual smelter-refinery stages to produce the respective metal products.

Bulk flotation of a combined copper-nickel sulfide concenterate is the beneficiation method used in the models. Reagent consumptions are estimated to be approximately 0.1 lb/mt crude of both xanthate collector and alcohol frother. Fresh water (other than potable) requirements are estimated at 160 gal/mt crude ore.

Differential flotation of separate copper and copper-nickel concentrates is definitely a possibility, but not yet shown to recover as much of the metal as does bulk flotation. This beneficiation method would require additional equipment and reagents and probably result in higher operating cost than the bulk system. Until differential flotation is researched more fully the models will deal only with bulk flotation. (Additionally, the smelting-refining model is based on a bulk concentrate feed and therefore ties directly to these processing models. Consideration of differential flotation would require extensive revision of the pyrometallurgical scheme and the smelting-refining model.)

Tailing disposal and water systems are of extreme importance in the processing model variations under consideration. For the range of 4 to 20 X 10^6 mtpy crude ore and productive lives of 20 to 40 years, tailing ponds of between 1000 and 5000 acres would be necessary with an effective pond height of 100 feet. Seepage control systems are included and it is assumed that although the tailing ponds will seep (and a number of engineering sources indicate that they must seep), most of the seepage can be collected and returned to the system.

The quality of seep waters will be the subject of another report. Studies at the MRRC show both xanthate and frother compounds break down readily in the discharge products, but the ultimate end products are not yet known. Further investigation should clear up this question, but at this point the end products are thought to be basically C, H_2, and O_2 products.

A water system must be able to return a maximum amount of clarified water from the tailing pond to the plant water supply system for reuse. Additionally, such a system must supply sufficient fresh water to make up for that lost during the process, about 25% of the total required. About 650 gal/mt crude ore is needed to process Minnesota copper-nickel material, of which 160 gal/mt must be replaced by the fresh water intake system.

Capital and operating cost data were generated for the processing models in 1977 dollars. Manpower and other miscellaneous data were also collected and the total summarized in the text. Capital costs are reported on a unit basis of $/mtpy crude ore capacity, and also as a total capital investment. Operating costs are generated on a unit basis of $/mt crude ore. All costs are direct out-of-pocket costs and no inclusion is made of interest, taxes, and insurance-type cost items.

It appears from the data that considerable cost savings can be realized by scaling the plant capacity upwards and by using autogenous grinding instead of conventional grinding. Depending on the processing plant system and capacity, capital investment varies between \$125 and \$248 X 10^6, with unit capital cost variation of \$11.1 to \$17.7/mtpy crude ore. Operating costs vary between \$1.80 and \$2.69/mt crude ore. On a unit basis the most expensive operation, with all facilities on the surface, is the 7.94 X 10^6 mtpy plant with conventional grinding. The lease expensive on a unit basis is the 20.00 X 10^6 mtpy plant with autogenous grinding.

Manpower requirements vary between 227 and 414 persons, when primary crushing is included, depending on the plant size and flowsheet selection. Autogenous grinding systems require 10% to 15% less manpower than conventional grinding due to less total operating facilities. Total manpower per ton of crude also decreases as the plant size increases, due to increased efficiency.

Total power requirements range from 184 to 552 X 10^6 kwh/yr with primary crushing, depending on the operation size and the grinding system used. Autogenous systems are less efficient and require almost 20% more power for comparative grinding than do corresponding conventional systems, but the difference is easily overcome in other operating cost areas.

The construction schedule of a project of this magnitude must be closely coordinated between the engineering, purchasing, and construction phases to ensure that the desired product is obtained when planned and within the costs projected. Overall time requirements would range from 3 to 3½ years for the model variations given, and require maximum construction forces of 675 to 1000 personnel.

INTRODUCTION TO THE REGIONAL COPPER-NICKEL STUDY

The Regional Copper-Nickel Environmental Impact Study is a comprehensive examination of the potential cumulative environmental, social, and economic impacts of copper-nickel mineral development in northeastern Minnesota. This study is being conducted for the Minnesota Legislature and state Executive Branch agencies, under the direction of the Minnesota Environmental Quality Board (MEQB) and with the funding, review, and concurrence of the Legislative Commission on Minnesota Resources.

A region along the surface contact of the Duluth Complex in St. Louis and Lake counties in northeastern Minnesota contains a major domestic resource of copper-nickel sulfide mineralization. This region has been explored by several mineral resource development companies for more than twenty years, and recently two firms, AMAX and International Nickel Company, have considered commercial operations. These exploration and mine planning activities indicate the potential establishment of a new mining and processing industry in Minnesota. In addition, these activities indicate the need for a comprehensive environmental, social, and economic analysis by the state in order to consider the cumulative regional implications of this new industry and to provide adequate information for future state policy review and development. In January, 1976, the MEQB organized and initiated the Regional Copper-Nickel Study.

The major objectives of the Regional Copper-Nickel Study are: 1) to characterize the region in its pre-copper-nickel development state; 2) to identify and describe the probable technologies which may be used to exploit the mineral resource and to convert it into salable commodities; 3) to identify and assess the impacts of primary copper-nickel development and secondary regional growth; 4) to conceptualize alternative degrees of regional copper-nickel development; and 5) to assess the cumulative environmental, social, and economic impacts of such hypothetical developments. The Regional Study is a scientific information gathering and analysis effort and will not present subjective social judgements on whether, where, when, or how copper-nickel development should or should not proceed. In addition, the Study will not make or propose state policy pertaining to copper-nickel development.

The Minnesota Environmental Quality Board is a state agency responsible for the implementation of the Minnesota Environmental Policy Act and promotes cooperation between state agencies on environmental matters. The Regional Copper-Nickel Study is an ad hoc effort of the MEQB and future regulatory and site specific environmental impact studies will most likely be the responsibility of the Minnesota Department of Natural Resources and the Minnesota Pollution Control Agency.

PROCESSING MODEL INTRODUCTION

Processing of Minnesota copper-nickel ore involves crushing and grinding to liberate the valuable minerals, separation of the valuable from the worthless minerals, transportation of the mineral concentrates to the next stage of the overall system, and disposal of the waste products. Processing is the link in the system which takes run-of-mine ore, prepares a product suitable for a smelting complex, and disposes of the waste products resulting from such an operation. Materials consumed include crude ore, energy, water, chemical reagents, and materials such as grinding media and wear surfaces on equipment.

Based on information from AMAX, International Nickel Company (INCO), and the Minnesota Department of Natural Resources (MDNR) report on Minnesota resource potential (1), both open pit and underground mining operations are possible. The Mineral Resources Research Center (MRRC) report (2) is based on two identical open pit mining operations, each at a rate of 11.33 X 10^6 metric tons per year (mtpy) crude ore to supply 680,000 mtpy of concentrate to a single smelter producing 150,000 mtpy of copper-nickel white metal matte. The MRRC study is based on testwork conducted at the U.S. Bureau of Mines (USBM)(3). INCO (4) proposed a 12.2 X 10^6 mtpy open pit operation producing 372,000 mtpy of bulk concentrate. Although INCO did not propose a smelter in conjunction with their operation, they did not rule out the possibility.

AMAX, on the other hand, is considering both open pit and underground mining operations and probably will propose a combination of both to supply their requirements. They are considering processing schemes (5) ranging from 22,700 metric tons per day (mtpd) (7.94 X 10^6) to 55,600 mtpd (20.0 X 10^6 mtpy), the former being underground mining and the latter being open pit and sufficiently large to support a smelting complex producing 100,000 mtpy of copper-nickel metal.

The upper limit on the operation size is generally dependent upon the smelting capacity following processing and the needs of the company for the resource being developed. The lower limit is generally placed by economics of mining and processing small quantities of a low grade material.

Consideration of ore grade must be paramount in evaluation of Minnesota copper-nickel as the Duluth Complex definitely is a low grade deposit. MDNR estimates (1) place the resource at 4.4 X 10^9 mt averaging 0.66% copper and 0.20% nickel. On the basis of copper only, this is higher than the average of all U.S. copper deposits presently being mined, but the difficulty of removing the valuable metal places the resource in the low-grade category. AMAX latest estimates in their Minnamax Project 1977 Study dated August, 1977, indicate their materials average as follows:

Open Pit - 780 X 10^6 mt averaging 0.46% copper and 0.11% nickel
Underground - 305 X 10^6 mt averaging 0.79% copper and 0.18% nickel
Total - 1085 X 10^6 mt averaging 0.55% copper and 0.13% nickel

INCO and MRRC data for open pit ores averaged about 0.45% copper and 0.15% nickel. Based on the available information, ore grades were selected for our models which indicate expected average values as follows:

Open Pit - 0.45% copper and 0.11% nickel
Underground - 0.80% copper and 0.20% nickel

These levels will be used throughout the models; however, it is fully realized that variations could be significant, particularly when massive or semimassive sulfide ore is encountered which may contain upwards of three percent copper.

Our models will cover a size range from 4 X 10^6 mtpy to 20 X 10^6 mtpy, with specific examples at 7.94 X 10^6, 11.33 X 10^6, and 20.00 X 10^6 mtpy. The lower tonnage rate corresponds with the underground mining model and the higher rate with the open pit mining model, while the total range indicates operation sizes reasonably possible for northeastern Minnesota.

Other general considerations which must be dealt with in designing a processing system include crushing, grinding, and processing schemes, and the tailings disposal and water systems. These will be detailed in subsequent sections of the report.

Major sources of information were AMAX and INCO company reports, MRRC and USBM reports and continuing research, and numerous meetings between responsible representatives of the above organizations and Regional Copper-Nickel Study staff. Since available company data proved to be the most valuable resource for the processing model, much of this report is based on this data. However, verification of this information is being and will be made before final documents are produced.

PROCESSING DESIGN CRITERIA

As described in the introduction, process design will encompass a tonnage range of 4 to 20 X 10^6 mtpy crude ore. Typical examples of operations within that range which will be detailed are 7.94, 11.33, and 20.00 X 10^3

mtpy crude from both underground and open pit mines. All parameters for processing will be unitized based on these three models, and the complete range of tonnages can then be determined based on the units developed. As an example, the overall process operating cost in 1977 dollars per metric ton of crude ore ($/mt crude ore) and the capital cost in 1977 dollars per annual metric ton crude ore ($/mtpy) are two unit costs which may be scaled up or down depending on the size of the operation selected.

Detailed data for 7.94 X 10^6 mtpy and 20.00 X 10^6 mtpy crude ore processing plants is available from AMAX (5). Data from MRRC (2) for 11.33 X 10^6 mtpy crude is readily available, and that from INCO (4) for 12.2 X 10^6 mtpy is much less detailed. Only the 7.94 X 10^6 mtpy example is detailed for an underground mining operation; however, underground and open pit operations covering the range from 4 to 20 X 10^6 mtpy crude ore will be considered.

Basic criteria for mill design consists of certain fundamental concepts which must be established before detailed information can be developed. For the models these design criteria areas consist of:

1) Primary Crushing
2) Secondary Crushing
3) Grinding
4) Flotation
5) Water Reclamation
6) Tailing Disposal
7) Materials Handling

The following paragraphs will deal generally with each of these main areas to explain the basic criteria considered in designing the models, and in plant design in general. Materials Handling will be discussed in each of the areas 1) through 6), as it pertains to that area.

Primary Crushing

Primary crushing is used to reduce the run-of-mine ore to a maximum size of six to eight inches 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 on the surface outside the ultimate pit limit and is a processing function.

The processing models will not describe the underground primary crushing systems as they will be covered in the underground mining model. 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 costs). Once the ore is raised to the surface and dumped it becomes a processing function.

In contrast, crude ore from an open pit mine becomes a processing function or responsibility when it is dumped into the primary crusher. The primary crushing system here consists of the crusher, feeders, dust control, and conveyors to transport crushed ore to the coarse ore storage.

Primary crushing is generally found more economical if done in one large unit rather than in two 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 several smaller units.

Gyratory crushers are generally the most productive of all primary crushers and are currently the type installed, except where unusual circumstances dictate another selection is advisable. If a gyratory is not selected, a jaw crusher is the best alternative.

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 screens to crush the primary crusher product to about -3/4" for feeding the conventional rod mill-ball mill grinding system. All necessary conveyors, storage bins, and dust collection facilities are included in this operation to provide suitable feed material for the grinding system.

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.

If autogenous grinding is considered in place of conventional grinding, secondary crushing is not utilized (see Autogenous Grinding Section, page 9).

Grinding

Grinding copper-nickel material for subsequent mineral separation requires about the same energy as does taconite ore. Typical levels are 15 to 20 kwh/mt crude ore. Energy requirements are determined in the laboratory by measuring the energy needed to reduce the ore from a coarse size to a finer size. The Bond Index (6) is one such method which evaluates the ore energy requirement in reducing the 80% passing size of the feed to an 80% passing size in the product. The Bond Index is determined and the Work Index (6) is then calculated for the size distribution needed. An appropriately sized motor is selected and a grinding mill is then fitted to the motor size and 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 grinding system. Both appear suited to Minnesota copper-nickel material and will be discussed in the design.

Grinding Parameters--Estimation of power requirements and grinding mill size is based on the Bond Indices determined by AMAX and supplied to Stearns-Roger for design purposes (5). Basis for the criteria given below is 150M grind initially with 325M final 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 74 microns.). Current testwork at the MRRC indicates a coarser grind of about 65M is beneficial for initial flotation, followed by regrinding the concentrate to 270M before final separation for maximum recovery of copper and nickel. If the coarser grind approach is followed, grinding mill requirements and power consumption could decrese dramatically.

Grinding estimation parameters used in the design of the Study processing model variations are as follows:

Primary crusher discharge - 8 inch

Secondary crusher discharge - 3/4 inch

Conventional grinding discharge 80% - 150M

Rodmill open circuit grinding to - 35M

Ballmill closed circuit grinding to - 150M

Bond Index 15

Autogenous grinding discharge 80% - 150M

Autogenous mill closed circuit grinding to - 35M

Pebble mill closed circuit grinding to - 150M

Bond Index - Autogenous 22

Pebble 19

Regrind mill grinding discharge 80% - 325M

Regrind mill closed circuit grinding to - 325M

Bond Index 20

Conventional Grinding--Conventional systems consist of coarse grinding in rod mills followed by fine grinding in ball mills to produce liberation of 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. Ball mills grind primarily by attrition of the ore due to the tumbling action of the balls as the mill rotates. Rod mills are generally effective in grinding from a feed size of 3/4" 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. Power consmption and steel wear both as 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 with as little overgrinding as possible.

Autogenous Grinding--Autogenous grinding makes use of the ore characteristics to crush and grind itself to 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 than with conventional grinding.

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") 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, and to consume themselves in the process.

Autogenous and pebble mill grinding have some advantages over conventional rod and ball mill grinding. First, less capital investment and 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 to grind to grain boundaries rather than across grins, less power is consumed, less fines are produced, and flotation reagent consumption should be lower. The potential advantage of lower power consumption, however, is often offset by variations in ore characteristics which cause wide variations in mill performance and result in the same or more power consumed to grind the ore, and less control over the system.

Both conventional rod mill-ball mill grinding and autogenous mill-pebble mill grinding appear applicable to Minnesota copper-nickel ore, and will be considered in the models.

Flotation

Theory--Flotation is the heart of the process separating copper-nickel minerals from the Duluth Complex rocks. Basically, the system incorporates treating a ground ore-rock mixture in a slurry with chemicals called collectors which adhere to select mineral surfaces to render them water repellent, while the 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 to the air bubbles which are strengthened by the addition of a frothing chemical and rise to the surface where they are removed as concentrate. Wetted minerals remain in the pulp and are discharged as tailing.

Chemical Reagents--Testwork to date on bulk flotation procedures at the MRRC indicate reagent levels of less than 0.1 lb/mt for both collector and frother are sufficient for maximum mineral recovery. The modeling considers the reagent preparation section much more extensively than what is necessary for only these two reagents, primarily because additional reagents may be necessary for pH control, flocculation, etc. Additionally, differential flotation procedures would require much more extensive reagent systems than simple bulk flotation, and the cost figures developed reflect this possibility.

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

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.

In the Study models the flotation scheme designed by AMAX (5) for bulk flotation of copper-nickel sulfides is used, as this method appears to be the most comprehensive and well founded to date. The scheme is similar to that selected by MRRC for LCMR pilot plant testwork. The system incorporates 500 ft^3 rougher and scavenger flotation cells with 5 and 15 min retention time, respectively; 180 ft^3 first cleaner cells with 10 min retention; 180 ft^3 first cleaner scavenger cells with 8 min retention; 100 ft^3 second cleaner cells with 5 min retention; and 50 ft^3 third cleaner with 4 min retention. The middling is thickened and reground in a ball mill before final cleaning, and both concentrate and tailing products are thickened before leaving the processing system.

The overall design is based first 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 and a considerable equipment reduction is effected as the tailing material is removed from the circuit.

Bulk Flotation--Bulk flotation recovers the copper and nickel values as a combined sulfide concentrate which then requires 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 to effct an acceptable recovery of copper and nickel. The Regional Copper-Nickel Study processing models will be based on this technique as will subsequent smelting and refining models.

Differential Flotation--Differential flotation, on the other hand, separates the metal values into a copper concentrate and a nickel concentrate which require separate pyrometallurgical treatment to recover the separate metals. Normally, one metal sulfide is depressed with a reagent combination while the other is floated. Then the remaining sulfide is reactivated and floated as before. To date only cursory investigations in differential flotation have been done at MRRC, but apparently laboratory investigation by AMAX has indicated differential flotation may be preferable over bulk flotation. It is more difficult than bulk flotation and requires more equipment and more chemical reagents, but if successful it provides a more desirable product for subsequent smelter treatment. Until more detailed data is available on differential flotation, the models will incorporate bulk flotation and produce a bulk copper-nickel concentrate.

WATER SYSTEM

The water system is perhaps the most critical of the nonseparation phases of the process. If the system fails production immediately stops and sanitary and fire protection facilities become inoperative. Process 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 potable water and sufficient fire protection. A constant and sufficient supply of water must be assured; therefore, many dollars are spent building water management areas, water reclamation systems, and in designing the overall process for realistic water requirements.

Potable Water--Potable water needs are estimated at 500 gpm for the 7.94 X 10^6 mtpy plant and 1250 gpm for the 20.00 X 10^6 mtpy plant (all facilities), and would be supplied from wells or local municipal water supplies. Fire protection water would also be supplied by this source.

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.

AMAX considers 160 gal of fresh or make-up water necessary to process each metric ton of crude ore to replace the water lost in the processing system, about 25% of the total required. Losses consist of water retained in the tailing solids (17-18%) and miscellaneous losses such as evaporation and seepage (7-8%). With 75% of the process water returning to the system as tailing thickener overflow and decant water from the tailing pond, the 7.94 X 10^6 mtpy plant requires 2500 gpm fresh water and the 20.00 X 10^6 mtpy plant requires 6300 gpm fresh water for processing.

After the separation process the products must be disposed of in an orderly fashion. Flotation concentrate (froth) would be thickened and pumped to a smelting facility. Additionally, filtering and drying stages could be included; however, in these models filtration will be treated as an optional facility and drying will be considered a smelter function.

Tailing Disposal

The flotation tailing (sand) disposal system must be a well-maintained system as failure would cause the entire plant to be shut down. Generally, this material is first thickened to remove a good portion of the water for direct recycle to the plant. The thickened tailing (50-60% solids) is then pumped to the tailing pond for further settling and water recovery. If underground mining operations require material for backfilling mined-out stopes, the coarse portion of the tailing may be removed for this purpose by screens or cyclones and the remainder is then transported to the pond. The coarse portion is then stored for use as backfill material.

When the total tailing material is transported to the disposal pond it is either pumped directly into the basin for settling or separated into coarse and fine fractions by a cyclone. The coarse fraction is used to construct the tailing dikes and the fine material is allowed to settle behind the dikes, forming the majority of the disposal pond.

Tailing pond areas ranging from 1000 to 5000 acres would be necessary for the models under consideration.

Miscellaneous equipment and/or facilities not as critical as the above described major stages of a processing system are nonetheless important and would result in overall system failure if they did not function properly. These areas will be dealt with in the models.

FLOWSHEET DEVELOPMENT

Figures 1 through 6 are processing flowsheets for Minnesota copper-nickel ore which have been developed over the past few years. Brief histories of each will be given to illustrate development in the State.

USBM Flowsheet

The USBM bulk flotation flowsheet (3) shown in Figure 1 was developed on the INCO bulk sample material, analyzing 0.35% copper and 0.11% nickel. Both laboratory and 750 pounds per hour (lb/hr) crude ore pilot plant operations were used to design the flowsheet and establish operating parameters. Microscopic examination indicated a 200 (M) grind was required to liberate the sulfide minerals. Overall, copper recoveries ranged from 85 to 90% at concentrate grades of 15 to 11% copper, respectively. Corresponding nickel recoveries remained at 60% over the stated copper recovery range, with grade variations of 3.1 to 2.1%. Weight recovery was also uniform at about 2.5% of the feed material.

Bulk concentrate grades of 12 to 14% copper and 2.4 to 2.7% nickel appeared to give optimum combinations of grade and recovery. Copper grades above 15% did not appear practical because of increased copper and nickel losses in the tailing.

INCO Flowsheet

The flowsheet proposed by INCO and given in their July, 1975, report (Figure 2) was developed on the same material as used by the USBM, and at about the same time (4). Crude ore grade averaged 0.46% copper and 0.17%

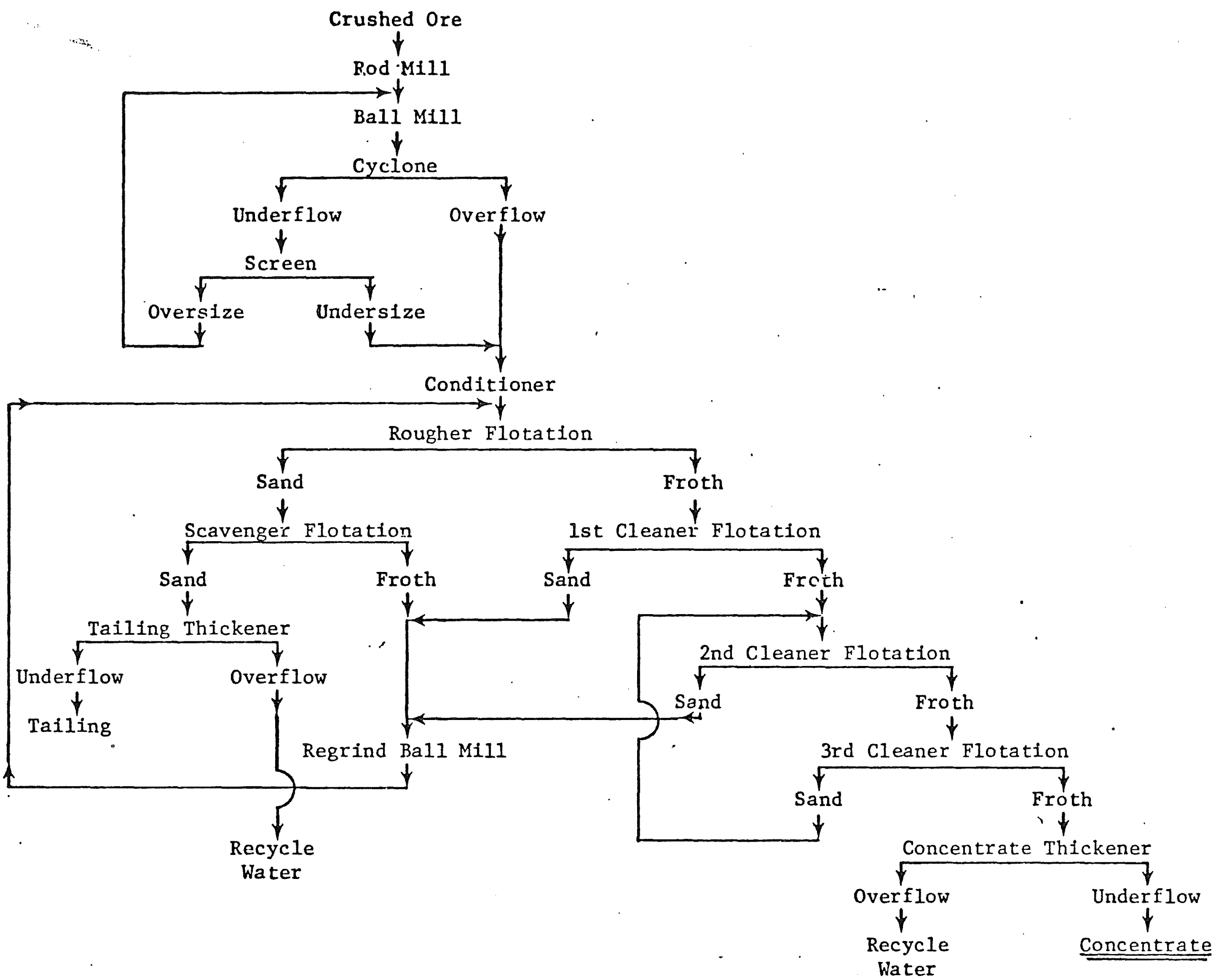


Figure 1
US Bureau of Mines Flowsheet
INCO Cu-Ni Ore

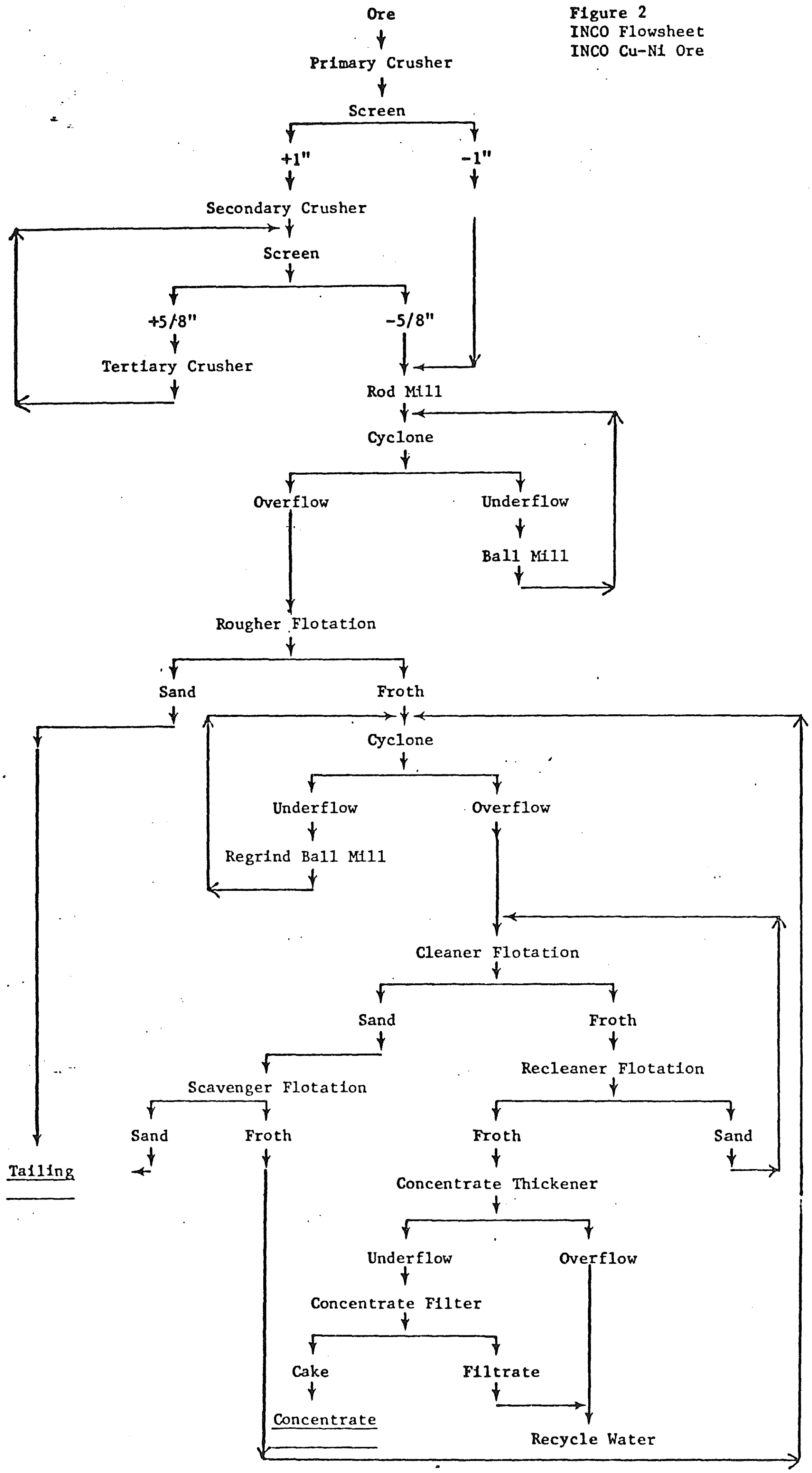


Figure 2
INCO Flowsheet
INCO Cu-Ni Ore

nickel. Bulk concentrate weight recovery was 3 percent and metal recoveries were 88% copper and 65% nickel, at grades of 13.3% copper and 3.6% nickel.

No more detail is available on the INCO flowsheet, other than overall reagent and water consumption data. The flowsheet is conventional and similar in many respects to the USBM flowsheet.

MRRC Flowsheet

In conjunction with the Bureau study, the University of Minnesota MRRC, under a Bureau grant, completed an economic model of Minnesota copper-nickel (2). The study presented the capital and operating costs of mining and concentrating a copper-nickel ore from northeastern Minnesota, and the production of a white metal matte from the concentrate in a smelter.

MRRC's model processing or beneficiation system (Figure 3) was based on the USBM flowsheet, but was modified according to the authors' expertise in design and information from manufacturers. They also considered the INCO data in establishing flowsheet parameters. MRRC assumed a crude ore grade of 0.45% copper and 0.15% nickel and a resulting concentrate grade of 14.0% copper and 3.0% nickel. Corresponding recoveries were determined to be 3.0% weight, 93.3% copper and 60.0% nickel. The MRRC report contains a considerable amount of detail which has been considered in the Study models.

AMAX Flowsheet

The AMAX flowsheet (Figures 4 and 5) was developed by company personnel

Figure 3
MRRC Flowsheet
Basis: USBM Report
INCO Cu-Ni Ore

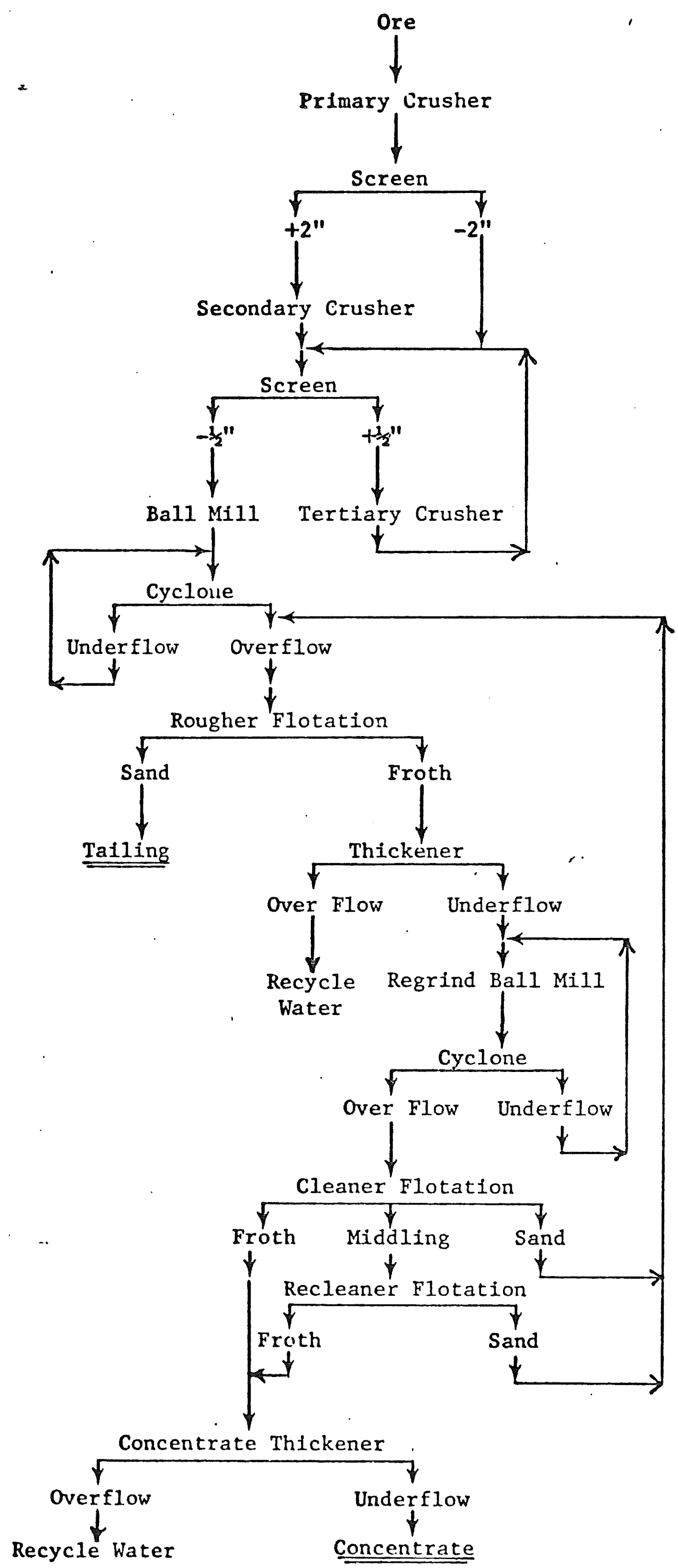
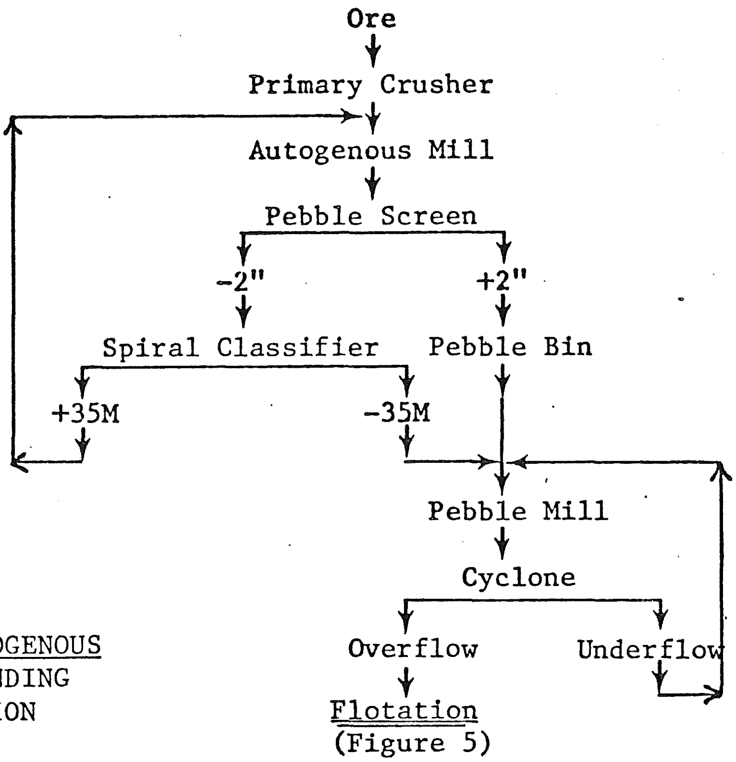


Figure 4
AMAX Flowsheet
Crushing and Grinding
AMAX Cu-Ni Ore



<u>AUTOGENOUS</u>
GRINDING
OPTION

<u>CONVENTIONAL</u>
GRINDING
OPTION

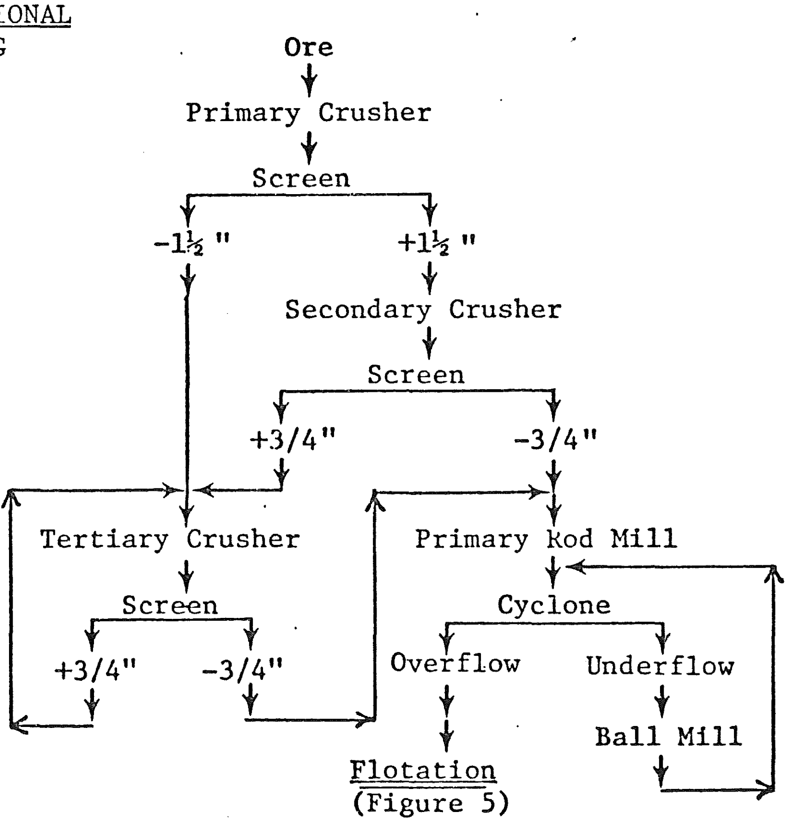
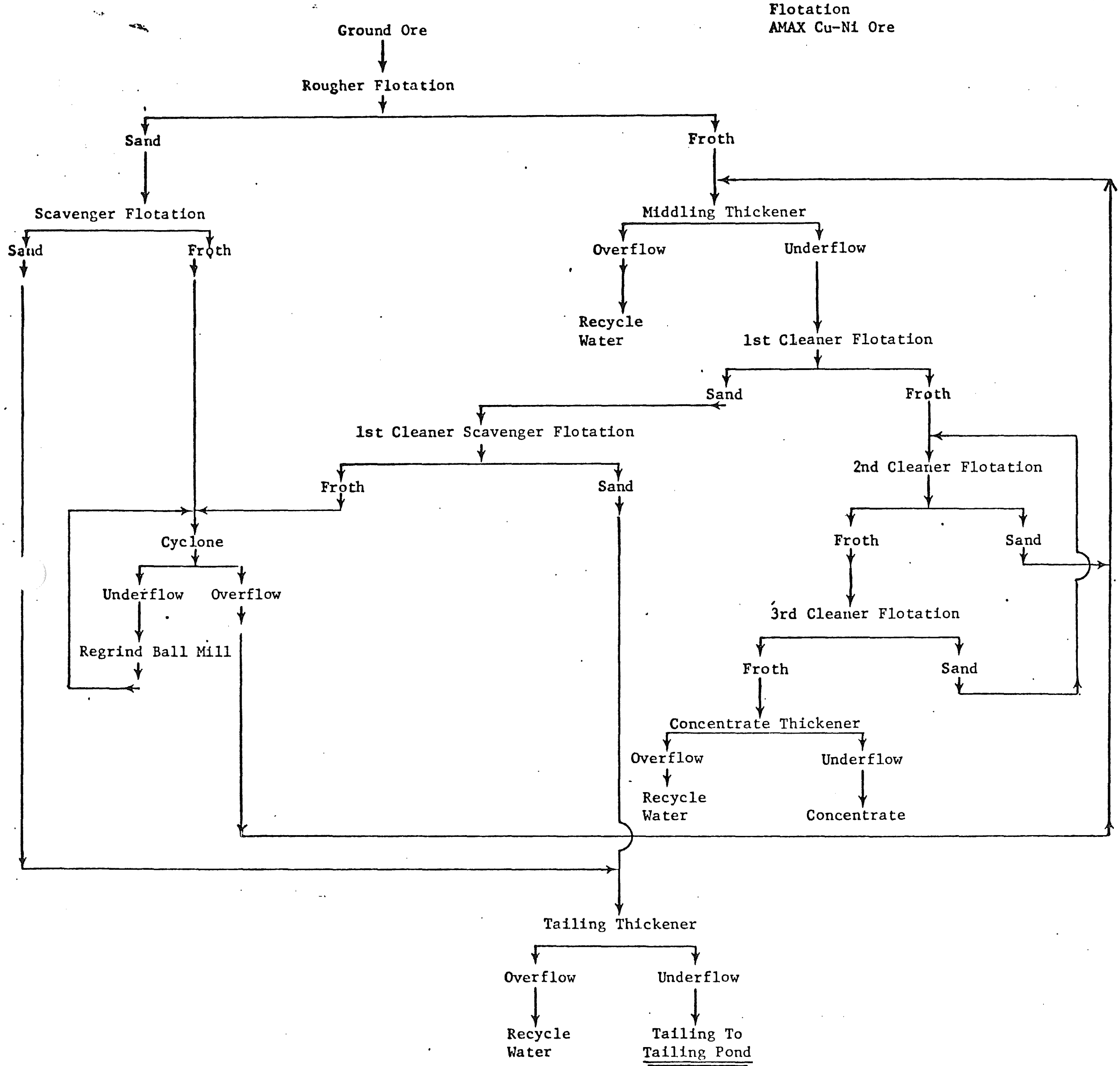


Figure 5
AMAX Flowsheet
Flotation
AMAX Cu-Ni Ore



based on laboratory testwork and detailed in a report by Stearns-Roger, Inc. (5) for AMAX. Several options were costed out-conventional and autogenous grinding, and 25,000 and 63,000 short tons per day (stpd) operations. A metallurgical balance was not included in the data, except for weight distribution through the system. Overall weight recovery was listed at six percent in the concentrate.

Generally, crude ore grade varies between 0.8 and 1.0% combined copper and nickel. Values listed in a previous publication (7) are 0.83% copper and 0.15% nickel in the crude, 14.7% copper and 2.8% nickel in the concentrate, with an overall weight recovery of 5.0%.

Table 1 contains a summary of this and previously discussed testwork. Because of the detail developed in both the MRRC and AMAX reports, considerable reliance will be placed on these sources in developing study processing models.

MRRC Flowsheet-Current Testwork

The final flowsheet shown (Figure 6) is that being used by MRRC in pilot plant testwork for the Legislative Commission on Minnesota's Resources (LCMR). Samples to be tested include those from AMAX, INCO, USS, and Erie's Dunka Pit. The autogenous option will be tried and appears promising based on preliminary information supplied to AMAX by Allis Chalmers (8).

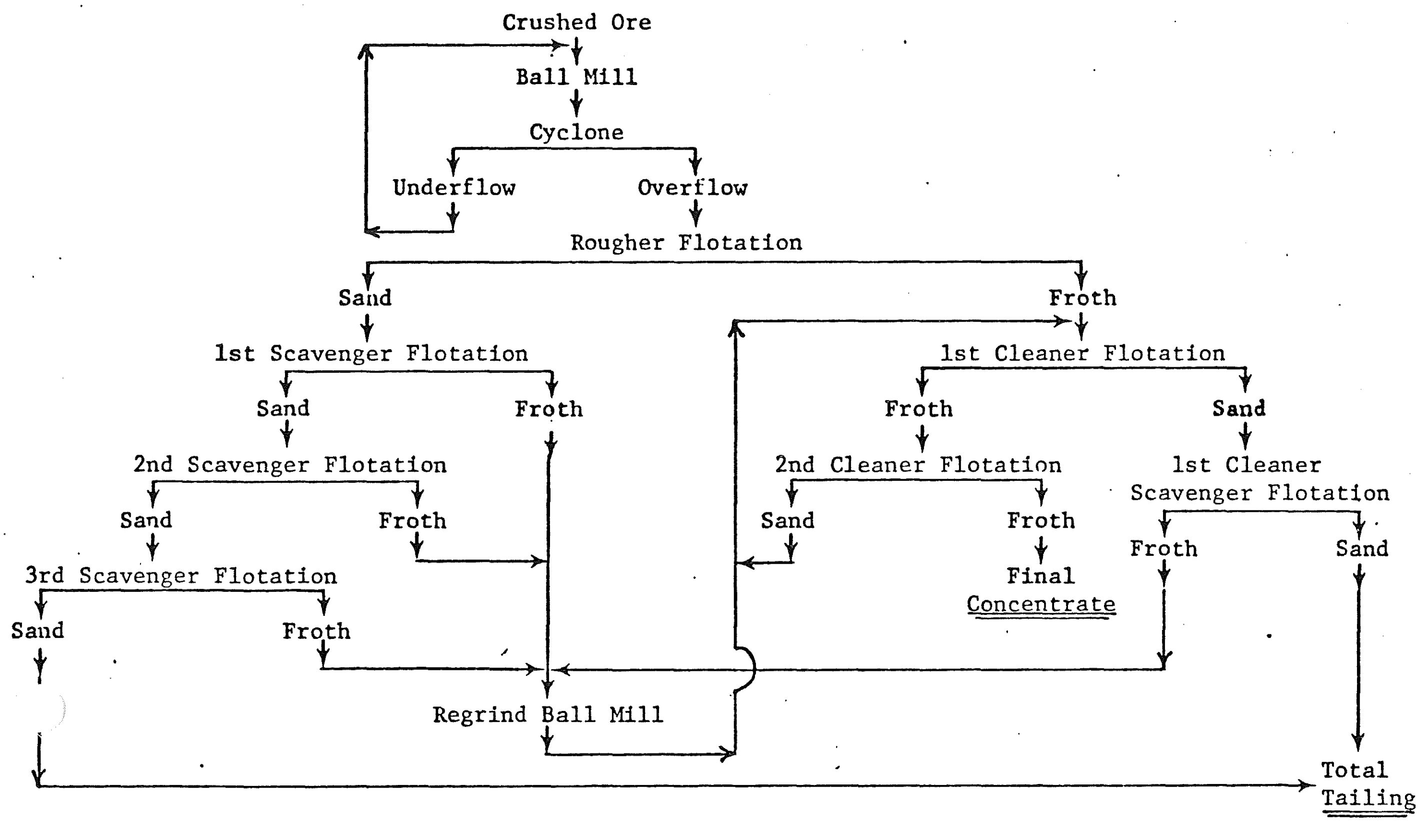
At this time differential flotation has been proposed by AMAX and will be considered in future MRRC testwork. Although the concept and reasoning is

Table 1. Summary of testwork on Minnesota copper-nickel material.

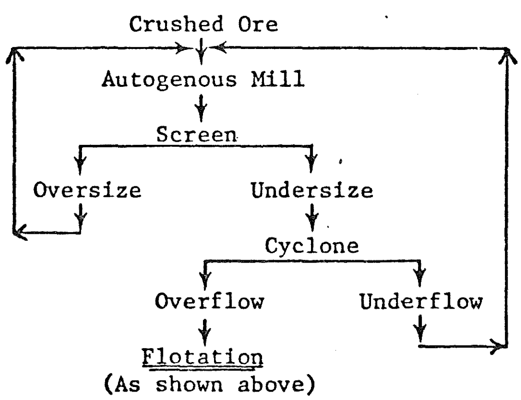
SOURCE	CRUDE ORE % Cu	% Ni	CONCENTRATE % Wt Rec	% Cu	% Ni	% Cu Rec	% Ni Rec
USBM	0.35	0.11	2.5	11-15	2.1-3.1	85-95	60
INCO	0.46	0.17	3.0	13.3	3.6	88	65
MRRC	0.45	0.15	3.0	14.0	3.0	93.3	60.0
AMAX	0.83	0.15	5.0	14.7	2.8	88.6	93.3

Figure 6
MRRC Flowsheet (LCMR)
Minnesota Cu-Ni Ores

Conventional Grinding System



Autogenous Grinding Option



sound, a practical application has not yet been developed and tested. Thus, processing models will consider only bulk flotation of Minnesota copper-nickel ore until such time as more detailed information is available on the differential approach.

Anticipated Concentrate Grade and Metal Recovery

Obviously lacking in the processing discussion to this point is a clear picture of the quality of the crude ore to be treated and the products generated--both concentrate and tailing. The introduction section specifies crude ore grades for these models at:

Open Pit - 0.45% Cu and 0.11% Ni

Underground - 0.80% Cu and 0.20% Ni

There is really no point in discussing these grades further as they are assumed from and based on various data sources as identified in the introduction. However, the concentrate produced from these crude ores is open to question and greatly affects the smelter modeling as can be seen in the detailed elemental balances in that model. The equipment designed into the processing model is capable of handling a variation in tonnage rates of at least 10% which would compensate for normal grade variations of the crude ores described above. In addition, the concentration equipment described for the small operation (7.94 X 10^6 mtpy) has been increased to compensate for the additional concentrate weight recovery expected from the richer underground crude ore.

An example of anticipated concentrate quantities and grade will be detailed, however, one must realize that depending on process variables such as size of grind, reagent quantities and material flow, a wide range of products are possible. In the final analysis the operator will consider the grade of concentrate the company requires and adjust the system to produce that product. If the desired product cannot be obtained or if losses or costs are too high, the company must reduce its requirements or abandon the operation.

Typically, from an open pit ore averaging 0.45% Cu and 0.11% Ni, 3.0% weight recovery as concentrate would results in a product containing 12-14% Cu and 2-3% Ni. This represents metal recoveries of 80-93% Cu and 55-82% Ni which appear to agree with current testwork at the MRRC and past work of both the USBM and AMAX.

An underground operation would produce crude ore averaging 0.80% Cu and 0.20% Ni. Expected recoveries in a processing plant from such material would be higher in weight but not necessarily higher in copper and nickel content. Typically, we can assume a 5.5% weight recovery of this crude as concentrate assaying 12-14% Cu and 2-3% Ni, representing 83-96% Cu recovery and 55-83% Ni recovery.

Thus, the total range of possible concentrates from typical Cu-Ni ores at assumed constant grade levels of 12-15% Cu and 2-3% Ni represent metal recoveries in the range of 80-96% Cu and 55-83% Ni. Table 2 summarizes these assumptions.

PROCESSING PLANT MODELS

Basic Flowsheet Considerations

The flowsheet to be used in all processing models for Minnesota

Table 2. Summary of anticipated processing results, Minnesota copper-nickel ores.

CRUDE ORE			CONCENTRATE PRODUCT					TAILING PRODUCT				
TYPE	% Cu	% Ni	%Wt Rec	% Cu	% Ni	%Cu Rec	%Ni Rec	% Wt	% Cu	% Ni	% Cu Loss	% Ni Loss
Open Pit	0.45	0.11	3.0	12.0	2.0	80	55	97.0	0.09	0.05	20	45
				14.0	3.0	93	82		0.03	0.02	7	18
Underground	0.80	0.20	5.5	12.0	2.0	83	55	94.5	0.15	0.10	17	45
				14.0	3.0	96	83		0.03	0.04	4	17

copper-nickel ore is shown in Figure 7 in simplified form. The two crushing-grinding options previously described are shown in their proper perspective.

The variable of underground vs open pit mining would affect only the primary crushing aspect of the flowsheet. For either crushing-grinding system the primary crusher would be underground in that type of mining option, and on the surface with an open pit operation. For both mining options, all facilities from the coarse ore storage on would be located on the surface. Of course, since open pit operations average larger in capacity than underground operations, their corresponding processing facilities would also be larger in scale.

Briefly described, the flowsheet consists of size reduction of mined ore either by conventional three-stage crushing followed by rod mill-ball mill grinding, or by single stage crushing followed by autogenous and pebble mill grinding. Both systems result in ground material with the sulfide minerals sufficiently liberated to produce the desired recovery through six flotation stages, including regrinding of intermediate (middling) material to liberate the contained sulfide. Final concentrate (froth) and tailing (sand) products are then distributed to the smelter and tailing pond, respectively.

Process Model Scale and System Variations

To compare the economics and potential impacts of scale, consideration must be given to the designed tonnage rate and the method of grind. (As described earlier, only bulk flotation of Cu-Ni concentrate will be considered in the models). Accordingly, the following ten model variations were selected to

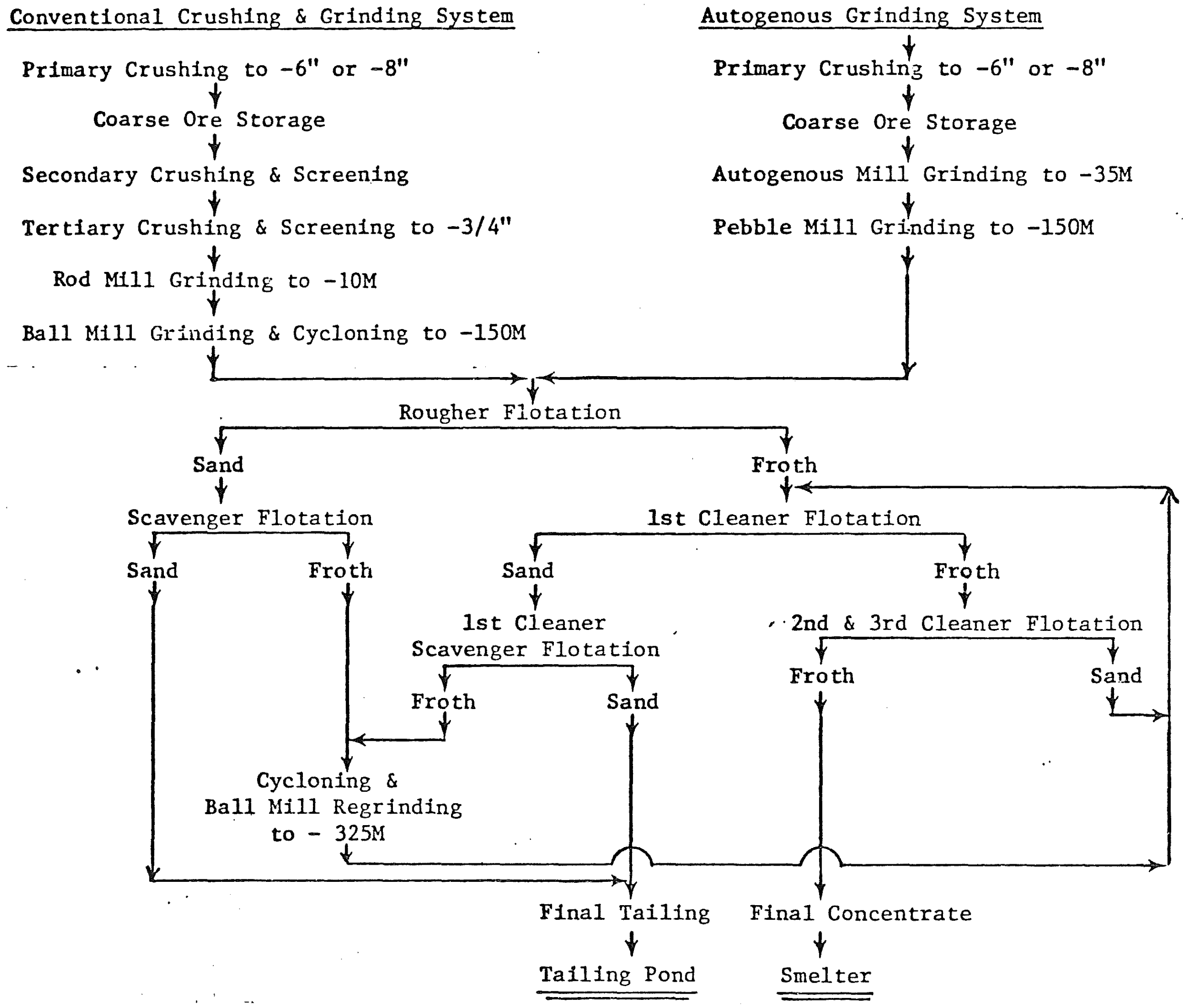


Figure 7
Simplified Model Processing Flowsheet
Minnesota Cu-Ni Ore

coincide with both the mining and smelting models, and to develop the unit capital and operating cost data necessary to scale processing operations between 4 and 20 X 10^6 mtpy crude ore:

Processing Model Variations

Model 1: 7.94 X 10^6 mtpy crude ore, primary crushing, open pit mining

A. conventional grinding

B. autogenous grinding

Model 2: 7.94 X 10^6 mtpy crude ore, no primary crushing, underground mining

A. conventional grinding

B. autogenous grinding

Model 3: 11.33 X 10^6 mtpy crude ore, primary crushing, open pit mining

A. conventional grinding

B. autogenous grinding

Model 4: 11.33 X 10^6 mtpy crude ore, no primary crushing, underground mining

A. conventional grinding

B. autogenous grinding

Model 5: 20.00 X 10^6 mtpy crude ore, pri ary crushing, open pit mining

A. conventional grinding

B. autogenous grinding

Plant Facility Capital and Operating Costs

Capital Cost Considerations--The processing plant capital expenditure estimates do not include the tailing disposal and water system, general and administrative facilities and an optional concentrate filtration system, all of which are dealt with separately. Basically, this capital estimate includes only what is found within the confines of the plant itself.

It was convenient to divide the processing plant capital expenses into eight categories or areas of primary function for comparison of relative cost variation with size and grinding system. These areas are summarized below:

1) Primary crushing - all equipment and materials including primary crusher, feeders, conveyors, dust collectors, support equipment.

2) Coarse ore storage - all necessary feeders, conveyors, weightometers, dust collectors.

3) Secondary crushing - all necessary screens, crushers, conveyors, feeders, bins, weightometers, dust collectors.

4) Conventional grinding - all necessary feeders, weightometers, conveyors, rod mills, ball mills, hydrocyclones, pumps, dust collectors, samplers, flotation machines, thickeners, regrind ball mills, distributors, tanks.

5) Autogenous grinding - primary autogenous mills, pebble screens, splitters, conveyors, classifiers, pumps, secondary pebble mills, cyclones, samplers, distributors, flotation machines, thickeners, regrind ball mills.

6) Tailing thickening - pumps, samplers, blenders, distributors, thickeners.

7) Reagents - mix tanks, pumps, storage tanks, unloading facilities, storage bins, feeders, ball mill, cyclone, dust collectors.

8) Ancillary - air compressors, air receivers, air dryers.

Within each of the above areas capital cost estimates include all site preparation such as excavation, facility installation such as buildings and structures, piping and electrical, equipment as installed, contractor engineering and supervision fees, 4% Minnesota state tax, and 10% contingency.

As described earlier, eight categories or areas of the plant which were used to generate capital expenditures are those areas contained within the plant itself. Table 3 summarizes these capital costs by area for each of the ten model variations described above. Costs are on the basis of capital cost (1977 $) per annual metric ton of crude ore capacity, $/mtpy, and consist of total direct capital required for a turn-key operation (i.e. equipment, buildings, construction, and installation). The grinding circuit options are further broken down to compare conventional and autogenous system capital costs.

This information will not be discussed at this point, other than to point out the obvious conclusions (from Table 3) that the unit capital cost decreases with an increse in tonnage; and that the grinding and flotation areas account for 60 to 80% of the total capital expenditure for the plant.

Table 3. Unit capital cost breakdown for processing plant variations by area or function. (1977 dollars)

PARAMETER	MODEL VARIATION 1 A	1 B	2 A	2 B	3 A	3 B	4 A	4 B	5 A	5 B
Design Capacity, 10^6 mtpy Crude Ore	7.94	7.94	7.94	7.94	11.33	11.33	11.33	11.33	20.00	20.00
Primary Crushing Included	yes	yes	no	no	yes	yes	no	no	yes	yes
Grinding System[a]	conv	auto	conv	auto	conv	auto	conv	auto	conv	auto
Capitol Cost, Unit basis, $/mtpy										
Primary crushing[b]	1.1	1.1			1.0	1.0			0.8	0.8
Coarse ore storage	0.6	0.7	0.6	0.7	0.6	0.7	0.6	0.7	0.5	0.6
Secondary crushing	2.3		2.3		2.1		2.1		1.6	
Conventional grinding	6.1		6.1		5.7		5.7		4.6	
Autogenous grinding[c]		6.4		6.4		5.9		5.9		4.8
Bulk flotation	3.4	3.4	3.4	3.4	3.0	3.1	3.0	3.1	2.2	2.3
Tailing thickening	1.1	1.1	1.1	1.1	1.0	1.0	1.0	1.0	0.7	0.7
Reagent system	0.4	0.4	0.4	0.4	0.3	0.3	0.3	0.3	0.2	0.2
Ancillary	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1
Total $/mtpy Crude Ore	15.1	13.2	14.0	12.1	13.8	12.1	12.8	11.1	10.7	9.4
Total Connected HP, 10^3 HP	44.3	44.8	43.4	43.9	61.7	61.9	60.3	60.6	106.3	105.9

[a]Conv - Conventional rod mill-ball mill grinding; auto - autogenous mill-pebble mill grinding

[b]When no primary crushing is indicated it is considered a mining function such as in the case of an underground mining operation.

[c]Autogenous grinding replaces both secondary crushing and conventional grinding.

Major Equipment Capital Cost, 20.00 X 10^6 mtpy Crude Ore Plant--Table 4 breaks down the major equipment needs for each of the eight areas, using the 20.00 X 10^6 mtpy model size as an example (the majority of the information contained in these tables was obtained from the AMAX report by Stearns-Roger previously noted). Using this tabulation, equipment costs for the major areas of a processing plant can be compared as follows:

EQUIPMENT AREA	CAPITAL COST DISTRIBUTION BY GRINDING SYSTEM, %	
	Conventional	Autogenous
Primary Crushing	7.0	7.6
Coarse Ore Storage	4.2	4.5
Secondary Crushing	16.2	-
Concentrator		
Conventional Grinding	67.1	-
Autogenous Grinding	-	82.0
Tailing Thickening	4.2	4.5
Reagent	1.1	1.2
Ancillary	0.2	0.2
Total	100.0%	100.0%

Operating Cost Considerations--Operating costs from AMAX data include supervision, operation and maintenance manpower requirements, supplies, materials, and power. Operating costs were generated on the basis of \$/mt crude ore treated and will be compared in that manner.

Since the tailing pond and water system personnel were considered part of the processing system, these operating costs are included in the processing

Table 4. Major equipment by area or function for a 20.00 X 10^6 mtpy crude ore Minnesota copper-nickel processing plant.

		ESTIMATED COST $\$10^6$(1977)
1) <u>PRIMARY CRUSHING</u>		
1 - Primary gyratory crusher, 60" X 109", 1000 HP motor complete		2.4
Miscellaneous feeders, conveyors, tripper conveyors		1.0
Miscellaneous dust collectors, cranes, chutes, bins		0.4
Installation Labor		0.6
TOTAL		4.4
Connected HP	2450	
2) <u>COARSE ORE STORAGE</u>		
15 - Pan feeders, 48" X 12', 25 HP		0.7
5 - conveyors		1.1
Miscellaneous weightometers, tramp iron magnets, metal detectors		0.1
Miscellaneous dust collectors, chutes, bins		0.3
Installation Labor		0.4
TOTAL		2.6
Connected HP	1570	
3) <u>SECONDARY CRUSHING</u> (Conventional Grinding Only)		
20 - Sizing screens		0.7
15 - Standard and short head cone crushers		4.4
Miscellaneous conveyors, feeders, weightometers, pumps		2.0
Miscellaneous dust collectors, chutes, bins, cranes		1.7
Installation Labor		1.3
TOTAL		10.1
Connected HP	9860	

Table 4 (contd.)

4) CONVENTIONAL GRINDING CONCENTRATOR		ESTIMATED COST $\$10^6$(1977)
10 - Rod mills, 14' diam X 18' long, 1750 HP complete		8.1
10 - Ball mills, 16½' diam X 28' long, 4500 HP complete		15.7
5 - Regrind ball mills, 9½' diam X 18' long, 700 HP		1.9
55 - Hydrocyclones and cyclones		0.3
40 - Rougher flotation cells, 500 ft^3 each		0.7
140 - Rougher scavenger flotation cells, 500 ft^3 each		2.6
70 - First cleaner flotation cells, 180 ft^3 each		0.7
60 - First cleaner scavenger flotation cells, 180 ft^3 each		0.6
60 - Second cleaner flotation cells, 100 ft^3 each		0.4
60 - Third cleaner flotation cells, 50 ft^3 each		0.2
2 - Concentrate thickeners, 100' diam		0.3
5 - Middling thickeners, 100' diam		0.6
Miscellaneous pumps		1.0
Miscellaneous feeders, conveyors, distributors, samplers, tanks, bins, dust collectors, etc.		4.3
Installation Labor		4.5
TOTAL		41.9
Connected HP	85,050	

5) AUTOGENOUS GRINDING CONCENTRATOR		
6 - Autogenous mills 28' diam X 15', 3500 HP, complete		15.3
12 - Pebble mills 15½' diam X 31', 2500 HP, complete		12.2
6 - Regrind ball mills, 9½' diam X 18', 700 HP, complete		2.3
12 - Spiral classifiers, 78", 48 HP, complete		1.8

Table 4 (contd.)

		ESTIMATED COST $\$10^6$(1977)
5) <u>AUTOGENOUS GRINDING CONCENTRATOR</u> (contd.)		
42 - Rougher flotation cells, 500 ft^3 each		0.8
138 - Rougher scavenger flotation cells, 500 ft^3 each		2.6
72 - First cleaner flotation cells, 180 ft^3 each		0.8
60 - First cleaner scavenger flotation cells, 180 ft^3 each		0.6
60 - Second cleaner flotation cells, 100 ft^3 each		0.4
60 - Third cleaner flotation cells, 50 ft^3 each		0.2
2 - Concentrate thickeners, 100' diam		0.3
3 - Middling thickeners, 125' diam		0.4
Miscellaneous pumps		0.7
Miscellaneous feeders, conveyors, distributors, samplers, tanks, bins, dust collectors, screens, etc.		3.7
Installation Labor		5.1
TOTAL		47.6
Connected HP	94,340	
6) <u>TAILING THICKENING</u>		
5 - Tailing thickeners, 275' diam, 25 HP, complete		1.4
10 - Tailing thickener underflow pumps (2 lines 5 pumps each) 7240 GPM, 16 X 14", 600 HP		0.3
Miscellaneous pumps		0.1
Miscellaneous samplers, blenders, distributors, launders, etc.		0.3
Installation Labor		0.5
TOTAL		2.6
Connected HP	6520	

Table 4 (contd.)

		ESTIMATED COST $\$10^6$(1977)
7) REAGENTS		
Reagent mixing tanks and storage tanks		0.2
1 - Ball mill, 6' diam X 8' long, 150 HP		0.1
Miscellaneous pumps, bins, feeders, cyclones, dust collectors, etc.		0.3
Installation Labor		0.1
TOTAL		0.7
Connected HP	310	
8) ANCILLARY		
Air compressors and dryers		0.1
Installation Labor		0.1
TOTAL		0.1
Connected HP	550	

SUMMARY	CONVENTIONAL GRINDING ($\$10^6$)	AUTOGENOUS GRINDING[a] ($\$10^6$)
Total Equipment	55.0	51.6
Installation Labor	7.4	6.7
GRAND TOTAL	$\$62.4 \times 10^6$	$\$58.3 \times 10^6$
Connected HP	106,310	105,930

[a]Autogenous grinding eliminates secondary crushing and replaces conventional grinding concentrator categories. In addition, minor variations in both the coarse ore storage and tailing disposal areas not treated in this table account for minor differences in the summary figures.

operating cost comparison. These facilities will be described in separate sections of this report.

A comparison is described in Tables 5 and 6 where operating costs for each model variation are summarized with primary crushing (open pit mining) and without primary crushing (underground mining). Unit operating costs with primary crushing decrease by 9 to 10% in both the conventional grinding system and the autogenous grinding system, when increasing the plant capacity from 7.94 to 20.00 X 10^6 mtpy. However, 24 to 25% unit operating cost savings are realized when considering autogenous systems over conventional grinding systems in the same tonnage range. Operating data for the models without primary crushing indicate the same cost differentials and are included only as a point of interest as the total operating costs are comparable when the underground crushing cost is included.

Plant Facility Summary of Capital and Operating Costs--Table 7 summarizes the total operating and capital cost data generated in the previous tables for each of the ten model variations. In total, savings are realized by increasing the plant capacity and also by using an autogenous grinding system rather than a conventional grinding system, as follows:

20.00 over 7.94 X 10^6 mtpy Crude Ore Capacity Advantage	
Capital cost, $/mtpy basis	29%
Operating cost, $/mt basis	10%
Personnel, mt/man-year basis	49%

Autogenous Over Conventional Grinding Advantage	
Capital cost $/mtpy basis	12%
Operating cost, $/mt basis	25%

Table 5. Operating costs for plant variations with primary crushing. (1977 dollars)

	CONVENTIONAL GRINDING						AUTOGENOUS GRINDING					
	Model 1A		Model 3A		Model 5A		Model 1B		Model 3B		Model 5B	
ITEM	No.	$/mt Crude	No.	$/mt Crude	No.	$/mt Crude	No.	$/mt Crude	No.	$/mt Crude	No.	$/mt Crude
Personnel[a]												
Supervision & Technical	35	0.113	43	0.098	62	0.081	33	0.106	41	0.093	61	0.080
Operations	106	0.270	130	0.231	191	0.190	82	0.209	102	0.181	154	0.155
Maintenance	63	0.166	69	0.128	83	0.088	54	0.142	59	0.109	71	0.075
TOTAL	204	0.549	242	0.457	336	0.359	169	0.457	202	0.383	286	0.310
Operating Supplies												
Rods & Balls		0.813		0.813		0.813		0.216		0.216		0.216
Reagents		0.157		0.157		0.157		0.157		0.157		0.157
Other		0.011		0.011		0.011		0.011		0.011		0.011
TOTAL		0.981		0.981		0.981		0.384		0.384		0.384
Maintenance Supplies												
Crusher Steel		0.044		0.044		0.044		0.011		0.011		0.011
Mill Liners		0.141		0.141		0.141		0.194		0.194		0.194
Tailing Line Replacement		0.020		0.020		0.010		0.020		0.020		0.010
Replacement Parts & Materials		0.143		0.143		0.143		0.126		0.126		0.126
TOTAL		0.348		0.348		0.338		0.351		0.351		0.341
Power Consumption, Total, 10^6 kwh/yr	(184)		(262)		(463)		(219)		(313)		(552)	
Unit Cost		0.463		0.463		0.463		0.552		0.552		0.552
General Expenses		0.033		0.033		0.033		0.033		0.033		0.033
Allocated Plant Services	12	0.117	17	0.108	28	0.085	10	0.117	14	0.108	25	0.085
GRAND TOTAL	216	2.491	259	2.390	364	2.259	179	1.894	216	1.811	311	1.706

[a]Includes tailing disposal and water system personnel.

Table 6. Operating costs for plant variations without primary crushing. (1977 dollars)

	CONVENTIONAL GRINDING				AUTOGENOUS GRINDING			
	Model 2A		Model 4A		Model 2B		Model 4B	
ITEM	No.	$/mt Crude	No.	$/mt Crude	No.	$/mt Crude	No.	$/mt Crude
Personnel[a]								
Supervision & Technical	35	0.113	43	0.098	33	0.106	41	0.093
Operations	94	0.240	121	0.215	70	0.179	93	0.167
Maintenance	55	0.145	64	0.117	46	0.121	54	0.098
TOTAL	184	0.498	227	0.430	149	0.406	188	0.358
Operating Supplies								
Rods & Balls		0.813		0.813		0.216		0.216
Reagents		0.157		0.157		0.157		0.157
Other		0.011		0.011		0.011		0.011
TOTAL		0.981		0.981		0.384		0.384
Maintenance Supplies								
Crusher Steel		0.033		0.033				
Mill Liners		0.141		0.141		0.194		0.194
Tailing Line Replacement		0.020		0.020		0.020		0.020
Replacement Parts & Materials		0.143		0.143		0.126		0.126
TOTAL		0.337		0.337		0.340		0.340
Power Consumption, Total, 10^6 kwh/yr		(180)		(258)		(214)		(308)
Unit Cost		0.454		0.454		0.542		0.542
General Expenses		0.033		0.033		0.033		0.033
Allocated Plant Services	12	0.117	17	0.108	10	0.117	14	0.108
GRAND TOTAL	196	2.420	244	2.343	159	1.822	202	1.765

[a]Includes tailing disposal and water system personnel.

Table 7. Processing plant cost summary. (1977 dollars)

PARAMETER	PROCESSING MODEL-ANNUAL CAPACITY, 10^6 mtpy CRUDE ORE									
	1		2		3		4		5	
	A	B	A	B	A	B	A	B	A	B
Design Capacity, 10^6 mtpy Crude Ore	7.94	7.94	7.94	7.94	11.33	11.33	11.33	11.33	20.00	20.00
Primary Crushing Included	yes	yes	no	no	yes	yes	no	no	yes	yes
Grinding Method[a]	conv	auto	conv	auto	conv	auto	conv	auto	conv	auto
Total Personnel[b]	216	179	196	159	259	216	244	202	364	311
Total Power Consumption, 10^6 kwh/yr	184	219	180	214	262	313	256	308	463	552
Capital Cost, Installed, Total $\$10^6$	119.9	104.8	111.2	96.1	156.4	137.1	145.0	125.8	214.0	188.0
$/mtpy Crude	15.1	13.2	14.0	12.1	13.8	12.1	12.8	11.1	10.7	9.4
Operating Cost, $/mt Crude[b]										
Materials	1.31	0.72	1.30	0.70	1.31	0.72	1.30	0.70	1.31	0.72
Personnel	0.67	0.57	0.62	0.52	0.57	0.49	0.54	0.47	0.44	0.42
Power	0.46	0.55	0.45	0.54	0.46	0.55	0.45	0.54	0.46	0.55
General	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03
Total	2.49	1.89	2.42	1.82	2.39	1.81	2.34	1.77	2.26	1.71

[a]Conv - conventional grinding, Auto - autogenous grinding
[b]Includes tailing pond operating cost and personnel.

Personnel, mt/man-year basis 20%

Obviously, if it is within company needs and ability, one must consider the largest possible operation and also the alternative of autogenous grinding over conventional grinding.

Other Processing Cost Areas

General and Administration Costs--The General and Administration (G & A) cost data includes all areas of a mining operation which cannot be directly applied to any one function. This category is not a mining cost, a processing cost, or a smelting cost as it serves all areas as an overhead cost. Included are the general offices, warehouses, yards, shops, laboratories, garages, gate houses, change houses, safety and first aid, power lines, access roads, rail spurs, potable water supply, and all miscellaneous necessities of such an operation.

In summary, the cost and personnel breakdown of this category is as follows:

	MODEL SIZE, 10^6 mtpy CRUDE ORE		
COST BASIS, 1977 Dollars	7.94	11.33	20.00
Total Capital, $\$10^6$	30.0	34.0	46.0
Total Operating, $\$10^6$	6.5	6.6	6.9
Unit Capital, \$/mtpy Crude Ore	3.78	3.00	2.30
Unit Operating, \$/mt Crude Ore	0.82	0.58	0.35
Total Employees	192	194	200

In the real life situation these costs are apportioned out to the various functions on some equatable basis, such as on the ratio of total operating costs. A rough breakdown suggested by AMAX shows the following distribution of the total G & A capital and operating cost:

Mining	50%
Mill	25%
Smelter	20%
Refinery	5%
Total	100%

Applying this breakdown to the G & A cost and personnel data results in capital, operating and personnel additions to the processing plant data, shown in Table 8. Apparently, although capital requirements increase more proportionately to the model capacity (about $\$10^6$ for each 10^6 mtpy crude capacity), the operating cost and manpower needs do not follow the same trend. One rational is the need for the facility regardless of operation size and doubling or tripling the model size does not appreciably increase operating personnel and supply requirements.

Plant Services--A service area called Plant Services is required which includes overall project maintenance from a central location. This category consists of both salaried and hourly personnel performing major maintenance over and above that which can be handled by each area within itself.

The costs consist only of labor and materials, since capital costs are included in General and Administration, and are distributed to each function within the total complex according to the same schedule. These

Table 8. General and administration cost breakdown for processing plants.

ITEM	MODEL SIZE, 10^6 mtpy CRUDE ORE 7.94	11.33	20.00
Capital Cost			
Total $\$10^6$	7.5	8.5	11.5
Unit \$/mtpy	0.94	0.75	0.57
Operating Cost			
Total $\$10^6$/yr	1.6	1.7	1.7
Unit \$/mtpy	0.20	0.15	0.09
Personnel	48	49	50

costs are already included in processing plant models as "allocated plant services" and will not be considered further in this writing.

Optional Concentrate Filtration--The possibility exists that a copper-nickel operation in Minnesota would develop as far as the concentration stage and leave the smelting and refining to be done elsewhere at a remote location. In that case, the final concentrate would at least be filtered in preparation for shipment to a distant smelter. Capital and operating cost estimates for the range of tonnage rates being considered are shown in Table 9. Operating and maintenance cost will range around $0.80/mt concentrate filtered and capital expenditure will vary between $8 and $10 per mtpy concentrate in the design range.

This facility would be attached to the mill building. Thickened concentrate would be fed to the filtration system and filter cake at 10 to 15% moisture would be the product. Water removed as filtrate could be recycled to the plant process water system directly or with the tailing streams, as needed. The filter cake would be loaded into railroad cars and shipped to a distant smelter. Additionally concentrate drying prior to shipment may be desirable or necessary, but this facility will not be considered in these models.

Tailing Disposal and Water System--Tailing disposal treatment will be treated in this model development in general terms only. Specialists in the field of tailing disposal are conducting detailed studies which will be reported on separately.

Basic Design Considerations--Economically speaking, tailing ponds are generally located as close to the processing facilities as possible, in

Table 9. Capital and operating cost summary, optional concentrate filtration system.

ITEM, Crude Ore Basis	MODEL SIZE, 10^6 mtpy CRUDE ORE 7.94	11.33	20.00
Capital Cost, Total $\$10^6$	4.0	4.3	5.2
Unit \$/mtpy	0.5	0.4	0.3
Operating Cost, including Labor, Supplies, and Power @5.5 kwh/mt Concentrate			
Total $\$10^6$/yr	0.04	0.03	0.03
Unit \$/mtpy	0.04	0.03	0.03

areas where the natural relief is conducive to efficient damming and solids storage, where borrow material is available for dam construction, where soil conditions permit effective dam construction and prevent excessive seepage, and finally in areas where other land use is not a limiting factor.

Basically, a tailing disposal system and water reclamation system (including fresh make-up water) can be fully integrated or separated completely, as dictated by existing circumstances. In a fully integrated system, tailing from the mill is placed behind a retaining dam which also collects sufficient run-off from the surrounding watersheds to supply the needed make-up water. As the tailing solids settle the clarified effluent and the collected fresh water run-off combine and are decanted off the tailing pond area, stored in a holding pond, and subsequently used in the plant complex. The potable water supply more than likely would be independent of the reclaim system, originating in a well or series of wells (this cost is considered in the General and Administration section).

At the other extreme, separated water facilities would have reclaimed water returning from the tailing pond as above to a holding pond. Fresh make-up water collected from a watershed not connected to the tailing pond system would also be pumped to the holding pond and the total used to supply plant process water needs. Potable water would originate as stated above.

Process Water Requirements--The water system required for our models must be capable of completely supplying make-up water requirements for the plant complex. The major water source will return water from the tailing pond; however, during initial stages of production when there is no return water and during emergency situations such as a failure of the return water system, the major water source would be the fresh water supply.

Hickok and Associates prepared a study report (9) for AMAX in which they detail prospective tailing disposal and water supply areas for a 7.94 X 10^6 mtpy operation. Their design water requirement was a maximum of 7000 gpm for 350 days/yr of operation, or 11,300 acre-feet of water annually. In another AMAX report (5), water requirements for 7.94 X 10^6 mtpy crude were stated as 2500 gpm excluding potable water. Considering 2500 gpm as the normal fresh water requirement for processing and 7000 gpm as a maximum fresh water requirement, the following needs can be determined:

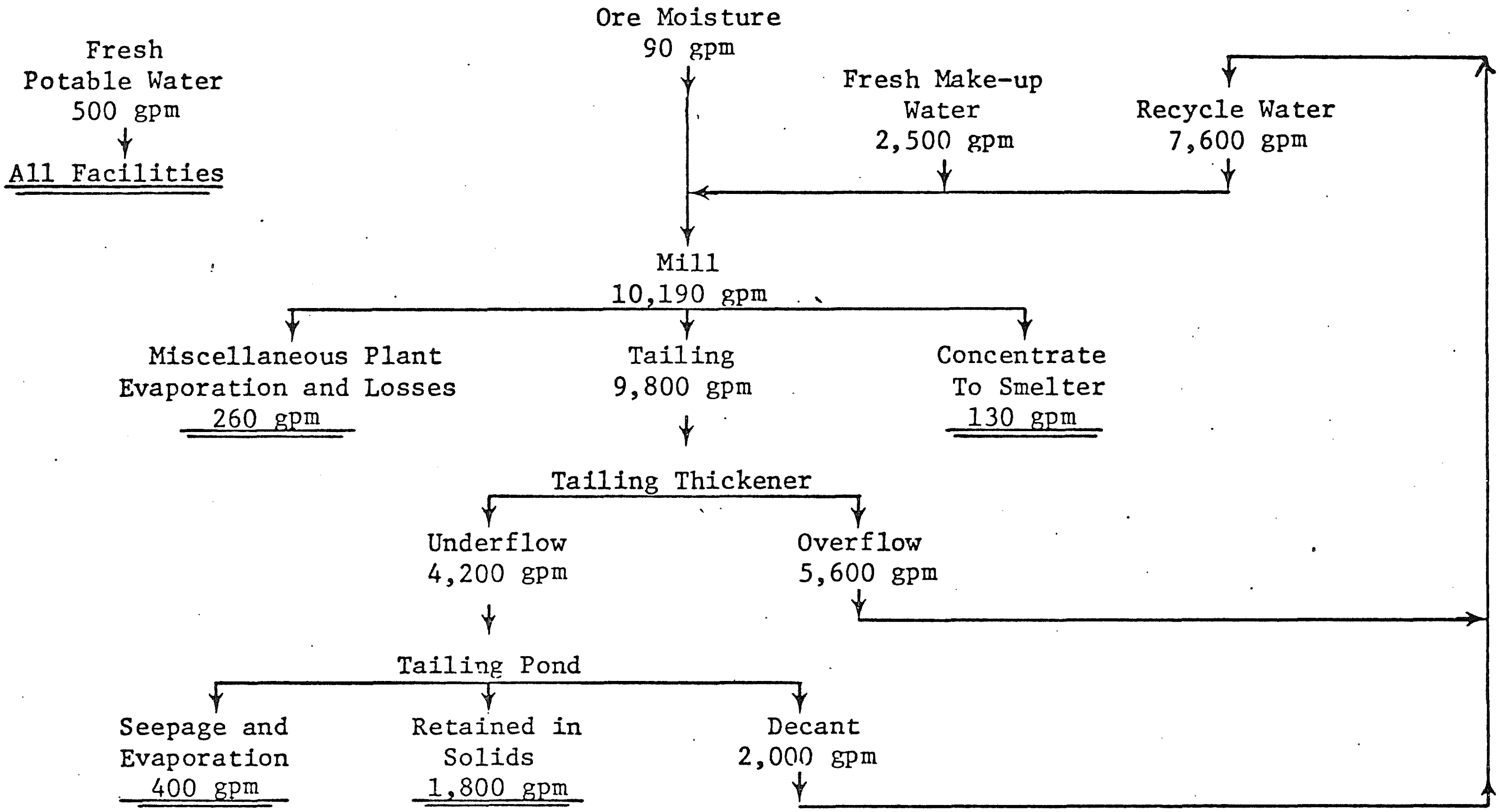
	Normal Fresh Water Use		Maximum Fresh Water Use	
Model Size, 10^6 mtpy	GPM	Acre-ft/yr	GPM	Acre-ft/yr
7.94	2,500	4,050	7,000	11,300
11.30	3,500	5,750	10,000	16,100
20.00	6,300	10,150	17,600	28,300

This water use is then 160 gal/mt crude on the average and 445 gal/mt crude for a maximum use value.

Figure 8 is a water balance for the 7.94 X 10^6 mtpy operation, showing approximately the path of water through the processing system. The quantities as shown would be proportional for the other model sizes and essentially identical with either conventional or autogenous grinding consideration.

Of the total fresh make-up water requirement, the major loss (72%) is due to water retained in the tailing solids. Of the remaining 28% in losses, savings can be realized by filtering the concentrate prior to delivering it to the smelter, by reducing the miscellaneous plant losses in such areas as

Figure 8
Process Water Balance for 7.94 X 10^6 mtpy Crude Ore Plant.



wash-down water, and by improving the tailing pond seepage collection system.

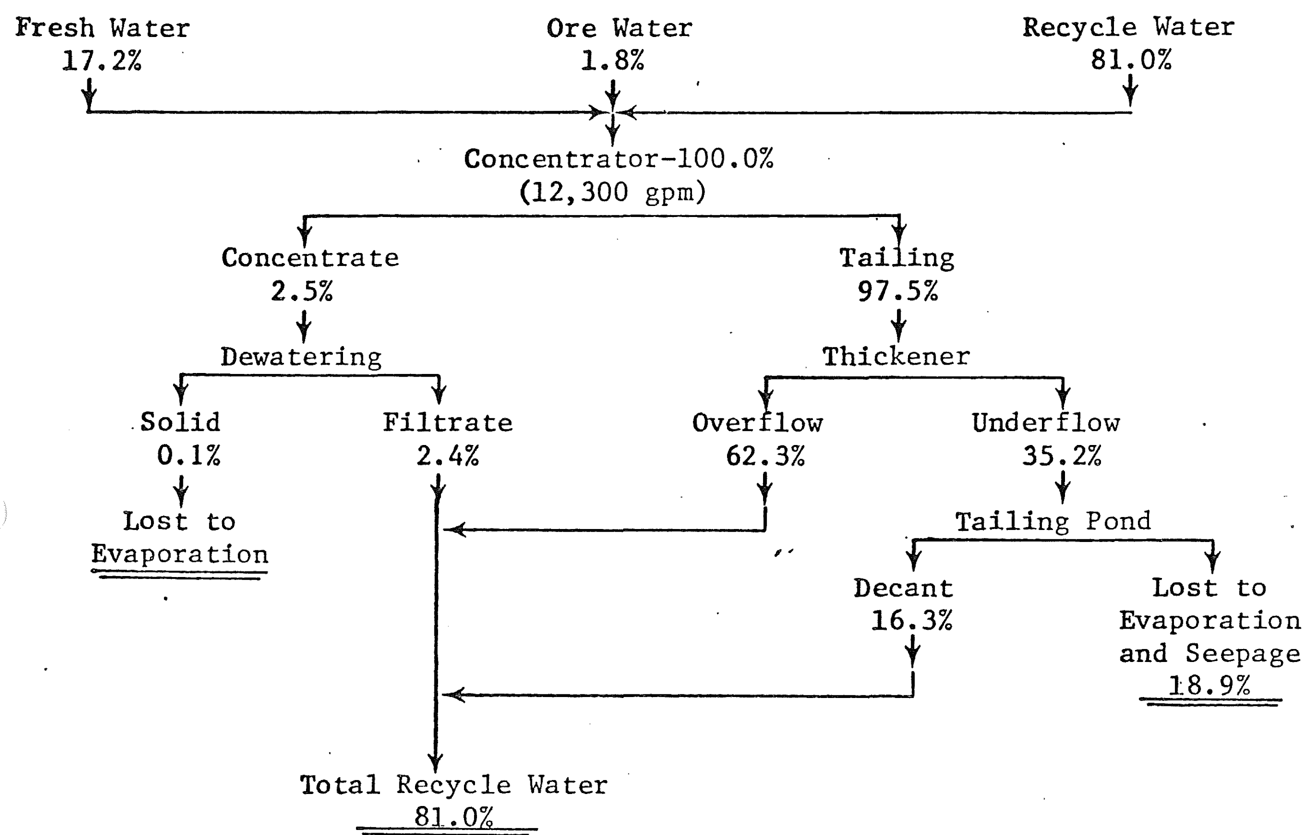
Nebeker and Cooper (10) provided an in-depth view of the water requirements for a hypothetical 7.94 X 10^6 mtpy crude copper concentrator in Arizona. Fresh water to replace that lost in the system was shown as 2120 gpm, or 135 gal/mt crude ore (Figure 9). Considering the additional complexity of the Minnesota copper-nickel ore treatment, the estimate of 160 gal/mt crude compares favorably with the Arizona figure.

Water Quality Considerations--In designing any water system, primary consideration must be given to the effects on the existing hydrologic environment of any change in the local conditions. Effects on existing downstream conditions must be minimized as to water volume and quality. Tailing dams seep, in fact they must seep or the build-up in hydrostatic pressure could cause a failure of even the strongest dam. However, controls can be placed downstream of the dam to insure minimal influence of seepage on the existing hydrology. The dam can be built with a semi-impervious core and seepage through the dam can be collected by secondary dams and/or wells downstream and pumped back to the water circulation system.

Chemical Reagents--Water quality is a serious question, particularly in seep waters that are not collected and recycled. Investigations at the MRRC indicate that chemical reagents necessary in the flotation system are basically of two types:

1) Collectors such as xanthates with long carbon chains which completely ionize and tend to form insoluble metal salts. Collector ions are strongly

Figure 9.
Water Balance for a Hypothetical Arizona 7.94 X 10^6 mtpy Copper Concentrator.
(values are given as a percent of total required-12,300 gpm)



absorbed on metal sulfide surfaces and therefore tend to remain with the concentrate (approximately 90%) leaving residual concentrations of 1-2 ppm in the water.

2) Alcohol type frothers such as MIBC are used to stabilize the froth carrying the concentrated sulfide away from the gangue minerals. Frothers do not ionize and are not absorbed which results in approximately 90% residual concentration in the water system, or 8-15 ppm.

Both collectors and frothers appear to decompose readily and reportedly would be absent in tailing pond waters within a few weeks. At this time their products of decomposition are not known.

<u>Heavy Metals</u>--The question of heavy metal ions and other potentially harmful contaminants other than chemical reagents in the tailing material, and the possibility of leaching of these elements into tailing water, has not yet been determined. Leaching characteristics and the potential impacts of leached elements in waste waters will be discussed in Study reports by the Physical and Biological Science groups. At this point, concentrations of potentially harmful elements appear to be within recognized limits of safety.

<u>Size and Location Considerations</u>--The tailing disposal system must handle 90 to 95% of the crude ore processed in the plant. Assuming no use of this material other than disposal is considered, annual amounts for each model are:

Model Size, 10^6 mtpy	Tailing Disposal, 10^6 mtpy
7.94	7.54
11.33	10.76
20.00	19.00

Using a settled solids density of 80 lb/ft^3 as suggested by AMAX and an operating life of 20 to 40 years, volume requirements for the above models are:

	Tailing Disposal, Requirements	
Model Size, 10^6 mtpy	Annual Volume $10^6 ft^3$	Total Volume $10^9 ft^3$
7.94	208	4.2 - 8.3
11.33	297	5.9 - 11.9
20.00	524	10.5 - 21.0

Translating these volume requirements to simple geometric terms of a square tailing pond with an effective height of 100 ft, areas required would be:

	Tailing Pond Area Range	
Model Size, 10^6 mtpy	$10^6 ft^2$	Acres
7.94	42 - 83	960 - 1900
11.33	59 - 119	1350 - 2730
20.00	105 - 210	2410 - 4820

Therefore, tailing ponds will range in size between 1000 and 5000 acres for the models as developed, providing the effective height is 100 ft and the anticipated operating life ranges between 20 and 40 years.

Tailing Pond Dam Construction Material--A typical tailing product will contain 50 to 60% + 200M material, which would be the primary material used in constructing permanent tailing pond dikes. Typically, tailing pumped to the pond area is separated into coarse and fine fractions in cyclones and the coarse used to form the retaining dikes. Fines are discharged behind the dikes and allowed to settle.

Another viable source of dike material is waste rock from the mine area. Economics dictate, but if local borrow material is not available and the coarse fraction of the plant tailing is not sufficient, waste rock would have to be used. If this is done space requirements for waste rock storage are reduced as are potential impacts from such a structure.

Capital and Operating Cost Data--Tailing pond and water systems for the 7.94 and 20.00 X 10^6 mtpy models are outlined in Tables 10 and 11, and both capital and operating costs generated for the three model sizes in Table 12, based on AMAX data (11). These systems contain tailing lines, water system and emergency tailing pond categories, and consist of the following major cost areas:

Tailing line - Starter dikes constructed of borrow material to contain the tailing product until permanent dikes are built.

- A seepage interceptor system to collect downstream seepage waters emitted from the pond system and return them to the pond.

- Dual tailing disposal line, urethane coated to improve wear characteristics, to carry the tailing slurry from the plant to the pond distribution dystem.

- Tailing cyclones to remove the coarse fraction for constructing permanent tailing dams.

- A tailing spigot system to distribute the tailing slurry within the confines of the pond.

- Water reclaim line and pump barge system to decant clarified water from the pond and return it to the plant system.

Table 10. Tailing pond and water system capital costs - 7.94 X 10^6 mtpy plant.

Tailing System		Cost, $\$10^6$
Starter dike, 30' high, 28' wide, 3,200' long		0.54
Seepage control system		0.25
Dual tailing slurry line, 16" diam, 35,000' long, urethane coated		2.98
Tailing cyclone, 50 units, urethane coated		0.19
Tailing spigot system		0.11
Miscellaneous site preparation		0.50
TOTAL		$4.57
Reclaim Water System		
Reclaim water line, 20" diam, 37,000' long		1.17
Reclaim water pump barge, 500 HP		0.72
Power Line substation		0.52
Miscellaneous site preparation		0.50
TOTAL		$2.91
Fresh Water System		
Fresh water line, 12" diam, 40,000' long		1.21
Pump system, 180 HP, 2,500 gpm		0.04
TOTAL		$1.25
Emergency Tailing Pond		
Retaining dike, 10' high, 5,500' long		0.13
Miscellaneous pipes, valves, etc.		0.03
Miscellaneous site preparation		0.26
TOTAL		$0.42
GRAND TOTAL	680 HP	$9.15

Table 11. Tailing pond and water system capital costs - 20.00 X 10^6 mtpy plant.

Tailing System		Cost, $\$10^6$
Starter dike, 30' high, 28' wide, 9,000' long		1.55
Seepage control system		0.25
Dual tailing slurry line, 30" diam, 93,500' long, urethane coated		7.36
Tailing cyclone, 130 units, urethane coated		0.50
Tailing spigot system		0.29
Miscellaneous site preparation		0.80
TOTAL		$10.75
Reclaim Water System		
Reclaim water line, 30" diam, 47,500' long		1.97
Reclaim water pump barge, 500 HP		0.72
Power Line substation		0.55
Miscellaneous site preparation		0.80
TOTAL		$4.04
Fresh Water System		
Fresh water line, 20" diam, 21,000' long		0.67
Pump system, 350 HP, 6,300 gpm		0.07
TOTAL		$0.74
Emergency Tailing Pond		
Retaining dike, 10' high, 5500' long		0.13
Miscellaneous pipes, valves, etc.		0.03
Miscellaneous site preparation		0.26
TOTAL		$0.42
GRAND TOTAL	2350 HP	$15.95

Table 12. Tailing pond and water system cost summary.

ITEM	PLANT SIZE, 10^6 mtpy CRUDE ORE 7.94	11.33	20.00
Capital Cost Summary, $\$10^6$			
Direct Field Costs	9.2	11.1	16.0
Indirect Field Costs	0.9	1.1	1.6
Total Field Costs	10.1	12.2	17.6
Engineering & Project Management	1.0	1.2	1.7
Total Field & Engineering	11.1	13.4	19.3
Sales Tax (4% Minnesota Sales Tax)	0.2	0.2	0.3
Contingency (15% of Total Field & Engineering)	1.6	2.0	2.9
Grand Total, $\$10^6$	12.9	15.6	22.5
$/mtpy Crude Ore	1.6	1.4	1.1
Connected HP	680	1150	2350
Operating Cost Summary, $/mt Crude Ore			
Personnel[a]	12@0.03	14@0.02	18@0.02
Power Consumption, Total 10^6 kwh/yr (@1.63 kwh/mt crude)	(12.9)	(18.5)	(32.6)
Power, 1.63 kwh/mt Crude @2¢/kwh[a]	0.03	0.03	0.03
Maintenance (3% of Capital)[b]	0.05	0.04	0.04
Slurry Pipeline Replacement (15 year life)[a]	0.02	0.02	0.01
Total $/mt Crude Ore	0.13	0.11	0.10
Contingency (10%)	0.01	0.01	0.01
Grand Total, $/mt Crude Ore	0.14	0.12	0.11

[a] Included in mill operating, manpower and costs.
[b] Included in plant services operating costs.

- Power substation.

- Site preparation for these facilities.

Water system (as described in the following section but included in this list for completeness)

- A water pipe line and pumping system to provide fresh make-up water for processing, to replace that lost in the

Emergency tailing pond (this system is necessary to prevent emergency discharges of tailing material to unprotected areas)

- Retaining dike to contain emergency discharges.

- Piping, valves, etc. necessary to bypass regular tailing line to the emergency line.

- Miscellaneous site preparation for the system.

The 7.94 and 20.00 X 10^6 mtpy operation data is basically revised and adjusted data from AMAX (11), and the 11.33 X 10^6 mtpy operation was simply calculated between the other model levels. Obviously, these facilities are expensive ranging between \$13 and \$23 X 10^6 in capital expenditures and 11¢ to 14¢/mt crude in operating costs. These values include no reclamation or water treatment other than described in the tables.

Summary of Cost Data

Table 13 summarizes all cost parameters attributed to the processing plant design for the ten variations considered. In addition, an optional filtration plant is included (but not used in the total) in the event

Table 13. Summary of total processing plant model cost and employment data. (1977 dollars)

PROCESSING COST ITEMS, CRUDE ORE BASIS	MODEL 1 A	1 B	2 A	2 B	3 A	3 B	4 A	4 B	5 A	5 B
Plant Capacity, 10^6 mtpy Crude Ore	7.94	7.94	7.94	7.94	11.33	11.33	11.33	11.33	20.00	20.00
Primary Crushing Included	yes	yes	no	no	yes	yes	no	no	yes	yes
Grinding Method[a]	conv	auto	conv	auto	conv	auto	conv	auto	conv	auto
Plant										
Total Capital, $\$10^6$	119.9	104.8	111.2	96.1	156.5	137.1	145.0	125.8	214.0	188.0
Unit Capital, $/mtpy	15.1	13.2	14.0	12.1	13.8	12.1	12.8	11.1	10.7	9.4
Unit Operating, $/mt	2.49	1.89	2.42	1.82	2.39	1.81	2.34	1.77	2.26	1.71
Personnel	216	179	196	159	259	216	244	202	364	311
Power Consumption, 10^6 kwh/yr	184	219	180	214	262	313	258	308	463	552
Connected HP X 10^3	44.3	44.8	43.4	43.9	61.7	61.9	60.3	60.6	106.3	105.9
General and Administration										
Total Capital, $\$10^6$	7.5				8.5				11.5	
Unit Capital, $/mtpy	0.9				0.8				0.6	
Unit Operating, $/mt	0.20				0.15				0.09	
Personnel	48				49				50	
Tailing Disposal and Water System										
Total Capital, $\$10^6$	12.9				15.6				22.5	
Unit Capital, $/mtpy	1.6				1.4				1.1	
Unit Operating, $/mt[b]	(0.14)				(0.12)				(0.11)	
Connected HP X 10^3	0.7				1.2				2.4	
Optional Filtration Plant (not included in Total)										
Total Capital, $\$10^6$	4.0				4.3				5.2	
Unit Capital, $/mtpy	0.5				0.4				0.3	
Unit Operating, $/mt	0.04				0.03				0.03	
Total										
Total Capital, $\$10^6$	140.3	125.2	131.6	116.5	180.5	161.2	169.1	149.9	248.0	222.0
Unit Capital, $/mtpy	17.7	15.8	16.6	14.7	15.9	14.2	14.9	13.2	12.4	11.1
Unit Operating, $/mt	2.69	2.09	2.62	2.03	2.52	1.96	2.49	1.92	2.35	1.80
Personnel	264	227	244	207	308	265	293	251	414	361
Productivity, mt/man-shift	116	135	125	148	141	164	149	174	186	213
Connected HP X 10^3	45.0	45.5	44.1	44.6	62.9	63.1	61.5	61.8	108.7	108.3

[a] Conv - conventional, auto - autogenous.

[b] Tailing disposal and water system unit operating costs are built into Plant, General, and Administration data and not added into the total figures a second time.

concentrate is produced for subsequent smelting and refining in a distant facility and must be semi-dried before being transported.

All important parameters have been estimated and the totals are plotted in Figure 10 which compares unit capital and operating costs for the operations ranging from 7.94 to 20.00 X 10^6 mtpy design capacity. Of major importance then is the economy of scale data and the further advantage of an autogenous grinding system over a conventional grinding system. In scaling the cost data from an annual capacity of 7.94 to 20.00 X 10^6 mtpy, overall savings are estimated at:

Unit Capital Cost	30%
Unit Operating Cost	13%

Considering autogenous grinding over conventional grinding, estimated savings are:

Unit Capital Cost	11%
Unit Operating Cost	23%

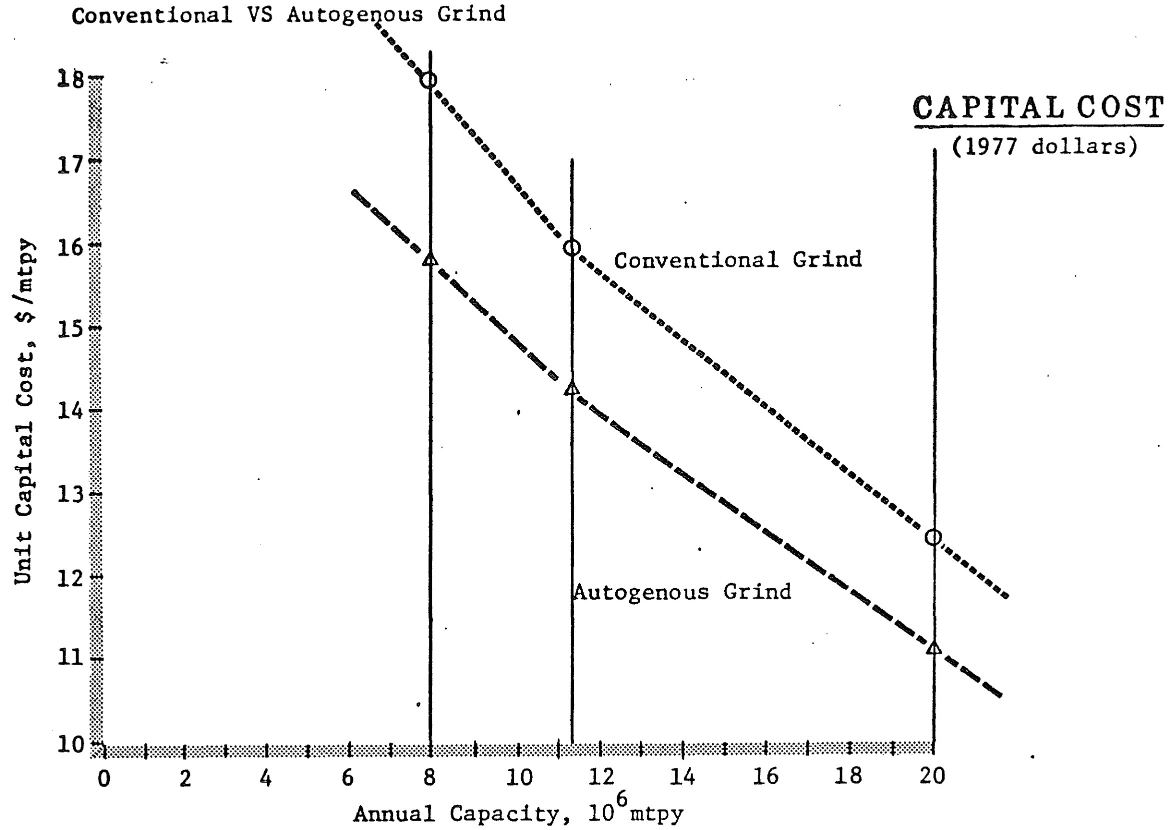
And finally, in the extreme comparison between 7.94 X 10^6 mtpy conventional grind and 20.00 X 10^6 mtpy autogenous grind, unit savings would be:

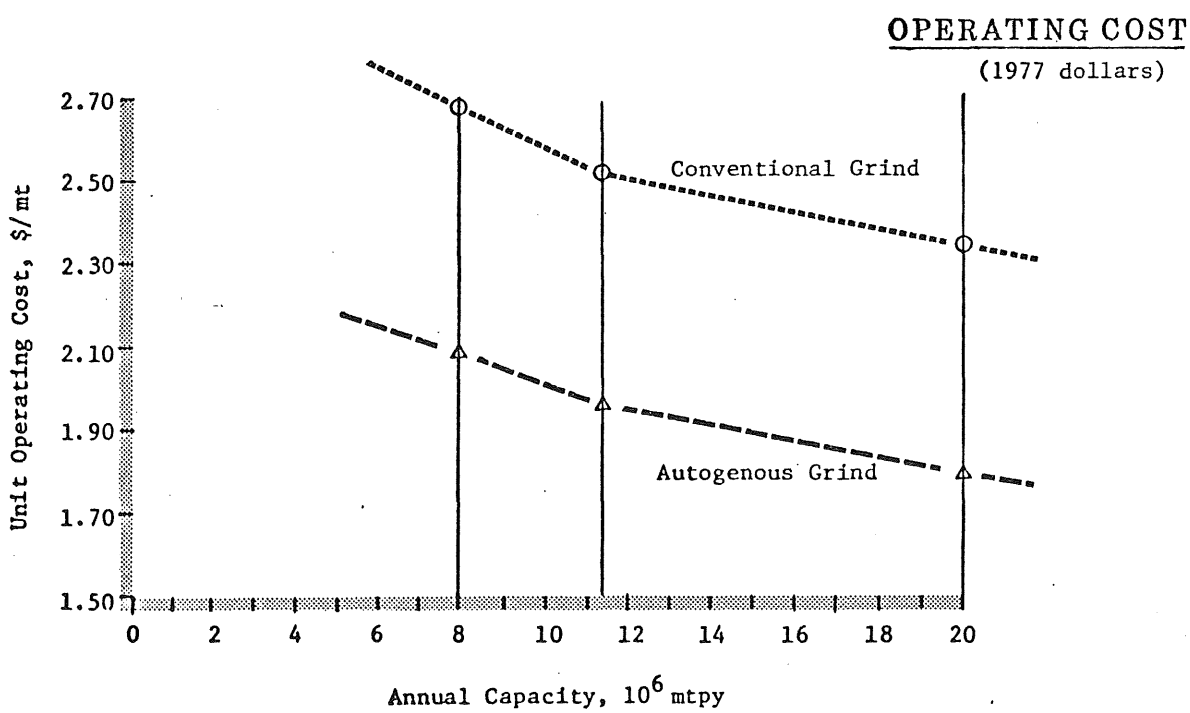
Unit Capital Cost	37%
Unit Operating Cost	33%

PROJECT CONSTRUCTION SCHEDULE

A bar graph construction schedule is shown in Figure 11 broken into three major categories: Engineering, Purchasing, and Construction.

Figure 10
Total Processing Unit Capital and Operating Cost Comparison
Conventional VS Autogenous Grind





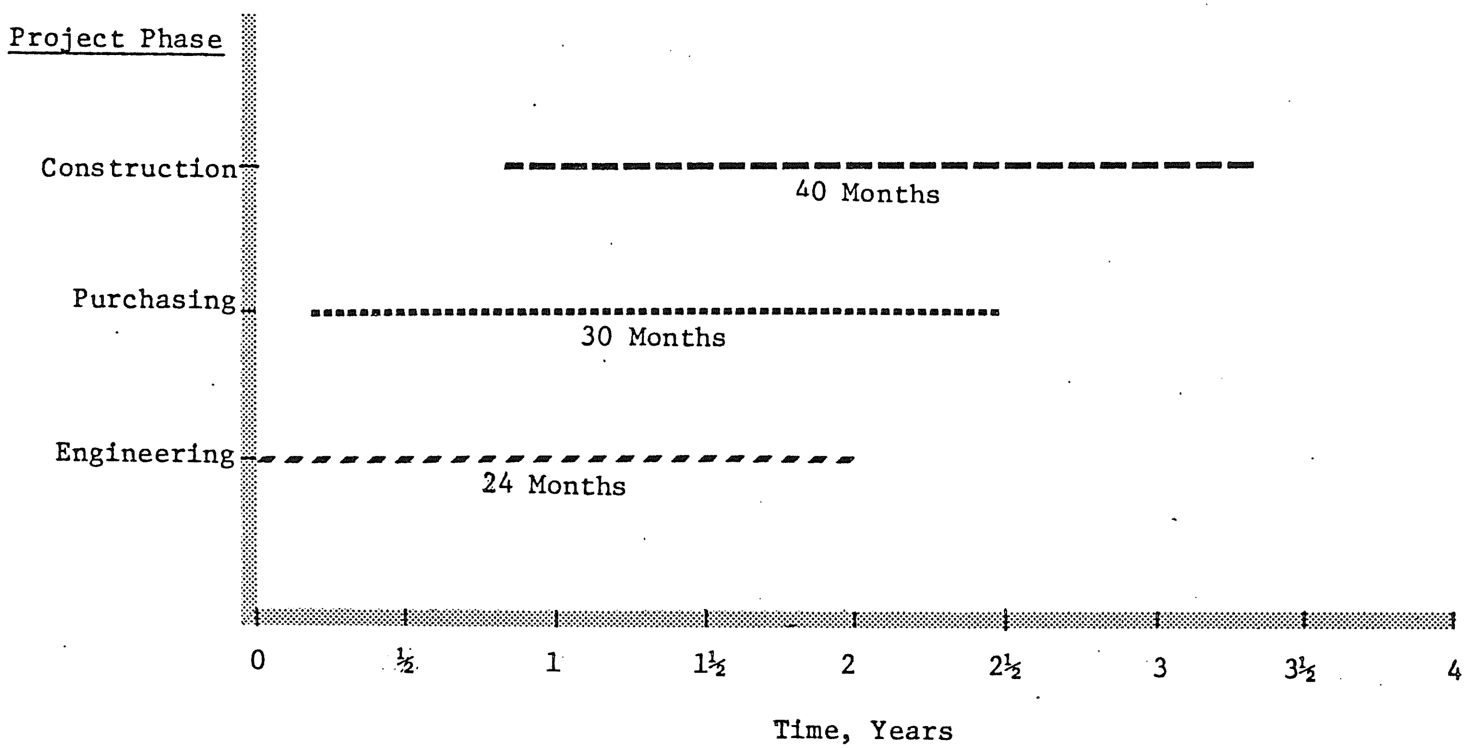


Figure 11

Typical Plant Construction Schedule

Engineering will require about two years to complete, assuming sufficient process design criteria is available at the start. This would include completed geological exploration, processing testwork and pyrometallurgical testwork. Additionally, all major company decisions would have been made, all governmental agency verdicts would have been given, and the required permits would have been obtained.

Purchasing includes ordering and taking delivery of all items and would require 2-1/3 years to complete. Certain major equipment such as grinding mills and their gears have the longest delivery and are thus the controlling feature. Such equipment would have to be specified and ordered soon after engineering is started to meet the construction schedule.

Field construction would require 2 to 2-1/2 years and should begin before the first year of engineering is complete. Ideally, the concentrator building should be erected and enclosed during the summer months, allowing inside construction during the winter months.

In total, the project construction time would be 3 to 3-1/2 years, depending on the size of the model variation.

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