Preliminary Report

Mining and Metallurgical Technology

Regional Cu-Ni Study

Minnesota EQC

by

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FORWARD

This preliminary technology assessment report is the first effort of the Mining and Metallurgical Technology team of the Regional Copper-Nickel Study, in presenting the data necessary to evaluate the possibility of copper-nickel mining in Minnesota. This is the groundwork, the basic information which will be modified, cultivated and molded during the next two years into a final study report for consideration by the 1979 legislative session.

Our mission is not to recommend mining plans for copper-nickel, but solely to offer the best possible information concerning such mining and its total impacts, so the Legislature will be able to make wise and just decisions concerning the future of Minnesota's copper-nickel resources.

The best estimate to date 1/ indicates about 6.5 billion tons of resource material averaging 0.6% Cu and 0.2% Ni as sulfide minerals, contained in the contact area between the Duluth Complex intrusive and the underlying rock formations. The mineralized zone is very irregular. It outcrops in a semicircular band from south of Duluth northwest through the Aurora-Hoyt Lakes Area, then northeast through Ely and toward Grand Marais. The contact dips at a fairly steep angle of 20 to 40° to the southeast, thus both open pit and underground mining operations would be possible.

The mineralized zone sulfides containing copper and nickel are chalcopyrite, cubanite and pentlandite, all of which respond to a simple beneficiation flowsheet involving crushing, grinding and flotation. Flotation procedures at natural pH require only a collector such as xanthate and an alcohol-type frother to recover up to 90% of the copper and 65% of the nickel as combined sulfide concentrates.

1/ P.K. Sims, MGS, 1972.

These concentrates could then be treated by a number of secondary metallurgical processes to produce whatever final stage produce is desired. A fair value of precious metals recovered with the sulfides is also recoverable.

This report begins with a general discussion of the regional geology as condensed from the published literature. More information will become available as current studies on hydrology, surficial geology, bedrock geology and mineralogy are completed. These studies should be extremely helpful as, for example, a more detailed outline of the potential regional ore zone will be generated.

The next section is a series of professional opinions on mine and mill design. Due to the limited information available and the time restraints placed on the team, the author had to be rather bold in his assumptions and conclusions in order to produce a report of this caliber. Qualifications exist and are pointed out at the beginning of this section, read them carefully.

The final section of this report concerns secondary metallurgy - pyrometallurgical or hydrometallurgical techniques which are necessary to produce a saleable product from the flotation concentrate. Let me assure you, "SMELTER" is not the dirty word it once was. Read this summary of the techniques available and consider the claims for SO_2 and particulate matter removal.

Basically, this report is a summary of the processing techniques available as applied to Minnesota's great resource potential of copper-nickel. Again, remember the qualifications when you read and digest our findings.

The Mining and Metallurgical Technology team has spent a great deal of time and effort on this report. We now want your input, particularly as it affects your area of interest in the regional study. To those of you in the Study group who will be attending our September 10 seminar, please prepare your questions and comments for the meeting, and we will attempt to answer them at that time.

David L. Veith, Coordinator
Mining & Metallurgical Technology

Section A

Geology

GEOLOGY

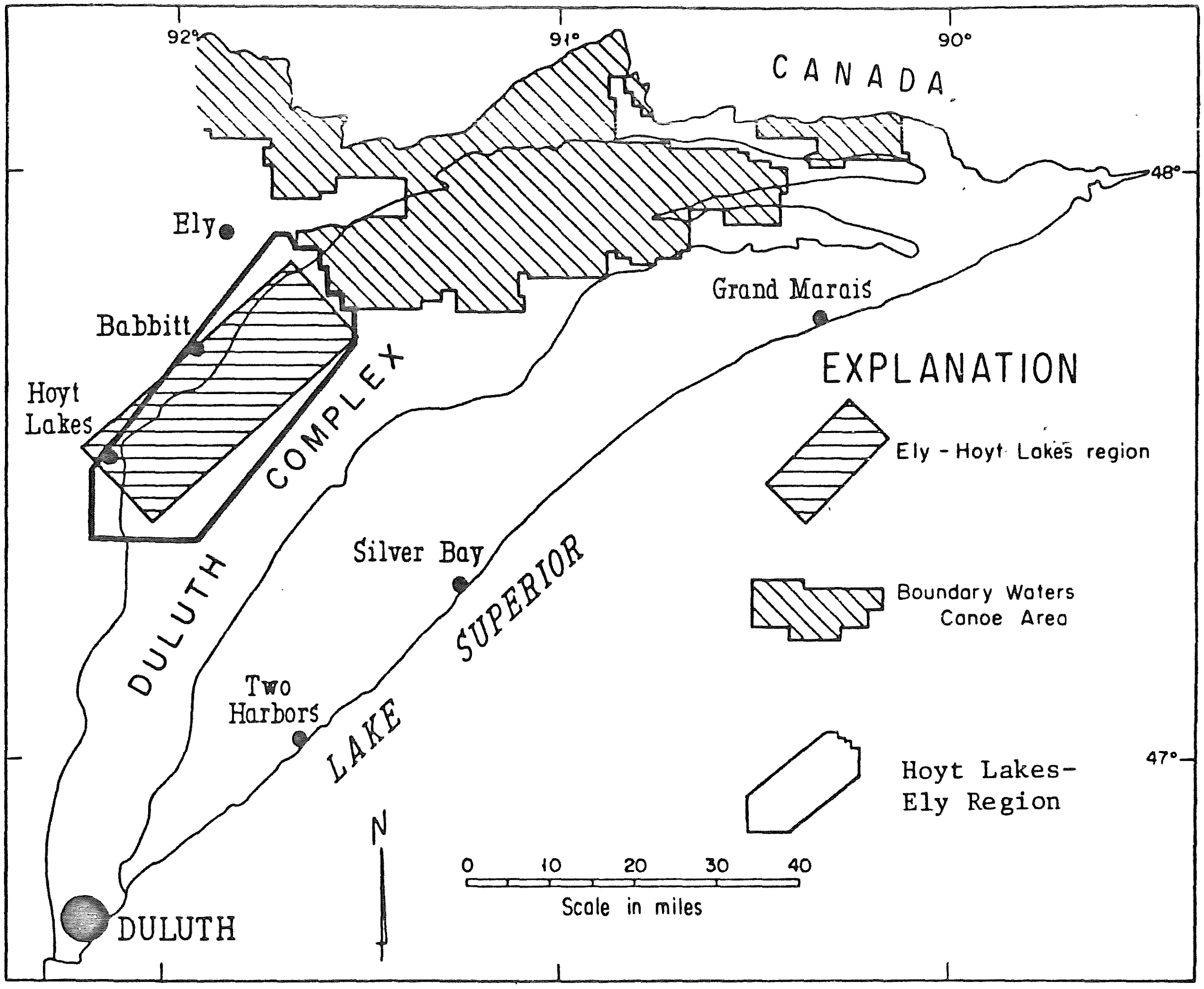
This section attempts to describe the geology of the Hoyt Lakes - Ely Area with particular emphasis on the geologic conditions influencing mine design and its potential environmental impacts.

The Minnesota Land Management Information Survey (MLMIS) has generated a computer based area delimiting the most economically attractive area for copper-nickel mining consideration, hereafter in this report designated as the Hoyt Lakes-Ely Area (Map A1). Based on 24 drill hole assays in this area, Bounichsen has estimated, using a cutoff grade of 0.5% Cu + Ni, that >2 billion tons of mineralized rock averaging 0.8% Cu + Ni probably exists at depths of 3000 ft. or less (Table A1). Besides Cu and Ni, significant amounts of economically recoverable Co, Ti, Al, Au, Ag, Rh, Pt and Pa with the possibility of other Platinum Group Metals (ie: Ir, Ru and Os) exist within the Duluth Complex.[A20] Investigators have also reported trace amounts of Va, Cr, Zn, Pb and Mo.[A10]

GEOCHRONOLOGY

Bedrock (Precambrian)

There are three major rock groups that have to be considered closely in this study, characterizable by age as well as rock type(s) (Table A2). The earliest are early Precambrian greenstones, metavolcanics, metasediments, granitic rocks and migmatite dated at 2.6 to 2.7 billion years old. In the middle Precambrian, 1.8 to 2.0 billion years ago, the Animikie group of rocks (consisting of the basal Pokegawa Quartzite, Biwabik Formation and Virginia Formation) was deposited in a subsiding basin lying roughly where the present Lake Superior basin is. By late Precambrian times, roughly 1.1 billion years ago, the "Keweenawan rift" began splitting this basin



Map A1 Map of northeastern Minnesota, showing the location of the Ely-Hoyt Lakes region relative to the Duluth Complex and to the Boundary Waters Canoe Area.

Table A1 Estimated tonnage, value, and grade of copper-nickel deposits in the Ely-Hoyt Lakes region for calculations A and B.

	Calculation A ($33.2\ mi^2$)		Calculation B ($15.3\ mi^2$)	
Cutoff grade	0.25%	0.50%	0.25%	0.50%
Tons* of mineralized material	14.30×10^9	5.85×10^9	5.54×10^9	2.24×10^9
Tons of metal (Cu + Ni)	78.60×10^6	49.18×10^6	29.81×10^6	18.42×10^6
Tons of copper**	58.95×10^6	36.89×10^6	22.36×10^6	13.82×10^6
Tons of nickel**	19.65×10^6	12.29×10^6	7.45×10^6	4.60×10^6
Gross value of metal***	$117.9 Billion	$73.8 Billion	$44.7 Billion	$27.6 Billion
Average grade (Cu + Ni)	0.55%	0.84%	0.54%	0.82%

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* Short tons (2000 lbs.)

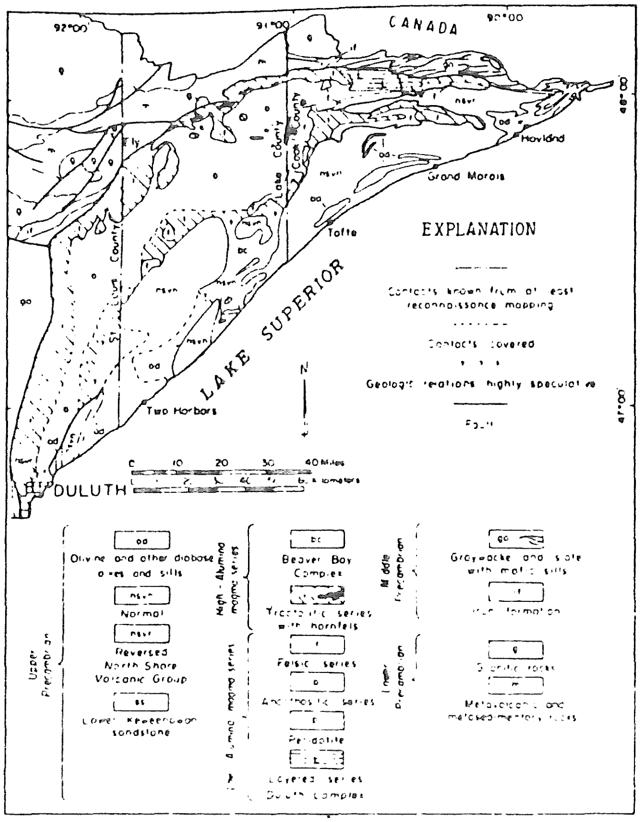
** Assuming Cu:Ni ratio of 3:1

*** Assuming a copper price of 50¢/lb. and a nickel price of $1.50/lb.

Table A2

Time-stratigraphic framework of Precambrian rocks in Minnesota

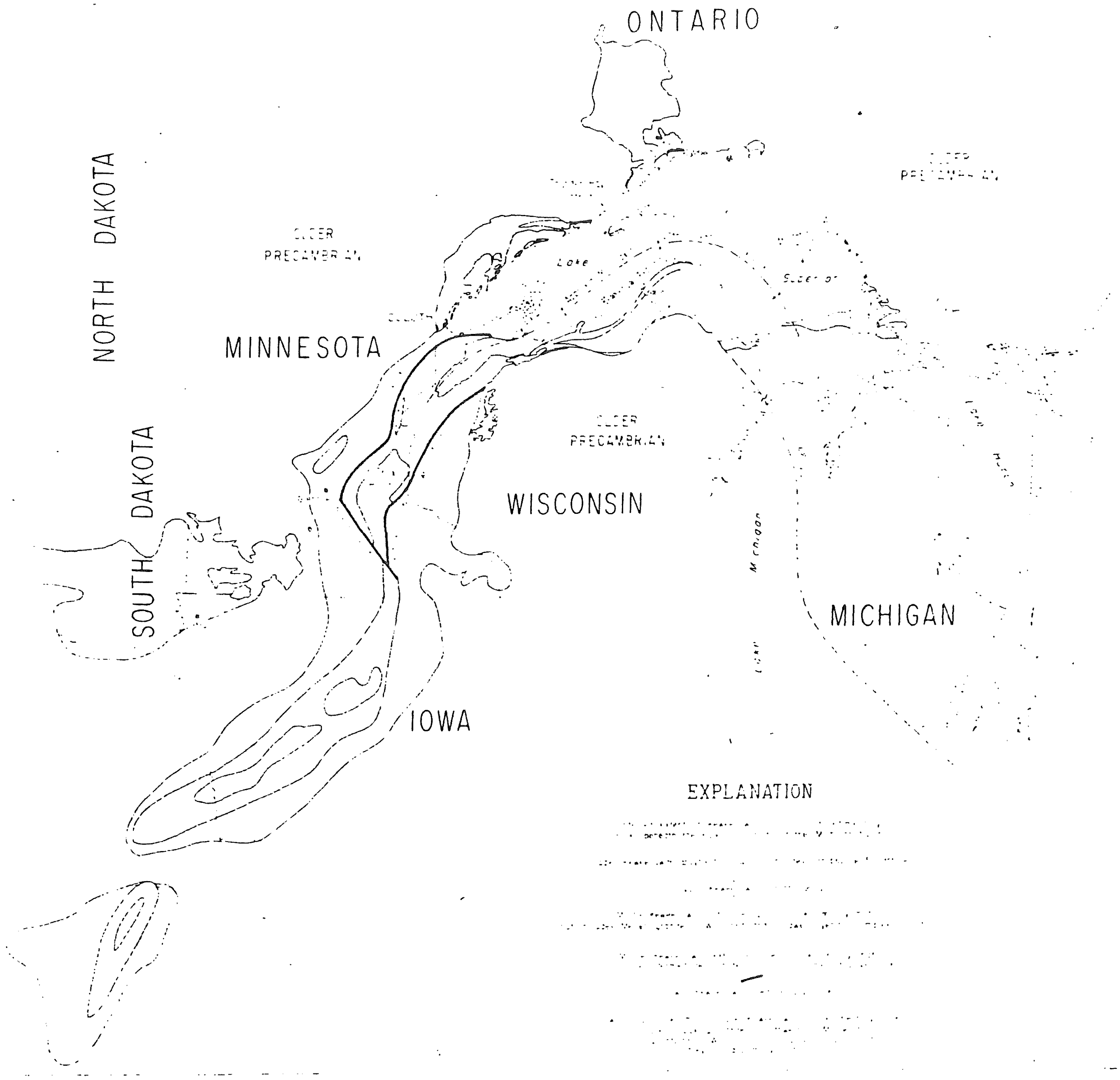
Era	Lithology	Event (Approximate age)	Intrusive Rocks
Paleozoic	------------------ Unconformity ------		
600 m.y.	Clastic rocks having dominantly fluvial attributes		
	------disconformity------		Duluth Complex and smaller mafic intrusions in northeastern Minnesota
Late Precambrian	Mafic lava flows, tuffs, and interbedded sedimentary rocks		
	-------Unconformity------		
	Clastic rocks having shallow-water attributes	Deformation and metamorphism (?) (ca. 1,200 m.y.)	Diabasic gabbro dikes in northern and east-central Minnesota
		Unnamed period of mild deformation and metamorphism in NE. Minn. about 1,600-1,700 m.y.	
1,800 m.y.	------------------Unconformity------	Penokean orogeny (ca. 1,850 m.y.)	Granitic plutons in east-central Minnesota and small plutons of granitic and mafic rocks in Minnesota River Valley
	Clastic rocks and iron-formation having shallow-water to deep-water, or miogeosynclinal, attributes		
Middle Precambrian	-------disconformity------		
	Clastic and carbonate sedimentary rocks having shelf-like attributes		Dikes of diabasic gabbro and diorite in northern Minnesota (stratigraphic position relative to the shelf-like strata unknown)
2,600 m.y.	------------------Unconformity------	Algoman orogeny (ca. 2,700 m.y.)	Small syenitic plutons in northern Minnesota; Granitic rocks of Vermilion, Giants Range, and Saganaga batholiths in northern Minnesota, McGrath gneiss in east-central Minnesota, and Sacred Heart and Ortonville plutons in southwestern Minnesota
Early Precambrian	------------Unconformity?------		
	Gneisses of igneous and sedimentary derivation	Mortonian event (ca. 3,550 m.y.)	Morton Gneiss and Montevideo gneiss, at least in part, in southwestern Minnesota



Generalized bedrock geologic map of the Duluth Complex and associated rocks.

(Map A2). The Keweenawan rocks are part of the Mid-Continent Gravity High, a belt of fault bounded blocks cutting across older, less dense rocks in a six state path. This continental rifting was accompanied by great volumes of lava flows and intrusive bodies, including the North Shore Volcanics and the Duluth Complex ("Duluth Gabbro"), which is subdivided into the Anorthositic and Troctolitic Series. Since Cu-Ni mineralization is confined largely to this third major group of rocks and its two subdivisions, their structural and petrologic evolution merit further comment. Keweenawan rifting, faulting, fracturing and accompanying igneous activity occurred during two stages. In the first stage, as the volcanic pile increased in thickness, upwelling magma tended to travel between the formations of the Animikie Group and along the contact between the Animikie Group and earlier Precambrian rocks. During an assumed period of tectonic acquiesence, low-Al magma gravity segregated as crystal mushes into rocks of the Anorthositic Series: peridorites, anorthosite and late local granophyre rocks (Figure A1). Before the Anorthositic Series completely crystalized, the second stage of rifting disrupted the initial volcanic edifice and created new magma chambers of a second parent magma, high in Al. Isolated magma chambers produced the Troctolitic Series of rocks that were intruded as pulses as indicated by interfingering layers of different (compositionally and texturally) troctolitic gabbro. Textural relationships indicate a crystallization sequence as follows (earliest forming crystals first).[A1]

Anorthositic Series Rocks:	Troctolitic Series Rocks:	Textural Changes:
spinel	spinel (?)	large, euhedral crystals
plagioclase	olivine	↓
plagioclase & olivine	olivine & plagioclase	subhedral & interstitial crystals
olivine	plagioclase	
clinopyroxene	clinopyroxene, pyrrhotite, pentlandite	↓
magnetite & ilmenite	magnetite & ilmenite, chalcopynite, cubanite	grains filling voids, exsolution & replacement in previously formed grains
biotite		



Map A2 Generalized geologic map of the Lake Superior region showing the known and inferred distribution of Upper Precambrian rocks. Compiled by Campbell Craddock (1971) from various sources including the following: Ontario Department of Mines Maps 2108, 2137, 2198, and 2199; Bayley and Muehlberger, 1968; Sims, 1970; Dutton and Bradley, 1970; Leith and others, 1935; King and Zietz, 1971; Hamblin, 1958; Halls, 1966; Coons and others, 1967; Hinze, 1963; Meshref and Hinze, 1970; White, 1966a; Farnham, 1967, unpub. Ph.D. thesis, Univ. Minn.; and Craddock, this chapter.

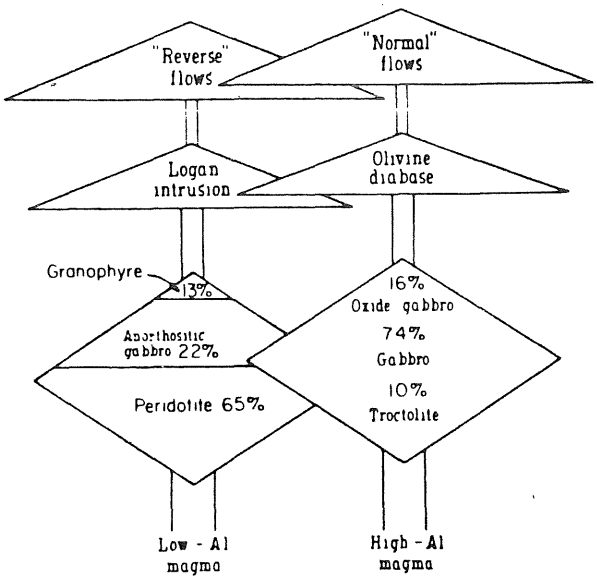


Figure A1 **Schematic diagram summarizing results of petrologic mixing calculations discussed in text.**

Boucher has suggested that a later, Cu-bearing solution entered the still porous, fractured rock after the clinopyxoxene, pyrrhotite and pentlandite crystallized filling voids and replacing some pyrrhotite with intergrowths of chalcopyrite and cubanite.

Apparently, by the end of the Precambrian (570 million years ago) rifting had abated, resulting in approximately 8 km. of actual separation. Although faulting is thought to have occurred mainly during emplacement, there is no reason to assume motion along these faults closed entirely with the cessation of magmatic activity in Middle Keweenawan time. Many of the faults associated with the Midcontinent Gravity High in southeastern Minnesota were periodically active during Cambro-Ordovician time, and perhaps during Pleistocene time.[A35]

Later erosion removed much of the volcanic edifice and led to the present exposure of crystalline rocks that outcrop in an arcuate shape for 150 miles from Duluth towards the Arrowhead Region of Minnesota. Repeated glaciation in the Pleistocene left 0 to 85 ft. (ave. < 30 ft.) of till, ice-contact deposits, outwash, moraines, crevasse fillings and other glacial features.

Unconsolidated deposits (Pleistocene-Recent)

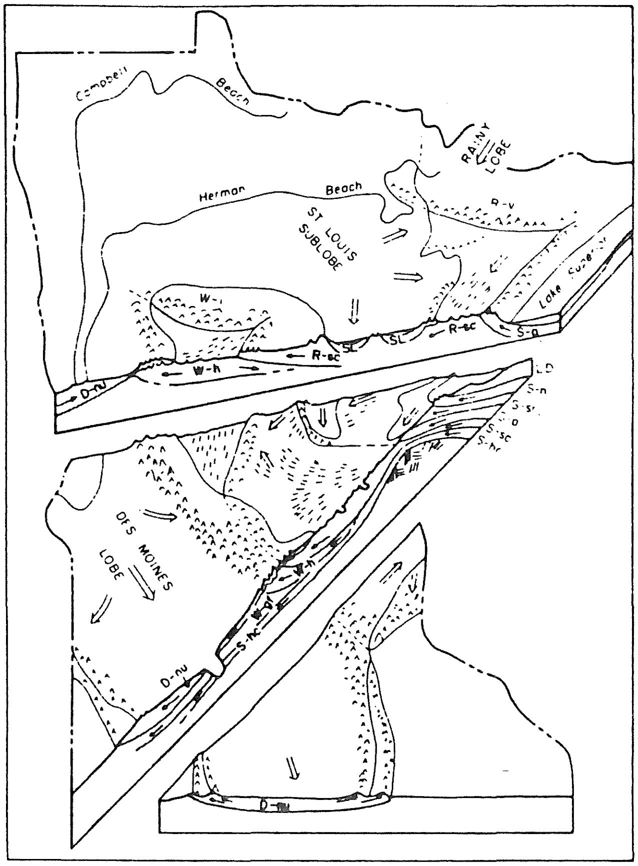
In Minnesota, the sequence of glaciation is represented as a series of glacial phases, each named on a local basis and associated with a particular lobe(s). Any definite time correlation (based on carbon dating, stratigraphic, and/or geomorphic relationships) between lobes is indicated by combined names, (e.g., Automba-Vermilion phase of the Superior and Rainy Lobes)(Map A3).

Itasca-St. Croix phase of the Wadena, Rainy and Superior Lobes:

About 20,000 years ago, the Rainy Lobe moved southwestwardly, from Canada, across the upland of crystalline rocks, while the Superior Lobe moved up and out of the Lake Superior Basin into the Minneapolis lowland (Map A4). In central Minnesota, the two lobes flowed side by side as one lobe depositing the St. Croix Moraine at the limit of advance, about 200 miles south of Ely (Map A5). As the lobe retreated, it split into the two original lobes. The toimi drumlin field was produced by the retreating Rainy Lobe as a series of roughly 1400 drumlins trending S-45°W (Map A6).[A37] Drumlins formed as the thinning ice lobe retreated; some sort of knob or surface irregularity started a "plastering" action in which layer upon layer of bouldarly clay was deposited as the debris-choked ice lost its carrying capacity.[A37]

The long axis of a drumlin and tapered end indicate the direction of ice flow at that area. The Toimi drumlins average about 50 feet high and one mile long, consisting of grey, sandy, stony (mostly gabbroic rock) till.[A37]

This till is loosely consolodated; presumably any cementing clay or silt fraction has been eroded away. The drumlins are buried under younger drift south of the St. Louis River and truncated on the north by the Vermilion moraine.



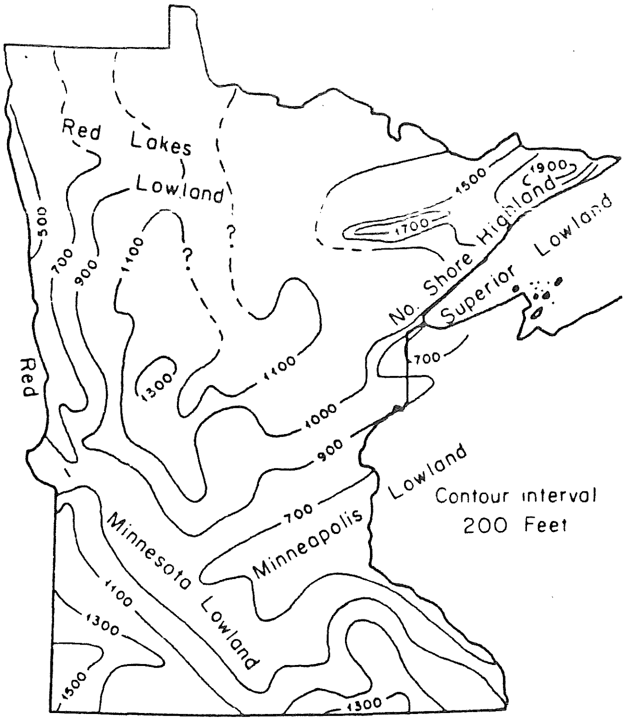
Map A3 Block diagram of Minnesota showing composite of main phases of glaciation, with three cross-sections. Arrows show direction of ice movement. Main moraines and drumlin fields are also shown. Initials stand for the following ice lobes and phases:

Des Moines lobe
- D-nu New Ulm phase

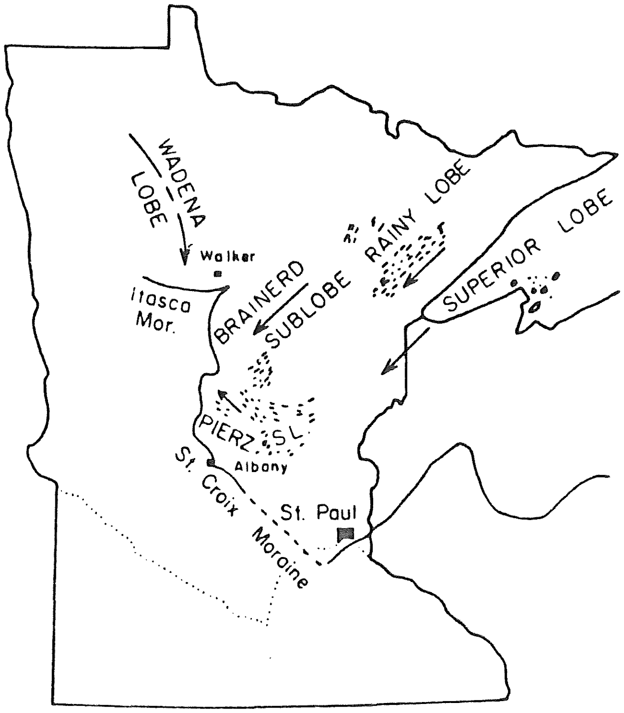
Wadena lobe
- W-i Itasca phase
- W-h Hewitt phase
- W-gf Granite Falls phase

Rainy lobe
- R-v Vermilion phase
- R-sc St. Croix phase

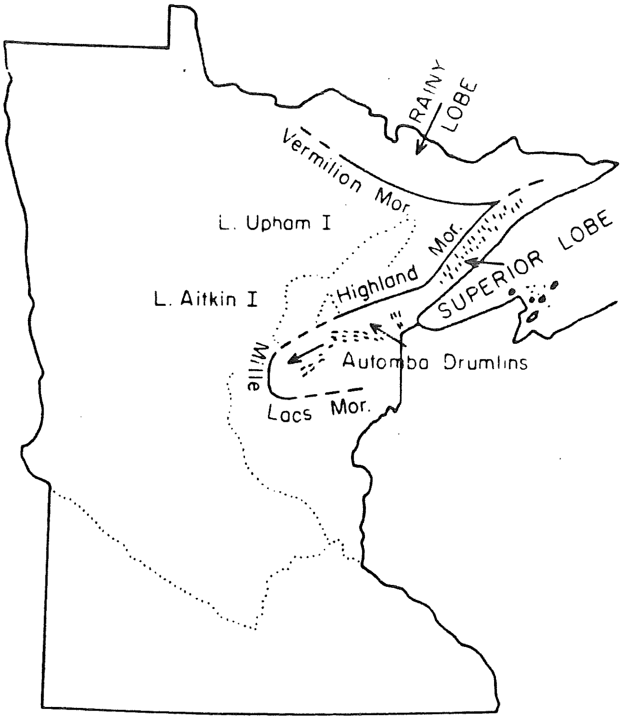
Superior lobe
- LD Lake Duluth
- S-n Nickerson phase
- S-sr Split Rock phase
- S-a Automba phase
- S-sc St. Croix phase
- S-hc Hawk Creek phase



Map A4 Bedrock topography



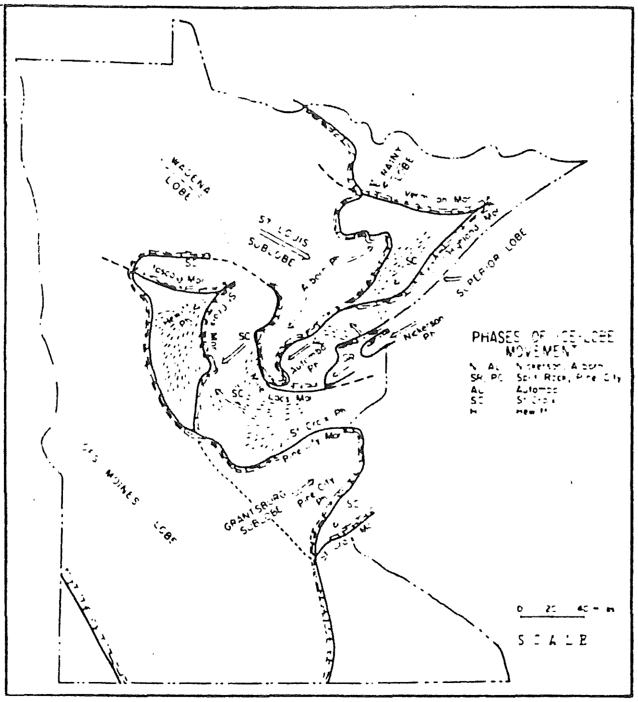
Map A5 St. Croix phase: advance of Superior and Rainy lobes to St. Croix moraine; formation of drumlin fields by sublobes



Map A6 Automba phase of Superior and Rainy lobes: advance to Vermilion moraine

Automba phase of the Superior and Rainy Lobes:

Following the readvance of the Rainy and Superior Lobes, the Vermilion and Highland moraines were deposited contemporaneously. Near Isabella, the two moraines meet at a 30° angle (Map A7). Eskers developed at this junction and blocks of "stagnant" ice were left stranded within and on moraines and outwash plains (Figures A2 and A3).[A37]



Map A7 Composite map showing main phases of Wisconsin glaciation in Minnesota. Short dashes show drumlin fields.

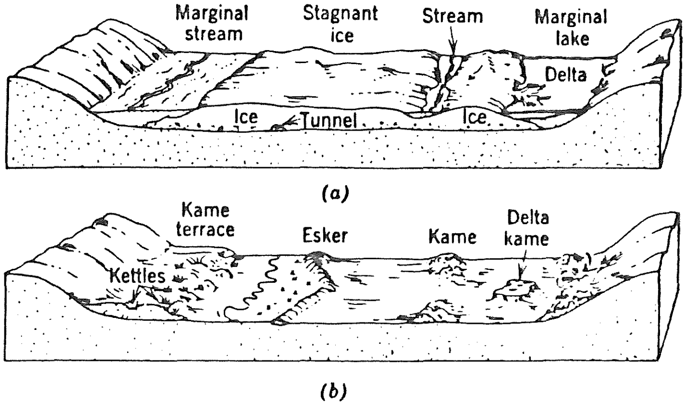


Figure A2 Ice-contact deposits associated with a mass of stagnant ice. The upper diagram shows the deposits being formed, the lower diagram after the ice has disappeared.

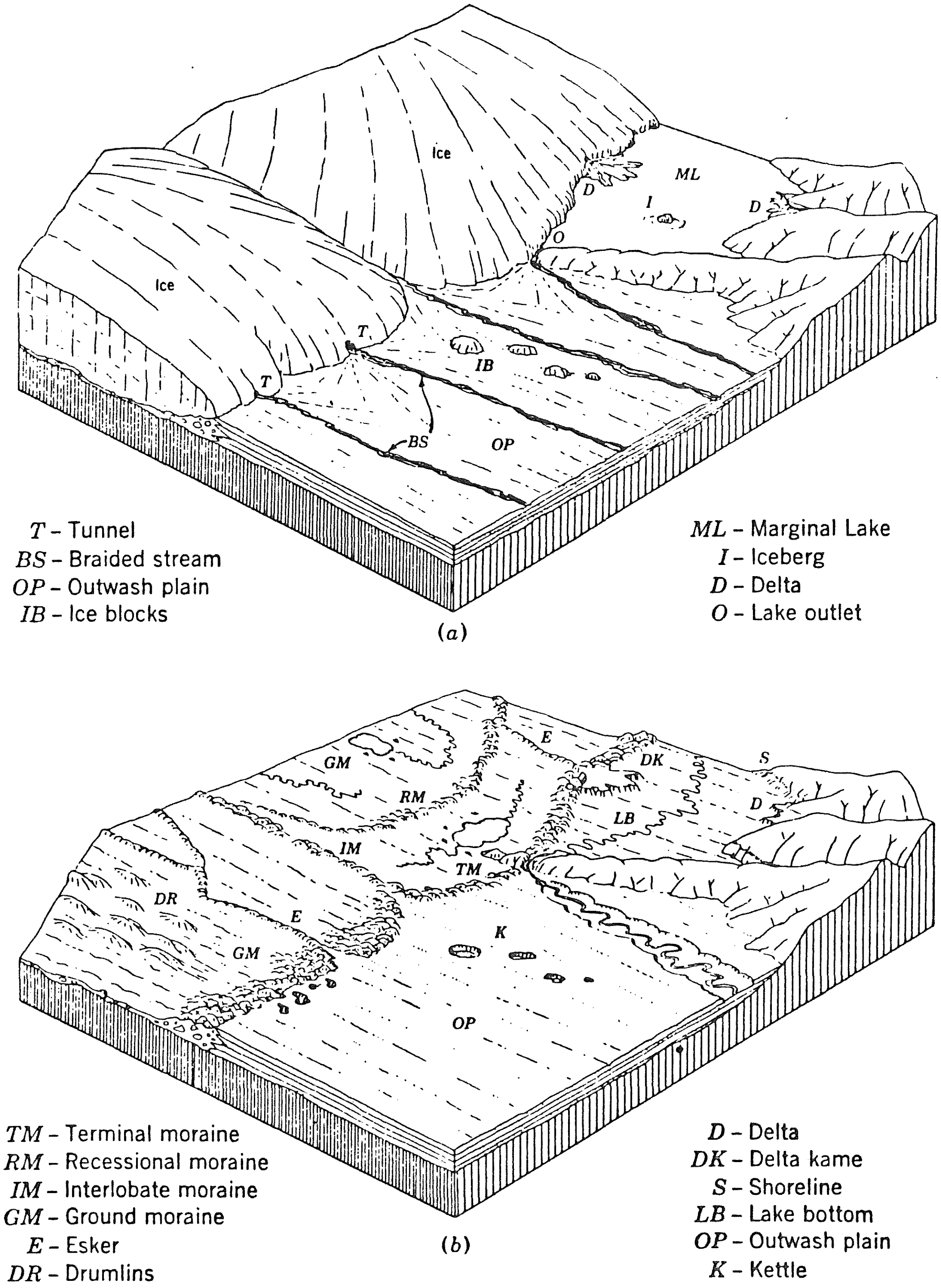


Figure A3 Landforms produced by deposition near the margin of a continental glacier.

Post glaciation:

The stagnant ice blocks, melted slowly since only a few feet of glacial drift is needed to insulate a mass of underlying ice; sometimes a cover of soil and forest can develop over stagnant ice. Eventually, the buried ice melted, forming depressions or "kettles" that filled with water. Lakes with no rapidly eroding outlet streams, filled with sediments, eventually becoming bogs and marshes. Radiocarbon dating of early sediments indicate that some kettle lakes did not come into existance until thousands of years after ice had left the region.[A38] (Table A3)

Table A3. Quaternary history of Hoyt Lakes-Ely Area Wisconsin Glaciation

Years ago	Glacial Feature
15,000 (St. Croix Phase)	Rainy Lobe advanced in S 20° W direction creating Roche Moutonnées preferentially in gabbro and glacial striations
(interstade)	Rainy Lobe retreated depositing ground moraine (thickening to the south) in between a series of east-west trending end moraines; glacial erratics incorporated within and on top of glacial debris; stagnant ice blocks deposited, later forming kettle lakes and stagnation gravel deposits. Superior Lobe melt waters draining to the east; Perched deposits of well-bedded sand and gravel deposited.
(Vermilion Phase) and end of glaciation to present	Rainy Lobe readvanced depositing the Vermilion Moraine and a veneer of glacial debris northward from the moraine; Glacial outwash deposits formed and sedimentation of lakes, bogs, etc.

Within the Hoyt Lakes-Ely area, the first advance of the Rainy Lobe (about 15,000 years ago) moved S 20 ° W, making glacial straitions in the exposed bedrock and preferentially scouring gabbronic bedrock into streamlined landforms called Roches Moutonnées.

During the following interstade, ground moraine was deposited between a series of east-west trending end moraines (Map A8). Generally, these deposits thicken to south (Map A9). As the ice melted, large glacial erratics of local bedrock were deposited within and on top of glacial debris. The glacial ice apparently did not transport debris for great distances, as the lithic character of the drift changes abruptly as contacts in the bedrock are crossed. Large stagnant ice blocks were stranded, later melting to form kettle lakes and stagnation gravel deposits. Apparently, in these deposits, the finer fraction (clay and silt size) was carried off by meltwaters and the coarse fraction was concentrated.

Glacial meltwaters from the Superior Lobe apparently were draining to the west and had established a large watershed draining from the east through the present topographic low now occupied by the Stoney River. Perched deposits of well-bedded sand and gravel developed as sand and gravel from the Superior Lobe were carried by meltwaters and deposited between blocks of stagnant ice.

In the Vermilion Phase, the Rainy Lobe advanced again and upon retreating, deposited the Vermilion Moraine and veneer of drift to the north (Table A4).

As the ice left for the last time, glacial outwash deposits were formed (Qo) and favorable topographic lows filled with locally derived silt and clay (Qsal). Vegetation was established and peat as much as 12 feet deep has accumulated in bogs (Map A10).

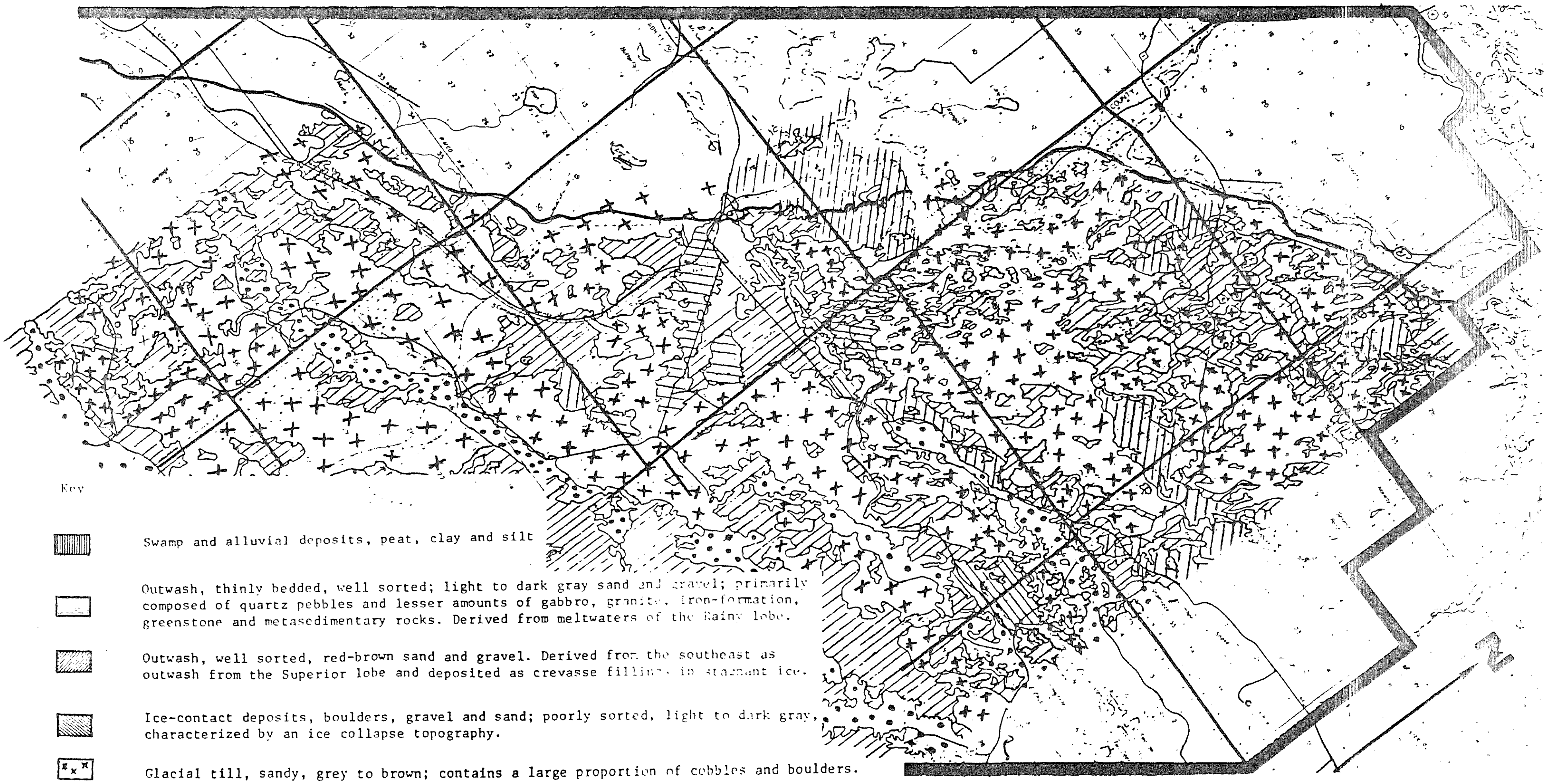
HOYT LAKES-KAWISHIWI AREA

SCALE IN STATUTE MILES

[illegible], MINN JULY 1, 1975

Map A8

Reconnaissance Surficial Geology



Key

▥ Swamp and alluvial deposits, peat, clay and silt

☐ Outwash, thinly bedded, well sorted; light to dark gray sand and gravel; primarily composed of quartz pebbles and lesser amounts of gabbro, granite, iron-formation, greenstone and metasedimentary rocks. Derived from meltwaters of the Rainy lobe.

▨ Outwash, well sorted, red-brown sand and gravel. Derived from the southeast as outwash from the Superior lobe and deposited as crevasse fillings in stagnant ice.

▩ Ice-contact deposits, boulders, gravel and sand; poorly sorted, light to dark gray, characterized by an ice collapse topography.

⊠ Glacial till, sandy, grey to brown; contains a large proportion of cobbles and boulders.

⁙ End moraine, esker or crevasse filling.

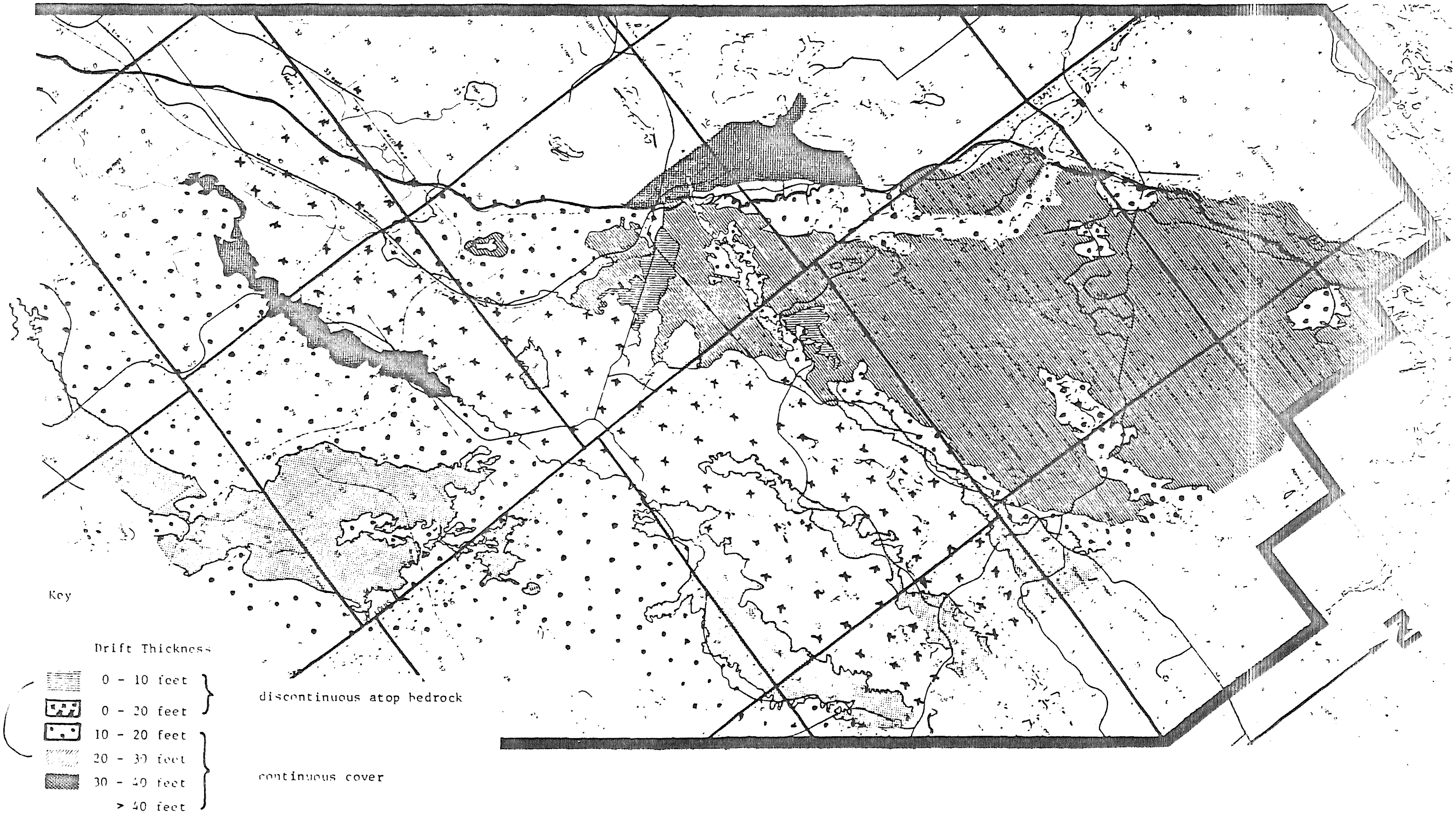
Map A9

HOYT LAKES-KAWISHIWI AREA

[illegible]

[illegible] MINN JULY 1, 1975

Drift Thickness



Key

Drift Thickness

▨	0 - 10 feet	discontinuous atop bedrock
▨	0 - 20 feet	
▨	10 - 20 feet	continuous cover
▨	20 - 30 feet	
▨	30 - 40 feet	
	> 40 feet	

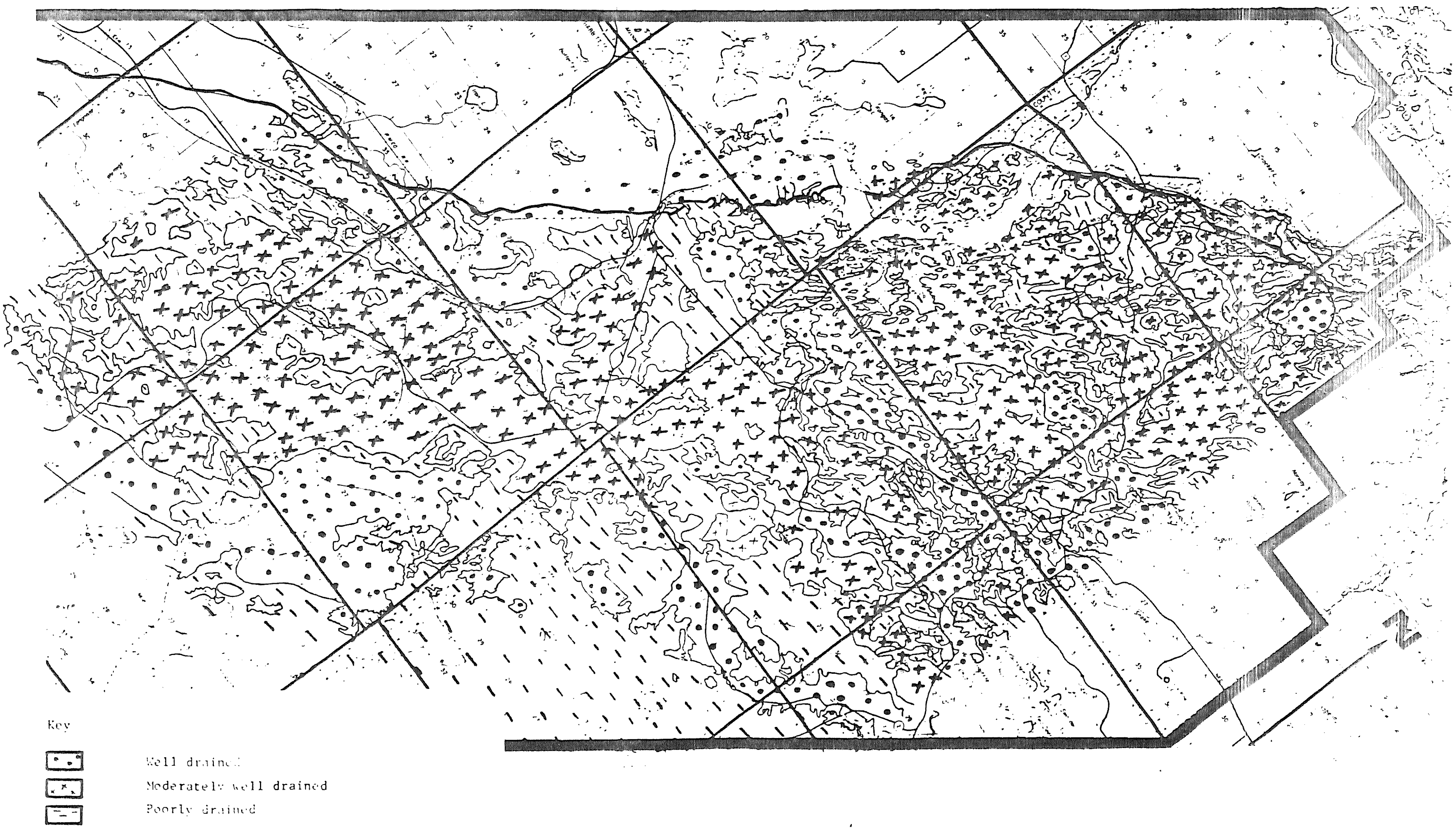
?

HOYT LAKES-KAWISHIWI AREA

SCALE OF STATUTE MILES

[illegible], MINN JULY 1, 1975

Drainage



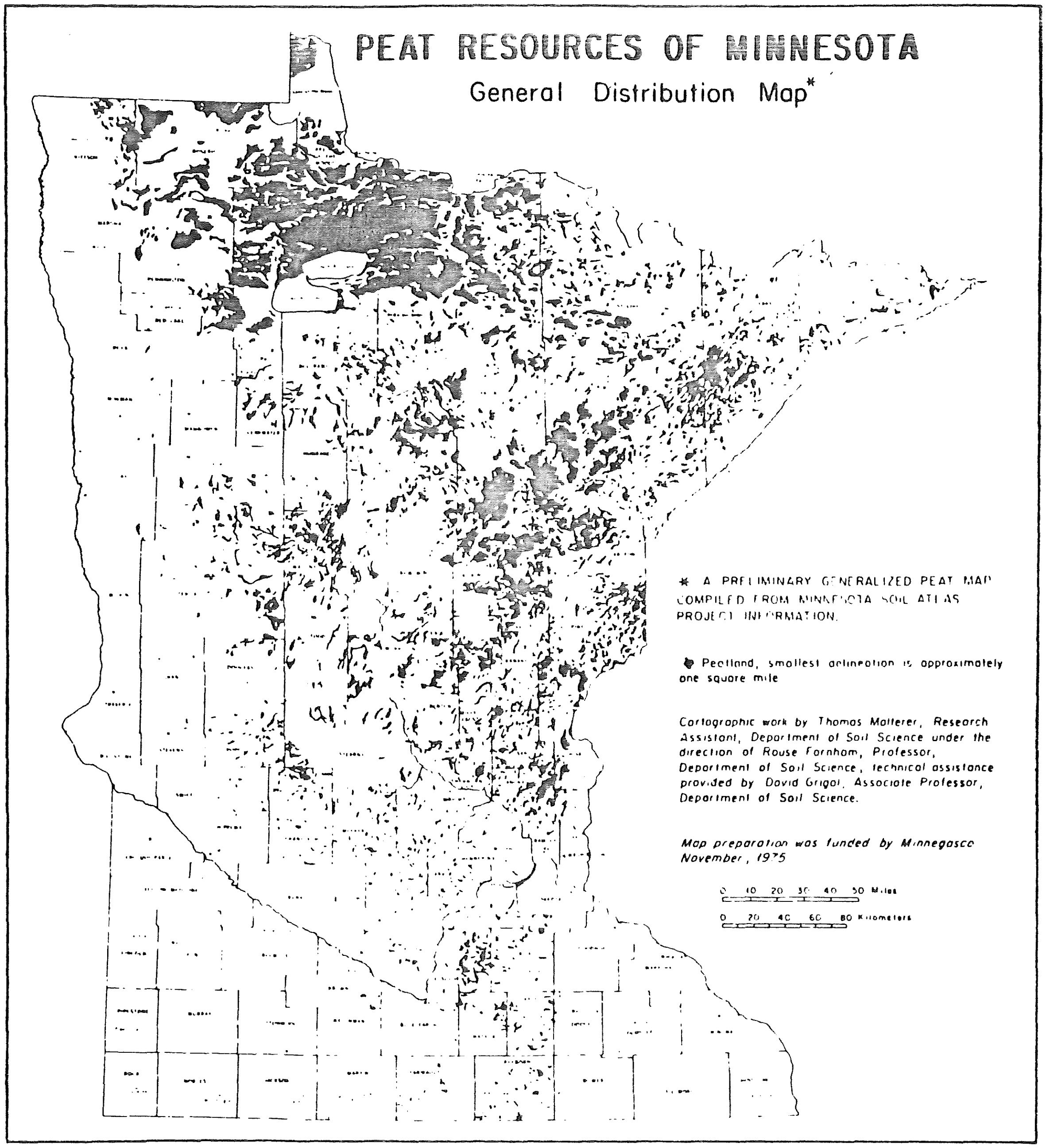
Key

- ● Well drained
- × Moderately well drained
- ⁻ Poorly drained

Table A4. Late glacial and postglacial vegetational and climatic history in northern Minnesota

Years ago	Climate	Vegetation
11,500	warming and drying trend	Northward spread of boreal spruce forest from southern Minnesota; final glacier has left the state by this time.
10,000		Forest successions leading to a pine forest principally.
8,000		Prairie vegetation in central Minnesota.
7,000	--------	Prairie-forest border reaches maximum limit, about 75 miles northeast of present position.
	cooler and wetter trend	Gradual invasion of prairie by forest and the advance of conifer into deciduous forest.
		Poorly draining areas expand into "blanket bogs" where peat accumulates (perenially wet conditions inhibit decomposition of plant debris).
0	--------	Continuing lake sedimentation from soil erosion and the conversion of lakes to bogs (mats of sedge, heath plants, and conifers spread towards centers of shallow lakes).

The geometry of sand and gravel deposits, resulting from Pleistocene glacial activity, varies according to the depositional environment in which the body was formed. Many of the smaller deposits formed as outwash fans, kames, valley trains, and eskers. Major deposits were laid down along channels of glacial rivers. In many places, post-glacial rivers reworked the primary sand and gravel deposits, forming gravel-rich deposits. Because of the high bulk-low unit value of sand and gravel, their utilization depends more on economic and technologic factors than distribution.

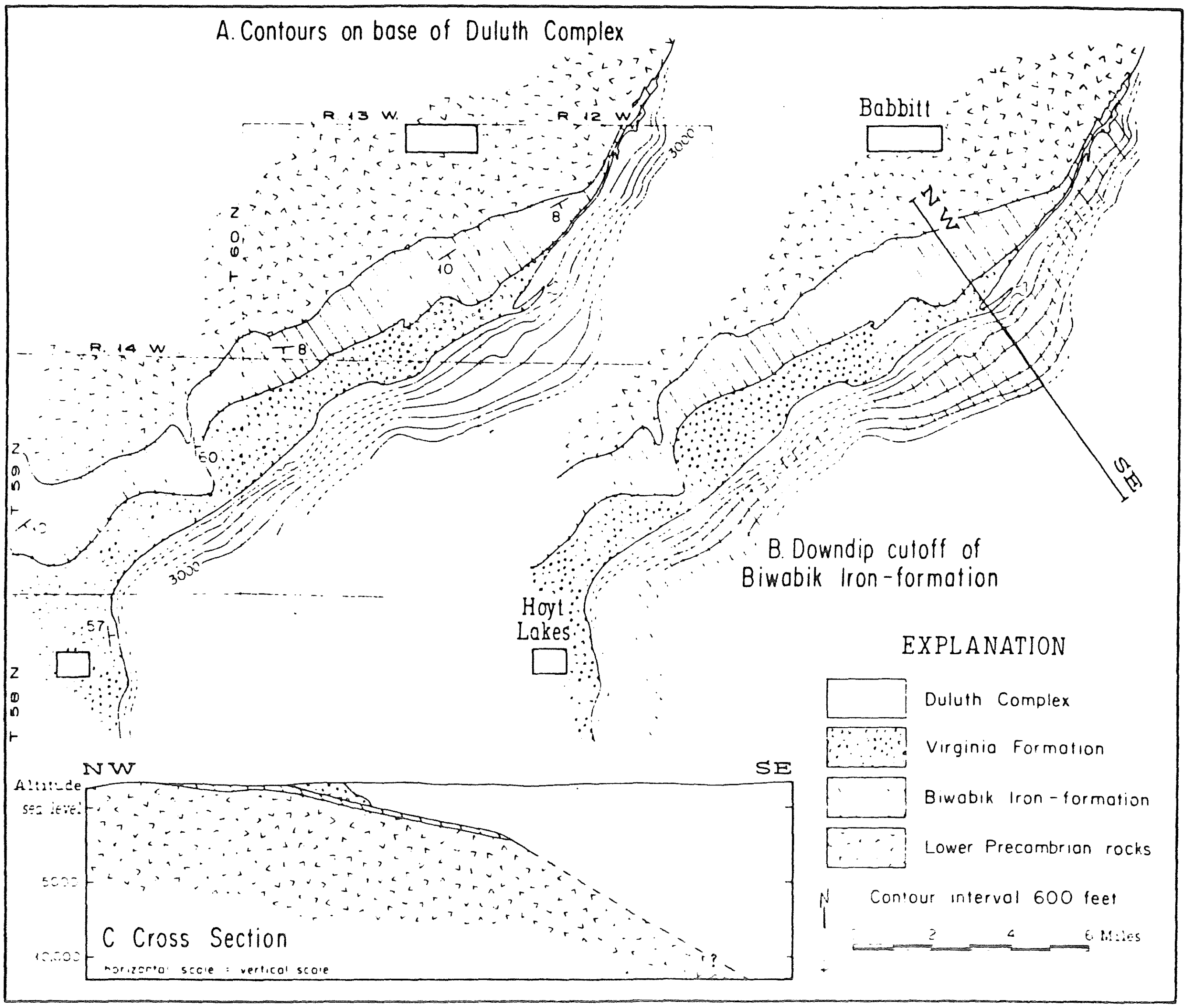


Map A10 The distribution of peatlands in Minnesota.

Since sand and gravel will most likely be consumed near the place where it is mined, local problems may result in urban areas unless planning is adequate.

STRUCTURE

Researchers have compiled drill core data to define the structural attitude of the basal contact of the Duluth Complex (Map A11). Preliminary work of this type and of others indicates a moderate, though locally variable, dip of the basal contact: 10-60° (generally 25°),[A3] 20-45°,[A20] 20-30°,[A19] and "almost vertical to flat", all in a southeasterly direction.



Map A11 Structural contours and cross-section of Babbitt-Hoyt Lakes region.

In the northern part of the study area, the Duluth Complex rests on the Giant's Range granite. The upper contact of the granite is estimated to be about 20-45° SE. In the southern part, the Duluth Complex lies over a wedge of metamorphosed Virginia and Biwabik Fm. that pinch out down dip. Drill core data supports this "pinching out" at depth and it has been suggested that structural activity at the time of emplacement removed the Biwabik Fm. at dept. The Biwabik Fm. dips 5-15° SE throughout the study area except in the Dunka River area where it dips at 15-35° SE at the surface and 5-15° SE at depth. Bonnichsen believes this indicates moderate deformation and rotation due to forceful emplacement locally.

Lineaments studied from high level aerial photographs (1:24,000)(Map A12), have been studied in the area and represent stream courses or topographic alignment due to faulting or contacts between rocks of different lithologies.[A11] Data collected on joint frequency and orientation tends to confirm the idea that lineaments are a surficial expression of faulting. In fact, faults probably control the distribution of rock units of the Duluth Complex, spatially and stratigraphically. Studies indicate that joint frequency is directly related to lineaments. Besides faults and lineaments trending northeast to north-northeast in the Babbit area, there are extensive north-northeast trending lineaments that are subparallel to known lower Precambrian faults suggesting a reactivation of old faults during the emplacement of the Duluth Complex.[A35] In the lower 2000-3000 ft. of the Complex where it lies directly over the Giant's Range Granite, numerous near vertical, late-stage granitic and syenitic dikes exist.[A9]

Generally, the attitude of the basal contact was probably determined by the pre-existing structure of older, country rocks. The complex was

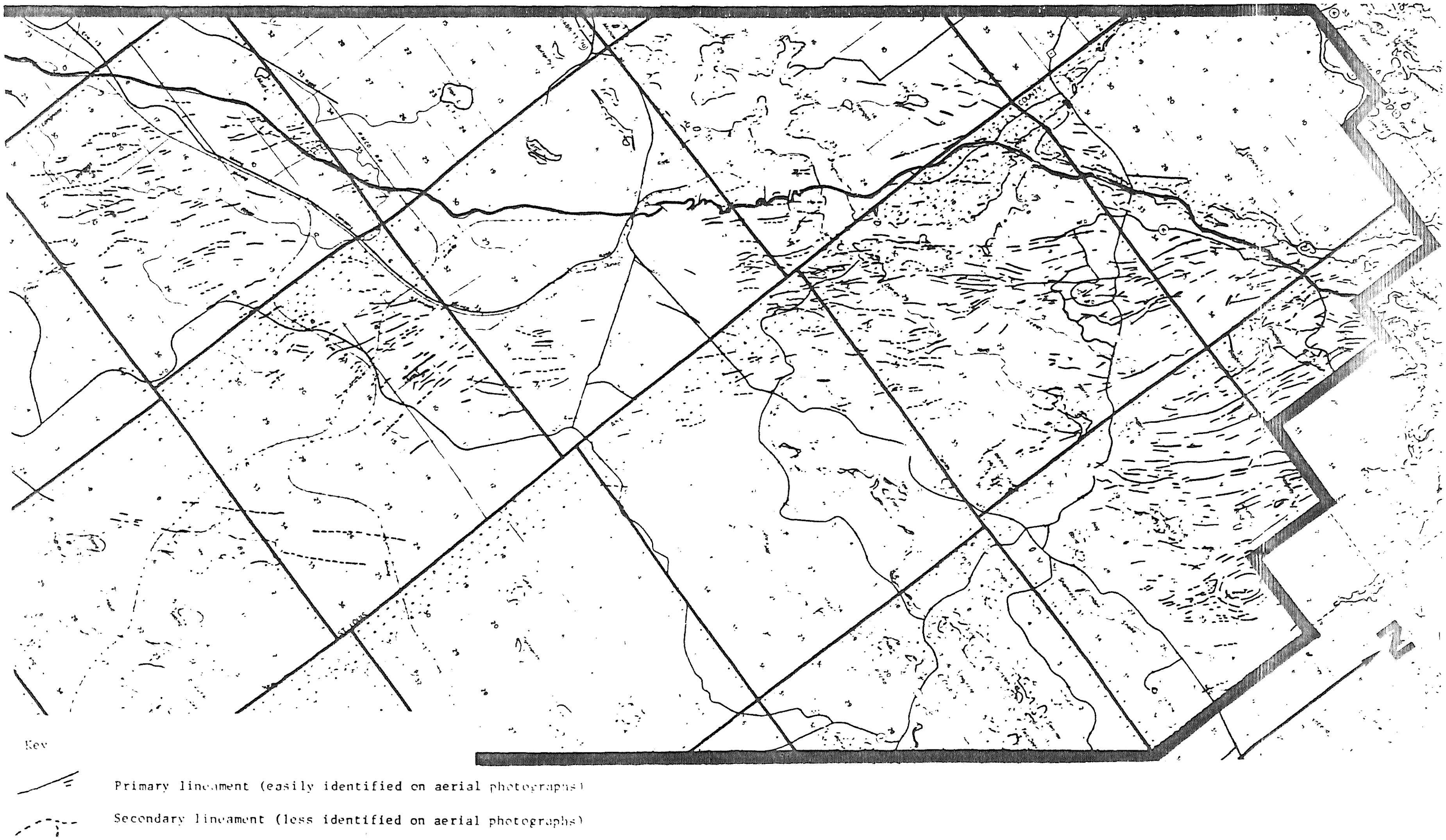
Map A12

HOYT LAKES-KAWISHIWI AREA

SCALE OF STATUTE MILES

HIBBING, MINN JULY 1, 1975

Lineaments



Key

Primary lineament (easily identified on aerial photographs)

Secondary lineament (less identified on aerial photographs) emplaced as sheet-like intrusions from the southeast, cutting into the Virginia Fm. and riding up on the Virginia-Biwabik contact, stoping, plucking and assimilating on the way.

MINERALIZATION

Economic Mineralization

Sulfide mineralization occurs both in the Giant's Range Granite and within the Duluth Complex. Wherever the Duluth Complex is in direct contact with the Giant's Range Granite, a several hundred foot thick metamorphic zureole is evident. Generally, the metamorphism recrystallized feldspars to aggregates of smaller grains and converted hornblende and biotite to hypersthene and augite. Significant mineralization (>1% Cu)~~in places)~~ exists in places, as disseminated chalcopyrite, bornite, sphalerite and millerite. These deposits are considered to be epigenetic in origin resulting from residual liquids from fractional crystallization of the Duluth Complex. Cu;Ni ratios are typically 5:1 in contrast to ~~Cu:Ni ratios of~~ 3:1 in the disseminated basal Duluth Complex deposits.[A9]

? The second area of mineralization is considered to be mostly due to syngenetic magmatic segregation and is confined to the Troctolitic Series, specifically at and/or near the contact of the Troctolitic Series and middle or early Precambrian rocks.

Available data indicate potentially economic Cu-Ni mineralization is confined to the Troctolitic Series, particularly at and near the basal contact. Locally, mineralized zones extend for a "few hundred feet". Numerous inclusions, with dimensions ranging from inches to thousands of feet, exist in the Troctolitic Series. Generally, these inclusions of the Anorthositic Series, Biwabik and Virginia Fm. are barren. Cu-Ni sulfides are either disseminated (discontinuous lenses or layers) or as scattered bodies of massive sulfides at or near the base, generally only inches across. Individual disseminated deposits vary in shape, from lenses parallel to the base, to lenses or very elongate bodies inclined to the base, to irregular lenses. From available information, it appears that the lateral boundaries of the Cu-Ni concentrations are gradational, except where these abut against large, non-mineralized inclusions. However, the upper limits of some deposits are abrupt and generally coincide with the contacts between intrusive units.

Controls for the Cu-Ni concentrations are poorly known. Basin-like depressions on plunging synformal structures at the base of the Gabbro appear to have localized some Cu-Ni concentrations. Juxtaposition of sulfides with the various lithologic units in the footwall and abundant inclusions of Virginia hornfels may be considered as possible ore controls. In the Dunka River area and at other locations, economic copper and nickel sulfide concentrations also occur locally in older granitic rocks to depths of a few hundred feet beneath the Duluth Complex.

Specifically, Cu-Ni sulfides have been found as (1) scattered bodies of massive sulfides at or near the base of the troctolite, (2) myrmeketic intergrowths with late-stage pyroxene and plagioclase, (3) interstitial grains, associated with or surrounded by clinopyroxene, magnetite-ilmenite or plagioclase, and as (4) fine grained inclusions in biotite, amphiboles and in the borders of plagioclase and clinopyroxene grains. (Figure A4) The last two modes of occurrence are the most important. An estimated 70% of sulfides occur as interstitial grains and nearly 30% occur as inclusions. Occurrences of sulfides are confined to a sulfide zone 1-2 miles from the contact on the surface. This basal zone is characterized by inclusions and a heteorgeneous

MN
only or
world-
wide?

texture and grain size. Local subtle to well defined discontinuous layering exists in this zone as well. The local variability in texture, grain size and modal composition is due to inclusions of a fine grained gray sandy textured hornfels (.5-1mm diam. grain size), well-banded Biwabik Formation and basic group (possibly recrystallized olivine-rich gabbro and/or troctolite). Generally, the grain size of the sulfides varies directly with the grain size of the silicates.

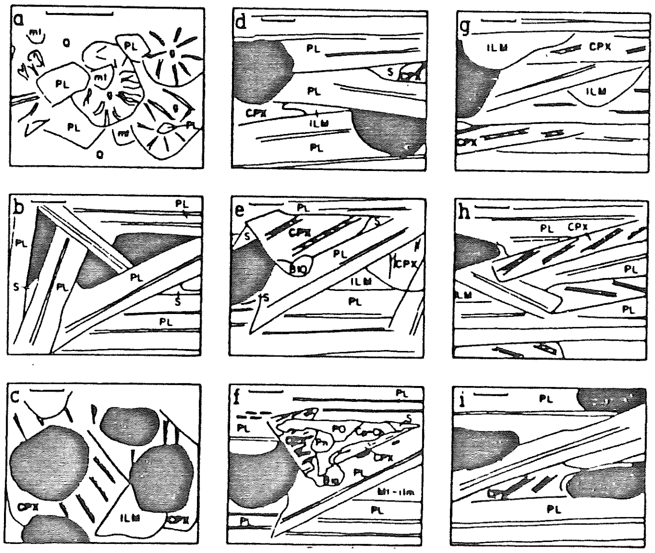


Figure A4 **Diagrammatic sketches of typical textures in rocks of the Duluth Complex.**

The bar scale in each diagram is approximately 2 mm in length.

Mineral Formulas

Below are listed most of the minerals occurring in the rocks of the Duluth complex and the contact zone in the study area.

Major minerals

Plagioclase Series (pl): $(Ca,Na)(Al,Si)AlSi_2O_8$: a continuous solid-solution feldspar series with end members albite ($NaAlSi_3O_8$) to anorthite ($CaAlSi_3O_8$). Series members in order of increasing anorthite percent are: albite (An_0-An_{10}); oligoclase (An_{10}-AN_{30}); andesine (An_{30}-An_{50}); labrodorite (An_{50}-An_{70}); bytownite (An_{70}-An_{90}); anorthite (An_{90}-An_{100}); this constitutes the major portion of rocks in the troctolitic series.

Olivine: $[(Mg,Fe)_2SiO_4]$: continuous solid-solution series with forsterite (Mg_2SiO_4) and fayalite (Fe_2SiO_4) as end members. Intermediate varieties are called olivine; alters to serpentine or hematite.

Pyroxene group: a closely related group of mafic minerals commonly divided into two groups based on crystal structure; occurs interstitially to plagioclase and olivine.

Orthopyroxenes (opx): (orthorhombic crystal structure) enstatite ($MfSiO_3$) to hypersthene $[(Mg,Fe)SiO_3]$ forming a continuous series not extending to pure Ferrosilite $[FeSiO_3]$.

Clinopyroxenes (cpx): (monoclinic crystal structure) the main clinopyroxenes are clino-enstatite ($MgSiO_3$), clino-hypersthene $[(Mg,Fe)SiO_3]$, diopside ($CaMgSi_2O_6$), hedenbergite ($CaFeSi_2O_6$), augite (intermediate between diopside and hedenbergite), and pigeonite (intermediate between clino-enstatite and augite). The most common pyroxenes in the Duluth complex rocks are augite-pigeonite and slightly less hypersthene.

Major sulfides (in order of abundance):

Pyrrhotite: $Fe_{n-1}Fn$, n = 0-.2, gangue (non-ore) mineral

Chalcopyrite: $CuFeS_2$, copper ore mineral

Cubanite: $CuFe_2S_3$ copper ore mineral

Pentlandite: $FeNi_9S_8$ nickel ore mineral

Other accessory and minor minerals are listed on the next page.

apatite: $Ca_5(PO_4)_3$ (F,Cl,OH)

biotite: $K(Mg,Fe)_3(AlSi_3O_{10})(OH)_2$, common accessory mineral associated with ores

bornite(bo): Cu_5FeS_4, exsolution with cp as small grains along fractures in pl and as lamellae in cp, may alter to cp, covellite or malachite

bravoite: $(Fe,Ni)S_2$

? chalcopyrite: Cu_2S

chlorite series: $(Mg,Fe,Al)_6(Al,Si)_4O_{10}(OH)_8$, alteration product of biotite and other ferromagnesium minerals.

covellite: CuS alteration product of primary sulfides

u
cyprite: Cu_2O

cummingtonite-grunerite: $(Mg,Fe)_7(Si_8O_{22})(OH)_2$

digenite: $Cu_{18}S$ also $Cu_{2-x}S$

graphite: C

hematite: Fe_2O_3, typically gossan cap mineral

haycockite: $Cu_4Fe_5S_8$

hornblende: $(Ca,Na)_3(Mg,Fe^{+2},Al,Ti)_5(Si,Al)_8O_{22}$ $(OH,F)_2$, an alteration product of clinopyroxene; locally, associated with ore

ilmenite: $FeTiO_3$, se magnetite

mackinawite: FeS

magnetite: Fe_3O_4 both magnetite and ilmenite and intergrowths of the two are associated with main silicates in mineralized zones.

malachite: $Cu_2(CO_3)(OH)$

millerite: NiS

mooihoekite: $Cu_9Fe_9S_{16}$

native copper: Cu

phase "X": $Cu_3Fe_4S_6$

pyrite(py): $(Fe,Ni,Co)S_2$, locally important; upon oxidation breaks down to limonite

serpentine: $Mg_6Si_4O_{10}$ $(OH)_8$, an alteration produce of olivine

? sphalerite(sph): An(Co)S , inclusions in cp/cb intergrowths; also (Zn,Fe)S spinel group: $(-,Mg()-,Ti,Al)_2O_4$, (-) appropriate cations

talc: $Mg_3(Si_4O_{10})(OH)_2$

talnakhite: Cu_{10} $(Fe,Ni)_{18}S_{32}$

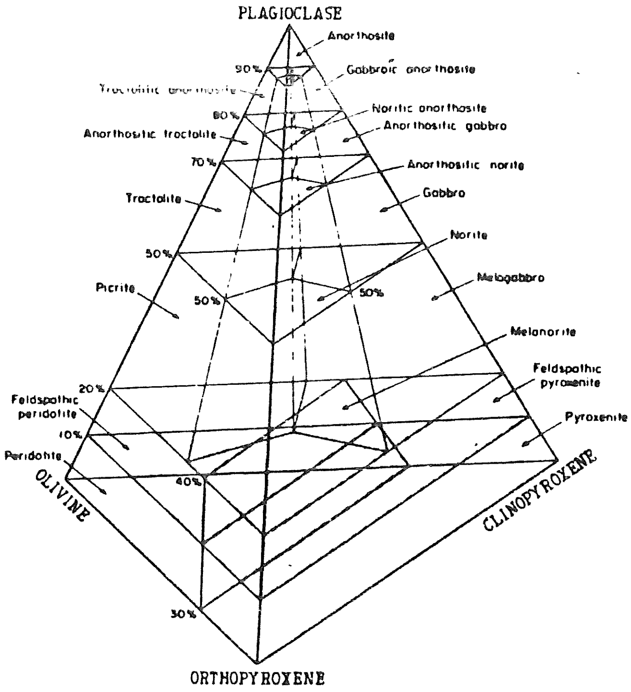
tenorite (melaconite): CuO

triolite(tr): FeS, commonly in exsolution with po, nonmagnetic

valleriite: CuFeSi: $Mg,Al,Fe(OH)_2$

violarite: $(Ni,Fe)_3S_4$, common alteration product of pn.

other secondary nickel sulfides



Classification schemes used to name various rock types in the Duluth Complex.

In all classification schemes, the root name, such as troctolite, is determined by the relative abundances of the essential minerals named at the corners. Compound names, such as augite-troctolite indicate the presence of non-essential minerals in relatively appreciable quantities (generally greater than 10%). Textural modifiers such as "poikilitic" also have been used as modifiers of the root name.

Table A5. Average modal composition of gabbroic rocks; in vol.% (**5-10% Hydrou Silicates)

	1.		2.		3.		4.		5.	6.	7.		8.	9.
	ave.	range	ave.	range	ave.	range	ave.	range	range	range	range	range	ave.	ave.
plagioclase	60	40-79	60.2	20.8-82.7	58.8	6.1-95.1	76.4	65.6-88.0	55-80	65-75	60-75	75-95	86	50-70
olivine	14	0.34	24.9	0.0-76.3	21.1	0-53.9	7.7	tr-19.3	12-30	5-10	10-25	4-8	---	10-20
hypersthene	6	0-45	1.1	0.0-15.9			4.9	0.6-23.9		-----	0-3	0-1	---	
(OPX)									2-14					15-30
augite	7	0.40	5.4	0.0-27.9	6.0	0-47.7	5.5	2.6-9.7		10-20	2-4	0-2	---	
(CPX)														
biotite	2	0-8	2.1	0.0-5.9	.9	0-5.2	1.5	.4-3.5	1-4	1-5	0-2	0-2	---	minor
serpentine	3	tr-20	(1)	tr-45.8	1.5	0-16.6			0-5	**	-----	---	---	-----
chlorite	3	0-5							0-5		-----	---	---	-----
magnetite	2	0-5								1-5			---	
ilmenite			3.5	0.0-18.4	4.1	0-50-7	2.3	1.3-3.9	6-17		0-3	0-1	---	5-10
sulfides	2	0-8								5-15			---	5-10
other	3	0-12	1.4	0.0-34.2	4.8		1.1	0-4.3	0-5	**	-----	---	---	-----

1. Anderson. (average of 18 samples-nos. 2-19 from Anderson-Table 2. Samples ranged from mineralized to normal to altered gabbros in rock type).
2. Hardyman. (average of 49 samples from Table 1, app. 1 in Hardyman's thesis. Rock types range from troctolite to olivine-gabbro to picrite.)
3. Boucher. (average of 72 samples from U.S.B.M.-H-IA, excludes hornfels, includes anorthosite, troctolite and olivine gabbro to picrite.)
4. Phinney. (average based on Phinney's average data from 318 thin sections of anorthositic gabbro, troctolite and norite from the South Kawishiwi and contact zones.)
5. Vifian, Iwasaki and Bleifuss. 1967. (ranges given by the authors-p. 2 for a bulk sample-ranging from olivine gabbro to troctolite.)

"other" may include considerable sericite, appatite, amphibole (hydrous equivalents of the pyroxenes), symplecite, talc $Mg_3Si_4O_{10}(OH)_2$, etc.

6. Lawver. 1975. (grab samples from Spruce Road Test Pit)
7. Hays. 1974. (unidentified source(s)) Troctolitic series. Anorthositic series.
8. Bonnichsen. (according to Phinney, 1969)
9. Weiblen and Hall. (Mineralized gabbro samples from Inco's Spruce Rd. Test Pit)

Table A6. Average plagioclase and olivine compositions.

	1c.	2c.	24	33
plagioclase-mole % in An- $CaAlSi_3O_8$	57	51	50-65	35.3-70.3
olivine-mole % Fo- Mg_2SiO_4	53	39	50	45.2-51.5

1. Hardyman, 1969. (plagioclase ave.-24 samples; olivine-35 samples)
2. Renner, 1969. (plagioclase ave.-24 samples; olivine ave.-5 samples)
3. Weiblen & Morey, 1976. (weight %(?) Inco's Spruce Road Test Pit #1, 3 samples)

Table A7. Average pyroxene compositions.

	1d.		2d.		24		33	
	CPX	OPX	CPX	OPX	CPX	OPX	CPX	OPX
mole % enstatite $MgSiO_3$(Eu)	40	62	38	51	35	61	37.5	57.7
mole % ferrosilite $FeSiO_3$(Fs)	18	36	19	47	20	36	18.8	39.7
mole % wollastonite $CaSiO_3$(Wo)	45	1	43	2	45	3	43.7	2.6

1d. Hardyman. (35 CPX (clinopyroxene) samples-29 OPX samples.)
2d. Renner. (15 CPX samples-22 OPX samples.)

Table A8. Average modal composition of hornfels and contact rocks.

	1a.		2a.		3a.	
	ave.	range	ave.	range	ave.	range
plagioclase	54.4	45.7-65.8	48.8	10.3-75.8	44.6	40.6-48.7
olivine			8.0	0-17.0		
OPX (hyp.)	29.6	2.0-44.9	12.8	0-40.6	10.8	6.3-15.4
CPX (augite)	3.3+	.4-13.2	17.4	0-65.4		
biotite	1.9	.1- 7.4	.7	0- 2.2	17.6	13.6-20.6
opaques (ilm)	4.5	2.0-10.6	3.3	0- 8.5	13.9	4.3-23.5
(sulf)	.9+	.1- 9.6				
other	3.0	0-11.3	3.3	0-16.4	13.5	

1a. Renner, 1969. (ave. of 16 samples from "D-series" rocks from the Duluth complex-Virginia Fm. contact zone.)
2a. Boucher, 1975. (ave. of 9 samples from U.S.B.M.-DDH-IA. hornfels)
3a. Hardyman, 1969. (these averages are based on only 2 samples of hornfels and are thus rather poor as averages)

Table A9. Opaque compositional distribution (vol. %)

	1b.	2b.*	3b.	4b.			33	33			
				Cu	Fe	S		Cu	Fe	S	Ni
chalcopyrite	31.5	44.8	22.7	34	33	35	40.6	14.8	47.3	36.8	1.1
cubanite	22.5	13.4	11.2	22	43	35	0.7				
pyrrhotite	22.4	37.1	24.3	--	61	39	55.4				
pyrite	2.3										
violarite	3.5										
pentlandite	4.9	5.7	3.4				3.3				
oxides	12.9	*	37.6								
other		tr	.8	(42	27	31)-bornite					
ave. vol.% opaque in samples	---	10.6*	8.9								

1b. Anderson, 1956. (ave. vol.% for opaques from 10 samples)
2b. Hardyman, 1969. (ave. vol.% for sulfides from 15 thin sections)
3b. Boucher, 1975. (ave. vol.% for opaques from 16 thin sections)
4b. Hardyman, 1969. (elemental compositions for chalco., cuban., pyrr. and bornite averaged from Hardyman's microprobe anal. on 7,5,4 and 2 grains of the respective sulfides.)

* these averages are based on total sulfides, not total opaques.

33 Modal Abundance Data from Inco Test Pit #1. Interstitial Sulfides
Column 9. 24 random modal analyses

Table A10. Average Weight % Electron Microprobe Analyses of Sulfide Minerals.

	po'	po"	pn	cp	cb	mackinawite
S	36.67	39.41	33.37	32.90	36.94	36.06
Fe	62.65	60.02	30.55	31.29	39.46	54.95
Co	0.28	0.03	1.84	0.08	-----	0.96
Ni	.014	0.24	34.67	0.06	0.06	7.68
Cu	-----	0.01	-----	35.76	21.97	-----

Note: Inco Test Pit #1. Each analysis is ave. of 3-5 point analyses. po' and po" are intergrown po, po' is monoclinic and po" is hexagonal crystal form.

Table A11. Theoretical Composition of Major Sulphide Minerals - % by Weight.

	po	pn	cp	cb
S	36.5	20	35.0	35.4
Fe	63.5	38	30.6	41.2
Co	----	—	----	----
Ni	----	40	----	----
Cu	----	--	34.6	23.4

ENGINEERING DATA

Weathering: an estimated 6-12" deep gossan exists where the Duluth Complex is exposed with a friable 2-3" upper layer.

Stratigraphy & Permeability: from available core information and literature, the following rock types can be expected:

(1) mineralized Troctolitic Series rock (this includes ore and lean ore)

(2) non-mineralized Troctolitic Series rock

(3) Anorthositic Series Rock

(4) Virginia, Biwabik, Giant's Range Granite

(5) and inclusions of (3) and (4) that could be partially assimilated and/or metamorphoses depending on dimensions.

Overburden would consist primarily of:

(1) Pleistocene glacial deposits - largely sand and gravel, although numerous boulders exist, from 0 to 85' deep but generally less than 30'.

(2) Recent deposits of peat, lake and stream sediments

Since the thin glacial drift acts as the aquifer for the area, erosional glacial features such as buried river channels should be anticipated as well as the permeable outwash deposits and ice contact features (such as eskers, kames, and features made by stagnant ice blocks).

Rock permeability is very low. The USGS Water Resources Department lists the Duluth Complex as having a low water yield capacity with massive areas of imperviousness and adds that below 300-400' the complex is reportedly less fractured than above this depth. Farm wells in the complex yield "little water" on the order of 1 gpm. Reportedly, in the 400' drift of INCO's shaft, seepage was approximately 16 gpm.

In-situ Rock Stress: measured lateral in-situ stresses of the Duluth Gabbro generally are higher than vertical pressures. This stress distribution is related to the intrusion and crystallization of the magma at depth.

Rock Defects: drill core data indicates no major alteration zones except surficially, 13-60 ft. zones of fracturing, breakage, serpentinization and occasional slickensides and shearing. Zones of serpentinization conceivably could act as surfaces for future fault and water movement.

Table A12. Density Measurements of Duluth Complex Rocks.

Troctolitic Series: (SPT, AGU, AGO, ST)	mean = 2.85 gm/cc range = 2.72-3.13 gm/cc standard deviation = 0.077 gm/cc
Flow-Structure Troctolite: (ST)	mean = 3.01 gm/cc range = 2.92-3.10 gm/cc standard deviation = 0.126 gm/cc
Anorthositic Series: (AGU)	mean = 2.85 gm/cc range = 2.72-3.13 gm/cc standard deviation = 0.084 gm/cc

Note: ( ) indicate rock types included in measurements according to Phinney's classification (Phinney, 1972). For comparism, the density of plagioclase series ranges from 2.62-2.76 gm/cc.

Table A13 - Physical property results

	(a) Maximum differential stress, MN/m² (=145 psi)				(b) Young's modulus, GN/m² (=145x10³ psi)				(c) Hardness and Density	
test Zone type	Brazilian	Uniaxial compression	Triaxial compression 17.24 MN/m²	Triaxial compression 39.48 MN/m²	Uniaxial compression	Triaxial compression 17.24 MN/m²	Triaxial compression 34.48 MN/m²	Dynamic	Shore hardness	Density gm/cm³
Mineralized a. troctolite	14.5 (2.0) n = 11	214.0 (11.3) n = 5	318.4 (40.0) n = 3	362.6 (37.9) n = 2	48.3 (2.4) n = 5	39.8 (1.3) n = 3	79.5 (1.4) n = 2	93.0 n = 1	60.0 (4.2) n = 9	3.03 (.04) n = 11
Nonmineralized	15.7 (3.4) n = 6	179.7 (44.2) n = 4	347.8 (9.9) n = 3	387.5 (57.4) n = 3	46.4 (5.5) n = 4	38.9 (.4) n = 3	69.0 (25.6) n = 3	82.4 (11.5) n = 3	62.1 (7.3) n = 6	2.99 (.08) n = 13
Hornfels	17.1 (8.0) n = 4	279.8 (34.8) n = 5	434.6 (291.3) n = 2	580.6 (90.5) n = 3	52.4 (5.2) n = 5	37.4 (7.8) n = 2	80.6 (4.5) n = 3	105.2 n = 1	54.7 (5.4) n = 9	3.09 (.11) n = 10
Semi-massive	---	152.9 n = 1	263.3 n = 1	356.7 n = 1	39.6 n = 1	34.2 n = 1	34.4 n = 1	77.1 (.1) n = 2	---	3.88 (.06) n = 2
Biwabik iron formation	20.6 (5.7) n = 4	304.8 n = 1	411.1 (374.3) n = 2	330.5 (188.7) n = 2	53.4 n = 1	33.4 (7.0) n = 2	35.4 (4.9) n = 2	---	52.4 (18.4) n = 4	2.85 (.17) n = 5
	Straight pull									
Troctolite	3.5 (.5) n = 3	221.9 (32.1) n = 4	---	---	54.1 (10.0) n = 4	---	---	69 n = ?	78.8 (5.7) n = 5	3.11

Note: Laboratory tests are made on intact rock specimens so it should be remembered that test data often represents only the stronger properties of the specimen. Allowances for these differences are made by an experienced engineer in the design stage.

Results are given in this order: average
(standard deviation)
n = number of samples

REFERENCES FOR ILLUSTRATIONS
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Section B

Mining and Beneficiation

Appendicies

Preface

It must be emphasized that this study and resulting report has had many drawbacks. No recommendations are either made or intended. No competent engineer would base any report or decision to a client for a multi-million dollar project on the sketchy and limited information that was available for this "preliminary report" during the short time period available for its preparation.

In addition, no land ownership information was available for this preliminary study hence the possibility of the land indicated for use as Minesite, Millsite, disposal areas, etc., might not be available for use as outlined in figures B2, B3 and B4. A Minesite package as shown could include a multiplicity of ownerships including several private landowners in addition to the State and Federal Government.

No proper engineering study can be made without a physical examination and assessment of the area involved. Time limitations did not permit an on sight inspection of any of the "Minesite" area. No exploration nor resource data was available for the selection of "hypothetical" Minesites No. 1 and No. 2 and "hypothetical" Minesites No. 3, 3A, and 4 were selected solely on the basis of the analyses of the drill cores of one drill hole for each site.

The conflict of interest in land use between the taconite operations and potential copper-nickel operations was neither studied nor investigated. The following are some of the items that should be studied and discussed to promote cooperation toward joint uses:

1. Land use for waste rock and tailings disposal areas west of the Duluth complex contact.
2. The feasibility of building a second railroad by expanding the present railroad bed of either Erie Mining Co. R.R. or Reserve Mining Co. R.R. to a dock (or a smelter) on the North Shore at Lake Superior.
3. Joint uses of the following:
    a. Railroads and truck haul roads

b. Power Lines and power line rights of way

c. Water resources

    1. Potable and non potable water supplies

    2. Water treatment and purification facilities

    3. Sewage treatment facilities

    4. Effluent disposal to the watershed

Other areas of common interest will become apparent as this study continues and all should be given careful consideration.

It is hoped that after a proper "on sight" examination and assessment of the "Minesite Area" and more information is made available to the "Copper-Nickel Study Group", a more meaningful analysis of the potential for Copper-Nickel mining and its impact on the environment can be made.

George Weaton, P.E.
August 6, 1976

Mineralogy-Duluth Gabbro Complex - Brief Description

Major	Minor
1. Pyrrhotite ($Fe_{1-x}S$)	1. Bornite (Cu_5FeS_4)
2. Chalcopyrite ($CuFeS_2$)	2. Sphalerite (ZnS)
3. Cubanite ($CuFe_2S_3$)	
4. Pentlandite ($FeNi_9S_8$)	

Cu-Ni Ratio: ± 3:1 in mineralized parts of basal intrusive unit. 1:1 to 8:1 in underlying granite.

Trace Elements of possible economic potential: Cobalt, Silver, palladium, gold, platinum, rhodium.

Typical ore at Contact: ± 1.0% Combined Cu-Ni.

Typical ore drops at depth to ± 0.5% Combined Cu-Ni

Average % Concentration: Copper=0.45% - 1%; Nickel=0.05% - 0.35%.

Troctolites are a possible source of alumina (Calcic Felspar). 53 bore hole samples by the U.S.G.S. averaged 11.8% Al_2O_3 (52.9% Al; 47.1% O) ore 6.24% Al.

Outline of Considerations
Mining, Beneficiation and Waste Disposal in the Minesite Area

A. Exploration

1. Geological, Geophysical, and Geochemical Exploration
2. Sampling by Core drilling, Shaft Sinking, tunnelling, and trenching
3. Chemical, physical, and metallurgical testing for mining and processing feasibility and design.

B. Development

1. Preparation of the ore body for mining, including initial shaft sinking and underground development for mining; or, pre-stripping for surface mining
2. Construction of all facilities for mining and concentration of the ore
3. Site preparation for areas to be used for stockpiling of waste materials, (surface stripping, lean ore, and/or waste rock)
4. Site preparation for area to be used for tailings disposal and the settling ponds and water recycling system
5. Construction of water appropriation facilities including possibly, a reservoir. Construction of settling ponds and water treatment facilities for effluents from the beneficiation plant, mine and sewage disposal system
6. Construction of transportation facilities such as roadways, conveyor lines, and railroads
7. Construction of auxiliary facilities, such as power plants, transmission lines, pipe lines, pumping stations, sewage disposal plant, off site equipment, supply and repair facilities, expansion of townsites, etc.

C. Mining

1. Extraction of waste rock after initial mine development
2. Construction of primary crusher underground
3. Disposal of waste rock and lean ore
4. Disposal of water from underground-possible use for mill make-up water
5. Transportation of ore from mine to secondary crushing facilities
6. Construction of repair, maintenance and warehousing facilities of mining equipment on site

7. Extraction of ore
8. Reclamation or backfilling of mined out areas
9. Reclamation of waste disposal areas on the surface as mining progresses

D. Concentrating

1. Secondary and tertiary crushing facilities
2. Concentration of the ore-flotation
3. Disposal of tailings
4. Water appropriation facilities for the concentrator
5. Water discharge and treatment for effluents from the concentrator and tailings facilities
6. Reclamation of tailings disposal areas as deposition progresses

Mining, Beneficiation and Waste Disposal in the Minesite Area

I. Preliminary Planning

In studying the plan map of the "Minesite Area", it was determined that for simplification, it was expedient to divide the area into four(4) segments as follows:

NE Area	207 square miles - That area north of the Laurentian Divide and east of the Gabbro Contact.
NW Area	67 square miles - That area north of the Laurentian Divide and west of the Gabbro Contact.
SW Area	47 square miles - That area south of the Laurentian Divide and west of the Gabbro Contact.
SE Area	229 square miles - That area south of the Laurentian Divide and east of the Gabbro Contact.

Total area of the Minesite is 550 square miles. **(Fig. 1)**

Each area has been reviewed as to the location of physical objectives that must be considered in the location of a mine site and particularly those objectives which must be avoided or cannot be included in any minesite or disposal area for a mine operation. Major rivers, lakes over 40 acres in size, railroad rights of way, highway right of way, cities, open pit taconite mine areas and a 1 MILE WIDE buffer zone between the Minesite Area (550 sq.mi.) and the Boundary Waters Canoe Area (BWCA) were all considered to be areas which must be excluded in planning for the location of either an open pit mine or an underground (shaft) mine.

Table I shows the areas which must be considered unavailable for use in a mining operation,

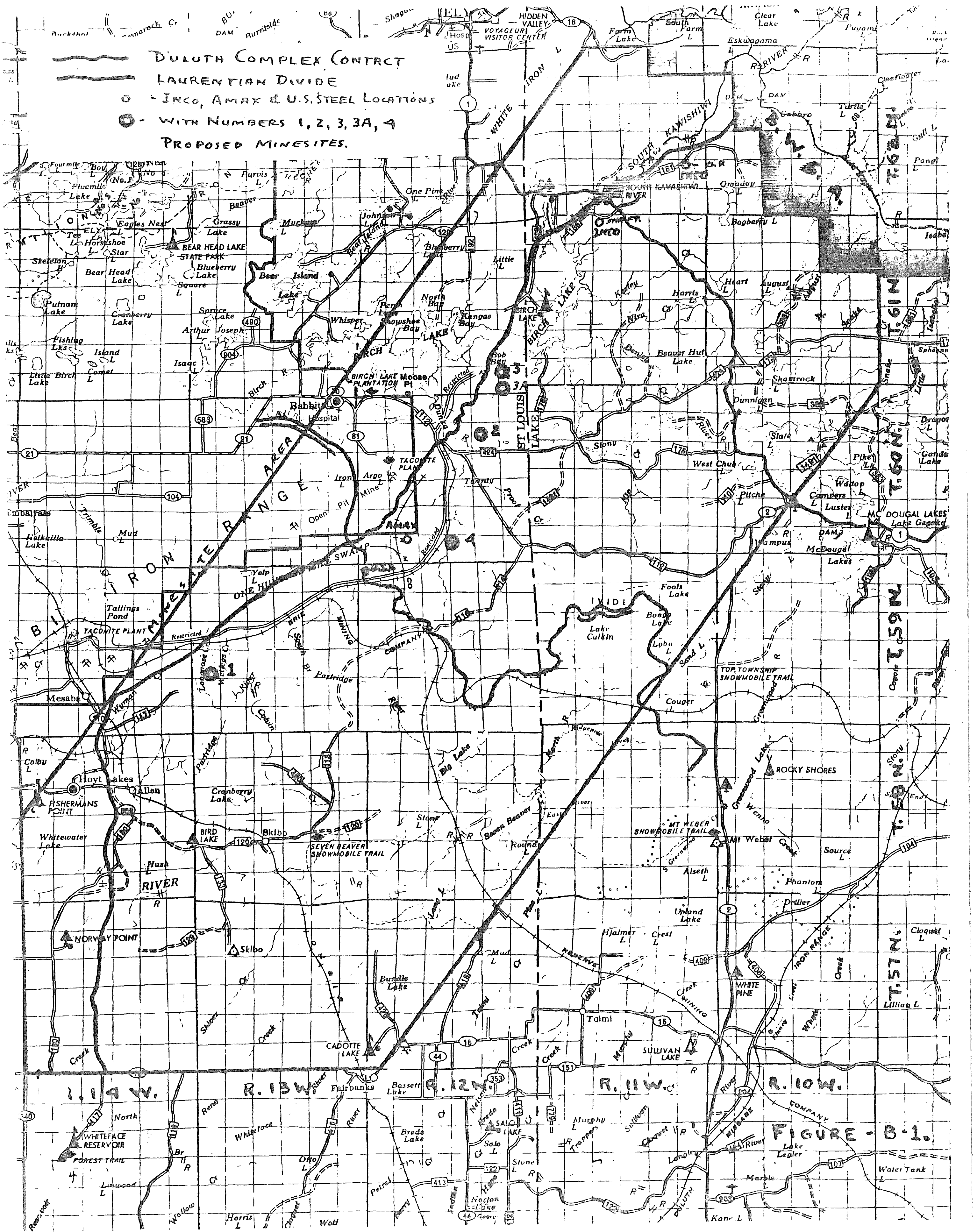

DULUTH COMPLEX CONTACT
LAURENTIAN DIVIDE
O - INCO, AMAX & U.S. STEEL LOCATIONS
O - WITH NUMBERS 1, 2, 3, 3A, 4
PROPOSED MINESITES.
1
2
3
3A
4
INCO
AMAX
T. 62 N.
T. 61 N.
T. 60 N.
T. 59 N.
T. 58 N.
T. 57 N.
R. 14 W.
R. 13 W.
R. 12 W.
R. 11 W.
R. 10 W.
ELY
HIDDEN VALLEY
VOYAGEUR VISITOR CENTER
Farm Lake
South Farm L
Clear Lake
Eskwagama L
SOUTH KAWISHIWI RIVER
Gabbro L
Omaday L
Bogberry L
Isabella
WHITE IRON L
BEAR HEAD LAKE STATE PARK
Bear Head Lake
Eagles Nest
Grassy Lake
Bear Island Lake
Johnson
Blueberry Lake
One Pine
Whisper L
Snowshoe Bay
North Bay
Kangas Bay
BIRCH LAKE
BIRCH LAKE PLANTATION
Babbitt
Hospital
Moose Pt
Heart L
August L
Harris L
Beaver Hut Lake
Shamrock
Dunnigan L
Slate L
Stony River
West Chub L
Pitcha L
Campers L
Luster L
MC DOUGAL LAKES
McDougal Lakes
Wampus L
ST LOUIS
LAKE
TACONITE PLANT
Argo L
Iron L
Open Pit Mine
Tommy L
ONE HUNDRED MILE SWAMP
Yelp L
Tailings Pond
Fools Lake
Bonga Lake
Lake Culkin
Lobo L
Sand L
Cougar L
TOFTE TOWNSHIP SNOWMOBILE TRAIL
ROCKY SHORES
Greenwood Lake
Mt Weber
MT WEBER SNOWMOBILE TRAIL
Wenho
Source L
Phantom L
Driller L
Upland Lake
Alseth L
Hjalmer L
Crest L
Cloquet L
Lillian L
IRON RANGE
WHITE PINE
Tolmi
SULLIVAN LAKE
MINING COMPANY
RESERVE
ERIE MINING COMPANY
Mesaba
Hoyt Lakes
Allen
Colby L
FISHERMANS POINT
Whitewater Lake
Cranberry Lake
BIRD LAKE
Sklbo
SEVEN BEAVER SNOWMOBILE TRAIL
Stone L
Round L
Seven Beaver
Big Lake
Long L
Pine L
Hush L
RIVER
NORWAY POINT
Burdle Lake
Mud L
CADOTTE LAKE
Fairbanks
Bassett Lake
SALO LAKE
Brede Lake
Murphy L
WHITEFACE RESERVOIR
FOREST TRAIL
Linwood L
Otto L
Stone L
Norton Lake
Harris L
Kane L
Lake Leppler
Water Tank L
Marble L
Langley
DULUTH MISSABE

FIGURE - B-1.

Minesite Areas Unavailable for Mining Use, Square Miles

Table I

Item	NE Area	NW Area	SW Area	SE Area	Total
Rivers	6.3	14.6	2.8	14.6	38.3
Lakes	13.4	12.4	1.8	11.8	39.4
Railroad R/W	1.5	0.6	4.3	8.9	15.3
MN Hwy#1 R/W	3.4	1.0	----	----	4.4
Cities	----	----	3.0	----	3.0
O.P. Tac. Mines	----	8.0	17.0	----	25.0
BWCA Buffer Zone	11.0	6.0	----	----	17.0
Sub Totals	35.6	42.6	28.9	35.3	142.4
Total Area	207.0	67.0	47.0	229.0	550.0
Area Available	171.4	24.4	18.1	193.7	407.6

The area west of the Gabbro Contact is not mineralized and hence not potentially a mineable area. However, some areas could be adapted for tailings and/or stripping and rock dump purposes. Any potential minesite would necessarily be located east of the Gabbro Contact and, due to the dip of the Contact (20 to 45), would necessarily be within a band ± 1½ miles wide. Beyond this point, the depth of a shaft to the contact would be excessive and costly.

Discounting the possible site locations for mining in the areas listed in Table I, we have: (1) In the NE Area 6.0 square miles of possible open pit mine site locations (room for 2 open pit mines), and 24.0 sq. mi. of possible underground mine site locations; and (2) another 38.0 sq. mi. of possible underground mine site locations in the SE Area. A total of 41 linear miles along the contact would possibly be potential underground mine locations and 6.0 miles additional for open pit possibilities, including the location chosen by Inco. Hence, the possible area now available for the disposal of waste material (surface stripping, waste rock, lean ore and tailings) is thereby reduced to 141.4 sq. mi. in the NE Area, and to 155.7 sq. mi. in the SE Area. The NW Area and SW Area would not be effected by possible mine locations. Therefore, of the 550 acres of Minesita Area, a maximum of 339.6 sq. mi. could possibly be available for the disposal of waste materials.

II. Proposal No. 1, Underground Mine. (Fig.82)

a.) Shaft Site: SW-SE of Sec. 19 T.59NR 13W

b.) Mill Site: W½-NE of Sec. 19 T.59NR 13W

c.) Shaft Depth: Approximately 1800 feet

d.) Accessability: State Highway #110 to 3 miles north of Hoyt Lakes. East on Forest Service Road 3.5 miles to end. Shaft site is 1 mile N.E. Railroads: D.M. & I.R. is located 4 miles SW of shaftsite

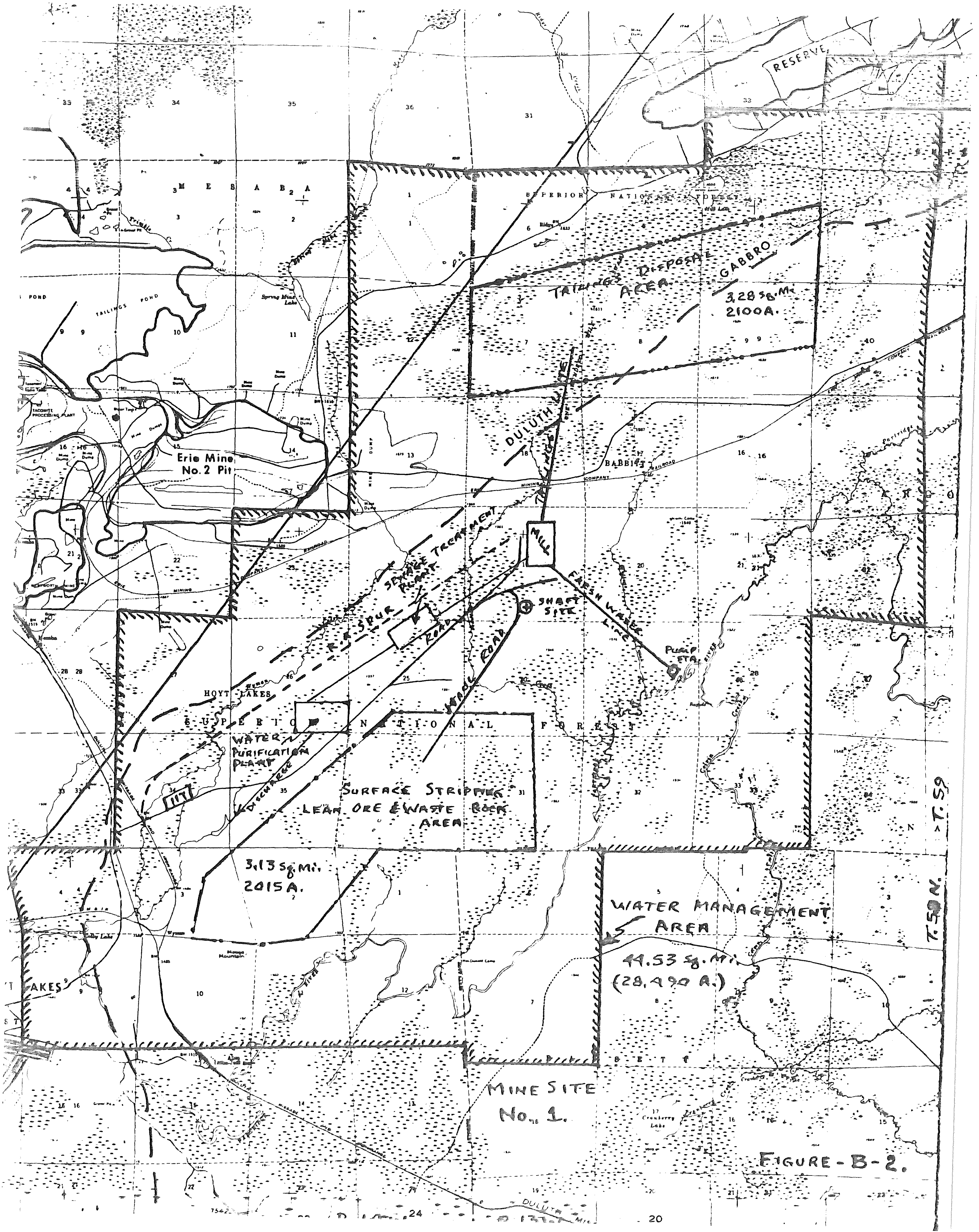
e.) Electric Power: Erie Mining Co./MP & L Electric Power Line is located 3.5 miles due west of proposed plant location

f.) Water Source: Potable Water: Wells drilled on the property should provide ample potable water supplies.

Mill Water: The water source for plant operations could pose a problem. The flow of the Partridge River 1.3 miles SE of the plantsite is not known at this time. The potential supply from wells on the site is also unknown at this time. The next potential possibilities in order of accessability would be Cranberry Lake, 5 miles south of the plantsite; Colby Lake, 5.5 miles SW; the St. Louis River, 6.8 miles south; and Big Lake, 9 miles SE. All drainage is to the St. Louis River and thence to Lake Superior.

g.) Waste Disposal Area (Lean Ore, Waste Rock and Surface Stripping)

A designated area covering 3.15 sq. mi. (2016 acres) located as follows:



MINE SITE No. 1.

FIGURE - B - 2.

SE Diag. ½ Sec. 35, & Sec. 36 of T59N R13W; W½ Sec. 31 T59N R12W, Sec. 2 and E½ Sec. 3 T58N R13W will store 295 million tons of waste materials (rock and surface) at a maximum height of 50 feet over a period of 20 years. The average haulage distance would be approximately 2½ miles. This area is located southwest of the shaft site.

h.) Tailings Disposal Area: Parts of sections 4,5,6,7,8 and 9 of T.59N R.13W located 1½ miles north of the plantsite has been designated for the tailings disposal area and reclaim water pond. This area covers 3.28 sq. mi. (2099 acres) and will store some 230 million tons of tailings at a depth of 50 feet for the 20 year mine life period. The average pumping distance from the plant to the disposal area will be approximately 3.5 miles.

All water systems from the plant will be closed systems and so constructed to prevent leakage or seepage into the ground water or streams of the area.

i.) Water Management Area: In order to exert proper control and monitoring, the area exposed to mining and possible contamination of ground water and soil as well as the lakes and rivers of the area, it is recommended that a geochemical survey be made of the water management area (both land and water)to establish the base level of all pertinent metals and chemicals before mine development work is begun. For the water management area of this particular mine operation, 44.52 sq. mi. (28,500 acres) has been designated as follows:

T.58N. R.13W - Sec. 6,7
R.14W - Sec. 1,2,3,4,9,10,11,12
T.59N. R.13W - Sec. 3,4,5,6,7,8,9,10,15,16,17,18,19,20,21,22,28,29, 30,31,32,33
R.14W - Sec. 1,12,13,23,24,25,26,27,34,35,36
T.60N. R.13W - S½ Sec. 33, Sec. 34

Rivers and creeks in the above area include:

The Partridge River	- 12 miles
Wyman Creek	- 7.5 miles
Langley Creek	- 3.5 miles
Colvin Creek	- 2.5 miles
Watleg's Creek	- 2.0 miles
Creek entering Colby Lake	- 1.0 miles

This area also includes the mine site, plant site, waste disposal and tailings disposal areas. All creeks and rivers listed above flow generally to the southwest with the exception of Colvin Creek which flows northward from Cranberry Lake before joining the Partridge River in Sec. 28 T.59NR 13W, making this particular creek a good monitor base. All the above listed creeks and rivers should have adequate water monitoring stations established along their length and careful analyses should be made of all samples before any work begins in order to establish the base data for controls to be established. The tailings disposal area and the rock dump area are located in areas of low relief and swampy ground so that the effluents of these areas, as well as those from the plant, should be kept under complete control. A settling pond should be established between the tailings area and the plant (about 6 acres) for recycling water from this area.

Production from this mine should approximate 11.3 million metric tons per year as suggested by Lawver, Wiegel and Shultz in their paper, "Study of Economic Feasibility for Copper-Nickel Development in Northern Minnesota", delivered at the University of Minnesota Mining Symposium, Duluth, in January, 1976, and published in the proceedings of this meeting (Appendix "A"). The method of mining recommended is "Block Caving". The possibility for this material (Gabbro) caving to the surface would be remote as the caving stopes would arch to support the more than 1500 feet of rock above the ore body. The feasibility of pumping coarse tailings into these voids should also be studied. Tailings from this processing plant are composed of a high percentage of Calcic Felspar and, if it can be beneficiated, could be a source of Alumina (Al_2O_3) and hence metallic aluminum. A total mine production of 211 million tons (metric) of tailings containing an average of 11.8% Al_2O_3 (52.9% Al; 47.1% O) is equal to 1,298,000 metric tons/year Al_2O_3 or 1881 tons Al metal per day.

III. Proposal No. 2, Underground Mine. (Fig.B 3)

a. Shaft Site: NE of the NW¼ of Sec. 11 T.60NR.12W

b. Mill Site: SW-NW and NW-SW Sec. 14 T.60NR.12W

c. Shaft Depth: approximately 1800 feet

d. Accessability:

1. Roads - State Highway #81 east from Babbitt to Forest Service Road #112; 1 mile south to F.S. Road #424, 6 miles east and then construct mine road 1.5 miles NE to mine and mill site.

2. Railroads:

a. Erie Mine RR passes 0.75 miles west of mill site. Investigate possibility of building a 150 car siding to arrange with Erie Mining Company to have the D.M. and I.R. RR. haul 285 cars of concentrate per month, 20 miles to the connection with the DM and I.R. RR. in Sec. 17 T.59NR.14W.

b. Transport concentrates from mill loading station on Forest Service Road #424. Truck haul west 20.5 miles to the D.M. & I.R. RR. at Embarrass. Construct a loading station for RR. loading at Embarrass.

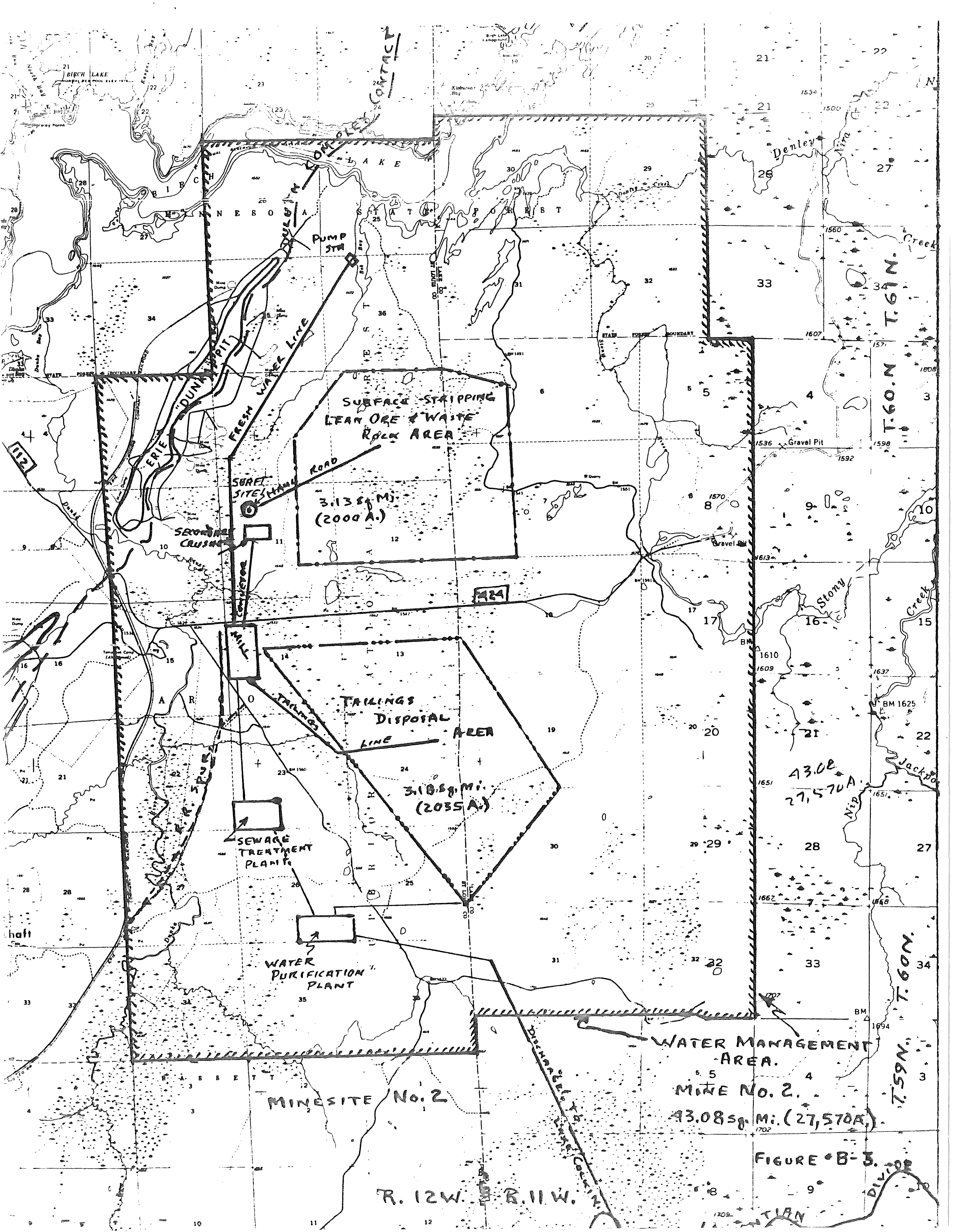

DULUTH COMPLEX CONTACT
BIRCH LAKE
MINNESOTA STATE FOREST
PUMP STA
ERIE
DUNKA PIT
FRESH WATER LINE
SURFACE STRIPPING
LEAN ORE & WASTE ROCK AREA
HAUL ROAD
SURF. SITE
3.13 Sq. Mi.
(2000 A.)
SECONDARY CRUSHER
CONVEYOR
MILL
TAILINGS DISPOSAL AREA
TAILINGS LINE
3.18 Sq. Mi.
(2035 A.)
R.R. SPUR
SEWAGE TREATMENT PLANT
WATER PURIFICATION PLANT
DISCHARGE TO DUNKA RIVER
43.08
27,570 A.
WATER MANAGEMENT AREA.
MINE No. 2.
43.08 Sq. Mi. (27,570 A.)
MINESITE No. 2
R. 12W.
R. 11 W.
T. 61 N.
T. 60 N.
T. 59 N.
T. 60N.
Denley
Nira
Creek
Stony
Creek
Jackpot
Nip
Gravel Pit
Gravel Pit
STATE FOREST BOUNDARY
BASSETT
ARGO
DIVIDE
112
424

FIGURE B-3.

e. Electric Power Source: An electric power line from the Erie plant to Dunka Mine is located 0.75 miles west of the proposed mill. It might be necessary to arrange with Erie Mining Co. and MP & L Co. to increase the capacity of this line. The towers for the transmission line are already there and in use. Arrangements would have to be made with Erie Mining Co.

f. Water Source: (1) Birch Lake. A 3.5 mile pipe line could be built from **Bob's Bay on Birch Lake (NE-NW Sec. 36 T.61NR. 12W). A pumping station can be built at that location to provide ample water for processing the ore.** Permits would have to be obtained from the various State departments for the use of this water and it would have to be used in a completely closed circuit. The potable water supply could be obtained from wells drilled on the property.

g. Waste Disposal Area (Lean Ore, Waste Rock and Surface Stripping) A designated area of 3.13 sq. miles (2000 acres) located as follows: Parts of Sections 6 and 7. T.60NR11W and all of Sec. 1, and parts of Sections 2, 11 and 12 of T.60NR12W. This is an area while not entirely flat will lend itself well to the separation of the various products to be disposed of such as the surface stripping, lean ore and waste rock. A road that is located in SW-SW Sec. 6 and W½-NW Sec. 7-T60NR11W can be relocated. This disposal area is located ½ mile to 2.3 miles due east of the shaft site. Average haul will be 1.3 miles.

h. Tailings Disposal Area: The center of this designated area is approximately 3 miles SE of the mill site and was selected for its low relief. The area covers parts of Sections 18, 19 and 30 of T.60N.R.11W and parts of Sections 13, 14, 23, 24 and 25 of T.60N.R.12W. This area will be surrounded by a perimeter canal for collection of any seepage outside the tailings dikes and the water system will be a completely closed circuit. Tailings will be pumped approximately 3 miles to the area for distribution over the area.

i. Water Management Area: For the mine and mill to be located as designated, a water management area has been set up to include the mine and plant and the two disposal sites. This area covers a total of 43.08 sq. miles (27,571 acres) and covers the following areas:

T61N.R.11W - Sec. 29,30,31,32
R.12W - Sec. 25,26,35,36
T60N.R.11W - Sec. 5,6,7,8,17,18,19,20,29,30,31,32
R.12W - Sec. 1,2,3,10,11,12,13,14,15,22,23,24,25,26,27,34,35,36

Also included in this water management area are the following: The Dunka River and Tributaries 10.5 miles; Stony River and Tributaries 9.0 miles; Creek to Bob Bay of Birch Lake 2.0 miles; Denley Creek 1.0 mile and 4 miles of the south shoreline of Birch Lake. Drainage flow of this area is to the NE into Birch Lake and into the South Kawishiwi River and thence into the Boundary Waters Canoe Area (BWCA).

j. Method of Mining: Block caving is recommended since at a depth of approximately 1900 feet and a zone thickness of 200 feet, the rock overbreak above the ore zone should arch to support the rock above to prevent caving to the surface. The feasability of using coarse tailings for fill above the ore zone should be studied to permit backfilling when an area of the mine has been exhausted of economically mineable ore.

Production from this mine should approximate that suggested for Minesite Proposal No. 1.

Three items should be considered seriously in the planning for the operation of this mine:

1. The feasibility of using coarse tailings for backfill in the open stopes when mining is completed.
2. The feasibility of the beneficiation of the calcic felspar in the tailings to produce an alumina product.
3. The feasibility of pumping the discharge effluents (after purification) into the watershed of the St. Louis River, south of the Laurentian Divide, approximately seven miles either to a tributary of the Partridge River in the E½ of the NE¼ of Sec. 29 T.59NR. 11W or to the North River in the N½ of the SE¼ of Sec. 7 T.59NR. 12W.

All water systems will be operated in closed circuit and all effluents will be treated in a water treatment plant before any discharge into a watershed area. This is also true for the sewage disposal plant to be built on the property for the personnel employed.

It is very important that a geochemical survey be conducted over the entire water management area to establish the base mineral content of the soils before any construction or development of a mining operation begin. It is equally important to survey the rivers and lakes of this area. Any water treatment plant should be capable of treating the effluents of the disposal areas and/or plant and mine to within reasonable limits of the condition in which they are found prior to the development of the property.

IV. Proposal No. 3 - Open Pit Mine. (Fig.B4)

a.) Mine Site (Open Pit Limits)

W½-SW Sec. 25, E½ Sec. 35, W½-NW, NW-SW Sec. 36-61-12, NW-NE, NE-NW Sec. 2-60-12

Depth, - 1110 feet, Length. - 1.5 miles, Width. - 0.5 miles

b.) Mill Site:

1. Primary Crusher - SW-SW Sec. 36-61-12
2. Secondary Crusher - SE-SE Sec. 1-60-12
3. Mill site - Lots 3 and 4 Sec. 18-60-11

c.) Pit Dimensions:

1. Length 1.5 miles (NE-SW)
2. Width 0.5 miles (NW-SE) Maximum
3. Depth 1110 feet

d.) Accessability:

1. Roads - Forest Service Road 112 from Babbitt to Forest Service Road No. 424-3.6 miles east to Mill site.
2. Railroads - Erie Mining Co. railroad is located 3 miles west of Mill site in NW-NW Sec. 15-60-12.
3. Open Pit Site is 2½ mi. NW of Mill site. Roads and conveyors will be used to transport men, material, and ore from mine to crushing plants and to the mill.

e.) Electric Power:

Reserve Mining Company Power Transmission line is located 3 miles west of the millsite. The possibility of tying into this line should be investigated with both Reserve Mining Co. and MP & L.

f.) Water Sources:

1. Potable Water Supply: Water wells drilled on the property should provide an ample water supply for the personnel working on this project.

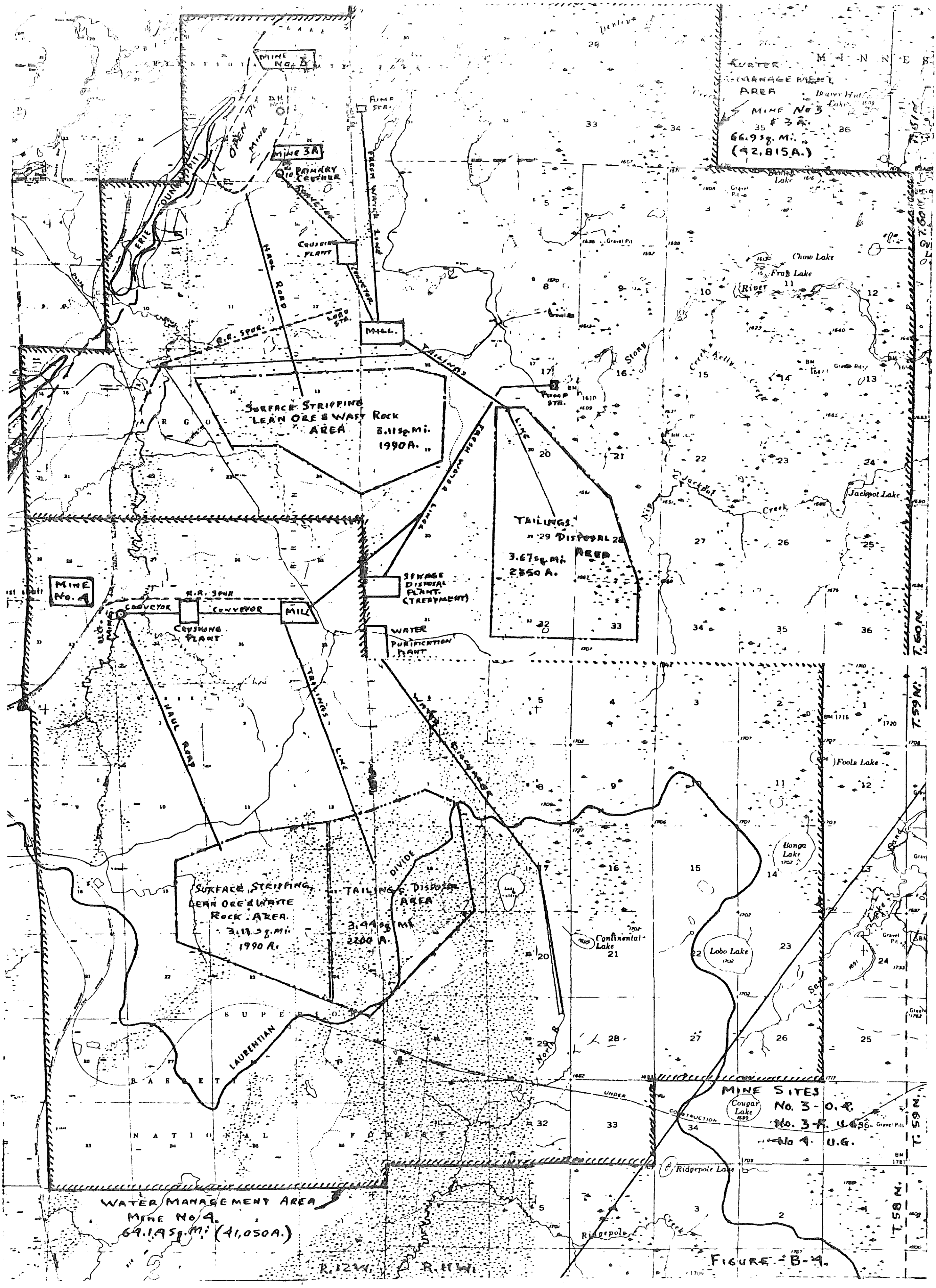

MINE No. 3
OPEN PIT MINE
ERIE MINING CO. PIT
MINE 3A
PUMP STA.
PRIMARY CRUSHER
CONVEYOR
FRESH WATER LINE
CRUSHING PLANT
HAUL ROAD
MILL
LOAD STA.
R.R. SPUR
TAILINGS LINE
SURFACE STRIPPING LEAN ORE & WAST ROCK AREA 3.11 sq. Mi. 1990 A.
PUMP STA.
FRESH WATER LINE
TAILINGS DISPOSAL AREA 3.67 sq. Mi. 2350 A.
SEWAGE DISPOSAL PLANT (TREATMENT)
WATER PURIFICATION PLANT
MINE No. 4
R.R. SPUR
CONVEYOR
CRUSHING PLANT
CONVEYOR
MILL
HAUL ROAD
TAILINGS LINE
LAKE DISCHARGE
SURFACE STRIPPING LEAN ORE & WASTE ROCK AREA 3.11 sq. Mi. 1990 A.
TAILINGS DISPOSAL AREA 3.44 sq. Mi. 2200 A.
DIVIDE
LAURENTIAN
WATER MANAGEMENT AREA MINE No 3 & 3A 66.9 sq. Mi. (42,815 A.)
WATER MANAGEMENT AREA MINE No. 4. 64.14 sq. Mi. (41,050 A.)
MINE SITES No. 3 - O.P. No. 3-A. U.G. No 4. U.G.
Chow Lake
Frog Lake
River
Stony
Kelly Creek
Jackpot Creek
Jackpot Lake
Dunnigan Lake
Beaver Hut Lake
Fools Lake
Bonga Lake
Continental Lake
Lobo Lake
Cougar Lake
Ridgepole Lake
Ridgepole Creek
North R.
Gravel Pit
UNDER CONSTRUCTION
SUPERIOR NATIONAL FOREST
BASSETT
ARGO
T.60 N.
T.59 N.
T.58 N.
R.12 W.
R.11 W.

FIGURE B-4.

2. Mill Water: Establish a pumping station on the south end of the tip of Birch Lake that extends into the SE-NW of Sec. 36-61-12 and pump 3.75 miles through a pipe line to the mill.

Alternate water supply for mill - Dunka River, 2.5 miles west of the millsite. Water available. Unknown.

g.) Waste Disposal Area (Lean Ore, Waste Rock & Surface-Stripping):

A designated area approximately 3 miles south of the proposed open pit of 3.11 sq. mi. (1990 acres) located as follows: Parts of Sections 13, 14, 23, 24 of T.60NR.12W and parts of Sections 18 and 19 of T.60NR.11W. This area will store approximately 290 million tons of waste materials (rock and surface stripping) at a height of 50 feet over the 20 year life of the mine. The average haulage distance would be approximately 4.5 miles. This area should be underlain by an impermeable pad to control any seepage or leaching that could occur, and the area would be surrounded by a perimeter canal draining to the south point of the area. Here the effluent would be collected and either recycled or pumped to the purification plant for disposal into the watershed south of the Laurentian Divide. The purification plant and Sewage Disposal Plants should be located in the area of Sec. 31 T.60NR.11W (lots 12 & 13).

h.) Tailings Disposal Area:

The area designated for tailings disposal is designated as follows: All of Sec. 29 and parts of Sections 20, 21, 28, 32 and 33 of T.60NR.11W and totals approximately 3.67 sq. mi. (2350 acres). This area will store some 270 million tons of tailings at a depth of 50 feet for the 20 year mine life period. The average pumping distance from the mill to the disposal area will be approximately 2.5 miles. The area will be surrounded by a perimeter canal draining to the south point of the area where it will be collected and recycled or pumped to the purification plant for treatment before disposal.

into the watershed on the south side of the Laurentian Divide. A settling basin of approximately 6 acres will be located between the tailings area and the plant for the recycle water.

All water systems will be completely closed circuits, and so constructed as to prevent any leakage or seepage into the ground water or streams of the area. A complete geochemical survey should be made of the entire area before any work commences in order to establish the base from which proper levels of control can be established. A complete monitoring of all rivers and lakes of the area should also be done before work begins for the same purpose. Permanent monitoring stations should be established on Birch Lake, along the Dunka and Stony Rivers and along Nip, Jackpot, Niva, Denley and Kelley Creeks.

i.) Water Management Area:

In order to exert the proper control over the area exposed to mining and possible contamination of the soil, groundwater, lakes and rivers of the area; it is recommended that a complete geochemical survey be made of the water management area designated as the following 75.82 sq. mi. (48,525 acres):

T.61NR.11W - Sec. 27, 28, 29, 30, 31, 32, 33 & 34
R.12W - Sec. 24, 25, 35, & 36
T.60NR.11W - Sec. 3, 4, 5, 6, 7, 8, 9, 10, 15, 16, 17, 18, 19, 20, 21, 22, 28, 29, 30, 31, 32, 33 & 34
R.12W - Sec. 1, 2, 3, 10, 11, 12, 13, 14, 15, 16, 21, 22, 23, 24, 33, 34, 35 & 36

Rivers and Creeks included in the above area are:

The Dunka River and tributaries - 14 lineal miles
The Stony River and tributaries - 18 lineal miles

Nip Creek - 7 lineal miles

Jackpot Creek - 6 lineal miles

Niva Creek - 4 lineal miles

Denley Creek - 3 lineal miles

Kelley Creek - 2 lineal miles

Also, 6.5 miles of the shoreline of Birch Lake

Complete monitoring of the above should be completed with all analyses before work is begun. Then permanent monitoring stations should be established at strategic locations for control points. All rivers and creeks listed drain into the Birch Lake Watershed and hence into the B.W.C.A. Therefore, it is essential that good controls are provided, based on the data acquired before the beginning of any operation and continued monitoring during operations. This area for water management includes the mine site, mill site, waste rock and tailings disposal areas. The last two sites being located in areas of low relief and reasonably close to the Laurentian Divide to make it feasible to dispose of effluents (after purification) into the watershed of the St. Louis River.

Production from this mine should approximate the 11.3 million metric tons per year (41,450 D.S.T/day) as suggested by Lawver, Wiegel and Shultz in their paper. The method of mining would be open pit, based on the information from a drill hole located in NW-NW of Sec. 36 T.61NR. 12W which indicates 140 feet of 0.197% Cu and 0.076% Ni ore, equivalent to 0.422% combined Cu-Ni value between the depths of 970 feet and 1110 feet below the collar of the hole (El. ± 1470 ft.). Also, 40 feet of 0.543% Cu and 0.235% Ni (1.241% combined Cu-Ni value) between the depths of 610 and 650 feet below the collar. The pit limits would be approximately 1.5 miles long in a SW-NE direction and 0.5 miles in a SE-NW direction, and would cover approximately 0.74 sq. mi. A proper control barrier would have to be maintained between the NE side of the pit and Bob[illegible] Bay of Birch Lake to prevent seepage in either direction.

When this open pit mining has been completed, processing of additional Copper-Nickel ores indicated by another drill hole in SW-SW Sec. 36-61-12 could use the surface facilities described above. This underground operation would mine 50 feet of 0.208% Cu; 0.014% Co and 0.148% Ni at 2080 feet to 2130 feet below the collar (El. 1450), and 47 feet of 0.324% Cu; Trace Co, and 0.089% Ni between 2133 and 2180 feet below the collar. Combined equivalent analyses are 0.647 and 0.587% Cu respectively. This area could be mined by room and pillar method or (if the dip is over 40° - Rill or shrinkage stoping). Additional waste rock and tailings disposal areas would be needed for this operation.

V. Proposal No. 4 - Underground Mine (Fig.B4)

a.) Shaft site location - NW-NW Sec. 34 T.60NR. 12W

1. Primary Crusher, underground

b.) Mill site

1. Secondary Crushers - NE-NE Sec. 34-60-12
2. Mill site N½-NW Sec. 36-60-12

c.) Shaft depth - 3625 feet (including skip pocket)

d.) Accessability:

1. Roads - Forest Service Road 112 SE from Babbitt to Forest Service Road 424-1 mile east to F.S. Road 114-3.5 miles SE to mill site. A mine road would be built 3 miles west to the shaft site.
2. Railroads - The Erie Mining Company railroad is located 3.5 miles west and the Reserve Mining Company railroad is located 2 miles south of the Erie facility, or 5.5 miles from the mill site. A railroad spur would have to be built to service the plant.
3. Build a 1 mile conveyor from the shaft pocket to the secondary crusher. and a second conveyor 2 miles to the fine ore bins at the mill. These conveyors would parallel the mine road from the shaft site to the mill.

e.) Electric Power Source:

The Reserve Mining Co. power transmission line is located 2 miles NW of the shaft site (4.5 miles from the mill site). The Erie Mining Co. power transmission line is located 4 miles SW of the secondary crushing plant site. Feasibility studies should be made with both of the mining companies and MP & L to determine what steps would be necessary to provide adequate electric power for this operation and which line would be best to furnish this power.

f.) Water Source:

Potable water supplies can probably be furnished by wells located on the property. These wells should furnish adequate water for the personnel employed.

Mill water source could be located on the Stony River. A pump station could be located 4.5 miles NE of the mill in the NW-NE of Sec. 17 T.60NR.11W and a pipeline from the pump station to the mill. Permits from the various State agencies would be necessary for the use of this water. All water must be in a completely closed circuit.

g.) Waste Disposal Area (Surface Stripping, Lean Ore & Waste Rock):

A designated area of 3.11 sq. mi. (1990 acres) has been located in an area of low relief and swamp land covering all of Section 14 and parts of Sections 11, 12, 13, 15, 22, 23 and 24 of T.59NR.12W. This area will store some 290 million tons of surface material, lean ore, waste rock at a height of 50 feet, maximum, over the period of the mine life (20 years). The average haulage distance would be about 4 miles and drainage would be to a tributary of the North River, south of the Laurentian Divide after purification at the treatment plant located in Section 31 T.60NR.11W.

h.) Tailings Disposal Area:

The center of this designated area is approximately 4 miles SE of the millsite and was selected for its low relief and its proximity to the Laurentian Divide. It is 3.44 sq. mi. in area (2200 acres) and will provide room for approximately 325 million tons of tailings at a depth of 50 feet for the 20 year life of the mine. The area is located in parts of Sections 7, 18 and 19 of T.59NR.11W and parts of Sec. 12, 13 and 24 of T.59NR.12W. It will be surrounded by a perimeter canal to collect seepage outside the tailings dikes and the water system will be a completely closed circuit. Drainage from this canal will be pumped to a settling pond (about 6 acres) between the tailings area and the mill for recycling as make-up water. An alternate disposal of this effluent could be through the purification plant for disposal into the west branch of the North River, south of the Laurentian Divide.

i.) Water Management Area:

For a mine and mill operation to be established in the above area, a water management area of approximately 55.76 sq. mi. (35,686 acres) has been set up to include the mine, mill and all disposal areas as well as the lakes and rivers in the surrounding area. It is located as follows:

T.59NR.11W - Sec. 2, 3, 4, 5, 6, 7, 8, 9, 10, 11, 14, 15, 16, 17
18, 19, 20, 21, 22, 23, 26, 27, 28, 29, 30, 31, 32 & 33
T.59NR.12W - Sec. 1, 2, 3, 4, 9, 10, 11, 12, 13, 14, 15, 16, 21, 22
23, 24, 25, 26, 27, 28, 33, 34, 35, & 36
T.60NR.12W - Sec. 25, 26, 27 & 28

Also included in the water management area are the following:

Dunka River and tributaries - 8 miles
North River and tributaries - 2 miles
Lobo Lake shoreline - 2½ miles
Bonga Lake shoreline - 2 miles
North End Big Lake shoreline - 1.5 miles
Lake Culkin shoreline - 1.5 miles
Continental Lake shoreline - 1 mile
Fools Lake (west half) shoreline - ½ mile

The Dunka River, its tributaries, Lobo Lake and Bonga Lake all drain into Birch Lake (the two lakes via Sand Lake and Sand River, Stony Lake and Stony River). All other lakes and the North River and tributary drain into the St. Louis River watershed. All effluents from both mine and mill and sewage disposal plant which are not recycled will pass through the water purification plant before being discharged into the west branch of the North River. Water discharged from the plant must meet PCA standards before discharge.

j.) Method of Mining:

Drill hole analyses show four zones of mineable grade of Copper-Nickel ore as follows:

Depth	Thickness	%Cu	%Ni	%Combined
1793 to 2080 ft.	287 ft.	0.25	0.10	0.547
2580 to 2602 ft.	22 ft.	0.305	0.10	0.602
2782 to 2894 ft.	112 ft.	(company	report)	0.542
3093 to 3521 ft.	428 ft.	0.239	0.072	0.453

The dip of the ore body, according to the sketchy information available, seems to be approximately 17° to the SE from the Contact. It is suggested that the top zone and the bottom zones be mined by the sub-level caving method and that the middle two zones be mined by inclined room and pillar methods. The distance between the ore zones are great enough to support the workings of two or three of the ore zones simultaneously and for the high volume production necessary to support this mine, these would be the most economic methods to do it safely.

One fact must be remembered, all assumptions for the design and operation of this mine are based on the information available at this time (August 2, 1976) and the analysis of only one drill hole.

Production of this mine is based on the tonnage suggested for proposals 1, 2 & 3.

Water Usage

1. The mine and mill water supply is to be a closed system with make-up water provided by precipitation in excess of evaporation, and amsmall input of some fresh (river or lake) water.

   Selection and preparation of the tailings area will require care and study to assure a minimum of water seepage from the impoundment area. Perimeter canals to accept seepage waters from the tailings would be channeled to adequate settling basins from which the effluents could be combined with the regular water recycling system from the tailings pond.

   The tailings area will require partial revegetation throughout the life of the mine, although the major part of the tailings should be water covered to prevent wind erosion. Complete revegetation will be done when the mine has been exhausted.

   Lean ore and stripping dumps will also require revegetation at the completion of operations.

2. The mine will need water for the drilling equipment, automotive equipment, sewage disposal system, dust control, and a potable drinking supply for personnel.

3. The mill will need water for use by the grinding, classifying, thickeners, flotation machines, screens, and other milling equipment as well as for dust control, sewage treatment and a potable drinking supply for personnel. Water will also be needed for major dust control of the tailings basin.

4. The collection of the water will be provided by pumps from the original make-up source (lake, river and/or well) to a mainstorage tank located in close proximity to the mill and high enough to provide a gravity feed to milling operations. A separate, but similar, water system will be required for the potable water supply with a smaller storage tank. The main-nonpotable supply tank should be large enough in capacity that it could also be used for a source of water in case of fire. At the end of each use of water in the mill, water will be drained into sumps and pumped to settling basins where the contained material will settle and the collected water can be recycled. Water used in mining equipment, for the most part, will be lost. It is possible also that the mine could generate some water from the rock through fractures and fissures. Inco's shaft and stope (1) operation produced less than 300 g.p.m. at a depth of 1100 feet. This water would probably be used in the mine for dust control and/or pumped to the surface for use in the mill. The discharge from the milling operations would be directed to tailing thickeners and settling basins for clarification, and any excess to a treatment plant for further purification before being discharged into the watershed of the particular area. If heavy metals are present in the effluents, further treatment will be required for their removal.

(1) Inco Maturi Shaft Sec. 5 T.61NR. 11W

5. Water recycling would be maximized and discharge to the watershed minimized as much as possible. According to the Hays Report of August, 1974, a total of 720 gallons water per ton* of ore treated would be required. Therefore, with a rated capacity of 11.3 million metric tons per year, the water consumption will be 17,066 gallons per minute. Assuming 85% recycle and 15% make-up, the fresh water consumed will then be 2560 gallons per minute.

6. 3.5 million gallon water storage tank would furnish the make-up water. This water would come from the primary source (lake, river and/or wells). This storage tank would also supply water for emergency fire protection.

7. Both the rock and stripping dump areas and the tailings disposal area would have perimeter canals surrounding them with drainage to a sump at the low end of the areas where the seepage water would be collected for recycling, and/or purification before discharge into the watershed.

8. Depending on the amount, and quality, of water generated from the mine, this water, with a little treatment, could possibly be used as a source of potable water supply. Otherwise, the source of the potable water would come from either Birch Lake, Colby Lake, Big Lake, the Partridge River or the St. Louis River. Very little treatment would be required for water from any of these sources. A separate storage tank would be required for this water.

*short ton of 2000 lb.

9. All effluents from the mine and plant would be treated in a neutralization process before being discharged into the watershed of the area. Particular care should be exercised in the areas north of the Laurentian Divide since this watershed passes into the BWCA. This is especially true of heavy metals or deleterious chemicals are detected in the effluents.

Waste Disposal (14.7 MMT/Y)*

1. Waste rock, lean ore and surface stripping would be segregated and stacked to a height of 50 feet on a ± 2000 acre tract selected in the most acceptable (environmentally) location as close to the mine as possible for economic considerations and controlled disposal of this material. A perimeter canal would surround the area with ditches at strategic points from the stacks draining into the perimiter canal. A geochemical survey should be made of the area before any dumping takes place to provide a base of data to determine the possible contamination of the ground by dumping this material on this 2000 acre tract. This is particularly necessary at the bottom of the perimeter canal after it has been dug. The overburden would be deposited in the same area as the waste rock and lean ore, but in separate stacks. As mining proceeds, it is possible that some of the waste rock could be used as backfill for some of the mined out areas. That of course, would depend on the mining method used. It would not be possible if block caving were used.

Tailings (11.0 MMT/Y)*

1. The tailings disposal area of ± 2000 acres (50 feet deep) would be selected in the most environmentally acceptable area as close to the mill as possible. This area too would have a perimiter canal to receive seepage that would be generated from the tailings draining into a sump or pond at the low end of the area. This water would be pumped back along with the recycle water from

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*Million Metric Tons per Year

the tailings area itself to the mill for reuse. The area would be covered with water to the extent practical and revegetation would be promoted in dry areas continually to prevent wind and water erosion. Continuous monitoring of the seepage water would be made, and if any is allowed to join the watershed of the area, neutralization and purification would be done before this would take place. The geochemical survey of the bottom of the canal after it has been dug would establish the base information to control ground contamination.

Effluents

1. Effluents from the mine would only be an excess of water produced by the mine through faults, fractures or fissures in the rock. This would be used for dust control in the mine with the excess being pumped to the surface to supplement the make-up water for the mill. If the mine water is sufficiently pure, it could add to the potable water supply instead of going to the mill. Drill water excess will enter the mine water excess to be pumped to the surface.
2. Effluents from the mill would be water, probably contaminated by both reagents and dust. The mill floor should be made of finished concrete sloped in such a manner as to drain all water into sumps conveniently located so that the water may be treated and recycled to the plant. It is customary to hose down most mills each shift hence drainage of the floor is an important factor. A level iron grill flooring is usually laid above the floor 4" to 6" for personnel traffic to prevent slipping on the wet concrete floor. All water used in the mill will be in closed circuit so that the only water leaving the mill would be that discharged to the tailings area and that contained in the filtered concentrate.

Pollution Control

1. Air -- In the mine, all drilling will be done wet, hence the only pollution problems would be dust from blasting and loading operations, and gases from

blasting and mobile equipment with combustion engines. Water will be used to control the dust from blasting and loading operations, forced air ventilation and dust collectors for blasting operations, and exhaust scrubbers for mobile equipment with combustion engines. Additionally, forced air ventilation would be used to provide dillution of any gases as well as fresh air for the personnel working underground.

In the mill, dust collectors would be installed in the crushing area where the major portion of the dust would be generated in the gyratory crushers, the secondary and tertiary crushers, and the fine ore bins. From the fine ore bins, water will be used through the circuit with the addition of reagents in the flot circuit to the concentrate dewatering and drying. Adequate ventilation of the mill building will provide fresh air for the personnel. Offices and electronic equipment rooms will be air conditioned and temperature and humidity controlled.

Tailings Area: A maximum amount of the tailings area will be covered with water at sufficient depth to prevent wind erosion, hence no air contamination would result. That area not possible to maintain inundated will be continuously revegetated to prevent wind and/or water erosion. Chemical treatment of the tailings surfaces may be used where difficulty is encountered in promoting vegetation growth.

Roads: A sprinkler truck will be used to keep dust on the plant roads under control. These roads would include those which would not normally be blacktopped such as those surrounding the rock dump and tailings areas and those providing access to these areas. Roads in the plant area would be blacktopped.

2. Water - Waters that are not to be recycled, namely those that will be discharged into the watershed or into the groundwater, will be put through a neutralization process to remove as much of the contaminating material as is technically possible. This will be done after it passes through a

proper settling basin (approximately 6 acres in size) strategically located as to reception from the plant, purification plant and discharge to the watershed and/or ground water table. Monitoring of the inflow and outflow of the purification plant will be continuous. Water used for the sewage system must be also treated to the extent possible to prevent contamination of the watershed. At least a secondary sewage disposal plant would be necessary since approximately 860 people will be employed at each mine and mill, and both will be operating on a year around basis. (300 days)

3. Ground Pollution - This type of pollution is difficult to describe in concrete terms. Contamination of the ground by effluents such as could occur in the canals surrounding the rock, stripping and tailings areas, can be monitored to give a fairly accurate indication of how much contamination is occurring by geochemical surveys. Esthetic pollution, which is more difficult to define, will occur in practically every phase of the operation from the original prospecting through to the exhaustion of the mine. Efforts must be made to have all construction coordinated so as to blend into the adjacent area to make it as esthetically pleasing and least offensive as possible. There will be destruction of trees, bypassing of creeks, diversion of water channels, filling in of swamps, etc., all of which will be necessary for the operation of a mine and mill to succeed. Contributors to this esthetic pollution would be the building of access roads; clearing of mine site and mill site; preparation of the rock and tailings disposal sites; construction of the mine shaft facilities and/or open pit and facilities; construction of the mill; power line and substation; water lines and storage tanks; sewage system and disposal plant; and water purification facility. All this work should be done in a manner which is in harmony with the area in which it is located so that it blends well with its background. At the end of the

operation, the area, before abandonment, should be completely rehabilitated in such a way that when nature takes over, the scars will be healed and the area will again be a part of the former ecological landscape.

Remarks:

Because of the lack of concrete information in many areas of the "Minesite Area" such as the dip of the beds, drill hole analyses, depth of the overburden, flow of the rivers and creeks, etc., many assumptions were necessary in this report. The proposed mines are hypothetical since they are not based on known ore reserves. These assumptions and locations were chosen to indicate the major conditions that will have to be encountered and solved if a viable ore reserve is found in the Minesite Area.

In calculating the combined Copper-Nickel analysis, nickel assays were converted to copper values on the basis of the 74 cents per pound price of copper and the $2.20 price per pound of nickel hence, nickel assays were multiplied by 2.97 to convert to the copper equivalent. This figure was then added to the copper analysis to give the combined Cu-Ni value.

To develop a viable underground mine, an average Cu-Ni value would need to be at least 0.75% combined. With the exception of two analyses (A 10 ft. layer in Sec. 16-57-14 and a 40 ft. zone in Sec. 36-61-12), we have no drill hole analyses that will average this mineable grade.

Between Section 10 T.60NR.12W and Section 17 T.59NR.13W and also between Section 9 T.58NR.14W and Section 16 T.57NR.14W, we presently have very little geological or exploration data to work from.

From the information at hand at this time, the area of greatest potential for development would seem to be located along the contact from Section 4 T.58NR.14W to Section 24 T.62NR.11W and from the contact east between 1.5 and 2 miles for underground development and from the contact to approximately 0.5 miles east for an open pit development.

For underground mining, three methods of mining can be considered for large volume mining, depending on the thickness of the ore zone, the dip of the zones, and the distance between ore zones (barren zones).

1. Room and pillar for zones up to 100 ft. thick and the dip of the zone under 20°.
2. Sub-level caving for zones up to 500 ft. thick and the dip of the zone under 20°.
3. Block caving for thick zones and where the dip is greater than 20°.

For multi-level mining where more than one method of mining is employed, the barren zone between ore zones should be greater than 150 feet to assure proper safe support for the upper level workings.

Depending on the presence of heavy metals and chemicals in the run-off waters from the surface stripping dumps, lean ore and waste rock piles, it may be necessary to lay an impervious mat over the entire waste rock area. For this purpose, the possibility of setting up a ready-mix concrete plant in the area should be investigated. There are several sand and gravel pits in the area, particularly in the NE segment, north of the Stony River.

If two or more mines are operated at the same time and in the same general area, the possibility of combined sewage treatment plants, water purification plants and water management areas should be studied as suggested in proposed mine sites III and IV of this study. **(Figure B-4)**

All efforts should be made to dispose of effluents which are not recycled after purification into the St. Louis River Watershed on the south side of the Laurentian Divide rather than through the Birch Lake Watershed and into the B.W.C.A.

Appendix A

A Summary of the Paper
"Study of Economic Feasibility for
Copper-Nickel Development in Northern Minnesota"
presented at the 49th Annual Meeting of the
Minnesota Section, A.I.M.E. and the
37th Annual Mining Symposium
Duluth, Minnesota
January 14-16, 1976

Copper-Nickel in NE Minnesota

On the Contact between the Duluth Complex and Giants Range Granite.

Dip - 20° to 40° to the Southeast

Favorable Zone: 30 to 35 miles long and up to 1 mile below the surface - Zone width approximately 2 miles.

Major Ore Minerals: Chalcopyrite ($CuFeS_2$), Cubanite ($CuFe_2S_3$) Pentlandite ($(Fe,Ni)_9S_8$)

Minor Minerals: Pyrrhotite ($Fe_{1-x}S$), Bornite (Cu_5FeS_4), Sphalerite (ZnS) -

In lenses parallel to the base of the gabbro.

Economic Analysis

2 identical mines operate at 11.3 million metric tons/yr. for 20 years.

Mining

Rotary Drilling; ANFO-Slurry Blasting; Electric Shovels (16 cu. yd.); 150 Ton Diesel-Electric Trucks

Milling-Crushing Circuit:

Primary Gyratory Crusher; Screens (2"); Secondary Crushers; Screens (½"); Tertiary Crushers;

Concentrating Circuit:

6-4500 HP Ball Mills; in closed circuit with Hydrocyclones; $\overset{80\%}{(-150M)}$; 114-500 cu.ft. Rougher Flotation Machines (Sodium Isopropyl Xanthate Collector; Polypropylene glycol, Methyl Ether Frother), Thickener, Regrind Circuit-5-700 HP Ball Mills in closed Circuit w/hydrocyclones to 80% (-325M); overflow to Cleaner Flotation (7-300 cu.ft. cells).

Tailings recirculated to rougher flot & middlings to reclean.

Recleaned tailings to rougher and recleaner conc. and combined with cleaner conc., thickened, filtered and dried in a fluid bed dryer for shipment to smelter.

Tailings Site = 2000 acres ($3^1/_8$ sq. mi.) at 50 ft. depth. - Preparation costs $7 million. Revegetation costs $200,000/yr. plus $2 million at termination.

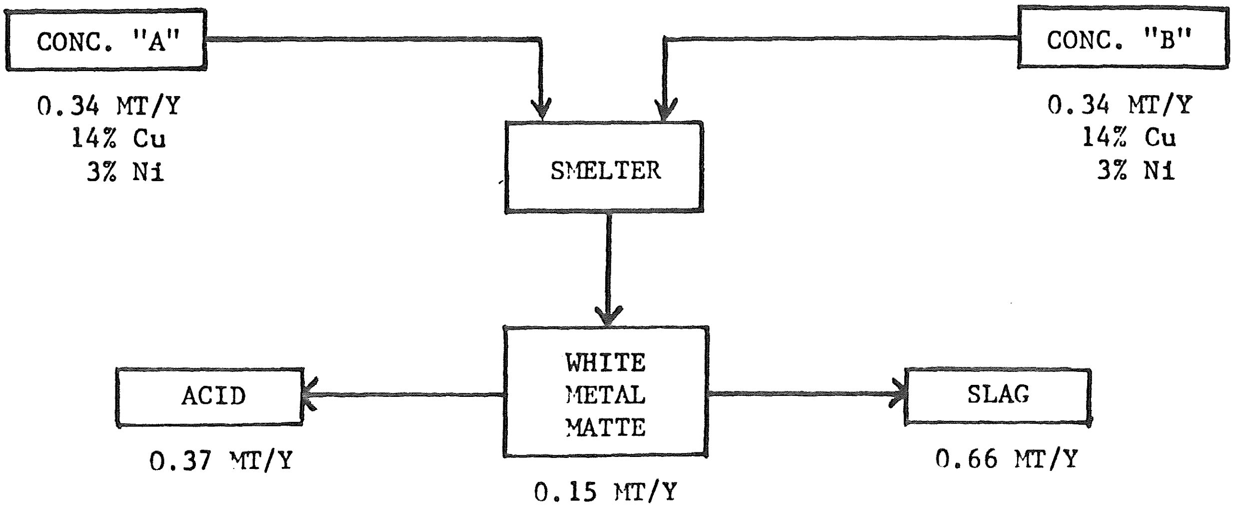
Lean Ore & Rock Disposal Area = 2000 acres at 50 ft. height - Revegetation cost $1 million at termination.

Smelting:

1. Fluid-bed roasters in tandem w/submerged arc resistance type electric smelting furnaces.
2. Outokumpu type flash smelting furnaces
3. Inco-type Oxygen flash smelting furnaces

depending on 1. Nature of the feed
2. Price of electric power and fossil fuels
3. Nature of local markets

Smelter Product Flows

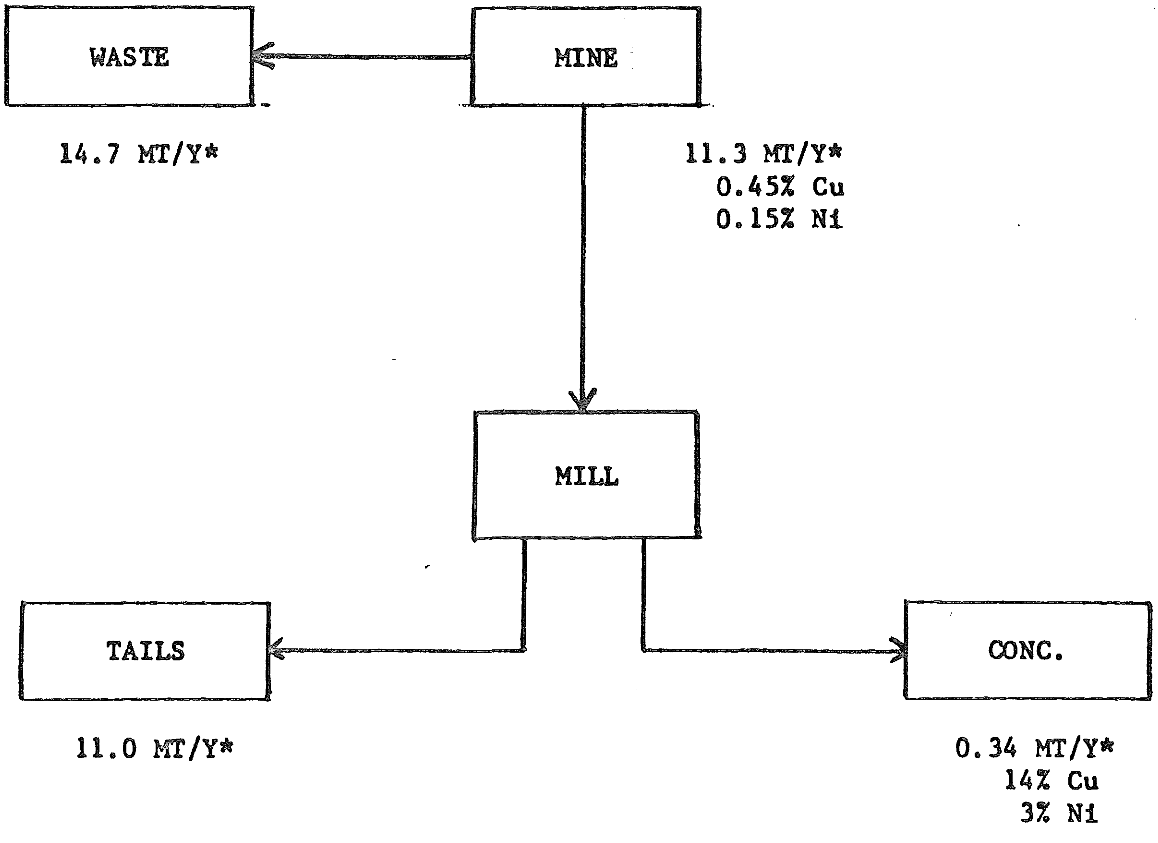


62.4% Cu
12.9% Ni
21.0% S
3.5% Fe
0.25% Co

4.90 oz./T Ag
0.15 oz./T Au
0.18 oz./T Pt
0.44 oz./T Pd
0.002 oz./T Rh

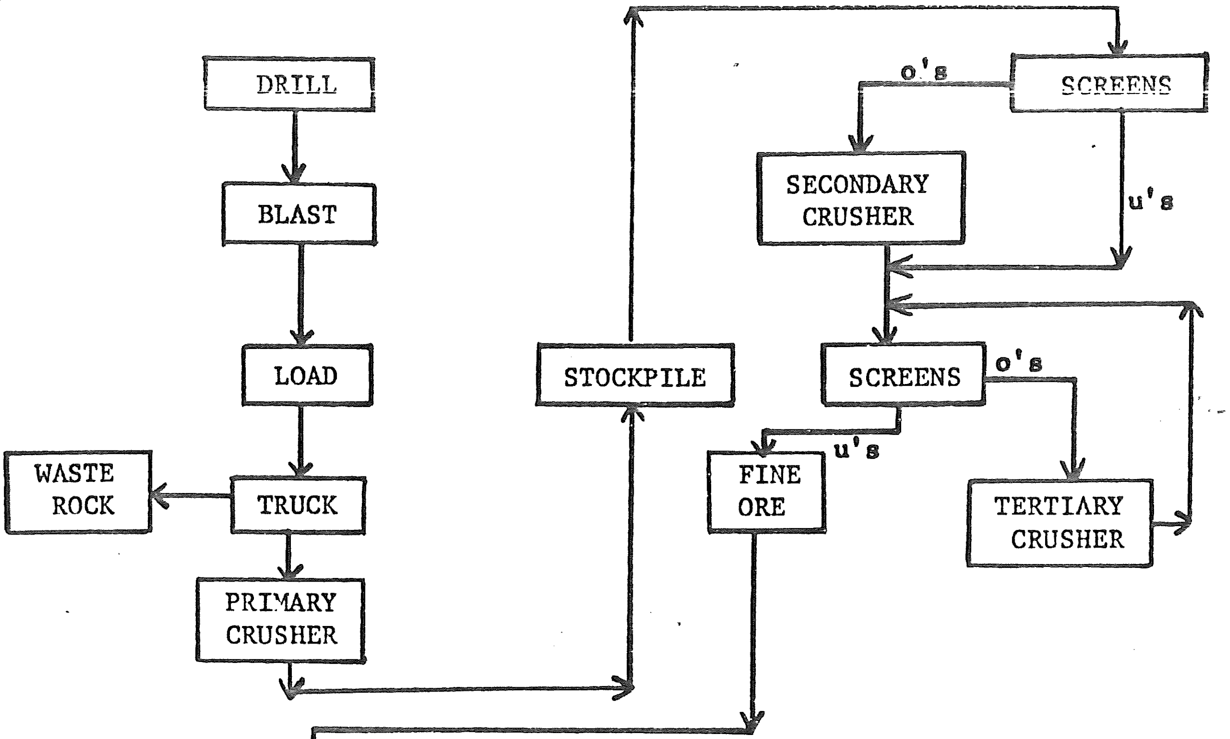
Mine and Mill

Flow Sheets

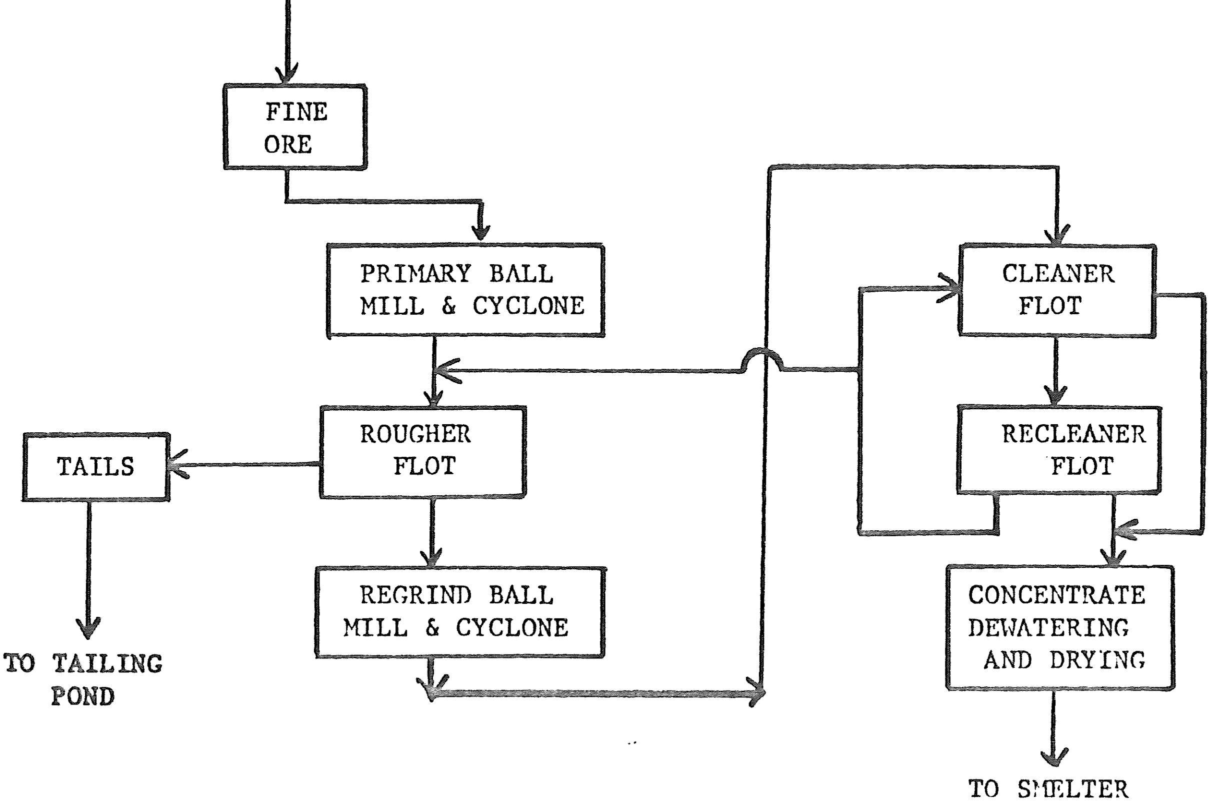


* Million Metric Tons Per Year

Mining & Crushing Flow Sheet



Grinding & Flotation Flow Sheet



To mine 11.3 million metric tons per year (Open Pit)

Mining - 3 - Rotary Drills

ANFO-Slurry Blasting

6-Electric Shovels-15 cu. yd. buckets

23-150 ton Diesel-Electric Trucks

Milling - 1-Primary Gyratory Crusher

Screens-2" and -½"

2-Secondary Crushers

5-Tertiary Crushers

6-4500 HP Ball Mills w/Hydrocyclones

5-700 HP Regrind Ball Mills w/Hydrocyclones

2-Thickeners

114-500 cu.ft. Rougher Flotation Cells

7-300 cu.ft. Cleaner Flotation Cells

-Filters

-Fluid Bed Dryer

Personnel - 860 for each mine and mill

450 mining

270 mill

140 technical

Mining & Milling Capital Costs
(Millions of Dollars)

Predevelopment Expenses	4.7
Mining	33.6
Milling	86.7
Tailings Site Preparation	7.0
Total-1 mine + 1 mill	132.0
-2 mines + 2 mills	264.0

Mining Operating Costs per Year
(Millions of Dollars)

Labor and Supervision	8.83
Energy-Explosives	2.12
Fuels	0.71
Electrical	0.51
Operating and Maintenance Supplies	2.65
Surface Stripping & Drilling (Contract)	1.49
Total - 1 mine	16.31
- 2 mines	32.62

Milling Operating Costs per Year
(Millions of Dollars)

Labor and Supervision	5.46
Energy-Electrical	3.59
Metal Wear	8.85
Flotation Reagents	1.21
Operating & Maintenance Supplies	1.62
Tailings Area Revegetation (Contract)	0.20
Total - 1 mill	20.93
- 2 mills	41.86

Rail Transportation Costs

Concentrates-Ely to Duluth	$4.50/M.T.
Slag -Duluth to Ely	$2.00/M.T.
Concentrates-Ely to Sudbury Area	$18.75/M.T.

Smelter Capital Costs
(Millions of Dollars)

Predevelopment Expenses	10
Smelter	90
Acid Plant	30
Miscellaneous	20
Total	150

Smelter Operating Costs per Year
(Millions of Dollars)

390 Workers	
Labor & Supervision	6.48
Energy-Electrical	5.21
Operating Supplies	2.02
Maintenance Supplies	1.10
Total	14.81

General Office Operating Costs
(Millions of Dollars per Year)

20 workers	
Labor & Supervision	0.51
Supplies	0.05
Total	0.56

Unit Cost for Mining = $0.63/M.T. Ore & Waste
$1.44/M.T. Ore only
$48.26/M.T. Final Conc.
Milling = $1.85/M.T. Crude Ore
$62. Final Conc.

Combined Unit Cost for mining and milling is $110/M.T. of Final Conc.

Total Unit Cost for mining, milling, and smelting is $600/M.T. White Metal Matte

Unit Cost of smelting = $22/M.T. Conc. or $99/M.T. White Metal Matte

Note: Energy cost for production of Matte = 30% in mining
30% in milling
40% in smelting

The **smallest** economic smelter (Duluth area) = 150,000 Metric Tons **metal/yr.**
Mill capacity = 340,000 Metric Tons **Conc./yr.**
= 930+ Metric Tons Conc./day

Concentrates to smelter = 680,000 Metric tons/yr. = 411 Metric tons metal/day
= 150,000 M.T. metal/yr.
@ 62.4% Cu & 12.9% Ni

Smelter Flow Sheet

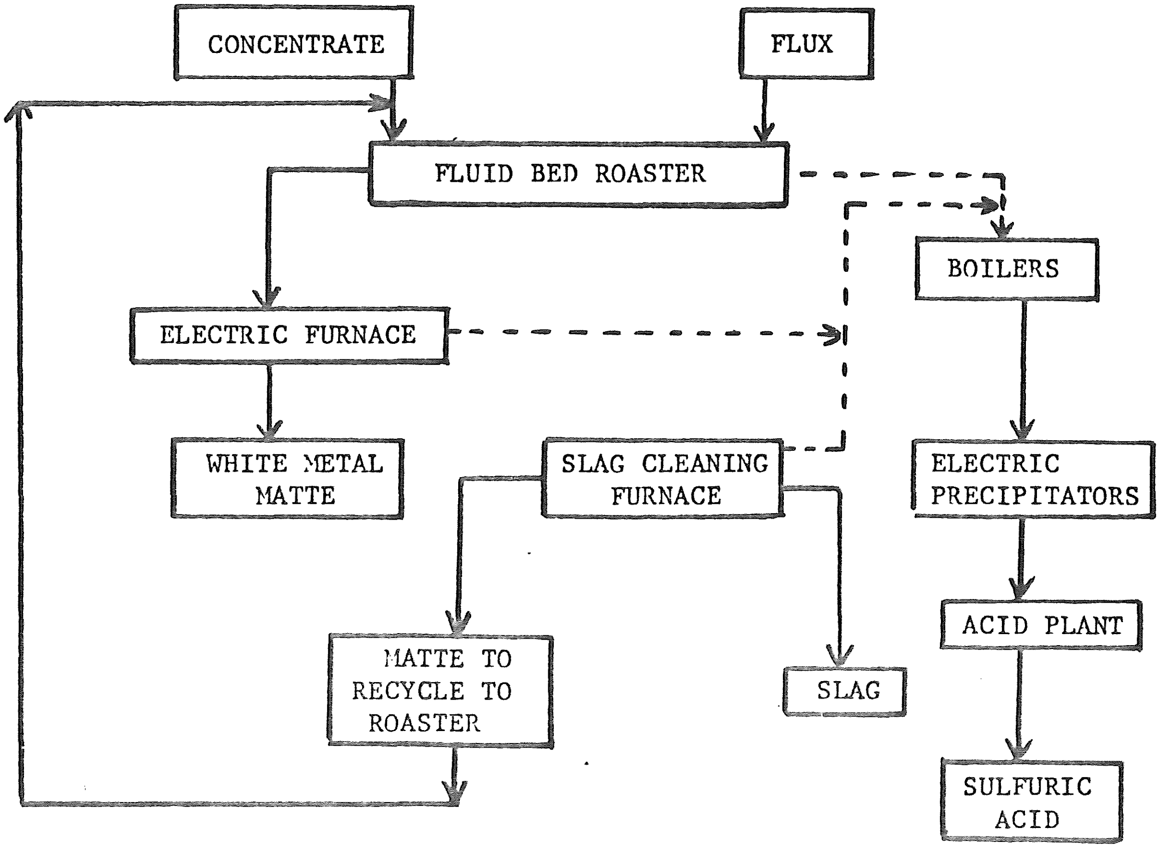


TABLE 8

SIMPLIFIED FINANCIAL ANALYSIS - MINES AND MILLS
(Millions of Dollars)

Capital Investment			
Two Mines			$76.6
Two Mills			187.4
			$264.0
Working Capital			14.8
			278.8
Product Sales **(Per Year)**			
Metal Values			
Copper		$109.7	
Nickel		52.6	$162.3
Expenses **(Per Year)**			
Operating Costs			
Mining	32.6		
Milling	41.9	74.5	
Transportation			
Conc. to Sudbury		14.3	
Royalty		5.4	
State Copper-Nickel and Royalty Tax		0.6	
Depreciation*		17.9	112.7
Net Income Before Taxes **(Per Year)**		49.6	49.6
Depletion Allowance		24.8	
Taxable Income		24.8	
Income Taxes: State @ 12%	3.0		
Federal @ 48%	10.5		13.5
Net Income After Taxes			36.1
Depreciation			17.9
Cash Flow**			$54.0

* Includes $93.6 million in mining equipment replacements in addition to initial capital investment of $264 million, depreciated over 20 year mine life.

**A detailed year by year cash flow evaluation for the two year construction period and 20 year operating life of these facilities indicates a 16.2% discounted cash flow rate of return on this investment.

TABLE 9

SIMPLIFIED FINANCIAL ANALYSIS - MINES, MILLS AND SMELTER
(Millions of Dollars)

Capital Investment			
Two Mines			$ 76.6
Two Mills			187.4
One Smelter			150.0
			$414.0
Working Capital			16.3
			$430.3
Product Sales (Per Year)			
Metal Values			
Copper	$123.8		
Nickel	68.3		
Cobalt	2.0		
Precious Metals	16.9	$211.0	
By-Product Acid		12.8	$223.8
Expenses (Per Year)			
Operating Costs			
General Office	0.5		
Mining	36.6		
Milling	41.9		
Smelting	14.8	89.8	
Transportation			
Conc. to Duluth	3.4		
Matte to Sudbury	3.1		
Slag to Ely	1.5		
Acid to Chicago	6.0	14.0	
Royalty		5.4	
State Copper-Nickel & Royalty Tax		0.6	
Depreciation*		25.4	135.2
Net Income Before Taxes		$88.6	88.6
Depletion Allowance		28.0	
Taxable Income		$60.6	
Income Tax: State @ 12%	7.3		
Federal @ 48%	25.6		32.9
Net Income After Taxes			55.7
Depreciation			25.4
Cash Flow**			$81.1

* Includes $93.6 million in mining equipment replacements in addition to initial capital investment of $414 million, depreciated over 20 year mine life.

**A detailed year by year cash flow evaluation for the two year construction period and twenty year life of these facilities indicates a 16.8% discounted cash flow rate of return on this investment.

Metal Prices

Metal	Price Used	Latest Price	Unit	Date	Metal	Price Used	Latest Price	Unit	Date
Copper	$0.65/ 0.75	$0.74	/lb	7/2/76	Gold	$175	$112.50	/TroyOz.	7/30/76
Nickel	$2.00	$2.20	/lb	6/30/76	Platinum	$175	$165-175	/ " "	7/27/76
Cobalt	$3.00	$4.45	/lb	6/30/76	Palladium	$100	$45-50	/ " "	5/20/76
Silver	$5.00	$4.53	/Troy oz.	7/30/76	Rhodium	$175	$300-310	/ " "	10/10/75

Copper Value in Concentrates ($/lb) = 0.95 [Refined Cu Value ($/lb)-0.20]

Nickel Value in Concentrates ($/lb) = 0.90 [Refined Ni Value ($/lb)-0.70]

Depreciation:

Mine & Mill	**-10 years**
Smelter	**-14 years**

Depreciation of capital costs assumed to be 95%-5% remaining in salvage value.

Depletion Allowance:

1. **Cost Depletion: -A portion of the mineral acquisition costs, prorated on the basis of annual production vs. proven reserves.**
2. **Percentage Depletion: -22% for Nickel; 15% for Copper-of gross income from the property during the tax year, but cannot exceed 50% of the taxable income**

Royalty: $1.00 per acre per year through 1976

$5.00 per acre per year 1977 to production/or through 1986

4.8% of finished Copper & Nickel Metal Value - 3 through 1986

5.3% of finished Copper & Nickel Metal Value - 3 through 1996

5.8% of finished Copper & Nickel Metal Value - 3 through 2006

State Taxes:

Copper-Nickel Tax: Base Rate $0.025/ton crude sent to concentrator with 10% escalation for each 0.1% Cu-Ni content exceeding base rate of 1.0%.

Royalty Tax: -1% of total mineral royalties is paid to the State.

Occupation Tax: -1% of value of concentrate produced less certain deductions, (2/3 of this tax is set aside if converted to semi-finished or refined metal in Minnesota.)

State Corporate Income Tax: 12% on income allowing corporate deductions of: Operating Costs; Transportation Costs; Interest; Royalties; Depreciation; and depletion allowance; also deduction of the Occupation Tax.

Federal Income Tax:

48% of adjusted income after deductions; Operating Costs, Transportation, Depreciation, Interest, Royalty, depletion and State Taxes.

Appendix B

DRILL HOLE DATA

D.H. No.

PD II-2 - SE-SW Sec. 16 T.57NR.14W. El Col. ± 1550 depth-1702' @45°
Contact @ ± 1200 ft.
Depth 900'-910' - El. 650-640 = 0.25 Cu; 0.040 Ni; 0.002 Ag; 0.005 Mo;<0.002 Co
= 0.144 Combined*

PD II-3 - NE-NE Sec. 16 T.57NR.14W El. Col. ± 1540 depth 510' @45°
Contact @ not known
Depth 110'-150' - El. 1430-1390 = 0.213 Cu; 0.089 Ni = 0.475 Combined

PD II-4 - SE-NE Sec. 16 T57NR.14W El. Col. ± 1527 depth 623' @45°
Depth 105-623 - 518 ft. Trace Calcopyrite, disseminated El. 1422-904

PD II-5 - SE-SW Sec. 16 T.57NR.14W El Col. = ± 1550 depth 593' @45°
Depth 100'-110'; El. 1450-1440 - 0.27 Cu; 0.08 Ni = 0.507 Combined

PD II-6 - NE-SE Sec. 16 T.57NR.14W El. Col. =± 1530 depth 1323' @45°
Depth 100-110; El. 1430-1420 - 0.27 Cu; 0.08 Ni = 0.507 Combined
220-230; El. 1310-1300 = 0.82 Cu; 0.37 Ni - 1.919 Combined
80-130; El. 1450-1400 - 0.139 Cu; 0.060 Ni = 0.318 Combined
Contact ± 1323

BA-2 - NW-NW Sec. 34 T.60NR.12W El. Col. =± 1560 depth 3574
Depth 1793-2080-El. (233)-(520)** = 0.25 Cu; 0.10 Ni = 0.547 Combined
2580-2602-El. (1020)-(1642) = 0.305 Cu; 0.10 Ni = 0.602 Combined
2782-2894-El. (1222)-(1334) = 0.542 (Company Report Combined Cu-Ni)
3093-3521-El. (1533)-(1961) = 0.239 Cu; 0.072 Ni = 0.453
Contact =± 3521

G8161 - SE-SW Sec. 10 T.60NR.12W El. Col. 1550' depth 353
Depth 183-195 = El = 1367-1355 = 0.20 Cu; 0.08 Ni = 0.438 Combined

G8162 - SE-NE Sec. 10 T.60NR.12W El. Col. ± 1550 depth 250
Depth 225-250 = El. = 1325-1300 = 0.139 Cu; 0.045 Ni = 0.274 Combined

D.H. No.

59029 - NE-NE Sec. 36 T.60NR.13W El. Col. ± 1600 depth 6301
Depth 47-185 = Gabbro-No Analysis-El. 1558-1415

A4-14 - SW-NE Sec. 20 T.59NR.13W El. Col. ± 1550 depth 3463'
Depth 1710-1800-El. (160)-(250) = 0.238 Cu; 0.071 Ni = 0.452 Combined
2316-2317- " (766)-(767) = 0.26 Cu; 0.10 Ni = 0.557 "
2431-2444- " (881)-(894) = 0.283 Cu; 0.091 Ni = 0.553 "
2725-2755- " (1175)-(1205) = 0.51 Cu; 0.20 Ni = 1.104 "
2885-2915- " (1335)-(1365) = 0.35 Cu; 0.16 Ni = 0.825 "
2325-3397- " (1775)-(1847) = 0.283 Cu; 0.078 Ni = 0.515 "
2725-2915- " (1175)-(1365) = 0.135 Cu; 0.057 Ni = 0.304 "
Contact ± 3405 El. (1885) I.F. = 3452 El. (1902)

A4-11 - NE-NW Sec. 16 T.59NR.13W El. Col. ± 1500 depth 2179'
Depth 251-335-El. 1249-1165-0.264 Cu; 0.064 Ni = 0.454 Combined
1335-1405-El. 165-95-0.426 Cu; 0.134 Ni - 0.825 "
1515-1525-El. (15)-(25)-0.37 Cu; 0.09 Ni = 0.637 "
1335-1525-El. 165-(25)-0.176 Cu; 0.054 Ni = 0.337 "
1804-1825-El. (305)-(325)-0.32 Cu; 0.07 Ni = 0.528 "
Contact ± 2180 - El. (680)

B-3 - NW-NW Sec. 36 T.59NR.14W El. Col. ± 1550 depth=2477'
Depth 1010-1087-El. 540-463 - 0.326 Cu; 0.117 Ni = 0.673 Combined
Contact ± 1087 El. 463

A4-12 - SE-NE Sec. 25 T.59NR.14W El. Co. ± 1500 depth-2655
Depth 1026-1079 - El. 474-424 - 0.298 Cu; 0.096 Ni = 0.583 Combined
Contact 1414 El. ± 86 I.F. 2629 ± (1129)

No. 9 - NE-NE Sec. 36 T.61NR.12W El. Co. ± 1440 depth 2696'
Depth 2670-2694 - El. ± (1230)-(1254)-0.271 Cu; 0.016 Co; 0.098 Ni = 0.563 Combined
Contact ± 2696 + El. ± (1256)

D.H. No.

No. 10 - SW-SW Sec. 36 T.61NR.12W El. Col. ± 1450 depth 2225'

Depth 1990-2000 - El. (540)-(550) - 0.32 Cu; 0.01 Co; 0.10 Ni = 0.617 Combined

2080-2130 - El. (630)-(680) - 0.208 Cu; 0.014 Co; 0.148 Ni = 0.647 Combined

2133-2180 - El. (683)-(730) - 0.324 Cu; Tr. Co; 0.089 Ni = 0.587 Combined

1990-2180 - El. (540)-(730) - 0.152 Cu; Tr. Co; 0.066 Ni - 0.348 Combined

Contact 2160 El. (710)

No. 11 - NW-NW Sec. 36 T.61NR.12W El. Col. ± 1470 depth 11751

Depth 610-650 - El. 860-820 - 0.543 Cu; 0.235 Ni = 1.241 Combined

800-810 - El. 670-660 - 0.22 Cu; 0.08 Ni = 0.458 Combined

970-980 - El. 500-490 - 0.22 Cu; 0.09 Ni = 0.487 Combined

1000-1110 - El. 470-360 - 0.231 Cu; 0.089 Ni = 0.495 Combined

970-1110 - El. 500-360 - 0.197 Cu; 0.076 Ni = 0.422 Combined

Contact 1110 El. ± 360

* Cu-Ni Equivalent = Assay for Copper + 2.97 times assay for Nickel at $2.20/lb. the price for Nickel is 2.97 times the price for Copper (74¢/lb).

**Numbers in parenthesis are below sea level elevations.

Section C

Secondary Metallurgy

SECONDARY METALLURGY

Ore-dressing is but one of the many vertical stages in the process of isolating metal from its host rock, that is, concentrating or enriching the mineral portion of the ore. Further treatment, however, is required to eliminate the remainder of the gangue and to break down the mineral into its component elements for the freeing of the metal. Just as different minerals require special methods to effect their concentration, so the extraction of various metals requires a diversity of treatment. The extractive processes can be classified under three general headings:

1. Pyrometallurgy, in which the reaction is carried out by the application of heat supplied by the burning of fuel.
2. Hydrometallurgy, in which a solvent is employed to leach out the metal from the ore.
3. Electrometallurgy, in which electrical energy is utilized either for its heating value to effect the decomposition of the material, or for electrolytic dissociation of the metal in either aqueous solution or a salt bath.

Pyrometallurgy

Fire treatment usually plays an important role in a least one stage of the various chemical changes through which a material has to pass in becoming suitable for market. Both the quantity of heat as well as the intensity of heat play an important role in metal recovery. Following is a review of the operations of roasting, smelting, and converting as used in the treatment of copper-nickel sulfide ores.

Roasting

Roasting is defined as a metallurgical operation wherein a material is heated to an elevated temperature in a desired atmosphere to create a certain desired chemical change. Stated differently, during roasting, an ore is heated under such conditions and to such a temperature (below that of fusion of its mineral constituents) that the metal components of the ore which is sought for recovery, will be chemically changed from the form in which it occurs in the ore to some other form which will be amenable to some definite subsequent treatment. Roasting processes are generally of the oxidizing, reducing, sulfating, or chloridizing type. Oxidizing roasting involves oxidation of a portion of the sulfur and iron sulfide content of the ore or concentrate to iron and sulfur oxides.[C26]

Typical types of roasters include:

1. mechanical roasters
2. agglomeration roasters
3. fluid roasters

Mechanical Roasting The most widely used mechanical roasters today are of the superimposed multiple-hearth type. The ore is continuously raked by rotating rabble arms from hearth to hearth. During its passage down the furnace, the rabbling action continually exposes fresh surfaces to the oxidizing action of the air. In the case of sulfides, provided sufficient sulfur is initially present, once roasting has commenced, the heat generated by the reaction is sufficient to run the sulfur down to a low content without the need for extraneous heat. The gases leaving the top of the roaster contain from 2% to 6% SO_2 (low for sulfuric acid manufacture) and carry away approximately 6% of the roaster calcine product. Sulfur elimination in the roaster is varied by regulation of air flows and charge retention time on the hearths.[C26] (Figure C1)

Agglomeration Roasting The chain, circular, and Dwight-Lloyd types of traveling grates can be considered as proven industrial machines. Reactions are usually performed with the beds supported horizontally, though in some unusual cases, grates have been mounted on inclines to elevate the discharge end. Wind boxes and, in some instances, hoods used for confining draft that is either induced or formed by blowers, are used as auxiliaries for traveling grates. External igniting or firing is performed by the use of open torches in cases of sintering, by incandescent refractory heat reflectors in cases of coal combustion with autogenous ignition,

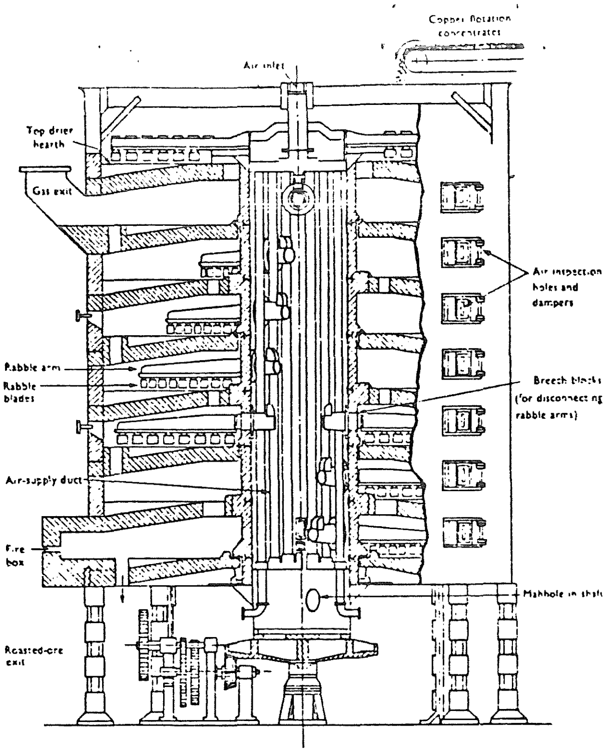


Figure C1 WEDGE MECHANICAL ROASTER

or by elaborate combustion chambers mounted in the hoods of the traveling grates when substantial quantities of heated draft are used. In some processes, recirculation and recuperation of draft are used by compartmenting and connecting the wind boxes and hoods with blower arrangements.

The circular type of grate operates by having the grate carriers under compression and tension induced by a friction or gear-driven mechanism outboard of the ring of grate members. The circular grate revolves in a horizontal plan and offers the unique feature of having grate components in continuous use throughout the processing period. Discharge is accomplished by a tripping mechanism which can be operated by gravity or by a separate gear-tilting mechanism. This type of grate was applied very early in the sintering of ores. For this service, relatively heavy-duty components are required.

The Dwight-Lloyd type of traveling-grate is operated by a tail end drive which pushes pallet cars (containing grate bars and side walls) through the processing zone. Pallets made of castings are arranged as a train of cars under compression.

Individual pallets are removable to allow the insertion of spare pallets. The relatively heavy narrow pallet and grate members serve as a heat sink and consequently allow processing bed temperatures as high as 3000 F. The Dwight-Lloyd machine has been extensively applied for high-temperature downdraft operations in the production of metallic and nonmetallic sinter and pallets. (Figure C2)[C1]

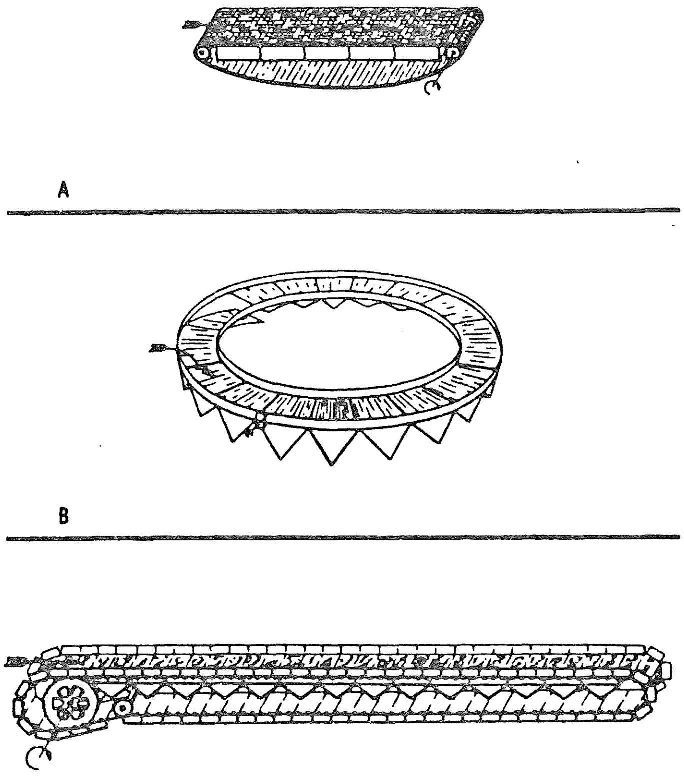


Figure C2 Simplified illustrations of various types of traveling grates: chain or stoker type (A), circular type (B), Dwight-Lloyd (C).

Fluid Roasting Fluid bed roasting is characterized by a gas-solid reaction in a dense suspension of solids maintained in a turbulent mass by the upward flow of gases that affect the reaction. The roaster is essentially a cylindrical refractory-lined steel shell used to contain the suspended solids.

Air is forced into the roaster through tuyeres in a refractory-lined steel construction plate that is placed at the bottom of the shell. Calcine is continuously exhausted from the reactor and recovered in cyclone dust collectors. Once started, combustion is self-sustaining. The heated suspension of solids in the reactor diluted by the air/gas streams acts in many ways like that of a fluid. Material will flow like water into and out of the reactor. It will exert a gravity head which is used to effect movement of the mass. Upon the introduction of heat either in the form of gas or fuel, the fluidized bed, because of its violent motion, quickly reaches a uniform temperature throughout. The temperature of the reactor can be controlled by regulating the volune of combustion air and by the admittance of spray water to the chamber. (Figure C3)[C12]

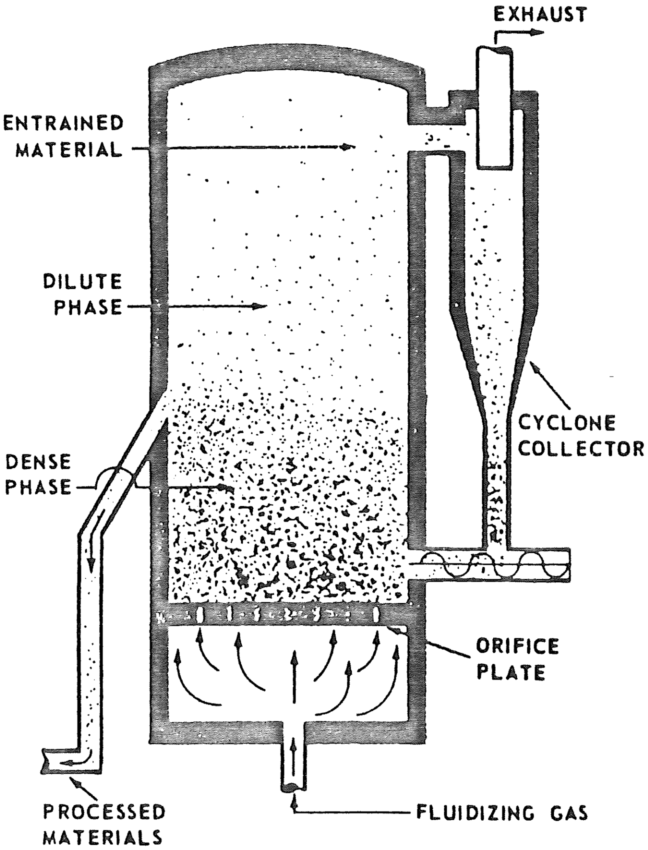


Figure C3

Fluid bed processing is essentially a gas-solid contacting mechanism utilized in materials processing. In minerals processing it has been most widely used in roasting reactions involving nonferrous metal concentrates, although it may be employed in any sort of reaction involving drying, oxidizing, reducing, or agglomeration whereby a gaseous phase must be contacted with particles.

Smelting

Smelting can be regarded as a process for metal extraction involving the reduction of ore to metal or matte, later to be collected by gravitational flow. As the gangue material usually melts at temperatures in excess of that of the metal or matte, it is generally necessary to add fluxes which form with the gangue a slag sufficiently liquid at the furnace temperature. Smelting temperatures are commonly attained in practice by burning coal, coke, pulverized fuel, gas, or by the heat obtained from electrical energy.

Treatable materials in the formation of mattes (artificial sulfide solutions) include:

1. naturally occurring sulfides
2. partially roasted sulfides
3. mixtures of sulfides and oxides

In reference to a predominantly Cu-bearing calcine, two theories exist concerning the chemical reactions taking place during smelting. The older version is based on a neutral or slightly oxidizing atmosphere wherein copper has a greater affinity for sulfur than does iron. Little or no sulfur is essentially lost in the system. All of the copper combines with the available sulfur to give the stable Cu_2S form. The remaining sulfur combines with available iron to give the stable FeS form. The remaining iron and gangue combine as a slag. In the newer version, however, the affinities of both copper and iron for sulfur are considered to be approximately equal at 1300 C. It is believed that the difference in the affinity of oxygen by copper as opposed to iron is responsible for the decisive and relatively clean-cut separation of copper into the matte.

1. $2Cu\ (\ell) + FeS\ (\ell) = Fe\ (\gamma) + Cu_2S\ (\ell)$

$$K = 3$$

2. $Fe\ (\gamma) + Cu_2O\ (\ell) = FeO\ (\ell) + 2Cu\ (\ell)$

$$K = 1300$$

3. $Cu_2O\ (\ell) + FeS\ (\ell) = Cu_2S\ (\ell) + FeO\ (\ell)$

$$K = 3900$$

The above chemical equations, with their corresponding equilibrium constants, indicate iron's greater affinity for available oxygen than that of copper for oxygen. The iron tends to combine with oxygen, whereas the copper tends to remain, in combination with sulfur.

For copper-nickel bearing calcines, smelting occurs in a similar fashion with copper sulfides and nickel sulfides forming the matte.

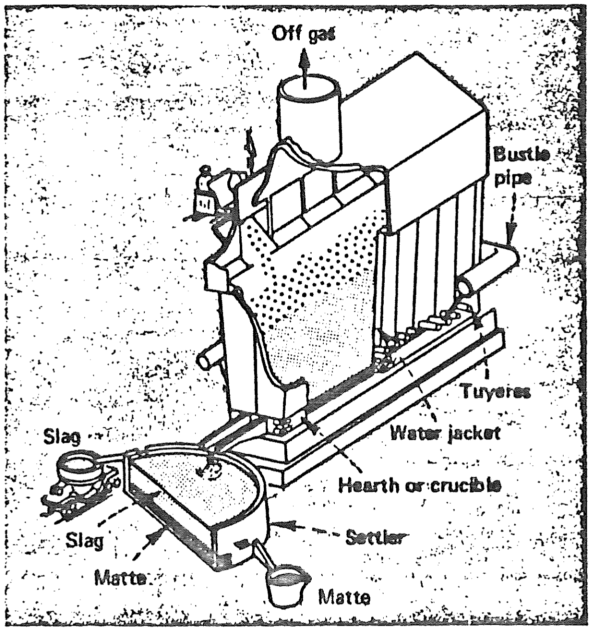
With this in mind, a review of available smelting apparatus is in order. Typical types of smelting equipment include:

Conventional Blast Furnace	Smelting Furnaces
Momoda Blast Furnace	
Reverberatory Furnace	
Electric Furnace	
Outokumpu Furnace	Flash Smelting Furnaces
INCO Furnace	
Noranda Furnace	Continuous Smelting Furnaces
WORCRA Furnace	
Mitsubishi Furnace	
Britcosmaco Furnace	
USBM Furnace	

Processing operations for the conventional smelting furnaces include only the smelting phase. Flash smelting equipment combines both phases of roasting and smelting in a single confined unit. Continuous smelting apparatus, on the other hand, combines the phases of roasting, smelting, and converting (with slag cleaning in some instances) in a single contained unit.

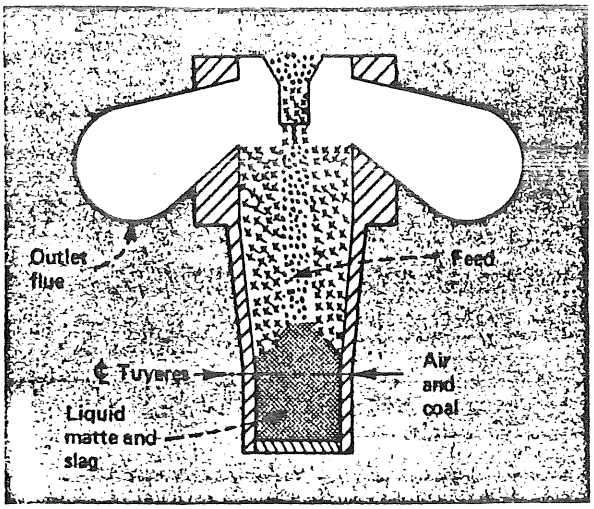
<u>Conventional Blast Furnace</u> The blast furnace consists essentially of a tall vertical hollow shaft, rectangular in cross-section, surmounting a hearth or crucible.

Air required for combustion is supplied by a blower; it passes into the blast main surrounding the furnace and from there to the tuyeres, hence into the furnace. Where the air enters the furnace, very high temperatures are generated by the combustion of fuel, and as the products of combustion ascend, they yield up their heat to the charge. Gases and fumes escape from the top of the furnace through a flue located just below the charging floor. Metal and slag separate according to their specific gravities and are tapped from the crucible. The blast furnace, although simple in construction, is elastic in operation and is capable of high efficiencies. Coarse or sintered ore, however, is required for efficient working. (Figure C4)[C26]



CONVENTIONAL BLAST furnace has walls of water-cooled jackets and is charged by side-dump cars Figure C4

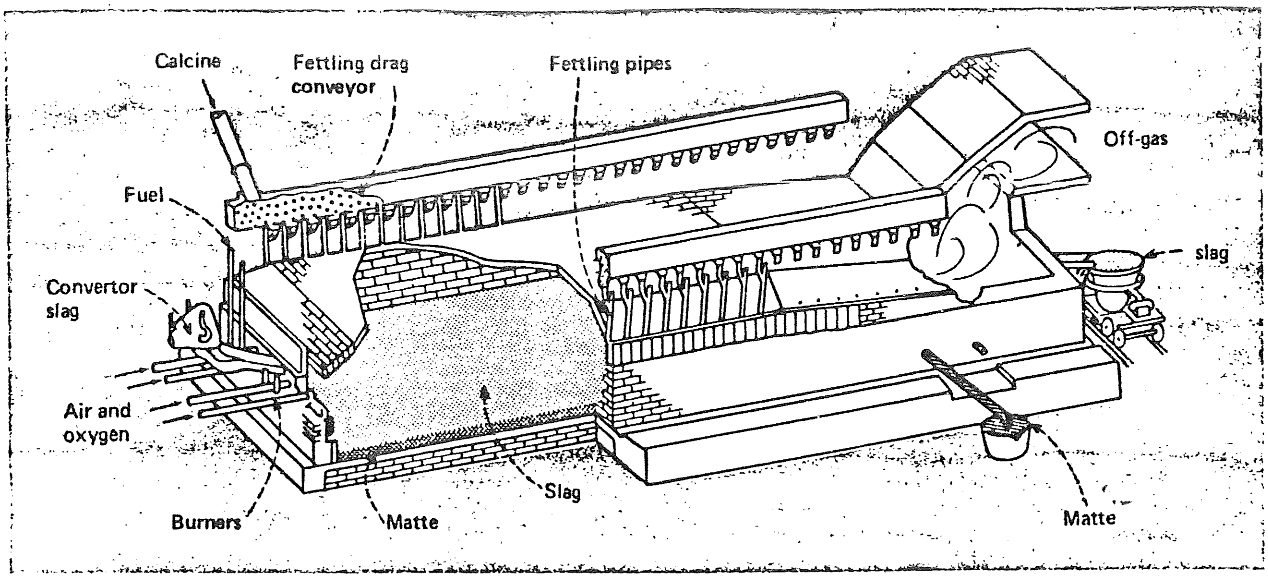
Momoda Blast Furnace The Momoda process involves the feeding of plasticized concentrates directly into a furnace without prior briquetting or sintering. The plasticizing step consists of thorough kneading of fine copper concentrates, flue dust, and other fine material such as cement copper into a stiff plastic mass containing 10-15% water. The furnace is charged alternately at approximately ½ hour intervals through a central hopper opening, first with plasticized concentrates, and then with a coarse mixture of coke, silica flux, limestone, crushed converter slag, and crushed skull material. The plasticized concentrate represents 50-60% of the total charge to the furnace. (Figure C5)



SPECIALLY-DESIGNED hopper feeds Momoda furnace without briquetting or sintering of concentrates Figure C5

Reverberatory Furnace The reverberatory furnace is composed of a shallow hearth with side and end walls surmounted by an arch roof. The hearth is well constructed, being built of concentrate overlaid with silica brick and sintered sand. Side walls are of silica brick with frequently two or more courses of magnesite brick at the level of the slag line. The arch, or roof, is composed of a span of silica or magnesite bricks held in place by iron plates which run along the longer sides of the furnace and are bolted to cast-iron uprights set in the foundations. The plates take up the thrust, the weight resting on the side walls. Horizontal tie-rods across the top of the furnace, bolted to the vertical uprights, serve to hold the furnace rigid. Heating is usually by pulverized fuel or gas injected into the furnace via burners inserted in the end wall. The ore is charged either through holes in the roof or through openings in the side walls. Metal is withdrawn through a tap-hole in the side of the furnace, slag either being tapped from the opposite side or, as in the case of copper, running continuously through a tap-hole in the front end. (Figure C6)[C26]

Electric Furnace Electric furnaces are rectangular in cross-section, with a firebrick sprung-arch roof and a basic-brick inverted-arch bottom. Walls are of



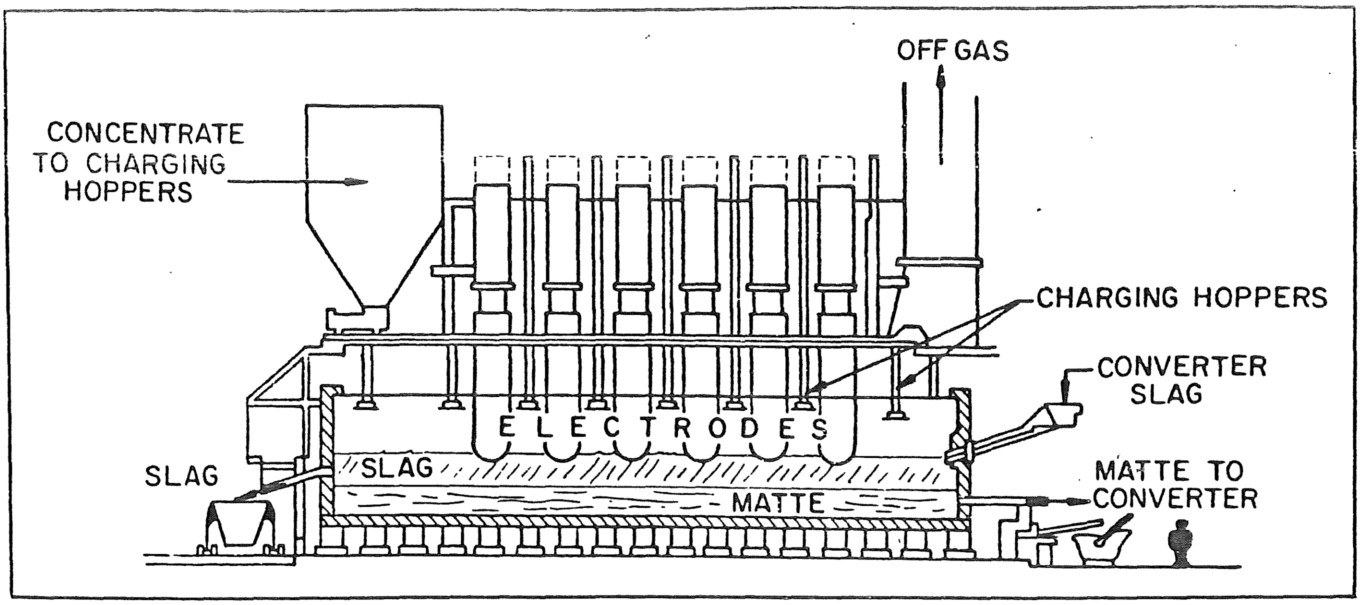
CUTAWAY of Inco reverberatory furnace shows key features of furnace-type common at copper producing installations Figure C6

magnesite basic brick to above the slag line, followed by three or four courses of transitional chrome magnesite brick, followed by firebrick to the roof level.

Matte is tapped from either of two regularly-rotated outer matte taphole is used for the draining of the furnace.

Slag is tapped from one of three regularly-rotated tapholes situated at one end of the furnace. A converter slag-return launder enters above bath level at the opposite end of the furnace.

Power is supplied to the bath by inline Soderberg electrodes, either three, four, or six in number, extending down from the center line of the furnace. Surrounding each immersed electrode is an annular ring of superheated slag, and the free-flowing charge of calcine or dried concentrates is charged by drag chain conveyors to as close to the electrodes as possible. The heat for smelting is generated by electrical resistance in the slag, and correct slag composition (35.5%-38.5% silica) must be maintained for proper slag resistivity. The depth of the slag bath must be raised as the power rating of the furnace is increased, in order to provide the electrical resistance required for high power inputs. Normal slag depths are about 24" for a 6000 kva furnace, 40" for a 30,000 kva furnace, and 50-60" for a 50,000 kva furnace.[C26] (Figure C7)



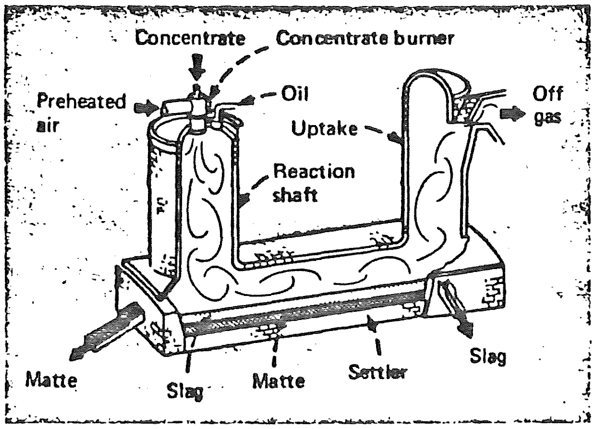
Electric furnaces have been successfully used in areas where hydro-electric power is cheap and fuel is expensive. Figure C7

Flash Furnaces Flash furnaces for copper smelting are of two types:

1. The Outokumpu Furnace
2. The INCO Furnace

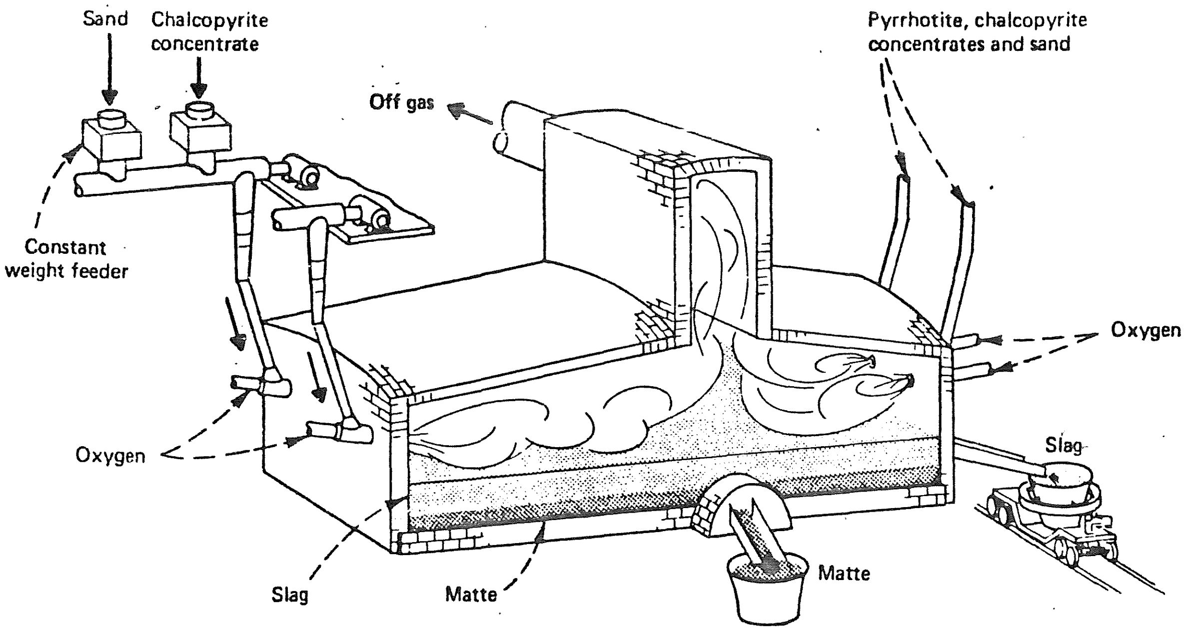
In the flash smelting process, flotation concentrates are injected along with flux and either preheated air or oxygen into a combustion chamber maintained at smelting temperature by "flash" combustion of iron and sulfur while the particles are in suspension. The heat of the exothermic oxidation reactions is used in the smelting, with any heat deficiency augmented by a small addition of fuel, preheating of combustion air, enrichment of the combustion air with oxygen, or a combination of the above.

Outokumpu Furnace Concentrate, dried to less than 0.2% moisture is mixed with heated air in a burner at the top of a reaction shaft. The melted droplets collect in the settler and develop the two normal liquid layers--slag and matte. The off gas has a sulfur dioxide content of approximately 14% and a dust burden of 6-10% of the charge. The matte grade is ordinarily 50-60% in Cu. The slag, which contains too much copper to discard, must be cleaned by settling in an electric furnace, or by casting, slow cooling, and treating by conventional flotation-concentration methods. The high matte grade substantially reduces the amount of converter processing which follows. However, the amount of secondaries which can be smelted as coolant in the converter is limited. The amount of heat available to flash smelting depends upon the iron and sulfur content. (Figure C8)[C26]



OUTOKUMPU FURNACE produces 14% SO_2 offgas, an ideal gas grade for sulfuric acid plants Figure C8

INCO Furnace The INCO Flash furnace uses commercially-pure oxygen in the flash smelting of sulfide concentrates. The concentrates are dried to approximately 0.1% moisture and partially oxidized with the oxygen in burners located in the ends of the furnace. A furnace of modest proportions offers a high throughput. The gas volume is very small, containing approximately 80% SO_2 which can be scrubbed and liquified by compression and cooling. The slag developed in the furnace is low enough in value to discard, but the converter slag is not returned for cleaning. Thus, the slag-cleaning facilities must be included in the smelter design. The oxygen flash furnace is dependent solely upon the oxidation of a portion of the iron and sulfur for the furnace's energy so that its application generally covers the same concentrate grade ranges as the Outokumpu furnace. (Figure C9)[C26]



INCO FLASH FURNACE utilizes 95% oxygen for combustion of concentrates, produces 75% to 80% SO_2 offgas Figure C9

Continuous Furnaces Continuous smelting furnaces include the following:

1. The Noranda Furnace
2. The WORCRA Furnace
3. The Mitsubishi Furnace
4. The Britcosmaco Furnace
5. The USBM Furnace

In continuous copper smelting, each metallurgical stage of roasting, smelting, and converting, with slag cleaning, is accomplished in an individual unit with continuous tapping and transfer of matte and metal.

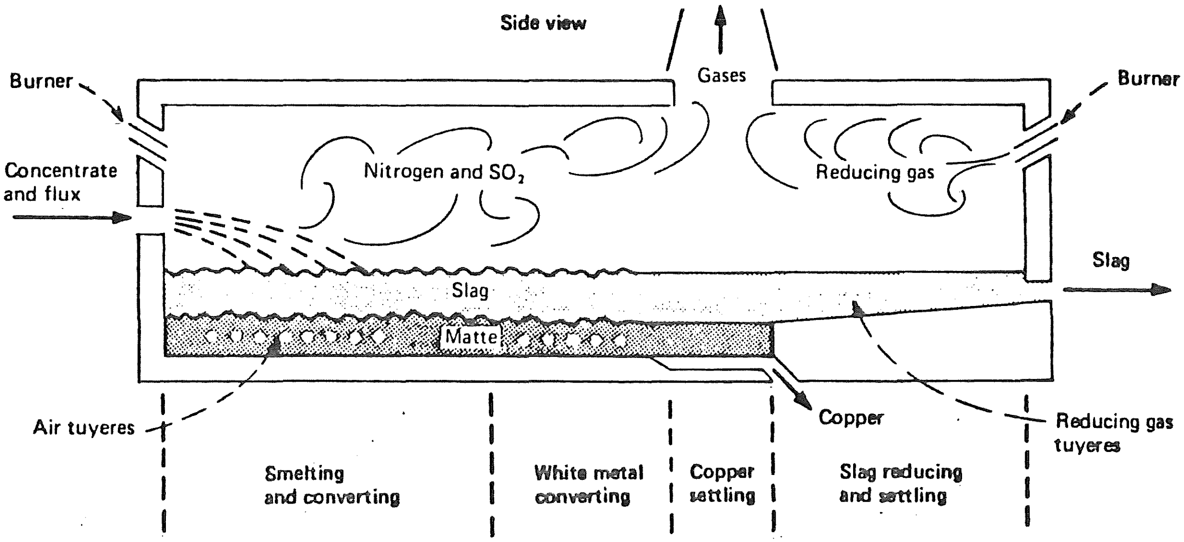
Noranda Furnace The Noranda process is based on two considerationg: by combining smelting and converting in the same furnace, the fuel consumption can be minimized, and dynamic conditions can be created in the bath which will allow the production of copper while charging copper concentrates in the furnace.

Pelletized concentrates and flux are fed through a slinger belt into a cylindrical furnace of the same diameter but twice the length of a large converter. As oxygen-enriched air is introduced through tuyeres, three liquid layers are produced and coexist in the furnace: slag, matte, and copper.

Matte and slag flows are controlled as they move slowly to the tapping ports. Oxidizing gas is introduced into the matte to oxidize the FeS.

Continued injection of the gas into the resulting white metal gradually oxidizes Cu_2S to metallic copper, which is tapped periodically after it separates by settling.

Slag tapped from the reactor is a low-silica, high-copper product which is treated by milling to yield a high-grade concentrate, which is recycled, and a low-copper (about 0.5%) tailing which is discarded. (Figure C10)[C26]



NORANDA PROCESS produces rich off-gases to facilitate SO_2 recovery Figure C10

<u>WORCRA Furnace</u> In the WORCRA process, concentrate and flux are introduced in a mildly oxidizing smelting zone by pneumatic or mechanical injection at an appropriate angle to ensure particle penetration into the liquid and to aid the continuous circulation of matte and slag in the "bowl". Some concentrates may be added in the converter zone closer to the slag exit, where they

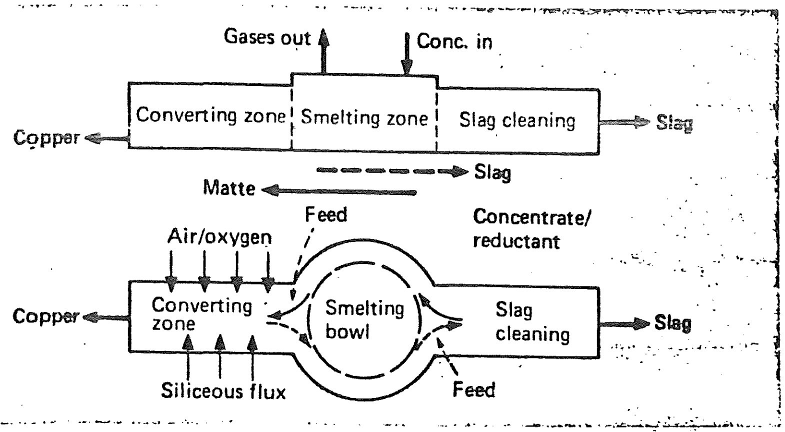
help to control magnetite formation in the slag.

A form of hot solvent-extraction is achieved by forcing the slag to move generally countercurrent to the matte. Unwanted nonvolatile components, particularly iron, are thus continuously being transferred to the slag after oxidation. Conversely, and particularly in the smelting and slag cleanup zones, valuable copper in the slag can be caused to revert to the matte phase by interaction with ferrous sulfide in the matte. In this separate, but connected, slag-cleaning zone, additions of concentrates or pyrites are made to cause both separation or settling of entrained matte, which is continuously returned by gravity to the smelting zone via a sloping hearth.

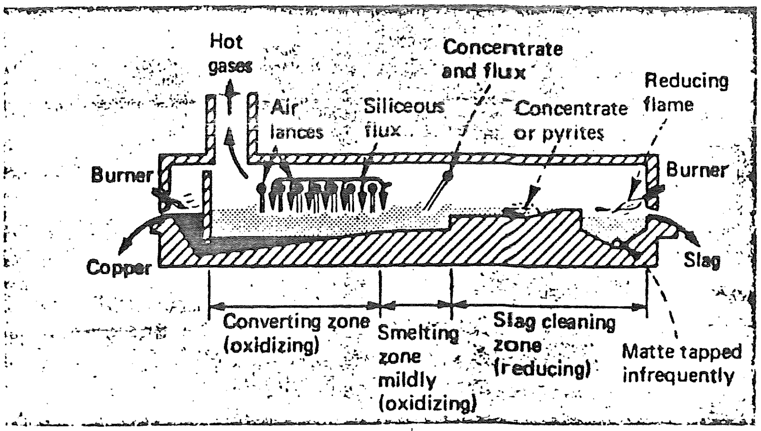
As the matte moves slowly through the smelting and converting zones, it is sequentially lanced with air (or enriched air), causing conversion to white metal and thence to copper. Significant differences in specific gravity of these phases aid in the separations. The hearth in the converting zone continues to slope downward to an underpass through which copper passes continuously to a "copper well" which overflows with the blister copper product.

Furnace gases rich in SO_2 can be treated for waste-heat utilization and dust recovery in conventional equipment prior either to venting or to conversion in a sulfuric acid plant. In a commercial-scale plant, the low SO_2 gases generated by the burners that heat the slag-cleaning zone could be used separately for concentrate drying or combined with the rich smelter gas. (Figures C11 and C12)[C26]

Mitsubishi Furnace The Mitsubishi process is a relatively recent development and utilizes three separate furnaces. In the first furnace, concentrates are dispersed in air jets which impinge against the surface of the melt, and are smelted and partially converted to approximately a 60% copper matte. This matte then flows into a second furnace and is converted by air lances,

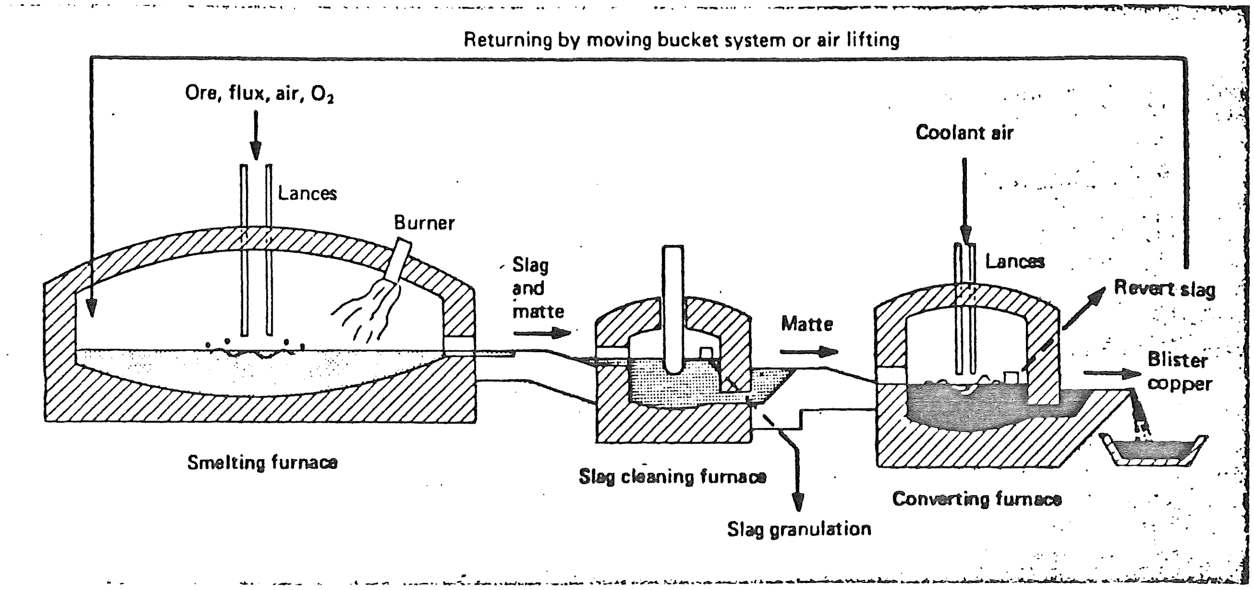


WORCRA PROCESS SCHEMATIC shows concept Figure C11



WORCRA FURNACE unites three operating zones Figure C12

while the slag is treated pyrometallurgically in an electric furnace. The basic elements of the Mitsubishi process are similar to those of the flash smelting process, with the exception that the three furnaces are interconnected and transfer of slag and matte between them is by gravity flow.[C26] (Figure C13)



MITSUBISHI PROCESS depends on material flow between three vessels Figure C13

Britcosmaco Furnace This process is intended to bring together the best features currently available in pyrometallurgical methods for the treatment of sulfide concentrates. It is based on minimizing the use of oxygen to not only oxidize sulfur, but also produce a slag of high enough oxygen potential to allow metallic copper to be made directly in the process.

Dry concentrate and flux are fed into the main smelting shaft with sufficient preheated or oxygen-enriched air to provide for autogenous smelting, producing enriched white-metal and slag phases. These collect on the hearth in two layers.

As the volume of slag increases, it flows along the phase-reaction section of the hearth. Contact with matte and low-grade matte causes more copper to be rejected. As the slag progresses toward the top hole, it is subjected to even greater reducing action immediately under the secondary smelting shaft. In the final stretch between the shaft and the tap hole, fine particles of matte disperse through the slag, reducing oxygen potential even further and causing additional rejection of copper.

Meanwhile, at the other end of the hearth, copper falling down the main smelting shaft is oxidized up to the state of metal, with variations in oxidation level evening out as the particles work their way through the slag. The copper dissolves at the top of the enriched white-metal layer, is precipitated from the bottom as metal, when it is removed via a bottom-tapping xyphon arrangement. (Figure C14)[C26]

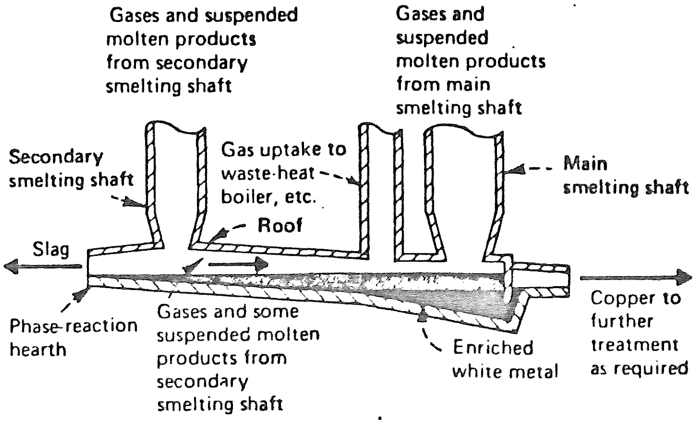


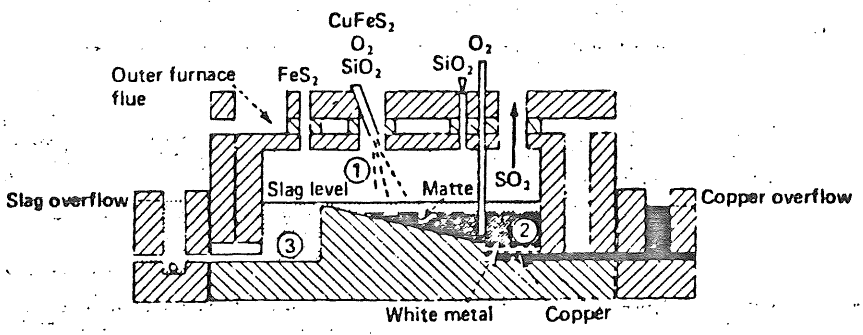
Figure C14 BRITCOSMACO PROCESS needs low energy input

USBM Furnace The USBM process consists of a furnace with a sloping hearth. Sulfide concentrate and flux are blown into the furnace through a burner. Flash smelting produces approximately a 50% copper matte, which flows down the hearth countercurrent to slag flow.

At the converting end of the furnace, oxygen is introduced via a watercooled lance immersed in the matte layer, where conversion to blister copper proceeds. Injection of oxygen into the matte by a lance passing through the slag layer concentrates heat in the matte, where it is needed, and keeps the slag at lower oxygen potential, thus lowering its copper content.

Slag formed in the smelting and converting reactions flow to the slag well, undergoing cleaning enroute by:

1. contact with lower-grade matte
2. the roast reaction of copper sulfide and copper oxide
3. the reduction of copper oxide by sulfur and ferrous sulfide
4. the reduction of magnetite (Figure C15)[C26]



Verticle elevation section through center

Common reactions by zone

① Flash roasting sand smelting

$2CuFeS_2 + O_2 \rightarrow Cu_2S + 2FeS + SO_2$
$2FeS + 3O_2 \rightarrow 2FeO + 2SO_2$
(Fe_2O_3 and Fe_3O_4 also formed)
$3Fe_2O_3 + FeS \rightarrow 7FeO + SO_2$
$3Fe_3O_4 + FeS \rightarrow 10FeO + SO_2$
$2FeO + SiO_2 \rightarrow Fe_2SiO_4$

② Converting

$2FeS + 3O_2 \rightarrow 2FeO + 2SO_2$
$2Cu_2S + 3O_2 \rightarrow 2Cu_2O + 2SO_2$
$Cu_2S + 2Cu_2O \rightarrow 6Cu + SO_2$
$2FeO + SiO_2 \rightarrow Fe_2 \cdot SiO_4$
$6FeO + O_2 \rightarrow 2Fe_3O_4$

③ Slag cleaning

$2Cu_2O + Cu_2S \rightarrow 6Cu + SO_2$
$Cu_2O + FeS \rightarrow Cu_2S + FeO$
$2Cu_2O + S \rightarrow 4Cu + SO_2$
$FeS + 3Fe_3O_4 \rightarrow 10FeO + SO_2$

BUMINES AUTOGENOUS scheme combines multiple processing in single furnace Figure C15

Converting

The process of converting is essentially an oxidation process, no extraneous fuel being utilized. Air for oxidation is admitted a foot or so above tuyeres so when the converter is in the normal operating position, the air is forced by pressure from the blower through the matte.

The conventional-type converter lies in a horizontal plane and is capable of being rotated on its long axis for purposes of charging and emptying. Liquid matte from the smelting phase is charged to the converter together with silica flux. In the case of a copper rich matte, oxidation commences and slag begins to form, the oxidation of the iron producing the heat necessary to maintain the slagging action.

$$2FeS + 3O_2 = 2FeO + 2SO_2 + 224{,}000 \text{ calories}$$

The iron oxide produced combines with the silica flux to form an iron silicate slag which is poured off, more matte and flux added, air readmitted, and the action commences.

$$2FeO + SiO_2 = 2FeO \cdot SiO_2 \text{ (slag)}$$

When sufficient copper sulfide has been accumulated in the converter, this in turn is oxidized, the product being blister copper (98% - 99% Cu), which is transferred to furnaces for casting into anodes prior to electrolytic refining.

$$\begin{array}{l} 2Cu_2S + 3O_2 = 2Cu_2O + 2SO_2 \\ Cu_2S + 2Cu_2O = 6Cu + SO_2 \\ \hline 3Cu_2S + 3O_2 = 6Cu + 3SO_2 \end{array}$$

In the case of a copper-nickel sulfide matte, separation of the non-magnetic copper-nickel sulfides is generally affected by flotation, the copper sulfide being conveyed to the copper smelter, the nickel sulfide being sintered for the production of nickel oxide.

Typical types of converters include:

1. The Pierce-Smith Sideblown Converter
2. The Hoboken Horizontal Siphon-Type Converter

3. The Kennecott Converter-Smelter Vessel

<u>Pierce-Smith Sideblown Converter</u> The Pierce-Smith Converter is an efficient machine whose high air-flows allow both large copper throughputs and the smelting of culky copper-bearing materials and scrap that might otherwise be difficult to dispose of. Outlet gases are generally low in SO_2 concentration, a consequence of excessive air infiltration into the off-take hood over the converter mouth. (Figure C16)[C26]

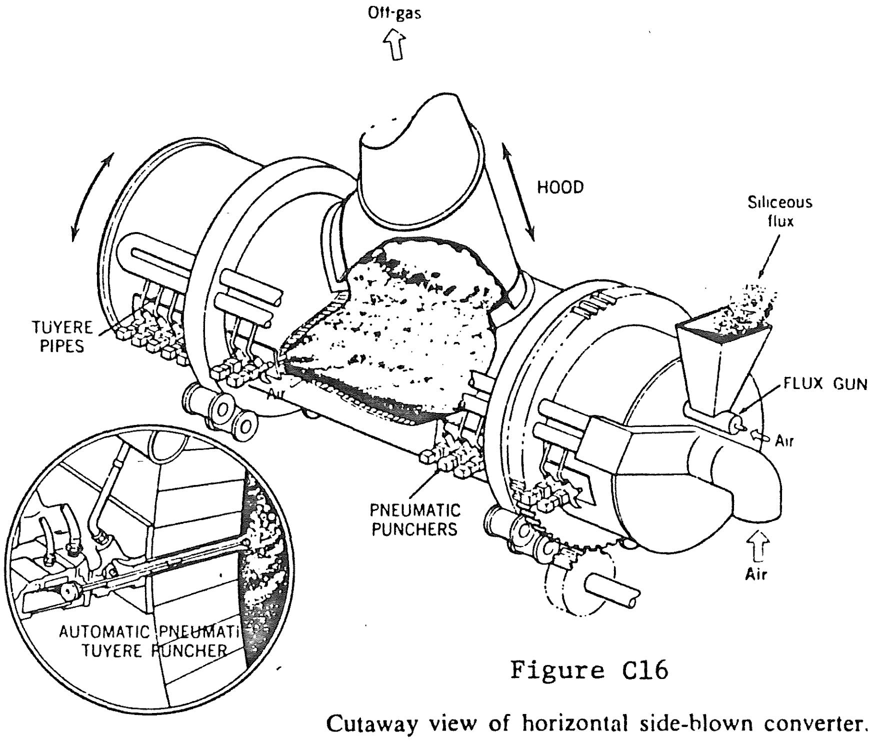
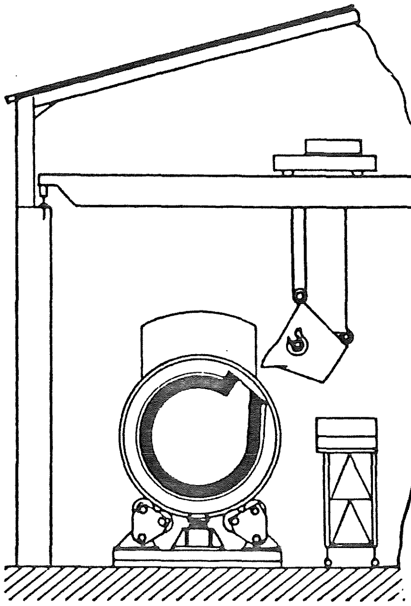


Figure C16

Cutaway view of horizontal side-blown converter.

<u>Hoboken Horizontal Siphon-Type Converter</u> In this unit, reaction gas is taken off the vessel constantly through one end, and matte charged to the converter through the small pouring-mouth while the converter is in the upright blowing position. Dilution of the exhaust gas is minimized by controlling the converting draft to give zero suction at the converting mouth.

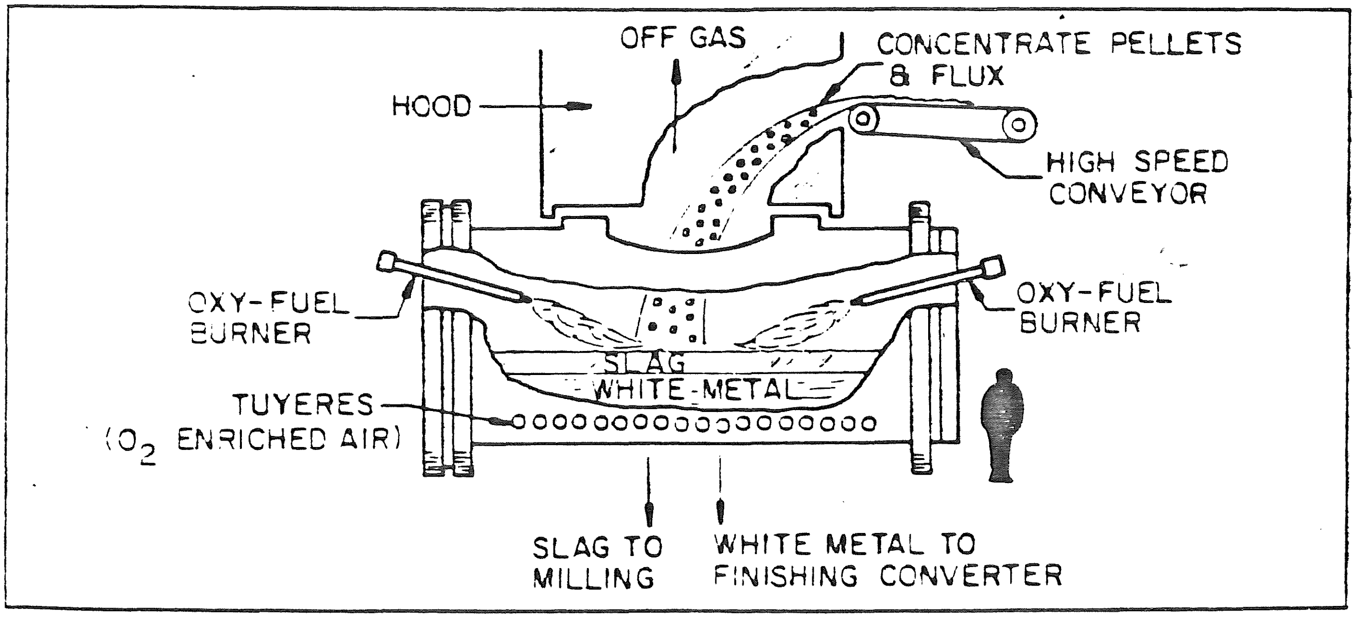
When the converter is being poured or skimmed of slag, blowing air is discontinued and the weak off-gases are vented to an escape stack rather than to the acid plant. Significantly, an average of 8% SO_2 gas is consistently obtained during converting operations, and if three or more Hoboken converters are operating on a continuous basis, it should be possible to supply an acid plant with a mixed exhaust gas containing all of the converter off-gases while avoiding excessive fluctuations in gas volume and SO_2 concentration. (Figure C17)[C26]



HOBOKEN CONVERTER can improve sulfur recovery but lacks Peirce-Smith unit's smelting efficiency Figure C17

Kennecott Converter-Smelter Vessel This unit is based on the modification of existing converters to allow them to double as smelting and converting vessels. Pelletized concentrates and flux are fed through the mouth of the converter and are smelted and converted to high-grade matte through the use of oxy-fuel burners and oxygen-enriched tuyere air.

In the KCS process, a high bath of high-grade matte is maintained in the vessel at all times; slgg is skimmed periodically and treated by milling; and at hourly intervals, excess high-grade matte is transferred to a finishing converter for blowing to copper. The process may provide a low-capital alternative for existing smelters which for some reason do not want to continue operating their reverberatories. (Figure C18)[C25]



Kennecott's converter process offers an attractive low-cost alternative for existing smelters. Figure C18

Refer to table C1 for a statistical review of some of the features of each piece of pyrometallurgical equipment and/or apparatus; refer to figure C19 which illustrates in a very generalized combined flowsheet the various choices in current and developing technology offered to smelter designers.

Figure C20 compares and contrasts the quality of metal product produced from the various pyrometallurgical systems. The reverberatory and electrical furnaces produce a matte of composition A, unless they are preceeded by fluidized or hearth roasters, in which case matte is converted slightly to the composition of point B. The flash smelting furnace and also the converting furnace of the Missubishi process produce a matte of approximately 55-60% copper, near point C. In the Noranda process reactor, the matte can be converted all the way to point E (98% Cu), unless there is an electro-refining problem due to the retention of high bismuth or antimony content in the anodes

Table C1

Salient Features of Pyrometallurgical Equipment and Apparatus

APPARATUS	DESCRIPTION & DIMENSIONS	CAPACITY	GAS GRADES	DISADVANTAGES	ADVANTAGES
Multiple Hearth Roaster	Cylindrical-brick-lined vessel		2-6% SO_2	More expensive to operate in comparison to other calcining operations. Calcine Carryover of 6% in outlet gases.	
Fluid Bed Roaster	Cylindrical-refractory-lined steel shell	1600 TPD Concentrate	12-14% SO_2	Processing difficulties. Excessive calcine carryover of 80% in outlet gases	No moving parts in the combustion chamber. Maintenance requirements are at a minimum. Uniform bed temperatures and composition are maintained in the fluid bed. Little excess air is required.
Sintering Machine	Traveling straight line grate 25 ft. - 35 ft. long 5 ft. - 8 ft. wide		1.5-2% SO_2	Requires a very careful adjustment of the physical and chemical conditions of the charge. Sulfur control is not very practical.	Excellent hearth efficiency.
Momoda Blast Furnace		500 TPD charge	7.3% SO_2 9.5% CO_2 3.1% O_2	Lacks capacity for large smelter operations.	Lower BTU requirements per ton of charge when compared with the BTU requirements per ton of wet charge in reverberatory.
Reverberatory Furnace	A long horizontal furnace with a roof designed to reflect flame down onto a charge on the hearth 120-150 ft. long 30-40 ft. wide High temperatures 1400-1700°C Tapping temperature 1100-1300°C Gas temperatures 1100°C	1600 TPD charge	.5-1.0% SO_2 11-20% of sulfur content removed	Thermal efficiency of the reverberatory is low.	Furnace possesses flexibility with respect to feed. Capable of handling large tonnages of hot charge per day.
Electric Furnace	Rectangular in cross-section with a firebrick sprung-arch roof and a basic-brick inverted-arch bottom		2-4% SO_2	High hydrostatis heads attributable to deep layered slags constitute a definite furnace operating hazard. Bath runaways are a major hazard. Copper matte is more difficult to produce than nickel matte.	

Table C1 continued

Flash Furnace (Outokumpu)		1400 TPD concentrate	14% SO_2	Slags are too high in copper content to be discarded. An additional electric furnace is needed to treat both flash furnace slag and converter slag.	Its small physical size supports a large throughput. It is very economical in its use of fuel, the greater portion used for air heating, and therefore, of low quality. The gas produced is a good feedstock for an acid plant. Direct production of elemental sulfur in the flash-smelting emission system is being developed.
Flash Furnace (INCO)			75-80% SO_2	Slags are too high in copper content to be discarded. An additional electric furnace is needed to treat both flash furnace slag and converter slag.	
Pierce-Smith Sideblown Converter	Cylindrical steel vessel lined with basic refractory bricks. 13 ft. diameter; 30 ft. length		2-6% SO_2	Relatively low concentration of SO_2 in outlet gas.	Very efficient as a smelting machine under certain modified conditions.
Hoboken Horizontal Siphon-Type Converter		56.5 TPD blister copper	8% SO_2	Capital costs are high than Pierce-Smith converter. Capacity per unit size is lower than comparable units. Efficiency as a smelting machine is low.	
Mitsubishi Process (Continuous)		500 TPD-1500 TPD concentrates	12-15% SO_2		The system is pollution free, all exit gases have a SO_2 content greater than 10% which will permit recovery of sulfur as sulfuric acid. Capital investment is 70% of that needed to build a conventional smelter. Operating costs are reduced due to a smaller work force.
WORCRA Process (Continuous)			9-12% SO_2		Capital costs 20-30% below that of reverberatory-converter plants. Lower operating costs, with savings dependent upon local costs for fuel, power, and labor. Economic viability at low annual throughputs-possible 10,000 to 20,000 tons per year of copper. Efficient recovery of byproduct sulfur from continuous SO_2 emission. Process makes metal rather than matte directly from concentrates.

Table C1 continued

			Most of the exothermic oxidation reactions are generated and continued within the liquid bath. Injection of oxygen-containing gases via lances creates turbulence and continuous flow in the smelting and converting zones. In the converting zone, slag moves by gravity, generally countercurrent to matte and metal. Copper Content of the slag is reduced to throwaway levels as the slag flows through the smelting and slag-cleaning zones; there is no revert slag. Offgases generated in the smelting and converting stages are combined and leave the furnace continuously. SO_2 concentration is mostly in the 9 to 12% range.
Noranda Process (Continuous)	Horizontal cylindrical furnace with a raised hearth at one end, and a depression near the middle in which the copper collects. 13 ft. diameter; 60 ft. length	800 TPD- 1600 TPD concentrate	
Kennecott Converter-Smelter Vessel		400-500 TPD concentrate	

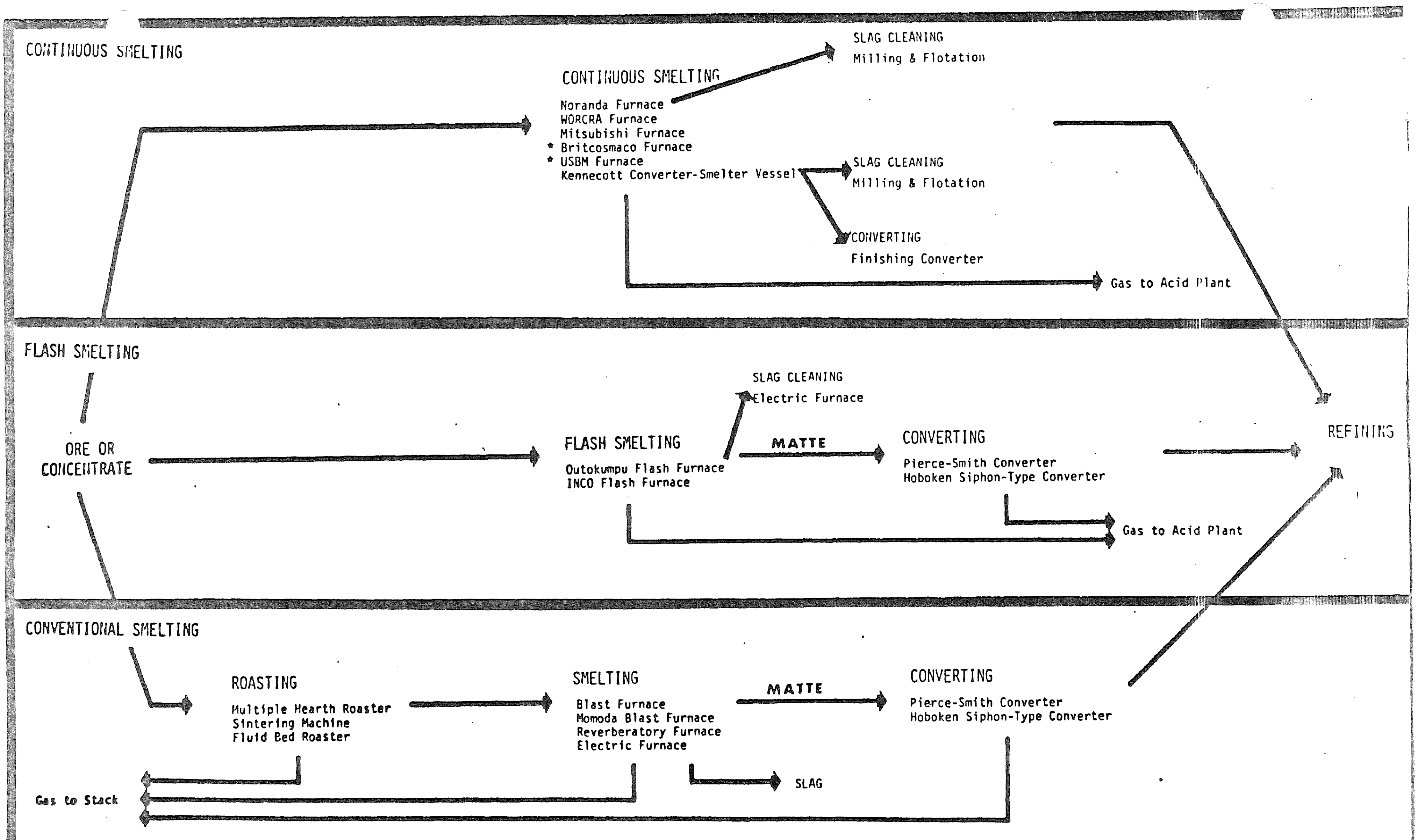


Figure C19 **COMBINED FLOWSHEET** shows variety of choices in current and developing technology offered to smelter designers.

ENERGY REQUIREMENTS AND SULFUR ELIMINATION CHARACTERISTICS OF SEVERAL MATTE SMELTING SYSTEMS

(Approximate values for a concentrate assay: Cu—30%, Fe—28% and S—32%)

Matte smelting system	Smelting energy/ ton conc	Matte grade	Gas eliminated: proportion & concentration Roaster	 Furnace	 Converter
Reverberatory green charge	3.2mm Btu[a]	34%	—	30% @ 2% SO_2	70% @ 6%
Reverberatory calcine from multi hearth roaster	1.8mm Btu[a]	45%	40% @ 5% SO_2	10% @ 2% SO_2	50% @ 6%
Reverberatory calcine from fluid bed reactor	1.8mm Btu[a]	45%	45% @ 14% SO_2	5% @ 2% SO_2	50% @ 6%
Electric furnace calcine	500 KW hr	45%	50% @ 14% SO_2	1% @ 4% SO_2	50% @ 6%
Flash furnace Outokumpu	0[a, b]	48%	—	55% @ 12-14% SO_2	45% @ 6%
Flash furnace oxygen (Inco)	46mm Btu[b, c]	48%	—	55% @ 80% SO_2	45% @ 6%

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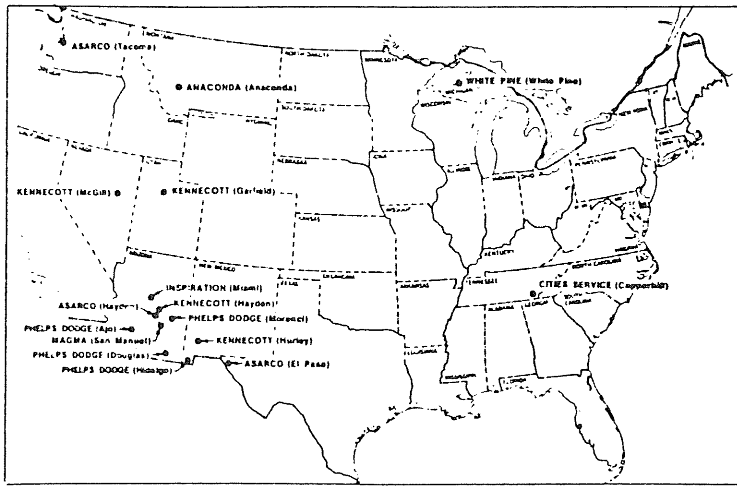
[a] Smelting energy with waste heat steam credit

[b] Energy for concentrate drying included

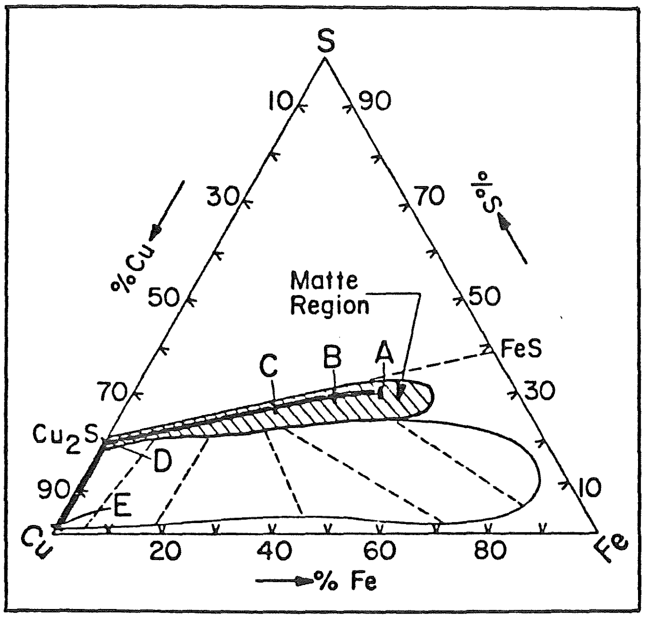
[c] Equivalent of electrical energy used in oxygen plant

Table C2



Location	Company	Capacity, Short tons metal
Ajo, Az.	Phelps Dodge Corp.	70,000
Douglas, Az.	Phelps Dodge Corp.	165,000
Morenci, Az.	Phelps Dodge Corp.	200,000
San Manuel, Az.	Magma Copper Co.	180,000
Hayden, Az.	Kennecott Copper Corp.	80,000
Hayden, Az.	Asarco, Inc.	130,000
Miami, Az.	Inspiration Consolidated Copper Co.	125,000
White Pine, Mi.	White Pine Copper Co.	90,000
Anaconda, Mt.	Anaconda Co.	200,000
McGill, Nv.	Kennecott Copper Corp.	80,000
Hurley, N.M.	Kennecott Corp.	100,000
Copperhill, Tenn.	Cities Service Co.	15,000
El Paso, Tx.	Asarco, Inc.	100,000
Garfield, Ut.	Kennecott Copper Corp.	240,000
Tacoma, Wash.	Asarco, Inc.	110,000
	Total	1,885,000

Primary copper smelters in the United States Figure C21 produced. In this case, the conversion in the reactor proper is stopped at the white metal stage of point D. The Kennecott converter-smelter process also converts concentrates to the white metal stage of point D.[C25]



Copper matte composition diagram. Figure C20

Finally, table C2 summarizes, in tabular form, the degree of sulfur removal as well as the concentration of exit gases for several of the matte smelting systems.

The fifteen operating smelters in the United States vary in annual capacity from 15,000 to 240,000 tons of metal (Figure C21). Thirteen of these smelters use reverberatory furnaces, of which seven roast concentrates before charging, while the others practice green-feed charging. Because of air pollution regulations for smelter dioxide emissions, roasting may be adopted at some of the other smelters to obtain gas streams that are suitable for sulfuric acid production. Other methods to abate air pollution and improve efficiency are being developed.

As a final note, table C3 summarizes some of the new pyrometallurgical facilities that either have gone on stream in the last five years or will go on stream within the next five years.

ENERGY REQUIREMENTS AND SULFUR ELIMINATION CHARACTERISTICS OF SEVERAL MATTE SMELTING SYSTEMS

(Approximate values for a concentrate assay: Cu—30%, Fe—28% and S—32%)

Matte smelting system	Smelting energy/ ton conc	Matte grade	Gas eliminated proportion & concentration		
			Roaster	Furnace	Converter *
Reverberatory green charge	3.2mm Btu[a]	34%	—	30% @ 2% SO_2	70% @ 6%
Reverberatory calcine from multi-hearth roaster	1.8mm Btu[a]	45%	40% @ 5% SO_2	10% @ 2% SO_2	50% @ 6%
Reverberatory calcine from fluid bed reactor	1.8mm Btu[a]	45%	45% @ 14% SO_2	5% @ 2% SO_2	50% @ 6%
Electric furnace calcine	500 KW hr	45%	50% @ 14% SO_2	1% @ 4% SO_2	50% @ 6%
Flash furnace Outokumpu	0[a b]	48%	—	55% @ 12-14% SO_2	45% @ 6%
Flash furnace oxygen (Inco)	46mm Btu[b c]	48%	—	55% @ 80% SO_2	45% @ 6%

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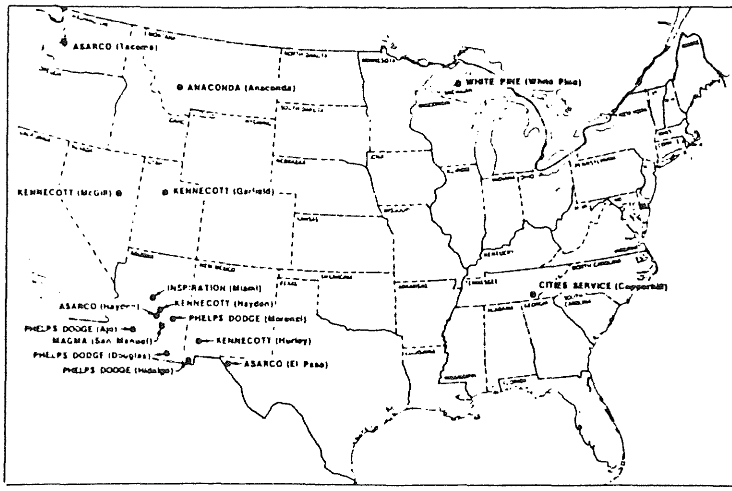
[a] Smelting energy with waste heat steam credit

[b] Energy for concentrate drying included

[c] Equivalent of electrical energy used in oxygen plant

Table C2



Location	Company	Capacity, Short tons metal
Ajo, Az.	Phelps Dodge Corp.	70,000
Douglas, Az.	Phelps Dodge Corp.	165,000
Morenci, Az.	Phelps Dodge Corp.	200,000
San Manuel, Az.	Magma Copper Co.	180,000
Hayden, Az.	Kennecott Copper Corp.	80,000
Hayden, Az.	Asarco, Inc.	130,000
Miami, Az.	Inspiration Consolidated Copper Co.	125,000
White Pine, Mi.	White Pine Copper Co.	90,000
Anaconda, Mt.	Anaconda Co.	200,000
McGill, Nv.	Kennecott Copper Corp.	80,000
Hurley, N.M.	Kennecott Corp.	100,000
Copperhill, Tenn.	Cities Service Co.	15,000
El Paso, Tx.	Asarco, Inc.	100,000
Garfield, Ut.	Kennecott Copper Corp.	240,000
Tacoma, Wash.	Asarco, Inc.	110,000
	Total	1,885,000

Primary copper smelters in the United States Figure C21

Table C3 New Generation of Copper Smelters

Plant Location	Process	Designed Capacity (Tons of Cu/Year)	Start-Up	Comments
Copperhill, Tennessee	Electric Furnace		1972	
Noranda, Quebec	Noranda Continuous Process	50,000	1973	Worlds First Continuous Copper Furnace
Onahama, Japan	Reverberatory Furnace		1973	99.7% fixation of the imput sulfur, ½ as sulfuric acid using a magnesium hydroxide scrubbing process, ½ as gypsum. Ready markets for these two by-products exist.
Inspiration, Arizona	51-MVA Electric Furnace (Dry Charge) with Hoboken Sypon Converters	150,000?	1974	Worlds largest electric copper smelting furnace
Naoshima, Japan	Mitsubishi Continuous Smelter		1974	
Chile	Reverberatory Furnace		1974	Uses auxiliary oxygen-oil roof burners to increase smelting capacity; higher SO_2 strength in outlet gases
Lakeshore, Arizona	Roast-Leach-Electrowin Process		1975	
Butte, Montana	36-MVA Electric Furnace (Calcine Charge) w/38-Ft. Diameter Fluid Bed Roaster	210,000	1975	Worlds largest calcine-charged electric copper smelting furnace
Hidalgo, New Mexico	Autokumpu Flash Furnace with reduction of SO_2 to Elemental Sulfur	100,000	1977	First North American Outokumpu Flash Smelter. First copper flash smelter to employ SO_2 reduction
Garfield, Utah	Noranda Continuous Smelting	298,000	1977	2nd Noranda Process Smelter, 1st in U.S.
Timmins, Ontario	Mitsubishi Continuous Smelting	130,000	1978	2nd Mitsubishi process smelter, 1st in North America
Kamloops, British Columbia	Top Blown Rotary Converter	28,000	--	First commercial use of TBRC for copper smelting

Hydrometallurgy

Wet methods are frequently resorted to in the case of low-grade ores or those ores whose concentration by ore-dressing presents difficulties or is not economic. The essential operation is leaching, the ore being introduced to the solvent (sometimes under pressure) which abstracts the metal leaving the gangue unaffected. After filtration from the gangue, the metal is recovered either by a chemical precipitant or electrolysis. It is essential that the metal be in a form for dissolution and, to effect this, preliminary treatment is often necessary. To ensure that the leaching solution effects its solvent action, the ore must be in a fine state and hence the first step is frequently crushing. The ore having been rendered into a suitable form can then be leached.

Leaching--Generalized Look

Leaching involves the extraction and/or separation of particular minerals, metals, or elements. In hydrometallurgical operations an ore or concentrate is brought into contact with a liquid which is capable of dissolving the valuable metal without attacking or effecting the host or gangue rock.

In order for an ore to be successfully treated by a leaching system, the material to be leached should have the following characteristics:

1. The metal to be recovered should be soluble in the solvent used.
2. It is hoped that the gangue minerals will be insoluble in the leaching solvent.
3. The dissolved metal should be readily recoverable from the pregnant liquor.

Hydrometallurgy offers many features which make it attractive as a means of metal extraction. The following generalizations are readily revealed in contemporary literature which deal specifically with leaching systems.

1. In leaching generally a high rate of extraction is realized. This is important, extremely important in contemporary mining where lower and lower cut-off grades are anticipated and/or expected.

2. Energy requirements are very, very, very low except when preliminary roasting is essential or necessary.

3. The equipment utilized is relatively simple and inexpensive when compared with equipment used in pyrometallurgical operations. (When considering small scale operations only.) The principal expense in our case, is the cost of the chemical reagent used, which incidentally, is a direct function of rock competency, leachability, and certain environmental characteristics.

Leaching is a heterogeneous process involving three vertical stages of development. The first or initial vertical stage involves the diffusion of the solvent through the pores into the interior of the solid particle. The second vertical stage involves the chemical process of solution at the reaction site within the solid particle. The final vertical stage involves the transport of the dissolved substance in the solvent away from the particle surface. Steps one and three are generally considered to be the rate controlling phases, that is to say, the weak links in the chain of the leaching cycle. Therefore, the choice of a particular leaching system depends principally on the chemical and physical characteristics of the material, or;

1. the grade of the ore
2. the solubility of the material
3. the size of particular particles
4. the relative size and intensity of the operations
5. the mode of occurrence of the minerals

Materials suitable for leaching are classified into two broad classes, the first category being the low-grade ores, the second category being other materials, further classified as

a) concentrates

b) calcines

c) matte

d) speiss

e) certain alloys

Solvents Solvents typically used in commercial operations should be:

1. cheap and readily available in large quantities
2. regenerative in nature (regenerated in the precipitation cycle of the leaching system)
3. non-reactive with the gangue minerals.

Specific solvents include:

1. water - natural
   connate
   interstitial
   acid mine type of water
   brine

2. acids - dilute sulfuric acid
   nitric acid (concentrated form)
   hydrochloric acid

3. bases - ammonium hydroxide
   ammonium carbonate

4. salts - sodium cyanide
   potassium cyanide

Mechanics of Leaching Leach solutions are applied to copper-bearing rock under three circumstances in conventional practices:

1. Ore is mined, crushed, and processed in a tank.
2. Unsorted ore is mined, placed in a heap or dump, and leached in-place.
3. Ore is broken or fractured and the copper is leached from the rock in-situ.

The basic principles involved are applicable to all three methods of extraction. The leach solutions must pass downward through the ore and contact and dissolve the copper-bearing minerals. Adequate arrangements are necessary to collect the resultant copper-bearing liquors without appreciable loss. Provision usually must be made to pump the liquors to the surface where the copper can be recovered.

Underground leaching opens wide possibilities for complex extraction from ores. All valuable components in the orebody can be recovered either simultaneously or successively.

Methods of Leaching Physical-chemical and microbiological methods for winning metals and other substances can be utilized in complementary fashion in combination with conventional mining. It is possible to combine the washing out of the ore through boreholes with leaching, to combine underground leaching with drilling and blasting, or underground leaching with microbiological methods.

Following is a review of some of the operations involving the treatment of copper-nickel sulfide ores by a hydrometallurgical process.

Ammonia Leaching - Sulfide Ores The ammonia leach process can be used to extract and recover both nickel and copper, from a copper-nickel sulfide concentrate, ore, or matte. To adapt the ammonia-leaching process for the treatment of copper-nickel sulfide concentrates, it is necessary to include the following steps:

1. Roast the copper-nickel concentrate to remove sulfur.
2. Reduce the calcine with hydrogen to produce metallic nickel (and copper) and Fe_3O_4.
3. Digest the reduced calcine with ammonia-ammonium-carbonate leach solution to overcome the leach-inhibiting action of any metallic iron produced during reduction.
4. Leach the digested calcine with ammonia-ammonium-carbonate solution to extract nickel and copper, leaving the iron oxides as an insoluble residue.
5. Purify the filtered leach solution to remove copper by precipitation with reduced nickel metal.

From this point, the copper and nickel can be recovered and the ammonia leach solution regenerated by any one of a number of accepted methods. Figure C22 identifies the various steps in the recovery of nickel and copper from sulfide ores and concentrates.[C7]

Ammonia Leaching - Copper-Nickel-Cobalt Sulfides An ammonia pressure leaching process for producing high purity copper, nickel, and cobalt from a sulfide concentrate can be used successfully at a cost appreciably lower than if conventional smelting and electrolytic refining were used (small scale operation).

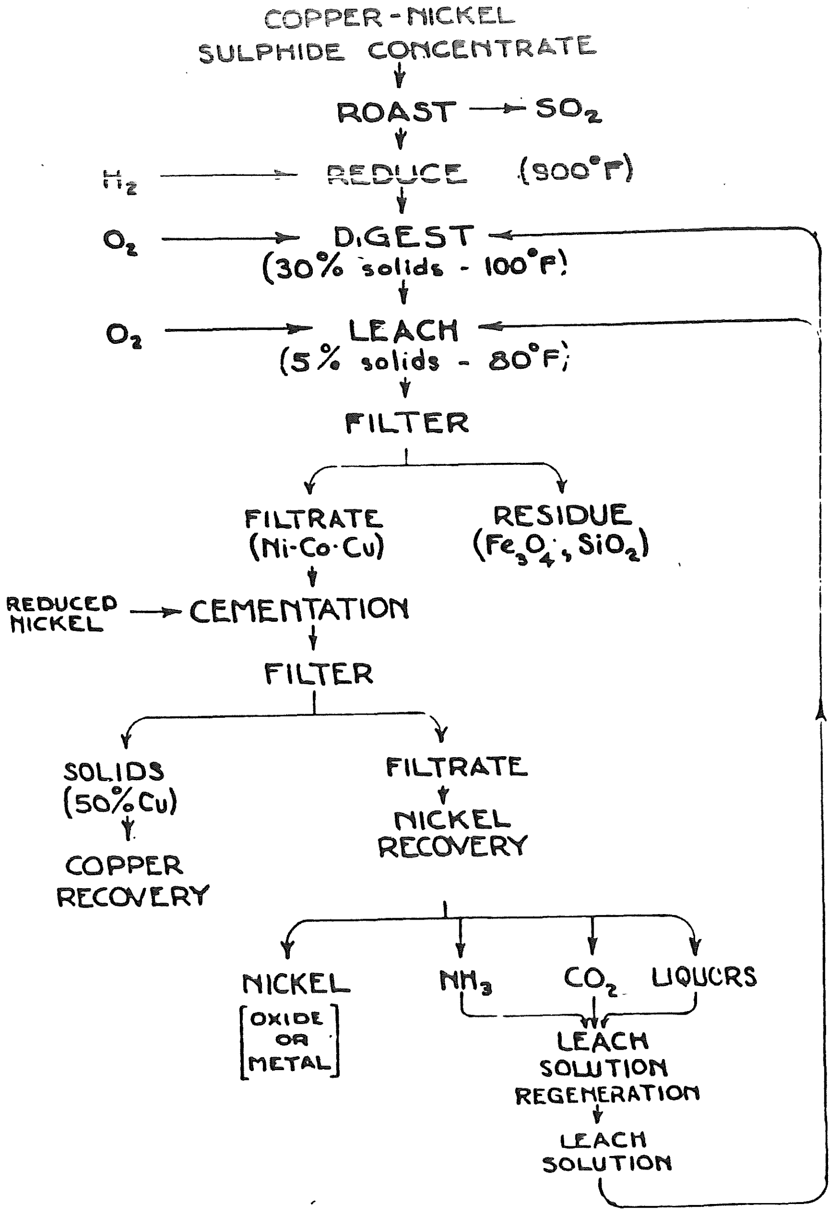


Figure C22 **Proposed flow-sheet for adapting the ammonia-leaching process to the treatment of copper-nickel sulphides.**

The process involves treating the copper-nickel-cobalt-iron sulfide concentrates under pressure with air and ammonia to dissolve the variable metals and most of the sulfur, leaving the iron and other impurities in a tailing which is discarded. The copper is first separated from the solution as sulfide by a boiling operation, the nickel is then precipitated by hydrogen under pressure, the cobalt removed and purified, and the end solutions evaporated to produce ammonium sulfate.

The leaching operation is carried out at relatively low temperatures of the order of 150-220°F under total pressures of less than 125 pounds per square inch.. The copper sulfide is precipitated by bailing the pregnant leach solution at atmospheric pressure. Hydrolysis and hydrogen reduction of nickel are carried out at higher temperatures and pressures.

The total leaching time in continuous operation is less than 24 hours, while the subsequent operations such as boiling, hydrolysis, and metal precipitation are all of short duration, of the order of one-half to two hours each.

Figure C23 shows in outline form the main production and re-cycle streams. Essentially, the leaching operation comprises a reaction between the sulfide minerals, oxygen, ammonia, and water. Although the mechanism has not fully been established, it is recognized that a number of reactions take place in sequence, some being more rapid than others. The principle reactions in a batch operation are presumed to be:

1. Ammonia Leaching -- Ammonia, oxygen, and water react with the sulfide in an autoclave at 125 pai and 150-220 F to yield metal amines, hydrated iron oxide, and ammonium thiosulfate,

   $$2NiS + 8FeS = 14O_2 + 20\ NH_3 + 8\ H_2O = 2Ni(NH_3)_6\ SO_4 + 4Fe_2O_3\ x\ H_2O + 4(NH_4)_2\ S_2O_3$$

   and dissolved oxygen reacts further with the ammonium thiosulfate to produce ammonium sulfate and ammonium trithionate.

   $$2(NH_4)_2\ S_2O_3 + 2O_2 = (NH_4)_2\ S_3O_6 + (NH_4)_2\ SO_4$$

   Further oxidation of the trithionate produces more ammonium sulfate and ammonium sulfamate.

   $$(NH_4)_2\ S_3O_6 + 2O_2 + 4NH_3 + H_2O = NH_4\ SO_3\ NH_2 + 2(NH_4)_2\ SO_4$$

   At the end of the second leaching stage, the hydrated ferric oxide, silica, and insoluble compounds are filtered, washed, and discarded. The two stage continuous counter current leach is carried out in a continuously operated compartmented autoclave. The first stage treats fresh concentrate with recycle liquor from the second stage and the partly leached solids go to the second stage where they are supplied fresh ammonia. Oxygen demand is lower in the second stage and increasing amounts of sulfate and sulfamate are formed.

2. Copper Removel -- Boiling in a continuous still removes excess free ammonia and recomposes thiosulfate and trithionate accordingly.

   $$S_3O_6^{--} + Cu^{++} + 2H_2O = 2SO_4^{--} + CuS + 4H^+$$

$$S_2O_3^{--} + Cu^{++} + H_2O = SO_4^{--} + CuS + 2H^+$$

$$8Cu^{++} + 2S_2O_3^{--} + 4H_2O = 8Cu^+ + S_3O_6^{--} + SO_4^{--} + 8H^+$$

$$2Cu^+ + S_3O_6^{--} + 2H_2O = Cu_2S + 2SO_4 + 4H^+$$

Insufficien sulfide ion is produced to precipitate nickel, and the copper sulfide contains only about 1% nickel while the purified solution contains less than 0.005 grams of copper per liter. While all the copper may be precipitated by the reactions above, it is convenient to remove the last traces of copper with small amounts of H_2S.

3. Hydrolysis -- After removal of copper through the reactions above, remaining thionate and thiosulfate and sulfamate are removed to avoid high sulfur content in the nickel metal product. Oxidation under pressure converts the thionate to sulfate and hydrolyzes the sulfamate.

$$S_2O_3^{--} + 2O_2 + H_2O = 2SO_4^{--} + 2H^+$$

$$S_3O_6^{--} + 2O_2 + 2H_2O = 3SO_4^{--} + 4H^+$$

$$NH_4SO_3NH_2 + H_2O = (NH_4)_2\ SO_4$$

4. Hydrogen Reduction -- As shown by the reactions below, both cobalt and nickel compounds react the same way when treated with hydrogen under pressure.

$$Ni(NH_3)_2\ SO_4 + H_2 = Ni^{\circ} + (NH_4)_2\ SO_4$$

$$Co(NH_3)_2\ SO_4 + H_2 = Co^{\circ} + (NH_4)_2\ SO_4$$

Separate nickel precipitation, leaving cobalt in solution, is possible because the nickel reaction proceeds more readily. To minimize cobalt precipitation due to overlap of reaction conditions, about 1 gram of nickel is left per liter of solution.[C22]

<u>The Arbiter Process</u> The Arbiter process is a hydrometallurgical process capable of treating copper sulfide concentrates. A number of options are available to the operator, depending on the characteristics of the feed, the market for sulfur compounds, and the cost of energy and/or other supplies. The system allows for the treatment of mixed concentrates containing appreciable amounts of zinc, nickel, and cobalt, or lesser amounts of other materials.

The Anaconda plant which hosts the arbiter process is based on a moderate-temperature, low-pressure leach of concentrates in an oxygen-ammonia-ammonium sulfate system. The process is outlined in figure C24. Slurried concentrate is leached with ammonia,

FLOW SHEET OF AMMONIA PRESSURE LEACHING PROCESS FOR TREATING NICKEL-COPPER-COBALT CONCENTRATE

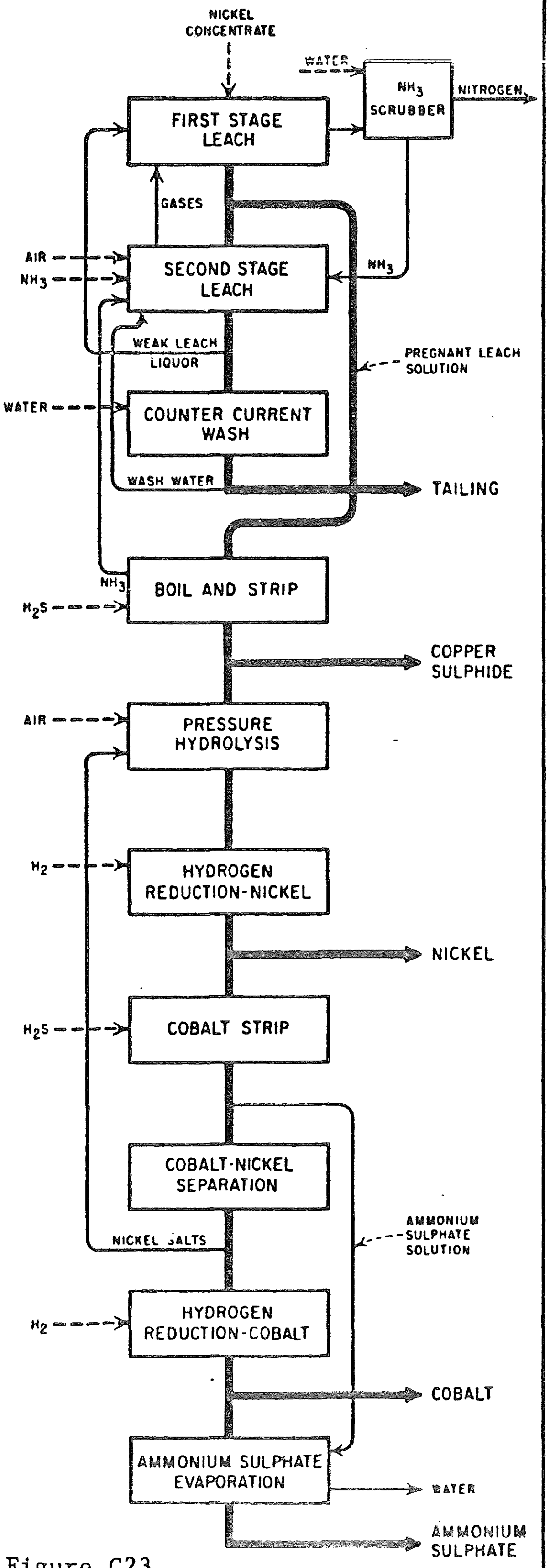
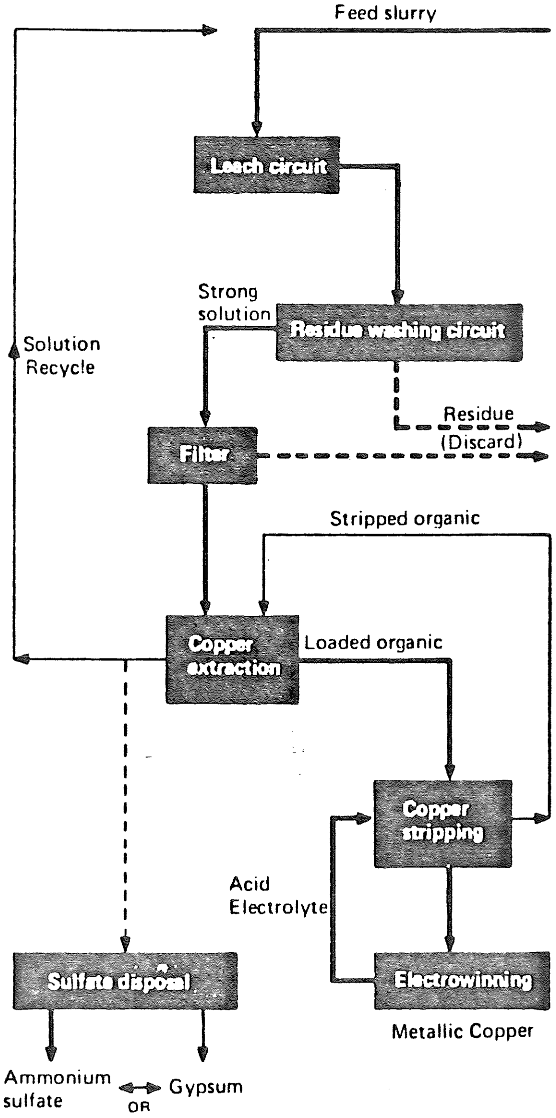


Figure C23

the pregnant solution filtered, the copper then extracted with an organic solvent. The copper is stripped from the organic by means of an electrolyte, and recovered by electrowinning.[C26]



ARBITER PROCESS of Anaconda Copper is hydrometallurgical, using ammonia to leach feeds Figure C24

The Cymet Process The Cymet process is a hydrometallurgical process utilizing electrochemical techniques to put metal values into solution for eventual recovery. The process has been demonstrated in a semi-works plant at a capacity of 50 TPD of chalcopyrite concentrate.

The chemistry of the process (specifically for a chalcopyritic ore) is presented in figure C25 in simplified form. Sulfide concentrates are ground in ball mills to a fineness of at least 95% - 200 mesh, to facilitate contact in the leaching step. Leaching with ferric chloride anolyte solution from the iron electrolytic cells produces the following reaction:

$$6FeCl_3 + 2CuFeS_2 = 1CuCl + 8FeCl_2 + 4S$$

Leached slurry goes to the anode compartment of electrolytic dissolution cells, which are diaphragm-fitted cells with a woven permeable synthetic material that separates anode from cathode compartments. Anodes are titanium coated with conducting oxides.

The chemical reaction at the anode is:

$$CuFeS_2 + 2HCl - 3e = CuCl + FeCl_2 + 2S + 3H^+$$

Slurry discharge from this compartment is recycled to the final leach stage in countercurrent flow.

Copper is recovered from the leach and anolytic solutions by electrowinning in the cathode compartments of the cell. Cathodes are round copper rods, uniformly spaced and oriented parallel to anodes in the opposite compartment. Metals are precipitated as elemental powders, which are removed in slurry and thickened prior to filtration. The chemical reaction is:

$$3CuCl = 3e = 3Cu + 3Cl^-$$

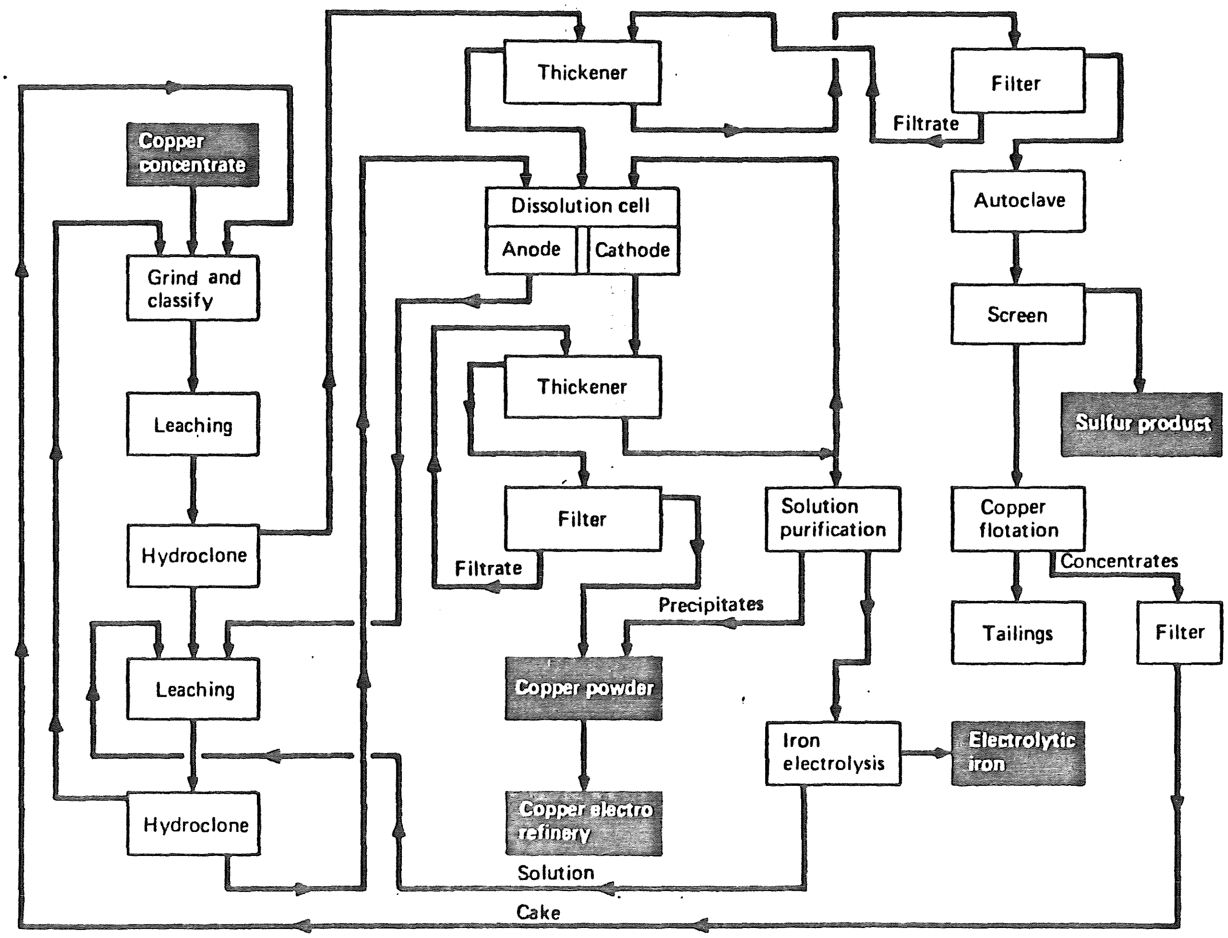
The copper powder eventually goes to electrorefining. (Figure C26)[C10]

The Treadwell Process In this method, copper concentrate is leached in a solution of calcium cyanide and calcium hydroxide. The underflow from the leach cells is washed with water, and the solid tailings rejected. Produdt from the washing step is precipitated and filtered in the presence of copper sulfite, then separated into:

1. a rejected calcium carbonate stream
2. concentrated copper cyanide and copper sulfide that are combined with the rich liquor from the first leach stage

LEACH	$6\ Fe$ [illegible] $+ 3\ CuFeS_2 \rightarrow$ [illegible] $+ 8\ FeCl_2 + 4$ [illegible]
CELL ANODE	$CuFeS_2 + 3\ HCl - 3\ e \rightarrow CuCl + FeCl_2 + 2\ S^0 + 3\ H^+$
CELL CATHODES	$3\ CuCl + 3\ e \rightarrow 3\ Cu^0 + 3\ Cl^-$ $3\ H^+ + 3\ Cl^- \rightarrow 3\ HCl$
IRON CELLS, CATHODES	$3\ FeCl_2 + 6\ e \rightarrow 3\ Fe^0 + 6\ Cl^-$
IRON CELLS, ANODES	$6\ FeCl_2 + 6\ Cl^- - 6\ e \rightarrow 6\ FeCl_3$
SUMMARY	$3\ CuFeS_2\ (\pm)\ 9\ e \rightarrow 3\ Cu^0 + 3\ Fe^0 + 6\ S^0$ Theoretical Current: 573.8 Ampere Hours Per Pound Copper
IRON ALTERNATE	$9\ FeCl_2 + 2.25\ O_2 + (H_2O) \rightarrow 1.5\ Fe_2O_3 \cdot (H_2O) + 6\ FeCl_3$
SUMMARY	$3\ CuFeS_2\ (\pm)\ 3\ e + 2.25\ O_2 \rightarrow 3\ Cu^0 + 1.5\ Fe_2O_3\ (H_2O) + 6\ S^0$ Theoretical Current: 191.3 Ampere Hours per pound of copper Theoretical Oxygen Required: 0.38 pounds per pound copper

Figure C25 *Chemistry of the Cymet process, a hydrometallurgical process employing electrochemical techniques to put metal values into solution and recover these values*



CYMET PROCESS is one frontrunner in hydrometallurgical recovery of copper Figure C26

In the second leaching step, copper values are recovered in contact with calbium sulfhydrate. After precipitation with sulfuric acid, liquid and solids are separated by filtration. Filtrate is neutralized with calcium hydroxide and recycled to the first leach step, while copper sulfide and calcium sulfate solids are separated by flotation. Part of the sulfide is calcined, reacted with SO_2 (generated) to provide the copper sulfite. Balance goes to a reverberatory furnace for conversion to copper.

The CLEAR Process The CLEAR process involves ferric chloride as an agent in leaching. Refer to figure C27 for a flowsheet analysis of the process.[C6]

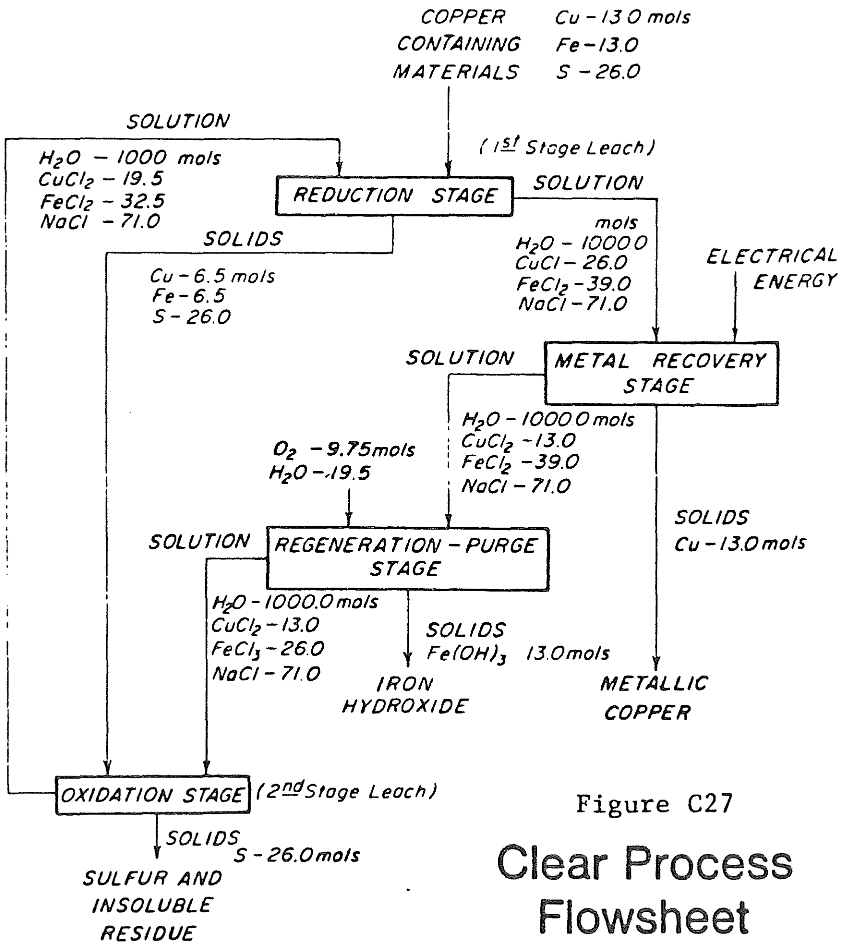


Figure C27

Clear Process Flowsheet

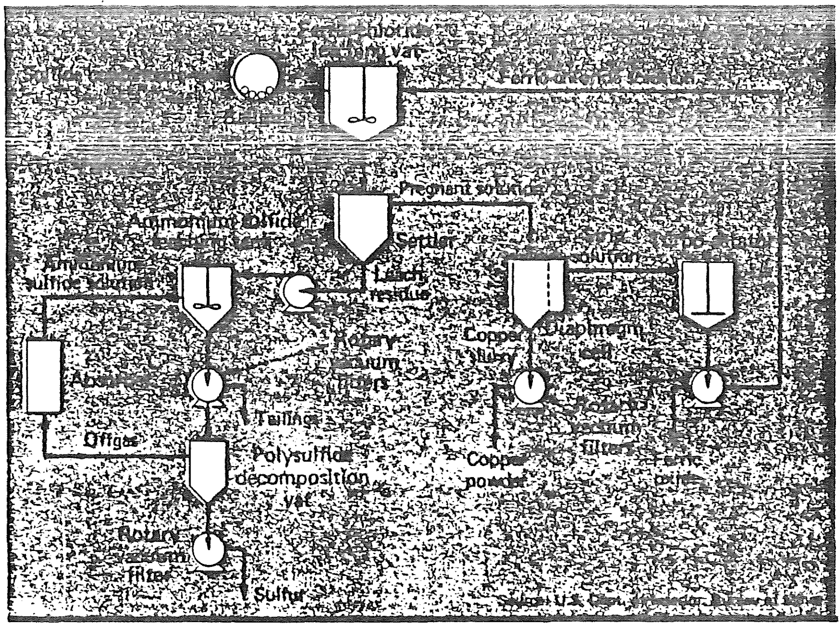
The USBM Revised Ferric-Chloride Process In this process, sulfide concentrate, which is ground to -325 mesh by a ballmill, goes to a leaching tank held at approximately 220°F. A leach solution, containing 36% ferric chloride and 1% hydrochloric acid, dissolves 99% of the copper, 96% of the silver, and 84% of the iron in the concentrate. Slurry from the leaching tank is cooled and sent to a settler.

Overflow solution, approximately 5.5% copper, goes to a diaphragm-type electrolytic cell operating at 105°F. Copper and silver plate onto the cathode, and are dislodged directly into the liquor by periodic tapping. Copper slurry thus formed at the bottom of the cell passes to a rotary vacuum filter which separates the metal from the liquid. The copper is then melted in a furnace to remove impurities, and cast into anodes for further purification.

Liquid from the top of the diaphragm cell is pumped to turboaerators. Oxidation there at approximately 175°F regenerates ferrous chloride to the ferric state, and also yields solid ferric oxide which is removed by a rotary vacuum filter. An evaporator eliminates excess water from the solution, and the hydrochloric acid is added before recycling the stream to the leaching tank.

Underflow from the settler passes to another rotary vacuum filter. Cake from this unit goes to a leaching vat maintained at 80°F, where an aqueous ammonium-sulfide solution dissolves free sulfur to form ammonium polysulfide. Solids then are separated out by a rotary vacuum filter and sent to a tailings dump. Sulfur is precipitated from the liquid by heating to decompose the polysulfide. (Figure C28) C19

The Lurgi-Mitterburg Process In this process, sulfide concentrate goes to a vigration mill. The mill reduces the size of particles plus causes a high degree of distortion in crystal latices, enhancing subsequent leaching. Ground concentrate then is slurried with spent sulfuric-acid electrolyte from a downstream electro-



Revised ferric-chloride process now features much-improved economics Figure C28

winning unit, and pumped into an autoclave. There, at an oxygen partial pressure of 10-20 atm. and a temperature of approximately 240°F, copper values dissolve, forming copper sulfate. Following depressurization and discharge from the autoclave, thickening and filtration remove leach residue, including all the iron (in the form of oxides) and sulfur (in elemental form). Electrowinning in rubber-lined cells then plates the copper out of solution onto cathodes. The spent electrolyte is recycled.

Precious metals in the concentrate remain in the residue from leaching. An extraction scheme has been devised. The metals dissolve readily in hot sodium-chloride solution as long as a certain oxidation potential is maintained by continuously dosing a small quantity of chlorine.[C19]

The Nitric Technique Being developed by DuPont and Kennecott Copper Corporation, the process uses nitric and sulfuric acids in leaching sulfide concentrates. Concentrate is ground to the -325 mesh size, then slurried with a nitric/sulfuric-acid solution, and sent to a two-step leaching system featuring countercurrent flow.

In the first step, preferably taking place at about 230-240 F and 10-15 psig., the acid solution dissolves most of the copper and iron, and produces necessary ferrous ions for a subsequent nitrate-removal step. Then, in the second step, held at the same temperature and pressure levels, a preferably threefold excess of nitric and sulfuric acids accelerates the leaching of the partially treated solids. Operation is controlled so that substantially all the copper goes into solution and remains there along with the iron.

Following separation of leach residue, the pregnant solution passes to an autoclave; heating to approximately 285-390°F drives off nitrate as nitric oxide, which is recycled to a nitric-acid recovery circuit. Next, the solution goes to another autoclave for removal of iron that otherwise would impair the efficiency of electrowinning. Contact with ammonia at 320-390°F precipitates iron from solution as an easily filterable jarosite mineral. Electrowinning of the purified stream then yields product copper.[C19]

Refining

The metal obtained from pyro- and hydro-metallurgical operations generally contains impurities originally present in the ore which have remained with the metal during smelting and/or leaching. Blister copper for example rarely exceeds 99.5% purity and frequently runs as low as 98.5%, the chief impurities being sulfur, iron, oxide, arsenic, and bismuth, all of which tend to have various deleterious effects. Seurral techniques are available to further treat metals for purification and refining. The major techniques are described below.

Fire Refining   Fire refining can be broadly divided into two sections:

1. The metal is liquified in a reverberatory furnace, the impurities being oxidized by the application of air or oxidizing agents and slagged off. Alternatively, chemical reagents such as soda ash or caustic are added which combine with the impurities and are removed by skimming.
2. Distillation is used when the metal concerned is volatile at an economic temperature and capable of condensation.

The essential part of fire refining is carried out by oxidizing the impurities, the oxide slags so formed being removed by skimming. Partial oxidation of the copper also occurs, this being corrected by a subsequent exposure of the metal to a reducing action. The readiness with which such impurities are removed is related both to the heat of formation of the metallic oxides and also to the extent of the solubility of the oxides in molten copper. Sulfur, iron, and zinc are eliminated without undue difficulty, but nickel, selenium, te-lurium, and bismuth are very difficult elements to slag and have usually been dealth with by electrolysis. In practice, it is found that the bulk of impurities are eliminated early in the process, but the removal of the remainder often requires repeated and prolonged slagging.[C5]

Electro-metallurgy   Electrometallurgy involves the application of electric current by two means:

1. electrolysis

2. utilization of the heat liberated by the passage of an electric current either by means of resistance or by arcing.

Electrolysis Electrolysis is concerned not only with the refining of metals, but also plays an integral part in the operational recovery (winning) of many metals, and thus it can be conveniently considered from these two aspects.

In electro-refining, the metal to be refined is made the anode of the system, pure metal deposited at the cathode under electrolytic action. The impurities associated with the metal either remain behind, attached to the anode, or fall off and form a slime at the bottom of the cell.

Reactions are based upon the fact that if two metals are placed in an appropriate solution and a direct current passed from one metal (the anode) to the other (the cathode), the former dissolves and the latter increases in weight. The metal dissolved from the anode acts as a carrier of the current and on reaching the cathode gives up its charge and deposits thereon.

Electro-winning In electro-winning, prior treatment of the ore or concentrate has isolated the metal either in the form of a purified solution or else as a solid chemical compound, such as a chloride or oxide. Thus, a soluble electrode is untenable, an insoluble anode being the rule. With solutions, the insoluble anode may be of lead or a lead alloy, the metal in solution plating out in the usual manner on the cathode. In the case of a solid chemical compound, however, electrolysis takes place in a fused-salt bath, an insoluble anode of graphite or similar compound being used. The liberated metal does not plate out on the cathode, but either floats on the surface of the electrolyte or else sinks to the bottom of the cell.[C5]

In summary, the new generation of smelting equipment captures the energy of the sulfides for autogenous smelting, producing either a high-grade matte in the case with the

Outokumpu flash smelter, or converting the concentrated sulfide minerals directly into a crude metal, as is the case with the newer smelting processes - Noranda, WORCRA, and Mitsubishi. These processes use air, oxygen, or oxygen-enriched air for more efficient combustion, producing a rather strong SO_2 gas which is amenable to conversion into sulfuric acid, reduction to elemental sulfur, or neutralization with lime to produce gypsum.

The development of hydrometallurgical processes as possible alternatives to smelting or roasting has also been spurred in recent years. Some processes leach concentrate directly while others involve special pre-treatments. Some processes dissolve copper selectively, others do not. Most, if not all systems, include a primary leach step. Sulfur is oxidized at least to the element with oxygen from the air directly or indirectly, or electrolytically directly or indirectly, or with both. Where iron dissolves, precipitation at a later stage is required (as a ferric compound). Dissolving and precipitating steps usually utilize the -ic and -ous states that both copper and iron have. Copper can selectively be deposited in the presence of some impurities, while other impurities must be selectively removed from solution. Some systems utilize pressurized atmospheres at elevated temperatures while others operate at or near atmospheric pressure and room temperature.

Distinct advantages of the pyrometallurgical processes are summarized below:

1. Recovery of metal by smelting is very high. In the case of copper, recovery is usually above 98%.
2. All new smelting processes consume little or no energy, a scarce and expensive commodity. All produce energy for internal use.
3. Pyrometallurgical processes do not consume large quantities of water, whereas hydrometallurgical operations can be large consumers of water, another scarce and expensive commodity in many areas.
4. It is believed that pyrometallurgical facilities have larger production per unit than hydrometallurgical facilities.
5. Disposal of slurried residues in hydrometallurgical process plants necessitates the construction of tailings dams. Soluble salts in the residues may find their way into ground waters and contaminate them. This is not the case with waste slags from smelter operations.

Distinct advantages of the hydrometallurgical processes are summarized below:

1. From a sulfur point of view, many systems are essentially pollution-free. Water is processed and reused, all sulfur is captured and recovered in elemental form, acid form, or neutralized to form gypsum.
2. Costs are comparable to the costs of pyrometallurgical operations when considering small scale operations.
3. Greater flexibility in the handling of different feedstocks.

Distinct disadvantages of the hydrometallurgical processes are summarized below:

1. Due to the nature of ores, the rate of dissolution of valuable metals is unsatisfactory. Relatively high capital costs and operating costs exist.
2. Precious metals cannot be recovered efficiently by known hydrometallurgical.
3. Pollution control of aqueous and particulate effluents, and the disposal of leach residues are more difficult and more costly than control of gaseous emissions from modern pyrometallurgical plants.

The new hydrometallurgical processes will have to compete against the energy-improved, pollution-free smelting processes currently being developed as substitutes for conventional reverberatory smelting. Hydro-processes will have to perform at least as well as the new pyro-processes if they are to eventually take the lead.[C17]

Sulfur Dioxide Recovery Systems

The domestic base-metal-producing industry has been challenged with the task of reducing the amount of sulfur oxides and other pollutants emitted by smelters treating sulfide ores and concentrates. This challenge has been imposed by legislation aimed at reducing air pollution. Standards for both ambient air quality and allowable emissions have been proposed and already are in effect in some states.

The principle means for the removal of SO_2 from stack gases in the late 1960's and early 1970's included:

1. the use of tall stacks
2. the use of tall stacks coupled with a contact sulfuric acid plant
3. a closed-loop control system

The use of tall stacks was considered at that time to be the simplest technique. Discharges to the atmosphere at such heights diluted SO_2 values when dispersed into the lower atmosphere. Even if other processes for removal of SO_2 from the smelter gas are adopted, the tall stack will still be an important control method.

The second method combines or couples the tall stack with a contact sulfuric acid plant. Acid plants require at least 3.5-5.0% SO_2 in the feed gas for acid production and cannot economically treat weaker gases. Production of sulfuric acid by the contact process is a well established process.

The third method, that of controlling sulfur dioxide pollution utilizing the "closed-loop control," will be discussed later.

Process selection will be dictated by economic conditions for sulfur byproduct disposal and by variation in the nature of the off gases, such as sulfur dioxide concentrations, impurities carried over from smelting operations, and fluctuations in gas flow rates. Other factors, both economic and noneconomic, such as geographic location, nearness of markets for sulfurous products, available land for product disposal, and potential water pollution problems will influence the choice of method for controlling emissions.[C3]

The Contact Sulfuric Acid Process The contact sulfuric acid process is a well-established chemical process for removing sulfur dioxide from smelter gases.

Smelter gases are first cleaned of particulates, sulfur trioxide removed in a mist cattrell precipitator, the remaining gas stream dried with strong sulfuric acid. The sulfur dioxide in the dry gas is then oxidized to sulfur trioxide in a catalytic converter. The sulfur trioxide is absorbed in strong (98%) sulfuric acid to yield the product of the plant. The tail gas is treated to remove droplets of acid and normally is vented to the atmosphere. Tail gases will contain from 0.2-0.5% SO_2. (Figure C29)[C3]

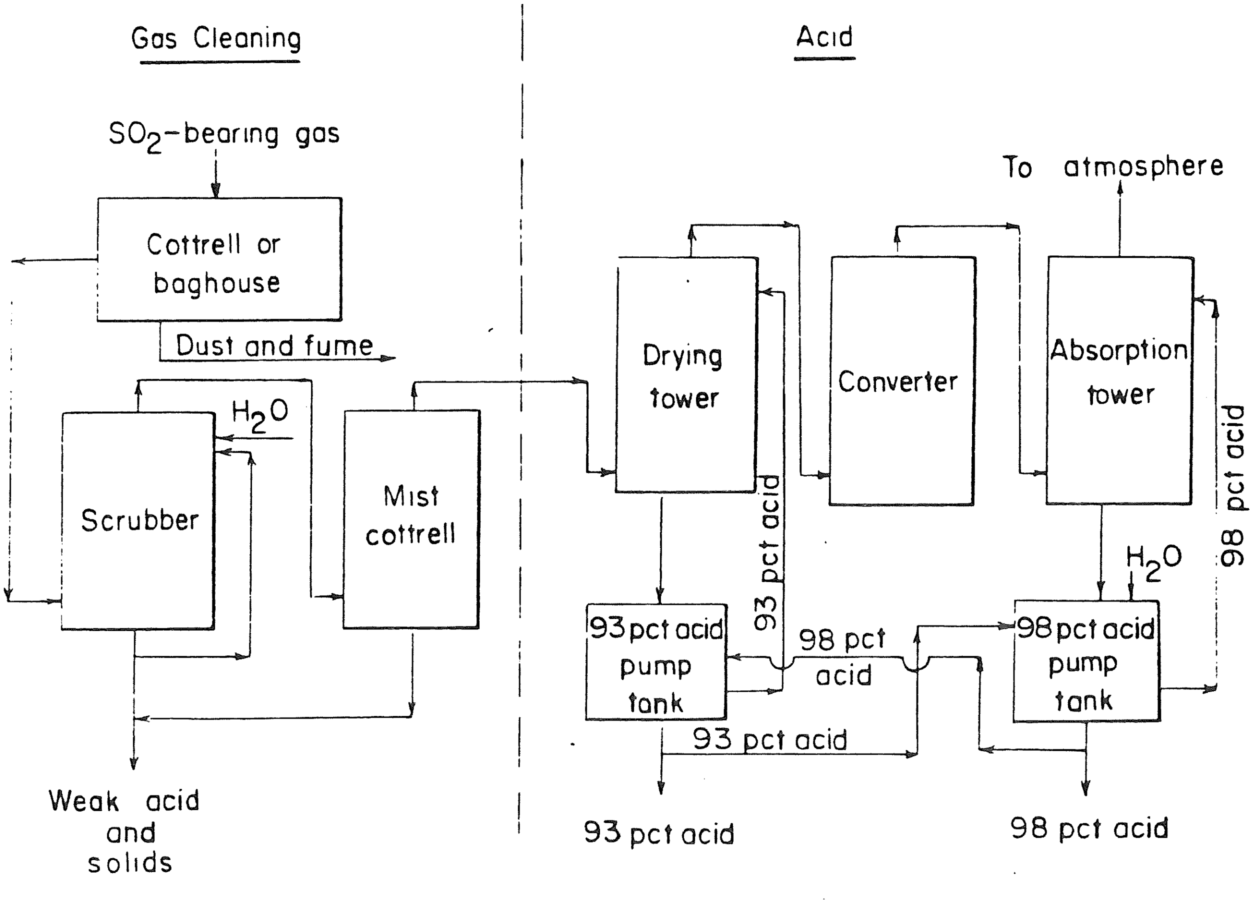


Figure C29 Contact Sulfuric Acid Process.

The Cominco Absorption Process. This process is an ammonium sulfite-bisulfite bsorption process which involves chemical regeneration with sulfuric acid to release SO_2 and form ammonium sulfate. Figure C30 outlines the process.

Sulfur dioxide-bearing gas, free of sulfur trioxide and particulates, is absorbed by an ammonium sulfite-ammonium bisulfite solution. The sulfur dioxide in the gas reacts with ammonium sulfite to form the bisulfite. Ammonia is added to convert part of the bisulfite to sulfite which is recycled to the absorption scrubbers. The remaining bisulfite solution is diverted to the stripper, acidified with sulfuric acid, and stripped with air to produce approximately a 25% sulfur dioxide gas and a solution of ammonium sulfate containing approximately 10% of the feed sulfur. The process will remove 90% of the sulfur dioxide from dilute flue gases, even at concentrations as low as 0.5%. Tail gases contain as little as 0.03% sulfur dioxide.[C3]

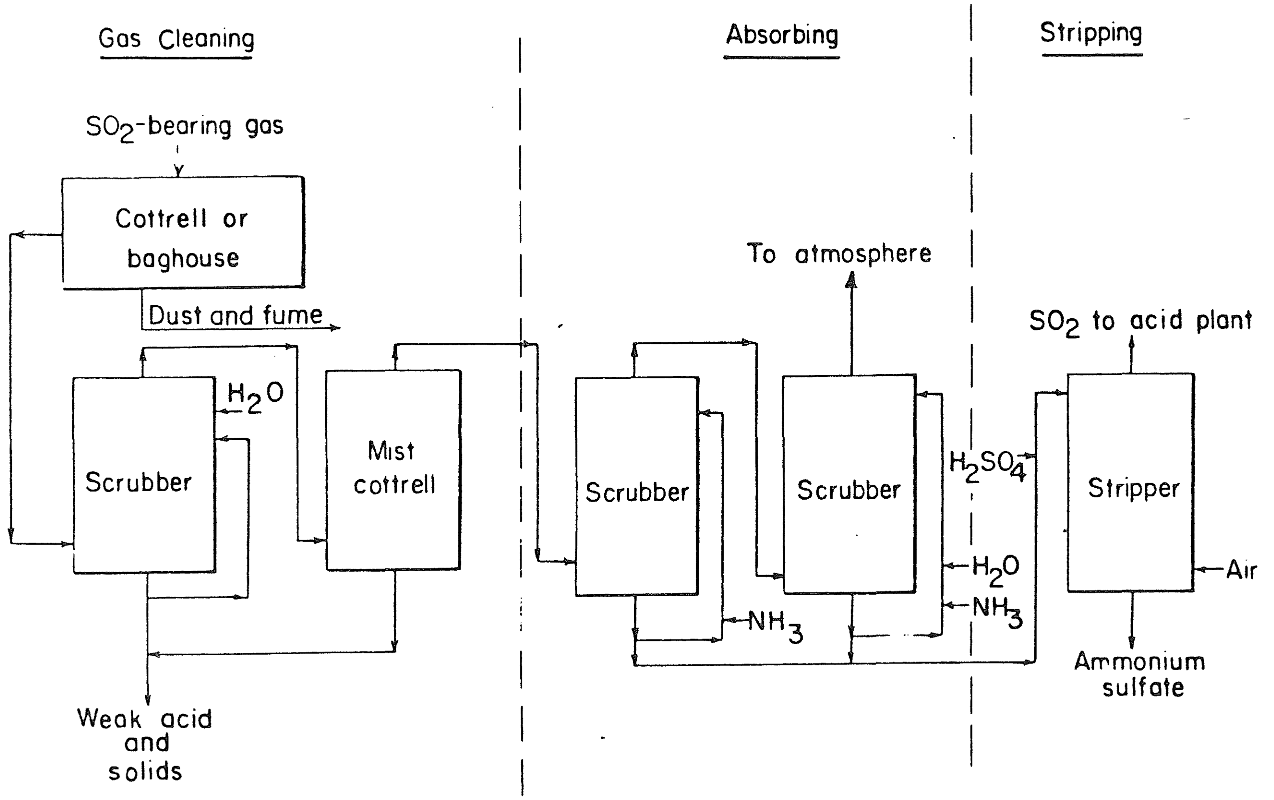


Figure C30 Cominco Absorption Process.

The Dimethylaniline (DMA) Absorption Process This process, developed by the American Smelting and Refining Company is used for recovering sulfur dioxide from smelter gases containing 4-10% SO_2. Outlined in Figure C31, flue gases are first cleaned, the cleaned gas stream then passing through a bubble-cap tower where the SO_2 is absorbed by dimethylaniline. Tail gases from the DMA absorption section in the bottom of the tower are scrubbed with a dilute sodium carbonate solution in the middle section of the absorption tower, thereby recovering residual sulfur dioxide from the gas stream. The sodium carbonate also neutralizes the sulfuric acid used for DMA vapor recovery as well as any acid formed through sulfur dioxide oxidation. The upper section of the absorption tower scrubs with dilute sulfuric acid to remove any remaining DMA vapor.

The loaded DMA solution is stripped with steam in the stripping section of the stripping column. Dimethylaniline and sulfur dioxide are recovered from the combined aqueous scrubber solutions, by steam distillation in the lower section of the stripping tower. The hot gas stream leaving the stripper, containing sulfur dioxide, steam, and dimethylaniline vapor, is cooled in the upper or rectifier section of the stripping column. In the presence of the sulfur dioxide, dimethylaniline vapor is recovered as water soluble dimethylaniline sulfate. This leaves essentially pure sulfur dioxide which can be liquified. The process can recover from 90-95% of the sulfur dioxide from a gas stream containing 2-4% sulfur dioxide, and as much as 95-98% from a gas stream containing 8-10% sulfur dioxide.[C3]

The Monsanto Cat-Ox Process The Cat-Ox process is particularly suited for gas streams lean in sulfur dioxide. As indicated in figure C32, the incoming lean gas is partially heated by exchange with product gas effluents from the converter. The gas is then preheated to approximately 800-900°F, and finally is oxidized to sulfur trioxide in the presence of vanadium pentoxide catalyst. Conversions are of the order of 90%.

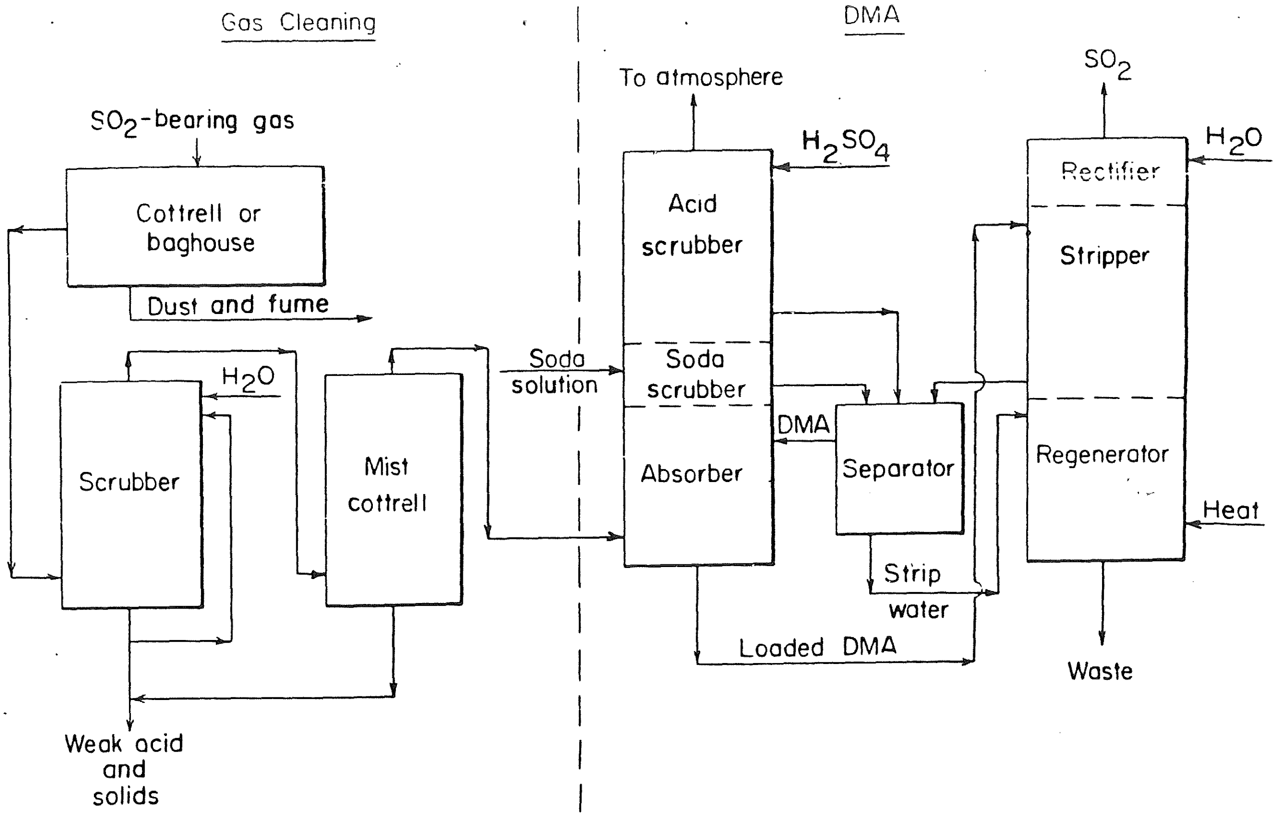
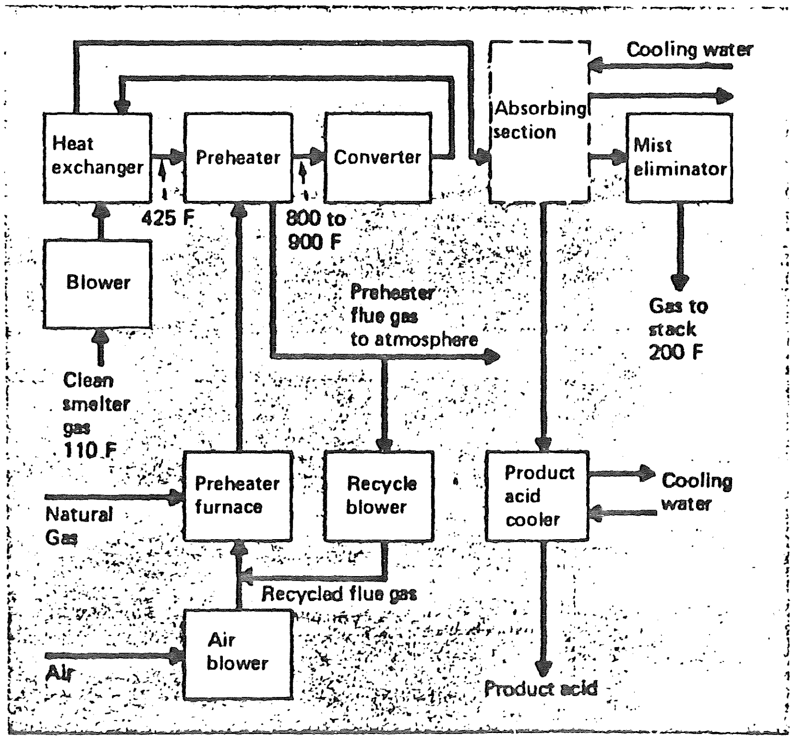


Figure C31 DMA Process.

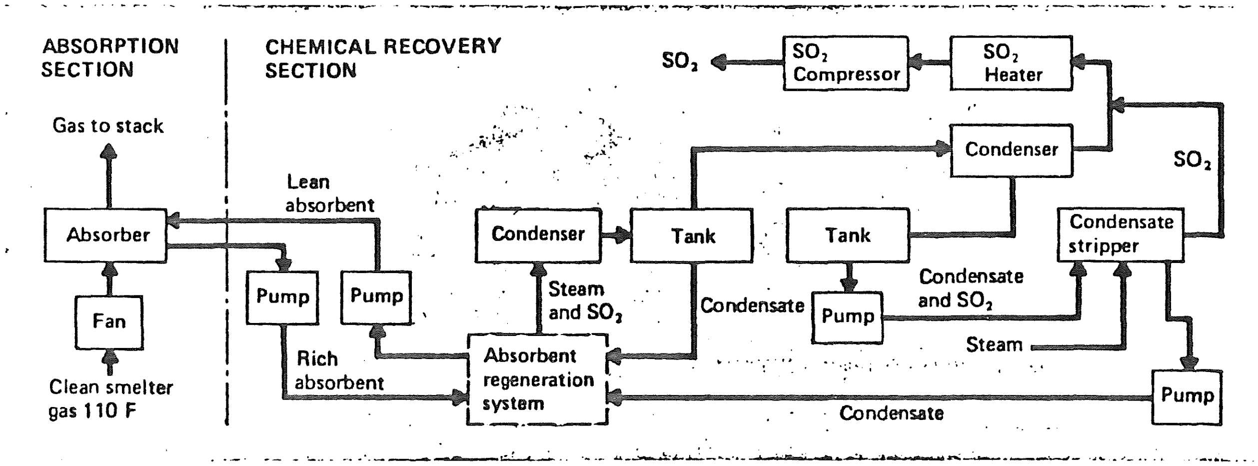


CAT-OX PROCESS works on lean SO_2 feed Figure C32

Product gases from the converter are heat-exchanged against feed gases, the sulfur dioxide then absorbed in water. Product acid ranging in concentration from 80-93% is then recovered in a cooler, with offgases going to the stack after cleansing in a mist eliminator. Critical areas in the process include:

1. fly-ash removal upstream of the unit
2. life and efficiency of the catalyst
3. quality and concentration of the acid product
4. corrosion in the heat exchanger unit[C26]

The Wellman-Power Gas Process The Wellman-Power Gas system (Figure C33) is based on a cyclical absorption-desorption phase, with sodium sulfite solution as the absorbent. During the absorption cycle, the sulfite reacts with sulfur dioxide to form sodium bisulfite. The reaction is reversed in desorption, with concentrated sulfur dioxide released and sulfite regenerated and recycled back into the absorption system.



WELLMAN-POWER GAS absorbs SO_2 with Na_2SO_3 Figure C33

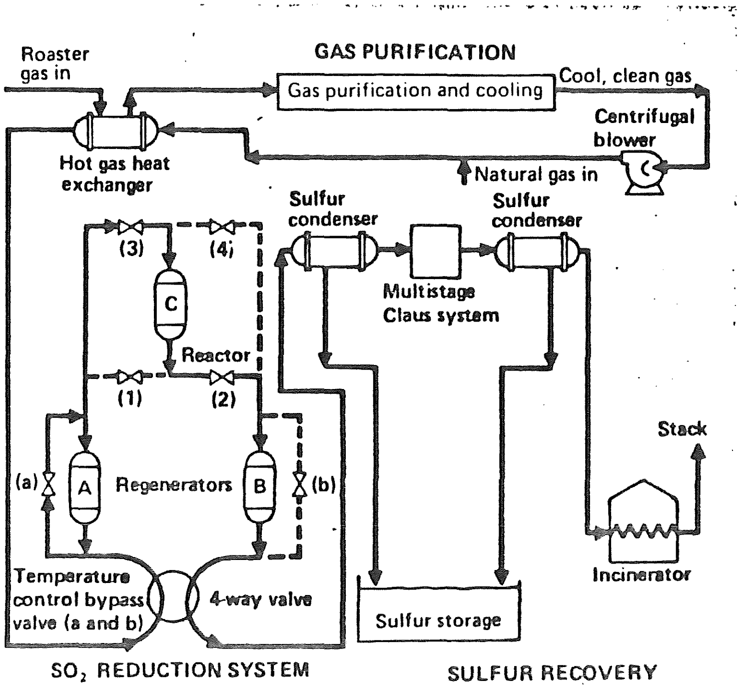
Gases coming from the smelter must be cleaned of particulate matter before treatment. Upon entering the absorber, the gases come in contact with the sodium sulfite solution and the sulfur dioxide is tripped out by converting the absorbent to the biosulfute form. Exit gases contain approximately 500 ppm when input gas has a sulfur dioxide concentration of 1-2%.

Desorption is accomplished quite easily by stripping the enriched bisulfite solution with steam in a countercurrent tower (7-15 # stream per # SO_2).

Any entrained stream in the sulfur dioxide gas is removed in a two-stage condensation process which removes the bulk of the water vapor at a relatively high temperature, and most of the remainder at approximately 120°F. The product sulfur dioxide gas contains approximately 10% water vapor, which makes it suitable as feed to a sulfuric acid plant or a sulfur dioxide reduction plant.[C26]

The Allied SO_2 Process The Allied SO_2 process involves the removal of sulfur dioxide from relatively rich sulfur dioxide gas streams. The process involves direct catalytic reduction of the sulfur dioxide to sulfur using natural gas as the reductant.

Gas purification is optional, but generally is considered necessary in order to remove particulates and excess water vapor. The cool, clean gas leaving this step is preheated by exchange with hot roaster gases before going to the reduction section.(Figure C34)



ALLIED SO_2 process preserves reaction heat Figure C34

In the reduction section, approximately half of the sulfur dioxide is converted to elemental sulfur by reaction with methane, creating in the process H_2S. In the subsequent sulfur recovery system, sulfur formed in the reactor is condensed and the remaining sulfur dioxide and H_2S are reacted in a multistage claus conversion system to product additional sulfur via the reaction:

$$2\ H_2S + SO_2 = 2\ H_2O + 3S$$

All sulfur recovered in the condensers is sent to storage and exit gas incinerated to remove traces of H_2S before it is exhausted to the atmosphere.[C26]

The Bureau of Mines Citrate Process In this process, sulfur dioxide bearing gases are cleansed of dust and fume by precipitation or filtering, followed by water scrubbing. The wet gases have the mist removed by another precipitator, then pass into an absorber system where they are contacted with a partially neutralized solution of citric acid. 90-95% of the sulfur dioxide is claimed to be removed.

The sulfur dioxide stream next goes to a reactor unit where hydrogen sulfide reacts with the absorbed sulfur dioxide to yield elemental sulfur. This is separated from the citric acid solution by filtration, with the solution recycled to the absorber.

A small portion of the sulfur product is diverted to a H_2S generator, where it is burned with steam and methane to produce H_2S and CO_2, the latter passing through the process and exiting from the reactor. (Figure C35)[C26]

The Limestone Scrubbing System A process for removal of sulfur dioxide from effluent gases by exposure of such gases to materials such as limestone exists. Designed systems convert the sulfur dioxide to an innocuous form.

Flue gas is precleaned of fly ash, then blown into the bottom of a spray tower where it is sprayed with a slurry containing 7-10% powdered limestone. As contact is made, the limestone reacts with the sulfur dioxide to yield insoluble calcium sulfite and sulfate. Slurry effluent drains by gravity into a delay-and-mixing

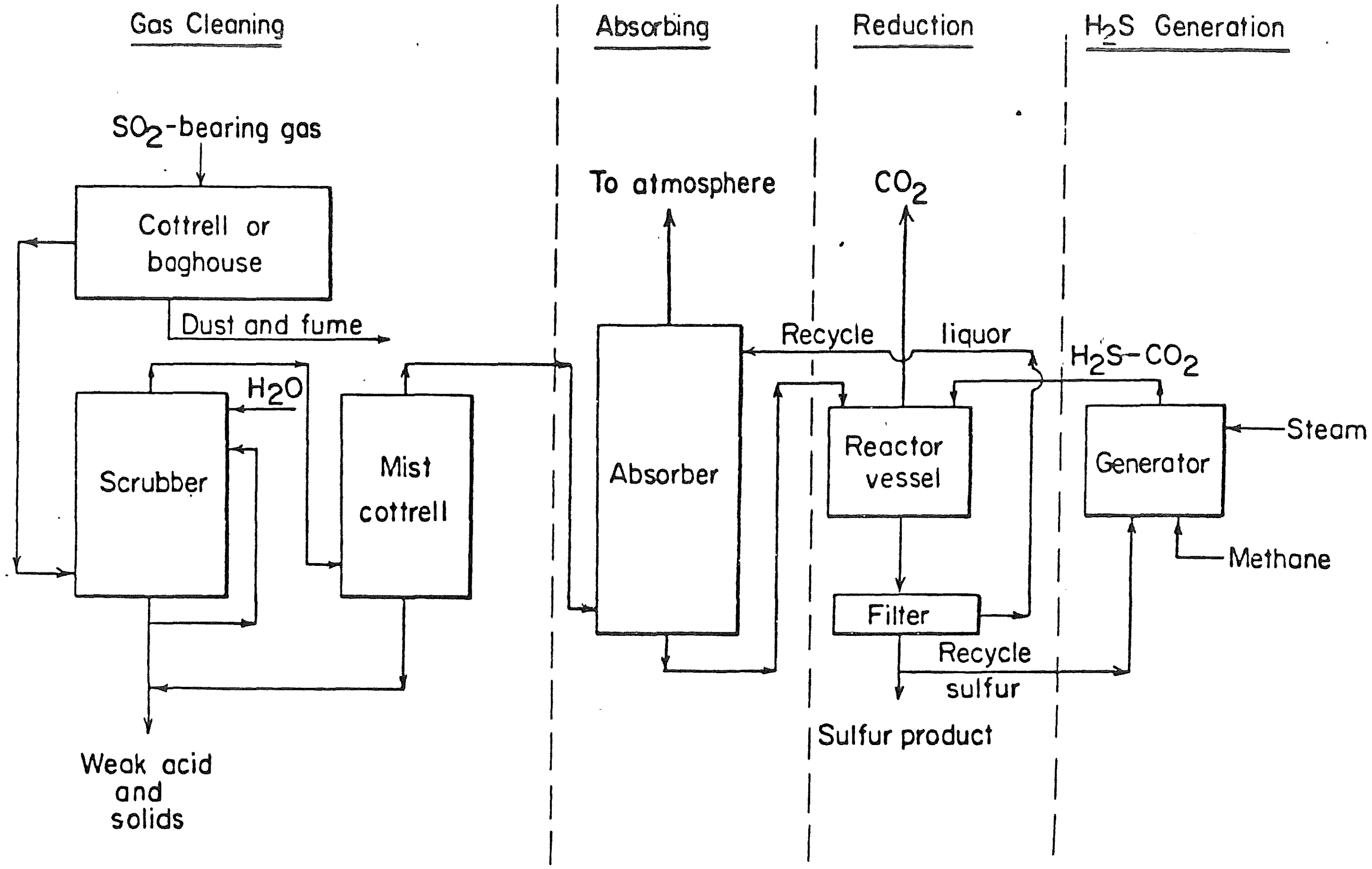


Figure C35 Citrate Process (U.S. Bureau of Mines).

tank, wherein supersaturated calcium salts settle out before the solution is recycled back to the scrubber. Fresh limestone is added in this tank.

A portion of the recycle stream is directed to a clarifier that returns a fairly clear overflow stream to a wet electrostatic precipitator into which the treated gases flow after scrubbing. Underflow from the classifiers is filtered so that precipitated calcium salts can be eliminated from the system and filtrate returned to the mixing tank.

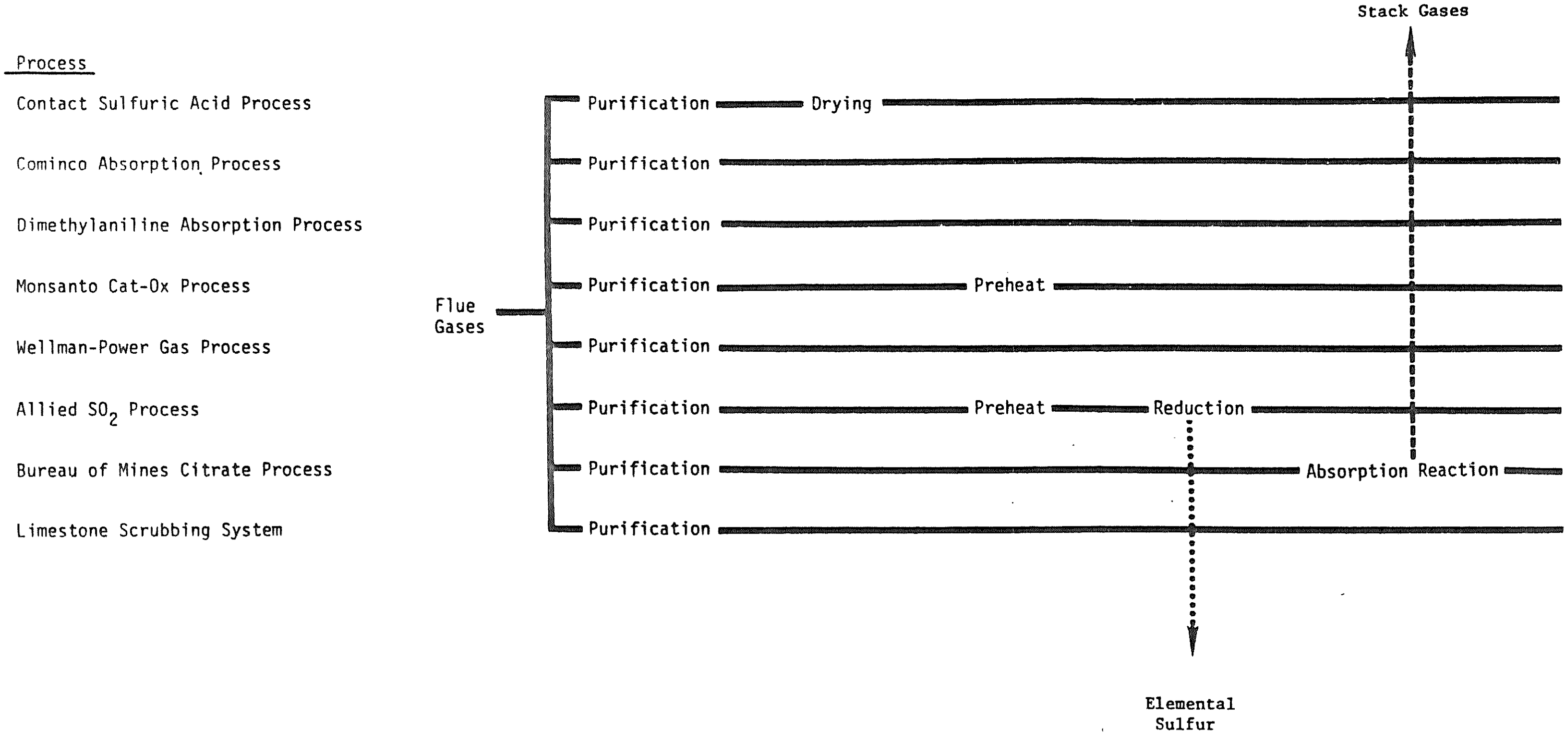
Other limestone-based processes exist, many in various stages of development.[C3]

The Closed-Loop Control System for SO_2 Emissions. This approach to control of sulfur dioxide emissions involves the use of special automated instrumentation for the continuous measurement of sulfur dioxide in the air. The system includes designed monitoring stations, degital computers which question each monitor and printing out average SO_2 values for each station periodically, a full-time meteorologist, plus supporting staff with extensive weather data recording and receiving equipment.

Two techniques are used to limit sulfur dioxide emissions. The first involves reaction to increasing concentrations of sulfur dioxide, the second involves utilization of meteorological data in the prediction of low winds and inversions.

A third technique in the developmental stage involves continuous measurement of sulfur dioxide emissions with the simultaneous prediction of downwind SO_2 concentrations as ground level.

Refer to figure C36 (a generalized flow diagram) to compare and contrast the various processes for sulfur dioxide concentration and/or removal from pyrometallurgical effluent gas streams. Also, refer to table C4 for a listing of the additives and/or catalysts which are necessary for efficient removal of gaseous sulfur dioxide.[C3]


Process
Contact Sulfuric Acid Process
Cominco Absorption Process
Dimethylaniline Absorption Process
Monsanto Cat-Ox Process
Wellman-Power Gas Process
Allied SO2 Process
Bureau of Mines Citrate Process
Limestone Scrubbing System
Flue Gases
Purification
Drying
Purification
Purification
Purification
Preheat
Purification
Purification
Preheat
Reduction
Purification
Absorption Reaction
Purification
Stack Gases
Elemental Sulfur

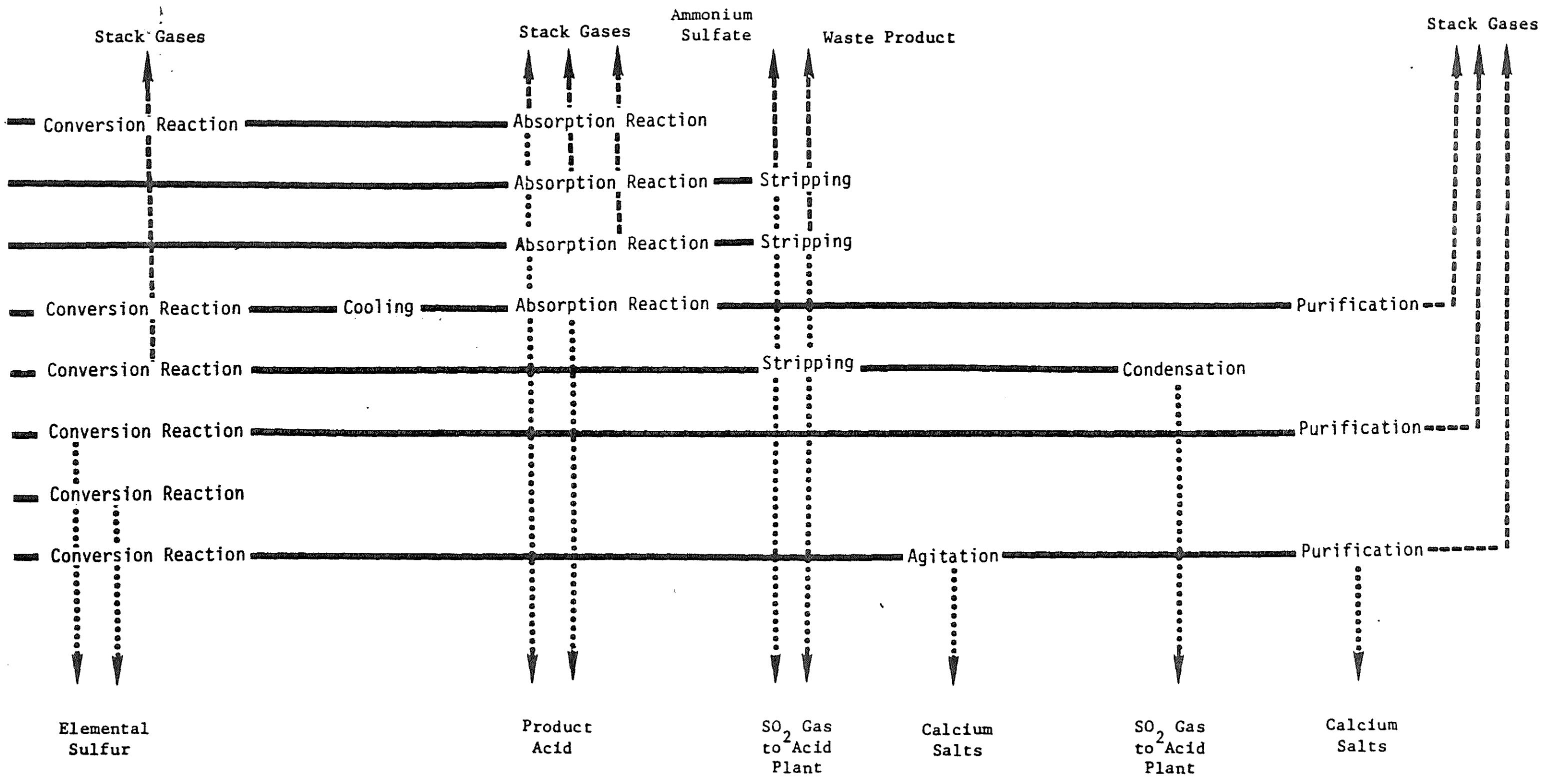


Figure C36 Generalized Flowsheet shows variety of choices in current and developing SO_2 control systems

Table C4

Additives and Catalysts Necessary for Process Concentration and/or Conversion of Sulfur Dioxide Gas

Process	Drying	Reduction	Conversion Reaction	Absorption Reaction	Stripping
Contact Sulfuric Acid Process	Sulfuric Acid			Sulfuric Acid	
Cominco Absorption Process				Ammonium Sulfite-Ammonium Bisulfite, Ammonia, H_2O	Sulfuric Acid, Air
Dimethylaniline Absorption Process				Dimethylaniline, Dilute Sulfuric Acid, Sodium Carbonate	Steam
Monsanto Cat-Ox Process			Vanadium Pentoxide	H_2O	
Wellman-Power Gas Process			Sodium Sulfite		Steam
Allied SO_2 Process		Methane, Allied Catalyst	Hydrogen Sulfide		
Bureau of Mines Citrate Process			Hydrogen Sulfide	Partially Neutralized Solution of Citric Acid	
Limestone Scrubbing System			Limestone		

Copper[C23]

Uses Copper is utilized as a pure metal, alloyed with ainc to form brass, and alloyed with tin to form bronze. Copper is used as a pure metal principally:

1. in electrical mining for motors
2. transformers
3. generators and instruments
4. as copper and brass tubing
5. as sheet for roofing, gutters, decorative applications, and coinage
6. in copper and alloy coatings and forgings for bearings, bushings, jewelry, and mechanical posts
7. ordnance
8. in chemicals for insecticides, pigments, and agriculture

Supply-Demand The domestic demand for copper is distributed libezally among the following industries:

Industry	Percent of Total Market
Electrical Equipment and Supplies	60%
Construction	15%
Industrial Machinery	10%
Transportation	8%
Ordnance	2%
Miscellaneous	5%

World copper production for 1974 was 8.1 million tons. U.S. production for the same period totaled 1.6 million tons.

Figure C37 indicates the supply-demand relationships in 1973 and table C5 reveals copper-supply-demand data for the period 1964-74.

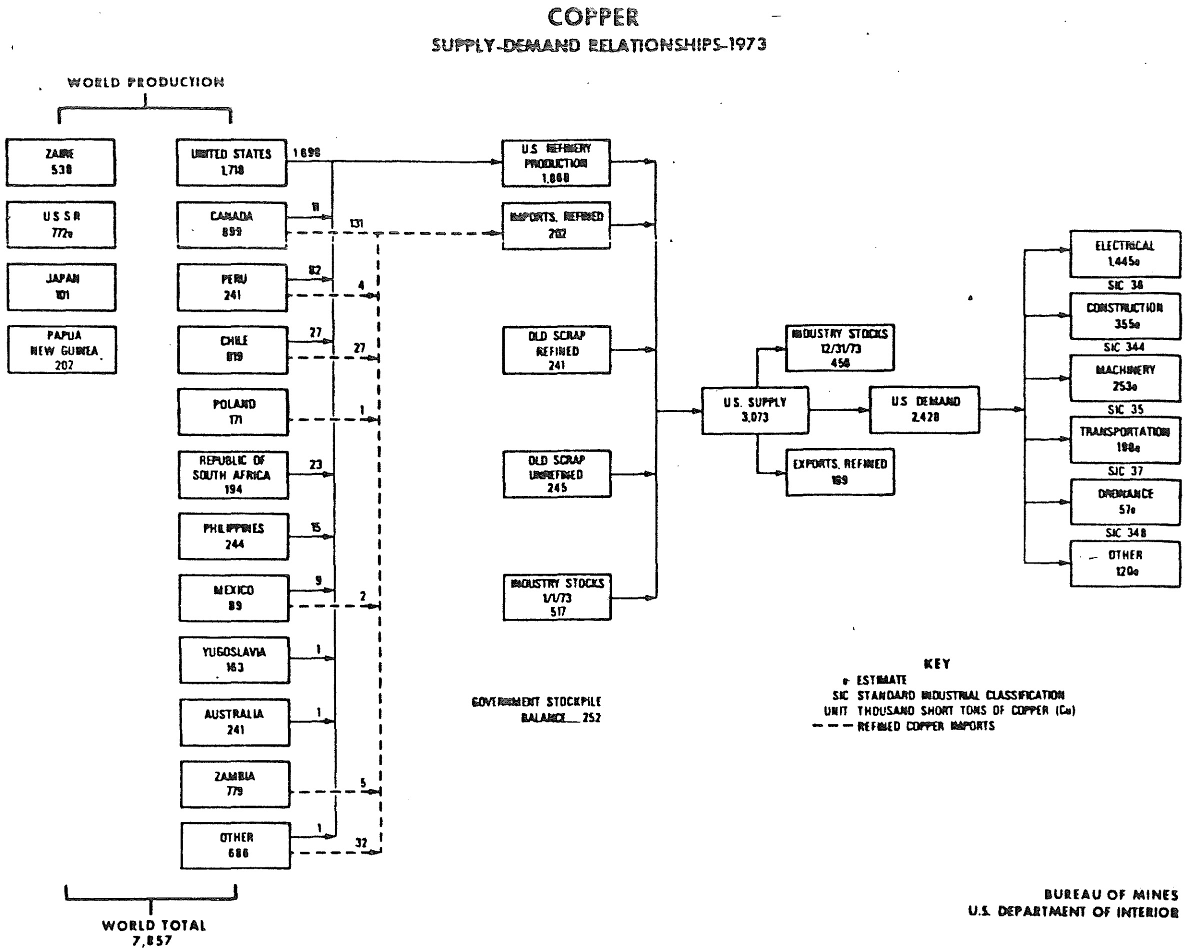


Figure C37 Supply-demand relationships for copper, 1973.

Table C5 **Copper supply-demand relationships, 1964–74**

(Thousand short tons)

	1964	1965	1966	1967	1968	1969	1970	1971	1972	1973	1974
World production (primary):											
Mine production:											
United States	1,247	1,352	1,429	954	1,205	1,545	1,720	1,522	1,665	1,718	1,597
Rest of world	3,865	3,967	4,056	4,270	4,436	4,679	4,918	5,131	5,649	6,139	6,512
Total	5,112	5,319	5,485	5,224	5,641	6,224	6,638	6,653	7,314	7,857	8,109
Components of U.S. supply (primary and old scrap):											
Refined production:											
Domestic mines	1,259	1,336	1,353	847	1,161	1,469	1,521	1,411	1,680	1,698	1,421
Old scrap	186	214	242	190	231	284	278	208	212	241	268
Imports of ore, blister, etc	396	376	358	286	276	274	244	181	193	170	234
Sales of Government stockpile excesses	27	120	400	149	----	----	----	----	----	----	252
Imports of refined	140	137	164	331	400	131	132	164	192	202	314
Old scrap (unrefined)	288	299	293	293	290	291	226	237	246	245	215
Industry stocks, Jan. 1	527	467	498	602	507	563	541	645	586	517	456
Total U.S. supply	2,823	2,949	3,308	2,698	2,865	3,012	2,942	2,846	3,109	3,073	3,160
Distribution of U.S. supply:											
Industry stocks, Dec. 31	467	498	602	507	563	541	645	586	517	456	597
Exports (refined)	316	325	273	159	241	200	221	188	183	189	127
Industrial demand	2,040	2,126	2,433	2,032	2,061	2,271	2,076	2,072	2,409	2,428	2,436
U.S. demand pattern:											
Electrical	941	1,028	1,178	1,113	1,046	1,193	1,101	1,113	1,252	1,445	1,250
Construction	420	415	410	277	316	341	328	351	432	355	425
Machinery	312	305	316	208	239	254	251	243	300	253	340
Transportation	222	227	226	145	193	198	173	192	227	198	257
Ordnance	43	45	182	188	164	172	119	69	78	57	42
Other	102	106	121	101	103	113	104	104	120	120	122
Total demand	2,040	2,126	2,433	2,032	2,061	2,271	2,076	2,072	2,409	2,428	2,436
Total U.S. primary demand (industrial demand less old scrap)	1,566	1,613	1,898	1,549	1,540	1,696	1,572	1,627	1,951	1,942	1,953

The forecasted demand for copper in the year 2000 is expected to range from 4.1-7.5 million short tons. The most probable demand figure is 60 million tons. Copper demands for the rest of the world are expected to range from 16.9-28.5 million tons in 2000. A summary of forecasted U.S. and world demand is included in table C6.

Table C6 **Summary of forecasts of U.S. and rest-of-world copper demand, 1973–2000**

(Thousand short tons)

	1973	2000 Forecast range		Probable		Probable average annual growth rate 1973–2000 (percent)
		Low	High	1985	2000	
United States:						
Primary	1,942	2,900	5,300	2,700	4,200	2.9
Secondary	486	1,200	2,200	900	1,800	5.0
Total	2,428	4,100	7,500	3,600	6,000	3.4
Cumulative (primary)	----	65,000	92,000	28,000	80,000	----
Rest of world:						
Primary	6,058	12,700	21,400	10,000	18,000	4.1
Secondary	2,158	4,200	7,100	3,000	6,000	3.9
Total	8,216	16,900	28,500	13,000	24,000	4.0
Cumulative (primary)	----	246,000	337,000	97,000	301,000	----
World:						
Primary	8,000	15,600	26,700	12,700	22,200	3.9
Secondary	2,644	5,400	9,300	3,900	7,800	4.1
Total	10,644	21,000	36,000	16,600	30,000	3.9
Cumulative (primary)	----	311,000	429,000	125,000	381,000	----

Reserves and Resources Based on available information, the world reserves of copper in ore are estimated at 450 million tons of copper. In addition, an estimated 1600 million tons of copper is available in other global resources. In the latter category, copper contained in oceanic manganese nodules accounts for 20% of known copper reserves and 20% of other copper resources. A detailed assessment of world copper resources is shown in table C7. The reserve classification includes all economically recoverable material in identified deposits. The other resource classification applies on an approximately equal division to deposits not yet discovered and to deposits from which the copper is not now economically recoverable.

Domestic reserves of copper in ore are estimated at 90 million short tons. Five states - Arizona, Utah, New Mexico, Montana and Michigan - account for more than 90% of the total reserves. Nearlyyall of the reserves are in deposits where copper is the dominant value; in the remainder, copper occurs as a coproduct of mixed or

Table C7 World copper resources

(Million short tons of copper)

	Reserves	Other[1]	Total
North America:			
United States	90	320	410
Canada	40	130	170
Other	20	30	50
Total	150	480	630
South America:			
Chile	90	130	220
Peru	30	40	70
Other	10	70	80
Total	130	240	370
Europe:			
U.S.S.R	40	90	130
Other	20	40	60
Total	60	130	190
Africa:			
Zaire	20	30	50
Zambia	30	70	100
Other	10	20	30
Total	60	120	180
Asia: Total	30	170	200
Oceania: Total	20	60	80
Sea nodules	---	400	400
World total	450	1,600	2,050

[1] Includes undiscovered (hypothetical and speculative) deposits.

complex base metal ores. Areas containing major potential for developing domestic copper resources include the porphyry copper-bearing basin and range province of the Southwest United States and the copper-nickel bearing Duluth Gabbro of Minnesota.

Excluding deep sea resources, the amount of copper estimated to exist in the rest of the world comprises reserves of 360 million tons and other resources of 880 million tons. Six countries - Chile, Canada, the U.S.S.R., Peru, Zambia, and Zaire - account for 69% of the rest of the world's reserves and 56% of other resources. Remaining reserves are divided among many other countries including Australia and Papua New Guinea, Finland, Iran, Japan, Panama, the People's Republic of China, the Philippines, Poland, the Republic of South Africa, Sweden, and Yugoslavia.

Economics The price of copper, in terms of constant 1973 dollars, for the 1954-74 period, is summarized in table C8. Prices shown in the table are domestic producer quotations for delivered electropytic wirebar refined copper.

The price of copper (Domestic Refined) for the week of July 5, 1976 was quoted at $.74 in terms of 1976 dollars.

Costs of producing copper are variable, largely depending upon location and physical characteristics of the ore deposit. For a representative large open pit copper operation, cost components in terms of percentages of the price of copper are estimated to be:

15% mining

25% ore beneficiation

25% freight, smelting, and refining

35% discovery, development taxes, marketing, and general overhead, including profit

Table C8 **Time-price relationship for copper**

Year	Average annual price, cents per pound	
	Actual prices	Constant 1973 dollars
1954	29.5	50.8
1955	37.3	63.3
1956	42.5	69.8
1957	30.1	47.6
1958	26.3	40.6
1959	30.7	46.6
1960	32.1	48.0
1961	30.0	44.2
1962	30.8	44.9
1963	30.8	44.3
1964	32.6	46.2
1965	35.4	49.3
1966	36.6	49.6
1967	38.6	50.7
1968	42.2	53.2
1969	47.9	57.6
1970	58.2	66.4
1971	52.0	56.8
1972	51.2	54.1
1973	59.5	59.5
1974[1]	77.3	70.1

[1] The price was 63.625 to 65 cents per pound in December 1975.

Nickel[C4]

Uses More than 90% of all nickel consumed is in the form of metal, principally in alloys. Its ability to resist corrosion and to impart corrosion resistance, strength, and specific physical properties in alloys leads to its wide and variable use in many producer and consumer goods. Nickel's chemical properties lead to its use in batteries, dyes, and pigments, as a catalyst, and in insecticides. Major end used include:

1. chemicals and allied products and petroleum refining and allied products
2. fabricated metal products
3. aircraft and parts
4. motor vehicles and equipment
5. electrical machinery, equipment, and supplies
6. household appliances
7. machinery
8. general building contractors
9. chip and boat building and repairing
10. miscellaneous

<u>Supply-Demand</u> The domestic demand for nickel is distributed among the following industries:

Industry	Percent of Total Market
Chemicals	15%
Petroleum	9%
Fabricated Metal Products	10%
Transportation	21%
Electrical	13%
Household Appliances	7%
Machinery	7%
Construction	9%
Miscellaneous	9%

World nickel production for 1974 was 823.2 thousand tons. U.S. production for the same period totaled 14.1 thousand tons.

Figure C38 indicates the supply-demand relationships in 1973 and table C9 reveals nickel-supply-demand data for the period 1964-74.

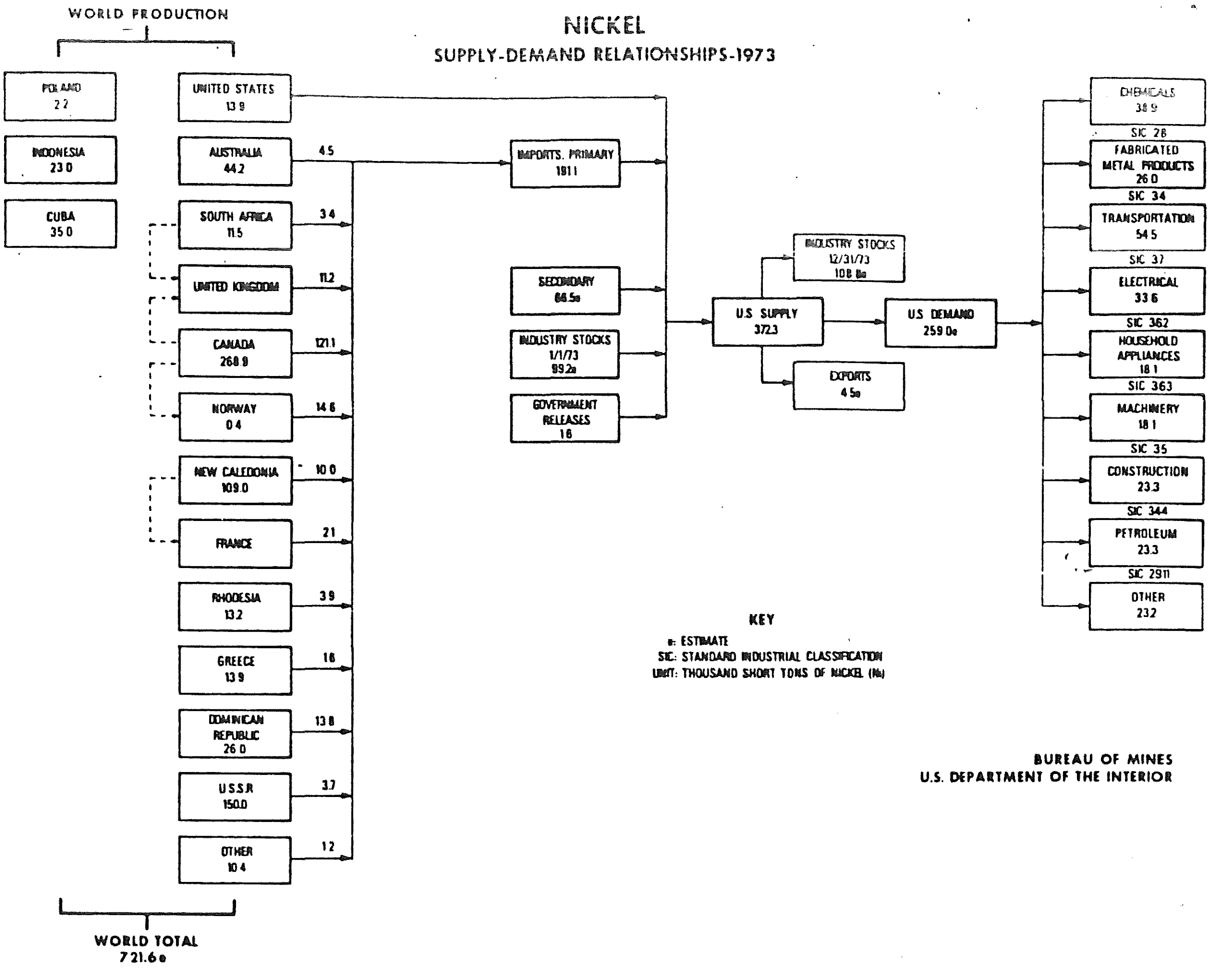


Figure C38 Supply-demand relationships for nickel, 1973

Table C9 **Nickel supply-demand relationships, 1964–1974**

(Thousand tons)

	1964	1965	1966	1967	1968	1969	1970	1971	1972	1973	1974[p]
World mine production:											
United States	12.2	13.5	13.2	14.6	15.2	15.8	15.6	15.6	15.7	13.9	14.1
Rest of world	396.8	454.8	426.9	480.2	532.9	516.9	676.8	685.0	666.2	707.7	809.1
Total	409.0	468.3	440.1	494.8	548.1	532.7	692.4	700.6	681.9	721.6	823.2
Components of U.S. supply:											
Domestic mines	12.2	13.5	13.2	14.6	15.2	15.8	15.6	15.6	15.7	13.9	14.1
Secondary	50.9	51.4	63.1	52.3	36.6	71.0	48.7	72.9	74.0	66.5	70.0
Net Government release	2.9	16.3	103.6	23.3	3.2	4.3	2.1	14.9	1.8	1.6	4.6
Imports	129.0	163.0	141.0	142.6	143.7	125.8	156.3	142.2	173.9	191.1	220.7
Industry stock, Jan. 1	17.2	17.2	14.1	44.5	39.6	37.2	31.9	56.0	82.0	99.2	108.8
Total U.S. Supply	212.2	261.4	335.0	277.3	238.3	254.1	254.6	301.6	347.4	372.3	418.2
Distribution of U.S. supply:											
Industry stock, Dec. 31	17.2	14.0	31.3	34.6	37.3	31.9	56.0	82.0	99.2	108.8	150.3
Exports	12.0	5.6	11.8	8.0	6.5	1.4	6.3	4.6	2.6	4.5	4.3
Industrial demand	183.0	241.8	291.9	234.7	194.5	220.8	193.3	215.0	245.6	259.0	263.6
U.S. demand pattern:											
Chemicals	10.5	16.8	21.4	25.9	21.9	34.0	28.9	32.1	36.8	38.9	39.7
Petroleum	5.8	9.1	11.7	14.1	11.9	17.0	15.4	19.4	22.2	23.3	23.7
Fabricated metal products	15.8	22.5	28.7	40.2	24.5	18.4	18.4	22.0	24.6	26.0	26.5
Transportation:											
Aircraft	14.5	23.9	30.4	16.5	24.2	14.8	11.6	15.1	16.9	18.1	18.4
Motor vehicles and equipment	27.2	38.4	36.4	23.9	22.0	23.3	23.1	23.6	27.5	28.5	29.0
Ship & boat building and repairs	5.3	9.1	11.7	7.3	9.1	8.5	5.7	6.6	7.1	7.9	8.0
Total	47.0	71.4	78.5	47.7	55.3	46.6	40.4	45.3	51.5	54.5	55.4
Electrical	17.0	28.5	34.7	20.2	20.2	29.7	24.9	27.8	31.9	33.6	34.2
Household appliances	16.9	24.9	28.9	18.4	18.2	13.4	12.5	14.7	17.2	18.1	18.4
Machinery	21.6	33.6	42.6	12.9	12.6	16.0	14.3	14.8	17.2	18.1	18.4
Construction	6.8	9.8	13.8	11.0	10.3	15.8	18.4	19.6	22.2	23.3	23.7
Other	41.6	25.2	31.6	44.3	19.6	29.9	19.3	19.3	22.0	23.2	23.6
Total U.S. primary demand (industrial demand less secondary)	132.1	190.4	228.8	182.4	157.9	149.8	143.6	142.1	171.6	192.5	193.6

[p] Preliminary.

The forecasted demand for nickel in the year 2000 is expected to range from 420-640 thousand short tons. The most probable demand figure is 550 thousand tons. Nickel demands for the rest of the world are expected to range from 1190-1710 thousand tons in 2000. A summary of forecasted U.S. and world demand is included in table C10.

Table C10 **Summary of forecasts of U.S. and rest of world nickel demand, 1973–2000**

(Thousand tons)

	1973	2000 Forecast range		Probable		Probable average annual growth rate 1973–2000 (percent)
		Low	High	1985	2000	
United States:						
Primary	193	300	450	260	385	2.6
Secondary	66	120	190	100	165	3.5
Total	259	420	640	360	550	2.8
Cumulative (primary)	---	6,600	8,300	2,700	7,600	---
Rest of world:						
Primary	597	990	1,450	810	1,205	2.6
Secondary	144	200	260	160	245	2.0
Total	741	1,190	1,710	970	1,450	2.5
Cumulative (primary)	---	21,200	26,200	8,500	23,600	---
World:						
Primary	790	1,290	1,900	1,010	1,590	2.6
Secondary	210	320	450	260	410	2.5
Total	1,000	1,600	2,350	1,300	2,000	2.6
Cumulative (primary)	---	27,800	34,500	11,200	31,200	---

Reserves and Resources U.S. nickel reserves (measured, indicated, and inferred) totaled approximately 200,000 tons in lateritic material containing 0.8-1.3% nickel. All are in deposits at the operating mine near Riddle, Oregon. There are other nickeliferous lateritic deposits in Oregon, California, and Washington, some parts of which are as high grade as the reserves at Riddle, however, they are not mineable at a profit under current economic conditions. The large low-grade sulfide deposits in the Duluth Gabbro of northeastern Minnesota are currently classified as resources.

World nickel reserves have been estimated by the Bureau of Mines at nearly 50 million tons, however, the estimates are based on fragmentary information and are considered to be low. The general distribution and order of magnitude of principle nickel reserves are given in table C11, Total resources are estimated at 100 million tons of nickel.

Combined sulfide and laterite world resources averaging approximately 1% nickel are estimated by the U.S. Geological Survey to total 70 million tons of nickel in

Table C11 **Assessment of world nickel resources**

(Thousand tons)

	Reserves	Other resources[1]	Total resources[1]
North America:			
United States	200	15,000	15,200
Canada	8,000	9,900	17,900
Total	8,200	24,900	33,100
Central America and Caribbean Islands:			
Cuba	4,200	13,800	18,000
Dominican Republic	900	------	900
Guatemala	500	500	1,000
Puerto Rico	----	100	100
Total	5,600	14,400	20,000
Europe: U.S.S.R.	10,000	------	10,000
Oceania:			
Australia	2,500	2,000	4,500
Indonesia	3,700	4,300	8,000
New Caledonia	15,400	1,100	16,500
Philippines	4,500	3,500	8,000
Total	26,100	10,900	37,000
World total[2]	49,900	50,200	100,100

[1] Derived from U.S. Geol. Survey Prof. Paper 820, 1973.

[2] Excludes small quantities of reserves in Brazil, Rhodesia, Republic of South Africa, and Burma, and an unknown quantity of low-grade laterites that exist in tropical and semitropical areas. Also excludes nickel associated with copper deposits of Botswana and seabed manganese nodules.

7 billion tons of material. Among the world's resources containing less than 1% nickel are U.S. sulfide deposits, estimated to contain 7 billion tons of material averaging 0.2% nickel. In addition, the peridotites and serpentinites containing 0.2-0.4% nickel are widely disseminated throughout the world.

Economics Table C12 shows the time-price relationship for nickel over a 20-year period in actual cents per pound and in constant 1973 dollars. The price of nickel for the week of July 5, 1976 was quoted at $2.20 per pound in terms of 1976 dollars.

Table C12 **Time-price relationship for nickel[1]**

Year	Average annual price, cents per pound	
	Actual prices	Constant 1973 dollars
1954	64.50	111
1955	64.50	110
1956	74.00	122
1957	74.00	117
1958	74.00	114
1959	74.00	112
1960	74.00	111
1961	81.25	120
1962	79.00	115
1963	79.00	114
1964	79.00	112
1965	77.75	108
1966	85.25	116
1967	94.00	123
1968	103.00	130
1969	128.00	154
1970	128.00	146
1971	133.00	145
1972	140.00	148
1973	153.00	153
1974	174.24	158

[1] Price changes (cents per pound): 1974—Jan. 4, 162; June 28, 185; Dec. 20, 201; and 1975—January-May, 201.

Cobalt C24

Uses Cobalt is used principally in heat and corrosion-resistant materials, high-strength materials, and permanent magnets. Other important uses include hard facing alloys for wear and abrasion resistance, and tool and die steels. More than 25% is used in nonmetallic compounds. In the metallic form, cobalt is used primarily in permanent magnets, cemented carbides, and various alloys. Cobalt compounds are used in salts and dryers, pigments, and catalysts. In summary, cobalt finds its use in:

1. electrical equipment and supplies
2. aircraft and surface engines and parts
3. machine tools
4. construction and mining
5. paints and related products
6. miscellaneous chemical products
7. miscellaneous

Supply-Demand The domestic demand for cobalt is distributed amont the following industries:

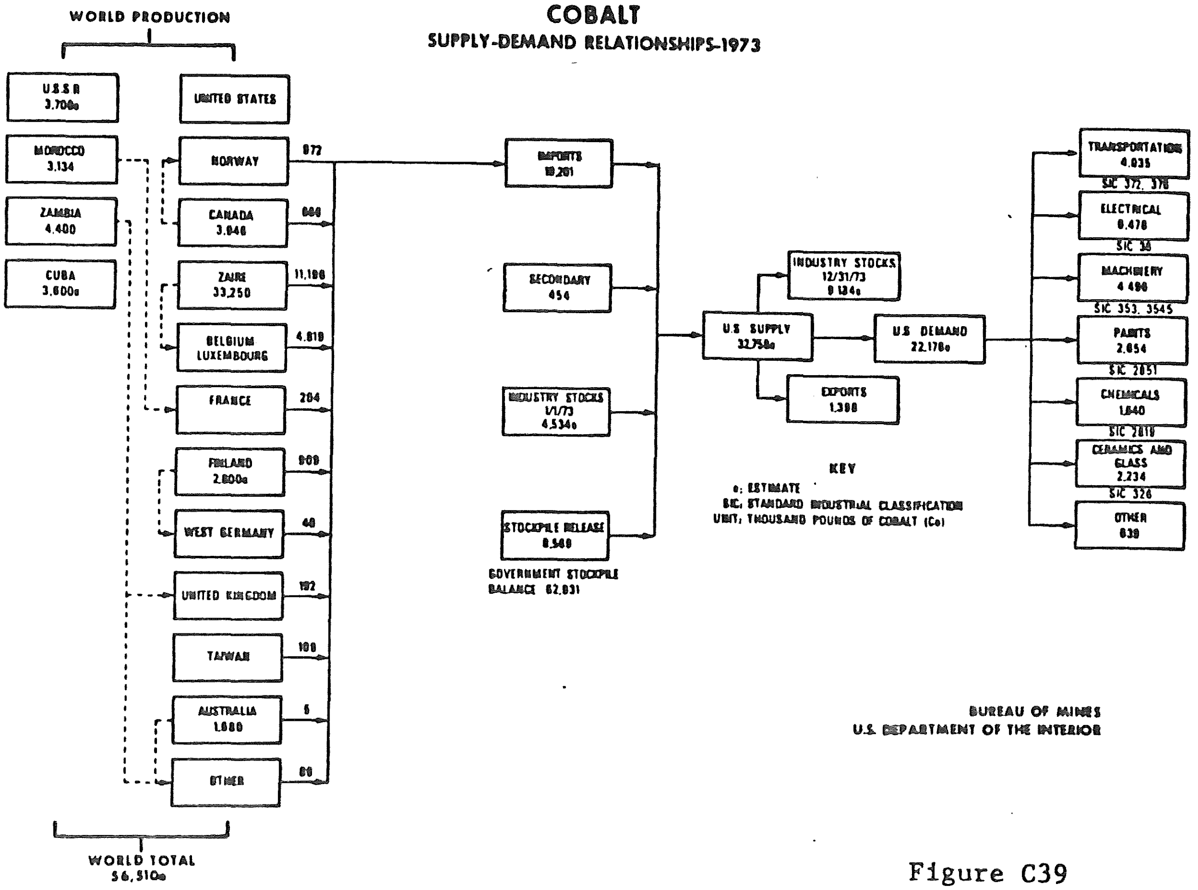
Industry	Percent of Total Market
Paints	12%
Chemicals	7%
Ceramics and Glass	10%
Transportation-Aircraft	18%
Electrical	29%
Machine Tools	11%
Construction Machinery	9%
Miscellaneous	4%

The U.S. industry relies almost entirely on imports of cobalt for its supply. In 1974, imports constituted 63% of domestic supply. The remainder came from national stockpiles and secondary production. Except for one nickel-cobalt refinery in Louisiana, which processes an imported matte, no primary cobalt is producte in the United States.

Virtually all cobalt is recovered as a byproduct of copper or nickel. In 1974, world mine production of cobalt was 67 million pounds, of which 38.7 million pounds was from Zaire. No mine production has been reported in the U.S. since 1971. Zambian production was 7 million pounds, Canada 4.2 million pounds, Morocco 3.9 million pounds, and the U.S.S.R. an estimated 3.8 million pounds.

Azire accounts for 60-65% of total world cobalt, followdd by Zambia, 10%, and Canada, 9%. This trend is expected to continue for the next few years.

Figure C39 indicates the supply-demand relationships in 1973 and table C13 reveals cobalt-supply-demand data for the period 1964-1974.


COBALT
SUPPLY-DEMAND RELATIONSHIPS-1973

BUREAU OF MINES
U.S. DEPARTMENT OF THE INTERIOR

Figure C39

Table C13 **Cobalt supply-demand relationships, 1964–74**

(Thousand pounds)

	1964	1965	1966	1967	1968	1969	1970	1971	1972	1973	1974[p]
World production:											
United States	1,084	1,186	1,215	1,168	1,176	1,003	697	690	---	---	---
Rest of world	38,150	40,624	46,782	44,028	41,968	43,556	52,590	47,908	51,290	56,510	67,008
Total	39,234	41,810	47,997	45,196	43,144	44,559	53,287	48,598	[e]51,290	56,510	67,008
Components of U.S. supply:											
Domestic mines	1,084	1,186	1,215	1,168	1,176	1,003	697	690	W	W	W
Secondary	148	87	48	120	143	328	69	125	197	454	270
Net Government release (purchase)	733	(1)	762	6,189	4,953	6,007	5,162	1,683	5,945	8,569	8,938
Imports	12,443	15,408	18,823	8,215	9,068	12,911	12,417	10,912	13,915	19,201	15,878
Industry stocks, Jan. 1[1]	1,099	1,420	3,600	6,400	6,552	5,888	5,128	5,733	5,235	4,534	9,184
Total U.S. supply	15,507	18,100	24,448	22,092	21,892	26,137	23,473	19,143	25,292	32,758	34,268
Distribution of U.S. supply:											
Industry stocks, Dec. 31[1]	1,420	3,600	6,400	6,552	5,888	5,128	5,733	5,235	4,534	9,184	9,467
Exports	108	116	100	200	1,420	1,497	1,478	365	1,293	1,398	1,348
Industrial demand	13,979	14,384	17,948	15,340	14,584	19,512	16,262	13,543	19,465	22,176	23,453
U.S. demand pattern:											
Nonmetal											
Paints	1,246	1,259	1,451	1,044	1,451	1,285	2,152	2,042	2,323	2,654	2,778
Chemicals	699	665	753	696	882	1,382	538	707	1,270	1,640	1,938
Ceramics and glass	1,529	1,428	1,580	1,135	1,281	858	1,890	1,768	1,982	2,234	2,308
Total nonmetal	3,474	3,352	3,784	2,875	3,614	3,525	4,580	4,517	5,575	6,528	7,024
Metal											
Transportation: Aircraft	3,208	3,514	4,482	3,575	2,544	4,046	3,297	2,518	4,294	4,035	5,235
Electrical	3,972	3,976	5,387	4,160	5,100	4,024	4,589	3,633	6,069	6,478	5,571
Machinery:											
Machine tools	1,628	1,672	2,152	1,628	1,616	1,911	1,882	1,376	1,717	2,459	2,820
Construction machinery	1,418	1,582	1,685	1,623	998	1,268	1,179	1,080	1,190	2,037	2,192
Total	3,046	3,254	3,837	3,251	2,614	3,179	3,061	2,456	2,907	4,496	5,012
Coating and plating	---	---	---	---	---	3,351	---	---	---	---	---
Other	279	268	458	1,479	712	1,387	735	419	620	639	611
Total metal	10,505	11,032	14,164	12,465	10,970	15,987	11,682	9,026	13,690	15,648	16,429
Total U.S. primary demand (total demand less secondary supply)	13,831	14,297	17,900	15,220	14,441	19,184	16,193	13,418	19,268	21,722	23,183
Total U.S. demand for primary metal	10,357	10,845	14,116	12,345	10,827	15,659	11,613	8,901	13,693	15,194	16,159

[1] Includes consumers' and producers' stocks after 1964. Producers' stocks not available prior to 1965.

[p] Preliminary.

The demand for primary cobalt in the United States for the year 2000 is forecast at between 29.7 and 52.1 million pounds. The most probable demand figure is 44.7 million pounds. Cobalt demands for the rest of the world are expected to range from 69.8-113.0 million pounds in 2000. A summary of forecasted U.S. and world demand is included in table C14.

Reserves and Resources Most cobalt resources are only available as byproducts of mining for more abundant elements. According to the U.S. Geological Survey, Minnesota possesses the largest deposits (identified resources) in the United States, approximately 1 billion pounds of cobalt in the form of sulfides in the Ely area. Missouri is second with 150 million pounds in the southeast Missouri

Table C14 **Summary of forecasts of U.S. and rest of world cobalt demand, 1973–2000**

(Thousand pounds)

	1973	2000 Forecast range		Probable		Probable average annual growth rate 1973–2000 percent
		Low	High	1985	2000	
United States:						
Primary	21,722	29,700	52,100	27,100	43,000	[1]3.1
Secondary	454	1,400	2,000	800	1,700	5.1
Total	22,176	31,100	54,100	27,900	44,700	3.1
Cumulative (primary)[1]	---	651,000	896,000	278,000	804,000	---
Rest of world:						
Primary	37,309	69,800	113,000	56,400	94,500	3.5
Secondary	2,600	3,200	4,600	3,200	4,200	1.8
Total	39,909	73,000	117,600	59,600	98,700	3.4
Cumulative (primary)	---	1,430,000	1,890,000	564,000	1,690,000	---
World:						
Primary	59,031	99,500	165,100	83,500	137,500	3.4
Secondary	3,054	4,600	6,600	4,000	5,900	2.5
Total	62,085	104,100	171,700	87,500	143,400	3.3
Cumulative (primary)	---	2,081,000	2,786,000	842,000	2,494,000	---

[1] Calculated from the 20-year primary demand trend value of 18,888 for 1973.

lead district. Maine, Alaska, and Pennsylvania also have large identified resources.

Much of the world's identified resources are in the form of lateritic nickel ores in tropical regions. However, most cobalt currently comes from sulfide deposits in Zaire, Zambia, and Canada. An assessment of world cobalt resources is shown in table C15.

Table C15 **Assessment of world cobalt resources**

(Million pounds)

	Estimated reserves	Other resources[2]	Total resources[2]
North America:			
United States[1]	0	1,684	1,684
Canada	386	164	550
Cuba	744	1,568	2,312
Total	1,130	3,416	4,546
Africa:			
Morocco	28	NA	28
Zaire	1,500	160	1,660
Zambia	766	NA	766
Total	2,294	160	2,454
Oceania	1,480	460	1,940
Europe:			
U.S.S.R.	450	NA	450
Finland	50	NA	50
Total	500	—	500
World total	5,404	4,036	9,440

NA Not available.

[1] No resources of cobalt are considered reserves because it is currently (1974) uneconomical to mine cobalt in the United States.

[2] Source: U.S. Geol. Survey Prof. Paper 820, 1973.

Economics Cobalt prices have risen staeily from $1.50 per pound in 1964 to over $4.00 in 1976. Factors affecting this increase include:

1. inflation
2. removal of cobalt from a list of commodities prohibited from being shipped to Sino-Soviet nations
3. a 30% increase in U.S. industrial consumption as a result of the Vietnam War
4. the announced nationalization of mines in Zaire
5. increased demands because of the nickel shortage due to the Canadian nickel strike
6. currency realinement

The price is expected to increase as new uses for cobalt are found.

Sulfur [C14]

Uses Sulfur is unusual as compared with most major mineral commodities in that by far the largest portion of it is used as a chemical reagent rather than as a component of a finished product. Its predominant use as a process chemical generally requires that it be first converted to an intermediate chemical product prior to its initial use by industry.

Sulfuric acid is the most important of these intermediate products. Ninety percent of the sulfur consumed in the United States in 1974 was either converted to sulfuric acid or produced directly in this form. Other intermediate products included carbon disulfide and sulfur dioxide. Four percent of the total consumption was used directly in the elemental form.

The distribution of U.S. sulfur consumption is as follows:

1. agricultural products
2. plastic and synthetic products
3. paper products

4. paints
5. nonferrous metal production
6. explosives
7. petroleum refining
8. iron and steel production
9. miscellaneous

Supply-Demand The domestic demand for sulfur is distributed among the following industries:

Industry	Percent of Total Market
Agriculture (Fertilizers)	55%
Plastic & Synthetic Products	6%
Paper Products	4%
Paints	4%
Nonferrous Metal Production	5%
Explosives	3%
Petroleum Refining	2%
Iron & Steel Production	1%
Other	20%

World production of sulfur in all forms exceeded world demand by a substantial amount. In 1974, world production of sulfur in all forms was 50.9 million tons. United States production for the same period totaled 11.4 million tons.

Figure C40 indicates the supply-demand relationships in 1973 and table C16 reveals sulfur-supply-demand data for the period 1964-1974.

The forecasted demand for sulfur in the year 2000 is expected to range from 18-26 million tons. The most probable demand figure is 23 million tons. Sulfur demands for the rest of the world are expected to range from 71-104 million tons in 2000. A summary of forecasted U.S. and world demand is included in table C17.

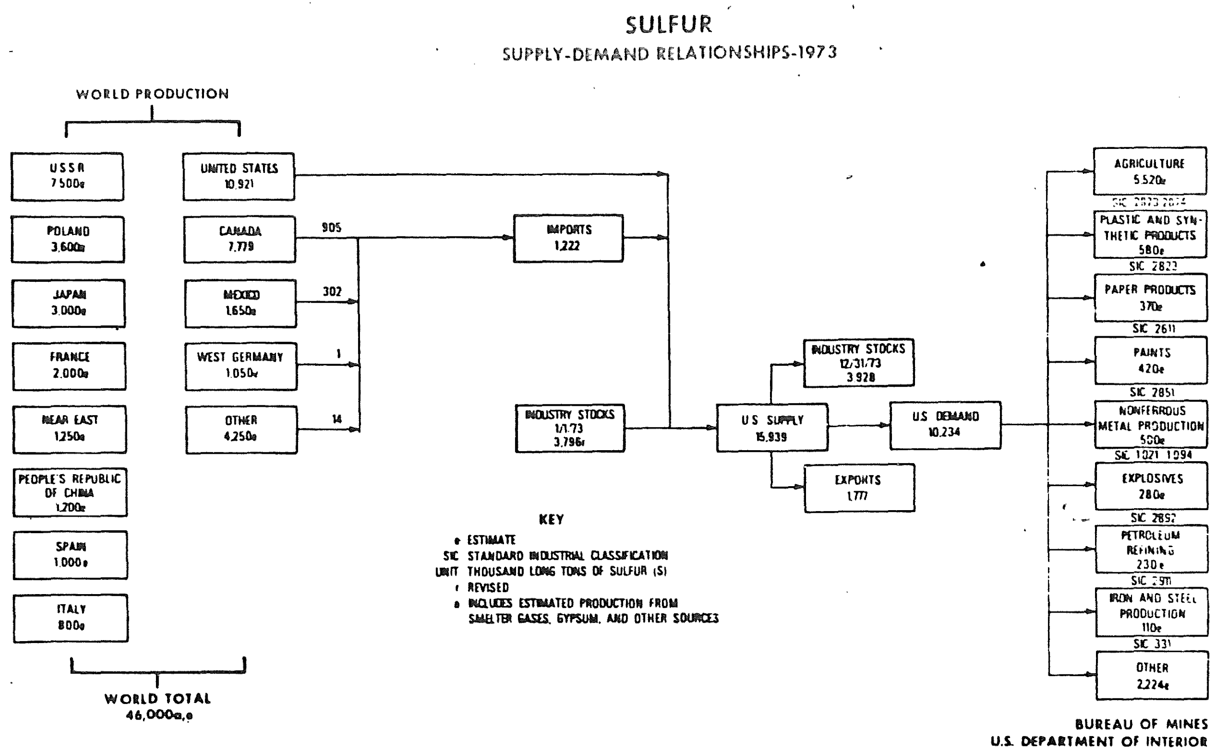


Figure C40 Supply-demand relationships for sulfur, 1973.

Table C16 **Sulfur supply-demand relationships, 1964–74**

(Thousand long tons)

	1964	1965	1966	1967	1968	1969	1970	1971	1972	1973	1974
World production:											
United States	7,087	8,196	9,141	9,121	9,735	9,545	9,557	9,580	10,218	10,921	11,419
Rest of world	20,813	21,804	22,559	24,579	25,565	27,655	29,643	31,320	32,782	35,079	39,481
Total	27,900	30,000	31,700	33,700	35,300	37,200	39,200	40,900	43,000	46,000	50,900
Components of U.S. supply:											
Domestic mines	7,087	8,196	9,141	9,121	9,735	9,545	9,557	9,580	10,218	10,921	11,419
Imports	1,582	1,646	1,674	1,639	1,754	1,795	1,667	1,429	1,188	1,222	2,150
Industry stocks, Jan. 1	4,683	4,403	3,425	2,704	1,954	2,655	3,338	3,829	4,120	3,796	3,928
Total	13,352	14,245	14,240	13,464	13,443	13,995	14,562	14,838	15,526	15,939	17,497
Distribution of U.S. supply:											
Industry stocks, Dec. 31	4,403	3,425	2,704	1,954	2,655	3,338	3,829	4,120	3,796	3,928	3,957
Exports	1,928	2,635	2,373	2,193	1,602	1,551	1,433	1,536	1,852	1,777	2,601
Demand	7,255	7,981	9,145	9,251	9,072	9,169	9,227	9,173	9,854	10,234	10,880
Apparent surplus (+), deficit (−) of supply[1]	−234	+204	+18	+66	+114	−63	+73	+9	+24	----	+59
U.S. demand pattern:											
Agriculture (fertilizers)	3,090	3,810	4,425	4,735	4,470	4,465	4,680	4,800	5,210	5,520	5,980
Plastic and synthetic products	500	555	555	495	550	570	495	515	540	580	610
Paper products	415	430	440	400	390	365	350	320	340	370	390
Paints	505	510	520	505	490	445	420	380	390	420	440
Nonferrous metal production (Cu-U)	235	265	310	260	300	370	390	410	430	500	560
Explosives	215	230	260	265	250	260	255	255	260	280	290
Petroleum refining	155	165	190	195	180	190	195	200	220	230	240
Iron and steel production	330	295	275	220	170	125	120	105	110	110	110
Other	1,810	1,921	2,170	2,176	2,272	2,379	2,322	2,188	2,354	2,224	2,260
U.S. primary demand	7,255	7,981	9,145	9,251	9,072	9,169	9,227	9,173	9,854	10,234	10,880

[1] The difference between total U.S. distribution of supply and total U.S. supply.

Table C17 Summary of forecasts of U.S. and rest-of-world sulfur demand, 1973–2000

(Thousand long tons)

	1973	2000 forecast range		Probable		Probable average annual growth rate 1973–2000 (percent)
		Low	High	1985	2000	
United States.						
Total	10,234	18,000	26,000	14,500	23,000	3.0
Cumulative	----	380,000	460,000	150,000	430,000	----
Rest of world:						
Total	32,766	72,000	104,000	50,500	87,000	3.7
Cumulative	----	1,400,000	1,700,000	500,000	1,500,000	----
World:						
Total	43,000	90,000	130,000	65,000	110,000	3.5
Cumulative	----	1,780,000	2,160,000	650,000	1,930,000	----

Reserves and Resources Based on available information, the world reserves of sulfur amount to 2000 million tons. U.S. reserves total 230 million tons. A major portion of the reserves is in the sour natural gas deposits in the Near East, Canada, and Western Europe. Other large reserves are in the elemental (native) sulfur deposits in Mexico, Poland, the U.S.S.R., and the Near East. Pyrite deposits are an important element of reserves in Western Europe and the U.S.S.R. Additionally, there are substantial reserves in the petroleum deposits in the Near East.

An assessment of world reserves and identified resources is shown in table C18. A similar assessment for the United States by type of deposit is shown in table C19. Quantities are based on complex factors relating to the geological occurrence of sulfur deposits and the technical and economic problems associated with the recovery of sulfur from widely varying sources.

Economics Table C20 shows the time-price relationship for sulfur during the past 22 years in terms of actual and constant 1973 dollars per ton. These prices are based on the average reported prices for elemental sulfur (Frasch and recovered) f.o.b. mine/plant and reflect essentially 90% of the shipments of sulfur in all forms during this period. Rapid increase in the price of sulfur in recent years can be attributed to:

1. rapid expansion in fertilizer manufacturing, both domestically and worldwide

2. a continuing dependency upon Frasch sulfur for the large forward commitments that were required for this purpose
3. the high profitability of the fertilizer sector which permitted high sulfur prices to be passed on to consumers
4. the recognition that Frasch sulfur production costs has increased substantially
5. logistic problems which restricted deliveries from other sources.

Table C18 Identified world sulfur resources

(Million long tons)

	Reserves	Other	Total
North America:			
United States	230	400	630
Canada	410	1,100	1,510
Mexico	90	60	150
Other	5	----	5
Total	735	1,560	2,295
South America	30	30	60
Europe:			
U.S.S.R	150	450	600
Poland	70	10	80
France	35	10	45
West Germany	20	5	25
Spain	20	450	470
Italy	15	25	40
Other	185	285	470
Total	495	1,235	1,730
Africa	20	----	20
Asia:			
Japan	20	40	60
Near East	600	300	900
China, People's Rep. of	25	25	50
Other	50	200	250
Total	695	565	1,260
Oceania	25	10	35
World total	2,000	3,400	5,400

Table C19 Identified U.S. sulfur resources

(Million long tons)

Type of deposit	Reserves	Other	Total
Salt domes and evaporites	150	50	200
Nonferrous metal sulfides	40	60	100
Natural gas	25	10	35
Petroleum	10	135	145
Pyrites	5	55	60
Gypsum and anhydrite	----	50	50
Volcanic	----	30	30
Tar sands	----	10	10
Total	230	400	630

Table C20 Time-price relationship for sulfur[1]

Year	Average annual price, dollars per long ton	
	Actual prices	Constant 1973 dollars
1954	26.65	45.87
1955	27.94	47.44
1956	26.49	43.50
1957	24.41	38.62
1958	23.82	36.76
1959	23.46	35.60
1960	23.13	34.57
1961	23.12	34.10
1962	21.75	31.70
1963	19.99	28.76
1964	20.19	28.64
1965	22.47	31.30
1966	25.77	34.92
1967	32.64	42.83
1968	40.12	50.59
1969	27.05	32.55
1970	23.14	26.42
1971	17.47	19.07
1972	17.03	17.98
1973	17.84	17.84
1974	28.88	26.18
1975 [p]	46.50	38.72
1976 [e] (first quarter)	55.00	43.55

[e] Estimate. [p] Preliminary

[1] Frasch and recovered sulfur.

REFERENCES FOR ILLUSTRATIONS
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