Volume 3-Chapter 2

MINERAL RESOURCE POTENTIAL

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

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

1) Where reasonable, an effort has been made to use the metric system (meters, kilograms, kilowatt-hours, etc.) of units which is widely used in the physical and biological sciences, and is slowly becoming accepted in the United States.

2) In several areas, notably engineering discussions, the use of many English units (feet, pounds, BTU's, etc.) is retained in the belief that this will better serve most readers.

3) Notable among the units used to promote the metric system is the metric ton, which consists of 2205 pounds and is abbreviated as mt. The metric ton (1000 kilograms) is roughly 10% larger (10.25%) than the common or short ton (st) of 2000 pounds. The metric ton is quite comparable to the long ton (2240 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 2½ acres (actually 2.4710 acres). Thus, one square mile, which consists of 640 acres, contains some 259 hectares.

The attached 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 (m)	= 3.28 feet = 1.094 yards
1 centimeter (cm)	= 0.3937 inches
1 kilometer (km)	= 0.621 miles
1 hectare (ha)	= 10,000 sq. meters = 2.471 acres
1 square meter (m^2)	= 10.764 sq. feet = 1.196 sq. yards
1 square kilometer (km^2)	= 100 hectares = 0.386 sq. miles
1 gram (g)	= 0.037 oz. (avoir.) = 0.0322 Troy oz.
1 kilogram (kg)	= 2.205 pounds
1 metric ton (mt)	= 1,000 kilograms = 0.984 long tons = 1.1025 short tons
1 cubic meter (m^3)	= 1.308 yd^3 = 35.315 ft^3
1 liter (l)	= 0.264 U.S. gallons
1 liter/minute (l/min)	= 0.264 U.S. gallons/minute = 0.00117 acre-feet/day
1 kilometer/hour (km/hr)	= 0.621 miles/hour
degrees Celsius (°C)	= (5/9)(degrees Fahrenheit -32)

Standard abbreviations.

ha	- hectare	ppm	- parts per million
st	- short ton of 2000 lb	ppb	- parts per billion
lt	- long ton of 2240 lb	um	- micron or 10^{-6} meters
mt	- metric ton of 2205 lb	%	- percent by weight unless
mtpy	- metric ton(s) per year		otherwise noted

ELEMENT	SYMBOL	ELEMENT	SYMBOL	ELEMENT	SYMBOL
Actinium	Ac	Holmium	Ho	Rhenium	Re
Aluminum	Al	Hydrogen	H	Rhodium	Rh
Americium	Am	Indium	In	Rubidium	Rb
Antimony	Sb	Iodine	I	Ruthenium	Ru
Argon	Ar	Iridium	Ir	Samarium	Sm
Arsenic	As	Iron	Fe	Scandium	Sc
Astatine	At	Krypton	Kr	Selenium	Se
Barium	Ba	Lanthanum	La	Silicon	Si
Berkelium	Ba	Lawrencium	Lw	Silver	Ag
Beryllium	Be	Lead	Pb	Sodium	Na
Bismuth	Bi	Lithium	Li	Strontium	Sr
Boron	B	Lutetium	Lu	Sulfur	S
Bromine	Br	Magnesium	Mg	Tantalum	Ta
Cadmium	Cd	Manganese	Mn	Technetium	Tc
Calcium	Ca	Mendelevium	Md	Tellurium	Te
Californium	Cf	Mercury	Hg	Terbium	Tb
Carbon	C	Molybdenum	Mo	Thallium	Tl
Cerium	Ce	Neodymium	Nd	Thorium	Th
Cesium	Cs	Neon	Ne	Thulium	Tm
Chlorine	Cl	Neptunium	Np	Tin	Sn
Chromium	Cr	Nickel	Ni	Titanium	Ti
Cobalt	Co	Niobium	Nb	Tungsten	W
Copper	Cu	Nitrogen	N	Uranium	U
Curium	Cm	Nobelium	No	Vanadium	V
Dysprosium	Dy	Osmium	Os	Xenon	Xe
Einsteinium	Es	Oxygen	O	Ytterbium	Yb
Erbium	Er	Palladium	Pd	Yttrium	Y
Eutopium	Eu	Phosphorus	P	Zinc	Zn
Fermium	Fm	Platinum	Pt	Zirconium	Zr
Fluorine	F	Plutonium	Pu		
Francium	Fr	Polonium	Po		
Gadolinium	Gd	Potassium	K		
Gallium	Ga	Praseodymium	Pr		
Germanium	Ge	Promethium	Pm		
Gold	Au	Protactinium	Pa		
Hafnium	Hf	Radium	Ra		
Helium	He	Radon	Rn		

Volume 3-Chapter 2 <u>MINERAL RESOURCE POTENTIAL</u>

2.1 INTRODUCTION AND SUMMARY OF FINDINGS

The previous chapter characterized in some detail the mineral, chemical, and

elemental composition of the Duluth Complex rocks likely to be directly involved

in, or disturbed by the development of a copper-nickel mining industry in the

region. This chapter focuses on the quantities and composition of material

likely to be affected by such activities. Then a detailed discussion follows of

the various products likely to result from mining. These products include not

only the marketable metals and other valuable by-products, but the waste rock and

tailing materials produced as well. The concentrate, as an intermediate product,

will also be discussed. The results of this chapter are summarized in the

introduction to Chapter 1 (section 1.1.2).

2.1.1 <u>Resource Areas</u>

As part of the Regional Copper-Nickel Study, the MDNR conducted an investigation

to define and estimate the mineral resource potential of the Study Area (Listerud

and Meineke 1977). The data base consisted of assays from about 500 drill cores,

mostly within 2 miles of the outcropping or suboutcropping basal contact of the

Duluth Complex. The types of data available for these holes varied

significantly. Some of the holes did not meet designated minimum data level

requirements and could not be used in the resource estimate. In order to qualify

for use, the hole had to contained mineralization that met one or more of the

following 3 specifications: Type 1 is a minimum vertical thickness of 50 ft

assaying greater than 0.5% Cu; Type 2 is a minimum vertical thickness of 100 ft

assaying greater than 0.25% Cu in the top 100 ft of the core or core less than

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100 ft in length if the base of the Complex was reached by drilling less than 100 ft and the core was mineralized throughout; Type 3 is a minimum vertical thickness of 50 ft containing greater than 10% TiO_2. Holes that do not indicate any of these types of mineralization qualify for use in the estimate if they were drilled all the way to the footwall. Three hundred twenty-four holes were used in the resource estimate because they met the criteria outlined above, and the locations of these drill holes are shown on Figure 1.

Figure 1

A standard method, described in Parks (1949), of using perpendicular bisectors for polygon contruction was used for the MDNR estimate (Listerud and Meineke 1977). The average polygon was 33.7 ha (0.13 mi^2). The total area measured in the MDNR study was 10,930 ha (42.2 mi^2) or 38.2% of the 70,790 ha (273.3 mi^2) in the 6-mi wide band of resource development zones (Figure 2) and 50.9% of the 53,048 ha (204.8 mi^2) of the Duluth Complex in the development zones.

Figure 2

The development zones mentioned above were derived from a smaller area termed the resource zones (Figure 2). These zones form a 3-mi wide strip bounded on one side by the Duluth Contact, and laying to the south and east of the Contact. The zones contain the mineralized polygons (see Figure 17 given in Chapter 1), and represent an area that is likely to contain all actual mining which takes place within roughly 3,000 to 4,000 ft of the surface. Based on available information, no copper-nickel mines developed in the Study Area are expected to fall outside of these resource zones. To facilitate discussion, 7 resource zones were rather arbitrarily defined along the Contact. The band was divided along

2
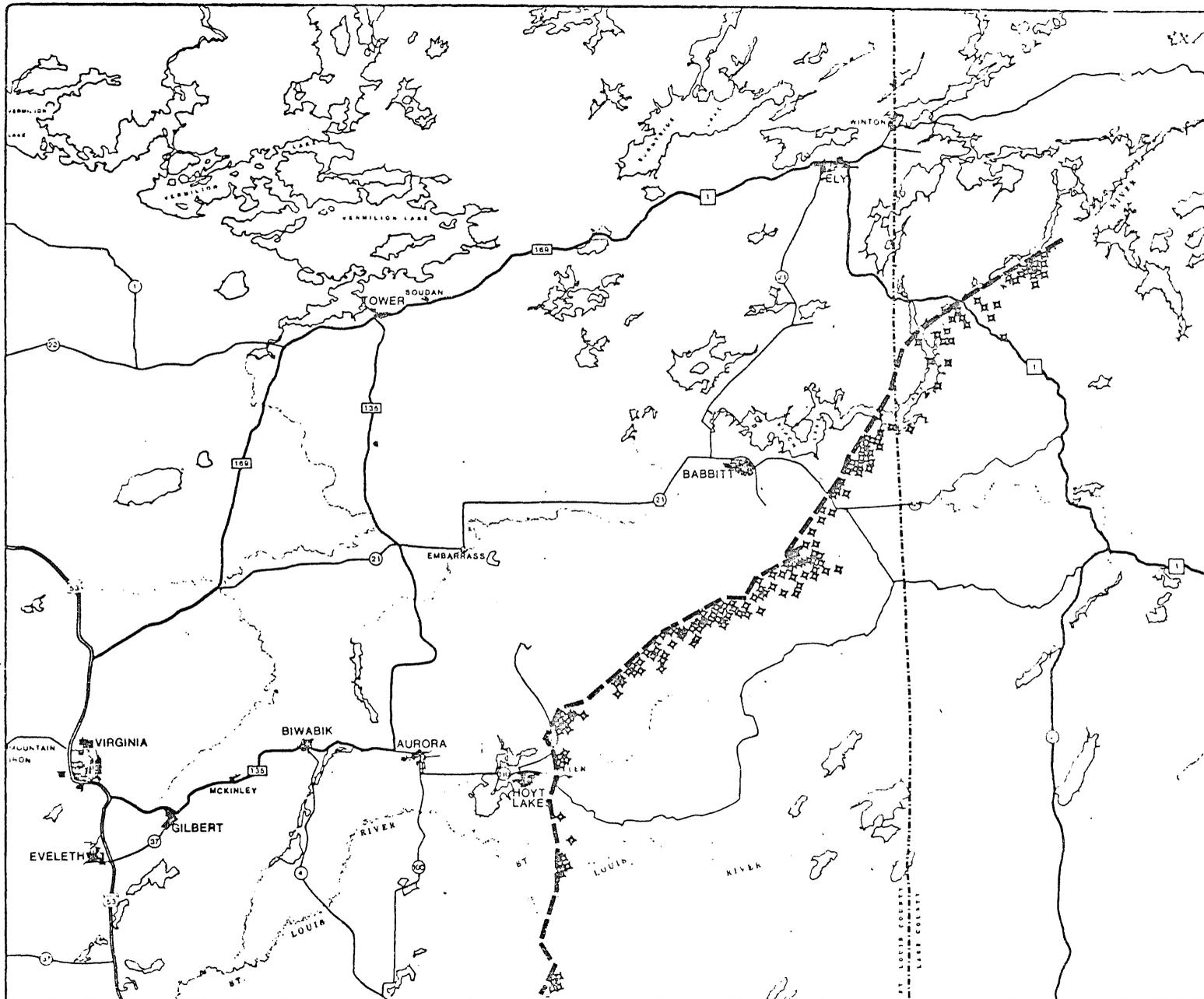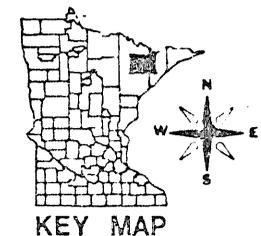
KEY MAP

1 422 400

FIGURE 1

MEQB REGIONAL COPPER-NICKEL STUDY

MINERAL RESOURCE ESTIMATE DRILL HOLE LOCATION MAP

LAURENTIAN DIVIDE

DULUTH CONTACT

DEVELOPMENT ZONE

RESOURCE ZONES

KEY MAP
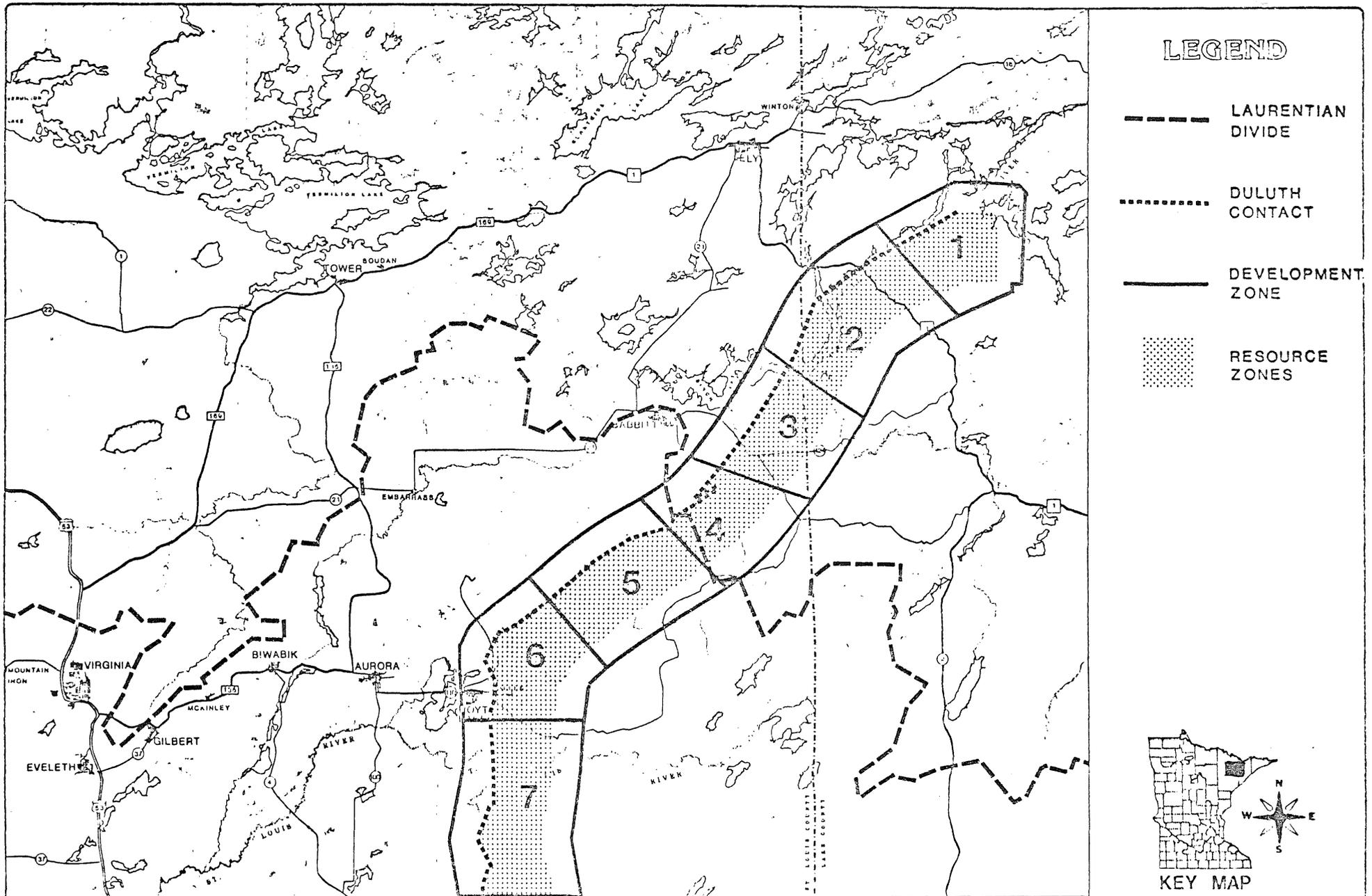
MEQB REGIONAL COPPER-NICKEL STUDY

FIGURE 2

MN CU-NI DEVELOPMENT AND RESOURCE ZONES

discontinuities in the pattern of mineralized polygons so that each of the 7

zones contains a potential locus of mining activity, generally reflecting the

possible extent of a single mining operation. Zone 7 was created, in the absence

of adequate mineralized polygon information, on the basis of exploration interest

shown by mining companies as reflected in state mineral lease holdings. With

this exception, current mineral rights holdings, company interests, watershed

divides, and vegetation boundaries were not used to designate the zones.

With the identification of the resource zones, it was convenient to further

define a broader area likely to contain the major support elements (waste piles,

mill, tailing basins, etc.) needed to service any mines in the resource zones.

Thus, a set of 7 developent zones (Figure 2) were created by adding a strip $1\frac{1}{2}$ mi

wide completely encircling the resource zones. The resulting set of 6-mi wide

zones provides a useful framework for discussing the direct impacts of a mine-

mill complex, as well as an on-site smelter/refinery operation. See the mine

lands discussion, Volume 5-Chapter 5, for further information on the development

zones. It should be noted that due to their size, tailing basins may well extend

beyond the boundaries of the development zones.

2.1.2 Tonnages and Grades of Mineralization

The mineral resources in the study area, shown by the MDNR estimate, are quite

substantial. The calculations for resources grading greater than 0.5% Cu total

over 4.0×10^9 mt (4.4×10^9 st). Of the 324 total holes used in the esti-

mate, 116 contained mineralization which met the greater than 0.5% Cu criterion.

That represents 36% of all holes considered, and those polygons cover 31% of the

total area measured. The indicated near-surface mineralization grading greater

than 0.25% Cu is over one billion metric tons (1.1×10^9 st). The resource

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total for material greater than 10% TiO_2 is over 180 X 10^6 mt (200 X 10^6 st).

Figure 3 shows the major concentrations of the 3 types of mineralization considered in this study. Four major concentrations of the greater than 0.5% Cu mineralization can be seen on the map. These are, from north to south, the INCO Spruce Pit area in resource zone 1 (635 X 10^6 mt); the INCO-Hanna-Duval block in resource zone 2 (2.09 X 10^6 mt); the AMAX area in zone 4 (726 X 10^6 mt); and the U.S. Steel Dunka area in zone 5 (272 X 10^6 mt). Scattered small areas of this type of mineralization also occur elsewhere along the contact in zones 3, 5, and 7.

Figure 3

There are 2 major areas of near-surface mineralization indicated on Figure 3, the INCO Spruce Pit area and the AMAX area. The Spruce area data indicates over 327 X 10^6 mt of material grading greater than 0.25% Cu, and the AMAX area estimate is over 281 X 10^6 mt of similar grade material. These 2 mineralized zones are indicated by contiguous polygons on the map. Significant tonnages (272 X 10^6 mt) are indicated in the Dunka Pit area as several isolated polygons. A few small scattered indications do occur elsewhere along the contact, but the above 3 areas account for over 90% of near-surface, greater than 0.25% Cu resource.

The titanium resources indicated total over 200 X 10^6 mt greater than 10% TiO_2 and are located in 3 small areas. The largest of the 3 areas is the southernmost, the Water Hen area, with an estimate of over 91 X 10^6 mt. This type of mineralization may be a significant resource in the study area, but it appears that little exploration for this specific type of resource has been done.

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LEGEND

---- DULUTH CONTACT

≤ .50% COPPER

≤ .25% COPPER (NEAR SURFACE)

≥ 10% TiO2

B.W.C.A.

INCO SPRUCE PIT AREA

INCO, HANNA DUVAL AREA

DUNKA PIT AREA

AMAX AREA

USS DUNKA AREA

WATER HEN AREA

WINTON

ELY

TOWER   SOUDAN

VERMILION LAKE

BABBITT

EMBARRASS

VIRGINIA

MOUNTAIN IRON

BIWABIK

AURORA

ALLEN

HOYT LAKE

MCKINLEY

GILBERT

EVELETH

ST. LOUIS COUNTY
LAKE COUNTY

T63N
T62N
T61N
T60N
T59N
T58N
T57N

KEY MAP

N
W   E
S

1 422 400

MILES

KILOMETERS

MEQB REGIONAL COPPER-NICKEL STUDY

FIGURE 3

MINERALIZATION LOCATION MAP

Although copper and nickel sulfides are the major resources in the Duluth Complex, there are other economic minerals (see Volume 2-Chapter 5, section 5.3.4). Of this secondary group, ilmenite (titanium) is most likely to be considered for mining.

In any estimation of resources the choice of the cutoff grade is important. By adjusting the cutoff grade downward, the tons of the resource increase, but the cutoff may also be lowered below the level of economic feasibility. MDNR investigated the effect of lowering the cutoff grade by conducting a smaller study using only those holes for which complete assay data was available. This amounted to 122 holes or 38% of the total number of holes used for the main estimate in the large study. The same area of influence was used in this calculation that was used for each hole in the main estimate. The average area for each of these 122 polygons is 5.7 ha (0.22 mi^2), which is significantly larger than the average for the whole study because of a lack of complete assay data for the most heavily drilled areas. Although the polygons are larger on the average than in the complete study, the average thickness of the greater than 0.5% Cu zones in these holes is 126 ft, reasonably close to the value of 134 ft in the complete estimate.

The tonnage of greater than 0.25% Cu material in the near-surface mineralization estimate is over 344 X 10^6 mt, or 34% of that for the larger study. The average grade calculated for this mineralization is 0.34% Cu. The greater than 0.5% Cu estimate for these holes was over 2.3 X 10^9 mt of resource. This estimated tonnage is 59% of the total estimated in the Copper-Nickel Study Area. The average grade of the 2.3 X 10^9 mt is 0.66% Cu. If the cutoff grade in the greater than 0.5% Cu estimate were lowered to greater than 0.25% Cu the tonnage would increase to over 6.0 X 10^9 mt. The overall grade of this material is about 0.45% Cu.

Figure 4 shows the frequency distribution of copper assay values reported by Listerud and Meineke (1977). Based on 5,293 individual assays, those from zero to 1.4% Cu account for 99.9% of the total number available to MDNR. By assuming the area under the curve from 0.5% Cu to 1.4% Cu represents 4.0×10^9 mt of greater than 0.5% Cu resource, several things can be done. The average grade can be calculated by finding the grade that divides the area in half, which results in a value of approximately 0.65% Cu. Lowering the cutoff grade to 0.25% Cu increases the tonnage to over 12.7×10^9 mt, with an average indicated grade of 0.39% Cu.

Figure 4

Another way to look at the resource tonnage present is to distinguish between those resources likely to be recovered by open pit mining, and those likely to require recovery by underground techniques. This is done by splitting the amount present into the categories shown on Table 1 by resource zone. For purposes of this estimate, the use of open pit extraction methods will be assumed not to exceed a depth of 1,000 ft below surface level. Resources below 1,000 ft can be generally considered underground mining resources. The average grade numbers are from the above mentioned MDNR report. The average grade of 0.34% Cu (Table 1) for the near-surface resource grading 0.25 to 0.5% Cu was determined from Figure 4 using the graphical method of MDNR. Table 1 shows a total ore tonnage of almost 4.6×10^9 mt. This estimate is 9% lower than that of the MDNR because the zone estimate was made using the MDNR's computerized MINESITE hectare grid which underestimates surface area relative to the planemetric area method of the MDNR report. MINESITE and the MDNR report used the same drill core information.

Table 1

FIGURE 4

GRADE-FREQUENCY CURVE, BASED ON 5293 DATA POINTS.
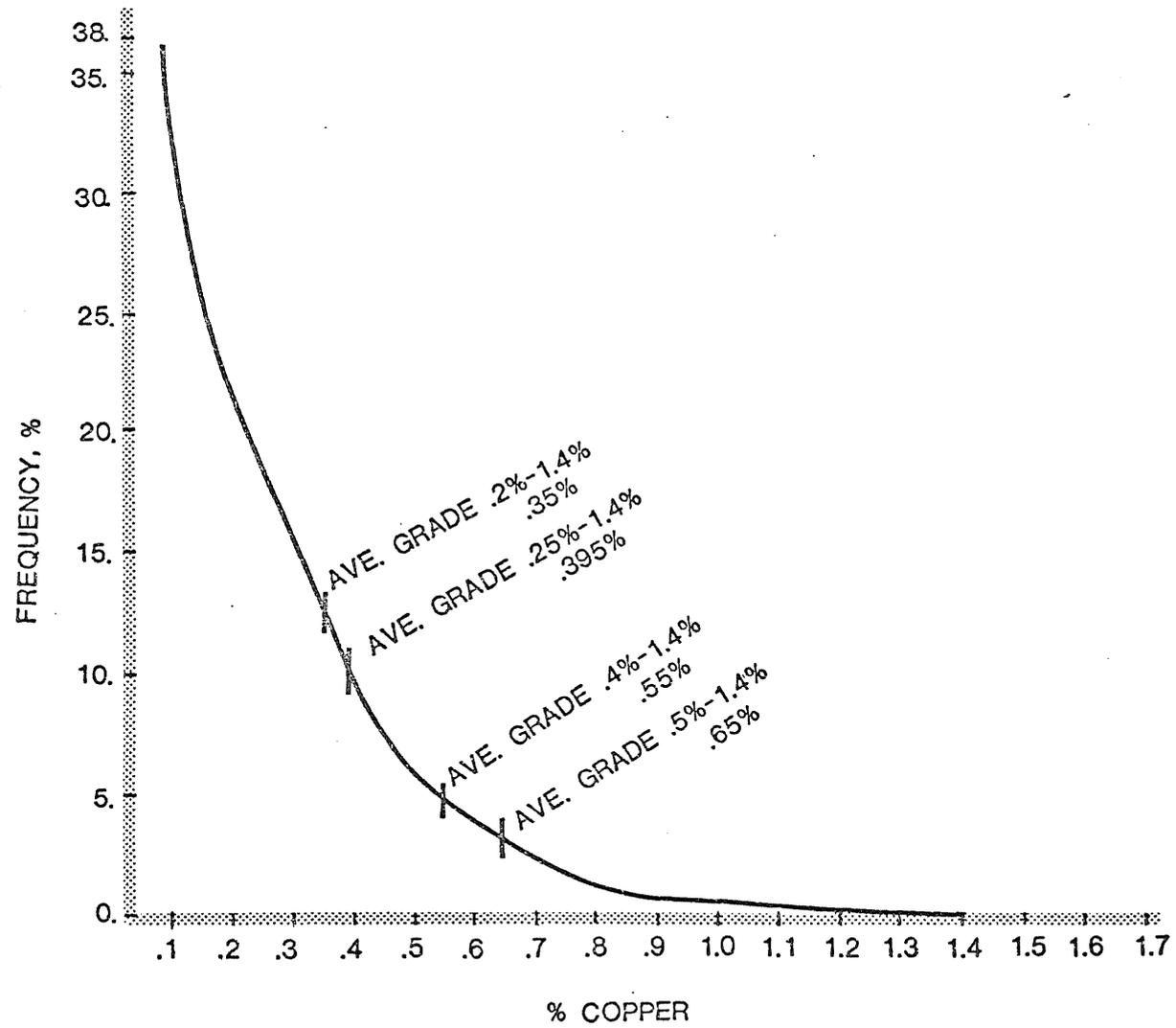
SOURCE : LISTERUD AND MEINEKE (1977)

Table 1. Copper ore tonnage and grade resource estimate, by resource zone.

RESOURCE ZONE	OPEN PIT RESOURCE, 10^6 mt				UNDERGROUND RESOURCE, 10^6 mt		TOTAL RESOURCE, 10^6 mt	
	0.25-0.50% Cu Near Surface	0.50% Cu above 1000 ft	Total above 1000 ft Resource	% Cu	0.50% Cu below 1000 ft	% Cu	Grand Total	Average % Cu
1	107.2	369.9	477.1	0.59	371.9	0.66	849.0	0.62
2	-----	339.6	339.6	0.66	1614.7	0.66	1954.3	0.66
2&3[b]	14.6	112.5	127.1	0.62	----	----	127.1	0.62
3	245.8	18.9	264.7	0.36	76.1	0.66	340.8	0.43
4	183.4	49.1	232.5	0.41	545.2	0.66	777.7	0.59
5	38.1	73.4	111.5	0.55	232.8	0.66	344.3	0.62
6	59.6	52.0	111.6	0.50	48.9	0.66	160.5	0.55
7	-----	11.2	111.2	0.66	-----	----	11.2	0.66
TOTAL	647.7	1026.8	1675.3	----	2889.6	----	4564.9	----
AVERAGE % Cu	0.34	0.66	0.54		0.66		0.62	

[a]Resources of 0.25-0.50% Cu average 0.34% Cu, 0.50% Cu, or better average 0.66% Cu.
[b]Zones 2 and 3 underwater resources (under Birch Lake) but within 1,000 ft of the surface.

Figures 3 and 5 show the distribution of the MDNR polygons by Cu content, along with the surface and the 1,000 ft basal contact contours of the Duluth Gabbro Complex. Zone 2 contains the largest amount of greater than 0.5% Cu whereas Zone 7 contains the least. Zone 3 has the largest amount of 0.25 to 0.50% Cu near surface resources.

Figure 5

Table 2 shows the copper metal contained in the ore by resource zone and depth. Assuming that open pit extraction will stop at the 1,000 ft level, there is 9.0 X 10^6 mt of contained copper accessible by the open pit method and 19.1 X 10^6 mt of contained copper accessible by underground mining techniques.

Table 2

In presenting estimates of the amount of nickel present in the Study Area, it is useful to discuss the Cu/Ni ratio found in the samples assayed. This is discussed in the MDNR report. The ratio most frequently mentioned when discussing the Duluth Complex is 3:1. This is the ratio assumed by Bonnichsen (1974) for his resource estimate. With much more assay data available, the MDNR determined the actual Cu/Ni ratios and plotted the distribution of the values. It was assumed that all assays used represent the total metal content of the rock. A total of 4,912 individual ratios were calculated and plotted for this study. The copper and nickel values for intervals less than 5 ft were weighted and averaged with the adjacent values. The ratios were plotted in 0.5 intervals (0 to 0.50, 0.51 to 1.0, etc.) and the histogram is shown in Figure 6. It is interesting that the peak occurs from 1.5 to 3.0, much lower than anticipated. This graph is based on individual assays and not strictly on material that could

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LEGEND

- ☐ 1 NONMINERALIZED POLYGONS
- ▨ 2 ≥ 0.8 CU
- ☐ 3 ≥ 0.26 CU AND < 0.80 CU NEAR SURFACE
- ▦ 4 BOTH #2 AND #3 OCCUR

- ∙∙∙∙∙∙∙ BASAL CONTACT OF GABBRO
- ∙∙∙∙∙ DEPTH TO BASAL CONTACT 1000' CONTOUR

▬▬▬ DEVELOPMENT ZONES

① DEVELOPMENT ZONE NUMBERS

KEY MAP

1 422 400

MILES

KILOMETERS
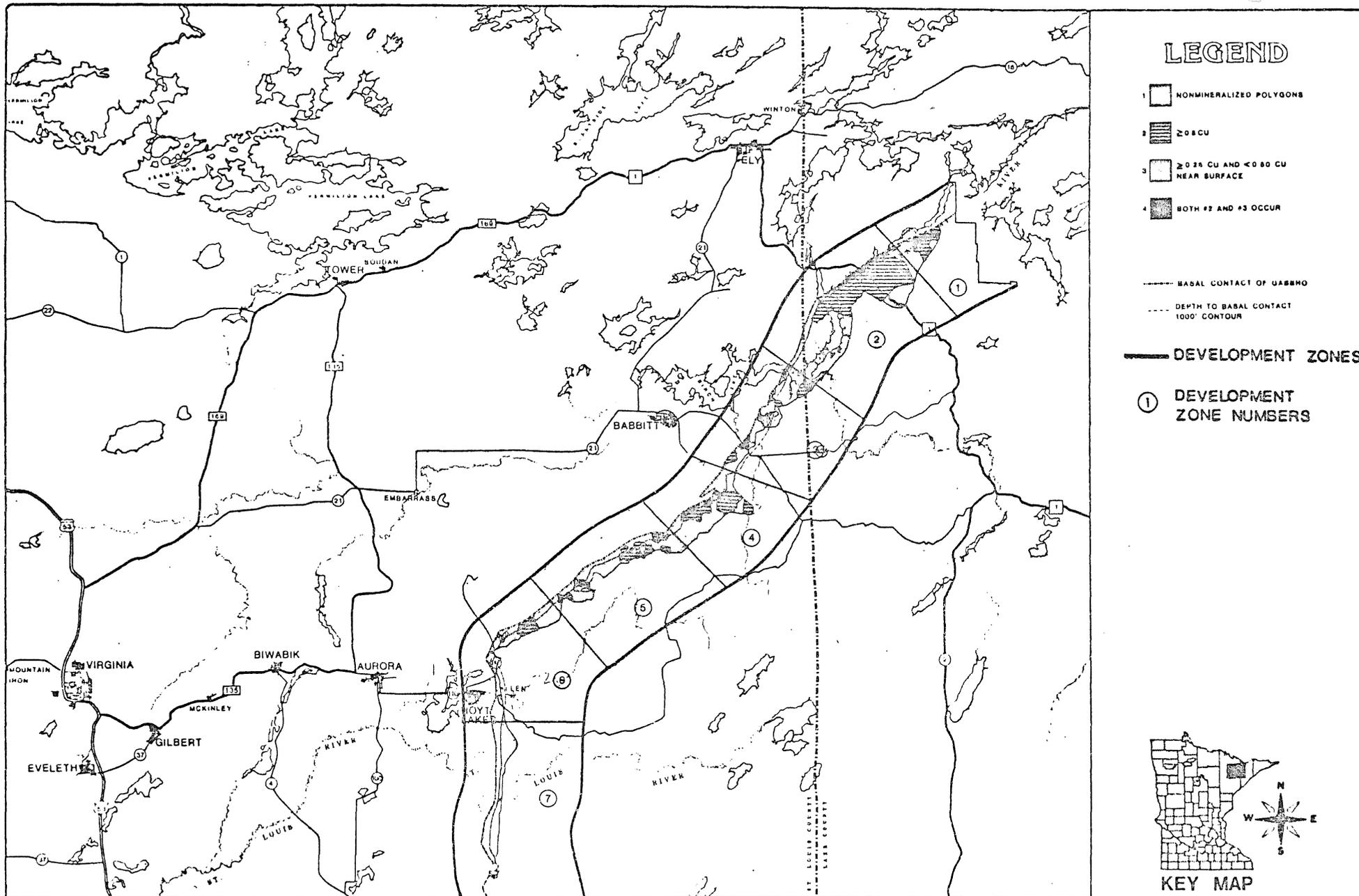
MEQB REGIONAL COPPER-NICKEL STUDY

FIGURE 5

MDNR POLYGON DISTRIBUTION

Table 2. Contained copper metal by resource zone.

RESOURCE ZONE	OPEN PIT RESOURCE, 10^6 mt			UNDERGROUND RESOURCE, 10^6 mt	TOTAL RESOURCE, 10^6 mt
	0.25-0.5% Cu Near Surface	0.50% Cu above 1000 ft	TOTAL[a] above 1000 ft	0.50% Cu below 1000 ft	
1	.36	2.44	2.80	2.46	5.26
2[c]	---	2.24	2.24	10.66	12.90
2&3[b]	.05	0.74	0.79	----	0.79
3	.84	0.13	0.97	0.50	1.47
4	.62	0.32	0.94	3.60	4.54
5	.13	0.48	0.61	1.54	2.15
6	.20	0.34	0.54	0.32	0.86
7	---	0.07	0.07	----	0.07
TOTAL	2.20	6.76	8.96	19.08	28.04

[a]For modeling purposes, material within 1,000 ft of surface is considered to be accessible by open pit mining.

[b]Zones 2 and 3 underwater resources within 1,000 ft of surface in addition to resources listed for the individual zones.

[c]Of the 2.24 X 10^6 mt above 1,000 ft, only the copper below 600 ft is thought to be minable by underground methods. Underground extraction may be deemed necessary due to the proximity to Birch Lake.

or would be mined, and this may be an important influence.

Figure 6

The weighted average Cu/Ni ratio for the mineralized zones (greater than or equal
to 0.5% Cu) of 26 drill cores is 3.33. The ratios for individual holes vary from
1.26 to 6.33. The near-surface mineralization (greater than or equal to 0.25%
Cu) has an average Cu/Ni ratio of 3.59 based on data from 8 drill cores. The
ratios from these 34 holes were plotted against the depth of the mineralized
zones, distance from the contact, percent copper, and percent nickel.
Examinations of these plots show no prominent relationships between the Cu/Ni
ratio and depth or distance from the contact, although it appears that the more
extreme values are more likely to occur at greater depths and further from the
surface basal contact. The ratio plotted against percent copper shows no trends,
but a strong, almost linear negative reationship appears to exist between nickel
content and the Cu/Ni ratio (see Listerud and Meineke 1977 for more details on
this). An examination of the individual core data showed no consistent variation
of Cu/Ni ratio within the mineralized zones, in one core the ratio decreased
steadily downward while in another it increased downward.

Based on the data used in the MDNR study, no apparent areal variation in Cu/Ni
ratio occurs. However, independent studies based on other data indicate that the
Cu/Ni ratio does vary from area to area within the Duluth Complex (Listerud and
Meineke 1977). The Spruce Road Area of INCO (zone 1) was calculated by them to
have a ratio of 2.71 for 248×10^6 mt within their proposed pit. The 2.0 X
10^9 mt of greater than or equal to 0.5% Cu estimated for the INCO-Hanna-Duval
area (zone 2) has a ratio of 3.24. This was calculated using all of the 17
mineralized holes that define the mineralization. The ratio used for the 725 X

FIGURE 6

COPPER-NICKEL RATIO HISTOGRAM
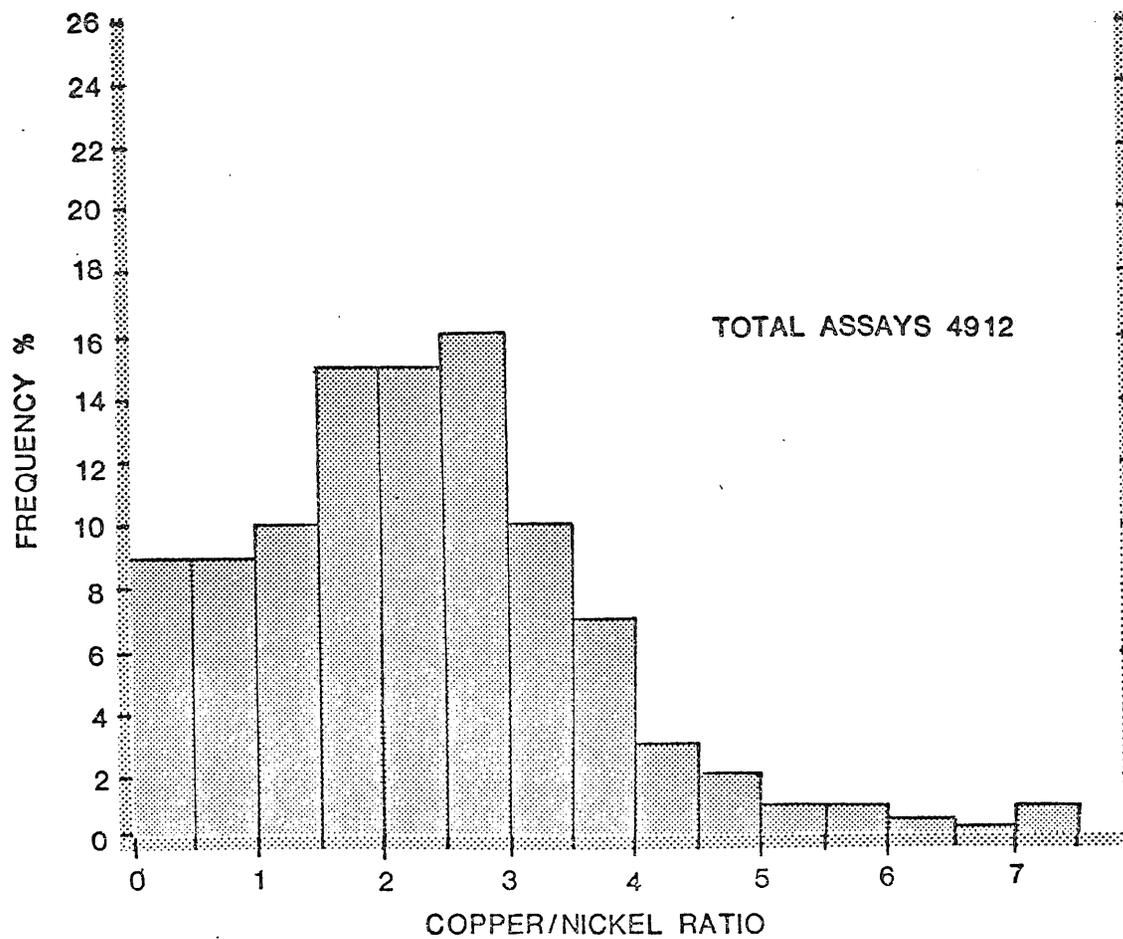
SOURCE : LISTERUD AND MEINEKE (1977)

10^6 mt estimated in the AMAX area (zone 4) is 4.00. This is based on the ratio determined by AMAX for their 300 to 340 X 10^6 mt of potential underground resources. The semi-massive sulfides that AMAX has defined (AMAX 1977) have a higher Cu/Ni ratio than most of the rest of the resource. According to information from AMAX, possible underground semi-massive reserves of 4.8 X 10^6 mt exist grading 2.84% Cu and 0.52% Ni, for a Cu/Ni ratio of 5.46. The U.S. Steel Dunka area (zone 5) is estimated to have about 272 X 10^6 mt of resource greater than or equal to 0.5% Cu. The ratio determined by U.S. Steel for that resource is 3.20. From Table 23 of Chapter 1, ratios can be determined from the various samples collected for the Regional Study. The disseminated average contains 0.545% Cu and 0.125% Ni for a ratio of 4.36, much higher than the 3.33 found in the MDNR study. The semi-massive average, at 2.849% Cu and 0.318% Ni, has a ratio of 8.96, even higher than that found by AMAX. The study of 91 core samples yielded a median composition of 0.486% Cu and 0.096% Ni, or a ratio of 5.05.

The conclusion from all of this seems to be that, though the MDNR study ratio of 3.33 appears to be valid as a regional average, there can be considerable departures from this value on a local basis. For example, indications are that an operation in resource zone 1 might produce 40 to 50% more nickel per unit of contained copper than a similar operation in resource zone 4. Since nickel is likely to have a market value 2 to 3 times that of copper for the foreseeable future, such variations could easily make the difference between an economically viable operation, and an unprofitable one. The variable nature of the data presented here indicate that the nickel content must be carefully investigated at each specific potential mining location.

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For purposes of the resource estimate being discussed here, the MDNR ratio of 3.33 for the region is used for internal consistency. The totals in Table 3 can thus be read as a valid estimate of nickel ore tonnages, in which the average copper grade is converted to nickel grade by dividing by 3.33.

Table 3

The results shown in Table 3 indicate contained nickel in millions of metric tons. Again, as this discussion has shown, these values must be interpreted with care, and cannot be scaled down for application at the level of the resource zone. They do indicate, however, a total of 2.69×10^6 mt of contained nickel accessible by open pit methods, and 5.73×10^6 mt, or 68% of the total, recoverable by underground methods.

In dealing with the problem of classifying the material examined in the MDNR study into categories which adequately and concisely define the mineralization the MDNR used the classification system of Brobst and Pratt (1973)(Figure 7). The terminology is defined below, quoted from Brobst and Pratt (1973).

Figure 7

RESERVES: "Known identified deposits of mineral-bearing rock from which the mineral or minerals can be extracted profitably with existing technology and under present economic conditions."

RESOURCES: "Include not only reserves but also other mineral deposits that may eventually become available--either known deposits that are not economically or technologically recoverable at present, or unknown deposits, rich or lean, that may be inferred to exist but have not yet been discovered."

Table 3. Contained nickel metal in resource zones.[a]

| | OPEN PIT RESOURCE, 10^6 mt | | | UNDERGROUND RESOURCE, 10^6 mt | TOTAL RESOURCE, 10^6 mt |
	0.25-0.50% Cu near surface	0.50% Cu above 1000 ft	Total[b] above 1000 ft	0.50% Cu below 1000 ft	
10^6 mt nickel metal	0.66	2.03	2.69	5.73	8.92
Calculated ore grade[c] % Ni	0.10	0.20	0.16	0.20	0.18

[a]The information in this table assumes a Cu/Ni ratio of 3.33.

[b]For modeling purposes, material within 1,000 ft of the surface is considered to be accessable by open pit mining.

[c]This number is calculated from the 2.69 X 10^6 mt of metal and the total ore tonnage shown in the third column of Table 1.

FIGURE 7

CLASSIFICATION OF MINERAL RESOURCES

LEGEND

POTENTIAL RESOURCES=

CONDITIONAL + HYPOTHETICAL + SPECULATIVE
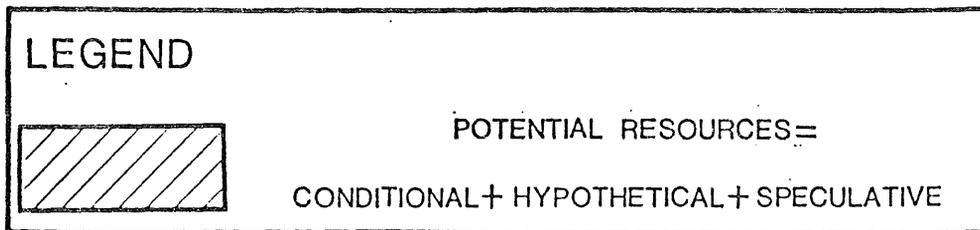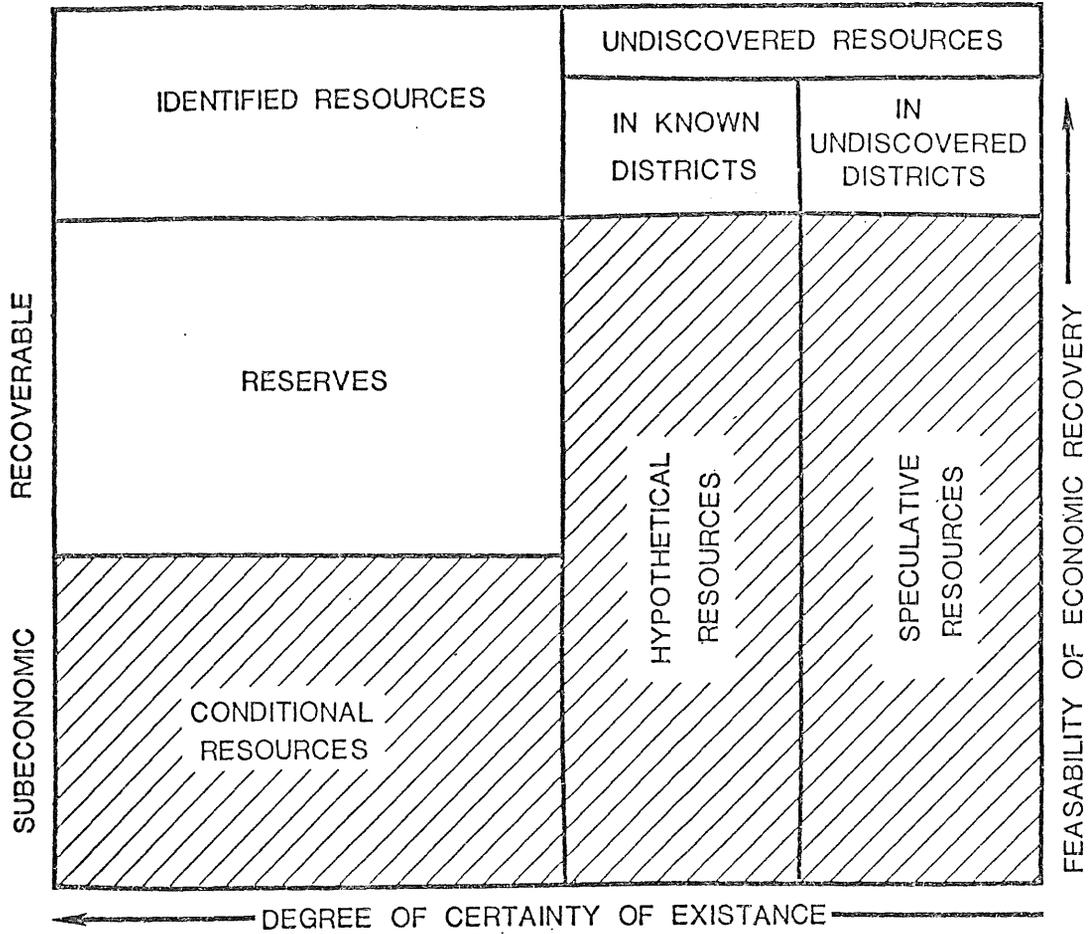
SOURCE : BROBST AND PRATT, 1973

CONDITIONAL RESOURCES: "Resources that may eventually become reserves when conditions of economics or technology are met."

HYPOTHETICAL RESOURCES: "Undiscovered resources that we may still reasonably expect to find in known districts."

SPECULATIVE RESOURCES: "undiscovered resources that may exist elsewhere.

INDENTIFIED RESOURCES: "Specific bodies of mineral-bearing rock whose existence and location are known. They may or may not be evaluated as to extent or grade."

The resources estimated by MDNR would appear to belong in the conditional resources category. The mineralized areas are all-identified in at least one drill core and the blocks with the major tonnages are defined by a minimum of 15 drill cores each. Several bulk samples for metallurgical testing have been taken from the Duluth Complex. Two exploration shafts have been sunk, one 1,100 ft and one 1,700 ft, and one proposal for an open pit mine has been made and subsequently withdrawn. All of this activity, and the 1,000 to 1,500 holes drilled, has yet to result in a producing mine. Therefore, the estimate of 4.0 X 10^9 mt of material containing greater than 0.5% Cu and a Cu/Ni ratio of 3.33 must presently be classified as a subeconomic, identified resource (or conditional resource). However, as Bonnichsen (1974) has pointed out, the Duluth Complex remains the single, largest known resource of copper and nickel in the United States and the potential for eventual development and mining is good. The Bureau of Mines (Mineral Commodity Profile-Cobalt 1977) indicates that this resource is also the largest identified cobalt resource in the United States. Besides copper, nickel, and cobalt, the Duluth Complex contains small but recoverable quantities of gold, silver, and platinum-group metals (see sections 2.4.1 and 2.5.1).

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2.1.3 Resource Life

By splitting the resources of the area at the 1,000 ft depth, open pit and underground resources may be looked at separately. As was indicated in Table 1, some resources may not be accessible because of their proximity to water. These considerations have to be taken into account with any discussion of resource life.

Table 4 shows the mine life for both open pit and underground mines producing ore from each resource zone. The number of years shown in Table 4 for the different zones are directly related to the amount of material present per zone. This table is presented to give only an order of magnitude feeling of the possible mining life of a particular region. The figures are obtained simply by dividing the resource tonnage figures shown in Table 1 by the various model production rates. Clearly, this greatly oversimplifies the complex problem of determining the actual mineable reserves at any specific location and by a specific mining method. The numbers can only be used as general indicators, and to illustrate what is known about the relative potentials of the various resource development zones with respect to each other. The main assumptions made are that all of the resources in each zone are extractable, and that the total production by zone will conform to the modelled mining extraction rates.

Table 4

The combined zone 2 and 3 shown in Table 4 indicates the loss in years of production from open pit mining due to the proximity of water, and the addition of years of production to underground mining, assuming mining could go as high as 600 ft below the surface.

Table 4. Life span (in years) per development zone assuming a given mine production per zone and all resources extractable.

MINE MODEL (10^6 mtpy)	20.00 OPEN PIT	11.33 OPEN PIT	12.?5[a] UNDERGROUND	5.35[a] UNDERGROUND
ZONE				
1	23.9	42.1	23.2	53.5
2	17.0	30.0	100.6	232.4
2&3	(3.8)[b]	(6.8)[b]	(3.2)[c]	(7.3)[c]
3	13.2	23.3	4.8	10.9
4	11.6	20.5	33.9	78.5
5	5.6	9.8	14.5	33.5
6	5.6	9.8	3.0	7.0
7	0.6	1.0	---	---

[a]These numbers are calculated assuming 23% of resource will be left in place for underground mining roof support.

[b]These figures show the number of mine production years lost because resource is underwater and within 600 ft of surface.

[c]These figures are the number of mine production years gained by assuming 40% (600-1000 ft level) of resource in this zone can be extracted by underground techniques.

Zones 1, 2, 3, and 4 have the longest open pit model life, with each one over 20 yr at the lower model rate of 11.33 X 10^6 mtpy. The resources in zone 3 are scattered over a large area, which inhibits development. This is not the case in zones 1, 2, and 4. Zone 7 has the shortest model span at one year, at the model rates.

Zone 7 is also the shortest for underground mining as it has no identified resource below 1,000 ft. Combining years of production, zone 6 has the next lowest resource model life. Although zones 1 and 4 have resource model lives of over 50 yr for the 5.35 X 10^6 mtpy model mine, the underground resource in zone 2 is much larger, over 100 yr for the 12.35 X 10^6 mtpy model mine. The underground numbers use a 77% recovery factor, because the models assume that 23% of the rock in a mining area will be left in place for roof support or is not economically accessible to underground mining techniques.

2.2 ORE TYPES

As was discussed in previous sections, mining in northeastern Minnesota is expected to be by open pit and/or underground methods, so models of ore from these 2 mine types were derived from the disseminated average data presented earlier. This can be done because most of the type of mineralized rock that will be present both above and below the 1,000 ft depth falls into the disseminated category as discussed in section 1.4.3.

The fact that separate models are required for ore that would be mined respectively by open pit and underground techniques, is simply a reflection of the fact that the more expensive underground methods require a higher average ore grade to be economically viable. In addition, the possibility of the presence of a small amount of higher grade semi-massive ore at certain locations is recognized in a

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third ore model, which stresses the compositional pattern expected for this material.

2.2.1 Ore Chemistry

Table 5 lists data that was available to the Regional Study concerning the copper content of resource material that is considered possibly mineable with current open pit and underground technology. The values include data from deposits world-wide, as well as within the Study Area. Typically, values of 0.5% Cu for open pit mining and 0.8% Cu for underground mining are quite reasonable based on this data. These general values were thus chosen as the basis for modeling the ore for the hypothetical mine models presented in the technical assessment report, Volume 2-Chapter 5.

Table 5

The models were designed to support a smelting/refining complex producing 100,000 mtpy of copper and nickel metal. Using averages of recovery values determined by the USBM, MRRC, INCO, and AMAX, and mine sizes chosen to cover the expected range of Minnesota operations, model values for ore grades were calculated as:

    0.494% Cu for open pit material
    0.800% Cu for underground material

Additionally, the nickel levels were calculated based on the corresponding recovery data and the Cu/Ni ratio of 4.33 found for the disseminated average (Table 25 of Chapter 1). These calculations resulted in:

    0.114% Ni for open pit material
    0.185% Ni for underground material

These levels were then used in the mining models, as separate open pit or

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Table 5. Available data on the composition of potentially mineable mineral resources.

DATA SOURCE	MINING METHOD	% Cu		AVE. OR TYPICAL VALUES	
		Range	Average or Typical Value	% Ni	% S
MRRC[f]	Underground	0.58-0.81	0.71	0.16	1.49
AMAX[b]	"	---	0.82	0.19	2.01
World[a]	"	0.70-7.0	0.90	--	--
AMAX[e] (semi-massive)	"	---	2.84	0.52	7.38
AMAX[c]	Open Pit	---	0.46	0.11	1.0
INCO[d]	"	---	0.47	0.15	1.08
World[a]	"	0.35-7.0	0.50	--	--

[a]World data-from Table 1, mining Section II.B.2.
[b]Based on disseminated resource (AMAX 1977).
[c]Based on possible open pit reserves (AMAX 1977).
[d]Based on Spruce Road sample (INCO 1975).
[e]Based on semi-massive resource (AMAX 1977).
[f]MRRC data on bulk flotation pilot plant work with AMAX sample (MRRC 1978).

underground operations, or as a combination, to yield operations of 12.35, 16.68, and 20.00 X 10^6 mtpy, all supplying the model smelter. Also, 5.35 and 11.33 X 10^6 mtpy models were detailed to illustrate how smaller operations could exist but not by themselves support the modeled smelter.

It should be mentioned that for practical purposes the ore models contain 0.5% Cu and 0.8% Cu for open pit and underground models, respectively. The use of the value 0.494% Cu for the open pit is merely a reflection of the fact that the parameter is part of a material balance calculation needing precision for internal consistency. Predictive precision should not be attributed to such numbers, wherever they appear in the modeling work presented in this report.

With the given copper values as the basis for the ore models, it remains to model the remainder of the chemical constituents expected to be present in the ore. The open pit model ore was calculated from the disseminated average by extrapolating the sulfide-related elements proportional to the copper value of 0.494%. The same method was used to get the underground model ore composition, based on the copper value of 0.800%. The non-sulfur related elements were then adjusted proportionally so the total equaled 100%. It should be noted that the amount of water present is not included but is approximately 0.50 wt% (Weiblen and Stevenson 1978).

Table 6 shows the resultant chemistry of the 2 model ore types. Also shown are the chemistries of the disseminated average and the semi-massive average as discussed earlier and given in Table 25 , section 1.4.3.2 of Chapter 1. Comparison of the modeled values for Cu, Ni, and S with the data shown in Table 5 indicates that the models are reasonable, and generally consistent with available data from sources outside of the Regional Study.

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Table 6

Column 1 of Table 6 is the average chemistries of the disseminated average ore. This is the same data that was shown in Table 25 of Chapter 1. The S, Cu, and Ni values for the semi-massive average are not as close to the AMAX semi-massive analysis as are the disseminated open pit and underground models to these respective published comparisons. However, the average of column 4 is close enough to the published value to be used as a useful model of semi-massive ore.

Figure 8 is a normalization plot of the three model ores (open pit, underground, and semi-massive). As was shown previously (section 1.4.3.2), and again in Figure 8, the semi-massive ore is significantly higher in sulfur-related elements than are the 2 disseminated ore models. The accuracies of the analyses shown in previous figures and Figure 8 are difficult to estimate but are believed to be relatively good compared to the variability of the mineralized gabbro.

Figure 8

To place the modeled ore chemistry values into perspective, Table 7 shows the actual range of observed values in various input or feed samples used by MRRC for bench-scale test work. All the samples used were included in deriving these ranges.

Table 7

Elements not shown in Table 6 but for which plasma spectrometry or neutron activation analysis (NAA) data is available is shown in Table 8. The errors for the data shown in Table 8 are perhaps as high as 10 times those of Table 6. Because

Table 6. Model ore composition: chemistry.

ELEMENT	1 AVERAGE DISSEMINATED[a]	2 MODEL FOR OPEN PIT	3 MODEL FOR UNDERGROUND	4 AVERAGE SEMI-MASSIVE[a]	UNITS
$Si(SiO_2)$	22.29(47.73)	22.32(47.79)	21.96(47.01)	20.68(44.29)	PCT
$Al(Al_2O_3)$	9.21(17.40)	9.22(17.42)	9.08(17.14)	7.08(13.37)	PCT
$Fe(FeO)$	8.69(11.18)	8.70(11.19)	8.56(11.01)	8.66(11.14)	PCT
$Mg(MgO)$	4.54(7.52)	4.55(7.53)	4.47(7.41)	3.10(5.73)	PCT
$Ca(CaO)$	5.53(7.74)	5.54(7.75)	5.45(7.63)	2.88(4.03)	PCT
$Na(Na_2O)$	2.18(2.94)	2.18(2.94)	2.15(2.90)	1.26(1.70)	PCT
$K(K_2O)$	0.35(0.42)	0.35(0.42)	0.34(0.41)	0.53(0.64)	PCT
$Ti(TiO_2)$	0.92(1.53)	0.92(1.53)	0.90(1.51)	1.00(1.67)	PCT
$P(P_2O_5)$	0.03(0.07)	0.03(0.07)	0.03(0.07)	0.03(0.07)	PCT
$Mn(MnO)$	0.12(0.16)	0.12(0.16)	0.12(0.16)	0.10(0.13)	PCT
$Cr(Cr_2O_3)$	0.03(0.05)	0.03(0.05)	0.03(0.05)	0.03(0.04)	PCT
B	571.2	571.9	562.7	127.2	PPM
Ba	704.3	705.1	693.8	394.4	PPM
Be	0.54	0.54	0.53	1.74	PPM
Sr	277.7	278.0	273.6	181.9	PPM
V	166.2	167.2	163.7	222.6	PPM
Th	4.3	4.3	4.2	3.4	PPM
Zr	96.1	96.2	94.7	79.1	PPM
S	1.129	1.095	1.658	6.528	PCT
Cu	0.545	0.494	0.800	2.849	PCT
Ni	0.125	.114	0.185	0.318	PCT
Fe	1.249	1.235	1.834	7.335	PCT
Co	0.012	0.011	0.017	0.030	PCT
Zn	139.0	132.9	204.1	236.8	PPM
Pb	5.3	4.6	7.8	0	PPM
Ag	2.7	2.4	4.0	7.1	PPM
As	10	9.5	14.7	10	PPM
Hg	.080	.075	.117	.070	PPM
Mo	1.6	1.4	2.3	1.4	PPM
Cd	10	9.5	14.7	10	PPM

[a]See Table 25, Section 1.4.3.2 of Chapter 1.

FIGURE 8

COMPARISON OF MODEL ORES AND SEMI-MASSIVE AVERAGE ORE TO THE DISSEMINATED AVERAGE ORE

LEGEND

+ SEMI-MASSIVE ORE
△ UNDERGROUND ORE
⊙ OPEN PIT ORE
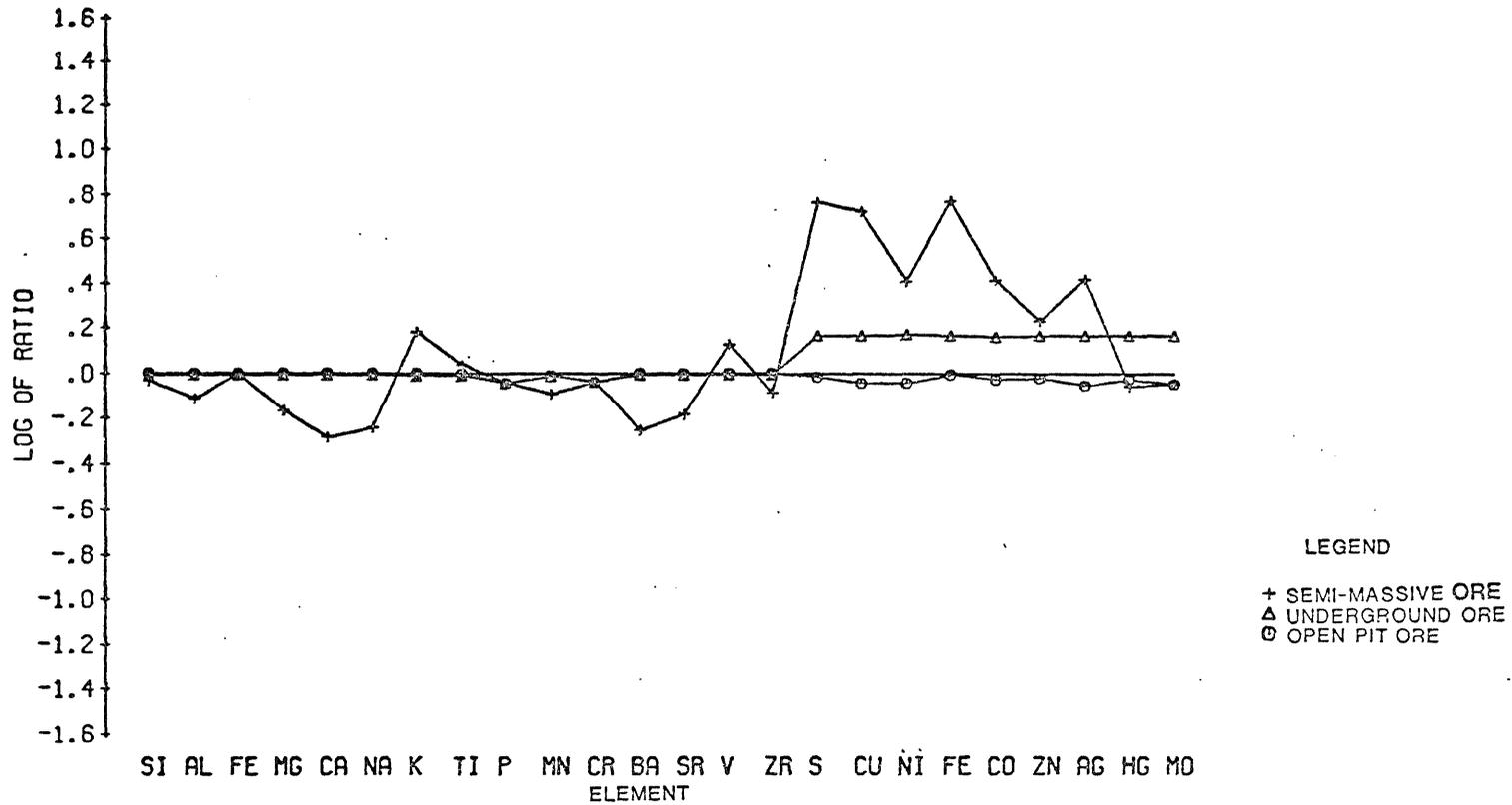
Table 7. Range of values observed for constituents in bench-scale feed samples.[a]

ELEMENT	RANGE OBSERVED	UNITS
Si[b]	No reliable analyses	---
Al	5.68-11.00	PCT
Fe(O)[c]	See entry (under sulfide) below	---
		PCT
Mg	2.46-6.09	PCT
Ca	0.753-6.64	PCT
Na	0.90-2.54	PCT
K	0.084-0.746	PCT
Ti	0.25-1.57	PCT
P	0.00007-0.248	PCT
Mn	0.0566-0.131	PCT
Cr	0.0077-0.0672	PCT
B	0.006-2660	PPM
Ba	136-1800	PPM
Be	0.005-2700	PPM
Sr	149-350	PPM
V	52.5-273	PPM
Th	0.06-14	PPM
Zr	21-123	PPM
S	0.86-10.55	PCT
Cu	0.236-6.03	PCT
Ni	0.0588-0.462	PCT
Fe(S)[c]	4.61-23.4	PCT
Co	0.0046-0.0363	PCT
Zn	63-302	PPM
Pb	0.16-62	PPM
Ag	0.002-16.9	PPM
As	0.14-120	PPM
Hg	0.070-0.090	PPM
Mo	0.03-11.0	PPM
Cd	0.07-4	PPM

[a] Includes data from all samples tested.
[b] No reliable data available, values adjusted to close the material balance in the models.
[c] Range shown is for total iron independent of whether it is in the oxide or sulfide form.

of this, no extrapolation to model ores was made.

Table 8

A comment should be made here concerning precious metals in the ore. These are important to the overall operation since they are recovered at little added cost, principally from the spent electrolyte in the copper refinery, and may provide significant income to the company as by-product credits. The important precious metals in the Duluth Complex resources appear to be silver, gold, platinum, and paladium. Silver is included among the constituents in the ore models presented here. The other metals, however, are present in such low concentrations in the mineralized samples that they are measured very imprecisely with available analytical techniques, or are below detection limits all together. Consequently, it is more useful to model the precious metals content of the concentrate produced in the processing step, based on analyses of actual bench scale and pilot plant concentrates. Accordingly, this is discussed further in section 2.4.1, where the model concentrate is presented.

2.2.2 Ore Mineralogy

The mineralogy of the model ores and other samples presented in the following sections is based on the average disseminated sample mineralogy shown in Table 9 and extrapolated to fit the amount of sulfide minerals present in the samples.

Table 9

The mineralogy of the 2 model ores shown in Table 9 was extrapolated from the average disseminated ore by adjusting the total sulfide content to conform, proportionally, to the amount of copper shown in Table 6. The basic assumption is

Table 8.   Representative concentrations of trace elements in Duluth
           Complex resource material.[a]

ELEMENT	ORE CONCENTRATION (ppm by weight)
F	12
Cl	420
Au	45
Sb	0.7
Sn	0.5
W	0.5

[a]The values shown are selected as being representative of
the small amount of data available. Actual concentrations may be
expected to vary locally by as much as 1 to 2 orders of magnitude
from these values.

Table 9. Ore mineralogy[a].

MINERAL	AVERAGE[b] DISSEMINATED ORE	MODEL OPEN PIT ORE	MODEL UNDERGROUND ORE	AVERAGE[b,c] SEMI-MASSIVE ORE
Plagioclase	58.786	58.838	58.112	47.519
Sericite	1.320	1.321	1.305	0.079
Olivine	17.883	17.899	17.678	6.589
Clinopyroxene	5.689	5.694	5.624	15.518
Orthopyroxene	1.517	1.518	1.500	9.609
Monocrystalline amphibole	2.471	2.473	2.443	0.011
Fibrous amphibole	0.366	0.366	0.362	0.011
Chlorite	2.202	2.204	2.177	0.287
Serpentine	1.498	1.499	1.481	0.008
Iddingsite	0.090	0.090	0.089	0.029
Talc	0.087	0.087	0.086	---
Biotite	2.624	2.626	2.594	4.780
Smectite	0.031	0.031	0.031	---
Opaques	4.592	4.504	5.682	13.031
Chalcopyrite-cubanite[d]	1.305	1.258	1.916	2.194
Pentlandite[d]	0.069	0.067	0.101	0.260
Pyrrhotite	1.017	0.980	1.493	7.471
Ilmenite-magnetite[d]	2.197	2.199	2.172	2.882
Graphite[d]	0.003	0.003	0.003	0.225
Spinel	0.001	0.001	0.001	---
Myrmekite	0.084	0.084	0.083	---
Apatite	0.134	0.134	0.132	0.094
Epidote	0.285	0.285	0.282	---
Allanite	0.025	0.025	0.025	---
Calcite	0.042	0.042	0.042	0.003
Quartz	---	---	---	0.017
Cordierite	0.272	0.272	0.269	2.415

[a]Values are given in percent composition by volume.
[b]See section 1.4.3.1 of Chapter 1.
[c]Percentages based on DP9002 and AX9004 mineralogy.
[d]Values are included in the total for opaques.

that the open pit ore, underground ore, lean ore, and waste rock are on a continuous grade curve such that the total amount of sulfides present may change but not the relative proportions of the sulfide minerals. This, of course, means that the non-sulfide mineral content increases or decreases proportionally as the amount of sulfide minerals change. Each of the model ore compositions just presented and the semi-massive ore average will now be briefly discussed.

2.2.3 Open Pit Disseminated Ore Model

2.2.3.1 Mineralogy--Since the copper content of the model open pit ore is close to that of the average disseminated material, their mineralogies are also similar. The sulfide mineral content for the model is derived by scaling the disseminated average sulfide content downward. The scale factor used is the ratio of the copper contents of these 2 materials, taken from Table 9. The ratio is 0.906, indicating a decrease of about 10% in sulfide content. In terms of overall material composition, this is a decrease of only 0.086% by volume for the sulfide content, and it is offset by a corresponding increase in the non-sulfide minerals content.

2.2.3.2 Chemistry--The chemistry of the model ore for the open pit is shown in Table 6 and is plotted against the average disseminated ore in Figure 8. The figure illustrates that since the Cu value (0.545%) found for the disseminated average and the modeled value of 0.494% Cu in the open pit model ore are close, the plots of the 2 analyses are very similar, especially in regard to the non-sulfur related elements (Si through Zn). There is only a small departure from the average disseminated composition in the sulfide-related elements. Note that with a lower copper value the related elements also are lower than the disseminated average. This is a result of the assumption that the interelement ratios

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of sulfide-related elements remains constant. In a statistical analysis of the chemistries of the 91 drill core samples, no interelement trend was found to negate this assumption.

2.2.3.3 <u>Physical Characteristics</u>--It is important to realize that the bulk of the ore being modeled is composed of non-sulfide minerals. This is clearly seen from the chemistry data which indicates that 97.0% by weight of the open pit ore model consists of constituents that are present in the non-sulfide minerals. This is to be compared with a value of 96.9% for the disseminated average. As a result of this small difference, the macroscopic physical properties of these two materials would be virtually identical. The model ore, therefore, has a density of 3.07 gm/cm^3, and an unconfined compressive strength around 30,000 psi. Fracture density is 4 to 5/10-ft interval (Van Eeckhout and Gerken 1977). As a result, the rock is quite competent and should be able to sustain pit slopes of 45° or greater. The modeled amphibole content of this ore is 2.84% by volume.

2.2.4 <u>Underground Disseminated Ore Model</u>

2.2.4.1 <u>Mineralogy</u>--The mineralogy of the underground disseminated ore model is shown in Table 9. The model is again based on the disseminated average mineralogy from Table 21 of Chapter 1. The sulfide mineral content has been adjusted upward by the ratio of the copper content of the 2 materials (1.47) as shown in Table 6. The non-sulfide mineral content is then scaled down proportionally to balance the increase. The basic assumption is that the underground model ore, as mentioned earlier, is on a continuous grade curve with the average disseminated ore.

Since the Cu content of the model underground ore is almost 50% higher (0.800% Cu) than that of the disseminated average (0.545% Cu), the amount of sulfide

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minerals (chalcopyrite, cubanite, pentlandite, and pyrrhotite) increases by the same amount (neglecting small differences in density). Although the relative increase is large for the sulfide minerals, the net effect on the non-sulfide minerals is much smaller (1.1% decrease). This means that the net effect of modeling an underground ore at 0.800% Cu is relatively large with respect to the sulfide minerals but almost negligible with respect to the non-sulfide minerals.

2.2.4.2 Chemistry--Because of the small change in non-sulfide elements (Si through Zn), the plot of the underground model chemistry versus the disseminated average is so close to both the disseminated average and the open pit model that they nearly plot on top of one another in Figure 8. The sulfur-related elements, however, because of the nearly 50% increase from the disseminated average, are shown as being considerably higher in amount than the disseminated average. As was explained above with the open pit model chemistry, the evenness of the sulfur-related element portion of the plot is due to the assumption of constant interelement ratios. In reality, this assumption may not hold true. For example, as discussed earlier, available information indicates that the Cu/Ni ratio varies both locally and from one resource zone to another. This certainly could have important economic implications. However, no data is available to indicate systematic variations that could be incorporated in the model, necessitating the simplifying assumption used here. This caution should be born in mind in applying this model to specific situations.

2.2.4.3 Physical Characteristics--As discussed with the open pit model, the physical characteristics of this model material are dominated by the non-sulfide minerals. The distribution of these are quite close to those of the disseminated average, if not identical. The density is essentially unchanged, at 3.07 gm/cm^3. The model contains 2.81% amphibole by volume. Fracture density is 4

to 5/10-ft interval. A USBM report (Van Eeckhout and Gerken 1977) notes that available data indicates that during caving this material might typically form blocks as cubes modeled to be 5.24 ft on a side. The report quotes a study which concludes that 50% or more of the ore in a mine must cave in blocks of 5 ft or less on a side in order for block caving to be economical. Further, this material has an uncontained compressive strength of about 30,000 psi while block caving is thought to require 2,000 to 5,000 psi (or less) in the absence of adequate fracture frequency. Therefore, present indications are that this ore is marginal in terms of cavability and block caving is not likely to be a preferred mining method. On the contrary, the relatively high strength of this rock would aid in making large stopes and small pillars, and should allow for fairly high ore recovery. Promising mining methods in this context include sublevel stoping in the steeply dipping zones, and room-and-pillar in the flatter areas.

These rock characteristics have important environmental implications in terms of subsidence. Given the fact that underground mining is likely to be employed at depths below 1,000 ft, and with the rock competence noted above, it is reasonable to expect that surface subsidence can be avoided by proper mine planning. There will be no major areas of subsidence as is seen, for example, in many operations in the southwestern United States or in the abandoned Pioneer Mine near Ely, Minnesota. A cautionary note must be added, however. Existing resource data indicates the possibility of mineable resources in areas which are not amenable to open pit methods. A prime example of this is the area under Birch Lake. If underground mining is employed in such an area, there would be an interest in extracting resources closer to the surface than 1,000 ft. This would also be the case if open pit mining is not allowed, but underground methods are permitted and are used to extract minerals which would normally be removed by

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open pit mining. In such cases, care would have to be taken to avoid the possibility of surface subsidence following mining, unless such subsidence is deemed acceptable.

In addition, if a high density of fracture zones is encountered in such a situation, not only would the possibility of subsidence increase, but water infiltration from the overlying lake or other surficial and groundwater system could greatly exceed estimates based on data available to date. This water may be of poor quality and create discharge problems. Just such an occurrence posed serious problems at INCO's operation at Shebandowan in Canada where nickel is being mined beneath a lake. Clearly, detailed drilling and mine planning in such a situation would be of particular importance from the perspective of preventing these potential environmental impacts. However, in many cases, such conditions may pose risks which cannot be significantly reduced by additional studies.

2.2.5 Semi-Massive Ore Average

2.2.5.1 Mineralogy--The mineralogy of the semi-massive ore is more difficult to calculate than the previous 2 models because of the lower number of samples of semi-massive ore that were characterized by the Minnesota Geologic Survey. Only 2 samples (DP9002 and AX9004) were characterized. With such a low number, reliability of the average is a problem.

The semi-massive average departs from the open pit and underground disseminated models in that it is not an extrapolation from the disseminated average analysis. Instead, it specifically illustrates the changes in ratios of various constituents, particularly among the sulfide minerals. The semi-massive ore mineralogy shown in Table 9 was calculated from DP9002 and AX9004 by calculating a point between the 2 samples at which the Cu value matched that of the average

chemistry shown in Table 6. Recall that this chemistry is based on 4 samples (DP9002, AX9004, AX9006, and AX9007). It is this chemistry that is used for the average and so it is reasonable that the mineralogy shown in Table 9 be calculated in part from this chemistry.

As was discussed earlier (section 1.4.3.1 of Chapter 1), the major differences between the disseminated samples and the semi-massive samples is the relative decrease in alteration products (especially amphibole), plagioclase, and olivine, and the relative increase in graphite, cordierite, biotite, and pyroxene (especially orthopyroxene). There also is a substantial increase in sulfide minerals, especially pyrrhotite. In relation to the increase in pyrrhotite and other sulfides, the ratio of the content of copper-bearing sulfide minerals (chalcopyrite and cubanite) to pyrrhotite goes from close to 1 in the disseminated samples to less than 0.5 in the semi-massive samples. The semi-massive average has a ratio of 0.29.

The increase in graphite, biotite, and cordierite in the semi-massive average is due to a higher number of inclusions of Virginia Formation. This increase in inclusions increases the amount of pyroxene, especially orthopyroxene, and decreases the amount of olivine present.

2.2.5.2 Chemistry--The average semi-massive ore chemistry is shown in Table 6 and is plotted against the disseminated average feed in Figure 8. The composition of the 4 samples that make up these values range in sulfur-related elements, from 3.88% to 10.50% S, 0.782% to 6.030% Cu, and 0.178% to 0.462% Ni (see Table 23 of Chapter 1). This is a larger range than for the disseminated samples. The AMAX semi-massive estimate shown in Table 5 is based on over 10 times the number of analyses used in this study. Note the very close agreement

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between the AMAX numbers and the average semi-massive feed, the Cu value is especially close. Therefore, the chemistry of the semi-massive average is considered to be a good model of the semi-massive ore present in the Duluth Complex.

As was discussed in section 1.4.3, the differences in chemistry shown in Figure 25 of Chapter 1 are characteristic of semi-massive samples. The Cu/Ni ratio of the semi-massive average (8.96) is higher than that for the disseminated average (4.36) even though the contents of both Cu and Ni are higher. Sulfur, Fe (that calculated to be in sulfide phases), Co, Zn, and Ag are also higher in concentration in the semi-massive than in the average disseminated ore.

Aluminum, Mg, Ca, and Na are lower than in the disseminated average, while K is higher (see section 1.4.3 for a more complete discussion).

2.2.5.3 Physical Characteristics--The total amount of amphibole present in the semi-massive average is 0.022% by volume, much less than the disseminated average. Because of the lower amount of alteration minerals (0.428% by volume versus 8.423% by volume for the disseminated average), the rock should be tougher. However, available data (Van Eeckhout and Gerken 1977) indicates the material is substantially weaker than the disseminated ore. This may be due to mechanical weakening of the rock structure resulting from the presence of the sulfide minerals themselves. Whatever the cause, if the semi-massive material is consistently less competent than the disseminated and waste material, alterations in the mining methods mentioned previously may be required. Sublevel caving might be applicable in these areas. In any event, the occurrence of semi-massive ore appears to be on a small scale, only 4.8×10^6 mt estimated, and all at the Minnamax site. This amounts to roughly 0.1% of the total resources described by Listerud and Meineke (1977).

The sulfide minerals typically have a density in the range of 4 gm/cm^3. As a result of the increased sulfide content, the density of the semi-massive average ore is 3.25 gm/cm^3. Contrasted with the disseminated average density of 3.07 gm/cm^3, this is a 5.6% increase. This increase is probably not large enough to pose problems such as overloading transport systems designed for the lighter disseminated ore, since safety margins of 10% to 20% are typically used by mining equipment designers.

2.3   MINE SOLID WASTES

All the raw material moved during the course of mining can be divided into one of 2 basic categories, ore or waste. The extraction of quantities of ore for further processing is the objective of the entire mining operation, and in the process a variety of wastes are generated. Broadly defined, mine solid wastes consist of a geological material, whether surficial or bedrock, which must be moved to allow access to the ore, but which does not contain the valuable metals, in this case nickel and copper, in sufficient concentrations to allow profitable extraction. The waste category can be further divided into 3 classes:

1) Overburden 2) Waste rock 3) Lean ore

Overburden consists of any surficial material, possibly including soil, overlying the bedrock in areas to be mined by open pit methods. This material must be moved aside to provide access to the bedrock below. The remaining 2 classes of wastes both consist of bedrock (or host rock) material of insufficient grade to qualify as ore. To understand the distinction between waste rock and lean ore, it is useful to recall some of the concepts used in defining ore. Any known mineral reserve is defined both in terms of the number of tons of material and the average grade of the material. These quantities are in turn dependent on the

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cutoff grade chosen. Material with a metal content below the selected cutoff grade is rejected as waste. The determination of the correct cutoff grade, average grade, and reserve tonnage is a complex process involving evaluation of detailed information on the geology of the deposit and the costs of appropriate recovery and processing methods. To illustrate these variables, models were developed and presented in the technical assessment report, Volume 2-Chapter 5. The average grades and cutoff grades used there vary according to whether the mine is open pit or underground. The values used are:

MINE TYPE	AVERAGE GRADE (% Cu)	CUTOFF GRADE (% Cu)
Open Pit	0.494	0.25
Underground	0.800	0.60

The higher average and cutoff grades for the underground mine are reflections of the higher costs associated with producing a given tonnage of material using underground methods rather than open pit methods.

The concept of a cutoff grade provides an easy definition for mine wastes. However, it is important to realize that the major factors in determining the specific value used include worldwide economic conditions affecting the supply and demand for copper and nickel, as well as technical developments that affect the costs of mineral recovery and processing. The worldwide conditions are clearly beyond the control of the operators of any one mine; however, technical developments include site specific improvements which do depend to some extent on individual mine operations. Both conditions change with time and, as a result, the appropriate cutoff grade, and corresponding average grade and reserve tonnages, will also change. The trend worldwide is towards a decrease in cutoff grades as higher grade deposits are exhausted and extraction techniques improve. This is reflected in proper mine planning by segregating waste material of a

grade reasonably close to the current cutoff grade from material substantially below the cutoff. The former waste is termed lean ore, and the latter is then waste rock. The 2 are segregated into separate piles with the expectation that the lean ore piles may be treatable as ore at some point during the life of the mill.

The above discussion illustrates the concepts behind the lean ore-waste rock distinction. It also illustrates the complexity of the quantification of grade distinctions between these 2 classes, not to mention estimates of the tonnages of each type of material to be produced by a given mine. These must be determined by a detailed study of each proposed mine development. For example, it is apparent that much of the material defined as lean ore for the model underground mine used here would qualify as ore for the model open pit mine. Further, the ability to distinguish between the 2 classes of wastes is ultimately limited by the technology used to move the material. For example, open pit wastes are only able to be segregated with the fine degree of selectivity afforded by the 15 to 18 yd^3 bucket of an electric shovel. As a result, there is no attempt here to specifically model both the tonnages and grades for waste rock and lean ore piles likely to be generated by copper-nickel mining. However, the fact that such piles will exist is acknowledged by developing 2 models having successively lower copper, and thus sulfide, content. These models are useful not merely to illustrate the economic potential of lean ore piles, but to facilitate environmental impact assessment as well. Since lean ore storage is potentially less permanent than waste rock storage, it may be appropriate to consider different reclamation methods for these piles. Further, the higher sulfide content of the lean ore piles may imply the potential for increased release of heavy metals due to leaching. Correspondingly, increased measures to protect the

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water quality in the area may be indicated. These topics are discussed in the mineral processing section, Volume 2-Chapter 3, and in the water resources report, Chapter 4 of this volume. Each of the 3 classes of wastes introduced above will now be discussed in turn.

2.3.1 Overburden

The classification of overburden as a mine waste assumes first that overburden disposal will in fact be required, which is only true if open pit mining methods are employed. Second, the quantity of overburden to be disposed of must exceed the constructive requirements for this material which are created by the mining operation itself. Only if these 2 assumptions are met will overburden actually become a waste. Accordingly, this section will focus on a semi-quantitative assessment of the types and amounts of overburden likely to be generated as a solid waste from the development of a copper-nickel mining industry in the Study Area.

Overburden is defined as all surficial material overlying bedrock which must be removed during the course of open pit mining. This material may result from weathering of the country rock or deposition by forces such as wind, water, and glaciers. Within the Study Area, the surficial material is derived almost exclusively from glacial action, and includes unsorted tills, more sorted glaciofluvial deposits, and soils (see section 1.3).

Overburden, consisting of solum, organic deposits, glaciofluvial materials, and till, may be regarded either as a resource or as a waste product, depending on needs and availability of materials at the site of a mining operations. The solum, or soil layer, is an important resource in the context of reclamation. It differs from the parent material in its higher proportion of organics and

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nutrients, but especially in the presence of micro-organisms and seeds.
Topdressing with soil is, therefore, more effective for revegetation purposes
than merely topdressing with till (see Volume 2-Chapter 2). Because the depth of
the solum is less than that of parent materials, the supply of soil is likely to
be less in all parts of the Study Area than the demand for it by the reclamation
needs of a copper-nickel mining operation. Organic deposits (peat) can be used
as pads for waste rock piles and as part of the mix in topdressings. Although
the pH levels of peats vary depending on their origin, some are sufficiently
neutral to allow their use as absorbents for heavy metals (see Volume 3-Chapter
4). Both glaciofluvial sands and gravels and till can be used as construction
materials (see section 1.3.3). Their usefulness is dependent on engineering and
design criteria at a specific site. The geographic distribution of various types
of surficial material is illustrated on the map in Figure 11, section 1.3.3.

A model overburden budget was prepared in order to asses the relationship between
supply and demand for materials in each of the 7 development zones. Available
overburden was estimated by calculations using available data on maximum soil
depth for the dominant soil association in the zone (USDA 1973), and an average
of measured drill hole depths for till, glaciofluvial materials, and organic
deposits within each zone (Olcott and Siegel 1978). For the purpose of
calculating the available volume (in m^3), it was assumed for each case that the
open pit was excavated entirely in the material in question, except that the
surface soil layer was subtracted from the depths of till and glacio-fluvial
deposits. Estimates for volumes of peat may be excessive because the available
data did not record the depth at which boreholes drilled in peat intercepted
other surficial materials such as till or sands. Depth of each type of material
in a given zone was multiplied by the modeled area of the open pit (in m^2, see

Volume 2-Chapter 5) to calculate the potential volume of material available from pit excavation. The results are presented in Table 10. It should be noted that the figures in the table are based on different numbers of drill holes in each zone and that glacial materials are extremely variable in depth. Therefore, the table can only serve as a guide to indicate which types of materials may be in short supply or in excess in the various zones.

Table 10

Table 11 presents a summary of modeled construction and reclamation needs for overburden. On the basis of discussions presented in Volume 2-Chapter 2 and in section 1.3.3 of this chapter, it is assumed that sands and gravels will be preferred over tills for construction purposes and that soils, then tills and peats will be preferred over sands and gravels for reclamation purposes.

Table 11

A comparison of available overburden from pit excavation (Table 10) with needs (Table 11) on a zone-by-zone basis aids in defining geographic areas in which all available overburden is likely to be a useful resource and those in which excess overburden is likely to be a waste product. Figure 9 highlights these relationships for each of the 7 zones. The model does not take into account the actual area covered by each surficial material within each zone or the distribution of surficial materials within the zones. Inclusion of such information would only be meaningful in the context of site-specific studies.

Figure 9

Table 10. Available overburden for mine models in development zones.[a]

| | SOIL | | | AVAILABLE OVERBURDEN, $10^6 m^3$, BY MODEL | | | |
| | | | | 16.68 X 10^6 mtpy Combination 159.0 ha | | 20.00 X 10^6 mtpy Open Pit 211.7 ha | |
Zone	Type	Depth,m	Thickness,m				
1	Soil[b]	0.70	0.70	1.1		1.5	
	Sand/gravel[c]	0	0	0		0	
	Till[d]	4.73	4.03	4.5	11.4	8.5	26.3
	Peat[e]	3.66	3.66	5.8		7.8	
2	Soil	0.70	0.70	1.1		1.5	
	Sand/gravel	4.06	3.36	5.3		7.1	
	Till	3.05	2.35	3.7	17.1	5.0	22.9
	Peat	4.37	4.37	7.0		9.3	
3	Soil	0.70	0.70	1.1		1.5	
	Sand/gravel	13.80	13.10	20.8		27.7	
	Till	3.04	2.34	3.7	38.0	5.0	50.7
	Peat	7.80	7.80	12.4		16.5	
4	Soil	0.70	0.70	1.1		1.5	
	Sand/gravel	15.20	14.50	23.1		30.7	
	Till	3.05	2.35	3.7	42.0	5.0	55.9
	Peat	8.84	8.84	14.1		18.7	
5	Soil	1.28	1.28	2.0		2.7	
	Sand/gravel	0	0	0		0	
	Till	9.49	8.21	13.1	34.5	17.4	45.9
	Peat	12.19	12.19	19.4		25.8	
6	Soil	1.28	1.28	2.0		2.7	
	Sand/gravel	15.58	14.30	22.7		30.3	
	Till	4.78	3.50	5.6	45.8	7.4	61.0
	Peat	9.75	9.75	15.5		20.6	
7	Soil	1.28	1.28	2.0		2.7	
	Sand/gravel	3.05	1.77	2.8		3.8	
	Till	18.29	17.01	27.1	68.1	36.0	90.7
	Peat	22.76	22.76	36.2		48.2	

[a]Only models with open pit facilities are included.

[b]Soil depths are based on maximum depths of predominant soil associations within each zone.

[c]Depths of sand/gravel are based on average depths of all boreholes in glaciofluvial material within each zone, minus depth of the soil.

[d]Depths of till are based on average depths of all boreholes in till within each zone, minus the depth of the soil.

[e]Depths of peat are based on average depths of all boreholes in organic material within each zone and may be higher than actual peat depth because the data used to generate the map shown in Figure 11, section 1.3, does not indicate whether depths include underlying surficial materials such as till or sands and gravels.

Table 11. Total modeled overburden needs for construction and reclamation.

MODEL NEEDS[b]	MINE MODEL[a]		
	12.35 X 10⁶ mtpy Underground	16.68 X 10⁶ mtpy Combined	20.00 X 10⁶ mtpy Open Pit
Construction Materials			
Starter dam (sand, gravel, till)	0.38	0.46	0.54
Drain (waste rock, till, crushed rock)	1.53	1.84	2.06
Sub-total: construction	1.91	2.30	2.60
Reclamation Materials			
Topdressing for tailing basin (soil, peat, till)	1.40	1.99	2.44
Topdressing for waste rock/lean ore piles (soil, peat, till)	0.08	0.49	0.80
Topdressing for slag pile (soil, peat, till)	0.02	0.02	0.02
Sub-total: reclamation	1.50	2.50	3.28
Total Model Needs 10⁶m³	3.41	4.80	5.86

[a]See Volume 2-Chapters 2 and 5.
[b]See Chapter 1, section 1.3.3 and Volume 2-Chapter 2.

FIGURE 9

THE RELATIONSHIP BETWEEN OVERBURDEN SUPPLY
AND DEMAND IN THE SEVEN DEVELOPMENT ZONES

⊠ DEMAND GREATER THAN SUPPLY AVAILABLE
FROM OPEN PIT EXCAVATION

☐ DEMAND APPROXIMATELY EQUAL TO SUPPLY
AVAILABLE FROM OPEN PIT EXCAVATION

▦ SUPPLY FROM OPEN PIT EXCAVATION IN EXCESS OF DEMAND FOR
RECLAMATION OR (IN THE CASE OF SAND AND GRAVEL) FOR CONSTRUCTION

▨ SUPPLY FROM OPEN PIT EXCAVATION GREATLY IN EXCESS OF DEMAND
SO THAT OVERBURDEN MAY BE CONSIDERED A WASTE MATERIAL

TYPE OF OVERBURDEN AND MINE MODEL	DEVELOPMENT ZONES						
	1	2	3	4	5	6	7
12.35×10^6 MTPY UNDERGROUND							
SOIL							
SAND / GRAVEL	DOES NOT APPLY						
TILL							
PEAT							
16.68×10^6 MTPY COMBINED							
SOIL							
SAND / GRAVEL							
TILL							
PEAT							
20.00×10^6 MTPY OPEN PIT							
SOIL							
SAND / GRAVEL							
TILL							
PEAT							

The following assumptions were used in the construction of the table:

1) Total demand for soil is assumed to be equal to total demand for topdressing, which probably reflects the maximum demand for this material.

2) Demand for sand and gravel is assumed to be equal to total construction demand. This level of demand may overestimate requirements for dam construction but underestimates requirements for construction of haul roads and plant sites.

3) Demand for till is assumed to be equal to total demand for both construction and reclamation needs, which may overestimate demand because other materials may be preferred for both construction and reclamation.

4) Demand for peat is restricted to total demand for topdressing. This figure probably overestimates demand because peat is generally mixed with other materials in topdressing.

5) An underground mine would not produce overburden, therefore, construction and reclamation materials would have to be supplied from borrow pits.

Assumptions pertaining to supply are the same as those used to generate Table 10, section 2.3.1. It is assumed that the pit is excavated entirely in the material in question, except that the thickness of overlying soil is discounted from the thickness of till and sand/gravel before calculation of volume.

As can be seen from Figure 9 and Table 11, an underground mining operation, by definition, generates no overburden. Although this situation alleviates any cost of disposal of this material, it also leaves unfilled any needs for topdressing material for reclamation purposes. This problem is discussed in Volume 2-Chapter 2.

From the tables, it can be seen that disposal of the soil fraction of overburden should present no problem, because the supply from pit excavation is either exceeded by demand for topdressing or nearly equal to it in all zones. Reclamation considerations discussed in Volume 2-Chapter 2 may require the excavation of soil as a borrow material in zones 1-4 for a combined mine and in all zones for an open pit mine, if topsoil is used exclusively for all areas that require topdressing. However, the depth of sand and gravel deposits in all zones except 1 and 5 appears to be sufficient to meet construction needs if the pit is excavated in these materials. The restricted geographic distribution and small area of such deposits may limit their availability. This issue is discussed further in Volume 2-Chapter 2.

Both construction needs and reclamation needs can be met by till where other more suitable materials are unavailable. Adequate supplies of till to meet all needs are available in zones 5, 6, and 7 for both the combined and open pit models. As can be seen from Tables 10 and 11, the disposal of excess till as a waste is likely to be required in these zones. An excess of between 10 and 30 X 10^6 m^3 of till (20 to 60 X 10^6 mt) may occur in these zones. Till supplies from pit excavation in zone 3 seem to be inadequate to meet total needs for both the combined and open pit models. On the other hand, it appears that there should be little problem disposing of excess till in zones 1-4 because the excess is only about twice the total need. The difference could be made up by topdressing 2 to 3 times as deeply on all surfaces requiring topdressing.

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2.3.2 Waste Rock

The question of the various amounts of waste rock (and lean ore) likely to result
from Minnesota copper-nickel mining operations have been discussed, and modeled
elsewhere (Volume 3-Chapters 2 and 3). Accordingly, this section will focus
exclusively on a discussion of the likely composition, both mineralogical and
chemical, of these materials. Compositional models of waste rock and, later,
lean ore are given. These are important in subsequent impact assessments of the
potential environmental impacts which may result from the creation of these waste
materials.

2.3.2.1 Mineralogy--The mineralogy of the model waste rock (Table 12) is similar
to that of the disseminated average ore shown in Table 9. The model waste rock
composition was calculated by the same method used for the disseminated open pit
and underground ores (see section 2.2.3). The basis for this calculation is the
assumption that the copper content of the waste rock is 0.100%. A value of
0.100% Cu was chosen for the waste rock model because for an open pit ore cutoff
grade of 0.2% Cu, the actual value of the average waste rock copper content would
be below the midpoint of the range. This is because of the concave upward nature
of the copper grade distribution plot shown in Figure 4. By using 0.100% Cu, a
"worst case" can be modeled. In other words, assuming the tonnage distribution
vs. grade of material mined follows the curve shown in Figure 4, any waste rock
pile produced using a cutoff of 0.20% Cu or even a little higher (0.25% Cu in the
case of the open pit model) would not exceed an average copper content of 0.100%.
This material would constitute waste rock for both the open pit model and the
underground model mines.

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Table 12

Table 12. Model waste rock and lean ore mineralogy (percent by volume).

	MODEL WASTE ROCK	MODEL LEAN ORE	LEAN ORE[a]
Plagioclase	59.962	59.417	59.112
Sericite	1.346	1.334	2.176
Olivine	18.241	18.075	10.510
Clinopyroxene	5.803	5.750	11.185
Orthopyroxene	1.547	1.533	3.716
Monocrystalline amphibole	2.520	2.498	3.567
Fibrous amphibole	0.373	0.370	0.288
Chlorite	2.246	2.226	1.136
Serpentine	1.528	1.514	0.257
Iddingsite	0.092	0.091	0.075
Talc	0.089	0.088	---
Biotite	2.676	2.652	1.738
Smectite	0.032	0.031	0.021
Celadonite	---	---	---
Opaques	2.683	3.567	5.098
Chalcopyrite-cubanite[b]	0.239	0.733	0.769
Pentlandite[b]	0.013	0.039	0.037
Pyrrhotite[b]	0.187	0.571	0.844
Ilmenite-magnetite[b]	2.241	2.221	3.447
Graphite[b]	0.003	0.003	---
Spinel	0.001	0.001	---
Myrmekite	0.086	0.085	---
Apatite	0.137	0.135	0.085
Epidote	0.291	0.288	0.953
Allanite	0.025	0.025	---
Calcite	0.043	0.042	0.056
Quartz	---	---	---
Cordierite	0.277	0.272	0.027

[a] Lean ore is represented by data from sample AX9001.
[b] Values are included in the figure for opaques.

With 0.100% Cu as a basis and extrapolating from the sulfide mineral content of

the disseminated average (Table 9), the model waste rock contains 0.439% by

weight sulfide minerals with the same internal proportions as that shown for the

disseminated average (disregarding rounding errors). This 82% decrease in

sulfide content results in a corresponding increase of only 2.0% by weight for

the non-sulfide minerals. Therefore, as was seen with the calculated mineralogy

of the disseminated open pit and underground models, although the amount of con-

tained sulfide minerals changes drastically, the change in the other minerals is

very small.

2.3.2.2 Chemistry--The chemistry of the model waste rock is shown in Table 13

and is plotted against the average disseminate ore in Figure 10. The figure

illustrates that since the Cu value (0.100%) and the other sulfur-related ele-

ments (S thru Mo) are all proportionately lower than the disseminated average,

their plot is a straight line well below the disseminated average comparison line

(unsymboled straight line at $\log_{10}$ of 0.0). Also note that because of the

relatively small change in the non-sulfur related elements (Si thru Zr) there is

no large displacement from the disseminated average reference line. Phosphorus

and Cr, as on previous plots of this type, are below the disseminated average

because of rounding errors only. The other non-sulfur related elements plot

above the reference line. As in the cases of the open pit and underground ore

models, since no available data indicates systematic variations of interelement

ratios with a change in Cu content, the simplifying assumption of proportionality

is used. This assumption should be born in mind in applying this model to

specific situations.

Table 13, Figure 10

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Table 13. Model waste rock and lean ore chemistry.

ELEMENT	WASTE ROCK MODEL	LEAN ORE MODEL	AX9001 LEAN ORE SAMPLE[a]	UNITS[b]
$Si(SiO_2)$	22.86(48.95)	22.66(48.50)	22.80(48.81)	PCT
$Al(Al_2O_3)$	9.45(17.86)	9.36(17.69)	8.98(16.96)	PCT
$Fe(FeO)$	8.92(11.47)	8.83(11.36)	9.22(11.86)	PCT
$Mg(MgO)$	4.66(7.72)	4.62(7.65)	4.31(7.15)	PCT
$Ca(CaO)$	5.67(7.94)	5.62(7.87)	5.55(7.76)	PCT
$Na(Na_2O)$	2.24(3.02)	2.22(2.74)	2.00(2.70)	PCT
$K(K_2O)$	0.36(0.43)	0.35(0.43)	0.32(0.39)	PCT
$Ti(TiO_2)$	0.94(1.57)	0.93(1.55)	1.41(2.35)	PCT
$P(P_2O_5)$	0.03(0.08)	0.03(0.08)	0.02(0.05)	PCT
$Mn(MnO)$	0.13(0.16)	0.12(0.16)	0.12(0.16)	PCT
$Cr(Cr_2O_3)$	0.03(0.05)	0.03(0.05)	0.03(0.05)	PCT
B	586.0	578.6	662.0	PPM
Ba	722.6	713.4	1173.0	PPM
Be	0.6	0.5	1.0	PPM
Sr	284.9	281.2	279.0	PPM
V	170.5	168.3	276.5	PPM
Th	4.5	4.4	6.40	PPM
Zr	98.6	97.3	80.0	PPM
S	0.207	0.634	0.655	PCT
Cu	0.100	0.306	0.306	PCT
Ni	0.023	0.070	0.073	PCT
$Fe(S)$	0.229	0.701	0.696	PCT
Co	0.002	0.007	0.009	PCT
Zn	25.5	78.0	157.0	PPM
Pb	1.0	3.0	0[c]	PPM
Ag	0.5	1.5	1.35	PPM
As	1.8	5.6	0	PPM
Hg	0.015	0.045	-1[d]	PPM
Mo	0.3	0.9	0.5	PPM
Cd	1.8	5.6	0	PPM

[a]Analysis also in Table 23, Chapter 1.
[b]PCT=weight percent
 PPM=parts per million by weight.
[c]0=below detection limits.
[d]-1=not analyzed.

FIGURE 10

COMPARISON OF WASTE ROCK AND LEAN
ORE MODELS TO DISSEMINATED AVERAGE ORE

LEGEND

△ LEAN ORE
⊙ WASTE ROCK
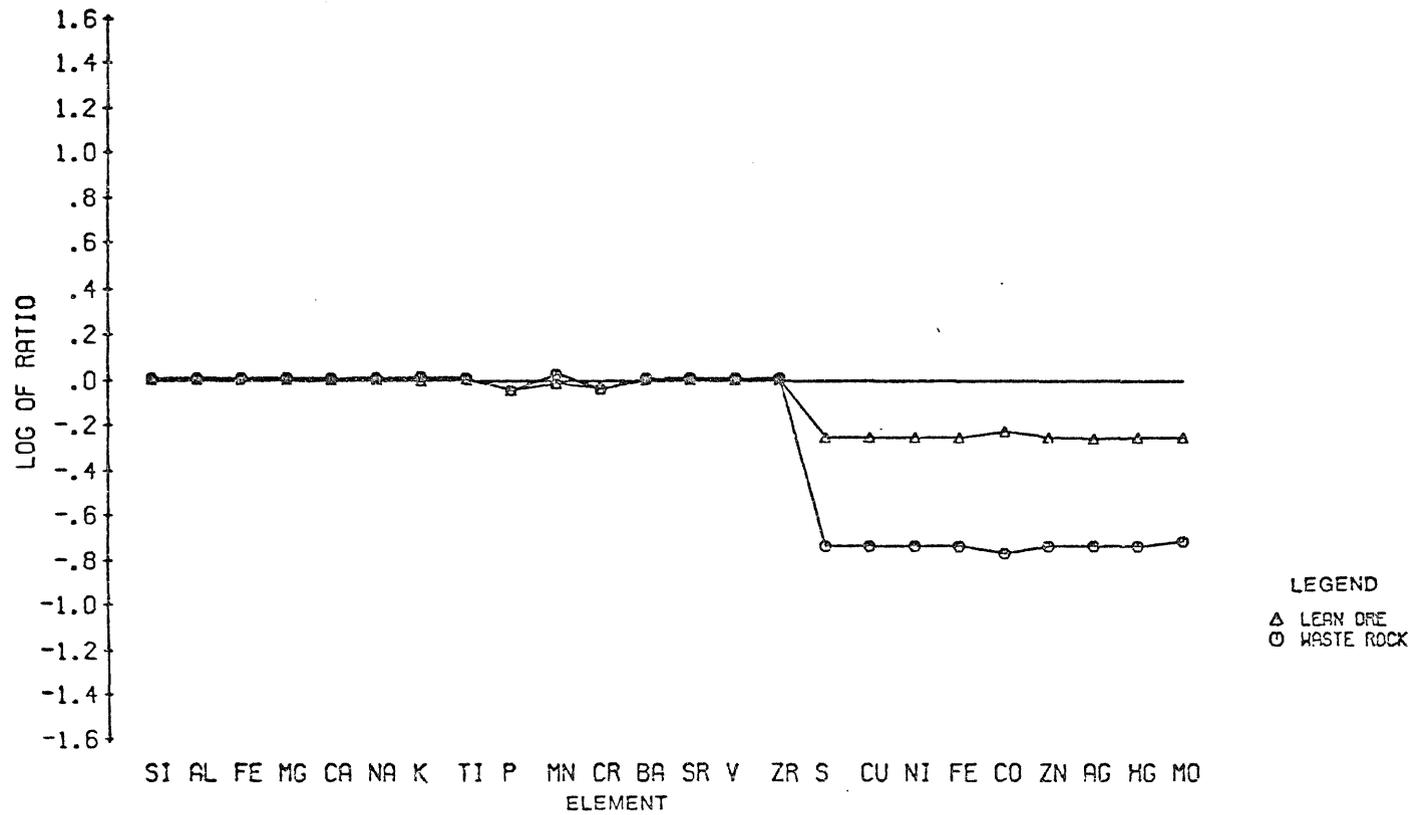
2.3.2.3 Physical Characteristics—As with the open pit and underground model

ores, the bulk of the model waste rock is composed of non-sulfide minerals (Table

12). This is also shown by the chemistry which indicates that 99.6% by weight of

the model waste rock consists of constituents that are present in the non-sulfide

minerals. The value for the disseminated average ore is 96.9% by weight. As a

result of this small difference, the macroscopic physical properties of these 2

materials would be virtually identical. Based on this small difference, the

waste rock model has a density of 3.0 to 3.1 gm/cm^3 and an unconfined

compressive strength around 30,000 psi. As with the open pit model ore, the rock

is quite competent and should be able to sustain pit slopes of 45° or greater.

The modeled amphibole content is 2.89% by volume.

Waste rock encountered in an underground mine would be bypassed if possible, so

the major source of waste rock would be from an open pit mine, from which it is

removed and piled on a waste rock pile. Depending on the method of transport

employed (truck, conveyor, or skip), the waste could require crushing. To get

semi-quantitative numbers of the size distribution of waste rock material, the

values 4 ft (1.22 m), 2 ft (0.61 m), and 8 in. (0.203 m) can be used as reaso-

nable top sizes for material from an open pit, underground, and conveyable

(primary crusher) material from either an open pit or underground mine, respec-

tively. These will be used in conjunction with Figure 11 (Lapakko 1978) to

estimate mine waste size distributions. Figure 11 represents the best available

method of making sizing estimations for blasted and crushed products. By using a

value of 30 um as the upper size limit (see Air Quality-Impacts section, Chapter

3) and Figure 11, an estimate can be made of the weight percent of air

suspendable particulates present in the 3 blasting-crushing processes. This

model results in values of 0.11, 0.17, and 0.35% by weight for the open pit,

underground, and primary crusher, respectively.

Figure 11

Another point of interest that is dependent on the size distribution of the waste rock is the leachability of the material (i.e. Cu, Ni, S, etc.). As the material is blasted or ground finer, the surface area per unit weight increases, thus exposing more surface to leaching. For a further discussion on the effect of the size distribution on the chemistry of possible leach water (Hoffman, Eisenreich 1979).

2.3.3 Lean Ore

2.3.3.1 Mineralogy--The calculated mineralogy of the lean ore model is also shown in Table 13. For comparison, the calculated mineralogy is shown next to the mineralogy determined by MGS for an underground mine lean ore sample taken from AMAX (AX9001). Note that, as expected, the amount of sulfides present are between the disseminated ore average and the waste rock. The lean ore model mineralogy was calculated from the disseminated ore average by the same technique used for the waste rock model, but here using a Cu value of 0.306% by weight. The value of 0.306% Cu was used merely as a convenient reference, because that was the amount of Cu in the underground lean ore sample (AX9001, see section 1.4.3.2). Note that for the model open pit mine, this material qualifies easily as ore, illustrating the impossibility of defining a unique lean ore model applicable to all situations. The observed mineralogy of sample AX9001 is also shown in Table 12. The similarity of values for the sulfide minerals does indicate the validity of the calculation technique of scaling the content of a fixed mixture of sulfide minerals. The non-sulfide mineral differences between

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FIGURE 11

PERCENT PASSING VS PARTICLE SIZE

▲ : ACTUAL DATA, CONE CRUSHER

● : ACTUAL DATA, JAW CRUSHER
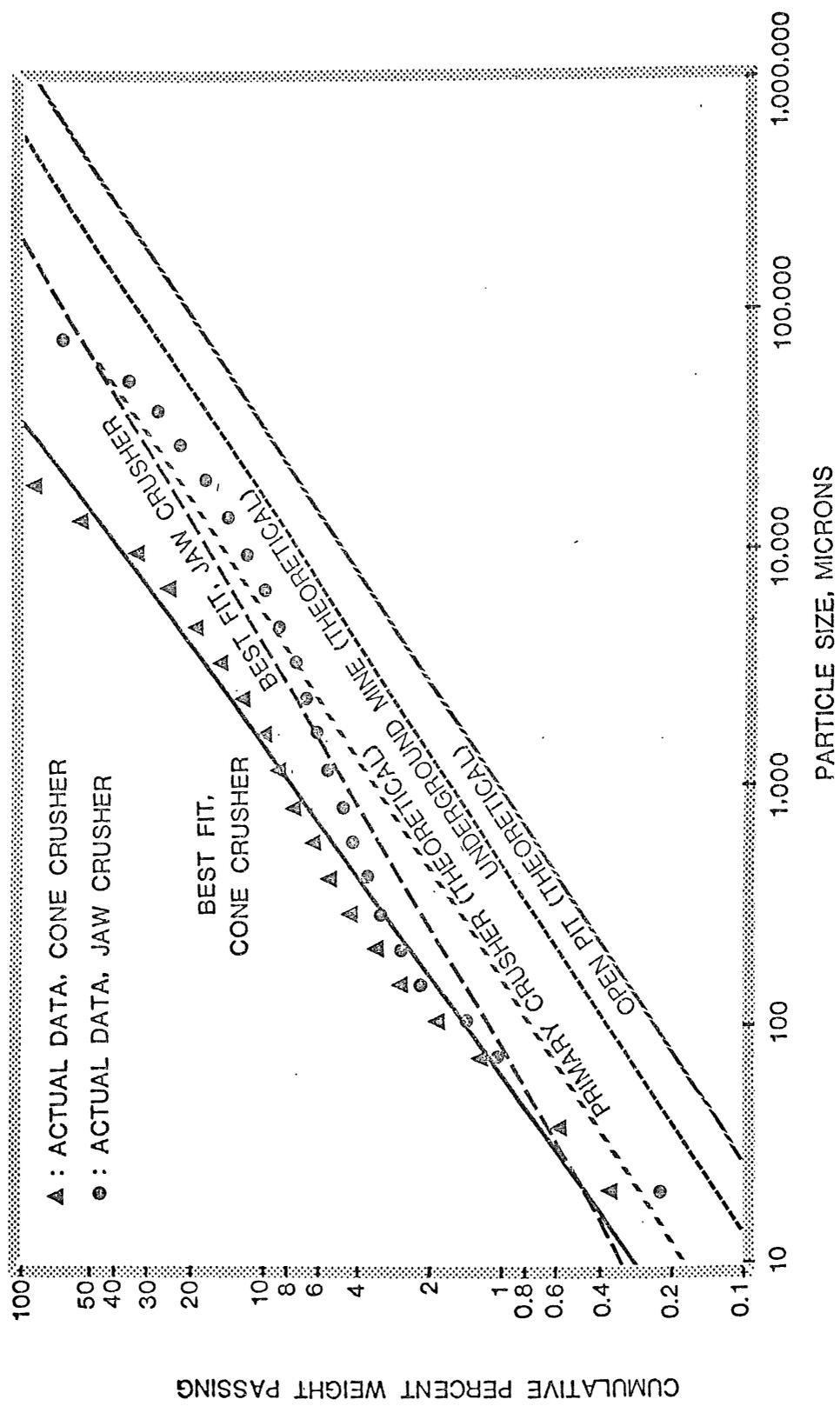
SOURCE : LAPAKKO, 1978

AX9001 and the model lean ore are most pronounced in the amounts of olivine and clinopyroxene that are present. Sample AX9001 is not used as the model lean ore because the model lean ore has to fulfill, as do the other models, the requirement of being non-local, i.e. the models are intended to represent the entire range of host rocks in the Study Area for the sulfide minerals. As was discussed in section 2.2, the sulfur and related elements are gradational in amount; and as was seen in section 1.4.1, the content of sulfides in a host rock is independent of host rock type. This is why the Cu content of AX9001 was used as a reference, but not its complete mineralogy.

The model lean ore contains 1.343% by weight sulfide minerals with the same proportions as shown for the disseminated average (disregarding rounding errors). This 44% decrease in sulfide mineralogy from the average disseminated ore results in a proportional increase of 1.1% by weight for the non-sulfide minerals. Therefore, as with the other model ores and the model waste rock, the change in non-sulfide mineral content is small compared to the sulfide mineral content.

2.3.3.2 Chemistry--The chemistry of the model lean ore is shown in Table 13 and is plotted against the average disseminated ore in Figure 10. The figure illustrates that since the Cu value (0.306% Cu) and the other sulfur-related elements (S thru Mo) are all proportionally lower than the disseminated average, their plot is a straight line below the disseminated average comparison line. Also note that because of the relatively small change in the non-sulfur related elements (Si thru Zr), there is no large displacement from the disseminated average reference line. As in the cases of the open pit, underground, and waste rock models, since no available data indicates systematic variations with a change in Cu content of the elements, the simplifying assumption of proportionality is used.

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An examination of the value of Cu, S, Ni, Fe(S), and Co in Table 13 for both the model lean ore as well as the one sample of underground lean ore (AX9001) shows that for these elements the simplifying assumption of proportionality is good. The most important difference is with Zn. Using the simplifying assumption, the model Zn value is half that of the lean ore sample. This shows that care should be taken in regard to the values presented as modeled constituents. As a result of local variation, real sample values can be significantly higher or lower than the modeled values, especially when only trace amounts of the elements are involved.

Elements such as Zn are important from a leaching standpoint and this indicates the difference between lean ore and waste rock in a mining operation. The 2 types of material removed from the mine but not immediately processed (waste rock and lean ore) should be treated separately as is indicated by the amounts of potentially leachable elements (such as Zn) present in the models of these 2 materials. See the discussion on leaching in Chapter 4 for the importance of the modeled threefold increase of sulfur-related elements in the lean ore model compared to the waste rock model.

2.3.3.3 Physical Characteristics--As with the open pit and underground model ores, the bulk of the model lean ore is composed of non-sulfide minerals (Table 12). This is also shown by the chemistry which indicates that 98.9% by weight of the model lean ore consists of constituents that are present in the non-sulfide minerals. The value for the disseminated average ore is 96.9% by weight. As a result of this small difference, the macroscopic physical properties of these 2 materials would be virtually identical. Therefore, the lean ore model has a density of 3.0 to 3.1 gm/cm^3 and an unconfined compressive strength around 30,000 psi. The modeled amphibole content is 2.87% by volume.

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Lean ore would be treated as waste rock in an underground mine, and as such would be bypassed if possible. The lean ore would be removed from an open pit mine and segregated on a lean ore pile for possible future processing depending on changes in economics and technology. As with the waste rock, the potential for dust generation is present. Using Figure 11 and the top sizes of 4 ft, 2 ft, and 8 in., the model indicates that 0.11, 0.17, and 0.36% by weight, respectively, of the lean ore will be below the 30 um size used to characterize particles suspendable in winds of sufficient speeds (see Air Quality-Impacts section in Chapter 3).

2.3.4 Conclusion

An important comment must be made concerning the relative amounts of waste rock and lean ore which might be generated by any mining operation. Analytical data for the region as a whole, as indicated in Figure 4, shows a general exponential increase in tonnage of material with decreasing copper grade. Thus, it might be expected that if the cutoff grade used in a mining operation is, say, 0.2% Cu, then the average grade of the resulting composite waste rock/lean ore piles would be considerably below 0.1%. The use of 0.1% Cu would appear to be a conservatively high estimate for purposes of assessing potential air and water impacts. However, this involves applying a regional grade vs. tonnage rela- tionship based on widely-spaced samples, to the local scale on which an actual mining operation would take place. There the local mechanisms responsible for creating the sulfide deposit may have led to a tonnage vs. grade curve that departs radically from the smooth form implied by Figure 4. Data simply is not available to resolve this question. In fact, all that can be rigerously concluded is that, by definition, the average grades of both the waste rock and lean ore piles will be below the cutoff grade used for the mine.

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The sulfide mineral composition of the waste rock and lean ore piles is very important to the assessment of potential air and water impacts. On the one hand, the piles might have a total average sulfide content only slightly below that fixed as a result of the cutoff grade chosen. Such piles would be huge, in the case of an open pit mine, and may pose serious water pollution problems. On the other hand, the bulk of the waste may be essentially barren of sulfides, posing a greatly reduced threat of environmental impacts. In this case, the sulfide-bearing wastes would be a relatively small quantity, and might be easily segregated into lean ore piles and managed to prevent environmental impacts.

To resolve this question at the site of a potential mine, it is crucial that detailed drilling be carried out not only to determine the mineralogical and chemical composition of the potentially mineable ore, but of any waste rock or lean ore that might have to be stockpiled as well. This work should involve sampling on a fine enough scale to reliably model the arithmetic average and distribution of the sulfide content of the wastes. From such information plans could be made to optimize the segregation of wastes to both facilitate future access to lean ore, and minimize the potential for environmental impacts resulting from the presence of sulfides in the waste. Such a plan, for example, might include a series of piles of successively lower sulfide content requiring successively less stringent environmental safeguards, rather than simply 2 types of piles as modeled here.

2.4 MINERAL PROCESSING OUTPUTS

The principal products of the mineral processing operation fall into 2 categories, concentrate and tailing. Depending on whether bulk flotation is used, or a more sophisticated method such as selective or differential flotation, one or

more concentrates may be generated. These concentrates contain the bulk of the constituents of economic interest, and are sent to the smelter for further processing. The process wastes from the various concentration operations are combined to form the tailing waste. The tailing is made up almost exclusively of non-sulfide minerals on a total volume basis, and is generally not processed further. It is important in relation to disposal and related leaching and dust generation problems.

Another aspect of the mineral processing operation which is of concern for environmental reasons is the fiber generation process. The initial phases of mineral processing require crushing and grinding of the ore in order to physically liberate the sulfide minerals from the nonsulfide minerals in preparation for the flotation operation. This crushing and grinding may act to release any minerals in the ore which are present in an asbestiform habit. Further, and possibly more importantly, it may act to mechanically generate fiber particles, or cleavage fragments, from certain minerals present in the ore in a massive (non-asbestiform) habit. All fibers so generated will then be present in the tailing and/or concentrate and might eventually be released to the air or water. There is no clear information to allow the assessment of the special health impacts which may result from such release, but the possibility of a hazard clearly exists, warranting a close examining of this topic. Therefore, the question of fiber generation is discussed in a special section here, in addition to the following 2 sections which describe the general nature of the bulk concentrate and tailing likely to result from the processing of Duluth Gabbro copper-nickel ore. Models are also presented for the composition of the concentrate and tailing, to aid in subsequent impact assessment.

2.4.1 Concentrate

As in the discussion of Duluth Gabbro ore, lean ore, and waste rock, this discussion relies heavily on studies conducted for the Regional Copper-Nickel Study by the Mineral Resources Research Center (MRRC) of the University of Minnesota. The various Duluth Gabbro samples discussed earlier were subjected to bench-scale concentrating tests using bulk flotation. The resulting products were studied particularly intensively in terms of their chemical composition. Unlike the mineralized Duluth Gabbro samples used as feed for the bench tests, relatively little data was gathered on the mineralogy of the products. Instead, their mineralogy can be inferred reasonably well from the knowledge of their chemistries, coupled with a detailed understanding of the mineralogy of the ore samples. Accordingly, it is most appropriate here to discuss the chemistry of the products first, and follow this with a discussion of mineralogy.

2.4.1.1 Chemistry--Much of the data here is based on the elemental analysis of 2 bench-scale concentrates produced at MRRC by a standardized flotation test conducted on each of 12 mineralized Duluth Gabbro samples, as discussed earlier. In order to illustrate the variability of the elemental composition of the resulting concentrates, Table 14 shows the observed range of values for the constituents of principal interest. Also shown is the range of observed concentrate weight recoveries. Note that the major constituents of economic interest, Cu, Ni, Fe, S, and Co, typically vary by a factor of 2 to 4 from the lowest to the highest observed values. On the other hand, trace constituents of environmental and metallurgical interest, such as As, Pb, Zn, and Cd, vary over a range of 2 to 4 orders of magnitude or more. Clearly, predictive knowledge of these minor constituents is limited only to the general order of magnitude of expected composition.

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Table 14

Using bench scale test results, as well as information from pilot plant runs made by USBM, INCO, and MRRC, the chemistry data on selected concentrates produced were used to obtain a model of the concentrates likely to result from processing the various ores discussed earlier (Table 15, column 2). In order to simplify the calculations for the feed to the model smelter, just one model concentrate was produced to be used for both the open pit and underground ore models. This model is based on the processing of disseminated ore. For comparison, chemistry information is included in Table 15 on the actual average composition of the disseminated concentrate generated by MRRC and the concentrate results from processing the semi-massive samples (columns 1 and 3, respectively).

Table 15

The chemistry of the disseminated concentrate average is an average of the 65 M (mesh) and 200 M grind process samples. The 2 types of grinding and flotation (65 M and 200 M) are discussed in Iwasaki et al. (1978). Briefly the 200 M grind method reduces the ore to minus 200 M and then the ore is sent to flotation and a concentrate is produced. The 65 M grind method reduces the ore to minus 65 M, and the ore then undergoes preliminary flotation. The resultant concentrate is then reground to minus 270 M and further concentrated by flotation.

The results are a relatively high Cu content and low weight recovery for the 65 M grind, and a relatively low Cu content and high weight recovery for the 200 M grind. The 200 M grind also has a higher Cu recovery. For the purposes of interest here, the results of these 2 methods are combined. The resulting

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Table 14. Range of values observed for constituents in bench-scale
           concentrates.[a]

ELEMENT	COMPOSITIONAL RANGE OBSERVED	UNITS
Si	no reliable analyses[c]	---
Al	0.22 - 3.55	PCT
Fe(oxide)	see entry (under sulfide) below	---
Mg	0.098 - 4.7	PCT
Ca	0.025 - 2.4	PCT
Na	0.04 - 0.737	PCT
K	0.0001 - 0.17	PCT
Ti	0.02 - 0.229	PCT
P	0.00007 - 5.7	PCT
Mn	0.004 - 0.066	PCT
Cr	0.0000007 - 0.192	PCT
B	0.006 - 994	PPM
Ba	21 - 367	PPM
Be	0.005 - 0.50	PPM
Sr	4.6 - 104	PPM
V	8.1 - 145	PPM
Th	0.06 - 12	PPM
Zr	4 - 50	PPM
S	19.18 - 36.35	PCT
Cu	7.47 - 26.20	PCT
Ni	0.69 - 3.29	PCT
Fe(sulfide)[b]	18.7 - 47.6	PCT
Co	0.031 - 0.232	PCT
Zn	0.019 - 2280	PPM
Pb	0.16 - 230	PPM
Ag	14 - 58	PPM
As	0.14 - 60	PPM
Hg	0.14 - 0.19	PPM
Mo	0.03 - 119	PPM
Cd	0.07 - 30	PPM
Weight Recovery[d]	1.50 - 24.34	PCT

[a]Includes data from all samples tested, based on analysis of fourth
cleaner concentrate (Iwasaki et al. 1978).

[b]Range shown is for total iron, independent of whether it is in the
oxide or sulfide form.

[c]Silicon values in the ore models were chosen to balance the total
constituents to 100%.

[d]Includes very high weight recoveries in semi-massive samples.
Omitting 3 samples (AX9004, 6, and 7) gives a more representative weight
recovery range of 1.50 - 10.46% weight.

Table 15. Model concentrate: chemistry.

ELEMENT	DISSEMINATED BENCH CONCENTRATE AVERAGE	DISSEMINATED CONCENTRATE MODEL	SEMI-MASSIVE BENCH CONCENTRATE AVERAGE	UNITS
$Si(SiO_2)$	6.85(14.67)	7.25(15.53)	4.53(9.69)	PCT
$Al(Al_2O_3)$	1.71(3.24)	1.81(3.43)	0.62(1.17)	PCT
$Fe(FeO)$	2.39(3.08)	2.53(3.26)	0.98(1.26)	PCT
$Mg(MgO)$	1.46(2.43)	1.55(2.57)	0.31(0.57)	PCT
$Ca(CaO)$	1.08(1.52)	1.14(1.61)	0.32(0.45)	PCT
$Na(Na_2O)$	0.37(0.50)	0.39(0.53)	0.17(0.23)	PCT
$K(K_2O)$	0.08(0.09)	0.08(0.10)	0.02(0.02)	PCT
$Ti(TiO_2)$	0.09(0.16)	0.10(0.17)	0.06(0.10)	PCT
$P(P_2O_5)$	0.01(0.03)	0.01(0.03)	0.01(0.02)	PCT
$Mn(MnO)$	0.03(0.04)	0.03(0.04)	0.01(0.01)	PCT
$Cr(Cr_2O_3)$	0.04(0.06)	0.04(0.06)	0.01(0.01)	PCT
B	293.0	310.1	251.9	PPM
Ba	122.4	129.6	35.6	PPM
Be	0.06	0.06	0.02	PPM
Sr	47.4	50.2	16.3	PPM
V	80.8	85.5	29.8	PPM
Th	2.8	3.0	0	PPM
Zr	24.3	25.7	12.8	PPM
S	26.235	25.870	30.894	PCT
Cu	14.580	13.825	12.010	PCT
Ni	2.618	2.647	2.380	PCT
$Fe(S)$	30.424	30.001	40.890	PCT
Co	0.1320	0.132	0.190	PCT
Zn	1134.4	1137.0	589.00	PPM
Pb	64.0	60.7	28.3	PPM
Ag	36.2	34.3	19.4	PPM
As	31	31	21.9	PPM
Hg	0.177	0.174	0.160	PPM
Mo	28.5	28.1	33.5	PPM
Cd	40	40	40	PPM

average disseminated concentrate is plotted against the disseminated average ore in Figure 12 along with the concentrate model and the semi-massive concentrate average. As can be seen both from Figure 12 and Table 15, the average disseminated concentrate and the model concentrate are very close in compositions. The S, Cu, Co, and Ni values in the model concentrate were iterated from pilot plant analyses run by USBM, INCO, and MRRC. The Cu/Ni ratio of the open pit and underground ore models along with the input requirements for a smelter producing 100,000 mtpy of combined Cu and Ni metal was also used. These considerations were then applied to the average disseminated concentrate and the model values for S (25.87%), Cu (13.825%), Co (0.132%), and Ni (2.647%) were arrived at. Then Fe (S), Mo, & Hg were scaled based on the amount of S; Pb and Ag were scaled based on Cu; and Zn was scaled based on Co. These elements were scaled in this way because each reference element indicated (S, Cu, Co) had the corresponding best linear fit when data from all the concentrate bench scale tests were correlated separately. Because of a lack of infomation, As and Cd were derived from INCO pilot plant work. The non-sulfide elements as a group were then adjusted proportionally to total 100%.

Figure 12

The open pit model concentrate makes up 3.1763% by weight of the feed to the processing plant, while the remaining 96.8237% by weight is tailing. The underground model concentrate comprises 5.1438% by weight and the remaining 94.8562% by weight is tailing. This difference in weight percent is a direct reflection of the higher copper content in the underground ore model. The concentrate weight recoveries of 3.176% and 5.144% for the processing of open pit and underground ore respectively are representative values based on data from

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FIGURE 12

COMPARISON OF CONCENTRATES TO DISSEMINATED AVERAGE ORE
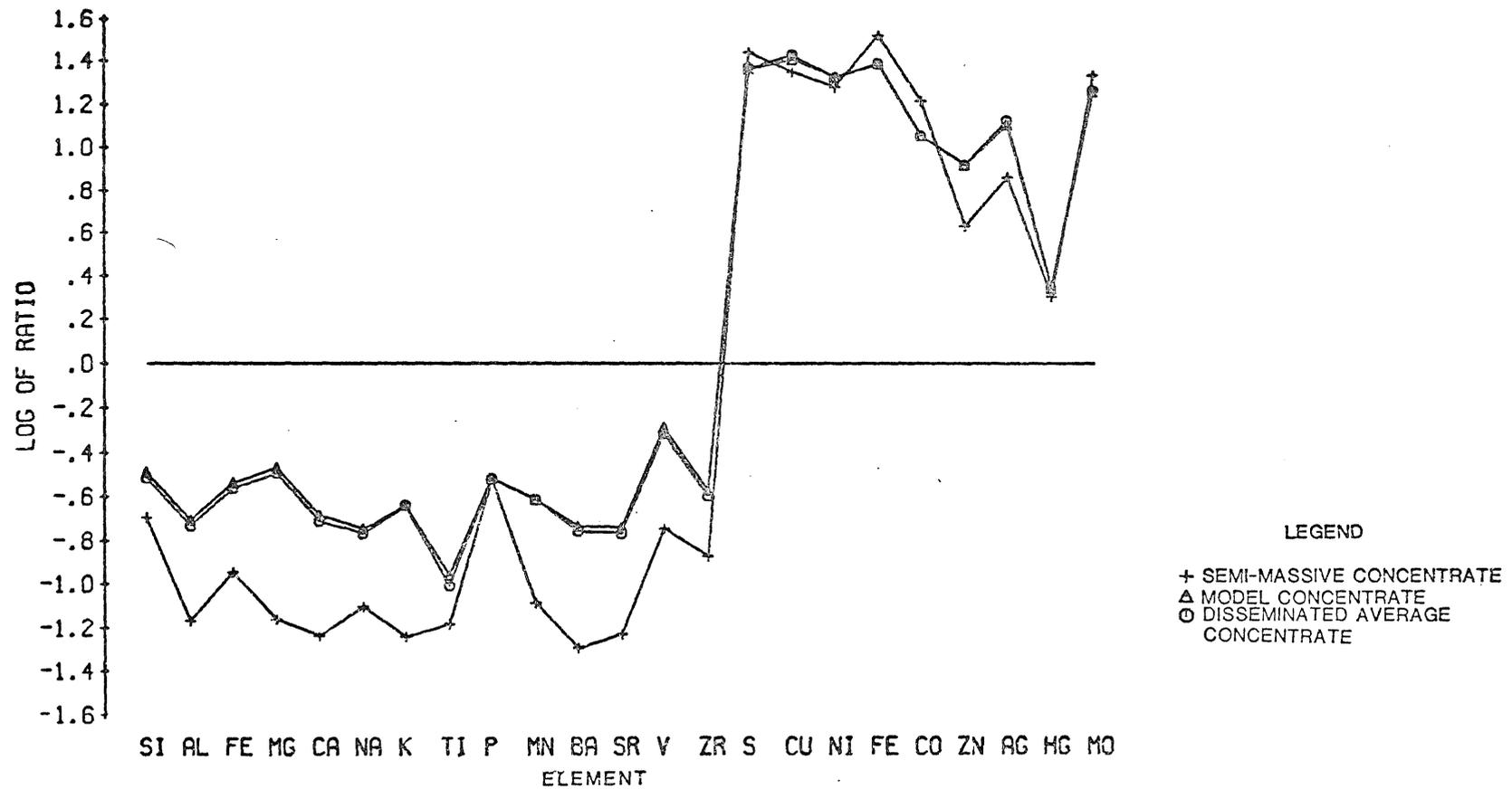
USBM and MRRC processing work. Using these weight recoveries and the given chemistries for the various ores (Table 6) and the concentrate model (Table 15), the unit recoveries for each constituent are calculated and shown in Table 16. Recoveries are also shown for the semi-massive concentrate. These values indicate the percent of each element present in the ore that leaves the mill in each of the 2 modeled products. The recovery of copper in the concentrate is reasonably high, almost 89%, for the disseminated ores. The nickel, at approximately 74% is somewhat lower. The available data indicated that the recoveries of both of these metals are higher for the semi-massive ores, as the table indicates. The cobalt recoveries are quite low, as represented by the 38-40% values in the model. There is a need for research into the problem of improving this cobalt recovery, considering the high economic value and strategic importance of this metal to the U.S. The major problem with the depressed Co and Ni recoveries is their low initial concentrations in the ore.

Table 16

The elements included in the chemistry modeling of the processing products are those for which considerable data is available from work performed at the USBM, MGS, and MRRC. These elements clearly account for all the major constituents in the ore, as well as many of the minor or trace (less than 100 ppm) constituents. There are other elements of interest for which considerably less data is available, either because of levels below detection limits, or because quite expensive analytical techniques are required to measure them. These elements typically are present in only trace amounts in the ore, and thus are not discussed in the section dealing with ore chemistries. However, some of these constituents are preferentially recovered in the concentrate and must be mentioned here. Included in these elements are two basic groups:

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Table 16. Concentrate and tailing percent recoveries.[a]

ELEMENT	OPEN PIT MODEL CONCENTRATE	OPEN PIT MODEL TAILING	UNDERGROUND MODEL CONCENTRATE	UNDERGROUND MODEL TAILING	SEMI-MASSIVE BENCH AVERAGE CONCENTRATE	SEMI-MASSIVE BENCH AVERAGE TAILING
Si	1.0	99.0	1.7	98.3	3.1	96.9
Al	0.6	99.4	1.0	99.0	1.3	98.7
Fe(oxide)	0.9	99.1	1.5	98.5	1.8	98.2
Mg	1.1	98.9	1.8	98.2	1.6	98.4
Ca	0.7	99.3	1.1	98.9	1.6	98.4
Na	0.6	99.4	0.9	99.1	2.0	98.0
K	0.7	99.3	1.2	98.8	0.5	99.5
Ti	0.3	99.7	0.6	99.4	1.1	98.9
P	1.1	98.9	1.7	98.3	3.3	96.7
Mn	0.8	99.2	1.3	98.7	1.4	98.6
Cr	4.2	95.8	6.9	93.1	4.1	95.9
Ba	0.6	99.4	1.0	99.0	2.7	97.3
Sr	0.6	99.4	0.9	99.1	1.4	98.6
V	1.6	98.4	2.7	97.3	2.3	97.7
Zr	0.8	99.2	1.4	98.6	2.6	97.4
S	75.0	25.0	80.2	19.8	69.3	30.7
Cu	88.9	11.1	88.9	11.1	96.6	3.4
Ni	73.7	26.3	73.6	26.4	93.9	6.1
Fe(S)	77.1	22.9	84.1	15.9	69.7	30.3
Co	38.1	61.9	39.9	60.1	86.8	13.2
Zn	27.2	72.8	28.6	71.4	31.6	68.4
Pb	41.9	58.1	40.0	60.0	17.4	82.6
Ag	45.4	54.6	44.1	55.9	72.1	27.9
As	10.4	89.6	10.8	89.2	65.5	34.5
Hg	7.4	92.6	7.6	92.4	28.3	71.7
Cd	63.7	36.3	14.0	86.0	53.6	46.4
Model Value % Weight of Ore	3.1763	96.8237	5.1438	94.8562	14.76	85.24

[a]Each value represents the percent of the corresponding element in the ore as it leaves the processing operation in the product shown. All values are normalized to 100%.

1) Minor or trace constituents of economic interest. Those include Au, Ag, Rh, Pt, Pd.

2) Minor or trace elements which may pose operating problems or environmental problems in the smelting/refining processes. These include Sb, Sn, W, Bi, F, Cl, Se, Te.

In order to provide a general estimate of the amounts of these elements likely to be present in the concentrate, available data has been examined and representative values selected. Table 17 is based on an average of values from USBM, INCO, and Cornell University data (Iwasaki et al. 1978), and neutron activation analysis results for the IP9002 concentrate. The asterisked values (*) are based on plasma emission spectrometer results (Bi, Cl, Se, Te and F) and neutron activation analysis results (Au, Sb, Sn, and W).

Table 17

Fluorine and Cl are based on ore information that has been extrapolated to fit the non-sulfide elemental abundances for the model concentrate. These estimates are of interest in assessing potential problems from trace and minor elements, as discussed in section 4.7.4 of the technology assessment report.

Figures 13 and 14 show the model ore, tailing, and concentrate for the open pit and underground ores, normalized against the average disseminated ore. Some elements such as Zn, and Hg are not as strongly enriched in the concentrate as are some of the major elements such as S, Cu, Ni, and Fe(S). The weak enrichment may be due to the low amount and/or the dispersed nature of these elements in the minerals.

Figures 13 & 14

Table 17. Minor and trace elements in concentrate.[a]

ELEMENT	REPRESENTATIVE CONCENTRATION(ppm)
Ag	26.8
Au	0.80 (0.90)[b]
Rh	15.8
Pt	1.00
Pd	2.93
Sb	1.7*
Sn	less than 740*
W	less than 900*
Bi	100*
F	3.2*
Cl	111.6*
Se	less than 4*
Te	less than 3.3*

[a]Values shown are generally representative of levels found in a variety of analyses, and indicate the general order of magnitude of concentration expected in a bulk concentrate produced from Duluth Complex ores.

[b]Values are based on analysis performed for the Regional Study using plasma emission spectrometry or neutron activation analysis (see text).

FIGURE 13

COMPARISON OF OPEN PIT ORE, CONCENTRATE AND
TAILING MODELS TO DISSEMINATED AVERAGE ORE

LEGEND

+ MODEL CONCENTRATE
△ OPEN PIT TAILING
⊙ OPEN PIT ORE
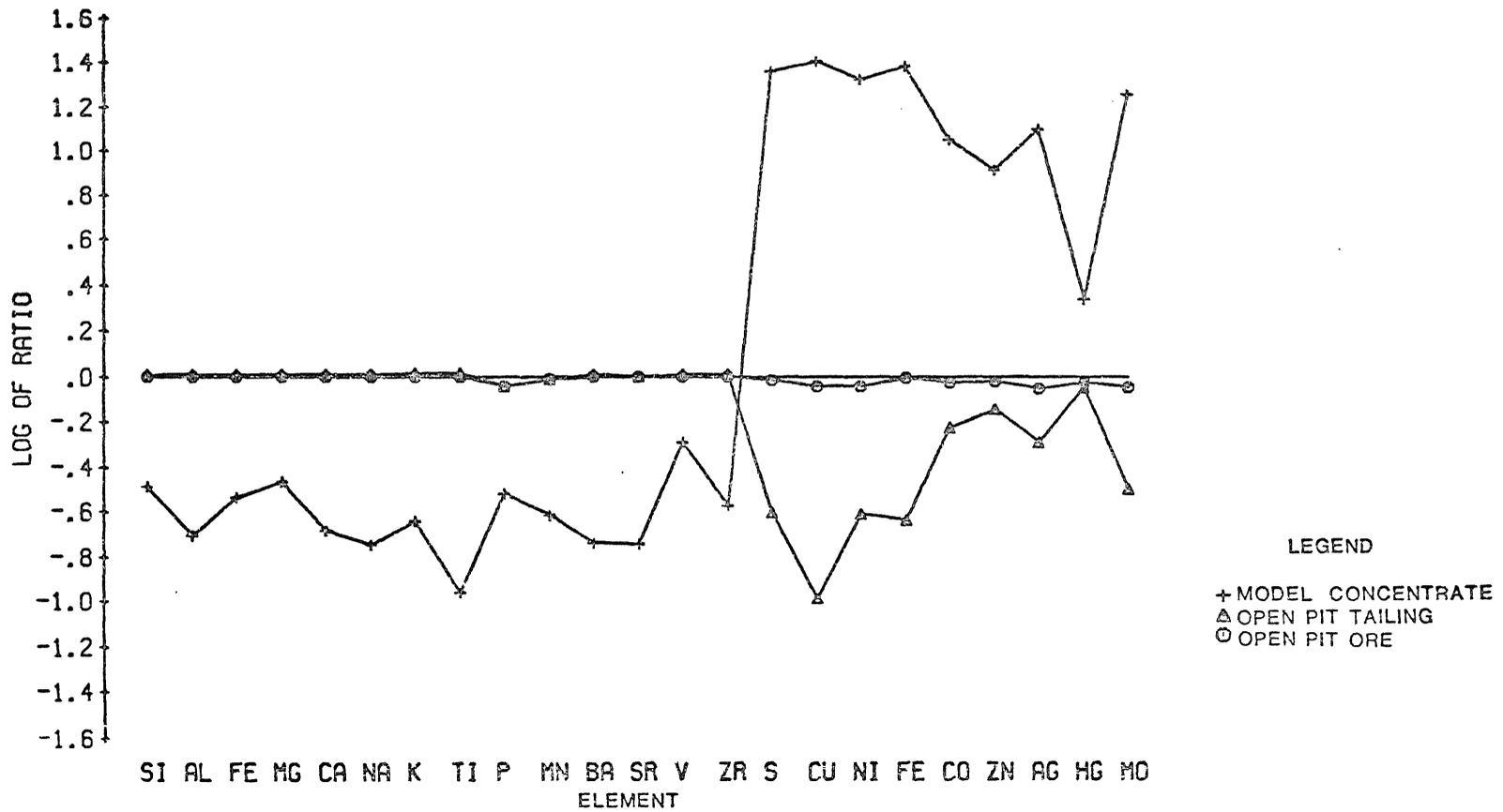
FIGURE 14

COMPARISON OF UNDERGROUND ORE, CONCENTRATE
AND TAILING MODELS TO DISSEMINATED AVERAGE ORE

LEGEND
+MODEL CONCENTRATE
△UNDERGROUND TAILING
⊙UNDERGROUND ORE
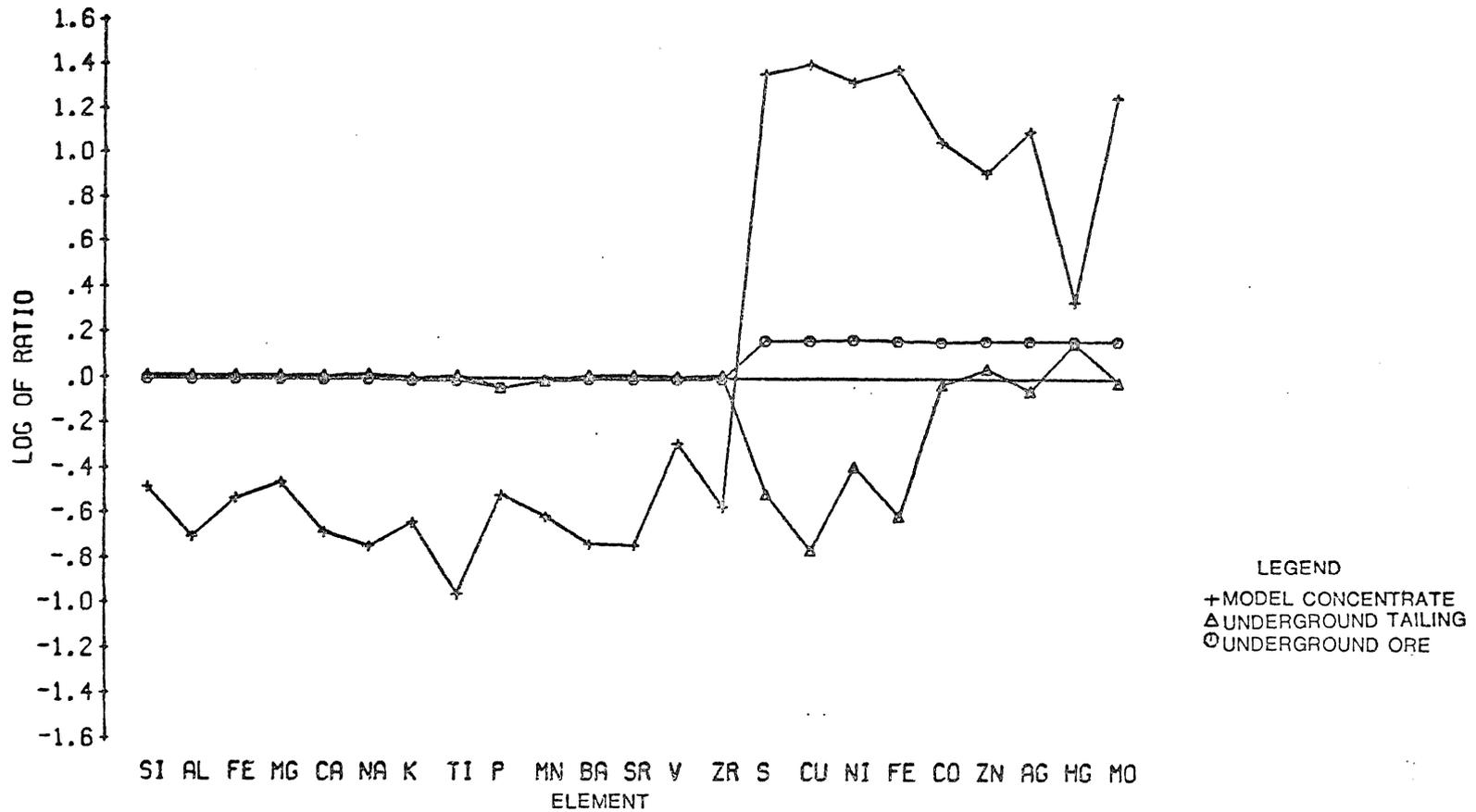
It is interesting to compare these figures with the curve in Figure 12 which includes the average semi-massive concentrate normalized against the average disseminated ore. The trend is similar to that of the disseminated concentrate. The non-sulfide element pattern is the same but lower, while the sulfide-related elements pattern is very close to that of the disseminated model. Recall that the analysis shown in Table 15 of the semi-massive concentrate is based on averaging 65 and 200 M grind runs of semi-massive concentrate as with the disseminated average.

Note in Figure 12 that the semi-massive concentrate is lower in most non-sulfide elements while it is higher in S, Fe(S), Co, and Mo than the disseminated concentrates. The disseminated concentrates are higher in Cu, Ni, Zn, and Ag. The major reason for more S, Fe(S) and less Cu, Ni, and non-sulfide elements is the higher proportion of pyrrhotite (Fe_7S_8-FeS) (see Table 9) in the semi-massive concentrate than in the disseminated concentrate. The semi-massive concentrate makes up 14.76% by weight (Table 16) of the semi-massive ore output. This increase is due, as with the case of the underground model, to the higher sulfide content of the ore.

Figure 15 shows the averages of semi-massive data for ore, tailing, and concentrate. As mentioned above, the general trend shown by the disseminated concentrate is also seen for the semi-massive concentrate. For Ti and Ba, both the concentrate and tailing are on the same side of the ore plot, which is a physical impossibility. This inconsistency helps point out the errors associated with the analysis of the sample. In both cases at least one of the 2 values is close to the ore plot so that the trend established for the disseminated sample is consistent.

Figure 15

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FIGURE 15

COMPARISON OF SEMI-MASSIVE ORE CONCENTRATE
AND TAILING TO DISSEMINATED AVERAGE ORE

LEGEND

+ SEMI-MASSIVE CONCENTRATE
△ SEMI-MASSIVE TAILING
⊕ SEMI-MASSIVE ORE
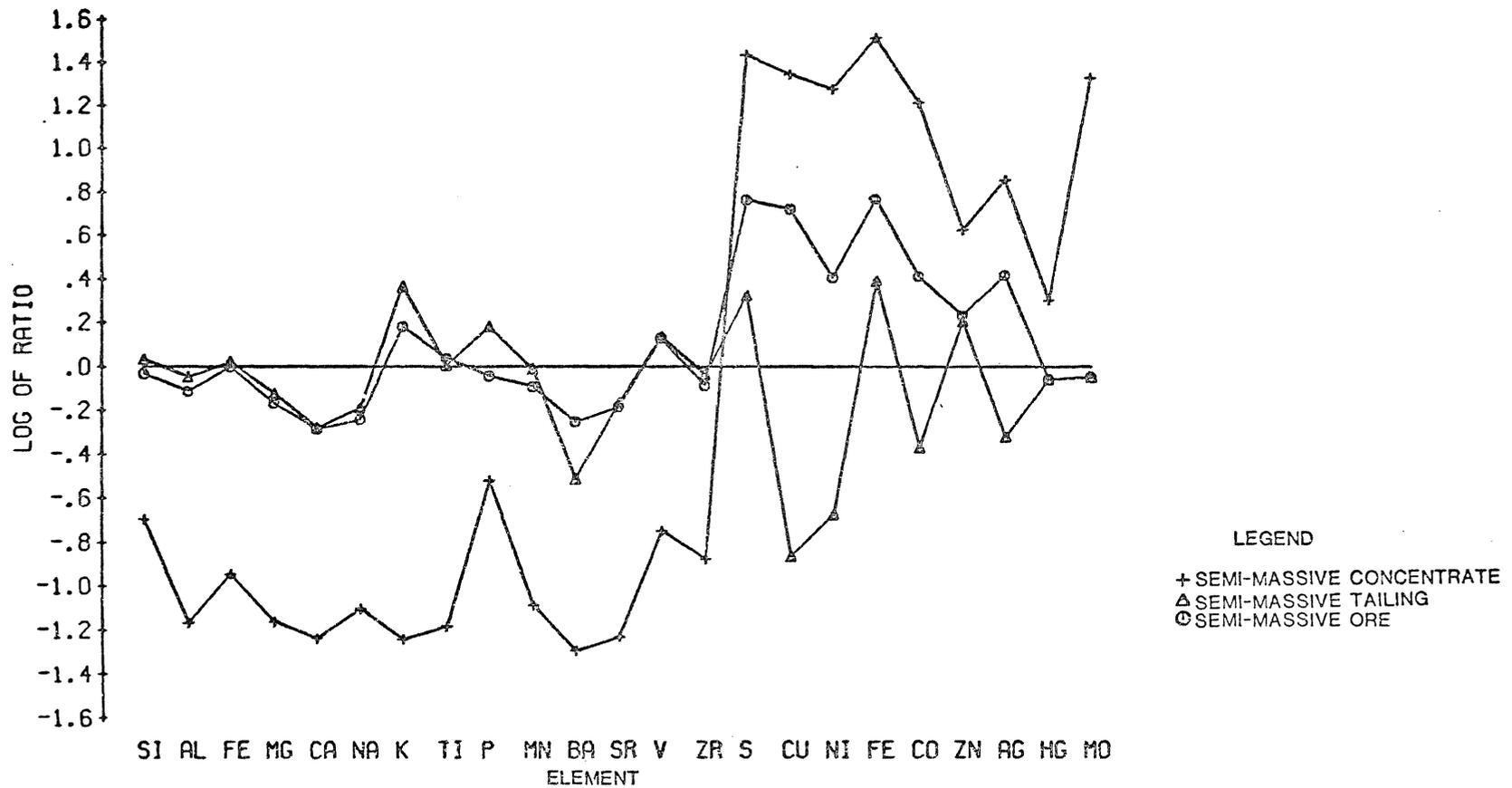
Table 16 shows the elemental recoveries for the model tailing and concentrate for the open pit and underground ores, and the semi-massive averages. Since the open pit and underground models are calculated, no normalization was needed but in the case of the semi-massive averages normalization was required. This is because the tailing analyses for the semi-massive ore averages is an average of actual data and not calculated from the ore and concentrate models. Therefore, the recovery values shown in Table 16 for the semi-massive models have been normalized, by element, to total 100%. In most cases this produced little change, however, in some the change was greater than 30%. Table 18 lists the actual calculated values. As can be seen from a comparison of Tables 16 and 18, care should be taken in using the recovery values for the semi-massive concentrate since, for instance, the measured and normalized Cu recoveries are 62.2% and 96.6%, respectively. Table 18 should be used with the knowledge that the middlings (see Iwasaki et al. 1978) are not included in the data on which this table is based. Actual recoveries of sulfide constituents in the concentrate will be higher when these middlings are included, as would be the case in any continuous operation. This information on semi-massive ores is included merely to illustrate generally the nature of the resulting processing products in comparison to those derived from the disseminate ores, which constitute by far the bulk of the known resource (roughly 99.9%).

_____

Table 18

2.4.1.2 Mineralogy--The calculated mineralogy of the disseminated model concentrate and those of the MRRC disseminated concentrate average and the semi-massive concentrate average are shown in Table 19. The mineralogies of all of these concentrates are based on the Cu contents shown in Table 15, in the pre-

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Table 18. Measured average percent recoveries for semi-massive concentrate and tailing.[a]

ELEMENT	SEMI-MASSIVE AVERAGE CONCENTRATE	SEMI-MASSIVE AVERAGE TAILING
Si	3.2	99.5
Al	1.3	99.8
Fe(oxide)	1.7	90.2
Mg	1.5	94.6
Ca	1.6	85.8
Na	2.0	96.1
K	0.6	130.3
Ti	0.9	79.3
P	4.9	142.1
Mn	1.5	102.3
Cr	4.9	113.7
Ba	1.3	46.6
Sr	1.3	89.2
V	2.0	86.6
Zr	2.4	95.4
S	69.9	30.9
Cu	62.2	2.2
Ni	110.5	7.2
Fe(S)	82.3	35.7
Co	93.5	14.2
Zn	36.7	79.2
Pb	0	0
Ag	40.3	15.6
As	32.3	17.0
Hg	33.7	85.2
Cd	59.0	51.1

[a]The values shown are averages of actual analyses of bench scale test feeds and products (see Iwasaki et al. 1978). Various middling fractions are not accounted for which, along with sampling and analytical errors, explains why the recoveries shown do not total 100%.

vious section. These copper values are used as references, and are compared to the copper contents for the appropriate ores, as given in Table 15 (model ore composition: chemistry). The model and average mineralogies were constructed by first scaling the sulfide minerals using the ore mineralogy information in Table 9 (model minerology). The appropriate content of each sulfide mineral was calculated using the formula:

$$\text{concentrate volume \%} = (\text{ore volume \%}) \times \frac{(\text{concentrate weight \% of Cu})}{(\text{ore weight \% of Cu})}$$

The non-sulfide minerals are than scaled appropriately to account for the remaining weight of each concentrate.

Table 19

For all 3 analyses shown in Table 19, the amount of total sulfide minerals presented comprises over 50% of the concentrate or an increase of about 20 times over that found for the ore. The dominant mineral group is now the sulfides whereas before concentrating it was the silicates, mainly plagioclase. Plagioclase still dominates the non-sulfide mineral fraction. The inter-mineral ratio within each of the 2 mineral groups (sulfide minerals and silicate minerals) are assumed to be the same for the disseminated concentrate model and the disseminated and semi-massive concentrate averages as for the ore models. In reality, a change in the inter-mineral ratios is expected but there is not enough information available to predict, for example, if there will be proportionally more plagioclase in the concentrate than in the tailing.

2.4.1.3 Physical Characteristics--The grain size of the concentrate, because of processing, will be essentially 100% minus 65 M (-210 um) initially, and then 80 to 90% minus 270 M (-53 um) due to regrinding.

Table 19. Calculated concentrate mineralogy (percent by volume).

MINERAL	DISSEMINATED CONCENTRATE AVERAGE[a]	DISSEMINATED MODEL CONCENTRATE[a]	SEMI-MASSIVE CONCENTRATE AVERAGE[b]
Plagioclase	27.226	27.387	18.746
Sericite	0.611	0.615	0.031
Olivine	8.282	8.331	2.599
Clinopyroxene	2.635	2.650	6.122
Orthopyroxene	0.703	0.707	3.791
Monocrystalline amphibole	1.144	1.151	0.004
Fibrous amphibole	0.170	0.171	0.004
Chlorite	1.022	1.026	0.113
Serpentine	0.694	0.698	0.003
Iddingsite	0.042	0.042	0.011
Talc	0.040	0.040	-----[e]
Biotite	1.215	1.222	1.886
Smectite	0.014	0.014	-----
Opaques	55.812	55.553	65.693
Chalcopyrite-cubanite[c]	29.906	29.761	14.251
Pentlandite[c]	1.581	1.574	1.689
Pyrrhotite[c]	23.306	23.193	48.527
Ilmenite-magnetite[c]	1.018	1.024	1.137
Graphite[d,c]	0.001	0.001	0.089
Myrmekite	0.039	0.039	-----
Apatite	0.062	0.062	0.037
Epidote	0.132	0.133	-----
Allanite	0.012	0.012	-----
Calcite	0.019	0.019	0.001
Quartz	-----	-----	0.007
Cordierite	0.126	0.127	0.953

[a]Based on Table 9 disseminated average mineralogy.
[b]Based on Table 9 average semi-massive ore mineralogy.
[c]These values are included in the total number listed for opaques.
[d]Actual amount may be as much as 10 times that shown (Iwasaki et al. 1978).
[e]Values not shown are less than .001%.

2.4.2 Tailing

Two samples of tailing were analyzed for each of MRRC's bench scale flotation runs. They are called the rougher tailing and first cleaner tailing. The rougher tailing is that tailing produced from the first flotation step while the first cleaner tailing is produced from the second flotation step or first cleaning of the rougher concentrate. In Figure 16, the rougher tailing disseminated average, the cleaner tailing disseminated average, and their combined product are normalized against the average disseminated ore. The first cleaner tailing is higher than the rougher tailing in all sulfur-related elements (S through Mo) and appears rather similar to the disseminated average ore. This is because it is produced from the rougher concentrate (Iwasaki et al. 1978) so the starting material is enriched in the minerals that contain these elements. As can be seen from Figure 16, the combined tailing plots closer to the rougher tailing than to the first cleaner tailing. This is because the rougher tailing makes up approximately 95% by weight of the combined disseminated tailing.

Figure 16

2.4.2.1 Chemistry--As noted before, much of the data presented here is based on the analysis of product produced at MRRC by bench-scale test. To convey an idea of the variability of the results of these analyses, Table 20 shows the observed range of values for the constituents of principal interest. The major constituents (especially S, Fe, Cu, Ni, and Co) which are selectively removed with the concentrate, are seen to vary typically by several orders of magnitude from the lowest to the highest values observed in the tailing samples. This clearly illustrates the importance of properly designing and operating a flotation plant tailored for the specific ore being processed if low residual levels of valuable

FIGURE 16

COMPARISON OF

MEASURED TAILING AVERAGES TO DISSEMINATED AVERAGE ORE

LEGEND

◇ COMBINED TAILING
✕ CLEANER TAILING
+ ROUGHER TAILING

⊘ DISSEMINATED ORE AVERAGE
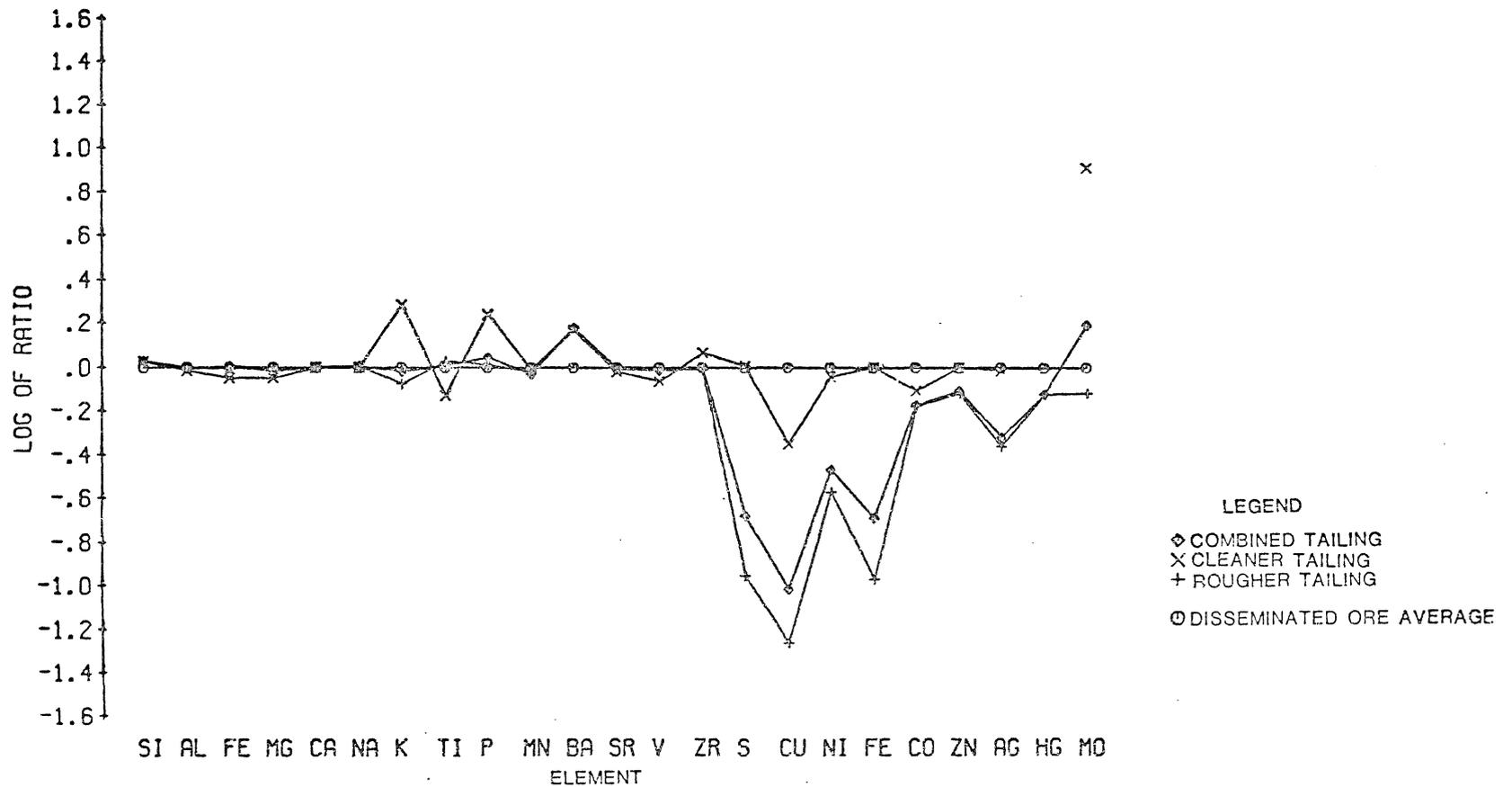
constituents in the tailing material are to be obtained. This is also important for elements of environmental concern such as sulfur. Residual sulfur levels of 2 to 3% in the tailing basin may pose a serious potential for acid formation and resultant discharge of water at low pH containing high levels of leached heavy metals. Proper flotation design should be able to achieve residual sulfur levels near the bottom of the observed range (0.03 to 0.1%), thereby greatly reducing or eliminating the possibility of acidification in the basin.

Table 20

Following up on the various averages and models presented in the discussion of concentrates, the chemistry of the average disseminated tailing, open pit model tailing, underground model tailing, and semi-massive average tailing are shown in Figure 17 and listed in Table 21. These show that the combined tailing average is close to that of both the open pit and underground model tailing. Since a single model concentrate is used, the open pit model tailing has a lower S and sulfur-related elemental composition than does the underground model tailing. Although this is a model-dependent relationship, under actual processing conditions both the concentrate and the tailing of an underground mine would likely be higher in S and sulfur related elements than the corresponding products from open pit mine ore.

Figure 17, Table 21

The elemental composition of the disseminated tailing models were calculated by subtracting the appropriate weight percent of model concentrate from the model ores and then adjusting the remainder to total 100%. The correlation (Figure 17) between the model tailing and the combined disseminated average bench-scale

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Table 20. Range of values observed for constituents in bench-scale tailing.[a]

ELEMENT	CONCENTRATION RANGE OBSERVED	UNITS
Si	no reliable analyses[c]	---
Al	5.04-11.5	PCT
Fe(oxide)	see entry (under sulfide) below	---
Mg	1.27-6.02	PCT
Ca	0.958-6.84	PCT
Na	0.927-3.0	PCT
K	0.094-3.52	PCT
Ti	0.0838-1.88	PCT
P	0.00007-0.222	PCT
Mn	0.069-0.223	PCT
Cr	0.0000007-0.144	PCT
B	0.006-2790	PPM
Ba	176-2400	PPM
Be	0.005-20.4	PPM
Sr	121-358	PPM
V	51.3-316	PPM
Th	0.06-372	PPM
Zr	40-300	PPM
S	0.028-2.87	PCT
Cu	0.0022-0.755	PCT
Ni	0.000009-0.26	PCT
Fe(S)[b]	1.01-31.5	PCT
Co	0.000007-0.0138	PCT
Zn	81-250	PPM
Pb	0.16-110	PPM
Ag	0.002-8.2	PPM
As	0.14-120	PPM
Hg	0.07[d]	PPM
Mo	0.03-34	PPM
Cd	0.07-51	PPM

[a]Includes data from all samples tested, based on analysis of rougher and first cleaner tailing products.
[b]Range shown is for total iron, independent of whether it is in the oxide or sulfide form.
[c]Silicon values in the ore models were chosen to balance the modeled constituents to 100%.
[d]The value for Hg is based on analyses from 3 samples, all of which yielded the same concentration.

FIGURE 17

COMPARISON OF OPEN PIT, UNDERGROUND AND SEMI-MASSIVE AVERAGE
TAILING PRODUCTS TO THE DISSEMINATED AVERAGE ORE

LEGEND

SEMI-MASSIVE
X AVERAGE TAILING
+ UNDERGROUND TAILING
△ OPEN PIT TAILING
⊕ DISSEMINATED
   AVERAGE TAILING
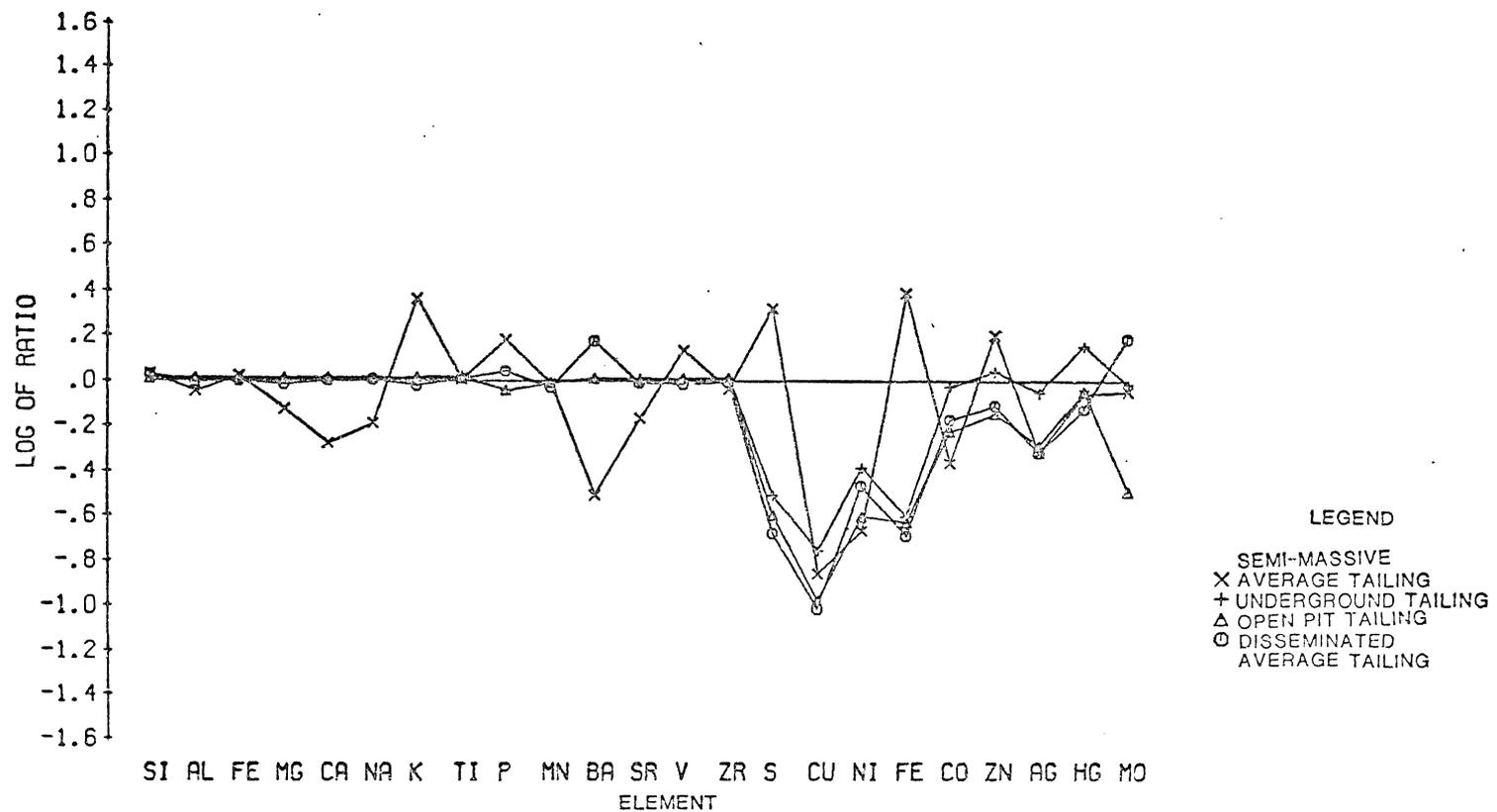
Table 21. Model tailing composition: chemistry.

ELEMENT	DISSEMINATED BENCH AVERAGE TAILING		MODEL OPEN PIT TAILING		MODEL UNDERGROUND TAILING		SEMI-MASSIVE BENCH AVERAGE TAILING		UNITS
$Si(SiO_2)$	23.65	(50.64)	22.83	(48.87)	22.76	(48.72)	24.14	(51.68)	PCT
$Al(Al_2O_3)$	9.05	(17.09)	9.46	(17.88)	9.47	(17.88)	8.29	(15.66)	PCT
$Fe(FeO)$	8.72	(11.22)	8.90	(11.45)	8.89	(11.43)	9.16	(11.78)	PCT
$Mg(MgO)$	4.36	(7.22)	4.65	(7.69)	4.63	(7.67)	3.44	(6.35)	PCT
$Ca(CaO)$	5.53	(7.74)	5.68	(7.95)	5.68	(7.96)	2.90	(4.06)	PCT
$Na(Na_2O)$	2.19	(2.96)	2.24	(3.02)	2.25	(3.03)	1.42	(1.91)	PCT
$K(K_2O)$	0.33	(0.40)	0.36	(0.43)	0.35	(0.43)	0.81	(0.97)	PCT
$Ti(TiO_2)$	0.94	(1.58)	0.95	(1.57)	0.94	(1.58)	0.93	(1.55)	PCT
$P(P_2O_5)$	0.004	(0.008)	0.03	(0.07)	0.03	(0.07)	0.05	(0.12)	PCT
$Mn(MnO)$	0.11	(0.15)	0.12	(0.16)	0.12	(0.17)	0.12	(0.15)	PCT
$Cr(Cr_2O_3)$	0.03	(0.05)	0.03	(0.05)	0.03	(0.05)	0.04	(0.06)	PCT
B	1120.3		580.5		576.4		707.6		PPM
Ba	1056.2		724.0		724.4		215.7		PPM
Be	0.74		0.56		0.56		2.0		PPM
Sr	272.0		278.3		285.7		190.4		PPM
V	160.4		169.9		167.9		226.2		PPM
Th	3.5		4.34		4.3		5.5		PPM
Zr	95.2		98.5		98.4		88.5		PPM
S	0.234		0.281		0.346		2.365		PCT
Cu	0.052		0.056		0.094		0.074		PCT
Ni	0.042		0.031		0.051		0.027		PCT
Fe(S)	0.253		0.290		0.308		3.070		PCT
Co	0.008		0.007		0.011		0.005		PCT
Zn	109.1		99.9		153.6		220.5		PPM
Pb	2.2		2.8		4.9		23.2		PPM
Ag	1.3		1.4		2.4		1.3		PPM
As	2		8.8		13.8		2		PPM
Hg	0.06		0.0717		0.114		0.07		PPM
Mo	2.4		0.5		1.5		2.0		PPM
Cd	6		8.5		13.3		6		PPM

tailing is quite good; considering that the data for the combined composition is from bench-scale test averages and the models are calculated by subtraction of concentrate from model ores. For the open pit model the tailing is modeled as making up 96.825% by weight of the products from the processing plant. For the higher grade underground ore the tailing makes up 94.858% by weight of the processing plant output.

Figures 13 and 14 show ore, tailing and concentrate of the open pit and underground models normalized against the disseminated average ore. Because the amount of material concentrated from the ore is relatively small, but rich in sulfur-related elements, the non-sulfide related elements of the tailing are close to the ore composition. The sulfide related elements, except Hg, are, of course, quite depleted in the tailing as compared to the ore. The relatively low Hg depletion is likely due either to the presence of Hg in both sulfide and non-sulfide minerals, or simply to analytical inaccuracies since Hg is present in such small amounts (in the range of 100 ppb). Note in Figures 13 and 14 that for the Cu and Ni, the percent recovery is not the same. Sulfur, Ni, and Fe(S) have about the same percent recoveries (Table 16). The relatively high recovery of Co, Zn, Pb, and As in the disseminated tailing indicates a need to investigate the possibility that leaching of these metals might pose water quality problems upon the discharge of tailing basin water (see Water Quality Section, Chapter 4).

Figure 17 also shows the semi-massive average bench-scale tailing (Table 21) normalized against the average disseminated ore. Note that the trend is similar to that of the disseminated tailing models. The semi-massive tailing is higher in S, Fe(S), Zn, and Pb than disseminated tailing models. While Zn is slightly higher, S, Fe(S), and Pb are almost an order of magnitude higher. This, at least for S and Fe(S), is due to the relatively high pyrrhotite (see Table 9) content

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of the semi-massive ore. Depending on the proportions of disseminated ore to semi-massive ore tailing that is pumped into a tailing basin, the S and Fe(S) (pyrrhotite) content of the tailing basin could be elevated such that water quality problems would result.

Figure 15 shows the ore, tailing, and concentrate for the semi-massive bench-scale tests. As stated earlier the trends are the same for the semi-massive and disseminated samples.

A comparison of Table 13 (lean ore and waste rock models) with Table 21 shows that the amounts of sulfur and sulfur-related elements is comparable between the disseminated tailing models and the waste rock. Also shown by the comparison is that the lean ore model is higher in these constituents by 2 to 3 times relative to the disseminated tailing models. The semi-massive tailing has the same relationship to the lean ore and waste rock models as do the disseminated tailing models except for S and Fe(S). These 2 elements are higher in the semi-massive average tailing than in the lean ore model by a factor of 4.

2.4.2.2 Mineralogy--The mineralogies of the disseminated average bench tailing, open pit model tailing, underground model tailing, and semi-massive average bench tailing are shown in Table 22. Each of the mineralogies was calculated using the same method (extrapolation based on Cu content) as was used for the ore models. The dominant minerals are the silicates.

As discussed earlier, the semi-massive tailing comes from a mineralogically distinct sample group so that its mineralogy is different from that of the disseminated tailing samples and models shown in Table 22. Because the elemental compositions of the 2 disseminated tailing models (open pit and underground) are quite similar (see Figure 17 and Table 21) their mineralogies are also very

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similar; as both have less than 0.5% by volume sulfide minerals (chalcopyrite-cubanite, pentlandite, and pyrrhotite) compared to 2.391% by volume for the average disseminated ore (see Table 22). Plagioclase is still the dominant mineral (approximately 60% by volume). Generally speaking, because of the relatively small amount of sulfide minerals in the ore, the bulk of the mineralogic composition of the disseminated tailing models is very close to that of the disseminated ore average. The overall volume increase in non-sulfide minerals is only 2.2% for the open pit model and 2.0% for the underground tailing model.

Table 22

For the semi-massive material, the change between the ore and tailing non-sulfide mineralogy is higher (10.7%) than for the disseminated models. The calculated sulfide mineralogy of the semi-massive tailing indicates a smaller residual amount of sulfide minerals than the underground tailing model and a little more than the open pit tailing. The actual production trend would have the semi-massive tailing with the higher sulfide content, consistent with the elevated sulfur content seen in the chemistry of the semi-massive average tailing (Table 21). The unrealistic values calculated in the mineralogy are the result of the normalized Cu recovery values shown for the semi-massive concentrate (Table 15) as discussed earlier.

2.4.2.3 Physical Characteristics--The grain size of the tailing is the same as that for the concentrate, able to pass through a 65 M screen. The amphibole content of the semi-massive tailing model is 0.024% by volume while that of the open pit and underground tailing models is 2.899 and 2.894% by volume, respectively.

Based on measurements made at MRRC, as well as information available from private industry, a representative bulk tailing density of 90 lb/ft^3 was chosen for

Table 22. Calculated tailing mineralogy (volume percent).

MINERAL	DISSEMINATED AVERAGE BENCH TAILING	MODEL OPEN PIT TAILING	MODEL UNDERGROUND TAILING	SEMI-MASSIVE AVERAGE BENCH TAILING
Plagioclase	60.088	60.078	59.978	52.618
Sericite	1.349	1.349	1.347	0.087
Olivine	18.279	18.276	18.246	7.296
Clinopyroxene	5.815	5.514	5.804	17.183
Orthopyroxene	1.551	1.550	1.548	10.640
Monocrystalline amphibole	2.526	2.525	2.521	0.012
Fibrous amphibole	0.374	0.374	0.373	0.012
Chlorite	2.251	2.250	.2.247	0.318
Serpentine	1.531	1.531	1.528	0.009
Iddingsite	0.092	0.092	0.092	0.032
Talc	0.089	0.089	0.089	-----
Biotite	2.682	2.682	2.677	5.293
Smectite	0.032	0.032	0.032	-----
Celadonite	-----	-----	-----	-----
Opaques	2.478	2.493	2.657	3.698
Chalcopyrite-cubanite[a]	0.125	0.134	0.225	0.057
Pentlandite[a]	0.007	0.007	0.012	0.007
Pyrrhotite[a]	0.097	0.104	0.175	0.194
Ilmenite-magnetite[a]	2.246	2.245	2.242	3.191
Graphite[a]	0.003	0.003	0.003	0.249
Spinel	0.001	0.001	0.001	-----
Myrmekite	0.086	0.086	0.086	-----
Apatite	0.137	0.137	0.137	0.104
Epidote	0.291	0.291	0.291	-----
Allanite	0.026	0.026	0.026	-----
Calcite	0.043	0.043	0.043	0.003
Quartz	-----	-----	-----	0.019
Cordierite	0.278	0.278	0.278	2.674

[a]These values are included in the total number listed for opaques.

modeling purposes. This value represents the average density of tailing in place in a disposal basin, and is assumed to be the same for tailing material from processing either open pit or underground ore. This parameter is important in determining the area required for tailing disposal. For example, assuming a density of 90 lb/ft^3 and an average tailing basin depth of 70 ft, the model of the large 20.00 X 10^6 mtpy open pit mine and mill operation would require 4,016 acres of land for disposal of tailing over the total of 23 yr of full capacity operation. See Volume 2-Chapter 5 for further discussion of this model. Clearly, assuming the average depth of the basin is held constant, the disposal area requirements will change proportionally to variations of the actual average density from the 90 lb/ft^3 model value.

2.4.3 Integrated Mine-Mill-Smelter/Refinery Models

To aid in characterizing the nature of possible copper-nickel development operations, a set of hypothetical development models were prepared, and are discussed in detail in the Technical Assessment section, Volume 2 of this report. The smelter/refinery model was designed to produce 100,000 mtpy of copper and nickel metal, with specific recovery efficiencies assumed for each metal. The values used for the smelter/refinery operation are:

Metal	Recovery Efficiency (overall, %)	Annual Production (mtpy)	Resulting Input Requirements to Meet Production (mtpy)
Cu	96.31	84,584	87,825
Ni	91.68	15,416	16,815

It must be noted that the large number of significant figures are retained for calculational purposes only, to meet the requirements for internal consistency in the material balance. They do not imply comparable predictive accuracy in the models used.

The smelter/refinery input requirements for Cu and Ni are met by appropriate quantities of the model concentrate from the disseminated ores. The amount of concentrate needed to supply the annual needs of the model smelter/refinery operation is:

Metal	Input Requirements (mtpy)	Concentrate Grade (%)	Annual Concentrate Requirements (mtpy)
Cu	87,825	13.825	635,259
Ni	16,815	2.647	635,259

The annual concentrate requirement of 635,259 mtpy can be met either by ore from an open pit mine, underground mine, or an appropriate combination. Looking at the extremes, if the ore is supplied either by an open pit or an underground mine, with the modeled weight recoveries given for the concentrate in each case, the ore requirements are:

Annual Concentrate Requirements (mtpy)	Mine Type	Concentrate % Weight Recovery	Annual Ore Requirements
635,259	open pit	3.1763	20,000,000
635,259	underground	5.1438	12,350,000

This model serves to illustrate the functional relationship between the various phases of a totally integrated mining development, and at the same time illustrates the typical quantities of material involved in the various phases. Thus, for example, a very large open pit mine producing 20.00×10^6 mtpy of ore is required to provide feed for a mill which will generate some 635,000 mtpy of concentrate and discard over 19×10^6 mtpy of tailing. The concentrate is processed by a smelter/refinery operation to produce 100,000 mtpy of copper and nickel metal for market.

The model presented here is useful not only for the picture just presented, but for its value in examining future proposals for specific developments. At that time, improved information on the modeled parameters should be available from the developer of the mining proposal. These values will most certainly differ from those selected for use in the model given here. However, it will then be a simple matter to substitute the new values into the model and calculate the new product requirements for the various phases of the operation.

2.4.4   Mineral Fibers in Processing Products

As discussed earlier, in section 1.4.3.5, present information indicates that the concentration of asbestiform amphibole minerals in Duluth Complex ore is expected to be quite low, on the order of 0.1 ppm by weight. On the other hand, amphibole minerals in non-asbestiform habits are expected to be present in significant amounts, possibly ranging as high as 13% by volume and averaging 2 to 3% by volume based on Regional Study data. The concern thus far focuses on the possibility that this amphibole content will result in the creation and subsequent release of fiber-like cleavage fragments of amphibole which might then constitute a potential health hazard.

As discussed earlier, a set of 9 mineralized Duluth Gabbro samples from potential mining locations within the resource zones was used as a basis for this investigation (see Stevenson 1978 for more information). As noted earlier, in an attempt to simulate possible processing conditions, each of the 9 samples was ground to 2 finenesses; a coarser 65 M grind, and a finer 200 M grind (Iwasaki et al. 1978). Of the resulting 18 samples, all 9 of the 200 M grind and 3 of the 65 M grind were used for the fiber generation study. Samples of rougher flotation tailing slurries and one concentrate sample produced in bench-scale tests were

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agitated and then sampled using a standard sedimentation sizing technique with an Andresen pipette, to include only particles less than 37 um (microns) in diameter. For environmental reasons, the focus of concern was on the tailing material, which would be generated in large quantities and disposed of in the vicinity of the mine. The concentrate, on the other hand, is a small quantity by comparison (typically 3-5% by weight of the ore) and is fed to successive processes involving heating to temperatures that are expected to destroy any fibers present. Consequently, only one concentrate sample was tested for fiber content.

Samples of the water with the suspended tailing or concentrate particles as well as some bulk samples were sent to the Minnesota Department of Health (MDH) for fiber analysis. There, the particles were collected on Nucleopore filters and prepared for Transmission Electron Microscopy using the Jaffe-Wick method. Transmission electron microscope (TEM) with a tilting stage and an attached x-ray energy dispersive analysis system was used for fiber counting. The mineralogy of the fibers present in the various process samples was investigated at MDH by using both energy dispersive x-ray spectroscopy (EDS) and electron diffraction analysis. These methods allow information on the chemistry of a fiber, from EDS analysis, to be combined with measurements of its crystal structure from electron diffraction analysis. All of this data was typically obtained on 10-20% of the fibers observed in a sample. Comparisons of the resulting data on each fiber were then made to similar data taken from mineral standards of known composition to identify the fiber's mineralogy with reasonable certainty. The methods used are explained in greater detail by the MDH staff in appendix 2 of the report "Ambient Concentrations of Mineral Fibers in Air and Water in Northeast Minnesota" (Ashbrook 1978).

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The actual samples analyzed by MDH are shown with a description of each in Table 23. The samples that are directly comparable are the first 12 samples in the table and AX9002-200 No. 2. Note that AX9002-200 No. 2 and AX9002-200T-1A are taken from different subsamples of the same (AX9002) process samples under identical conditions.

Table 23

Table 24 shows the amphibole, non-amphibole, ambiguous, and total fiber concentrations for the samples shown in Table 23. The 95% confidence limits based only on the accuracy of the grid counting method used are given in Stevenson (1978), and typically are only within 30% of the given value, depending on the number of fibers observed. In general, the counts shown can be considered only as a general order of magnitude figure for the fiber concentration. Since the amount of crysotile present was very low, the data for crysotile is only shown in the total fiber column. For a particular fiber to be placed in the amphibole category, it had to give an electron diffraction pattern characteristic of amphibole minerals. A fragment with a clearly non-amphibole, non-crysotile diffraction pattern is classified as non-amphibole, non-crysotile. A fragment which clearly has a crysotile diffraction pattern is classified as crysotile. Mineral fibers classified as ambiguous have diffraction patterns or chemical ratios which cannot be used to place the fiber in one of the 3 previous categories.

Table 24

A calculation (Ashbrook 1978) was done for sample AX9002-200T, and the concentration found was 1.98×10^9 fibers per gram. This corresponds to an

Table 23. Description of samples analyzed for mineral fibers by MDH.

AX9001-200T-1A   200 M grind sample of AX9001 tailing slurry at <37 um.

US9001-200T-1A   200 M grind sample of US9001 tailing slurry at <37 um.

AX9002-200T-1A   200 M grind sample of AX9002 tailing slurry at <37 um.

AX9002-65T-1A   65 M grind sample of AX9002 tailing slurry at <37 um.

AX9003-200T-1A   200 M grind sample of AX9003 tailing slurry at <37 um.

AX9005-200T-1A   200 M grind sample of AX9005 tailing slurry at <37 um.

IP9003-200T-1A   200 M grind sample of IP9003 tailing slurry at <37 um.

IP9002-200T-1A   200 M grind sample of IP9002 tailing slurry at <37 um.

IP9002-65T-1A   65 M grind sample of IP9002 tailing slurry at <37 um.

DP9002-200T-1A   200 M grind sample of DP9002 tailing slurry at <37 um.

DP9002-65T-1A   65 M grind sample of DP9002 tailing slurry at <37 um.

AX9004-200T-1A   200 M grind sample of AX9004 tailing slurry at <37 um.

AX9002-200F   200 M grind of AX9002 feed, all sizes.

AX9002-200C   200 M grind of AX9002 concentrate, at <37 um.

AX9002-200T   200 M grind of AX9002 tailing slurry, all sizes.

AX9002-200 No. 5   200 M grind of AX9002 tailing slurry, at <37 um;
                      following reagitation of the beaker after 90% of the water
                      had been decanted and replaced by distilled water.

AX9002-200 No. 2   200 M grind of AX9002 tailing slurry, at <37 um;
                      sample AX9002-200T-1A is the same type of sample.

AX9002-200T-1F   200 M grind of AX9002 tailing slurry after settling
                      for 24 hr, without agitation.

AX9002-200T-1G   200 M grind of AX9002 tailing slurry after settling
                      for 48 hr, without agitation.

AX9002-200T-1H   200 M grind of AX9002 tailing slurry after settling
                      for 48 hr, reagitating and taking a sample at <37 um.

AX9002-200T-1J   MRRC distilled water sample.

Table 24. Concentrations of fibers (10^{12} fibers per liter).

SAMPLE	AMPHIBOLE	NON-AMPHIBOLE	AMBIGUOUS	TOTAL
AX9001-200T-1A	3.03	0.774	0.852	4.65
US9001-200T-1A	0.230	0.546	0.357	1.13
AX9002-200T-1A	0.588	0.912	0.366	1.86
AX9002-65T-1A	0.585	1.63	0.717	2.93
AX9003-200T-1A	10.6	5.31	1.87	18.1[a]
AX9005-200T-1A	0.414	0.867	0.414	1.69
IP9003-200T-1A	1.64	1.43	0.819	3.96
IP9002-200T-1A	1.88	2.63	4.32	8.85
IP9002-65T-1A	0.675	0.675	0.591	2.03[b]
DP9002-200T-1A	0.207	1.71	0.621	2.54
DP9002-65T-1A	0.0492	0.738	0.148	0.933
AX9004-200T-1A	0.182	1.23	0.318	1.73
200 M grind:				
Range	0.182-10.6	0.546-5.31	0.318-4.32	1.13-18.1
Average concentration	2.09	1.71	1.10	4.95
65 M grind:				
Range	0.049-0.675	0.675-1.63	0.148-0.591	0.933-2.03
Average concentration	0.436	1.01	0.368	1.61
AX9002-200F[c]	---	---	---	1.24
AX9002-200C[c]	---	---	---	1.08
AX9002-200T[c]	---	---	---	1.04
AX9002-200 No. 5 tailing (90% removed)	3.48	3.66	1.29	8.46
AX9002-200 No. 2 tailing	0.570	0.660	0.483	1.71
AX9002-200T-1F	0.477	0.510	0.375	1.36
AX9002-200T-1G	2.32	4.29	1.43	8.04
AX9002-200T-1H	0.981	2.08	1.83	4.89
AX9002-200T-1J	0.122×10^{-5}	0.735×10^{-5}	1.10×10^{-5}	2.2×10^{-5}

[a]One fiber of crysotile was observed.
[b]Two fibers of crysotile were observed.
[c]Only total fiber concentrations were determined for these samples.

average for the 2 minus 37 um samples (AX9002-200T-1A and AX9002-200 No. 2) of

1.78×10^{12} fibers/l of water. Therefore, using the conversion factor of

0.00111 times the fibers per liter values shown in Table 24, an approximation of

the number of fibers per gram can be made. Again, this is simply an order of

magnitude estimate, but is an important parameter needed for the estimation of

the potential fiber concentration in the air as a result of dry dust liftoff from

a tailing basin (see Volume 3-Chapter 3).

Since the transmission electron microscopy techniques used in this study did not

reveal whether fibers observed in processing products originated from asbestiform

or non-asbestiform minerals, it is useful to discuss the observed concentration

of fibers from a mineral present in the rock in a habit that is known to be non-

asbestiform. Such a reference mineral in this case is plagioclase, a feldspar

which is not known to be carcinogenic. Table 25 shows, in the first 3 columns,

the percentage of fibers contributed by each of the 3 classifications to the

total fiber concentration in the samples. In addition, the percentages for

plagioclase alone are shown in the fourth column. These percentages are also

included under the "non-amphibole" classification in the second column. By

averaging the amphibole and plagioclase percentages for all the samples, it is

found that the samples tested contained 2.2 amphibole fibers for each plagioclase

fiber. From Table 21 of Chapter 1, showing the mineralogical composition of

these samples, it is further calculated that on the average, the volume percent

of amphibole in the samples was 0.048 times that of plagioclase. Dividing the

first figure by the second reveals that on the average, a given amount of

amphibole in the samples formed 46 times as many fibers as an equal amount of

plagioclase.

Table 25

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Table 25. Fiber percentage of total.[a]

	TOTAL FIBER DISTRIBUTION			
SAMPLE	Amphibole	Non-Amphibole[b]	Ambiguous	Plagioclase[b]
AX9001-200T-1A	65	17	18	10
US9001-200T-1A	20	48	32	22
AX9002-200T-1A	32	49	19	30
AX9002-65T-1A	20	56	24	16
AX9003-200T-1A	60	29	12	10
AX9005-200T-1A	25	51	24	22
IP9003-200T-1A	42	37	21	18
IP9002-200T-1A	21	30	49	11
IP9002-65T-1A	35	35	39	6
DP9002-200T-1A	8	67	25	20
DP9002-65T-1A	5	79	16	29
AX9004-200T-1A	11	71	18	18
AX9002-200 No. 5 tailing (90% removed)	41	44	15	20
AX9002-200 No. 2 tailing	33	39	28	10
AX9002-200T-1F	35	38	27	23
AX9002-200T-1G	29	53	18	22
AX9002-200T-1H	20	43	37	20
AX9002-200T-1J[c]	6	38	56	0

[a]Includes all samples for which fiber counts were divided into the categories shown.
[b]Plagioclase percentages are included in the non-amphibole total as well.
[c]Distilled water blank.

The above result indicates that the tendency to form fibers, or fiber-like particles is much greater for the amphiboles than for plagioclase. The value of 46 cannot be taken as statistically significant by the nature of the work done, but it is certainly fair to say that amphibole fiber concentrations from crushing and grinding a given quantity of non-asbestiform amphibole will be from one to 2 orders of magnitude higher than those from processing an equal amount of plagioclase. On the other hand, it is important to note that due to the large amount of plagioclase present in the samples, the number of fibers or fiber-like fragments of plagioclase in the bench-scale tailing samples are often equal to or greater than the number of amphibole fibers. Thus, in reality there may be as many plagioclase fibers present in the tailing material from a full-scale mining operation as there are amphibole fibers. There was, however, a difference observed between the median aspect ratios (length divided by width) of plagioclase and amphibole fibers. The median aspect ratio for plagioclase fibers was 5.56, while the amphibole fibers in 200 and 65 M grind products had median aspect ratios of 6.70 and 6.92, respectively. Within the amphibole minerals themselves, trends in the median aspect ratios were observed. The amphibole minerals observed in the 200 M grind tailing samples were hornblende, cummingtonite, and actinolite. These mineral fibers had median aspect ratios of 5.08, 6.70, and 8.48, respectively. Thus, actinolite appears to form fibers with the highest median aspect ratio, while the median aspect ratio of hornblende is lower than that observed for plagioclase in the same samples. The proportions of these 3 amphibole groups contributing to the amphibole fiber counts in the 200 M grind samples were hornblende 29.7%, cummingtonite 54.7%, and actinolite 15.5% (see Table 26).

Table 26

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Table 26. Percent distribution of amphibole fibers in 200 M grind tailing
samples.

SAMPLE	TOTAL FIBER DISTRIBUTION			% OF AMPHIBOLE CONTRIBUTION TO TOTAL FIBER COUNT
	Hornblende	Actinolite	Cummingtonite	
AX9001-200T-1A	5.1	7.7	87.2	65.2
US9001-200T-1A	36.4	9.1	54.5	20.4
AX9002-200T-1A	31.0	17.2	51.7	31.6
AX9003-200T-1A	23.5	14.7	61.8	59.7
AX9005-200T-1A	40.0	10.0	50.0	24.5
IP9003-200T-1A	18.8	81.2	0.0	42.0
IP9002-200T-1A	87.5	0.0	12.5	21.3
DP9002-200T-1A	0.0	0.0	100.0	8.2
AX9004-200T-1A	25.0	0.0	75.0	10.5
200 M grind: range	0.0-87.5	0.0-81.2	0.0-100.0	8.2-65.2
average	29.7	15.5	54.7	31.5

The effect of the degree of grinding on fiber formation was investigated by comparing the fiber contents of 65 M and 200 M grind samples. There was insufficient data to justify quantification of the results, but certain qualitative observations can be made. As expected, the more a sample is ground, the higher the fiber concentration in the resulting products. However, there does not appear to be any systematic change in the observed median aspect ratios, or the proportion of amphibole fibers comprising the total observed fiber concentration. When the aspect ratios of the fibers formed during either of the test procedures used here are compared to the aspect ratios of ground material known to be truly asbestiform (Wylie 1978), the results confirm the thin-section observations. The low median aspect ratios observed for amphibole and non-amphibole fibers alike confirm that they are not truly asbestiform, but rather are acicular crystal fragments or cleavage fragments.

A series of samples were taken from a beaker containing a slurry of 200 M grind tailing material to investigate the settling properties of fibers. The results indicate that the longer fibers (longer than 2 um) do tend to settle out in from one to 2 days. However, these fibers comprise a small fraction of the total fibers present, and no significant change in overall fiber count was observed over a 48-hr period. Further, upon carefully decanting and replacing 90% of the water with distilled water after 48 hr and reagitating, it was found that the resulting water actually contained a higher fiber concentration than was initially present. A significant fraction of the fibers present appear to be trapped by larger mineral fragments during settling, and are then present in the mass of tailing material. This trapping effect may even be enhanced initially by the presence of residual processing reagents. Whatever the source of the trapping effect, the observed results indicate that resuspension of the tailing

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at a later date, followed by settling, results in less trapping and a higher fiber concentration in the residual water.

The above observations have important implications for the long-term disposal of tailing material. Assuming the hypothesis given, no matter what treatment the recycled or discharged tailing water undergoes to control the release of fibers, the tailing material itself retains a significant fraction of the fibers generated. If tailing material is exposed to wind erosion, the resultant fugitive emissions will most certainly contain mineral fibers. If the tailing material is subject to water erosion, the resulting discharges will most certainly contain mineral fibers. In terms of water treatment for fiber removal, continuous pilot plant work at MRRC indicates that the use of a flocculant in the clarification of process water prior to recycling reduces the fiber content by 4 to 5 orders of magnitude. However, this means the fibers are then essentially all retained in the tailing material, to pose the potential for future release when active control practices cease.

All the information generated by the above studies presents quite a dilemma when an attempt is made to use it in the assessment of potential environmental impacts. As currently defined, mineral fibers will most certainly be present in the products of mineral processing. Typical tailing slurries may contain from 10^{12} to 10^{13} fibers/l of which some 20 to 30% or more may be amphibole fibers. This corresponds to some 10^9 to 10^{10} fibers/gm of dry tailing material in a basin. When compared with estimates of the amphibole fibers present in the tailing produced by Reserve Mining at Silver Bay, the results show that the processing of Duluth Complex material may produce roughly 1/3 the concentration of amphibole fibers present in Reserve's tailing material (Stevenson 1978). However, it has also been found that the fibers observed do not originate

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from minerals in the ore which are present in asbestiform habits. The occurrence of such habits is expected to be rare. The aspect ratios of the resulting fibers are thus quite low in comparison to those formed from truly asbestiform minerals. In fact, it has been observed that a large fraction of the fibers are plagioclase. This is quite a common mineral in the earth's crust, and it is expected that such fibers may naturally occur in most areas as a result of cleavage fragment formation by the forces of wind and water acting in nature.

Since the mechanisms by which some fibers are harmful to human health are not yet clearly understood, this study points out the importance of continued research into the pathology of mineral fibers. Even such a basic mineral constituent as plagioclase forms fiber fragments which meet the definitions of mineral fibers as used in this study. If it is important to control these fibers, or fibers of selected mineralogy or aspect ratios, the pathology must be known to allow a specific control program to be designed and instituted.

2.5 METAL PROCESSING OUTPUTS

To complete the picture presented by the discussions of ore, concentrate, and waste materials just given, it is appropriate to discuss the final outputs from the last stage of metal processing, the smelter/refinery stage. This completes the material balance by showing the fate of the constituents in the concentrate fed to the smelter. This topic is dealt with in more detail in Chapters 4 and 5 of the technical assessment report (Volume 2). The results are briefly summarized here.

Even though a concentrate model has been given as a feed for a hypothetical smelter, the various output products will not be presented as specific models. Rather, their composition and quantities will be discussed in summary form, with

emphasis being placed on typical ranges of composition and quantity. The reasons

for this are two-fold:

1) A variety of technologies and process flow-sheets are applicable to the
recovery of metal from a concentrate. As noted earlier, this includes the
possibility that the bulk concentration approach, used in this report, would not
be used. Rather, 2 separate concentrates may be produced. Even without this
complication, the range of possible treatment options available imply a range of
possible compositions for the various outputs, since the ultimate fate of a given
elemental constituent is in part dependent on the treatment used.

2) The additives to the concentrate(s) input material also may vary. Depending
on process requirements, varying amounts of silica flux, lime or limestone, coal
and/or coke, and possibly other constituents must be added. The outputs then
will vary accordingly.

Table 27 summarizes these inputs and outputs. Each will be discussed in turn.

Table 27

2.5.1 Metals

The basis for the models used in the technical assessment discussions is a

smelter/refinery facility producing 100,000 mt of copper and nickel metal per

year. Accordingly, the production of such a facility will yield 80,000 to 90,000

mtpy of cathode copper and 10,000 to 20,000 mtpy of cathode nickel, all likely to

be in the form of large ingots. In addition, 350 to 450 mtpy cobalt would be

generated, probably in the form of a metallic powder. Finally, precious metals

would be recovered from the copper refinery slimes and spent electrolyte.

Silver is recovered from the spent electrolyte, while gold, platinum, and pala-

dium are recovered from the slimes. Only a small number of mineral samples

available to the Study were subjected to precious metals analysis (Iwasaki et al.

1978), so the results must be interpreted with caution. Table 28 shows, to one

significant figure, a preliminary estimate of the annual production of precious

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Table 27. Basic inputs and outputs for a smelter/refinery operation.

INPUTS	OPERATION	OUTPUTS
Concentrate(s) Silica flux Lime/limestone Fuel (coal, coke, etc.)	Sulfuric acid plant and emission controls Smelting Converting Refining	Sulfuric acid Copper, nickel, cobalt Precious metals (Au,Ag,Pt,Pd) Slag Sludge Other (metallic dusts)
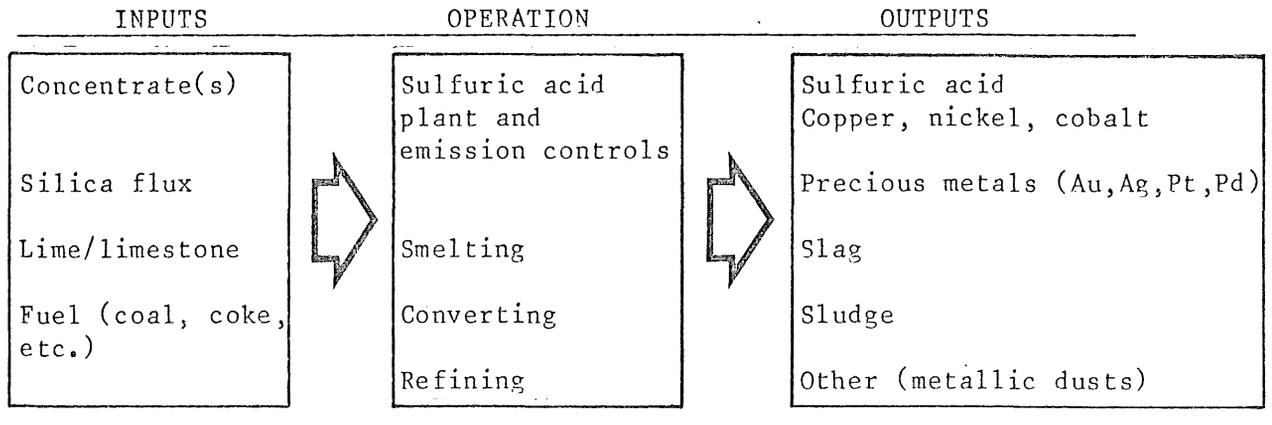
metals from an operation processing 635,000 mtpy of the model concentrate. The
values are the averages of 6 analyses of bench scale and pilot plant con-
centrates, and they assume 100% smelting/refining recovery of the precious metals
contained in the processing concentrate. These values represent a considerable
annual income to the mining company, as is discussed in Volume 5-Chapter 17 of
this report.

Table 28

2.5.2 Sulfuric Acid

The primary gas streams from the smelter and copper converters, as well as other
gas streams, contain high levels of sulfur in the form of SO_2. These gas
streams would be sent to a sulfuric acid plant, where typically 97% to 98% of
incoming SO_2 would be converted to sulfuric acid (98% pure). This acid would
likely be sold, if a suitable purchaser is found, or neutralized with lime for
disposal. Depending principally on the SO_2 control system and the acid plant
conversion efficiency, from 400,000 to 500,000 mtpy of acid would be generated.

2.5.3 Slag

The iron contained in the concentrate is removed in the molten state by combining
it with silica (SiO_2) to form an iron silicate slag which is skimmed from the
smelter or converter, leaving the valuable metals behind. After appropriate
cleaning operations to recover trapped Cu, Ni, and Co, the slag is disposed of,
usually in a pile near the smelter. Also, some of the slag may be sold, for
example, as road building material if a market is available. To generate the
slag, sufficient silica must be added as a flux to combine with the iron.
Calcium is also added, typically as lime, to improve the separation between the

Table 28. Annual production of precious metals from a model
smelter/refinery operation: a preliminary estimate.

	Au	Ag	Pt	Pd
Troy ounces	30,000	900,000	10,000	50,000

slag and the metal matte. The added silica and calcium, along with the Al and Mg present, accompany the iron to the slag. Typically, 100,000 to 150,000 mtpy of silica flux and 20,000 to 30,000 mtpy of lime would be added. In addition, coal and coke might be added, principally as fuels. In the event these fuel forms are used, rather than natural gas or electricity, some 30,000 to 50,000 mtpy of coal and coke would be required. These fuels contain SiO_2, Al, Mg, and Ca which would go to the slag. The resulting slag production would amount to some 500,000 to 650,000 mtpy.

2.5.4 Sludge

Sludges would be produced from several portions of the operation, depending on the facilities chosen. Examples are sludges from the electrolytic refineries and the sulfuric acid plant. If SO_2 gas scrubbers are used, these would produce sludges, as would water treatment plants. Many sludges could be recycled as a flux to the smelter since they contain calcium. They may also be disposed of on land. Sludge generation might range between 10,000 and 40,000 mtpy, assuming no neutralization of sulfuric acid is required. In the extreme, if all the sulfuric acid must be disposed of by neutralization to calcium sulfate, an additional 550,000 to 700,000 mtpy would be created, but such a possibility is highly unlikely. This would occur only if no market for the acid can be found, which is not expected to be the case since sulfuric acid is a valuable commodity with many commercial applications.

2.5.5 Other

A range of miscellaneous other outputs might also occur. These would be present in small quantities and will not be discussed other than to mention some of them briefly. Included here might be non-recyclable metallic dust, a weak acid stream

from the sulfuric acid plant, SO_2 and particulates which are released to the atmosphere, and blowdown from any cooling towers or water treatment facilities. Many of these are discussed in detail in the technical assessment, air, and water sections of this report (Volume 2-Chapter 4).

Table 29 summarizes the outputs discussed above.

Table 29

Table 29. Summary annual outputs from a model smelter/refinery operation producing 100,000 mtpy of copper and nickel metal.

OUTPUT CONSTITUENT	QUANTITY RANGE
Metals	
Cu	80,000-90,000 mtpy
Ni	10,000-20,000 mtpy
Co	350-450 mtpy
Au	30,000 troy oz/yr
Ag	900,000 troy oz/yr
Pt	10,000 troy oz/yr
Pd	50,000 troy oz/yr
Sulfuric Acid	400,000-500,000 mtpy
Slag	550,000-650,000 mtpy
Sludge	10,000-40,000 mtpy

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