Volume 2-Chapter 5

INTEGRATED DEVELOPMENT MODELS

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 disucssions, 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 mertric 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 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 m^3	=	1.308 yd^3 = 35.315 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)(degrees Fahrenheit -32)

Volume 2-Chapter 5 INTEGRATED DEVELOPMENT MODELS

5.1 INTRODUCTION AND SUMMARY OF FINDINGS

Chapters 1-4 of this volume evaluate various exploration, mining, processing, smelting and refining technologies and operating practices that are applicable in the exploitation of Minnesota's copper-nickel resources. Unlike an Environmental Impact Statement (EIS), the Regional Copper-Nickel Study does not have definite development proposals to evaluate which specify the size, design and operating procedures of the proposed development. Therefore, in order to assess the environmental, social and economic impacts of copper-nickel development on a quantitative basis, realistic hypothetical models are presented in this chapter so that impacts can be consistantly asessed in the following chapters.

The task in this section is to apply the wealth of information just presented on mining and metallurgical technology to a framework which will organize the information into a form suitable for use by the Study and by its audience. This organization must meet 2 basic requirements simultaneously in order to serve the needs of the Regional Copper-Nickel Study:

1) All of the aspects of a large mining operation which may cause significant environmental, social, or economic impacts must be quantified in a format that will facilitate the impact assessment process.

2) The above quantification of mining variables must be done in an internally consistent manner. That is, when taken together, the values used to represent a mining development must form a total picture of an integrated, coherent, reasonably-sized operation which has all the necessary facilities and resources needed to function as a viable economic entity.

Both of the above requirements are essential if a truly interdisciplinary impact assessment is to be done. In general, impact mitigation is a matter of trade-offs, with reductions in one type of impact leading to increased impacts in another area. Properly constructed development models will allow such trade-offs to be clearly identified.

To meet these needs, a set of hypothetical development models are presented in this chapter. Included are models for each phase of an operation, the mine, the mill, and the smelter/refinery complex. Each of these 3 phases will be represented by one or more models, as needed to bring out both the range of basic technological approaches available and the range of operating capacities which may be applicable to such a development in Minnesota. These operating phase models can then be combined to generate a set of integrated development models. In many cases, a variety of differing technologies are available, and specific selections for modelling purposes will be made with the goal of generating representative models, rather than of predicting or recommending the choices which might actually be made by a company developing a specific ore deposit. It is, therefore, crucial that this feature be clearly recognized. The models developed here are representative, not predictive. The subsequent impact assessment process, on the other hand, will be predicated on the development models as given sources of environmental, social, and economic changes, and will assess the consequences using predictive models. The emphasis here will be on the identification of cause and effect relationships. If a certain action is taken, then certain reactions or impacts will necessarily follow.

Once the above relationships are determined based on a reasonable set of development models, the models then become valuable tools for future use. When, and if, specific mine development proposals are made by private industry, they will most certainly differ from the development models given here in many respects. However, using the development models as points of reference, the implications of specific proposals can be quickly assessed by comparing their parameters to those of the models, and correspondingly scaling the predicted impacts as dictated by the impact assessment models. It is with these ends in mind that the following models are presented. Readers who would like more information concerning model details and the sources of information used as the basis for the models are referred particularly to the following Regional Copper-Nickel Study reports:

1) Preliminary Report-Mining and Metallurgical Technology. David L. Veith, Michael G. Pojar, George F. Weaton, Susan Hakomaki. August 15, 1976.

2) Preliminary Report-Exploration Models. William W. Ryan. June, 1978.

3) Preliminary Report-Details of the Open Pit Mine Model. Steven P. Oman. July, 1977.

4) Preliminary Report-Details of the Underground Mine Models. Steven P. Oman and William A. Ryan. February, 1978.

5) Preliminary Report-Processing Model. David L. Veith. April, 1978.

6) Preliminary Report-Tailing Basin Design. William A. Ryan. June, 1978.

7) Preliminary Report-Metallurgical Technology, Pollution and Pollution Control in the Nonferrous Metals industry. Michael G. Pojar. 1977.

8) Preliminary Report-Metallurgical Technology, Smelter-Refinery Model. Michael G. Pojar. 1978.

First, in order to describe the potential impacts of copper-nickel mining in realistic terms, model operations were constructed to span the range of capacities thought possible for the Study Area. This resulted in models ranging from 5.35×10^6 to 20.00×10^6 mtpy ore for mining and processing; and a smelter/refinery complex capable of producing 100,000 mtpy of copper plus nickel metal. There are 2 underground mining models, 5.35×10^6 and 12.35×10^6 mtpy ore, which are large when compared to existing underground mines around the world. However, with the low grade nature of the resource, the complex recovery processes required, and the current trend toward technology which favors large scale operations in the mining industry, the range stated is suitable for impact analysis. There are 2 open pit mining models, 11.33×10^6 and 20×10^6 mtpy ore, which again depict larger-than-average operations. However, the same reasoning used above for underground applies to the open pit modelling. A combination of 5.35×10^6 underground and 11.33×10^6 open pit, totalling 16.68×10^6 mtpy ore was also developed to evaluate the possibility of both mining methods being employed simultaneously.

In comparison, INCO proposed an open pit operation producing just over 12×10^6 mtpy ore, and Amax's most recent thinking at this writing is a combination mining operation consisting of a 14×10^6 mtpy open pit and a 5×10^6 mtpy underground operation. Thus the model operations herein presented are in line with industrial considerations for processing Minnesota copper-nickel resources.

In addition to providing a realistic approach to mining operations as dictated by the ore quality and the difficulty of valuable mineral separation, the larger operations were designed to produce adequate feed to allow for the production of 100,000 mtpy of copper and nickel metal from the modelled smelter/refinery operation which is a reasonable size for a new, modern smelter. Thus, the

12.35 X 10^6 mtpy underground, the 16.68 X 10^6 mtpy open pit-underground combination and the 20 X 10^6 mtpy open pit operations each result in sufficient ore to meet the above metal production requirement.

To evaluate the potential impacts due to more than one of the modelled operations in a given region, any combination of operations can be located there, dictated only by the resource and surface area available in that region. For example, resource zone 1 of the Study Area (see Figure 1, Chapter 1) has an estimated 2.4 X 10^6 mt of recoverable copper available to open pit operations, and 1.6 X 10^6 mt of recoverable copper available to underground mining operations, for a total of 4.0 X 10^6 mt of recoverable copper. Since the modeled 100,000 mtpy smelter/refinery produces approximately 85,000 mtpy of copper and 15,000 mtpy of nickel from Minnesota ore, there would be approximately 28 yr of open pit operation and 19 yr of underground operation for a 100,000 mtpy smelter before the resource in zone 1 would be exhausted. This combination could be run consecutively as 28 yr of open pit production followed by 19 yr of underground operation for a total life of 47 yr at full rated production, or simultaneously with each mine producing a portion of the smelter feed such that both operations are exhausted at the same time. In either case, the total life would be the same. Other alternatives include more than one smelter, more than 2 mining operations, larger than 100,000 mtpy metal operations, etc.

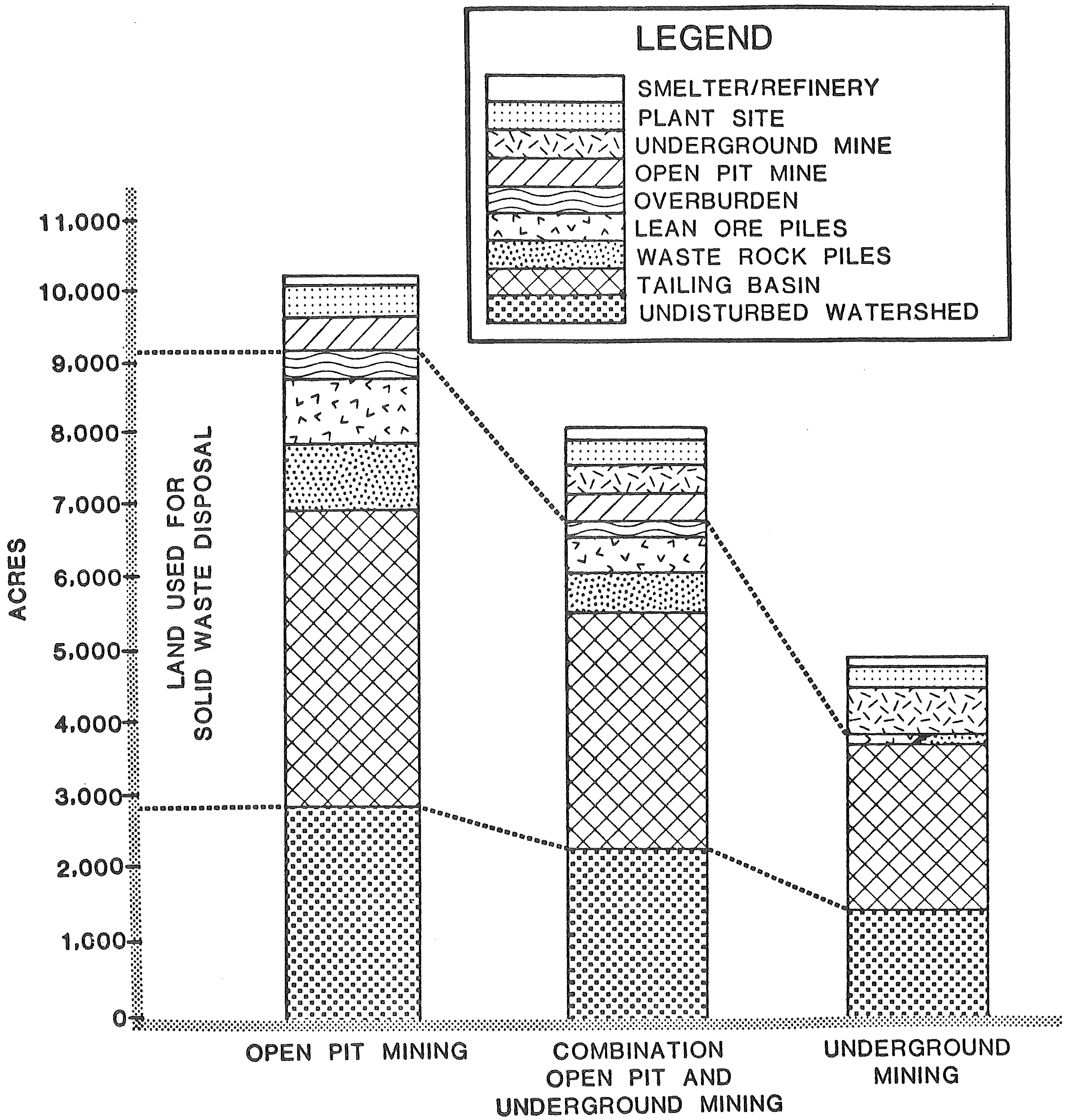
The timing and sequencing of operations in the Study Area, locations of the facilities, and interactions between mining companies all play an important part in the total development of Minnesota's copper-nickel resource. The MDNR resource estimate discussed in Volume 3-Chapter 2, shows some 4 billion mt of resource grading 0.5% copper or greater, and containing an estimated 25.8 mt of copper, of which more than two-thirds is as an underground mining resource. Thus, on the basis of resource distribution alone, one would expect underground development to exceed open pit development. In reality, as open pit development is generally less expensive than underground development, economics dictate open pit mining to be the best choice initially. Underground mining might then follow, utilizing many of the same facilities which have been capitalized by the earlier open pit operations. (See Volume 5-Chapter 14, Economic Analysis of Copper-Nickel Models).

After studying the modeling results presented in this chapter, the reader will have a clearer picture of what copper-nickel development may be like in northeastern Minnesota. The first thing that becomes obvious is the land use implications of developing mineral resources, especially low grade resources such as copper-nickel. Large quantities of land must be disturbed for solid waste disposal purposes (Figure 1), but the total amount of land disturbance can be reduced by a factor of 2 if underground mining methods can be used in lieu of open pit mining methods. (Note: The following discussion deals with the models resulting in 100,000 mtpy metal production only.)

Figure 1

The comparison of open pit mining verses underground mining presents other changes in impact levels similar to the land use example presented above. Water is an important resource for the operation of a copper-nickel mining operation. It is used as a transport medium, it can interfere with mining activities, and it can become contaminated as a result of mining and processing and pose a water pollution hazard. During the operating life of the mine and plant water can be managed on the site to the point that total recycle and reuse is possible, if

FIGURE 1 MINNESOTA CU / NI DEVELOPMENT MODELS*
LAND REQUIREMENTS



* SEE TEXT FOR MODEL SIZES

the size of disturbed land areas producing contaminated water is not too large. The potential for producing excess water in the system is 10 times greater for the open pit mine than the underground mine because of the large quantity of solid wastes produced. The large open-pit mine model produces almost 4 times more solids wastes than the underground mine model presented in this chapter.

The quantity of land disturbed and the quantity of ore and wastes that have to be hauled by truck both have environmental implications other than water quality. The production of fugitive dust is strongly related to the amount of disturbed land and the number of large, off-the-road trucks used to haul ore and waste rock since these are major sources of dust at mining operations. Once again open pit mines are more significant contributors to fugitive dust emissions than underground mines because underground mines do not produce the large quantities of waste rock which must be hauled to surface disposal sites, and they do not disturb as much surface area as do open pit operations.

It is clear from the above discussion, the models presented in this chapter, and the generalized impacts comparison presented in Figure 2, that underground mining has many environmental advantages compared to open pit mining. This is especially important since a majority of Minnesota's copper-nickel resources are at a depth requiring underground mining. Unfortunately, in the context of equivalent ore deposits, there are economic factors which make underground mining significantly less attractive economically than open pit mining. While the capital costs estimates for the large open pit mine model are 32% greater than the underground mine, the annual operating costs for the underground mine are 20% greater (Figures 3 and 4). One of the primary reasons for the higher underground operating costs is the labor intensive nature of such operations (Figure 5). Open pit operations reduce labor requirements by using larger equipment which results in higher capital costs and greater energy use (Figure 6). Notice that underground operations, while less energy intensive, use a greater percentage of the more expensive electrical energy (Figure 7).

Figures 2-7

Information is also presented for a mine model which combines open pit and underground mining methods. If maximum utilization of copper-nickel resources occurs, then this approach is the most likely option to be followed. As mentioned previously, the developer may wish to begin production utilizing only open pit methods in order to take advantage of the lower production costs and the shorter construction period, and then phase in underground mining after all or a portion of the costs of the processing plant and other surface facilities have been recovered. Parallel application of both methods is an alternative approach and the one used in the combination models presented in this chapter.

In order to produce salable products, smelting and refining systems will be required. The addition of these operations at the minesite will increase the impacts previously discussed. In comparison to the open pit mining operation (mine and plant), the smelter/refinery complex requires 40 times less land (Figure 1) and produces 76 times less solid wastes, but requires up to 3 times more water flowing in the total system. Smelting and refining are energy intensive processes and consume 33% more energy than an open pit mine and plant operation (Figure 6). By the time the contained copper and nickel reach the smelting stage the volume of material to be processed has been reduced by over 98% when compared to the amount of material removed from the open pit mine (Figure 8). This feature and the automation found in modern smelters and refineries are the principal reasons that these operations require half the number

FIGURE 2

COMPARISON OF OPEN PIT AND UNDERGROUND MINING IMPACTS

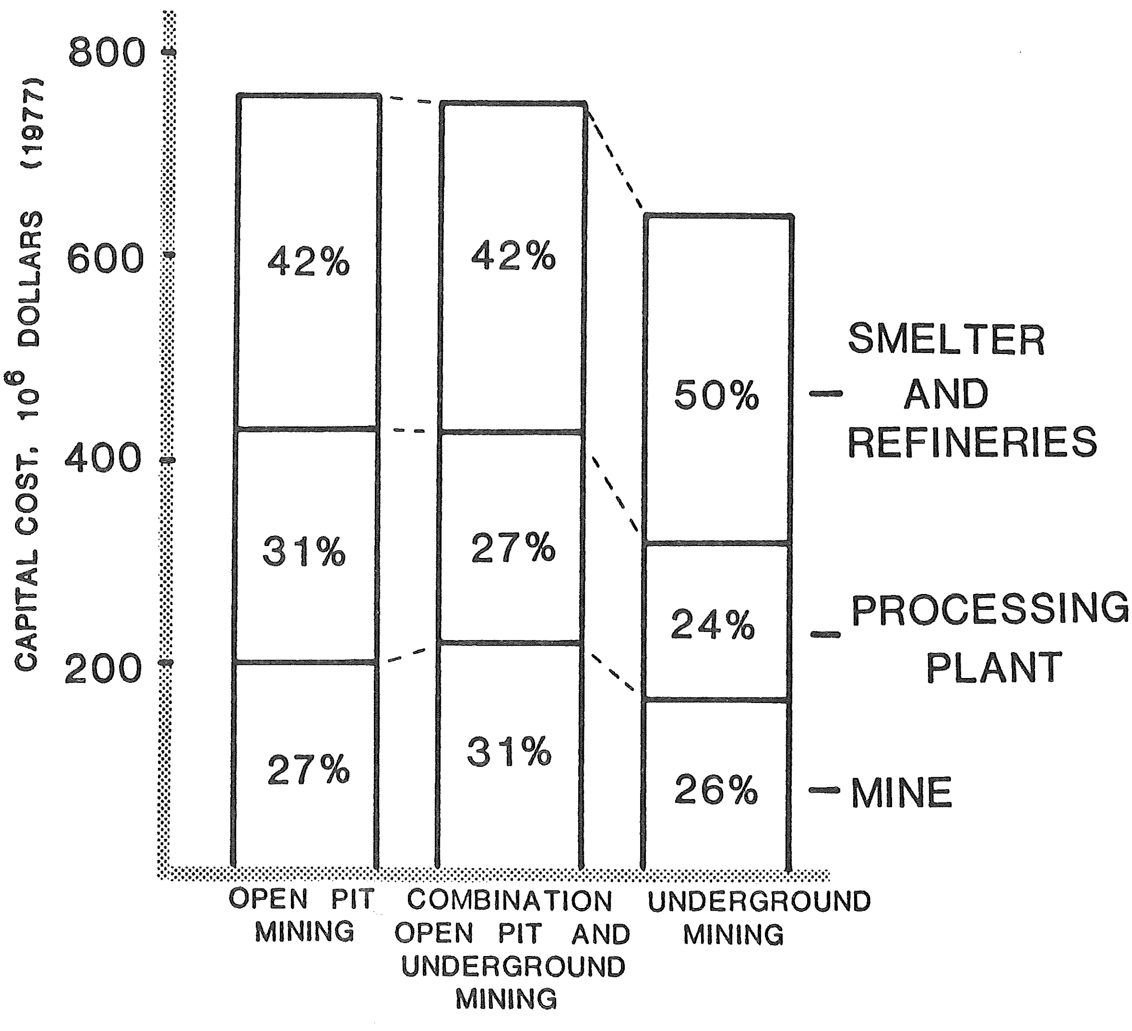
	MINING METHOD*	
I M P A C T	OPEN PIT	UNDER-GROUND
MORE LAND DISTURBED		
HIGHER WATER DISCHARGE POTENTIAL		
MORE SOLID WASTE PRODUCTION		
MORE FUGITIVE DUST EMISSION		
LESS EMPLOYMENT OPPORTUNITIES		
GREATER ENERGY DEMAND		
LESS RECLAMATION POTENTIAL		
GREATER CAPITAL COST		
GREATER OPERATING COST		
GREATER TOTAL PRODUCT COST		
GREATER OCCUPATIONAL HAZARD		
LOWER RESOURCE RECOVERY		

* THE SHADED AREA INDICATES THE MINING METHOD WITH THE HIGHER LEVEL OF THE INDICATED IMPACT PER UNIT OF ORE PRODUCED.

FIGURE 3

MINNESOTA CU/NI DEVELOPMENT MODELS*

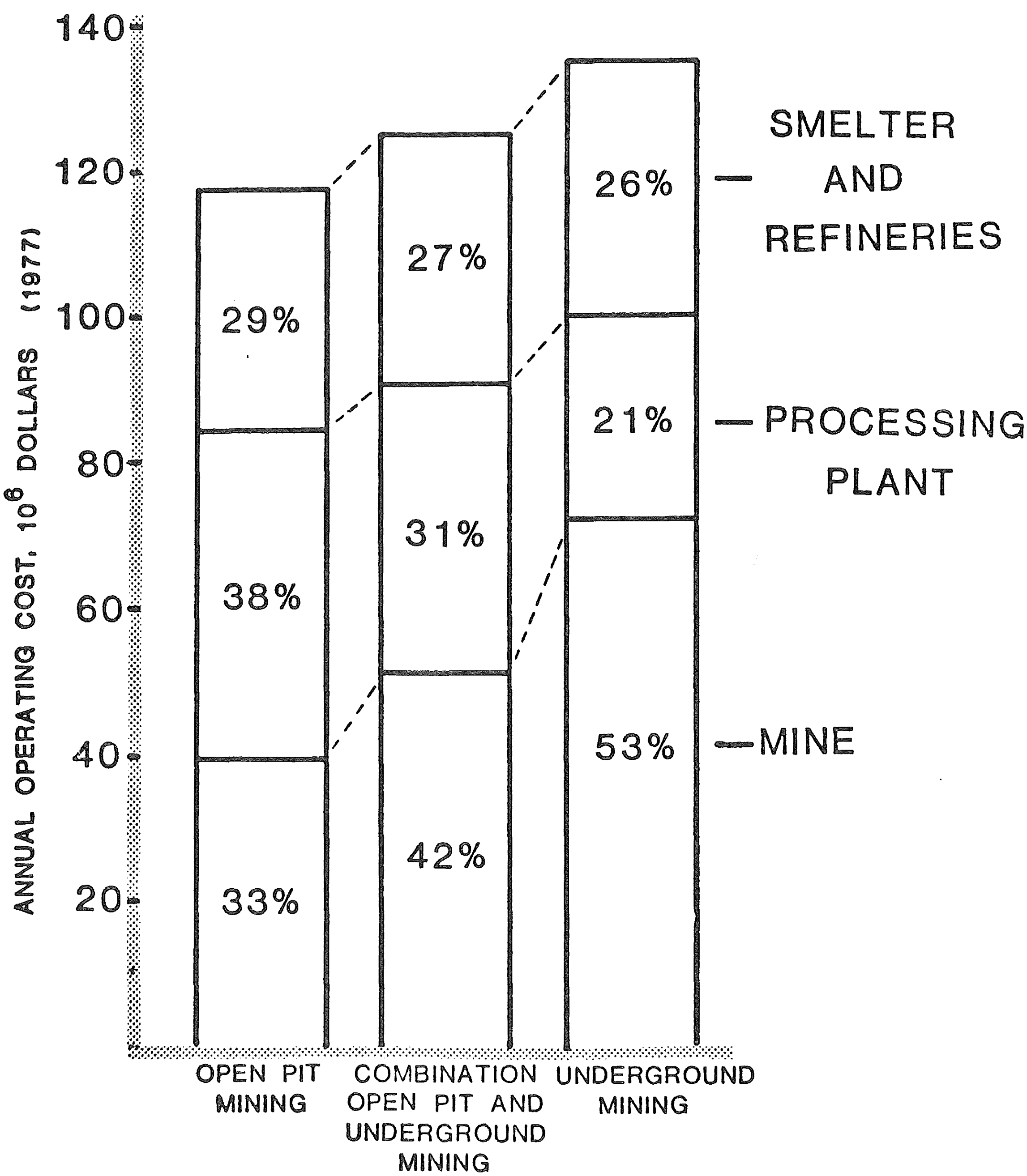
CAPITAL COST COMPARISON



* SEE TEXT FOR MODEL SIZES

FIGURE 4

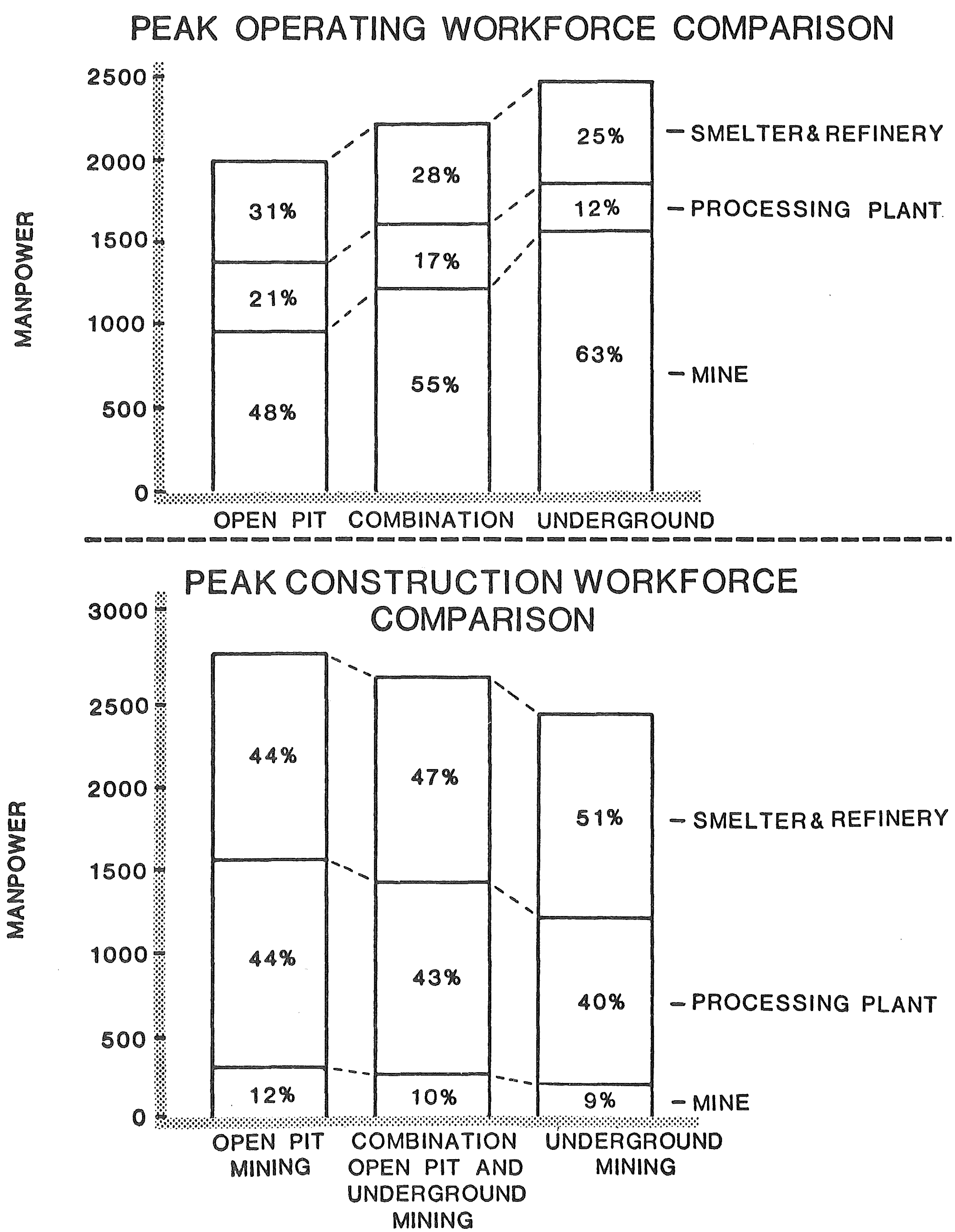
MINNESOTA CU/NI DEVELOPMENT MODELS*
ANNUAL OPERATING COST COMPARISON



* SEE TEXT FOR MODEL SIZES

FIGURE 5

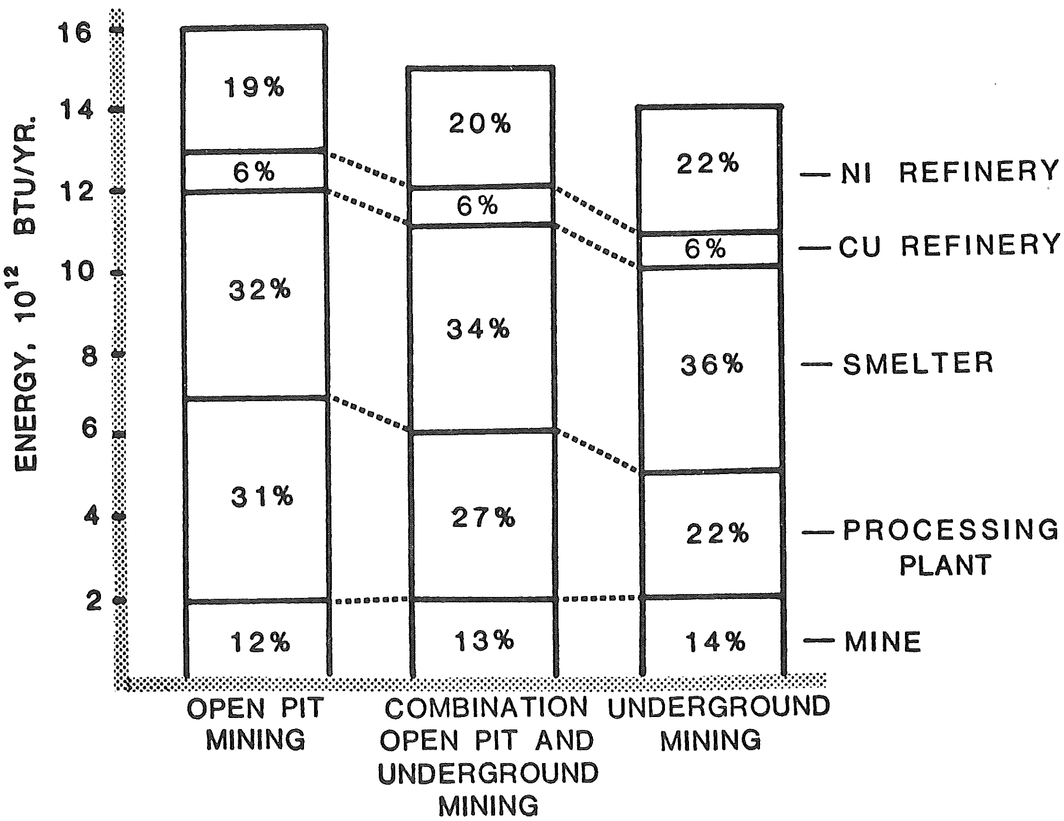
MINNESOTA CU / NI DEVELOPMENT MODELS*



* SEE TEXT FOR MODEL SIZES

FIGURE 6

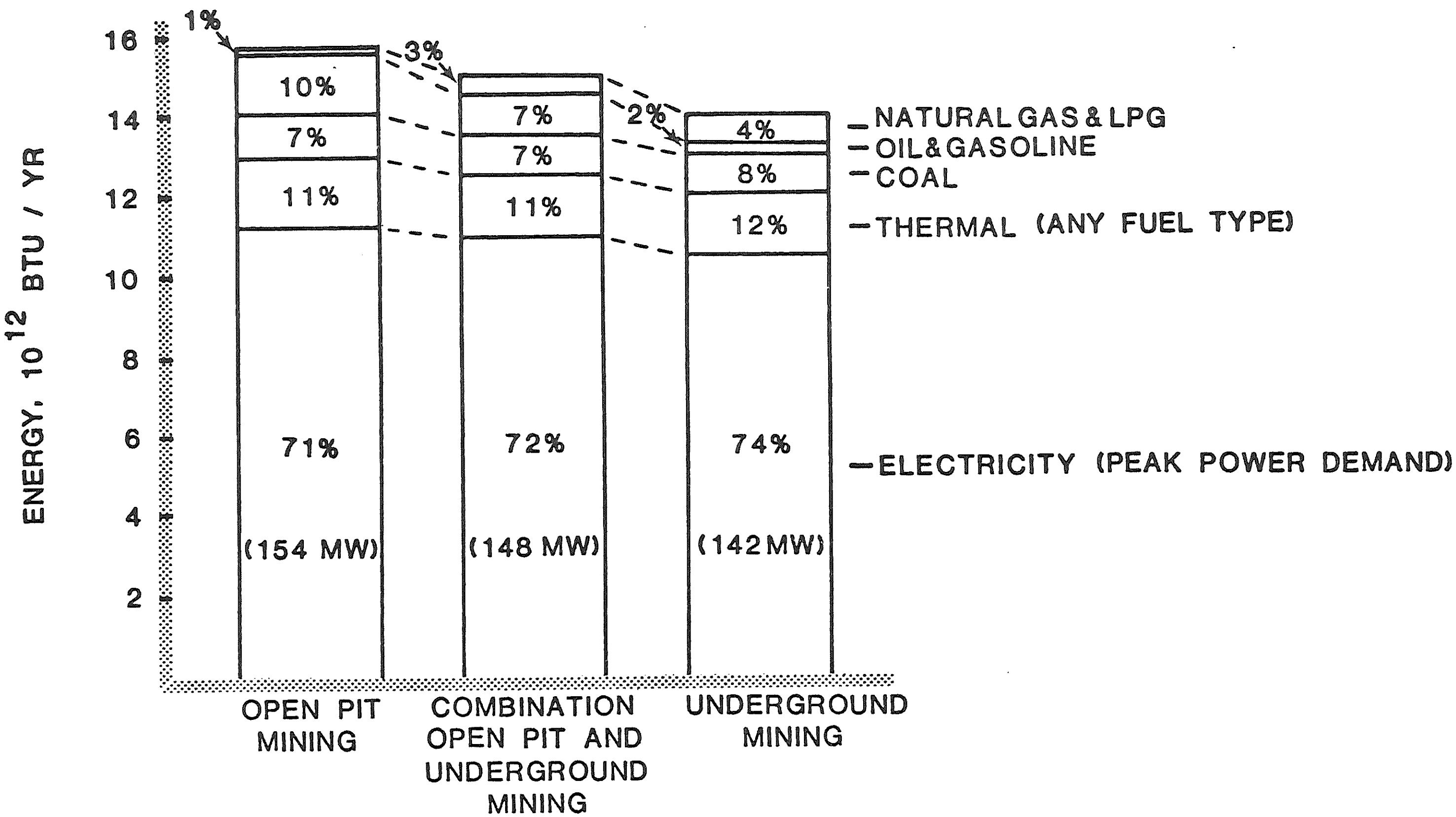
MINNESOTA CU/NI DEVELOPMENT MODELS*
ANNUAL ENERGY REQUIREMENT COMPARISON
(10^{12} BTU/YR)



*SEE TEXT FOR MODEL SIZES

FIGURE 7

MINNESOTA CU/NI DEVELOPMENT MODELS*
ANNUAL ENERGY REQUIREMENT
BY FUEL TYPE



* SEE TEXT FOR MODEL SIZES

of workers that a large open pit mine and plant require (Figure 5).

Figure 8

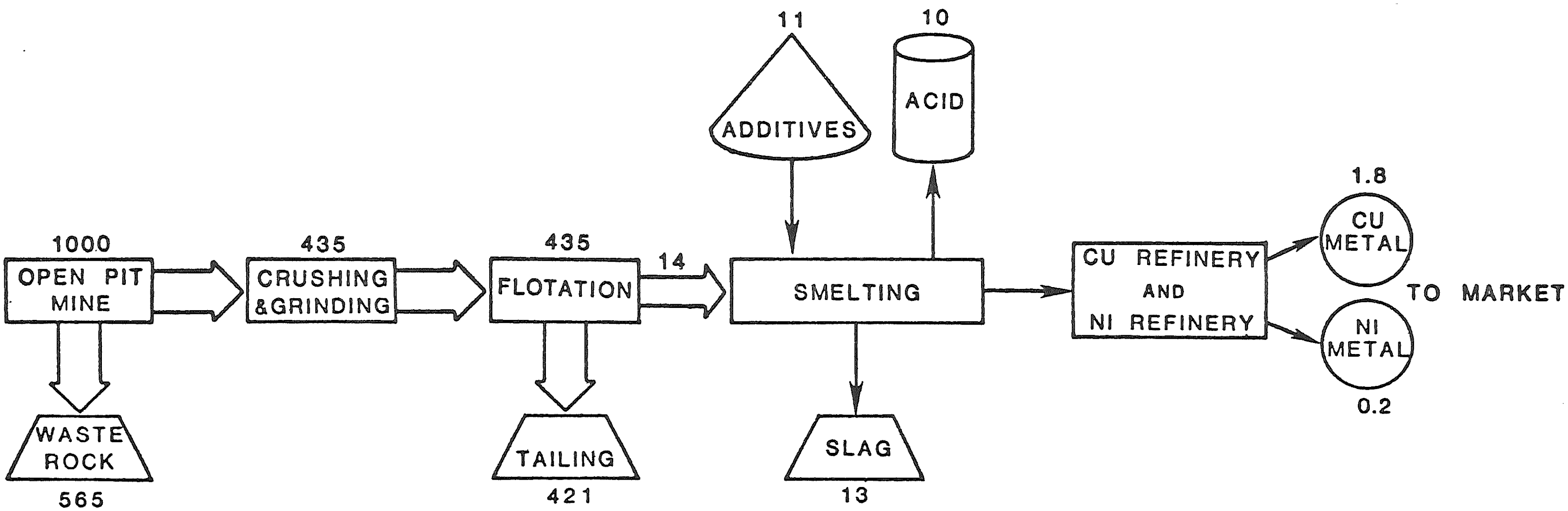
A major factor to be considered when assessing the impacts of copper-nickel operations is the extent to which all stages of the operations are located on a common site. While the mine must be where the resource is located and the processing plant must be near the mine because of the large volume of ore that must be transported, there is much more flexibility in siting the smelting and refining facilities. Isolating the smelter/refineries from the mine site will increase both capital and operating costs for the operation. The cost increases are primarily due to the inability to share certain facilities, equipment, and personnel with the mining and processing phases of the operation, therefore requiring duplications, as well as the increased cost of transporting the concentrate to the smelter. For example, if a smelter/refinery complex is located in Duluth instead of at the minesite, it is estimated that the total capital and operating costs would each increase approximately 15%. Other costs, such as taxes, utility services and pollution control could also increase or decrease depending on the location selected. In addition, if the smelter/refinery complex is located closer to sulfuric acid and copper/nickel metal primary consumers, then the transportation savings for shipping these products would partially offset the increased concentrate shipping costs. The flexibility of siting these facilities presents many opportunities for mitigating the environmental, social and economic impacts caused by these operations.

The ability to control pollutants emitted from copper-nickel operations and the approximate costs of such controls is another relationship addressed by the models presented in this chapter. For example, the control of sulfur dioxide

FIGURE 8

INTEGRATED MINNESOTA CU-NI OPERATION UTILIZING BULK FLOTATION PROCESS

(BASED ON 1000 UNITS MOVED FROM AN OPEN PIT MINE)



emissions from a smelter has a direct affect on the extent of environmental and health impacts associated with the operation of such a facility. This chapter presents data on the effectiveness of alternative smelter air pollution control systems and the cost of these systems (detailed descriptions of these systems can be found in Chapter 4 of this Volume). This information indicates that a factor of 6 reduction in SO_2 emissions is possible with the application of state-of-the-art emission controls and would result in a 3% increase in smelter capitol costs and a 14% increase in smelter operating costs.

The different copper-nickel development alternatives that may be proposed for northeastern Minnesota are great in number and could differ significantly from the models presented. The corresponding impact assessments provide valuable reference points for future evaluation of specific development proposals. In addition, alternative models not presented in this chapter can be created and evaluated by the reader with the information presented in this report.

5.2 DEVELOPMENT MODEL COMPONENTS

5.2.1 Variable Classes

In creating the hypothetical development models to describe potential Minnesota operations and their resulting impacts in the Study Area, it was necessary to generate values for a large number of variables from data sources presently available. These data sources varied from educated guesses by experienced engineers and scientists to detailed historical information or detailed estimates from consultants experienced in the particular field in question. Therefore, the accuracies involved ranged from as much as orders of magnitude to as little as $\pm 30\%$. This latter is considered to be the best one can expect from any estimate based on the type of raw data available and the conditions under which the estimates were made. Mining companies may spend several millon dollars on feasibility studies for a specific and highly explored mineral deposit, and expect only 20 to 30% accuracy.

In order to organize the information involved in the generation of the models, the variables were placed in classes and each class was ranked according to the accuracy involved in its contained variables. As a result, 5 classes of variables were developed as follows:

Class I - Illustrative Variables

Class II - Geological Variables

Class III - Operating Variables

Class IV - Emission Variables

Class V - Economic Variables

Each of these classes will be discussed in turn. Table 1 summarizes the specific variables assigned to each class.

Table 1

5.2.1.1 Class I--Illustrative Variables: These variables, including production capacity, total ore produced, life of operation, and technology selection, fix the basic scope and nature of each model by making specific selections from a wide range of possible values for each variable. The concept of accuracy does not apply to these variables in the usual sense, as they were selected generally to illustrate or represent reasonable potential developments in the resource area. No accuracy discussion is therefore necessary; however, it must be stressed that these variables are real in the sense that they reflect plausible potential developments in the Study Area. For example, a mine capacity ranging from 5.35 to 20.00 X 10^6 mtpy of ore is within the probable development range; technology choices of flotation, smelting and refining to recover the valuable metals also reflect reasonable and generally conservative state-of-the-art assumptions. No models are predicated on the success of radically new, untested technology, though the applicability of technology which is proven in closely related applications is assumed.

5.2.1.2 Class II--Geological Variables: These variables represent those parameters which must meet economic criteria set by the laws of supply and demand in the world metal markets to allow a mineral resource to qualify as a mineral reserve, with the valuable constituents recoverable with existing technology and at a profit. The resource data, such as that available from the MDNR drill core study (Volume 3-Chapter 2), cannot demonstrate that such reserves exist. A great deal of detailed data is required to achieve the degree of geological

Table 1. Summary list of development model variables by class.

Class I - Illustrative Variables

a. Production capacity
b. Life cycle
c. Total ore produced
d. Equivalent years at full production
e. Construction personnel
f. Technology selection

Class II - Geological Variables

a. Ore grade
b. Material estimates
c. Mine recovery data

Class III - Operating Variables

a. Process recoveries-product analyses
b. Operating personnel
c. Energy requirements
d. Water requirements and management practices
e. Operating surface area requirements

Class IV - Emission Variables

a. Atmospheric particulates
b. Sulfur dioxide
c. Water

Class V - Economic Variables

a. Capital costs
b. Operating costs

definition needed to accomplish this, and when the work is finally done the results may simply prove that in fact there is no economically recoverable deposit in the context of current costs and prices. However, for purposes of the modelling being done here, a great leap is made and sufficient reserves are assumed to exist, as an illustrative variable, so that the model operations can proceed. Values for the quality of resource needed will, to a large degree, be based on actual test data. Thus, the big assumption here, and this without a basis of firm data, is that sufficient quantities of resource exist in sufficiently continuous deposits to allow an actual operation to proceed. In the modeling it is assumed that sufficient minerals having a certain grade, continuity and depth do exist. The results of the modeling and the economic feasibility analysis in Volume 5-Chapter 14 determine the copper and nickel prices that are required for such a resource to then be called a reserve, and for the operation to proceed.

Variables such as grade, depth of resource, depth limits of open pit and underground mining activities, and metal recoveries were all obtained from the best available information on the Duluth Complex. Based on several hundred drill hole records made available by mining companies to the MDNR, as well as several bulk samples supplied to the University's Mineral Resources Research Center (MRRC), the resource was described. Basic engineering data from several mining companies and MRRC indicated the range of mining parameters, and representative values were selected for use in the models. Overall, geological variables should be accurate to within $\pm 30\%$; however, some parameters such as metal recovery in the mine could be within $\pm 10\%$.

5.2.1.3 Class III--Operating Variables: Parameters such as resource recovery percentages, operating personnel, energy requirements, water use and consumption, and operating area requirements fall into this variable category. Estimates are based on data varying from average values for the copper industry to detailed values from specific requirements in analagous operations. In some cases, an accuracy of ±10% is the limit, but most have ±30% as their accuracy.

5.2.1.4 <u>Class IV--Emission Variables:</u> When all of the phases of a potential copper-nickel development are considered, a range of solid, liquid, and gaseous emissions to the land, water, and atmosphere might occur and result in environmental impacts. Thus, it is necessary to model these emissions if the environmental impact analysis process is to proceed. The mining and processing phases in particular may result in the release of contaminated water to the surface and groundwater of the region. Modelling the quantity and quality of these discharges involves technical considerations beyond the scope covered in this section of the report. The need for detailed information on meteorology, hydrology, and water chemistry to complete these models has led to the inclusion of this information in the water resources impacts section of this report (Volume 3-Chapter 4). Similarly the fugitive dust emissions to the atmosphere are discussed in the section on air resources impacts (Volume 3-Chapter 3). The exception comes in the case of the smelter/refinery phase of the operation. Unlike mining and processing, this phase involves activities which occur in a relatively small area and which can be under a high degree of control by the metallurgical engineer. As a result, the air and water emissions from this phase are most appropriately presented here, as a part of the model of the smelter/refinery operation. Therefore, air and water emission data is listed under this variable class for the smelter/refinery operation. Basically, the data has been accumulated from state-of-the-art information which in many cases is of necessity an estimate, based on the professional judgement of those experienced in the field. Thus, accuracies will range from a factor of 2 to one or 2 orders of magnitude.

5.2.1.5 Class V--Economic Variables: Parameters of interest here are capital and operating costs and the timing of such expenditures. Each of the models were detailed to the extent possible for economic evaluation. The source of information here ranged from detailed economic estimates by consultants and contractors to average values gleaned from the literature, resulting in an estimated ±30% accuracy for these variables.

5.2.2 Mine Development Models by Phase

In this section, models will be presented separately for each phase of the overall operation. The phases of interest are explicitly the mine, the processing plant, and the smelter/refinery complex. Exploration activities are included as part of the mine phase. The smelter/refinery phase, which in reality might be subdivided with the copper and/or nickel refinery remotely located from the smelter, is treated as one unit for simplicity. Data on the models for each phase are presented in the organizational framework of the 5 classes of variables just discussed. As noted within each classification, data is presented on:

1) the exploration and mining phase
2) the processing phase
3) the smelting/refining phase

5.2.2.1 Class I--Illustrative Variables: As the classification suggests, these variables were selected to illustrate plausible potential developments. They include:

Table 2

Production capacity of the mining operations ranges from 5.35 to 20.00 X 10^6 mtpy ore. Of this, underground mining examples are 5.35 and 12.35 X 10^6 mtpy ore, and open pit mining examples are 11.33 and 20.00 X 10^6 mtpy ore, with one combination of the smallest open pit and underground totalling 16.68 X 10^6 mtpy ore.

Mine life cycle was taken as a total of 30 yr for each model, consisting of construction, start-up, full production, and shut-down periods.

Total ore produced over the life of the operation combines the production during start-up, full production, and shut-down, and ranges from 123.1 to 500.0 X 10^6 mt.

Equivalent years at full production is a measure of the total production over the life of the operation. This is the total production divided by the rated production capacity. The range is 23 to 25 yr.

Construction personnel peak value was estimated and then varied over the construction period to obtain construction personnel requirements and payroll costs. The range on peak construction personnel is 180 to 318. See section 5.3.2 of this chapter on the year-by-year requirements for the 20.00 X 10^6 mtpy example, for a discussion of the variation in construction personnel during the construction period.

Technology selection was based on the geological location and orientation of the Duluth Complex resources, which indicates that both open pit and

Table 2. Illustrative variables-exploration and mining phase.

VARIABLE	MODELS				
Production Capacity, 10^6 mtpy ore	5.35	11.33	12.35	16.68	20.00
Total life, yr	30	30	30	30	30
Life cycle, yr					
Pre-production construction period	4.0	3.0	4.0	3.0	3.0
Production	26.0	27.0	26.0	27.0	27.0
Start up period	4.0	2.0	4.0	5.0	2.0
Full production period	20.0	23.0	20.0	20.0	23.0
Shut down period	2.0	2.0	2.0	2.0	2.0
Equivalent years at full production	23.0	25.0	23.0	24.4	25.0
Total ore produced, 10^6 mt	123.1	283.3	284.1	406.3	500.0
Construction personnel, peak	180	180	280	360	318
Technology selection	underground sublevel stope	open pit truck haul	underground sublevel stope	combined	open pit truck haul

underground technologies are applicable. With this decision made, the requirements for both of these methods were determined from listings in various mining handbooks. Open pit mining is typically performed using large blast-hole drills, electric shovels, diesel-electric haul trucks, and a fairly conventional mine design. For underground mining, standard sublevel open stoping methods were selected based on available information concerning the configuration and physical properties of the Duluth Complex. Ore transportation is by LHD units and train. Underground primary crushing reduces the ore to an acceptable size for conveyor haulage to shaft hoisting facilities.

Processing, Table 3

Table 3

Production capacity of the processing plants is the amount of concentrate produced annually and is directly proportional to the ore grade. It is given as 5.14% of the weight of underground ore, 3.81% of the combination ore weight, and 3.18% of the weight of open pit ore fed to the mill. The range then is 275.3×10^3 to 635.3×10^3 mtpy for the plant sizes given. Over the total life of the operations, this range varies from 6.33×10^6 to 15.88×10^6 mt of concentrate.

Plant life cycle totals 30 yr as does the mine life, with full production ranging from 24.5 to 26.5 yr and equivalent years at full production ranging from 23 to 25.

Construction personnel peak was estimated and then used as a reference to model requirements over the total construction period. The peak range is 750 to 1,250 workers.

Table 3. Illustrative variables-processing.

VARIABLE	MODELS				
Plant capacity 10^6 mtpy ore	5.35	11.33	12.35	16.68	20.00
Production Capacity					
Concentrate, 10^3 mtpy	275.3	360.0	635.3	635.3	635.3
Total Concentrate, 10^6 mt	6.33	9.00	14.61	15.33	15.88
Life cycle, yr					
Pre-production lag time	3.0	1.0	3.0	1.5	1.0
Pre-production construction period	2.5	2.5	2.5	2.5	2.5
Production period	24.5	26.5	24.5	26.0	26.5
Start up period	1.0	1.0	1.0	1.0	1.0
Full production period	22.5	24.5	22.5	24.0	24.5
Shut down period	1.0	1.0	1.0	1.0	1.0
Equivalent years at full production	23.0	25.0	23.0	24.4	25.0
Construction personnel peak	750	960	990	1150	1250
Technology selection	conventional crushing and grinding, bulk flotation, conventional tailing disposal basin, maximum water recycle.				

Technology selection was based on a variety of operating and test information, and incorporates 3-stage crushing, conventional rod mill-ball mill grinding, bulk copper-nickel sulfide flotation, tailing disposal in a basin, and maximum water recycle.

Smelting/Refining, Table 4

Table 4

Production capacity is based on a set level of 100,000 mtpy of copper and nickel metal and the metal distribution and estimated recovery levels within the system. The smelter-refinery data applies only to the mining operations large enough to support such a facility. The 12.35, 16.88 and 20.00 X 10^6 mtpy ore operations were specifically sized to meet the feed requirements of this smelter model. Metal production is 84,584 mtpy copper and 15,416 mtpy nickel, based on a concentrate feed of 635,259 mtpy.

Plant life cycle totals 30 yr as for the other facilities, with full production ranging from 22.5 to 24.5 yr and equivalent years at full production ranging from 23 to 25 yr depending on the mining operation.

Construction personnel peak was estimated and then scaled over a total construction period identical to the processing plant facility. The level is the same, 1,250, for each smelter/refinery complex.

Technology selection was based on best estimates of existing technology available to process a bulk copper-nickel concentrate to produce copper and nickel metal products. This technology includes flash smelting of the bulk concentrate followed by converting and electrorefining of the final

Table 4. Illustrative variables-smelting/refining.

VARIABLE	MODEL		
Mine ore capacity, 10^6 mtpy	12.35	16.68	20.00
Mill concentrate capacity, 10^3 mtpy	635.3	635.3	635.3
Production Capacity			
Metal, 10^3 mtpy	100,000	100,000	100,000
Cu metal, 10^3 mtpy	84.584	84.584	84.584
Ni metal, 10^3 mtpy	15.416	15.416	15.416
Total Production			
Metal, 10^6 mt	2.300	2.440	2.500
Cu metal, 10^6 mt	1.945	2.064	2.115
Ni metal, 10^6 mt	0.355	0.376	0.385
Life Cycle, yr			
Pre-production lag time	3.0	1.5	1.0
Pre-production construction period	2.5	2.5	2.5
Production period	24.5	26.0	26.5
Start up period	1.0	1.0	1.0
Full production period	22.5	24.0	24.5
Shut down period	1.0	1.0	1.0
Equivalent years at full production	23	24.4	25
Construction personnel peak	1250	1250	1250
Technology selection	flash furnace, slag cleaning, electrorefining, leach-electro-winning, acid plant		

copper product. The copper smelter slag will contain the nickel, and the slag will be cleaned in an electric furnace to produce the final discard slag and a copper-nickel matte product. This matte is converted and refined by leaching and electrowinning techniques to produce copper, nickel, and cobalt products. The smelter includes a sulfuric acid plant to control the release of sulfur dioxide gas to the environment.

5.2.2.2 Class II--Geological Variables, Table 5: As discussed earlier, these variables apply to the exploration and mining phases of the overall operation as they dictate how and whether or not exploration proceeds, and which mining techniques can profitably be applied to remove the ore. Obviously, these variables also affect the subsequent stages of processing, smelting and refining, but only indirectly as a result of their effects on exploration and mining techniques. Therefore, the geologic variables are considered as directly affecting only the first phase.

Table 5

Ore grade, as noted earlier, was based on data from bulk samples submitted for testwork, and on drill hole data. The average grades of 0.800% Cu for underground ore and 0.494% Cu for open pit ore were selected as best meeting both the operating requirements of the processing and smelting/refining stages, and the available geological data. A copper to nickel ratio of 4.3:1 was assumed to remain constant over the range of ore grades chosen, and average values were also used for sulfur, iron, and cobalt.

Material estimates-Based on the mine life, production rate, and both surficial and bedrock geological information on the mineralized zone (physical), mining

Table 5. Geological variables-exploration and mining phase.

VARIABLE	MODELS				
Production Capacity 10^6 mtpy	5.35	11.33	12.35	16.68	20.00
Ore grade, % Cu	0.800	0.494	0.800	0.592	0.494
% Ni	0.185	0.114	0.185	0.137	0.114
% S	1.658	1.095	1.658	1.276	1.095
% Fe (total)[a]	10.394	9.935	10.394	10.082	9.935
% Co	0.017	0.011	0.017	0.013	0.011
Total material over life of operation					
Overburden, 10^6 cu yd[b]	N.A.[e]	19.0	N.A.	19.0	25.3
Waste rock-lean ore, 10^6 mt[c]	12.2	368.2	28.2	380.4	650.0
Stripping ratio	N.A.	1.3	N.A.	1.3	1.3
Mine recovery data					
Ore left in place, %[d]	23	0	23	7	0
Extraction of ore, %	77	100	77	93	100
Dilution of ore, %	10	5	10	7	5

[a]Total Fe, oxide and sulfide.
[b]In place.
[c]Assumed 50% waste rock and 50% lean ore.
[d]Consists of 10% of ore not available for extraction plus 15% loss of available ore during the extraction stage for underground mining operations.
[e]N.A. = not applicable

plans were developed and the resulting volumes of overburden, waste rock-lean ore, and ore were determined. Included in the calculations was an assumed stripping ratio of 1.3 mt of waste rock-lean ore for every metric ton of ore removed. For purposes of discussion, the non-ore rock removed from the mine models was assumed to split evenly between waste rock and lean ore. These materials would be stored in separate piles.

The values determined by this method are highly subject to variation, as essentially homogenous conditions had to be assumed throughout the mining area in order to calculate the numbers. Obviously, the area is not homogenous and values of overburden, waste rock-lean ore, and open pit surface areas would change greatly with a change in the dip of the contact, the depth to ore, overburden thickness, etc. Overall, the geological variables can be assumed to be within ±30% of the true regional average value; however, one must realize that local conditions could effect a change, by a factor of 2 or 3, in any of the parameters shown.

Mine recovery data is based on average industrial values for the underground sub-level open stoping and open pit mining methods. The selected underground method generally results in about 10% of the true ore material being left in place as it is not accessible, and 85% recovery of the remaining 90% for a total average of 77% extraction of the ore material. An additional 10% extraction termed "dilution" is obtained and is composed of sub-ore grade material which must be removed to expose and remove the ore; otherwise it would be left in place. Open pit mining, on the other hand, results in total extraction of the ore plus about 5% dilution by sub-ore grade material.

Accuracies of these parameters can be considered as ±10%; however, local conditions and mine management practices could seriously change the values, particularly in the underground mining situation.

5.2.2.3 Class III--Operating Variables: Values for these variables are based on the results of intensive investigations of the operating parameters for all phases of a hypothetical copper-nickel mining operation. Each phase was considered separately in the detail allowable from available literature, contractor information, and information supplied by private corporations involved in actual mining activities.

Caution must be exercised in taking the numbers at face value as many are listed to several decimal places which was necessary for calculational purposes. For example, as discussed previously, the model concentrate analysis (also given in Table 7) is listed as 13.825% Cu and 2.647% Ni. This does not imply a claim of accuracy to the third decimal place in this figure. However, it does mean that this value represents a parameter which is part of a material balance that should, in principle, be closed. In terms of the model variable itself, a more reasonable value would be 13.8 ($\pm$4)% Cu. The resulting range, from 9.8% to 17.8% would generally cover the range of values obtained in metallurgical testwork. Concentrate with values in excess of 20% Cu have been obtained in some instances, but not with representative ore material, nor with acceptable metal recoveries.

Exploration and Mining, Table 6

Table 6

Operating personnel is estimated only for the mining operation during the full production period. Exploration personnel levels are very erratic and short-lived and do not enter into the mine operating cost picture since

Table 6. Operating variables-exploration and mining at full production

VARIABLE		MODELS				
Mine Capacity, 10^6 mtpy		5.35	11.33	12.35	16.68	20.00
Operating personnel		674	546	1555	1220	964
Energy requirements						
Electricity,	10^6 kwh/yr	57.7	18.3	133.0	76.0	32.2
	10^9 BTU/yr	605.9	189.0	1396.5	798.0	338.1
Diesel fuel,	10^6 gal/yr	0.8	6.3	2.0	7.1	11.1
	10^9 BTU/yr	111.0	873.8	277.4	984.7	1539.5
Propane,	10^6 gal/yr	1.2	0	2.8	1.2	0
	10^9 BTU/yr	110.4	0	257.6	110.4	0
Gasoline,	10^6 gal/yr	0	0.2	0	0.2	0.4
	10^9 BTU/yr	0	25.0	0	25.0	50.0
Total,	10^9 BTU/yr	827.3	1087.8	1931.5	1918.1	1927.6
Water requirements						
Discharged only, 10^9 gal/yr[a]		0.02	0.19	0.02	0.21	0.23
Surface area, acres						
Waste rock-lean ore piles		51	1119	96	1170	1988
Overburden piles		N.A.[b]	137	N.A.	137	173
Miscellaneous mine		40	40	40	80	40
Open pit mine		N.A.	393	N.A.	393	523
Underground mine		316	N.A.	710	316	N.A.
Total, direct		407	1689	846	2096	2724
Undisturbed area (40% of direct total)		163	676	338	838	1090

[a]24 hr/day, 365 day/yr
[b]N.A. = not applicable

exploration is a capitalized cost item in terms of the mine. Mine operating personnel figures are based on a detailed analysis of equipment requirements to remove the required amounts of materials and the manpower necessary to operate this equipment. Accuracy levels of these estimates are considered to be ±30% with the techniques and technology used in the estimates. Any major departure from the stated conditions could effect a manpower change of ±50%, but such changes do not appear likely.

Major factors such as mine size, production rate, mine type, mining method and degree of automation must all be considered when determining manpower requirements. More subtle factors such as experience and age of miners, local miners, union regulations, working conditions, condition of the equipment and worker productivity also affect the manpower requirements.

For modelling purposes, underground manpower requirements range from 674 to 1,555 (±30%), and open pit operations range from 546 to 964 (±30%). On the basis of ore production, this is 3.8 mt/man hour for underground mining and 10.0 mt/man hour for open pit mining, both within the nationwide ranges for these types of mining operations.

Energy requirements-As shown in the table, energy requirements are separated by type: electricity, diesel, propane, and gasoline. In the model open pit operations, 18% of the total energy consumption is electrical, 80% is diesel, and 2% is gasoline. Underground the energy split is 73% electrical 14% diesel, and 13% propane, the difference of course is due to different types and amounts of mining equipment. In the open pit operations haulage is by diesel-electric trucks, where the underground haulage is principally by electric train and hoist. Total energy needs average about 148 X 10^9

BTU/mt ore for underground operations and 96 X 10^9 BTU/mt ore for open pit operations, with expected accuracy levels of $\pm 30\%$. Electrical needs in the Regional Study Area will likely be met by the combustion of low sulfur coal at large central power stations located in Minnesota and North Dakota and owned by private utilities. It is unlikely that the power stations would be located in the Study Area due to water limitations.

Water requirements-The only water of concern in these operations is the unwanted inflow from any groundwater encountered in driving drifts and developing stopes in underground mining and from the glacial overburden drainage and direct precipitation in open pit mining. Available data indicates that such sources will be small, ranging from 30 to 40 gpm for the model underground operations and 360 to 450 gpm for the open pit operations. In underground mining operations, unknown, localized extensive fracturing could result in excessive water production, but there is no way of predicting such occurences until actual operations exist. Volume 3-Chapter 4 contains a discussion of the implication of excessive mine water production and resulting water quality impacts. No water of consequence will be consumed in mining operations and the total inflow, less evaporation, will be discharged. Because of the uncertainty of groundwater conditions and the variability due to location in the Study Area, actual water inflow could vary by orders of magnitude. Minor amounts of water will be used for dust suppression in the open pit mines, and for cooling during drilling in the underground mines.

Operating surface areas required for mining operations and disposal of waste materials are calculated based on regular shapes and assuming given volumes of materials are to be removed and disposed of. Model waste rock-lean ore piles store up to 50 X 10^6 mt of material each, and all overburden is contained in one pile for each operation. Accuracy levels for the acreage values given are $\pm 30\%$, but changes in the ratio of waste rock-lean ore to ore, or of overburden to ore could change the values shown considerably, as could changes in the heights of storage piles. The estimated ratio of waste rock-lean ore to ore is 1.3:1 for open pit mining and 1:10 for underground mining. An area is also included under the heading of "undisturbed area" to represent space not actually used for a specific function but which is in fact lost to other uses since it lies in and around the actual mining areas. Access to this land would necessarily be restricted by the mine operator, in many cases for public safety purposes. This area is modeled by assuming it is 40% of the sum of the direct land requirements.

Processing, Table 7

Table 7

Process recoveries-product analyses-The product of the processing phase is the concentrate. Concentrate grade and recovery values are based on bench scale and pilot plant tests as mentioned earlier, and on projections by private industry, state and federal agencies. Based on this information, the modeled chemical analyses of products should be within $\pm 30\%$, and recovery values within $\pm 10\%$ for both copper and nickel. The behavior of cobalt is the least known of the 3 primary valuable metals, but the stated values should be within $\pm 30\%$.

As mentioned earlier, analysis values shown are carried out to several decimal places for mass balance purposes only, and do not indicate any

Table 7. Operating variables-processing at full production.

VARIABLE	MODELS				
Mine Capacity, 10^6 mtpy ore	5.35	11.33	12.35	16.68	20.00
Concentrate grade,	5.14	3.18	5.14	3.81	3.18
% Cu	13.825	13.825	13.825	13.825	13.825
% Ni	2.647	2.647	2.647	2.647	2.647
% Co	0.132	0.132	0.132	0.132	0.132
% S	25.87	25.87	25.87	25.87	25.87
% Fe (total)	32.53	32.53	32.53	32.53	32.53
ppm precious metals[a]	47	47	47	47	47
Concentrate % weight (of input ore)	5.14	3.18	5.14	3.81	3.18
Concentrate recoveries[b]					
% Cu	88.89	88.89	88.89	88.89	88.89
% Ni	73.75	73.75	73.75	73.75	73.75
% Co	39.88	38.18	39.88	38.69	38.18
% S	80.20	75.13	80.20	77.25	75.13
% Fe (total)	16.09	10.41	16.09	12.29	10.41
Tailing, % weight (of input ore)	94.86	96.82	94.86	96.19	96.82
Tailing grade					
% Cu	0.094	0.056	0.094	0.068	0.056
% Ni	0.052	0.031	0.052	0.038	0.031
% Co	0.011	0.007	0.011	0.008	0.007
% S	0.346	0.281	0.346	0.302	.0281
% Fe (total)	9.195	9.193	9.195	9.193	9.193
Operating personnel	183	308	302	379	414

Table 7 continued.

	MODELS				
	5.35	11.33	12.35	16.68	20.00
Energy requirements					
Electricity, 10^6 KWH/yr	121.6	262.3	280.7	383.0	463.0
, 10^9 BTU/yr	1276.8	2754.2	2947.4	4021.5	4861.5
Thermal , 10^9 BTU/yr	31.4	70.3	72.7	100.0	124.1
Total , 10^9 BTU/yr	1308.2	2824.5	3020.1	4121.5	4985.6
Water requirements[c]					
Total requirement[d], 10^9 gal/yr (total system flow)	3.48	7.37	8.03	10.84	13.00
Recycle, 10^9 gal/yr	3.35	7.10	7.74	10.45	12.53
Make-up (loss)[e], 10^9 gal/yr	0.13	0.27	0.29	0.39	0.47
Operating area requirements					
Plant, acres	120	240	260	340	400
Tailing basin, acres	1067	2348	2309	3279	4016
Total, acres	1187	2588	2569	3619	4416
Undisturbed area, acres (40% of total)	475	1035	1028	1448	1766

[a]Precious metals such as Au, Ag, Rh, Pt, Pd, see Volume 3-Chapters 1 and 2 for more details.

[b]Percent of metal recovery from the ore fed to the plant.

[c]Water balance includes tailing pond contribution and losses as discussed in the previous chapter and in Volume 3-Chapter 4 (see Figure 9).

[d]Based on a water requirement of 650 gal/mt ore.

[e]Includes water in ore fed to the plant.

claim of accuracy. For discussion purposes, the concentrate analysis should be:

13.8 ($\pm$4.0)% Cu

2.6 ($\pm$0.8)% Ni

0.13 ($\pm$0.04)% Co

Tailing-Metal content and weight distribution assigned to the tailing product is dictated by a mass balance as simply the difference between ore and concentrate, and therefore the accuracies are the same as stated for these, $\pm$30%.

Operating personnel-This value is for the full production period and is based mainly on private industry estimates for similar plant operations. It is assumed to be accurate to within $\pm$30%. Technology changes could of course greatly affect the operating personnel requirements and the currently estimated range of 183 to 414 for plants ranging from 5.35 to 20.00 X 10^6 mtpy ore could then change by more than $\pm$30%. However, these values compare favorably with other processing operations and are considered accurate.

Energy requirements-Total energy requirements are broken down by fuel type; however, 98% is electrical, with the remainder provided by any suitable thermal source such as propane, fuel oil, or smelter waste heat. The electrical requirements are based on equipment needs and are considered to be accurate within $\pm$30%. Individual company preferences in flowsheet design could affect the power needs by this amount. Again, electrical generation would probably be central station coal-fired units operated by utility companies.

Water requirements-Water is needed to provide an ore-water, or pulp mixture at 30% solids which is suitable for transport of the ore slurry and flotation of the concentrate, as well as transport of the tailing to the disposal basin. The total amount of water needed in the processing system is 650 gal/mt ore, which amounts to 3.5 to 13.0 X 10^9 gal/yr for the 5.35 to 20.00 X 10^6 mtpy operations, respectively. This is the total amount of water flowing through the processing system over the time period of one year, with an estimated accuracy of ±10%.

Figure 9 illustrates the largest open pit operation water system where the processing plant is closed with the tailing basin only. Here, the basin is assumed to be 20% covered with water indicating evaporative losses of about 9% of the total requirement. A more realistic basin coverage would be 80% to provide water storage and to mitigate dust lift-off, which would result in evaporation losses some 50% greater (see Volume 3-Chapter 4, section 4.4 for a more detailed discussion of the water budget).

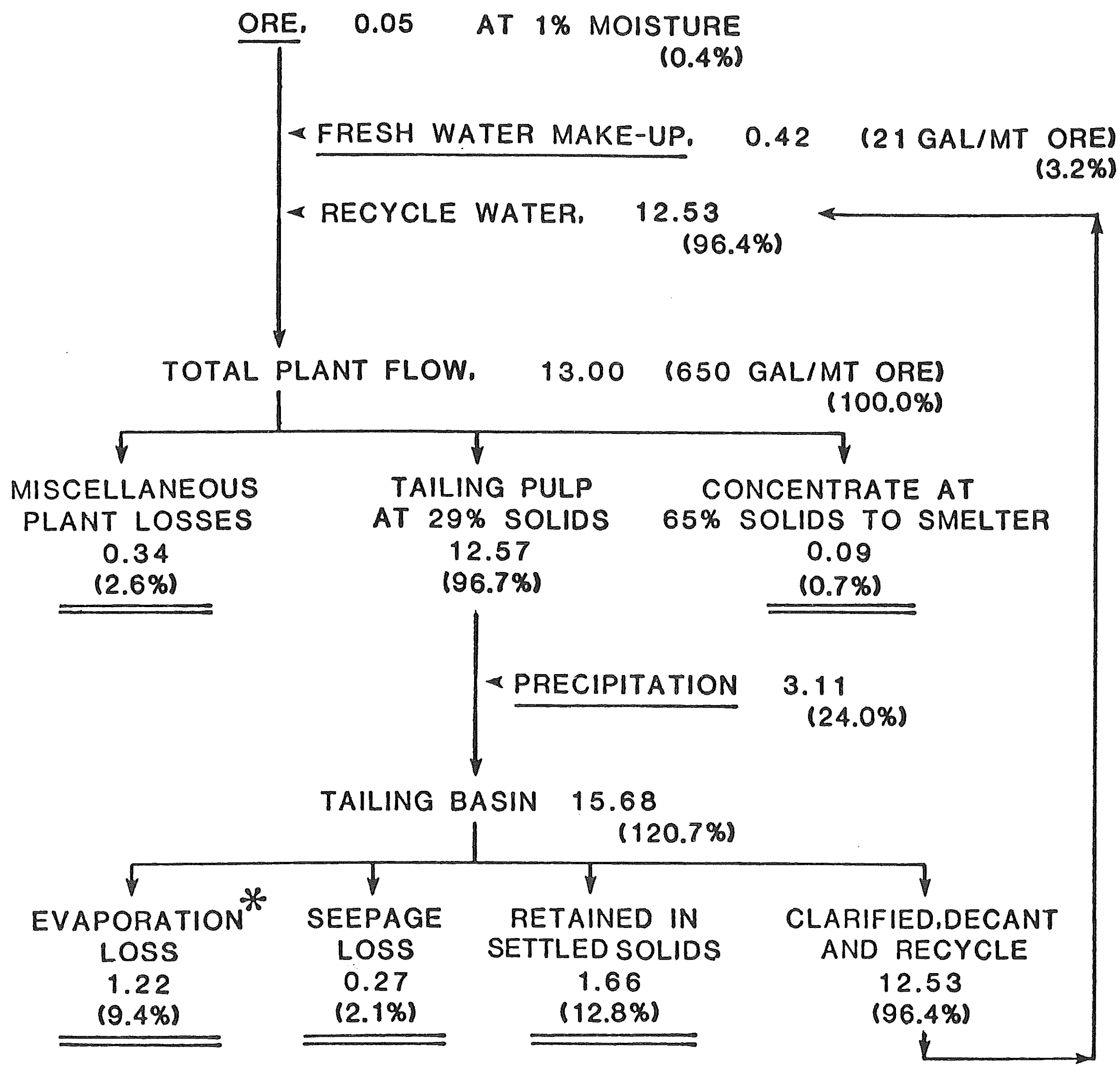
Figure 9

Operating area requirements-Plant area requirements are general estimates only and could vary by ±50%. The tailing basin areas are based on assuming that the total tonnage of material produced is stored in a regular, circular basin of 70 ft average thickness at a compacted solids density of 90 lb/ft^3. Depending on local topographic conditions and the actual life of the operation modeled, the range of 1,067 to 4,016 acres could easily vary ±30%. An undisturbed area is included as before, at 40% of the above area requirements.

FIGURE 9 TYPICAL PROCESSING WATER BALANCE MODEL-20X10^6 MTPY ORE OPERATION

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

(FLOWS SHOWN IN 10^9 GPY WATER)



*EVAPORATION SHOWN HERE IS BASED ON A MINIMAL 20% WATER COVERAGE FOR THE BASIN. A MORE REALISTIC COVERAGE OF 80% WOULD INCREASE THIS LOSS BY ABOUT 50% WITH A CORRESPONDING INCREASE REQUIRED IN THE FRESH WATER MAKE-UP.

Smelting and Refining, Table 8

Table 8

Process recoveries-product analyses-Product or metal recovery data depends on the chemical content of the concentrate being treated and on the equipment selected for the various integrated operations within the smelter/refinery complex. Recovery of copper and nickel from the concentrate are assumed accurate within ±10% based on consultant and literature information. Cobalt recovery is less well-known and assumed to be ±20%. Precious metal recovery is similarly uncertain, but is assumed to be 100% for purposes of discussion. Overall amounts involved appear to be small and are expected to go with the copper to the refinery where excellent recovery is possible.

Operating personnel-This was determined from industry values, and estimated at 621 total personnel for the smelter and both refineries at full production. Equipment selection and the degree of automation would seriously influence this variable. An accuracy of ±30% is, therefore, used for the personnel value.

Energy requirements-As shown in the table, energy needs are listed by fuel type for the total smelting/refining complex. Energy needs are highly dependent on the equipment selection and the degree of emission control exercised in the operation. For the modeled smelter/refinery, the energy source distribution is:

Table 8. Operating variables-smelting and refining at full production.

VARIABLE		MODELS[a]		
Mine Capacity, 10^6 mtpy		12.35	16.68	20.00
Metal Recoveries, % Cu			96.30	
, % Ni			91.67	
, % Co			50	
, % Precious metals			100.00	
Operating personnel			621	
Energy requirements				
Electricity,	10^6 kwh/yr		580	
	10^9 BTU/yr		6090	
Propane,	10^6 gal/yr		1	
	10^9 BTU/yr		92	
Coal,	10^6 mt/yr		40	
	10^9 BTU/yr		1060	
Natural gas,	10^6 ft^3/yr		222	
	10^9 BTU/yr		222	
Thermal,	10^9 BTU/yr		1631	
Total,	10^9 BTU/yr		9095	
Operating surface area, acres				
Smelter			50	
Refineries			100	
Slag disposal			25	
Total			175	
Undisturbed area, acres			70	
Water requirements[b], 10^9 gal/yr				
Total[c]			22.87	
Make-up (losses)[d]			1.81	
Internal recycle			21.06	

[a]Data is identical for each model size as is the concentrate production and composition, and therefore the smelter/refinery performance is the same for each model.

[b]Maximum internal recycle assumed (see Figure 10).

[c]Inclues $0.1x10^9$ gpy water in concentrate not included in make-up water.

[d]Two-thirds of this requirement may be met by external recycling of discharge water suitably purified in a water treatment plant, along with water contained in the concentrate.

Source	Energy Distribution, % Smelter	Cu Refinery	Ni Refinery	Total
Electrical	44	6	17	67
Propane	0	0	1	1
Coal	12	0	1	12
Natural Gas	0	0	2	2
Thermal	0	4	14	18
Total	56	10	34	100

The total energy value of 9.1 X 10^{12} BTU/yr, or 91 X 10^6 BTU/mt metal produced is based on industrial information and USBM data and is distributed as described above. Changes in operating parameters could easily change the energy needs by $\pm$50%, but the stated values within $\pm$30% are reasonably accurate for the conditions modeled. The distribution is highly dependent on technology selection so that, for example, use of an electric smelting furnace rather than the flash furnace used in the model might virtually eliminate coal as a significant energy source. This must be born in mind in interpreting the modeled distribution.

Operating surface area-Estimates of area requirements of the facilities were based on industry information and can be considered accurate within $\pm$30%. Operation size and equipment selection will affect the area needs, as will the extent to which the metals are refined on site and the degree of emission control utilized.

Water requirements-Values listed in the table and shown in Figure 10 are for the Case 2 (Chapter 4) condition of maximum internal recycle of both process and non-contact cooling waters. Even assuming closed-cycle cooling, depending on the process selected and the regulations concerning discharge waters, the total flow requirements could vary by $\pm 50\%$ with make up requirements varying by an order of magnitude. If once through cooling is used (Case 1, Chapter 4), both the appropriation and the discharge values would increase by more than an order of magnitude, with evaporative losses decreasing considerably.

Figure 10

5.2.2.4 Class IV--Emission Variables, Table 9A-C: Emission variables listed in Table 9A-C are for particulate, water, and SO_2 emissions from the smelter/refinery complex. The levels shown are based on specific assumptions for:

Operation size

Chemistry of feed materials

Equipment and processes selected

Type and degree of emission control

Fuel types (air only)

Maintenance programs

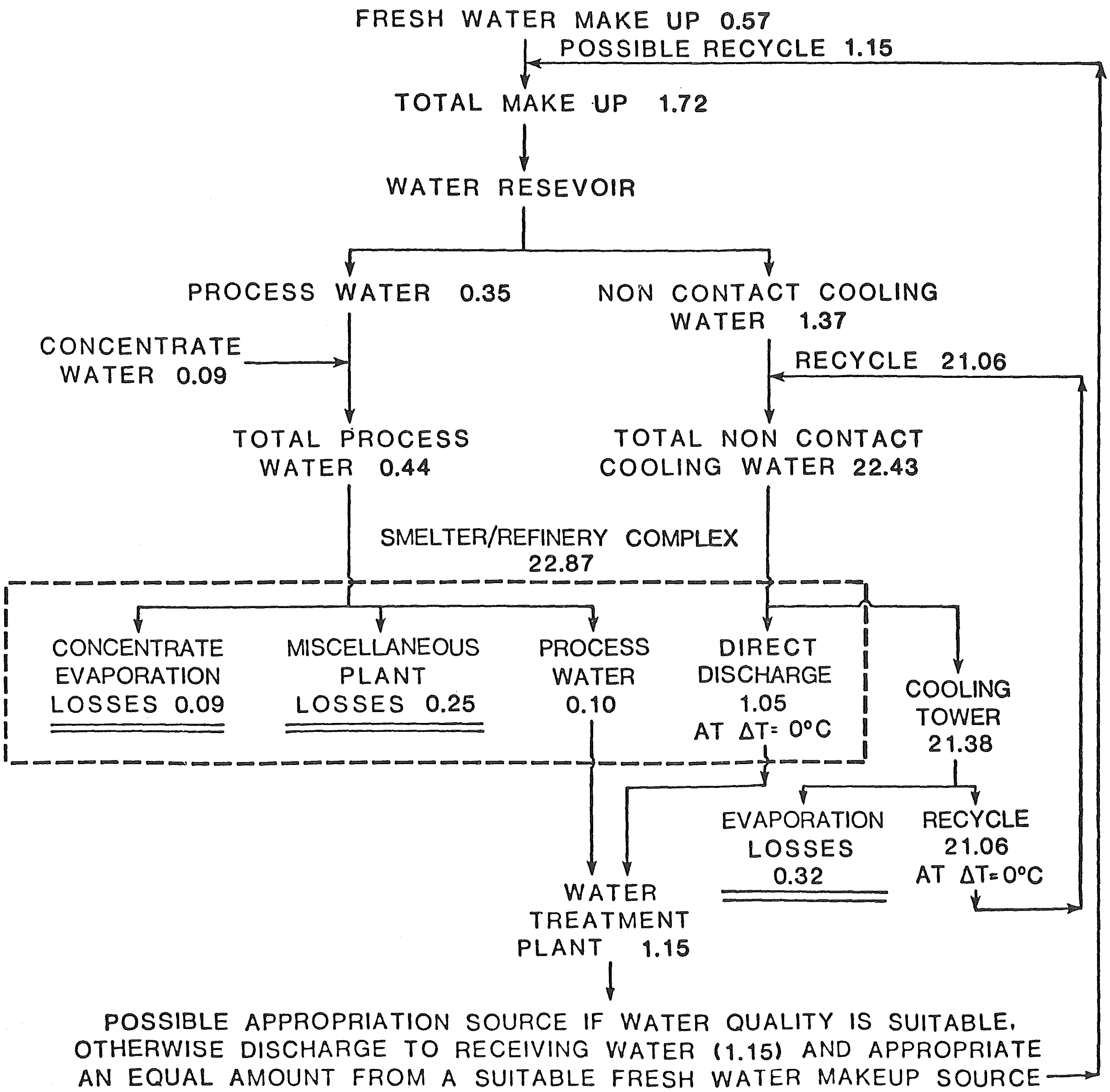
Tables 9A-C

Sources of emission information were primarily the literature and results produced by a metallurgical and pollution control consultant as discussed in Chapter 4 of this volume. In some cases, a profession estimate had to be made based on performance data from other industrial applications. Since the levels of most critical elemental constituents were very low in the concentrate material feeding the smelter/refinery complex (see Volume 3-Chapter 1), it was difficult to estimate their distribution in the final products of the operation. Thus, variations of one to 2 orders of magnitude would not be unreasonable for

FIGURE 10

WATER BALANCE FOR 100,000 MTPY MODEL SMELTER/REFINERY COMPLEX ASSUMING MAXIMUM INTERNAL RECYCLE, (FLOWS SHOWN IN 10^9 GPY WATER)

CASE 2*



* SEE CHAPTER 4

Table 9A. Emission variables-smelter/refinery complex producing 100,000 mtpy Cu and Ni metal.

Stack Particulate Emission Models[a]

CONSTITUENT	BASE MODEL, mtpy[c]	OPTION 1 AND OPTION 2 MODEL, mtpy[d]
Cu	263.5	39.53
Ni	50.45	7.57
S	496.63	74.50
As	17.44	0.01
Cd	1.84	0.01
Co	2.53	0.38
Be	0.02	0.00007
Pb	9.8	0.02
Hg	0.07	0.00006
Zn	23.82	0.33
Fe	581.76	87.26
Sb	0.003	0.0005
Cl	0.39	0.06
F	0.02	0.003
SiO_2	638.51	95.79
Al_2O_3	101.83	15.28
MgO	57.48	8.62
CaO	100.18	15.03
Other[b]	39	14
TOTAL	2,385	358

[a]The models assume the particulates will have the same composition as the smelter feed. Normal operating conditions are assumed.

[b]Includes oxides of Na, K, Ti, P, Mn, Cr, and Fe.

[c]Includes 97% particulate removal efficiency for ESP units and 99.9% particulate removal efficiency for acid plant.

[d]Includes all of [c] plus 85% particulate removal efficiency for scrubbing units. There is no distinction between options 1 and 2 in terms of particulate removal efficiency (Chapter 4).

Table 9B. Emission variables-smelter/refinery complex producing 100,000 mtpy Cu and Ni metal.

Water Emission Model[a]

PARAMETER	VALUE
Flow (gpm)	4,065
pH	2.7
TDS (mg/l)	79,700
$SO_4^=$ (mg/l)	12,800
As (mg/l)[b]	3.0
Cd (mg/l)	2.3
Co (mg/l)[c]	2.40
Cu (mg/l)	16.6
Fe (mg/l)	17.2
Hg (mg/l)[b]	0.017
Ni (mg/l)[c]	39.8
Pb (mg/l)[b]	5.2
Zn (mg/l)	450

[a]Process water effluent stream. Unless otherwise noted, model values are based on data from selected domestic operations (Chapter 4).

[b]Values adjusted downward to reflect 100% of the constituent present in the modeled smelter feed.

[c]Value based on worst case model of waste rock/lean ore leachate.

Table 9C. Emission variables-smelter/refinery complex producing 100,000 mtpy Cu and Ni metal.

SO_2 Gas Emission Model

EMISSION POINT	MODEL VARIATION, mtpy sulfur			
	Basic[a]	Base Case[b]	Option 1[c]	Option 2[d]
Fugitive	4,960	495	495	495
Stack	1,177	5,642	2,256	501
Total	6,137	6,137	2,751	996
Cost Effect[e]				
Total Capital	---	+10%	+12%	+13%
Operating	---	+26%	+40%	+44%

[a]Acid plant control of strong SO_2 gas to 650 ppm SO_2 only.

[b]Same as a plus redirection of weak SO_2 gas with secondary hooding.

[c]Same as b plus scrubbing of collected weak SO_2 gas to 650 ppm SO_2.

[d]Same as c with acid plant control of strong SO_2 gas to 300 ppm SO_2, plus scrubbing of acid plant tail gas and collected weak SO_2 gas to 143 ppm SO_2.

[e]Cost effect is for the smelter only. Basic case smelter contains only an acid plant and the associated total capital and annual operating costs (smelter only) are $\$193.2 \times 10^6$ and $\$15.12 \times 10^6$, respectively (see Chapter 4).

elemental loadings, depending on the technology used to produce the final metal products. Variations in operating practice, material recycling and maintenance will also greatly affect element distribution estimates.

Emission models listed in Tables 9A-C assume a flash smelter with a double contact acid plant treating the strong SO_2 gas, and various treatments of weak SO_2 and acid plant tail gases to further reduce the emission levels. Chapter 4 deals in detail with these models and treatments and a more summary discussion is given later in this section.

Particulate Removal and SO_2 Conversion Efficiencies--In addition to the controlling factors listed above for the emission variables, particulate and SO_2 control also depend on:

Volume of gas flows

Temperature of gas flows

Particle size (particulates only)

Particulate loading

Chemistry of gas flows (SO_2 only)

The major sources of information for these variables included a smelter consultant and data from the literature. Values are expected to be within ±30% in all cases.

5.2.2.5 Class V--Economic Variables, Table 10: Economic data includes both capital and operation costs (1977 dollars) for each facility and for all phases including exploration/mining, processing, and smelting/refining. In the 5.35 and 11.33 X 10^6 mtpy ore models there are no corresponding smelter cost data included, as neither of these operations are large enough to support the modeled smelter. For additional economic analysis, custom or toll smelting/refining is assumed. However, the 2 in combination result in the 16.68 X 10^6 mtpy operation which does support a smelter/refinery facility capable of producing 100,000 mtpy of copper and nickel metal.

Table 10

Exploration and mining cost data is the combined total of all facilities listed for those models. Data was accumulated from all sources listed for the variables detailed in Class I-IV discussions and are estimated to be within $\pm 30\%$. Mining costs include primary crushing in the underground operations, but this facility is charged as a processing function in the open pit operations.

Similarly, processing cost data includes both capital and operating costs for all facilities described in the previous variable discussions. The processing phase models use a single processing facility to produce a bulk concentrate suitable as new feed to the smelting/refining operation in the 12.35, 16.68, and 20.00 X 10^6 mtpy operations, with disposal of the resulting tailing product. In the 5.35 and 11.33 X 10^6 mtpy operations, the modeled concentrate quality is identical, but the quantities would not support subsequent treatment by themselves--only in combination as the 16.68 X 10^6 mtpy operation.

Processing data was primarily based on extrapolations from detailed capital and operating cost data supplied by the mining industry and consulting firms. The accuracy, as claimed by the consultants, is $\pm 30\%$.

Smelting and refining cost data is not a compilation of the component parts of the system, as such data was not available. Sources such as the literature, consultants, and industry provided assistance in designing the 100,000 mtpy of

Table 10. Economic variables-all facilities (1977 dollars).

DEVELOPMENT PHASE	MINE CAPACITY AT FULL PRODUCTION, 10^6 mtpy ore 5.35[a]	11.33[a]	12.35	16.68	20.00
Exploration/Mining					
Capital cost, total $\$10^6$	116.5	120.9	177.5	237.4	205.9
\$/annual mt ore	21.78	10.67	14.37	14.23	10.30
Operating cost, $\$10^6$/yr	32.30	23.24	74.35	54.26	40.76
\$/mt ore	6.04	2.05	6.03	3.26	2.04
Processing					
Capital cost, total $\$10^6$	89.3	164.7	154.3	203.2	230.9
\$/annual mt ore	16.69	14.54	12.49	12.18	11.55
Operating cost, $\$10^6$/yr	13.97	27.55	29.29	38.88	45.44
\$/mt ore	2.61	2.43	2.37	2.34	2.27
Smelting/Refining					
Capital cost, total $\$10^6$	N.A.[b]	N.A.	324	324	324
\$/annual mt ore	N.A.	N.A.	26.23	19.42	16.20
Operating cost, $\$10^6$/yr	N.A.	N.A.	33.3	33.3	33.3
\$/mt ore	N.A.	N.A.	2.70	2.00	1.67
Total Facilities					
Capital cost, total $\$10^6$	205.8	285.6	655.8	764.6	760.8
\$/annual mt ore	38.47	25.21	53.10	45.84	38.04
Operating cost, $\$10^6$/yr	46.27	50.79	136.94	126.44	119.50
\$/mt ore	8.65	4.48	11.09	7.58	5.98

[a]Exploration, mining, and processing only, as concentrate production is insufficient to support modeled smelting and refining facilities.

[b]N.A. = not applicable

copper and nickel metal complex, and in costing out such an operation. Since the data was not as detailed or as well documented as for either the mining or processing sections, an accuracy of $\pm 50\%$ is felt to be a more reasonable estimate.

A special cautionary note must be added concerning the cost estimates for the smelting and refining operations, especially since these constitute such a large portion of the overall modeled costs (e.g. some 43% of estimated capital costs for the large open pit model). Taken as a whole, the smelter/refinery facility envisioned for this operation, as noted earlier, will likely be unique in the world, due to the unique ratio of copper to nickel, as well as the low grade, of the concentrate expected to result from the processing operation. Consequently, there is no reliable basis upon which to estimate costs, without the assumption of some risk that the technology upon which the estimates are based will not perform satisfactorily, requiring possibly costly modifications unforeseeable now. The estimates given here assume the applicability of conventional technology, and thus the results of any economic analyses based on them must be used with extreme caution. This is one topic on which reliable data is sorely lacking, and cost information provided in the future by the industry will be extremely useful.

5.3 INTEGRATED DEVELOPMENT MODELS

The previous section details the 3 phases or components of exploration/mining, processing and smelting/refining facilities. In order to assess the cummlative impacts of these development phases it is necessary to combine them into integrated models. The integrated development models each consist of complete and compatible component parts necessary to produce a refined, finished product, in this case copper and nickel metal, with all operations located within a reasonable distance of each other such that certain necessary facilities and staff can be shared by each phase of the operation. Examples of shared facilities and staff include administration and security personnel, warehouse facilities, general maintenance staff, parking areas, fire protection equipment, potable water supply, etc.

Since the establishment of such complete operations may be the goal of some companies interested in developing the resources in Minnesota, only by combining the individual component parts into an integrated model can a total assessment of potential impacts be made. This becomes obvious, for example, in the consideration of water quality impacts resulting from runoff discharged from lean ore storage piles. The potentially high levels of metals in this water might cause significant biological impacts if discharged. However, the water could be collected and diverted to a near-by processing plant as part of the mill make-up water, where the process chemicals and ore minerals may act to reduce metal levels in the water. Alternatively, the water might be effectively isolated from the environment as interstitial water retained with the tailing in a well-sealed basin. This single example illustrates but one of many possibilities in which a picture of the total, integrated development is needed in order to properly assess the potential for impacts.

If copper-nickel development takes place in Minnesota, any single operation would probably require both a mine and a processing plant, as economics alone would dictate against long distance haulage of ore. However, if the development of a mining district proceeds and the industry becomes better established, more isolated mines could be opened to supply ore to an existing processing plant. Under these conditions different economic considerations come into play and relatively isolated mines become more feasible. Concentrate, on the other hand, consists of only 3 to 5% of the original ore weight, is considerably enriched in metal values, and could be economically transported a greater distance than could the ore material from the mine. Thus, the smelter/refinery site might be some distance removed from the processing plant. In fact, it could be outside of the state or the nation, such as in Canada.

From a strictly technical point of view, and ignoring existing production capacity and market conditions, the most economical approach is the totally integrated facility producing finished, marketable products. This approach was developed and is described for mine capacities of 20.00, 16.68, and 12.35 X 10^6 mtpy ore, each resulting in 100,000 mtpy of finished metal products.

Additionally, information was developed for an isolated 11.33 X 10^6 mtpy open pit mine and for the isolated 100,000 mtpy copper and nickel metal smelter/refinery. Both of these approaches are designed to illustrate the additional needs of material transportation between isolated components and the corresponding inability to share certain common facilities.

5.3.1 Model Description

Each model phase or component has already been discussed in the variable descriptions of the previous section. In summary form, each model will be briefly described, and the value for each variable introduced previously is given as a total in Table 11 which summarizes the contributions from each phase to the totally integrated models.

Table 11

A comment concerning the water requirements is necessary, as the total values shown in Table 11 do not summarize the individual mine, processing, and smelter/ refinery components shown previously. The components were combined to provide overall water balances as shown in Figures 11, 12, and 13 for each of the model operations producing 100,000 mtpy metal. This is one example of how total systems can be designed and integrated to provide the necessary water flow at each process point.

Figures 11,12 & 13

The smelting/refining portion of the water flowsheet incorporates maximum treatment of both process and non-contact cooling waters, with total recycle either internally or through the tailing basin, to provide the most efficient water use in combination with the processing and mining component systems.

In decreasing order of ore capacity, the models are:

5.3.1.1 20.00 X 10^6 mtpy Open Pit Model--This is the largest model (based on ore production) used as an example of potential developments in Minnesota. With a total life of 30 yr and an effective life of 25 yr, 20.00 X 10^6 mtpy of ore are produced resulting in 635,259 mtpy of concentrate and 100,000 mtpy of copper and nickel metal.

Table 11. Data summary for common site mine, processing plant, smelter, and refineries.

	RATED ANNUAL CAPACITY, 10^6 mtpy ore		
	12.35	16.68 (1 proc. plant)	20.00
Capital cost, $\$10^6$ total	665.73	764.78	761.03
, $/annual mt ore	53.91	45.85	38.05
Operating cost, $\$10^6$/yr	137.05	126.41	119.47
, $/mt ore	11.10	7.58	5.97
Construction manpower, peak	2520	2760	2818
Operating manpower, at full production	2478	2220	1999
Energy requirement, 10^{12} BTU/yr	14.25	15.27	16.21
Electricity, KWH/mt ore	81.8	62.4	54.0
Fossil fuel, 10^3 BTU/mt ore	294.7	260.8	243.5
Water requirement			
Process, 10^9 gal/yr[a]	32.76	35.57	37.73
, 10^3 gal/mt ore	2.65	2.13	1.89
Make-up water, 10^9 gal/yr[b]	1.01	0.80	0.76
Potable, 10^9 gal/yr	1.3	1.8	2.1
, gpm	110	150	180
Total life of operation, yr	30	30	30
Actual production life, yr	26	27	27
Effective full production life, yr	23	24.4	25
Area requirement, acres[c]	5026	8246	10241

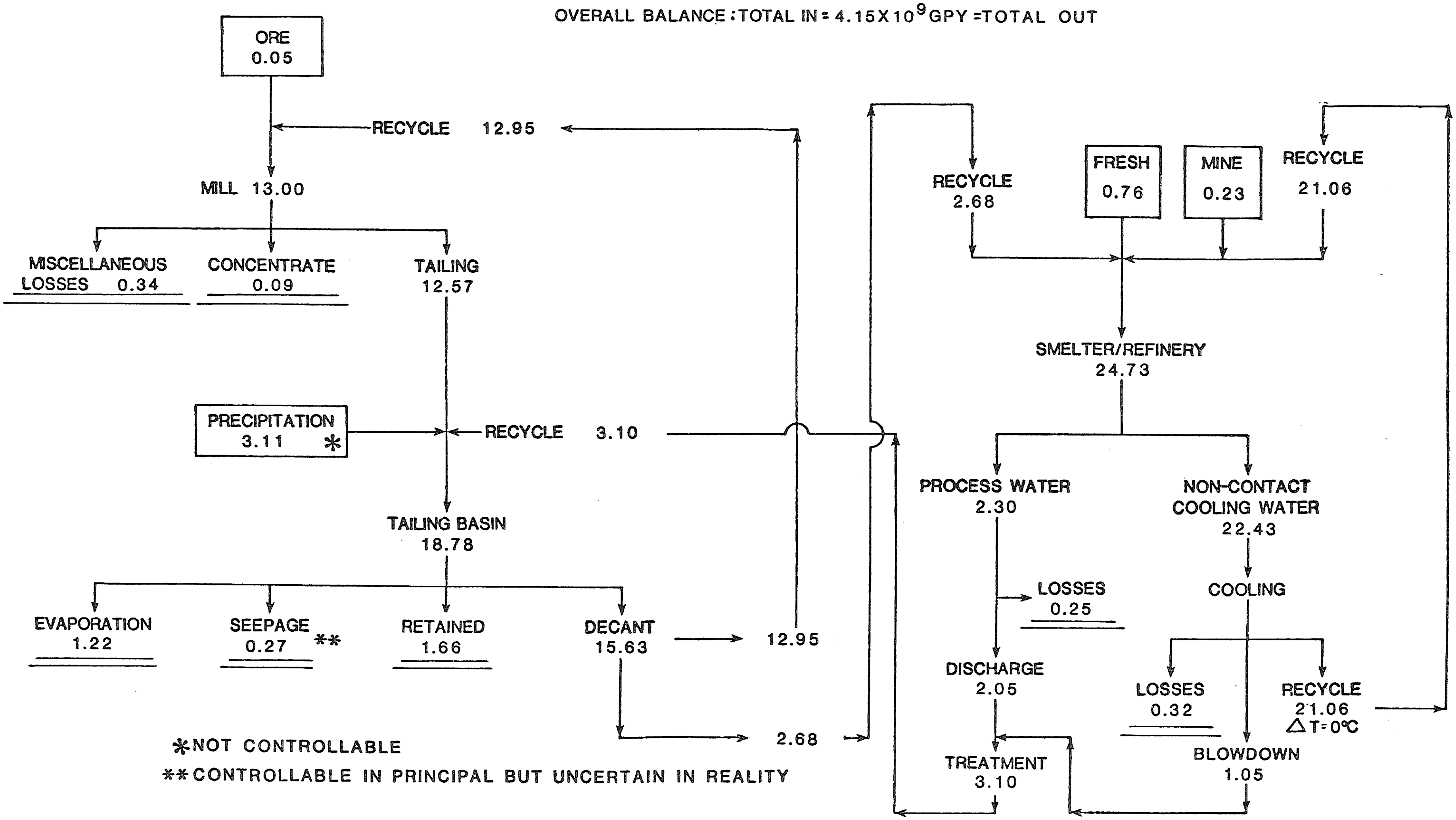
[a]Process includes total water needs for processing, smelting, and refining.

[b]Excludes water in ore, mine discharge water, and precipitation on tailing basin as detailed in Tables 6, 7, and 8 of this chapter.

[c]Direct area plus undisturbed area.

FIGURE 11

EXAMPLE OF TOTAL WATER BALANCE
20 X 10^6 MPTY ORE INTEGRATED OPERATION
(FLOW IN 10^9 GPY WATER)



*NOT CONTROLLABLE

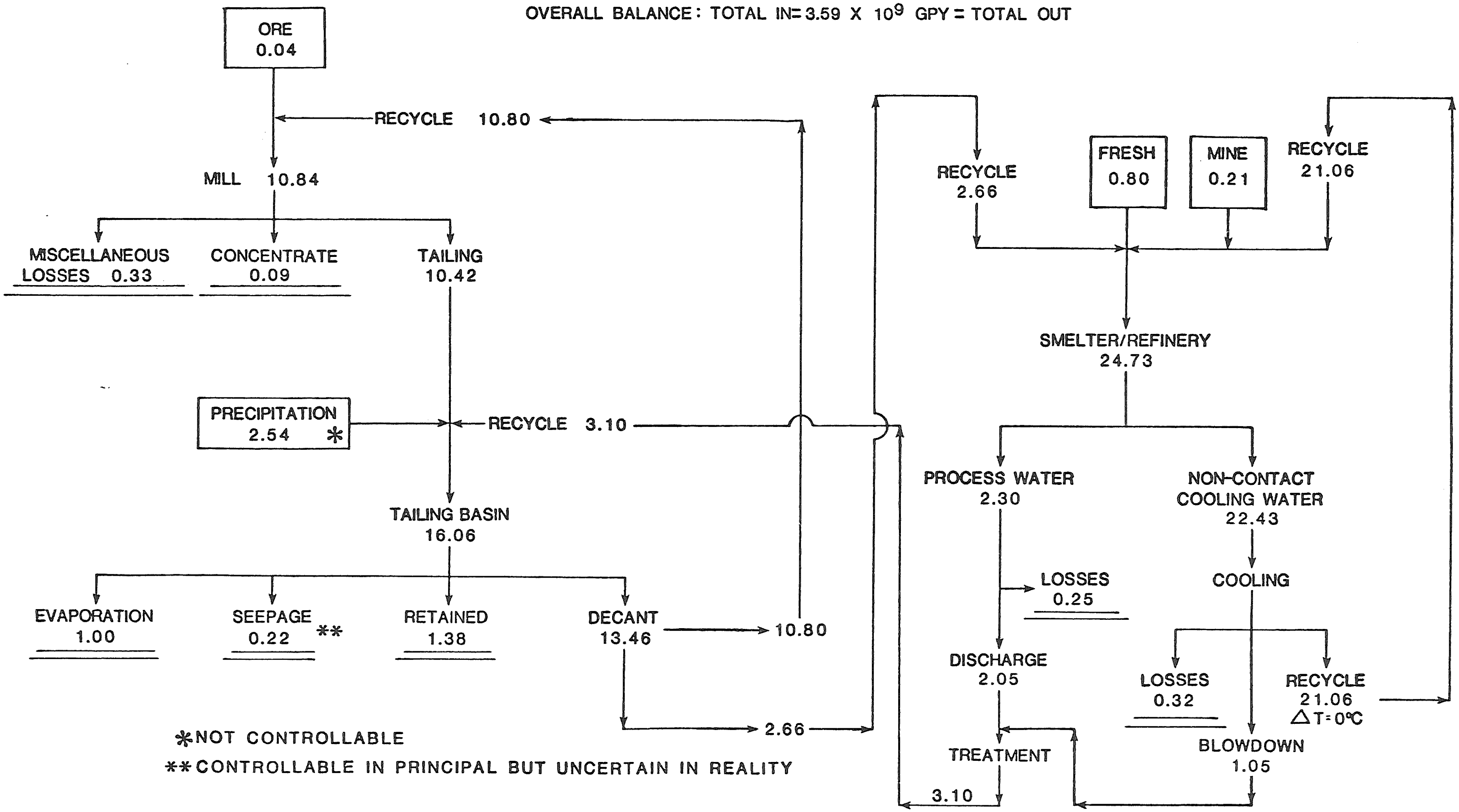
**CONTROLLABLE IN PRINCIPAL BUT UNCERTAIN IN REALITY

FIGURE 12

EXAMPLE OF TOTAL WATER BALANCE
16.68 X 10^6 MPTY ORE INTEGRATED OPERATION

(FLOW IN 10^9 GPY WATER)

OVERALL BALANCE: TOTAL IN = 3.59 X 10^9 GPY = TOTAL OUT



*NOT CONTROLLABLE

**CONTROLLABLE IN PRINCIPAL BUT UNCERTAIN IN REALITY

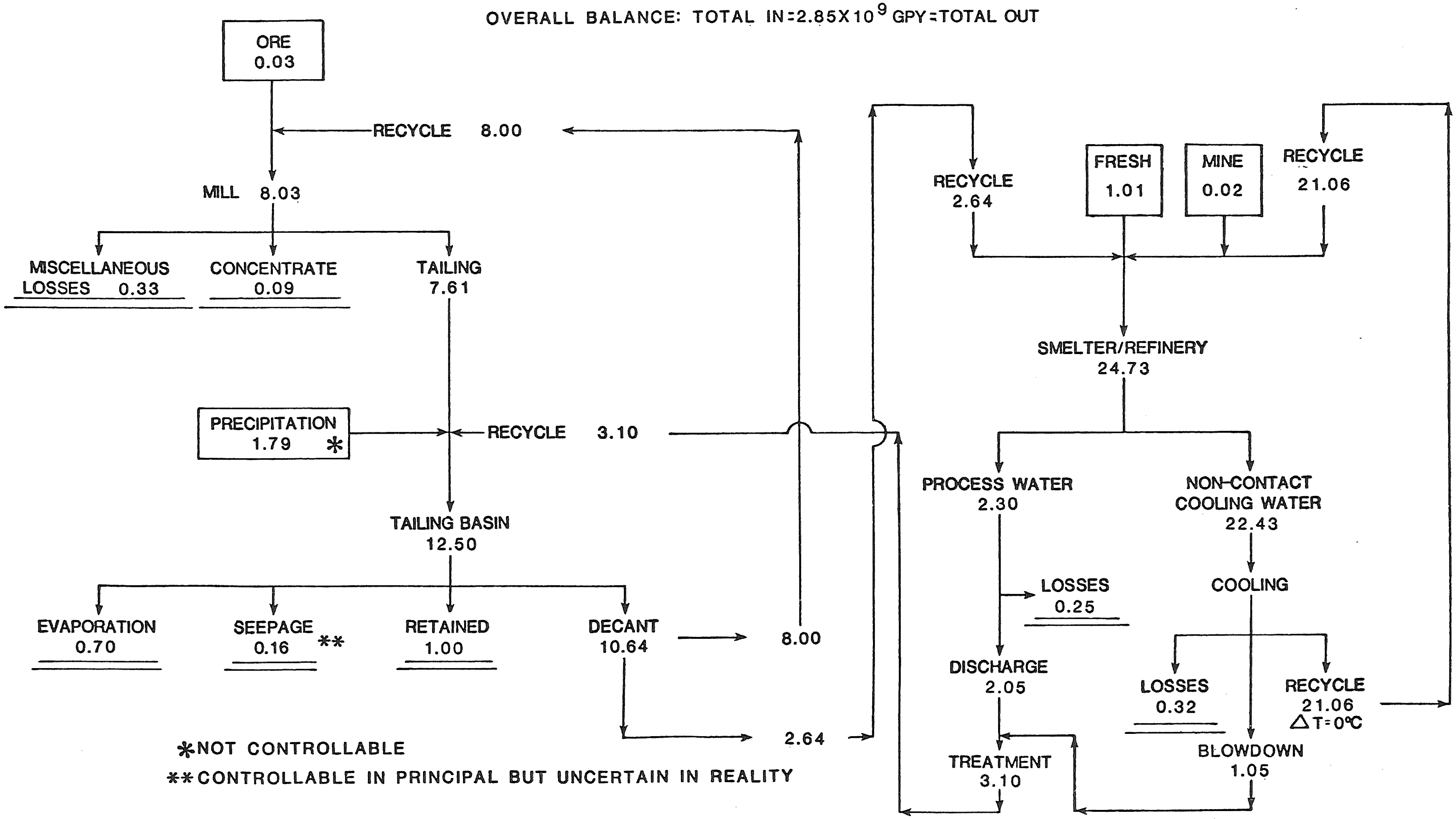
FIGURE 13

EXAMPLE OF TOTAL WATER BALANCE

12.35×10^6 MPTY ORE INTEGRATED OPERATION

(FLOW IN 10^9 GPY WATER)

OVERALL BALANCE: TOTAL IN $= 2.85 \times 10^9$ GPY = TOTAL OUT



*NOT CONTROLLABLE

**CONTROLLABLE IN PRINCIPAL BUT UNCERTAIN IN REALITY

The production method is open pit mining, conventional crushing and grinding, bulk sulfide flotation, tailing disposal in a basin one mile distant from the processing facility, with maximum water reclamation and recycle to the processing facility. The concentrate is flash-smelted, converted, and refined into copper, nickel, and cobalt products suitable for sale. Precious metals are also recovered. The smelter has secondary hooding to direct weak SO_2 gases to the stack, and strong SO_2 gases are ducted to a sulfuric acid plant, resulting in the production of some 450,000 mtpy of sulfuric acid.

5.3.1.2 16.68 X 10^6 mtpy Combination Open Pit and Underground Model--This model is the second largest (based on ore production) example of potential development in Minnesota. The total project life is 30 yr, but the effective life is 24.4 yr in which an average of 16.68 X 10^6 mtpy ore is mined. Resulting production is 635,259 mtpy concentrate from a single processing facility, and 100,000 mtpy refined copper plus nickel metal.

This method combines an 11.33 X 10^6 mtpy open pit mine and a 5.35 X 10^6 mtpy underground mine, with conventional crushing and grinding, bulk flotation in a single large plant, tailing disposal one mile from processing, and maximum water reclamation for processing. As for all examples, in underground mining the primary crushing is done underground, and in open pit mining the primary crushing is done outside of the mine, on the surface. All subsequent operations are carried out in surface facilities. Metal production is identical to the 20.00 X 10^6 mtpy operation.

5.3.1.3 12.35 X 10^6 mtpy Underground Model--As the third largest example of potential Minnesota operations, this model also results in 635,259 mtpy concentrate and 100,000 mtpy metal, but from a 12.35 X 10^6 mtpy ore underground mine. The total life is the same, 30 yr, but the effective life is only 23 yr due to more extensive development requirements for underground mining compared to open pit mining. Underground mining employs primary crushing in the mine followed by all subsequent operations described above, which are carried out on the surface.

The above described models are totally integrated examples of potential copper-nickel developments in Minnesota, each consisting of mine, processing, smelting, and refining facilities resulting in the production of 100,000 mtpy of combined copper plus nickel metal. These models form the basis then for all subsequent environmental concerns described in Volumes 3-5 of this report.

5.3.1.4 Model Variations--Additionally, many variations of the above models are possible, or variations of components of the models, which would result in intermediate products such as ore or concentrate and which would necessitate subsequent facilities elsewhere to complete the processing of mined ore to finished metal.

To illustrate this point, several specific examples were investigated and are summarized in Tables 12 and 13 for comparison to the previously developed examples. These variations are discussed along with summations of selected previously described models in the following sections.

Tables 12 & 13

5.35 X 10^6 mtpy Underground Model--Consisting of an underground mine and associated processing plant, this model by itself is not large enough to support the smelter/refinery model. As the underground portion of the 16.68 X 10^6 mtpy combination example, but with a smaller processing facility, this model has a

Table 12. Possible exploration, mining, and processing variations of major model components.

VARIABLE	EXPLORATION, MINING AND PROCESSING MODEL VARIATION BY DESIGN AND CAPACITY, 10^6 mtpy ore: 5.35 Underground Mine Plus Plant	11.33 Open Pit Mine Only	11.33 Open Pit Mine Plus Plant	16.68 Combination Mine: Common Plant	16.68 Combination Mine: Individual Plants
Production Comparison					
10^3 mtpy concentrate	275.3	0	360.0	635.3	635.3
10^6 mt total concentrate	6.33	0	9.00	15.33	15.33
Personnel Requirements					
Peak construction	930	210	1140	1430	2070
Full production	857	600	854	1599	1711
Energy Requirements, Full Production					
Fossil fuels, 10^9 BTU/yr	252.8	1544.3	969.1	1220.1	1221.9
Electricity, 10^6 kwh/yr	179.3	18.3	280.6	459.0	459.9
Total, 10^{12} BTU/yr	2.14	1.56	3.91	6.04	6.05
Area Required, acres					
Direct total	1594	1689	4277	5715	5871
Undisturbed (40% of direct)	638	676	1711	2286	2349
Total	2232	2365	5988	8001	8220
Cost Data					
Capital, $\$10^6$ total	205.8	146.7	285.6	440.6	491.4
Operating, $\$10^6$/yr	46.27	28.92	50.79	93.14	97.06

Table 13. Smelter/refinery complex variation by location (100,000 mtpy copper plus nickel metal).

	ON SITE[a]	IN REGION[b]	IN DULUTH	OUT-OF-STATE[c]
Capital cost, $\$10^6$	324.21	372.84	372.84	372.84
Operating cost, $\$10^6$/yr	33.27	36.67	38.26	49.49
Operating personnel	621	649	649	649

[a]Base case example.
[b]Regional Copper-Nickel Study Area.
[c]Canada (Sudbury area).

total life of 30 yr with an effective life of 23 yr during which 275,259 mtpy of concentrate are produced. This is equivalent to 43,330 mtpy of metal if the operation were associated with a smelter/refinery complex. Methodology is identical to the 12.35 X 10^6 mtpy model except that concentrate is the final product.

<u>11.33 X 10^6 mtpy Open Pit Model</u>--This model consists of an open pit mine and associated processing facility. Not large enough by itself to support the modeled smelter facility, it is the open pit portion of the 16.68 X 10^6 mtpy model, but with a smaller processing plant. With a total life of 30 yr and an effective life of 25 yr, concentrate production amounts to 360,000 mtpy, equivalent to 56,670 mtpy of copper plus nickel metal if it were treated in the modeled smelter/refinery complex. The method is identical to that described for the 20.00 X 10^6 mtpy open pit operation, except that the concentrate is the final product.

<u>Isolated 11.33 X 10^6 mtpy Open Pit Model</u>--This model is identical to the mining portion of the model described above, but it is isolated from any subsequent treatment and therefore includes the requirement of transporting ore to some processing plant. In addition, facilities which would normally be shared (safety, first aid, parking lot, etc.) with other phases of an integrated operation must now be developed for the mine only.

<u>Combination 5.35 X 10^6 mtpy Underground and 11.33 X 10^6 Open Pit Models</u>--This model consists of a combination of the models described above, and results in 2 mines plus 2 processing plants producing concentrates which could then be combined to supply any of the smelter/refinery options listed below to result in the production of 100,000 mtpy of copper plus nickel metal.

This combination is listed in Table 12 and is compared to the corresponding portion of the 16.68 X 10^6 mtpy integrated model, which has both mines combining their ores to supply one large processing plant. The difference then reflects the scale economies of using one large processing plant, rather than 2 smaller ones.

Isolated 100,000 mtpy Copper plus Nickel Smelter/Refinery--The final illustrated variation of the basic models is a smelter/refinery complex isolated from mining and processing facilities. This facility is identical to the smelter/refinery described above, except that it is remotely located from the supporting mining and processing facilities to illustrate the additional requirements of support facilities and of transporting concentrate to such a smelter/refinery.

Tables 12 and 13 summarize the major variables for the above models and repeat some of the data presented in the previous section for comparison. What these tables allow is a discussion of combinations not previously described, such as isolated 5.35 and 11.33 X 10^6 mtpy mines, each with its own processing facility, feeding a smelter/refinery complex located away from the area.

Considerable discussion can be generated over the location of the smelter/refinery complex. Table 13 estimates the principal differences depending on location. These differences in dollars and personnel reflect additional facilities necessary to haul concentrate to the smelter, to develop a separate water management system, and to establish all facilities which would otherwise be shared in a totally integrated operation. Costs of water treatment facilities are not included.

5.3.2 Year-By-Year Detailed Example--20.00 X 10^6 mtpy Open Pit Integrated Operation

Each of the previously described model operations was detailed on a year-by-year basis for all major parameters and then summarized for this report. To illustrate this process, the following discussion details the 20.00 X 10^6 mtpy open pit mining operation, fully integrated with on-site processing, smelting, and refining facilities resulting in 100,000 mtpy copper plus nickel metal.

The total 30-yr life of the operation is illustrated in Figure 14 which shows each year from the beginning of construction through shut-down of the entire facility. Specifically shown is each phase of the operation and its corresponding manpower requirement for mining, processing, and smelting/refining. Construction of each facility is sequenced such that the entire operation will reach full production in the shortest possible time and with the least number of interfacing problems.

Figure 14

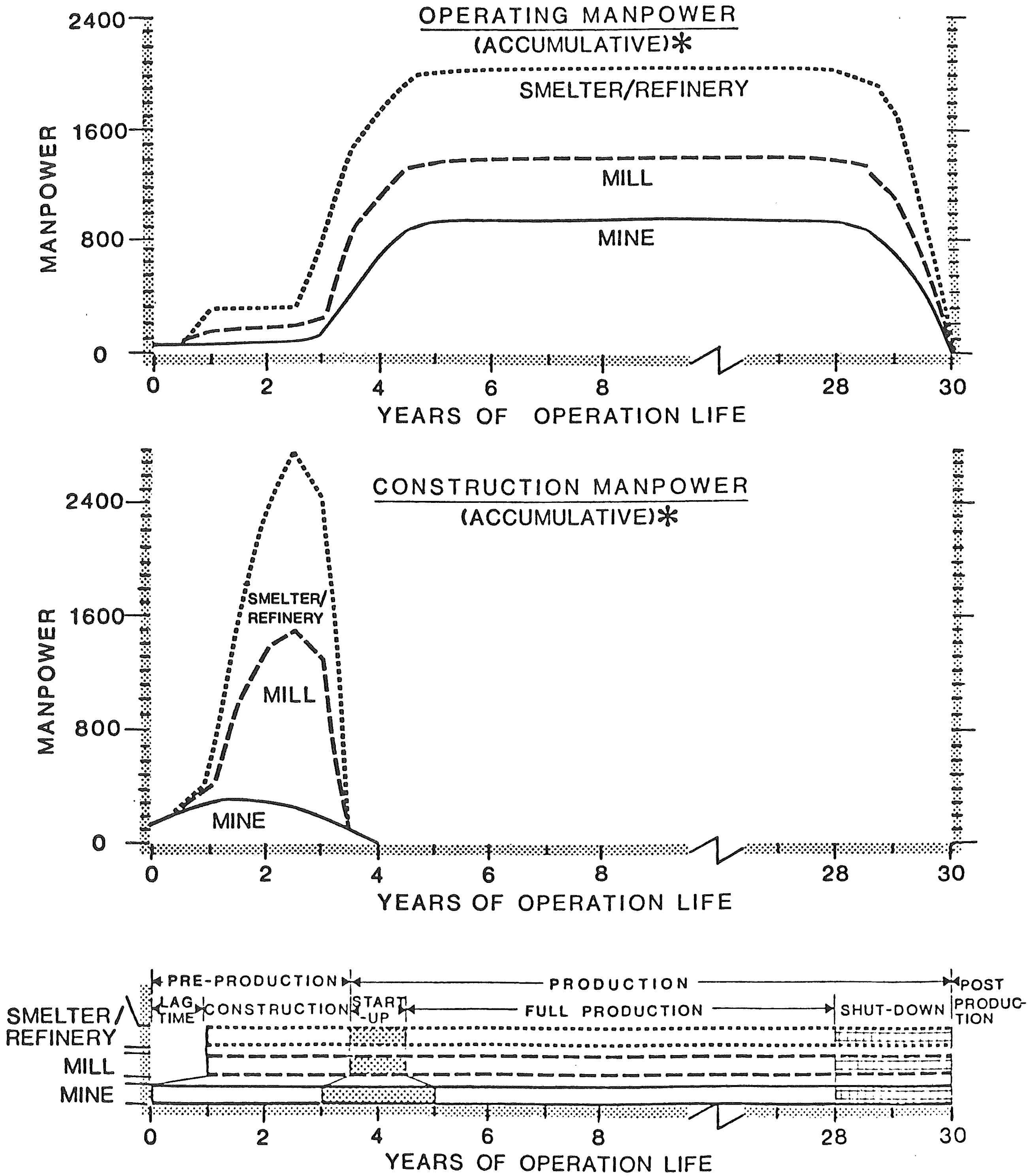
Construction personnel are tied to an overall schedule which is completed within 4 yr of the starting date. Operating personnel are generally brought into their respective facilities ahead of actual production, where they are trained and prepared to run the operation once construction is completed.

Major summary values over the 30-yr life of the 20.00 X 10^6 mtpy operation (as detailed in following tables) are:

Ore produced	500 X 10^6 mt
Metal produced	2.5 X 10^6 mt
Capital cost	\$757 X 10^6
Operating cost	\$3.0 X 10^9
Energy consumed	404 X 10^{12} BTU

FIGURE 14

TYPICAL 20X10^6 MTPY ORE INTEGRATED OPERATION TIME LINE



*EACH GRAPH INCLUDES THE MANPOWER REQUIREMENTS OF ALL THE OPERATIONS LISTED BELOW IT

Operating manpower	1,999
Direct land used	7,315 acres
Total fresh water consumed	23.5×10^9 gal

Table 14 presents the percentage distribution of the above parameters for the mining, processing, and smelting/refining facilities of this totally integrated operation. Note that mining is more labor intensive than the other phases, mainly due to the large amounts of material being handled and the inability of the system to be automated to the extent possible with processing and smelting/refining.

Table 14

Processing consumes the majority of the land used, principally for tailing disposal, and more than 85% of the water, which is needed to transport material throughout the system. Energy, on the other hand, is mainly consumed in the smelter/refinery facility and particularly in the energy intensive nickel refinery.

The following tables list the year-by-year and total values for the integrated model variables:

Table 15. Capital and operating costs

Table 16. Energy requirements

Table 17. Manpower requirements

Table 18. Land requirements

Table 19. Water distribution

Table 20. Material distribution

Table 14. Summary 20.00 X 10^6 mtpy integrated model.

COMPARATIVE PARAMETER	% DISTRIBUTION OF PARAMETERS BY PHASE		
	Mining	Processing	Smelting/ Refining
Capital cost	26.7	30.5	42.8
Operating cost	34.1	38.0	27.9
Energy consumption	12.8	30.9	56.3
Manpower needs at full production	48.2	20.7	31.1
Direct land use	37.2	60.4	2.4
Fresh water consumed[a]	0	86.3	13.7

[a]Assigning all tailing basin losses to processing plant.

For a detailed breakdown of these totals, on a year-by-year basis for each phase of the integrated operation, the reader is referred to the technical assessment reports referenced earlier.

Tables 15 through 20

As mentioned earlier, the year-by-year schedule is shown in Figure 14 for construction and operating personnel, and for the construction startup, full production and shut down phases of the operation. Similar schedules would hold for each of the other modeled operations, although the time for each segment of the total model would vary as shown in the summary tables.

5.3.3 Smelter/Refinery Emission Control Options

The following discussion summarizes the various emission control options detailed in Chapter 4, for the smelting and refining of 635,259 mtpy of bulk copper-nickel concentrate analyzing 13.825% Cu, 2.647% Ni, 30.001% Fe, and 25.870% S. In each case, the refined product will be 100,000 mtpy of metal (84,584 mt of copper plus 15,426 mt of nickel), plus cobalt, precious metals, and sulfur recovered as byproducts.

Concentrate will be pumped to the smelter as a 65% solids slurry, dried and flash smelted. Copper-nickel matte coming from the flash furnace will be further treated in Pierce-Smith converters by a modified process involving selective fluxing, blowing, slagging, and skimming to effect the proper segregation and separation of the matte into a slag (containing the bulk of the recoverable nickel and cobalt) and blister copper. The blister copper will then be fire-refined and prepared for casting in anode furnaces, to be shipped as anodes to the copper refinery for final purification by electrorefining procedures.

Table 15. Total capital and operating costs for 20 X 10^6 mtpy integrated model.

YEAR	CAPITAL $\$10^6$	OPERATING $\$10^6$	YEAR	CAPITAL $\$10^6$	OPERATING $\$10^6$
1	29.87	0	17	23.34	117.51
2	203.29	0	18	0.74	117.51
3	200.69	0	19	0	117.51
4	190.51	34.14	20	0	117.51
5	0	107.57	21	11.00	117.51
6	11.00	120.11	22	12.67	117.51
7	12.67	120.11	23	0	117.51
8	0	120.11	24	0	117.51
9	1.92	120.11	25	1.92	117.51
10	0.74	120.11	26	11.73	117.51
11	11.00	120.11	27	0	117.51
12	12.67	120.11	28	0	117.51
13	0	120.11	29	0	88.13
14	0	117.51	30	0	33.43
15	0	117.51			
16	21.39	117.51	TOTAL	757.13	2986.80

Table 16. Total energy requirements, 20 X 10^6 mtpy integrated model.[a]

YEAR	ELECTRICAL 10^6 KWH	ELECTRICAL 10^{12} BTU	GASOLINE 10^{12} BTU	DIESEL 10^{12} BTU	NATURAL GAS 10^{12} BTU	COAL 10^{12} BTU	PROPANE 10^{12} BTU	MISC. THERMAL 10^{12} BTU	TOTAL 10^{12} BTU
1	0	0	0	0	0	0	0	0	0
2	4.8	0.05	0.01	0.23	0	0	0	0	0.29
3	4.8	0.05	0.01	0.23	0	0	0	0	0.29
4	143.3	1.50	0.02	0.62	0.03	0.13	0.01	0.27	2.58
5	941.6	9.89	0.05	1.39	0.19	0.93	0.08	1.55	14.08
6	1080.1	11.34	0.06	1.78	0.22	1.06	0.09	1.76	16.31
13	1080.1	11.34	0.06	1.78					16.31
14	1075.2	11.29	0.05	1.54					16.02
28	1075.2	11.29	0.05	1.54	0.22	1.06	0.09	1.76	16.02
29	806.6	8.47	0.04	1.16	0.17	0.80	0.07	1.35	12.05
30	268.7	2.82	0.01	0.39	0.06	0.27	0.02	0.53	4.10
TOTAL[b]	26938.6	282.86	1.45	41.39	5.55	26.50	2.30	43.88	404.11

[a]Energy requirements for construction are not included.
[b]Totals may not equal actual sums of columns due to rounding.

Table 17. Total manpower requirements, 20 X 10^6 mtpy integrated model.

YEAR	CONSTRUCTION	OPERATING	TOTAL
1	229	100	329
2	1568	355	1923
3	2765	355	3120
4	113	1465	1578
5	0	1935	1935
6		1999	1999
28		1999	1999
29		1935	1935
30	0	942	942

Table 18. Total direct land requirements, 20 X 10^6 mtpy integrated model.[a]

	MINING		PROCESSING		SMELTER/ REFINERY		TOTAL	
YEAR	Acres	ha	Acres	ha	Acres	ha	Acres	ha
1	40	16	0	0	0	0	0	16
2	96	39	2208	893	75	30	2379	962
3	96	39	2208	893	75	30	2379	962
4	125	51	0	0	12	5	137	56
5					13	5	138	56
6					0	0	125	51
12	125	51					125	51
13	119	48					119	48
14	74	30					74	30
29	74	30					74	30
30	64	26	0	0	0	0	64	26
TOTAL	2724	1107	4416	1786	175	70	7315	2963

[a]Undisturbed but controlled areas (at 40% of the direct area requirements) are not included here.

Table 19. Total water distribution, 20 X 10^6 mtpy integrated model.

YEAR	PROCESSING-SMELTING/REFINING TOTAL[a] 10^9 gal	PROCESSING-SMELTING/REFINING FRESH 10^9 gal	MINE DISCHARGE 10^9 gal
1	0	0	0
2	0	0	0
3	0	0	0
4	4.64	4.64	0.12
5	33.09	0.66	0.23
6	37.73	0.76	
28	37.73	0.76	
29	28.30	0.57	
30	9.43	0.19	0.23
TOTAL	943.25	23.54	6.10

[a]Maximum internal recycle in smelter/refinery.

[b]Does not include 0.05 X 10^9 gal/yr water in ore, 0.23 X 10^9 gal/yr mine discharge water, and 3.11 X 10^9 gal/yr precipitation water on tailing basin. If any of these inflows change, the fresh water requirement will change accordingly.

Table 20. Total material distribution, 20 X 10^6 mtpy integrated model.

YEAR	ORE 10^6 mt	LEAN ORE & WASTEROCK 10^6 mt	OVERBURDEN 10^6 yd^3	CONCENTRATE 10^6 mt	TAILING 10^6 mt	CU+NI METAL[a] 10^6 mt	DISCARD SLAG 10^6 mt	SULFURIC ACID 10^6 mt
1	0	0	0	0	0	0	0	0
2	0	0	2.1	0	0	0	0	0
3	0	0		0	0	0	0	0
4	5.0	6.5		0.079	2.421	0.012	0.076	0.056
5	15.0	19.5		0.556	16.944	0.088	0.532	0.394
6	20.0	26.0		0.635	19.365	0.100	0.608	0.450
13			2.1					
14			0					
28	20.0	26.0		0.635	19.365	0.100	0.608	0.450
29	15.0	19.5		0.476	14.524	0.075	0.456	0.338
30	5.0	6.5	0	0.159	4.841	0.025	0.152	0.113
TOTAL	500.0	650.0	25.2	15.881	484.119	2.500	15.200	11.250

[a]84% Cu and 16% Ni.

Slag from the flash furnace will be cleaned, along with the copper converter slags, in 2 electric furnaces to yield a nickel-copper matte/alloy and a discardable slag. This nickel-copper matte will undergo another converting step and then be sent to the nickel refinery where leaching and electrowinning techniques will be used to recover nickel, copper, and cobalt as separate products.

The variations detailed in Chapter 4 are summarized in the following sections 5.3.3.1 through 5.3.3.4.

5.3.3.1 Basic Model--The basic model incorporates no control of weak SO_2 gases, but has acid plant control of strong SO_2 gases to a concentration of 650 ppm SO_2. Emission totals are (in mtpy S): 4,960 as fugitives, 1,177 as acid plant tail gas, total 6,137. The model is shown in Figure 15.

Figure 15

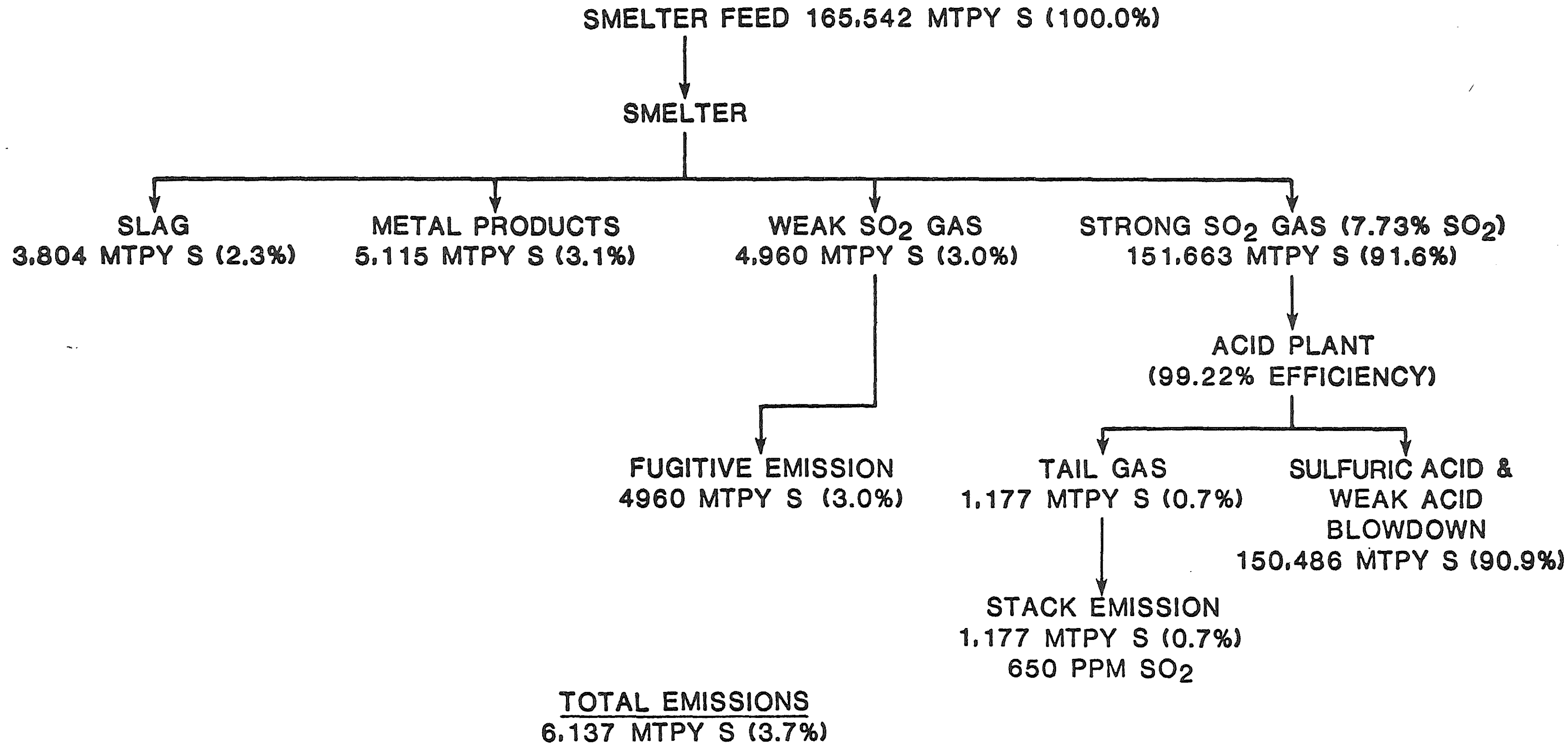
Current new source performance standards (NSPS) regulations do not specify their applicability to weak SO_2 gas streams. This case assumes no treatment with all weak SO_2 gases (4,960 mtpy S) emitted as fugitives. Table 21 summarizes the sulfur balance for this example.

Table 21

As basic as the above example may seem, irrespective of whether such an approach would be allowed by regulatory authorities, it is not realistic since the uncontrolled fugitive emissions would prevent effective and efficient operation of the facility on the part of the personnel involved. Some control of the weak SO_2 gases is necessary to provide a reasonable environment for the workers. Such an example follows.

FIGURE 15

BASIC MODEL SULFUR BALANCE FOR FLASH FURNACE WITH ACID PLANT CONTROL OF STRONG SO_2 GAS STREAM TO 650 PPM SO_2. NO WEAK SO_2 GAS CONTROL*



*NORMAL OPERATING CONDITIONS ARE ASSUMED

Table 21. Basic model sulfur balance for flash smelter facility with acid plant control of strong SO_2 gases only.[a]

PRODUCT	SO_2 REMOVAL EFFICIENCY, %	GAS STREAM ppm SO_2[b]	SULFUR PRODUCTION mtpy	% Distribution
Smelter feed			165,542	100.0
Slag product			3,804	2.3
Metal products			5,115	3.1
Weak SO_2 gas fugitives			4,960	3.0
Strong SO_2 gas to acid plant		7.73%	151,663	91.6
Acid plant	99.22			
Sulfuric acid and acid blowdown			150,486	90.9
Acid plant tail gas (to stack)		650	1,177	0.7
Total Emissions			6,137	3.7

[a]Normal operating conditions are assumed.
[b]PPM SO_2 unless otherwise noted.

5.3.3.2 <u>Base Case Smelter/Refinery Model</u>--This model incorporates secondary hooding of weak SO_2 gases to direct 90% of this stream to the stack. Acid plant control treats strong SO_2 gases to a concentration of 650 ppm SO_2. Emission totals (in mtpy S) are: 495 as fugitives, 5,642 as stack discharge, total 6,137, as in 5.3.3.1 above.

Figure 16 illustrates the general flows involved in the base case smelter/ refinery complex and Figure 17 provides a sulfur balance for this case (see Chapter 4).

<u>Figures 16 & 17</u>

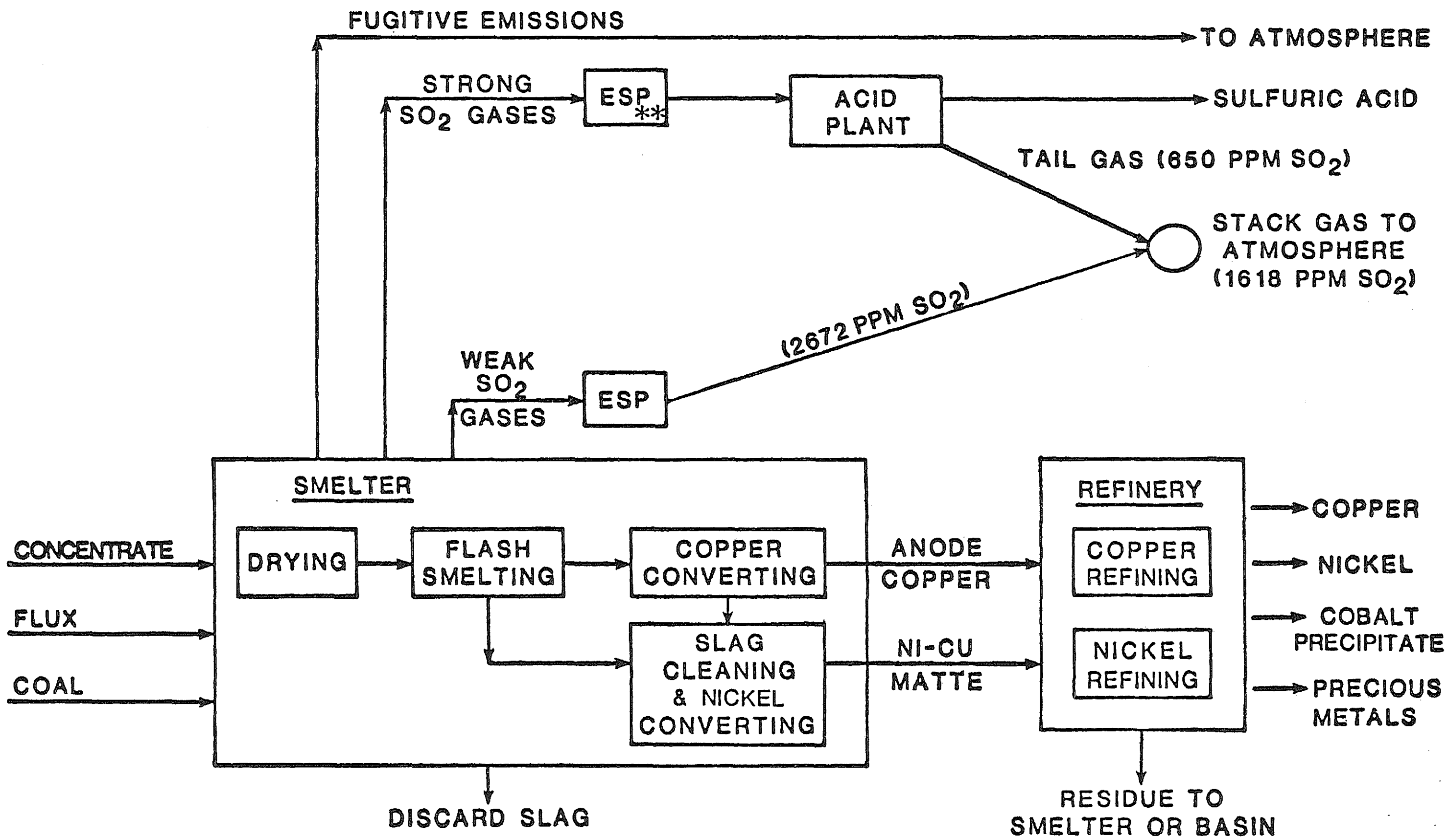
It must be stressed that this base case does not improve the overall emission picture. However, what it does do is to redirect most of the weak SO_2 gases to the stack discharge, thereby providing a greater dispersing mechanism than in the previous example. This redirection is realized through the use of secondary hooding to collect 90% of the weak SO_2 gases and channel them through the stack along with the acid plant tail gas.

The net result is a cleaner environment within and in the immediate vicinity of the facility, as only 495 mtpy sulfur is emitted as fugitives rather than the 4,960 mtpy sulfur as illustrated above in the basic model. The collected 4,465 mtpy sulfur combines with the acid plant tail gas SO_2 content of 1,177 mtpy sulfur for a total stack discharge of 5,642 mtpy sulfur in a gas containing 1,618 ppm SO_2 (see Table 22).

<u>Table 22</u>

FIGURE 16

GENERALIZED FLOWSHEET WITH SO_2 EMISSION CONCENTRATION, BASE CASE COPPER-NICKEL SMELTER/REFINERY USING FLASH SMELTING FURNACE*

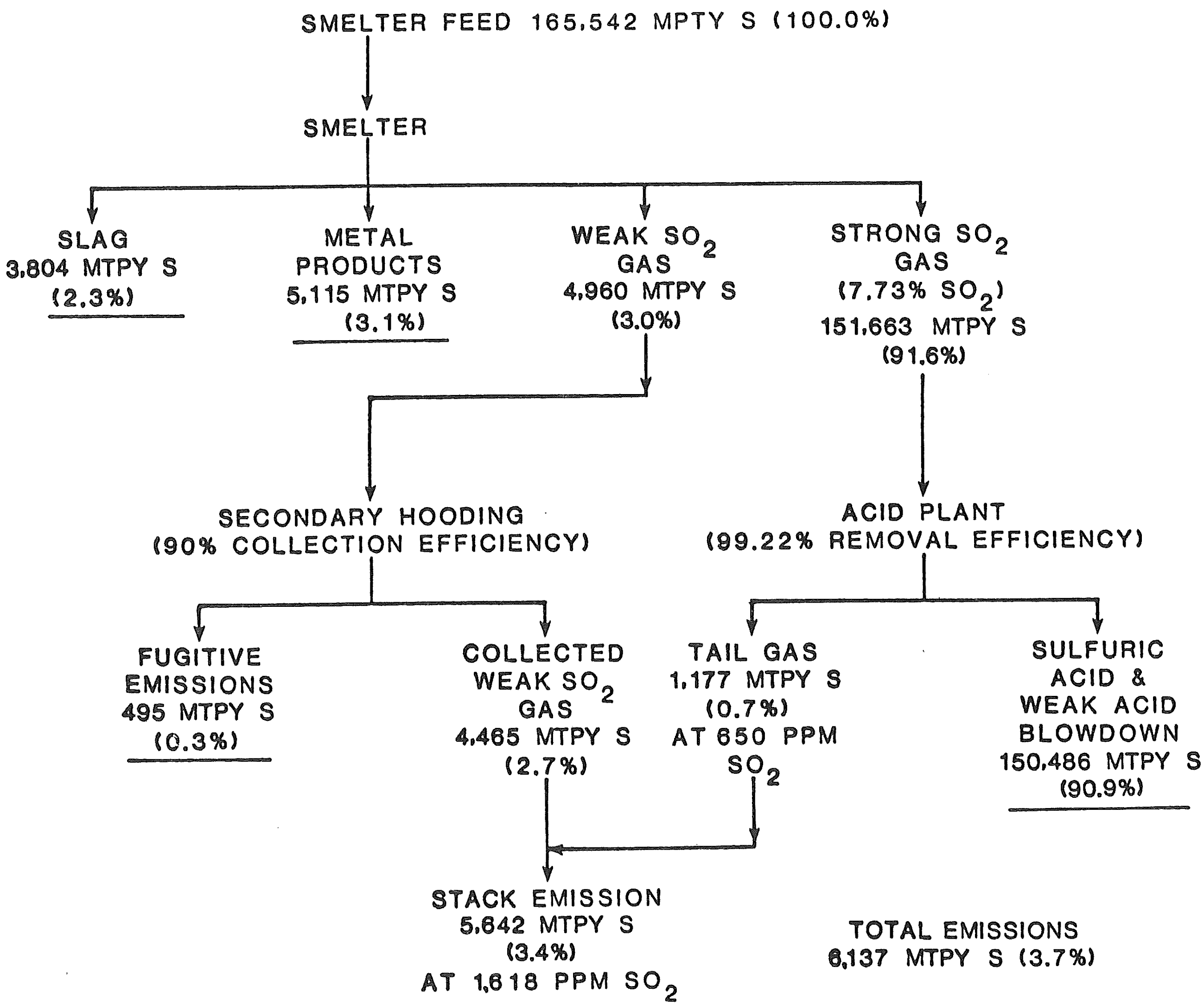


*NORMAL OPERATING CONDITIONS ARE ASSUMED

**ELECTROSTATIC PRECIPITATOR

FIGURE 17

BASE CASE MODEL SULFUR BALANCE FOR FLASH FURNACE WITH ACID PLANT CONTROL OF STRONG SO_2 GAS TO 650 PPM SO_2, SECONDARY HOODING COLLECTION OF WEAK SO_2 GAS TO REDIRECT IT TO THE STACK DISCHARGE*



* NORMAL OPERATING CONDITIONS ARE ASSUMED.

Table 22. Model sulfur balance for base case flash smelter with acid plant control of strong SO_2 gases to 650 ppm SO_2, secondary hooding of weak SO_2 gases, acid plant tail gas, and hooded weak SO_2 gas to stack at 1618 ppm SO_2.[a]

PRODUCT	SO_2 REMOVAL EFFICIENCY, %	GAS STREAM ppm SO_2[b]	SULFUR PRODUCTION mtpy	SULFUR PRODUCTION % Distribution
Smelter feed			165,542	100.0
Slag product			3,804	2.3
Metal products			5,115	3.1
Weak SO_2 gas to secondary hooding			4,960	3.0
Secondary hooding[c]	(90)			
Weak SO_2 gas to stack		2,672	4,465	2.7
Weak SO_2 gas as fugitives			495	0.3
Strong SO_2 gas to acid plant		7.73%	151,663	91.6
Acid plant	99.22			
Sulfuric acid and acid blowdown			150,486	90.9
Acid plant tail gas to stack		650	1,177	0.7
Total stack emissions		1,618	5,642	3.4
Total Emissions			6,137	3.7

[a]Normal operating conditions assumed.
[b]PPM SO_2 unless otherwise noted
[c]90% collection efficiency.

It is not specifically stated in the regulations whether the current NSPS regulations for a primary copper smelter require treatment of low SO_2 concentrations of gases such as from the electric slag cleaning furnace. In this particular model, such treatment is not provided, and the weak, or low SO_2 concentration gases, following 97% particulate removal, are assumed to be ducted directly to the stack for discharge to the atmosphere. For evaluating the potential impacts associated with smelter stack gas emissions, this can be considered as the "worst case" or minimum control alternative. It is comparable to the Phelps Dodge Hildago smelter which is the only flash smelter operating in the U.S., and is also the newest greenfield smelter in the United States.

Modeled particulate emissions are shown in Figure 18, according to the discussion given in Chapter 4. Electrostatic precipitators are assumed to remove 97% of the entrained particulate matter, and the acid plant system removes 99.9% of the particulates in the strong SO_2 gases. These efficiencies are the same wherever the equipment is used in all examples.

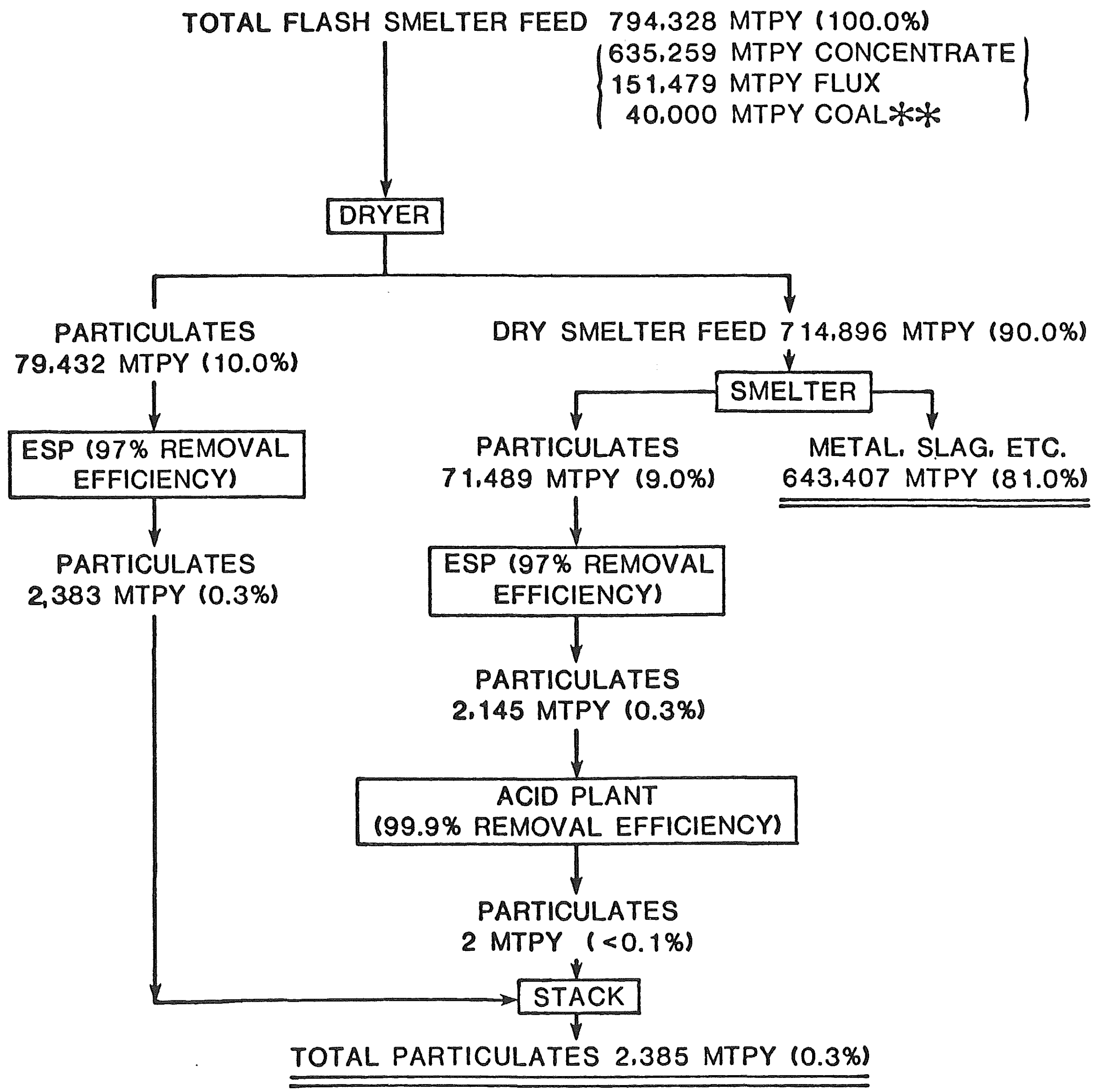
Figure 18

In order to examine the effectiveness of mitigating measures which would further reduce these emissions, two options were detailed which control the SO_2 levels with various degrees of removal efficiency. These are discussed below.

5.3.3.3 Option 1 Model--This model incorporates the same weak SO_2 gas handling as in the Base Case Model, plus a scrubber to reduce the weak SO_2 gas emissions to 650 ppm SO_2. The same acid plant control of the strong SO_2 gases gives an emission level of 650 ppm SO_2. Emission totals are (in mtpy S): 495 as fugitives and 2,256 as stack discharge at 650 ppm SO_2, total 2,751.

FIGURE 18

MODEL FOR STACK EMISSIONS PARTICULATE BALANCE FOR BASE CASE SMELTER / REFINERY COMPLEX*



*NORMAL OPERATING CONDITIONS ARE ASSUMED, IGNORING FUGITIVES WHICH ARE UNKNOWN

**ONLY 7,590 MTPY OF THE 40,000 MTPY COAL ASSUMED TO REPORT AS PARTICULATE MATTER (SEE CH. 4)

Figure 19 illustrates the general flows in this option; Figure 20 and Table 23 detail the sulfur balance.

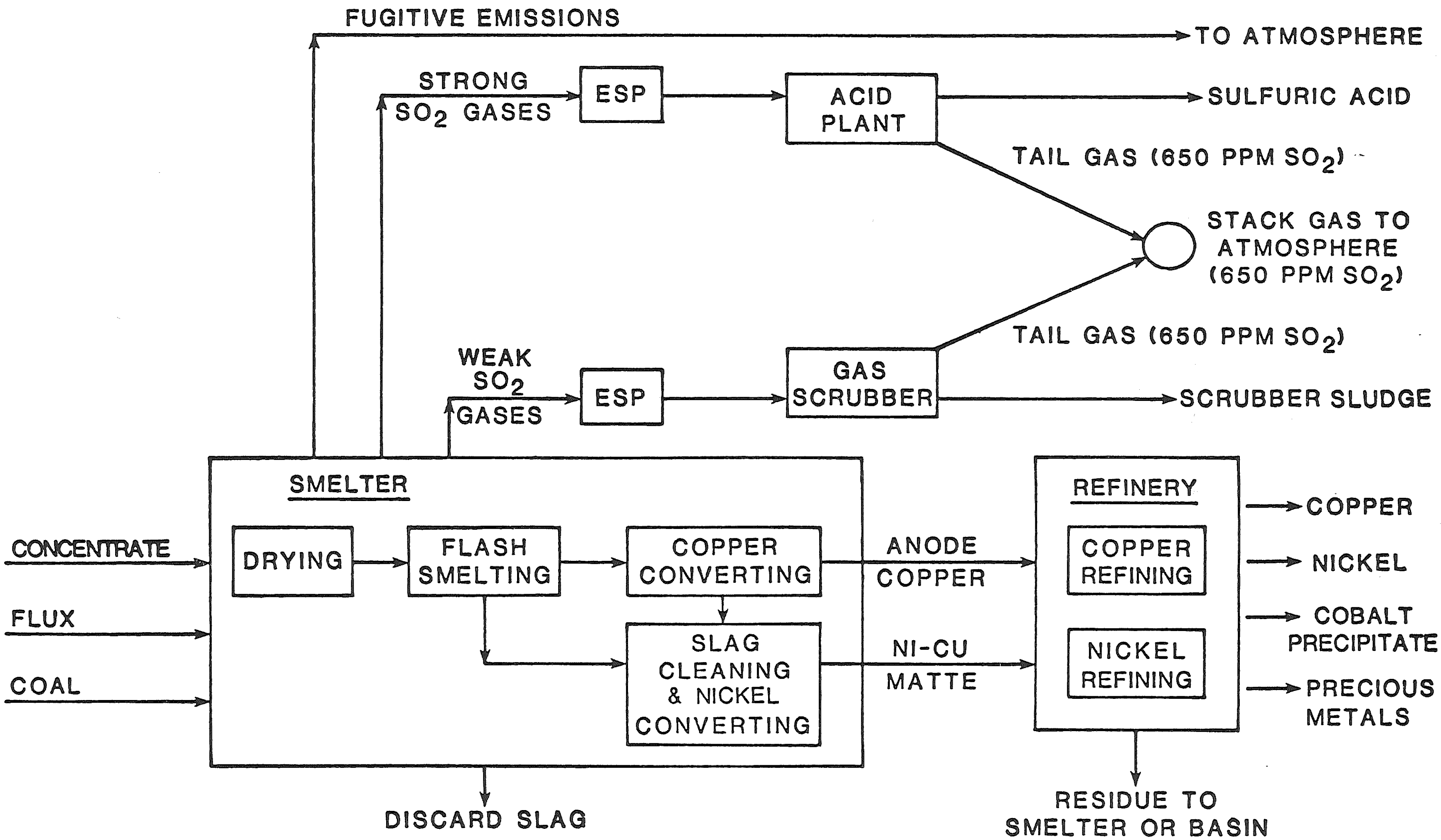
Figures 19 & 20, Table 23

In the Option 1 case situation, the strong SO_2 gases from the flash furnace and the copper converters are cycled through electrostatic precipitators (97% efficiency in particulate matter removal assumed) and fed into a DC/DA acid plant for SO_2 conversion to H_2SO_4. The acid plant removes essentially all (99.9%) of the remaining particulate matter in the gas stream and has a sulfur removal efficiency of 99.22% during steady state operation, leaving 650 ppm by volume SO_2 in the acid plant tail gases. All weak SO_2 gases from the flash furnace, copper converters, slag cleaning furnaces, and nickel converters are cycled through electrostatic precipitators (97% efficiency in particulate matter removal assumed), then through a venturi pre-scrubber with 70% efficiency in particulate matter removal and a TCA tail gas scrubber with 50% efficiency in particulate matter removal. The scrubber has an assumed sulfur conversion efficiency of 75.8% during steady state operations, resulting in a gas containing 650 ppm by volume SO_2 which is then combined with the acid plant tail gases and released to the atmosphere. Scrubber slurry from the TCA will be treated by lime precipitation along with weak acid blowdown from the acid plant. The resultant sludge material will be either recycled back to the smelter for use as flux material or disposed of as a solid waste.

The obvious advantage of this model system over the base case model is that use of a scrubber means that both weak and strong SO_2 gases are cleaned to 650 ppm SO_2 for discharge through the stack, for a 60% reduction in stack SO_2 discharge. The fugitive discharge remains as before, and the total SO_2 discharge then is reduced by 55%.

FIGURE 19

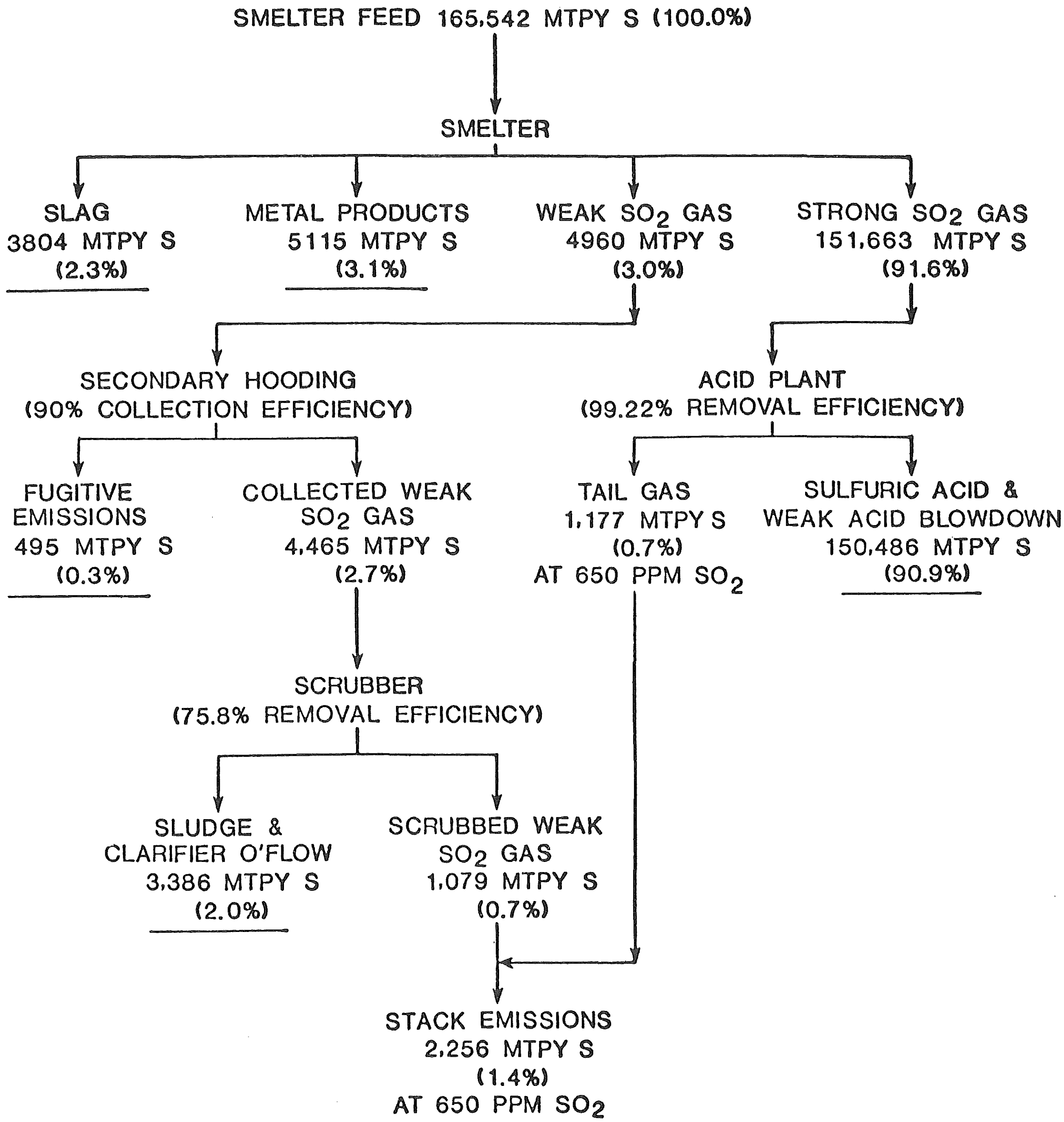
GENERALIZED FLOWSHEET WITH SO_2 EMISSION CONCENTRATIONS <u>OPTION 1</u> COPPER-NICKEL SMELTER/REFINERY USING FLASH SMELTING FURNACE*



*NORMAL OPERATING CONDITIONS ARE ASSUMED

FIGURE 20

OPTION 1 MODEL SULFUR BALANCE FOR FLASH FURNACE WITH ACID PLANT CONTROL OF STRONG SO_2 GAS TO 650 PPM SO_2, SECONDARY HOODING COLLECTION OF WEAK SO_2 GAS FOLLOWED BY SCRUBBING TO 650 PPM SO_2*



TOTAL EMISSIONS 2,751 MTPY S (1.7%)

* NORMAL OPERATING CONDITIONS ARE ASSUMED

Table 23. Option 1 model sulfur balance for flash smelter with acid plant control of strong SO_2 gases to 650 ppm SO_2, secondary hooding collection of weak SO_2 gases, and scrubbing of weak SO_2 gases to 650 ppm SO_2 with total stack discharge at 650 ppm SO_2.[a]

PRODUCT	SO_2 REMOVAL EFFICIENCY, %	GAS STREAM ppm SO_2[b]	SULFUR PRODUCTION mtpy	SULFUR PRODUCTION % Distribution
Smelter feed			165,542	100.0
Slag product			3,804	2.3
Metal products			5,115	3.1
Weak SO_2 gas to secondary hooding			4,960	3.0
Secondary hooding[c]	(90)			
Weak SO_2 gas to scrubber		2,672	4,465	2.7
Weak SO_2 gas as fugitives			495	0.3
Scrubber	75.8			
Scrubbed weak SO_2 gas to stack		650	1,079	0.7
Sludge & clarifier overflow			3,386	2.0
Strong SO_2 gas to acid plant		7.73%	151,663	91.6
Acid plant	99.22			
Sulfuric acid and weak acid blowdown			150,486	90.9
Acid plant tail gas to stack		650	1,177	0.7
Total stack emissions		650	2,256	1.4
Total Emissions (stack & fugitive)			2,751	1.7

[a]Normal operating conditions are assumed.
[b]PPM SO_2 unless otherwise noted.
[c]90% collection efficiency.

5.3.3.4 Option 2 Model--This model uses the secondary hood collection of weak SO_2 gases and acid plant control of strong SO_2 gases to 300 ppm SO_2. The acid plant tail gases and weak gas stream are combined and scrubbed to 143 ppm SO_2 assuming state-of-the-art control technology. Emission totals (in mtpy S) are: 495 as fugitives and 501 as stack discharge, total 996.

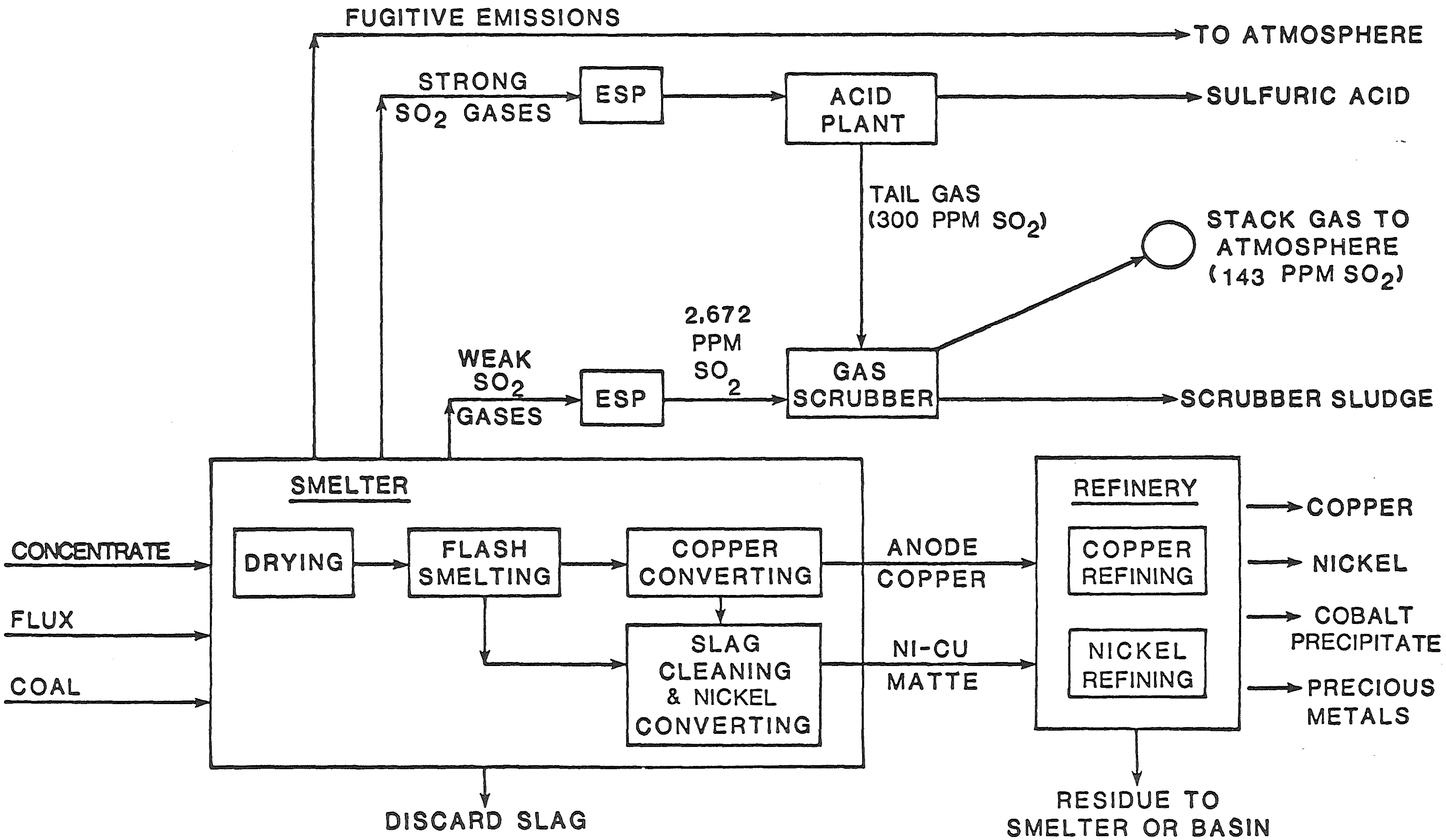
Figure 21 illustrates the general flows in this option, while Figure 22 and Table 24 detail the sulfur control scheme and balance. Figure 23 illustrates the anticipated particulate matter control, which is identical for both option 1 and option 2 models.

Figures 21, 22, & 23, Table 24

Option 2 represents the application of the best state-of-the-art control methods for a new, 1985 generation copper-nickel smelter operation. For SO_2 removed, it assumes 99.64 % acid plant efficiency and a 90% efficiency in the TCA scrubbers. The literature supports the potential for this high efficiency performance for the individual components as specified in this treatment case. The actual concentration (ppm) of stack gas emissions will vary greatly depending on the amount of dilution air; therefore, all model cases should be compared on the basis of component performance efficiencies and overall mass balance sulfur collection and losses.

In this optional situation, the strong SO_2 gases from the flash furnace and the copper converters are cycled through electrostatic precipitators (97% efficiency in particulate matter removal) and fed into a DC/DA acid plant for SO_2 conversion to H_2SO_4. The acid plant removes essentially all (99.9%) remaining particulate matter in the gas stream and has a sulfur removal effi-

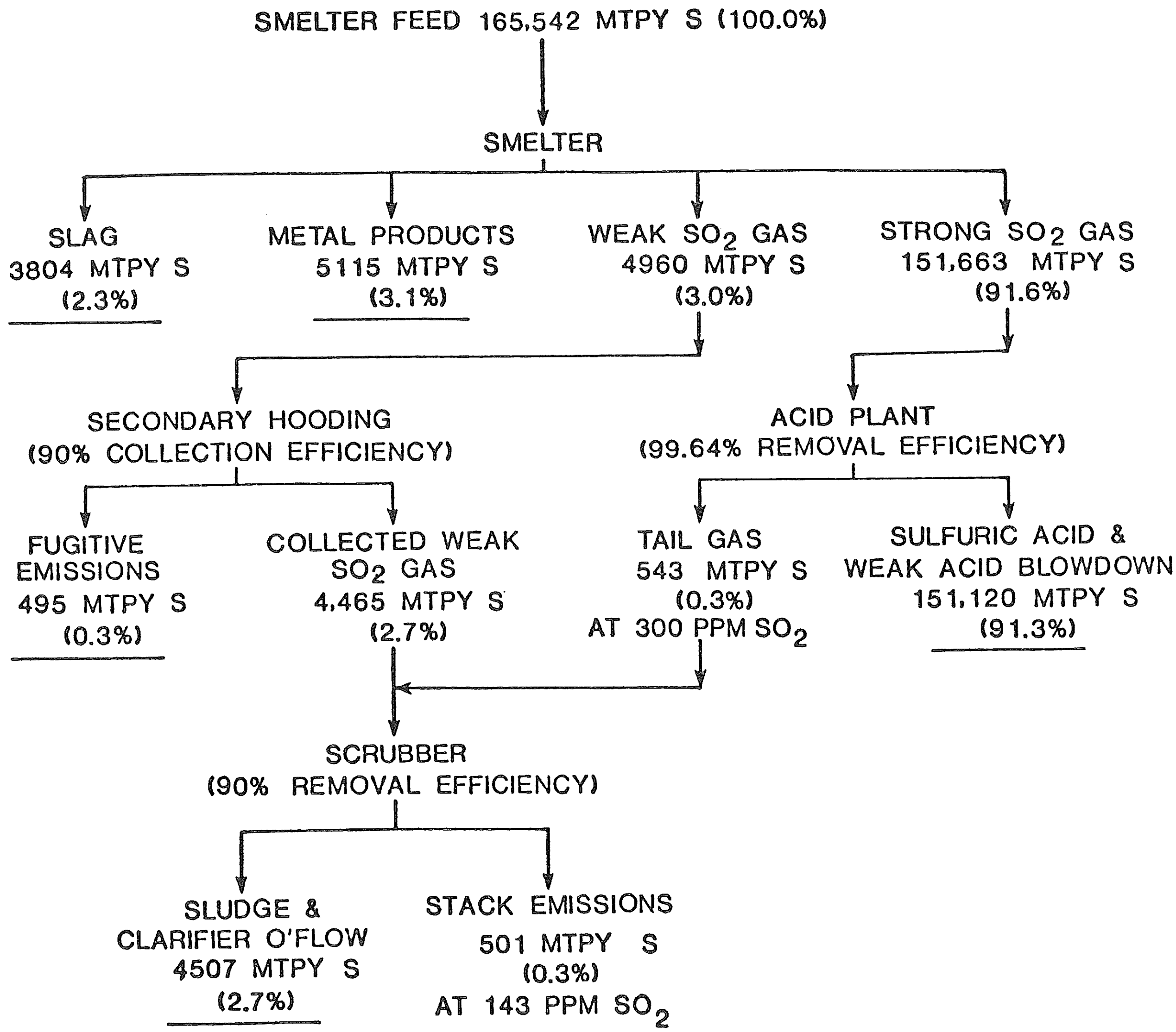
FIGURE 21 GENERALIZED FLOWSHEET WITH SO_2 EMISSION CONCENTRATIONS OPTION 2 COPPER-NICKEL SMELTER/REFINERY USING FLASH SMELTING FURNACE WITH STATE-OF-THE-ART TECHNOLOGY *



*NORMAL OPERATING CONDITIONS ARE ASSUMED

FIGURE 22

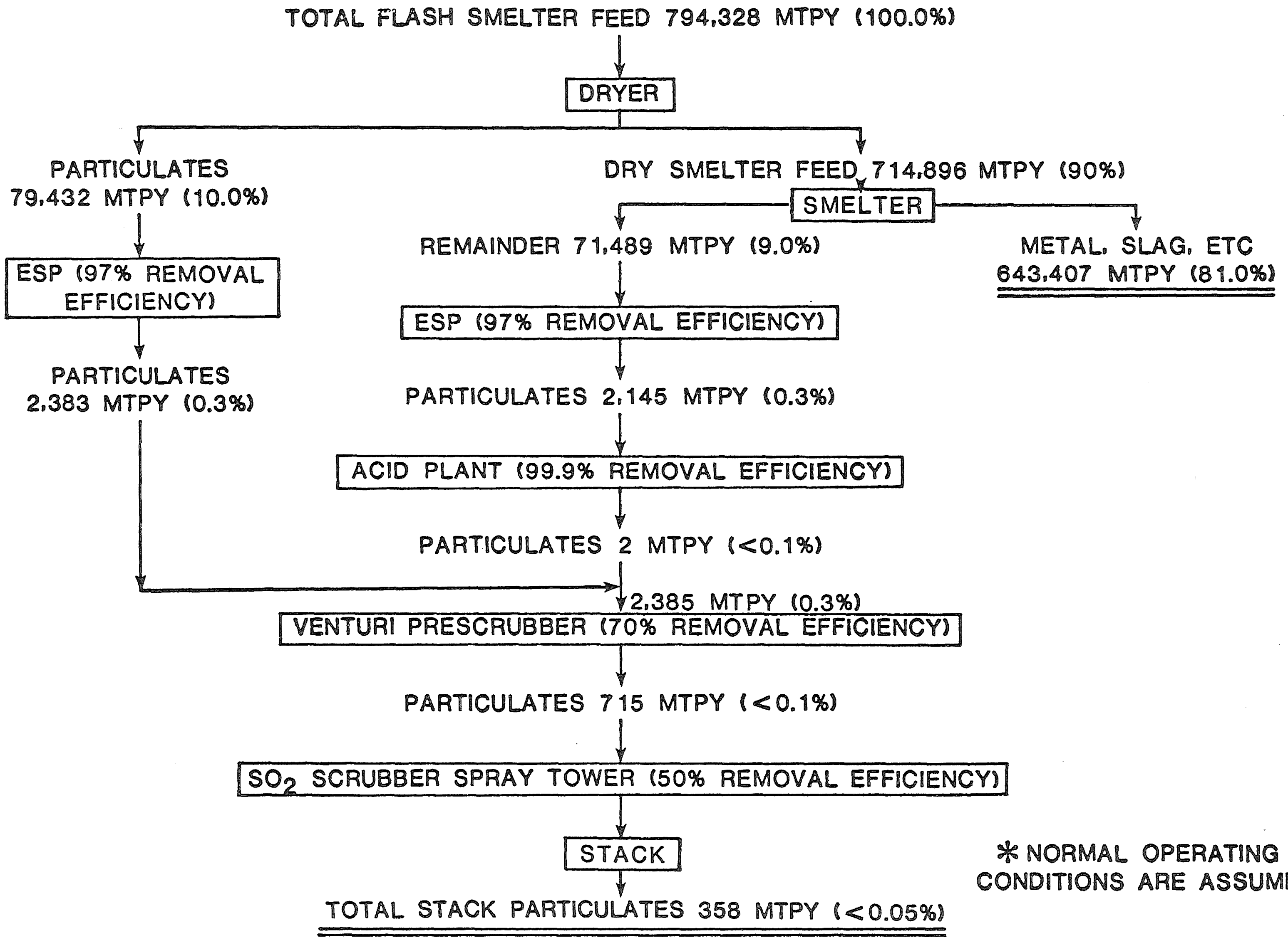
<u>OPTION 2</u> MODEL SULFUR BALANCE FOR FLASH FURNACE WITH ACID PLANT CONTROL OF STRONG SO_2 GAS TO 300 PPM SO_2, SECONDARY HOODING COLLECTION OF WEAK SO_2 GAS, AND SCRUBBING OF ACID PLANT TAIL GAS PLUS COLLECTED WEAK SO_2 GAS TO 143 PPM SO_2*



TOTAL EMISSIONS, 996 MTPY S (0.6%)

* NORMAL OPERATING CONDITIONS ARE ASSUMED

FIGURE 23
MODEL FOR STACK EMISSIONS
PARTICULATE BALANCE FOR OPTIONS 1 AND 2*



* NORMAL OPERATING CONDITIONS ARE ASSUMED

Table 24. Model sulfur balance for option 2 flash smelter with acid plant control of strong SO_2 gases, secondary hooding of weak SO_2 gases, scrubbing of acid plant tail gas, and hooded weak SO_2 gases to 143 ppm SO_2.[a]

PRODUCT	SO_2 REMOVAL EFFICIENCY, %	GAS STREAM ppm SO_2[b]	SULFUR PRODUCTION mtpy	SULFUR PRODUCTION % DISTRIBUTION
Smelter feed			165,542	100.0
Slag product			3,804	2.3
Metal products			5,115	3.1
Weak SO_2 gas to secondary hooding			4,960	3.0
Secondary hooding[c]	(90)			
Weak SO_2 gas to scrubber		2,672	4,465	2.7
Weak SO_2 gas as fugitives			495	0.3
Strong SO_2 gas to acid plant		7.73%	151,663	91.6
Acid plant	99.64			
Sulfuric acid and weak acid blowdown			151,120	91.3
Acid plant tail gas to scrubber		300	543	0.3
Scrubber	90			
Scrubber feed		1,435	5,008	3.0
Sludge & clarifier overflow			4,507	2.7
Scrubber stack emission		143	501	0.3
Total Emissions (stack & fugitives)			996	0.6

[a]Normal operating conditions are assumed.
[b]PPM SO_2 unless otherwise noted.
[c]90% collection efficiency.

ciency of 99.64% during steady state operation, leaving 300 ppm by volume SO_2 in the acid plant tail gases. All weak SO_2 gases from the flash furnace, copper converters, slag cleaning furnaces, and nickel converters are cycled through electrostatic precipitators (97% efficiency in particulate matter removal), then combined with the acid plant tail gases, and finally cycled through a venturi pre-scrubber (70% efficiency in particulate matter removal) and a TCA tail gas scrubber (50% efficiency in particulate matter removal and a sulfur conversion efficiency of 90% during steady state operation) before release to the atmosphere. Stack gas emissions of 143 ppm SO_2 reflect the application of state-of-the-art control technology. Scrubber slurry from the TCA will be treated by lime precipitation along with weak acid blowdown from the acid plant. The resultant sludge material will be either recycled back to the smelter for use as flux material or disposed of as solid waste.

5.4 ADDITIONAL DEVELOPMENT MODEL VARIATIONS

This section briefly describes some of the additional possible variations in the basic models described thus far, and the potential effects of such variations. The intent is to allow the reader to adjust the general models given here by simple multipliers, additions or subtractions, and thus obtain a reasonable picture of other, more specific situations.

The described variations are not generally dealt with in the model descriptions of this chapter. However, these variations can be evaluated and adjustments made to the basic models, resulting in a multitude of possible development options.

5.4.1 Relationship Between Capital and Operating Costs and Hoisting Depth in Underground Mining

The basic underground mining costs of development and ore production as discussed in Chapter 2 are independent of mining depth; however, the cost of shaft sinking, hoisting equipment, and ventilation equipment, and operation of these facilities, do increase with depth.

Figures 24 and 25 show the approximate relationship between capital and operating costs, and shaft depth, for both production and service shaft installations. The information used to generate the curves was obtained from industry and consultant data. The production shaft is a 24 ft diameter, concrete lined shaft, with either a single or a double hoisting facility. The service shaft is scaled to the production facility.

Figures 24 & 25

The model underground mining shaft facilities generally fit the curves shown, as does shaft cost data supplied by industry for Minnesota operations. Operating costs are constant on a per foot basis, totalling 0.011¢/ft of hoisting depth metric ton of ore and varying between 11¢ and 33¢/mt of ore hoisted when increasing the depth from 1,000 to 3,000 ft for a given operation.

Capital costs vary with depth according to the straight line shown in Figure 24, in the depth range applicable to Minnesota copper-nickel. Increasing the hoisting depth from 1,000 to 3,000 ft increases the capital cost by 50% to 60% for each shaft facility graphed.

5.4.2 Variation in Waste Rock-Lean Ore Storage Pile Design

Permanent and temporary materials storage are major considerations in any mining operation. From the mine itself, the major materials requiring such con-

FIGURE 24

RELATIONSHIP BETWEEN CAPITAL COST AND HOISTING DEPTH PRODUCTION & SERVICE SHAFTS

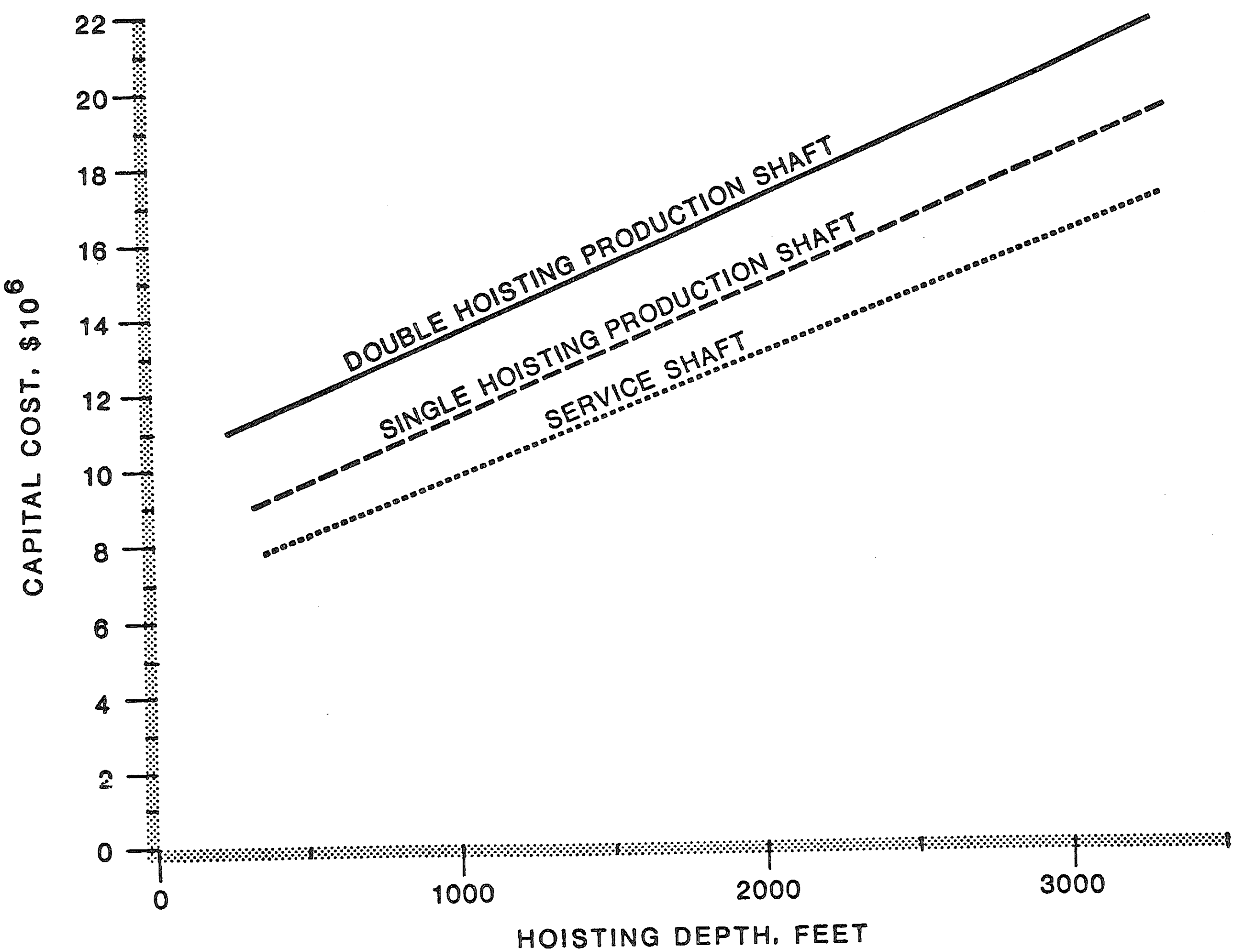
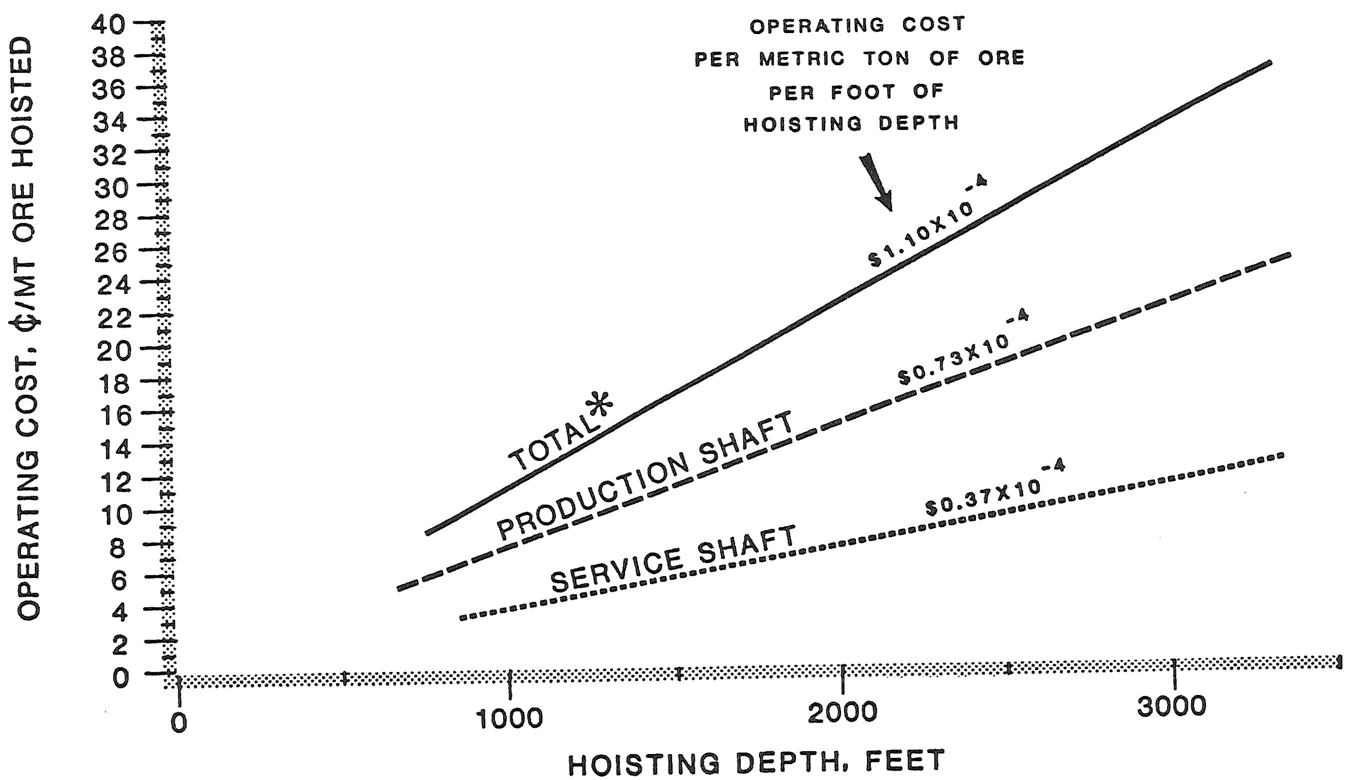


FIGURE 25

RELATIONSHIP BETWEEN OPERATING COST AND HOISTING DEPTH PRODUCTION & SERVICE SHAFTS



*CHARGES SERVICE SHAFT COSTS TO ORE PRODUCTION sideration are waste rock and lean ore, with minor consideration given to overburden since the quantity is generally small and storage is temporary. Presumably, overburden would be used to reclaim disturbed areas.

The base case mine models include multiple waste rock-lean ore storage piles containing up to 50 X 10^6 mt of material per pile. In plan view, the piles are square, with a height of 61 m (200 ft) and a bank slope of 2.5 to 1. Table 25 lists the storage pile data as designed for each model size, and indicates an overall storage capacity of up to 0.8 X 10^6 mt of material per hectare of land area. Table 26 summarizes the same type of data for the 20 X 10^6 mtpy open pit operation only. Area requirements are listed both for 13 piles of 50 X 10^6 mt each and for a single pile of 650 X 10^6 mt, for pile heights of 30.5, 61.0, and 121.9 m (100, 200, and 400 ft), respectively. The same data is illustrated in Figure 26.

Tables 25 & 26, Figure 26

The advantages of multiple pile storage over single pile storage include more flexibility in pile location and in scheduling delivery to each pile. The disadvantages are the need for 11 to 12% more total area for storage and probably the need for more equipment and manpower to work 13 piles rather than only one pile.

Increasing pile height has a marked advantage in area requirements (Figure 26); however, equipment needs and manpower to support the equipment will increase in order to elevate the material to the new pile height. Additionally, the visual impact of increased pile height may be a disadvantage.

Table 25. Mine model waste rock-lean ore storage piles.

Base Case Data: Square piles, 60.96 m high, maximum of 50 X 10^6 mt per pile, bank slope 2.5:1.

	MODEL CAPACITY, 10^6 mtpy ORE				
	5.35	11.33	12.35	16.68[a]	20.00
Number of piles	1	8	1	9	13
Total waste rock-lean ore, 10^6 mt	12	368	28	380	650
Base area of piles, ha[b]	21	453	39	474	805
Storage capacity, 10^6 mt/ha	0.6	0.8	0.7	0.8	0.8

[a]Assumed sum of 5.35 and 11.33 X 10^6 mtpy operations.
[b]1 ha = 2.471 acres.

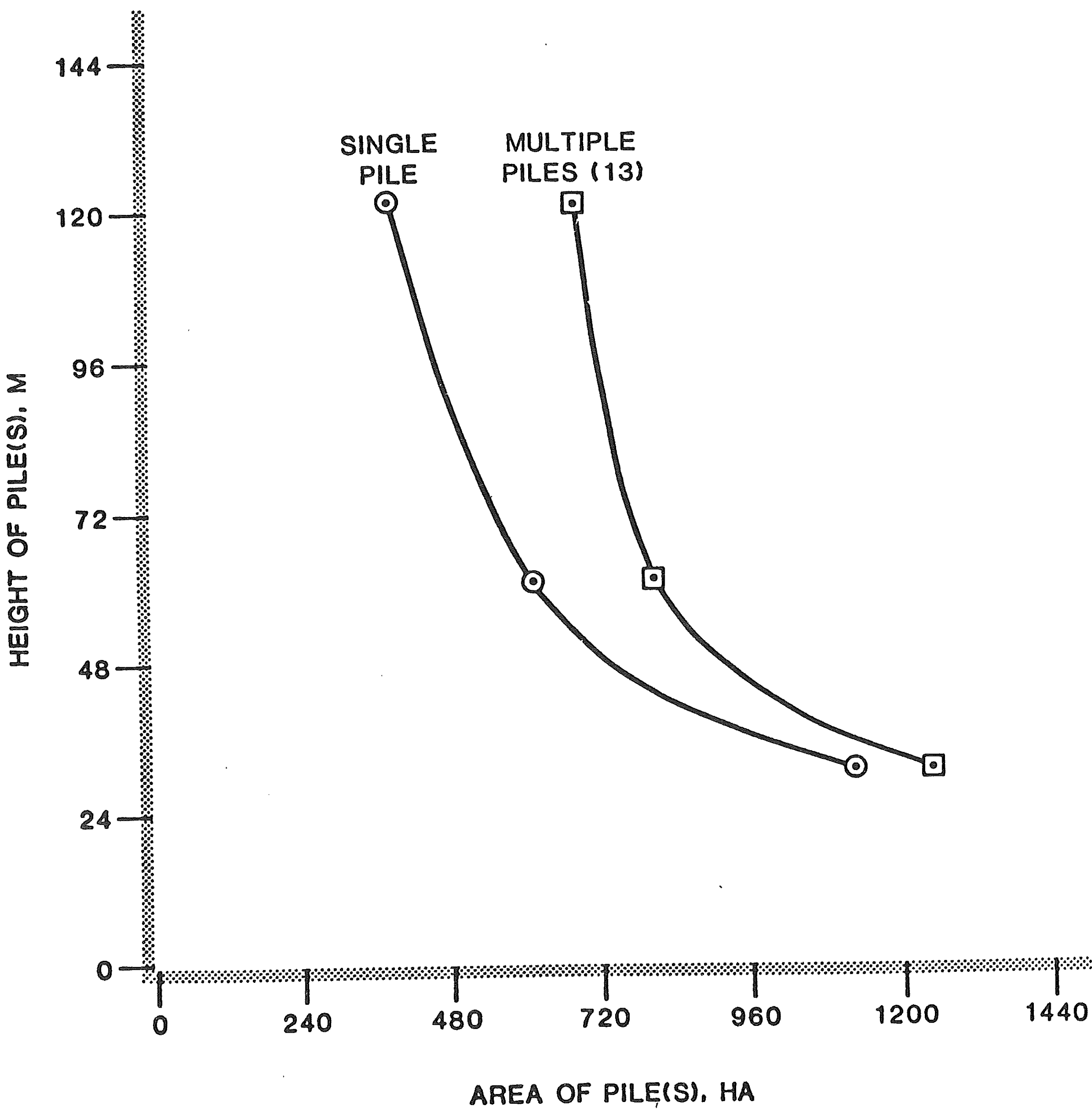
Table 26. Variation in waste rock-lean ore storage pile configuration with number of piles and pile height. Example: 20 X 10^6 mtpy ore operation, Bank slope 2.5:1

COMPARISON	REQUIREMENTS: SQUARE PILES, Single Pile	REQUIREMENTS: SQUARE PILES, Multiple Unit Piles	REQUIREMENTS: ROUND PILES, Single Pile	REQUIREMENTS: ROUND PILES, Multiple Unit Piles
Number of piles	1	13	1	13
10^6 mt per pile	650	50	650	50
Base area, ha				
30.48 m height	1,117	1,251	1,111	1,229
60.95 m height	605	805	597	772
121.92 m height	387	678	359	629

FIGURE 26

COMPARISON OF AREA REQUIREMENTS FOR STORAGE OF WASTE ROCK-LEAN ORE 20 X 10^6 MTPY OPERATION

SINGLE PILE VS 13 PILES AT VARIOUS HEIGHTS



5.4.3 Stripping Ratio Effect on Area Requirements for Waste Rock-Lean Ore Storage Piles

As used in this report, the stripping ratio is the ratio of metric tons of waste rock-lean ore which must be removed in order to remove one metric ton of ore material. The base case open pit mine models have a stripping ratio of 1.3:1. Thus, over the life of the 11.33 X 10^6 mtpy ore model, 283 X 10^6 mt of ore will be recovered and 368 X 10^6 mt of waste rock-lean ore material must be removed and stored. For the 20.00 X 10^6 mtpy ore model, 500 X 10^6 mt of ore will be recovered, and 650 X 10^6 of material must be stored.

Base case unit storage piles contain 50 X 10^6 mt of material each at a height of 61.0 m and covering an area of 61.9 ha. Therefore, a total of 8 piles are needed for the small open pit and 13 piles for the large open pit, to store all the waste rock-lean ore removed to expose the ore.

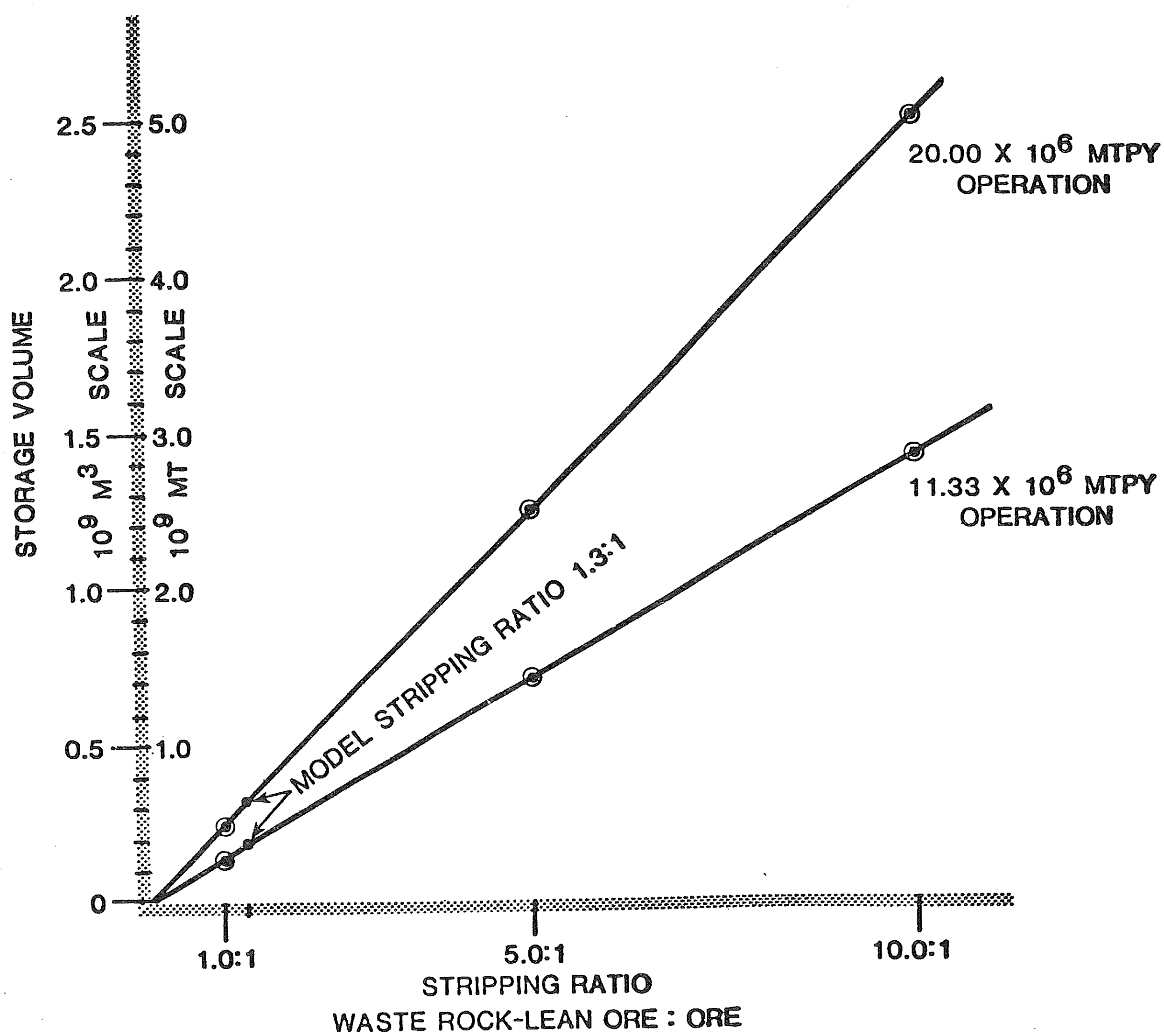
If the stripping ratio varies, the amount of material to be stored varies accordingly, as shown in Figure 27. For the 20 X 10^6 mtpy example with a stripping ratio of 1.3:1, the required storage volume is 325 X 10^6 m^3 (650 X 10^6 mt). At a 5:1 stripping ratio, the volume increases to 1,250 m^3, and to 25,000 X 10^6 m^3 at a 10:1 stripping ratio. Corresponding quantities of material divided by the unit pile storage capacity of 50 X 10^6 mt per pile results in 13, 50, and 100 storage piles with total areas of 805, 3,100, and 6,200 ha, respectively.

Figure 27

The same approach for the 11.33 X 10^6 mtpy ore operation results in similar changes in storage area requirements. Table 27 summarizes the results for both

FIGURE 27

COMPARISON OF WASTE ROCK-LEAN ORE STORAGE VOLUME REQUIRED WITH VARYING STRIPPING RATIOS*
(ASSUME 25 YEARS OF PRODUCTION)



*IN PLACE ROCK DENSITY OF 2.0 MT/M^3

open pit operations.

Table 27

Another variable to be considered in stockpile arrangement is storage in multiple piles of a given height and size (tonnage), versus storage in a single pile of a specified height, sized to accommodate all of the waste material. Such a comparison is shown in Figure 28 for the 20 X 10^6 mtpy operation, which indicates that the area required to store a given amount of material decreases more rapidly and to a much lower total for the single pile relative to the multiple pile approach. The base case height of 61.0 m (200 ft) shows 805 ha required for the 13 pile arrangement, but only 605 ha (25% reduction) would be needed for a single pile storage. This advantage approaches a 45% reduction when the pile height reaches 122 m (400 ft). The variation for the 11.33 X 10^6 mtpy operation is analogous.

Figure 28

5.4.4 Variation in Tailing Basin Configuration With Dam Height

The approach to tailing basin design, construction, and eventually operation, are important both from the environmental point of view and in terms of total costs for the facility. The variation in basin configuration can dramatically affect the cost of construction, just as distance from the basin to the processing facility affects the cost of moving tailing to the basin and of recycling clarified water to the processing plant (see sections 5.4.5 and 5.4.6).

Table 27. Variation in storage pile area requirements for 20.00 and 11.33 X 10^6 mtpy open pit operations with stripping ratio, square pile design, 61 m height.

OPERATION SIZE, 10^6 mtpy	STRIPPING RATIO	NUMBER OF STORAGE PILES	TOTAL AREA OF STORAGE PILES, ha
20.00	1.3:1	13	805
	5:1	50	3,100
	10:1	100	6,200
11.33	1.3:1	8	456
	5:1	29	1,753
	10:1	57	3,507

FIGURE 28

AREA COVERED BY WASTE ROCK PILES

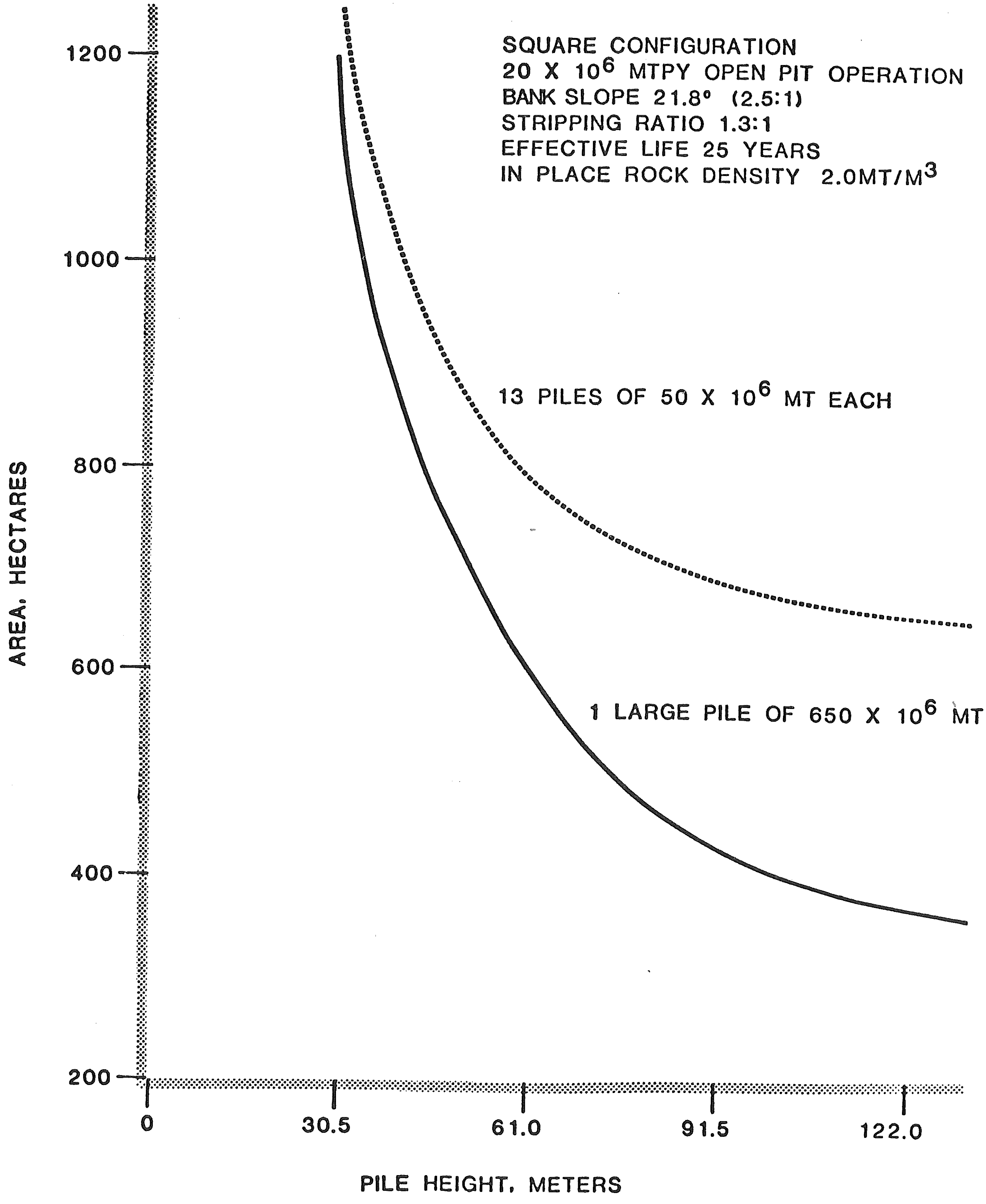


Table 28 illustrates the variation in physical parameters for the 20.00 X 10^6 mtpy open pit model tailing basin. The model dam height is 75 ft, with 70 ft of retained tailing behind the dam. Variations are shown for a range of dam heights from 55 to 155 ft, or an effective tailing storage height from 50 to 150 ft.

Table 28

The 2 most important construction factors (illustrated by the comparison in Table 29) are the volume of the dam itself and the area occupied by the total basin. In ranging from the lowest to highest dam height considered, the dam volume varies from 3% to 13% of the total tailing basin volume. The total area occupied by the basin decreases by a factor of 2.8 over the same range. These points are further illustrated in Figure 29, where the dam volume and total basin area are plotted against dam height.

Table 29, Figure 29

The importance of the dam volume is primarily economics even though the cost per cubic meter to construct the dam is essentially a constant which is applied to the variable of dam volume. As a result of volume changes alone, this cost changes by a factor of 4 in going from the 55 ft to the 155 ft dam height.

In contrast, the basin area requirements decrease by a factor of 2.8 over the same increase in dam height, requiring the purchase and control of less land, and less area to be concerned about from the standpoint of dust lift-off and water seepage control.

Table 28. Variation in tailing basin characteristics with dam height (20×10^6 mtpy ore model, circular basin).

	DAM HEIGHT (ft)			
	55	75[a]	105	155
Actual Area, acres				
Total	5,555	4,010	2,865	1,987
Dam	300	337	386	455
Retained tailing	5,255	3,673	2,479	1,532
Volume, 10^9 ft^3				
Total basin	11.90	11.90	11.90	11.90
Dam	0.39	0.58	0.91	1.56
Retained tailing	11.51	11.32	10.99	10.34
Radius Values, ft				
r_2-outside of dam base	8,776	7,457	6,303	5,248
r_1-inside of dam base	8,536	7,137	5,863	4,608
Dam Base Width, ft	240	320	440	640

[a]Model value 4016 acres, 7,462 ft radius to outside of dam.

Table 29. Approximate correction factors for adjusting tailing basin configuration with a change in tailing dam height.[a]

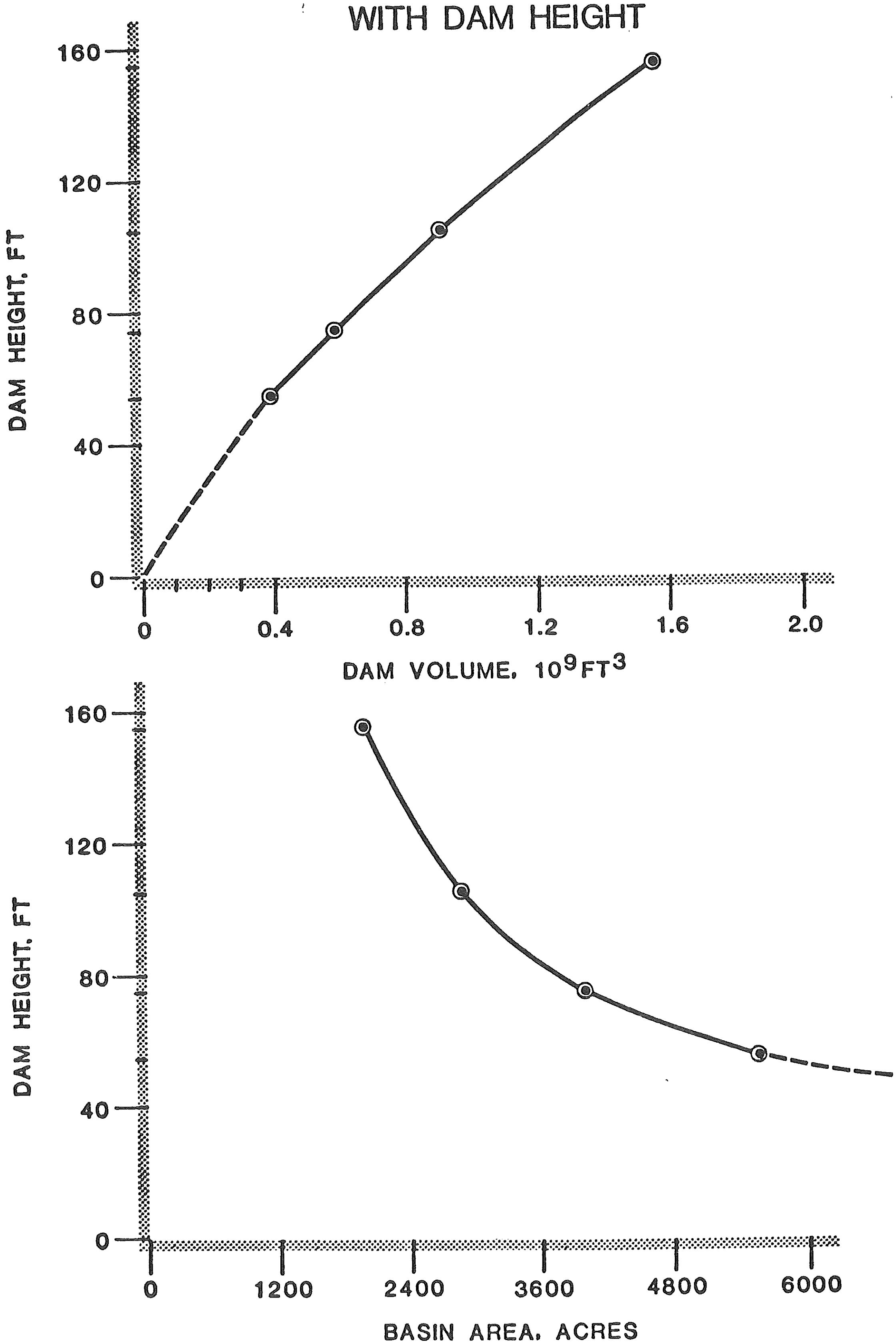
	DAM HEIGHT (ft)			
	55	75[b]	105	155
Area Occupied by Structure, acres				
Dam	0.9	1.0	1.2	1.4
Retained tailing	1.4	1.0	0.7	0.4
Total	1.4	1.0	0.7	0.5
Volume of Structure, ft^3				
Dam	0.7	1.0	1.6	2.7
Retained tailing	1.0	1.0	1.0	0.9
Total	1.0	1.0	1.0	1.0
Radius of Structure, ft				
Outside toe of dam	1.2	1.0	0.9	0.7
Inside toe of dam	1.2	1.0	0.8	0.7
Dam Base Width, ft	0.8	1.0	1.4	2.0

[a]Based on calculations for the 20.00 X 10^6 mtpy open pit model tailing basin with a 75 ft dam height and a 70 ft basin height.

[b]Model dam height.

FIGURE 29

RELATIONSHIP OF DAM VOLUME AND TOTAL BASIN AREA WITH DAM HEIGHT



Tables 29 and 30 were developed to illustrate the relationships discussed above, which can be applied to all models with the dam and basin configuration shown in Figure 30. This configuration was used for all basin considerations. Finally, Tables 31 through 35 list the basic basin criteria for each model, for each dam height. Table 35 is a repeat of Table 28, included for completeness in the comparison. All model tailing basins in Tables 31 through 35 are adjusted to the acreage listed in the model discussions of the previous section.

Tables 30-35, Figure 30

5.4.5 Tailing Dam Capital Cost Variation With Overhaul Distance

The 20.00 X 10^6 mtpy ore model is used to illustrate the changes in capital cost for the tailing dam when the overhaul distance is increased over that used in the models (2 mi). The data listed in Table 36 indicates that the starter dam and tailing sand costs remain independent of distance from the processing facility; however, the waste rock drain capital cost increases by \$0.50/yd^3-mi as the tailing basin is moved away from the processing plant. For example, the overall tailing dam cost then increases according to the table, by 68% with 4 mi of additional overhaul, and by 168% with 10 mi of additional overhaul. These costs must be combined with the tailing material and reclaim water cost increases with distance to obtain the overall cost increase picture.

Table 36

5.4.6 Tailing Material and Reclaim Water Transportation Capital and Operating Cost Variation With Pumping Distance

Table 30. Percent distribution of area and volume of tailing basin components with a change in dam height.

	DAM HEIGHT (ft)			
	55	75	105	155
Area Occupied by Structure, %				
Dam	5.4	8.4	13.5	22.9
Retained tailing	94.6	91.6	86.5	77.1
Total	100.0	100.0	100.0	100.0
Volume of Structure, %				
Dam	3.3	4.9	7.7	13.1
Retained tailing	96.7	95.1	92.3	86.9
Total	100.0	100.0	100.0	100.0

Table 31. Tailing basin variation with dam height.
Model Description: 5.35×10^6 mtpy underground mine model.

	DAM HEIGHT (ft)			
	55	75	105	155
Area Requirements, acres				
Dam	80	90	103	121
Retained Tailing	1,398	977	659	408
Total	1,478	1,067	762	529
Volume of Structure, 10^9 ft^3				
Dam	0.1	0.1	0.2	0.4
Retained Tailing	2.8	2.8	2.7	2.5
Total	2.9	2.9	2.9	2.9
Radius of Structure, ft				
Outside toe of dam	4,527	3,864	3,250	2,708
Inside toe of dam	4,287	3,526	2,810	2,068
Dam Base Width, ft	240	320	440	640

Table 32. Tailing basin variation with dam height.
Model Description: 11.33×10^6 mtpy open pit mine model.

	DAM HEIGHT (ft)			
	55	75	105	155
Area Requirements, acres				
Dam	176	197	226	266
Retained Tailing	3,076	2,151	1,451	897
Total	3,252	2,348	1,677	1,163
Volume of Structure, 10^9 ft^3				
Dam	0.2	0.3	0.5	0.9
Retained Tailing	6.5	6.4	6.2	5.8
Total	6.7	6.7	6.7	6.7
Radius of Structure, ft				
Outside toe of dam	6,715	5,706	4,82	4,016
Inside toe of dam	6,475	5,386	4,382	3,376
Dam Base Width, ft	240	320	440	640

Table 33. Tailing basin variation with dam height.
Model Description: 12.35 X 10^6 mtpy underground mine model.

	DAM HEIGHT (ft)			
	55	75	105	155
Area Requirements, acres				
Dam	173	194	222	262
Retained Tailing	3,025	2,115	1,428	882
Total	3,198	2,309	1,650	1,144
Volume of Structure, 10^9 ft^3				
Dam	0.2	0.3	0.5	0.9
Retained Tailing	6.4	6.3	6.1	5.7
Total	6.6	6.6	6.6	6.6
Radius of Structure, ft				
Outside toe of dam	6,659	5,658	4,783	3,983
Inside toe of dam	6,419	5,338	4,343	3,343
Dam Base Width, ft	240	320	440	640

Table 34. Tailing basin variation with dam height.
Model Description: 16.68 X 10^6 mtpy open pit mine model.

	DAM HEIGHT (ft)			
	55	75	105	155
Area Requirements, acres				
Dam	245	275	315	372
Retained Tailing	4,297	3,004	2,027	1,253
Total	4,542	3,279	2,342	1,625
Volume of Structure, 10^9 ft^3				
Dam	0.3	0.5	0.7	1.3
Retained Tailing	9.3	9.1	8.9	8.3
Total	9.6	9.6	9.6	9.6
Radius of Structure, ft				
Outside toe of dam	7,936	6,743	5,699	4,747
Inside toe of dam	7,696	6,423	5,259	4,107
Dam Base Width, ft	240	320	440	640

Table 35. Tailing basin variation with dam height.
Model Description: 20.00 X 10^6 mtpy open pit mine model.

	DAM HEIGHT (ft)			
	55	75	105	155
Area Requirements, acres				
Dam	300	337	386	456
Retained Tailing	5,264	3,679	2,483	1,534
Total	5,563	4,016	2,869	1,990
Volume of Structure, 10^9 ft^3				
Dam	0.4	0.6	0.9	1.6
Retained Tailing	11.5	11.3	11.0	10.3
Total	11.9	11.9	11.9	11.9
Radius of Structure, ft				
Outside toe of dam	8,782	7,462	6,307	5,252
Inside toe of dam	8,542	7,142	5,867	4,612
Dam Base Width, ft	240	320	440	640

FIGURE 30

MODEL BASIN AND DAM CONFIGURATION

CROSS SECTION

(DISTANCE VALUES IN FEET)

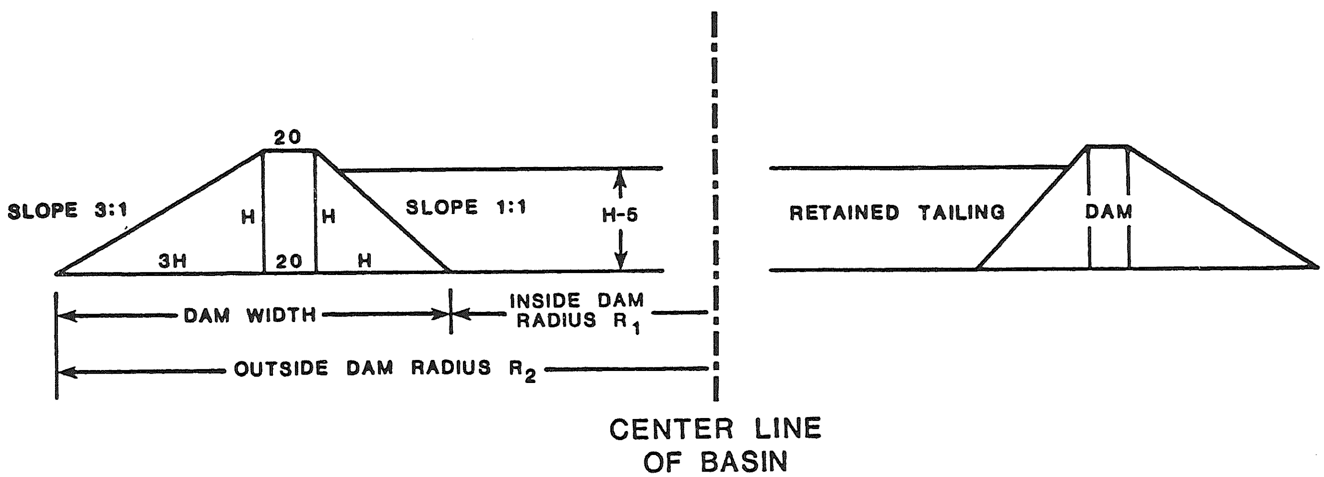


Table 36. Tailing dam capital cost variation with overhaul distance.

ITEM	UNIT COST $/yd^3$	VOLUME $10^6 yd^3$	ADDITIONAL[a] OVERHAUL DISTANCE (mi)	ITEM TOTAL 10^6 $	GRAND TOTAL 10^6 $
Starter dam	2.60	0.9	--	2.4	--
Tailing sand	0.20	18.8	--	3.8	--
Waste rock drain	0.50	2.5	0	1.2	7.4
	1.00	2.5	1	2.5	8.7
	1.50	2.5	2	3.7	9.9
	2.50	2.5	4	6.2	12.4
	3.50	2.5	6	8.7	14.9
	5.50	2.5	10	13.6	19.8

[a]Overhaul is the additional distance waste rock must be hauled from the mine beyond the 2 mi assumed in the basic model.

In addition to the increased capital cost for dam construction due to additional overhaul of waste rock from the mine for basin drain material, increasing the distance between the processing plant and the tailing basin has a marked effect on the capital and operating costs of the tailing and recycle water transportation systems. Using the information developed by Golder and Associates and the system design outlined and discussed in Chapter 3, Table 37 and Figure 31 were generated for the 12.35, 16.68, and 20.00 X 10^6 mtpy model operations with distances between processing plant and tailing basin of 1,000 ft, one mile, 5 mi, and 10 mi. As discussed in Chapter 3, tailing pumped 1,000 ft and one mile are transported unthickened, and material pumped 5 mi and 10 mi is thickened in the plant before transporting. This method was determined to be most economical, as is reflected in the cost data of Table 37.

Table 37, Figure 31

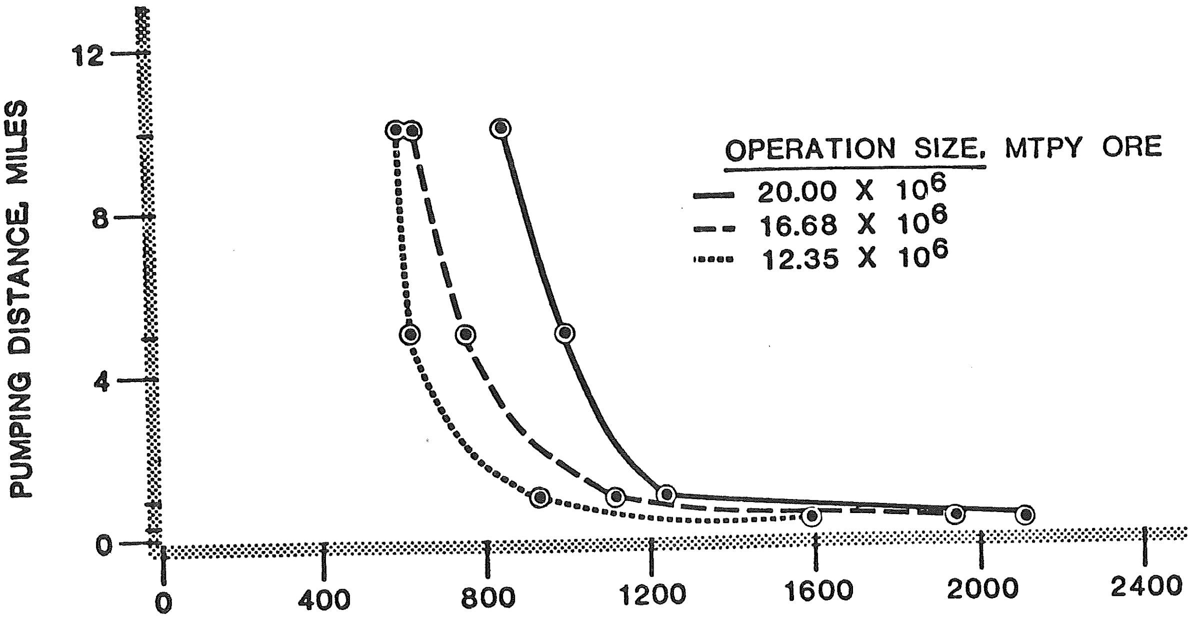
The capital costs for transportation of tailing and recycle water vary considerably depending on the size of the operation and the pumping distance, as do the associated operating costs. The data indicates a considerable reduction in the cost per foot of tailing line with increased distance (capital cost decreases by a factor of 2.5 to 3.0 and operating cost decreases by a factor of 6.7 to 8.1 when increasing the tailing line from 1,000 ft to 10 mi). However, the total capital and annual operating costs increase tremendously with increasing pumping distance (factors of 20 and 7, respectively, over the same range). These cost changes must be combined with the overhaul distance cost adjustments for the full cost picture.

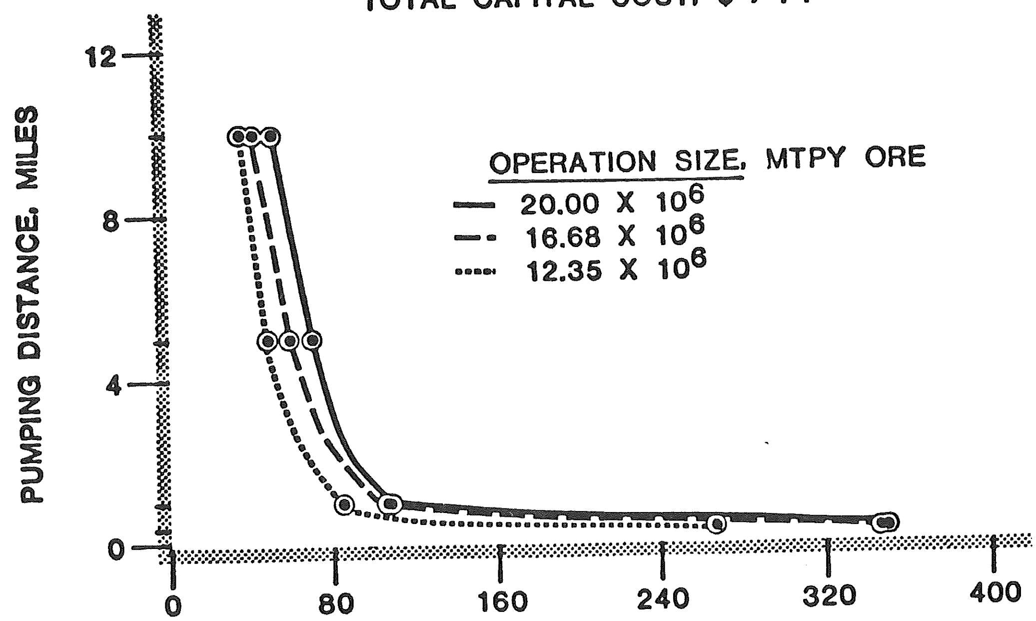
Table 37. Tailing material and water reclamation capital and operating cost variation with pumping distance.

	OPERATION SIZE, 10^6 mtpy ore					
	12.35		16.68		20.00	
	Total $\$10^6$	Unit \$/ft	Total $\$10^6$	Unit \$/ft	Total $\$10^6$	Unit \$/ft
Total Capital Cost						
Line length - 1000 ft	1.59	1593	1.95	1947	2.11	2111
1 mi	4.94	935	5.89	1115	6.53	1239
5 mi	16.52	626	19.94	755	26.32	997
10 mi	31.56	598	33.54	635	44.93	851
Annual Operating Cost						
Line length - 1000 ft	0.27	266	0.35	350	0.35	347
1 mi	0.45	85	0.56	106	0.56	107
5 mi	1.30	49	1.56	59	1.87	71
10 mi	1.92	36	2.29	43	2.76	52
Annual 10^6 mt tailing	11.71		16.04		19.36	

FIGURE 31

CAPITAL AND OPERATING COST VARIATION OF TAILING AND WATER TRANSPORTATION SYSTEMS WITH PUMPING DISTANCE





5.5 REFERENCES

All applicable reference material is listed at the end of each preceeding chapter of this volume.