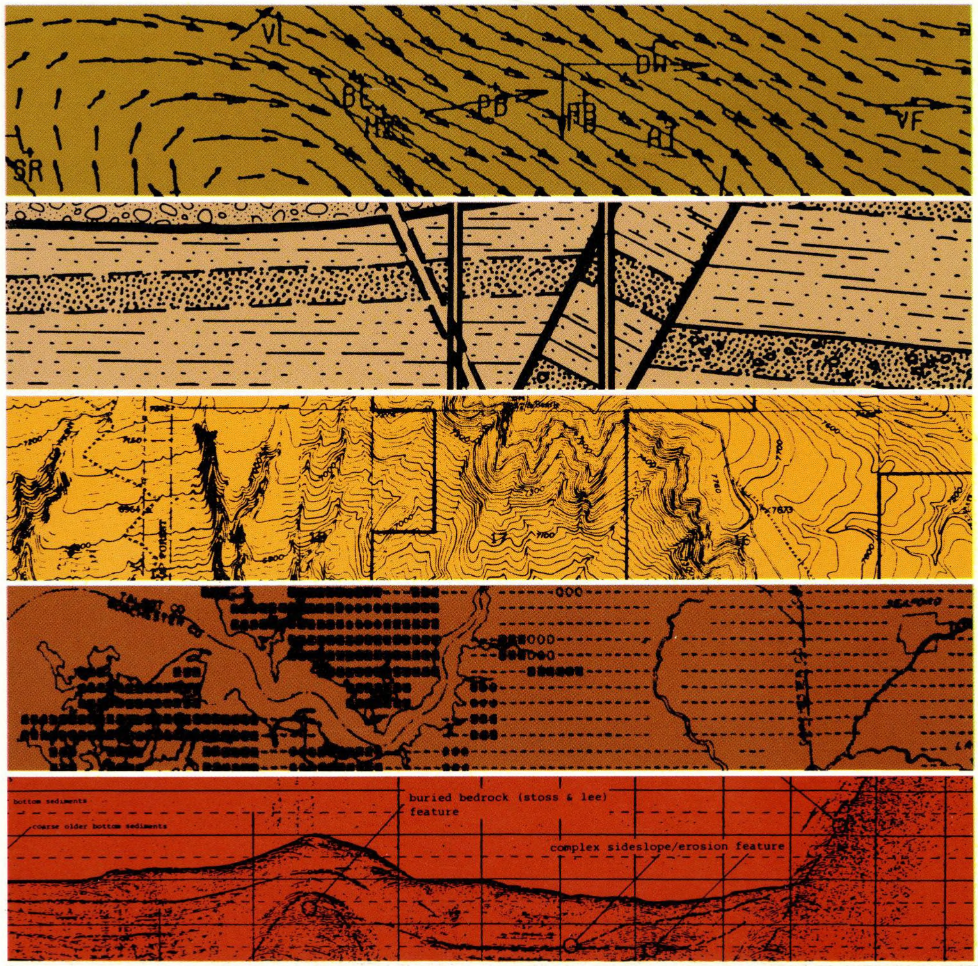
VL
DW
BE
PB
VF
SR
buried bedrock (stoss & lee)
feature
complex sideslope/erosion feature

Dames & Moore

FINAL REPORT

ENVIRONMENTAL, SOCIAL AND ECONOMIC EFFECTS OF CONTINUED RELIANCE ON LAND MINING TO PRODUCE METALS AVAILABLE FROM MANGANESE NODULES

September, 1980

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80 3947

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FOREWORD

This study was funded by the Marine Minerals Division of the National Oceanic and Atmospheric Administration (NOAA) as part of their investigations into the developing deep seabed mining industry. NOAA is presently preparing a programmatic environmental impact statement (PEIS) and draft regulations intended to promote orderly development of the industry. This study provides information on the likely effects of continued reliance on land mining to produce copper, nickel, cobalt, and manganese if deep seabed mining does not take place or is greatly delayed. The information will assist in the PEIS process in two ways: it will help to place in perspective the environmental effects associated with deep seabed mining and it will help to elucidate the need for deep seabed mining.

The scope of this investigation is limited to considerations of land mining and processing (to commercial grade purity) of the four metals available from manganese modules. Assessment of, or comparison with, deep sea mining and processing is not included. These topics are the subject of other studies which will form the basis for future decisions about potential environmental impacts and regulatory control philosophy. This study is not an investigation of the economic conditions which might prevail in the future, or which might be necessary to the existence of a significant large-scale deep seabed mining industry.

The report is divided into three parts. Part I is an investigation of the expected demand and supply for copper, nickel, cobalt, and manganese during the period 1980 through 2010. This supply/demand analysis is based on an independent review of other projections, interviews with government and private industry minerals experts, and industry plans for new or expanded mines and process facilities worldwide. Whenever possible, both supply and demand estimates are given for individual countries, market economies, and central economy countries. Three levels of demand are projected - low, high, and most likely - and compared to present reserves, resources, and expected mine capacity.

Part II describes the current methods used in mining and processing copper, nickel, cobalt, and manganese ore deposits. Future technological advancements which may materially affect the method or efficiency of mining and processing are also considered.

Part III is an analysis of environmental and socioeconomic effects of continued reliance on land mining and processing during the period 1980 through 2010. Because of the great difficulty in predicting where such activities would take place, and consequently what effects may occur, the analysis was divided into (1) a quantitative, non-site specific, aggregate impact analysis and (2) a site specific review of conditions at a typical array of existing or soon-to-be developed mine and process facilities. Also included is a description of generic types of environmental effects which normally accompany the proposed activities. Part III is developed using supply and demand forecasts of Part I and the mining and processing techniques described in Part II.

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PART I

SUPPLY OF AND DEMAND FOR METALS AVAILABLE FROM MANGANESE NODULES

September, 1980

Dames & Moore

PREPARED FOR

NATIONAL OCEANIC
AND ATMOSPHERIC ADMINISTRATION
U.S. DEPARTMENT OF COMMERCE
WASHINGTON, D.C.

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1.0 COBALT: SUPPLY-DEMAND

1.1 INTRODUCTION

To determine whether world cobalt reserves and resources are sufficient to meet projected demand for this metal during the period 1985-2010, the economic geology of cobalt, as well as the factors which could, and do impact on its production must be clearly understood. Unlike many other metals of strategic importance, the future availability of cobalt derived from land-based sources of supply will be contingent not only in large part on the demand for other metals--in this case copper and nickel, but also on geologic, technical, and political considerations.

In Part I of the discussion which follows, the economic geology of cobalt is reviewed. The six major geologic environments in which cobalt occurs are discussed and examples of economically significant deposits belonging to each generic type described. Part II considers the mode of occurrence and geologic setting of the major cobalt deposits of nations known or thought to have significant reserves - resources of this metal. Each major cobalt-bearing deposit or district already in production, or likely to be brought on-stream during the next thirty years, is discussed from the standpoint of existing and/or potential methods, rates, and timing of exploitation.

1.2 COBALT SUPPLY

1.2.1 Part I - The Economic Geology Of Cobalt

An important factor in any discussion of minerals availability is the definition of resource terminology. The most commonly utilized system divides resources into two categories: identified resources, and undiscovered resources. An identified resource is one whose location, quality, and quantity are known from geologic evidence supported by actual measurements. That portion of identified resources from which a metal can be economically and legally extracted at the time of determination is said to be a mineral reserve. Undiscovered resources are generally placed in one of two categories: hypothetical resources, and speculative resources. Hypothetical resources are those that may be reasonably expected to exist in a known mining district under known geologic conditions. Speculative resources may occur either in known types of deposits in a favorable geologic setting where no discoveries have been made, or in as yet unknown types of deposits that remain to be recognized (Figure 1-1).

Although cobalt occurs in a wide variety of geologic environments, economic occurrences of this metal are uncommon. From a geochemical standpoint, the principal concentrations of cobalt occur in mafic and ultramafic rocks (Table 1-1). However, with regard to the discovery of mineable deposits, cobalt is most often found in economic concentrations in association with copper, nickel, and/or silver mineralization. Consequently, nearly all of the world's cobalt production is derived as a byproduct of the mining of these and other base and precious metals (for example, chromium, iron, lead, uranium, and zinc).

Cobalt occurs as a major or trace constituent in several hundred minerals. However, fewer than ten of these minerals are of economic significance (Table 1-2). The cobalt minerals of greatest economic importance are the pyritic cobalt-bearing minerals cobaltite, linnaeite, and carrollite; the cobalt arsenides skutterudite and safflorite; the cobalt oxides heterogenite

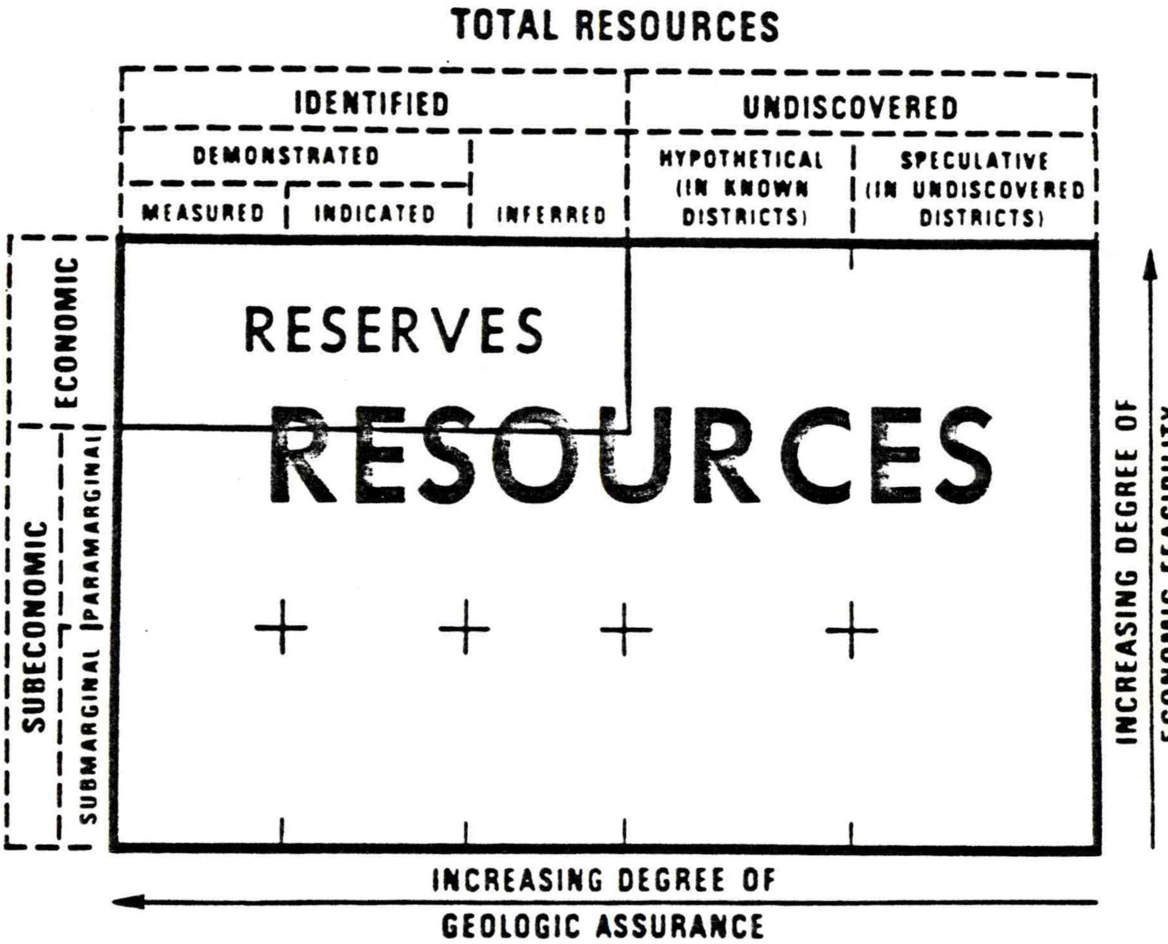


Figure 1-1

Source: USGS, "Principles of the Mineral Resource Classification System of the U.S. Bureau of Mines and the U.S. Geological Survey," Geological Survey Bulletin 1450-A, 1976.

Table 1-1

AVERAGE COBALT CONTENT OF IGNEOUS ROCKS

Rock Type (listed in order of increasing differentiation)	Cobalt (ppm)	Nickel (ppm)	Nickel/ Cobalt
Ultramafic rocks......................	270	1,900	7
Gabbro................................	51	133	2.6
Basalt................................	41	102	2.5
Diabase...............................	31	65	2.3
Intermediate igneous rocks............	14	27	1.9
Felsic rocks..........................	5	5.7	1.1

Source: USGS Professional Paper 820, 1973, p. 145.

Table 1-2

MAJOR COBALT MINERALS OF ECONOMIC IMPORTANCE

Mineral	Percent Cobalt
Linnaeite, Co_3S_4	58.0 (theoretical)
Siegenite, $(Co,Ni)_3S_4$	20.4 - 26.0
Carrollite, $(CO_2Cu)S_4$	35.2 - 36.0
Cobaltite, (Co,Fe)AsS	26.0 - 32.4
Safflorite, $CO,Fe)As_2$	13.0 - 18.6
Glaucodot, (Co,Fe)AsS	12.0 - 31.6
Skutterudite, $(Co,Fe)As_3$	10.9 - 20.9
Heterogenite, CoO (OH)	64.1 (theoretical)
"Asbolite," (Manganese oxides + Co)	0.5 - 5.0
Erythrite, $(Co,Ni)_3(AsO_4)_3$ $8H_3O$	18.7 - 26.3
Gersdorffite, (Ni,Co)AsS	(low)
Pyrrhotite, $(Fe,Ni,Co)x\text{-}1^S$ x	1.00 (maximum)
Pentlandite, $(Fe,Ni,Co)_3S_4$	1.50 (maximum)
Pyrite, $(Fe,Ni,Co)_9S_8$	13.00 (maximum)
Sphalerite, Zn(Co)S	0.30 (maximum)
Arsenopyrite, Fe(Co)AsS	0.38 (maximum)
Manganese oxide minerals	0.10 - 1.00 (or more)

Source: USGS Professional Paper 820, 1973, p. 146.

and asbolite; the cobalt arsenate erythrite; and, when cobaltiferous, the copper, nickel and iron sulfides pyrite, pentlandite, pyrrhotite, and bravoite.

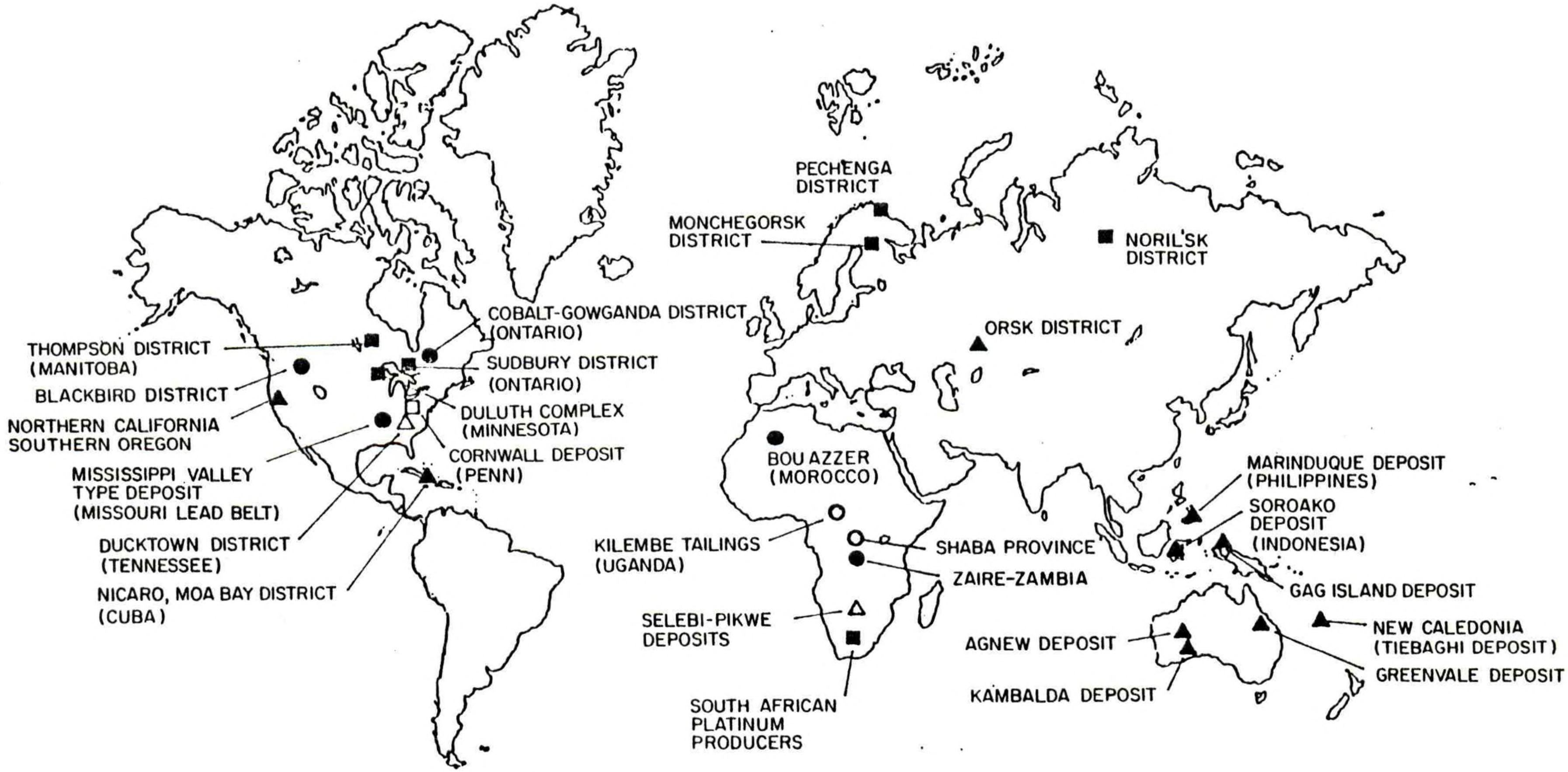
Seven types of cobalt deposits have been identified by the United States Geological Survey (Professional Paper 820). These classifications, which are based on the environment of genesis or occurrence of known deposits, are as follows:

- o Hypogene deposits associated with mafic intrusive igneous rocks.
- o Contact metamorphic deposits associated with major intrusive rocks.
- o Laterite deposits.
- o Massive sulfide deposits in metamorphic rocks, chiefly of volcano-sedimentary origin.
- o Hydrothermal deposits of several varieties.
- o Stratabound deposits.
- o Deposits formed as chemical precipitates.

Significant examples are shown on Figure 1-2.

With the exception of the last classification (deposits formed as chemical precipitates) which refers to cobalt contained in deep-sea manganese nodules, the geologic setting and economic geology of selected major cobalt-bearing deposits of each of the six listed types will be discussed in the following section.

IMPORTANT COBALT RESERVES & RESOURCES



6 TYPES OF COBALT DEPOSITS

- ■ HYPOGENE DEPOSITS ASSOCIATED WITH MAFIC INTRUSIVE IGNEOUS ROCKS
- □ CONTACT METAMORPHIC DEPOSITS ASSOCIATED WITH MAJOR INTRUSIVE ROCKS
- ▲ LATERITE DEPOSITS
- △ MASSIVE SULFIDE DEPOSITS IN METAMORPHIC ROCKS
- ● HYDROTHERMAL DEPOSITS
- ○ STRATABOUND DEPOSITS

Figure 1-2

1.2.1.1 Hypogene Deposits Associated With Mafic Intrusive Igneous Rocks

Cobaltiferous hypogene deposits associated with mafic intrusive igneous rocks are found in many regions of the world. The principal deposits of this type occur at Sudbury (Ontario, Canada), Noril'sk (U.S.S.R.), and in the vicinity of Ely, Minnesota (the Duluth Gabbro Complex).

1.2.1.1.1 Sudbury District, Ontario, Canada

The cobaltiferous copper-nickel sulfide orebodies of the Sudbury district, Ontario, occur at or near the margin of the Sudbury Irruptive---an elliptically shaped, layered body consisting essentially of micropegmatite underlain by quartz gabbro and quartz norite. The Sudbury ores are comprised of massive sulfide bodies, breccia sulfides, veins, and disseminated or stringer sulfides in silicates. The principal sulfide minerals are pyrrhotite, pentlandite, chalcopyrite, and cubanite. Although cobalt is generally intimately and uniformly mixed throughout the nickeliferous sulfides, particularly pentlandite, it also occurs as discrete cobaltiferous sulfides among which are the minerals of the cobaltite-gersdorffite series (CoAsS-NiAsS). The average nickel and cobalt contents of Sudbury ore are 1.5 percent and 0.07 percent, respectively; the nickel to cobalt ratio varies from 20:1 to 33:1. (For a more complete discussion of the economic geology of the Sudbury district, refer to Section 4 of Part I).

1.2.1.1.2 Thompson District, Manitoba, Canada

Cobaltiferous nickel deposits associated with ultramafic rocks are found in the Thompson district of Manitoba. The deposits occur along a 90 mile trend in or near lenticular pods of alpine-type peridotite within metasediments and gneisses. The biotite schist in which the sulfides occur is as much as 200 to 300 feet wide. The orebody, which pinches and swells from 2 to 85 feet, is conformable with the enclosing sediments. This would suggest that the nickel-cobalt mineralization was derived from the peridotite. Unlike the ores of the Sudbury district, the nickel to copper ratio of the Thompson district is high, in excess of 15:1.

1.2.1.1.3 Duluth Complex, Minnesota, United States

The Duluth Complex consists of a series of sheet-like intrusions into and beneath the Precambrian Keweenawan volcanics. The major portion of all known copper-nickel mineralization in the Duluth Complex occurs in the lowermost several hundred feet of the intrusive in a series of gabbroic rocks which in large part comprise what is termed the "basal zone." The copper-nickel mineralization associated with the Duluth Complex exists as either disseminated or massive sulfides, the former occurrence being of greater economic significance than the latter (despite the fact that the massive zones are known in some instances to contain over 10 percent copper). The primary minerals are pyrrhotite, chalcopyrite, cubanite, pyrite, sphalerite, and bornite. According to the Minnesota Department of Natural Resources (MDNR), the average grade of the disseminated mineralization can be considered for the basis of ore estimation to be 0.66 percent copper and 0.20 percent nickel. Associated with the copper-nickel mineralization of the Duluth Complex are cobalt, gold, silver, and platinum group metals--all in possibly recoverable quantities.

1.2.1.1.4 Noril'sk District, U.S.S.R.

The cobaltiferous copper-nickel sulfide mineralization of the Noril'sk district consists primarily of segregations of pyrrhotite, pentlandite, and chalcopyrite which occur as a relatively persistent layer in a differentiated gabbro-dolerite intrusion. Detailed data are not available concerning the geology of the Noril'sk district. However, one deposit is reported to contain 0.75 percent copper, 0.5 percent nickel, and 0.313 percent cobalt.

1.2.1.2 Contact Metamorphic Deposits Associated With Major Intrusive Rocks

Contact metamorphic deposits include those occurrences of magnetite, chalcopyrite, and cobaltiferous pyrite formed by contact metamorphism of carbonate rock by diabase intrusives. The Cornwall deposit in Pennsylvania is perhaps the best known of the cobalt-containing occurrences of this type.

1.2.1.2.1 Cornwall Deposits, Pennsylvania, United States

The Cornwall deposits consist of two major iron orebodies and numerous smaller pods of mineralization which occur as limestone replacements above a diabase sheet. The major metallic minerals in order of decreasing abundance are magnetite, pyrite, chalcopyrite, and hematite. Although cobalt has been detected through spectrographic analysis in the magnetite, the majority of the cobalt exists in the pyrite, which, in numerous instances, has been found to contain more than 2 percent cobalt. Prior to the closure of operations at Cornwall, mine run ore averaged between 0.02 and 0.056 percent cobalt.

1.2.1.3 Laterite Deposits

Cobaltiferous nickel-laterite deposits occur in many areas throughout the world. These deposits, which often contain 40 to 50 percent iron, 1 to 2 percent nickel, and 0.01 to 0.10 percent cobalt, are formed by the weathering of peridotite and serpentine bodies. The largest and, at present, the most commercially significant cobaltiferous nickel deposits are located in Australia, eastern Cuba, Indonesia, New Caledonia, and the Philippines. Additionally, deposits of potential economic importance occur in northern California and southern Oregon, United States. Because the economic geology of lateritic nickel deposits will be discussed in detail in Section 4, Part I, there will be no review of this type of occurrence in this section.

1.2.1.4 Massive Sulfide Deposits In Metamorphic Rocks

Relatively few economically or geologically significant deposits of cobaltiferous massive sulfides in metamorphic rocks have been discovered. As is the case for most cobalt occurrences, the cobalt contained in those massive sulfide deposits identified to date is associated with other mineralization--most commonly copper, iron, nickel and zinc. The Ducktown district of Tennessee is the only major United States deposit of this type.

1.2.1.4.1 Ducktown District, Tennessee

Eight copper-zinc orebodies of varying size and tonnage have been discovered in the highly folded and metamorphosed Precambrian meta-graywackes and micaceous schists of the Ducktown district. Each of these deposits is generally tabular in shape, and is conformable with the regional trend of the strike and dip of the bedding of the host rock. The major sulfide minerals of the district are pyrrhotite, pyrite, chalcopyrite, sphalerite and magnetite. Although trace amounts of cobalt have been detected in the ore, no specific cobalt minerals have been identified. However, recent studies of the mineralogy at Ducktown indicate that pyrite (which is assumed to be the principal host of the cobalt) can contain as much as 0.4 to 0.5 percent cobalt. To date, no effort has been made to recover the cobalt from the sulfide concentrates.

1.2.1.5 Hydrothermal Deposits

Hydrothermal deposits of cobalt constitute one of the world's major sources of cobalt. Hydrothermal cobalt deposits are of two major types: vein deposits, and replacement deposits. The principal domestic vein deposit is located in the Blackbird district, Idaho. Other major vein deposits occur in the Cobalt-Gowganda region of Ontario, Canada; Bou Azzer, Morocco; and in the Soviet Union. Significant replacement deposits of cobalt are located at Outokumpu, Finland, and are associated with lead-zinc deposits of the United States Missouri Lead Belt.

1.2.1.5.1 Blackbird District, Idaho

The Blackbird district of east-central Idaho contains numerous narrow pods of copper-cobalt mineralization which occur in the sheared and faulted quartzites, schists, and phyllites of the Precambrian Yellowjacket Formation. The degree of foliation and shearing of the host rocks is believed to be of importance in the localization of the copper-cobalt deposits. The principal sulfide minerals of the Blackbird deposit are cobaltite and chalcopyrite with associated pyrrhotite and pyrite (much of which also contains cobalt). The potential copper-cobalt reserves of the Blackbird district are believed to be large. The average grade of copper in the district is approximately 1.3 percent; the cobalt content of the ore varies from 0.5 to 0.8 percent.

1.2.1.5.2 Cobalt And Gowganda Districts, Ontario

The cobaltiferous nickel-silver deposits of the Cobalt and Gowganda districts of Ontario occur in steeply dipping Keewatin basic and intermediate lavas (greenstones) and interflow sediments; the Precambrian conglomerates, graywackes, and quartzites of the Coleman Formation; and the Nipissing diabase sill. The deposits consist of relatively short veins; a typical orebody is comprised of several anastomosing veins varying from a few tenths of an inch to a foot or more in width. Native silver and the cobalt-nickel arsenides smaltite-chloanthite, safflorite, rammelsbergite, and niccolite are the most abundant and characteristic minerals in the Cobalt district deposits.

In many instances, the silver-nickel-cobalt minerals are intimately intergrown. Although cobalt grades of up to 10 percent are common, the average grade of all ore is considerably lower.

1.2.1.5.3 Bou Azzer, Morocco

Numerous occurrences of cobalt mineralization have been located in the Bou Azzer district of southern Morocco. The majority of the deposits occur either adjacent to, and/or along a contact between serpentinite and a Pre-cambrian diorite with granite and mica schist. The mineralization exists as veins, some of which are over 1,600 feet long and vary in thickness from several feet to over 100 feet. The principal minerals of economic interest are the cobalt arsenides and arsenates: skutterudite, erythrite, cobalt-bearing loellingite as well as chalcopyrite, molybdenite, and gold--which occurs not only as microscopic inclusions in the arsenides, but is also disseminated locally throughout the deposits. According to Battelle Memorial Institute (1960), typical ore from the Bou Azzer Vein Number 7 (one of the district's major deposits) contains 14 percent cobalt, 1.2 percent nickel, and 23 grams per ton gold. At the Aghbar deposit, the ore averages 9 percent cobalt, 1.2 percent nickel, and 4 grams per ton gold. The average grade of the district, however, is believed to be 1.2 percent cobalt.

1.2.1.5.4 Dashkesan and Magadan Districts, U.S.S.R.

At least two major (from the standpoint that they are believed to be in production) Soviet cobalt deposits have been described by Soviet geologists as being hydrothermal vein deposits of the "Blackbird-type". These occurrences are: the Northern deposit in the Dashkesan region in Azerbardzhan S.S.R. and, the Seimchan group of deposits in the Magadan district. At the Northern deposit, cobalt occurs as "segregation veinlets and lenses" along a "fracture-gouge clay chlorite" zone in a "granitoid intrusion" and diabase dikes. Mineralization, consisting of cobaltite, glaucodot, and cobalt-bearing arsenopyrite, is reported to extend over a mile along strike and nearly 2,000 feet down-dip.

The Upper Seimchan deposits occur in metamorphosed sediments which have been intruded by granites and ultramafics. Cobalt mineralization, consisting primarily of cobaltite with associated skutterudite and smaltite, occurs as fracture and fault controlled disseminations and veinlets in and adjacent to faults and fractures.

1.2.1.5.5 Cobalt Associated With Mississippi Valley Lead-Zinc Deposits

Significant concentrations of cobalt and nickel exist in association with lead-zinc mineralization of the Missouri Lead Belt. The deposits of the Missouri Lead Belt are stratabound in character and occur in every formation from the Cambrian Lamotte Sandstone to the lower Ordovician, partly dolomitized Jefferson City limestone. The highest cobalt-nickel grades tend to occur in those lead-zinc deposits which are in the lower part of the Bonneterre Dolomite, particularly where it transitions with the Lamotte Formation (mainly a fine grained pure orthoquartzite with occasional siltstone sections).

1.2.1.6 Stratabound Deposits

Stratabound cobaltiferous deposits occur in many diverse geologic environments. As in the case of virtually all other types of cobalt occurrences, stratabound orebodies are generally mined not for their cobalt content, but for some other metal such as copper, lead or zinc. From a tonnage standpoint, the most significant cobaltiferous stratabound deposits are the copper-cobalt occurrences of Zaire and Zambia, and the Kupferschiefer copper-rich shales of central Europe.

1.2.1.6.1 Zairian (Katangan) And Zambian (Rhodesian) Copper-Cobalt Deposits

Zaire. The copper-cobalt deposits of Shaba Province of Zaire (formerly known as the Belgium Congo and the Republic of the Congo) are distributed in three groups (West, Middle, and East Sectors) along a 200 by 40 mile wide curve extending from Kolwezi in the northwest to the region southeast of Lubumbashi (Figure 1-3). Zairian copper-cobalt deposits occur primarily as individual layers of finely disseminated, evenly distributed sulfides and oxides which extend throughout the Serié des Mines Formation of the Precambrian Katanga Group--a continuous sedimentary series consisting predominately of folded, sheared, and fractured carbonates and fine clastics.

A notable feature of Zairian copper-cobalt mineralization is its lack of uniformity. Not only do the metal contents of the ore-bearing strata of Zaire vary considerably (from 0.1 to 0.5 percent cobalt to as much as 1.0 to 2.0 percent cobalt), but economically exploitable zones of the Serié des Mines are often laterally separated by non-ore bearing facies. Mineralized zones vary from 12 to 45 feet in thickness. The major cobalt-bearing minerals are carrollite, linneaite, heterogenite and asbolite. In addition to cobalt and copper, Zairian ores contain small quantities of gold, platinum group metals, and selenium.

Zambia. Zambian copper-cobalt deposits are located along a 125-mile-long belt in the predominately fine to coarse clastics of the Serié des Mines

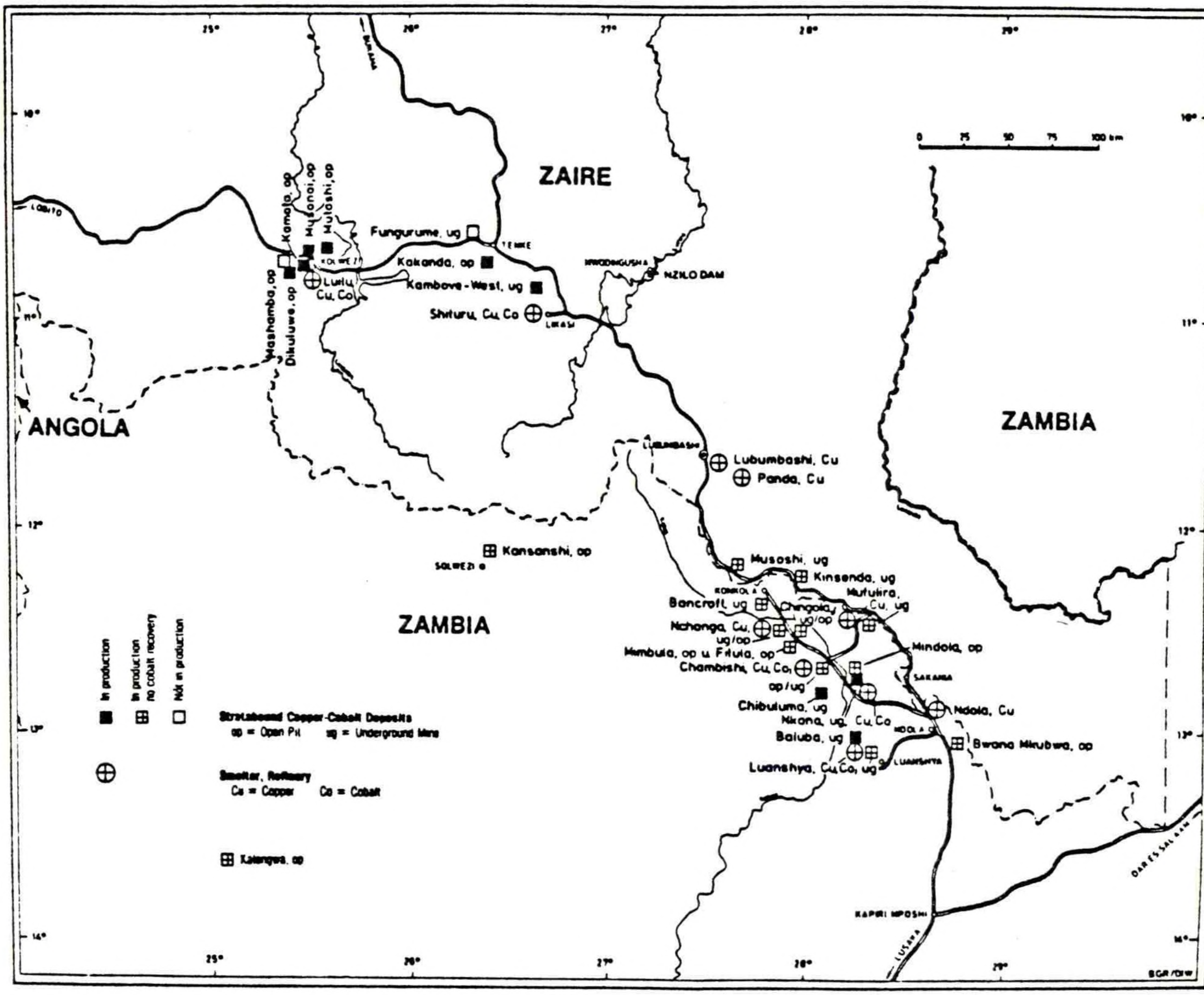


Figure 1-3
COPPER-COBALT DEPOSITS IN ZAIRE AND ZAMBIA

Source: Kruszona et al., 1979, Appendix 1.

Formation (Figure 1-4). As in the case of the Zairian deposits, Zambian copper-cobalt mineralization is finely disseminated as thin layers throughout the host rock. The principal cobalt-bearing minerals are carrollite and linnaeite; the average grade of the orebodies is 0.1 to 0.18 percent cobalt and 2.5 to 5.0 percent copper.

1.2.1.6.2 Kupferschiefer

The Kupferschiefer Formation near Mansfeld, Germany, is a thin (approximately 2 feet thick), widespread (underlies much of northern Europe) bed of black shale (with alternating layers of carbonate, clay, and organic matter) which contains an average of 0.3 percent copper and lesser amounts of cobalt, lead, molybdenum, nickel, selenium, silver and vanadium. At some locations, for example, the Lubine district of Poland, the average copper content of the Kupferschiefer is 1.5 percent. Throughout much of Europe, the Kupferschiefer is overlain by a thick sequence of limestones and evaporites.

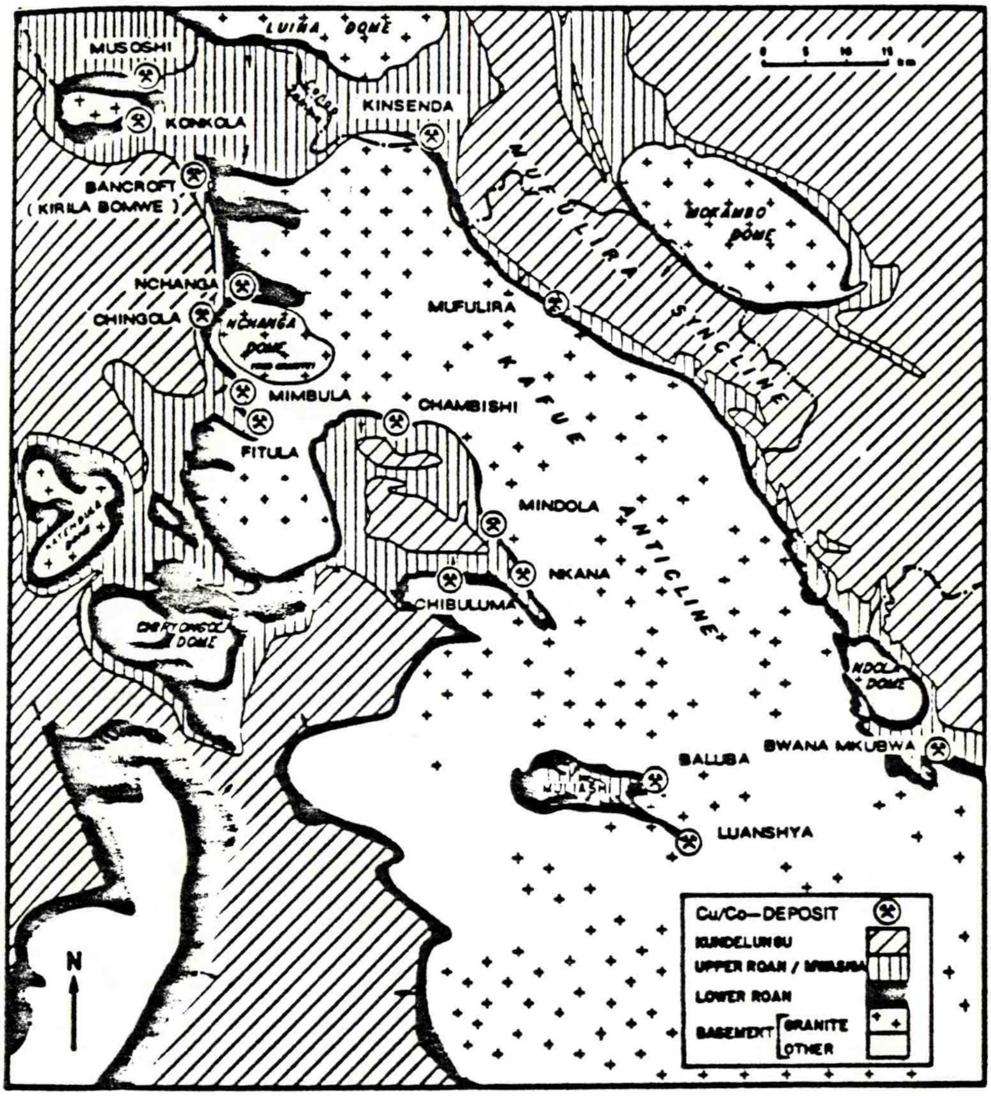
1.2.2 Part II - Cobalt Reserves and Resources

World cobalt reserves and resources (exclusive of cobalt associated with deep-sea manganese nodules) as determined by the Federal Institute for Earth Sciences and Raw Materials (West Germany) and the German Institute for Economic Research are presented in Table 1-3. Table 1-3 also gives estimates of world cobalt reserves and resources compiled by the United States Geological Survey (USGS) in conjunction with the United States Bureau of Mines (USBM). As is evident from a comparison of the reserve-resource data, significant differences exist between the German and American estimates.

In many instances, the estimates made by the USGS-USBM of the cobalt reserves-resources of a particular nation are similar to the estimates prepared by the German group (for example, Canada, Cuba, Zaire and Zambia). What discrepancies may exist between USGS-USBM and German total estimated reserves-resources are largely due to differences in the parameters established by the two groups for each reserve-resource category.

Figure 1-4

GEOLOGICAL STRUCTURE OF THE ZAMBIAN COPPERBELT



Source: Kruszona et al., 1979, p. 37.

Table 1-3

ESTIMATED WORLD RESERVES-RESOURCES OF
COBALT
(thousand short tons)

	U.S. BUREAU OF MINES, 1977			GERMAN INSTITUTE FOR ECONOMIC RESEARCH (Kruszona, et. al., 1979)		
	Estimated Reserves	Other Identified Resources	Total Identified Resources	Proven & Probable Reserves	Potential Reserves	Proven, Probable & Potential Reserves
NORTH AMERICA						
Canada	33.00	242.00	275.00	242.44	33.06	275.50
United States	--	842.00	842.00	--	440.80	440.80
TOTAL	33.00	1084.00	1117.00	242.44	473.86	716.30
SOUTH AND CENTRAL AMERICA						
Brazil				33.06	77.14	110.20
Columbia				--	33.06	33.06
Cuba	120.00	1036.00	1156.00	881.60	330.60	1212.20
Dominican Republic				--	55.10	55.10
Guatemala				16.53	--	16.53
Puerto Rico				--	49.59	49.59
Venezuela				--	16.53	16.53
TOTAL	120.00	1036.00	1156.00	931.19	562.02	1493.21
EUROPE-CHINA						
Albania				--	5.51	5.51
China				--	22.04	22.04
Finland	20.00	5.00	25.00	22.04	5.51	27.55
Greece				--	220.40	220.40
Spain				--	22.04	22.04
USSR	230.00	20.00	250.00	220.40	165.30	385.70
Yugoslavia				--	82.65	82.65
TOTAL	250.00	25.00	275.00	242.44	523.45	765.89
OCEANIA-ASIA						
Australia	54.00	271.00	325.00	148.77	181.83	330.60
Burma				--	16.53	16.53
India				49.59	--	49.59
Indonesia				622.63	330.60	953.23
New Caledonia	300.00	125.00	425.00	424.27	495.90	920.17
Papua-New Guinea				--	44.08	44.08
Philippines	210.00	15.00	225.00	468.35	1928.50	2396.85
Solomon Islands				--	22.04	22.04
TOTAL	564.00	411.00	975.00	1713.53	3019.48	4733.09
AFRICA						
Botswana	29.00	6.00	35.00	27.55	5.51	33.06
Morocco	14.00	1.00	15.00	11.02	5.51	16.53
Rhodesia				33.06	--	33.06
Uganda				11.02	11.02	22.04
Zaire	500.00	250.00	750.00	330.60	253.46	584.06
Zambia	125.00	258.00	383.00	495.90	55.10	551.00
TOTAL	668.00	515.00	1183.00	909.15	330.60	1239.75
WORLD TOTAL	1635.00	3071.00	4706.00	4038.75	4909.41	8948.24
GEOPOLITICAL SUB-UNITS						
United States		842.00	842.00	--	440.80	440.80
Central Economy Countries	350.00	1056.00	1406.00	1102.00	606.10	1708.10
World exclusive of U.S.	1635.00	2229.00	3864.00	4038.75	4468.61	8507.44
World exclusive of Central Economy Countries	1285.00	2015.00	3300.00	2936.75	4303.31	7240.14

However, in instances where major differences exist between the two estimates of total identified resources for a particular nation (for example, Indonesia, New Caledonia and the Philippines), the German data are believed to be a more reliable indicator of the probable reserve-resource position of that country. There are several reasons why the German data will be used in this study rather than USGS-USBM estimates:

- o Most of the producers and consumers of cobalt contacted in the conduct of this investigation were strongly of the opinion (based on their perception of the world cobalt supply) that the data prepared by the German group more accurately reflect the current world cobalt reserve-resource situation.

- o During the course of this investigation large reserves-resources of cobalt were documented which were excluded from the USGS-USBM estimates but included in the German effort. Consequently, for several nations, USGS-USBM reserve-resource estimates are incomplete, and thus, are only indicative of the theoretical world cobalt supply situation.

1.2.2.1 Estimates of World Cobalt Reserves-Resources

As indicated in Table 1-3, United States, market economy nations, and total-world cobalt reserves and resources are restricted to relatively few nations. The following facts serve to substantiate this point:

- o Of total world proved, probable, and potential reserves, three-fourths (6.67 million tons) exist in only six countries (Cuba, Indonesia, New Caledonia, the Philippines, the United States and Zaire).

- o Of the estimated 4.04 million tons of proved and probable cobalt reserves, approximately 3.83 million tons, or 91 percent, occur in only nine nations, two of which, Cuba and the Soviet Union, account for over one-fourth of the world's total.

- Forty-six percent of all proved and probable reserves of cobalt exist in just three countries, Cuba, Indonesia and Zaire.

- With regard to potential cobalt reserves, nearly 40 percent of total world reserves occur in the Philippines.

1.2.2.2 Geological Classification of World Cobalt Reserves-Resources

From a geologic standpoint, world cobalt reserves and resources can be divided essentially into one of two broad classifications: cobaltiferous sulfides and/or lateritic ores. As indicated on Table 1-4, nearly three-fourths of the world's identified cobalt reserves and resources occur with nickel laterites; the remaining one-fourth is contained in copper-nickel sulfide ores. As discussed in Section 1.2.1, less than 1 percent of the world's cobalt reserves and resources exist as deposits in which cobalt is the major or primary metal of economic interest.

1.2.2.3 Factors Important in Assessing World Cobalt Availability

Although the total world cobalt reserve-resource base is moderately large vis-a-vis other strategic metals, the critical question concerning its supply-market availability is whether sufficient production capacity either exists, or is likely to be brought on-line in the future to meet present and anticipated demand. From Section 1.2.1 and Table 1-4, it is apparent that virtually all of the world's cobalt production is derived as a byproduct or coproduct of the mining of nickel and/or copper ores. Consequently, assuming that no new discoveries are made of cobalt deposits similar to the Blackbird or Bou Azzer orebodies, the availability of cobalt on the market is, and will likely continue to be, largely contingent on the demand for nickel and copper.

To determine whether a company can or will produce cobalt from its copper and/or nickel orebodies, a number of factors must be considered, among which the most important are:

Table 1-4

WORLD COBALT PRODUCTION AND RESERVES BY GEOLOGIC TYPE

		Reserves	
	Production	proved and probable	potential
Nickel-copper deposits (Sudbury type)	21	14	14
Copper-cobalt deposits (Katanga type)	55	21	7
Vulcanogenic-sedimentary	7	1	1
Nickel laterite ore deposits	17	64	78
World total	100	100	100

Source: Kruszona et al., 1979, p. 52.

o Does the particular copper and/or nickel orebody contain cobalt?

As discussed earlier in this chapter, not all copper and/or nickel occurrences are cobaltiferous. For example, whereas the nickel laterites of Cuba and New Caledonia contain cobalt, those of South America, for the most part, are deficient in this metal.

o Can the cobalt contained in cobaltiferous copper and/or nickel ores be extracted?

Whether the cobalt contained in copper and/or nickel ores can be recovered is contingent not only on economic considerations, but also on the extractive technology utilized at the concentrator and smelter. Because cobalt nearly always is obtained as a byproduct of the mining and smelting of other metals, the extraction of cobalt is usually considered to be of secondary importance; the primary emphasis almost without exception is placed on achieving maximum recovery of the major metals, for example copper and nickel. From a metallurgical standpoint, cobalt can be extracted from cobaltiferous nickel laterites only if hydrometallurgical processes are used to beneficate the ores. Should other methods be employed to process cobaltiferous nickel laterites, recovery of the contained cobalt is significantly more difficult--if not impossible. If the end product of beneficiation is nickel matte, only limited extraction of the contained cobalt is possible. When ferro-nickel is produced as the principal end-product, cobalt separation cannot be accomplished. Insofar as 55 percent of the world's nickel derived from laterites is processed into ferro-nickel (and because a large number of planned nickel mine-mill-smelter complexes are designed to produce ferro-nickel), much of the cobalt content in lateritic concentrator feed is not and will not be recoverable. Hence, much of the cobalt classified as "potentially recoverable world cobalt reserves and resources from nickel laterites" will not impact on future cobalt supply.

As discussed in Section 1.2.3.4, cobalt recovery from copper-bearing ores is often extremely low. For example, in Zaire and Zambia, only one-half of the cobalt originally contained in the concentrator feed is recovered. Consequently, estimates of world cobalt reserves and resources, as well as data concerning world mine cobalt production, are only indicative, at best, of the total potential market availability of the metal.

Table 1-5 gives world mine production of cobalt by country during the period 1975-1979. As is evident from this table, world cobalt production has been relatively stable over the past five years. This is somewhat surprising in light of the depressed demand for nickel, and to a lesser extent, for copper during several of the years under consideration (nickel: 1974-1979; copper: 1975-1978). It is important to note that mine production of cobalt is not equivalent to world production of cobalt metal. As indicated at the bottom of Table 1-5, for every ton of cobalt metal produced for world trade, approximately 1.4 to 1.5 tons of contained cobalt metal in ore must be mined. Assuming 100 percent extractive efficiency (that is, all of the cobalt believed to be contained in either reserves or resources could be recovered--which is almost certainly not possible), cobalt reserves-resources provide only a rough guide to the adequacy of total estimated world cobalt resources to meet projected cumulative demand for the metal.

1.2.3 World Cobalt Production

In the section which follows, the economic geology and the existing and projected cobalt production capacity of nations known and/or thought to possess large cobalt reserves-resources will be discussed. Additionally, the problems attendant with the development of new production capacity will be addressed. The data presented in the following section is summarized in Table 1-6.

1.2.3.1 United States

The economically significant cobalt occurrences of the United States, that is, those which have been or which might reasonably be expected to be

Table 1-5

WORLD MINE PRODUCTION OF COBALT BY COUNTRY, 1975-1979

(tons contained cobalt)

COUNTRY	1975	1976	1977	1978e
Australia	2,986.4	3,835.0	3,802.0	3,800.0
Botswana	88.2	165.3	193.4	200.0
Canada	1,493.2	1,515.3	1,697.1	1,281.0
Cuba	1,801.8	1,801.8	1,763.2	2,000.0
Finland	1,542.8	1,410.6	1,400.0	1,500.0
Morocco	2,159.9	947.7	1,002.8	2,000.0
New Caledonia	2,264.6	1,983.6	2,000.0	4,600.0
Philippines	115.7	512.4	700.0	1,200.0
U.S.S.R.	1,950.5	1,950.0	2,093.8	2,000.0
Zaire	15,428.0	12,122.0	14,998.0	12,000.0
Zambia	2,628.3	2,424.4	2,700.0	2,500.0
MINE PRODUCTION TOTAL	32,459.4	28,668.6	32,350.0	34,000.0
Ratio	1.42	1.41	1.55	
METAL PRODUCTION TOTAL	22,784.0	20,271.0	20,861.0	NA

e = estimate

Ratio of mine production of contained metal to actual marketable metal.

GEOPOLITICAL DISTRIBUTION OF MINE PRODUCTION OF COBALT

United States	--	--	--	--
World Exclusive Of United States	28,707.1	24,916.3	28,475.0	30,000.0
Central Economy Countries	3,752.3	3,752.3	3,875.0	4,000.0
World	32,459.4	28,668.6	32,350.0	34,000.0

Source: USBM, Mineral Commodity Summaries, 1979, p. 39; Kruszona, et al., 1979, p. 60.

Table 1-6

PROJECTED MINE PRODUCTION OF CONTAINED AND EXTRACTABLE COBALT

(tons cobalt)

	ExF	Actual 1977	1980	1985	1990	1995	2000	2005	2010
Australia	C --	3,759	3,800	3,800	3,800	3,800	3,800	3,800	3,800
	E	--	3,040	3,040	3,040	3,040	3,040	3,040	3,040
Botswana	C	182	300	300	300	300	300	--	--
	E	--	270	270	270	270	270	--	--
Canada	C	1,637	1,650	2,700	3,500	4,000	5,000	5,000	5,000
	E	--	1,320	2,160	2,800	3,200	4,000	4,000	4,000
Cuba	C	1,600	1,600	4,500	4,500	4,500	5,700	5,700	5,700
	E	--	960+	2,700+	2,700+	2,700+	3,420+	3,420+	3,420+
Finland	C	1,300	1,400	1,500	1,500	1,500	1,500	1,500	1,500
	E	--	1,120	1,200	1,200	1,200	1,200	1,200	1,200
Indonesia	C	100	475	475	2,275	2,275	2,275	2,275	2,275
	E	--	427	427	1,867	1,867	1,867	1,867	1,867
Morocco	C	1,721	2,000	2,000	--	--	--	--	--
	E	--	1,500	1,500	--	--	--	--	--
New Caledonia	C	700	700	700	2,700	2,700	2,700	2,700	2,700
	E	--	630	630	2,230	2,230	2,230	2,230	2,230
Philippines	C	1,195	1,700	1,700	1,700	1,700	1,700	1,700	1,700
	E		1,360	1,360	1,360	1,360	1,360	1,360	1,360
Republic of South Africa	C	280	280	500	1,000	1,000	1,000	1,000	1,000
	E	--	250	450	900	900	900	900	900
Uganda	C	--	--	1,680	1,680	--	--	--	--
	E	--	--	1,344	1,344	--	--	--	--
United States	C	--	--	2,950	3,700	4,200	3,950	1,750	1,750
	E	--	--	1,915	2,365	2,615	2,490	950	950
U.S.S.R.	C	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000
	E	--	1,200	1,200	1,200	1,200	1,200	1,200	1,200
Zaire	C	11,600	12,241	18,166	23,413	24,607	24,607	24,607	24,607
	E	--	12,241	18,166	23,413	24,607	24,607	24,607	24,607
Zambia	C	2,500	3,450	4,526	6,557	6,891	6,891	6,891	6,891
	E	--	3,450	4,526	6,557	6,891	6,891	6,891	6,891
TOTALS: CONTAINED		28,274	31,596	47,497	58,625	59,473	61,423	58,923	58,923
EXTRACTABLE.		--	27,768	40,888	51,246	52,080	53,475	51,665	51,665

C = Contained
E = Extractable

ExF = Extraction Factor

For percentages of extractable cobalt used in the determination of each country's potential cobalt production base, please refer to the following pages and supporting text which give the assumptions used to derive these data.

ASSUMPTIONS USED IN THE COMPILATION OF TABLE 1-6

Australia - Production from Australia's three mines likely to remain constant for at least the next twenty years. Production data are for cobalt contained in concentrates.

Botswana - Production not expected to increase substantially over the next ten to twenty years. Production data for contained cobalt in matte from Selebi-Pikwe mine-smelter.

Canada - Production data are an estimate of Canadian mine output of cobalt metal.

Cuba - Cuban mine cobalt production. No adjustment has been made for metal losses during beneficiation of ore and processing of concentrates.

Finland - Production data are for cobalt contained in concentrates from Finnish mines.

Indonesia - Production data reflect cobalt content of matte from Soroako (475 tons); cobalt contained in concentrates from Gag Island (1,800 tons). Cobalt production could increase slightly in late 1990's if deposits at Obi and Gebe Islands are brought into operation.

Morocco - Production data are for cobalt contained in concentrates; presently identified reserves are believed to be insufficient to warrant existing production levels for more than five years.

New Caledonia - Production data are for recoverable cobalt contained in matte produced by SLN smelter. SLN smelter in New Caledonia has capacity to process approximately 700 tons contained cobalt in matte which is sent to AMAX's Louisiana refinery and to SLN's facility in France (which was destroyed by a fire in 1979). Data regarding Tiebaghi deposit are in terms of cobalt contained in concentrates.

Philippines - Production data are for contained cobalt in nickel-cobalt sulfides from Surigao mine on Nonoc Island.

Republic of South Africa- Production data are for recoverable cobalt from nickel and nickel-copper mattes.

(Continued)

ASSUMPTIONS USED IN THE COMPILATION OF TABLE 1-6

Uganda	-	Production data are for cobalt contained in sulfides in tailings.
United States	-	Production data are primarily for cobalt contained in ores and concentrates.
U.S.S.R.	-	Estimate is for cobalt contained in ores (mine production). Insofar as nickel production is expected to be increased over the next twenty years, cobalt output could also possibly rise if nickel ores are cobaltiferous.
Zaire	-	Production data have been adjusted as indicated on page 1-41, Table 1-7 to reflect recoverable cobalt content of ore.
Zambia	-	Production data have been adjusted as indicated on page 1-41, Table 1-7 to reflect recoverable cobalt content of ore.

ASSUMPTIONS USED TO ESTABLISH COBALT RECOVERIES FROM MINES, MILLS, AND SMELTERS

- 50 percent of the cobalt metal content of cobaltiferous nickel and/or nickel-copper ores ultimately can be recovered as metal.
- 80^{+} percent of the cobalt metal content of cobaltiferous nickel and/or nickel-copper concentrates ultimately can be recovered as metal.
- 90^{+} percent of the cobalt metal content of cobaltiferous nickel and/or nickel-copper matte ultimately can be recovered as metal.

For these three assumptions to be valid:

- The deposits must be exploited in an efficient manner with one of the primary goals of the operating company being the optimum recovery of cobalt.
- The cobaltiferous nickel and/or nickel-copper concentrates must be processed to a product such as nickel and/or nickel copper-matte from which the cobalt can be economically extracted.

Australia - 80 percent of cobalt contained in concentrates is recoverable as metal.

Botswana - Approximately 90 percent of the cobalt in matte is recoverable.

Canada - Approximately 80 percent of the cobalt content of mine production can be recovered.

Cuba - Only 60 percent of cobalt contained in mine production can be recovered. Uncertainties exist as to efficiencies of cobalt recovery by Soviet facilities, as well as to the type of end products of Soviet nickel smelters and refineries. Although it is possible that cobalt reserves could be higher, how much so is not known.

Finland - Because of the low cobalt grade of Outokumpu Oy's ores, plant efficiency will only be 90 percent.

Indonesia - Ninety percent of cobalt contained in matte from the Soroako deposit is recoverable; 80 percent of cobalt contained in concentrates from the Gag Island mine is recoverable.

Morocco	-	Seventy-five percent of the cobalt content of Bou Azzer ores is recoverable.
New Caledonia	-	Ninety percent recovery is possible of the cobalt from the cobaltiferous nickel matte produced by the SLN smelter (700 tons per year); 50 percent of cobalt contained in the concentrates from the Tiebaghi deposit is recoverable.
Philippines	-	Eighty percent of cobalt contained in concentrates is recoverable as metal.
Republic of South Africa	-	Ninety percent of cobalt contained in matte is recoverable.
Uganda	-	Eighty percent of the cobalt content of the pyritic concentrates can be recovered.
United States	-	Only 70 percent of the cobalt content of Blackbird ores is recoverable; only 60 percent of the cobalt contained in Gasquet Mountain laterite ores recoverable; because of the extremely low-grade of cobalt in the ores of the Duluth Complex, only 50 percent of contained metal is recoverable; only 55 percent of the cobalt content of Missouri lead-zinc-copper concentrates - tailings is recoverable as metal.
U.S.S.R.	-	Sixty percent of Soviet mine production of cobalt is recoverable as cobalt metal.
Zaire	-	Please refer to text in Section 1.2.3.4 and Table 1-7.
Zambia	-	Please refer to text in Section 1.2.3.4 and Table 1-7.

* * *

Note: The percentage recovery indicated above is highly variable and in large part is contingent upon the type of orebody under consideration. According to Kruszona et al., 1979, in the case of sulfide ores (Sudbury type), ultimate metal recovery can be as much as 80 percent; for oxide ores (laterite type), 50 to 55 percent. In the case of Bou Azzer-type ores, up to 75 percent of the metal content can be recovered. In the production of cobalt concentrates from the cobaltiferous copper ores of Zambia, only about 30 percent of the metal is recoverable.

Sources: Kruszona, et al., 1979; various industry and government experts, 1978, 1980.

* * *

brought into production within the period 1980-2010, can be classified into one of five major categories:

- o Hydrothermal Deposits: Mississippi Valley-type deposits (Missouri Lead Belt); Blackbird district, Idaho.
- o Laterite Deposits: Nickel-cobalt laterites of Northern California and Southern Oregon.
- o Massive Sulfide Deposits in Metamorphic Rocks: Ducktown, Tennessee.
- o Hypogene Deposits Associated with Mafic Intrusive Igneous Rocks: Duluth Complex, Minnesota.
- o Contact Metamorphic Deposits Associated with Major Intrusive Rocks: Cornwall and Lebanon, Pennsylvania.

1.2.3.1.1 Hydrothermal Deposits

Blackbird District, Idaho. Over the period 1917-1960, deposits in the Blackbird district of Idaho have been intermittently worked for cobalt, copper, and, to a lesser extent, gold. Recent exploration efforts by Hanna Mining Company at the Blackbird deposit have delineated over 4 million tons of proven ore containing 0.76 percent cobalt and 1.2 percent copper. According to company sources, an additional "similar quantity of probable ore" also is indicated at the property. Assuming a minimum conservative total tonnage of 6 million tons of ore having an average grade of 0.50 percent cobalt and 1.0 percent copper, the Blackbird orebody could contain as much as 60,000 tons of copper and 30,000 tons of cobalt. Because of the recent increase in reserves at the Blackbird deposit, limited production is expected to begin from this property as early as the second half of 1980. Targeted annual production of 2,200 tons of contained cobalt metal is planned to commence in 1982. Should reserves be 6 million or more tons of ore, mine life is expected to be at least ten years.

Although exploration for other Blackbird-type deposits is proceeding throughout the United States, few other suitable geologic environments are known which could serve as a host to cobalt mineralization. This is not to say, however, that as exploration efforts continue, new districts will not be discovered containing orebodies similar to that at Blackbird. However, most economic-exploration geologists do not believe that with increased exploration many such new cobaltiferous areas will be discovered.

Cobalt Associated With Mississippi Valley Lead-Zinc Deposits. Numerous lead-zinc deposits in the Mississippi Valley are cobaltiferous. Although cobalt and nickel have long been recognized in the lead-zinc ores of southeastern Missouri, relatively few attempts have been made to recover the contained cobalt and nickel values. The only technically successful (albeit not truly economic in the strictest sense of the word) cobalt-nickel recovery operation was undertaken in 1944 by NL Industries with the assistance of a substantial government subsidy. Between that time and 1961, NL Industries produced lead, copper, and nickel-cobalt concentrates from its Madison mine in Fredericktown, Missouri.

According to research conducted by the USBM in conjunction with the University of Missouri - Rolla, the Missouri Lead Belt contains over 325 million tons of ore grading 5.9 percent lead, 1.1 percent zinc, 0.3 percent copper, 0.02 percent nickel, and 0.015 percent cobalt. Assuming complete mine-mill recovery, as much as 65,000 tons of nickel and 48,750 tons of cobalt exist in this resource. Other estimates of the nickel-cobalt content of the Missouri Lead Belt indicate total in-place nickel and cobalt at as much as 85,500 tons and 59,500 tons, respectively.

Aside from economic reasons, milling difficulties have prevented the recovery of cobalt and nickel from Missouri Lead Belt ores. Nearly all of the cobalt and nickel of the Missouri Lead Belt occur in the mineral siegenite $(Ni,Co)_3S_4$. To recover siegenite from lead-zinc-copper ores requires extremely fine grinding to liberate finely intergrown siegenite from chalcopyrite, sphalerite, galena, and dolomite. Assuming that fine grinding is accomplished, an additional problem exists insofar as siegenite has many of the same flotation properties as does chalcopyrite. Hence, to recover siegenite from the chalcopyrite requires that the copper be leached from the chalcopyrite using ferric chloride; the resultant insoluble residue contains the cobalt-nickel values.

Other approaches have been proposed to recover siegneite from the chalcopyrite. Recent research by the USBM indicates that after regrinding the copper concentrate to 94 percent minus 400 mesh, chalcopyrite can be floated off leaving a "tailings" assaying 20.4 percent copper, 5.2 percent nickel and 3.45 percent cobalt---a product which potentially could be sold as marketable feed material for cobalt-nickel recovery processes. This represents 67.2 percent of the nickel and cobalt originally in the copper concentrate (USBM RI8321).

Despite the metallurgical problems inherent in the recovery of cobalt and nickel from the ores of the Missouri Lead Belt, at least one company has demonstrated an interest in developing this potential source of cobalt-nickel supply. Anschutz Uranium Corporation recently announced its purchase of the Madison mine which is believed to contain nealy 14,000 tons of cobalt in tailings, residues, and in-place. According to Anschutz spokesmen, the company intends to begin cobalt extraction from existing mill tailing piles at an annual rate of 750 to 1,000 tons of cobalt. Assuming no major problems develop, recovery of cobalt from the tailings will occur in 1982. Actual mining of the Madison orebody is not likely to begin until 1985 at the earliest.

Although the USBM study of the recovery of cobalt and nickel from the Missouri Lead Belt demonstrates the technical feasibility of recovering these metals from lead-zinc-copper ores, and despite Anschutz Uranium Corporation's effort, significant quantities of cobalt and nickel are not likely to be recovered from this district prior to 1990. Beyond that time, total cobalt production from the Missouri Lead Belt possibly could be as much as 1,800 tons per year.

1.2.3.1.3 Massive Sulfide Deposits in Metamorphic Rocks

Several cobaltiferous massive sulfide deposits have been discovered in the Blue Ridge Mountains and Piedmont province of the southeastern United States. However, none contain sufficient concentrations of cobalt to warrant its economic recovery. Therefore, unless additional large massive sulfide deposits are discovered in the near future which contain anomalously high concentrations of cobalt (in conjunction with other economically recoverable metals), appreciable quantities of cobalt are not likely to be obtained from this source over the period 1985-2010.

1.2.3.1.4 Hypogene Deposits Associated with Mafic Intrusive Igneous Rocks

Numerous cobaltiferous hypogene deposits associated with mafic intrusive igneous rocks have been discovered in the United States. However, with few exceptions, these deposits are thought to be too small and/or of too low of grade to justify their exploitation. Additionally, the proximity of a number of these deposits to urban and/or scenic areas would very likely preclude their development--particularly in light of their "metallic marginality."

Duluth Complex, Minnesota. Of all the hypogene deposits associated with mafic intrusive igneous rocks mentioned as being potential sources of cobalt, the Duluth Complex of Minnesota is preeminent. Because of the interest by numerous mining companies in exploiting the Duluth Complex for its copper-nickel-cobalt mineralization, the Minnesota Department of Natural Resources (MDNR) in 1976 commenced a comprehensive study of the economic geology of the intrusive. As a result of this investigation, the MDNR determined that the Duluth Complex contains at least 4.4 billion tons of mineralized material from which 29 million tons of copper (at a cutoff grade of 0.50 percent copper) and 8.8 million tons of nickel could theoretically be obtained assuming 100 percent recovery. Should the average grade of cobalt in the Duluth Complex be 0.01 percent cobalt, over 440,000 tons of cobalt possibly are contained in this intrusive. Assuming that the Duluth Complex is developed, annual production from a single operation in this intrusion (given 100 percent recovery

1.2.3.1.2 Laterite Deposits

Whereas cobaltiferous nickel laterite deposits have been identified in several areas of the United States (California, Oregon, Washington, and North Carolina), cobalt is not being produced at present from any of these occurrences. The only domestic nickel producer is Hanna Mining Company at its Nickel Mountain deposit near Riddle, Oregon. However, because the process used to smelt the ore is not cobalt selective (the end product being ferronickel), economic recovery is not feasible.

Gasquet Mountain, California. The cobaltiferous nickel laterites of northern California are presently being investigated by at least one company to determine whether the deposits can be economically developed. At its Gasquet Mountain (California) deposit, California Nickel Corporation is reported to have outlined in excess of 50 million tons of ore containing approximately 1 percent nickel and 0.05 percent cobalt. Assuming that the ore reserves and grades are correct, the Gasquet Mountain deposit contains approximately 500,000 tons of nickel and 25,000 tons of cobalt. Although California Nickel Corporation plans to begin construction of its mill by 1982, significant nickel-cobalt production from this property is unlikely until the mid-to-late 1980's. Should the proposed mill be brought into production at an annual feed capacity of 1.5 million tons of ore, and assuming 100 percent recovery (which is virtually impossible), yearly output of nickel and cobalt from this deposit could be as much as 750 tons of cobalt and 15,000 tons nickel. The actual quantity of nickel, and more particularly, cobalt metal annually recovered will be significantly less than the indicated amounts. Given optimum conditions, and considering the existing Riddle operations of Hanna Mining Company, it is unlikely that more than 90 percent of the contained nickel and 50 percent of the contained cobalt ultimately could be recovered as metal.

which is virtually impossible) is estimated at 500 tons of cobalt, 110,000 tons of copper and 30,000 tons of nickel. However, because of the overall low grade of the ore, metal recoveries most likely will be low--on the order of 50 to 70 percent depending on the metal involved.

Whether the Duluth Complex will be exploited for its copper-nickel-cobalt content within the next ten to twenty years is uncertain. Because of its proximity to the Boundary Waters Canoe Area and the Superior National Forest, many environmental groups are concerned about the potential effects which mining the Duluth Complex could have on these scenic and recreational resources. Over the past decade, two companies, AMAX Inc. and INCO, have been particularly active in conducting extensive exploration of the intrusive. In the mid-1970's, AMAX and INCO advised Minnesota state authorities that they intended to file environmental impact statements as a preliminary step to mining the intrusive. The Minnesota Environmental Quality Board (MEQB) of the State Planning Agency thereupon established a five year moratorium on mining which was rescinded only after the completion of an extensive environmental and economic impact study in September, 1979. Because it was determined that the environmental effects of mining the Duluth Complex can be managed in an acceptable manner, AMAX was granted permission to begin a new environmental impact statement concerning its plans to begin a small-scale, $40 million mine and ore processing pilot plant. Assuming no difficulties are encountered either from an environmental, financial, or technical standpoint, AMAX hopes to begin full-scale mining of the Duluth Complex sometime after the mid-1980's.

Given the environmental constraints of the project, the fact that the operation cannot be economically viable for the content of one of the three metals alone, and the potential taxation policies which could be enacted which would significantly reduce the profitability of the project, large scale mining of the Duluth Complex is unlikely to begin before 1990 at the earliest.

1.2.3.1.5 Contact Metamorphic Deposits Associated With Mafic Intrusive Rocks

Several cobaltiferous contact metamorphic deposits of magnetite and chalcopyrite have been discovered in the United States. At Cornwall, Lebanon County, Pennsylvania, and at Morgantown, Berks County, Pennsylvania, cobalt exists primarily in the pyrite which contains an average of 1.2 to 1.4 percent cobalt at the Cornwall and adjacent mines, and 0.51 percent cobalt the Grace mine. Because of the relatively low cobalt content of the ore, neither of these deposits could have been mined (even at present high cobalt prices) for their cobalt alone. Although operations at both of these properties have ceased, the economic viability of presently mining these or geologically similar deposits would be contingent on their iron, and not cobalt content.

Other occurrences similar to those at Cornwall and Morgantown are known to exist in the United States, particularly along a 75 mile long, 5 to 20 mile wide trend in southeastern Pennsylvania. However, given the fact that other types of iron deposits are generally more economic to develop and mine than are contact metamorphic deposits such as those at Cornwall and Morgantown, significant quanitites of new domestic cobalt capacity are unlikely to be derived from these types of occurrences over the next thirty years.

1.2.3.2 Canada

Canadian cobalt reserves and resources are predominately associated with nickel-copper sulfide deposits. Virtually all of Canada's economically significant cobalt occurrences can be classified as either:

- Hypogene Deposits Associated with Mafic Intrusive Igenous Rocks: Sudbury District, Ontario; Thompson-Moak Lake area, Manitoba, or
- Hydrothermal Deposits: Cobalt-Gowganda region, Ontario.

With the exception of cobalt derived as a coproduct of silver mining, 99 percent of all Canadian cobalt production is largely contingent on the market for copper and nickel. In 1978, two companies produced cobalt as a byproduct of domestic nickel-copper mining: Falconbridge Nickel Mines, Ltd., and INCO, Ltd. One company, Agnico Eagle Mines, Ltd., produced cobalt as a byproduct of its silver mining operations in the Cobalt district of Ontario.

To assess the quantity of cobalt which possibly could be produced by Canadian mines over the next thirty years, future nickel production must be determined. However, the type of nickel produced bears heavily on total cobalt output from Canadian mines. Nickel is marketed primarily in two forms: Class I (high purity) and Class II (lower purity) products. Whereas Class I products can be used in virtually all nickel applications, the use of Class II nickel is restricted largely to steelmaking. In the production of Class I nickel, cobalt can be recovered insofar as significant refining is required to achieve high purity metal. However, most production processes for Class II nickel are not suitable for cobalt recovery. Insofar as demand for Class II nickel products has been growing at a rate greater than that for Class I nickel, total cobalt output from Canadian nickel smelters will very likely decrease over time. In 1977 and 1978, Canadian mine production of cobalt is estimated to have been 1,637 and 1,281 tons of cobalt, respectively.

1.2.3.2.1 Hypogene Deposits Associated With Mafic Intrusive Igneous Rocks

Sudbury-Thompson Districts. As dicussed in Chapter II, the world nickel market has been depressed since 1974. Only recently has nickel demand again begun to exceed production. Given this scenario, additional major expansions to Canadian nickel mine production capacity are unlikely in the near future (1980-1986). Of the new nickel projects expected to come on-stream between 1980 and 1990, none are in Canada. The only possibility of additional cobalt production from existing nickel facilities lies either in the reopening of mines placed on standby status during the 1974-1979 period of stagnant nickel demand, or expansions to existing and/or construction of new cobalt recovery facilities. With regard to potential increases in the efficiency of existing operations, both INCO, Ltd. and Falconbridge Nickel Mines, Ltd. recently announced plans to optimize their mill extraction circuits to improve cobalt recoveries. It is reported that after improvements are made to INCO's recovery circuits (expected to be completed by 1980), annual cobalt production by the company will double from approximately 750 tons cobalt to more than 1,500 tons cobalt. In a similar move, Falconbridge Nickel Mines plans to increase its cobalt recovery from 1,000 tons cobalt per year to 2,000 tons cobalt annually by 1981. Of the 3,500 tons of cobalt expected to be produced annually by both companies by 1982, the majority will be derived from Canadian mine production.

As discussed in Section 4 of Part I, over the long-term (1986-2010) it is possible that several cobaltiferous copper-nickel deposits will be brought into production. According to research conducted by the Mineral Policy Sector, Energy Mines and Resources, Canada, Canadian production of nickel by 1998 is projected to be approximately 260,000 tons, or virtually the same capacity as estimated to be installed in 1981.(1) If it is assumed that <u>at least</u> the same amount of cobalt will be contained in nickel ores brought into production in the future as is the case at the present, and that smelting and refining practices will remain fairly constant, Canadian cobalt production likely will remain essentially at 1973 levels--at least through the mid-to-late 1990's.

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(1) This projection is based not only on reasonably definite data regarding the timing and capacity of new production centers, but also includes information concerning likely extensions of known reserves, probable reactivation of standby mines, and possible development of "on the shelf" deposits. To summarize, this projection represents not an absolute, but rather a most-likely scenario.

1.2.3.2.2 Hydrothermal Deposits

Cobalt-Gowganda District. With regard to expansion to, and/or discovery of new cobaltiferous silver properties such as those in the Cobalt-Gowganda district, additional cobalt output possibly will be derived from these types of deposits. In light of projected increased demand for cobalt as well as recent high prices, many companies are considering the possibility of re-treating cobaltiferous tailings and/or residues from the mines of the Cobalt district. Previously, because area producers were penalized by smelters for the cobalt content of their ores, much of the cobalt associated with the silver ores of the Cobalt district was sent out with the tailings. However, given presently high prices and projected strong demand for cobalt, many smelter contracts now provide for compensation for cobalt content when it can be economically processed.

A second problem regarding the achievement of increased cobalt production from the Cobalt district concerns the lack of a central cobalt processing plant in the area to handle cobaltiferous ores, tailings, and residues. Should such a facility be constructed, more cobalt probably will be recovered from the ores of this district. However, most Canadian cobalt producers do not believe that such a plant will be constructed within the next decade.

1.2.3.3 Cuba

All of Cuba's cobalt production is derived as a byproduct of mining of cobaltiferous nickel laterites. Cuba presently produces approximately 1,600 tons of cobalt per year from deposits at Nicaro (on the northern coast of Oriente Province) and at Moa Bay (near the eastern tip of Cuba, also in Oriente Province). At both of these properties is an extraction-processing plant, each of which is capable of producing 21,000 tons of contained nickel per year.

Cuba is not only in the process of expanding its existing nickel production capacity by approximately 11,000 tons per year, but is also constructing a third mining and processing facility at Punta Gorda (2 miles east of Moa in Oriente Province) estimated to have an annual capacity of 33,000 tons contained nickel metal. In addition, a fourth plant of the same capacity, termed the Las Camariocas facility, is planned to begin production in 1984 in the Moa Bay region. Thus, by 1985 (when all of the expansions as well as new facilities are projected to be completed), Cuban nickel capacity could be as much as 119,000 tons contained nickel metal per year. Assuming that an approximate ratio exists between annual nickel and cobalt production of 26:1, by 1985 Cuban cobalt production possibly could be as much as 4,500 tons of cobalt metal. Should Cuban nickel production capacity be increased to 165,000 tons per annum (as some nickel experts predict based on Cuban announcements), yearly cobalt production could exceed 5,700 tons within twenty years. However, implicit in these assumptions is the fact that future Cuban processing facilities will be capable of recovering virtually all of the cobalt content of the feed--which is highly unlikely. Additionally, the efficiency of Soviet and Czechoslovakian smelters and refineries in recovering cobalt contained in Cuban nickel-cobalt exports is uncertain.

1.2.3.4 Zaire and Zambia

Zaire. All of the cobalt production in Zaire currently is obtained from strata-bound deposits. The principal mining company, Gecamines, operates seven mines in Zaire (5 open pit and 2 underground operations) which have a total production capacity of approximately 14 million tons per year of ore containing an average of 4.77 percent copper and 0.35 percent cobalt. Given this total mine production capacity, Gecamines theoretically should be able to produce 664,000 tons of copper and 49,000 tons of cobalt metal contained in extracted ore per year. Current cobalt production capacity in Zaire is estimated to be approximately 17,600 tons per year. (For additional data concerning Zairian cobalt production capacity, please refer to Footnote 1 on Table 1-7.) However, because of recovery problems which are encountered in both the mining and milling of the ore as well as processing of the resultant concentrates, only 11,240 tons of the cobalt originally contained in the ore produced in 1977 was extracted during cathode production. According to recent studies of the Zaire copper-cobalt mining industry, only one third of the cobalt content of all copper-cobalt/cobalt-copper concentrates produced by Gecamines is ultimately recovered (Kruszona, 1979).

To process the copper-cobalt ores produced by Gecamines, four flotation plants (at Kolwezi, Kamoto, Kambove and Kakanda) and one washing facility (at Mutoshi) are utilized. Whereas oxide concentrates from these plants are processed at the electrowinning facilities at Shituru and Luilu, sulfide concentrates are smelted at Lubumbashi.

To specifically increase future production of cobalt (as differentiated from projects directly concerned with raising copper output), Gecamines has recently initiated a four-year expansion plan called the P-2 Project. Through this program, Zaire hopes to add 6,600 tons to their country's current 17,600 ton capacity by 1982. To accomplish this task, Gecamines plans to:

- Bring the Mashamba and Dikuluwe orebodies into production using open pit methods.

- Construct a new concentrator at Kolwezi capable of processing the additional 4.4 million tons of ore annually produced by these two mines.

- Construct a copper smelter at Luilu capable of producing 6,600 tons of cobalt per year.

Table 1-7

PROJECTED COBALT PRODUCTION CAPACITY: ZAIRE AND ZAMBIA
(tons cobalt metal)

		1979	1980	1985	1990	1995	2000	2005	2010
ZAIRE	1) Approximate present production capacity.	12,000	12,000	12,000	12,000	12,000	12,000	12,000	12,000
	2)	—	—	5,300	5,300	5,300	5,300	5,300	5,300
	3) Incremental additions to capacity.	—	—	—	4,320	4,320	4,320	4,320	4,320
	TOTAL CAPACITY	12,000	12,000	17,300	21,620	21,620	21,620	21,620	21,620
	Pre-date capacity with 1.0 percent improvement	—	12,120	12,241	18,166	23,413	24,607	24,607	24,607
	1.0 percent improvement in performance	120	121	625	927	1,194	--	--	--
	New increment of capacity	--	--	5,300	4,320	--	--	--	--
	TOTAL ADJUSTED CAPACITY	12,120	12,241	18,166	23,413	24,607	24,607	24,607	24,607
ZAMBIA	4) Approximate present production capacity.	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500
	5)	—	900	1,800	1,800	1,800	1,800	1,800	1,800
	6) Incremental additions to capacity.	—	—	—	1,800	1,800	1,800	1,800	1,800
	TOTAL CAPACITY	2,500	3,400	4,300	6,100	6,100	6,100	6,100	6,100
	Pre-date capacity with 1.0 percent improvement.	--	2,525	3,450	4,526	6,557	6,891	6,891	6,891
	1.0 percent improvement in performance.	25	25	176	231	334	--	--	--
	New increment of capacity.	--	900	900	1,800	--	--	--	--
	TOTAL ADJUSTED CAPACITY	2,500	3,450	4,526	6,557	6,891	6,891	6,891	6,891
ZAIRE AND ZAMBIA	TOTAL CAPACITY	14,645	15,691	22,692	29,970	31,498	31,498	31,498	31,498

1) Although the current cobalt metal production capacity of Zaire is believed to be approximately 17,600 tons per year, and has been reported to have been as high as 19,400 tons annually in 1974, most industry experts believe that the country's present effective operating capacity is about 12,000 tons per year.

2) Gecamines' proposed new smelter at Luilu with a capacity of 6,600 tons of cobalt per year will be only 80 per cent efficient when it comes on-line in the early 1980's.

3) Tenké Fungurumé deposit having a reported cobalt production capacity of 7,200 tons per year will be exploited by 1990—however, actual cobalt recovery will be only 60 per cent efficient.

4) As in the case of Zaire, there is controversy concerning existing installed Zambian cobalt mine production capacity. Estimates vary from 2,000 to 8,400 tons of contained cobalt in ore per year. Much of the discrepancy is due to whether the cobalt data is for crude-ore output or concentrator-refinery output. Most industry experts believe that Zambian extractable cobalt production capacity is about 2,500 tons per year.

5) The 2,400 ton per year cobalt refinery at Chambishi (RCM) will begin operation at only 38 per cent of its designed capacity. Within several years, operating efficiency will be raised to 75 per cent.

6) The estimated NCCM 3,000 ton per year expansion of cobalt production capacity at Rokana will not come on line until at least 1985 at the earliest. When fully operational, the facility will be only 85 per cent efficient, yielding slightly 1,800 tons per year of additional cobalt production.

However, because of the civil unrest which occurred in Zaire in May 1978, neither of these projects are likely to be completed significantly prior to 1985.

In addition to these properties, the Societe Miniere de Tenke Fungurume (SMTF) plans to initiate development of a copper-cobalt deposit located in the Central Sector of the Zairian copperbelt. Ore reserves at the Tenke Fungurume deposit are estimated to be 56.2 million tons averaging 5.7 percent copper and 0.45 percent cobalt. Assuming total cobalt recovery, this deposit contains approximately 303,500 tons of cobalt metal. According to SMTF, annual copper and cobalt production from this property would be 110,000 to 143,000 tons and 7,200 tons, respectively. Although this project has been temporarily suspended due to rapid escalations in the original cost of development (from U.S. $600 million to over $1 billion), the development of the Tenke Fungurume orebody will likely commence within the next five years.

Zambia. Two major companies mine the stratabound copper-cobalt deposits of Zambia: Nchanga Consolidated Copper Mines, Ltd. (NCCM) and Roan Consolidated Mines, Ltd. (RCM). Whereas NCCM has nine mines in operation (5 open pit, 2 underground, and 2 open pit-underground unit operations), RCM manages six mines (4 underground, 1 open pit, and 1 combined open pit-underground operation). Of the fifteen mines belonging to the two companies, cobalt is recovered as a byproduct from only three underground copper-cobalt properties:

Mine/Owner	1977 Production (Tons of Ore)	Grade Copper/Cobalt	1977 Production Cobalt Content (Tons)
Nkana (NCCM)	4,718,000	1.60%/0.10%	4,717
Chibuluma (RCM)	626,000	3.21%/0.18%	1,127
Baluba (RCM)	1,576,000	1.39%/0.16%	2,521
TOTAL	6,920,000		8,365

Source: Kruszona, et al., 1979.

Production from RCM's Chibuluma mine (a small high-grade deposit containing approximately 7 million tons of ore) is processed on-site into copper and cobalt concentrates. The copper concentrates are shipped to a smelter at Mufulira; the cobalt concentrates are sent to Chambishi where they are converted into cobalt hydroxide. This material, in turn, serves as feed to NCCM's processing facility in Nkana where cobalt metal is produced. Ore from RCM's Baluba mine (a large, high-grade orebody with estimated reserves of 70 million tons) is concentracted by the company's mill at Luanshya. As is the case of concentrates from the Chibuluma mine, those of RCM's Baluba mine are also processed at the Chambishi conversion facility. Production from the Nkana mine is not only milled, but converted to cobalt metal (along with that of the Chibuluma and Baluba mines) on-site.

As in the case of Zaire, much of the cobalt originally contained in Zambian ores produced from the three mines is lost to tails or copper concentrates during processing. Of the approximately 8,400 tons of cobalt contained in mine production capacity during 1977, only one-fourth is estimated to have been recovered during metal production (Kruszona, et al., 1979).

In early 1978, installed Zambian cobalt production capacity was estimated to be 2,500 tons per year. (Some estimates place Zambian installed cobalt production capacity at considerably less than 2,500 tons per year--perhaps as little as 1,900 tons annually.) However, to increase cobalt production from Zambia, several projects are now either nearing completion or are in the development phase. RCM plans to increase the throughput capacity of the Luanshya concentrator to slightly over 11,000 tons from 4,600 tons per day. In addition to this expansion, RCM also intends to begin operation in 1980 of a new cobalt concentrate processing-leaching plant at Chambishi having an annual capacity of approximately 2,400 tons. When fully operational, this facility should increase Zambian cobalt production capacity to approximately 4,000 tons per year. Similarly, NCCM is contemplating the expansion of its cobalt production capacity at Rokana from 7,700 tons to 11,000 tons per year through construction of new roasting, leaching and electrowinning facilities. However, inauguaration of this increased capacity (which is estimated to be approximately 2,800 tons per year) is not expected until at least 1985 at the earliest. (It should be noted that some estimates place NCCM's new addition at Rokana at as much as 4,200 tons capacity per year.)

Summary. Future levels of cobalt production from mines in Zaire and Zambia are difficult to predict insofar as the political instability of southern Africa has tended to discourage foreign investment in the mining industry of this region. Additionally, foreign technical personnel are reluctant to return to the copperbelt of Zaire and Zambia because of the recent invasion by Katangan rebels.

However, despite these potential limitations to the installation of additional cobalt production capacity, by 2000, total combined cobalt output from refineries in Zaire and Zambia possibly could be as much as 29,637 tons (Table 1-7).

In developing projections of future installed cobalt production capacity, the following factors and assumptions have been taken into consideration:

Zaire

- The Tenke Fungurume deposit will be developed by 1990 at a production capacity of 7,200 tons per year cobalt. However, only 60 percent of the cobalt contained in the ore will be recovered.

- When Gecamines' new smelter at Luilu is brought on-line prior to 1985, its efficiency will be 80 percent of the stated operating capacity of 6,600 tons of cobalt per year, or approximately 5,300 tons cobalt annually.

Zambia

- The 2,400 ton per year cobalt refinery at Chambishi expected to be completed by RCM in 1979 will operate at an average efficiency of 75 percent. Due to startup difficulties, first year efficiency will be only 50 percent of average operating efficiency, or 900 tons of cobalt.

- The projected increase of approximately 2,800 tons in NCCM's cobalt production capacity will be only 65 percent efficient. Thus, the total increment of new capacity will be about 1,800 tons per year cobalt.

- Development of a new discovery of cobalt-bearing copper ore adjacent to the Chingola mine in the Chingola Division of Zambia will probably occur prior to 1990. Although drilling has indicated the existence of over 44 million tons, predominately of oxide ore, grading 0.4 percent cobalt, there is some question as to the rate at which this discovery will be exploited. This is despite the fact that initial announcements by NCCM personnel indicated the possibility of annual production of 5,000 tons of cobalt from the Chingola mine.

Zaire-Zambia

- Reserves of mines presently in operation will be sufficient to maintain existing production levels.

- Copper market conditions will remain sufficiently strong to justify maximum production from mines in Zaire and Zambia despite possible political or military disruption which could limit full utilization of installed capacity and/or transporation of concentrates and metal to consumers. Although demand for copper appears to follow a seven to ten year cycle, this factor will be ignored in this analysis.

- As discussed previously, cobalt production by Zaire and Zambia could be dramatically increased through the implementation of improved extractive technology. Therefore, given the present effort by the mining companies of both countries to utilize new and more efficient recovery techniques and equipment, an average 1 percent improvement in cobalt output per year possibly could be realized between 1980 and 1995.

1.2.3.5 Australia, Indonesia, New Caledonia and The Philippines

Virtually all of the cobalt either presently or likely to be produced by Australia, Indonesia, New Caledonia, and the Philippines is, or will be obtained as a byproduct of the mining of cobaltiferous nickel laterite deposits.(2) As in the case of Cuban nickel laterite deposits, the laterite orebodies of Australia, Indonesia, New Caledonia and the Philippines are mined primarily by open pit methods. Because most of the cobaltiferous nickel laterites of these nations are, to a greater or lesser extent, geologically similar, the following discussion will be restricted to consideration of the likelihood of the future discovery and exploitation of cobaltiferous nickel laterite orebodies in this region.

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(2) Of particular importance in assessing the impact which cobalt produced during mining of laterite nickel deposits will have on world cobalt supply is the way in which the cobaltiferous nickel ores are processed. As discussed in Section 1.2.2.3, separation of cobalt from ferro-nickel is not possible. Thus, much of the cobalt contained in the ores mined in Indonesia and New Caledonia cannot be recovered insofar as the Aneka nickel smelter in Indonesia and a large part of the capacity of SLN (Societe Metallurgique Le Nickel, SA) Doniambo smelter produce ferro-nickel.

As discussed in Section 4 of Part I, demand for nickel during the past five years has not been strong. Consequently, few mining companies have been willing to commit their financial resources to the development of any nickel orebodies--regardless of whether or not they contain economically extractable quantities of cobalt. As a result of the recent period of depressed demand for nickel, the general world overcapacity in nickel production capacity, and the relatively high producer inventories of nickel, only two new projects in the region are likely to be initiated during the period 1981-1990 which would have an impact on cobalt production. These are:

- o P.T. Pacific Nikkel's - Gag Island deposit, Gag Island, Indonesia.

- o BRGM-AMAX's - Tiebaghi deposit, northern New Caledonia.

<u>Australia</u>. Virtually all Australian nickel deposits are known to contain minor amounts of cobalt. This is significant insofar as Australia is presently the world's fourth largest producer of nickel. Although little information is available concerning the cobalt content of ores from Australian deposits, cobalt output will most likely continue to increase as new cobaltiferous nickel deposits are discovered and brought into production. At present, three mines produce the majority of Australian cobalt output:

- o Greenvale mine (Freeport Minerals).

- o Agnew mine (Western Selcast and Him Holdings.

- o Kambalda mine (Western Mining).

However, in the following discussion, only the Greenvale deposit will be considered.

At the Greenvale mine, ore feed to the concentrator averages 1.57 percent nickel and 0.12 percent cobalt. Although the property has been in operation four years, only one-half of the cobalt contained in the ore is reportedly recovered. In 1978, cobalt production from the Greenvale mine in recovered sulfides was slightly over 1,000 tons (out of plant capacity of over 1,300 tons of cobalt contained in recovered sulfides). The cobaltiferous concentrates are processsed by Nippon Mining Company in Japan where approximately 90 percent of the cobalt is recovered.

Indonesia. The principal cobalt producer in Indonesia is the Soroako nickel deposit which is operated by P.T. International Nickel Indonesia. When full production is achieved at this property in 1980, total output of cobalt will be approximately 475 tons of cobalt contained in matte. It is understood that the cobaltiferous matte will be shipped to INCO's nickel refinery in Wales where the cobalt will be extracted. Assuming 80 percent recovery, total cobalt output from the Soroako deposit will be about 380 tons of refined cobalt metal.

The nickel laterite deposit at Gag Island contains approximately 200 million tons of unusually high-grade ore averaging 1.4 percent nickel and 0.15 percent cobalt. Originally, the deposit was slated to begin production in 1979 at a rate of nearly 55,000 tons per year nickel and 1,800 tons per year cobalt. However, because of the depressed nickel market and the increasing cost of energy, development of the deposit was postponed. It is now generally thought that given the strengthening demand for nickel and cobalt, the deposit will likely be brought into production in 1988. Reserves at Gag Island are sufficient to enable production from the property for a period of at least 30 years.

New Caledonia. The Tiebaghi deposit at the extreme northern end of New Caledonia is estimated to contain 55 million tons of silicate ore with an average grade of 2.5 percent nickel (although in some areas of the deposit, grades as high as 3.0 percent nickel have been noted) and approximately 0.11 percent cobalt. Initial annual production capacity at this deposit is planned at 29,000 tons of contained nickel. However, there is the possibility that production will ultimately be increased to 35,000 tons of contained nickel per year. It is possible that mine cobalt output contained in concentrations from this property could be as much as 2,000 tons cobalt per year. Given the ongoing recovery of the nickel market and relatively high costs to develop the Tiebaghi deposit (estimated to be $600 million), production from this orebody is not likely to begin prior to 1990. How this ore will be processed is also unknown. Hence, unless a leaching-hydrometallurical process is used, much, if not all of the cobalt content possibly could be lost.

Philippines. Philippine potential cobalt reserves are, as indicated on Table 1-3, larger than those of any other country. Consequently, assuming that efficient extractive methods amenable to cobalt recovery are utilized at future mine-mill-smelter complexes, cobalt production from the Philippines could be significant.

The principal Philippine cobalt producer is Marinduque Mining and Industrial Corporation's Marinduque open pit mine at Surigao on Nonoc Island in Mindanao. This deposit contains substantial reserves grading 1.2 percent nickel and 0.1 percent cobalt. The mine has a capacity of approximately 35,000 tons of nickel and 1,700 tons of cobalt per year contained in sulfides. Whereas a large percentage of the nickel sulfides are processed to pure nickel on-site, the cobaltiferous sulfides are refined in Japan at Sumitomo Metal Mining's refinery. Although the property was brought on-stream five years ago (1974), full production capacity only recently has been achieved.

1.2.3.6 Africa

Botswana. The Phikwe-Selebi deposits in Botswana are mined through underground and open pit methods. The ore produced at the Phikwe and Selebi mines is processed on-site at a smelter which has a capacity of approximately 46,000 tons of nickel-copper-cobalt matte per year grading 0.66 percent cobalt. This is equivalent to slightly over 300 tons of cobalt annually. As lower grades of ore are mined at both deposits, cobalt production is expected to decrease.

Morocco. The Bou-Azzer deposit is the only orebody in the world presently mined primarily for its cobalt content. Confirmed and expected cobalt reserves of the Bou-Azzer deposit are estimated to be no more than 10,000 tons of contained cobalt metal. Therefore, at 1978 production levels of 2,000 tons contained cobalt in concentrates, it is unlikely that present levels of production can be maintained for more than 5 years--unless, of course, additional reserves are discovered.

Republic of South Africa. For several years, the Republic of South Africa has been a producer of small amounts of cobalt derived from the Rustenburg platinum mine. Of the estimated 200 tons of cobalt in the form of cobaltiferous nickel matte annually produced by the Rustenburg smelter, only one half has been recovered. Recently, however, Rustenburg Platinum Mines, Ltd. announced that it will begin production of cobalt sulfate at an initial rate of approximately 150 tons of contained cobalt per year. It is understood that given the expected continued strong demand for cobalt, other South African platinum producers are considering cobalt recovery projects which reportedly could cumulatively yield up to 1,000 tons cobalt per year. However, commodity analysts have expressed doubts as to whether this much cobalt production could be achieved--particularly insofar as future nickel production capacity appears insufficient to provide the feed necessary to produce this quantity of cobalt metal.

Uganda. At Kilembe, Uganda, stratabound cobaltiferous copper orebodies containing 2.1 percent copper and 0.18 percent cobalt occur in the Precambrian metamorphic rocks of the Kilembe Series. It is reported that the Government of Uganda has solicited applications from mining companies interested in processing nearly 1.2 million tons of cobaltiferous pyrite tailings (1.4 percent cobalt) from the Kilembe copper mine. According to industry sources, up to 16,800 tons of cobalt are contained in the tailings which have been stockpiled at the mine since it began production in 1956. Assuming a ten year recovery operation, annual cobalt production from this source could be as much as 1,680 tons of cobalt assuming nearly complete recovery. To date, only one company has agreed to purchase any of the tailings. A Swedish company, Tropiscale, will process 100,000 tons of the tailings.

1.2.3.7 Europe

Finland. Virtually all of the cobalt concentrates processed by Outokumpu Oy, Finland's state-owned copper-cobalt producer, are obtained from the Keretti copper mine. Keretti cobalt concentrate is reported to be low-grade, averaging only 0.70 percent cobalt.

At present, the capacity of Outokumpu Oy is estimated to be approximately 1,100 tons of cobalt. Plans are now underway, however, to increase the annual capacity of the cobalt plant to approximately 1,500 to 1,600 tons. To provide the necessary additional feed requirements to support this expansion, industry experts believe that cobaltiferous pyrite will be imported into Finland from West German sources.

1.2.3.8 Union Of Soviet Socialist Republics

Introduction. With the exception of Cuba, little is known about either the extent or the degree of exploitation of the cobalt reserves-resources of the Soviet Union and other Central Economy Countries. Consequently, what little data are available only provide, at best, an incomplete picture of cobalt self-sufficiency of the Central Economy Countries.

According to the USBM, the cobalt reserves of the Soviet Union are estimated to be approximately 11,000 tons of contained metal. As far as can be determined, the majority of Soviet cobalt reserves exist in nickel-copper sulfide deposits similar to those of the Sudbury district of Ontario, Canada. In addition, a portion of the cobalt reserves of the Soviet Union occur in Blackbird-type hydrothermal vein orebodies as well as in laterite nickel deposits.

Although data are incomplete, it is either definitely known or suspected that cobalt is a byproduct of nickel-copper mining at the following locations:

Noril'sk District, Northwestern Siberia. Within the vicinity of Noril'sk are large, mineralogically complex nickel-copper and nickel-platinum sulfide deposits, most of which are associated with differentiated gabbroic and doleritic (basaltic-diabasic) intrusions. The ores of the region also contain varying amounts of cobalt, gold, selenium, silver, and tellurium.

At Noril'sk is a major smelter-refinery complex consisting of at least fifteen plants which process ores from nearby mines. The annual capacity of the complex is estimated to be approximately 100,000 to 145,000 tons of nickel and 650,000 tons of copper. Although silver and platinum group metals are known to be major byproducts of this operation, Noril'sk is reported to be the largest cobalt-producing center in the Soviet Union. No data are available concerning the amount of cobalt produced annually.

Three deposits in the vicinity of the Noril'sk smelter-refinery are believed to provide much of the feed to the processing complex:

The Noril'sk deposit is being developed by an underground mine (the Zapolyarnyy) and an open pit operation (Medvezhiy Ruchey). The ore, which occurs at less than 800 feet below the surface, is reported to grade 0.75 percent copper, 0.5 percent nickel, and 0.313 percent cobalt.

The Talnakh deposit(s) is/are located at a depth of approximately 800 feet and is/are being mined by two underground mines: the Mayak and the Komsomol'skiy. Data are conflicting concerning the ore grades of the Talnakh deposits. However, the ore is believed to contain about 3.2 percent copper, 3.1 to 3.5 percent nickel, 0.11 to 0.14 percent cobalt, and up to 11 grams of platinum group metals (PGM) per ton. It is thought by the Ontario Ministry of Natural Resources (Canada) that in 1973, the Mayak mine was producing about half of the total output of the Noril'sk region.

The Oktyabr'skiy deposit was discovered in 1961 through detailed surveying of the Talnakh orebody. The deposit lies at a depth of approximately 1,100 feet below the surface and averages 1.12 percent nickel and 0.036 percent cobalt. Although the mine is not yet believed to be in full production (insofar as the property was only brought into operation in 1975), the Ontario Ministry of Natural Resources believes that, ultimately, 50 percent of the ore production of the Noril'sk region will be provided by this mine. As of 1977, the Oktyabr'skiy mine was the largest non-ferrous underground operation in the Soviet Union.

Kola Peninsula. Aside from the mines at Noril'sk, cobaltiferous nickel-copper sulfides are also mined at Pechenga and at Monchegorsk on the Kola Peninsula. At Monchegorsk is a major nickel-copper smelter and refinery. Feed for this complex is obtained not only from mines in the immediate vicinity of Monchegorsk, but also in Cuba. The capacity of the Monchegorsk smelter-refinery is not known. However, it is thought that the capacity of the Monchegorsk facility is considerably less than that of the Noril'sk complex.

Southern Urals. The only cobaltiferous nickel laterite deposit mentioned frequently in Soviet literature is that at Burukhtal in the southern Urals. The deposit, which was brought into production in 1976, is reported to contain 0.88 percent nickel and 0.08 percent cobalt.

Although other occurrences of cobaltiferous nickel laterites in the Ural Mountains have been exploited, data are not readily available concerning their grades or production history.

1.2.3.9 Summary - Future Production

Table 1-4 classifies world cobalt resources and reserves by geologic type. As discussed in Sections 1.2.1 - 1.2.8, nearly all of the world's cobalt production in 1978 was derived from one of three geologic environments: nickeliferous laterites, copper and/or nickel sulfide orebodies, and cobalt orebodies. Given the results of recently intensified exploration efforts by domestic and foreign mining companies, and in view of geologic investigations by the USGS and other government agencies, it is probable that:

- Additional ore deposits similar to those already in production will be discovered. However, as is presently the case, the overwhelming majority of new discoveries will be either cobaltiferous nickel laterites or cobaltiferous copper and/or nickel sulfide deposits.

- Given the relative comprehensiveness of previous worldwide exploration efforts for nickel and copper, most major cobaltiferous deposits of these two metals likely to be developed over the next ten to twenty years (1990-2000+) have already been discovered. This is particularly true in the case of nickel.

- New cobalt production capacity will be from expansions to present or future producers, or due to the discovery of extensions to orebodies now being exploited.

- Although not falling into a geologic classification per-se, a limited amount of future production will be derived from the tailings of existing or proposed mining operations. For example, stockpiled tailings from the copper mills in Uganda, Zaire, and Zambia, as well as the gold tailings of South African mines, contain a significant quantity of cobalt. Such is also the case with cobaltiferous tailings from lead-zinc-mills in the Missouri Lead Belt (United States).

1.3 COBALT DEMAND

1.3.1 Introduction

Cobalt is a hard, strongly magnetic, moderately malleable and ductile metal consumed in the production of a variety of alloys and chemicals. In its metallic form, cobalt is used to:

- o Enhance the coercive force and Curie temperature of magnet materials.
- o Increase the melting point, high temperature, and ultimate tensile strength of iron and other alloys.
- o Increase the wear and corrosion resistance as well as fatigue strength of metals.
- o Facilitate the bonding properties of materials.

Cobalt-based chemicals are used in the manufacture of temperature-resistant pigments and glazes; as catalysts to promote surface-drying, to denitride and desulphurize oil products, and to facilitate hydrogenation of olefins to aldehydes; and as an additive to fertilizer. Additionally, minor amounts of the isotope cobalt-60 are used in many medical and industrial applications.

1.3.2 Specific Applications.

Data are not available concerning the specific quantities of cobalt consumed either on an aggregate basis or for a particular nation or geographic-geopolitical area, or by a given end use. Consequently, with few exceptions (the United States being one), virtually all published statistics concerning cobalt consumption are merely the best estimates of industry experts and are rarely determined through the use of reliable-accurate information. In light of this caveat, Table 1-8 presents an estimate of Western World cobalt consumption by end use for the years 1966 and 1977, as compiled by Kruszona et al., (1979).

Table 1-8

COBALT CONSUMPTION IN THE MARKET ECONOMY COUNTRIES BY END USE: 1966 and 1977

(short tons cobalt metal)

	1966	1977
Magnet Alloys	4,398.8	7,934.4
Chemical Compounds	4,084.6	6,612.0
Super-alloys	3,770.4	7,934.4
Stainless steels	1,571.0	1,057.9
Other Metallic Applications	1,571.0	1,322.4
Hard Metals	314.2	1,586.9
TOTALS	15,710.0	26,448.0

Source: Kruszona, *et al.*, 1979.

As is evident from Figure 1-5, several changes have occurred in the end use patterns of Western World cobalt consumption. Whereas the use of cobalt in the manufacture of superalloys, hard metals, magnet alloys, and other metallic applications has increased, there has been a relative decrease in demand for cobalt in chemical applications and stainless steels. To predict whether continued shifts are likely in the future end use patterns of cobalt, as well as the overall quantity of the metal consumed by Western World nations, a brief review of each major end use is necessary.

1.3.2.1 Superalloys

A principal property of cobalt is its ability to impart heat resistance and hardness to metals. Consequently, nearly one-fourth of United States as well as a large percentage of world demand for cobalt in 1978 was for use in the manufacture of jet engine components. Cobalt alloys are utilized in cast parts such as turbine blades and nozzle guide vanes for jet engines. Approximately 5 percent of the weight of a typical commercial airline jet turbine engine (such as that used on a Boeing 707) consists of cobalt-containing material.

1.3.2.1.1 Outlook

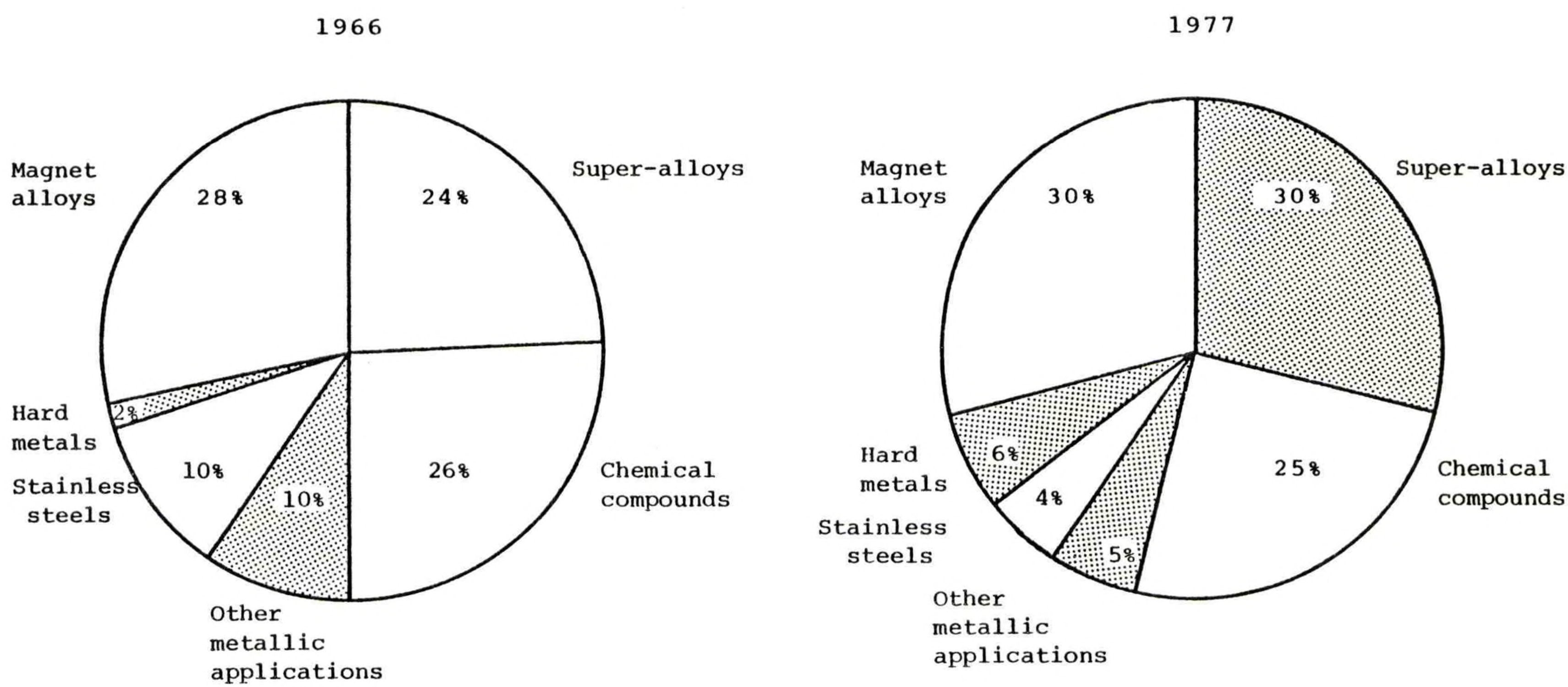
Despite claims by some aircraft industry representatives that few, if any, substitute materials exist for cobalt in its jet engine applications, numerous engine manufacturers are reducing cobalt consumption through the use of nickel-based alloys, ceramics, and the recycling of cobalt-bearing machinery scrap. Additionally, most aircraft power plant manufacturers are intensifying their research efforts with regard to the development of new powder metallurgical techniques and high-temperature, high-pressure methods of fabricating cobalt-containing compounds.

As evidence of the ongoing cobalt reduction effort by aircraft power plant manufacturers, newer jet turbines such as those used on the Boeing 727 contain only 37.4 pounds of cobalt. This is in contrast to older models of the type JT-3D-3B (Boeing 707-320) which required 132.0 pounds of cobalt.

Figure 1-5

DISTRIBUTION OF MARKET ECONOMY COUNTRIES' COBALT CONSUMPTION

BY END USE, 1966 AND 1977



It should be noted, however, that jet turbine engines for military aircraft, because of their higher performance requirements, contain significantly more cobalt than do the power plants used on commercial aircraft. For example, the Pratt & Whitney F-100 engine used on the F-15 and F-18 contains 180 pounds of cobalt. Consequently, should a war occur or should there be a general world-wide military buildup, cobalt consumption is likely to increase markedly.

Research by the United States Air Force and aircraft power plant manufacturers has indicated that by dropping non-cobaltiferous superalloy metals against a rapidly spinning disc in a helium gas environment an alloy can be produced with characteristics similar to those of a cobalt-based alloy. Based on this and other developments, aviation industry experts believe that a 20 percent reduction in cobalt use in aircraft engine components is possible over the next five years (1980-1985). However, regardless of the potential substitution of other metals, primarily nickel, for cobalt, and the increased recycling of cobalt-bearing scrap, overall demand for this metal in aerospace applications is unlikely to decrease over the near- to intermediate-term. This is particularly true insofar as many commercial airlines are now in the process of replacing existing older, less-fuel-efficient equipment with new aircraft having greater load capacities and lower fuel consumption per passenger-mile.

1.3.2.2 Magnet Alloys

Because cobalt enhances the coercive force and the Curie temperature of various metals, it is used in the fabrication of both hard and soft magnets. Whereas <u>permanent</u> or <u>hard magnets</u> are able to retain a magnetic field and have a high-coercivity, <u>soft magnets</u> are characterized by their inability to retain the magnetism induced by an external field after it has been removed. Both hard and soft magnets are used in a large number of electrical applications such as for loudspeakers, telephones, rotating electrical machinery, and measuring instruments.

1.3.2.2.1 Outlook

Because of the recent shortage of cobalt coupled with its escalating price, many magnet manufacturers have begun to substitute other materials such as barium and strontium for cobalt. One major manufacturer of high performance magnets for electronics uses has developed a magnetic material which provides five times the energy of a conventional magnet, but which contains only 20 percent of the cobalt originally required to obtain such efficiency.

Whereas the use of cobalt is desirable in developing high performance magnets, it is not critical in all applications. Consequently, some substitution of other metals and rare-earths for cobalt is likely to occur unless prices stablize and the supply-availability of cobalt increases.

Some cobalt industry experts have stated that if demand for the metal by magnet producers alone could be significantly reduced, a surplus of cobalt would develop in three to five years.

1.3.2.3 Chemical Compounds

Approximately one-fourth of the cobalt consumed by the Western World is for the manufacture of chemical compounds. Cobalt compounds are used by the glass and ceramics industries as both a colorant and a decolorizer. Cobalt is "one of the more satisfactory" colorants for use in glass and ceramic products insofar as its effect is extremely stable. When glass sands are found to contain iron, cobalt can be used to neutralize the yellow coloration which the iron could impart to the final glass products.

Cobalt is commonly used as catalyst to aid in the desulfurization of petroleum fractions, the hydrogenation of certain oils to edible products, and the drying of finishes based on oxidizing oils.

1.3.2.3.1 Outlook

Because of the relatively small quantities of cobalt used in glass and ceramic applications, consumption of the metal by this end use is not expected to impact substantially on total demand. However, as a catalyst, the consumption of cobalt is expected to remain strong. As stated by the USBM, substitutes for cobalt as a dryer in paints or as a catalyst is "usually not effective." Although manganese, when complexed with certain aromatic amines, has been determined to be a suitable substitute for cobalt in a select number of drying applications, its effectiveness diminishes with increasing humidity. In many instances, the best driers are achieved by using lead, cobalt, and manganese together in varying combinations insofar as each metal imparts unique drying characteristics.

Other elements such as aluminum, molybdenum, nickel, and tungsten can be used either partially, completely in place of, or in conjunction with cobalt as a catalyst in the production of petroleum products. Although the use of cobalt in this application is likely to continue to increase, efforts are now being made to recover and recycle cobalt contained in catalysts, thereby somewhat reducing primary demand for this metal.

1.3.2.4 High-Speed and Tool Steels - Cemented Carbides (Hard Metals)

Cobalt is a major constituent in very high-strength and abrasion-resistant steels. Cobalt metal and tungsten carbide powders are the basic materials used in the manufacture of cemented carbides.

1.3.2.4.1 Outlook

High-speed tool steels, the most highly alloyed of the tool and die steels, are used in the manufacture of cutting tools. The principal attribute of cobalt containing high-speed steels is its ability to retain sufficient hardness over long periods of time to cut metal at the increased temperatures generated by the friction of cutting. Although chromium, molybdenum, tungsten and vanadium are used in varying amounts along with cobalt in the manufacture of tool steels, cobalt is a necessary additive and tends to enhance and complement the qualities provided by these other elements. Consequently, demand for cobalt in this application is not likely to diminish over the foreseeable future. This is particularly true insofar as Western World production of high-speed tool steels has been increasing steadily over the past decade.

Cobalt serves as a binder in the production of extremely hard and wear-resistant materials made essentially of a refractory metal cemented together by a metal alloy (often comprised of cobalt). Although other materials exist which could be used as a substitute in this application, cobalt is still the preferred constituent. According to Kruzona et al.(1979), the average proportion of cobalt for which a binding agent cannot be substituted is approximately 9.5 to 10.0 percent.

1.3.2.5 Stainless Steels

According to the American Iron and Steel Institute, a stainless steel is any iron-based alloy which contains a minimum of 4 percent chromium with or without the addition of other elements. Typically, stainless steel will contain 66 to 71 percent iron, 18 to 20 percent chromium, 8 to 10.5 percent nickel (which is added to increase the corrosion-resistance and toughness of the alloys), and varying minor amounts (less than 2 percent) of manganese, carbon, phosphorous, sulfur, and silicon.

1.3.2.5.1 Outlook

As indicated on Figure 1-3, demand for cobalt in stainless steel is decreasing as a percentage of total Western World demand. Insofar as cobalt is usually only a minor addition to most stainless steels, and given the availability of alternative materials for use in corrosive environments (for example, plastics and plastic coatings, and other corrosion-resistant metal claddings), demand for cobalt in this application is unlikely to increase significantly in the near to intermediate future.

1.3.3 Substitute Alternative Materials: A Summary

As indicated in the foregoing discussion, substitute materials exist for cobalt in a number of applications. Nickel can be used instead of cobalt in some superalloys. For some applications, ceramic materials can be used in place of superalloys--regardless of their cobalt content. Ceramic magnets made from barium and strontium ferrite are suitable for use in a number of instances where cobalt-bearing magnetic materials are now employed. However, to date, no totally satisfactory substitute materials have been found for cobalt in some catalytic uses, or as a binder of carbides. Consequently, over the long-term, demand for cobalt probably could be reduced through substitution--particularly of ceramic materials. However, in virtually all uses to which cobalt is presently applied, its performance meets or exceeds that of existing possible alternatives, even though in many instances the alternative materials are often less costly or easier to obtain than cobalt.

1.3.4 SECONDARY SOURCES

Unlike many metallic minerals, very little cobalt is, or can be, recovered and/or recycled. This is in large part due to the fact that in most applications, only small amounts of cobalt are used. Consequently, recovery is economically unjustified, particularly in light of increased energy costs.

Because cobalt, in its application as an alloying element, is so disseminated throughout the host matrix, relatively few sources of cobalt-bearing scrap contain sufficiently significant concentrations of cobalt to warrant its recovery. This situation is somewhat compounded insofar as few cobalt-containing products, for example, jet engine components, completely wear out over short periods of time.

The largest potential source of cobalt-bearing recyclable scrap is derived from the forming and fabrication of cobalt-bearing superalloys. In the manufacture and shaping of some cobalt-containing components, much of the original ingot metal content of the form is machined away and therefore becomes scrap.

At present, according to the USBM, approximately 2 percent of United States, 4 percent of "Rest of World," and 3 percent of world demand for cobalt is satisfied by <u>reported</u> reclaimed-recycled metal. However, both the USBM and other industry experts believe that the actual amount of demand filled by reclaimed-recycled cobalt metal is probably closer to 5 percent. It is unlikely that this percentage of recycling will change appreciably over the near-to-intermediate-term.

1.3.5 Projections Of Cobalt Demand

As discussed previously in this section, data concerning the historic demand for cobalt in both the aggregate and by end use are generally unavailable. Consequently, it is extremely difficult to establish with any degree of accuracy consumption patterns which typify cobalt. Similarily, the sensitivity of cobalt demand to economic and political events is also indeterminable. However, given an analysis of the end uses for cobalt, demand for this metal probably would increase in time of war and/or economic prosperity.

Numerous projections of world cobalt demand have been developed. However, insofar as little information is available concerning existing levels of world cobalt consumption, most estimates, from a quantitative standpoint for a given year, are dissimilar. Therefore, the reader is cautioned to regard any projections of cobalt demand as merely indicative of possible levels of metal consumption and not as a definitive guide to future demand.

As is the case for most other metals, demand projections for cobalt are developed for relatively short periods of time (seldom exceeding twenty years), and are based in large part on historical demand patterns and annual rates of growth in consumption of the metal. Because of the aforementioned lack of demand data, many projections of cobalt consumption are, to some extent, established through analyses of the availability of supply--the premise being that supply is more or less equivalent to demand. Table 1-9 presents historical rates of annual cobalt demand for a select number of nations and geographic areas for the period 1951-1955 to 1971-1975.

In light of the data presented in Table 1-9, it is apparent that a wide diversity exists in the rates of growth for various nations and geographic areas. Cobalt consumption by developing countries (for example, Asia, Latin American, and China) tends to be at a greater rate of growth than for more industrialized nations.

As stipulated by the United States Department of Commerce in its request for proposal, USBM projections of United States, market economy, and total world demand for cobalt are to be used in this study as a primary source of data concerning future levels of cobalt consumption. However, insofar as several projections of cobalt demand recently have been made by various groups, and given the fact that USBM estimates are for the United States and world only, other demand projections have been incorporated in development of the proceeding analyses.

Tables 1-10 and 1-11 give twelve projections of cobalt demand for the United States, world, and Central Economy Countries over the period 1980 through 2010. Although numero s projections of cobalt demand were reviewed prior to the completion of Table 1-10, two principal sources of data were selected given their similarity to other estimates and completeness in documentation-methodology. These were:

United States and World:

- United States Bureau of Mines Mineral Commodity Profile: Cobalt, July 1977.

TABLE 1-9

HISTORICAL RATES OF ANNUAL COBALT DEMAND

Region	Growth Rate 1951-1955 to 1971-1975
Western Europe	4.6
Japan	14.5
Other Developed Lands:	
Australia Canada Israel New Zealand Republic of South Africa	5.8
U.S.S.R.	8.0
Eastern Europe	4.8
Africa (less the Republic of South Africa)	3.6
Asia (less Israel, Japan, and Mainland China)	18.2
Latin America	7.0
China	13.5
United States	2.5
World	4.9

Sources: Malenbaum, 1978, p. 76.

Table 1-10

ESTIMATES OF PRIMARY COBALT DEMAND-DOCUMENTATION
(thousand tons cobalt)

	BASE YEAR		Rate of Growth (%)	1980	1985	1990	1995	2000	2005	2010
UNITED STATES										
Medium	1972.5	11.72	3.27	14.92	17.52	20.57	24.16	28.37	33.31	39.12
	@ 5% recycle		3.27	14.17	16.64	19.54	22.95	26.94	31.64	37.16
USBM	1977 TL		3.32	11.09	13.06	15.37	18.09	21.30	25.07	29.51
	1976 TL		2.87	11.17	12.87	14.83	17.09	19.69	22.69	26.14

From: Malenbaum, Wilfred, World Demand for Raw Materials in 1985 and 2000, 1978, pp. 76, 88.

	BASE YEAR		Rate of Growth (%)	1980	1985	1990	1995	2000	2005	2010
Medium	1976	9.92	2.94	11.14	12.87	14.88	17.20	19.88	22.98	26.56
	@ 5% recycle		2.94	10.58	12.23	14.14	16.34	18.89	21.83	25.23

From: Kruszona, Manfred, Cobalt, 1979, p. 236.

	BASE YEAR		Rate of Growth (%)	1980	1985	1990	1995	2000	2005	2010
WORLD										
Medium	1972.5	25.82	3.27	32.86	38.59	45.32	53.22	62.51	73.39	86.19
	@ 5% recycle		3.27	31.22	36.66	43.05	50.56	59.38	69.72	81.88
USBM	1977		3.38	27.69	32.69	38.59	45.57	53.80	63.52	75.00
	1976		3.19	42.50	49.74	58.20	68.11	79.70	93.27	109.14

From: Malenbaum, Wilfred, World Demand for Raw Materials in 1985 and 2000, 1978, pp. 76, 88.

	BASE YEAR		Rate of Growth (%)	1980	1985	1990	1995	2000	2005	2010
Medium	1976	31.08	3.26	35.34	41.49	48.71	57.19	67.14	78.83	92.54
	@ 5% recycle		3.26	33.57	39.89	46.27	54.33	63.78	74.89	87.91

From: Kruszona, Manfred, Cobalt, 1979, p. 236.

CENTRAL ECONOMY COUNTRIES

Medium 1972.5 - USSR, Soviet Bloc plus Albania, Yugoslavia, China plus Mongolia, North Korea, and North Vietnam.

	BASE YEAR		Rate of Growth (%)	1980	1985	1990	1995	2000	2005	2010
	USSR	2.18	3.14	2.75	3.21	3.74	4.37	5.11	5.95	6.95
	E. Europe	1.22	3.36	1.56	1.84	2.18	2.57	3.03	3.57	4.21
	China	0.22	6.49	0.35	0.48	0.66	0.91	1.24	1.70	2.33
	TOTAL	3.62	3.57	4.66	5.53	6.58	7.85	9.38	11.22	13.49
	@ 5% recycle		3.57	4.43	5.25	6.25	7.46	8.91	10.66	13.82

From: Malenbaum, Wilfred, World Demand for Raw Materials in 1985 and 2000, p. 76.

- German Democratic Republic, Poland, Czechoslavakia, USSR, China, rest of Eastern Bloc.

	BASE YEAR		Rate of Growth (%)	1980	1985	1990	1995	2000	2005	2010
Medium	1976	7.6	2.91	8.52	9.84	11.35	13.10	15.12	17.44	20.13
	@ 5% recycle		2.91	8.09	9.35	10.78	12.45	14.36	16.57	19.12

From: Kruszona, Manfred, Cobalt, 1979, p. 236.

Table 1-11

ESTIMATES OF PRIMARY COBALT DEMAND-DOCUMENTATION
(thousand tons of cobalt)

	DOCUMENTATION BASE YEAR		Annual Rate of Growth (%)	1980	1985	1990	1995	2000	2005	2010
UNITED STATES										
Low	1977 TL	10.06	1.75	10.60	11.56	12.61	13.75	15.00	16.36	17.85
Medium	1977 TL	10.06	3.32	11.09	13.06	15.37	18.09	21.30	25.07	29.51
High	1977 TL	10.06	4.01	11.32	13.78	16.77	20.41	24.85	30.25	36.82
From: U. S. Bureau of Mines, Mineral Commodity Profile: Cobalt (Preliminary Data), 1979.										
REST OF WORLD										
Low	1977	15.00	2.06	15.95	17.66	19.56	21.67	24.00	26.58	29.44
Medium	1977	15.00	3.42	16.59	19.63	23.22	27.47	32.50	38.45	45.49
High	1977	15.00	4.22	16.99	20.89	25.69	31.59	38.85	47.78	58.76
From: U. S. Bureau of Mines, Mineral Commodity Profile: Cobalt (Preliminary Data), 1979.										
WORLD										
Low	1977	25.06	1.94	26.55	29.23	32.18	35.42	39.00	42.94	47.27
Medium	1977	25.06	3.38	27.69	32.69	38.59	45.57	53.80	63.52	75.00
High	1977	25.06	4.14	28.30	34.67	42.46	52.01	63.70	78.02	95.56
From: U. S. Bureau of Mines, Mineral Commodity Profile: Cobalt (Preliminary Data), 1979.										
UNITED STATES										
Low	1976 TL	9.98	1.27	10.49	11.17	11.90	12.68	13.50	14.38	15.31
Medium	1976 TL	9.98	2.87	11.17	12.87	14.83	17.09	19.70	22.69	26.14
High	1976 TL	9.98	3.67	11.52	13.80	16.53	19.79	23.70	28.38	33.99
From: U. S. Bureau of Mines, Mineral Commodity Profile: Cobalt, 1977.										
WORLD										
Low	1976	37.48	1.82	40.29	44.09	48.25	52.81	57.80	63.26	69.23
Medium	1976	37.48	3.19	42.50	49.74	58.20	68.11	79.70	93.27	109.14
High	1976	37.48	3.98	43.82	53.27	64.76	78.72	95.70	116.34	141.43
From: U. S. Bureau of Mines, Mineral Commodity Profile: Cobalt, 1977.										

o United States Bureau of Mines Mineral Commodity Profile: Cobalt, October 1979.

Central Economy Countries:

o Kruzona, Manfred, et al., Cobalt, 1979.

o Malenbaum, Wilfred, World Demand for Raw Materials In 1985 and 2000, 1978.

With the exception of USBM data, the majority of the projections selected were for total cobalt demand (inclusive of both primary and secondary metal consumption). Therefore, given USBM and other supporting information, all non-USBM estimates for total cobalt demand have been reduced by 5 percent to account for scrap recycling and reclamation (Table 1-10).

Table 1-12 presents the projected supply of and demand for cobalt by the United States, Central Economy Countries, market economy, and total world for the period 1980-2010. As is evident from this table:

- o Regardless of whether total or adjusted mine output for contained metal is used, United States demand for cobalt far exceeds any projected level of domestic supply. Therefore, unless the United States either reduces its demand for cobalt significantly, or discovers major new deposits of cobalt within its borders, the nation probably will have to continue to rely on foreign sources of cobalt supply through the study period.

- o Although large deposits of cobaltiferous nickel sulfides and laterites are known to exist in Central Economy Countries, the adjusted cobalt production of this group of nations is insufficient to meet projected demand over the period 1980-2010. Should the efficiency of cobalt

Table 1-12

PRIMARY COBALT SUPPLY-DEMAND
(thousand tons of cobalt)

	Annual Rate of Growth 1980-2010	1980	1985	1990	1995	2000	2005	2010
UNITED STATES								
Supply - Unadjusted	—	--	2.95	3.70	4.20	3.95	1.75	1.75
Adjusted	—	--	1.91	2.37	2.62	2.49	0.95	0.95
Demand -								
1) Low	1.75	10.6	11.56	12.61	13.75	15.00	16.36	17.85
2) Medium	3.32	11.09	13.06	15.37	18.09	21.30	25.07	29.51
3) High	4.01	11.32	13.78	16.77	20.41	24.85	30.25	36.82
WORLD								
Supply - Unadjusted	—	31.60	47.50	58.63	59.47	61.42	58.92	58.92
Adjusted	2.09	27.77	40.89	51.25	52.08	53.48	51.67	51.67
Demand -								
4) Low	1.94	26.55	29.23	32.18	35.42	39.00	42.94	47.27
5) Medium	3.38	27.69	32.69	38.59	45.57	53.80	63.52	75.00
6) High	4.14	28.30	34.67	42.46	52.01	63.70	78.02	95.56
CENTRAL ECONOMY COUNTRIES								
Supply - Unadjusted	—	3.60	6.50	6.50	6.50	7.70	7.70	7.70
Adjusted	2.57	2.16	3.90	3.90	3.90	4.62	4.62	4.62
Demand -								
7) Low	3.57	4.43	5.25	6.25	7.46	8.91	10.66	12.82
8) Medium	3.17	6.26	7.30	8.52	9.96	11.64	13.62	15.97
9) High	2.91	8.09	9.35	10.78	12.45	14.36	16.57	19.12
MARKET ECONOMY								
Supply - Unadjusted	—	28.00	41.00	52.13	52.97	53.72	51.22	51.22
Adjusted	2.05	25.61	36.99	47.35	48.18	48.86	47.05	47.05
Demand -								
10) Low	1.49	22.12	23.98	25.93	27.96	30.09	32.28	34.45
11) Medium	3.44	21.43	25.39	30.07	35.61	42.16	49.90	59.03
12) High	4.53	20.21	25.32	31.68	39.56	49.34	61.45	76.44

SOURCES OF PROJECTIONS

All supply projections compiled by Dames & Moore.

1) 2) 3) - United States Bureau of Mines, Mineral Commodity Profile: Cobalt, (Preliminary Data), 1979.

4) 5) 6) - United States Bureau of Mines, Mineral Commodity Profile: Cobalt, (Preliminary Data), 1979.

7) - Malenbaum, Wilfred, World Demand for Raw Materials in 1985 and 2000, 1978.

8) - Is an average of Demand Projections 7 and 9.

9) - Kruszona, Manfred, Cobalt, 1979.

10) 11) 12) - Derived through subtraction of Central Economy Countries data from World data.

Unadjusted - Unadjusted mine production data (no provisions made for efficiency of recovery).

Adjusted - Mine production data adjusted to account for probable losses due to mining, milling and smelting inefficiencies.

extraction by Central Economy Countries be increased to virtually total recovery, only low demand would be met and only for the years 1985-1990. Consequently, with regard to cobalt, the Central Economy Countries are in a similar position to that of the United States insofar as they will likely have to continue to rely on foreign exports of the metal to satisfy future demand.

- o Total Market Economy adjusted production of cobalt appears to be sufficient to meet this block's expected demand regardless of level (low, medium, high) through 1995. Beyond that time, shortfalls are likely to occur in supply, particularly with respect to medium and high levels of cobalt demand.

- o World demand for cobalt very likely will exceed adjusted supply by the year 2000 unless the projected rate of growth in demand drops to less than 2.22 percent. Should unadjusted supply be considered, demand is not anticipated to exceed production until 2005 and 2000, given a high and medium rate of growth in consumption, respectively.

In evaluating these projections, it is important to note that they represent a static situation which ignores the possibility of inventory accumulations. It is possible that many of the shortfalls projected in Table 1-12 could be delayed should inventory accumulations of cobalt be used to satisfy demand during periods in which installed production capacity proves to be inadequate, (Figure 1-6).

In summary, whether world production of cobalt will meet projected demand is contingent on several factors:

- o The future demand for copper and, more particularly, nickel.

- o The cost and availability of energy necessary to beneficiate and smelt lateritic ores.

COBALT

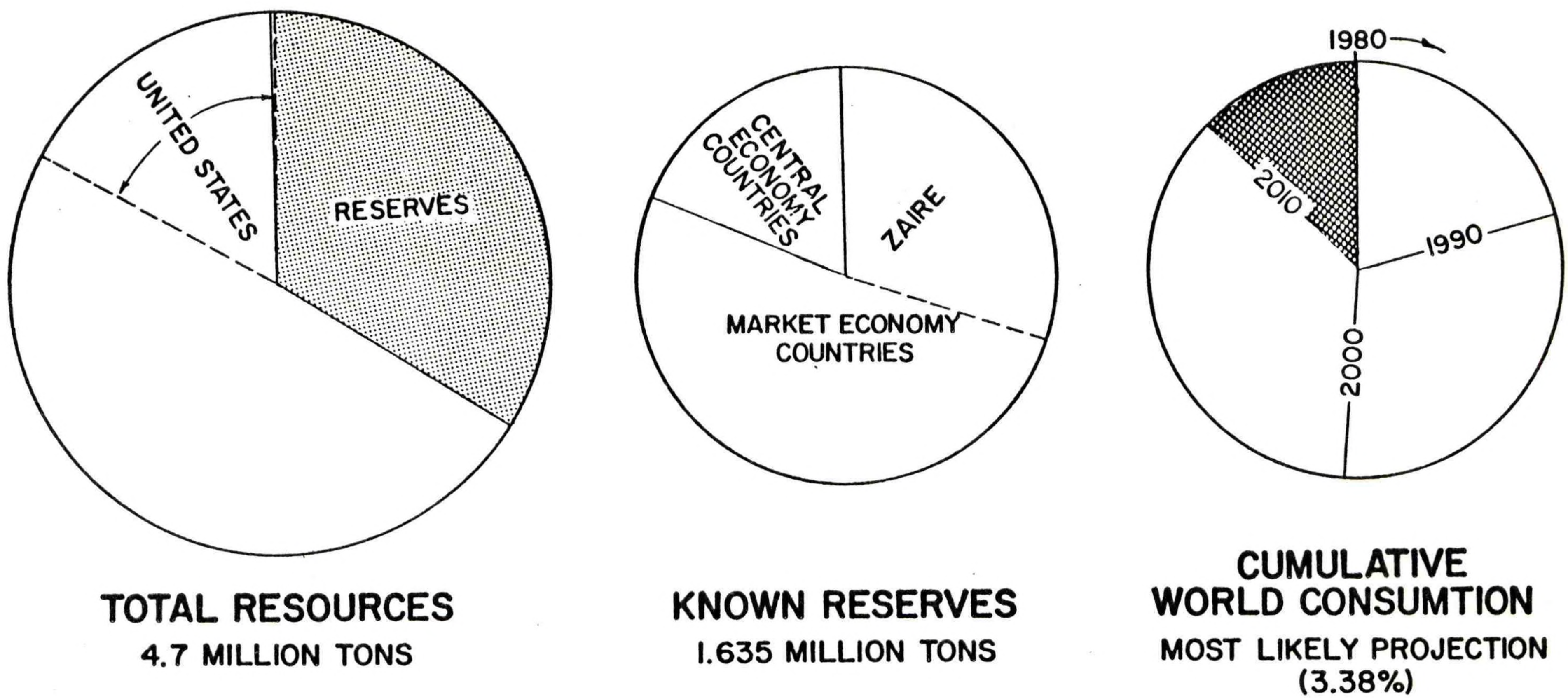


Figure 1-6

- The inauguration of new mines projected to come on-stream during the period 1980-2010, and the continued operation of those properties already on-line for the lifetimes stated in the supporting tables.

- The maintenance of political stability in southern Africa.

- The improvement of cobalt extractive technology.

- The degree of substitution of other metals and materials for cobalt in various applications.

2.0 COPPER: SUPPLY-DEMAND

2.1 INTRODUCTION

World reserves and resources of copper are large and widely distributed. Whether or not they are developed at a pace rapid enough to meet forecast demand will be largely a function of the market price of copper. Historically, changes in copper availability tend to lag behind the economic conditions which determine demand, resulting in a cyclical pattern of shortfalls and surpluses.

In Part I of the discussion which follows, the economic geology of world copper reserves and resources are reviewed. The five major geologic environments in which copper occurs are discussed and examples of significant deposits belonging to each generic type described. Part II reviews the reserves and resource position of the major copper producing regions of the world. Deposits under development are discussed, and major deposits awaiting development tabulated. Site specific mine capacity projections are made for the next 5 to 10 years: beyond that, likely candidates for production are suggested. An investigation of copper demand follows, with a look at past consumption as well as projected demand.

2.2 COPPER SUPPLY

2.2.1 Part I - The Economic Geology of Copper

In an optimal sense, the world's supply of copper has an upper bound limited only by the finite quantity of copper in the earth's crust averaging approximately 50 parts per million, or 0.005 percent. Economic and technological constraints, however, restrict mining to much greater concentrations of copper-bearing minerals.

Copper occurs in three major groups of minerals: sulfides, carbonates, and silicates, with sulfides being the most important and silicates the least. Since World War I, copper sulfides have become the principal ore mined, primarily chalcopyrite and chalcocite, and, to a lesser extent, bornite, covellite and enargite, along with the associated carbonates azurite and malachite. Chalcopyrite ($CuFeS_2$) is the most widely occurring copper sulfide mineral, and is the principal primary copper mineral in most porphyry copper deposits. Chalcocite (Cu_2S) occurs principally as a supergene mineral in enriched zones of sulfide deposits.

Most copper deposits can be classifed into one of five geological types:

- o Native copper deposits,
- o Porphyry and vein and replacement deposits,
- o Stratabound deposits in sedimentary rocks,
- o Massive sulfide deposits in volcanic rocks, and
- o Copper in nickel ore deposits.

2.2.1.1 Native Copper Deposits

Native copper was originally man's primary source of the metal. Most commonly associated with basaltic lavas, the deposition of copper is believed to have resulted from the reaction of hydrothermal solutions with iron oxide minerals. The largest known deposit of this type is on the Keweenaw Peninsula, Michigan, on the shores of Lake Superior. While no longer being exploited, over 5 million tons of copper have been produced since 1845. With the growth in demand for copper that came with industrialization, deposits of natuve copper were insufficient in size and quantity to meet demand. Consequently, other lower grade copper deposits were brought into production to satisfy demand.

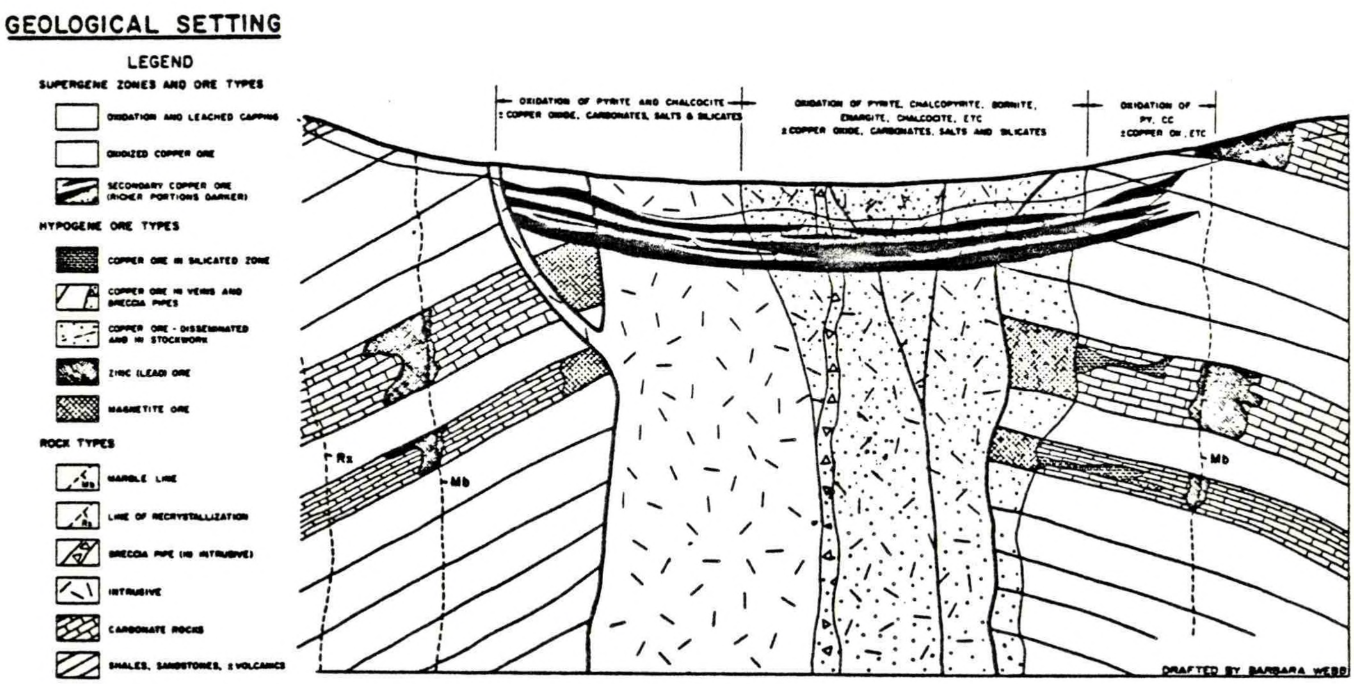
2.2.1.2 Porphyry Copper Deposits and Vein and Replacement Deposits

Porphyry copper deposits in the broadest sense are defined as large, relatively low-grade occurrences of disseminated copper mineralization genetically associated with intrusions of felsic igneous rocks. Vein and replacement-type deposits are generally associated with porphyry intrusions as localized concentrations in favorable host rocks. Although there are many theories of porphyry copper ore genesis, most deposits are thought to have formed as cupriferous magmatic intrusions cooled and crystallized, trapping part of the contained copper in place, thus producing the characteristic disseminated mineralization. Some copper is thought to have remained dissolved in the magmatic fluids and with changes in the fluid temperature, pressure and composition, was deposited in faults and fractures, thereby producing vein deposits. On occasion, some copper escaped these two processes and formed replacement-type deposits in nearby reactive host rocks. In many instances, supergene enrichment processes increased the grade of the original copper-bearing zone, thereby enabling some economically marginal porphyry copper deposits to be profitably mined. A typical deposit cross-section is shown in Figure 2-1.

This class of deposits is the most commonly occurring, and contains approximately two-thirds of the world's copper resources and is the source for approximately 46 percent of current copper production. Porphyry copper deposits are found primarily in three major belts; one along the western side of North and South America, another along the island arc in the southwest Pacific, and the third through the Alpide Belt in the Middle East (see Figure 2-2).

Figure 2-1

TYPICAL PORPHYRY COPPER DEPOSITS



Source: Titley and Hicks, Geology of the Porphyry Copper Deposits

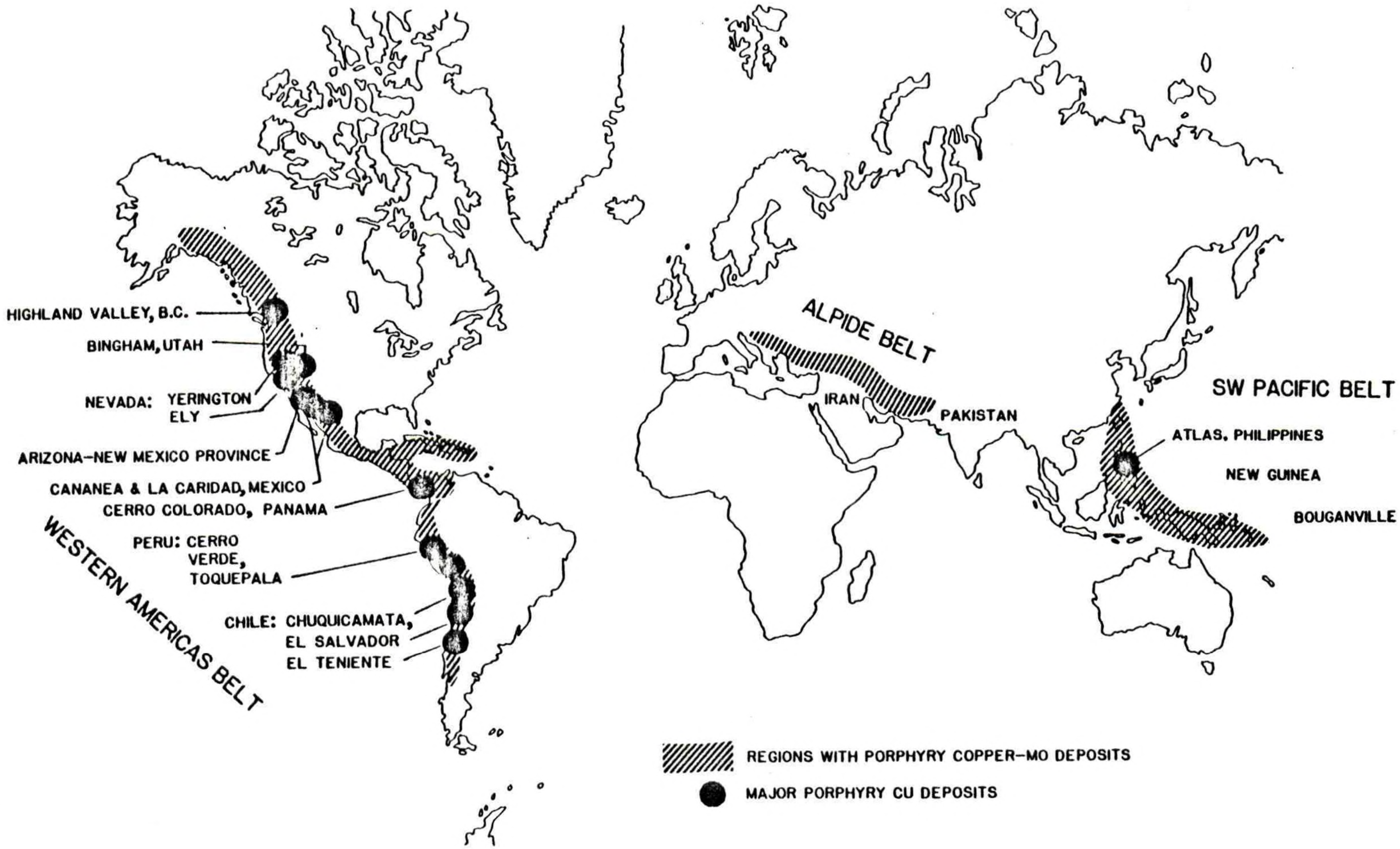


Figure 2-2

IMPORTANT COPPER PORPHYRY PROVINCES

Typical grades and tonnages vary greatly in porphyry copper deposits, but some generalizations can be made. The Andean porphyries (in Chile and Peru) are, on average, both the largest and highest grade deposits of this class, averaging 500 million to a billion tons of ore grading 1 to 2 percent copper (Table 2-1).

Table 2-1

CHILEAN COPPER RESOURCES, 1977

	MEASURED*			INDICATED*			HYPOTHETICAL*			TOTAL		
Operating Mines	Ore, million tons	Assay % Cu	Copper content, tons	Ore, million tons	Assay % Cu	Copper content, tons	Ore, million tons	Assay % Cu	Copper content, tons	Ore, million tons	Assay % Cu	Copper content, tons
Chuquicamata	624.2	1.23	8,613	910.5	0.99	9,013	8,853	0.46	40,724	10,387.3	0.56	58,350
Exotica	163.8	1.61	2,637	16.4	1.61	264	—	—	—	180.2	1.61	2,901
El Salvador	150.0	1.28	1,920	125.0	1.04	1,300	37	1.16	429	312.0	1.17	3,649
Andina	120.0	1.29	1,548	200.0	1.10	2,200	100	0.80	800	420.0	1.08	4,528
El Teniente	2,850.0	1.16	33,060	1,200.0	0.90	10,800	4,300	0.30	12,900	8,350.0	0.68	56,760
Disputada	100.0	1.40	1,400	—	—	—	—	—	—	100.0	1.40	1,400
Mantos Blancos	50.0	1.50	750	—	—	—	—	—	—	50.0	1.50	750
Subtotal	4,058.0	1.23	49,928	2,491.9	0.96	23,577	13,290	0.41	54,953	19,800.0	0.65	128,358
Deposits Under Development												
El Abra	—	—	—	1,368.0	0.59	8,071	—	—	—	1,368.0	0.59	8,071
Pelambres	—	—	—	430.0	0.78	3,350	—	—	—	430.0	0.78	3,350
Andacollo	—	—	—	300.0	0.69	2,080	—	—	—	300.0	0.69	2,080
Pampa Norte	—	—	—	242.0	0.70	1,694	—	—	—	242.0	0.70	1,694
Otros (Others)	—	—	—	—	—	—	800	0.90	7,200	800.0	0.90	7,200
TOTAL RESOURCES	4,058.0	1.23	49,928	4,832.0	0.80	38,772	14,090	0.44	62,153	22,940.0	0.66	150,753

* Terms used according to U.S. Bureau of Mines resource classification, Bulletin 667.
Source: Engineering Mining Journal, October 1978.

Deposits in the Southwest porphyry copper province in the United States and Mexico are generally more moderate in both size and tenor than those in the Andean Province, although do exist. Typically, grade and tonnage run 0.4 to 0.8 percent copper and 200 to 500 million tons of ore, respectively. Yerrington (Nevada), Lakeshore (Arizona), and Silver Bell (Arizona) in the United States, and La Caridad (0.8 percent copper, 660 million tons) in Mexico are representative of these deposits.

Canadian and Philippine prophyry copper deposits, on average, grade 0.3 to 0.5 percent copper and contain 50 million to 200 million tons of ore each. The Island Copper (0.5 percent copper, 230 million tons) and Maggie (0.34 percent copper, 198 million tons) deposits in Canada, and the Dizon deposit (0.45 percent copper, 90 million tons) in the Philippines are typical. Other deposits of the "island arc" type in the Southwest Pacific Belt average 0.5 percent copper, and 350 million tons of ore each. Examples in Papua New Guinea include Frieda River (0.45 percent copper, 366 million tons), Yandera (0.42 percent copper, 338 million tons) and Bougainville (0.46 percent copper, 800 million tons).

Porphyry deposits in the Alpide Belt are more difficult to characterize, encompassing such diverse orebodies as the Iranian supergiant, Sar Cheshmeh (1.13 percent copper, over 500 million tons) and a cluster of three small, low-grade porphyries near Saindak, Pakistan (0.3 to 0.7 percent copper, 30 to 200 million tons each).

Because of their relatively low-grade, an important consideration in the economic viability of many porphyry copper deposits is their frequent association with other recoverable minerals. Molybdenum, gold, and silver are commonly associated with copper porphyry deposits. Lead, rhenium, selenium, tellerium, and zinc are also sometimes associated with the copper mineralization.

Vein and replacement-type deposits found near, or in association with porphyry copper deposits are difficult to characterize in terms of grade and tonnage. However, these deposits are generally small in comparison to the associated porphyry deposits. In Arizona, the average copper metal content is somewhat less than 350,000 tons per deposit. Larger examples of vein and replacement-type deposits associated with porphyries include Carr-Fork (next to Bingham) in Utah, Magma in Arizona, and Butte, Montana. In many cases these deposits are regarded simply as higher grade regions of the porphyry itself.

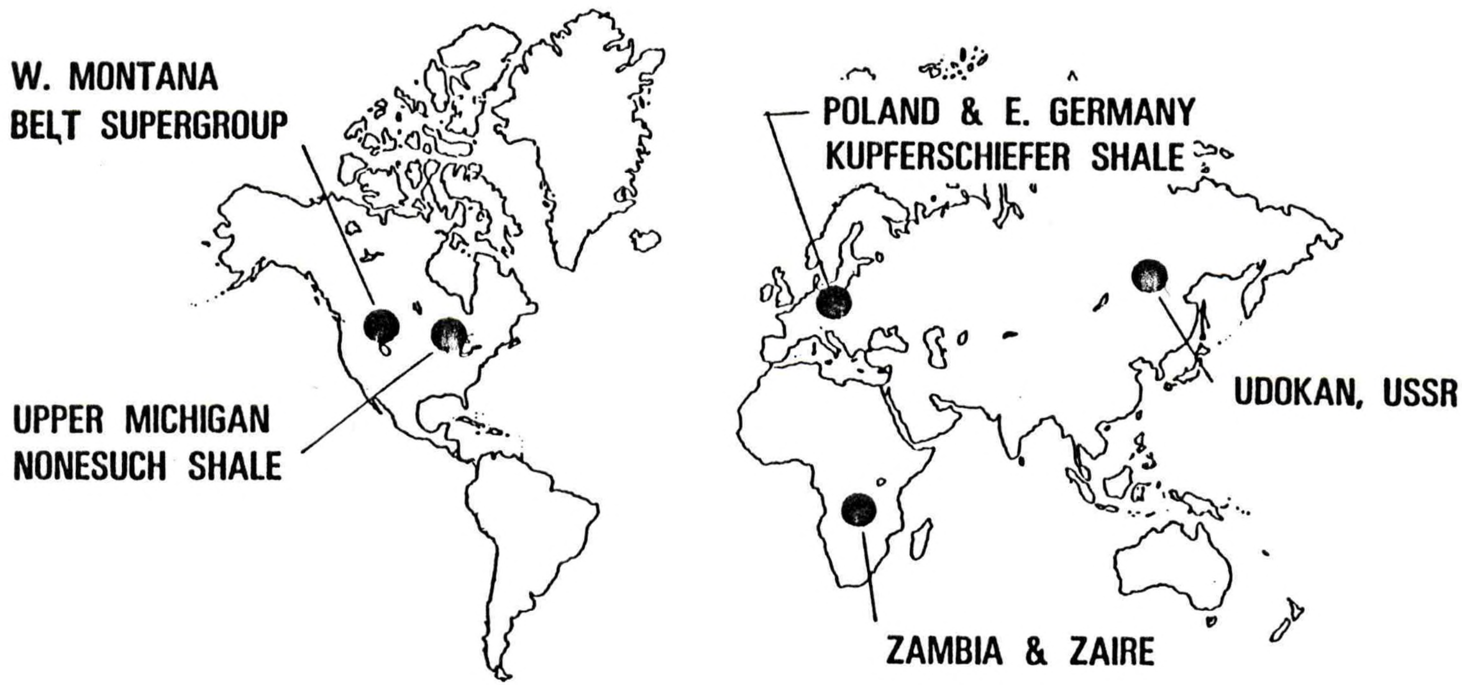
2.2.1.3 Stratabound Copper Deposits in Sedimentary Rocks (Figure 2-3)

Although stratabound deposits are apparently less geologically common in occurrence than porphyries, they are the second most important source of copper in terms of metal reserves, containing approximately 20 percent of the world's total identified reserves. The copper-bearing minerals, typically silicates, carbonates and sulfides, occur primarily in marine sediments. Many theories of ore genesis exist concerning stratabound copper deposits. One of the most commonly held theories suggests that these deposits were probably formed when atmospheric oxygen became sufficiently abundant (during the middle Precambrian) to oxidize, and thence mobilize and redeposit previously stable copper occurrences.

Stratabound copper deposits are generally smaller and of a higher grade than porphyry deposits. Their size varies greatly; deposits have been noted from one million to 100 million tons of ore, or larger.

The largest and best known developed deposits of this category are those of the African copper belt, which stretches across 300 miles of Zambia and Zaire. These two nations account for nearly 20 percent of the Market Economy Countries' annual copper production, and an even larger share of export capacity. The Zambian deposits, while larger, are lower grade than the neighboring Zairian deposits: 2 to 4 percent copper in sulfide minerals versus 4 to 6 percent copper in carbonate and silicate rocks.

Other important stratabound copper deposits include the metal rich Kupferschiefer Shale in the German Democratic Republic (East Germany) and Poland, and the White Pine District in Michigan. The Kupferschiefer is a 25 to 50 centimeter thick bed of bituminous shale, which forms the basis for Poland's growing copper production (as well as co-products zinc and lead). Four major mines are currently, or soon will be, exploiting thicker sections of the bed: Lubin, Polkowice, Rudna, and Sieroszowice. The average grade of the Kupferschiefer Shale in this region is said to be around 1.5 percent copper.



IMPORTANT COPPER RESERVES & RESOURCES
IN STRATABOUND SEDIMENTARY DEPOSITS

Figure 2-3

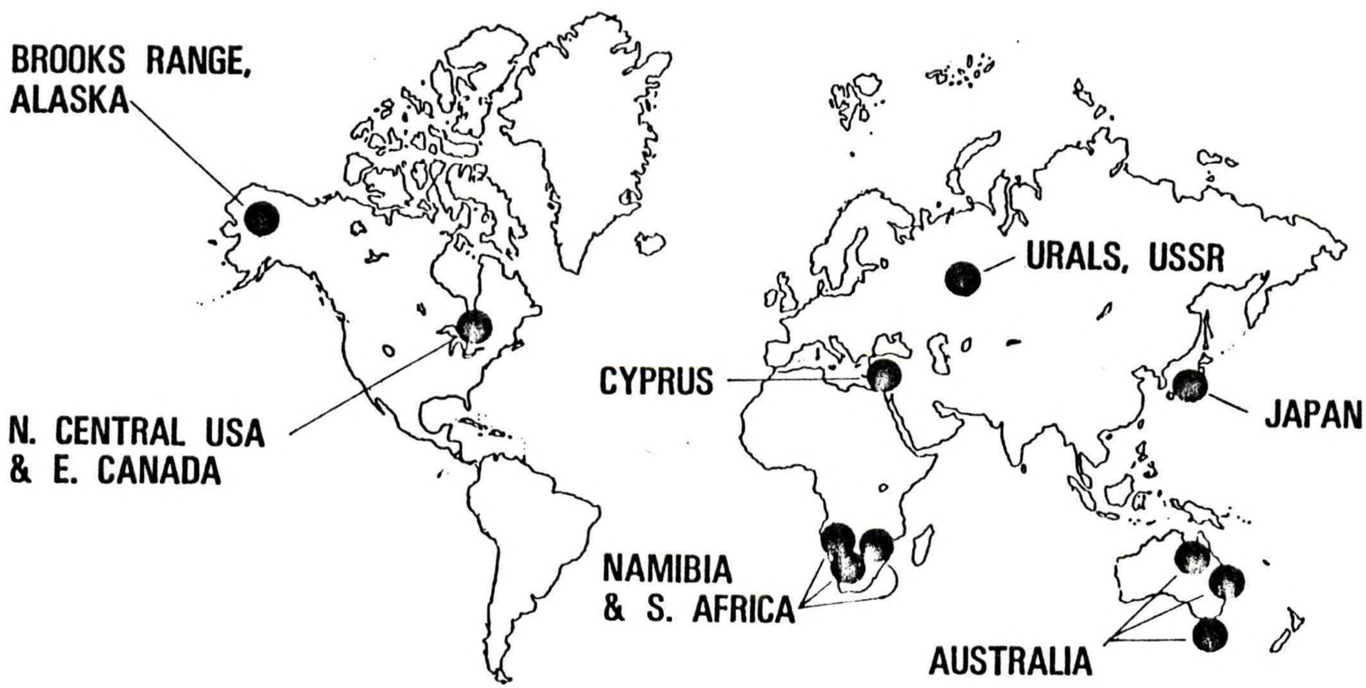
At the White Pine District in Ontonogan County, Michigan, copper occurs in the Nonesuch Shale Formation: a single belt 3 to 25 feet thick, spanning at least 160 miles of the Upper Peninsula of Michigan and northern Wisconsin. This is the only major stratabound copper deposit presently in production in the United States. The chalcocite mined here has accounted for slightly less than 5 percent of United States copper production in recent years, but this percentage is declining due to relatively high production costs versus those of open-pit prophyry copper mines.

One of the largest stratabound sedimentary copper deposits in the world is located in Eastern Siberia. The Udokan copper deposit has identified resources of over 800 million tons of ore with an average grade of 1.15 percent copper. The U.S.S.R. is planning to develop this deposit sometime during the period 1981 to 1990. Negotiations with British, French, and Japanese firms to develop the Udokan deposit on a joint basis have taken place. The problems which must be surmounted include the severely cold climate, permafrost, and seismicity. Dzhezkazgan in Kazakhstan is the other major stratabound copper deposit in the Soviet Union.

Stratabound sedimentary copper deposits also occur in the Belt Supergroup in western Montana and adjacent parts of Idaho. Although no major copper mining operations are currently underway in this district, the resource potential of the region is large. The USGS estimates that the area may contain as much as one billion tons of rock with an average grade of 0.5 to 1.0 percent copper, but few concentrations of economically mineable copper have been located to date. Additionally, any attempt to exploit large tonnages in many parts of this region would entail environmental conflicts.

2.2.1.4 Massive Sulfide Deposits in Volcanic Rocks (Figure 2-4)

Copper is more abundant in basalt and andesite than in other igneous rocks. Consequently, under certain conditions, the copper content of these rocks has been concentrated sufficiently to produce an ore grade deposit



IMPORTANT COPPER RESERVES & RESOURCES IN MASSIVE SULFIDE DEPOSITS

Figure 2-4

of a distinctly different form than the porphyries associated with felsic intrusions.

Approximately 12 percent of the world's copper reserves are found in massive sulfide deposits. According to the USGS, these deposits are generally:

- o Stratiform or lenticular, and concordant with the bedding of the surrounding rocks,
- o Of small lateral extent as compared to thickness,
- o Composed largely of sulfide minerals with small proportions of silicate gangue.

Mineralogically, the deposits consist mainly of pyrite and/or pyrrhotite and varying amounts of chalcopyrite, sphalerite, and galena.

Massive sulfide deposits are typically small, but they range in size from several hundred thousand tons to more than 50 million tons of ore. The ore grade of different massive sulfide deposits varies greatly, from 1 to 10 percent copper, with a mean in commercial ores of about 2.5 percent--reflecting the varied values of associated byproduct metals (for example, gold, silver, lead, zinc).

The most important massive copper sulfide deposits are located in Australia, Canada, and Japan, while other, lesser deposits occur in Cyprus (Tamasos), South Africa, and the Soviet Union.

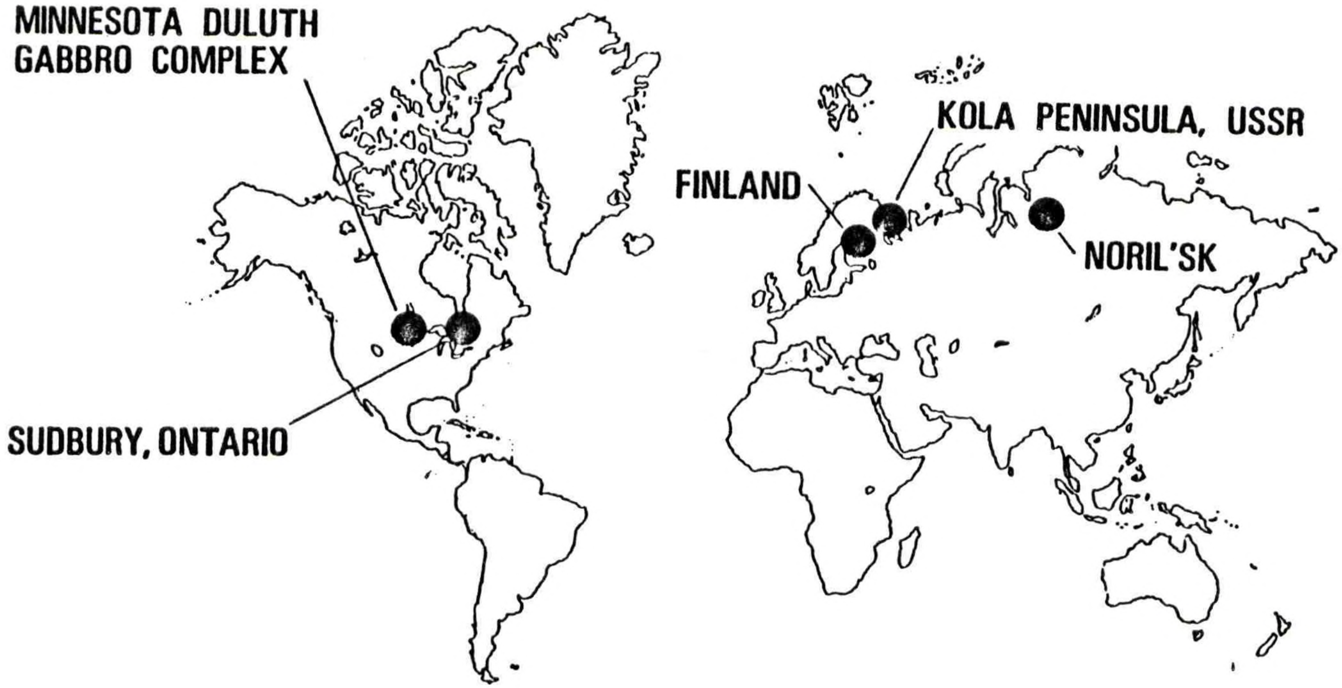
The Kuroko ores of Japan are the classic example of volcanogenic massive sulfide deposits. They are especially common in northeastern Japan, and occupy definite stratigraphic horizons. The deposits are generally flat, lenticular bodies, extremely fine-grained, and range in size from small nodules to irregular masses as much as 2400 feet long, by 1000 feet wide and 300 feet thick. In 1975, Japanese mines produced approximately 90,000 tons of copper, mostly from Kuroko ores.

The Roseberry deposit in western Tasmania (Australia), and the Mount Isa deposit in Queensland are noted massive sulfide copper producers, as are the deposits of Noranda (Quebec), Canada. In the Soviet Union, the many copper pyrite deposits in the Urals provided much of the early supply of copper for industrialization. The region still occupies second place in the Soviet Union for explored reserves and copper production.

The United States has large resources of massive sulfide copper located in the southwest Brooks Range in Alaska. The Ambler District contains at least six identified deposits averaging 10 million tons or more. The most noted deposit, Ruby Creek, may run 30 million tons at 4 percent copper (and lead, zinc, silver), but its development would be hindered by potentially bad water problems and great depth (+1000 feet). More promising are the Arctic, North, Sunshine, and Horace Creek deposits; a recent study indicated that several had economic potential. Perhaps more importantly, all would probably be undevelopable under recent and proposed Wilderness Act regulations and legislation.

2.2.1.5 <u>Copper in Nickel Ore Deposits</u> (Figure 2-5)

Many of the world's nickel deposits in mafic rocks are cupriferous. As such, this byproduct copper is an important contributor to supply, especially in Canada and the U.S.S.R. These deposits are generally interpreted to have been formed by a magmatic segregation process as the mafic intrusions cooled. (For a more complete discussion of the genesis of copper-bearing nickel deposits, please refer to Section 4 of Part 1.)



IMPORTANT COPPER RESERVES & RESOURCES
IN COPPER-NICKEL ORE DEPOSITS

Figure 2-5

The most important producer of copper from this type of deposit is the Sudbury District of Ontario. Other significant producing regions are located in Finland, Norway, Sweden and the U.S.S.R. The nickel-copper-cobalt ores of the Noril'sk deposit in north-central Siberia are an important source of copper for the U.S.S.R.

One potentially important domestic resource is the Duluth Gabbro Complex, on the northern edge of the Arrowhead region of Minnesota. A recent study by the Minnesota Department of Natural Resources evaluated all the presently available data. This assessment tentatively estimates that the Duluth Complex contains approximately 4.4 billion tons at an average grade of 0.66 percent copper (with a 0.50 percent copper cutoff). The potential for converting this large resource into a reserve, however, is still uncertain. The metal mineralization (in addition to copper, nickel, cobalt and titanium are present in recoverable quantities) is apparently disseminated in discontinuous lenses and layers. This would tend to make mining operations more difficult. Other obstacles to rapid development are detailed in Sections 1 and 4 of Part 1. Somewhat offsetting the many potential obstacles to mining the Duluth Complex is its relative proximity to the well developed infrastructure system that currently serves nearby taconite operations. Nonetheless, barring a national emergency and/or major price hikes in some or all of the commodities discussed above, it is unlikely any production will be obtained from the Duluth Complex before 1995.

2.2.2 <u>Part II - Copper Reserves and Resources</u> (Figure 2-6)

2.2.2.1 <u>Introduction</u>

The market supply of copper traditionally has been determined largely by two factors: the price of the non-ferrous metal, and the industry's production capacity (for the sake of simplicity, these are assumed to be all that is necesssary to derive the industry's production function). As demand (hence

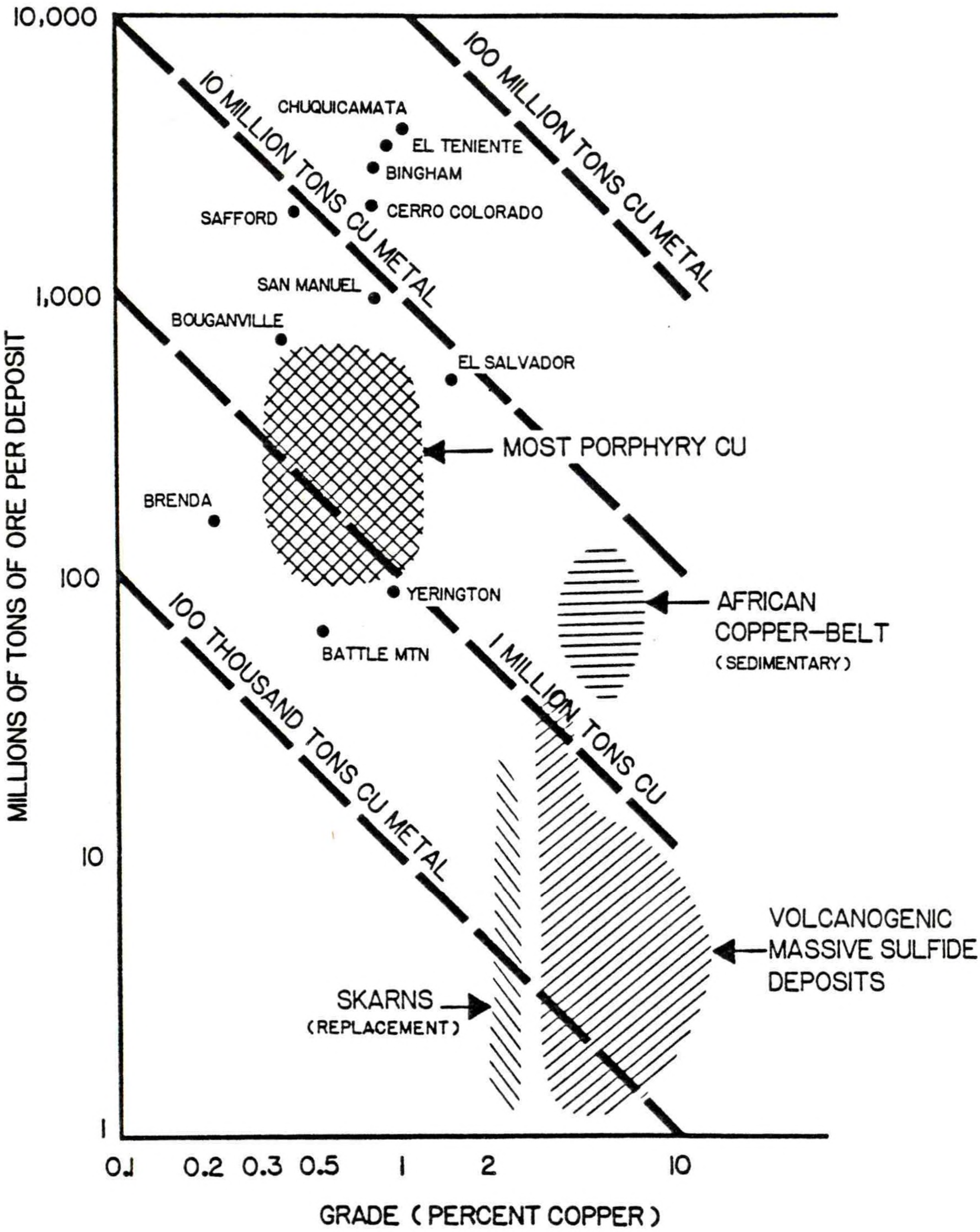

10,000
1,000
100
10
1
MILLIONS OF TONS OF ORE PER DEPOSIT
0.1
0.2
0.3
0.5
1
2
10
GRADE (PERCENT COPPER)
100 MILLION TONS CU METAL
10 MILLION TONS CU METAL
I MILLION TONS CU
100 THOUSAND TONS CU METAL
CHUQUICAMATA
EL TENIENTE
BINGHAM
CERRO COLORADO
SAFFORD
SAN MANUEL
BOUGANVILLE
EL SALVADOR
MOST PORPHYRY CU
BRENDA
YERINGTON
BATTLE MTN
AFRICAN
COPPER-BELT
(SEDIMENTARY)
VOLCANOGENIC
MASSIVE SULFIDE
DEPOSITS
SKARNS
(REPLACEMENT)

Figure 2-6

prices) rose, production as a percentage of capacity grew until maximum output was achieved. This inelasticity of supply was a short-run phenomenon, because as excess capacity and stocks disappeared, prices rose and the industry was induced to meet the rising demand by expanding existing operations and developing new ones. With the addition of new, and at least initially excess capacity, prices would stabilize, or even fall, and the industry's supply curve would become elastic once more. In this manner, prices and production capacity combined to yield a supply growth rate of approximately 3 percent per year over the last sixty years. This also helps to explain the cyclical nature of copper prices, production, and consumption. When prices and demand are high, producers are encouraged to invest in new projects. Typical lead times (discussed later below) tend to exacerbate the problems of oversupply and tight markets.

2.2.2.2 Projected Mine Capacity

Most estimates of mine capacity, including those presented in this chapter, are in terms of annual production of copper metal. Unfortunately, to derive these production estimates some subjective judgment must be involved insofar as the announced capacity of a particular property may tend to overstate or understate the actual capacity. Two factors are of special importance in their effect on metal production capacity: ore grade and recovery.

In many deposits the variability of mineralization allows short-term high-grading, or selective mining of richer ore, at the expense of future production. Selective mining of high- or low-grade ore can cause considerable variation in the effective production capacity of the mine. The ore grade tends to decline gradually in most mines, especially in porphyries with their gradational character. If mill capacity remains fixed, annual metal production tends to decline. This declining trend can be counteracted by increasing mine and mill capacity.

Copper recovery is the second important factor in metal production capacity. Eighty to ninety percent of the metal content is recovered in the traditional floatation process--the remainder is left behind in tailings. Exceptions do exist, however, as the mineralogical composition of ores varies from deposit to deposit. Generally, recovery characteristics are studied in great detail before production begins, but actual recovery quite often turns out to be higher or lower than predicted. One example of the latter unfortunate instance is the Exotica copper mine in Chile. In this case, production was begun only to be curtailed when actual recovery did not match laboratory and pilot plant predictions.

Capacity additions can be accomplished by two means: new projects and expansions of existing mines. In the past, the ratio of new installations to expansions of copper mines has been about 3:2. Currently, planned additions to capacity are listed in Table 2-19 at the end of this chapter.

Assessing the impact of planned new mines is relatively straightforward. Most large scale projects presently require at least five to ten years or more to progress from feasibility study to full production status. Bougainville (Papua New Guinea) and Cuajone (Peru), each large projects in countries favoring development of natural resources, were seven years in development. By simple logic then, any new mine slated to come on-line in the next four or five years must already be in the planning stages. Announcements of most of these new projects are published in several trade magazines, and are tabulated by interested government agencies and firms in the industry. Projections for the next five years are based on these announcements.

Additions to capacity through the expansion of existing facilities are more difficult to assess. Expansions can take place in much shorter periods of time, typically 18 to 36 months. In addition, capital investment is generally much less than for new projects. These two factors frequently allow additions to capacity to occur with little prior notice, and hence, are less predictable. In economic terms, additions to capacity through the expansion of existing facilities are the marketplace's solution to a short-run inelasticity of supply.

In direct contrast to additions to capacity are subtractions, or plant closures. Closures typically occur as the result of one of two events: 1) exhaustion of reserves, or 2) changes in the cost-price relationship of copper produced. The former can be predicted with some degree of certainty, the latter with very little assurance. Looking ahead to the next 15 years or so, it is unlikely that many major mines will have to close down because of ore depletion. Temporary shutdowns of higher marginal cost producers during periods of depressed demand/prices will be more likely. For instance, this occurred in Arizona in 1977 when many high cost producers temporarily shut down or curtailed production, laying off more than 20,000 workers.

Long-term projections are inherently less reliable than those of shorter duration. Therefore, this study details anticipated additions and subtractions to or from existing capacity only through 1985. Beyond the short-run projections, forecasters usually rely on a comparison between projected demand and reserve estimates. However, as was made clear earlier, the very definition of reserves versus resources makes this a difficult proposition without involving the price mechanism. Determining the price elasticity of supply and demand is essential, along with the cross-elasticities of various potential copper substitutes. These determinations were judged to be beyond the scope of this project. Instead, Dames & Moore has undertaken a careful analysis of potential deposits awaiting development, and has compiled a listing of these, by area and grade/tonnage wherever possible. This allows a projection of copper supply through production estimates as a function of demand for mine copper under the assumption that so long as reserves are available, supply will be sufficient to meet market demand over the long run. This is not to say, however, that periods of over- or undersupply will not occur, but simply that on average, supply and demand will be in equilibrium.

2.2.2.3 Reserve and Production Estimates

The estimates shown in this section have been compiled from many sources and are believed to be the most accurate figures currently available. Estimates are given by:

- Country and region, (Table 2-2).

- Deposit type and ore grade, (Table 2-3, 2-4).

- Level of deposit development, (Table 2-4).

2.2.2.3.1 Market Economy Countries

North and Central America. North and Central America possess nearly one-third of the world's copper reserves, and 37 percent of the Market Economies' total. Of this, the United States controls the largest share with 60 percent of the region's proven ore reserves. Canada has 20 percent, while Mexico, Panama, and Guatemala share the remainder.

Porphyry deposits account for the nearly 75 percent of North America copper reserves. They are found predominantly along the western side of the continent, with especially well developed deposits in British Columbia and the southwestern United States. Porphyries account for approximately 75 percent of United States reserves, 40 percent of Canadian reserves, and virtually all other reserves in the region.

Massive sulfides make up the next largest group of deposits in North America with approximately 10 percent of proven reserves. Numerous deposits occur in northeastern Canada (for example, the Noranda deposit, Quebec) and in the Brooks Range of Alaska (for example, the Arctic and Ruby Creek deposits).

Stratabound and copper in nickel ore deposits combine to share reserves equal to that contained in massive sulfides. The largest deposits of the former are found in Michigan (White Pine District) and Montana (Spar Lake deposit), while Sudbury, Ontario holds the greatest quantity of copper-nickel ores. Native copper, as discussed earlier, is known to occur in many areas, but the only substantial quantities in North American were mined on the

Table 2-2

WORLD COPPER RESERVES AND RESOURCES
(in 1000 short tons copper)

REGION	RESERVES	RESOURCES EXCLUSIVE OF RESERVES	TOTAL RESOURCES
NORTH AMERICA			
United States	107,000	320,000	427,000
Canada	35,000	120,000	155,000
Other	35,000	30,000	65,000
Total	177,000	470,000	647,000
SOUTH AMERICA			
Chile	107,000	130,000	327,000
Peru	35,000	40,000	75,000
Other	26,000	70,000	96,000
Total	168,000	240,000	408,000
EUROPE			
Total	7,000	40,000	47,000
AFRICA			
Zaire	26,000	32,000	58,000
Zambia	37,000	70,000	107,000
Other	13,000	20,000	33,000
Total	76,000	122,000	198,000
ASIA			
Total	30,000	70,000	100,000
OCEANIC			
Total	25,000	60,000	85,000
MARKET ECONOMY COUNTRIES			
Total	483,000	1,002,000	1,485,000
CENTRAL ECONOMY COUNTRIS			
Total	66,000	192,000	285,000
WORLD TOTAL*	549,000	1,194,000	1,743,000

* These figures exclude sea nodules

Table 2-3

WORLD COPPER RESERVES BY TYPE OF DEPOSIT
(in 1000 short tons)

REGION	Native Copper	%	Porphyry & Vein and Replacement	%	Stratabound Sedimentary	%	Massive Sulfide	%	Copper in Nickel Ores	%	TOTAL
NORTH AMERICA											
United States	5000	5	81000	75	5000	5	16000	15	-		107,000
Canada	700	2	14000	40	3500	10	6300	18	10500	30	35,000
Other	200	1	34800	99	-		-		-		35,000
Total	5900	3	129800	73	8500	5	22300	13	10500	6	177,000
SOUTH AMERICA											
Chile	-		100000	95	-		7000	5	-		107,000
Peru	-		33250	95	-		1750	5	-		35,000
Other	300	1	23250	90	-		2450	9	-		26,000
Total	300	4	156500	93	-		11200	7	-		168,000
EUROPE											
Total	-		2100	30	700	10	2100	30	2100	30	7,000
AFRICA											
Zaire	-		-		26000	100	-		-		26,000
Zambia	-		-		37000	100	-		-		37,000
Other	-		1000	8	5000	38	7000	54	-		13,000
Total	-		1000	1	68000	90	7000	9	-		76,000
ASIA											
Total	-		25000	83	-		5000	17	-		30,000
OCEANIA											
Total	-		17000	68	4500	18	3500	14	-		25,000
TOTAL Market Economics	6200	1	331400	69	81700	17	51100	11	12600	3	483,000
TOTAL Central Economics	-		13200	20	31000	47	10000	15	11800	18	66,000
WORLD TOTAL	6200	1	334600	63	112700	21	61100	11	24400	4	549,000

Table 2-4

WORLD COPPER RESERVES
LEVEL OF DEVELOPMENT
(in 1000 short tons metal)

	PRODUCING MINES		MINES UNDER CONSTRUCTION		OTHER KNOWN DEPOSITS		TOTAL	
	Copper Content	Ave. Grade	Copper Content	Ave. Grade	Copper Content	Ave. Grade	Copper Content	Ave. Grade
UNITED STATES	80,700	0.71	8,400	1.21	17,900	0.63	107,000	0.71
CANADA	23,700	0.70	700	1.21	10,600	0.48	35,000	0.62
MEXICO								0.67
PANAMA								0.60
OTHERS								0.76
NORTH & CENTRAL AMERICA								0.68
PERU	12,000	1.07	8,800	0.68	14,200	0.88	35,000	0.93
CHILE	78,500	1.11	4,400		24,100	0.84	107,000	1.07
ARGENTINA								0.59
OTHERS								1.10
SOUTH AMERICA								0.99
ZAIRE	19,300	3.90	2,700	5.02	4,000	4.1	26,000	4.05
ZAMBIA	35,900	3.06	1,100	2.22	—	—	37,000	3.02
SOUTH AFRICA	4,800	0.71	1,200	0.8	—	—	6,000	0.73
OTHERS								1.8
AFRICA								2.8
PHILIPPINES	11,800	0.54	4,000	0.44	4,200	0.46	20,000	0.50
PAPUA NEW GUINEA	10,000	0.47	—	—	6,000	0.85	16,000	0.61
AUSTRALIA	7,800	2.58	—	—	1,200	2.69	9,000	2.59
IRAN								1.09
INDIA								1.35
OTHERS								1.08
ASIA								0.75
OTHER MARKET ECONOMIES a)							85,000	--
CENTRAL ECONOMIES b) c)	38,400	1.57	17,100	1.72	10,500	1.5	66,000	1.6
WORLD d)	355,300	1.03	67,200	1.04	126,500	—	549,000	0.96

a) includes Yugoslavia
b) derived from estimate by Stolberg Ingenieurberatung GmbH, W. Germany
c) Bulgaria, Hungary, Poland, Romania, USSR
d) excluding China

Sources:- total reserve/country from World Reserve Table
- ratios from United Nations' Study: Copper, The Next Fifteen Years.

Keweenaw Peninsula in Michigan. Substantial reserves are reported to exist in the region, even though none are currently being worked.

At least 60 percent of all known reserves in North America are located on currently producing properties. This is not surprising, given that producing deposits are almost always more well defined than deposits awaiting development. Another 6 to 8 percent of North American reserves are at mines under construction; thus, over two-thirds of all proven copper reserves in North America are either in properties currently being worked or under development.

As indicated in Table 2-5, North America currently has over 30 percent of the world's total mine production capacity and 38 percent of the Market Economies' mine production capacity. The United States alone accounts for nearly 20 percent of total world capacity. In the next 4 to 5 years, increases in United States mine production will come predominantly from expansions of existing facilities, and reactivations of some properties temporarily shut down due to the last round of low prices. Canadian increases will likewise be largely from existing properties. Elsewhere in North and Central America, increases will probably come from new properties. Prime among these are the La Caridad deposit in Mexico (which commenced production this year) and the Cerro Colorado deposit in Panama (which will probably come on line before the end of the 1980's).

Beyond 1985, specific developments are very difficult to predict. As Table 2-6 indicates, however, an ample number of identified deposits appear to be likely candidates for development. Safford (Arizona) and Pinos Altos (New Mexico) in the United States, Coldstream (BC) and Corbet (Quebec) in Canada, Petaquila in Panama, and Santa Rosa in Mexico will probably be among the first deposits to be developed. Given the political instability in many of the countries known or thought to have economically significant copper deposits, deposits in North America probably stand a better chance of exploitation than those in other parts of the world. Additionally, in light of the

Table 2-5

WORLD COPPER MINE PRODUCTION CAPACITY 1979-1984
(thousands of short tons)

	MINE PRODUCTION						Mine Capacity 12/31/78	Additional Capacity Scheduled For Completion During						Total Planned Additions 1979-1984	Estimated Mine Capacity 12/31/84
	1973	1974	1975	1976	1977	1978		1979	1980	1981	1982	1983	1984		
NORTH AMERICA															
United States	1718	1597	1413	1606	1504	1496	1835	90	33	55	55			233	2068
Canada	908	905	809	806	837	713	945	25	31[e)]	103	10[e)]	65		234	1179
Mexico	89	91	86	98	99	96	125	175			20[e)]			195	320
Other	5	9	4	5	6	5	10							—	10
TOTAL	2720	2602	2312	2515	2446	2310	2915	290	64[e)]	158	85[e)]	65		662	3577
SOUTH AMERICA															
Chile	811	994	913	1108	1164	1141	1150	35	20	11	40	75		181	1331
Peru	223	233	200	242	376	404	425	37			30			67	492
Other	14	13	16	10	6	8	10		40	20			110	170	180
TOTAL	1048	1240	1129	1360	1546	1553	1585	72	60	31	70	75	110	418	2003
AFRICA															
Zambia	779	769	746	781	723	709	700							—	700
Zaire	538	550	545	490	531	467	550			130				130	680
S. Africa	194	197	197	217	226	231	240		6					6	246
Namibia	32	37	32	48	54	42	35							—	35
Other	88	93	88	88	71	62	80							—	80
TOTAL	1631	1646	1608	1624	1605	1511	1605		6	130				136	1741
ASIA															
Philippines	244	249	249	262	301	291	360	+92[e)]	26[e)]	18		43		179	539
Japan	100	90	94	90	90	81	85							—	85
Indonesia	42	71	70	76	63	65	70			20				20	90
Iran	3	4	4	7	7	7	5						160[?]	160	165
Other	91	88	85	96	98	95	110		+45[e)]	+?	60			105	215
TOTAL	480	502	502	531	559	539	630	+92[e)]	+71[e)]	+38	60	43	160[?]	464	1094
OCEANIA															
Australia	243	277	242	241	244	242	255	18	14	6				38	293
Papua New Guinea	201	203	190	195	201	219	215							—	215
Other	0													—	
TOTAL	444	480	432	436	445	461	470	18	14	6				38	508
W. EUROPE (inc. Yugoslavia)	325	311	326	329	329	329	380	30[e)]	45[e)]	58				133	513
TOTAL, MARKET ECONOMIES	6648	6781	6309	6795	6930	6703	7585	502	260	421	215	183	270	1851	9436
CENTRAL ECONOMY COUNTRIES															
U.S.S.R.	1168	1168	1213	1323	N.A.	N.A.			50					50	
Poland	168	204	254	342	N.A.	N.A.					50[?]			50[1]	
Bulgaria	53	55	61	60	N.A.	N.A.			33[e)]					33	
Romania	44	44	50	53	N.A.	N.A.								—	
China & North Korea	154	165	176	179	N.A.	N.A.								—	
Other Central Economy Countries	36	36	32	35	N.A.	N.A.			?					—	
TOTAL	1623	1672	1786	1992	N.A.	N.A.	N.A.		83[e)]		50			133	N.A.
TOTAL WORLD PRODUCTION	8271	8453	8096	8787	N.A.	N.A.	N.A.	502	343	421	265	183	270	1984	N.A.

N.A. = comparable data not available
e) = estimated tonnage
? = exact date in doubt
+ = additional unknown tonnage scheduled to come on line that year
— = not known
1) = United Nations' numbers used here because they include a greater number of countries than do the U.S. Bureau of Mines' figures.

PARTIAL LISTING OF KNOWN DEPOSITS AWAITING DEVELOPMENT
- By Country -

Country/Mine, Location	Estimated Tonnage	Average Grade	Copper Content
NORTH AMERICA			
UNITED STATES			
Copper Basin, Arizona	160×10^6	0.55	880,000
East Helvetia, Arizona	210×10^6	0.56	1,200,000
Florence, Arizona	800×10^6	0.40	3,200,000
Joe Bush, Arizona	90×10^6	0.70	630,000
Red Mountain, Arizona	Unknown	Unknown	--
Safford (Kennecott), Arizona	500×10^6	0.50	2,500,000
Safford (Phelps Dodge), Arizona	400×10^6	0.72	2,800,000
Vekol Hills, Arizona	100×10^6	0.56	560,000
West Casa Grande, Arizona	1×10^9	1.00	10,000,000
Spar Lake, Montana	43×10^6	0.74	320,000
Ann Mason, Nevada	495×10^6	0.40	2,000,000
Bear, Nevada	500×10^6	0.40	2,000,000
Pinos Altos, New Mexico	8×10^6	2.00	160,000
Mazama, Washington	470×10^6	0.37	1,700,000
North Fork, Washington	44×10^6	0.50	220,000
Sampson, Washington	100×10^6	0.50	500,000
Crandon, Wisconsin	70×10^6	NA	--
Ladysmith, Wisconsin	4×10^6	4.80	200,000
Deep Continental North	500×10^6	0.49	2,500,000
Deep Low Grade	820×10^6	0.74	6,100,000
Hall	54×10^6	0.46	250,000
Heddleston	93×10^6	0.48	450,000
Kirwin	70×10^6	0.75	525,000
Lights Creek	350×10^6	0.34	1,200,000
CANADA			
Berg, B.C.	400×10^6	0.40	1,600,000
Coldstream, B.C.	3.2×10^6	4.49	144,000
Davis-Keays, B.C.	1.4×10^6	3.38	47,000
Galaxy/Evening Star, B.C.	4×10^6	0.58	23,000
Galore Creek, B.C.	59×10^6	1.20	710,000
Huckleberry, B.C.	85×10^6	0.40	340,000
J-A Zone, B.C.	286×10^6	0.43	1,200,000
Jericho, B.C.	1×10^6	1.5	15,000
Krain (Keystone), B.C.	20×10^6	0.45	90,000
Maggie, B.C.	200×10^6	0.40	800,000
Ox Lake, B.C.	30×10^6	NA	--
Poison Mountain, B.C.	85×10^6	0.37	310,000
Rainbow (Sugarloaf), B.C.	20×10^6	0.55	110,000
Schaft Creek, B.C.	294×10^6	0.40	1,200,000

PARTIAL LISTING OF KNOWN DEPOSITS AWAITING DEVELOPMENT
- By Country -

Country/Mine, Location	Estimated Tonnage	Average Grade	Copper Content
CANADA (Continued)			
Stikine River, B.C.	138×10^6	1.0	1,400,000
Sustut, B.C.	60×10^6	1.25	750,000
Trojan, B.C.	170×10^3	1.56	2,600
Wheal Tamar (Ajax), B.C.	10×10^6	0.50	50,000
Reed Lake (Freeport), Man.	1×10^6	2.00	20,000
Reed Lake (HBMS), Man.	1.15×10^6	2.18	25,000
Stall Lake, Man.	670×10^3	5.38	36,000
Canoe Landing Lake, N.B.	6×10^6	0.50	30,000
Chester (Clearwater), N.B.	13×10^6	0.77	100,000
Devils Elbow Group, N.B.	400×10^3	1.2	5,000
St. Stephen, N.B.	782×10^3	0.59	4,600
Great Burnt Lake, NFLD	1×10^6	3.1	30,000
Hand Camp, NFLD	2.5×10^6	2.6	65,000
G.W. Group, NWT	100×10^3	8.40	8,400
High Lake, NWT	5.2×10^6	3.53	180,000
Izok Lake, NWT	12×10^6	2.83	340,000
June, NWT	1×10^6	2.50	25,000
Wreck Lake, NWT	4.1×10^6	3.07	125,000
Atikokan, Ontario	14×10^6	0.35	49,000
Ego, Ontario	500×10^3	1.84	9,000
Great Lakes Ni, Ontario	32.8×10^6	0.36	120,000
Shunsby, Ontario	1.25×10^6	1.40	18,000
Spanish River, Ontario	1.1×10^6	1.49	16,000
Abitibi, Quebec	1.22×10^6	0.88	11,000
Brouillan, Quebec "A_1"	35.4×10^6	0.39	140,000
Brouillan, Quebec "A_2"	9.75×10^6	2.26	220,000
Brouillan, Quebec "B"	6.8×10^6	3.22	220,000
Expo Ungava	4.1×10^6	0.96	39,000
Fabie Bay, Quebec	1.16×10^6	2.40	28,000
Magusi River, Quebec	4.1×10^6	1.20	49,000
Midrim, Quebec	428×10^3	0.70	3,000
Miller Copper, Quebec	750×10^3	0.97	7,300
Raglan, Quebec	3.96×10^6	0.98	39,000
	10.2×10^6	0.70	71,000
Soucy No. 1, Quebec	3.5×10^6	1.40	49,000
Zulapa, Quebec	1.68×10^6	0.48	8,100

PARTIAL LISTING OF KNOWN DEPOSITS AWAITING DEVELOPMENT
- By Country -

Country/Mine, Location	Estimated Tonnage	Average Grade	Copper Content
CANADA (Continued)			
Quandt Group, Sas.	900×10^3	2.00	20,000
Minto, Yukon	5.2×10^6	1.78	93,000
Williams Creek, Yukon	20×10^6	1.00	200,000
MEXICO			
El Arco, Baja	630×10^6	0.60	3,800,000
La Verde, Michoacan	81×10^6	0.70	570,000
Santa Rosa/Pilare, Sonora	143×10^6	0.83	1,200,000
Santo Tomas, Sinaloa	150×10^6	0.50	750,000
PANAMA			
Cerro Colorado	1.86×10^9	0.67	12,000,000
Petaquila/Botija	300×10^6	0.60	1,800,000
PUERTO RICO			
Lares/Utuado/Adjuntas	244×10^6	0.73	1,800,000
SOUTH AMERICA			
CHILE			
Cerro Colorado	110×10^6	1.20	1,300,000
Los Andes	Unknown	Unknown	--
Los Pelambris	440×10^6	0.80	3,500,000
Pampa Norte	440×10^6	0.80	3,500,000
Quebrada Blanca	165×10^6	1.00	1,650,000
San Jose d(el Abra)	550×10^6	1.00	5,500,000
PERU			
Antamina	Unknown	Unknown	--
Berenguela	17.6×10^6	1.26	222,000
Chalco/Ferrobamba	33×10^6	2.50	825,000
Michiquillay	460×10^6	0.75	3,500,000
Pashpap	53×10^6	0.86	460,000
Quellaveco	200×10^6	0.94	1,900,000
Santa Rosa	1.17×10^9	0.55	6,400,000
Tintaya	55×10^6	2.00	1,000,000
Toromochu	240×10^6	0.78	1,900,000

PARTIAL LISTING OF KNOWN DEPOSITS AWAITING DEVELOPMENT
- By Country -

Country/Mine, Location	Estimated Tonnage	Average Grade	Copper Content
ARGENTINA			
La Alumbrera, Catamarca	220×10^6	0.50	1,000,000
ECUADOR			
Caucha	55×10^6	0.50	280,000
AFRICA			
RHODESIA			
Avondale	Unknown	Unknown	--
SOUTH AFRICA			
Black Mountain, Aggeneys	94×10^6	0.72	680,000
ZAIRE			
Tenke Fungurume	56×10^6	5.7	3,200,000
ZAMBIA			
Lumwana	220×10^6	1.0	2,200,000
Kansashi	10×10^6	3.0	300,000
OCEANIA			
AUSTRALIA			
Andamooka, S.A.	Unknown	Unknown	--
Cadia, N.S.W.	Unknown	Unknown	--
Golden Grove, W.A.	11×10^6	3.5	390,000
PAPUA NEW GUINEA			
Ok Tedi	250×10^6	0.85	2,100,000
Freida River	366×10^6	0.45	1,600,000
ASIA			
PHILIPPINES			
Luzon	NA	NA	--
Mapula	60×10^6	NA	--
Suso-Damutan	NA	NA	--
Tawi-Tawi	165×10^6	0.39	640,000
Toledo (Tailings)	200×10^6	0.57	1,100,000

PARTIAL LISTING OF KNOWN DEPOSITS AWAITING DEVELOPMENT
- By Country -

Country/Mine, Location	Estimated Tonnage	Average Grade	Copper Content
INDIA			
Ambaji	NA	NA	--
JORDAN			
Wadi Araba	55×10^6	1.36	750,000
PAKISTAN			
Saindak - North	19×10^6	0.49	93,000
- South	54×10^6	0.48	260,000
- East	264×10^6	0.38	1,000,000
CENTRAL ECONOMY COUNTRIES			
U.S.S.R.			
Udokan, Siberia	1.2×10^9	2.0	24,000,000
CHINA			
Qamdo (Chamdu), Tibet	7.1×10^6	NA	--
Dexing (Tehsing), Kiangsi	8.8×10^6	NA	--

current tight risk-capital situation, a shift away from development of large-scale open-pit mining may be developing. If so, despite their large tonnages and low mining costs, porphyries may, at least temporarily, lose some of their attractiveness. Smaller, higher grade, massive sulfide and stratabound deposits may become of greater interest to the copper mining industry. This would seem to be borne out with the anticipated development of Pinos Altos in the United States, and both of the Canadian deposits mentioned. Early development of the Crandon and Ladysmith deposits in Wisconsin would also be indicative of this.

Further down the line, however, if the United States wishes to retain its share of the market and meet rising demand, mining interest will have to return to the porphyries. The simple fact that porphyry deposits contain such a high percentage of proven reserves, coupled with the typical high annual rates of relatively low cost production from large open pit mines, virtually assures porphyry deposits a major share of future production capacity.

During the next 30 years, it is unlikely the United States will resort to any unconventional sources of copper. The only possible exception to this might be the Duluth Gabbro Complex in Minnesota. Production from this deposit may begin in the 1990's, but, if so, it will not be necessarily solely because of the need for copper. Other copper reserves exist which are more accessible, and which have greater economic potential.

South America. South America closely trails North America in reserves, with 30 percent of the world's total, and 34 percent of the Market Economy Countries' total. Chile's reserves account for more than 65 percent of the region, and Peru's slightly over 20 percent. Argentina, Brazil, Columbia and Ecuador account for much of the remainder.

Porphyry and associated vein and replacement-type deposits account for nearly all of South America's copper reserves. The largest porphyry province is found in the Andes mountains of Chile and Peru. The remaining copper reserves occur mostly in mixed ores, particularly massive sulfides. In both Chile and Peru, porphyries and vein and replacement deposits account for over 95 percent of proven reserves. Elsewhere in South America, porphyries account for at least 90 percent of copper reserves.

Approximately 60 percent of South American reserves are in deposits currently being mined, and an additional 10 to 15 percent are in deposits currently undergoing development. A very important consideration when interpreting South American reserve figures is the high grade of most of their reserves. Given the vast tonnages of these high-grade reserves (average grade is at least 1.0 percent copper), it would not be at all surprising to suspect official figures of being low. Little economic incentive exists to mine or even drill out lower-grade deposits. This is at least one reason why Chilean resource estimates are so much lower than those of the United States, which has comparable reserve estimates. (The United States resource to reserve ratio is 4:1, versus 2.2:1 for Chile and 2.1:1 for Peru.)

South America accounts for 17 percent of the entire world's mine production capacity and 21 percent of the Market Economy Countries' (Table 2-5). Chile, by virtue of its long history of copper exploitation, has the largest share of productive capacity (72 percent). In the next 4 to 5 years, additions to capacity will be most likely derived evenly from expansion and re-openings of existing facilities, and the development of new deposits. Peruvian mining activity will probably be similar to Chile's during this period. Production increases from Brazil, however, will come largely from new projects, especially the Caraiba and Camague/Pedra Verde deposits.

In the years beyond 1984, the prospects for large production from new and existing developments in South America are quite good. In the late 1980's and early 1990's, likely new projects include the development of the El Abra and Quebrada Blanca deposits in Chile, and the Toromochu, Quellaveco, Michiquillay and Chalcobamba deposits in Peru. Also, it is probable that expansions will occur at all the major mines in the region.

Given an orderly market development, South America will become, in all likelihood, the largest copper producing region in the world during the time span covered by this report. Current South American reserve and resource figures almost certainly understate the region's true potential.

Africa. Copper reserves in Africa constitute only 14 percent of the world's total (16 percent of the Market Economies'). This seemingly low figure is probably due to two causes: 1) underexploration, and 2) the apparent absence of any large porphyry copper provinces. Zambia (formerly Northern Rhodesia) controls slightly less than one-half of proven African copper reserves. Zaire (formerly the Congo-Kinshasa), has roughly one-third, while a small group of countries including Angola, Namibia, Rhodesia, the Republic of South Africa, and Uganda possess most of the remainder.

Stratabound copper deposits make up the largest and best-developed portions of African reserves with virtually all of the reserves in the two major producing nations in this category. For descriptions of these deposits in Zambia and Zaire (beyond what was given in Section 2.2.1.3) see Section 1 of Part I.

The Republic of South Africa is Africa's only other major copper producer. The largest deposit is Palabora, with approximately 320 million tons of ore grading 0.68 percent copper. A somewhat uniquely mineralized deposit, it has been classifed as a massive sulfide for the purpose of this study.

Close to 90 percent of Africa's copper reserves are associated with deposits already in production and another 5 percent are located in deposits under construction.

Africa's current production capacity is approximately 17 percent of the world's total. As with reserves, Zaire, Zambia, and the Republic of South Africa are the major producers with 95 percent of the region's production capacity. This situation is unlikely to change in the next 4 to 5 years, or, for that matter, in the next 2 to 3 decades. What is likely, however, is that Zaire will overtake Zambia (in terms of production) by the end of the 1980's. This will be due to existence of apparently larger, richer orebodies in Zaire. Ore reserve estimates for Zaire probably understate the nation's true potential.

In the next 4 to 5 years, the only announced African copper developments are a major expansion to production in Zaire's Shaba region, and the opening of the first new mine near Aggeneys, South Africa. The Shaba expansion (1981) includes smelter and refinery expansions. The development of the Broken Hill (Aggeneys) orebody in South Africa foreshadows the development of a potentially large producing region. Three other large orebodies (containing copper, lead, zinc, silver) have been outlined near Broken Hill; Black Mountain, Big Syncline, and Gamsberg. The potential for sharing infrastructure development makes this area quite promising for development in the next 10 to 20 years.

One of the richest, large copper orebodies in the world exists at Tenke Fungurume in Zaire and will probably be developed by 1990. Tenke Fungurume's reserves are currently estimated at 56 million tons of ore grading 5.7 percent copper. Development has been temporarily deferred due to anticipated high capital investment costs.

Elsewhere in Africa, a number of deposits have been identified for development, although the date when each may be brought into production is uncertain. These include the Avondale deposit in Rhodesia, and the Lumwanna and Kansashi deposits in Zambia. Other additions to African copper production capacity will likely come from expansions of existing mines.

Asia. Asian copper reserves in Market Economy Countries amount to only about 5 percent of world reserves. Some two-thirds of Asian reserves are located in the Philippines. Reserves are also held by India, Iran, Japan, Oman, and Pakistan.

The vast majority of the reserves of Iran, Pakistan and the Philippines occur as porphyry-type deposits. Deposits in the first two countries are in the so-called Alpide Porphyry Belt, the latter, in the Southwest Pacific Belt (see Figure 2-3). Other reserves of interest are the massive sulfide Kuroko ores of Japan. Roughly one-half of Asian copper reserves are located in existing mines, while the other half is split roughly 20:30 between deposits undergoing development and those awaiting development.

Copper mines in the Philippines account for 60 to 70 percent of Asia's copper production, and have the potential to contribute an even larger share. In the next 4 to 5 years, the Philippines will probably increase their mine production capacity by over 50 percent as more island-arc type porphyries are developed.

In Indonesia, the Ertsberg-East extension is expected to come on-line in 1981 or 1982 and will help reduce the impact of the main orebody's anticipated exhaustion in the mid-to-late 1980's. In Pakistan, three medium-size, low-grade porphyries near Saindak may be developed by the end of the decade. With regard to the giant Sar Cheshmeh deposit in Iran, it is not known when production can be expected, if ever. In early 1979, development work on Sar Cheshmeh stopped after the revolution in Iran, at which time the primary American contractor reported that the mine and support facilities were 97 percent complete. Given the recent turmoil in Iran, it is unlikely that full production capacity (160,000 tons of copper per year) will be achieved prior to 1985. Considering how close the project was to completion, however, limited production conceivably could begin as early as 1981 or early 1982.

In the longer run, the best potential for additional production in Asia lies in the Philippines. However, for expansions to Philippine production capacity to occur, potential problems must be overcome, such as the high capital investment costs necessary to exploit these remote, lower-grade deposits.

Oceania. Copper reserves in Oceania amount to approximately 5 percent of the World's total. For the most part, the majority of the copper reserves occur in Australia and Papua New Guinea (PNG). Nearly 65 percent of Oceania's copper reserves are found in the island-arc porphyries of PNG, while the remaining 35 percent located in Australia is split between porphyries on the eastern coast, massive sulfides in Tasmania, and scattered deposits of sedimentary origin. Some 70 percent of reserves are located in currently producing mines.

Australian copper mines produced slightly over 52 percent of Oceania's copper output in 1978, and in the next 2 to 3 years should increase that share to almost 55 percent with the development of several new base metal mines (for example, the Woodlawn, Gunpowder, and Teutonic Bore deposits).

Although Papua New Guinea has a larger reserve base than that of Australia, the country is just beginning to develop them. As a result, increases in production capacity will have to come from new mines instead of expansions to existing mines. Forty percent or more of PNG's copper reserves are in undeveloped deposits. Their potential impact probably will not be felt earlier than the late 1980's. The most promising orebodies currently under study are the Ok Tedi (250 million tons at 0.85 percent copper) and Freida River (366 million tons at 0.45 percent copper) deposits. Many other large porphyries are known to exist (e.g., Yandera--338 million tons, 0.42 percent copper) and await eventual development.

Europe. Copper reserves in Europe are small, constituting 1.2 percent of the world's total, and less than 1.5 percent of the Market Economies' total. Significant recoverable quantities of copper occur in Finland, Norway, Spain, and Sweden. Not surprisingly, these nations are also the major European producers of the metal; in total, their production equals just 2 percent of the world copper production. Many European copper deposits, such as those of the Rio Tinto District in Spain, have been worked for centuries. Over the next two years, increased production will come from the Aipsa, Aznacollar, and Minerva deposits in Spain, and from expansions in production at the Aitik deposit in Sweden. Further increases in future copper production probably will come from lower-grade, deeper deposits, and entail relatively higher production costs.

2.2.2.3.2 Central Economy Countries (CEC)

An accurate analysis of the copper supply situation in the Central Economy Countries of the world is difficult to provide. What little information is available, is often dated or conflicting.

Since 1977, the USBM has maintained a reserve estimate of 65 to 66 million tons of copper for this group of nations. Soviet copper reserves are estimated to be 40 million tons, or 60 percent of the reserves of the CEC total. Poland has the second largest reserves, containing approximately 14 million tons of copper. The remaining 17 percent of CEC reserves, or 12 million tons, occur in Bulgaria, China, East Germany, North Korea, Romania, and Yugoslavia.

A slightly higher estimate of CEC copper reserves has been determined by the United Nations and presented in a recent report entitled Copper: The Next 15 Years (1979). According to this study, just under 70 million tons of copper exist in the Central Economy Countries, excluding Yugoslavia and China. If USBM estimates for these two countries are added to the United Nations estimate, up to 80 million tons of copper could be held by CEC.

Soviet copper reserves are evenly distributed in four of the five deposit types discussed at the beginning of this chapter.

Stratabound sedimentary deposits make up the largest single portion of reserves, constituting 31.8 percent of the Soviet total. Termed "low-grade cupriferous sandstones" in much of the Russian literature, the largest single stratabound sedimentary deposit is the Udokan orebody in Eastern Siberia. Although not yet exploited, the reserves of the Udokan deposit are estimated to be 800 million tons of ore grading 1.15 percent copper. Development of the orebody is planned for the coming decade, and is relatively certain, given the level of investment already made in the BAM railroad. Currently, this category of deposit accounts for some 15.3 percent of U.S.S.R. production, the largest share of which is concentrated in Dzhezkazgan, SSR.

Copper-nickel deposits make up the second largest portion of the U.S.S.R.'s reserves (29.2 percent) and are the third largest contributor in terms of production (18.7 percent). The two largest copper-nickel districts are Noril'sk and the Kola Peninsula, both located north of the Arctic Circle. Further details about these deposits are given in the chapters NICKEL and COBALT.

Massive copper sulfide deposits are only the third largest segment of U.S.S.R. copper reserves (21.5 percent), yet are the leader in production with 32.2 percent of the Soviet total. Copper bearing pyrite deposits in the Ural Mountains form the greatest share of this deposit category, but recent indications are that this sector is decreasing in importance as many older deposits are being depleted.

Porphyry copper and vein and replacement-type deposits account for only 17.7 percent of reserves, and 23.8 percent of production, the majority of which are located in Kazakhstan and Uzbekistan. Noted deposits occur at Kounrad (in the former), and Kalmakyr and Almalyk (in the latter). The Almalyk complex is one of the largest copper producing centers in the U.S.S.R.

The ore reserves and copper mines of Poland and East Germany are primarily in the Kupferschiefer Shale formation discussed in Part I of this chapter. Elsewhere in the CEC, porphyry deposits make up a larger percentage of the reserves than they do in the U.S.S.R.

In the new decade, the only major new additions to announced CEC production capacity are the Bor Velika Kriuaja deposit in Yugoslavia (33,000 tons of copper per year) scheduled for 1981, the Sieroszowice deposit in Poland (50,000 tons of copper per year--1982?), and the Elatsite deposit in Bulgaria (estimated 33,000 tons of copper per year in 1980). The largest share of expanded Soviet production will probably come from the Udokan and Noril'sk regions well into the 1990's.

One of the biggest uncertainties in the future, with regard to CEC copper production is the potential for development of Chinese copper deposits. Current Chinese copper production is estimated to be only 165,000 tons per year and, on this basis, the USBM attributes only 2 million tons of reserves to China. Generally promising geology, combined with recently announced discoveries (for example, the Qamado deposit in Tibet--which is reported to contain 7.1 million tons of ore), and the involvement of a United States engineering firm in the design of a 90,000-ton per year copper mine in Tehsing, Kiangsi Province, lend credence to the belief that Chinese reserves are almost certainly understated.

2.2.2.4 Secondary Copper

The supply of primary copper is augmented by other sources -- specifically secondary copper derived from new and old scrap. New scrap is defined as a byproduct of copper fabricating operations; old scrap accrues from discarded copper-containing goods and equipment. Both new and old scrap occur as different grades and are of varying compositions ranging from slags containing as little as 10 percent copper, to new copper wire of greater than 99.9 percent purity.

Because copper scrap is retrieved through an assortment of channels data compilation is difficult. Examples include home or runaround scrap which originates and is reused in manufacturing plants, prompt industrial scrap which is generated similarly but then sold to merchants, and customer return scrap which is returned to the refinery for re-casting, often for use by the original user. Local scrap markets also exist for such items as old auto radiators and used copper pipe.

New scrap is generally not treated as a net addition to supply, and the amount which is available is largely determined by the level of current consumption as well as the price differential between scrap and unrefined copper.

Old scrap is placed into categories based on the composition of the scrap. Most old scrap can be classified into one of four categories: 1) No. 1 Wire and Heavy Copper--this copper is 99.9 percent pure and is interchangeable with refined copper in many uses; 2) No. 2 Wire, mixed heavy and light wire; 3) Yellow Brass; and 4) Low-Grade scrap.

Given today's increased emphasis on ecological awareness, and projected higher prices for primary metals, most experts anticipate increased emphasis on incentives for recycling which should increase the proportion of copper supply furnished by old scrap. In 1977, the USBM calculated the then current portion of scrap to be approximately 22 percent, and predicted a 27 percent share in 1985, and 31 percent in the year 2000. Although this forecast may appear conservative, several factors indicate it may actually be over-optimistic. First, a very high percentage of new copper scrap is already being recovered, and the scope for improvement in this sector is limited. Second, a great deal of the easily recovered old scrap (for example, radiators, electrical transmission lines) is also currently retrieved. Studies in certain developed market economy countries indicate that the vast majority of unrecovered copper scrap arises in household waste found in low-grade alloys.

Therefore, it seems reasonable to conclude that the potential for improving recovery of secondary copper is likely to be of less importance to the supply picture than was earlier predicted.

2.2.2.5 Summary - Copper Supply

Copper occurs in three major groups of minerals: sulfides, carbonates, and silicates, with sulfides being the most important, silicates the least. Viable deposits are classified in five geologic categories, the most significant being porphyry and vein and replacement deposits, and stratabound sedimentary deposits.

Copper supply is viewed through an evaluation of reserves and resources. Reserve estimates are given by deposit type and country/region, and by level of deposit development (Tables 2-7 and 2-8).

Table 2-7

GEOLOGIC DISTRIBUTION OF WORLD COPPER RESERVES AND PRODUCTION

TYPE OF DEPOSIT	Percent of World Reserves	Percent of Current Production
Porphyry and Vein/Replacement	63%	46%
Stratabound Sedimentary	20%	>26%
Massive Sulfide	11%	15%
Copper in Nickel Ore	5%	10%
Native Copper	1%	2%

Table 2-8

WORLD COPPER RESERVES BY TYPE OF DEPOSIT
(in 1000 short tons)

COUNTRY	Reserves	Resources (inc. reserves)	Native Copper	%	Porphyry Copper & Vein and Replacement	%	Strata-bound Copper	%	Massive Sulfides	%	Copper in Nickel Ore	%
UNITED STATES	107,000	427,000	5,000	5	81,000	75	5,000	5	16,000	15		
CHILE	107,000	237,000			100,000	95			7,000	5		
CANADA	35,000	155,000	700	2	14,000	40	3,500	10	6,300	18	10,500	30
ZAMBIA	37,000	107,000					37,000	100				
ZAIRE	26,000	58,000					26,000	100				
PERU	35,000	75,000			33,250	95			1,750	5		
PHILIPPINES	20,000	40			18,000	90			2,000	10		
AUSTRALIA	9,000	15			1,000	11	4,500	50	3,500	39		
SOUTH AFRICA	6,000	15							6,000	100		
PAPUA NEW GUINEA	16,000	32			16,000	100						
OTHER MARKET ECONOMIES	85,000	322	500	.6	68,150	80	5,700	6.7	8,550	10	2,100	2.5
POLAND	14,000						14,000	100				
USSR	40,000 e)	258,000			7,200	18	12,800	32	8,800	22	11,200	28
OTHER CENTRAL ECONOMY COUNTRIES	12,000 e)				6,000	50	4,200	35	1,200	10	600	5
SEA NODULES		800,000										
WORLD TOTAL	549,000	2,541,000	6,200	1.04	344,600	62.8	112,700	20.5	61,000	11.1	24,400	4.4

Mine production capacity is detailed (by country) for the next five years; beyond that, likely developments are postulated, and deposits awaiting development tabulated. In the near future, additions to mine capacity will come largely from expansions to existing mines, and (possibly) newly-developed massive sulfide and stratabound sedimentary deposits, due to their generally lower capital investment requirements. In the long-run, however, major production increases must come out of porphyry deposits, where the bulk of reserves are located. Few major deposits are expected to reach ore exhaustion during the period covered.

2.3 COPPER DEMAND

2.3.1 Introduction

Copper is a moderately soft, non-magnetic, malleable and highly ductile metal. It has been used by man for at least 6,000 years and has been one of the most important materials in his industrial advancement. Copper's uses are manifold and wide-ranging, both in its alloyed and unalloyed states, yet in many applications, copper has been losing ground to alternative materials.

2.3.2 Uses

Statistical information on the end-use pattern of copper consumption is unfortunately limited to the major consumers among the developed Market Economy Countries. Most of the data available are even then comparable over time and by region only at a broad level of aggregation.

As discussed earlier in this report, the demand for copper tends to move in synchronization with the level of industrial production. The cause of this strong correlation can be found through examination of the major end uses of copper (Table 2-9). Not surprisingly, the major consumers of copper encompass the major sectors of industrial activity.

Table 2-9

1975 COPPER CONSUMPTION
(percent)

USE	UNITED STATES	WESTERN EUROPE	JAPAN
Electrical	52.0	54.3	52.0
Building Construction	17.0	15.5	8.8
Industrial Equipment	14.0	14.0	15.0
Transporation	10.0	10.7	17.1
Other	7.0	5.5	7.1
TOTAL	100.0	100.0	100.0

In examining the end-use patterns of copper consumption, it is also useful to distinguish between consumption by semi-manufacture, and between copper and copper-alloy products (Table 2-10). The principal alloys of copper are brass (copper plus zinc in the ratio of about 65:35) and bronze (copper plus variable amounts of tin). Both can be cast into complex and precise shapes. Brass is also noted for its resistance to corrosion. The pattern of consumption by type of semi-manufacture has shown little change in the last decade. Wire is the predominant product in industrial nations, (particularly in the copper rather than the alloy form) and the product for which demand has grown fastest. Reasons for this growth will become apparent from the following review of end-uses.

2.3.2.1 Electrical and Electronic Parts

The electrical conductivity of copper per unit volume is higher than that of any metal other than silver (for example, twice that of aluminum, ten times that of iron). In addition, nearly pure copper can be drawn into very fine wires, and can be easily joined (soldered). These attributes have made copper the metal of choice in most electrical applications, for example, as a primary constituent in generators, motors, power distribution lines, industrial controls, communications equipment, transformers and household wiring.

2.3.2.2 Building Construction

The durability of copper pipe (tubing) combined with the ease with which it can be fabricated has until recently made copper the pre-eminant choice of builders for plumbing installations. The primary copper alloys, brass and bronze, also see widespread use for decorative and protective purposes.

2.3.2.3 Industrial Equipment

Major demand for copper comes from the increase in exploration for natural resources, especially in the mining and oil industry, where drilling equipment utilizes copper alloys, for example, brass, in the manufacture of valves and pipefittings.

Table 2-10

APPARENT COPPER CONSUMPTION BY TYPE OF SEMI-MANUFACTURE, SELECTED COUNTRIES, ANNUAL AVERAGE 1973-75

Countries	Distribution by type (thousand tons)									
	Wire	(%)	Rods, bars & sections	(%)	Sheets, strips & plates	(%)	Tubes	(%)	Total consumption	(%)
Copper										
France	270.4	75.8	12.0	3.4	17.5	4.9	56.7	15.9	356.6	100
F.R. of Germany	389.0	74.8	26.2	5.0	40.8	7.9	62.1	11.9	518.1	100
Japan	748.7	82.1	20.3	2.2	67.5	7.4	75.2	8.3	911.6	100
U.S.A.	1,169.4	70.7	57.0	3.4	126.8	7.7	299.8	18.2	1,653.1	100
Copper Alloy										
France	10.6	5.6	114.5	60.3	52.5	27.7	12.3	6.4	189.8	100
F.R. of Germany	28.2	9.0	133.4	42.6	78.0	24.9	50.4	16.1	290.0	100
Japan	30.8	7.3	193.8	46.2	157.3	37.6	37.5	8.9	419.4	100
U.S.A	(0.7)		356.8	44.0	378.1	46.7	76.1	9.4	810.3	100

Source: World Bureau of Metal Statistics, World Metal Statistics, November 1976, in Copper: The Next Fifteen Years, U.N, Study by Gluschlie, W., Shaw, J.R., and Varon, B.

2.3.2.4 Transportation

Automobiles, trucks, locomotives, ships and airplanes all require fairly large quantities of copper in the form of wiring and electric motors. Automobile radiators are a large consumer of copper, and copper-nickel cladding in ship hulls is becoming popular.

2.3.2.5 Other

Ordnance, coinage, and consumer products make up the bulk of the remaining demand for copper. Ordnance uses are primarily as brass in shell casings, and in heavy gun mountings where corrosion resistance is required. Consumer products include uses in jewelry, appliances and other household items. Pigment and chemical uses are relatively small consumers of copper.

2.3.3 Substitution

In a purely economic sense, substitution occurs when the price of one commodity changes relative to another product which can be similarly used. This interchangeability is measured by the cross-elasticity of demand (i.e., $\frac{dQ_x}{dP_y} \cdot \frac{P_y}{Q_x}$). In the case of copper, potential substitutes include; aluminum, stainless steel, titanium and plastics. Although a detailed analysis of the various cross-elasticities of demand between these commodities and copper is beyond the scope of this study, a brief look at the major end-use sectors is enlightening.

2.3.3.1 Electrical and Electronic Parts

Large-scale substitution has already occurred in power transmission, and will probably occur in telecommunications. Because of the weight reduction achieved by switching from copper to aluminum wire, aluminum has been able to make major inroads in the overhead transmission lines market. Today, most overhead transmission lines in the developed countries are aluminum. For underground cables, aluminum has comparatively fewer advantages but still is less expensive than copper in low voltage applications. (Use of aluminum in high voltage applications is usually too costly because aluminum requires more cables in parallel to carry the same load as copper.) The use of superconductors for United States electric power transmission will probably begin in the next 25 to 30 years. Copper, not being a true superconductor, will have only limited use and will be displaced by other materials, possibly including cadmium, lead, niobium and sodium.

In the telecommunications field, the trend will probably be for a decline in usage of copper as microwave transmission facilities increase in number and as a number of new developments become commercial. Among the two most likely copper substitutes are optic (glass) fibers and laser-produced, electronic impulse communications devices. Both allow the transmission of a greater volume of messages per cross-sectional area. Optic fibers are coming into commerical use now, and should become common by the end of the decade in new installations.

2.3.3.2 Building Construction

The major threats of substitution for copper in the building construction sector occur in two areas: wiring and plumbing. For the past decade or longer, aluminum has been increasingly used as a substitute for copper in household wiring. Nonetheless, several detrimental qualities have inhibited widespread use. Prime among these is the tendency for aluminum to "cold flow" at connection points. This leads to oxidation and an increase in resistance, which often creates a "hotspot" and potential fire hazard.

Plastics, especially polyvinyl chloride (PVC), have made strong inroads on copper's plumbing market. PVC's are less expensive, lighter and easier to work with than copper. However, after initial substitution for copper tubing in both hot and cold water applications, it has generally been concluded that plastics should be restricted to cold water use.

The recent trend away from single family dwellings may also have some impact on copper demand.

2.3.3.3 Industrial Equipment

Depending on prices and applications, substitution of other materials for copper in industrial equipment and machinery may take place in isolated instances. Possible losses for the copper market include valves and bearings using stainless steel in place of brass, and aluminum in large heat exchangers (e.g., refrigeration plants, power stations). Unless osmotic methods are perfected, copper consumption will probably increase in the relatively new field of salt water desalinazation as more plants are built.

2.3.3.4 Transportation

While direct substitution for copper in automobile wiring is unlikely, the possibility exists that should copper prices continue to increase, there will be a tendency for manufacturers to substitute aluminum for copper in electric motor and generator windings. Because of their ease of repair, copper-based radiators are still preferred in the United States over those made from aluminum. However, in view of the increasing emphasis on weight reduction in automobiles a switch from copper to aluminum may occur. Of potential significance, and largely offsetting any losses of copper due to substitution, the advent of electric cars very likely would be a boon to copper consumption.

2.3.3.5 Other

Brass in shell casings is vulnerable to substitution by various steel alloys. Household electrical products may incorporate aluminum in small motor windings, but any impact should be minimal. A much larger factor is the continued growth of consumer demand in developed nations, and the huge potential for growth in the developing countries. Another possible area for increased copper use is the growing solar power industry.

2.3.4 Summary - Uses and Substitution

As indicated in the foregoing discussion, in a number of applications, alternative materials exist for copper. Aluminum can be used in many low voltage, and some high voltage applications. Aluminum is also seeing increased utilization in the field of heat transfer (radiators). Plastics (especially PVC) have broad appeal in building construction for cold water plumbing. New products and technology, such as optic fibers and cryogenic metallurgy show strong signs of creating new replacement markets immune to competition from copper.

Yet copper remains a superior product to these alternatives in many respects. Therefore, the primary determinant in questions of substitution will remain the relative prices of the materials involved. Two additional relevant considerations are price volatility and energy costs. Copper, unfortunately, has a long history of wide swings in price. Although the average price of copper over a given period of time may be equal to that of a competing material, if copper's price is the more volatile, the consumer may well be swayed toward the commodity with the more stable price.

In the area of energy costs, copper comes out strongly ahead of aluminum. Energy costs account for only 10 to 15 percent of copper production costs, versus around 30 percent for aluminum. Therefore, continued sharp escalations in the price of energy should give copper a competitive edge on aluminum.

In conclusion, copper's uses <u>are</u> manifold and wide-ranging, and will continue to be so in the foreseeable future. Substitution will occur in a number of areas, but will probably occur gradually over time. Copper stands to lose some ground to alternative materials, while gaining ground elsewhere. The probable path of copper consumption is forecast below.

2.3.5 Projected Copper Demand

2.3.5.1 Introduction

To establish the demand scenarios used in this chapter, demand forecasts from a number of sources were collected and reviewed, including the following:

- United States Bureau of Mines

- United Nations Centre for Natural Resources

- World Bank

- CRU

Industry trade journals, academicians and several other publicly available sources were also consulted. As a result of this review, USBM projections have been accepted for the purpose of this report.

2.3.5.2 Past Consumption

Knowledge of historical patterns and trends in metal demand is helpful in projecting future consumption. Table 2-11 below presents historical growth rates of refined copper consumption for the focal areas of this study during the period 1951-1955 to 1971-1975:

Table 2-11

HISTORICAL DEMAND GROWTH RATES FOR REFINED COPPER

Region	Growth Rate
United States	1.89
Market Economies	3.83
Central Economies	6.56
World	4.37

Source: Metallgesellschaft Aktiengesellschaft-Metal Statistics (annual), 1952-1976 in Malenbaum, 1978.

A more complete listing is presented in Table 2-14 at the end of this section.

To correctly interpret these figures, actual consumption data must be examined. Consumption in the focal groups is shown below in Table 2-12, while a more complete tabulation is given in Table 2-15.

Table 2-12

WORLD CONSUMPTION OF REFINED COPPER - 1976

(1000 short tons copper)

United States	1,965
Market Economies	7,092
Central Economies	2,306
World	9,398

Source: from Table 2-15

2.3.5.3 Future Consumption Projections

Table 2-13 gives projections of copper demand for the United States, Market Economy Countries, Central Economy Countries, and world over the period 1980 to 2010. Three principal sources of data were utilized:

United States and World:	United States Bureau of Mines, Mineral Commodity Profile: Copper, June 1977.
	United States Bureau of Mines, Mineral Trends and Forecasts, 1979.
Central Economy Countries and Market Economy Countries:	Malenbaum, Wilfred. World Demand for Raw Materials in 1985 and 2000, 1978.

Table 2-13 presents the three (Low, Most Likely, High) forecast demand scenarios for the United States and World, and the most probable level of demand by the Market and Central Economy Countries for the period 1980-2010. It should be emphasized that the growth rate values given are predicted annual averages and, therefore, in any given year, the actual growth rate may vary from the overall projection considerably.

2.3.5.4 Supply/Demand Intersections

Table 2-13 also notes reserves and resources in each grouping of nations. Given the difficulty in accurately forecasting copper mine production capacity (discussed in Section 2.2.2.2 of Part I), supply/ demand intersections have been calculated on a static, rather than a dynamic basis. Supply is viewed as a finite quantity of copper contained only in known reserves. Using this very restrictive assumption, world copper reserves should last 28 to 37 years at projected rates of growth for primary demand. (See Figures 2-7 and 2-8 for a

Table 2-13

COPPER SUPPLY-DEMAND POSITION 1980-2010

(in 1000 short tons)

UNITED STATES:

Reserves: 107,000 — Total Resources: 427,000

	Ave. Ann. Grth Rate	1980	1985	1990	1995	2000	2005	2010	Cumulative
Projected Demand:L	1.91	1849	2033	2234	2456	2700	2968	3263	75,392
M	2.87	1989	2291	2638	3039	3500	4031	4643	95,255
H	4.11	2057	2515	3076	3762	4600	5625	6879	122,256

MARKET ECONOMIES:

Reserves: 483,000 — Total Resources: 1,483,000

	Ave. Ann. Grth Rate	1980	1985	1990	1995	2000	2005	2010	Cumulative
Projected Demand:M	4.02	6695	8170	9984	12,167	14,811	18,011	21,881	392,340

CENTRALLY PLANNED ECONOMIES:

Reserves: 66,000 — Total Resources: 258 000

	Ave. Ann. Grth Rate	1980	1985	1990	1995	2000	2005	2010	Cumulative
Projected Demand:M	2.98	2623	3027	3496	4046	4689	5443	6328	128,072

TOTAL WORLD:

Reserves: 549,000 — Total Resources: 1,741,000

	Ave. Ann. Grth Rate	1980	1985	1990	1995	2000	2005	2010	Cumulative
Projected Demand:L	2.56	8864	10,059	11,415	12,954	14,700	16,681	18,930	403,051
M	3.76	9318	11,207	13,480	16,213	19,500	23,454	28,209	530,442
H	4.70	9825	12,359	15,546	19,556	24,600	30,944	38,925	648,783

graphic representation.) Adding the restriction of United States self-sufficiency would result in domestic reserve exhaustion occuring in (see Figure 2-9) 28 to 39 years. Therefore, it seems clear that in the case of copper, known reserves should be sufficient to meet any reasonable projection of demand over the period of study (Tables 2-16, 2-17, 2-18, 2-19).

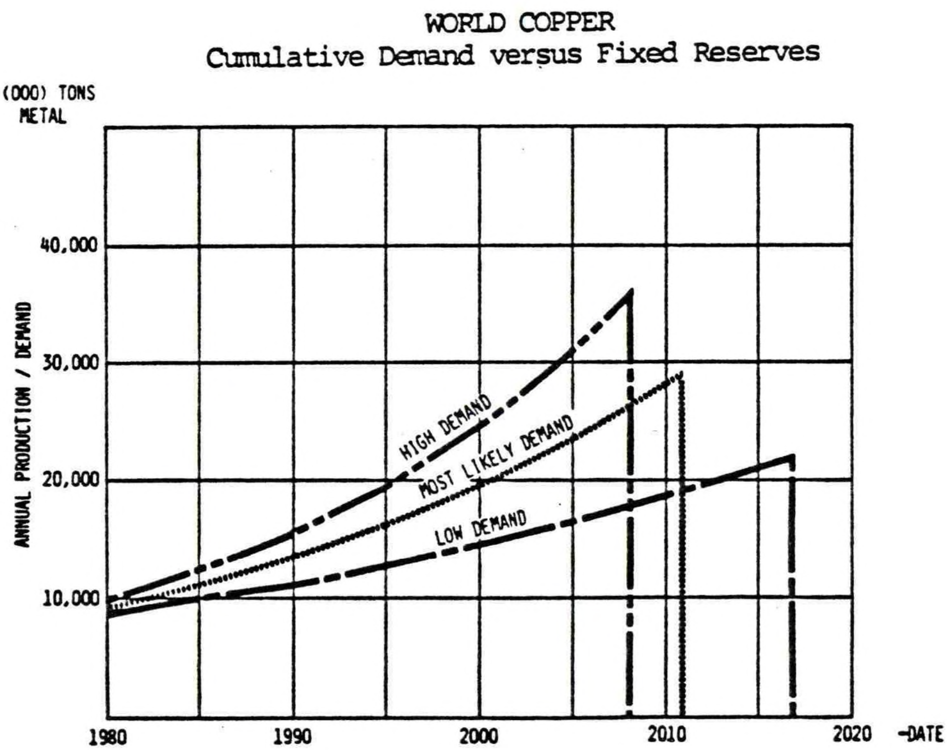


Figure 2-7

COPPER

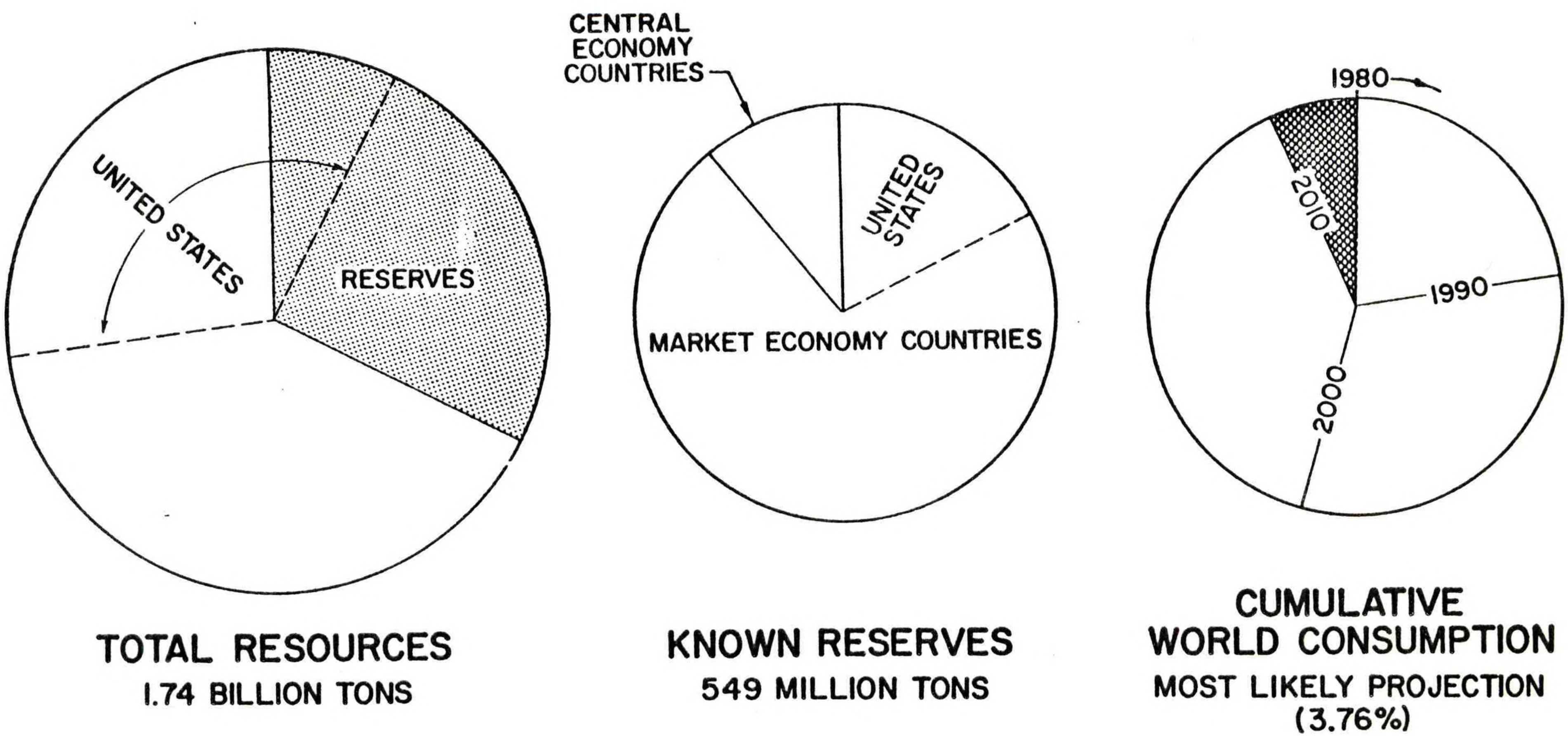


Figure 2-8

UNITED STATES SUPPLY DEMAND POSITION 1980-2010
CUMULATIVE DEMAND VERSUS FIXED DOMESTIC RESERVES-COPPER

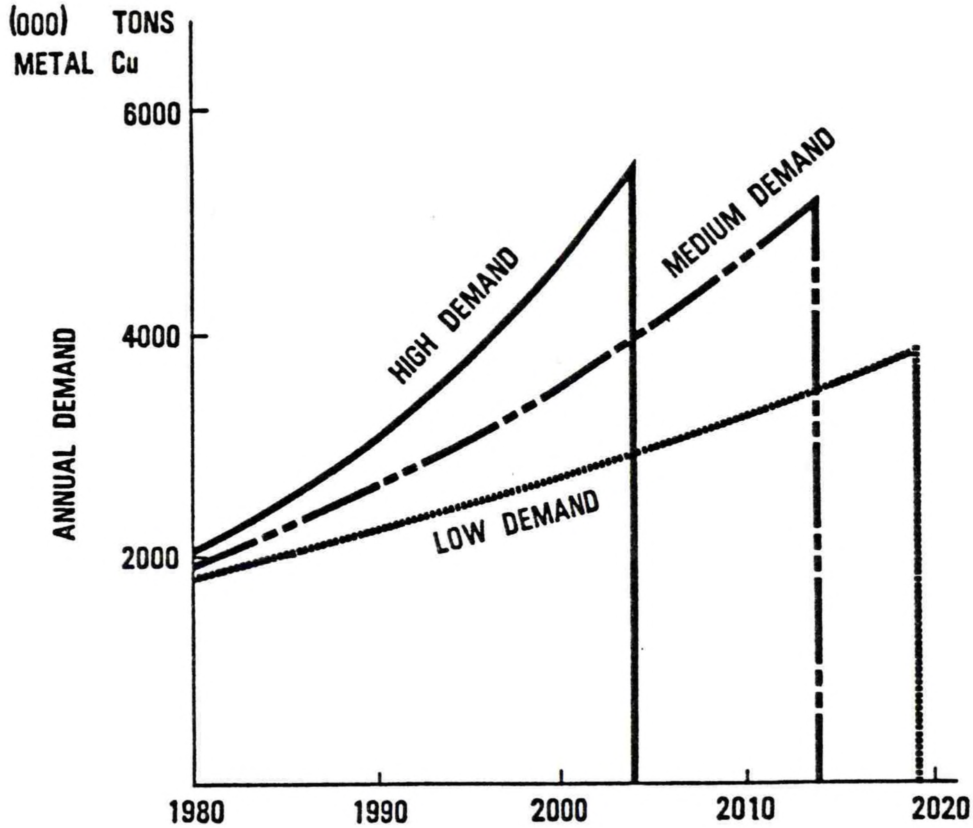


Figure 2-9

Table 2-14

WORLD DEMAND FOR REFINED COPPER
GROWTH RATES, 1951 TO 1971-1975

REGION	AVERAGE ANNUAL GROWTH RATE
United States	1.89
W. Europe	3.83
Japan	11.84
Other Developed Lands	3.82
Africa	6.69
Asia	5.97
Latin America	6.52
TOTAL MARKET ECONOMY COUNTRIES	3.83
U.S.S.R.	5.36
E. Europe	6.72
China	19.97
TOTAL CENTRAL ECONOMY COUNTRIES	6.56
World	4.37

Source: from data in Metallgesellschaft Aktiengesellschaft. Metal statistics (annual)

Table 2-15

WORLD CONSUMPTION OF REFINED COPPER-SELECTED YEARS
(in 1000 short tons of copper)

REGION	1929	%	1957	%	1965	%	1973	%	1975	%	1976	%	1977	%	1978	%
UNITED STATES	757	46	1282	37	1861	35	2448	32	1539	25	1965	28	2175	29	2391	30
JAPAN	83	5	195	6	473	9	1324	17	906	15	1158	16	1186	16	1377	17
WESTERN EUROPE	780	47	1695	48	2330	44	2921	38	--	—	2926	41	--	—	--	—
Germany	--	—	--	—	--	—	--	—	700	12	821	12	821	11	897	11
United Kingdom	--	—	--	—	--	—	--	—	497	8	505	7	550	7	557	7
France	--	—	--	—	--	—	--	—	402	7	405	6	426	6	362	5
Italy	--	—	--	—	--	—	--	—	320	5	355	5	359	5	367	5
Belgium	--	—	--	—	--	—	--	—	195	3	251	4	329	4	317	4
Other W. Europe	--	—	--	—	--	—	--	—	--	—	589	8	--	—	--	—
CANADA	14	1	118	3	226	4	255	3	216	4	237	3	251	3	272	3
AUSTRALIA, SOUTH AFRICA, NEW ZEALAND	17	1	41[a)]	1	88	2	218	3	--	—	--	—	--	—	--	—
OTHERS	--	—	164	5	316	6	480	6	1265	21	806	11	1453	19	1452	18
DEVELOPED COUNTRIES	1651	99	3331	95	5010	95	7166	94	--	—	6454	91	--	—	--	—
DEVELOPING COUNTRIES	11	1	164	5	284	5	480	6	--	—	638	9	--	—	--	—
TOTAL, MARKET ECONOMIES	1662	100	3495	100	5294	100	7646	100	6040	100	7092	100	7550	100	7992	100
TOTAL, CENTRAL ECONOMY COUNTRIES	--	—	514	—	998	—	2003	—	--	—	2306	—	--	—	--	—
TOTAL WORLD	--	—	4009	—	6292	—	9649	—	--	—	9398	—	--	—	--	—

-- = not available.

a) Australia only.

Source: American Bureau of Metal Statistics Yearbook, 1964 and 1970. "Resources for Freedom," Report of the President's Materials Policy Commission, June, 1952, V. II, p. 192. 1973 and 1976 data from Metal Statistics, 1966-1976, in Malenbaum, 1978. (Frankfurt, Metalgesellschaft, 64th ed., 1977, Lesemann, R.H., in "Copper," and Engineering and Mining Journal, March, 1978, 1979.)

Table 2-16

GROWTH OF WORLD COPPER RESERVE ESTIMATES
(in 1000 short tons of metal)

COUNTRY	1950	1960	1970	1975	1976	1977	1978	1979
UNITED STATES				90,000	90,000	93,000	93,000	107,000
CHILE				70,000	86,000	93,000	93,000	107,000
ZAMBIA				30,000	30,000	31,800	32,000	37,000
PERU				30,000	30,000	32,900	35,000	35,000
CANADA				40,000	40,000	34,400	34,000	35,000
ZAIRE				20,000	20,000	28,200	28,000	26,000
PHILIPPINES					16,000	18,500	19,000	20,000
PAPUA NEW GUINEA						9,800	10,000	16,000
AUSTRALIA					10,000	8,400	8,000	9,000
SOUTH AFRICA, REPUBLIC OF						3,000	3,000	6,000
OTHER MARKET ECONOMIES				95,000	88,000	88,000	82,000	85,000
TOTAL, MARKET ECONOMIES				375,000	410,000	441,000	437,000	483,000
USSR				---	40,000	40,000 e)	40,000 e)	40,000 e)
POLAND				---		14,000	14,000	14,000
OTHER CENTRAL ECONOMIES				---		11,000 e)	12,000 e)	12,000 e)
TOTAL, CENTRAL ECONOMY COUNTRIES				55,000	40,000	65,000	66,000	66,000
WORLD TOTAL	110,000	170,000	308,000	430,000	450,000	506,000	503,000	549,000

e=estimate

Sources: U.S. Bureau of Mines, Mineral Commodity Summaries (1975-1979). Tilton, J. E., The Future of Nonfuel Minerals, Brookings Institute, 1977, p. 10. U. S. Bureau of Mines, "Copper," in Mineral Facts and Problems, 1960. U. S. Bureau of Mines, "Copper," in Mineral Facts and Problems, 1970.

Table 2-17

Major Open Pit Copper Mines of the Market Economy Countries

UNITED STATES

Company	Mine or Location	Tonnage Size
Arizona		
Anamax Mining Co.	Twin Buttes	2
Asarco Inc.	Mission	2
	Silver Bell	2
	Sacaton	2
	San Xavier	2
Cities Service Co.	Miami	2
	Pinto Valley	1
Cyprus Johnson Copper Company	Benson	2
Cyprus Mines Corp.	Bagdad	2
Cyprus Pima Mining Co.	Pima	1
Duval Corp.	Mineral Park	2
	Esperanza	2
Duval Sierrita Corp.	Sierrita	1
Inspiration Cons. Copper Co.	Inspiration	2
	Christmas	2
	Ox Hide	2
Kennecott Copper Corp.	Ray	1
Phelps Dodge Corp.	Morenci	1
	New Cornelia	2
Ranchers Explor. & Devel. Corp.	Bluebird	2
Florida		
E. I. duPont de Nemours	Highland	2
Montana		
Anaconda Co.	Butte	1*
Nevada		
Anaconda Co.	Yerington	1
Duval Corp.	Battle Mountain	2
Kennecott Copper Corp.	McGill	2

Tonnage size ranges, annual ore production:
1- over 10 million tpy.
2- 1 to 10 million tpy.
3- 500,000 to 1 million tpy.
NA- Tonnage capacity not available.
*- Tonnage reflects combined open-pit and underground operations.
(U)- Under development.

Major Open Pit Copper Mines of the Market Economy Countries

UNITED STATES (continued)

Company	Mine or Location	Tonnage Size
New Mexico		
Kennecott Copper Corp.	Chino	2
Phelps Dodge Corp.	Tyrone	1
USNR Mining & Minerals Inc.	Silver City	3
U. V. Industries, Inc.	Continental	3*
Utah		
Kennecott Copper Corp.	Bingham Canyon	1

CANADA

Company	Mine or Location	Tonnage Size
British Columbia		
Bethlehem Copper Corp.	Ashcroft	2
Brenda Mines Ltd.	Brenda	1
Gibraltar Mines Ltd.	Gibraltar	1
Granby Mining Corp.	Phoenix	4
Granisle Copper Ltd.	Granisle	2
Highmont Mining Corp. Ltd.	Highmont	2
Lornex Mining Corp. Ltd.	Logan Lake	1
Noranda Mines Ltd.	Bell Copper	2
Similkameen Mining	Ingerbelle	2
Utah Mines Ltd.	Island Copper	1
Wesfrob Mines Ltd.	Tasu	2*
Manitoba		
Sherritt Gordon Mines Ltd.	Ruttan	2
New Brunswick		
Brunswick Mining & Smelting Corp. Ltd.	Bathurst	2*

Tonnage size ranges, annual ore production:

1- over 10 million tpy.
2- 1 to 10 million tpy.
3- 500,000 to 1 million tpy.
NA- Tonnage capacity not available.
*- Tonnage reflects combined open-pit and underground operations.
(U)- Under development.

Table 2-17 (Continued)

Major Open Pit Copper Mines of the Market Economy Countries

CANADA (continued)

Company	Mine or Location	Tonnage Size
Ontario		
Inco Ltd.	Clarabelle	2
Quebec		
Gaspe Copper Mines Ltd. (N.P.L.)	Copper Mountain	2
	Needle Mountain	2*
Noranda Mines Ltd.	Horne	3*
INTERNATIONAL		
AUSTRALIA		
Kanmantoo Mines Ltd.	Nairne	3
Mount Isa Mines Ltd.	Mount Isa	2*
Mount Lyell Mining & Railway Co., Ltd.	Queenstown	2*
Warman International Ltd.	Mount Morgan	2
BOTSWANA		
Bamangwato Concessions Ltd.	Selebi-Pikwe	2*
CHILE		
Corp. National Del Cobre	Chuquicamata	1
	Exotica	2
Cia. Minera Disputada De Las Condes SA	Grupo Disputada	2*
Emp. Minera De Mantos Blancos SA	Mantos Blancos	2*
CYPRUS		
Cyprus Mines Corp.	Skouriotissa	2
EIRE		
Avoca Mines Ltd.	Avoca	3*
Irish Base Metals Ltd.	Tynagh	3*

Tonnage size ranges, annual ore production:

1- over 10 million tpy.
2- 1 to 10 million tpy.
3- 500,000 to 1 million tpy.
NA- Tonnage capacity not available.
*- Tonnage reflects combined open-pit and underground operations.
(U)- Under development.

Table 2-17 (Continued)

Major Open Pit Copper Mines of the Market Economy Countries

INTERNATIONAL (continued)

Company	Mine or Location	Tonnage Size
FINLAND		
Outokumpu Oy	Vuonos	2*
INDONESIA		
Freeport Indonesia Inc.	Irian Jaya	2
ISRAEL		
Timna Copper Mines, Ltd.	Timna	3*
JAPAN		
Nittetsu Mining Co.Ltd.	Kamaishi	2*
MAURITANIA		
Soc. Miniere de Mauritanie	Akjoujt	2
MEXICO		
Cia. Minera de Cananea SA	Cananea	2
NICARAGUA		
Rosario Mining of Nicaragua, Inc.	Rosita	3*
PAPUA NEW GUINEA		
Bougainville Copper Ltd.	Panguna	1
PERU		
Southern Peru Copper Corp.	Toquepala-Ilo	1
PHILIPPINES		
Apex Exploration & Mining Co. Inc.	Wagas Mapula	2
Atlas Consolidated Mining & Devel. Corp.	Toledo	1*

Tonnage size ranges, annual ore production:

1- over 10 million tpy.
2- 1 to 10 million tpy.
3- 500,000 to 1 million tpy.
NA- Tonnage capacity not available.
*- Tonnage reflects combined open-pit and underground operations.
(U)- Under development.

Table 2-17 (Continued)

Major Open Pit Copper Mines of the Market Economy Countries

INTERNATIONAL (continued)

Company	Mine or Location	Tonnage Size
PHILIPPINES (continued)		
Baguio Gold Mining Co.	Sto Nino	2*
Benguet Consolidated Inc.	Dizon	2
CDCP Mining Corp.	Basay	2
Consolidated Mines Inc.	Isao-Pili	3
Marcopper Mining Corp.	Tapian	1
Marinduque Mining and Industrial Corp.	Sipalay	2
Philippine Iron Mines Inc.	Larap	3*
RHODESIA		
M.T.D. (Mangula) Ltd.	Mangula	2*
SOUTH AFRICA		
Palabora Mining Co.Ltd.	Palabora	1
SPAIN		
Rio Tinto Patino SA	Cerro Colorado	2
	Santiago	2
Tharsis Sulphur & Copper Co. Ltd.	Tharsis	3
Union Explosivos Rio Tinto SA	Riotinto	2*
SWEDEN		
Boliden AB	Aitik	2
TURKEY		
Etibank Genel Mudurlugu	Maden-Elazig	3
ZAMBIA		
Miniera Di Fragne	Mkushi Bona	2
Nchanga Consolidated Copper Mines Ltd.	Chingola	2*
	Rokana	2*
	Buoana Mkubwa	2
Roan Consolidated Mines Ltd.	Chambishi	2*

Tonnage size ranges, annual ore production:

1- over 10 million tpy.
2- 1 to 10 million tpy.
3- 500,000 to 1 million tpy.
NA- Tonnage capacity not available.
*- Tonnage reflects combined open-pit and underground operations.
(U)- Under development.

Table 2-18

Major Underground Copper Mines of the Market Economy Countries

UNITED STATES

Company	Mine or Location	Tons Per Day Mined
ARIZONA		
El Paso Natural Gas Co.	Lakeshore Mine Casa Grande, Ariz.	15,000
Magma Copper Co. Newmont Mining Corp.	San Manuel Mine San Manuel, Ariz.	52,000
MISSOURI		
Dresser Industries/ Cominco American Inc.	Magmont Mine Bixby, Mo.	4,200
CANADA		
Brunswick Mining & Smelting Corp.	Bathurst, New Brunswick	9,400
Inco	Coleman Mine Levack, ONTARIO	3,200
Inco	Copper Cliff South Mine Sudbury, ONTARIO	3,500
Inco	Crean Hill Mine Sudbury, ONTARIO	3,500
Inco	Creighton Mine Creighton, ONTARIO	9,500
Inco	Frood Mine Sudbury, ONTARIO	7,700
Inco	Garson Mine Garson, ONTARIO	4,800
Amax, Inc. Inco	Heath Steele Mines Ltd. Newcastle, NEW BRUNSWICK	3,500

Table 2-18 (Continued)

Major Underground Copper Mines of the Market Economy Countries

CANADA (continued)

Company	Mine or Location	Tons Per Day Mined
Texasgulf Canada	Kidd Creek No. 1 Mine Timmins, ONTARIO	14,000
Inco	Levack Mine Levack, ONTARIO	5,000
Inco	Levack West Mine Levack, ONTARIO	4,000
Inco	Little Stobie Mine Sudbury, ONTARIO	3,000
Gaspe Mines (Div. of l'Affinerie Canadienne de Cuiure Limitee)	Needle Mountain Mine Murdochville, QUEBEC	3,750
Noranda Mines Ltd. Geco Div.	Noranda Mines Ltd. Manitouwadge, ONTARIO	5,000
Inco	Shebandowan Mine Thunder Bay, ONTARIO	1,900
Inco	Stobie Mine Sudbury, ONTARIO	12,800
Falconbridge Nickel Mines Ltd.	Wesfrob Mines Ltd. Tasu, BRITISH COLUMBIA	3,800

Table 2-18 (Continued)

Major Underground Copper Mines of the Market Economy Countries

INTERNATIONAL

Company	Mine or Location	Tons Per Day Mined
AUSTRALIA		
Mt. Lyell Mining & Railway Co. Ltd.	Mt. Lyell Queenstown, Tasmania, AUSTRALIA	6,000
BOTSWANA		
Amax AAC of S.A. Botswana Government public	BCL Ltd. Selebi-Pikwe BOTSWANA	4,700
CHILE		
Government owned	Codelco, El Salvador Div. Santiago, CHILE	26,000
Chilean government	Codelco, El Teniente Div. Rancagua, CHILE	57,000
INDIA		
Indian government	Mosaboni Group Mines Mosaboni, Bihar, INDIA	4,000
PHILIPPINES		
Atlas Consolidated Mining & Development Corporation	Don Andres Soriano Toledo City, PHILIPPINES	20,000

Table 2-18 (Continued)

Major Underground Copper Mines of the Market Economy Countries

INTERNATIONAL (continued)

Company	Mine or Location	Tons Per Day Mined
QUEENSLAND		
M.I.M. Holdings, Ltd.	Isa Mine Mount Isa, QUEENSLAND	29,000
SOUTH AFRICA		
Messina (Tvl) Development Co.,Ltd.	Messina (Tvl.) Development Co. Ltd. Messina, North Tvl. SOUTH AFRICA	2,800
Newmont Mining Corp. Amax	O'okiep Copper Co. North-West Cape Province SOUTH AFRICA	5,770
Lonrho Falconbridge Superior Oil	Western Platinum Ltd. Marikana, Rustenburg Dist. Transvaal, SOUTH AFRICA	3,300
ZAMBIA		
Zambia Copper Investments Ltd. and Government	No. 1 Shaft- NCCM Ltd. Konkola Div. Chililabombwe, ZAMBIA	3,569
" "	No. 3 Shaft- Chililabombwe, ZAMBIA	2,157

Table 2-19

ANNOUNCED EXPANSIONS TO MINE PRODUCTION CAPACTIY
- By Country, By Year -
(in short tons of metal)

DATE	Country/Mine, Location	Estimated Capacity (tpy)	Net Increase or (Decrease)
	NORTH AMERICA		
	United States		
1979	Carr Fork. Utah	60,000	
	Palo Verde, Arizona	10,000	
	Battle Mountain, Arizona	7,000	
	Christmas,	6,000	
	Esperanza, (one-half)	7,500	
			90,500
1980	Cyprus - Pima,	25,000	
	Esperanza,	7,500	
	Hillsboro, New Mexico	N.A.	
			32,500+
1981	Oracle Ridge, Arizona	15,000	
	Troy, Montana	20,000	
	Ruth (tailings), Nevada	20,000	
			55,000
1982	Miami-East/Pinto Valley, Arizona	15,000	
	Chino (expansion)	40,000	
			55,000
			233,000+

DATE	Country/Mine, Location	Estimated Capacity (tpy)	Net Increase or (Decrease)
	CANADA		
1979	Kidd Creek, Ontario	20,000	
	Madeleine Mines,	10,000	
	Craigmont,	(5,000)	
			25,000
1980	Ruttan Lake, Manitoba	32,000 (e)	
	Craigmont,	(20,000)	
	Brunswick #12,	3,000	
	Sam Goosly,	7,000	
	Bell,	3,000	
	Sturgeon Lake,	(10,000)	
	Buchaus	(1,000)	
	Granduc	17,000	
			31,000 (e)
1981	Kidd Creek, Ontario	35,000	
	Highmont	25,000	
	Fraser	3,000	

ANNOUNCED EXPANSIONS TO MINE PRODUCTION CAPACTIY
- By Country, By Year -
(in short tons of metal)

DATE	Country/Mine, Location	Estimated Capacity (tpy)	Net Increase or (Decrease)
	CANADA (Continued)		
1981	Fraser	3,000	
	Detour	15,000	
	Lornex	30,000	
	Whitehorse	(5,000)	
			103,000
1982	Copper Mountain (3), B.C.	27,000 (e)	
	Whitehorse,	(7,000)	
	Madeleine Mines,	(10,000)	
			10,000 (e)
1983	Highland Valley, B.C.	65,000	
			65,000
			234,000 (e)
	MEXICO		
1979	La Caridad, Sonora	175,000	
			175,000
1982	Cananea, Sonora	20,000 (e)	
			20,000 (e)
			195,000 (e)
	SOUTH AMERICA		
	CHILE		
1979	Mina Sur (Exotica) (expansion)	35,000	
			35,000
1980	Lo Aguirre	20,000	
			20,000
1981	El Indio	6,000	
	Los Brochos - Disputada (expansion)	5,000	
			11,000
1982	El Salvador (expansion)	15,000	
	Andina (expansion)	25,000	
			40,000
1983	Andocollo	75,000	75,000
			181,000

ANNOUNCED EXPANSIONS TO MINE PRODUCTION CAPACTIY
- By Country, By Year -
(in short tons of metal)

DATE	Country/Mine, Location	Estimated Capacity (tpy)	Net Increase or (Decrease)
	PERU		
1979	Cerro Verde	37,000	
			37,000
1982	Cobriza (expansion)	30,000	
			30,000
			67.000
	ARGENTINA		
1984	El Pachon, San Juan Province	110,000	
			110,000
	BRAZIL		
1980	Caraiba	40,000	
			40,000
1981	Camague	12,000	
	Pedra Verde	8,000	
			20,000
			60,000
	AFRICA		
	ZAIRE		
1981	Shaba region	130,000	
			130,000
	SOUTH AFRICA		
1980	Broken Hill, Aggeneys	6,000	
			6,000

ANNOUNCED EXPANSIONS TO MINE PRODUCTION CAPACTIY
- By Country, By Year -
(in short tons of metal)

DATE	Country/Mine, Location	Estimated Capacity (tpy)	Net Increase or (Decrease)
	ASIA		
	PHILIPPINES		
1979	New Bataan	12,000	
	Baguia (expansion)	13,000 (e)	
	Basay	15,000	
	Dizon	25,000	
	Ino (Marindugne Island)	NA	
	North Davao	27,000	
			+92,000 (e)
1980	San Marcelino, Pua	26,000 (e)	
			26,000 (e)
1981	Billy Bueno	8,000	
	San Antonio	10,000	
			18,000
1983	Sipalay	35,000	
	New Bataan (expansion)	8,000	
			43,000
			+179,000 (e)
	INDONESIA		
1981	Ertsberg East	20,000	
			20,000
	IRAN		
1984	Sar Cheshmeh	160,000	
			160,000
	BURMA		
1982	Monywa	60,000	
			60,000

ANNOUNCED EXPANSIONS TO MINE PRODUCTION CAPACTIY
- By Country, By Year -
(in short tons of metal)

DATE	Country/Mine, Location	Estimated Capacity (tpy)	Net Increase or (Decrease)
	INDIA		
1980	Khetri/Kalihan, Rajasthan (expan.)	25,000 (e)	
			25,000 (e)
	OMAN		
1980	Laisal/Aarja/Bayda	20,000	
			20,000
	OCEANIA		
	AUSTRALIA		
1979	Kanmantoo	8,000	
	Gunpowder	10,000	
			18,000
1980	Woodlawn	14,000	
			14,000
1981	Teutonic Bore	6,000	
			6,000
			38,000
	EUROPE		
	SPAIN		
1979	Aipsa, Huelva Province	10,000 (e)	
			10,000
1980	Aznacollar	40,000 (e)	
			40,000
1981	Rio Tinto Minerva	20,000	
			20,000
			70,000 (e)

ANNOUNCED EXPANSIONS TO MINE PRODUCTION CAPACTIY
- By Country, By Year -
(in short tons of metal)

DATE	Country/Mine, Location	Estimated Capacity (tpy)	Net Increase or (Decrease)
	SWEDEN		
1980	Aitik (one-half)	5,000	
			5,000
1981	Aitik (one-half)	5,000	
			5,000
			10,000
	YUGOSLAVIA		
1979	Bucim/Cudar	20,000	
			20,000
1980	Bor Velika Krivaja	33,000	
			33,000
			53,000
	CENTRAL ECONOMY COUNTRIES		
	U.S.S.R.		
1980	Erdinet, Mongolia	50,000 (e)	
			50,000
	POLAND		
1982	Sieroszowice	50,000	
			50,000
	BULGARIA		
1980	Elatsite	33,000 (e)	
			33,000

3.0 MANGANESE: SUPPLY-DEMAND

3.1 INTRODUCTION

World reserves of manganese are large, but highly concentrated in just a few countries. Whether they are developed at a pace rapid enough to meet forecast demand will be largely a function of not only the market price of manganese, but also of the political and social stability of the producer nations. Manganese resources, although very large, are of questionable economic value becaue of their often low grade-tonnage characteristics, remote locations, and/or metallurgical characteristics.

In Part I of the discussion which follows, the economic geology of world manganese reserves and resources will be reviewed. The three major geologic environments in which manganese occurs will be discussed and examples of deposits belonging to each generic type given. Part II will review the reserves and resource position of the major manganese producing regions of the world. Deposits currently producing significant quanitities of ore will be described, and when possible, exhaustion dates predicted. Those few deposits known to be under or awaiting development will also be discussed. An investigation of manganese demand follows, with a brief look at recycling and substitution, and a review of past as well as projected demand.

3.2 MANGANESE SUPPLY

3.2.1 Part I - The Economic Geology of Manganese

3.2.1.1 Mineralogy

Manganese, the twelfth most common element in the earth's crust (about 0.1 percent) is distributed throughout the world, but never in the metallic state. Elemental manganese is a constituent of many crystalline rocks, from which it is often dissolved and redeposited to other locations. Because of its high affinity for oxygen, manganese most often occurs as an oxide mineral, or less frequently, in either manganese silicates or carbonates. Of the more than 125 minerals of manganese, relatively few are of commercial significance of major importance are:

- Pyrolusite: (MnO_2), generally with some H_2O; 63.19 percent manganese, 36.81 percent oxygen when pure. Pyrolusite is one of the most common manganese minerals. It usually forms in highly oxidizing environments such as bogs, in the oxidized zone of manganiferous ore deposits, and in deposits formed by circulating waters. It is associated with other manganese minerals and often contains minor amounts of silica, lime and barite. Manganite alters readily to pyrolusite. As such, some deposits of pyrolusite appear to be largely secondary after manganite. Important occurrences of pyrolusite are found in the Soviet Union, India, Ghana, Brazil, Cuba, and Central America. In the United States, it has been found at Batesville, Arkansas, and at Philipsburg, Montana.

- Psilomelane $(Ba^{2+},Mn^{2+})_3(O,OH)_6Mn_8^{4+}O_{16}$, contains 45 to 60 percent manganese and is apparently a colloidal form of pyrolusite which has absorbed impurities such as water, and the oxides of sodium, potassium, and barium. Psilomelane is commonly formed through the weathering of manganese silicates and carbonates. Psilomelane is frequently associated with pyrolusite, goethite, and limonite (all of which are iron oxides), and with hausmannite and braunite. It is the most abundant ore mineral in the Indian deposits, and is also found at Nikopol (Russia) and in Brazil.

- Manganite MnO(OH), contains 62.4 percent manganese when pure. It occurs as a low temperature vein mineral, in igneous acid rocks, and as replacements and deposits formed by meteoric waters. Manganite is frequently altered to pyrolusite, hence the aforementioned association. Other alteration products are psilomelane, braunite and hausmannite. Manganite is found in the Sandur District of India, and in scattered occurrences in the United States.

- Braunite, ($2Mn_2O_3MnSiO_3$), contains up to 69 percent manganese when pure, but never occurs in the pure form in nature: invariably, it is associated with silica (8 to 10 percent), either mechanically mixed or chemically combined, and is also always associated with other manganese minerals. Braunite may sometimes be primary, although it is usually secondary. In veins or lenses it results from metamorphism of manganese oxides and silicates. With psilomelane, pyrolusite and wad, braunite occurs as a weathering product. It is found in most of the Indian and Brazilian deposits.

- Hausmannite (Mn_3O_4), which contains 72.5 percent manganese when pure, is a primary mineral occurring in high temperature hydrothermal veins in association with acid igneous rocks. It is also found as a contact metamorphic mineral, and as a recrystallization product in metamorphosed sedimentary or residual manganese ore deposits. Hausmannite may be artificially produced by sintering manganese carbonates and oxides. Large deposits of hausmannite are quite rare, although occurrences are widely distributed. It occurs in the Brazilian manganese deposits, and in the United States it is an important constituent of the ores at Batesville, Arkansas.

- Rhodochrosite ($MnCO_3$), contains 47.6 percent manganese when pure and is best classified as a manganese carbonate with variable amounts of iron, calcium, and manganese carbonates. A pink to red, fairly soft mineral, it is comparatively rare, although often associated with gangue minerals in copper and lead-zinc vein deposits. Rhodochrosite also occurs in deposits of sedimentary origin, and in metamorphic deposits that were derived from sediments. Scattered occurrences of rhodochrosite are found at Butte, Montana, and in the French Pyrenees rhodochrosite has been mined as ore.

- o <u>Rhodonite</u> ($MnSiO_3$), contains 42 percent manganese. Its value is chiefly academic; its primary use is as an ornamental stone.

- o <u>Bementite</u> ($2MnSiO_3$ H_2O), contains 39 percent manganese, and is a hydrated silicate. Of minor importance, it has been mined at various localities in the Olympic Peninsula of Washington.

In addition to these manganese minerals of relatively well-defined composition, there are several others of more variable composition, most notably:

- o <u>Wad</u>, or "bog manganese," a soft, earthy, hydrous manganese oxide with variable manganese content. Not a true mineral, wad consists of a mixture of manganese oxides, iron oxides and detrital materials. As the name implies, wad is commonly found in swamps and lakes.

- o <u>Manganiferous Iron Ores</u>, contain variable mixtures of manganese and iron oxides usually of more than 40 percent iron and 5 to 10 percent (or less) manganese. An important domestic deposit of this ore occurs in the Cuyuna Range of Minnesota.

As discussed earlier, there are many more minerals of manganese. It should be noted, however, that the two silicates (bementite, rhodonite), the carbonate (rhodochrosite), and hausmannite (an oxide), are of little more than secondary importance. Psilomelane, pyrolusite, wad, and braunite are predominant to the virtual exclusion of all other manganese minerals.

3.2.1.2 Types of Deposits

The USGS classifies manganese deposits into three categories (Professional Paper 820):

- o Sedimentary deposits (including sea-floor nodules, here excluded).
- o Volcanogene deposits.
- o Hypogene deposits.

These three classes are intergradational, and in places it is difficult or impossible to distinguish between the different types of deposits.

3.2.1.2.1 Sedimentary Deposits

Sedimentary manganese deposits often contain both carbonate and oxide minerals, the origin of which are believed to have been due to the leaching of surrounding rocks during normal weathering with subsequent transportation in stream sediments to bodies of water followed by chemical precipitatation. The major difference between the formation of oxides and carbonate deposits is thought to be related to the final sedimentary environment. Carbonate deposits are thought to have been formed in reducing environments, whereas oxide deposits formed under strongly oxidizing conditions.

The largest manganese deposits in the world are stratiform oxide deposits, examples of which are those at Nikopol and Tchiaturi in Russia, the Kahlahari Basin in South Africa, and Urucum in Brazil. In many deposits the manganese occurs in a single bed; in others the manganese is in several separated beds. Most commonly, the deposits consist of lenses a thousand feet to a few miles long, and a few tens of yards to a mile or more wide. Thickness of the beds ranges from a fraction of an inch to about 100 feet. With the exception of the Kalahari Field, few ore grade primary oxide beds are more than 10 feet thick.

Ore grade in sedimentary manganese oxide deposits varies widely. Typical ore tenor runs 25 to 40 percent manganese; a few of the higher grade deposits run significantly richer. Tonnages range from 20 to 30 thousand tons to over a billion tons of ore. However, given the large tonnage mining techniques usually employed with this type of deposit, only the larger deposits are currently regarded as economically viable to develop in greenfield areas.

In general, carbonate deposits are uneconomic unless they are weathered and concentrated as oxides. High-grade oxide ores resulting from the weathering of such deposits rarely occur in bodies of more than 50 million tons. The most productive deposits are Nsuta in Ghana, Serro do Navio in Amapa, Brazil, and the Kisenge deposits in Zaire. The Moanda deposit in Gabon, by far the largest of this type yet discoved, has reserves of about 200 million tons of high-grade (+50 percent) secondary oxide ore. An exception to the "weathering requirement" is the Molango deposit in Hidalgo, Mexico, where the carbonate ore beds are both large and relatively high in grade.

A third type of sedimentary manganese deposit is associated with iron formations. The manganiferous beds commonly contain oxides concentrated in layers separate from the iron-rich beds, as at Urucum, Brazil. In other deposits, the manganese is distributed throughout many of the iron oxide layers, as in the Cuyuna Range, Minnesota.

3.2.1.2.2 Volcanogene Deposits

The volcanogene manganese deposit classification is really an intermediate category which combines elements of the previously discussed sedimentary-type deposits and the hypogene-type deposits which are addressed later in this section. Volcanogene deposits are closely associated with volcanic rocks, of both pyroclastic and flow origin, and are either subaerial or submarine. Although volcanogenic deposits are known to occur in continental volcanic rocks, the most important deposits are found in marine or lacustrine environments, and as such, are technically sedimentary deposits. Manganese oxide minerals predominate in volcanogenic deposits, but if deposition took place in an oxygen deficient area, carbonates would have formed, thereby enabling the manganese in solution to migrate elsewhere.

Manganese reserves in volcanogene deposits are typically small to medium in size. Very few deposits contain more than a million tons of recoverable ore, the largest probably being the Charco Redondo deposit in Cuba which produced about 5 million tons, and the Autlan deposits in Mexico which produced about 4 million tons before they were exhausted.

3.2.1.2.3 Hypogene Deposits

Hypogene manganese deposits (which are of limited commercial interest unless located in a non-remote area because of their typically small size) are thought to have originated from hydrothermal solutions. Some probably are derived from magmatic fluids, whereas others may have been generated by heated ground waters which obtained their manganese content while circulating through various acidic rocks. Both vein and replacement deposits are known to occur. Vein-type deposits (e.g., Butte, Montana) are predominately composed of manganese oxides; howver, a few occurrences are known to also contain carbonates. Replacement-type deposits (e.g., Philipsburg, Montana) may occur as either oxide or carbonate minerals. Near surface occurrences are typically oxidized by weathering processes.

3.2.2 Part II - Manganese Reserves and Resources (Figure 3-1)

3.2.2.1 Introduction

The market supply of manganese traditionally has been determined largely by the level of world steel production. Since more than 90 percent of the world's production of manganese is consumed by the iron and steel industries, this correlation has been and remains strong. As demand increased, manganese production rose commensurately. Given the dearth of cheap substitutes,

IMPORTANT MANGANESE RESERVES & RESOURCES

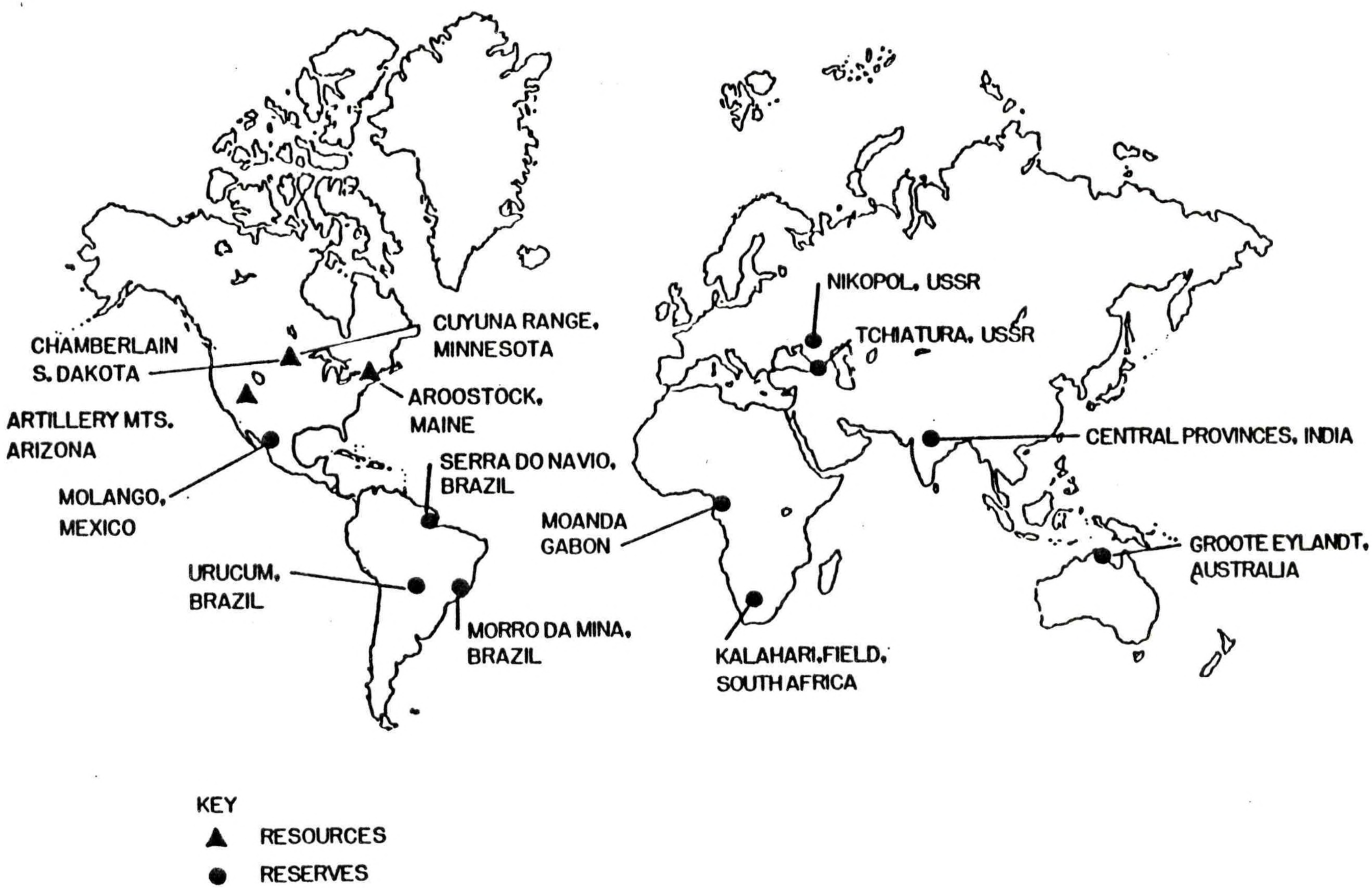


Figure 3-1

and the high proportion of captive (steel industry) mines, price has not been a major factor. Aggregate manganese demand was (and is) not very price elastic. But, because manganese ores are a heterogeneous group, the price elasticity of demand for ore from any given producer has been fairly high. However, that situation may be changing.

After World War II, a number of manganese deposits were located, developed, and for a brief period of time, were significant sources of supply. Among these were deposits located in:

o Angola	1949-1965
o Ghana	1955-1971
o Cuba	1930-1965
o Philippines	1950-1974
o Fiji	1955-1971
o Guyana	1945-1969
o Botswana	1952-1974
o Turkey	-1974
o United States	-1970

All are now effectively exhausted (Dykstra, 1979). In addition, many other currently producing deposits are known or thought to be nearing the end of their producible reserves. These include deposits in Brazil, Ghana, Zaire, and Morocco.

Offsetting the decline in number of producers, production has increased in several regions (e.g., South Africa, and the USSR) and two new, large deposits have been developed (Moanda in Gabon, Groote Eylandt in Australia).

The result has been an increase in the concentration of the manganese producing industry. This concentration, if not met with the development of new prospects (deemed unlikely by both those in industry and government), will probably result in the supply of manganese becoming less price elastic.

Such a reduction in the price elasticity of supply, when coupled with the essentially inelastic demand for manganese, would result in a greater price sensitivity to small shifts in the supply/demand equilibrium.

3.2.2.2 Projected Mine Capacity

Manganese mine production is usually given in one of two ways: either as the gross weight of ore produced, or in terms of the recoverable metal content. Although the latter figure is obviously the more important one, its reliability is not always as great as the former. Hence, both terms are used in various places in this report.

Interpreting capacity ratings is sometimes a problem, in that some subjective judgment is frequently involved. Rated capacity announcements may tend to overstate or understate actual mine capacity. Two factors have an important bearing on metal production capacity: ore grade and recovery.

In many manganese deposits, the variability in the lateral and horizontal grade of ore mineralization allows selective mining. Unfortunately, this variability in ore grade also permits short term highgrading (selective mining of richer ores) at the expense of future production. This activity can cause not only considerable variation in the effective production capacity of the mine, but can also lead to the premature abandonment of an orebody with medium- to low-grade ores still in place. Unlike some mineral deposits (e.g., copper porphyries), most manganese deposits are not gradational but tend to have rather distinct boundaries where the ore mineralization simply stops. An important variation occurs in deposits where secondary enrichment has taken place. In such a case, ore grades and mineralization may change markedly, but definite ore boundaries still exist.

Metal recovery is the second important factor in manganese production capacity. Higher grade ores may be used directly, or a simple beneficiation may be necessary (e.g., washing). (For a more detailed discussion of manganese ore beneficiation, please refer to Part II of this report.)

Capacity additions can be accomplished by two means: new mines, or expansion of existing mines. In the past, both have played important roles. The future is somewhat less clear, in that no major new deposits have been found in the last 20 years. As will be discussed further in a later section, increased production is most likely to come from expansions in South Africa, Gabon, and, perhaps, Brazil. Production changes in China and the Soviet Union are less predictable.

Lead times for new manganese mines are similar to that of other metals mines, on the order of 5 to 10 years. Unfortunately, in the case of manganese this knowledge is of little help in projecting future additions to capacity. This is due to three reasons:

- No large prospects (similar to today's large producing deposits) have been identified.
- Exploration is slow and difficult to conduct in remote tropical areas.
- Deposits found in remote areas must be large enough to support the development of a costly infrastructure.

Therefore, very few deposits can be accurately pointed to "on the horizon" as being developed, or awaiting development. Some authorities' supply projections reflect this problem as a shortfall in manganese supply during the late 1980's.

Additions to mine capacity through the expansion of existing facilities are therefore somewhat less difficult to predict, but remain difficult to accurately forecast. Factors helpful in this task are a knowledge of ore reserves, and the degree of existing infrastructure development/utilization.

In direct contrast to additions to capacity are subtractions, or plant closures. Closures typically occur as the result of one of two related occurrences: 1) the exhaustion of reserves, or 2) the premature depletion of economic reserves (due to earlier high-grading and/or price changes). Although the former can be predicted with some degree of certainty, the latter is determinable with very little assurance. In general, those deposits which have undergone secondary enrichment by selective leaching or oxidation--as for example Kuruman (India), Serra do Navio (Brazil) and Nsuta (Ghana)--lend themselves to "high-grading" because of their physical configuration. Typically, they have high-grade zones resulting from differential leaching/enrichment, grading down in quality to the manganiferous protore. Barring another zone of secondary enrichment at depth, the richest ore generally will be mined first, often leaving an uneconomic "reserve" beneath.

Projecting the future production capacity of the manganese industry is, therefore, a difficult proposition. This study attempts to identify not only the specific areas which will be developed, but also the areas which are being depleted. As discussed earlier in this section, the discernible trend is for an increased concentration of the manganese industry in just a few producing regions.

3.2.2.3 Reserve Estimates

The estimates shown in the tables in this section have been obtained from two sources: USBM, and the United States Steel Corporation. As can be seen in the accompanying table (USBM vs. U.S. Steel), large differences exist in some figures. Wherever possible, explanations for these differences will be elaborated on in the text.

Reserve estimates are given in Table 3-1 (and Table 3-16 at the end of this chapter) by:

TABLE 3-1

ESTIMATED WORLD RESERVES-RESOURCES OF MANGANESE

(in million short tons)

		GROSS WEIGHT OF ORE				MANGANESE CONTENT	
		Measured, Indicated and Inferred		Measured		Measured	
Notes		USBM 1979	U.S. Steel 1977	USBM 1979	U.S. Steel 1977	USBM 1979	U.S. Steel 1977
1.	NORTH AMERICA						
	Mexico	21	14	21	14	5	3
	Others	--	--	--	--	-	-
	TOTAL	21	14	21	14	5	3
	SOUTH AMERICA						
2.	Brazil	95	103	61	37	24	12
	Others	--	--	--	--	--	--
	TOTAL	95	103	61	37	24	12
	EUROPE						
	Morocco	--	3	--	1	-	.5
	Others	--	-	--	-	-	-
	TOTAL	--	3	--	1	-	.5
	AFRICA						
3.	South Africa	2,200	1,064+	722	722	274	274
4.	Gabon	165	213+	165	213	63	63
5.	Ghana	22	34	--	1	--	1
	Zaire	7	9	--	9	--	3
6.	Upper Volta	-	14	--	0	--	--
	Others	-	-	-	-	-	-
	TOTAL	2,394	1,334	887	945	337	341
	ASIA						
7.	India	65	118	8	8	4	3
	Japan	--	6	-	6	-	2
	Others	--	--	-	-	-	-
	TOTAL	65	124	8	14	4	5
	OCEANIA						
8.	Australia	330	539	209	209	53-84	36
	Others	--	--	--	--	--	--
	TOTAL	330	539	209	209	53-84	36
9.	OTHER MARKET ECONOMY COUNTRIES	9	324	4	1	1	--
	MARKET ECONOMIES TOTAL	2,914	2,441	1,190	1,221	424-453	398
	CENTRAL ECONOMY COUNTRIES						
10.	USSR	3,000	1,383	1,383	1,383	337	337
11.	China	50	32	--	32	--	11(?)
	Hungary	--	(?)4	--	(?)4	--	2(?)
	Other	27	--	--	--	--	-
	TOTAL	3,077	1,419	1,383	1,419	337	349
12.	WORLD TOTAL	5,991	3,860+	2,573	2,640	761-792	747

NOTES TO Table 3-1

ESTIMATED WORLD RESERVES-RESOURCES OF MANGANESE

Notes:

1. The USBM has not published any tabulation of measured reserves. The figures presented here were in the form of a release dated March 8, 1979. U. S. Steel figures presented here are from the testimony of Robert L. L'Esperance, Director-Corporate Explorations and Investigations, Resource Development, United States Steel Corporation, on April 22, 1977, before the Subcommittee on Oceanography of the Committee on Merchant Marine and Fisheries of the House of Representatives.
2. USBM figure is from 1978 Brazilian Government official statistics.
3. The USBM has no specifically defined measured ore reserve figure. The U. S. Steel figure has been accepted as appearing to be a reasonable estimate with the possibility that actual measured reserves may be larger.
4. USBM figures are from COMILOG, the Moanda Mine operator (1977), and are not specifically defined as measured, but rather as recoverable shipping product. The Bureau of Mines states that they accepted U.S. Steel's figures for manganese content because "...U.S. Steel presumably has access to firsthand information as part owner."
5. U. S. Steel's measured reserve figure for manganese (both ore and content) is for all oxide ore. The USBM figure for measured, indicated, and inferred ore reserves is for carbonate ore (from Caemi International, sales agents for operator, 1977).
6. Oxide ore only. Lower-grade carbonate ores are estimated at an additional 14.3 million tons.
7. USBM figure for measured content is from the Geological Survey of India and the Indian Bureau of Mines (1972) without recovery factor. With recovery factor, this would probably be about 2 million tons.
8. The range values given for USBM measured content is from the gross weight figure used in Monograph #5, Australian Institute of Mining and Metallurgy, 1975. It is the same gross weight figure as that of the U. S. Steel table. USBM content range uses the percent range of the U. S. Steel table, which is the same as that in page 93 of Australian Mines Handbook, 1976-1977, for the same gross weight. The USBM figure for measured, indicated, and inferred is from the parent company of the operator (1974).
9. U. S. Steel figure is for indicated or inferred material (mostly resources) credited to Bulgaria, United States, Chile, Mali, Israel, Canada, etc.
10. See note 3, above.
11. The USBM states it has no indication as to what part of the reserves are measured.
12. The plus sign denotes the possibility of additional inferred reserves.

- o Country and region.
- o Geopolitical status.
- o Deposit.

Manganese reserves and production (Tables 3-2, 3-3) are discussed on a country-by-country basis, in an approximate order of significance, followed by a review of domestic resources.

3.2.2.3.2 Union of Soviet Socialist Republics (U.S.S.R.)

Estimates of Soviet manganese reserves vary widely, depending on the source and the method of categorization. Smirnov (1974) states that 70 percent of the world's reserves are located in the U.S.S.R. U.S. Steel and USBM estimates of the Soviet share of manganese reserves are considerably lower, but still indicate that the U.S.S.R. controls the largest quantity of manganese ore in the world. U.S. Steel calculates (and the USBM accepts with the caveat that actual figures may be higher) measured reserves in the U.S.S.R. to contain 45 percent of the world's manganese metal. Total reserves (measured plus indicated plus inferred) are estimated by the USBM to include 50 percent of the world's manganese ore, while U.S. Steel believes that indicated and inferred reserves are uncertain and, therefore, should not be included, thereby reducing the Soviet Union to roughly 36 percent of the world's total in this category.

The vast bulk of Soviet reserves are located in the Tchiaturi and Nikopol regions, both of which have been exploited since the late 19th Century. Other, less well-developed reserves are located in western Siberia, the northern Urals, the Ukraine and Central Kazakhstan.

Tchiaturi (also known as Chiaturi, Chiatura). The Tchiaturi District is located in the Kutais region of the Republic of Georgia, on the south slope of the Caucasus Mountains. The mining area is approximately 19 miles long by 6 miles wide. Discovered in 1870, the original mineralized zone had been eroded by the Kivirila River into seven plateaus. The manganiferous beds are almost

Table 3-2

WORLD MANGANESE MINE PRODUCTION
(in 1000 short tons of ore)

COUNTRY	1970	1973	1974	1975	1976	1977	1978 e)
UNITED STATES	--	--	--	--	--	--	--
AUSTRALIA	828	1678	1678	1714	2375	1529	1700
BRAZIL	2071	2378	2000	1800	2400	990 e)	1200
GABON	1602	2115	2357	2458	2372	2040	2100
INDIA	1819	1692	1595	1688	1862	1955	2000
SOUTH AFRICA, REPUBLIC OF	2954	4603	4129	6359	6010	5564	4600#
OTHER MARKET ECONOMIES	2169	2766	1716	1818	1567	1500	1500
TOTAL MARKET ECONOMY COUNTRIES	11443	15232	13475	15837	16586	13578	13100
CENTRAL ECONOMY COUNTRIES	8641+	10158*	10700	11100	10700	10691	10700
WORLD TOTAL	20084	25390	24175	26937	27286	24269	23800

e) estimate
*Except Yugoslavia.
#Figure from World Mining, October, 1979.
Sources: Mineral Commodity Summaries, 1974-1979.
Tinsley, C.R., "Manganese," Engineering Mining Journal, March, 1979.

Table 3-3

WORLD MANGANESE PRODUCTION
(in 1000 short tons manganese content)

	1973	1975	1977
UNITED STATES	31	19	27
AUSTRALIA	806	857	730
BRAZIL	1,141	875	475
GABON	1,058	1,233	1,020
INDIA	626	624	684
SOUTH AFRICA, REPUBLIC OF	1,901	2,588	2,292
OTHER MARKET ECONOMIES	690	705	571
TOTAL, MARKET ECONOMY COUNTRIES	6,253	6,901	5,799
CENTRAL ECONOMY COUNTRIES	4,485	3,909	3,775
WORLD TOTAL	10,738	10,810	9,574

Sources: USBM Mineral Commodity Profile: Manganese, 1977, 1979.
DeHuff, G.L., USBM, Manganese, pp. 653-668.

horizontal, and are continuous. Although the entire bed averages 6 to 7 feet in thickness, the ore zone is uniformly 3 feet thick. The ore deposit itself is sedimentary in origin, and has primary oxide, carbonate, and hypergene oxide zones. Total tonnage for the deposit is estimated to be approximately 230 million tons of ore, with primary oxides and carbonates accounting for 47 percent and 39 percent of the reserves, respectively. Oxide grades average 45 to 52 percent manganese with highs up to 90 percent reported, making this deposit the major source of high-grade manganese in the Soviet Union. Unfortunately, the ore is soft and disintegrates readily to fines, thus making physical concentration difficult. This characteristic makes the ore unsuitable for use as blast furnace feed unless it is either mixed with other ore, or pelletized. (This may be of some importance in explaining the historical tendency for Tchiaturi ores to be the major source for Soviet manganese exports, and the reputed slow changeover in the U.S.S.R. from open hearth to blast furnace steel production.)

The Tchiaturi deposits have been mined for close to 100 years, and the mining operations are well developed. Mining takes advantage of the river canyons which exposed the ore bed; adits give entry to the mines. Mining is thought to be primarily by the longwall retreating method, using narrow pillars and allowing the roof to cave behind the workings. Tchiaturi is the second largest mine producer of manganese in the Soviet Union, behind Nikopol, with about 40 percent of Soviet production capacity.

<u>Nikopol Deposits</u>. The Nikopol district is located in the southern Ukraine on the Dneiper River, about 100 miles upstream from the Gulf of Odessa on the Black Sea. The district has many mines, and is divided into eastern and western fields, 10 to 15 miles apart, by a barren zone of crystalline rocks. The Nikopol manganese ore occurs as nodules, oolites, and concretions of pyrolusite and psilomelane in a horizontal bed of sandy clay, and lies at a depth of 50 to 250 feet. The ore bed averages 6.5 feet thick, and has an average grade of between 20 and 30 percent manganese. Total tonnage for the deposits is variously estimated at between 400 and 800 million tons of ore, but is thought to be inflated by the inclusion of material grading as low as

10 to 15 percent manganese. Actual "ore" reserves are probably closer to 250 million tons, grading less than 30 percent manganese (Dykstra, 1979).

Production began in 1886, and continues by surface mining methods. Mine production from Nikopol may account for over 50 percent of the Soviet Union's manganese production. Mine product was reported to be sorted, crushed and deslimed before concentration by tables, flotation, or magnetism. Recovery was said to be 75 percent, but some doubt exists on the behalf of western observers as to the accuracy of this figure.

Other Deposits. Numerous other deposits have been reported in Russian literature over the years, but very little is known about them. The Siberian deposits are thought to be mostly small in size, sufficiently low in grade to require concentration, relatively high in phosphorous, and most importantly, rather widely scattered. This last factor might be the economically prohibitive one, given the necessity for developing a vast new infrastructure. The Atasu region in Central Kazakhstan has many reported manganese occurrences, but only four or five have any economic significance. Reserves are estimated at 134 million tons, but the ore grade is low (20 to 30 percent manganese) and iron content high (around 50 percent). This region has been exploited since 1956, but production figures are unavailable.

Production at the Mazul (or Mozulsky) deposit (frequently mentioned in older literature) apparently ceased in 1956, after 20 years of development work. Indications are that the low grade of the deposit and concentration problems were the cause of abandonment. Other, scattered reserves and resources are reported but aside from the information presented in a table at the end of this section, little is known about them.

In conclusion, several other factors must be included in the assessment of U.S.S.R. manganese reserves. Since the turn of the century, the Soviet Union has been the major producer of manganese in the world. Until 1949, the Soviet Union was the primary supplier of manganese to the United States. At that point, export restrictions were imposed and alternate sources were found.

The U.S.S.R. remained a large exporter, however, primarily to countries in the COMECON. In recent years (since 1975), the level of exports to both COMECON and the Market Economy Countries has declined: Soviet exports to the Market Economies have apparently ceased, and Central Economy Countries have become a major importer from the west (see Table 3-4). Inferentially, this leads to the supposition that either Soviet reserves are not as great as once thought, or that production problems may exist.

As discussed earlier, the vast bulk of Soviet reserves are in the Nikopol and Tchiaturi deposits, both of which were well defined shortly after the turn of the century. Given the lenticular nature of these manganese deposits as well as the mass of geological data available concerning them, virtually no chance remains of expanding the size of the reserves associated with them. In view of the length of time these deposits have been worked, underground haulage distances are undoubtedly increasing, along with surface mining stripping ratios and attendant costs of production. More than likely, these factors are reducing the productivity of these mines with time. Other deposits have apparently yet to be proven economic, even in the non-profit Soviet economy, resulting in essentially no growth in Soviet manganese production.

If reserves are not as great as formerly stated ("70 percent of the world's manganese reserves"), and/or production and exports do not grow, a major source of manganese for the world's steel producing nations will have disappeared, leading to an even greater concentration in the industry.

3.2.2.3.3 Republic of South Africa (Figure 3-2)

South Africa claims to have the largest known reserves and resources of manganese in the world. However, there is considerable disagreement concerning the accuracy of reserve estimates. Major elements of this debate will be discussed later in this section.

TABLE 3-4

MANGANESE ORE TRADE
COMECON-MARKET ECONOMY COUNTRIES

	USSR EXPORTS					COMECON		
	USSR Production	To COMECON		To Market Economies		Imports[1] from Market Economies		Net-COMECON
Year	All Grades (000)m.t. Ore	Total ore	Contained manganese	Total Ore	Contained manganese at 35% (E)	Total ore	Contained manganese	Exports (Imports) cont'd Mn
1972	7,000	957	?	310	108	116	59	49
1973	8,245	1,069	?	244	85	109	55	30
1974	8,155	1,133	?	301	105	121	61	44
1975	8,459	1,174	?	97	34	126	64	(30)
1976	8,636	1,179	?	73	25	248	125	(100)
1977	8,595	1,150	?	202	71	253	127	(156)
1978	8,700			100(E)	35	340	172	(137)
1979				50(E)	17	350(E)	180	(163) (E)

1) This does not include occasional purchases from India.
Represents deliveries from Gabon and Brazil.
E) Estimate.

Source: Industrial Minerals, October, 1979.

REPUBLIC OF SOUTH AFRICA

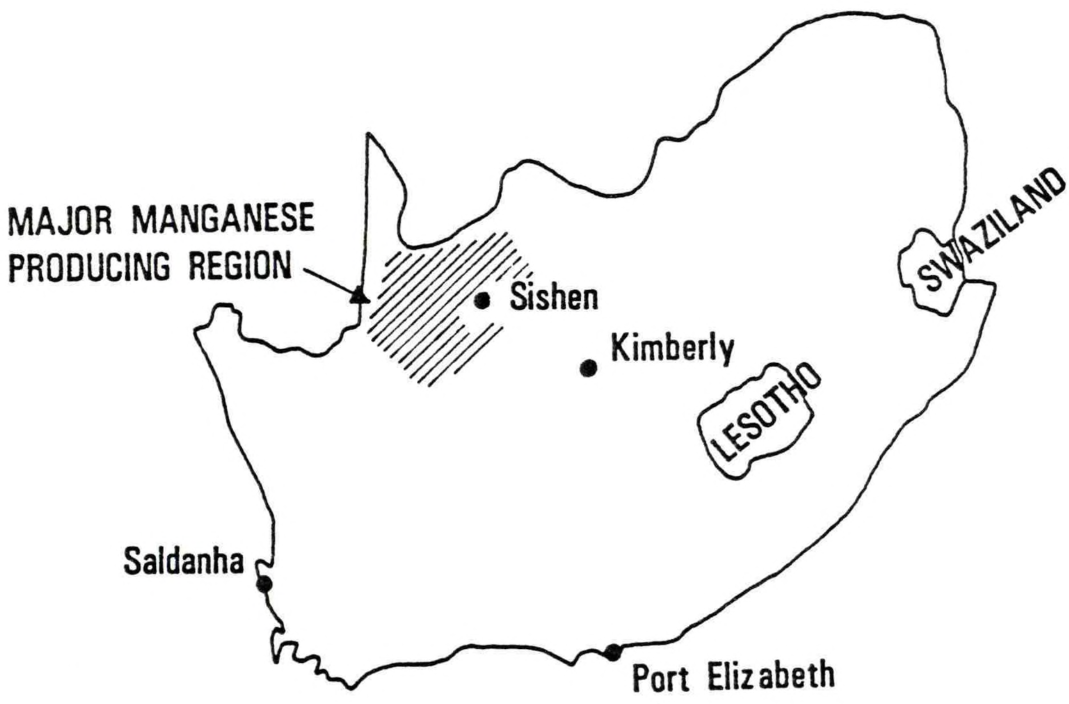


Figure 3-2

South African manganese deposits are fairly well distributed in the Cape Province and the Transvaal. The only deposits of current economic significance, however, are those in the Postmasburg and Kalahari Fields of the northern Cape Province, and those in the western Transvaal. The deposits in the Transvaal are primarily of local importance in the production of chemical-grade manganese ore for domestic use, principally in the extraction of uranium. Reserves are of nominal significance and are not embroiled in the debate surrounding the vast majority of South African reserves.

The manganese deposits of the northern Cape Province occur in the rocks of the Griquatown and Gamagara formations. The rocks are folded into a series of gently rolling faulted synclines. The beds dip gradually to the west, passing under the recent Kalahari sands. In general, the orebodies are horizontal, or have a dip of less than 15 degrees, although more intense folding has taken place in the northern Kalahari sector (e.g., the Black Rock mine). All the Cape Province deposits are thought to be secondary chemical replacements and enrichments associated with long periods of weathering and erosion.

The Postmasburg Field. The southern, or Postmasburg Field, extends northward from the west of the town of Postmasburg for about 40 miles to Sishen. The field is divided into an Eastern Belt, running along the Klipfontein Hills, and the Western Belt which lies roughly along the Gamagara Hills.

The Eastern Belt deposits are characterized by their intricate shapes, inconsistent sizes and by their occurrence in siliceous breccia. Braunite is the most common ore mineral. Due to a drop in grade, adverse mining conditions, and the gradual drop in demand for siliceous manganese ore, production in this area has been curtailed in favor of the Western Belt.

The Western Belt is characterized by ores that grade from manganiferous iron ores through ferruginous ore to high-grade manganese ore. In the north, the Western Belt terminates in an area of high-grade hematite iron ores south of Sishen.

Most of the manganese mining in the Postmasburg Field is by open-pit methods, although an increasing quantity of ore is produced in shallow underground operations. Although specific reserve estimates for the field are difficult to obtain, in 1976 the South African Geological Survey estimated ore reserves at some 6 million tons grading 20 to 30 percent manganese. (Precise figures are made difficult by the irregular size and shape of the orebodies.)

The Kalahari Field. The northern, or Kalahari Field was the last major manganese field found in South Africa. The only outcrop was that at Black Rock - a banded iron formation on an isolated ridge rising some 60 feet above the surrounding semidesert plains, about 70 miles north of the last outcrops of the Gamagara Hills. Systematic drilling followed extensive geophysical prospecting (mainly gravitational and magnetic surveys) which helped detail the surrounding geology; by virtue of their association with the iron formation, the deposits at Hotazel and Mamatwan were reportedly delineated by magnetometer surveys.

The Kalahari ore deposits are found primarily in two parallel beds, dipping gently to the west. The upper bed tends to be of slightly higher grade (+44 percent) manganese than the lower bed (±38 percent). As would be expected, the near-surface deposits apparently have been enriched to a greater degree than the deeper bed, although increased manganese concentration is also noted along zones of structural deformation in the lower bed. The ores tend to be low in phosphorus and silica, and rather high in lime and magnesia, to the extent that the ore is nearly self-fluxing.

Reserve Estimates. Estimates of South African manganese reserves, as mentioned previously, vary considerably. Most of that variance is in reserve estimates for the northern Cape Province, where the bulk of reserves are located.

The following are a few of the more widely publicized estimates that have surfaced in recent years:

- The USBM, in a tabulation dated 26 March, 1979, estimated world measured, indicated, and inferred reserves at 2.2 billion tons of ore. As the USBM admittedly has no specifically defined "measured ore reserve" figure, it has accepted U.S. Steel's estimate of 722 million tons as being "a reasonable estimate with the possibility that actual measured reserves may be larger."

- U.S. Steel Corporation, in testimony by Mr. Robert L'Esperance, Director of Corporate Explorations and Investigations, submitted to the United States Congressional Subcommittee on Oceanography, on 22 April, 1977, estimates measured, indicated, and inferred reserves at 1.064 billion tons of ore. Measured reserves alone are given as above, at 722 million tons.

- British Steel Corporation, in a 1978 internal memorandum quoted by Mr. Franz R. Dykstra, in a statement before the Subcommittee on Energy and Development of the Senate Committee on Energy and Natural Resources, 29 March, 1979, estimated South African reserves to be 777 million tons of ore, with an additional 700 million tons of resources.

And finally, the South African Minerals Bureau (SAMB) estimates of proven, inferred, and estimated reserves have been given as:

- 343 million tons at +44 percent manganese.

  7.9 billion tons at 30 to 40 percent manganese.

  5.1 billion tons at 20 to 30 percent manganese.

The SAMB provides the following definitions for their manganese reserve estimates:

> "Proven reserves include that ore within existing mine areas which can be determined with a high degree of reliability by extrapolation from present working faces and by advanced close drilling, varying from 250 foot to 4000 foot centers. It should also be noted that exploration to determine 'proven reserves' has in most cases been stopped when enough ore has been proved on the property to maintain production for the foreseeable future....
>
> "Inferred reserves include ore beyond the limits of the proven reserves which can be determined with reasonable reliability, taking cognizance of the geology of the deposits and confirmed by drilling at wider spacing.
>
> "Estimated reserves are for those mines for which figures are not available because of company policy, but where tonnage and grades have been estimated by interpolation from the known tonnages and grades on adjoining properties. The values can be accepted with a high degree of reliability.
>
> "All figures for proven, inferred and estimated reserves include ore that can be mined economically at present prices and with present mining technology."

The total of all grades is given at 13.37 billion tons of ore, ranging from 20 to greater than 44 percent manganese, and having a weighted average of 31.33 percent.

It is obviously very difficult to reconcile differences such as these, given that estimates differ by more than an order of magnitude (from 777 million tons to 13.3 billion tons of ore). As is highlighted above, a large part of this variance may be due simply to differences in the definitions of what constitutes a manganese reserve. For example, South Africa includes some 5 billion tons of manganese material grading 20 to 30 percent manganese in its reserves (because at least one South African firm does sell material of this grade as ore to some users), whereas the USBM and U.S. Steel do not. Another example illustrating this difference in terminology is highlighted in L'Esperance's testimony when he stated, "Indicated ore requires some exploration work. Taking 2500-foot hole spacing as a maximum reasonable on the underground ore..." Just above, the South African definition of 'proved' reserves includes that ore "proven" with drill spacings up to 4000 feet apart.

Given the large differences in estimates of South African manganese reserves, it is possible to conclude only that South African manganese reserves are extensive, and that a precise and agreeable estimate is not likely to be forthcoming in the near-future.

Mine Production. Mine production of manganese in South Africa was approximately 4.6 million tons in 1978. Among the larger mines currently operating are Wessels, Mamatwan, and the just-opened Middelplaats, all of which are in the Kalahari Field. Other producers in the same field include the Adams (nearing depletion), Black Rock, Devon, and Gloria mines. Newer mines and future mining operations are likely to be carried out underground as the deposit-containing formations are followed down-dip. With the opening of the Middelplaats mine this past summer (having a planned annual production of 200,000 tons in 1979, 500,000 tons in 1980, 800,000 tons in 1981, 1 million tons in 1982, and 1.1 million tons per year thereafter), no further additions to South African manganese mining capacity have been announced.

All manganese ore from the Kalahari Field is presently exported via Port Elizabeth, some 720 miles south of the Cape fields. The ore is shipped by rail through Hotazel and Kimberly, a trip which takes 36 to 48 hours. Originally the rail line to Port Elizabeth as well as the bulk-loading equipment at that site, were designed to handle both manganese and iron ore exports. Reportedly, however, since the inauguration of the Sishen-Saldanha rail link, all iron ore is now routed to the loading terminal at Saldanha Bay, thereby freeing the Port Elizabeth facility to handle only manganese ore exports. In addition, the South African government expects the facilities at Saldanha Bay to be able to handle manganese exports within two to three years.

3.2.2.3.4 Gabon (Figure 3-3)

The Moanda deposit, located in the mountainous eastern portion of Gabon, near the Congo border, accounts for 100 percent of Gabon's manganese reserves and production. The deposit's discovery in 1951 and subsequent development were closely related to the Soviet cutoff of manganese supplies to the United States during the Korean War.

Moanda's measured ore reserves account for approximately 15 percent of the Market Economy Countries' total, ranking Gabon a close third behind South Africa and Australia in this category. Although the surrounding area has been actively explored by several companies and governmental agencies, no other economically significant deposits have been located. The deposit, which is a weathering product of an earlier sedimentary carbonate-facies manganese deposit, contains approximately 200 million tons of high-grade (+48 percent) secondary oxide ore occuring as an essentially flat capping of an erosional plateau.

Production from the Moanda mine (the largest single manganese mine in the world) places Gabon as the third largest manganese producer in the world, behind the Soviet Union and South Africa. For the last several years, production has been near the present mine capacity of 2.5 million tons per year (16 percent of Market Economies total), all of which is exported.

GABON

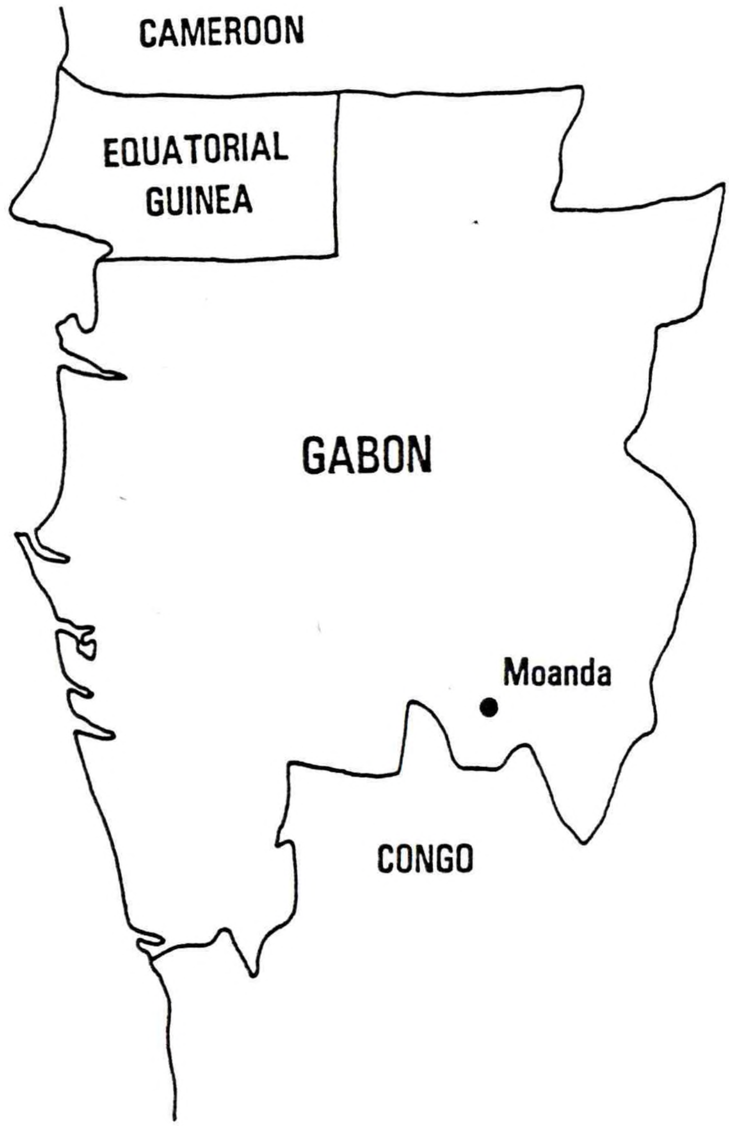


Figure 3-3

The Moanda mine is also the world's largest source of battery-grade manganese oxide. Ore reserves are all extractable by open pit. Once mined, the ore is carried approximately 40 miles by aerial tram into the Congo for transshipment. It had been hoped that the completion of the Trans-Gabon railway, and the associated port of Santa Clara (originally scheduled for 1981), would allow production from the Moanda mine to be doubled and transportation to become completely internal. Unfortunately, the latest indications are that the railway will not be completed until 1984-85 at the earliest, and that the plan to build a port capable of handling 250,000 dwt vessels at Santa Clara has apparently been shelved.

At the present rate of production, the measured reserves of the Moanda deposit will be exhausted sometime during the period 2045-2065. If the Trans-Gabon railroad is completed by 1985 and Moanda's production increases to a planned capacity of 4 million tons per year, ore exhaustion is predicted for between 2023 and 2035.

3.2.2.3.5 Australia

Measured manganese ore reserves in Australia amount to approximately 5 percent of the world's total, and 10 percent of the Market Economy Countries' total. Except for a few small deposits, most of the reserves are located on Groote Eylandt, an island in the Gulf of Carpenteria. The island, approximately 80 square miles in size, is bisected by a ridge separating the western mineralized zone from a barren, eastern region. Measured ore reserve estimates vary from between 150 million to 209 million tons grading 25 to 40 percent manganese, the difference apparently being the value (or lack thereof) attributed to the low-grade manganiferous overburden.

Since the discovery of the Groote Eylandt deposit in 1960, the Australian mainland has been carefully surveyed for any possible extensions of the ore zone, without success.

Mine production at Groote Eylandt is from three open-pit operations. The ore is blended and concentrated on the island, then trucked twelve miles to the port of Milner Bay, which can accommodate ships up to 35,000 dwt. An estimated 1.3 million tons of ore was mined from Groote Eylandt in 1978, a drop of 24 percent from the record levels of a year earlier.

Australian manganese mines produce a little over 7 percent of the world's manganese, and about 12.3 percent of the Market Economies' total. No increases in production capacity have been announced, or are known to be under consideration. Given the known size of Groote Eylandt's well-defined reserves, deposit exhaustion is calculated to occur sometime between 2007 and 2035, depending on production growth rates.

3.2.2.3.6 India

India's measured manganese ore reserves are seemingly quite small--less than 1 per cent of the Market Economy Countries' reserves--but total reserves (measured plus indicated plus inferred) are on the order of 5 per cent. The principle deposits are in the Balaghat, Bhandara, Chindwara, and Nagpur districts in the Central Provinces, and also in Bihar, Madras and Bombay.

The deposits of the Central Provinces are thought to be the result of an oxidation of crystalline "gondite" schists containing spessartite (a manganiferous garnet) and rhodonite. Most of the silica and alumina contained in the spessartite are believed to have been removed during weathering, resulting in the development of residual manganese deposits of chiefly psilomelane, pyrolusite, and wad. The deposits apparently are metamorphosed sediments that later underwent surface alteration to form residual enrichment deposits. As a result, a series of high-grade lenses within low-grade protore environments was formed.

Until the country gained independence, India supplied about one-third of United States requirements for metallurgical-grade ore. Since that time, its importance as a supplier of manganese has shrunk. This is largely due to two factors: 1) a deterioration in mine efficiency, and 2) an increasing emphasis on domestic steel production. Reportedly, since the end of the British colonial administration, the deliberate trend in the Indian manganese ore industry has been to intensify hand labor use (in an effort to preserve jobs) and defer the development of (new) reserves. Mines are also said to be deteriorating due to a lack of expert supervision, and continued high-grading.

The increasing emphasis on domestic steel production has required the diversion of formerly exportable manganese to domestic steel production. This increased internal demand, plus a tightening of supply, has led the government to ban all exports of ore grading over 46 percent manganese, and to place ceilings on exports of other grades.

An official Indian Government study has estimated that measured manganese ore reserves will last for 26 years if conserved for domestic use, but would be depleted in 12 years if export controls had not been emplaced.

3.2.2.3.7 Brazil (Figure 3-4)

Brazilian measured manganese ore reserves are estimated to contain between 37 million tons (by U.S. Steel) and 61 million tons (USBM - Brazilian Government official figures), or between 3 and 5 per cent of measured reserves in the Market Economy Countries. Reserves are widely scattered, with the largest concentrations in the states of Amapa, Minas Gerais, and Mato Grasso.

Located about 150 miles from the mouth of the Amazon River, the Serra do Navio deposit in Amapa was discovered in 1941. Reserves are estimated at 24 million tons of ore running 35 to 45 percent manganese. The Serra do Navio mine is the source of 90 percent of Brazil's manganese exports, but may be worked out within 12 to 15 years.

BRAZIL

Figure 3-4

More than 80 scattered deposits have been located in the state of Minas Gerais (near the eastern coast, north of Rio de Janeiro). Excluding the largest deposit, the Morro de Mina, they contain only 10 million tons of ore grading 18 to 51 percent manganese. The Morro de Mina deposit has proven reserves estimated at 6 million tons of carbonate ore averaging 31.8 percent manganese. Six lens-shaped orebodies are known, and are thought to be the weathered residual concentrations of ancient crystalline rock. Production from the Morro de Mina mine is consumed domestically at a ferroalloy plant in Minas Gerais.

Manganese resources in the state of Mato Grosso are largely contained in the deposits at Morro du Urucum. These deposits are found in a mesa about 15 miles south of Corumba, near the Bolivian border. The ore is found in two horizontal beds within the Band' Alta Formation, which consist of a series of banded hematite beds associated with subordinate beds of manganese oxide, siltstone, sandstone and jasper. At or near the base of the formation is a persistent bed of cryptomelane (a manganese oxide related to psilomelane) which varies in thickness from 8 inches to nearly 20 feet. One hundred to 150 feet higher is a second, less widespread bed of the same material ranging up to 7.5 feet in thickness. Reserve estimates have been increased over the last 35 years, but only in the indicated and inferred categories. Measured ore reserves are thought to be on the order of 4.5 million tons; indicated and inferred reserve estimates have roughly doubled to 33 and 28 million tons, respectively, since originally estimated (Geo. Survey Bulletin 946-A, 1945). Ore grade is high, 46 percent manganese, as is iron content (11 to 12 percent) and alkali content (3.5 to 4.5 percent).

Production from the Urucum deposit has been hampered largely by transportation problems, and for several years was suspended. When the mine's reopening was announced in 1978, it was planned that 100,000 tons of ore out of the 150,000 tons produced per year would be available for export. What ore has been mined has been shipped 1000 miles down the Paraguay River to Rosario (Argentina). Any attempt to increase production must first tackle this transportation bottleneck.

Other significant manganese resources are located in the states of Para and Bahia. Development work has been minimal, and given the inaccessibility of the deposits, significant production is unlikely to occur in this century.

The Brazilian government has repeatedly expressed concern over the country's declining high-grade ore reserves. Internal consumption is growing, and is expected to climb more rapidly in the future as Brazil expands its steel production capacity. Export controls are considered likely during the 1980's.

3.2.2.3.8 Ghana (formerly the Gold Coast)

Many manganese occurrences have been noted in Ghana, but the only one of significance is the Nsuta deposit, located at Dagwin, 34 miles from the port of Sekondi, on the Sekondi-Kumasi rail-line. Although historically of great importance, Nsuta's high-grade oxide ores are almost exhausted. Reserves are estimated at about 1 million tons of oxide ore, and production is by open-pit. The deposit is of sedimentary origin and is the result of the weathering of a manganese carbonate. An estimated 20 to 30 million tons of this lower-grade (about 33 percent manganese) carbonate protore remains. Plans to establish a nodulizing plant to make use of the manganese carbonates by 1980 have been stalled due to Ghana's balance of payments difficulties.

3.2.2.3.9 Mexico

Manganese deposits occur quite widely throughout Mexico. Since 1942, the country has been a small but consistent producer of manganese, with ore coming mostly from small oxide deposits. Most of the country's reserves of 15 to 16 million tons of ore are located in the Molango deposit in Hidalgo, Mexico. The ore at Molango is a sedimentary carbonate grading 38.5 percent manganese. The beds are as much as 200 feet thick and extend along strike for about 30 miles. Open-pit reserves are becoming depleted and mining operations are beginning to move underground. The ore is crushed, roasted, and then pelletized. The final product is a high lime manganese oxide suitable for steel furnaces, and is primarily consumed internally. Ore reserves are thought to represent no more than 15 to 20 years supply at present production rates.

3.2.2.3.10 Zaire

The manganese ore reserves of Zaire are less than 1 percent of the Market Economy Countries' total. Reserves held by Zaire's state owned Societe Miniere de Kisenge amount to approximately 8 to 9 million tons grading 35 to 40 percent manganese. Mine production frequently has been restricted over the last few years due to disruptions in neighboring Angola which caused the shutdown of the Benguela Railway; other routes are apparently uneconomical to use. Unless dependability of the Benguela Railway improves, the future of the Kisenge deposit may be in jeopardy. Plans to upgrade the Kisenge ore have been shelved.

3.2.2.3.11 Morocco

Manganese deposits have been worked for many years in Morocco, but most were small and rapidly depleted. The only remaining deposit of note is Imini, in the southern region, 80 miles southeast of Marrakech, on the south flank of the Atlas Mountains.

Discovered in 1918, the deposits at Imini have been worked steadily since. The sedimentary oxide ores occur as three mineralized beds of relatively high grade (+47 percent) manganese. Mining has been by both underground and surface methods, with the former of greater importance. Adits from the mountain slopes were used to gain entrance to the deposits; room-and-pillar methods are used to produce manganese ore of various grades. Mine production is hauled by truck to Marrakech, then by rail to Casablanca.

At present, manganese reserves are reportedly nearing exhaustion, with between 1 and 3 million tons of ore remaining.

3.2.2.3.12 Upper Volta

The manganese deposit at Tambao in Upper Volta is repeatedly mentioned in the literature over the last ten years. The deposit is estimated to contain 14.6 million tons of 54 percent manganese oxide, and 14.3 million tons of 48 per cent manganese carbonate ore. Initial development plans were made by a five-nation, 14 company consortium to begin production in 1981 and to reach full production of 700,000 tpy by 1983. However, development work on the necessary rail and port facilities fell behind schedule and serious cost overruns occurred. Recent studies indicate that a disproportionately high percentage (55 percent) of the cost of ore delivered to port would be tied up in rail freight and capital charges for the new railway. In addition, not only has the Union Carbide Corporation announced its desire to withdraw from the consortium, but Japanese participants indicate a reluctance to contract for 60 per cent of the project's output for a period of 20 years. Consequently, the entire project is currently in doubt, and production cannot be expected before the late 1980's, at the earliest.

3.2.2.3.13 Cuba

Many small manganese deposits have been found in Cuba--the highest concentration being in Oriente Province in the foothills of the Sierra Maestra Range on the eastern end of the island. Although reserves are largely depleted, pockets of ore undoubtedly remain. However, the deposit tonnages are so small as to render them largely uneconomic.

3.2.2.3.14 Canada

Known deposits of high-grade manganese ores in Canada are small and/or largely exhausted. Only one relatively large deposit may be brought into production in the next thirty years, the Manuels Killigrews orebody in Newfoundland. This deposit, a sedimentary manganese carbonate, is estimated to contain 10 million tons of material grading 10 percent manganese. Previous attempts at exploitation (the last being in 1942) failed because of still unresolved metallurgical problems. Other known occurrences are mostly small bog manganese deposits (less than 100,000 tons each) located in Nova Scotia and New Brunswick.

3.2.2.4 Domestic Resources (Table 3-5)

Table 3-5

DOMESTIC MANGANESE RESOURCES

RESOURCE-Name, State	Amount of Ore (in million short tons)	Manganese Content		Iron Content
		Percent	Amount	(percent)
Aroostock, Maine				
Maple-Hovey Mountains	287	8.9	26	20.7
North and South Districts	90	6-12	~8	17
	377	~9	~34	20
Artillery Peak, Arizona	175	3.9	7	3
Batesville, Arkansas	196	4	8	1-3
Chamberlain, South Dakota	2,240	~0.6	13	~1
Cuyuna Range, Minnesota	272	8	22	32
San Juan Mountains, Colorado	48	8	4	low

Source: NMAB-323

3.2.2.4.1 Introduction

Domestic manganese occurences have been repeatedly investigated in the years since World War II. Major studies include:

- Resources for Freedom, V. II, pp. 151-155. A Report to the President by the President's Materials Policy Commission (Paley Report), June, 1952.

- o Manganese: 1950 Materials Survey, compiled for the Materials Office, National Security Resources Board, by the USBM and USGS, October, 1952.

- o Symposium Sobre Yacimientos de Manganeso, Twentieth International Geological Congress, v. 3, America, Mexico, 1956.

- o Low-Grade and Nonconventional Sources of Manganese, David B. Brooks, 1966.

- o Manganese Recovery Technology, National Materials Advisory Board Panel on Manganese Recovery Technology, NMAB-323, 1976.

The essential conclusion of these as well as other studies is that there are no known deposits of manganese ore (+35 percent manganese) which can be exploited at current or even substantially higher prices. Although small quantities of ferruginous manganese ores (10 to 35 percent manganese) and moderate to large quantities of manganiferous iron ores (5 to 10 percent manganese) do exist, they are not substitutes for the more important manganese ores. Except as indicated below, they are not beneficiatable to higher grade products.

Under the influence of government subsidies and/or defense-related demands, some domestic production has been prompted. However, for most of this century, the United States has imported virtually all of its manganese ore. Sporadic production has occurred from the following deposits/locations:

- Artillery Peak, Arizona
- Batesville, Arkansas
- Leadville, Colorado
- Cartersville, Georgia
- Cuyuna Range, Minnesota
- Butte, Montana

- Philipsburg, Montana
- Pioche District, Nevada
- Three Kids District, Nevada
- Franklin, New Jersey
- Crimora, Virginia

The deposits at Three Kids (Nevada) and those in Montana are now reportedly exhausted. Consequently, there is no current domestic production of manganese ore (+35 percent manganese). All domestic production is a coproduct of iron or zinc, or is a low-grade manganiferous material used in coloring brick.

In addition to the aforementioned deposits (which have at one time or another produced manganese), several unexploited domestic deposits are known to exist:

- Gaffney-Kings Mountain, North and South Carolina
- San Juan Mountains, Colorado
- Chamberlain, South Dakota
- Aroostock, Maine
- Boulder City, Nevada

3.2.2.4.2 Cuyuna Range, Minnesota

The Cuyuna District, in central Minnesota, extends southwest from Aitkin to Todd Counties, a distance of about 75 miles. Topographically, the Cuyuna District is not a "range," but a low, swampy plain covered with a mantle of glacial drift 25 to 300 feet thick, making the deposit amenable to open-pit mining. The manganiferous iron ores were probably formed by oxidation of the primarily iron-bearing formation. Both the enclosing rocks and the iron-bearing beds have been strongly folded and faulted, and dip steeply.

The primary ores of the Cuyuna Range are of two types: 1) green carbonate slate, and 2) cherty iron formation. The green carbonate slate (an approximately 50-foot bed) is stratigraphically located some 100 to 150 feet beneath the cherty iron formation. It contains 3 to 8 percent manganese, 20 to 30 percent iron, and 25 to 40 percent silica. Upon oxidation, the slate has yielded the so-called "brown ores" which contain about 9 percent manganese, 44 percent iron, 6 percent silica, and 0.27 percent phosphorous. The cherty iron formation is apparently much thicker, and often contains more manganese and less iron. Oxidized zones yield the so-called "black ores" which contain about 15 percent manganese, 37 percent iron, 15 percent silica, and 0.09 percent phosphorous.

The total manganese resources of the Cuyuna Range are estimated to be 272 million tons of ore, averaging 8 percent manganese and 32 percent iron. According to NMAB-323, the manganese deposits of the Cuyuna Range". . . represent the most promising domestic land resource, are sufficient to fill the nation's manganese requirements for more than a decade, and have good possiblities for enlargement." In addition, the range is part of an area in which iron ores have been mined for three-quarters of a century, thus a large infrastructure base is already in place.

In the late 1950's, the government sponsored a semicommercial pilot operation run by the Manganese Chemicals Corporation (a subsidiary of Pickands Mather and Company) in Riverton, Minnesota, for treatment of Cuyuna Range manganiferous ores. Using an ammonium carbamate leach process, the plant could process 200 tons of ore per day (equivalent to an annual capacity of about 7,000 tons of manganese). Early in 1962 the plant closed, apparently for economic and technical reasons. Difficulties were encountered due to the intimate interlocking of manganese minerals with gangue minerals. In addition, ammonia recovery was apparently poor, resulting in high operating costs. ($0.35-.50/lb Mn, in the 1950's).

In discussions with various government and industry experts, it is apparent that the metallurgical problems encountered in the early test plant remain unsolved. Reports that ammonia requirements would be staggering in a full-size operation indicate that an appropriate extraction technique remains to be found.

3.2.2.4.3 Artillery Mountains (Peak), Arizona

The manganese deposits of Mohave County in west central Arizona are located in an approximately 25-square-mile area on the west side of the Bill Williams River, near the town of Alamo. Two basic types of manganiferous deposits have been described in the area:

- o Stratified oxide deposits in the clays, sandstones, conglomerates, and other sediments of the Artillery, Chapin Wash, and Sand Trap Formations.

- o Fracture fillings.

Of the two, only the stratified oxide deposits of the Chapin Wash Formation are thought to be of any potential significance.

The Chapin Wash Formation has two essentially parallel manganiferous zones, which have a stratigraphic separation of approximately 900 feet. The Formation underlies most of the valley between the Artillery and Rawhide Mountains in the southern part of Mojave County. The upper bed outcrops along the Artillery Mountains on the northwest side of the valley, and is found at or near the top of the Chapin Wash Formation. The lower zone of mineralization outcrops mainly along the Rawhide Mountains on the southwest side of the valley. Although it can be traced along strike for nearly four miles, and is as much as 350 feet thick, mineralization consists only of widely separated manganiferous lenses scattered among thicker and more frequently barren zones. The entire formation is interbedded with fanglomerates, clays, and tuffs. In addition to interfingering with barren beds, the deposits grade laterally into barren rock. The manganese content is said to range from zero to 30 percent, only small quantities containing more than 20 percent manganese. The largest part of the deposits contain less than 5 percent manganese, along with 3 percent iron, 0.08 percent phosphorous, 1.1. percent barium, and trace amounts of copper, lead, and zinc.

Total manganese resources are estimated at 175 million tons grading 4 percent manganese and 1 to 3 percent iron. The USBM has conducted extensive pilot-plant work on Artillery Peak ore in the past. Two flotation methods, fatty-acid and oil emulsion, have been successfully demonstrated (from a technical standpoint) on select high grade (10.6 percent manganese) samples. The former yielded a concentrate assaying 36 percent manganese and a middling assaying 15 percent (with 85 percent recovery), while the latter yielded a concentrate assaying 36 percent manganese (with no middling product) at 80 percent recovery.

Nonetheless, the Artillery Mountain deposits are not regarded as promising prospects for development. The scattered nature of the deposits, remote location, and low-grade will tend to prohibit or at least severely restrict the development of the district, barring a major national emergency.

3.2.2.4.4 Aroostock, Maine

The manganese formations of Aroostock County in northeastern Maine are bedded deposits enclosed in steeply dipping slates. The manganiferous zones are typically a few feet to 150 feet in thickness and can be traced for a mile or more along strike. Both the manganiferous beds and the wall rocks are intricately folded and faulted, leading to a high degree of variability of manganese grades. The area is overlain by a 3 to 50 foot layer of glacial drift.

The known deposits are grouped in three apparently distinct districts, the largest of which is the Central District, also known as the Maple and Hovey Mountain area, some 25 miles south of Presque Isle. The deposit is a canoe-shaped syncline, with several folds along the trough. Three mineralized zones have been noted. The upper zone is about 40 feet thick, and consists of red and purple slates interbedded with some greenish-slate and banded hematite. Manganese content averages 5 to 6 percent, and iron 10 to 15 percent. A transition zone averaging 6 to 7 percent manganese and 20 percent iron separates the upper zone from a 40-foot thick banded hematite zone averaging 12 percent manganese and 26 percent iron. Resources in the Central District are estimated at 287 million tons of ore averaging 8.9 percent manganese and 20.7 percent iron, making this the largest single deposit in the United States. The North District (located to the west of the towns of Caribou and Presque Isle) contains some twenty deposits, while the South District (located around the town of Houlton) has one major deposit at Littleton Ridge. Combined resources in the northern and southern districts were estimated in 1976 at 90 million tons of ore grading 6 to 12 percent manganese and 17 percent iron.

The Aroostock deposits were looked at with some intensity during the period immediately following World War II, primarily by the state of Maine. Manganese content is variable throughout, due to stratigraphic folding, complicating any attempt at mining. The ores are also characterized by very fine grain size and intimate interlocking of minerals, leading various researchers to conclude that physical beneficiation methods cannot be successfully applied. In addition, because the Aroostock ores occur partly as refractory silicate minerals, they are not readily amenable to hydrometallurgical extraction methods. Pyrometallurgical methods probably would be too energy intensive to be economically practical (Dykstra, 1979).

3.2.2.4.5 Batesville, Arkansas

The Batesville manganese district, in north-central Arkansas, is near the town of Batesville. Manganese is found along an east-west belt, 4 to 8 miles wide and 24 miles long. Manganese-bearing bodies are scattered at irregular intervals throughout the belt. Some deposits have been secondarily enriched. Manganese "buttons" occur in the Cason shale and its residual clay, and as masses of manganese oxides in residual clays occupying solution depressions in the weathered Fernvale (and other) Limestones. The clays attain as much as 80 feet in thickness, and the ores range from 25 to 50 percent manganese.

Present resources are estimated at 196 million tons grading 4 percent manganese and 1 to 3 percent iron. The ore is a combination of oxides and carbonates. Intermittent ore production occurred from 1849 to the 1950's. More recently, physical beneficiation tests have been made on samples somewhat higher in grade than average, and although several techniques yielded concentrates containing 40 percent manganese or more, recoveries were poor, due to the fine, intimate mixture of the manganese and gangue minerals. The potential for future large-scale production of manganese appears very unlikely.

3.2.2.4.6 Chamberlain, South Dakota

The Pierre Shale is an essentially flat-lying horizon of alternating bentonite, bentonitic shale, and manganese-iron carbonate concretions. Manganese occurs in concretionary nodules (average size is 1 to 8 inches thick, by 1 to 24 inches in diameter) scattered throughout the shale formation which is 12 to 86 feet in thickness. Outcrops of the manganese-bearing portion of the shale are exposed along the Missouri River (and its tributaries), south of Pierre. The nodules are distinct from the shale at the weathered surface, but tend to grade into the shale as depth increases below the surface. The manganese in the concretions appears chiefly as a complex manganese-iron-magnesium carbonate; other constituents include geothite, bentonite, calcium carbonate, and some form of silica. Some concretions have shells of manganese oxide.

The manganese content of the nodules tends to increase with decreases in the iron content. Assay values are clustered around two points: 14.5 percent manganese - 12.6 percent iron, and 18.7 percent manganese - 3.2 percent iron. Overall, the average manganese content of the nodules is 15.51 percent, with iron at 9 percent.

Outcrops have been noted along some 523 miles of river banks, with an average (weighted) width of 190 feet. By stripping on a 1/2:1 ratio (overburden:ore), the USBM estimates that this width can be extended to 365 feet. The area is calculated to contain over 2 billion tons of shale (dry) holding 77 million tons of nodules, yielding some 12 million tons of contained manganese. Each ton of shale thus contains about 77 pounds of nodules, or 12 pounds of manganese, meaning that the entire formation has an average grade of only 0.6 percent manganese.

It is apparent, therefore, that unless the nodules can be segregated from the bulk of shale, the grade of the deposit as a whole is really submarginal. Unfortunately, attempts to separate out the nodules have been notably unsuccessful.

3.2.2.4.7 Other Deposits

Several of other deposits deserve description, but only in an abbreviated form:

- San Juan Mountains, Colorado. Many large, steeply-dipping veins outcrop in the San Juan Mountains of southwestern Colorado at elevations of 11,000 to 13,000 feet. There is a known vertical extent of at least 2,400 feet in the Sunnyside mine. Rhodonite and rhodochrosite are the principal manganese minerals and are associated closely with gold-silver and lead-zinc-copper sulfide mineralization. Ores have been mined for base metals, but manganese recovery has been only on an experimental basis. NMAB-323 concluded that although an estimated 48 million tons grading 8 percent manganese exist, the deposits should not be considered a promising resource because of "the relatively high cost of underground vein mining at high elevations, the refactory nature of rhodonite, and the long distance to markets."

- Leadville, Colorado. The manganese deposits at Leadville are contained in a 900-foot range of the 1,550 foot thick Leadville Limestone. The manganese minerals form a roughly continuous zone surrounding the lead and zinc orebodies. Manganese and iron oxides run 40 to 55 percent (combined), along with 6 to 20 percent silica, less than 0.1 percent phosphorous, and traces of silver and zinc. Resources in the Leadville region are small, on the order of a million tons of ore.

- Franklin, New Jersey. Manganese occurs uniquely here as a constituent of the mineral franklinite $(FeZnMn)_2O_3$, which contains 10 to 17 percent manganese. Upon smelting, the franklinite concentrate leaves a residue similar in composition to an iron-manganese ore, grading 12 to 40 percent manganese and 40 percent iron. This material is used to make spiegeleisen (a manganese-iron ferroalloy).

- Boulder City, Nevada. Bedded manganese oxides occur in a bed of silty gypsum, 60 to 65 feet thick. Located about 5 miles southeast of Boulder City, resources of this area were estimated in 1942 by the USGS and the USBM to be approximately 14 million tons of material grading 3 percent manganese, and 1 million tons at 7.5 percent manganese.

3.3 MANGANESE DEMAND

3.3.1 Introduction

Elemental manganese is a brittle, gray-white to silver metal. Rarely used in its pure elemental form, commerical manganese ores are generally classified on the basis of their manganese content and their end-use. Traditionally, they are categorized as follows:

- Manganese ores are those which contain 35 percent or more manganese and are used primarily in the production of ferroalloys essential to metallurgy.

- Ferruginous manganese ores are those containing 10 to 35 percent manganese and traditionally have been used in the production of spiegeleisen.

- Manganiferous iron ores contain 5 to 10 percent manganese and are consumed as a premium grade iron ore.

After a review of the major uses of manganese, recycling and substitution are briefly examined and, finally, future consumption is forecast.

3.3.2 Uses

Manganese ores have three applications:

- Metallurgical.
- Battery/Chemical.
- Other uses.

3.3.2.1 Metallurgical

The principal use (95 percent) of manganese is in metallurgy. Small quantities of manganese are essential to the production of steel and iron, and are also commonly used in the production of aluminum and magnesium alloys.

3.3.2.1.1 Steel

In the United States, each ton of steel produced typically requires 11 to 15 pounds of manganese ore, excluding any manganese fed directly into the iron blast furnace or contained in steel scrap added to the steel furnace. This quantity varies considerably, depending on the type of steel being produced, ranging from less than 3 pounds per ton for silicon sheet steels to more than 200 pounds per ton for manganese (Hadfield) steels (Table 3-6).

Table 3-6

TYPICAL MANGANESE CONSUMPTION IN STEELMAKING
(pounds of manganese per short ton of raw steel)

	Type of Steel Produced				
Manganese	Carbon	Low Alloy	Full Alloy	Silicon Sheet	Stainless
Standard Ferromanganese	8.9	14.1	14.1	--	3.4
Silicomanganese	1.5	0.5	0.1	--	--
Refined Manganese*	1.6	3.5	--	1.2	15.0
Other**	--	--	--	1.6	7.0
Total	12.0	18.0	14.2	2.8	25.4

Source: NMAB-23, 1976

*Includes medium-carbon and low-carbon ferromanganese, electrolytic manganese metal, and various grades of nitrided manganese alloys.

**Generally ferromanganese silicon.

Typical manganese consumption in steelmaking also varies significantly by geographic region (Table 3-7). This variance is primarily a function of differences in raw materials and end-product mix.

Table 3-7

TYPICAL MANGANESE CONSUMPTION IN STEELMAKING IN VARIOUS GEOGRAPHIC AREAS
(pounds of manganese per short ton of raw steel)

Manganese	North America	Europe	Far East	Other
Standard Ferromanganese	11.9	11.9	7.5	12.5
Silicomanganese	1.8	1.8	4.7	2.7
Refined Manganese*	1.8	1.5	1.3	1.3
Total	15.5	15.5	13.5	16.5

Source: NMAB-323, 1976

*Includes medium-carbon and low-carbon ferromanganese, electrolytic manganese metal, and various grades of nitrided manganese alloys.

In steelmaking, manganese is used to:

- o Form manganese sulfides and thus reduce hot shortness due to sulfur (a condition which occurs when iron sulfides melt at rolling temperatures).

- o Prevent precipitation of carbides at grain boundaries, thereby preventing weak spots.
- o Increase strength by solid solution hardening.
- o Increase toughness at higher strength levels and low temperatures when substituted for carbon equivalent.
- o Increase hardenability.
- o Substitute for nickel in austenitic stainless steels.

To accomplish these tasks, manganese ore is used chiefly in the form of ferromanganese or silicomanganese (Table 3-8). Some electrolytic metal is also used, as is a very limited direct use of manganese ore in steel production.

Manganese use in a typical basic oxygen process steel blast furnace is shown in Figure 3-5.

Some producers of pig iron also add manganese in the form of scrap, slag or "sweetening ore" to their furnance burden. However, the principal use of manganese ore is for the production of various ferroalloys.

Ferroalloys are added to steel either in the furnace at the end of the steelmaking process, or after the metal has been tapped from the furnace into the ladle. Neither of these points in the process offer any appreciable opportunity for removal of any of the carbon, silicon, phosphorous, or other elements that may be present in the addition agent (ferroalloy) and are therefore added to the steel along with the manganese. For this reason, the different grades of ferromanganese listed in Table 8 have been developed so as to allow limits to be set and observed on certain objectionable elements. The 1950 Manganese Materials Survey compiled for the Materials Office, National Security Resources Board, by the USBM and USGS provided the following descriptions of commonly used ferroalloy addition agents:

Table 3-8

PRINCIPAL MANGANESE ALLOY SPECIFICATIONS
(Percent)

	Mn	C(max)	Si	P(max)	S(max)
Standard ferromanganese:					
Grade A	78.0-82.0	7.5	[1] 1.2	0.35	0.050
Grade B	76.0-78.0	7.5	[1] 1.2	0.35	0.050
Grade C	74.0-76.0	7.5	[1] 1.2	0.35	0.050
Medium-carbon ferromanganese:					
Grade A	80.0-85.0	1.5	[1] 1.0	0.30	0.020
Grade B	80.0-85.0	1.5	[1] 1.5	0.30	0.020
Low-carbon ferromanganese:					
Grade A	85.0-90.0	([2])	[1] 2.0	0.20	0.020
Grade B	80.0-85.0	0.75	[1] 5.0-7.0	0.30	0.020
Silicomanganese:					
Grade A	65.0-68.0	1.5	18.5-21.0	0.20	0.04
Grade B	65.0-68.0	2.0	16.0-18.5	0.20	0.04
Grade C	65.0-68.0	3.0	12.5-16.0	0.20	0.04
Ferromanganese-silicon	63.0-66.0	0.08	28.0-32.0	0.05	--
Spiegeleisen:					
Grade A	16.0-19.0	6.5	1.0- 3.0	0.080	0.050
Grade B	19.0-21.0	6.5	1.0- 3.0	0.080	0.050
Grade C	21.0-23.0	6.5	1.0- 3.0	0.080	0.050

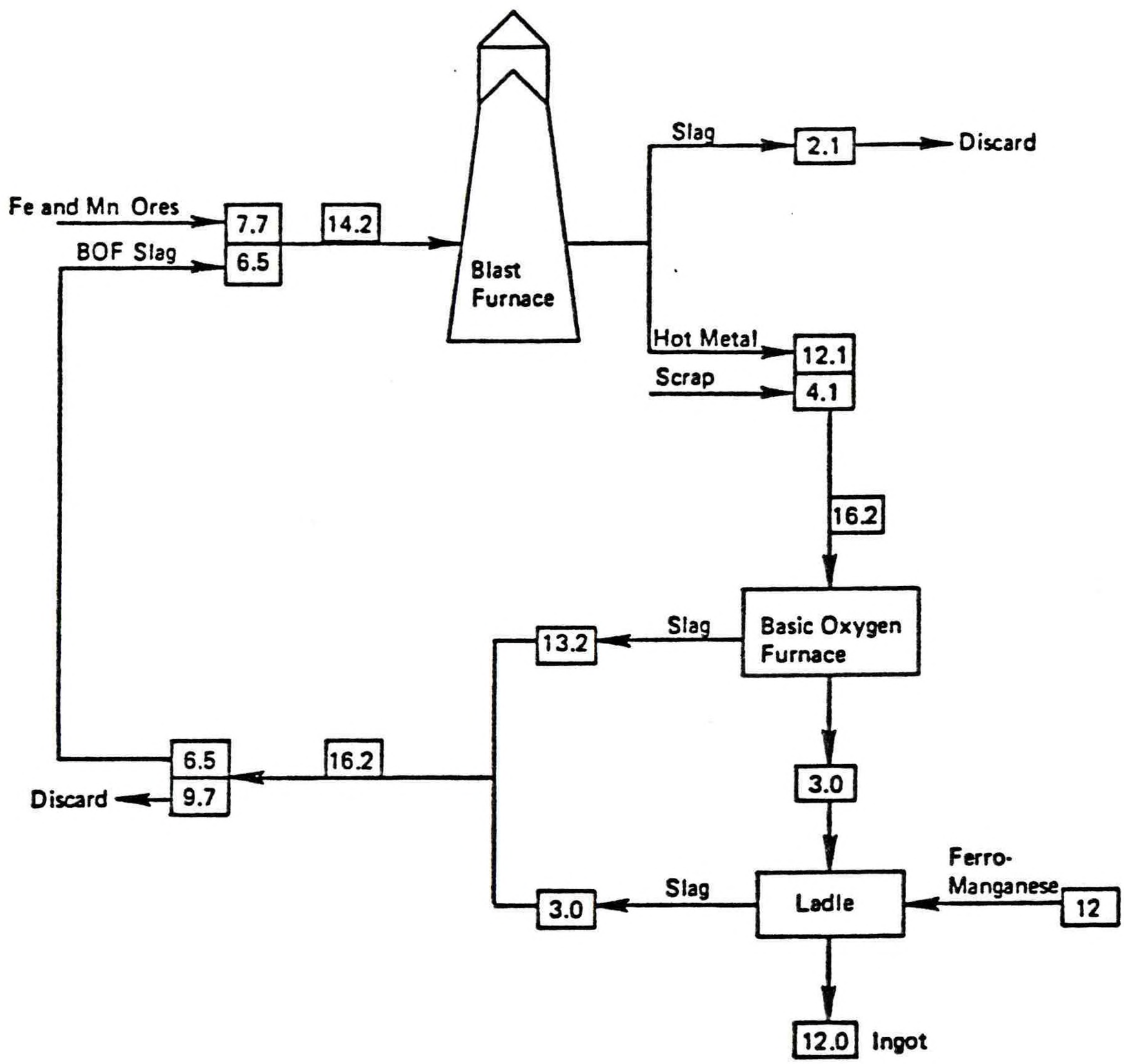
[1] Maximum.

[2] As specified. May have any of the following maximum percentages: 0.50, 0.30, 0.15, 0.10, and 0.070.

Source: USBM, Mineral Commodity Profile: Manganese - 1979

Figure 3-5

MANGANESE USE IN STEEL



Empirical flowsheet showing approximate manganese balance for production of steel by blast furnace - basic oxygen process (pounds of manganese per ton of raw steel).

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source: NMAB-323

Standard Ferromanganese - Standard (or high-carbon) ferromanganese may be considered the base alloy of which the other grades are variations. It is by far the most widely used type and usually is made up to run between 78 to 79 percent manganese, although slightly higher or lower amounts of manganese may be present without making the product unusable. The more closely the alloy can be held to a constant analysis, the easier it is for the steelmaker to use, since calculations can be standardized and error reduced to a minimum.

Medium-Carbon Ferromanganese - Medium carbon ferromanganese is generally used in making ordinary low-carbon steels and may be used in Hadfield steel (which contains 13 percent manganese) under certain conditions where a high percentage of returned scrap is being used.

Low-Carbon Ferromanganese - This grade is used for adding manganese to steels with such a low carbon content that neither high- nor medium-carbon alloy can be tolerated. The alloy is particularly useful in adding manganese to some chrome-nickel steels where the carbon content must be kept below 0.10 percent. Several grades of low-carbon ferromanganese are available in gradually increasing carbon content.

Silicomanganese - The low carbon:manganese ratio in this alloy makes it useful for adding manganese to low-carbon steels in which silicon may be tolerated. The presence of silicon in the alloy has the advantage of producing a somewhat cleaner steel as well as reducing the time required from the addition of alloy to the pouring of the ingot. These effects are due to silicon being a more powerful deoxidizer than manganese.

Spiegeleisen - Spiegeleisen or spiegel may be considered a high-manganese pig iron which is suitable for use in open-hearth or Bessemer practice. Spiegel is ordinarily used as a furnace addition and is therefore furnished in large chunks or pigs. Several grades are available, all of which have relatively low maximum limits for phosphorus and sulfur.

3.3.2.1.2 Aluminum

Manganese is alloyed with aluminum in relatively small amounts to add hardness and stiffness. The manganese does not enter into a true solution in aluminum, because of its higher melting point, but appears to be distributed along the grain boundaries of the metal. Manganese additions to aluminum are made as manganese-aluminum briquets, manganese-aluminum master alloys (95 percent aluminum, 5 percent manganese), or as electrolytic metal. Total manganese consumption for use in aluminum is very small, perhaps on the order of a few tens of thousands of tons of ore per year.

3.3.2.1.3 Magnesium

Manganese is sometimes alloyed with magnesium in small amounts to add hardness, stiffness and a certain amount of corrosion resistance. Since the melting point of manganese is so far above that of magnesium, it must be added as a flux, typically manganese chloride. Different flux compositions are available to allow for different alloys to be created. The total amount of manganese consumed by the magnesium industry is small.

3.3.2.2 Battery/Chemical

3.3.2.2.1 Battery

The common dry-cell battery (Figure 3-6) requires a special high grade of manganese dioxide (MnO_2) ore to function properly. Typically, battery specifications require manganese dioxide contents of 75 to 85 percent. The manganese, along with calcined petroleum coke, graphite and ammonium and zinc chlorides, form the depolarizing mix which surround the carbon element which acts as the positive pole. In operation, the manganese dioxide functions as a depolarizer by neutralizing the hydrogen film which otherwise would build up around the carbon electrode. Oxygen from the manganese dioxide

THE COMMON DRY CELL BATTERY

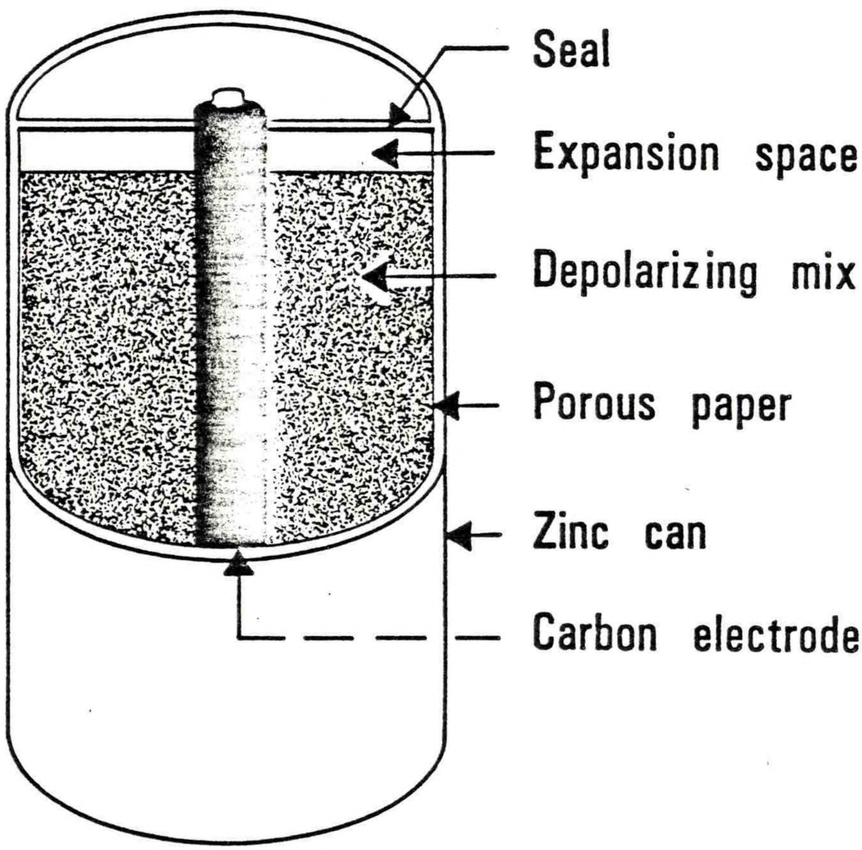


Figure 3-6

combines with the hydrogen almost instantly, permitting the continuous flow of current from the battery.

Natural battery grade ore, synthetic manganese dioxide, or a blend of both are generally used for battery-grade ores. Chemically, the ore should have a high oxygen availability and must not contain any soluble forms of copper, nickel, cobalt, or arsenic, as these impurities are electronegative to zinc and will cause corrosion of the cell wall. Interestingly, the only effective method for determining which ores are "battery-grade" ores is an empirical test wherein the ores are manufactured into a test cell, stored, and analyzed at regular intervals over several years.

Total consumption for battery usage is very small, relative to metallurgical demands for manganese.

3.3.2.2.2 Chemical

Manganese ores are used in a number of chemical processes. Although battery-grade ores are generally suitable for these purposes, their higher cost usually precludes use. Major chemical compounds include:

- Hydroquine for use in photographic developers and plastics.
- Potassium permangonate used as a water purifier.
- Manganese sulfate used as fertilizer supplement.
- Manganese chloride used as a brown textile dye, as a magnesium alloy flux, and as a part of an anti-knock additive to gasoline.

- Manganese oxide used as an additive to livestock and poultry feeds, and as a component of welding rods or fluxes.

- Manganese persulfate used as an oxidizing agent in the manufacture of organic compounds.

Total consumption for chemical uses is also very small relative to metallurgical demand.

3.3.2.2.3 Other

Manganese ores are also used in the glass and ceramic industries as both a coloring and a "decoloring" agent. (Manganese will neutralize the iron content common in most glass sands which would otherwise cause glass to carry a pale green color.)

Manganese dioxide also is used in the sealant in the base of incandescent light bulbs, and as a part of frits for bonding glass and porcelain to metal.

These uses, and others, are of relatively inconsequential importance to the total demand for manganese.

3.3.3 Recycling and Substitution

3.3.3.1 Recycling

In the process of making steel, slags are created which carry off the various undesirable impurities contained in the furnace charge. Table 3-9 illustrates the range of composition of basic oxygen furnance (BOF) slags.

Table 3-9

COMPOSITION OF BOF SLAGS

Slags Constitutent	Amount (%)
Fe	12-25
Mn	3-7
SiO_2	10-20
CaO	35-45
MgO	3-10
S	0.05-0.2
P	0.2-0.9

Source: NMAB-323, 1976

In current practice, because of its phosphorous content only 40 to 45 percent of the BOF slag produced is recycled into the blast furnace, primarily to reclaim the contained iron, although manganese and basicity values are also desirable. (Phosphorous in even minute amounts makes steel brittle at low temperatures, so every effort is made to exclude it throughout the production process.) If some slag was not constantly being discarded, phosphorous would build up excessively in the pig iron. As shown in Figure 3-7 the quantity of manganese in the discarded slag (9.7 pounds per ton of steel) approaches the quantity added to the steel in ferroalloy form (12 pounds per ton).

Reclaiming manganese from discarded steelmaking slags has been proposed for many years. As the steel industry relies less on open hearth furnaces (with a consequent increasing dependence on basic oxygen furnaces) the economic advantages of such reclamation grow worse due to the lower manganese

WORLD MANGANESE

Cumulative Demand versus Fixed Reserves

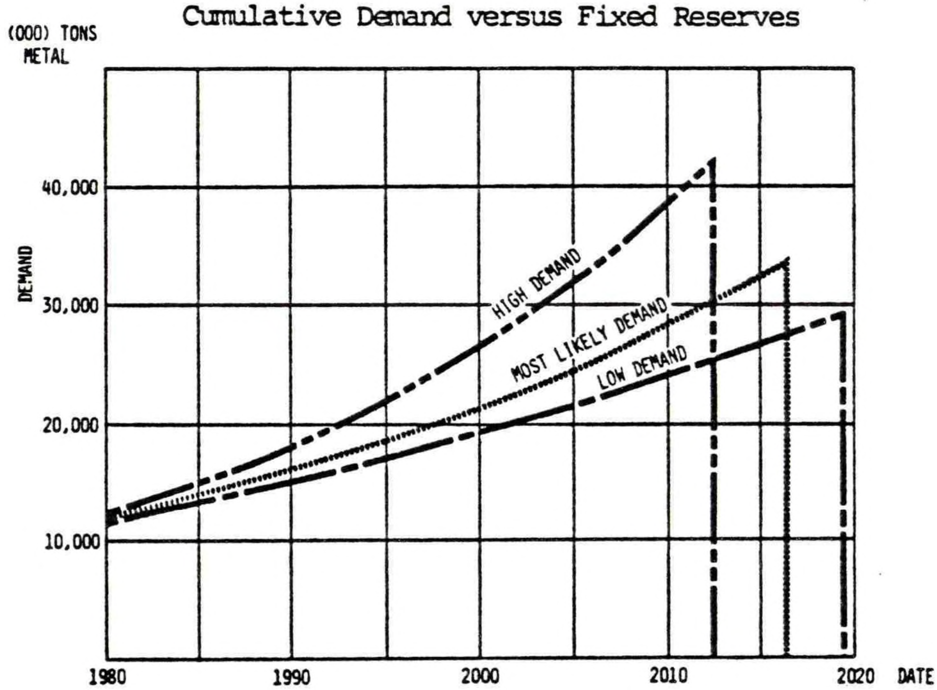


Figure 3-7

content of BOF slags and the intermingling of BOH and BOF slags. NMAB-323 therefore concludes correctly that steelmaking slags should no longer be considered a significant manganese resource.

3.3.3.2 Substitution

Two forms of substitution have been proposed for manganese usage in the steel industry:

- o Alternative materials containing no manganese.

- o Low-grade manganese substitutes.

3.3.3.2.1 Alternative Materials

Two National Materials Availability Board reports (317 and 323) concluded that the potential for conservation of manganese through substitution of other metals is limited. All of the known substitutes (e.g., titanium, zirconium, and rare earth metals) for desulferization have limited availabilites and are more costly than manganese. Alloy substitutes for manganese are known (e.g., chromium, molybdenum, nickel), but each has a more limited range of usefulness. In addition, the known alloy substitutes are more costly than manganese.

3.3.3.2.2 Low-Grade Manganese Substitutes

Much research has been aimed toward the identification of substitutes for standard ferromanganese utilizing other manganese addition agents. The most frequently proposed substitutes for standard ferromanganese are: spiegeleisen, silicomanganese and downgraded ferromanganese. A brief review of the literature highlights several problems with this approach, primarily in two areas. First is the increased waste in steel production in the form of scrap and slag. According to Brooks (1966), more manganese is lost in waste products from the production and consumption of spiegeleisen than in the use of standard ferromanganese. In addition, unwanted "tramp" elements are sometimes introduced into the steel, and the efficiency of manganese recovery reduced.

The second area of difficulty in using low-grade manganese substitutes is that their production requires the consumption of added amounts of other materials. For example (Brooks, 1966), "the coke consumption rate required to produce materials like spiegeleisen is more than double the rate (per pound of contained manganese) required to produce standard ferromanganese." Similarly, though silicomanganese can be made from rhodonite, the process takes substantially more electric energy than normal production from silica sand and manganese. Therefore, the net result from the substitution of any lower-grade material must be viewed carefully.

3.3.4 Summary - Uses and Substitution

As indicated in the foregoing discussion, manganese has a number of applications. Steelmaking, however, is by far the largest and most important consumer of manganese. Potential substitutes for this use exist, but manganese remains superior to each of them in terms of range of functions, cost, and current availability.

3.3.5 Projected Manganese Demand

3.3.5.1 Introduction

To establish the demand scenarios used in this chapter, Dames & Moore reviewed numerous demand forecasts for manganese. From these, three principal sources of data were selected:

United States and World:	United States Bureau of Mines, Mineral Commodity Profile: Manganese, July, 1979 and October, 1977
	United States Bureau of Mines, Mineral Trends and Forecasts, 1979
Central Economy Countries and Market Economy Countries:	Malenbaum, Wilfred, World Demand for Raw Materials in 1985 and 2000, 1978.

Industry trade journals and several other publicly available sources were also consulted.

3.3.5.2 Past Consumption

Knowledge of historical patterns and trends in manganese demand is helpful in projecting future consumption. Table 3-10 presents historical growth rates of manganese ore consumption for various geographic areas and blocks of nations during the period 1951-1955 to 1971-1975.

Table 3-10

HISTORIC DEMAND GROWTH RATES FOR MANGANESE ORE

(percentage)

REGION	AVERAGE ANNUAL GROWTH RATE
United States	0.44
Market Economies	4.63
Central Economies	3.67
World	4.20

Sources: Derived from tables in Malenbaum, 1978

A more complete listing is presented in Table 3-14 at the end of this section.

To correctly interpret these figures, actual consumption data must be examined to note the relative magnitude of demand in different regions. Demand in the same groups is shown below in Table 3-11, while a more complete tabulation is given in Table 3-15.

Table 3-11

AVERAGE WORLD CONSUMPTION OF MANGANESE ORE 1971-1975

(in 1000 short tons)

(5-year average)

REGION	AMOUNT
United States	2126
Market Economies	12626
Central Economies	9453
World	22079

Source: Malenbaum, Wilfred, World Demand for Raw Materials 1985 and 2000, 1978

3.3.5.3 Future Consumption Projections

Table 3-12 presents the three forecast demand scenarios for the United States and World, and the most probable level of demand by the Market and Central Economy Countries for the period 1980-2010. It should be noted that the growth rate values given are predicted annual averages and that, therefore, in any given year the actual growth rate may vary considerably from the overall projection.

3.3.5.4 Supply/Demand Intersections

Table 3-1 noted manganese reserves and resources of the major producing nations. United States, World, Market, and Central Economy Country groupings are shown in Table 3-12. Given the difficulty in accurately assessing both current and future mine production capacity (discussed in Section 3.2.2.2.),

Table 3-12

MANGANESE SUPPLY-DEMAND POSITION 1980-2010
(1000 short tons Mn content)

UNITED STATES:

Reserves: 0 — Total Resources: 88,000

	Ave. Ann. Grth Rate	1980	1985	1990	1995	2000	2005	2010	Cumulative
Projected Demand:L	0.07	1505	1571	1641	1713	1790	1868	1951	51,774
M	1.64	1539	1670	1811	1965	2130	2312	2508	60,012
H	2.26	1568	1753	1960	2192	2450	2741	3065	67,756

MARKET ECONOMIES:

Reserves: 410,000—440,000 — Total Resources: NA

	Ave. Ann. Grth Rate	1980	1985	1990	1995	2000	2005	2010	Cumulative
Projected Demand:M	2.60	6757	7590	8670	9894	11,412	12,840	14,598	309,310

CENTRALLY PLANNED ECONOMIES:

Reserves: 337,000—348,000 — Total Resources: NA

	Ave. Ann. Grth Rate	1980	1985	1990	1995	2000	2005	2010	Cumulative
Projected Demand:M	3.31	5255	6288	7364	8630	9988	11,886	13,969	271,774

TOTAL WORLD:

Reserves: 747,000—792,000 — Total Resources: NA

	Ave. Ann. Grth Rate	1980	1985	1990	1995	2000	2005	2010	Cumulative
Projected Demand:L	2.40	11,827	13,316	14,993	16,880	19,000	21,399	24,093	523 320
M	2.93	12,012	13,878	16,034	18,524	21,400	24,726	28,567	594 547
H	3.87	12,344	14,925	18,045	21,818	26,400	31,894	38,562	703 686

supply/demand intersections have been calculated on a static, rather than dynamic basis. Under the assumptions that reserves are fixed (no additions to reserves) and that production will meet demand until ore reserves are exhausted, world identified, measured reserves of manganese would be sufficient to last 32 to 40 years at projected rates of demand growth (Figures 3-7 and 3-8).

If manganese mine production capacity fails to grow at a rate sufficient to fulfill the above assumptions, a shortfall will occur much sooner. This could come about because of:

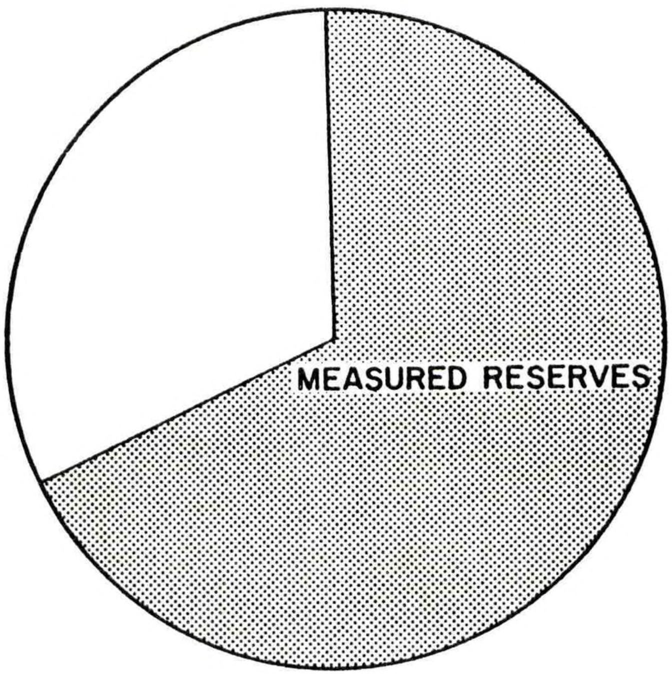
- o The increasing concentration of reserves and production capacity in just a few nations (U.S.S.R., Republic of South Africa, Gabon) limits the number of sources of manganese.

- o The lingering effects of the last major economic recession (1973-1975) which have left steel demand essentially flat. This in turn has kept real manganese prices from increasing sufficiently to trigger mine expansions and developments (Table 3-13).

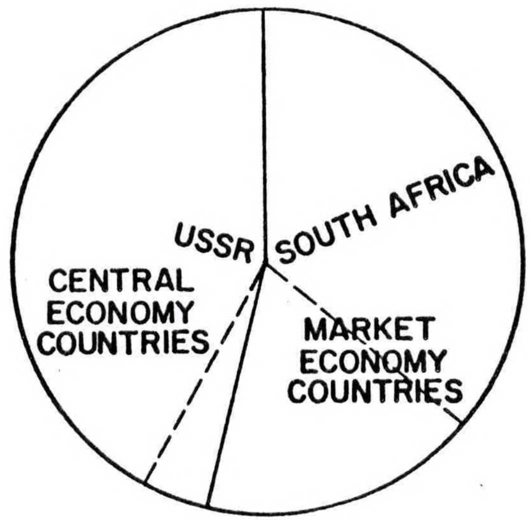
- o The supposed short-term inelasticity of manganese supply response to a rapid increase in manganese prices or demand. That is, given the concentration of production capacity (a large mine may produce 15 to 20 percent of the world's output) and typical new mine development lead times, an increase in demand for manganese beyond current mine capacity cannot be quickly met.

Another, more serious possibility exists. Any country that depends entirely on foreign sources of supply of a critically important raw material (as the United States does for manganese) is potentially vulnerable to economically or politically motivated shortages. In the past, the United States

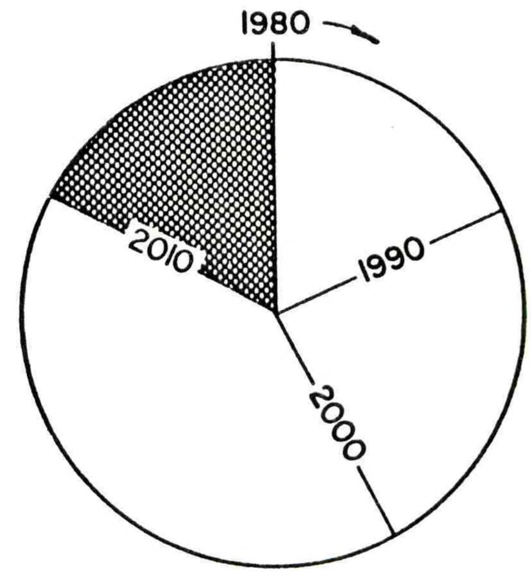
MANGANESE



MEASURED, INDICATED AND INFERRED RESOURCES
3.86 BILLION TONS
(GROSS WEIGHT OF ORE)



MEASURED RESERVES
747 MILLION TONS
(CONTAINED METAL)



CUMULATIVE WORLD CONSUMPTION
MOST LIKELY PROJECTION
(2.93%)

FIGURE 3-8

Table 3-13

TIME - PRICE RELATIONSHIP FOR MANGANESE IN ORE

YEAR	Average annual price, dollars per long-ton unit (22.4 pounds) Actual Prices	Price in constant 1977 dollars
1958	1.21	2.59
1959	.98	2.06
1960	.94	1.94
1961	.94	1.92
1962	.91	1.83
1963	.81	1.60
1964	.69	1.34
1965	.73	1.39
1966	.76	1.40
1967	.67	1.20
1968	.60	1.03
1969	.50	.82
1970	.54	.84
1971	.60	.88
1972	.60	.85
1973	.65	.87
1974	.90	1.10
1975	1.38	1.54
1976	1.45	1.54
1977	1.48	1.48
1978	1.40	1.31

Source: USBM Mineral Commodity Profile - Manganese, 1979.

reduced its manganese vulnerability by obtaining its supplies from sources that differed widely in both geographical location and political ideology; however, it seems clear that this approach will become increasingly difficult in the future. Because South Africa and the U.S.S.R. so completely dominate known world manganese reserves (and resources), they eventually also must dominate world manganese supply. The apparent cessation of manganese exports from the U.S.S.R. further narrows the potential sources of supply.

It might be argued that additional ore discoveries in other parts of the world would tend to reduce the future dominance of South Africa; however, the known ore reserves in South Africa are so vast that there is little economic incentive for additional exploration in other areas. Thus, it is entirely possible that a natural manganese monopoly will evolve and the United States will depend increasingly on South Africa for its future manganese requirements. As a consequence, U.S. vulnerability to manganese shortages will increase greatly.

Table 3-14

WORLD DEMAND FOR MANGANESE ORE
GROWTH RATES, 1951-55 TO 1971-75

REGION	AVERAGE ANNUAL GROWTH RATE
United States	0.44
W. Europe	4.71
Japan	8.34
Other Developed Lands	3.19
Africa	8.21
Asia	10.74
Latin America	9.33
TOTAL MARKET ECONOMY COUNTRIES	4.63
U.S.S.R.	3.39
E. Europe	2.51
China	9.25
TOTAL CENTRAL ECONOMY COUNTRIES	3.67
World	4.20

Source: derived from data in Malenbaum, Wilfred, World Demand for Raw Materials in 1985 and 2000, 1978.

Table 3-15
WORLD DEMAND FOR MANGANESE ORE
(1000 metric tons)
5-year averages

Region	1934-38	1951-55	1956-60	1961-65	1966-70	1971-75*
1. W. Europe	1345	1647	2288	2664	3715	4134
2. Japan	222	336	527	758	1038	1667
3. ODL	132	421	674	619	704	789
4. U.S.S.R.	1732	3297	4400	5786	5772	6420
5. E. Europe	359	705	1041	1096	1134	1157
6. Africa	122	99	169	221	308	480
7. Asia	110	180	229	669	1157	1384
8. L. America	18	180	184	673	687	1071
9. China	4	170	864	919	920	998
10. U.S.	629	1768	1755	1909	2090	1929
Totals						
11. World	4673	8803	12133	15314	17525	20030
12. Non-U.S. World	4044	7035	10378	13405	15435	18101
13. Poor Nations	254	629	1446	2483	3072	3934
14. Rich Nations	4418	8174	10625	12831	14455	16096

Source: Malenbaum, 1978; Production, Exports and Imports (1934-1971); Metal Bulletin Limited (1969).

*Preliminary in particular the figure for ODL (3) is uncertain due to inventory adjustments

Table 3-16

WORLD MANGANESE RESERVES AND RESOURCES (metric tons)

Country	Deposit Name	Reserves Million t	Grade	Mn Content of Reserve Million t	Product Equivalent (45% Mn) Million t	Resources Million t	Grade	Comments
USSR	Nikopol							
	Bol'shoi Tokmaksk	716	23.7	168	286			
	Shlonersk	-	-	-	-			
	Chiatura					1575		
	primary oxide	98	45-52	47	79			
	carbonate	82	10-30	16				
	hypergene oxide	29	30-35	9	37			
	Chkhari-Adzhemesk							
	Labinsk(oxide + carbonate)	-	-	-	-		8.8-25.2	
	Mangyshlak(oxide + carbonate)	-	-	-	-	60	3.8-47.0	
	Northern Urals carbonate	50	21.2	11	e 20	-	-	
	Ulutelyaksk	8	10	1	-	-	-	
	Akkermansk	-	-	-	-	4	-	
	Attasujsk Group Murdzhinsk	122	19-30	30	e 50	-	-	
	Dzhezdin-Ulatausk Group	-	-	-	-	13	-	
	Primagnitogarsk Group (oxide + silicates)	-	-	-	-	3	15-50	
	Usinsk							
	oxide	-	-	-	-	-	-	
	carbonate	95	3.9-11.1	7	e 24	-	-	
	Malij Khingan	10	21-35	3	e 4	-	-	
	Udako-Selemazhinsk Group	-	-	-	-			
	Arginsk Group	-	-	-	-	35	32-50	Small deposits scattered along strike for over 100 km.
	Gornoshorsk Group	-	-	-	-			
	Nizhneudinsk	-	-	-	-	50	13.6-17.7	
	Karsakpay	45	32	14	e 20	-	-	
	Total USSR	1255		306	520	1740		

Table 3-16, continued

WORLD MANGANESE RESERVES AND RESOURCES (metric tons)

South Africa	Wessels (Mn shale)	200	35-45	80	175	?	Underground, low Mn:Fe ratio
	Mamatwan (Mn shale)	200	30-40	70	155	?	Open pit, high Mn:Fe ratio
	Hotazel (Mn shale)	small	35-45	-	-	+ 3000	Open pit/underground, nearing exhaustion of upper better grade ore horizon.
	Lohathla (Mn shale)	± 100	>40	40	90	?	Open pit, near surface, enriched shale, collapsed breccias in dolomite, low Mn:Fe ratio
	Santoy/Belgravia (Black Rock Mine) (Mn shale)	20	+ 40	8	18	?	Open pit/underground, upgraded to + 43% Mn by hand sorting.
	Nehwaning (Mn shale)	5	44.6	2	5	10	Underground
	Gloria (Mn shale)	30	+ 38	11	25	+ 600 ?	Underground; in process of development.
	Adams/Devon (Mn shale)	very small	40			nil	Open pit, virtually worked out.
	Bergheim (Mn shale)	?	42			?	Prospect, adjacent to Wessels, structurally complex.
	Mukulu (Mn shale)	?	30-35			?	Prospect
	Kongoni) Tolelo) (Mn shale)					large	Prospects, very deep, no plans for more work.
	Bishop) Gloucester) Paling) Japies) (Manganiferous ore in collapsed domolite sink holes)	small	32-45			?	Small production, high-cost, labor intensive, no significant future production planned.
	Rooninekke (Mn/Fe shale)	small	22-24			2	Open pit, ore contains 20-30% Fe
	Pensfontein (Mn shale)	small	45			?	Open pit
	Kupstewel (Mn shale)	small	45			?	Open pit
	Langdon (part) (Mn shale)	small	+ 48			small	Open pit
	Devon (part)						Open pit, worked in conjunction with Langdon, data for Devon not available.
	Copani Manganese Mine	small	40				Underground, produced for U_3O_8 processing.

Table 3-16, continued

WORLD MANGANESE RESERVES AND RESOURCES (metric tons)

South Africa (cont'd)	Rand London Mn Mines) Roodepan Mn Corp.)							Small battery grade producers, no data available
	Middleplaats (Mn shale)	100	38	38	85	200		Undergroupd; capitalization + 40 x 10^6 rand; completion expected mid-1979; first new large Mn mine in competion with the two established producers.
	Total South Africa	655		249	553	3810 +		
Gabon	Moanda	193	51.7	57	110	377	24.5-45.4	Open pit
Australia	Groote Eylandt (oxide)	190	25-40	33	73	300		Open pit
	Woodie Woodie) Mount Sydney) (Mn deposits Ripon Hills) Pilbara, Skull Springs) W.A.) Ant Hill)					60 13	17) 19.1)	All significant production terminated in June 1975. Rippon Hills ore contained 25% Fe and Ant Hill ore contains 29.4% Fe.
	Mount Rove) (Ferro Mn Mount Nicholas) deposits, W.A.)						6.9) 1.9)	Mount Rove ore contains 47% Fe and Mt. Nicholas ore contains 51.7% Fe.
	Peak Hill (Fossil bog ores)					1	± 42	Not currently in production as reserves depleted; 15-20% Fe
Ghana	Nsuta oxide carbonate	1	52	1	1	30	46	
Morocco	Imini	1	47	1	1	2	47	
Brazil	Amapa Serra do Navio	24	33.8-46.9	7	15	4	28.5-45.3	
	Minas Cerais Morro da Mina	6	31.8	2	3			Sil-carbonate; calcined product plus minor fines
	Lafaiete Vicinity Agua Preta Olaria I Estiva Olaria II Jacuba Barroso Sao Dimas							

Table 3-16, continued

WORLD MANGANESE RESERVES AND RESOURCES (metric tons)

Brazil (Cont'd)	Fazenda Peroba	-	-	-	-	)			
	Agua Boa	-	-	-	-	)			
	Lagoinha	-	-	-	-	)			
	Cocuruto	-	-	-	-	)			
	Marimbondo	-	-	-	-	)			
	Paivu	-	-	-	-	)			
	Pasto da Serra	-	-	-	-	)			
	Fazenda Perobu	-	-	-	-	)	1	20-36	
	Maracuja	-	-	-	-	)			
	Collatino	-	-	-	-	)			
	Mina do Amaral	-	-	-	-	)			
	Sabugo (Padre Joao)	-	-	-	-	)			
	Buraco do Bicho	-	-	-	-	)			
	Jurema	-	-	-	-	)			
	Michaela	-	-	-	-	)			
	Sao Concalo	-	-	-	-	)			
	Sabino	-	-	-	-	)			
	Nedio	-	-	-	-	)			
	Pequiri	-	-	-	-	)			
	Bandeirinhas	-	-	-	-	)			
	Unha do Cato	-	-	-	-	)			
	Mina da Pussagem	-	-	-	-	)			
	Pazenda do Rocha	-	-	-	-	)			
	Agua Limpa	-	-	-	-	)			
	Dist. Sao Joao del Rei								
	Geriminal	-	-	-	-	)			
	Sumare	-	-	-	-	)			
	Cha	-	-	-	-	)			
	Pasto do Catiglinao	-	-	-	-	)			
	Penedo	-	-	-	-	)	1	36-46	
	Cachoeira	-	-	-	-	)			
	Cagenga	-	-	-	-	)			
	Santo Antonia	-	-	-	-	)			
	Aquino	-	-	-	-	)			
	QuadriLatero Ferrifero								
	Conta Historia	-	-	-	-	)		32-39	Ore contains 21-27% Fe
	Timbopeba	-	-	-	-	)		42	Ore contains 17% Fe
	Antonio Pereira	-	-	-	-	)		35	Ore contains 25% Fe

Table 3-16, continued

WORLD MANGANESE RESERVES AND RESOURCES (metric tons)

BRAZIL (Cont'd)	Lagoa Grande	-	-	-	-	)	4	35-45	Ore contains about 60% combined Mn+Fe
	Fazenda Caieira	-	-	-	-	)		30	Ore contains 28% Fe
	Rodrigo Silva	-	-	-	-	)		40	
	Fazenda Condarela	-	-	-	-	)		35	
	Serra do Bateteiro	-	-	-	-	)		25-28	Ore contains 30% Fe
	Dist. Dourada								
	Fuzenda Ponte Nova	-	-	-	-	)	very small	17	
	Dist. Sao Domingos do Prata								
	Lucas-Saude	-	-	-	-	)		35	Some battery grade & metallurgical grade has been produced
	Mina do Portao	-	-	-	-	)	<1?	13-47	
	Sao Bartolomeu	-	-	-	-	)		24-40	
	Dist. Serra do Cipo								
	Santana Pirapama	-	-	-	-	)		38-46	Ore contains 5-15% Fe and 0.2-0.5% P
	Ponte de Pedra	-	-	-	-	)	small	38	Ore contains 10% Fe
	Forquilha	-	-	-	-	)		20-40	Ore contains 21-33% Fe
	Chapeu de Sol	-	-	-	-	)			
	Dist. Rio Padro de Minas								
	Fazenda Tabutinga	-	-	-	-	)		38	
	Fazenda Jardim	-	-	-	-	)	very	20.7-50.0	
	Coqueiros	-	-	-	-	)	small	20-51	
	Alagoas	-	-	-	-	)		30-35	
	Cedro	-	-	-	-	)		35.9-40.0	
	Capao	-	-	-	-	)		35.5-45.7	
	Dist. Mateus Leme e Andradas								
	Serra Azul	-	-	-	-	)		22	The ore is related to the itabirites of the Minas Series
	Andradus	-	-	-	-	)	?	18-45.7	The ore contains 7.8-31.0% Fe
	Other Minas Gurais Occurrences								
	Caete	-	-	-	-	)			
	Betim	-	-	-	-	)			
	Santa Barbara	-	-	-	-	)			
	Barbacena	-	-	-	-	)			
	Rio Espera	-	-	-	-	)			
	Ouro Preto e Marina	-	-	-	-	)			
	Congonhas	-	-	-	-	)			
	Itabirito	-	-	-	-	)			
	Pitangui	-	-	-	-	)	+ 2		

Table 3-16, continued

WORLD MANGANESE RESERVES AND RESOURCES (metric tons)

Brazil (Cont'd)	Diamantina	-	-	-	-	)			
	Jaboticatubas	-	-	-	-	)			
	Itabira	-	-	-	-	)			
	Volta Grande	-	-	-	-	)			
	Caldus	-	-	-	-	)			
	Miguel Burnier	-	-	-	-	)			
	Para								
	Igarape Azul	-	-	-	-	)	40	41.2-44.2 p	About 24 million t lump and the rest granular material
							25	28.2	Resources material this line is manganiferous shale
	Buritirama	-	-	-	-		11	47	Several types of Mn oxides (i.e., pulverent, massive, friable, etc.)
	Serra do Sereno	-	-	-	-		3	19-50	
	Mato Grosso								
	Urucum	4	46 p	2	4		55	? -46	Ore contains 11-12% Fe and 3.5-4.5% alkalies. (Note: about 30 million tons of the resources are considered ore even though only 4 holes have been drilled in Urucum).
	Other deposits	-	-	-	-		41	? -46	Hypothetical resources (DNPM)
	Bahia								
	Marau	-	-	-	-		2	-	
	Other deposits	-	-	-	-		5	-	Hypothetical resources (DNPM)
	Goias								
	Diverse Occurrences	-	-	-	-	)			
	Itabirai	-	-	-	-	)		46-54	
	Uruacu (Agua Branca)	-	-	-	-	)		30-45	
	Acaba Saco	-	-	-	-	)		27-48	
	Corrego D'Anta	-	-	-	-	)		31-48	
	Caldas Novas	-	-	-	-	)	<1	36	
	Paulo Meiras	-	-	-	-	)		41	
	Alexania e Luziania	-	-	-	-	)		40	
	Mara Rose e Porangatu	-	-	-	-	)		30	
	Fazenda Genipapo	-	-	-	-	)		-	
	Fazenda do Se. Antonio Vicente	-	-	-	-	)		26.7-41.6	

Table 3-16, continued

WORLD MANGANESE RESERVES AND RESOURCES (metric tons)

Brazil (Cont'd)	Arraias	-	-	-	-	)	40-53	
	Cavalcante	-	-	-	-	)	37-47	
	Dist. Sao Joso da Alianca							
	Pedra Preta	-	-	-	-	)	46	
	Tocantinzinho	-	-	-	-	)	42	
	Pedro de Amolar	-	-	-	-	)	48	
	Jatobazinho	-	-	-	-	)	42	
	Fazenda Pontezinha	-	-	-	-	)	42	
	Jataizinho	-	-	-	-	)	40	
	Olhos D'agua	-	-	-	-	)	46	
	Fazenda Polonia	-	-	-	-	)	46	
	Liborio	-	-	-	-	) very small	45	
	Fazenda Ribeiro	-	-	-	-	)	-	
	Carestia	-	-	-	-	)	-	
	Confluencia	-	-	-	-	)	-	
	San Lourenco	-	-	-	-	)	48	
	San Lourenco (Ladeirao)	-	-	-	-	)	-	
	Fazenda Morro	-	-	-	-	)	48	
	Buritizinho	-	-	-	-	)	45	
	Vaoinho	-	-	-	-	)	44	
	Vereda	-	-	-	-	)	46	
	Lagoas	-	-	-	-	)	-	
Mexico	Molango (carbonate)	13	38.5 p	3	8	-	-	Beginning to switch from an open pit to an underground operation.
U. Volta	Tambao	-	-	-	-	13	54	
Zaire	Kisenge	8	35-40	3	6	-	-	
India	Jamda Koira Valley Orissa	-	-	-	-	36	30 +	
	Sonah Orissa	-	-	-	-	1	30 +	
	Madhya Pradesh Maharashtra	7	46 +	3	7	42	38-46+	
	Sandur Valley	-	-	-	-	14	38-46+	
	North Kanara Tumkur, Chitradurga Dist. Mysore	-	-	-	-	2	30 +	
	Goa	-	-	-	-	1	30 +	
	Srikakulam Dist. Andra Pradesh	-	-	-	-	1	30 +	
	Gujrat-Rajasthan	-	-	-	-	3	30 +	

Table 3-16, continued

WORLD MANGANESE RESERVES AND RESOURCES (metric tons)

China	All deposits	29	20-50	? 10	? 22	-	-
Hungary	All deposits	4 ?	? 45 p	-	? 4	-	-
Japan	All deposits (75 mines)	6	24	1	? 3	-	-
Others	Bulgaria, USA, Chile Mali, Israel, Canada, etc.	-	-	-	-	293	-
	Free World Total	1108		362	784	5195	
	Communist Block Total	1288		316	546	1740	
	World Total	2396		678	1330	6935	

e = estimate

Source: United States Steel Corporation, in testimony by Robert L. L'Esperance, Director-Corporate Explorations and Investigations, before The Subcommittee on Oceanography in regard to H.R. 3350, 95th Congress, the "Deep Seabed Hard Minerals Act," and H.R. 3652, 95th Congress, the "Ocean Mining Incentive Act of 1977," April 22, 1977.

4.0 NICKEL: SUPPLY-DEMAND

4.1 INTRODUCTION

World reserves and resources of nickel are extensive. However, despite the potentially large nickel resource base, it is uncertain whether many of the identified nickel deposits can be economically brought into production to meet estimated long-term demand (1980-2010) for this metal. Additionally, there is some question concerning the availability of energy necessary to process low-grade nickel ores.

In the discussion which follows, the economic geology of world nickel reserves and resources will be reviewed. This section will in turn be followed by an analysis of the reserve and resource position of nations known or thought to possess nickel deposits. The last part of this chapter will consider the future development of nickel occurrences from the standpoint of possible mining methods, rates of production, and timing of exploitation. An investigation of nickel demand will follow, with a look at past consumption as well as projected demand.

4.2 NICKEL SUPPLY

4.2.1 Part I - The Economic Geology of Nickel

Unlike many metals, nickel is widely distributed in the earth's crust. According to Krauskopf (1967), nickel is the fifth most abundant element of the earth. Although nickel is common in many geologic environments, economically significant concentrations of the metal are most commonly localized in ultramafic igneous rocks, which contain as much as 0.1 to 0.3 percent nickel.

Nickel occurs as a major or trace element in many minerals. However, as indicated in Table 4-1, the primary nickel-bearing minerals of economic importance are the nickel iron sulfides pentlandite, garnierite (which is not a true mineral, but rather a mixture of nickel, serpentine, nickeliferous talc and other silicites), and nickeliferous pyrrhotite and pyrite.

There are essentially two major categories of land-based nickel deposits: sulfide deposits and nickeliferous laterite deposits Figure 4-1). In the following section, the geologic setting and economic geology of selected, major nickel-bearing deposits of each of the two types will be discussed.

4.2.1.1 Sulfide Deposits

4.2.1.1.1 Introduction

Nickeliferous sulfide deposits are most commonly found at or near the base of peridotite or norite intrusives. The major orebodies generally consist of disseminations, massive bodies, and/or veins and stringers of pyrrhotite and pentlandite. Most nickel sulfide deposits are elongate, lenticular, or sheet-like and often extend many thousands of feet in depth. From a purely geologic standpoint, many sulfide nickel deposits are believed to have been formed by the process of magmatic segregation. Through this mechanism, immiscible liquid nickel sulfide droplets were segregated from the host mafic or ultramafic magma early in its crystallization. As the magma

Table 4-1

MAJOR NICKEL MINERALS OF ECONOMIC IMPORTANCE

MAJOR NICKEL MINERALS OF ECONOMIC IMPORTANCE

Mineral	Formula	Occurrence
Native nickel-iron	Ni_3Fe	Rare
Pentlandite	$(Fe,Ni_9S_8$	Common
Bravoite	$(Fe,Ni)S_2$	Minor; secondary
Violarite	Ni_3FeS_4	Do
Vaesite	NiS_2	Rare
Polydymite	Ni_3S_4	Do
Millerite	NiS	Minor; secondary
Heazlewoodite	Ni_3S_2	Rare
Siegenite	$(Ni,Co)_3S_4$	Minor
Linnaeite	$(Co,Fe,Ni)_3S_4$	Do
Geradorffite	NiAsS	Do
Niccolite	NiAs	Do
Rammelsbergite	$NiAs_2$	Rare
Chloanthite	$(Ni,Co)A_{3-4}$	Do
Smaltite	$(Co,Ni)A_{3-4}$	Do
Skutterudite	$(Co,Ni)A_3$	Do
Maucherite	$Ni_{11}As_8$	Do
Breithauptite	NiSb	Do
Ullmannite	NiSbS	Do
Parkerite	$Ni_3Bu_2S_2$	Do
Annabergite	$Ni_3(AsO_4)_2 \cdot 8H_2O$	Rare; secondary
Morenosite	$NiSO_4 \cdot 7H_2O$	Do
Zaratite	$NiCo_3 \cdot 2Ni(OH)_2$	Do
Garnierite	Nickel-magnesium hydrosilicate	Common; secondary.

Source: USGS Prof. Paper 820, 1973, p. 438.

IMPORTANT NICKEL RESERVES & RESOURCES

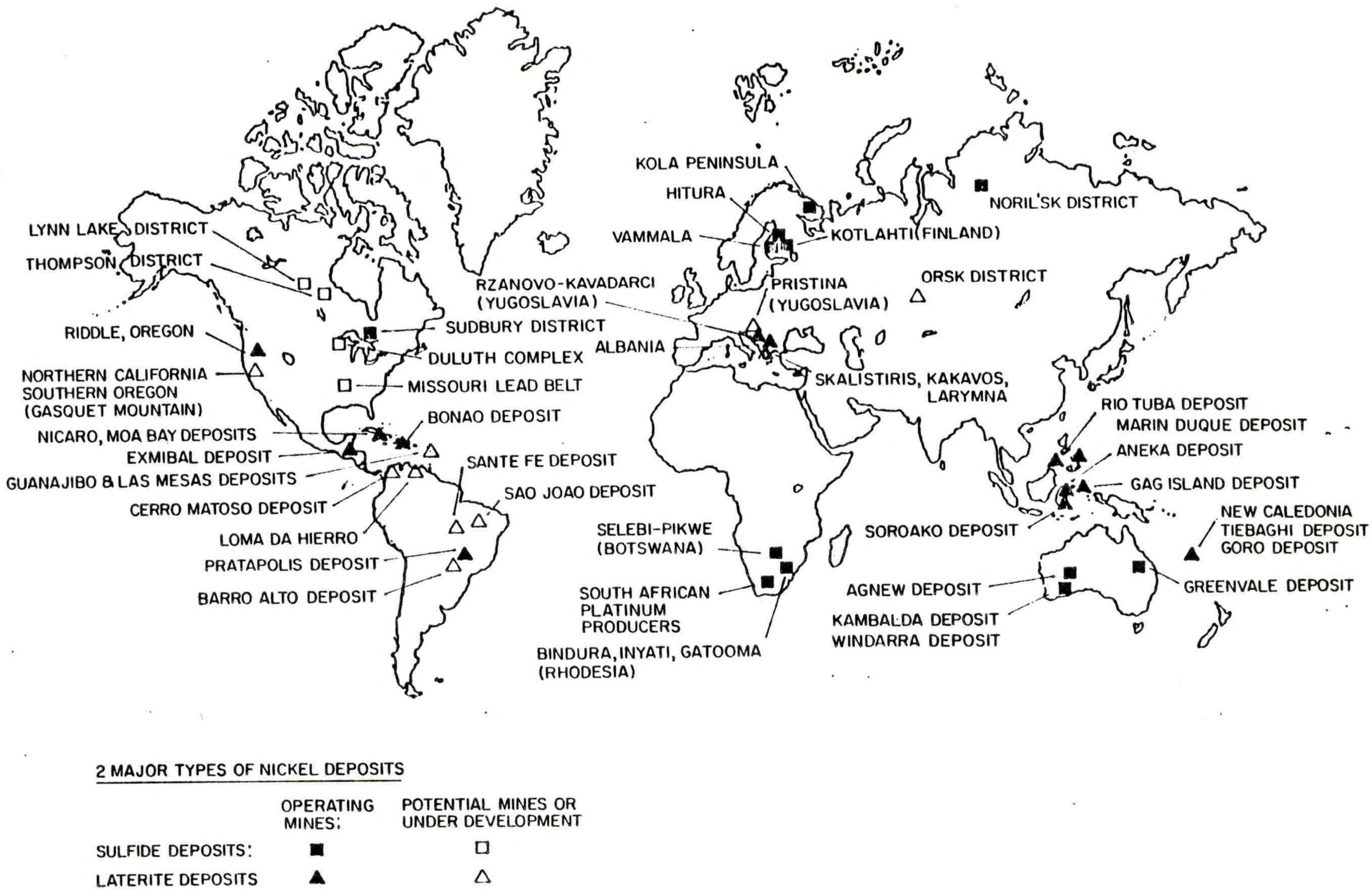


Figure 4-1

cooled, the nickel sulfide droplets collected as a layer at the base of the intrusive. In some instances, subsequent deformation of the intrusive may have remobilized the nickel sulfide layer thereby causing it to migrate and fill faults and fractures in the ultramafic and associated host rocks. Although this mode of origin has been used to explain the genesis of numerous nickel sulfide deposits, many other theories have been suggested to account for the occurrence of nickel sulfide deposits in similar or other geologic environments.

Some of the larger and perhaps most studied nickel sulfide deposits occur in the Sudbury district of Ontario, Canada. As of 1978, this district had one open pit and twelve underground nickel-copper mines either operating or on a standby basis.

4.2.1.1.2 Sudbury District, Ontario

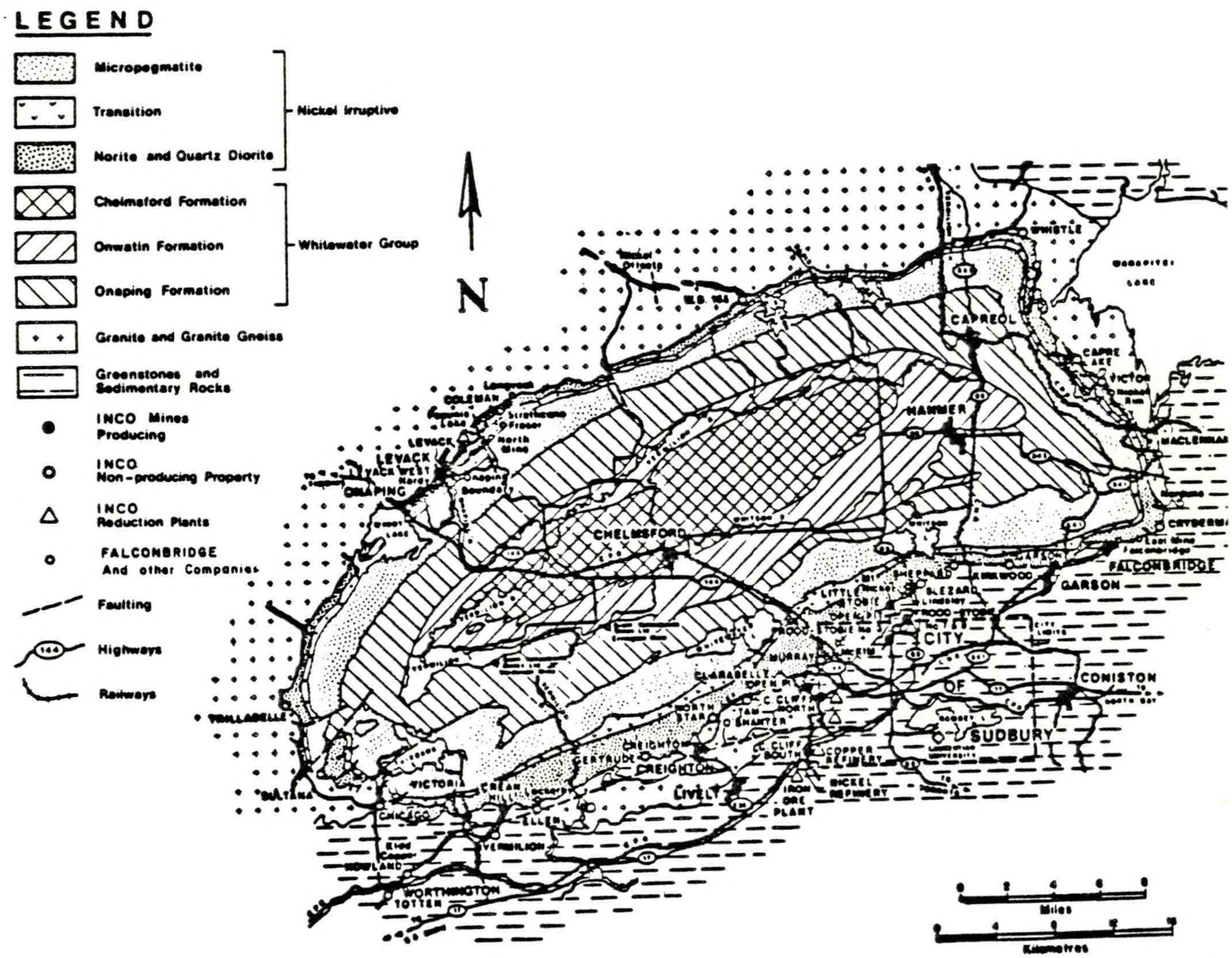
The copper-nickel ores of the Sudbury district occur in a multi-layered intrusive which, in its simplest form, consists of a lower layer of norite (a rock consisting primarily of pyroxene and calcic plagioclase feldspar with interstitial quartz and alkali feldspar) overlain by a granitic layer composed of micropegmatite. At the contact of the norite with the micropegmatite is a sublayer of quartz diorite with which most of the copper-nickel ores are associated. Overlying the Sudbury Intrusive is the Whitewater Formation which consists of a series of tuffs, sandstones, slates, and carbonate-rich sediments.

In plan, the Sudbury Intrusive is shaped like a bowl of a spoon. At the surface, the axes of the bowl are approximately 37 miles long by 17 miles wide (Figure 4-2). Drilling at the center of the intrusive indicates that it extends to a depth of at least 10,000 feet below the surface, or to the top of the Whitewater Formation.

The principal copper-nickel mineralization consists of pyrrhotite, pentlandite, and chalcopyrite. Mineralization is localized and does not

Figure 4-2

THE GEOLOGY OF THE SUDBURY BASIN, ONTARIO, CANADA



Source: Ontario Ministry of Natural Resources,
Mineral Policy Background Paper No. 4, 1977, p. 85.

occur uniformly throughout the district at the norite-micropegmatite contact. Through exploration, the most economically significant mineralization has been found to occur:

- o Where the norite or contact phase (quartz diorite sublayer) penetrates the Footwall Rock (which consists of a series of greenstones, granites, and gneisses).
- o In areas of shearing and associated breccia zones at the base of the intrusive.
- o In contact breccias at the base of the intrusive.

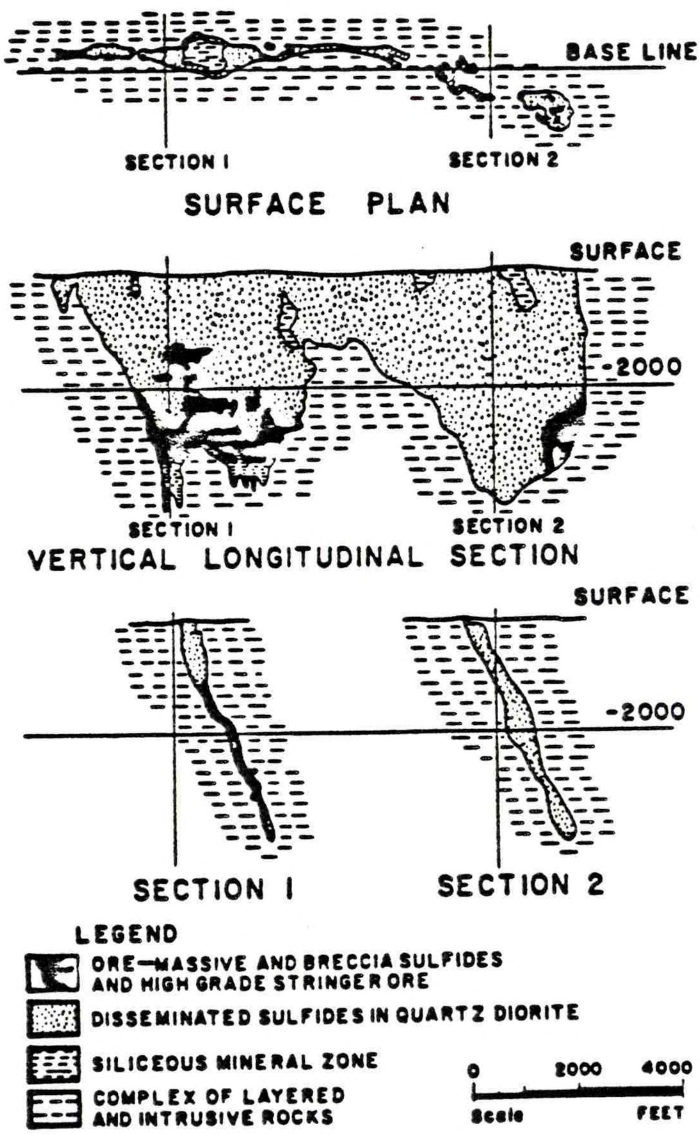
Although in several localities the economically significant mineralization is controlled by just one of these relationships, more often than not several come into interplay and have been identified as responsible for localizing mineralization.

In the following section, three typical copper-nickel deposits in the Sudbury Basin will be briefly discussed to provide the reader with prospective as to the nature and mode of occurrence of mineralization.

Frood-Stobie Deposit. The Frood-Stobie deposit, which occurs at a depth of more than 2000 feet below the surface, is an example of ore control by penetration of the Sudbury Intrusive into the Footwall Rock or basement rocks. As is evident from Figure 4-3 , the Frood-Stobie deposit lies approximately one mile from the nearest edge of the Sudbury Intrusive. Mineralization of the Frood-Stobie deposit occurs as blebs and disseminations of copper-nickel sulfides at the bottom of a quartz-diorite body which is approximately 10,000 feet in length and 800 feet wide. The primary ore minerals are pyrrhotite, chalcopyrite, and pentlandite. Accessory minerals include niccolite, maucherite, and gersdorffite.

Figure 4-3

PLAN VIEW, VERTICAL LONGITUDINAL SECTION, AND CROSS-SECTIONS OF THE FROOD-STOBIE MINE



Source: Boldt, J. R., The Winning of Nickel: Its Geology, Mining, and Extractive Metallurgy, 1967, p. 38.

Creighton Deposits. The Creighton deposits occur from the surface to a depth of over 3,800 feet in a depression at the base of the Sudbury Intrusive in a zone of intense shearing and brecciation. The mineralization consists of massive and disseminated sulfides at the intersection of shears in the Footwall Rocks (Figure 4-4). According to Boldt (1967), the typical Creighton orebody "is made up essentially of an upper part consisting of disseminated sulfides in contact phase norite and a lower part consisting of massive or breccia sulfides in the Footwall Rocks."

Murray Deposits. The Murray deposits occur in, and adjacent to, a contact breccia zone which (as indicated in Figure 4-5) occupies the lower portion of a depression at the base of the Sudbury Intrusive. Mineralization, which extends from the surface to a depth of over 3,000 feet, averages approximately 1,500 feet in length and 500 feet in thickness.

4.2.1.2 Lateritic Deposits

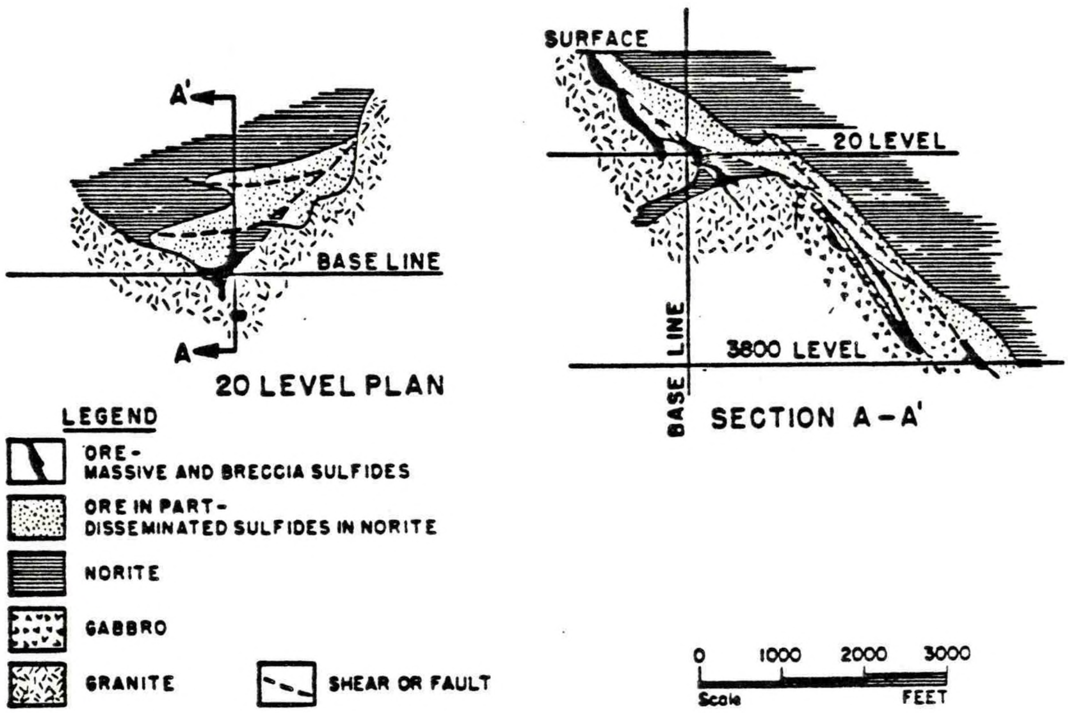
4.2.1.2.1 Introduction

Although nickeliferous laterite deposits are located throughout the world, most occur in areas which presently have subtropical climates. Virtually all nickel laterites are formed by the weathering of various ultramafic intrusives--principally dunite (a rock consisting primarily of olivine), peridotite (a rock composed essentially of olivine and pyroxene), pyroxenite (a rock comprised predominantly of pyroxene) and/or serpentine (serpentinized peridotite). The composition and nickel content (as well as the content of various accessory elements) of the resultant laterite are contingent upon the type of rock which has been weathered, the intensity of weathering, and the topography of the region.

In simple terms, lateritic weathering of a rock occurs when ground water containing carbon dioxide comes in contact with an olivine-rich rock. As the olivine is decomposed, magnesium, iron, and nickel are taken into solution. The silica content of the rock is also dissolved, but becomes a colloidal

Figure 4-4

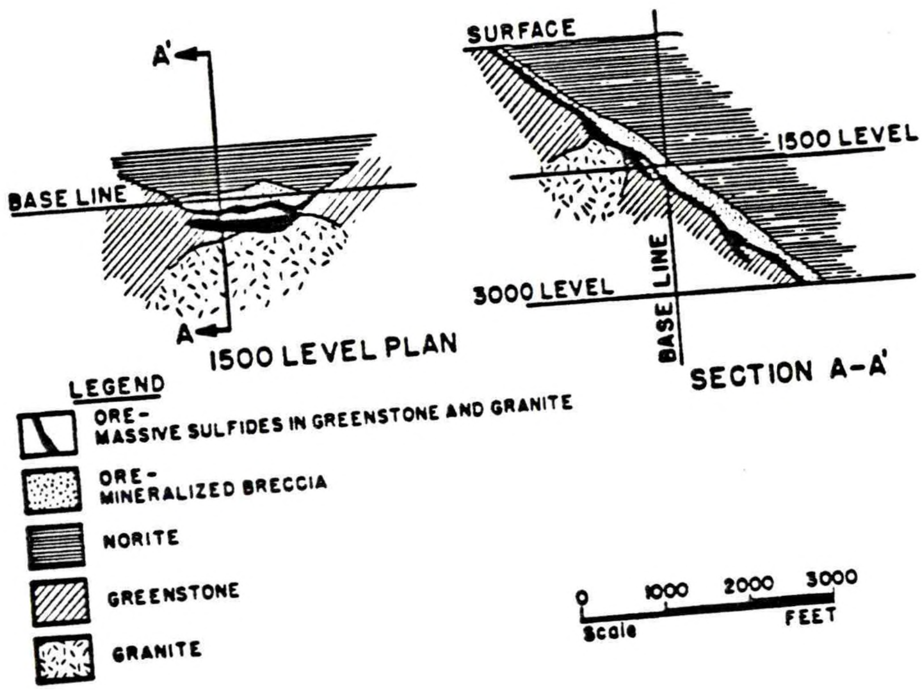
GENERALIZED GEOLOGICAL PLAN AND SECTION, CREIGHTON MINE



Source: Boldt, J. R., The Winning of Nickel: Its Geology, Mining, and Extractive Metallurgy, 1967, p. 36.

Figure 4-5

GENERALIZED GEOLOGICAL LEVEL PLAN AND SECTION, MURRAY MINE



Source: Boldt, J. R., The Winning of Nickel: Its Geology, Mining, and Extractive Metallurgy, 1967, p. 32.

suspension consisting of sub-microscopic particles of silica. With time, the iron in solution oxidizes and is precipitated as ferric hydroxide which, upon loss of water, alters to form the minerals goethite and hematite. The magnesium, nickel, and silicon will continue to remain in solution and will be carried below the zone of iron precipitation as long as the water remains sufficiently acid. Once the acidity of the mineralized solution is reduced, the magnesium, nickel, and silicon will begin to precipitate as hydrous silicates--generally in a zone lying just above the fresh, unaltered host rock.

The process of leaching is continuous insofar as when fresh rock is exposed to erosion, leaching can begin anew. If regional topography is too extreme, erosion may occur at too rapid a rate to permit effective leaching of the host rock. To generate an economically significant concentration of nickeliferous laterite, leaching induced enrichment must occur over a long period of time--from several thousand to over a million years.

Aside from the intensity of leaching and the nature of the solutions involved, the concentration of nickel in the laterite is in large part contingent upon the nickel content of the unweathered host rock.

Essentially two types of lateritic nickel deposits are known: <u>limonitic or nickeliferous iron laterites</u> and <u>nickel silicate laterites</u>. The limonitic or nickeliferous iron laterite develops from the weathering of serpentinite. The iron and nickel mineralization is pervasive throughout the deposit with little, if any, separation of the iron and nickel into distinct zones aside from the iron-rich layer which typically overlies the entire deposit. Exactly in what form the nickel occurs in the laterite is unknown.

As implied by its name, the nickel in nickel-silicate laterites occurs as either the hydrosilicate garnierite or as nickel-bearing talc or antigorite. Nickel silicate laterites developed from the weathering of peridotite display a much greater degree of separation between the iron-nickel mineralization than is the case with limonitic or nickeliferous iron laterites. In most nickel-silicate laterites, the nickel will tend to concentrate in a distinct zone just above the fresh, unaltered host rock.

As indicated in Table 4-2, from a geochemical standpoint, each type of nickel laterite deposit typically contains different quantities of iron, silicon dioxide, and nickel.

Table 4-2

IRON, SILICON, AND NICKEL CONTENTS OF NICKEL LATERITES

EXAMPLES	Fe	SiO_2	Ni
Limonitic or Nickeliferous Iron Laterite o Cuba	45.0-55.0%	Very Low	0.9-1.3% Averages 1.0%
Nickel Silicate Laterite o Indonesia o New Caledonia o Riddle, Oregon	30.0% or less	As much as 30%	Usually exceeds 1.5% Averages 1.6%

In the section which follows, two examples of nickel-bearing laterites will be discussed. The primary emphasis of each review will be to briefly describe the geologic setting and economic geology of the deposit under consideration.

4.2.1.2.2 Limonitic or Nickeliferous Iron Laterite - Guatemala

The limonitic-iron lateritic nickel deposits of eastern Guatemala occur in the vicinity of Lake Izabal, approximately 50 miles inland from the Caribbean Coastline.

The deposits were formed from weathering of a partly serpentinized peridotite. The typical soil profile consists of an upper layer of limonitic soil which progressively grades downward to a limonitic nickel-rich zone. With increasing depth, the limonitic nickel-rich zone becomes more serpentiferous until such point that the bed or host rock is encountered. In most localities the nickel-rich zone, which contains approximately 1.5 percent nickel, averages 30 feet in thickness. However, in several areas, the nickel-rich zone can achieve a thickness of up to 100 feet. The ratio of nickel-rich material to overburden is about 2:1. Although the host minerals to nickel are unknown, it is thought that the nickel is associated with limonite in the limonitic ore and with residual serpentine.

From the standpoint of exploration, the extent of erosion in the Lake Izabal region can be determined rapidly by observing the extent of tree cover. Whereas trees will grow in the low-nickel limonitic soil which is essentially considered to be overburden, they do not thrive in the soils of the nickel-rich zone.

4.2.1.2.3 Nickel-Silicate Laterite - New Caledonia

Nearly one-third of the Island of New Caledonia is underlain by a variety of ultramafic intrusives, many of which have been partially or completely serpentinized. Consequently, given the tropical climate of the South Pacific region, it is not surprising that large areas of New Caledonia contain lateritic nickel deposits.

The lateritic nickel deposits of New Caledonia are believed to have formed in two stages. During the first stage, silica and magnesium were removed from the peridotite and serpentinite through leaching. This resulted in a limonitic lateritic material containing up to 1.5 percent nickel. Continued weathering of the limonitic lateritic material not only resulted in the removal of more silicon and magnesium, but also caused the nickel concentrated in the previous stage to enter into solution. As the nickeliferous solutions were carried downward, the nickel was reprecipitated at depth, thereby forming a nickel-rich zone which contains an average of 3.5 percent or more nickel. The principal nickel-bearing mineral - rock of economic importance is garnierite--a nickeliferous variety of serpentine. In most New Caledonian nickel laterities, the nickel-rich zone lies from one to twenty feet below the surface. As in the case of Guatemalan nickel laterite deposits, the ratio of overburden to ore (nickel-rich material) is approximately 2:1.

4.2.2 Part II - Nickel Reserves and Resources

4.2.2.1 Introduction

World nickel reserves and resources have been determined by many groups and organizations. Table 4-3 gives two of the most recently developed estimates of world nickel reserves and resources as calculated by the USBM (1979) and the United Nations Committee on Natural Resources (1979). As is evident from a comparison of the results of these two studies, fairly close agreement exists with regard to the extent of nickel reserves-resources for many individual countries, geopolitical blocs, and the world. Consequently, because of this similarity, a discussion of the specific assumptions implicit in the derivation of each estimate is unnecessary.

As described in Part I of this chapter, virtually all nickel deposits of economic significance occur either as laterites or sulfides. Table 4-4 classifies world nickel reserves and resources by geologic type.

Table 4-3

ESTIMATED WORLD RESERVES-RESOURCES OF NICKEL

(thousands of tons metal content)

	RESERVES		RESOURCES		TOTAL	
	USBM 1979	United Nations 1979	USBM 1979	United Nations 1979	USBM 1979	United Nations 1979
NORTH AMERICA						
United States	200	220	14,900	331	15,100	551
Canada	8,700	8,375	12,500	8,265	21,200	16,640
TOTAL	8,900	8,595	27,400	8,596	36,300	17,191
SOUTH AND CENTRAL AMERICA						
Brazil	460	2,149	3,640	992	4,100	3,141
Columbia	900	1,543	600	220	1,500	1,763
Cuba	3,400	3,747	14,200	18,293	17,600	22,040
Dominican Republic	1,100	1,102	100	771	1,200	1,873
Guatemala	300	551	900	1,102	1,200	1,653
Puerto Rico		--	900	1,102	900	1,102
Venezuela		661	700	496	700	1,157
TOTAL	6,160	9,753	21,040	22,976	27,200	32,729
EUROPE-CHINA						
Albania		220		331		551
China		827		NA		827
Finland		882		--		882
Greece		2,204		2,535		4,739
USSR	8,100	9,918	13,200	5,510	21,300	15,428
Yugoslavia		882		661		1,543
TOTAL	8,100	14,933	13,200	9,037	21,300	23,970
OCEANIA						
Australia	5,600	2,755	3,200	7,163	8,800	9,918
Indonesia	7,800	9,698	55,000	5,510	62,800	15,208
New Caledonia	15,000	14,877	31,000	33,060	46,000	47,937
Philippines	5,700	7,163	10,600	33,060	16,300	40,223
Other		551		3,747		4,298
TOTAL	34,100	35,044	99,800	82,540	133,900	117,584
AFRICA						
Botswana		551		--		551
Republic of South Africa		882		3,802		4,684
Rhodesia		661		5,675		6,337
Other (Burundi-Madagascar)		331		5,290		5,620
TOTAL	2,300	2,425	6,700	14,767	9,000	17,192
WORLD TOTAL	59,560	70,750	168,140	137,916	227,700	208,666
GEOPOLITICAL SUB-UNITS						
United States	200	220	14,900	300	15,100	500
Central Economy Countries	11,500	15,594	27,400	24,795	38,900	40,389
World exclusive of U.S.	59,360	70,530	153,240	137,616	212,600	208,166
World exclusive of Central Economy Countries	48,060	55,156	140,740	113,121	188,800	168,277

Table 4-4

ECONOMICALLY RECOVERABLE NICKEL
RESERVES AT THE END OF 1977
(short tons of metal content)

	TOTAL		SULPHIDE ORES		LATERITE ORES	
	Thousands of tons	Share (per- centage)	Thousands of tons a/	Share (per centage)	Thousands of tons a/	Share (per centage)
DEVELOPING COUNTRIES b/	43,584	61.6	1,212	5.7	42,372	85.8
DEVELOPED MARKET ECONOMIES	16,199	22.9	12,232	57.2	3,967	8.0
CENTRAL ECONOMY COUNTRIES c/	10,968	15.5	7,934	37.1	3,031	6.2
WORLD	70,748	100.0	21,379 (30)	100.0	49,370 (70)	100.0

Source: Annex I, Table 15, United Nations Committee on Natural Resources (1979).

a/ Figures in parentheses refer to percentages.

b/ Including Cuba.

c/ Excluding Cuba.

Of total estimated world nickel reserves, only 30 percent occur as sulfide ores; the remaining 70 percent are in laterites. With regard to total world nickel resources, the variance between the sulfide and lateritic components is even more pronounced. Less than 20 percent of world nickel resources exist as sulfide deposits.

From a geopolitical standpoint, approximately 20 percent of the world's nickel reserves occur in Central Economy Countries. Seventeen percent of all nickel resources are in Central Economy Countries. United States nickel reserves and, to some extent, resources are relatively small in comparison with world reserves and resources of this metal.

Although nickel deposits are widely distributed throughout the world, relatively few nations control much of the world's nickel reserve-resource base. (Table 4-3). Of total world nickel reserves, over 70 percent occur in just five nations (Canada, Indonesia, New Caledonia, the Philippines, and the U.S.S.R.). Similarly, nearly 70 percent of identified world nickel resources exclusive of reserves exist in four nations (Canada, Cuba, New Caledonia, and the Philippines). Whereas, Canadian and Soviet nickel reserves and resources are largely restricted to sulfide ores, the reserves and resources of Cuba, Indonesia, New Caledonia, and the Philippines are predominantly lateritic. Therefore, it is not surprising that much of the world's future nickel production will be derived from nickeliferous lateritic deposits.

4.2.2.2 Factors Important in Determining Future Nickel Production - Deposit Exploitation

As indicated in Table 4-3, differences of opinion exist regarding the quantities of contained nickel which presently can be developed economically. To some degree, the decision as to whether a particular nickel deposit should be classified as a reserve or a resource is subjective insofar as quite often, insufficient geological and other data are not available to enable an absolute determination. However, given information regarding the geology of a deposit, metallurgical considerations, legal, social and environmental factors, and financial requirements, it is possible to state with some degree of certainty whether a particular nickel occurrence can or will be brought into production in the foreseeable future. In the section which follows, the major nickel occurrences of those nations known or thought to have the largest such reserves-resources will be discussed in light of their geology, degree and/or likelihood of development, and factors which do and/or may limit future exploitation.

4.2.2.3 North America

4.2.2.3.1 United States

The nickel deposits of the United States are relatively limited in contrast to those of most other nations. As discussed in Section I, geologically significant domestic deposits of nickel (that is, those which contain sufficient amounts of extractable nickel and which are generally believed by most economic geologists to be potential sources of nickel over the near to long-term (1980-2010)), are found in essentially three areas:

- o The lateritic nickel-cobalt deposits of California and Oregon.
- o The nickel-copper sulfides of the Duluth Complex.
- o The nickeliferous and cobaltiferous lead-zinc deposits of the Missouri Lead Belt.

Although other nickel occurrences have been identified throughout the United States, only these three are of significantly large tonnage and/or high grade to warrant their consideration for future development given existing and projected advances in extractive technology.

Riddle, Oregon. The lateritic nickel-cobalt deposits of northern California and southern Oregon are of the nickel-silicate type. Although there have been and continue to be numerous investigations of the feasibility of exploiting these laterites, at only one location (Riddle, Oregon) is nickel presently being mined. At Riddle, the Hanna Mining Company mines by open pit methods the Nickel Mountain deposit--a 6,000-foot long by 3,500 foot wide "blanket" of nickeliferous laterite which overlies an unserpentinized peridotite. The thickness of this deposit varies; from several to as much as 200 feet. The nickeliferous layer occurs at an average depth of 50 to 60 feet below the surface and is reported to grade approximately 1.3 percent nickel.

Annual nickel production at the Nickel Mountain deposit is on the order of 12,000 to 13,000 tons per year in the form of ferro-nickel. About 90 percent of the nickel metal content of the mill-smelter feed is recovered as metallic nickel in ferro-nickel. According to USBM estimates, at the present rate of production the Nickel Mountain deposit will be mined out within ten to twelve years. However, the USBM understands that to extend the life of its Riddle operation, Hanna Mining Company not only has developed lower-grade reserves on the property, but has also acquired nickeliferous laterite deposits at Red Mountain, California (located approximately 6 miles east of Kalamath, California). The United States Bureau of Land Management (USBLM) has determined that the Red Mountain laterite deposit contains 50 million tons of nickel.

Gasquet Mountain Deposit, California. At Gasquet Mountain in northern California is a lateritic deposit of cobaltiferous nickel which contains 50 million tons of ore averaging 1 percent nickel and 0.05 percent cobalt. Given these grades and assuming complete recovery, approximately 500,000 tons of nickel and 25,000 tons of cobalt could be produced from this deposit.

In an attempt to exploit the Gasquet Mountain nickel-cobalt laterite, California Nickel Corporation and the USBM have developed a method of processing the low-grade ore from this property. Additionally, California Nickel Corporation has conducted an extensive investigation of the geology of, and mining methods required to exploit the deposit as well as the environmental problems likely to develop should production commence.

At present, California Nickel Corporation intends to begin construction of a mill in 1982 capable of producing as much as 15,000 tons of nickel and 750 tons of cobalt annually. However, any significant nickel-cobalt production from the Gasquet Mountain deposit is unlikely until the mid- to late 1980's.

Other Nickel Deposits

Duluth Complex, Minnesota. The Duluth Complex in northern Minnesota contains large quantities of copper, nickel, and cobalt. The Duluth Complex can be described best as a series of sheet-like intrusives into and beneath the Keweenawan volcanics. The majority of the copper-nickel-cobalt mineralization occurs in the lowermost several hundred feet of the intrusive as disseminated or massive sulfides primarily consisting of bornite, chalcopyrite, cubanite, pyrite, pyrrhotite and sphalerite. According to the Minnesota Department of Natural Resources (MDNR), the average grades of the disseminated mineralization (for the basis of reserve-resource estimation) is 0.66 percent copper, 0.20 percent nickel, and 0.01 percent cobalt. Assuming these grades, the Duluth Complex contains at least 29 million tons of copper (at a cutoff grade of 0.50 percent copper), 8.8 million tons of nickel, and 440,000 tons of cobalt.

Whether the Duluth Complex can be developed for its copper-nickel-cobalt content is uncertain. Although many mining companies have investigated the economic potential of the Duluth Complex, the environmental considerations attendant with its development are significant. This is in large part because the Duluth Complex is proximate to the Boundary Waters Canoe Area and the Superior National Forest.

According to recent statements by mining companies and state agencies, the Duluth Complex may be brought into production sometime after the mid-1980's. It is estimated that a single mining operation could produce 30,000 tons of nickel, 110,000 tons of copper and 500 tons of cobalt annually. (For a more detailed discussion of the economic geology of, and the environmental problems associated with the mining of the Duluth Complex, please refer to Section 1.2.3.1.4.)

Nickel Associated With Missouri Lead Belt Deposits. Significant concentrations of nickel (0.02 percent) and cobalt (0.015 percent) occur in association with lead-zinc mineralization in the Missouri Lead Belt. The nickeliferous deposits of the Missouri Lead Belt are somewhat anomalous insofar as they are neither lateritic orebodies nor hypogene deposits associated with mafic intrusive igneous rocks, as are most other nickel occurrences. Instead, the nickeliferous and cobaltiferous lead-zinc orebodies of the Missouri Lead Belt are best described as stratabound hydrothermal deposits. The highest cobalt-nickel grades to occur in those lead-zinc deposits confined to the lowermost part of the Bonneterre Dolomite, particularly where it transitions with the orthoquartzites and siltstones of the Lamotte Formation.

The Missouri Lead Belt is estimated to contain as much as 65,000 tons of nickel, much of which could be recovered if the ore could be ground finely enough to recover the host mineral siegenite from the chalcopyrite, dolomite, galena, and sphalerite. (For a more complete discussion of the problems encountered in the recovery of nickel from the ores of the Missouri Lead Belt, please refer to Section 1.2.3.1.1.)

Although the USBM has developed a process which permits the economic recovery of cobalt and nickel from the lead-zinc ores, it is not known whether any effort will be made by the mills of this district to recover the nickel content. Because of the recent and prolonged weakness in the nickel market, few lead-zinc producers have demonstrated any interest in recovery of nickel from their ores. The only apparent incentive to producers to extract nickel is its association with cobalt--a metal for which there is presently strong demand.

4.2.2.3.2 Canada

Canadian nickel reserves and resources, which occur as sulfide deposits primarily in Ontario, are among the largest occurrences of this type in the world. As indicated in Table 4-5, Canadian mine production of nickel over the past decade has been approximately 280,000 tons per year of contained nickel metal.

Table 4-5

NICKEL CONTENT OF ORES/CONCENTRATES: CANADA
(Thousand short tons)

1971	1972	1973	1974	1975	1976
294.2	259.0	274.4	296.4	266.7	288.7

Whether Canadian nickel production capacity will be increased in the near-to-intermediate future is contingent on a number of factors.

The majority of Canadian nickel mines are relatively deep underground operations, many of which have lower workings greater than 3,000 feet below the surface. Therefore, mining costs per ton of ore for Canadian nickel-copper sulfide deposits are considerably more than those experienced by operators of lateritic nickel orebodies. As such, from a mining standpoint, lateritic deposits have a cost advantage over deep sulfide deposits. However, largely offsetting this consideration is the fact that nickel-copper-sulfide ores require significantly less energy to beneficiate and process than do lateritic ores. A recent investigation of the energy inputs needed to process nickel ore determined that whereas sulfide ore required 10 KWH per pound of nickel produced, up to 38 KWH are required to process oxide material. Consequently, the possibility exists that given continuing increases in the cost of energy, sulfide copper-nickel deposits similar to those of the Sudbury district may become much more attractive exploration targets than lateritic nickel deposits. Should this be the case, the nickel production capacity of Canadian producers may be increased.

However, largely offsetting the competitive advantage which Canadian producers recently have achieved over foreign lateritic sources of supply is the fact that during the past few years, production of nickel has exceeded demand. Therefore, the market for nickel has not been sufficiently strong to warrant either the expansion of existing nickel production facilities or the development of new properties. Consequently, with few exceptions, most producers have "tabled" plans to bring new mine capacity on-line. Additionally, a number of nickel producers have placed previously active mines and mills on a standby basis to reduce excess production capacity.

Given this situation, there has been little incentive for Canadian, or for that matter, the nickel producers of any nation to explore for and/or develop new deposits.

According to a recent study conducted by the United Nations, of all nickel projects likely to be brought into production between 1981 and 1985, none are located in Canada. This projection is supported by USBM data which indicate that the development of any new nickel mines in Canada between 1981 and 1990 is unlikely.

Beyond 1990, insufficient data are presently available to make any firm predictions concerning the rates and/or timing of exploitation of deposits which have not yet been developed. Given a review of Canadian mining literature, it is possible that within the time frame of this study (1985-2010), any one of the deposits listed in Table 4-6 <u>might</u> be brought on stream (the assumption of course being that the economics of mine development will be favorable and that demand for nickel and copper will be sufficient to absorb their output).

Table 4-6
CANADIAN NICKEL DEPOSITS AWAITING DEVELOPMENT

Deposit	Geology	Tonnages & Grades	Work	Remarks
Quebec				
RAGLAN NICKEL DEPOSITS New Quebec Raglan Nickel M.L. 35 G/9 61°35.6'/74°17.5' (Cross Lake)	Nickel mineralization in Cape Smith-Wakeham Bay belt of ultrabasics and volcanics.	(i) Cross Lake: 5,800,000 ind. at 0.65% Cu, 1.44% Ni. (ii) C-1: 1,300,000 ind. at 0.96% Cu, 1.71% Ni. (iii) C-2: 2,700,000 ind. at 0.99% Cu, 1.65% Ni. (iv) Donaldson: 3,000,000 measured at 0.73% Cu, 3.06% Ni. (v) 3-Area: 1,100,000 ind. at 0.69% Cu, 2.80% Ni. (iv) 2-Area: 600,000 ind. at 0.72. Cu, 2,43% Ni. (vii) Katiniq: 5,300,000 ind. at 0.7% Cu, 2.4% Ni. (1970)	Surface drilling, some u/g development, bulk sampling and testing for several of the deposits.	Poor accessibility; some structural problems within certain deposits, some metallurgical difficulties all create impediments to exploitation at this time. Financial constraints of Falconbridge Nickel Mines may be involved.
Ontario				
NICKEL RIM MINE Nickel Rim M.L. 41 1/10 46°41'20"/80°48'10"	Po., pn., pc., and minor py in a series of irregular, disconnected bodies in the quartz diorite zone of the SE range of the Sudbury irruptive series.	Ind: 747,624 at 0.35% Cu, 0.90% Ni before dilution (1959)	Pre-1901: discovery, extensive stripping and trenching 1914-18: minor putting 1929-32: 265' shaft, 1 level, crosscutting, drifting, deep drilling, surface trenching, sampling, mag. survey 1942-47: surface & u/g drilling. "extensive" development. 1950-52: No. 2 shaft to 726', 3 levels, mill erection. 1952-58: production, shaft deepened to 1418', 4 additional levels.	Property east of Falconbridge. Company may now be controlled by Falconbridge Nickel M.L. Outlook for re-opening is dependent upon corporate priorities and is presently slim.

Table 4-6, continued
CANADIAN NICKEL DEPOSITS AWAITING DEVELOPMENT

Deposit	Geology	Tonnages & Grades	Work	Remarks
Ontario				
TEXMONT (FATIMA) Texmont M.L. 42 A/3 48°09.7'/81°12.5'	A steeply-dipping, partially serpentinized lenticular body of peridotite intrudes andesitic lavas to granite on the east, and iron formation and associated sediments on the west, all of which are intruded by later diabase and gabbro dykes. Sulphide mineralization, consisting of py., po., pn., minor violarite and cp., occurs within the peridotite mass in two principal zones.	ind: 3,800,000 at 1% Ni to 1600' (1967)	1951: prospecting, mapping, geophysics, drilling. 1956-59: property purchased by Fatima Mng. Co., drilling geochem. survey, 790' shaft, 5 levels. 1960: u/g development on two levels, drilling. 1965-66: Co. renamed Texmont M.L., Canico obtained 15% interest, surface drilling metallurgical tests by Mines Branch. 1970: property leased to Sheridan Geophysics. 1971-72: production, concentrates stockpiled.	Property 22 air miles S of Timmins. Concentrate refined by Sheridan Hydrometallurgical process, and some sales made to chemical industry of basic nickel sulphate. Mining operations suspended Dec/72 owing to "heavy tax on fuel oil" which raised costs sharply. Refinery continued operations processing stockpiled ore. In 1975, fuel oil tax was rescinded and assessment of re-opening mine is underway.
McWATTERS Coldstream M.L. 42 A/6 48°18'30"/81°02'10"	Sulphides occurring either as irregular veinlets, pods and disseminations or less commonly as massive zones in the central part of a zone of lenses of serpentinized periodotite-diorite intruding Keewatin mafic volcanics. Principal minerals are pyrite with pn. millerite and minor cp.	ind: (i) 477,768 at 0.73% Ni in Upper Zone. (ii) 165, 790 at 1.92% Ni in Lower Zone. (1964)	1961: mag, and EM surveys, drilling. 1962: drilling. 1964-65: drilling, metallurgical tests. 1967: deep penetration EM survey, drilling. 1970: property acquired by Tontine Mng. Ltd., which amalgamated with North Coldstream M.L. to form Coldstream M.L. in 1971. 1971: property optioned to Can. Jamieson M.L. which sold option to Sheridan Geophysics in 1972.	Property in Langmuir township Metallurgical tests indicated 85% recovery with a nickel concentrate averaging 15%. Development depends on corporate priorities of Sheridan Geophysics.

Table 4-6, continued
CANADIAN NICKEL DEPOSITS AWAITING DEVELOPMENT

Deposit	Geology	Tonnages & Grades	Work	Remarks
Ontario				
JUNEAU LAKE PROSPECT Coldstream M.L. Int. Mogul M.L. 42 L/5 50°2'/87°59'	N/A	Ind: 2,200,000 at 0.59% Cu, 0.87% Ni. (1969)	1968: mag. EM surveys, drilling. 1969: drilling.	Ore dressing research to improve nickel recovery required.
GREAT LAKES NICKEL Great Lakes Nickel Ltd. 52 A/4 48°04'40"/89°37'25"	Zoned, sill-like mass of gabbro and anorthositic gabbro. Middle zone of N arm of structure is anorthositic olivine gabbro with sygenetic sulphides (pn, po, cp) and diss. chromium spinel. Lower chill zone gabbro has diss. sulphide and mass. po stringers.	Ind: 32,800,000 at 0.36% Cu, 0.20% Ni. Inf: 40,000,000 at 0.36% Cu, 0.20% Ni (1974)	1952: discovery. 1954-67: drilling and other work. 1968: drilling and mine development (adits). 1970: feasibility study. 1972-74: activie development under agreement with Boliden Canada Ltd.	Boliden suspended operations Oct./74 pending clarification of financial obligations, corporate priorities, and governmental initiatives with regard to regional development.

Table 4-6, continued

CANADIAN NICKEL DEPOSITS AWAITING DEVELOPMENT

Deposit	Geology	Tonnages & Grades	Work	Remarks
Manitoba				
a) BOWDEN LAKE Falconbridge Nickel M.L. 63 J/15 50°55'40"/98°28'30"	Concordant series of ultrabasic units intrusive into paragneiss. Ultrabasics contain irregularly distributed nickel sulphides.	ind: 80,000,000 at 0.04% Cu, 0.6% Ni. (1971)	Regional geophysics, drilling, mineralogical work, metallurgical tests, feasibility study.	Development probably depends on corporate priorities. Fine-grained nature of deposit has created problems in the production of acceptable grade concentrate with low tailings loss.
b) BUCKO DEPOSIT	2 rock types: an anthophyllite-cummingtonite-biotite schist and serpentinite. Both types contain disseminated nickel sulphides.	ind: 30,000,000 at 0.04% Cu, 0.78% Ni. (1971)	Regional geophysics, drilling, u/g development, mineralogical work, metallurgical tests, feasibility study.	
HAMBONE DEPOSIT Inco 63 O/8 55°17'30"/98°20'25"	Sulphides occur within a large drag-fold within Precambrian granitic gneiss which is intruded by small serpentinite bodies.	ind: 3,600,000 at 0.81% Ni in one deposit and 1,200,000 at 1.10% Ni in two other zones. (1959?)	1955: staking. 1956-58: drilling, option to Maralgo M.L., geophysics, deep drilling. 1959: property sold to Inco.	Property 1/4 mi NE of the N end of Hambone Lake. Original claims adjoin those held by Canico. Inco corporate priorities will dictate development.
MYSTERY LAKE SOUTH Inco 63 P/13 55°49'15"/97°45'45"	Po. and pn. occurring as disseminations within serpertinized periodite lying mainly beneath the SW end of Mystery Lake.	ind: 250,000,000 at 0.6% Ni equivalent. (1952?)	1949: staking. 1950: option to Inco, additional staking, drilling. Company purchased deposit.	Property in the Thompson area. Development will be dictated by Inco corporate priorities.

Source: Energy, Mines and Resources Canada, MR 181, pages 20, 23, 26, 27, 31 and 33.

As is evident from foregoing discussion, Canadian nickel production capacity probably will not be increased significantly over the period 1985-2010. In the event that nickel demand does increase, Canadian producers will probably first bring into production mines and mills which were "mothballed" during the last five years before considering development of any new properties.

At present, INCO has nine underground mines and one open pit mine in operation in the Sudbury district. In addition to this production capacity, INCO also maintains four mines in the Sudbury district on a standby basis. To increase its ore reserves, INCO has initiated a deep drilling program at of all of its properties in the Sudbury Basin. According to INCO sources, plans are being made to extend active workings at its deepest mine, the Creighton, to 10,000 feet from 7,100 feet. In 1980, INCO plans to produce approximately 135,000 to 150,000 tons of nickel from its Sudbury operations. This is approximately 65 percent of INCO's total Ontario production capacity.

Falconbridge Nickel, which closed all but two of its mines (the Falconbridge and the Strathcona) during the recent production cutback, intends to have all of its other mines back into full operation within the next few years. By 1982, the company hopes to achieve an annual output of approximately 42,500 tons of nickel. (By way of comparison, in 1973, prior to the oversupply of nickel, Falconbridge produced nearly 50,000 tons of nickel.)

According to the Ontario Ministry of Natural Resources, Canadian mine production capacity of nickel by 1981 should be approximately 275,000 tons of nickel contained in ore per year. Should no new mines be brought into production until 2000, and assuming that expansions to existing production capacity are minimal, Canadian nickel production is unlikely to increase significantly over the next 20 years. Beyond that time, it is extremely difficult, if not impossible, to predict what expansions may occur or new discoveries may be made.

4.2.2.4 Australasia and Malaysia

The nickel reserves and resources of the nations comprising Australasia and Malaysia are restricted to lateritic deposits which, in many instances, are cobaltiferous. Table 4-7 lists the mine production of nickel in Australia, Indonesia, New Caledonia, and the Philippines in 1977.

Table 4-7

INSTALLED NICKEL PRODUCTION CAPACITY

(Thousands Tons Metal In Concentrates)

	1977
Australia	94.44
Indonesia	17.74
New Caledonia	127.28
Philippines	40.55
	280.01

Source: United Nations, 1979.

In most instances, the nickel output of these four countries is derived from relatively few mines. Interestingly, although these four countries presently account for nearly 40 percent of total world mine production of nickel, with the exception of New Caledonia, none of the other three nations had major operating nickel mines prior to 1967.

4.2.2.4.1 Australia

Australian nickel production is derived from both lateritic and sulfide nickel deposits. The following mines (Table 4-8) are Australia's major nickel producers:

Table 4-8

MAJOR AUSTRALIAN PRODUCERS OF NICKEL

Mine	Type of Deposit	Mining Method	Capacity per Year (Tons Ni contained in Concentrates)	Estimated Reserves
Greenvale, Queensland	laterite	open pit	26,400	40 million tons averaging 1.57% Ni 0.12% Co
Kambalda, Western Australia	sulfide	--	49,600	NA
Agnew, Western Australia	sulfide	--	10,000	50 million tons averaging 2.05% Ni
Spargoville, Western Australia	--	--	4,400	NA
Nepean, Western Australia	--	underground	3,300	NA
Windarra, Western Australia (on standby)	--	--	15,400	NA
Wingellina	--	--	--	
		TOTAL CAPACITY	109,100	

As far as can be determined, no additional nickel deposits in Australia are being planned for development in the next ten years. Several exploration projects are underway, however, which could lead to new producing properties--three of the most commonly mentioned being the Mt. Keith deposit, the Widgiemooltha sulfide nickel orebody, and the Rockhampton lateritic deposit.

4.2.2.4.2 Indonesia

Virtually all of Indonesia's nickel reserves and resources are contained in lateritic deposits. Although Indonesia has produced relatively small quantities of nickel over the past decade, with the inauguration of several new properties, the country will soon become one of the world's major nickel suppliers. Existing mine production is limited to two producers (Table 4-9).

Table 4-9

MAJOR INDONESIAN PRODUCERS OF NICKEL

Mine	Type of Deposit	Mining Method	Capacity per Year (Tons contained Nickel in Concentration)	Estimated Reserves
Aneka, Kendari, South Sulawesi	laterite	open pit	18,700	NA
PT INCO, Soroako, Central Sulawesi	laterite	open pit	17,600 Stage I 49,600 Stage II	250 million tons averaging 1.76% Ni
			36,300 - 68,300	

Only one deposit, that at Gag Island, is presently slated to be brought into production within the next ten years. The nickel laterite deposit at Gag Island contains approximately 200 million tons of ore averaging 1.4 percent nickel and 0.15 percent cobalt. Although Pacific Nikkel Indonesia originally planned to bring the Gag Island deposit into production in 1979 at an annual rate of 55,000 tons of nickel and 1,800 tons of cobalt, the project has been postponed until 1988 because of the recent depressed state of the world nickel market coupled with the high cost of financing.

Other deposits having potential as producers in the time frame 1985-2010 are the nickeliferous laterites on Obi and Gebe Islands (in the Halmahera group of islands between New Guinea and Borneo) which together contain 60 million tons of ore averaging 1.6 to 2.2 percent nickel. In the early 1970's, a Japanese consortium of mining companies planned to mine these deposits and construct a plant capable of producing 28,700 tons of nickel per year. However, the project was dropped due to projected extremely high development and operating costs.

4.2.2.4.3 New Caledonia

The lateritic nickel reserves and resources of New Caledonia are, according to the United Nations, larger than those of any other nation. New Caledonia nickel deposits are exploited by several open pit mines having a combined capacity estimated to be 150,000 tons of contained nickel per year. Actual mine production of nickel is variable and is contingent upon the world nickel market. Over one half of mine nickel production is processed at New Caledonia to nickel matte. The remaining production is converted into ferro-nickel. Whereas the matte is exported primarily to France and Japan, most of the ferro-nickel is sent to France, Japan, and the United States.

Two potential new producing properties are likely to come on-line over the next twenty years: the Tiebaghi deposit in northern New Caledonia and the Goro deposit in the southern part of the island.

Tiebaghi Deposit. The Tiebaghi deposit, estimated to contain 55 million tons of ore grading 2.5 percent nickel, is planned to be brought into production in 1990 at an initial rate of 29,000 tons of contained nickel per year. Ultimately, annual production at this property is slated to be increased to 35,000 tons of contained nickel in concentrates.

Goro Deposit. The Goro deposit is believed to contain 165 million tons of ore having an average grade of 1.6 percent nickel. In 1973, INCO submitted a proposal to BRGM (the French Bureau de Recherches Geologique et Minieres) to mine the low-grade deposit at an ultimate rate of 50,000 tons per year nickel and 1,500 to 3,000 tons per year cobalt. In 1977, INCO reached an agreement with BRGM wherein INCO will inform the French Government prior to June 1982 whether it plans to exploit the Goro deposit. It is understood that this decision will be based in large part on the results of ongoing geologic, mining, and metallurgical investigations which INCO presently is undertaking on the deposit. Assuming that INCO decides to develop this deposit, production is unlikely to begin prior to 1990.

4.2.2.4.4 Philippines

Philippine nickel reserves and resources are extensive. At present, Philippine nickel production capacity is approximately 47,000 tons of contained metal per year: 35,000 tons of which are possible from the Surigao open pit mine, Nonoc Island, Mindanao; 11,000 tons from the Rio Tuba mine at the southern end of Pulawan Island; and 1,100 tons from the Acoje mine in Zambales Province.

As far as can be determined, no other deposits are slated to be brought into production over the next decade. However, two deposits, one on Pulawan Island and the other in southern Mindanao, have been reported to contain nearly 2 million tons of ore averaging 1.5 percent copper and 1.1 million tons of ore grading 0.42 percent nickel. It has been estimated that approximately 19,000 tons of contained nickel could be produced from these deposits per year. No data are available concerning any plans to bring these deposits into production.

4.2.2.5 South and Central America

Nickel reserves and resources have been identified in Brazil, Colombia, Cuba, Dominican Republic, Guatemala, Puerto Rico, and Venezuela. Of these occurrences, the nickel deposits of Cuba and Brazil are the largest.

4.2.2.5.1 Brazil

Brazilian nickel reserves and resources, while not extensive in terms of total world reserves and resources, are significant enough to support several mining operations. At present, the principal nickel mines in Brazil are located 200 miles northwest of Rio de Janiero at Pratapolis, in the State of Minas Gerais. The annual capacity of this mine is approximately 3,300 tons of nickel contained in ferro-nickel. Plans have been made to increase the capacity of the mine to enable production of 5,500 tons per year nickel in ferro-nickel.

Several projects have been proposed to increase Brazilian nickel production. One of these, at Barro Alto in the State of Goias, will add 25,000 tons of contained nickel to total Brazilian output. The lateritic nickel deposit at Barro Alto is reported to contain 44 million tons of ore averaging 1.9 percent nickel. Although no date has been set for development of the Barro Alto deposit, production is unlikely to begin from this property prior to 1988.

Two other deposits, at Santa Fe in the state of Goias, and at Sao Joao do Piaui in the State of Piaui, have been identified. Whereas the Santa Fe deposit is said to contain reserves of nearly 20 million tons (no grade given), the Sao Joao deposit has identified reserves of 18 million tons grading 1.7 percent nickel. Although there have been no reports of pending development of the Santa Fe deposit, there are indications that the Sao Joao deposit may be brought into production at a rate of 11,000 tons contained nickel per year sometime in the future.

4.2.2.5.2 Colombia

As indicated on Table 4-3, Colombian nickel reserves and resources are relatively small. To date, no major production has come from Colombian deposits. However, in the State of Cordoba in northwest Colombia, the Cerro Matoso lateritic nickel deposit, which contains nearly 85 million tons of reserves averaging 1.5 to 2.6 percent nickel, is being prepared for development. When operations are initiated at the property in 1982, up to 24,000 tons of nickel contained in ferro-nickel will be produced annually. As far as can be determined, no other nickel properties in Colombia are pending development.

4.2.2.5.3 Cuba

The lateritic nickel reserves and resources of Cuba are among the largest in the world. Not suprisingly, installed Cuban nickel production capacity (estimated to be 41,000 to 44,000 tons contained nickel per year in 1979) is large by world standards. At present, Cuban nickel mines are located at Nicaro (on the northern coast of Oriente Province) and at Moa Bay (near the eastern tip of Cuba). At each of these locations is an extraction-processing plant capable of producing 21,000 tons of contained nickel in nickel oxide per year.

To increase nickel production capacity, Cuba has embarked on a major expansion program which includes:

- Expansion of each of the processing centers at Nicaro and Moa Bay by approximately 5,000 tons per year. Therefore, total combined production capacity of these two facilities should be 51,000 to 54,000 tons of contained nickel per year.

- Construction of a third mining and processing facility at Punta Gorda with a capacity of 33,000 tons of contained metal per year. Completion of this project is expected by 1982.

- Construction of a fourth, 33,000 ton contained-nickel processing plant at Moa Bay (the Las Comariocas unit). Completion of this facility is planned for 1984.

Should all of these properties be completed on schedule, Cuban nickel production capacity by 1985 could be as much as 119,000 tons per year of contained nickel.

In addition to these projects, Cuba is reported planning to construct a fifth processing unit having a capacity of approximately 46,000 tons per year contained metal. Assuming that this fifth, as yet unnamed unit is completed by 1990 (which is presently Cuba's target date), total Cuban nickel production by that year could be nearly 165,000 tons contained nickel annually.

4.2.2.5.4 Dominican Republic

By world comparison, the nickel reserves and resources of the Dominican Republic are relatively small. Nevertheless, since 1971, Falconbridge Nickel of Canada has operated an open pit nickel mine at Bonao having an annual capacity of 31,500 to 33,000 tons of nickel contained in ferro-nickel.

Because of rapidly rising fuel costs and the depressed nickel market, the Falconbridge mine has been operated over the past two years at only a fraction of its rated capacity. Despite the relatively rich grade of the ore (1.6 percent nickel), and large reserves (50 to 70 million tons), any near-future additions to existing production capacity are unlikely in the Dominican Republic.

4.2.2.5.5 Guatemala

As in the case in most other Latin American nations, the lateritic nickel reserves and resources of Guatemala are relatively small. However, despite a relatively limited reserve-resource base, Guatemala has the potential of becoming a major producer of nickel. At the Exmibal deposit near Lake Izabal, INCO Ltd. of Canada and Hanna Mining Company have recently begun mining nickeliferous laterites by open pit methods. It is planned that the Exmibal deposit, which contains approximately 55 million tons of nickeliferous laterite averaging 1.9 percent nickel, will be mined at a rate sufficient to provide feed for a process facility with a capacity of approximately 14,000 tons per year of nickel contained in matte. Full production capacity is not expected to be achieved until the early 1980's.

As far as can be determined, no additional nickel mines are being planned for Guatemala. However, there are indications that at some later date, production capacity at Exmibal may be increased--although to what level is not known at this time.

4.2.2.5.6 Puerto Rico

Two nickeliferous lateritic deposits have been discovered at Mayaguez in western Puerto Rico. Because the United States imports the majority of its nickel requirements, these deposits are of particular interest to domestic mining companies. Initial reports indicated that the Guanajibo and Las Mesas deposits together contain up to 100 million tons of ore averaging 0.88 percent nickel. Because of the relatively-low grade of the deposits, little action has been taken toward their development. However, the Ontario Ministry of Natural Resources reports that if exploited, these deposits could likely be mined at a rate of nearly 17,000 tons of contained nickel per year for as long as 33 years (the assumption being that the total reserves of both deposits are equivalent to approximately 550,000 tons of contained nickel). It is not known when, if ever, either of these deposits will be exploited.

4.2.2.5.7 Venezuela

Although Venezuela has three belts of serpentinized rocks which, in many instances, have been weathered to form nickeliferous laterites, only one deposit to date has attracted much interest by the nickel mining industry.

The Loma de Hierro deposit, located 100 miles southwest of Caracas, contains ore reserves estimated to be 60 million tons averaging 1.65 percent nickel. Although no date has been set regarding initiation of exploitation of this property, it has been estimated that up to 22,000 tons of nickel contained in ferro-nickel could be produced annually.

4.2.2.6 Europe-China

Approximately one-tenth of the world's nickel resources are located in Albania, China, Finland, Greece, the Soviet Union, Sweden, and Yugoslavia. Whereas the nickel reserves and resources of Finland and the Soviet Union occur predominantly in sulfide deposits, those of Albania, China, Greece, and Yugoslavia are primarily lateritic in nature.

4.2.2.6.1 Albania

The cobaltiferous lateritic nickel reserves and resources of Albania are small. However, mines at Kukes, Pogradec, and Korca have exploited Albanian nickel reserves for a number of years. The exact production capacity of Albania's nickel mines is unknown; nevertheless, over the period 1971-1976, annual nickel output is believed to be between 6,600 and 7,700 tons of contained nickel. It is reported that Albania has undertaken a program to nearly triple its nickel output through inauguration of a "highly mechanized" mine at Prenjas and Guri i kuq. When this increase in production will occur is unknown.

4.2.2.6.2 China

Very little is known of Chinese nickel reserves, resources, or production. Consequently, discussion of China's nickel production capacity is impossible.

4.2.2.6.3 Finland

Much of Finland's nickel reserves and resources occur in the Kotlahti, Hitura, and Vammala sulfide nickel-copper orebodies (Table 4-10).

Table 4-10

NICKEL DEPOSITS OF FINLAND

Deposit	Average Grade (Mill Grade) Nickel	Copper	Geology
Kotlahti (South Central Finland)	0.78	0.31	Kotlahti orebodies associated with complex basic and ultra-basic rocks. There are two major types of ore: disseminated and breccia. Pentlandite, chalcopyrite, and pyrrhotite are the sulfide minerals of economic importance.
Hitura (North Central Finland)	0.48	0.17	Hitura deposit consists of two parallel orebodies consisting of finely-disseminated pentlandite, pyrrohotite, and chalcopyrite at opposite contacts of a serpentinite intrusive with a micaceous gneiss.
Vammala (Southern Finland)	0.40	0.30	Vammala deposits occur in a serpentinized peridotite.

Total nickel production capacity of Outokumpu Oy (the state-controlled mining company) is unknown. However, based on previous levels of mine nickel output, the total capacity of all mines in Finland is probably approximately 9,000 to 10,000 tons of nickel in concentrates per year. Given the limited nickel resources of Finland, there is some question by industry experts whether this output can be maintained over the period 1990-2010.

4.2.2.6.4 Greece

The lateritic nickel reserves and resources of Greece are relatively high-grade. At some deposits, the ores contain as much as 3.9 percent nickel plus cobalt. Many of the lateritic nickel deposits of Greece are geologically unusual insofar as they consist of a relatively hard, moderately folded and faulted conglomeritic horizon which, in a number of localities, is overlain by limestone.

Greek laterite deposits are mined primarily by open pit methods. However, one deposit near Larynma has been mined by underground techniques. The largest deposit at Skalistiris-Kakavos is reported to contain approximately 500 million tons of reserves grading 1.0 to 1.3 percent nickel. Of total Greek mine production of nickel, 80 percent is obtained from this orebody. The remaining 20 percent of Greek nickel production is derived from the deposits in the vicinity of Larymna which average 1.0 to 1.6 percent nickel.

Greek nickel production capacity as of late 1978 is thought to be about 17,000 tons of contained nickel per year. There are reports that production capacity may be (or has already been) increased by as much as 13,000 tons per year in the near future. However, information is sketchy regarding whether this capacity has been installed.

4.2.2.6.5 Sweden

It is reported that large cobalt and platinium-bearing deposits of nickel have been discovered in northwest Sweden near Vaesterbotten Fells. According to representatives of Boliden Metal AB, the deposits contain "almost inexhaustable" reserves grading 0.2 percent nickel and 0.1 percent cobalt. Assuming that one open pit operation were to be established at an annual rate of 5.5 million tons, up to 8,300 tons of nickel and nearly 400 tons of cobalt could be produced per year. Boliden Metal AB is considering mining these deposits in the 1990's. However, given the low grade of the deposits and the high cost of development and energy, significant production is unlikely to occur until early 2000, if ever.

4.2.2.6.6 U.S.S.R.

Soviet centers of nickel mining and production can be broadly grouped into three geographic areas:

- o In the north-central part of the Soviet Union in the region surrounding the city of Noril'sk.
- o In the northern Kola Peninsula near the cities of Murmansk, Pechenga, and Monchegorsk.
- o In the southern Ural Mountains in the area adjacent to the town of Orsk.

Whereas nickel reserves and resources in the vicinity of Noril'sk and on the Kola Peninsula occur in sulfide-type deposits, the deposits in the southern Ural Mountains are largely lateritic. (For an expanded discussion of the Soviet nickel-cobalt mining industry, please refer to Section 1.2.3.8.)

Because of a lack of reliable information, it is difficult to establish with any degree of certainty present or future levels of Soviet nickel production capacity. However, the USBM estimates that in 1980, the Soviet Union will produce approximately 210,000 tons of nickel contained in concentrates. According to the Ontario Ministry of Natural Resources, the Soviets have initiated a plan to increase that country's nickel production capacity by 80 percent over 1975 levels. Toward this end, the capacity of the Noril'sk smelter-refinery complex is being expanded by as much as 110,000 tons of contained nickel. Whether the Soviets can achieve an almost doubling of nickel production capacity by 1980 is doubtful. Regardless, there is a good probability that within the next decade Soviet nickel production capacity will be considerably greater than at present levels.

4.2.2.6.7 Yugoslavia

Lateritic nickel deposits occur in southern Yugoslavia near its border with Greece. The deposit at Rzanovo-Kavadarci is estimated to contain approximately 26.4 million tons of ore grading 0.9 percent nickel. In addition, another 121 million tons of ore averaging 1 percent nickel, 2.03 percent chromium as Cr_2O_3, and minor amounts of cobalt also occurs at Rzanovo-Kavadarci. To exploit this deposit, an integrated mine and smelting project has been constructed which when fully operational in the early 1980's will be capable of annually producing not only 17,600 tons of nickel in ferro-nickel, but also 660,000 tons of iron concentrates (55 percent iron), and 2,600 tons of antimony metal in concentrates.

Other lateritic deposits have been discovered in southern Yugoslavia near Pristina which contain 27.5 million tons of ore reported to average 1.36 percent nickel. Although at one time these deposits were under consideration for development at a rate of 11,000 tons of nickel in ferro-nickel per year, plans have since been suspended. No additional information is available regarding a revised development schedule.

4.2.2.7 Africa

African nickel resources and reserves are not extensive. Of all the nations comprising the African continent, only three countries (Botswana, the Republic of South Africa, and Rhodesia) are thought to contain economically-significant nickel reserves and resources.

4.2.2.7.1 Botswana

All of Botswana's production of nickel is derived from the Selebi-Pikwe sulfide deposits which together contain nearly 50 million tons of ore having an average grade ranging from 0.7 to 1.45 percent nickel and 1.1 to 1.6 percent copper. The deposits are mined by both open pit and underground techniques. The initial capacity of the concentrator and smelting plant, which was started up in late 1973, is 46,000 tons of cobaltiferous copper-nickel matte per year containing 40 percent nickel, 37 percent copper and 0.66 percent cobalt. Given present rates of production, reported reserves are adequate to sustain existing capacity for about 30 years.

4.2.2.7.2 Republic Of South Africa

The Republic of South Africa has for many years been a major producer of nickel which is obtained largely as a coproduct-byproduct of platinum mining. As of mid-1977, South Africa had the capacity to produce over 24,000 tons of nickel per year from its major platinum mines. It is unlikely that nickel production from South Africa producers will be increased appreciably over the foresseable future.

4.2.2.7.3 Rhodesia

Rhodesia has been a substantial producer of nickel with production in 1977 being approximately 13,000 tons of contained nickel in concentrates. Nickel is derived from several mines near Bindura, Inyati, and Gatooma. Recent discoveries of nickel-bearing deposits have increased Rhodesian reserves and resources appreciably. It is estimated that within the next decade (particularly if peace is achieved throughout the country) nickel production capacity could double.

4.3 NICKEL DEMAND

4.3.1 Uses

Nickel is used primarily to add strength, toughness, and corrosion resistance to a variety of ferrous and non-ferrous metals. There are ten principal use categories for nickel. These are:

- o Stainless and Heat Resisting Steels
- o Alloy Steel
- o Superalloys
- o Nickel-Copper Alloys
- o Copper-Nickel Alloys
- o Permanent Magnet Alloys
- o Cast Iron
- o Electroplating
- o Chemicals and Chemical Uses
- o Other Uses

In the discussion which follows, each of these major use categories for nickel will be briefly described. This in turn will be followed by an analysis of estimates of nickel consumption and the assumptions implicit in each.

4.3.1.1 Stainless and Heat Resisting Steels

Stainless and heat resisting steels constitute the largest single use for nickel. According to the American Iron and Steel Institute, a stainless steel is any iron-based alloy which contains a minimum of 4 percent chromium with or without the addition of other elements. To increase toughness and corrosion resistance of stainless steel, nickel is often added and constitutes from 1.25

to 37 percent of the metal content. Typically, a stainless steel will contain:

66.35 to 70.85	percent iron
18.0 to 20.0	percent chromium
8.0 to 10.5	percent nickel
1.0	percent silicon (maximum)
2.0	percent manganese (maximum)
0.08	percent carbon (maximum)
0.03	percent sulfur (maximum)
0.045	percent phosphorus (maximum)

Stainless steels are most often used to form structural members in aircraft, trucks, and railroad cars where high strength is necessary. Stainless steels are also used where corrosion resistance is required such as in food processing applications, petroleum refining equipment, and for surgical instruments.

4.3.1.2 Alloy Steel

Nickel, as well as other alloying agents, are added to iron to form steels which are resistant to wear, cracking, and spalling. A typical nickel-bearing alloy steel contains:

95.15 to 96.2	percent iron
1.65 to 2.00	percent nickel
0.20 to 0.30	percent molybdenum
0.20 to 0.35	percent silicon
0.38 to 0.43	percent carbon
0.60 to 0.80	percent manganese
0.70 to 0.09	percent chromium
0.035	percent phosphorus (maximum)
0.04	percent sulfur

4.3.1.3 Superalloys

A superalloy is any metal developed for use in high-temperature applications. Superalloys are characterized by their high tensile and creep strength as well as their resistance to corrosion and oxidation at elevated temperatures (1,800°F or more) such as exist in jet engines. A typical superalloy will contain:

56.2 to 56.3	percent nickel
19.5	percent chromium
13.5	percent cobalt
4.3	percent molybdenum
3.0	percent titanium
2.0	percent iron (maximum)
1.3	percent aluminum
0.10	percent carbon (maximum)
0.001 to 0.1	boron

Most nickeliferous superalloys are used in turbosuperchargers and gas turbine engines.

4.3.1.4 Nickel-Copper Alloys

Nickel-copper alloys are those which contain more than 50 percent nickel. The Monel group of alloys are considered to be nickel-copper alloys. According to the USBM, Monel 400, a typical Monel alloy, contains approximately 66 percent nickel plus cobalt, and 31.5 percent copper.

The primary attribute of the nickel-copper alloys is their resistance to corrosion. Consequently, nickel-copper alloys are used extensively in water meters, pumps, propellers, condenser tubes and in various marine applications.

4.3.1.5 Copper-Nickel Alloys

Copper-nickel alloys are similar to nickel-copper alloys insofar as both are resistant to corrosion. Copper-nickel alloys include the cupronickels, nickel silvers, and nickel-bearing brasses and bronzes. A typical copper-nickel alloy contains:

87.6	percent copper
10.0	percent nickel
1.2	percent manganese
1.0	percent iron
0.1	percent carbon
0.1	percent silicon

Copper-nickel alloys are used primarily in piping, tubing, pumps, and valves for marine applications. A major use of copper-nickel alloys is in piping and equipment for desalination plants.

4.3.1.6 Permanent Magnet Alloys

Nickel is often used, along with various combinations of aluminum, cesium, chromium, cobalt, iron, manganese, molybdenum, platinum, samarium and vanadium to form magnet alloys. There are essentially two basic types of magnet alloys, termed soft and hard. Soft magnet alloys include those magnets which are comprised primarily of cobalt (34 to 50 percent cobalt) and iron (60 percent or more) with minor amounts of chromium, manganese and vanadium. Hard magnets are those which contain the aluminum-nickel-cobalt alloys with trace amounts of cesium, molybdenum, samarium, and platinum. A typical hard magnet alloy contains:

52.0 percent iron
24.0 percent cobalt
14.0 percent nickel
8.0 percent aluminum
3.0 percent iron

4.3.1.7 Cast Iron

The addition of nickel to cast iron improves the machineability and resistance to creep of the resultant metal. Through the addition of chromium and/or molybdenum, the strength and corrosion resistance of nickel cast iron is increased.

Nickel cast iron is used to manufacture cylinder blocks, pistons, industrial melting pots and molds.

4.3.1.8 Electroplating

To provide wear and corrosion resistance as well as an attractive appearance to the surface of other ferrous and non-ferrous metals, nickel is frequently applied through electroplating. The automobile industry is the biggest user of nickel plating, especially as an undercoating for automobile bumpers upon which chrome can be deposited.

4.3.1.9 Chemicals and Chemical Uses

Nickel is used in the manufacture of chemicals and catalysts, and in pigments, dyes, paint varnishes, and insecticides. Nickel and nickel salts are used in the hydrogenation of fats and oils as well as to arrest fermentation.

4.3.1.10 Other Uses

Nickel is used in the manufacture of batteries (particularly of the nickel-cadmium type) and fuel cells. Pure nickel is used by many countries in the minting of coinage.

4.3.2 Substitute - Alternative Materials

Depending on the corrosion, high or low-temperature requirements, or other properties needed for a part or fabrication, alternative materials are available which can be used in place of nickel in almost all applications. However, with few exceptions, use of the alternative material(s) would very likely lead to increased production costs and, at the same time, would not necessarily provide all of the qualities which nickel imparts.

Aluminum, cobalt, columbium, molybdenum, silicon, tantalum, titanium, tungsten, and vanadium can be used to some extent in place of nickel to add strength to, and increase the resistance of metal to oxidation at high temperatures and to improve creep resistance. However, in many instances, the addition or substitution of these elements does not always serve to replace nickel completely, but only to enhance its properties. Consequently, to produce stainless steel, the quantity of nickel required can only be reduced with a consequent increase in other alloy elements. It is noteworthy that for many applications, other alloy steels, plastics and plastic coatings, and alternative corrosion resistant metal claddings can be used in place of stainless steel with equal success.

Cobalt, chromium, columbium, molybdenum, vanadium, and various ceramic materials can be substituted to some extent for nickel in the production of a select number of high-nickel or nickel containing ferrous-based superalloys.

Copper, manganese, and molybdenum can be used in place of nickel in some types of iron casting. Cobalt, copper, and platinum can be substituted for nickel in a number of its uses as a catalyst.

With regard to nickel use in the manufacture of high and ultra-strength steels, overall demand for nickel has been decreasing due to advances in alloy development and improvements in heat treatment techniques and equipment.

To summarize, substitute materials do exist for nickel in most applications. However, the substitution of alternative metals, plastics, and ceramics for nickel will occur only where the replacement material is technically and economically satisfactory. For most applications, the cost and quantities required of the alternative material are greater than those incurred if nickel were used.

4.3.3 Secondary Sources - Recycling

A major portion of world nickel supply (approximately 19 percent in 1977) is derived from nickel scrap generated during the forming and shaping of fabricated products. As indicated in Table 4-11, nickel scrap provided one-fourth of total United States supply in 1977.

There are various types of nickel scrap. Home or runaround scrap is produced by steel mills, non-ferrous smelting and refining plants, and foundaries during the forging, forming, shearing and shaping of nickel-containing metals. This scrap is generally recycled within the fabricating facility and does not normally reach the outside market. In the making and working of many high-nickel and stainless steel alloys, up to 80 percent of the metal originally contained in the ingot input of the item becomes scrap.

Prompt scrap is generated by the steel or alloy consumer in the production of fabrications, forgings, castings and powder metal products. This scrap most often is sold to steel mills, smelters, and refiners by the manufacturer.

Table 4-11

NICKEL SUPPLY-DEMAND RELATIONSHIPS, 1968-1977

(thousand tons)

	1968	1969	1970	1971	1972	1973	1974	1975	1976	1977
World mine production:										
United States[1]	15.2	15.8	15.6	15.6	15.7	13.9	14.1	14.3	13.9	13.7
Rest of World	532.9	516.9	676.8	685.0	666.2	707.7	809.1	883.3	872.4	837.3
Total	548.1	532.7	692.4	700.6	681.9	721.6	823.2	897.6	886.2	851.0
Components of U.S. supply:										
Domestic mines	15.2	15.8	15.6	15.6	15.7	13.9	14.1	14.3	13.9	13.7
Secondary	36.6	71.0	48.7	63.1	67.5	65.9	64.5	41.6	46.6	50.0
Shipments of Government stockpile excesses	3.2	4.3	2.1	14.9	1.8	1.0	4.6	0.4	0.5	0.5
Imports	148.0	129.3	156.3	142.2	173.9	191.1	220.7	162.3	188.6	185.4
Industry stock, Jan. 1	39.6	37.2	31.9	24.7	57.3	77.9	71.3	87.3	102.4	121.0
Total U.S. supply	242.6	257.6	254.6	260.5	316.2	349.8	375.2	305.9	362.0	370.6
Distribution of U.S. supply:										
Industry stock, Dec. 31	37.3	31.9	24.7	57.3	77.9	71.3	87.3	102.4	121.0	145.0
Exports	6.5	2.3	6.5	4.6	3.0	5.0	4.3	7.4	15.8	16.0
Industrial demand	196.8	223.4	223.4	196.6	235.3	273.5	283.6	196.1	213.5	209.6
U.S. demand pattern:										
Chemicals	22.4	34.3	33.4	29.7	35.2	41.2	43.0	28.0	29.9	29.4
Petroleum	12.4	17.3	17.8	17.9	21.3	24.7	26.1	18.5	19.2	18.8
Fabricated metal products	25.0	18.7	21.3	20.3	23.6	27.5	26.1	17.3	19.2	18.8
Transportation:										
Aircraft	24.3	14.9	13.4	13.9	16.2	19.2	17.0	11.9	18.0	17.7
Motor vehicles and equipment	22.1	23.4	26.7	21.8	26.4	30.2	29.4	21.5	22.5	22.1
Ship and boat building and repairs	9.2	8.6	6.6	6.1	6.8	8.4	13.4	10.2	8.6	8.4
Total	55.6	46.9	46.7	41.8	49.4	57.8	59.8	43.6	49.1	48.2
Electrical	20.7	30.0	28.8	25.7	30.6	35.6	34.4	25.6	27.8	27.3
Household appliances	18.7	13.7	14.5	13.6	16.5	19.2	21.7	12.7	14.9	14.6
Machinery	13.1	16.3	16.5	13.7	16.5	19.2	23.0	16.4	17.1	16.8
Construction	10.8	16.1	21.3	17.9	21.3	24.7	28.5	18.4	19.2	18.8
Other	20.1	30.1	23.1	18.0	20.9	23.6	21.0	15.6	17.1	16.9
Total industrial demand	196.8	223.4	223.4	196.6	235.3	273.5	283.6	196.1	213.5	209.6
Total U.S. primary demand[2]	162.2	152.4	174.7	135.5	167.8	207.6	219.1	154.5	166.9	159.6

1 Refined metal from domestic ores.
2 Industrial demand less secondary.
Source: USBM Mineral Commodity Profiles: Nickel (Update of July 1977 Issue), 1979, p. 11.

Obsolete or old scrap is generated through the process of discarding or replacement of consumer goods, machinery, and transporation equipment.

Because of the refractory nature of many high-nickel bearing and/or nickeliferous superalloys, reclamation of the various alloying elements contained in the scrap is either technically difficult or economically prohibitive. Consequently, unless the exact composition of a high-nickel alloy is known, neither it nor the scrap generated in its production can be recycled within the United States. Instead, the scrap must be shipped to special refineries in Japan and Europe (principally West Germany).

According to projections of future demand for nickel made by the USBM (Table 4-12), nickel scrap will play an increasingly important role in satisfying domestic demand for nickel. The USBM estimates that by the year 2000 up to 28 percent of probable United States demand for nickel will be met by domestically produced nickel-bearing scrap. On a world exclusive of the United States basis, the USBM predicts that the amount of total demand satisfied by scrap will increase from 18 percent to 20 percent in 2000.

4.3.4 Projections of Nickel Demand

Demand for nickel is responsive to general economic conditions. As indicated in Figure 4-6, during periods of international tension such as World Wars I and II and the Korean and Vietnam conflicts, demand for nickel has escalated. Conversely, demand for nickel decreases during recessionary periods such as have occurred in the 1930's, late 1940's, and middle to late 1970's. Because of nickel's apparent sensitivity to world economic conditions, and given the five to seven year lead time required to develop a nickel property, it is not surprising that periods of nickel oversupply develop on a somewhat cyclical basis. According to the Ontario Ministry of Natural Resources, demand for nickel tends to increase and decrease in response to five to seven year cycles. This observation appears to have merit not only insofar as the most recent period of nickel oversupply occurred during the period

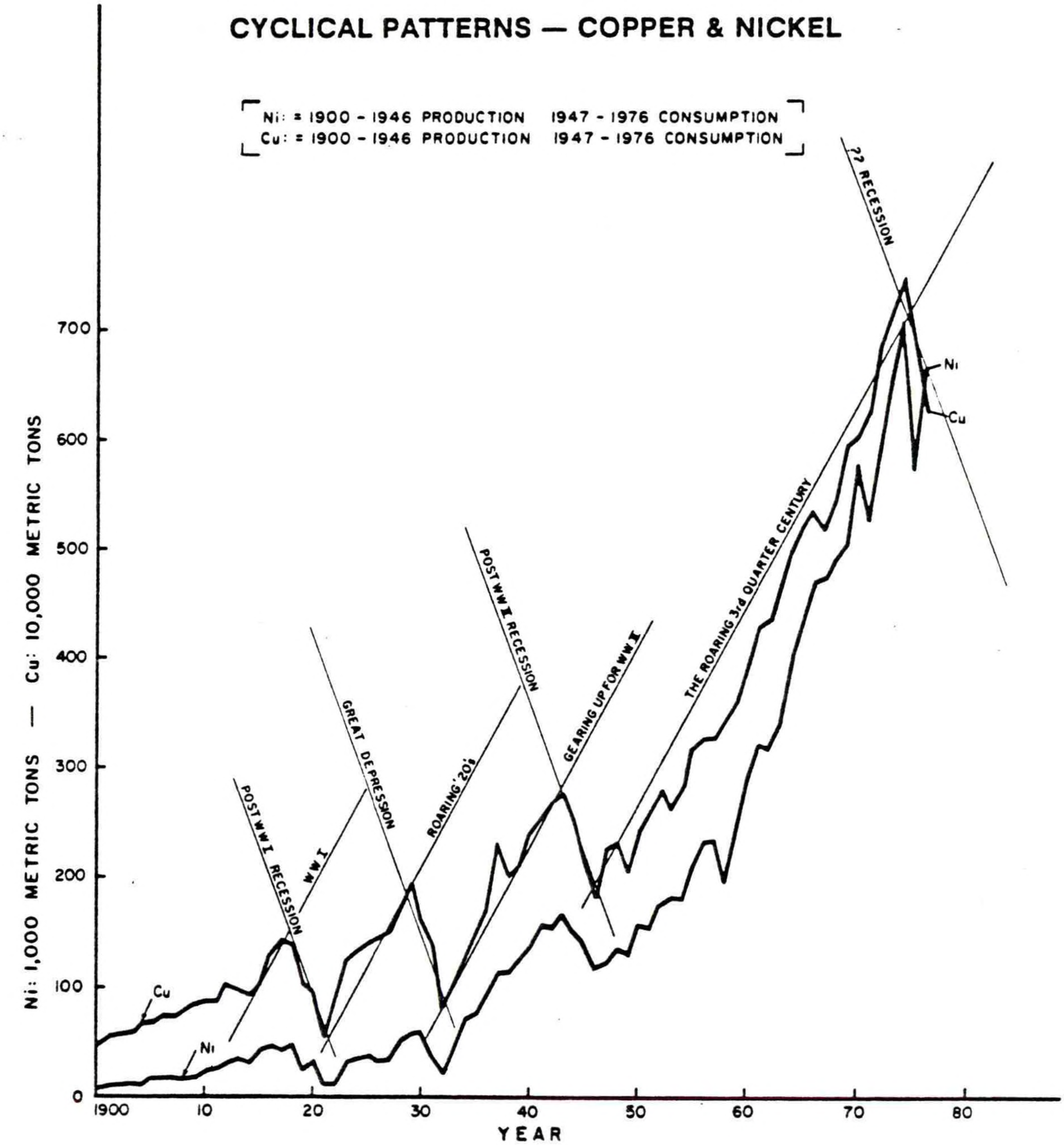
Table 4-12

PROJECTED DEMAND FOR NICKEL AS DETERMINED BY THE UNITED STATES BUREAU OF MINES

	Base Year 1977	Annual Rate of Growth	1980	1985	1990	1995	2000	2005	2010
REST OF WORLD	(Probable)								
Primary	551	@3.6% (given)	612.7	731.2	872.6	1041.4	1242.9	1483.3	1770.2
Primary	551	@3.64% (actual)	613.5	733.7	877.4	1049.4	1255.0	1500.9	1795.0
Secondary	124	@4.1%	139.9	171.0	209.1	255.6	312.5	382.0	467.0
Secondary	124	@4.14%	140.1	171.5	210.1	257.4	315.2	386.1	472.9
TOTAL	675	@3.70%	752.7	902.7	1082.5	1298.1	1555.4	1866.9	2238.7
TOTAL	675	@3.74%	753.6	905.4	1087.8	1306.9	1570.2	1886.5	2266.6
	(HIGH)								
Primary	551	@4.45%	627.9	780.6	970.5	1206.5	1500.0	1864.9	2318.4
Secondary	124	@5.34%	144.9	188.0	243.8	316.1	410.0	531.7	689.6
TOTAL	675	@4.63%	773.1	969.2	1215.2	1523.5	1910.0	2394.6	3002.2
	(LOW)								
Primary	551	@2.58%	594.8	675.6	767.3	871.6	990.0	1124.5	1277.3
Secondary	124	@2.10%	132.0	146.4	162.5	180.3	200.0	221.9	246.2
TOTAL	675	@2.50%	726.8	822.2	930.0	1052.0	1190.0	1346.1	1522.7
WORLD	(Probable)								
Primary	747	@3.53%	829.0	986.2	1173.3	1395.7	1660.4	1975.3	2349.8
Secondary	176	@4.37%	200.1	247.7	306.7	379.8	470.2	592.2	720.8
TOTAL	923	@3.70%	1029.4	1234.7	1481.0	1776.3	2130.6	2555.5	3065.2
	(HIGH)								
Primary	747	@4.26%	846.6	1043.0	1284.9	1582.9	1950.0	2402.3	2959.5
Secondary	176	@5.48%	206.5	269.6	352.0	459.6	600.0	783.3	1022.7
TOTAL	923	@4.52%	1053.8	1314.4	1639.3	2044.6	2550.0	3180.4	3966.7
	(LOW)								
Primary	747	@2.40%	802.2	903.3	1017.3	1145.5	1290.0	1452.7	1635.9
Secondary	176	@2.62%	190.3	216.7	246.8	281.0	320.0	364.4	415.0
TOTAL	923	@2.45%	992.5	1120.1	1264.1	1426.6	1610.0	1817.0	2050.6

From: Scott Sibley, 28 December 1979, USBM, personal communication.
Norm Matthews, 28 December 1979, USBM, personal communication.

Figure 4-6



Source: Ontario Ministry of Natural Resources, Mineral Policy Background Paper No. 4, 1977, p. 17.

1975-1980 (a five year time span), but also in light of length of lead time necessary to bring a deposit on-line.

Numerous projections of world nickel demand have been developed. However, given the advantage of hindsight, very few forecasters evidently have been able to predict with any degree of accuracy nickel demand several years, much less a decade or more, into the future. Consequently, because it is difficult to establish future demand for nickel with any degree of certainty, and insofar as the results of such efforts are often unreliable, the reader is cautioned to regard the nickel demand projections presented in this section as merely indicative of possible levels of metal consumption and not as a definitive guide to future demand.

In general, most projections of nickel demand are developed for relatively short periods of time (seldom exceeding twenty years), and are based in large part on historical annual rates of growth which are often tempered to account for factors determined to impact on consumption. Table 4-13 gives the historical rates of world annual nickel demand for a select number of periods.

Table 4-13

Period	Growth Rate (percentage)
1925-1929 to 1948-1950	10.7
1948-1950 to 1953-1955	8.0
1953-1955 to 1963-1965	5.4
1954 to	7.0
1954 to	6.1
1956 to 1971-1975	6.5
1967 to 1977	3.2

Source: Malenbaum, 1978; United Nations, 1979.

As indicated in Table 4-13, the annual rate of growth in world nickel demand since 1925 has decreased from nearly 11 percent to slightly more than 3 percent. Nevertheless, despite this reduction in the annual rate of overall world growth in demand, many geographic regions have, and continue to exceed the growth rates for the periods indicated (Table 4-14).

Table 4-14

Region	Growth Rate 1951-1955 to 1971-1975
Western Europe	6.0
Japan	16.7
Other Developed Lands:	
Australia Canada Israel New Zealand Republic of South Africa	8.5
U.S.S.R., China & Eastern European Nations[1]	7.0
Africa (less the Republic of South Africa)	19.8
Asia (less Israel, Japan, and Mainland China)	13.1
Latin America	15.8
United States	4.3
World[1]	6.5

[1] For period 1956-1975.

Source: Malenbaum, 1978, p.85.

Given the data presented in Table 4-13 and 4-14, it is not surprising that metals analysts have had difficulty projecting future demand for nickel--particularly if historical rates of growth are used as a basis for new estimates.

As stipulated by the United States Department of Commerce in its request for proposal, USBM projections of United States, market economy, and total world demand for nickel are to be used in this investigation as a primary source of data concerning future levels of nickel consumption. However, insofar as many projections of nickel demand recently have been made by various groups, and given the fact that USBM estimates are for the United States and world only, other demand projections have been incorporated in developing the proceeding analysis.

Table 4-15 gives twelve projections of nickel demand for the United States, world, and central economy countries over the period 1980-2010. Although many projections were reviewed prior to the completion of Table 4-15, four principal sources of data were selected based on similarity to other estimates and completeness in documentation-methodology. These were:

UNITED STATES: United States Bureau of Mines Mineral Commodity Profile: Nickel, May, 1979.

WORLD: Ontario Ministry of Natural Resources Mineral Policy Background Paper No. 4, Towards A Nickel Policy For the Province of Ontario, December, 1977.

Malenbaum, Wilfred, World Demand for Raw Materials In 1985 and 2000, 1978.

United Nations Committee on Natural Resources, Prospects For The Development of the Raw Materials Base For The Nickel Industry and The Demand for Nickel Over the Next 10-15 Years, March, 1979.

Table 4-15

PRIMARY NICKEL DEMAND DOCUMENTATION
(thousand tons nickel)

	Base Year		Annual Rate of Growth (%)	1980	1985	1990	1995	2000	2005	2010
UNITED STATES										
Low	1977 TL	196	1.87	207.0	227.3	249.4	273.6	300.1	329.3	361.2
Medium	1977 TL	196	3.21	215.5	255.0	295.6	346.1	405.4	474.8	556.0
High	1977 TL	196	3.68	218.4	261.7	313.5	375.6	450.0	539.2	645.9
From: U. S. Bureau of Mines, Mineral Commodity Profile: Nickel, 1979, p. 15.										
WORLD										
USBM	1977 TL	747	2.40	802.2	903.3	1017.3	1145.5	1290.0	1452.7	1635.9
		UN	Value							
Low	1976	738	2.21	805.4	898.3	1001.9	1117.5	1246.4	1390.2	1550.6
@ 20% recycle		—	—	644.3	718.6	801.5	894.0	997.1	1112.2	1240.5
From: Ministry of Natural Resources, Ontario (Canada), Mineral Policy Background Paper No. 4, 1977, p. 27.										
USBM	1977 TL	747	3.53	829.0	986.2	1173.3	1395.7	1660.4	1975.3	2349.8
		UN	Value							
Medium	1972.5	681	2.78	836.6	959.6	1100.6	1262.4	1448.0	1660.1	1905.1
@ 20% recycle		—	2.78	669.3	767.7	880.5	1009.9	1158.4	1328.1	1524.1
From: Malenbaum, Wilfred, World Demand for Raw Materials in 1985 and 2000, 1978, pp. 73, 85, 67.										
USBM	1977 TL	747	4.26	846.6	1043.0	1284.9	1382.9	1950.0	2402.3	2959.5
		UN	Value							
High	1974	783.2	3.85	982.3	1186.2	1432.6	1730.1	2089.5	2523.4	3047.5
@ 20% recycle		—	3.85	785.8	949.0	1146.1	1384.1	1671.6	2018.7	2438.0
From: United Nations Committee on Natural Resources: Nickel Study, 1979, p. 5.										
CENTRAL ECONOMY COUNTRIES										
		UN	Value							
Low	1976	194.7	2.8	217.4	249.6	286.6	329.0	377.8	433.7	497.9
@ 20% recycle		—	2.8	173.9	199.7	229.3	263.2	302.2	347.0	398.3
From: United Nations Committee on Natural Resources: Nickel Study, 1979, p. 17.										
		UN	Value							
Medium	1976	194.7	3.3	221.7	260.8	306.7	360.8	424.4	499.2	587.2
@ 20% recycle		—	3.3	177.4	208.6	245.4	288.6	340.0	399.4	469.8
From: United Nations Committee of Natural Resources, Nickel Study, 1979, p. 17.										
		UN	Value							
High	1976	194.7	3.8	226.0	272.4	328.2	395.5	476.5	574.2	692.0
@ 20% recycle		—	3.8	180.8	217.9	262.6	316.4	381.2	459.4	553.6
From: United Nations Committee of Natural Resources, Nickel Study, 1979, p. 17.										

TL = Trend Line Value

CENTRAL ECONOMY

COUNTRIES: United Nations Committee on Natural Resources, *Prospects For The Development of the Raw Materials Base for the Nickel Industry and The Demand For Nickel Over the Next 10-15 Years*, March, 1979.

With the exception of USBM data, the majority of the projections selected were for nickel demand inclusive of both primary and secondary metal consumption. Therefore, before any comparisons could be made between primary metal supply and demand, that part of the projected metal demand which would likely be satisfied by recycled scrap must be factored out. As discussed previously in this section, secondary sources of supply accounted for approximately 19 percent and 24 percent of United States and world nickel supply in 1977. Estimates by the USBM indicate that by 2000, nickel scrap will be used to satisfy 28 percent of United States demand and 22 percent of world demand (inclusive of the United States). Given these data, and in light of the uncertainties of the future projections, it is conservatively assumed that on an overall basis through the period 1980-2010, approximately 20 percent of world nickel demand will be satisfied by secondary scrap. Therefore, in keeping with this assumption, all demand estimates presented in Table 4-15 have been reduced by that amount in order to obtain an idea of possible primary metal demand.

Tables 4-16, 4-17, and 4-18 present the projected supply of, and demand for nickel by the United States, central economy countries, market economy, and world for the period 1980-2010. From this table, the following observations can be made:

- At no time during the period 1980-2010 is projected United States supply-production capacity of nickel expected to equal estimated domestic demand for that metal. Assuming an annual maximum installed production capacity in 2000 of 59,000 tons of metal, estimated low demand for that year is likely to be over five times available supply.

Table 4-16

PROJECTED MINE NICKEL PRODUCTION CAPACITY

	Actual Mine Production 1977	1980	1985	1990	1995	2000	2005	2010
UNITED STATES	13,114	13,000	14,000	29,000	44,000	59,000	59,000	59,000
ALBANIA	7,714	7,000	7,000	14,000	14,000	21,000	21,000	21,000
AUSTRALIA	90,915	109,100	109,100	109,100	109,100	109,100	109,100	109,100
BOTSWANA	13,885	20,000	20,000	20,000	20,000	20,000	--	--
BRAZIL	5,841	3,300	5,500	30,500	30,500	41,500	41,500	41,500
CANADA	265,362	275,000	275,000	275,000	275,000	275,000	275,000	275,000
CHINA	NA	NA	NA	NA	NA	NA	NA	NA
COLOMBIA	--	--	24,000	24,000	24,000	24,000	24,000	24,000
CUBA	40,554	42,500	52,500	118,500	164,500	164,500	164,500	164,500
DOMINICAN REPUBLIC	26,999	32,250	32,250	32,250	32,250	32,250	32,250	32,250
FINLAND	7,053	9,500	9,500	9,500	7,500	6,000	6,000	6,000
GREECE	18,073	17,000	17,000	30,000	30,000	30,000	30,000	30,000
GUATEMALA	--	--	14,000	14,000	14,000	14,000	14,000	14,000
INDONESIA	15,208	36,300	36,300	123,300	123,300	138,300	138,300	138,300
NEW CALEDONIA	131,028	150,000	150,000	179,000	179,000	185,000	185,000	185,000
PHILIPPINES	16,750	47,000	47,000	47,000	47,000	47,000	47,000	47,000
PUERTO RICO	--	--	--	--	--	--		
REPUBLIC OF SOUTH AFRICA	24,685	24,000	24,000	24,000	24,000	24,000	24,000	24,000
RHODESIA	17,852 (1)	13,000	13,000	26,000	26,000	26,000	26,000	26,000
USSR	148,770	185,000	210,000	265,000	320,000	320,000	320,000	320,000
VENEZUELA	--	--	--	--	22,000	22,000	22,000	22,000
YUGOSLAVIA	--	--	17,600	17,600	17,600	17,600	11,000	11,000
TOTALS	857,360	983,950	1,077,750	1,387,750	1,523,750	1,576,250	1,549,650	1,549,650

(1) Includes other African nations.

Source: Dames & Moore, 1980

Table 4-17

ASSUMPTIONS TO PROJECTED MINE NICKEL SUPPLY TABLE

United States - Assumes that Riddle, Oregon, operation will continue at full production capacity of 12,000-13,000 to 2000 + through utilization of Red Mountain ores.

Assumes Gasquet Mountain production at 15,000 tons of nickel per year commencing prior to 1990.

Assumes that ultimate Duluth Complex production at 30,000 tons of nickel per year will commence prior to 2000. Initial production will be on-line prior to 1995.

Assumes that 1,000 tons of nickel per year will be derived from Missouri Lead Belt commencing prior to 1985.

Albania - Assumes continuation of present capacity of 6,600-7,700 tons of nickel per year until 1990, at which time capacity will be reduced.

Australia - Assumes continuation of present capacity of 109,000 tons through 1990; then possible addition of new capacity will offset depletion reserves.

Brazil - Assumes that by 1985, production capacity of Pratapolis deposit will be increased to 5,500 tons; by 1990, Barro Alto deposit will be inaugurated at a rate of 25,000 tons per year; by 2000, either the Sao Joao or Santa Fe deposits will come on-line at a capacity of 11,000 tons per year.

Botswana - Assumes maximum production capacity at Selebi-Pikwe mine of 20,000 tons of nickel annually for a period of 20 or more years.

Canada - Assumes no major additions to capacity through 2000, new capacity will largely offset depleted deposits.

China - No data available to make reliable estimates.

Colombia - Assumes that Cerro Matoso deposit will be brought on-line in 1982 as planned at a production rate of 24,000 tons per year.

Table 4-17 (Continued)

ASSUMPTIONS TO PROJECTED MINE NICKEL SUPPLY TABLE

Cuba - Assumes that expansions to Nicaro and Moa Bay processing facilities will come on-stream by 1985 as planned--51,000 to 54,000 tons of contained nickel per year, respectively; assumes that Punta Gorda facility will not achieve full rated capacity until after 1985; assumes that fourth processing plant will not reach full capacity until shortly before 1990; assumes that fifth processing plant of estimated 46,000 tons wil come on-stream after 1990, but prior to 1995.

Dominican Republic - Assumes no new production aside from existing capacity.

Finland - Assumes no new production aside from existing capacity.

Greece - Assumes continuation of present production capacity at 17,000 tons of nickel per year until late 1980's, at which time production increases by 13,000 tons per year.

Guatemala - Assumes that full production at Exmibal mine not achievable until early 1980's.

Indonesia - Assumes continuance of present annual production capacity of 36,300 tons of nickel until late 1980's, at which time production will be increased to 49,600 tons of nickel; that Gag Island deposit will be brought on-line at a rate of 55,000 tons of nickel per year prior to 2000; that at least one of the deposits at Obi and Gebe will be brought on-line at a rate of 15,000 tons per year.

New Caledonia - Assumes that present capacity of 150,000 tons contained nickel per year can be maintained; that Tiebaghi deposit will be brought into production by 1990 at an initial rate of 29,000 tons of contained nickel annually--to be increased to 35,000 tons of contained nickel by 2000. Provision is made for development of Goro deposit.

Philippines - Assumes present capacity of 47,000 tons of contained nickel per year; no assumption made for development of deposits at Pulawan Island.

Table 4-17 (continued)

ASSUMPTIONS TO PROJECTED MINE NICKEL SUPPLY TABLE

Puerto Rico - Assumes no development of deposits near Mayaguez.

Republic of South Africa - Assumes continued production at 24,000 tons contained nickel per year.

Rhodesia - Assumes present mine output will remain constant until 1980's, at which time new capacity will come on-line.

U.S.S.R. - Assumes that plans to increase nickel production to 210,000 tons in 1980 will be delayed by several years, capacity not being reached until early 1980's; assumes 4 percent yearly increase in capacity between 1985-1995.

Venezuela - Assumes that Loma de Hierro deposit will come on-line in early 1980's at a rate of 22,000 tons per year contained nickel.

Yugoslavia - Assumes no increae in installed capacity over next 20 years.

Table 4-18

PRIMARY NICKEL SUPPLY - DEMAND
(thousand tons)

		Annual Rate of Growth 1980-2010	1980	1985	1990	1995	2000	2005	2010
UNITED STATES									
Supply		5.17	13.0	14.0	29.0	44.0	59.0	59.0	59.0
Demand	1) Low	1.87	207.0	227.3	249.4	273.6	300.1	329.3	361.2
	2) Medium	3.21	215.5	252.4	295.6	346.1	405.4	474.8	556.0
	3) High	3.68	218.4	261.7	313.5	376.6	450.0	539.2	645.9
WORLD									
Supply		1.53	984.0	1077.8	1387.8	1523.8	1576.3	1549.7	1549.7
Demand	USBM	2.40	802.2	903.3	1017.3	1145.5	1290.0	1452.7	1635.9
	4) Low	2.21	644.3	718.6	801.5	894.0	997.1	1112.2	1240.5
	USBM	3.53	829.0	986.3	1173.3	1395.7	1660.4	1975.3	2349.0
	5) Medium	2.78	669.3	767.7	880.5	1009.9	1158.4	1328.1	1524.1
	USBM	4.25	846.6	1043.0	1284.9	1582.9	1950.0	2402.3	2959.5
	6) High	3.85	785.8	949.0	1146.1	1384.1	1671.6	2018.7	2438.0
CENTRAL ECONOMY COUNTRIES									
Supply		2.67	234.5	287.1	415.1	516.1	523.1	516.5	516.5
Demand	7) Low	2.80	173.9	199.7	229.3	263.2	302.2	347.0	398.3
	8) Medium	3.29	177.4	208.6	245.4	288.6	340.0	399.4	469.8
	9) High	3.80	180.8	217.9	262.6	316.4	381.2	459.4	553.6
MARKET ECONOMY									
Supply		1.08	749.5	790.7	972.7	1007.7	1053.2	1033.2	1033.2
Demand	10) Low	1.95	470.4	518.9	572.2	630.8	694.9	765.2	842.2
	11) Medium	2.57	491.9	559.1	635.1	721.3	818.4	928.7	1054.3
	12) High	3.86	605.0	731.1	883.5	1067.7	1290.4	1559.3	1884.4

SOURCES OF PROJECTIONS

All supply projections compiled by Dames & Moore.

1) 2) 3)- United States Bureau of Mines, Mineral Commodity Profile: Nickel, 1979.

4)- Ministry of Natural Resources, Ontario (Canada), Mineral Policy Background Paper No. 4, 1977. (Assumes that approximately 20 per cent of total demand will be satisfied by recycling of scrap)

5)- Malenbaum, W., World Demand For Raw Materials in 1985 and 2000, 1978. (Assumes that approximately 20 per cent of total demand will be satisfied by recycling of scrap)

6), 8)- United Nations Committee on Natural Resources; Nickel Study, 1979. (Assumes that approximately 20 per cent of total demand will be satisfied by recycling of scrap)

7), 9)- High-low variance for Central Economy Countries not determined by United Nations. Assumption made that high-low variance could be as much as ± 0.5 per cent of United Nations' projected annual rate of growth of 3.3 per cent, based on 1976 estimated consumption.

10),11),12)- Derived through subtraction of Central Economy Countries data from World data.

- o For the most part, USBM projections for world high, medium-probable, and low scenarios are significantly higher in terms of quantities of nickel per year likely to be required than those estimates recently made by other organizations. Shortfalls in supply occur in all three scenarios (low, medium-probable, and high) if USBM projections are matched against projected supply. However, with some exceptions, if comparisons are made using projections made by other agencies, annual consumption of nickel, given a high rate of growth (3.85 percent), will exceed projected world supply of metal during the period 2000-2010. Consequently, unless a major unanticipated shortfall occurs in estimated world nickel production capacity, or should USBM projections prove to more accurate than those of the United Nations, Malenbaum, or the Ontario Ministry of Natural Resources, it is unlikely, assuming a moderate rate of growth in annual demand for this metal, that a deficit in supply will occur prior to 2010.

- o With only one exception (high demand, 2010), the expected nickel production capacity of nations considered to be central economy countries will exceed projected demands of this group of countries through 2010. A large part of the nickel supply available to central economy countries is expected to be produced by Cuba. Inasmuch as the production capacity of central economy countries is likely to exceed demand given low and medium rates of growth, excess nickel production probably will be sold to other market economy nations to obtain foreign capital necessary to buy non-Comecon goods.

- o Total estimated market economy nickel production capacity will, for most cases, be adequate to meet projected demand to the year 2010.

Although there appears to be sufficient nickel production capacity to forestall any shortfall in world supply until at least 1990 (assuming a high demand scenario using USBM data) or 1995 given other estimates of demand, it is important to note that the information presented in Table 4-18 represents a static situation. As inventories of nickel are accumulated, it is probable that the time at which shortfalls will occur, given presently installed and anticipated production capacity, will be delayed.

In summary, given existing and announced expansions to nickel production capacity, and moderate rates of growth in annual world demand for nickel, a shortfall in supply is unlikely to develop within the period 1980-2005. However, should the annual rate of growth in nickel demand increase to nearly 4 percent, shortfalls in nickel production capacity could develop as early as the year 1995 for market economy nations, 2000 for the world, and 2010 for central economy countries (Figure 4-7).

Whether any shortfalls in nickel supply will occur will depend largely on whether:

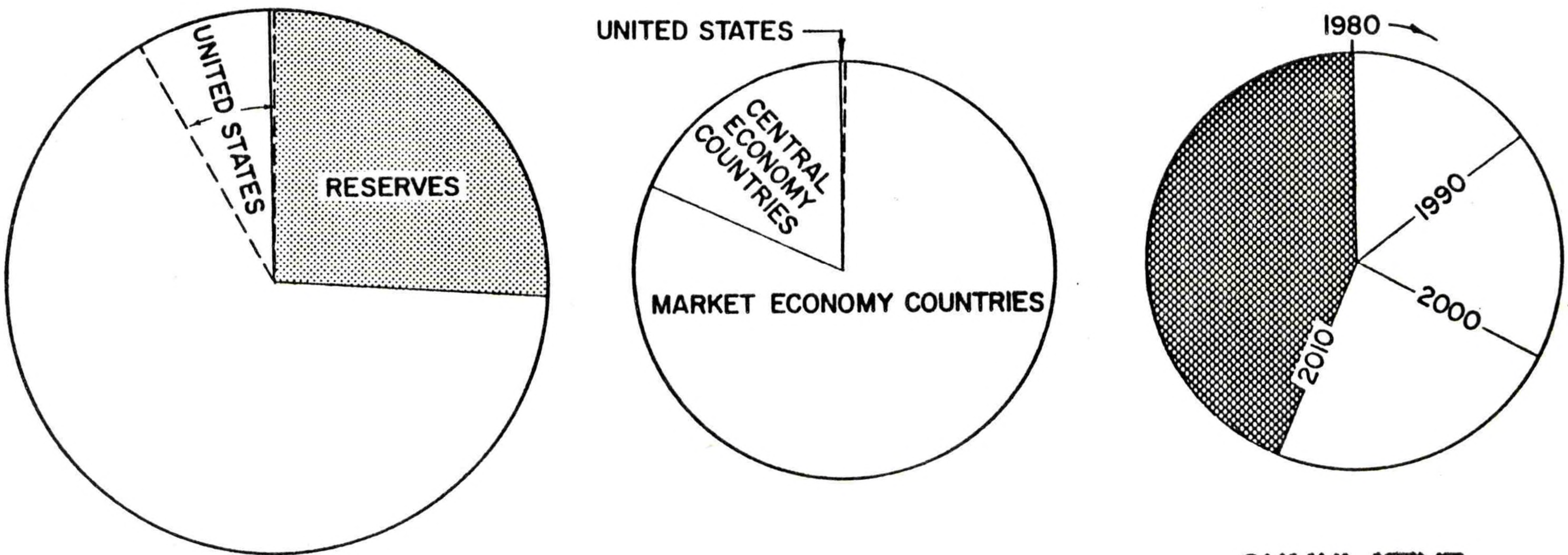
- New mines projected to come on-stream during the period 1980-2010 generally will be brought into production and whether those properties already on-line will continue to operate at full capacity.

- The cost and availability of energy necessary to beneficiate and smelt lateritic ore will restrict or prohibit the development of these types of nickel deposits.

- The necessary capital will be forthcoming to finance new mining operations.

- The nickel production expected to be derived from central economy countries (specifically Cuba) will be completely consumed by Communist nations, or if excess production, will be sold on the free market.

NICKEL



TOTAL RESOURCES
227 MILLION TONS

KNOWN RESERVES
59.56 MILLION TONS

CUMULATIVE WORLD CONSUMPTION
MOST LIKELY PROJECTION (2.78%)

Figure 4-7

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PART II

MINING AND PROCESSING METHODS
PRACTICED IN THE RECOVERY OF COPPER,
NICKEL, COBALT, AND MANGANESE ORE DEPOSITS

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1.0 INTRODUCTION

In this chapter, the reader is introduced to the ore genesis and important geologic environments of copper, nickel, cobalt and manganese ore deposits. Mining methods currently practiced in the extraction of each metal are described and whenever future mining trends are identified, their potential impact on the mining industry are addressed. Processing methods commonly utilized in freeing the metal content from the ore and minerals are presented and, similarly, any research or processes which may advance existing metallurgical practices are discussed. This chapter provides the data base by which the environmental impacts of surface based mining are aggregated and assessed in the following section.

2.0 COPPER

2.1 MINING METHODS

Existing copper deposits of economic significance may be classified into five geologic types: porphyry and associated vein replacement deposits, stratabound deposits, massive sulfide deposits, copper-nickel sulfide deposits, and native copper. The common mining techniques associated with porphyry and vein replacement, stratabound, and massive sulfide deposits are presented in this section. A review of the geology and mining techniques of the copper-nickel sulfide deposits may be found in the following nickel section. The mining of native copper, currently of limited importance, is not discussed since future supply from this deposit type will be relatively insignificant.

The geologic environments in which copper bearing ores are situated often dictate the system adopted for their extraction. A literature review of the important copper operations reveals that for a particular deposit type, a typical mining method or combination of methods is usually associated. The list below combines the most common mining methods with the three geologic environments considered in this section. An overview of each mining system follows:

DEPOSIT TYPE	MINING METHOD
Porphyry	Open-pit, block caving
Vein replacement	Open stoping
Stratabound	Open-pit, sublevel caving, sublevel stoping, room-and-pillar
Massive sulfide	Sublevel stoping, open stoping, sublevel caving

2.1.1 Open-Pit Mining

Open-pit mining is practiced in mineral deposits lying near or on the surface. Open-pit mining allows wide flexibility in production, selective mining of ore grades, and complete extraction of ore from within the pit limits. Other benefits include low manpower, high mechanization, high productivity, and in general, a safe working environment. Because of its many favorable factors, open-pit mining methods is usually selected before underground means if geologic and economic conditions permits.

Open-pit mining is often conducted in a series of 50 foot high benches in a descending manner. Usually the higher grade ore is mined first and accordingly, initial preproduction stripping is carried out to reach these zones. Once the initial cut is developed the upper benches are moved outward to permit a safe slope angle and operating space for machinery on the lower benches as the pit is developed downward.

In every open-pit mining operation certain tasks must be performed in the excavation of waste and ore material. Benches are drilled with large rotary drilling machines and loaded with Ammonia Nitrate Fuel Oil (ANFO) to fragment the rock to a size suitable for loading. The broken material (ore and waste) is then excavated by shovels ranging from 6 to 28 yd^3 bucket capacity and loaded into haulage trucks or ore cars pulled by diesel locomotives. Common truck sizes in porphyry copper operations range from 85 to 200-ton trucks. Ore car capacity is often between 40 and 42 yd^3. Ancillary equipment used in the pit are bulldozers, front-end-loaders, graders and water trucks. If trucks are used, a series of roads and ramps are constructed to transport material to either crushers, if ore, or to waste dumps if the material has no value. In addition low-grade ore may be stored in special dumps to be heap leached or stockpiled for further use.

The waste dumps are located at favorable sites near the ultimate pit limit. If the pit limit is reached while mining progresses downward it may be beneficial to backfill the higher bench areas with waste to avoid long haulage distances, a major cost compotent. Backfilling of the open-pit may also minimize the surface disturbances resulting from the excavation. Once the pit is mined out, dumps and roads may be recontoured to blend in with the landscape and seeded with grass and shrubs for restoration and reclamation. When completed, the deepest part of the pit is often flooded by ground water whereby an artificial lake is created.

2.1.2 Underground Mining Methods

Underground mining by its inherent characteristics is more selective, and in many respects, more difficult than surface mining. Like surface mines, ore must be drilled, blasted and transported; however, these activities must be done under more severe conditions. Underground mining requires detailed planning to ensure efficient productivity and material transport from the working areas to the surface facility. Underground mining can be thought of as essentially a materials handling science, and consequently, all efforts are directed towards optimizing this function. Underground methods used in the recovery of copper ore deposits are described in the following pages.

2.1.2.1 Block Caving

Block caving is ideally suited for massive, disseminated low-grade deposits of large horizontal dimensions in structurally weak rock. Advantages of this system include high worker safety, low mining costs, efficient supervision, and good ventilation. Its major disadvantages, however, are high development costs, high costs in maintaining open drifts in the drawing areas, and difficulty in varying the rate of production to meet a change in demand. Surface subsidence is common to caving methods and scars (gloryholes) make the surface unusable for post-mining use. In fact, in most instances, land restoration is easier and less costly over surface excavations than over areas which have subsided.

Block caving operations begin with the development of main haulage levels consisting of a series of parallel drifts on 240 foot centers. Approximately 60 ft above the haulage level a production level is constructed. The production level is composed of a series of parallel drifts driven on 50 foot centers. Finger raises are constructed up from each production drift to create a network of ore drawpoints. As the finger raises are constructed an undercut level is prepared 25 ft above.

The undercut level consists of 5 by 7-foot or 8 by 8-foot drifts parallel to and slightly offset from those on the production level; however, some operations may have them directly over the draw raises. Crosscuts are prepared perpendicular to the undercut drift on a new side of a block or adjacent to the caved portion in operating mines. Undercutting of the block is initiated by blasting out a pillar between drifts. Sufficient ore is then drawn off before the next adjacent pillar is blasted. Caved ore from the undercut horizon is drawn to the production level through transfer raises. These transfer raises funnel the ore from the draw raises to a common loading station where it is transported through a railroad system to the ore hoisting shafts. The ore trains often consist of 15-car trains of 12.5-ton capacity each. Ore is then hoisted to surface ore bins and later transferred to the milling facility.

2.1.2.2 Sublevel Caving

Sublevel caving is amenable to weak ground but can be used in competent ground provided either the capping rock will cave or sufficient room for drilling fan holes is available. Sublevel caving can produce high tonnages, particularly when used with modern Load Haul Dumps (LHD) and advanced drilling equipment. Ore extraction ranges from 60 to 85 percent, however, like all other mining systems dilution is inevitable.

In this mining system, sublevels are driven between (in the vertical dimension) and parallel to the main levels on 20 to 40 foot centers. The ore between the sublevels is blasted and collected into the sublevel for removal. This method requires that mining be done in a descending fashion. Ore is hoisted to the surface through vertical or inclined shifts.

2.1.2.3 Open Stoping

Open stoping refers to an underground opening from which ore minerals are removed without support (i.e. timber) of the walls or roof.

Numerous variations of open stoping mining are practiced around the world. One variation, open stoping with randomly spaced pillars, is used in the larger ore pockets or lenses where ore grade or thickness is variable. The pillars may be extracted in final mining if they contain good ore. When pillars are mined, approximately 75 percent of the ore body may be recovered. Open stoping is also used in vein deposits where the dip angle is such that broken ore will not flow under the action of gravity. Another variation, open stoping with pillars, uniformly placed has obtained extractions of 60 to 80 percent. Room-and-pillar mining is essentially the same method, however, it is used in flat-lying deposits of uniform grade and thickness.

2.1.2.4 Sublevel Stoping

Sublevel stoping is used in steeply dipping veins and bedded deposits where the ore dimensions are variable and the ore grade is fairly uniform. The foot and hanging wall, as well as the ore, should be fairly competent for this system to be cost effective.

Two common configurations are used; longitudinal and transverse to the strike of the deposit. In both cases the ore is mined from sublevels by benching or ring drilling. Haulage levels are opened under an ore pillar and draw points are established for ore transporting. Sublevels are opened and the ore is drilled and blasted between support pillars. The broken ore is then drawn out and hauled to shafts for hoisting to the surface.

Longitudinal stopes are developed in narrow, steeply dipping deposits. The stopes run parallel to the strike of the deposit and are of indefinite length. The width is limited by the thickness of the deposit. At Rokana, Zambia, a stratabound copper deposit, sublevels are normally opened 250 ft high by 30 ft wide at 60 ft intervals along the strike with 30 ft of stope and 30 ft of pillar. Production from the stope feeds two grizzlies which feed a single loading chute on the haulage level. Ore is moved to haulage levels by gravity in most stopes.

LHDs are used at drawpoints in some areas; however, the production costs are approximately 33 percent greater.

Transverse stoping is used for wide deposits (greater then 70 ft). In this method the stopes run perpendicular to the strike of the deposit and are limited in length to the thickness of the deposit. These stopes usually are outlined by a regular system of pillars with floor pillars the same as longitudinal stoping. The spacing of pillars is determined by the ability of the ore to form an unsupported span, but rarely exceeds 70 ft. The total recovery with transverse stopes is generally less than with longitudinal stopes since a greater percentage of the ore is left in the form of pillars, however less dilution from the sidewalls is experienced. In massive sulfide deposits, stopes have been mined with spans from 40 to 50 ft.

2.1.2.5 Shrinkage Stoping

Shrinkage stoping is used to mine veins and bedded deposits that are steeply dipping. Shrinkage stoping is basically an overhand stoping system in which a portion of the broken ore accumulates until the stope is completed. The increase in bulk as the ore is broken requires that 30 to 50 percent of the fragmented ore be periodically drawn through chutes or drawpoints to maintain a working floor. Ore material must be strong enough to stand unsupported across the width of the stope, and when broken as ore, should not compact to such a degree that it cannot be drawn. As the stope is being mined, both the hanging and footwall are stabilized by the broken ore in the stope below. Extractions from 75 to 85 percent are obtainable. Spans of 34 to 70 ft have been mined.

2.1.2.6 Ancillary Operations

The stoping operation is the heart of underground mining, but the auxiliary operations complete the total entity. The removal of ore cannot be divorced from problems of development work, hoisting and hauling, ventilation, efficient handling of supplies, water, support of workings and other phases. A few of these important operations are presented in the following discussion:

Development. Access to deep ore deposit is most often through shafts. The functions of shafts are to hoist ore and waste rock, to carry equipment and supplies, provide passage for the labor force, dewatering lines and ventilation. It is very common to find an additional shaft by which ventilation, escapeway and services for efficient production and safety are provided.

Development drifts are constructed from the shaft to the ore deposits. Drifts serve as main haulage levels, as access for supplies and manpower, and to delineate the working areas. Raises or vertical drifts, are made upward from the drifts for the same purposes.

Hoisting and Haulage. Mined ore is hauled to shafts by trains, conveyors, or rubber-tired equipment and then hoisted to the surface. Often the factor which determines the economics of underground systems is the method by which ore and waste is removed from the working place to the outside. Men, supplies, ore and waste must be transported in a rapid but safe manner.

Ventilation. Mine ventilation is necessary for the preservation of human life and for the safe conduct of underground operations. Ventilation should be carried out with minimum cost, but with sufficient capacity to ensure the health and safety of the mining force. In mine ventilation, air is coursed through the mine workings and, for auxiliary purposes, through vent tubings and ducts. Air is usually provided by large fans, although air flow may result due to natural conditions.

Handling of Supplies. Supplies underground must be continually transported to the working areas. Supplies involve explosives, timber or concrete for support, spare parts for equipment, and all other materials necessary for efficient production.

Water. Ground water has an important effect on the cost and progress of many deep excavations. Water can limit both the mining methods used and present hazards in others. The control of water in deep mining is effected by pumping and lowering water tables.

2.2 COPPER PROCESSING

2.2.1 Conventional Processing of Copper Sulfides

2.2.1.1 Crushing

Once ore is transported from the open pit or hoisted from underground workings, it is most often fed to a gyratory crusher and reduced to approximately 6-8 inches. This ore is discharged from the primary crusher and transported to stockpile areas. As material is needed, the broken ore is drawn at controlled rates from the stockpiles to secondary crushers. The secondary crushers further reduce the ore to an intermediate size, after which it is conveyed to a set of tertiary crushers for final sizing. Vibrating screens within the crushing circuit ensure that crushed ore leaves the plant at the correct size, approximately minus 3/4 inch. Oversize products are recirculated into the tertiary crushers for further size reduction. The crushed ore is then conveyed to ore storage bins.

2.2.1.2 Concentration

2.2.1.2.1 Grinding

Crushed ore is withdrawn from the storage bins and fed with water at controlled rates to the grinding circuits. Major circuits operating in parallel utilize a rod mill followed by two ball mills and a size classification system. The grinding circuits reduce the crushed ore to a fine particle size and slurried in preparation for flotation.

2.2.1.2.2 Flotation

Copper bearing minerals (sulfides) are separated from the waste material in the flotation process. Chemical reagents are mixed with the slurry and distributed to the flotation cells. The flotation units subject the slurry and chemical reagents to mechanical agitation and aeration so that a frothing action is achieved. As a result of the reagents, copper bearing mineral particles attach themselves to the injected air bubbles and float to the surface. The waste material is directed to the tailings area.

2.2.1.3 Smelter

Smeltering facilities consist of reverberatory furnaces, converters, anode furnaces, casting wheels, waste heat boilers and ancillary equipment. In the smelting process sulphur and other elements in the concentrate are separated from each other.

The first operation in the smelter is to melt the concentrate with a silica flux in a reverberatory furnace. A mixture of oil or pulverized coal and air keeps the furnace operating at temperatures as high as 3000°F. The concentrate is fluxed with silica-bearing sand and smelted to form two liquid layers; a heavy layer called matte and a lighter waste material called slag that is skimmed-off and discarded.

The matte containing 45 percent copper, 30 percent iron and 25 percent sulphur is tapped into large steel ladles and poured into converters where it is oxidized by a high velocity airstream. The iron is oxidized first and, with silica flux, forms a slag which is returned to the reverberatory furnace for the recovery of the entrained copper. The copper is then converted to blister copper, about 98.5 percent pure, which remains in the converter at the end of the cycle. Throughout these processes the sulphur is oxidized into a sulphur dioxide gas from which sulphuric acid may be produced at an adjoining plant.

The blister copper is transferred to the anode furnaces where the remaining traces of sulphur and oxygen are removed. The molten copper is channeled to a horizontal casting wheel for the casting of anodes which are 99.5 percent pure.

2.2.1.4 Refinery

The impure copper anodes (99.5 percent copper) are transported to the tankhouse and are suspended together with pure copper starting sheets (cathode) in electrolytic cells containing an acidified copper sulphate solution. A direct electric current is passed through the cell effecting electrolysis (i.e., copper dissolves from the anode and is plated out onto the cathode). Cathodes produced in this manner are 99.97 percent pure.

2.2.1.5 Refined Copper Casting Plant

A complete copper complex may include further refining facilities and rod casting plant. The cathodes from the tank house each weighing approximately 300 pounds are transported to the casting plant and melted in a vertical shaft furnace. No further refining of the copper occurs in the furnace and the molten copper is cast into either a wirebar or rod. Each Wirebar weighs approximately 300 pounds and is cast on a horizontal casting wheel containing about 14 copper molds. Wirebars are so named since they are exclusively used in the production of copper wire by means of initially rolling the bars followed by a drawing operation. The copper rod production is achieved through a continuous casting operation in conjunction with a fourteen stand rolling mill. The rod produced is a four ton coil, sizes ranging from 0.25 to 0.69 inch diameter. Wirebars and rods are then sent to fabricating plants elsewhere.

2.2.2 Leaching

Leaching involves the dissolution of copper minerals in any of a variety of lixiviants; however, dilute, low-cost sulphuric acid is the most important solvent used in leaching today. Vat leaching, heap leaching, dump leaching, and agitation leaching are the most common techniques. A description of each method follows.

2.2.2.1 Vat Leaching

Vat leaching encompasses either upward or downward percolation of a leaching solution through crushed ore (3/8 in) which has been carefully bedded in rectangular vats.

2.2.2.2 Heap Leaching

Heap leaching involves the placing of coarsely crushed ore over an impervious pad and distributing a sulphuric acid solution on the heap to dissolve the copper through downward percolation.

2.2.2.3 Dump Leaching

Dump leaching is widely used to extract copper from porphyry strip mine waste dumps. Dilute sulphuric acid-ferric sulphate solutions are sprinkled on the dump to leach the copper minerals.

2.2.2.4 Agitation Leaching

Agitation leaching is accomplished in open containers by agitation of finely ground material with dilute acid at atmospheric pressure. Agitation extracts approximately 95 percent of the copper, whereas vat and heap leaching recover approximately 80 percent.

2.2.2.5 Purification of Leaching Solution

The pregnant solution derived through leaching must be separated from impurities by purification. Common purification techniques are ion exchange and solvent extraction.

2.2.2.6 Electrowinning Processing

By passing an electric current through the purified solution the compound dissociates into positively charged metallic ions which are deposited on the cathode. The resulting cathodes then undergo conventional refining processes. Because impurities build up during electrolysis part or all of the electrolyte must be removed periodically from the cells and purified.

2.2.3 Recovery of Byproducts

Copper deposits frequently contain other recoverable metals such as molybdenum, gold, and silver. A brief description of the processes used in recovering these elements are found below.

2.2.3.1 Molybdenum

As described previously, copper ores are crushed, ground, and concentrated by flotation. This concentrate containing copper and molybdenum is submitted to a concentrator plant where molybdenum is floated while copper is depressed with either Anamol D or sodium ferrocyanide. Depressed copper is cleaned and returned to the copper circuit. Molybdenum is subjected to cleaner flotation and a process to eliminate contained copper. Ultimately, the molybdenum concentrate is filtered, dried, and packed in steel drums. The molybdenum concentrate contains about 56 percent molybdenum.

2.2.3.2 Gold and Silver

In the refining process gold and silver are recovered in the form of anode slimes.

3.0 NICKEL

The world's nickel ore deposits are classified in two general categories, each a function of the ores mineralogical makeup. These two deposit types are most often referred to as the sulfide ores, i.e., Sudbury, Ontario, and the Laterite ores, i.e., New Caledonia. Laterite deposits are presented first since their contribution in supplying future world nickel demand is expected to increase significantly.

3.1 LATERITE DEPOSITS

3.1.1 Geology

The lateritic nickel/cobalt bearing ores of economic significance are found as near surface deposits of relatively soft decomposed rock material. Their ore development begins with peridotite, a rock composed chiefly of olivine, a silicate of magnesium and iron which often contains as much as 0.3 percent nickel. Ground water rich in carbon dioxide acquired from both the atmosphere and decomposing vegetable matter attacks this olivine. The olivine decomposes, magnesium, iron, cobalt, and nickel go into solution, and a colloidal suspension of sub-microscopic silica particles develops. While in solution the iron is oxidized and precipitated as ferric hydroxide. Ultimately this precipitate loses water and forms the rust-like minerals goethite, $FeO(OH)$, and hematite, Fe_2O_3. Cobalt, often present in small quantities, is also precipitated at this time. Thus iron oxide and cobalt precipitate near the surface, and the magnesium, nickel, and silicon remain in solution so long as the water is acidic. When the water is finally neutralized by reaction with the rock and soil, these elements precipitate as hydrous silicates.

3.1.1.1 Laterite Profile

Several ore types of different physical and chemical composition are distinguishable on a typical laterite profile. Each horizon or zone displays the progressive weathering action as surface water percolates to depth. These horizons may be classified into six zones, identified as A to F. A description of each is found below in descending stratigraphic order.

Zone A - Upper Limonite. This uppermost zone consists of earthy ferruginous overburden (waste rock) which often exceeds 40 percent iron content. The nickel and cobalt ore concentrations found at greater depths were leached and transported from this horizon.

Zone B - Lower Limonite. Zones A and B are virtually identical in physical characteristics, however chemically, zone B may contain residual nickel values of ore grade. Iron content approaches 35 percent and cobalt presence is noticeable and commonly exists in economically recoverable values.

Zone C - Soft Serpentine. Zone C is the principal nickel containing horizon. Nickel content ranges between 1.2 to +3.0 percent. Up to 20 percent of this material is found as hard, unaltered, and therefore, unenriched boulders and lenses which do not meet ore grade criteria.

Zone D - Hard Serpentine. Zone D is the second most important nickel producing zone. This zone is often greyish in color, extensively weathered and fractured, and contains numerous veinlets of silica and garnierite. Approximately 20 to 50 percent of this material consists of unweathered peridotite.

Zone E - Serpentinized Peridotite. Zone E is composed of hard massive rock containing numerous silica veinlets. Approximately 50 to 75 percent of this material is unweathered peridotite.

Zone F - Ultramafic. Totally unaltered peridotite.

3.1.2 Mining

The recovery of all lateritic deposits is now, and will continue to be, through surface mining techniques. When compared to the extraction of almost every other mineral, lateritic mining is a relatively simple operation; stripping ratios (tons of overburden: tons of ore) are low, ranging from 0:1 to a known maximum of 3.2:1. The excavated material is virtually unconsolidated, and therefore, drilling and blasting is rarely practiced. Consequently, the greatest concentration of effort in the design of an efficient laterite mining operation is directed towards equipment and mining flexibility.

3.1.2.1 Sequence in Mining Lateritic Ore Deposits

A review of excavation techniques practiced around the world indicates that many operating parameters are common to all; however, certain variations of the general system occur at each individual mine. Probably the greatest operating variation is in topsoil removal and replacement, grading, sediment control, and other environmental matters. Due to the many political, social, economic, and cultural disparities found around the world, a common environmental approach does not exist, and consequently environmental regulations vary considerably.

Typically, the recovery of laterite ore deposits include the following unit functions.

1. <u>Land Clearing</u>. The areas to be mined are often cleared of vegetation using the slash-and-burn technique common to agricultural projects. Bulldozers are then used in the extraction of any remaining tree stumps which might interfere with overburden removal.

2. <u>Overburden Removal</u>. Essentially all overburden material is removed by mechanical means with little need for blasting. The principal overburden removal methods practiced around the world are:

- <u>Bulldozers</u>: Bulldozers are used to break up any material which may exist in a hard, consolidated state. As the rock is loosened, it is pushed into piles and loaded into haulage trucks by conventional excavating units. This particular overburden removal system is especially applicable when consolidated rocks and lenses are dispersed through the overburden material.
- <u>Scrapers</u>: Self-loading, elevating, push, or push-pull scrapers are used in overburden removal. An inherent problem with this

excavating system is the necessity of numerous haulage roads, which, because most lateritic deposits are found in extremely wet environments, may present difficult operating conditions and low equipment efficiencies.

- Front End Loaders/Hydraulic Shovels: Rubber tired front end loaders and more recently, crawler mounted hydraulic shovels are often used in direct exacavation of overburden material in bench heights up to 25 feet. Material excavated by these units are loaded into haulage trucks for transport to either waste dumps or backfill areas within the worked-out pit.
- Draglines: Draglines of relatively small bucket capacity (less than 10 yd^3) are employed in some locations having thin overburden on steep slopes.

3. Ore Mining. In general, the extraction of lateritic ore is carried out from a series of mining benches, beginning at the highest elevation and progressing downward. Good engineering practice concentrates on both the elimination of equipment operating on the wet ore material and flexibility in mining. The elimination of equipment on the ore zone is required since 20 to 50 percent of the ore weight is water. Also frequent tropical cloudbursts contribute additional water to the mining area. Together these two factors create difficult operating conditions for all excavating and hauling equipment. Mining flexibility is achieved by using numerous easily transportable excavating units at multiple working areas. By developing multiple areas, high ore and grade control is provided to the processing facilities. Additionally, this selective mining benefits the entire operation by upgrading the ore feed through elimination of waste and subgrade material.

Common excavating units used in mining operations around the world for the removal of ore zone material are:

- Draglines: Diesel powered draglines are found in a few ore mining applications. Excavated ore is either loaded directly into ore trucks, or windrowed for subsequent loading and haulage to the mill or stockpile area.

- Bulldozer: Dozers operating in the mineralized zone are used to push ore zone material to loading units below. This system is essentially the same as that described in the overburden removal section.
- Backhoe: Backhoes operating within the ore usually mine the lowermost portion of the ore profile. This zone is found to be extremely erratic in elevation and, as operating mines have discovered, the single greatest cause of waste dilution. For example, the Greenvale nickel mine in Greenvale, Autralia records that individual pits 50 feet in depth are found separated by just 15 to 30 feet in horizontal distance. By virtue of its high bucket breakout force, the backhoe can excavate ore material without tractor assistance. This equipment system is becoming increasingly popular since the machine operator can selectively excavate the mineralized material from around the interspaced and bottom waste material. In contradiction, the selective removal of ore and waste material is virtually impossible with the bulldozer method described above. Being a retreat mining system, water run-off and churned-up ore material is kept distant from the mining machinery. Excavated ore remains relatively dry (important in energy conservation during ore processing) and better working conditions are created when the backhoe is utilized.
- Hydraulic Shovels: This highly efficient excavating unit has recently been offered to the mining industry by numerous equipment manufacturers. In terms of cost, productivity and mobility, it lies between the rubber tired front-end-loader and the electric shovel. Like the backhoe, the hydraulic shovel offers selectivity in mining and high break-out force, however it must operate from within the ore zone itself. Application of the hydraulic shovel in the laterite mining industry should increase with time.
- Front End Loaders: This very mobile excavating unit is extensively utilized in the mining of lateritic ore deposits. However, because it must work from within the ore zone,

production effeciency is lost as a result of wet, muddy conditions. Its superior mobility however, still makes it an attractive primary excavating unit.

3.1.3 Extractive Metallurgy

The metallurgy associated with nickel oxide ore is more difficult than that of the sulfides. When sulfide ores are subjected to extractive treatment, well established beneficiating techniques such as flotation and magnetic separation play a major role. These processes collect a high percentage of the metal bearing minerals in a relatively small quantity of material by rejecting the waste rock. However, because of the manner in which the nickel-cobalt content is chemically disseminated in oxide ores, these beneficiation methods are not amenable. Screening, either wet or dry, may be used to reject the oversize, less weathered fragments, which contain less nickel. But, except for this simple screening process, no efficient means exists to physically concentrate nickel and cobalt.

Aware that chemical dissemination of the nickel in oxide ores prevents physical concentration of the metal values, metallurgists have approached the extractive task in different ways. Commercial pyrometallurgical techniques employ either melting and sulfiding to obtain phase separation of an iron-nickel matte from gangue, or melting and reduction to obtain phase separation of an iron-nickel metal from gangue. Since raw oxide ore may run 50 percent total water, and often contains substantial quantities of high melting point components, fusion of the ore requires a large amount of energy.

To eliminate those problems associated with pyrometallurgical processes, leaching methods were investigated. However, selectivity by hydrometallurgical means has been achieved commercially only by first changing the chemical nature of the feed, or by establishing special conditions of temperature and pressure. In the first case the nickel is selectively reduced to metal, then leached in an ammoniacal solution. In the second, sulfuric acid is used to leach the ore directly at elevated temperature and pressure, whereby nickel and cobalt are dissolved.

Together the pyrometallurgical and hydrometallurgical methods represent the extractive processes commercially practiced with nickel/cobalt oxide ores. Other pyrometallurgical, hydrometallurgical, and vapometallurgical methods have been demonstrated experimentally with some holding promise for the future. However, for the near future, the three basic processes for treating lateritic ores are:

1) Smelting, reduction roast
2) Ammonia leach
3) Sulfuric acid leaching

The traditional range of application of these processes is described as follows:

3.1.3.1 Pyrometallurgic Process

In general, pyrometallurgy is used in both the lower siliceous and higher nickel ores. Nickel recovery is approximately 95 percent, however, very little or no cobalt is recovered.

The basic steps involve drying, calcination (with or without prereduction), electric furnace smelting and conversion to produce a ferronickel product containing 20 to 50 percent nickel. Nickel matte may also be produced by the injection of sulfur prior to smelting. Today, laterite smelting is more widely practiced than hydrometallurgical means. Operating plants are in New Caledonia, Indonesia, Japan, Russia, Dominican Republic, U.S.A. and Guatemala.

3.1.3.2 Caron Process

The original plant using the Caron process is operating in Nicaro, Cuba. Variations of this process are currently operating in Australia (Greenvale) and the Phillipines (Marinduque) with smaller plants reportedly starting up in Brazil and India. The Caron process has been applied to the higher iron laterites (greater than 25 percent) having a lower nickel content. Nickel and cobalt recoveries are approximately 80 and 40 percent, respectively.

The basic process steps are drying, grinding, selective reduction in multiple hearth roasters, ammonium carbonate leaching, separation of cobalt, distillation to basic nickel carbonate and calcining to produce nickel oxide. At Nicaro, producer gas is used as the reductant and cobalt is not recovered. Greenvale utilizes direct fuel oil injection for reduction. Marinduque uses hydrogen in the roasters and also recovers metallic nickel by hydrogen reduction.

3.1.3.3 Sulphuric Acid Leaching

The only operating plant utilizing this process is in Moa, Cuba. Here ore is slurried and pumped to leaching towers where it is contacted with sulphuric acid at 440 degrees F. to dissolve nickel, cobalt and magnesium. The solids are separated and nickel and cobalt are recovered from solution by hydrogen sulphide precipitation. The sulphides are refined separately to recover metallic nickel and cobalt. Recoveries are 90-95 percent for nickel and 85-90 percent for cobalt. The high cobalt recovery places renewed emphasis on this process.

3.1.3.4 New Process Developments

Pyrometallurgy. Pyrometallurgical processes, inherently consume a high amount of energy and consequently, research is aimed at reducing this. However, to date energy reduction is apparently not feasible. Nickel recovery is high, so apparently economic advantages cannot be gained through this route. In general, cobalt content of the siliceous ores is low and the process inherently rejects most of the cobalt fed. Thus the trend to reduce operating costs appears to be towards cheaper sources of energy and physical upgrading and treatment of higher grade ores (if available).

Caron Process. The trend is towards improving recovery (there is considerable latitude) and in making the process applicable to a wider range of ores. Some 75-80 percent of the total energy for the process is expended in drying, grinding and roasting the ore, and therefore, the largest potential for saving is in this area.

Acid Leaching. There has been a significant development in the acid leach process, and its implementation is proposed for a new plant in New Caledonia. This new process differs from the Moa, Cuba operation by neutralizing excess acid with fresh, roasted ore. Thus, higher magnesia (and higher metal) ores can be treated. Chloride leaching is proposed with the resultant H_2S recycled to the sulfide precipitation steps. Estimated recovery of nickel and cobalt is 95 and 92 percent, respectively.

Other Developments. Many processes have been patented and/or described involving selective reduction roasting followed by a variety of methods to recover the metal values from the roasted ore. These include mild acid leaching, carbonyl and sea water leaching. They all, however, suffer from high capital cost and energy consumption at the front end of the processes.

3.2 SULFIDE NICKEL ORES (Copper-Nickel)

3.2.1 Geology

Sulfide Nickel ore deposits are found either within or in close proximity to bodies of rock high in iron and magnesium and low in silicon content. Typical rock classifications are horite and peridotite. Each of these rock types are of intrusive origin having ascended from deep within the earth in a molten state by forcing its way among rocks near the surface. These intrusions later solidified in shapes controlled by the path taken in their upward journey.

The close association of sulfide nickel ores with intrusive rocks of consistent chemical composition the world over suggests the nickel was a constituent of the molten rock, and, the nickel bearing sulfides were derived from the intrusive body either during or subsequent to the cooling period. During the cooling of these intrusives, geologists see an analogy to a familiar smelting furnace phenomenon. Experience in nickel-copper smelting has shown that matte, a melt consisting of iron, nickel, copper, and sulfur, is immiscible with slag, a silicate melt. It is observed that the matte separates and sinks in droplets toward the bottom of the furnace. In effect, the sulfur serves as a collecting agent for copper, nickel, and other metallic elements found in the melt. These conditions prevailing in metallurgy suggested to early students of sulfide nickel ore genesis the idea that the nickel ores had formed by sulfides settling to the bottom of a mass of molten norite and accumulating at its base. Because most theories of the origin of nickel sulfide ores were developed from studies in the Sudbury district, a study of their origin is essentially a study of the Sudbury ores. Although various mechanisms have been suggested as having brought the sulfide ore to its ultimate site of deposition, all but the most radical suggestions concur that the noritic, or gabbroic, magma and the sulfides came from the same ultimate source.

3.2.2 Mining

Virtually all nickel produced from the nickel-sulfide deposits is by underground means. To review the current mining technology and systems is to examine a typical mine in the Sudbury district, Ontario, Canada. The Creighton No. 3 and No. 9 mines are representative of current mining practices and to illustrate the more common methods, a brief description follows.

The Creighton complex is one of the oldest operations in the Sudbury district. Developing increasing depths has necessitated the sinking of North America's deepest continuous shaft (7,000 ft). Ventilation required in supplying the miners fresh air and to cool the working areas have presented considerable problems. To achieve high productivity and low unit costs revolutionary mining methods have been developed. A few of the important mining methods are:

3.2.2.1 Blast Hole Stoping

Mining operations commence with the delineation of stoping blocks. Stopes are the principal areas of ore mining and typically are 300-350 feet high, 80 feet wide, and 200 feet long. Fan drills operating from extraction drifts located at the bottom of the blocks make an undercut of approximately 60 feet in height. From the top of the stope block, 6-1/2 inch diameter blastholes are down-drilled approximately 300 feet. After all development and drilling work is completed, a five foot diameter raise is bored from the lower excavation lift to the upper drift. A "slot" is created by slashing the raises until they are as wide as the stope (80 ft) and about 12 feet wide in the direction of the stopes longest axis. Once this slot is developed, regular, systematic stope blasting begins by loading and blasting the 6-1/2 inch diameter holes developed earlier.

Bracken ore is loaded from the extraction drift by diesel powered front-end-loaders and trammed to various ore transfer raises for ore train haulage.

3.2.2.2 Mechanized Cut-and-Fill

A typical cut-and-fill stoping area is comprised of a series of six-25 foot wide stopes 200 feet long, having 20 foot wide rib pillars.

The pillars are eventually recovered by conventional undercut and fill methods, and most recently by vertical retreat methods. Cut and fill mining is carried out in captive stopes, that is, all equipment used for drilling and loading remains in the active stope until total extraction is achieved.

Cut-and-fill mining is an upward processing system. When a 10 foot slice of half of the slopes 200 foot length has been mined, a fill wall is erected and the worked out stope is filled with hydraulic sandfill. Simultaneously, mining continues in the other half of the stope. When the second half of the stope has been mined, the miners move on to extract the ore above the first mined out area using the fill as a floor. This method proceeds systematically up through the orebody until the entire stope has been excavated.

3.2.2.3 Vertical Retreat Mining (Used in Pillar Recovery)

In the vertical retreat mining method, topsill drifts are driven the full pillar width (20 ft) with jumbos and scooptram units. Because ore has been extracted and replaced with backfill, cemented sandfill stands on both sides. The immediate roof is supported by rockbolts and wire mesh. The extraction drifts, draw points and undercut drifts are driven at the bottom of the pillar approximately 75 feet below the topsill drifts. From the topsill drifts, 6-1/2 inch diameter holes are drilled down to the undercut drifts. Each hole is loaded with blasting agents at its bottom and ignited to ensure approximately 15 feet of breakage from the pillar bottom. This sequence continues in an ascending fashion until the total pillar is excavated.

3.2.3 Extractive Metallurgy

Sulfide ores are readily amenable to concentration by established mineral dressing methods. Low-grade nickel sulfide concentrates are first smelted in a reverberatory furnace with a suitable flux to obtain an impure copper-nickel-iron matte. This matte is blown with air in a converter using a silica flux to remove iron in a slag and part of the sulfur as sulfur dioxide. The matte is then slowly cooled for approximately 4 days to facilitate grain growth of synthetic mineral crystals of copper and nickel sulfides and a nickel-copper alloy. The mass is pulverized, and the sulfides are separated by flotation in the same manner as in the original ore. The alloy is extracted magnetically and refined electrolytically, leaving a residue containing precious metals. A portion of the nickel sulfide, separated by flotation and containing nearly 73 percent nickel can be dead-roasted to nickel oxide for sale. Part of the nickel sulfide is melted and cast into anodes for direct electrolysis. The balance of nickel sulfide is dead-roasted to oxide and further reduced to metal for refining either electrolytically or by the carbonyl method.

Sulfide concentrates can also be leached with ammonia. In this method the pregnant solution is heated to precipitate copper, and the nickel and cobalt are precipitated together by treating the pregnant solution with hydrogen.

4.0 COBALT

The important world cobalt deposits currently mined are the cobalt-bearing copper ore deposits, copper/nickel ore deposits, laterite ore deposits and the volcanogenic sedimentry ore deposits. In every case, except the volcanogenic sedimentry ore deposits, the recovery of cobalt is a byproduct of the principal mining and processing target, being either copper or nickel.

4.1 GEOLOGY

A brief description of the ore genesis for each ore type is found below. Acknowledging that the geology profession has proposed numerous hypothetical developmental sequences for each ore type, only the most widely accepted interpretations are presented.

4.1.1 Synsedimentary Marine-hydrothermal Cobalt-Bearing Copper Ore Deposits

The most important deposits of this type are distributed in Zaire and Zambia in what is termed the African Copper Belt. It is proposed that the principal copper and cobalt minerals were segregated from aqueous solutions in a synsedimentary process under strong reducing conditions. This segregation formed the stratabound ore bodies which occur as impregnations of young Precambrian carbonate and clastic rock series.

The predominate primary (pyritic) copper and cobalt minerals of this area are linneaite and carrollite. In the oxidation zones, which in parts reach to very deep levels, heterogenite and asbelane occur as secondary cobalt minerals. The cobalt grades are most often between 0.1 and 0.5 percent, but in some cases may reach 1.0 to 2.0 percent.

4.1.2 Liquid Magmatic Pyritiferous Nickel Copper Ore Deposits

During the differentiation and solidification of basic magmas, the separation of cobalt together with nickel starts in the liquid magmatic stage. If sufficient sulfur is present, sulfuric liquid segregation products occur. This process leads to the formation of impregnated ores as well as poorly defined lenticular, banded and layered massive ore bodies. In particular these occur in the deeper and marginal parts of large horitic complexes, the best example being the Sudbury Complex, Canada. Nickel and copper are the main elements of economic importance there. To the nickel and copper bearing minerals are linked cobalt, the platinum metals, gold, silver as well as selenium, tellurium and other elements. Actual cobalt minerals (sulfides and arsenides) occur only seldom; cobalt is most often found as isomorphous admixtures within the principal nickel and copper ore minerals.

4.1.3 Oxide and Silicate Nickel/Cobalt Laterite Ore Deposits

As was the case of nickel contained in the mineral olivine, (See nickel section, laterite geology) nickel is sometimes replaced by cobalt. These olivines can contain 0.04 percent cobalt. However, quantities of cobalt in the olivine mineral alone are not high enough in content to make ore. Only as a result of lateritic weathering processes do deposits of economically profitable nickel-cobalt ores develop.

In contrast to nickel, cobalt is concentrated in the upper parts of the lateritic profile. Cobalt content of known deposits vary between 0.1 and 0.25 percent and average 0.15 percent. In known oxide (limonite) ore deposits, typical nickel content of 1.1 percent to 1.6 is observed, with a cobalt grade of approximately 0.1 percent. In the region of the silicate ores the nickel content is often 1.4 to 3 percent. Here the cobalt grade is approximately 0.02 to 0.04 percent, and averages 0.03 percent.

4.1.4 Volcanogenic-Sedimentary Ore Deposits

Cobalt is often found in the massive sulfide ore zones of volcanogenic-sedimentary deposits. Cobalt minerals rarely occur on their own but are predominately isomorphously linked to pyrite. Examples for deposits of this type are the Cu-Co-Zn ores of Outokumpu in Finland and the pyrite ores in the Huelva District of Spain.

The only deposit in the world that, at present, is primarily mined for cobalt is the Bou Azzer Mine in Morocco. This deposit is within a metamorphously impregnated volcanogenic-sedimentary formation. The ore bodies occur mainly in serpentinites near the contacts with quartzite diorites nd metamorphic rocks. These lenticular ore bodies are steeply dipping and may extend up to 700 feet in length. Mainly arsenide minerals are involved in the ore mineral paragenesis. The primary cobalt mineral is skutterudite; of secondary importance are safflorite, glaucodote, arsenopyrite, loellingite, and rammelsbergite. In addition, native gold and silver as well as molybdenite, brannerite, chalcopyrite, galena, and zinc blende occur. The matrix consists of calcite, dolomite, quartz, talc, and chlorite. In the oxidation zone of varying depth, erythrite (cobalt bloom) is the predominate cobalt mineral. The average metal content of the ores is about 1.2 percent cobalt and 0.15 percent nickel; in addition, 5 to 15 ppm gold (in exceptional cases, up to 300 ppm gold) as well as up to 50 ppm silver are contained.

4.2 MINING

In all cases but one, the Bou Azzer Mine in Morocco, the mining of cobalt-bearing ore is a by-product of either nickel or copper mining. The major cobalt bearing copper ore deposits are found in Zaire and Zambia and mining systems used in these deposits have been identified within the copper chapter. The mining of liquid magmatic pyritiferous copper-nickel ore deposits (Sudbury District, Ontario) and the lateritic nickel-cobalt ore deposits (i.e.,New Caledonia, Guatemala, Cuba) have been presented in the nickel chapter.

Only the mining of the volcanogenic sedimentary ore deposits have not been discussed. Three operations mine this deposit type for cobalt ore. The Bou Azzer of Morocco is producing 300 tons of ore per day by underground vein mining. Ore grade averages 2 percent cobalt. In Finland in the Outokumpa mining district, two underground operations are mined by the room-and-pillar and inclined wall techniques. Average daily ore production is approximately 2,000 tons per day. Cobalt grade from the two mines is 0.3 percent and 0.15 percent. The principal mineral mined in these mines is chalcopyrite, a copper sulfide mineral.

4.3 ORE PROCESSING

There are currently a number of processes for extracting cobalt from ore. The most important is the Katanga process. This process produces cobalt by sulfatizing, cyclone roasting, and leaching. Copper is first separated and later cobalt is precipitated with milk of lime. In the electrolysis which follows, the cobalt is recovered and smelted in an electric furnace to form granulated metal, powder, oxides, or salts.

An older process which is still utilized in Zambia and Zaire operates in the following manner: in the electric furnace, the sulfide and oxide ores are reduced by smelting. In the regulus, a "red" smelt rich in copper segregates off (about 5 percent cobalt) while a "white" alloy smelt overlying it is separated out (around 42 percent to 45 percent cobalt). The white alloy is then dissolved with hydrochloric or sulfuric acid. The cobalt hydroxide that is formed is annealed and,after purifying, is smelted with coke in an electric furnace.

Cobalt in the sulfide nickel-cobalt ores of Canada is precipitated out during the electrolysis for nickel. The mixed concentrates are usually smelted in reverbatory furnaces with the resulting matte blown in converters. The cobalt is then concentrated in the matte and remains in the crude nickel until nickel electrolysis takes place. Cobalt is released in the electrolyte in the form of cobalt sulfate and precipitated out as hydroxide. This hydroxide is then processed to provide cobalt metal, oxides, or salts.

Two other important extraction processes are the Sherritt-Gordon process by which nickel is extracted from complex ores through leaching and reduction under pressure, and the Moa Bay process for treating lateritic nickel ores. All other processes in world cobalt production are of only minor importance.

4.4 FUTURE DEVELOPMENT OF COBALT PRODUCTION

Production of cobalt is linked almost exclusively with the production of sulfide nickel-copper ores, nickel laterite ores, and cobalt bearing copper ores. An increase in cobalt production is therefore possible only in the context of an increase in nickel production or a rise in the production of cobalt from cobalt bearing copper ores.

For the extraction of cobalt from nickel laterites, however, the type of metallurgical processing of this ore is of fundamental importance. In any case, the cobalt can be extracted only by the use of hydrometallurgical processes. If nickel matte is produced, the extraction of cobalt is possible only to a limited extent; if ferronickel is produced, the separation of the cobalt is not feasible.

At present about 55 percent of nickel laterite mined is turned into ferro-nickel. The production of ferro-nickel is also given primary importance in a large number of the newly planned nickel mining projects. In these circumstances even a considerable increase in nickel ore production would not give rise to a corresponding contribution to the production of cobalt. On the other hand, the production of cobalt concentrates could be considerably increased by improved technology in the beneficiation of cobalt bearing copper ore in Zaire and Zambia, even without an increase in copper production. Usually the recovery of cobalt there stands at about 30 percent, sometimes even lower. At present, however, all efforts are being made, even at the cost of copper recovery, to increase cobalt recovery.

In a number of countries in the western world, both expansions of existing nickel and copper mines, and the opening of new mines are being undertaken with provision for the extraction of cobalt as a by-product. These countries include: Zaire, Zambia, New Caledonia, Indonesia, and the Philippines.

5.0 MANGANESE

The mining techniques utilized in the extraction of manganese bearing ores vary extensively and, as with all other mineral recovery techniques, is usually selected after analysis of the deposit's physical geometry. Both surface and underground mining methods are used in this industry with the majority of ore produced by surface means. Bucket wheel excavator/conveyor and conventional truck/shovel and rail/shovel systems produce most of the surface mined ore. Underground mining includes room-and-pillar and more recently, longwall methods.

5.1 MINING METHODS EMPLOYED IN THE RECOVERY OF MANGANESE BEARING ORES

The largest producers of manganese are the USSR, South Africa, Gabon, India, Australia and Brazil. Specific data pertaining to actual mining techniques used in these countries are limited, however whenever possible, information was collected, analyzed and incorporated within this report. In the following sections mining systems for each country are presented. Information relating to specific mines or mining districts may also be found.

5.1.1 Soviet Union

The Soviet Union is by far the largest producer of manganese ore in the world. Production is currently around 10 million tons per year and although most is consumed in the country's domestic steel industry, exports total approximately 2 million tons per year.

Most of the manganese produced in the Soviet Union is from mines in the Nikopol Basin, located just north of the Crimean peninsula in the Ukraine. Much of this ore is mined by open-pit techniques. Here the manganese ore is found in a flat lying bed 5 to 13 feet in thickness and under 250 to 275 feet of clay and sandy overburden.

The Chiatura Basin in Georgia produced approximately 2.5 million tons of concentrates in 1977 from 23 underground and open-pit mines and eight concentrators. Over 80 percent or 2.0 million tons was extracted from underground operations. Of the total beneficiated, 66 percent contained 48.7 percent manganese and the rest 25.6 percent.

Of the total Soviet manganese output, approximately three-quarters came from the Nikopol Basin, and the rest from the Chiatura Basin in Georgia. Additionally, but of minor significance, approximately 300,000 tons are produced in the Kazakhstan. Here manganese ore occurs in a banded ironstone horizon up to 46 feet in thickness. This horizon is found in a sedimentary basin running north-south for approximately 25 miles.

5.1.2 South Africa

The bulk of South African manganese production comes from the Kalahari Field west of Kuruman in northern Cape Province. These deposits are of syngenetic origin with subsequent supergene enrichment and replacement.

Mining

Mining in South Africa is by both surface and underground means. A list of the major mines now operating are as follows:

Operating Manganese Mines

1. The Associated Manganese Mines of South Africa, Ltd.

   Mancorp Mine

   Location: Postmasburg, Cape Providence, South Africa
   Operations: Mine
   Product: Manganese, iron ore
   Mine: Open-pit
   Haulage: Rail

2. Rand London Manganese Mines (PTY), Ltd.

   Transvaal Operations

   Location: Transvaal, South Africa
   Operations: Mine

Product: Manganese dioxide

Mine: Open-pit and underground; room-and-pillar

3. Roodepan Manganese Corp. (PTY.), Ltd.

Roodepan Mine

Location: Digby Plain, Ventersdorp Dist. Trvl

Operations: Mine, concentrator

Mine: Open-pit

Daily Capacity: 100 TPD

Ore grade: 40% MnO_2

Concentrator: Extraction method, gravity

4. S. A. Manganese Amcor, Ltd.

Farm Wessels Mine

Location: Hotazel, Cape Province, South Africa

Operations: Mine, concentrator

Mine: Underground; room-and-pillar

Daily Capacity: 5500 TPD

Ore grade: 48%

Concentrator: Crushing, screening, washing

Hotazeh Mine

Location: Hotazel, Cape Province, South Africa

Operations: Mine, concentrator

Mine: Underground

Haulage: Truck

Daily Capacity: 3300 TPD

Ore grade: 48%

Concentrator: Crushing, screening

Lohathla Mine

Location: North Cape Province

Operations: Mine, concentrator

Mine: Open-pit

Haulage: Truck

Daily Capacity: 1700 TPD

Ore grade: 32% Mn, 22% Fe

Concentrator: Extraction method

Manatwan Mine

Location: North Cape Province

Operations: Mine, concentrator

Mine: Open-pit

Haulage: Truck

Daily Capacity: 660 TPD

Ore grade: 38% manganese

Concentrator: Extraction method

5. Middelplaats

Mine Name - Middelplaats

Owner - Anglo-American Corporation of South Africa

Location - South Africa - Kalahari Field in North Cape Province

Mineral - Manganese ore in banded iron stone

Reserves - 52,000,000

Grade - 38% Mn

Mine Life - 30 years from 1979

Type of Mining - Room and Pillar - Trackless

Plant - Screen and wash

Climate - Semi-arid

Employees - Approximately 700 at full production

Annual Production - 1,100,000 tpy ore at 38%

The newest and most modern underground mine in the Kalahara Field is the Middelplaats mine. This mine operates in a room and pillar fashion by first extracting approximately 12 feet of the upper ore horizon and later working in a two or three-bench system to recover the deposit's full thickness. Twenty-foot-square pillars are left to ensure adequate roof support and prevent subsidence. The mining operation is highly mechanized. Twin boom hydraulic jumbos carry out the drilling operations and after blasting, load-haul-dump units load 24-ton capacity rear-end dump trucks. These trucks are used in transporting the broken ore to an ore pass located above the primary crushing facility. Following this crusher the ore is hoisted to the surface. The primary crusher is the first step of a relatively simple ore treatment process and is used to reduce the ore to minus 3 inches. The ore is further reduced to minus 5/8 inches, washed, screened into two sizes, loaded into rail cars, hauled 590 miles to Port Elizabeth, and marketed.

5.1.3 Brazil

Manganese mining in Brazil uses both conventional truck/shovel and underground, room-and-pillar systems. The largest single producing mine is the Serra do Navio project. Manganese ore grading 30 percent is mined by electric shovels and hauled to the concentrator facility via off-highway trucks. Daily ore capacity is 17,500 tons, while approximately 60,000 tons per day of overburden waste material is excavated. The concentrator involves crushing, washing, sizing and heavy media concentration. A 235,000 tons per year agglomerating plant is also located adjacent to the concentrating facility.

One operation where manganese ore is mined by underground room-and-pillar techniques is the Morraria de Uracum mine. An ore zone approximately 10 feet thick is mined by conventional drilling/blasting/loading methods. In each room three men drill and slush the ore into waiting ore cars to be trammed to the surface by diesel locomotives.

5.1.4 Gabon

Gabon is the world's third largest producer of manganese. Its entire ore output is derived from the Comilo mine at Moanda. Reserves here are estimated at 230 million tons of saleable product grading 48 percent manganese dioxide. For the last several years production has remained around 2.0 million tons, all of which is exported. Production is expected to increase to 4.4 million tons per year when a newly constructed railway reaches Franceville, sometime in 1984.

5.1.5 Australia

Operating Manganese Mines

1. Groote Eylandt Mining Co. PTY., Ltd.

   Groote Eylandt Operations

Location:	Groote Eylandt, N.T.
Operations:	Mine, concentrator
Product:	Manganese ore
Mine:	Open-pit
Haulage:	Truck
Daily Capacity:	5500 TPD
Stripping Ratio:	1.1
Deposit type:	Quartz laterites and clays
Concentrator:	Wet scrubbing and heavy media separator

5.1.6 India

Operating Manganese Mines

1. Balani Ores, Ltd.

   Balani Operation

Location:	Barbil, Onissa, India
Operations:	Mine, concentrator
Mine:	Open-pit
Haulage:	Trucks and conveyor belts
Daily Capacity:	8,000 tons per day
Ore grade:	Iron - 60 to 63 percent
	Manganese - 20 to 30 percent
Deposit type:	Lateritic alteration - iron/manganese

2. Central Provinces Manganese Ore Co., Ltd.

   Balapur Hamesha Mine

Location:	Nagpur, India

Operations: Mine, concentrator

Mine: Underground

3. Chowgule and Company PVT, Ltd.

Sirigao Mines

Location: Bicholim-Sirigao, Goa, India

Operations: Mine, concentrator

Mine: Open-pit

Stripping Ratio: 1.5:2.0

Deposit type: Iron/Manganese

4. The Jeypore Sugar Co., Ltd.

Kuttinga Manganese Mines

Location: Orissa, India

Operations: Mine, smelter

Mine: Underground and surface

Smelter: Electric furnace producing ferromanganese

Capacity: 14,000 tons

5. Manganese Ore (India) Ltd.

Manganese Mine

Location: Nagpue, Maharashtra, India

Operations: Mine

Mine: Underground

6. The Orissa Minerals Development Co., Ltd.

Keonjaar Operations

Location: Keonjaar, Orissa, India

Operations: Mine

Mine: Open-pit

Daily Ore Production: 2300 TPD

7. Salem Magnesite PVT., Ltd.

Location: Omuliar Talnk, Salem District

Operations: Mine, agglomerating plant

Product: Raw, calcined and dead burnt magnesite

Mine: Open-pit

Haulage: Truck

Stripping
Ratio: 1.0:1.0

Daily
Capacity: 400 TPD

Ore Grade: 45%

8. Sesa GOA Private Limited

Harvalien Mine

Location: Fomento Sanguelim, GOE, India

Operations: Mine

Mine: Open-pit

Haulage: Trucks

Daily
Capacity: 220 TPD

Stripping
Ratio: 3.6:1.0

Deposit
Type: Iron/Manganese

5.1.7 Mexico

Compania Minera Autlan S.A. de C.V.

Location: Hidalgo State, Mexico

Surface area disturbance	- Pit	500M x 1500M	- 185 acres
	- Plant	500M x 500M	- 62 acres
	- Housing	500M x 300M	- 37 acres
	Total		284 acres

Emloyment - 350

Energy Requirement - gas-powered electric generators on site

H_2O Requirement - small

Mine Type - Surface and Underground

Stripping Ratio - 12:1 (ultimate)

Production - 384,000 metric tons from underground annually
- 391,397 metric tons from surface annually

Average Grade - 27% manganese

Proven Reserves - 21.3 million metric tons

Probable Reserves - 200.0 million metric tons

Possible Reserves - 1 billion metric tons

Mining Method

- Open pit - truck-shovel system
- Underground - Sublevel stoping
  - Adit and decline
  - Using LHD and trucks underground

The ore is upgraded into nodules grading 75% Mn

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PART III

ENVIRONMENTAL, AND SOCIOECONOMIC EFFECTS OF CONTINUED RELIANCE ON LAND MINING TO PRODUCE METALS AVAILABLE FROM MANGANESE NODULES

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PART III

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1.0 GENERAL IMPACTS OF MINING AND PROCESSING

1.1 BACKGROUND

Mining and processing of mineral ores requires the commitment of natural and human resources. The quantities of these resources utilized and the extent of the effects on the environment are dependent on many factors, for example: the mineral deposit itself; the type and size of the mine (open pit or underground); the processing methods; the natural and man-made characteristics of the area; and regulatory controls imposed on these activities.

Another factor of concern is the availability of the metals. Are present supplies adequate for anticipated demands and if not can substitutes be found for various ores, and/or can lower grade deposits be exploited. These supply/demand concerns have been discussed in Part I of this report. They are reviewed in Part III to the extent there is an effect on the environment. Mining and processing methods for each of the four metals and types of ores and ore bodies have been discussed in Part II.

The emphasis in Part III is on the impacts which will be created if land deposits are used exclusively through the year 2010 to supply world demand for copper, nickel, cobalt, and manganese.

The remainder of this chapter is devoted to a discussion of types of generic impacts to be expected from mining and processing the more important ore types of the four metals (Section 1.2). This discussion will form a background for the quantitative and site specific analyses which follow. Chapter 2.0 includes a brief summary, primarily in table form, of expected supply/demand relationships through 2010, and a compilation of aggregate impacts which can be expected as a result of continued worldwide and U.S. reliance on land-based mineral resources. Scenarios of typical mines and processing facilities are given in Chapters 3.0 and 4.0 to provide site specific examples of expected impacts.

1.2 AN OVERVIEW OF THE ENVIRONMENTAL IMPACTS OF LAND MINING AND PROCESSING COPPER, NICKEL, COBALT, AND MANGANESE ORES

Activities associated with production of copper, nickel, cobalt, and manganese from land-based resources can affect a wide range of natural and human resources. A general categorization of potential impacts is provided in Table 1-1 and Figure 1-1.

Among the most obvious effects of mining and processing ores are changes in land use and topography and in air and water quality. These changes, in turn, affect the habitats of aquatic and terrestrial animals and plants. Development of new mines and processing facilities also alters the socioeconomic conditions by introducing a fairly large labor force, often into small communities, putting stress on housing, roads, and public and private services. Beneficial effects include the increased availability of needed metals and an increase in local employment and in the tax base.

In the following sections, the general types of environmental impacts associated with mining, processing, and transporting metalic ores are described. This introduction provides a basis for the quantitative estimate of aggregate impacts in Chapter 2.0 and for the site specific impact reviews in Chapters 3.0 and 4.0.

1.2.1 Land Use

Land required for a mining operation includes not only the area of the mine, but also waste rock and tailings disposal areas, mine buildings, and access roads. The effects of surface mining on land use and surface topography vary widely depending on such factors as depth, size, and shape of the ore body and waste-to-ore ratio. Removal of overburden and waste rock to expose the ore body and maintain safe pit slopes requires surface storage or disposal of large volumes of overburden and mine waste, often covering several square miles during the life of a typical mine. Vegetation is destroyed in the pit, storage, and disposal areas, and also in areas where support facilities are erected. For some minerals, including copper, leaching of selected

TABLE 1-1

POTENTIAL MAJOR IMPACTS ASSOCIATED WITH LAND MINING (M), PROCESSING (P), AND TRANSPORTATION (T) OF MANGANESE, NICKEL, COPPER, OR COBALT

Air Quality

- SO_2 (P)
- Metal oxides (P)
- Particulates (P)
- Engine exhaust (M,T)
- Dust (M,T)
- Noise (M,P,T)
- Visible emissions (P)

Water Quality

- Drainage disturbance (M)
- Aquifer disturbance (M)
- Water use (P)
- Acidity, metallic salts (M,P)
- TSS, TDS (M,P)
- Heat (P)
- Aquatic habitat degraded (M,P)
- Accidental spills (T)

Land Disturbance

- Clearing (M,P,T)
- Subsidence (M)
- Waste disposal (M,P)
- Land use (M,P,T)
- Safety (M)
- Migration barrier (M)
- Soil fertility (M)

Economic Conditions

- Jobs, income (M,P,T)
- Tax revenues (M,P,T)
- Public service needs (M,P)
- Local inflation (M,P)
- Energy use (M,P,T)
- Land values (M,P)
- Transportation system stress (T)

Social Conditions

- Population (M,P)
- Industrial activity (M,P)
- Aesthetics (M,P)
- Safety/health (M,P,T)
- Life style (M,P)

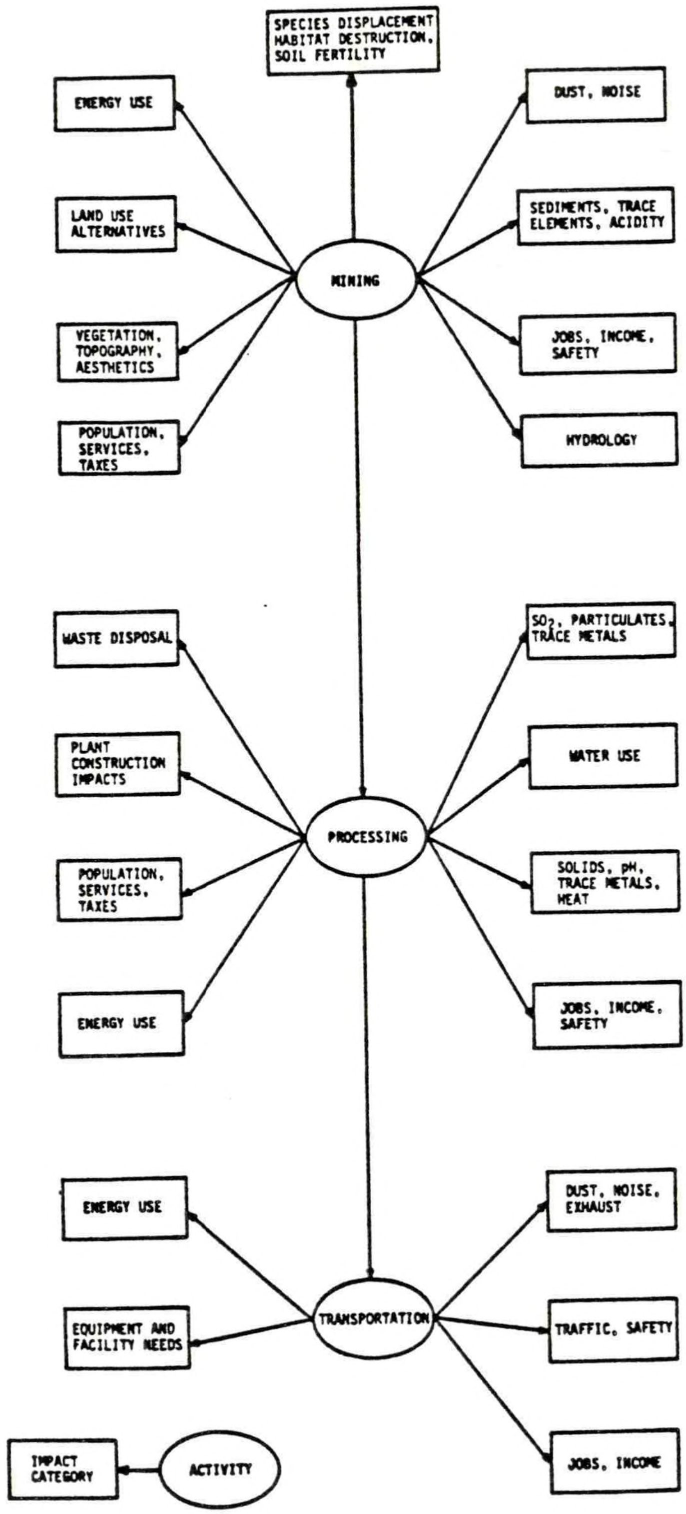


Figure 1-1. Schematic Representation of Land Mining Activities and Related Impacts.

waste dumps for recovery of low-grade ores may be undertaken. Individual copper waste dumps being leached are often hundreds of millions of tons in size. About 12 billion tons are available for leaching at the Bingham pit in Utah. Leaching of copper waste dumps may continue for decades. It is usually conducted by repetitive, alternate cycles of solution percolation, followed by air oxidation (National Research Council, 1979).

The principal effect of underground mining on the land surface, aside from destruction of vegetation and alteration of the terrain by roads, support facilities, and mine waste storage, is surface subsidence. Collapse of the surface may be part of a planned block-caving mining system, or may occur during or after completion of mining because of failure of underground support. Subsidence that occurs in mountainous or rugged terrain, such as that at San Manuel, Arizona (see Section 3.1.3), is less defacing than subsidence in level terrain. Some roads and structures have been damaged by subsidence (National Research Council, 1979).

At ore processing facilities tailings impoundment is the major use of land. Leach solution and water used in transporting the tailings to the disposal area are usually partially recovered for reuse in the mill, particularly in arid climates. Where operations continue for decades, vertical growth and lateral spread of accumulated tailings may result in a conspicuous terrain feature, covering several square miles. Tailings impoundments at major open pit copper operations in the U.S. now total about 50,000 acres. This compares with about 30,000 acres in waste dumps, which usually have a greater height than tailings ponds, and 13,000 acres of open pit excavation (National Research Council, 1979).

Most mill tailings are composed of finely ground rock or chemical precipitates that, when dry, are readily airborne. Most tailings lack the nutrients and microorganisms needed to sustain vegetative growth. Almost every mill tailing has its singular disposal or future land use problem. Water accompanying the tailings is initially alkaline, but long-term weathering converts sulfide minerals in the tailings to sulfates and sulfuric acid. Subsequent attempts at vegetating such surfaces, especially in the arid western United States, have had indifferent success (National Research Council, 1979).

Smelters and refineries also require land for structures and for slag disposal, though substantially less than for mining wastes. Often these facilities are located several miles (perhaps at great distances) from the mine.

Other demands on land resources are caused by the development of towns, roads, power plants and transmission lines, water supply systems, and other services required to support a large, labor intensive mine or process plant. In addition, land may be needed to build railroads, ports, terminal storage facilities and other transportation-related systems required to move the ore or refined metal to market.

1.2.2 Water Quality

Degradation of water quality is a pervasive and persistent environmental effect of many mining operations. The potential for damage is particularly severe in the mining and processing of ores that contain sulfide components subject to oxidation and solubilization. Inactive underground mines may also cause water contamination; though many do not. Water pollution from mining and available control methods have been reviewed by the U. S. Environmental Protection Agency (EPA, 1973); although the report is directed primarily to coal mining in the East, extensive coverage of metal and non-metal mining in all parts of the United States is included.

In the arid western United States, most open mine pits accumulate little or no water. Small flows may be used in sprinkling for dust control, pumped outside the pit for dissipation by evaporation, or added to the water used in milling the ore. Dewatering of a surface mine lowers the water table, sometimes to the detriment of other water users. In arid regions the flow of springs and streams, surface vegetation, fish, and wildlife may be adversely affected by lowering of the water table (National Research Council, 1979). Where subject to substantial inflow from surface water, diversion ditches or dikes are often provided to route runoff away from the mine pit and other surface facilities.

Water contaminated by contact with surface or underground mines will have high concentrations of dissolved solids, widely varying levels of pH, sulfates, and dissolved metals, and oil and grease. If discharged to surface streams directly (without neutralization and precipitation), substantial harm can result to aquatic life and downstream users. In the U.S., NPDES limitations placed on pH, total suspended solids (TSS), copper, lead, mercury, and zinc loadings encourage recycling and treatment of such waters (EPA, 1978). No such limitations may be placed on discharge in some other countries.

In addition to contamination by the mine, immense piles of waste rock (overburden) and low grade ore are usually produced during the course of surface mining. The overburden is usually not a significant source of contamination, but low grade ores are subject to natural and acid leaching of contained metals and sulphides. Acid leaching usually is accomplished in controlled situations with collecton and recirculation of the leaching solution.

Virtually the entire area of an active mining operation is susceptible to increased rates of surface erosion and potential siltation of streams and lakes. In the U.S., point source NPDES permit regulations generally require that the rainfall accumulation and runoff of a 10-year, 24-hour storm be contained so as to prevent uncontrolled release of process wastes. Such control systems may require very large collection ponds and pump systems and are not normally provided at most foreign mines. In any case, such control systems do not control non-point source pollution from the (frequently) many thousands of acres of non-process related lands.

In underground mines there is extensive exposure of ore and wall rock to an oxidizing and humid atmosphere. This promotes the formation of acids and solubilization of metals and chemicals. When active mining ceases, water treatment is often terminated and the control procedures to limit water entry may fail. Subsidence occurring long after mining ceases may provide new sources of water entry into the mine. Unless prevented by appropriate mine design, operating practice, and closure procedures, gravity drainage of contaminated mine waters may degrade both surface and ground water. The extent of ground water pollution from active and inactive underground mines is largely unknown. Dewatering of underground mines in preparation for and during mining tends to lower the area water table with adverse effects similar to those caused by dewatering of surface mines.

Though the mines themselves usually have an excess of water for disposal, this is not the case for processing facilities. Large quantities of water are needed for the milling operations (as a carrier) and in smelting and refining (primarily as a coolant). Though maximum recovery and recycling is normally attempted, substantial makeup quantities are needed to replace evaporation, water not recoverable from tailings, and miscellaneous losses. Local water sources may not be sufficient to supply this makeup in arid locations and, in any case, withdrawals may limit the degree of waste assimilation in surface supplies and reduce available potable water in surface and ground water supplies.

Drainage and seepage from tailings impoundments created by milling sulfide ores may cause surface and ground water contamination. As described for drainage from underground mines, the potential for damage remains and may be worsened when active operations cease. Current EPA regulations either allow no overflow of tailings solution, or require treatment for removal of contaminants before releasing the discharge to public waters. Regulations in other source countries often make no such restrictions. Orphaned tailings remain a long-term threat to water quality. Depending on the sealing characteristics of the clays in the tailings and the geologic nature of the impoundment terrain, damaging seepage of tailings solution into the ground water may or may not occur. Stream contamination by accidental discharges of waste solutions and tailings slurry (from failure of retaining dams and breaks in tailings pipes and flumes) occurs sporadically.

Process waste water which must be released to the enviroment may contain constituents similar to those in mine drainage - that is, high TDS, high or low pH, oil and grease, and a wide range of dissolved metals. In the U.S., NPDES limitations control the potential contamination of surface streams. For smelters and refineries, zero discharge is required except where excess rainfall occurs. In other countries regulations vary. There is a potential for significant contamination of receiving waters if no treatment (usually lime addition for pH control and precipitation) is employed.

Slag from copper and nickel smelting is basically inert, artificial rock that does not affect water quality. The slags are sometimes sold as crushed stone, or retained for future reprocessing. However, slag from copper smelting and refining, along with converter and reverberatory dust, and acid plant sludge, were included in the initial list of hazardous waste processes proposed by EPA as part of their RCRA regulations. However, only acid plant sludge was listed as a hazardous waste on May 19, 1980 (Federal Register, 1980).

As with land use, this discussion has not considered potential uses or degradation of water resources which would accompany secondary development around a new, or expanded, mine or processing facility. New residents would, of course, require water for domestic purposes. Power plants require substantial quantities of cooling water. Summed together, the demand on water resources in an arid region can be a prime limiting factor in development of mineral resources.

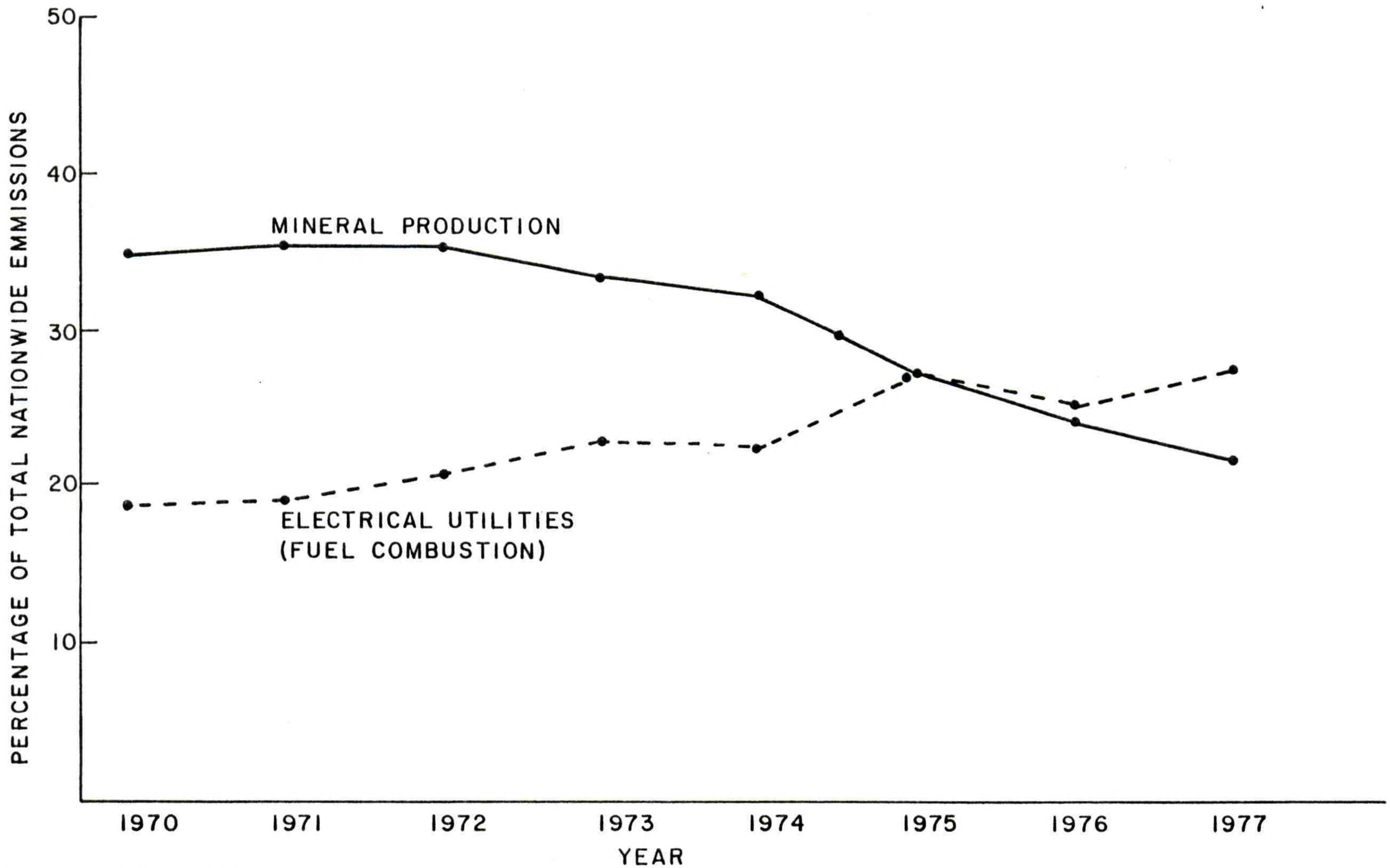
1.2.3 Air Quality

The production of some or all of the metals under consideration in this investigation usually results in a significant impact on air quality at some point during the production process. The most significant impacts occur during the mining and ore reduction stages (including smelting) of the process, although the level of impact is very dependent on the particular ore and method of processing and on the location of the activity.

1.2.3.1 Impacts of Mining and Milling

Particulate matter (common dust) accounts for virtually all of the air emissions produced by a typical surface mining operation. These particulates originate either from fugitive dust associated with blasting, wind erosion, general earth moving operations, traffic on haul roads, etc., or from processing equipment located at the mine such as a conveyor or a crusher. As an example of the magnitude of the emissions associated with mining operations, Figure 1-2 illustrates the estimated percentage of total nationwide particulate emissions (EPA, 1978) that were associated with mineral production for the years 1970-1977 in the United States. It should be noted that these estimates are for all phases of the mineral production process, and the majority of these emissions will originate at the mine in the form of fugitive dust. For comparison purposes, similar estimates are also shown for the particulate emissions from electric utilities (fuel combustion only). As can be seen, there is a distinct downward trend in mineral production emissions after 1972: however, in 1977 they still comprised nearly 25 percent of the total nationwide particulate emissions. It is expected that the primary reason for this apparent decrease in emissions is the implementation of various dust control strategies and installation of control equipment where possible.

It should be recognized that it is extremely difficult to estimate precisely the quantity of emissions from a given surface mining operation. This is largely because the emissions are not associated with any particular process, but rather they are a result of a combination of various natural and human-related activities which vary with time and location. As an example of this, Table 1-2 lists estimated quantities of fugitive dust from some typical mining operations. As can be seen there is a high degree of uncertainty associated with these estimates, with variations of up to three orders of magnitude for some



Source: EPA, 1978.

Figure 1-2. Estimated Percentage Annual Particulate Matter Emission for Mineral Production and Electric Utility Fuel Combustion.

TABLE 1-2

ESTIMATED QUANTITIES OF FUGITIVE DUST FROM SOME MINING OPERATIONS

Operation	Estimated Quantity of Dust
Overburden Removal	.0008-0.45 lb/ton of ore .048-0.16 lb/ton of overburden
Shovels/Truck Loading	up to 0.1 lb/ton of ore
Haul Roads	0.8-2.2 lb/mile traveled
Truck Dumping	.00034-0.04 lb/ton of ore
Waste Disposal	up to 14.4 ton/acre/year
Reclamation	Depends strongly on climate and soil

Source: PEDCo, 1976.

operations. The reason for this wide variation is that the absolute quantity of emissions is extremely dependent on the climate and the nature of the soil (i.e., one would expect a significantly greater quantity of fugitive dust at a surface mine located in an arid climate as compared to one located in a more humid climate).

With regard to the impact of these fugitive emissions on ambient air quality, the greatest impact is generally felt within only a few miles of the mine itself. The mass median diameter of fugitive emissions for surface mines is expected to comprise relatively large dust particles in the 10 to 35 μm size range (PEDCo, 1978), which is greater than the typical respirable particle size of 7 μm. Since these relatively large particles have a finite terminal settling velocity which increases with increasing particle size, they are expected to fall out of the particulate plume at relatively short downwind distances. For reasons such as these, fugitive dust from mines is generally considered (and observed) to be a localized problem associated with a particular mine, rather than a widespread regional problem. In other words, particulate matter concentrations at a given point are expected to be relatively independent of fugitive emissions which originate from surface mining operations located more than 10 km (6.2 miles) away, except possibly under very arid and windy conditions.

Particulate matter emissions from underground mining activities are extremely small in relation to fugitive emissions from surface mines. The major source of emissions at an underground mining operation is associated with ore transfer and processing (i.e., loading, unloading, conveying, crushing, sizing, etc.), with a small amount of emissions from mine air exhaust vents located at the surface.

The previous general discussion of the impact of fugitive emissions from surface mines should also apply to fugitive emissions from underground mines, except that the quantitity of emissions will be an order, or orders, of magnitude less than for the surface mine case. The difference in the two is primarily the large amount of dust created by the removal of the overburden and the disposal of large quantities of waste rock at surface mine sites (see Table 1-2). One additional point is the possibility that air quality will be harmed by radioactive material contained in the exhaust air from underground mines. Although this is not expected to be a major consideration for the type of mining operations of interest in this study, the problem is currently being studied by the Environmental Protection Agency and the Department of Energy.

There are also expected to be fugitive dust emissions associated with tailings from the ore processing operations. Most tailings consist of very fine particles which can cause substantial local nuisances in arid areas. Temporary control is usually attempted by physical methods, such as by water sprinkling or coverage with straw, bark, or gravel, and by applying chemical polymers which form a thin crust at the surface. Permanent control can only be achieved over large areas by establishing a growth of vegetation (Down and Stocks, 1977).

It should be noted that fugitive dust from mining and milling operations may contain very small amounts of heavy metals (e.g., such as lead, arsenic, cadmium, etc.). The emission of these substances is expected to occur only in very small quantities, particularly in relation to the total particulate emission rate.

1.2.3.2 Impacts of Smelting and Refining

The most widely used method of processing copper, nickel, and cobalt is by smelting the various mineral concentrates. Cobalt is also processed at many locations by using a hydrometallurgical or leaching process. Air quality impacts associated with these chemical leaching processes normally result in air quality impacts that are typically an order of magnitude less than for a smelting operation. Since blast furnace production of ferromanganese is not included in this study and little or no prior processing of manganese ore is required, no significant adverse air quality impacts are considered to occur. Inasmuch as the most significant and widespread air quality impacts are expected to occur during smelting processes, rather than from some of the other chemical processes, the following discussion is concentrated primarily on that aspect of ore processing.

Regulatory agencies in North America are probably more concerned with the emissions of SO_2 from smelters than with any other pollutant, the reason being that SO_2 is a noxious pollutant and is often emitted in very great quantities at these facilities. In light of this, regulatory imposed emission limitations are becoming more widespread, particularly for new and proposed facilities. In North America, the construction of new smelting facilities can typically be expected to be contingent upon the removal of approximately 90 percent of the SO_2 contained in the combustion gas. This is usually accomplished by using the SO_2 laden exhaust gas to produce sulfuric acid. Furthermore, ambient air quality regulations prohibit excessive emission of SO_2 from both new and old facilities from degrading the air quality to unacceptable levels.

The primary impact of large SO_2 emissions from a typical smelting operation is expected to occur on a local scale in the form of relatively high ground-level concentrations of SO_2. In most countries, ambient air quality regulations are enforced which prohibit excessive ground-level SO_2 concentrations which could pose a hazard to human health and, as a result, significant adverse effects to humans are not expected to occur. It should be noted, however, that the possibility does exist for some localized damage to soils and vegetation as a result of the operation of a large smelter.

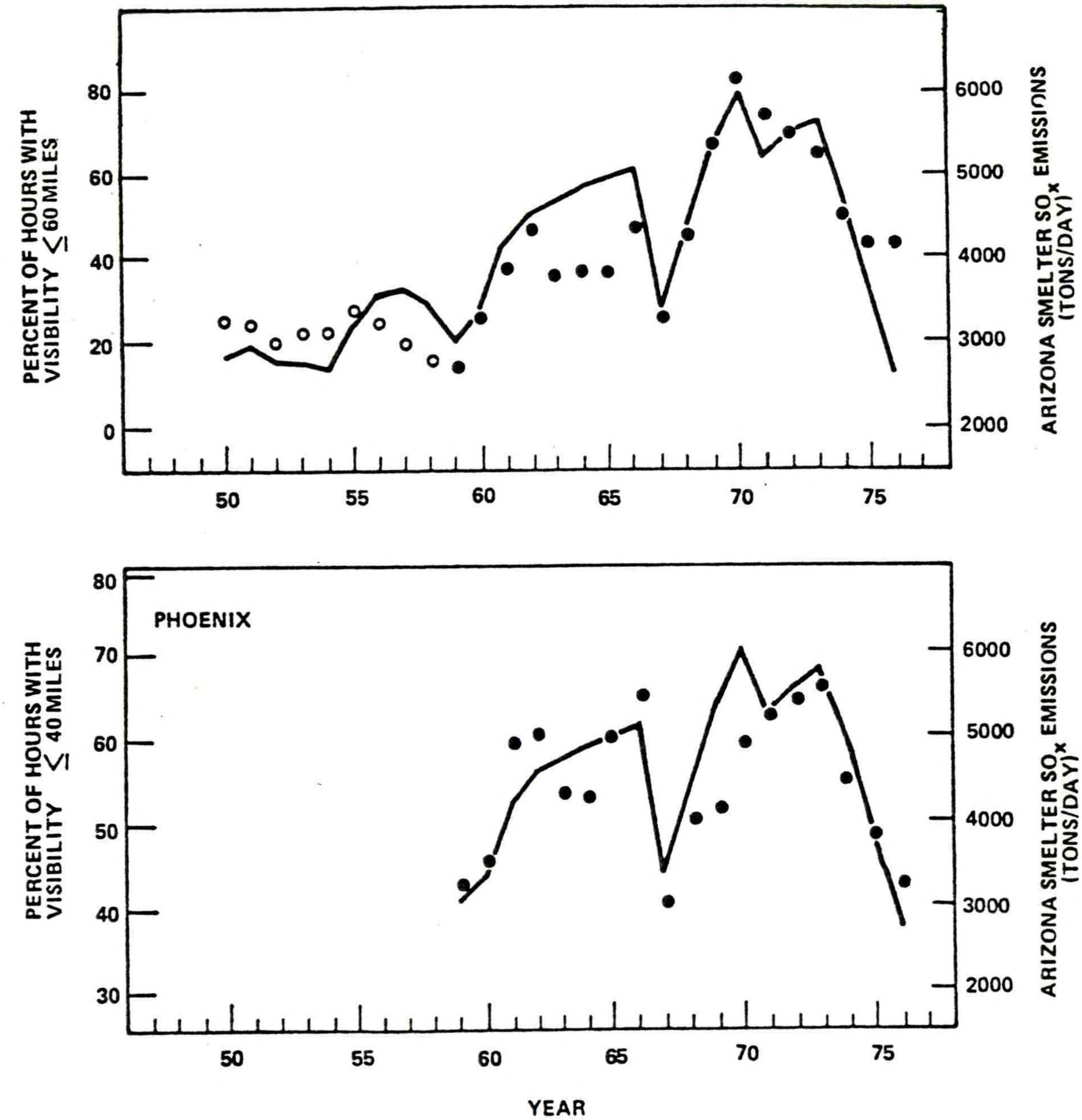
A secondary impact of SO_2 emissions from smelters that has been noted in recent years is that of reduced visibility within the same region that large smelters are located. Marians and Trijonis (1979) used multiple regression techniques to derive quantitative relationships between yearly extinction levels for six Arizona airport visibility data sets, yearly average emissions of SO_x from Arizona smelters, and statewide emissions of SO_x, NO_x, and non-methane hydrocarbons (NMHC). Their analyses showed that the total SO_x emissions from

Arizona smelters was by far the most significant variable for each of the data sets and the only significiant variable for five of the six data sets. The particularly close relationships between Arizona smelter SO_x emissions and visibility at Tucson and Phoenix are illustrated in Figure 1-3. The implication of the findings of Marians and Trijonis is that there is an approximate direct relationship between the number of hours of observed decrease in visibility and the emissions of SO_x.

This apparent relationship between visibility and smelter SO_x emissions in Arizona is not suprising in light of the extremely large emissions arising from smelter operations. For example, during the late 1960s and early 1970s, statewide Arizona NMHC and NO_x emissions represented less than 1 percent of the nationwide total NMHC and NO_x emissions, but statewide SO_x emissions (greater than 90 percent of which are believed to come from smelters) constituted over 6 percent of the nationwide total SO_x emissions (EPA, 1979b). It is therefore apparent that in regions where SO_x emissions either are, or are expected to be, relatively large, visibility could conceivably become a significant problem, depending on the level of SO_x emission in the particular region.

Unlike the fugitive dust emissions from mining operations which impact primarily on localized areas in the immediate vicinity of surface mine sites, gaseous emissions such as oxides of sulfur and nitrogen, can result in significant ground-level concentrations at very large distances from the source (i.e., on the order of hundreds of miles, depending on emission release height and meteorological conditions). During this long-range transport, these pollutants may cross geographical and/or political boundaries. This situation can in turn create national and international regulatory problems in that the air quality regulations of one state or country can have an indirect impact on the natural resources of another.

Aside from the rather obvious direct effects of long-range transport at distant locations, recent investigations by numerous agencies



● THE DOTS REPRESENT YEARLY PERCENT OF HOURS WITH REDUCED VISIBILITY (MEASURED ON LEFT AXIS). NOTE THAT THE TUCSON OBSERVATION SITE MOVED IN 1958; ALTHOUGH THIS MOVE DID NOT PRODUCE A STATISTICALLY SIGNIFICANT CHANGE IN REPORTED VISIBILITIES, OPEN DOTS ARE USED TO DISTINGUISH DATA PRIOR TO 1958.

—— THE LINES REPRESENT YEARLY SO_x EMISSIONS FROM ARIZONA COPPER SMELTERS (MEASURED ON RIGHT AXIS).

Source: Marians and Trijonis, 1979.

Figure 1-3. Historical Trends in Hours of Reduced Visibility at Phoenix and Tucson Compared to Trends in SO_x Emissions from Arizona Copper Smelters.

and individual investigators indicate that in various parts of the world, acid rain is becoming an increasingly important problem. Acid rain is a result of various chemical reactions, which convert airborne pollutants such as sulfur and nitrogen oxides into sulfuric (H_2SO_4) and nitric (HNO_3) acids which can fall to the surface of the earth as components of rain or snow. Other acids, such as hydrochloric acid (HCl) can also contribute to the acid rain problem, however sulfuric and nitric acids are the more commonly observed types. The most significant effects of acid rain which have been observed are damage to surface vegetation and a significant increase in lake water acidity.

Though unpolluted rainwater is normally slightly acidic with a pH of 5.4 (a pH of 7 is neutral), the average annual rainfall pH in the eastern United States is estimated to be less than 4.5 (Likens, 1976). Lakes that lack the buffering capacity, or the ability to chemically neutralize this acidity, face potential serious ecological harm. Indeed, hundreds of Norwegian lakes and lakes in the northeastern United States (e.g., in the Adirondacks of New York State) have become so acidic that they will no longer support fish life.

Although there is not a great deal of quantitative information available on the subject of acid rain, it should be kept in mind that substantial increases in regional SO_x (and to some extent NO_x) emissions could result in a significant increase in acid precipitation, not only locally, but even at relatively large distances from the source of emission.

As with land use and water resource impacts, potential impacts of secondary development have not been considered in detail. Though additional traffic associated with commuting workers and new communities may contribute to local emissions, the major potential is from combustion of fuels for power production. SO_2 emissions from coal combustion can be just as damaging to air quality as those from smelters.

1.2.4 Ecology

Disturbance of wildlife habitat, both onsite and in neighboring areas, is a common result of mining and processing metallic minerals. Damage is caused by: the intrusion of man, machines, odor, and noise; destruction of vegetation and other terrestrial habitats; and changes in water flow and in water and air quality. The extent of long-term damage depends on the nature of the mining operation and on the extent of controls. Deterioration of habitat may stem from destruction of wetlands, chemical pollution of surface and ground waters from mine drainage and process wastes; stream channelization and altered flood drainage; and air pollution. Tailings ponds may contain toxic solutions and oil scums that are hazards to wildlife, especially migrating water fowl (National Research Council, 1979).

Long-term effects depend on the adequacy of pre-mining planning and operating controls, and on the success of reclamation. After reclamation, an area damaged by mining may show increased carrying capacity for wildlife, especially if revegetated with mixed plant types. Lakes created by surface mining, such as those from abandoned open pit operations, may provide new habitat for waterfowl and other aquatic species. A steady increase in the deer and elk population in the vicinity of the Henderson molybdenum mine in Colorado is attributed to game management practices that compensate for mining's adverse effect on these species (National Research Council, 1979).

1.2.5 Socioeconomics

The potential socioeconomic impacts of mining and processing are sometimes difficult to define and quantify, particularly in remote areas. Consequently, it has not been until recently that these impacts have been considered as elements which must be addressed in the decision-making process concerning whether, where, and how to mine.

° Economic Impacts: These are measured in dollars and stem from changes in private and public revenue and cost flows related to a project. Economic impacts typically induce community growth.

° Social Impacts: These are less easily measured but are real nonetheless, and can be described as either aggravating or diminishing local social problems, or people's perceived social well-being. Social impacts often result from induced growth.

The magnitude of these impacts is determined largely by the actions (both dependent and independent) of three disparate groups:

° The government (all levels), which controls, to varying degrees, where mining may be permitted through zoning, taxation, and environmental policies. These policies often form the basis for the development or the exclusion of the mining industry.

° The mining industry, whose activities produce socioeconomic impacts in the course of meeting consumer demand for minerals (in this study, cobalt, copper, manganese, and nickel). These impacts are to some degree a function of the location, the technology employed, the duration of the project, and the size of the requisite labor force.

° The local population, whose size and distribution, economy, infrastructure, political structure, and community social structure and composition all may be directly or indirectly involved.

Ideally, the costs and benefits of each mining project should be borne equally by the affected parties. More typically, however, some share of costs/benefits commonly have accrued to third parties or to society in general. In the United States, society has become increasingly aware that payers of the external costs of development (negative externalities) are not necessarily benefactors to the same extent. In economic terms, the marginal social costs are greater than the marginal social gain. Litigation and other administrative actions to improve the balance have become common sources of delay in project developments.

A cursory examination of a typical mine development project reveals the dissociation of costs from benefits. The immediate economic beneficiaries of a given project are numerous and usually spread over a wide geographic area: local employees of the mining firm and out-of-region people who migrate to the area or commute to work; the local housing market and distant manufacturers of mobile homes; local merchants and merchants in distant areas benefitting from sales to dependents to whom some share of wages may be sent; income accruing to contractors and suppliers both local and regional; increase in the tax base primarily in the local community but also throughout the region. At the same time, the people who absorb the external, social costs of a project are typically local or area residents.

1.2.5.1 Local Socioeconomic Impacts

The size and proximity of nearby population centers is an important factor in determining the extent of socioeconomic impact of mining operations. Typically, population centers within 100 miles of the mine site will be affected during the development phase, and those within 30 miles or so, thereafter. (This admittedly broad generalization is a function of generally acceptable commuting distances.)

In terms of size, the larger the community, the less substantial the impact of mining. Small communities lack sufficient capacity in terms of labor, infrastructure, housing, community services, etc. to accommodate the induced growth resulting from mine construction and operation. In such communities a typical series of socioeconomic impacts occurs. The development of a mine brings with it an immediate demand for labor, only a small portion of which can be locally supplied. This unmet demand for labor is commonly met by in-migration of workers for the duration of the construction; these workers may move their families, but many do not. The permanent labor force for the operating mine is typically smaller than the construction work force, but often requires outside hiring, which induces new households to enter the community. (Typically, the combination mine employment-dependency multiplier in the United States is between 5 and 7. Thus, a mining operation bringing in 100 new miners to a community is likely to add a total of 500 to 700 people to the local population.) The increase in population often results in inadequate public and private services, which tends to reduce the living standards and lifestyles of many old as well as new residents, and, as a result, frequently generates high levels of labor turnover. This turnover of labor may reduce productivity in the local economy and undermine the social stability of the community.

Not all communities that host a mine development are so adversely affected. For the above-described impacts to occur, growth must be both rapid and large relative to existing population and employment levels. If these factors are absent or somehow mitigated, or if sufficient excess capacity exists in the social infrastructure to accommodate growth, the problems may be avoided. Also, in many areas, housing and services are provided by the mining company in a company townsite. The following section reviews the principal factors which directly influence local impact severity.

1.2.5.2 Site Specific Factors Affecting Degree of Impact

Four empirical measures are of prime importance in determining the potential severity of the local socioeconomic impact:

- The population size of the affected area at the time of impact;
- The population density of the surrounding region;
- The proximity to the nearest regional trade center; and
- The existing relationship between basic and secondary employment.

These affect other site specific characteristics which influence the impact severity, such as the availability of local workers. This, in turn, depends on local labor force participation rates that depend upon cultural and economic factors, as well as upon normal growth rates for the area. Obviously, many factors can combine to produce an infinite number of unique situations. However, through the use of the four basic measures outlined above, in conjunction with known site specific information, an assessment of the social and economic impacts of a particular mining operation can be made.

The first variable, population size, represents the absolute size of the labor force potentially available to fill newly created jobs. The second variable, population density, serves as a rough measure of the proximity of the labor force to the development site, and therefore, the availability of the commuting work force. Since the population growth associated with any new mining or processing facility is dependent upon the number of jobs created by the project and left unfilled due to shortages of available local workers, these two measures are of great importance. The third variable, the proximity of major trade centers, reflects the amount of economic activity in an area. Lastly, the existing relationship between basic and secondary employment is important as a measure of the proportion of the total work force employed in the provision of secondary goods and services.

A more detailed analysis of the ability of specific sites to cope with mining-induced growth would include identification of the following variables:

- Housing stock
- Educational facilities
- Medical facilities and public health
- Fire and police protection
- Road traffic
- Recreational facilities and usage
- Public utilities
  - power
  - water
  - sewage treatment
- Public finances
  - property taxes
  - any other taxes

The first seven variables listed are indicators of public services necessary to support growth in the community. The "public finances" variable describes the ability to pay for the requisite increases in public services.

Because of the wide range of communities which could be affected by mining and processing of metallic ores around the world, it is impossible to provide a comprehensive analysis of potential impacts. Using the concept detailed above, a qualitative judgement of potential impact severity is made for several specific sites in Chapters 3.0 and 4.0.

1.2.6 Energy Resources

Mining and processing of metallic minerals requires very large amounts of energy. The degree of energy use is strongly dependent on ore grade, which basically determines the amount of material to be moved per unit of metal content mined, and also on a host of other factors such as rock type and location, topography, and distance from market areas. Ore concentration and processing are also very energy intensive. Energy needs may be met by importing fuel stocks by truck, or electric power by transmission lines. In remote areas, new sources of electric power may have to be constructed. Use of excess public power supplies may hasten the need for new plant capacity and, ultimately, higher rates for the local populace. In today's increasingly energy-limited society, the cost and availability of energy supplies represent a significant new factor in the development of mineral resources. Some perspective on the quantities of energy required to meet the world and U.S. demand for copper, nickel, cobalt, and manganese is provided in Chapter 2.0.

1.3 THE LEGISLATIVE ENVIRONMENT

1.3.1 The United States

Concerns over impacts of growth on the natural and man-made environment have led to the enactment of numerous Federal, state, and local laws to prevent further deterioration and to repair damage already wrought.

1.3.1.1 Federal Regulations

The National Environmental Protection Act (NEPA) was passed in 1970. This act is "our basic national charter for protection of the environment. It establishes policy, sets goals, and provides means for carrying out the policy" (Federal Register, 1978, p.55990). NEPA is administered by the Council on Environmental Quality (CEQ). Agencies which are responsible for interpretation of other environmental regulations are responsible to CEQ.

The Clean Water Act (The Federal Water Pollution Control Act of 1972 and 1977 Amendments) is administered by the U.S. Environmental Protection Agency (EPA) or by state pollution control agencies to which permit authority has been delegated. Any new point source may be required to obtain a National Pollutant Discharge Elimination System (NPDES) permit before the facility can begin operation. The purpose of the act and the permits is to achieve a water quality level fit for swimming and for propagation of fish, shellfish, and wildlife by July 1, 1984; and to eliminate discharge of all pollutants to the nation's waters by 1985. Limitation guidelines have been established for the four minerals discussed here (EPA, 1975).

The Clean Air Act of 1970, and Amendments, is also administered by the EPA. Under this act, concentrations of atmospheric pollutants are limited to meet National Ambient Air Quality Standards (NAAQS). In areas where the air quality is better than the NAAQS, Prevention of Significant Deterioration (PSD) limits are set. These limits are expressed as maximum allowable increases over ambient conditions as they existed on August 7, 1977.

RCRA (The Resource Conservation and Recovery Act of 1976) is also administered by EPA. This act regulates the disposal of solid wastes and defines certain materials as hazardous or special which must be disposed of according to strict regulations; it also provides for monitoring and inspection of the waste disposal sites.

The Surface Mining Control and Reclamation Act of 1977 applies specifically to surface coal mines. For many reasons it is not now considered applicable to other surface mining activities. The report "Surface Mining of Non-Coal Minerals" was prepared by the National Research Council (1979) to serve as a basis for discussions regarding the applicability of the law to mining for non-coal minerals and for possible future regulations.

1.3.1.2 State and Local Laws

As noted in the previous section, the states may submit to EPA plans for regulations of effluents and emissions and may be given permitting authority. Not all states have, however, been given this authority and for those states the authority for issuance of these permits remains with EPA.

Most states have land use and mining laws, and most local governments require permits for construction, road building, sewage disposal, etc.

1.3.2 Other Countries

The United States is a mature nation of great wealth and highly diversified economy. Many developing nations, including those for which metals export is a major source of international credit, often find it economically infeasible to establish regulatory controls which would provide equivalent protection to human health and welfare and to the natural environment. Many countries with major mineral resources are still in the youthful stage of development; concerns over conflicting land use or preservation of wildlife or scenic areas are considered an infringement of personal liberty or otherwise have low priority. It is not surprising, therefore, that for most of the foreign sources discussed in Chapter 3.0 there is little concern for environmental regulation. Also, for several of the sites, the mines are far from population centers and/or are in areas that have few other land uses, or do not conflict with other land uses because of the large undeveloped acreages available. Generally, it may be concluded that development of identical mines and/or processing facilities in virtually any other country will have potential for greater adverse environmental effects than in the United States because of a less restrictive regulatory system.

2.0 AGGREGATE ENVIRONMENTAL IMPACTS

2.1 INTRODUCTION

Two approaches are used in assessing the potential environmental impacts of continued reliance on land mining of copper, nickel, cobalt, and manganese during the period 1980-2010. The first, presented in this section, consists of projecting worldwide supply sources and assigning approximate impact parameters per unit output. Although necessarily approximate, selection of appropriate categories and parameters provides an interesting overview of some of the most evident effects on a large scale. The second method, presented in Chapters 3.0 and 4.0, consists of describing specific mining and processing activities, either presently in operation or expected, as examples of the kinds of environmental disruption which can occur. By selecting a representative range of locations and activities, this method provides a more subjective interpretation of the consequences of mining and processing. These two methods of documentation provide the perspective needed to interpret potential consequences of future mining decisions.

The remainder of Chapter 2.0 presents the results of the aggregate impact analysis. Section 2.2 gives projections of worldwide and U.S. demand for the four metals during the period 1980-2010. Also presented are projections of world supply by deposit type, apportioned into surface and underground mine sources. Section 2.3 presents the aggregate impact parameters selected for the analysis. In Sections 2.4, 2.5, 2.6, and 2.7, these parameters are applied to the projected supply sources for copper, nickel, cobalt, and manganese, respectively, to calculate the degree of impact on land, energy, and other resources. Section 2.8 summarizes the impacts attributable to land mining for all the metals and provides some perspective to the significance of the calculated totals.

The basis for this aggregate analysis is free market access to available world supplies of copper, nickel, cobalt, and manganese.

This is most appropriate to a consideration of overall world demand impacts. The tables in this section provide estimates of impacts due to U.S. demand only as a fraction of world demand. That is, no attempt has been made to specify the subset of supply sources likely to provide ore for U.S. consumption. Therefore, in considering impacts attributed to U.S. demand, it must be remembered that the actual impacts will differ from those projected to the extent that there are differences in source deposit types, especially under the assumption of U.S. self-reliance (Chapter 4.0).

2.2 PROJECTED LEVELS OF DEMAND AND SUPPLY DURING THE PERIOD 1980-2010

Part I of this report provides high, low, and most likely estimates of demand for each of the metals of concern. For the purpose of projecting environmental impacts, the most likely world and U.S. demands are used. World demand for copper is projected to increase from 9.3 million tons in 1980 to 28.2 million tons in 2010; for nickel, from 669 thousand tons to 1.52 million tons; for cobalt, from 27.7 thousand tons to 75 thousand tons; and for manganese, from 12 million tons to 28.6 million tons (Table 2-1).

In order to evaluate potential environmental effects, it is necessary to project the sources for each metal by major deposit type and mining method. This is done to obtain some reasonable degree of uniformity for application of the impact parameters described in Section 2.3. Tables 2-2 through and 2-5 and Figures 2-1 through 2-4 provide the estimated supply breakdown for the individual metals, assuming supply meets demand. These breakdowns are based on a combination of present supply data, knowledge of worldwide reserves, and expected trends toward specific deposit types and mining methods. These projections are made for worldwide supplies, not for the supply mix likely to be used to meet U.S. demand.

Table 2-2 and Figure 2-1 give the supply breakdown for copper, allocated among porphyry, stratabound, copper-nickel sulfide, and massive sulfide deposits. Porphyries (e.g., southwestern U.S. and western South America) are expected to provide more than 50 percent of the total supply during the period, with most coming from surface mines (open pits) but rapidly increasing amounts from underground mines as the better surface grades are depleted. Stratabound deposits (e.g., Zambia and Zaire) will provide nearly 25 percent of the total copper supply during the period, with two-thirds coming from surface mines. Massive sulfides (e.g., Canada and Australia) and copper-nickel sulfides (e.g., Canada and Soviet Union) will provide the remainder of the copper, both almost entirely from underground mines.

TABLE 2-1

SUMMARY OF MOST LIKELY WORLDWIDE AND U.S. DEMAND FOR COPPER NICKEL, COBALT, AND MANGANESE METALS, 1980-2010
(1000 Short Tons)

	Copper		Nickel		Cobalt		Manganese	
Year	World	U.S.	World	U.S.	World	U.S.	World	U.S.
1980	9318	1989	669	216	27.69	11.09	12,012	1539
1985	11,207	2291	768	252	32.69	13.06	13,878	1670
1990	13,480	2638	881	296	38.59	15.37	16,034	1811
1995	16,213	3039	1010	346	45.57	18.09	18,524	1965
2000	19,500	3500	1158	405	53.80	21.30	21,400	2130
2005	23,454	4031	1328	475	63.52	25.07	24,726	2312
2010	28,209	4643	1524	556	75.00	29.51	28,567	2508
Cumulative	530,442	95,255	31,365	10,868	1448	574	594,547	60,012

TABLE 2-2

SUMMARY OF PROJECTED WORLDWIDE SUPPLY OF COPPER
BY MAJOR DEPOSIT TYPE, 1980-2010
(1000 Short Tons)

	Porphyry		Copper-Cobalt Stratabound		Copper-Nickel Sulfide	Massive Sulfide
Year	S	UDG	S	UDG	UDG	UDG
1980	3653	913	1623	800	932	1398
1985	4330	1443	1857	915	1043	1619
1990	5420	1806	2139	1054	1178	1885
1995	5832	3140	2478	1221	1339	2205
2000	7197	3875	2886	1421	1533	2589
2005	7479	6120	3375	1663	1766	3052
2010	9150	7487	3965	1953	2047	3608
Cumulative	185,017	110,902	80,269	39,536	43,115	71,603
By Deposit Type	295,919		119,805		43,115	71,603
Percent of Total Demand	55.8		22.6		8.1	13.5

S = surface mine. UDG = underground mine.

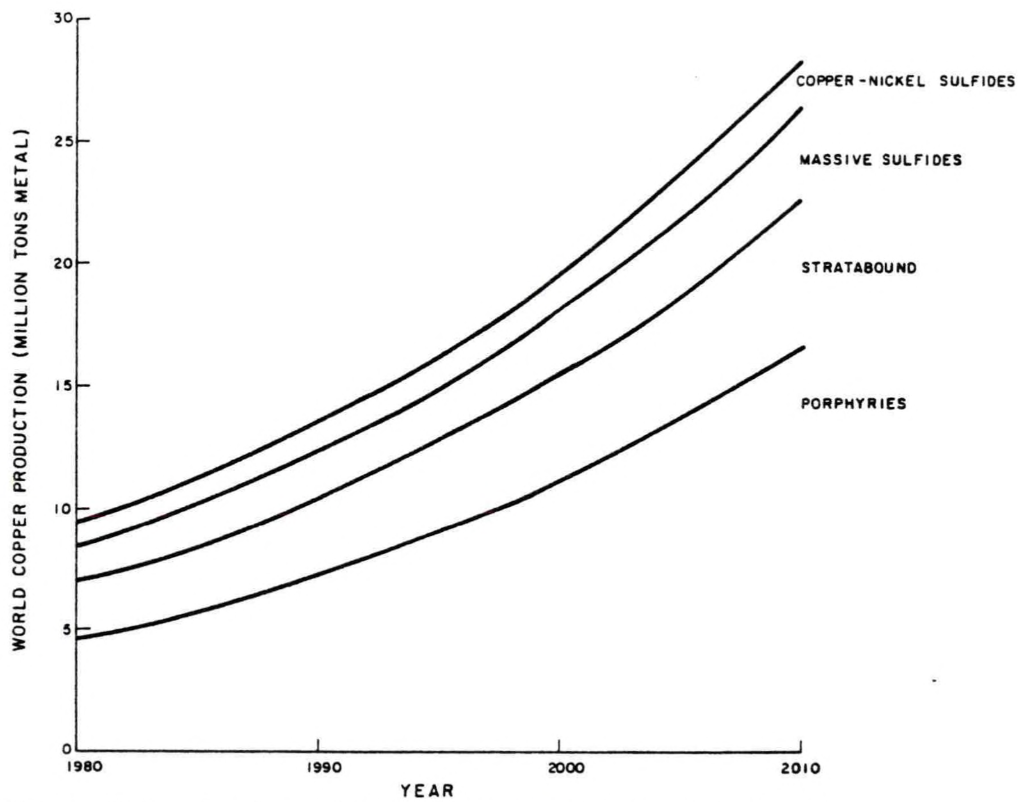


Figure 2-1. Projected Annual Copper Production from Land Resources.

Nickel is mined from two principal deposit types, copper-nickel sulfides (underground mines) and laterite deposits (surface mines). Laterites (subtropics) presently supply about 60 percent of the world's nickel and this percentage is expected to increase during the next 30 years (Table 2-3; Figure 2-2). As will be evident in Section 2.3, this trend could be affected by the high energy cost of processing nickel laterites.

Cobalt is mined as a by-product of copper and nickel in three major types of deposits (Table 2-4; Figure 2-3). More than half the world's supply comes from stratabound deposits as a by-product of copper production, mostly from surface mines. The remaining cobalt is obtained as a by-product from copper-nickel sulfides or from nickel-cobalt laterites, the former from underground and the latter from surface mines.

Manganese is mined from oxide or carbonate ores throughout the world. It is projected that the present 60/40 percentage split between underground and surface mines will be retained through 2010 (Table 2-5; Figure 2-4).

TABLE 2-3

SUMMARY OF PROJECTED WORLDWIDE SUPPLY OF NICKEL BY
BY MAJOR DEPOSIT TYPE, 1980-2010
(1000 Short Tons)

Year	Copper-Nickel Sulfide UDG	Nickel-Cobalt Laterite S
1980	268	402
1985	269	499
1990	264	616
1995	293	717
2000	324	834
2005	365	963
2010	412	1113
Cumulative	9281	22,075
Percent of Total Demand	30	70

S = surface mine. UDG = underground mine.

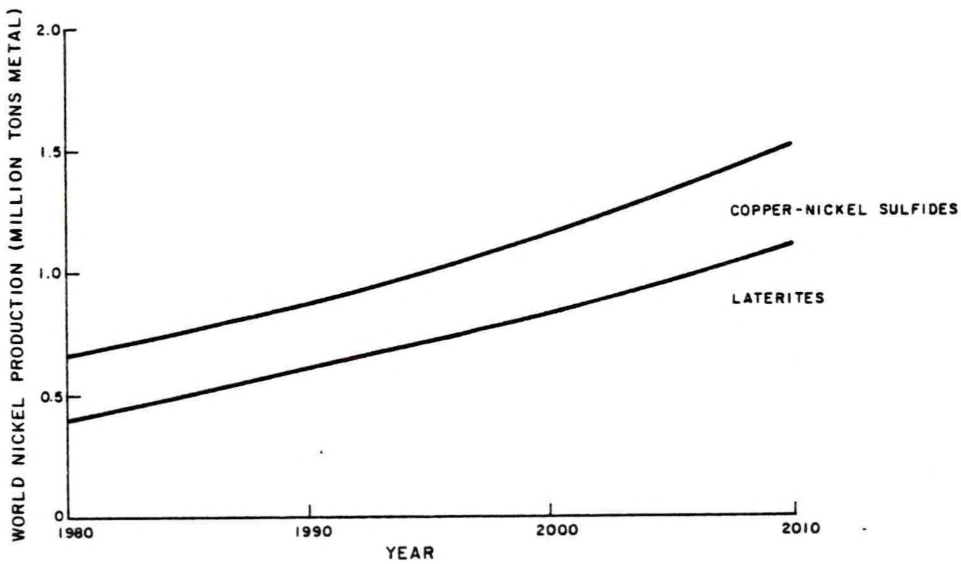


Figure 2-2. Projected Annual Nickel Production from Land Resources, 1980 - 2010.

TABLE 2-4

SUMMARY OF PROJECTED WORLDWIDE SUPPLY OF COBALT
BY MAJOR DEPOSIT TYPE, 1980-2010
(1000 Short Tons)

	Copper-Cobalt Stratabound		Copper-Nickel Sulfide	Nickel-Cobalt Laterite
Year	S	UDG	UDG	S
1980	10.2	5.0	6.9	5.5
1985	12.0	6.0	8.2	6.5
1990	14.2	7.1	9.6	7.7
1995	16.8	8.4	11.4	9.1
2000	19.8	9.8	13.5	10.8
2005	23.4	11.5	15.9	12.7
2010	27.7	13.6	18.8	15.0
Cumulative	533.6	262.8	362	289.6
By Deposit Type	796.4		362	289.6
Percent of Total Demand	55		25	20

S = surface mine. UDG = underground mine.

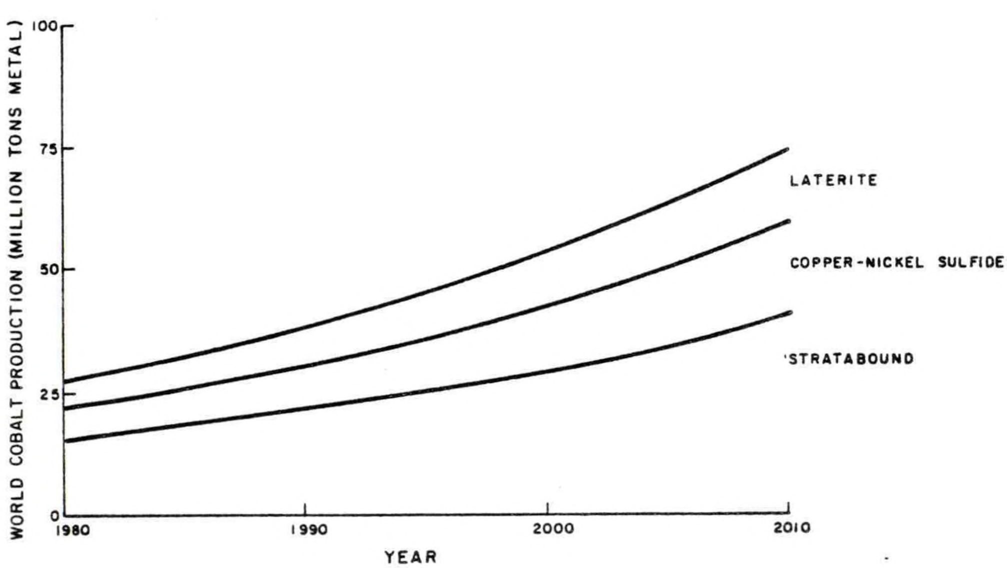


Figure 2-3. Projected Annual Cobalt Production From Land Resources, 1980 - 2010.

TABLE 2-5

SUMMARY OF PROJECTED WORLDWIDE SUPPLY OF MANGANESE, 1980-2010
(1000 Short Tons)

Year	Manganese S	Manganese UDG
1980	4805	7207
1985	5551	8327
1990	6414	9620
1995	7410	11,114
2000	8560	12,840
2005	9890	14,836
2010	11,427	17,140
Cumulative	237,819	356,728
Percent of Total Demand	40	60

S = surface mining. UDG = underground mining.

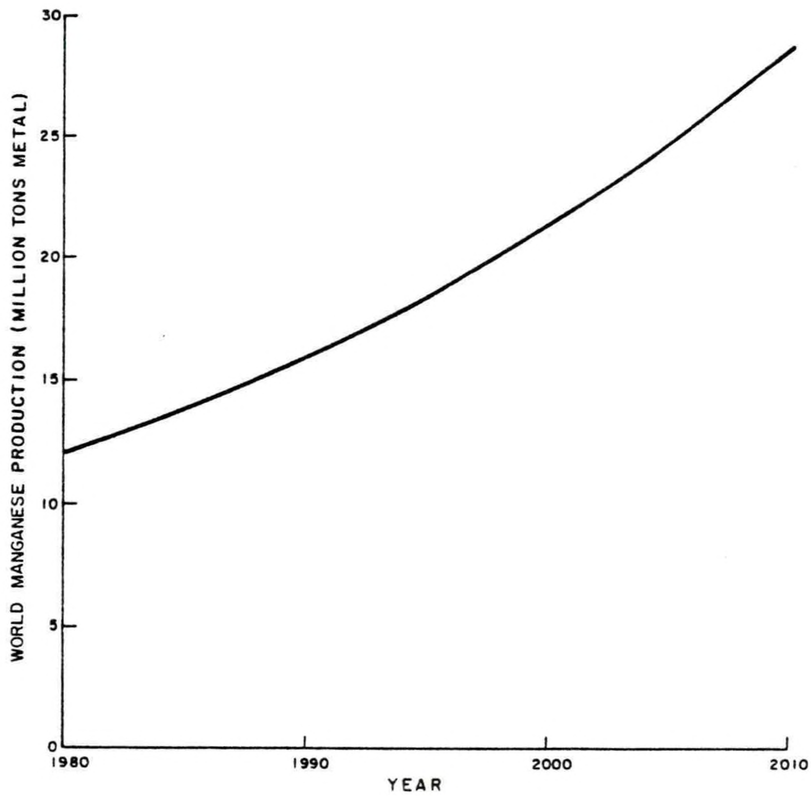


Figure 2-4. Projected Annual Manganese Production from Land Sources, 1980 - 2010.

2.3 IMPACT PARAMETERS

Parameters selected as indicators of aggregate worldwide impacts must be applicable to virtually all land-based mining and processing activities. Transportation impacts are not included since they are too site specific for such a broad generalization. Parameters were selected to describe five categories of impacts: land use disturbance, energy use, water use, air quality, and socioeconomics (Table 2-6).

TABLE 2-6

IMPACT CATEGORIES SELECTED FOR AGGREGATE ANALYSIS

Land Use Disturbance

- Mining
- Waste rock
- Process plants
- Tailings ponds

Energy Use

- Mining
- Processing

Water Use

- Processing

Air Emissions

- SO_2 emissions

Socioeconomics

- Employment
- Disabling injuries

2.3.1 Description of Impact Parameters

2.3.1.1 Land Disturbance

Mining - Surface mining of the ore body has obvious direct effects on the land. Open pit mining (as with most porphyry copper deposits) is carried out in a relatively confined location, producing a large, deep pit in conformance with the shape of the ore body. Laterites are strip mined from shallow deposits in a manner similar to coal. Land disturbances are less noticeable though more extensive than with open pits. Laterite mines are also more readily restored to original conditions.

Underground mining also has potential for surface disturbance. In addition to land use for head frames, warehouses, etc., certain mining methods have substantial potential for producing surface subsidence. This is particularly true with block caving, a common method of underground mining of porphyry copper.

Waste Dumps - Overburden, including low grade ore not economically recoverable, must be removed and discarded prior to mining the ore body. The volume of overburden in surface mining is frequently greater than that of the ore and is costly to move. The extent of land disturbance is often greater than with mining. With underground mines there is generally less waste rock to be moved; often the rock can be left in the mines.

Process Plant Facilities - These include benefication plants which concentrate the minerals from the mined ores and are usually located adjacent to the mine. Also included are smelters and refineries which process the concentrates into a commercially marketable metal form. The areal extent of these facilities is quite large and independent of the mining method. Smelters and refineries often serve several mines in a region and may be located quite distant from mining activities.

Tailings Ponds - During beneficiation, the metal- or product-bearing minerals are separated from the gangue or host rock. As the minerals usually constitute a small fraction of the ore body, large volumes of tailings are transported in a slurry form to ponds for settling and disposal. The size of tailings ponds is a function of the deposit type rather than the mining method.

Other Land Disturbances - Categories of land disturbance listed above are the most obvious and readily quantified. However, they are not all inclusive. Other land is required for water supply reservoirs, temporary material and waste storage, roads, and other uses. In addition, the mine and processing facilities are not laid out with the purpose of conserving land but to improve production efficiency (while avoiding use of land directly above potentially recoverable ore bodies). The total acreage of land impacts, direct and indirect, may be twice the sum of the components described. Quantification of this additional land is considered too site specific for inclusion in this analysis, however.

2.3.1.2 Energy Use

Mining - Diesel and electric energy is required to fuel drilling machines, shovels, trucks, rail cars, and draglines, and to perform a multitude of other types of work associated with mining the ore body. The lower the grade of ore the greater the volume of rocks to be removed and the greater the energy required to recover a given quantity of metal.

Processing - Conveyors, pumps, crushers, agitators, aerators, furnaces, and electrolysis are all substantial users of energy in processing the ores into a finished metallic form. In this analysis, energy use is expressed in British thermal units (Btus), with appropriate account taken for the losses in converting to electricity.

2.3.1.3 Water Use

Mining - Minimal quantities of water are needed for mining. In fact, water often must be pumped out of the mine and can be used in processing the ore.

Processing - Substantial quantities of water are used in flotation and leaching processes, in transporting waste rock to the tailings ponds, in controlling dust on roads, for noncontact cooling, and for many other purposes. Although the use of water will vary from site to site, the conservative assumption has been made that recycling and conservation will be practiced to the maximum extent possible.

2.3.1.4 Air Emissions

Although there are other potential air quality impacts from mining and processing activities, such as particulates and engine exhaust, SO_2 emissions from smelting operations was selected as the most significant, quantifiable source of impacts. As SO_2 emission control is frequently not applied, potential, or uncontrolled, emissions are estimated on the basis of actual measurements and the typical mineral content of the various ore types.

2.3.1.5 Socioeconomic Impacts

Employment - Estimates were made of the manpower required to mine the ore body, concentrate the minerals, and purify the metal content.

Disabling Injuries - Employment in processing plants and surface mines is statistically much safer than in underground mines. Data published by the U.S. Bureau of Mines on injury statistics were used (Given, 1973). It is likely that U.S. accident rates are considerably lower than those in other countries, however. Thus injuries may be understated on a worldwide basis.

2.3.2 Quantification of Impact Parameters

Data were obtained from the literature, from industry representatives, and from experienced mining and processing engineers to develop the impact parameters shown in Table 2-7. The parameters are necessarily educated estimates of industry-wide averages. It is recognized that individual mines and process plants may vary substantially from the estimates provided. The intent is to provide a reasonable projection of the total utilization of resources expected to result from continued reliance on land mining for copper, nickel, cobalt, and manganese over the next 30 years.

The parameters in Table 2-7 are expressed in units of resource use per 1000 short tons of metal content mined or processed. Any comparison among parameters must recognize the importance of ore grade, mining methods, mineral type, and other factors assumed in their development. Where two or more metals are produced from the same deposit type, some of the impacts are not cumulative (mining and waste dump acreage, for example). In these cases, the parameter is quantified under the primary metal only.

TABLE 2-7

SUMMARY OF UNIT IMPACT PARAMETERS USED IN AGGREGATE ANALYSIS OF WORLDWIDE MINING AND PROCESSING

	Resource Use Per 1000 Short Tons Metal									
	Mining Activities			Processing Activities						
	Land for Mine Site (Acres)	Land for Waste Dumps (Acres)	Energy (10^{10} BTU)	Land for Process Plants (Acres)	Land for Tailings Ponds (Acres)	Energy (10^{10} BTU)	Water Use (10^7 Gal)	Potential SO_2 Emissions (Tons)	Total Employment (Man years)	Typical Mine Size (Short Tons Metal Per Year)
Copper										
Porphyries - S	0.27	1.0	1.45	0.23	1.0	8.37	5.0	2000	20	125,000
- UDG	0.23	0.26	1.82	0.23	1.0	8.37	5.0	2000	35	125,000
Stratabound - S	0.75	0.95	0.42	0.59	0.15	1.70	0.78	1000	8	40,000
- UDG	0.40	0.40	0.40	0.59	0.15	1.70	0.78	1000	12	40,000
Cu-Ni Sulfides - UDG	a	a	a	0.08	a	4.80	a	2000	83	13,500
Massive Sulfides - UDG	0.4		1.4	0.50	0.27	5.70	1.3	2000	44	8,400
Nickel										
Cu-Ni Sulfides - UDG	0.34		4.12	0.26	0.51	12.3	2.36	3200	83	15,600
Ni-Co Laterites - S	1.62	0	2.05	0.75	0	61.7	1.69	0	67	30,000
Cobalt										
Cu-Ni Sulfides - UDG	a	a	a	0.03	a	10.6	a	1000	83	90
Stratabound - S	b	b	b	2.5	b	1.9	b	2000	8	500
- UDG	b	b	b	2.5	b	1.9	b	2000	12	500
Ni-Co Laterites - S	a	a	a	0.01	a	18.0	a	0	67	600
Manganese[c]										
- S	0.039	0	0.019	0.007		0.007	0.058	0	1.7	400,000
- UDG	0	0	0.034	0.007		0.007	0.058	0	1.7	400,000

[a]Included with impact parameter for nickel.
[b]Included with impact parameter for copper.
[c]The analysis includes only physical processing of manganese ores; specifically, smelting to high-carbon ferromanganese is excluded.
S = surface mines; UDG = underground mines

Table 2-7 also gives the typical mine size for which impact parameters were developed. Process plants could be regional or associated with a particular mine. Larger mines may be assumed to be somewhat more efficient (per unit production) in land use, employment, and possibly other resource uses. Smaller mines should be less efficient. The location and ownership of the facilities may be more important in this regard than size, however.

The following subsections briefly present some of the more important assumptions used in quantifying the impact parameters in Table 2-7.

2.3.2.1 <u>Copper</u>

<u>Porphyries</u>

Approximately 50 percent of present worldwide copper production, and more than 60 percent of future production, will come from porphyry deposits. These deposits are especially common in western North and South America. Because of their abundance and importance to U.S. supply, the derivation of aggregate impact totals is presented in some detail.

Data for copper porphyries are the most reliable, as this type of deposit is very common in the United States. The typical open pit mine for a porphyry deposit has an annual production rate of 125,000 tons of copper metal. The ratio of overburden to copper ore is about 2:1, with an ore grade of 0.60 percent copper. Based on a 20-year mine life, approximately 675 acres of land is consumed by the pit and a further 2500 acres is used for dumping the waste overburden. Surface structures require a negligibly small area (9 acres). Energy requirements for mining are about 7255 Btu/lb copper metal.

Processing of the copper ore requires almost as much land as the mine: 570 acres for plant facilities and 2500 acres for tailings ponds (for 125,000 tons per year capacity). Energy usage is estimated at nearly 42,000 Btu/lb, and water usage at 260 gallons (net after recycling) per ton of ore. Copper recovery is about 80 percent in the concentrate. SO_2 emissions from the smelting process are potentially 2 pounds per pound of copper produced, though they may be reduced by more than 90 percent with controls. Employment for the mining and processing facilities is estimated at 2500 workers.

Similar estimates can be made for a typical underground mining operation. Using a 20-year operating life and a capacity of 122,000 tons metal/year, the land use is 550 acres due to mining (from subsidence caused by block caving methods), 640 acres for waste dumps, and 12 acres for offices and other facilities. Energy use is estimated at 9080 Btu/lb of copper. Processing would be similar to that for open pit mined ore. Employment would be about 4200 workers, as underground mining is more labor intensive than open pit mining.

Using the above estimates as a basis, resource usage can be prorated per 1000 short tons of metal production. For example:

Land in waste dumps for open pit mining

$$= 2500 \text{ acres} \div 20 \text{ yrs.} \div 125{,}000 \text{ tons/year capacity.}$$

$$= 1.0 \text{ acres/1000 tons (see Table 2-7).}$$

Other parameters are calculated similarly.

Restating some of the assumptions:

- Average copper grade was assumed to be 0.60 percent with 80 percent recovery in concentrate (this grade will decrease in the future, requiring more land and energy).
- Stripping ratio: 2:1
- Tailings ponds: 50 feet deep (all ores).
- Energy use for surface mining - 7255 Btu/lb Cu;
  for underground mining - 9080 Btu/lb Cu;
  for processing (including beneficiation, smelting, and refinery) - 41,870 Btu/lb Cu.
- Water use - a minimum of 260 gal (1 ton) per ton of ore (all ores).

- Employment - combined total given for mining and processing (all ores).
- All land above underground workings assumed subject to substantial subsidence because of block caving methods.
- SO_2 emissions - based on data from Miami, Arizona, smelters which show potential SO_2 emissions (without control) of 1.94 tons per ton of copper smelted and a ratio of sulfur to copper content in typical porphyry minerals (chalcopyrite) of 1:1.

Stratabound

Estimates are based in part on a comparison of ore grade and stripping ratios for stratabound and porphyry deposits. Important assumptions are as follows:

- Subsidence is less likely with sublevel stoping and other methods than with block caving of porphyries.
- Ore grade - 4.77 percent Cu, with 70 percent recovery.
- Stripping ratio - 6:1
- SO_2 emissions - for a mixture of carbonate, oxide, and sulfide ores, a sulfur to copper ratio of 0.5:1 is estimated.

Massive Sulfides

Where other data were not available, estimates were based on comparisons of porphyry and massive sulfide ore grades.

- Ore grade assumed 2.5 percent, with 80 percent recovery.
- Subsidence potential similar to stratabound deposits.
- SO_2 emissions - based on typical minerals having a ratio of sulfur to copper content of about 1:1

Copper-Nickel Sulfides

Land uses associated with mining, waste dumps, and tailings ponds are not distinguishable from those attributable to production of nickel from these deposits. The same is true for water uses.

- Ore grade - 1.0 percent, with 90 percent recovery

- SO_2 emissions - based on typical minerals (e.g., chalcopyrite) having a sulfur-to-copper content ratio of 1:1.

2.3.2.2 Nickel

Copper-Nickel Sulfides

Mining is primarily underground and subsidence potential is not as great as for porphyry copper.

- Ore grade - 1.47 percent, with 75 percent recovery.

- SO_2 emissions - based on emissions and production data from Sudbury, Ontario, and on a typical nickel matte (pentlandite) consisting of 1.6 parts sulfur to 1 part nickel.

Nickel-Cobalt Laterites

Surface impacts are concentrated in mining activities. It is assumed that waste rock and tailings can be returned to the stripped lands.

- Ore grade - 1.8 percent, with 85 percent recovery.

- Stripping ratio - less than 1:1.

- Energy use in processing is very high, partly due to the need for drying the ore.

- SO_2 emissions - very low as the ore is an oxide, though sulfur is input to the smelting process.

2.3.2.3 Cobalt

Few of the impacts being quantified are directly attributable to mining or processing cobalt, since it is nearly always a by-product of nickel or copper mining. Some additional land and more employees are required for cobalt processing.

- Ore grade ranges from 0.1 percent in the copper-nickel sulfide and laterite deposits to 0.2 percent or more in the stratabound deposits. Recovery is usually less than 50 percent.

- Energy data are based on the ratio of cobalt to copper or nickel processing energy use (from Battelle, 1975a, 1975b, 1976), as appropriate for each deposit.
- SO_2 emissions - based on a typical ratio of sulfur to cobalt content in stratabound deposits (carrolite) of 1:1 and in copper-nickel deposits (cobalite) of 1:2.

2.3.2.4 Manganese

Surface mining of manganese deposits is carried out in a manner similar to laterites; waste rock is assumed backfilled. Underground mining has little or no subsidence potential. Processing steps considered include only crushing, screening, and washing; there are no tailings ponds, smelters, or refineries.

- Ore grade - 30 to 45 percent.
- Stripping ratio - normally less than 1:1.
  (Much lower grades occur in the United States, requiring much more land and energy to mine and process.)
- SO_2 emissions - none.

2.4 AGGREGATE IMPACTS OF COPPER MINING AND PROCESSING

Applying the impact parameters in Table 2-7 to the copper supply projections in Table 2-2 yields the estimate of cumulative resource impacts shown in Table 2-8 for each of the four major copper deposit types.

2.4.1 Land Use

According to Table 2-8, worldwide demand for copper is expected to require approximately 996,000 acres (1556 square miles) of land (direct impacts only) over the next 30 years. Slightly more than one-half is attributable to processing (beneficiation, smelting, and refining). Nearly 66 percent is due to development of porphyry deposits, with stratabound and massive sulfide deposits accounting for the remainder. Copper obtained from copper-nickel sulfides is essentially "free" as far as land resources are concerned, since the land is accounted for under nickel in Table 2-9.

Totals are also provided for the portion of copper demand attributable to the U.S. (18 percent). It is assumed for this analysis that U.S. supplies would be obtained in the same proportion as world supplies for each deposit type. The total land use committment is estimated at 179,000 acres or 280 square miles. As the United States is likely to supply much of its own copper, most of this land commitment would be within the U.S. (in which case a higher proportion would be taken from porphyry deposits).

2.4.2 Energy Use

Worldwide demand imposes an energy requirement of 39×10^{15} Btu (39 quads). Of this total, 76 percent is due to porphyry development. U.S. demand imposes an estimated requirement of 7 quads.

TABLE 2-8

CUMULATIVE IMPACTS OF CONTINUED LAND MINING FOR COPPER DURING THE PERIOD 1980-2010

	Land Use (Acres)			Energy Use (10^{15} Btu)	Water Use (10^{12} Gallons)	Potential SO_2 Emissions (1000 tons)	Employment (10^6 Man years)	Disabling Injuries (10^3)	No. Typical Mines	
	Mining	Processing	Total						S	UDG
Porphyry Supply										
World Demand	289,312	366,195	655,507	29.5	14.8	591,838	7.58	467	73	61
U.S. Demand [c]	51,954	65,760	117,714	5.3	2.7	106,280	1.36	84	13	11
Stratabound Supply										
World Demand	168,083	88,622	256,705	2.5	0.91	119,805	1.12	62	99	49
U.S. Demand [c]	30,183	15,914	46,097	0.45	0.61	21,514	0.20	11	18	9
Copper-Nickel Sulfides Supply										
World Demand	a	a	a	2.1 [b]	a	86,230	3.58	345	--	152
U.S. Demand [c]	a	a	a	0.38 [b]	a	15,485	0.64	62	--	27
Massive Sulfide Supply										
World Demand	28,641	55,135	83,776	5.1	0.93	143,206	3.15	304	--	430
U.S. Demand [c]	5,143	9,901	15,044	0.9	0.17	25,716	0.57	55	--	77
Totals										
World Demand	486,036	509,952	995,988	39.2	16.64	941,079	15.43	1178	172	864
U.S. Demand [c]	87,280	91,575	178,855	7.0	3.03	168,995	2.77	212	31	124

[a]See Nickel, Table 2-9.

[b]Mining energy use tabulated under nickel, Table 2-9.

[c]Assumes U.S. demand impacts are in proportion to world demand impacts for each source of supply.

S = surface mines; UDG = underground mines.

2.4.3 Water Use

Water requirements necessary to meet world demand for copper total 16.6×10^{12} gallons. Of that total, nearly 90 percent is attributable to porphyry deposits. U.S. demand for copper requires about 3×10^{12} gallons, much of that would be in the arid Southwest.

2.4.4 SO_2 Emissions

Potential SO_2 emissions resulting from worldwide copper demand are estimated at 941×10^6 tons. About 63 percent of that total is due to extraction from porphyry deposits. U.S. demand would result in potential (uncontrolled) SO_2 emissions of 169×10^6 tons. Copper smelted in this country, however, would be subject to Federal, state, and local emissions controls which should reduce this total by 80 to 90 percent or more.

2.4.5 Employment and Disabling Injuries

Cumulative employment required to satisfy worldwide demand for copper is estimated at 15.4 million man years of labor, approximately 50 percent from porphyry development. Employment required to meet U.S. demand is 2.8 million man years.

Disabling injuries attributable to worldwide copper demand are estimated at 1.18 million, with a majority coming from underground mining of stratabound and massive sulfide deposits. Total injuries resulting from U.S. copper demand are estimated at 212,000.

2.4.6 Number of Mines Required

A total of 172 surface and 864 underground mines are estimated to be required in production to meet world demand for copper by 2010. Only about 13 percent of these mines would be in porphyry deposits, even though porphyries would supply more than half the copper. An estimated 155 mines would be required to meet U.S. demand if supplies were taken in the same proportion for each deposit type. In reality, far fewer mines would likely be developed in the U.S. as most would be large mines in porphyry deposits.

2.5 AGGREGATE IMPACTS OF NICKEL MINING AND PROCESSING

Table 2-9 is a summary of impacts predicted to occur as a result of mining and processing of nickel during the period 1980-2010.

2.5.1 Land Use

Total land use to meet worldwide nickel demand is estimated at 62,600 acres, approximately 60 percent from mining activities. Laterite mining imposes the greatest demand on land resources.

Land required throughout the world to meet U.S. demand totals 21,700 acres. As nickel deposits in the U.S. are generally lower grade than in areas currently being mined, correspondingly more land would be required if selfsufficiency were assumed. Land requirements are much lower than required to meet copper demand, however.

2.5.2 Energy Use

An estimated 15.6 quads of energy would be required to meet world demand betwen 1980 and 2010, 90 percent for mining and processing of laterites. U.S. nickel demand imposes an energy requirement for 5.4 quads. The energy intensiveness of nickel production can be seen by comparing the following levels of energy use per unit metal production (mining and processing):

Copper: 0.074 quads/10^6 tons

Nickel: 0.53 quads/10^6 tons
(0.64 quads/10^6 tons for nickel from laterites and 0.16 quads/10^6 tons for copper nickel sulfides)

Managanese: 0.00035 quads/10^6 tons

2.5.3 Water Use

Potential water usage to meet worldwide nickel demand totals 1.7×10^{12} gallons, 86 percent for laterites. This total is 10 percent of the requirements to meet copper demand.

TABLE 2-9

CUMULATIVE IMPACTS OF CONTINUED LAND MINING FOR NICKEL DURING THE PERIOD 1980-2010

	Land Use (Acres)			Energy Use	Water Use	Potential SO_2 Emissions	Employment	Disabling Injuries	No. Typical Mines	
	Mining	Processing	Total	(10^{15} Btu)	(10^{12} Gallons)	(1000 tons)	(10^6 Man years)	(10^3)	S	UDG
Copper-Nickel Sulfides										
World Demand	3,156	7,146	10,302	1.5	0.23	29,699	0.77	74	--	26
U.S. Demand	1,090	2,467	3,554	0.5	0.08	10,253	0.27	26	--	9
Nickel-Cobalt Laterite Supply										
World Demand	35,761	16,556	52,317	14.1	1.43	0	1.48	37	37	--
U.S. Demand	12,416	5,748	18,164	4.9	0.50	0	0.51	12	13	--
Totals										
World Demand	38,917	23,702	62,619	15.6	1.66	29,699	2.25	111	37	26
U.S. Demand	13,506	8,215	21,721	5.4	0.58	10,253	0.78	39	13	9

S = surface mines.

UDG = underground mines.

2.5.4 SO_2 Emissions

Worldwide SO_2 emissions would total about 30×10^6 tons (uncontrolled), all from sulfide deposit processing. U.S. nickel demand would produce 10.3 million tons of SO_2 if no controls were imposed. As with copper, in excess of 90 percent SO_2 reduction would be expected from U.S. smelters. The total quantity of SO_2 emissions is estimated at less than 5 percent of that projected for copper production.

2.5.5 Employment and Disabling Injuries

Approximately 2.2 million man years of labor (65 percent for laterite deposits) would be required to meet the world demand for nickel; this is 14 percent of the labor required to supply copper. Disabling injuries are estimated to total 111,000.

2.5.6 Number of Mines Required

Worldwide, an estimated 37 surface mines and 26 underground mines are expected to be required in production by 2010. Meeting U.S. demand will require 22 mines.

2.6 AGGREGATE IMPACTS OF COBALT MINING AND PROCESSING

Compared to the use of resources required to meet world and U.S. demand for copper and nickel, effects of mining and processing cobalt will be very small. This is partially because much smaller volumes are required and partially because cobalt is mainly a by-product of copper or nickel and therefore comes relatively "free" in resource usage.

Total land use is projected to be 2000 acres, with 800 acres required to meet U.S. demand (Table 2-10). Energy and water usage would be very small. Potential SO_2 emissions are estimated at 1955 tons, 7 percent of nickel supply emissions and 0.2 percent of copper supply emissions (uncontrolled). Worldwide employment is estimated at 56,000, with 3800 disabling injuries between 1980 and 2010.

Because cobalt production is so small at most mines, more than 300 mines would be required in production by 2010. Since cobalt is produced mostly as a by-product or co-product of copper and nickel, it is interesting to compare the number of mines projected to be needed to supply world demand for these metals (Tables 2-8 and 2-9) with those for cobalt (Table 2-10). Comparing stratabound deposits first, it is projected that approximately 150 mines would be in copper production by 2010. This is more than the 82 expected to be needed to supply cobalt. For laterites, Table 2-9 projects that 37 mines should be in production to supply nickel; this exceeds the 25 needed for cobalt supply (Table 2-10). Only an estimated 26 copper-nickel sulfide mines are expected to be in production of nickel by 2010, however, compared to an apparent need for over 200 to supply cobalt (Table 2-10).

The implications of this comparison are that more cobalt will have to be supplied from stratabound and laterite deposits than projected in Table 2-4. If these mines can be developed with plans for cobalt extraction, sufficient capacity should be available worldwide to meet cobalt demand without forcing excess production of copper or nickel.

TABLE 2-10

CUMULATIVE IMPACTS OF CONTINUED LAND MINING FOR COBALT DURING THE PERIOD 1980-2010

	Land Use (Acres)			Energy Use	Water Use	Potential SO_2 Emissions	Employment	Disabling Injuries	No. Typical Mines	
	Mining	Processing	Total	(10^{15} BTU)	(10^{12} Gallons)	(1000 tons)	(10^6 Man years)	(10^3)	S	UDG
Stratabound Supply										
World Demand	a	1,991 [b]	1,991	0.015 [c]	a	1,593	0.007	0.41	55	27
U.S. Demand [c]	a	789 [b]	789	0.006 [c]	a	631	0.003	0.16	22	11
Cu-Ni Sulfide Supply										
World Demand	d	10.9 [e]	10.9	0.038	d	362	0.030	2.9	--	209
U.S. Demand	d	4.3	4.3	0.015	d	143	0.012	1.15	--	83
Laterite Supply										
World Demand	a	2.9 [b]	2.9	0.052 [c]	a	0	0.019	0.48	25	--
U.S. Demand	a	1.15 [b]	1.15	0.021 [c]	a	0	0.008	0.19	10	--
Totals										
World Demand	--	2,005	2,005	0.105 (c)	--	1,955	0.056	3.79	80	236
U.S. Demand	--	794	794	0.042 (c)	--	774	0.023	1.50	32	94

[a] Included under copper effects, Table 2-8.

[b] Includes only plant facilities. Mining, waste dumps, and tailings are included with copper, Table 2-8.

[c] Mining energy use included with copper, Table 2-8.

[d] Included under nickel effects, Table 2-9.

[e] Includes only plant facilities. Mining, waste dumps, and tailins are included with nickel, Table 2-9.

2.7 AGGREGATE IMPACTS OF MANGANESE MINING AND PROCESSING

Table 2-11 is a summary of impacts predicted to occur as a result of mining and processing manganese during the period 1980-2010. Although the quantity of manganese metal to be mined exceeds that of copper, the resource impacts are much smaller because of the high ore grade and the exclusion of metallurgical processing from this analysis. Land use impacts are projected to total 13,400 acres, less than 2 percent of that for copper. Land required to meet U.S. demand is only 1360 acres during the period 1980-2010. (However, U.S. self-reliance would require substantially more land because of low grade, see Chapter 4.0.). Energy and water usage are correspondingly small. There are no SO_2 emissions. (It should be noted,however, that blast furnace production of ferromanganese requires considerable amounts of energy and emits large quantities of particulates.) Approximately 70 mines would be required to meet worldwide demand by 2010; the output of 7 of those mines would meet U.S. demand. Employment worldwide would total 1 million man years, about 6.5 percent of the total for copper and 45 percent of the total for nickel.

TABLE 2-11

CUMULATIVE IMPACTS OF CONTINUED LAND MINING FOR MANGANESE DURING THE PERIOD 1980-2010

	Land Use (Acres)			Energy Use (10^{15} BTU)	Water Use (10^{12} Gallons)	Potential SO_2 Emissions (1000 tons)	Employment (10^6 Man years)	Disabling Injuries (10^3)	No. Typical Mines	
	Mining	Processing	Total						S	UDG
World Demand	9,274	4,161	13,435	0.21	0.35	0	1.01	68.6	29	43
U.S. Demand	936	420	1,356	0.02	0.04	0	0.01	6.9	3	4

S = surface mines.

UDG = underground mines.

2.8 SUMMARY OF AGGREGATE IMPACTS

The total projected resource uses required to supply worldwide and U.S. demands for copper, nickel, cobalt, and manganese during the period 1980-2010 are summarized in Table 2-12. It should be emphasized that this table does not specify where the mining and processing will take place. For example, under a likely supply/demand future, much of the U.S. copper demand would be met from domestic reserves. Very little of the U.S. nickel, cobalt, and manganese demand would be produced within the U.S., however, unless abnormal supply or trade restrictions were imposed by producing countries. In such a situation, the resource impacts within the U.S. to meet the same demand could be substantially larger than indicated on Table 2-12 because of the lower grade ores and lack of industry infrastructure. Such a supply restriction might also reduce demand and promote recycling or substitution, however. See Chapter 4.0 for a discussion of this situation.

The estimates in Table 2-12 are for direct impacts only. Land use would be substantially greater if indirect impacts were included at the mine sites and if secondary development and transportation impacts were included. Energy usage would also be greater if transportation were included. Water use estimates do not include demand due to secondary development. SO_2 emission estimates are without controls and thus could be substantially lower where air quality regulations are imposed (as in the U.S.); however, no account has been made for SO_2 emissions from power plants likely to be required in the vicinity of mines and process plants. Employment estimates do not include those in supporting industries or for transportation, nor do the estimates of injuries.

Each category of resource use is described separately in the following section and an attempt is made to place some perspective on the projections.

2.8.1 Land Use

During the 30 year period, more than 1 million acres (nearly 1680 square miles) of land will be utilized to meet worldwide demand for the

TABLE 2-12

SUMMARY OF IMPACTS ASSOCIATED WITH CONTINUED RELIANCE ON LAND MINING DURING THE PERIOD 1980-2010

	Land (Acres)	Energy (10^{15} Btu)	Water (10^{12} Gals.)	Potential SO_2 Emissions (1000 tons)	Employment (10^6 Man years)	Disabling Injuries (10^3)
World Demand						
Copper Supply	995,988	39.2	16.6	941,079	15.4	1,178
Nickel Supply	62,619	15.6	1.7	29,699	2.3	111
Cobalt Supply	2,005	0.11	0	1,955	0.06	3.8
Manganese Supply	13,435	0.21	0.4	0	1.01	68.6
Total World	1.074,047	55.1	18.7	972,733	18.8	1,361
U.S. Demand						
Copper	178,855	7.0	3.0	168,995	2.8	212
Nickel Supply	21,721	5.4	0.58	10,253	0.78	39
Cobalt Supply	794	0.04	0	774	0.02	1.5
Manganese Supply	1,356	0.02	0.04	0	0.10	6.9
Total U.S.	202,726	12.5	3.6	180,022	3.7	259

four metals; approximately 203,000 acres (317 square miles) of this amount would be needed to meet U.S. demand (Table 2-12; Figure 2-5). Approximately 93 percent of this land use will be due to copper production. Obviously, some of the land could be restored or reclaimed to other uses. For the most part, however, there would be a substantial change in its character and potential for other uses.

For comparison with the above figures, the state of Rhode Island has a total (land plus water) area of 1214 square miles; Arizona has an area of 113,909 square miles. Meeting U.S. demand for the four metals would thus require at least temporary commitment of land area equal to 26 percent of the state of Rhode Island, or 0.3 percent of Arizona. Most of this land use is due to copper production and would likely occur primarily within the U.S. since the nation will likely supply a large percentage of its copper demand from domestic resources.

As a further comparison, the total land area utilized by the mining industry (metals sector) during the period 1930-1971 (excluding smelters and refineries) was 524,000 acres (National Research Council, 1979). Of this amount, 166,000 acres were utilized for copper mining (U.S. Bureau Mines, 1974). Thus, there may be more than a doubling of the amount of land devoted to copper mining in the U.S. over the next 30 years. The percentage increase in world land use impacts would probably be well over 100 percent.

2.8.2 Energy Use

Energy use projections in Table 2-12 are given in quadrillion (10^{15}) Btus or quads. To provide some perspective on the magnitude of a quad, consider the following:

1 quad = the annual output of thirty-three 1000 MW power plants
= 180 million barrels of crude oil equivalent
= 45 million tons of coal equivalent

The total U.S. energy use in 1978 was about 78 quads.

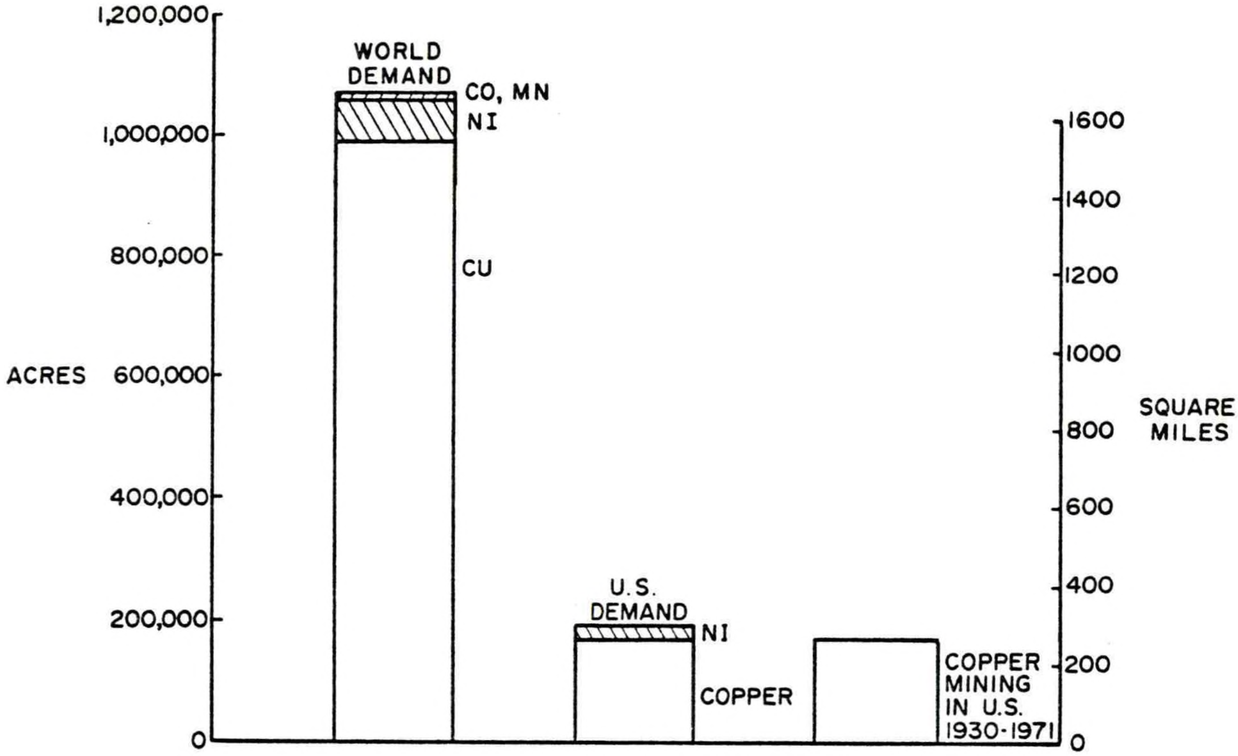


Figure 2-5. Cumulative Land Use for Mining and Processing, 1980 - 2010.

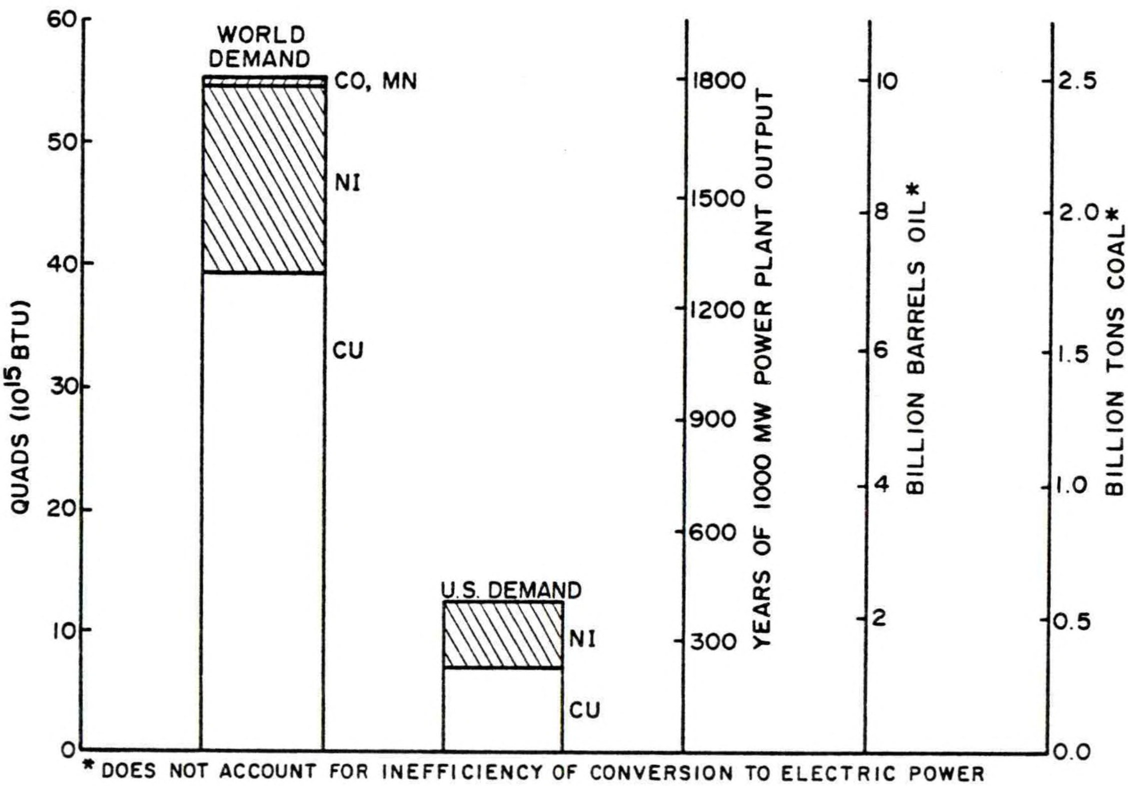


Figure 2-6. Cumulative Energy Use for Mining and Processing, 1980 - 2010.

Projected energy requirements to meet world demand for the four metals during 1980-2010 is 55 quads (Figure 2-6). This is the output of fifty-nine (59) 1000 MW power plants for 31 years; 875,000 barrels of oil per day; or 220,000 tons of coal per day. If oil and coal are converted to electricity (the most common form of energy used in the industry), the fuel requirement would be on the order of 2.5 million barrels of oil, or 640,000 tons of coal, per day. Approximately 71 percent of the energy use would be for copper production and 28 percent for nickel. Cobalt and manganese production would consume very little energy.

The energy use imposed by U.S. demand for these metals during 1980-2010 is 12.5 quads, or 0.4 quad per year. As a comparison, the Arizona electric utility industry produced about 0.55 quad of power in 1972.

2.8.3 Water Use

Projected water use to meet worldwide demand is estimated at 18.7×10^{12} gallons, or 57.4 million acre-feet (Table 2-12; Figure 2-7). U.S. demand imposes a requirement for 3.6×10^{12} gallons, or 11 million acrefeet. Ninety percent of this total is due to copper production and most of the remainder is for nickel. The total water withdrawal from surface and ground water supplies in Arizona during 1975 was 2.8×10^{12} gallons (Murray and Reeves, 1977). If the average surface runoff in all of Arizona is 1 inch per year, the amount of surface water available from in-state rainfall would be about 6 million acre-feet per year.

2.8.4 SO_2 Emissions

Total uncontrolled SO_2 emissions from production of copper, nickel, cobalt, and manganese to meet world demand during 1980-2010 is estimated at 973 million tons (Table 2-12; Figure 2-8). About 97 percent of this total is due to copper production, so that the 180 million tons of potential SO_2 emissions resulting from U.S. demand would occur mostly in the U.S. assuming essential self-reliance in

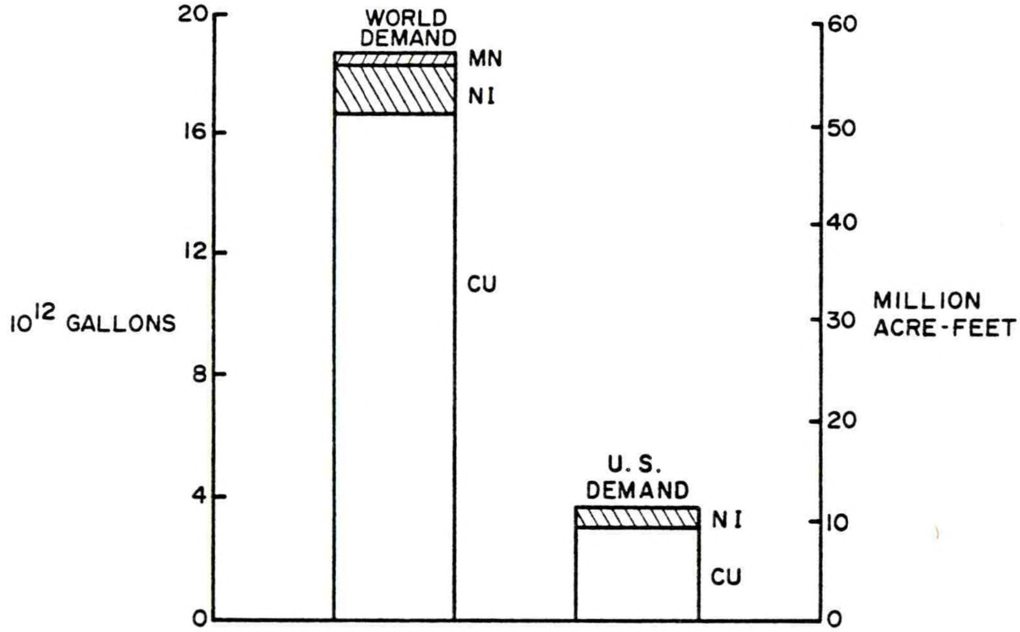


Figure 2-7. Cumulative Water Use for Processing, 1980 - 2010.

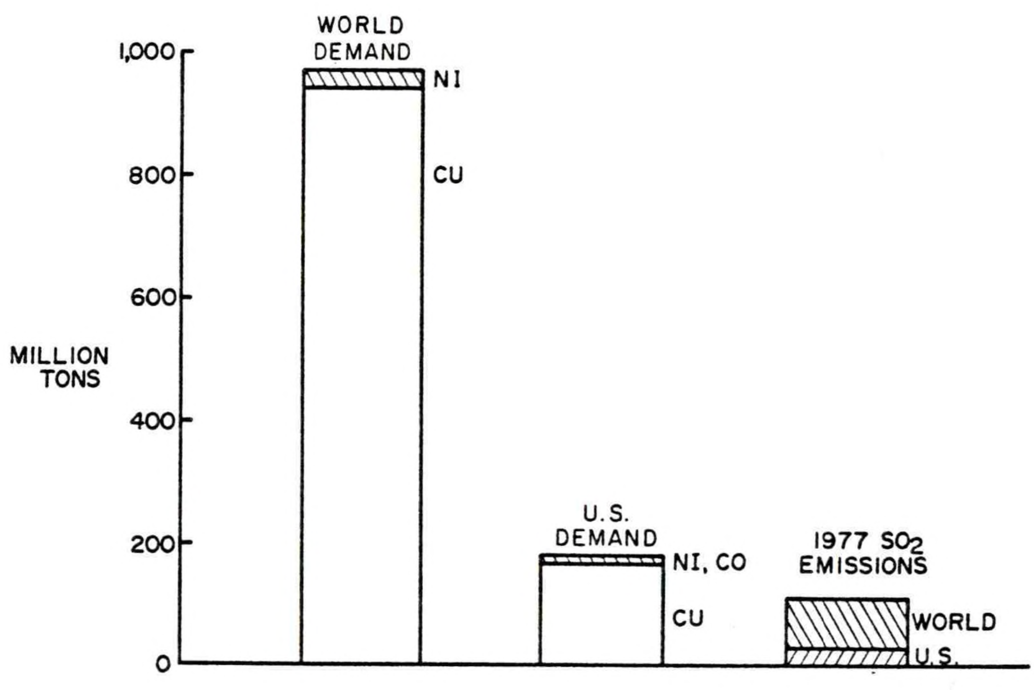


Figure 2-8. Cumulative SO_2 Emissions (Uncontrolled) from Processing, 1980 - 2010.

copper. Since SO_2 emission sources in the U.S. are likely to be subject to at least 90 percent control, this would reduce U.S. emissions to about 18 million tons. If little control is imposed on emissions outside the U.S., however, the total world emissions of SO_2 are likely to be on the order of 700 to 800 million tons, or an average of 23 to 26 million tons per year.

During 1977, the estimated emissions of SO_2 in the U.S. were 30 million short tons (EPA, 1978), compared to a worldwide total of about 110 million tons (Likens and others, 1979). Thus, the average annual SO_2 emission within the U.S. due to production of copper during 1980-2010 will be only about 2 percent (assuming 90 percent control) of the 1977 U.S. total (though it will be relatively concentrated in location). In contrast, however, without substantial control in other parts of the world, the average annual SO_2 emission rate due to meeting worldwide demand could be around 20 percent of the present world rate.

2.8.5 Employment and Injury Estimates

The estimated cumulative employment in worldwide production of copper, nickel, cobalt, and manganese is 18.8 million man years, 82 percent from copper production (Table 2-12; Figure 2-9). The employment due to U.S. demand is estimated at about 3.7 million man-years. For comparison, the total U.S. employment in primary metals industries in 1979 was 1.1 million (Williams, 1978). The total U.S. employment in mining (all types) during 1974 was 650,000, of which 25,000 was in Arizona. If the projected employment due to U.S. metals demand occurs primarily in the U.S. (as it should because of the dominance of demand imposed by copper), there would be an annual average of about 120,000 workers in the industry.

The estimate of disabling injuries due to worldwide demand for the four metals is 1.36 million, or an average of about 44,000 per year (Table 2-12; Figure 2-9). Injuries due to U.S. demand are about 19 percent of the world total. As high grade surface deposits of copper

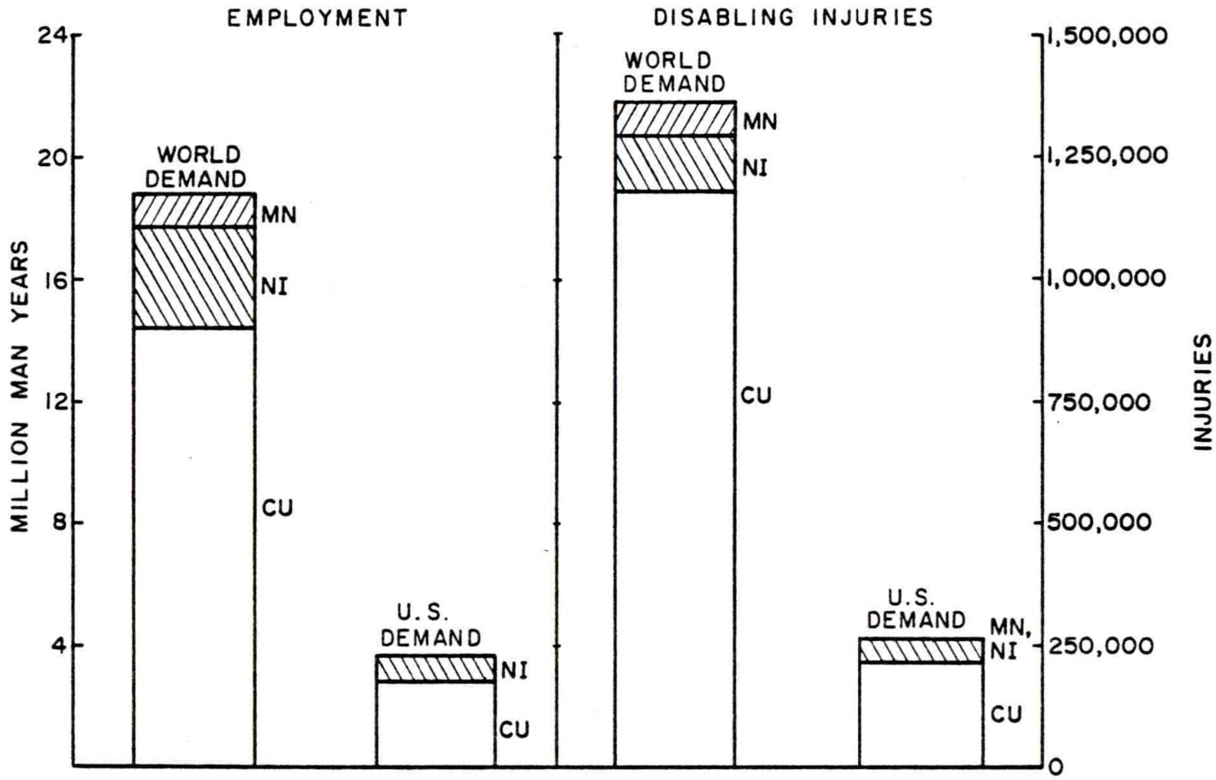


Figure 2-9. Cumulative Employment and Disabling Injuries for Mining and Processing, 1980 - 2010.

are depleted, more and more mining will be underground and consequently more injuries will occur.

2.8.6 Sensitivity to Predicted Demand

The aggregate impact estimates developed to this point are based on the projection of most likely world and U.S. demand for copper, nickel, cobalt, and manganese during 1980 to 2010. High and low demand projections were also made in Part I of this report. Cumulative demands for the period of interest are provided in Table 2-13 for the three levels of projected demand.

TABLE 2-13

PROJECTED RANGE OF WORLD AND U.S. DEMAND FOR COPPER, NICKEL, COBALT, AND MANGANESE

	Cumulative Demand Estimates, 1980-2010 (1000 Short Tons Metal)					
	Low		Most Likely		High	
	World	U.S.	World	U.S.	World	U.S.
Copper	403,051	75,392	530,442	95,255	648,783	122,256
Nickel	27,590	8,384	31,365	10,868	44,613	12,035
Cobalt	1088	421	1448	574	1692	661
Manganese	523,320	51,774	594,547	60,012	703,686	67,765

Applying the same unit impact parameters to these incremental differences in consumption predictions, a comparison of resource impacts can be made for each demand projection (Table 2-14). For most of the resources, the high and low impact estimates are within 25 percent of the most likely levels given in Table 2-12.

2.8.7 Sensitivity to Other Parameters

It should be noted that the estimates in Table 2-14 reflect the range of estimated demand levels more accurately than they reflect the

TABLE 2-14

RANGE OF ESTIMATED AGGREGATE IMPACTS OF MEETING WORLD AND U.S. DEMAND FOR COPPER, NICKEL, COBALT, AND MANGANESE

	Land (Acres)	Energy (10^{15} Btu)	Water (10^{12} Gallons)	Potential SO_2 Emissions (1000 tons)	Employment (10^6 Man Years)	Disabling Injuries (10^3)
World Demand						
High	1,325,504	70.4	23.2	1,194,993	23.4	1685
Most Likely	1,074,047	55.0	18.7	972,733	18.8	1361
Low	825,205	43.7	14.5	742,663	14.6	1056
U.S. Demand						
High	256,051	15.0	4.6	229,143	4.6	325
Most Likely	202,726	12.4	3.6	180,022	3.7	259
Low	159,997	9.7	2.9	142,233	2.9	205

range of possible resource impacts. No attempt has been made to quantify ranges for each unit impact parameter. This would be needed to develop an envelope of expected resource impacts.

A brief discussion of the sensitivity of the resource impact estimates to at least one of the parameters used in the analysis would be instructive, however. One very important parameter is the average grade of ore mined. At present, copper porphyries mined in the southwestern U.S. average about 0.6 percent copper. By 2000, the average grade may decrease to 0.3 percent (Kellogg, 1977). If this should occur, several of the impact parameters will increase dramatically (Table 2-7). The acreage required for waste dumps and tailings ponds (possibly also for the mine pit) will be doubled. Energy use (for mining at least) will be approximately doubled. Process plant area, water use, and employment will also increase. Similar increases will be observed for the other metals as ore grades decrease. As previously mentioned, nickel, cobalt, and manganese ore grades in the U.S. are generally lower (sometimes substantially) than those assumed in generating the parameters in Table 2-7. Resource impacts caused by U.S. demand could therefore be substantially higher for self-reliance in these metals than for use of foreign supplies.

3.0 EXAMPLES OF SITES EXPECTED TO SUPPLY U. S. DEMAND UNDER NORMAL MARKET CONDITIONS.

In this section of the report, specific sites have been selected to provide an overview of the types of sources (both U.S. and foreign) currently being used to provide copper, nickel, cobalt and manganese to the United States (Figure 3-1). These sources are also considered typical of those that would be used in the future if the U.S. is able to depend on its present mix of foreign and domestic supplies.

The individual mines/deposits and processing areas have been selected to encompass the range of ore type and mining methods providing U.S. supplies, and each is considered representative of a particular type. Comparisons of characteristics of each site are provided in Table 3-1.

For copper, cobalt, and manganese, one site is considered as a primary source and is described in more detail than other site(s) which provide information on additional sources. For nickel, two primary sites are discussed (see Table 3-1) because of the importance of the two major types of ores (see Part I). Where possible, to avoid repetition, comparisons and references are made to discussions in preceding parts of the report.

Detailed discussions of mining methods have been presented in Part II and of supply/demand relationships, in Part I. For this reason only brief descriptions of mining and processing methods and of overall commodity supply and demand are included here. Emphasis is placed on environmental effects (both natural and manmade) which would result if these mines, and by implication others like them, are used for U.S. supplies through the year 2010. The treatment of individual sites is not exhaustive. It is intended that these reviews will supplement the aggregate analysis of Chapter 2.0 in describing probable impacts of land-based mining/processing during the period 1980-2010.

The subsections which follow are devoted to examples of individual mines/processing areas for each mineral: Copper - Section 3.1;

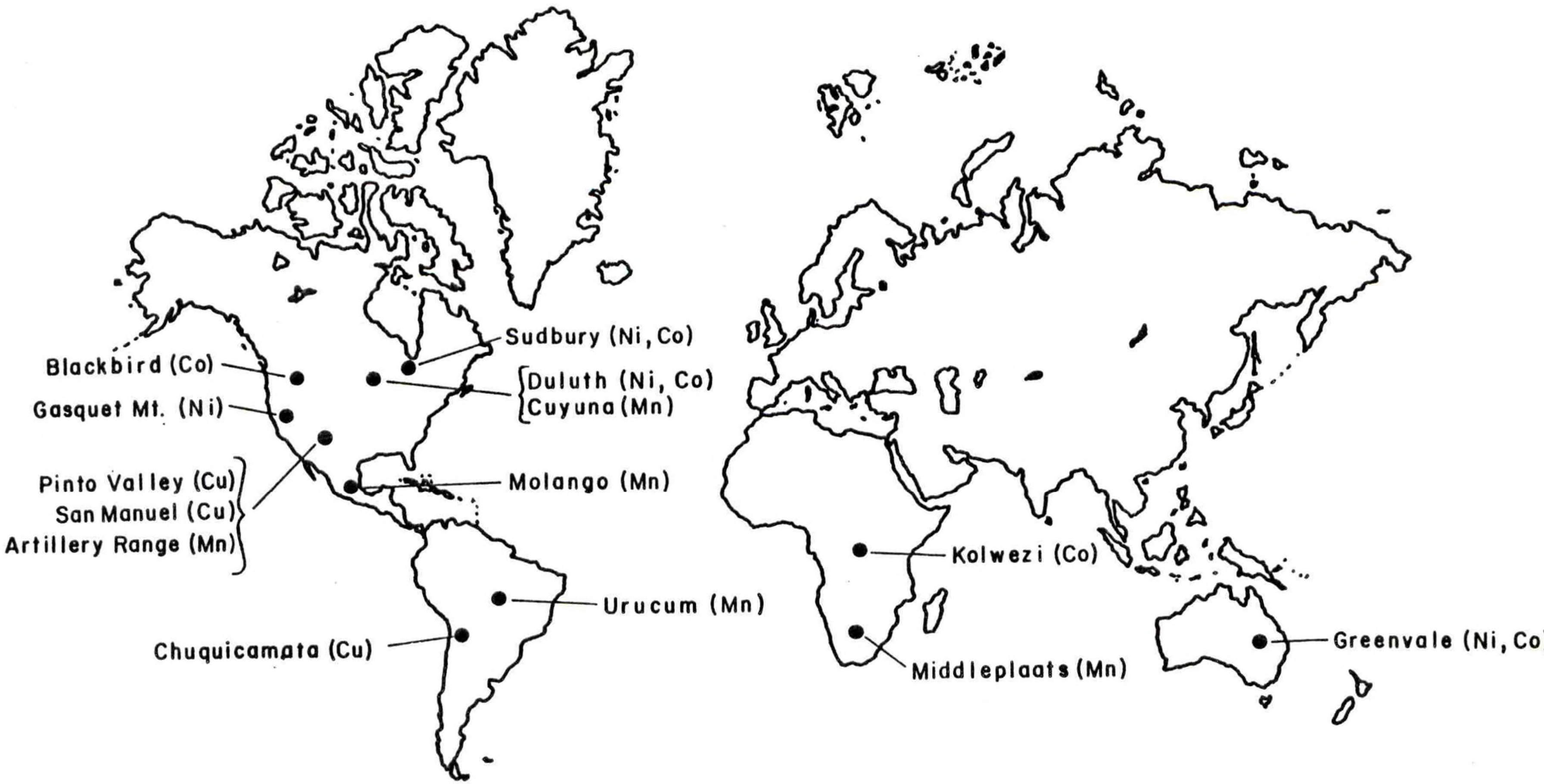


Figure 3-1. Site Specific Mine Locations.

TABLE 3-1

COMPARISONS OF SITE CHARACTERISTICS FROM
REPRESENTATIVE MINES/DEPOSITS AND PROCESSING AREAS

Characteristics / Site	Surface	Underground	Sulfide	Laterite	Oxide	Carbonate	Domestic	Foreign	Arid	Temperate	Tropic	Regulated	Less Regulated	Expansion	New Facility[a]	Rail Transport[b]	Vesel Transport	Report Section
Copper																		
Pinto Valley, AZ	X		X				X		X			X			X	X		3.1.1, 4.1.1
Chuquicamata, Chile	X		X					X	X				X	X		X	X	3.1.2
San Manuel, AZ		X	X				X		X			X		X		X		3.1.3, 4.1.2
Nickel																		
Sudbury, Ontario		X	X					X		X		X		X		X		3.2.1
Greenvale, Australia	X			X				X		X			X		X	X	X	3.2.2
Duluth Gabbro, Minn.	X	X	X				X			X		X			X	X		4.2.1
Gasquet Mountain, Calif.	X			X			X			X		X			X	X		4.2.2
Cobalt																		
Kolwezi, Zaire	X	X	X		X			X			X		X	X		X	X	3.3.1
Sudbury, Ontario		X	X					X		X		X		X		X		3.3.2
Greenvale, Australia	X			X				X		X			X		X	X	X	3.3.3
Duluth Gabbro, Minn.	X	X	X				X			X		X			X	X		4.3.1
Blackbird, Idaho		X	X				X			X		X			X	X		4.3.2
Manganese																		
Middelplaats, South Africa		X			X			X	X				X		X	X	X	3.4.1
Molango, Mexico	X	X			X	X		X			X		X		X	X	X	3.4.2
Urucum, Brazil		X			X			X			X		X		X		X	3.4.3
Cuyuna Range, Minn.	X				X		X			X		X			X	X		4.4.1
Artillery Range, AZ		X			X		X		X			X			X	X		4.4.2

[a] Or facilities on stream since 1970.

[b] For foreign facilities to point of loading for ocean transport.

Nickel - Section 3.2; Cobalt - Section 3.3; and Manganese - Section 3.4.

Each scenario presents a brief description of the mine and processing facilities and the existing baseline environmental conditions (including pertinent environmental regulations where this information is available). The more important impacts to the environment are also discussed.

3.1 COPPER

As noted in Parts I and II of this report, copper is found in several types of deposits of which porphyry copper has been the most important source. The U.S. is the largest producer of copper; within the U.S., Arizona provides 54 percent of production (1974). Chile is the second largest producer. The nature of porphyry copper deposits is such that about 80 percent of the mines are open pit, and 20 percent are underground. Accordingly, the examples selected include two open-pit mines - Pinto Valley in Arizona, and Chuquicamata, in Chile; and an underground mine, San Manuel, also in Arizona.

3.1.1 Pinto Valley, Arizona

Pinto Valley is an example of a typical domestic, open-pit porphyry copper mine. The mine is located 5.5 miles west of Miami, Arizona, and approximately 90 miles east of Phoenix in the Globe-Miami mining district (Figure 3-2).

3.1.1.1 General Setting

The Pinto Valley mine is located in an unsurveyed portion of T 1 N, R 14 E of the Inspiration, Arizona, 7 1/2-minute quadrangle (U.S. Geological Survey). Elevations within the quadrangle area vary from less than 2825 feet at Pinto Creek in the northwest to 5776 feet at Webster Mountain. Within a one-half mile radius of the Pinto Shaft, elevations range from 3450 feet in Pinto Creek to 4225 feet on an unnamed mountain to the south.

The area is characterized by incised mountains, alluvial-filled basins, and numerous intermittent streams which, in the vicinity of the mine, ultimately drain into Pinto Creek and eventually into Theodore Roosevelt Lake. The U.S. Geological Survey topographic map also indicates many springs throughout the area.

The rocks occurring within the area include Precambrian granite and schist, Precambrian to Cambrian limestone, sandstone, quartzite, shale, and conglomerate, Precambrian to Tertiary diabase, Carboniferous and Devonian limestone, shale, and sandstone, Laramide intrusives,

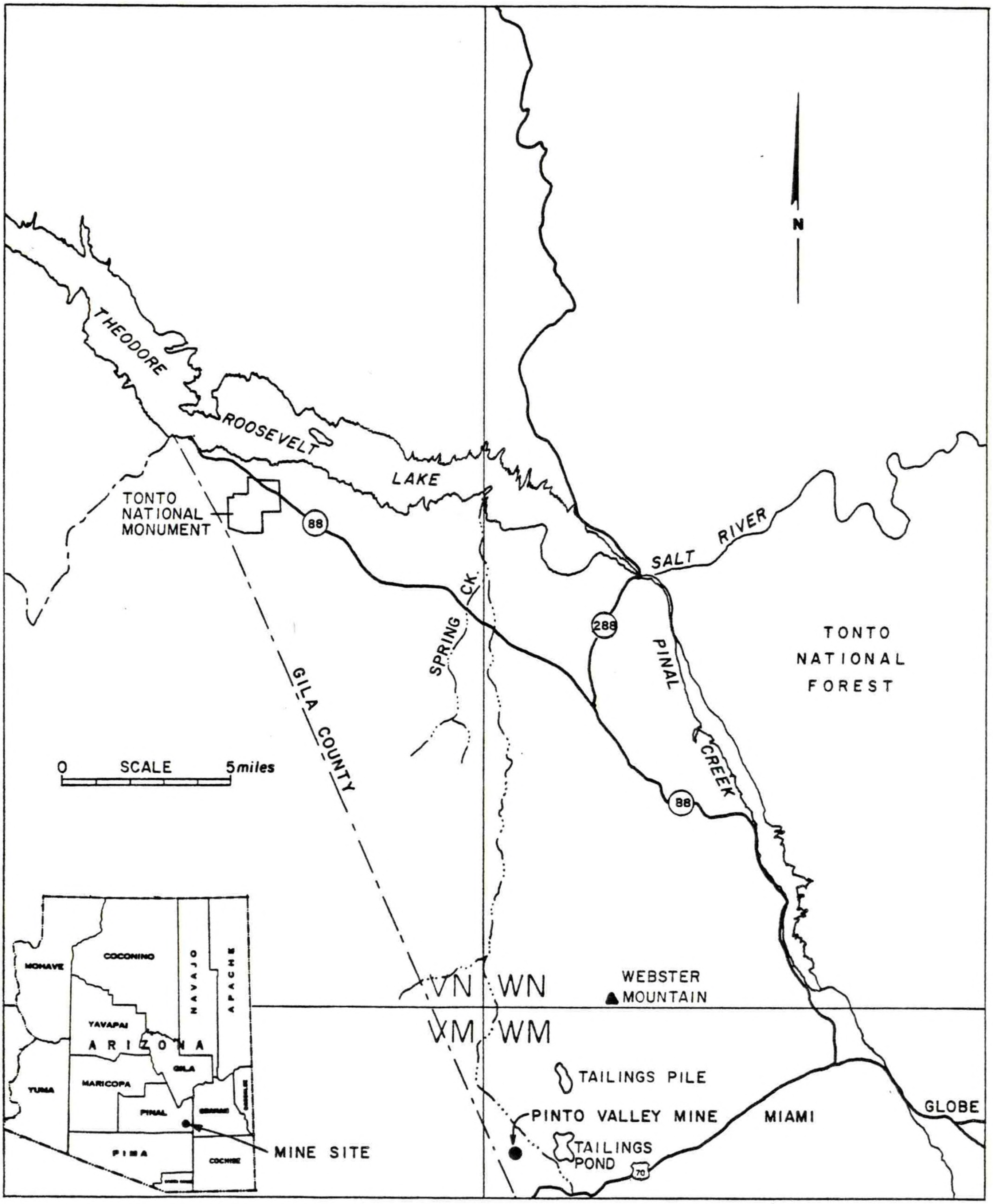


Figure 3-2. Location of Pinto Valley Mine.

Tertiary dacite, gravel, sand, and conglomerate, and Quaternary gravel, sand, and silt. There are several major faults trending NE-SW to the east of the mine area near the city of Miami.

Mining in the district began in 1874, but because of the remoteness of the region, initial interest centered on the many small gold and silver deposits. Development of the large, low grade, disseminated copper deposits (porphyries) began soon after the turn of the century, with production coming from several mines. Copper production from Pinto Valley commenced in mid-1975, and future increases are considered likely.

Throughout the area there are many mines, shafts, and adits. The Castle Dome mine and concentrator is located approximately 2 miles to the northeast of the Pinto mine and the Inspiration mine and mineworks are approximately 6 miles to the east-northeast. This area is rich in copper, gold, lead, molybdenum, silver, and zinc.

The approximate elevation of the Pinto Valley mine is 3750 feet. The predominant vegetative community is Interior Chaparral, with patches of Oak-Pine Series. The understories comprise principally annual grasses and forbs. Stands of Mixed Broadleaf Series, which include sycamore, cottonwood, ash, and willow, occur as riparian vegetation along Pinto Creek. Table 3-2 presents an annotation of these habitats given as communities, associations, and series (Brown and others, 1979).

Important biological features of this area include: the diversity of habitats and wildlife as discussed above; the intermittent riparian habitat of Pinto Creek; and the occurrence of game species such as mule deer, quail, and javelina. One of nine known nesting sites of the American bald eagle in Arizona is near the confluence of Pinal Creek and the Salt River, approixmately 15 miles from the mine. This area is also proximate to a known habitat of the endangered Arizona hedgehog cactus (Echinocereus triglochidiatus var. arizonicus).

TABLE 3-2

DESCRIPTION OF BIOTIC COMMUNITIES OCCURRING IN THE VICINITY OF THE PINTO VALLEY MINE

Interior Chaparral

Terrain: Foothills and mountain canyons, 3500-7000 feet elevation.

Climate: Semi-arid, 12-20 inches of precipitation annually, bimodally distributed, only a small part as snow. Warm summers, with highs often above 100. Mild winter days with freezing temperatures at night fairly common.

Soils: Udorthentic haplustralls, lithic torriorthids and haplargids, lithic camborthids and haplustalls.

Vegetation: Dense shrubby growth, of fairly uniform height from 3 to 7 feet. Dominants are sclerophyllous evergreen shrubs like the scrub oak, manzanita, sugar sumar, squawbush, and moutain-mahogany.

Fauna: Common bushtit, rufous-sided towhee, blue-gray gnatcatcher, brush mouse, and mule deer.

Oak-Pine Series

Terrain: Hills and mountain slopes, 400- 6500 feet elevation.

Climate: Bimodally distributed precipitation, with more than 50 percent of this in the summer. Mild summers with cool evenings and cool winters with subfreezing temperatures common.

Soils: Udorthentic and lithic haplustralls and lithic torriorthents, ustollic haplargids and aridic calcuistalls.

Vegetation: Open woodland formation dominated by silverleaf oak, Arizona oak, Emory oak, and Arizona cypress. Grasses and often prickly pear and other cacti form the understory.

Fauna: Bridled titmouse, painted redstart and black-tailed rattlesnake.

Mixed Broadleaf Series

Terrain: Alluvial floodplain adjacent to perennial streams.

Climate: Variable, compensate precipitation by riparian existence in hot-arid desert; ranges subtropical to temperature.

Soils: Typic torrifluvents.

Vegetation: Riparian deciduous forest; a mixture of sycamore, alder, willow, ash, and other deciduous trees.

Fauna: Representative species include, red-spotted toad, vermillion flycatcher, yellow warbler, summer tanager, hispid cotton rat, and raccoon.

There are numerous archaeological sites in the area, some of which indicate that prehistoric Indians may have worked two turquoise mines in Pinto Valley between 1200 and 1400 AD (Li and Carter, 1975). Other archaeological excavations have revealed 6 dwelling sites; 24 additional sites are being investigated.

Points of interest nearby include the Salt River Canyon, Tonto National Monument, Coolidge Dam, and Theodore Roosevelt Dam and Reservoir.

Three counties (Gila, Pinal, and Maricopa) with significant population centers are within a 30-mile radius of Pinto Valley, and so might be expected to experience some of the socioeconomic effects related to the mining operations. Estimated county populations (for July 1, 1980) are as follows:

- Gila 36,600
- Pinal 92,500
- Maricopa 1,490,100

Pinto Valley mine is located in Gila County which has a population density of only 8 people per square mile. Primary economic activities are mining and smelting, livestock ranching, timbering, tourism, and recreation. Unemployment (as of December 1979) in the county was around 8 percent of the work force. Globe, the county seat, had a population of 7935 in 1978; and Miami, 3915.

Pinal County has approximately twice the population density of Gila. Mining is prominent in the southern and eastern parts of the county but the northwestern corner of the county is much more involved in the manufacturing and retail trade sectors. Unemployment was 6.7 percent at the end of 1979.

Maricopa County, by far the most populous county in the state, has a population density of nearly 163 people per square mile. Economic activity is centered around Phoenix, the state's capital, and is heavily linked to the manufacturing (particularly electronics), retail trade, and service sectors. Unemployment was only 5.2 percent at the end of 1979.

Within Gila County 56 percent of the land is owned by the U.S. Forest Service; 2 percent by the Bureau of Land Management, 1 percent by the State of Arizona; 3 percent is in individual or corporate ownership; and 38 percent is Indian land.

3.1.1.2 Mine and Processing Facilities

The Cities Service Company's Pinto Valley mine was opened in July 1974 with an expected capacity of 62,500 tons per year of metallic copper. The ore body measures about 4800 by 2500 feet, with the longer axis striking about N 70°E; dips are nearly vertical.

Before mining could begin, 120 feet of overburden totaling 2.25 million tons were removed by a contractor as part of a pre-development program. From August 1972 until the mine opened for production in July 1974, Cities Service Company continued to remove overburden and develop the mine, removing an average of 128,000 tons per day (tpd) in 1973. The mine is expected to reach an ultimate size of 6000 by 3500 feet, and will extend to a depth of 1450 feet (Li and Carter, 1975).

Reserves are estimated at 350 million tons of ore averaging 0.44 percent copper; the major ore mineral is chalcopyrite. If the production target of 40,000 tpd of ore and 60,000 tpd of waste is maintained, the life of the mine would be 24 years. This production rate and a stripping ratio of 1.5:1 would require the ultimate removal of more than 500 million tons of waste and leach-grade material (Li and Carter, 1975).

A 40,000 tpd concentrator has been constructed adjacent to the mine. In this facility, run-of-mine ore goes through "three stages of crushing, grinding, and a standard flotation scheme, producing an average 720 tpd of copper concentrate and 4.8 tpd of molybdenum concentrate" (Li and Carter, 1975). Facilities were designed to permit a 30 percent increase in plant capacity.

The processed ore is stored in two 238,000 gallon tanks as a concentrate of 55 to 65 percent solids and is sent through a 10.7-mile long pipeline to a filter plant at the Inspiration Company's smelter in Miami. Makeup water for the processing plant comes from wells in the Pinto Creek drainage basin and from the Old Dominion mine. Water is reclaimed from the tailings thickeners (part of the flotation process) and from the tailings disposal area.

3.1.1.3 <u>Environmental Impacts</u>

As previously indicated, the Pinto Valley mine is a relatively large open pit surface mine with an ultimate pit size of approximately 6000 x 3500 ft. An average of approximately 100,000 tons/day of material is handled at this mine, of which 60 percent is rejected as waste rock. The remaining 40 percent undergoes further milling and processing at the mine, and the end product (copper concentrate) is transported via pipeline a distance of 10.7 miles in slurry form (55-60 percent solids) to the Inspiration Consolidated Copper Company's smelter complex near Miami, Arizona. Prior to the removal of material from the ore body, the overburden material is removed and relocated.

One of the obvious environmental impacts of the mine and processing plant is the requirement for large acreages of land for waste disposal. Although Pinto Valley is a new mine, evidence from previous mining activities indicates what is to come. Several square miles of land around Globe and Miami are covered with tailings and overburden. Just east of Pinto Valley shaft is the Cottonwood tailings site. The rugged, rocky terrain is ideal for tailing disposal as canyons can be filled with a minimum of dike construction and maintenance. If the unit impact parameters of Chapter 2.0 are applied to Pinto Valley, land resources required in a 24-year lifetime would total about 3000 acres, including 1200 acres each for waste rock and tailings. This very large committment of land will displace virtually all the vegetation and animal life. Reclamation of such land to provide habitat suitable for native species is not expected to be completely successful (Thames, 1977).

Energy and water use prediction can also be made on the basis of data in Chapter 2.0. Total energy usage for mining and processing should be on the order of 0.12 quads (0.12×10^{15} Btus). Water usage will be about 5×10^{10} gallons.

Concern is increasing in the Globe-Miami area over pollution of streams and water supplies as a result of releases of mine wastes (Thomas, 1979). Pinal Creek which is the next upstream tributary to the Salt River from Pinto Creek has recently been polluted by copper mine contaminants originating in the Globe-Miami area mines. Four reservoirs on the Salt River provide the water supply for Phoenix. To date no pollution has been reported to the drinking supply. Wastes are reportedly coming from mines near the Pinto Valley site, but not from the Pinto Valley Mine. Pollution of Pinal Creek (Figure 3-2) has reportedly killed all living things between the Miami-Globe area and Roosevelt Lake some 15 miles downstream on the Salt River. There is some concern over the eagle nesting site upstream on the Salt River from its confluence with Pinal Creek; the adults are apparently in good health, but no young have been observed. Any relation to the pollution problem has not been determined. With continuing mining and processing in the area over the next 30 years, such instances of pollution may become common.

The Pinto Valley mine complex employs a total of 500 men in its operating and maintenance work force. A reasonable employment multiplier for this occupation in Arizona is approximately 2.5, which means a total of 1250 jobs have been created in the area. In light of the area's long history of mining, the relatively great distance from Phoenix (or any other large population center), and locally higher unemployment rates, socioeconomic effects of possible future expansion of this or other mines will likely be localized to those few towns closest to the site.

Major production expansions at the Pinto Valley mine would require an influx of construction personnel, but these might generally be available locally, and in any case would probably be small in number as long as mining continues to be a surface operation. Another potential future impact-minimizing factor would be the pre-existence of a company supported tax-base, which conceivably could provide front-end financing of public planning for population increases.

As part of their contribution to the Miami-Globe area, Cities Service and other mining/processing companies have singly and jointly contributed to many public projects including a community hospital, a YMCA, an apprenticeship program, and a mine rescue station (since closed as the programs have been turned over to the various companies), which has now been turned over to the Gila County Archaeological Society for a museum.

Cities Service has funded a study to determine radiocarbon dates for three archeological sites in the area to be affected by the mine. Also the Arizona State Museum has excavated six prehistoric dwellings, and is continuing work on additional sites in a 16-mile area affected by the Pinto Valley mine (Li and Carter, 1975). Thus, development of the mine is contributing to knowledge of man's former occupation of the area.

The primary impact of the Pinto Valley Copper Mine on air quality is from fugitive dust arising from general mine activities. Major dust producing activities, a general discussion of which is contained in Section 1.2.3, are as follows:

- ° overburden removal
- ° ore body excavation - blasting, shoveling
- ° ore body transportation - haul roads
- ° storage piles - wind erosion
- ° ore processing - crushing, screening, etc.

The problem of fugitive dust at this mine is further complicated by the fact that the area is semiarid, as is typical of the southwestern

United States. In this climate, fugitive dust which originates from either human-related or natural processes is difficult to suppress when generated on a large scale such as at an open pit mining operation. The major impact of these emissions occurs on a local scale (say within 10 miles) as discussed in Section 1.2.3.

In addition to fugitive dust emissions, there is also a relatively small quantity of emissions associated with diesel engine exhaust at the mine. These emissions are also fugitive in nature in that they are not necessarily fixed in space. The emissions result primarily from numerous shovels, trucks, and generators interspersed throughout the mine site. No significant impact on air quality has been observed to occur as a result of the operation of these sources.

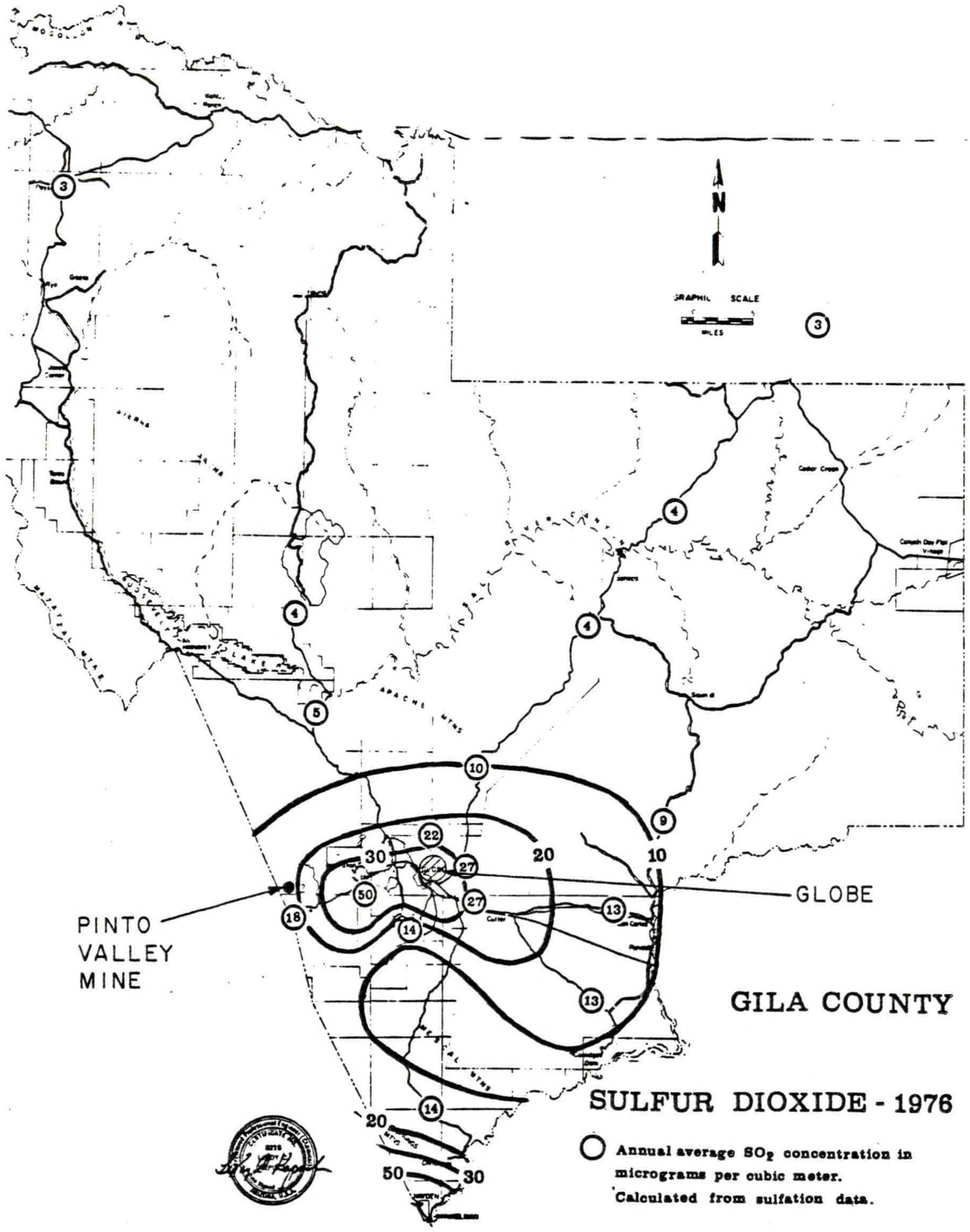
The most significant air quality impact usually associated with the production of copper results from the emissions of sulfur dioxide (SO_2) at copper smelters. Copper concentrate produced at the Pinto Valley mine is not refined at an onsite smelter, but rather at one operated by the Inspiration Consolidated Copper Company 10 miles to the east in Miami, Arizona. The Inspiration Consolidated smelter processes copper concentrate from several local mines, and it is estimated that approximately 30 to 50 percent of its total production is attributable to the Pinto Valley mine.

Conversations with an official of the Arizona Department of Health Services (Chelgrin, 1980) indicate that significant emissions of SO_2 result from the operation of seven copper smelters in Arizona with a total estimated SO_2 emission rate of 800 tpd. Of these seven smelters, four (including the Inspiration Consolidated Copper Company smelter) are located in the Pinal-Gila County air quality control district. According to the state, the average daily emissions of SO_2 from the Inspiration Consolidated smelter for the years 1976-1978 were as follows:

Year	Average Daily SO_2 Emissions (tons)
1976	122.4
1977	134.8
1978	138.0

As can be seen, the emissions from this facility are relatively constant. Though these totals can be expected to vary slightly as a result of equipment failure during the year, for the most part they are directly related to the quantity of copper produced at the smelter. The current level of SO_2 removal at the Inspiration Consolidated smelter is about 80 percent, but state officials indicate that a removal efficiency on the order of 95 percent should be possible within the next few years. The primary method of SO_2 removal is through the production of sulfuric acid at an onsite acid plant. The average daily SO_2 emissions from the Inspiration smelter represent about 16 percent of the statewide smelter emission rate of 800 tpd. If 30 to 50 percent of production at the Inspiration smelter is from the Pinto Valley mine, it can be assumed that SO_2 emissions directly associated with the operation of the Pinto Valley mine represent approximately 5 to 10 percent of the statewide emissions from smelter operations.

Since there are four major smelters in the same air quality control district (Gila and Pinal Counties), it is difficult to determine the absolute impact of this particular smelter on ground-level concentrations without a detailed analysis. There are, however, extensive annual average ambient SO_2 monitoring data available from which an estimate of the impact of the Inspiration Smelter can be made. Figure 3-3 shows the results of annual average SO_2 monitoring performed by the Pinal-Gila Air Quality Control District agency during 1976. These results, which are shown as isopleths of annual average SO_2 concentration, are based on the use of a large number of monitors in the Pinal-Gila area (approximately 70 ASTM recommended sulfational plates). The figure shows that in the Miami-Globe area where the Inspiration



Source: Pinal-Gila Counties Air Quality Control District, 1977.

Figure 3-3. Observed Annual Average SO_2 Concentrations in Gila County for the Year 1976.

smelter is located, there is a distinct node or area of locally high concentrations which appears to originate from the Inspiration smelter. Within approximately 20 miles of this smelter, concentrations are shown to be below 10 $\mu g/m^3$, except for an area to the south where the influence of another smelter is evident (a facility operated by ASARCO is located near Hayden in the most southerly portion of the county).

According to the Arizona Department of Health Services, annual average background concentrations should be approximately 3 to 5 $\mu g/m^3$. Therefore, within approximately 20 miles of the Inspiration Consolidated smelter, annual average SO_2 concentrations are seen to decrease to a level that would be about 2 to 3 times the ambient background values. Since 1974, there have been no recorded violations of the annual ambient air quality standard for SO_2 (80 $\mu g/m^3$) in the vicinity of this or any of the other three smelters in Pinal and Gila Counties. Information provided by the local agency indicates that 75 percent of the 10,000 square miles in these two counties experience annual average SO_2 concentrations of less than 15 $\mu g/m^3$. High annual and short-term concentrations (those which come close to the ambient air quality standards) are typically observed only in proximity to the actual smelting activities, usually within 2 to 3 miles of each smelter. Short-term violations of the air quality standards in the vicinity of the Inspiration smelter are usually, but not always, associated with equipment failure (i.e., such as an inoperative sulfuric acid plant).

If it is assumed conservatively that the emissions from the Inspiration smelter alone result in ambient SO_2 concentrations that approach or equal the ambient standards for SO_2, then the contribution to these concentrations that result from processing copper concentrate from the Pinto Valley mine should represent a 30 to 50 percent consumption of the standards.

On the basis of the above arguments it appears that the general area surrounding the Pinto Valley mine could conceivably support additional mining and smelting activities, depending on their location.

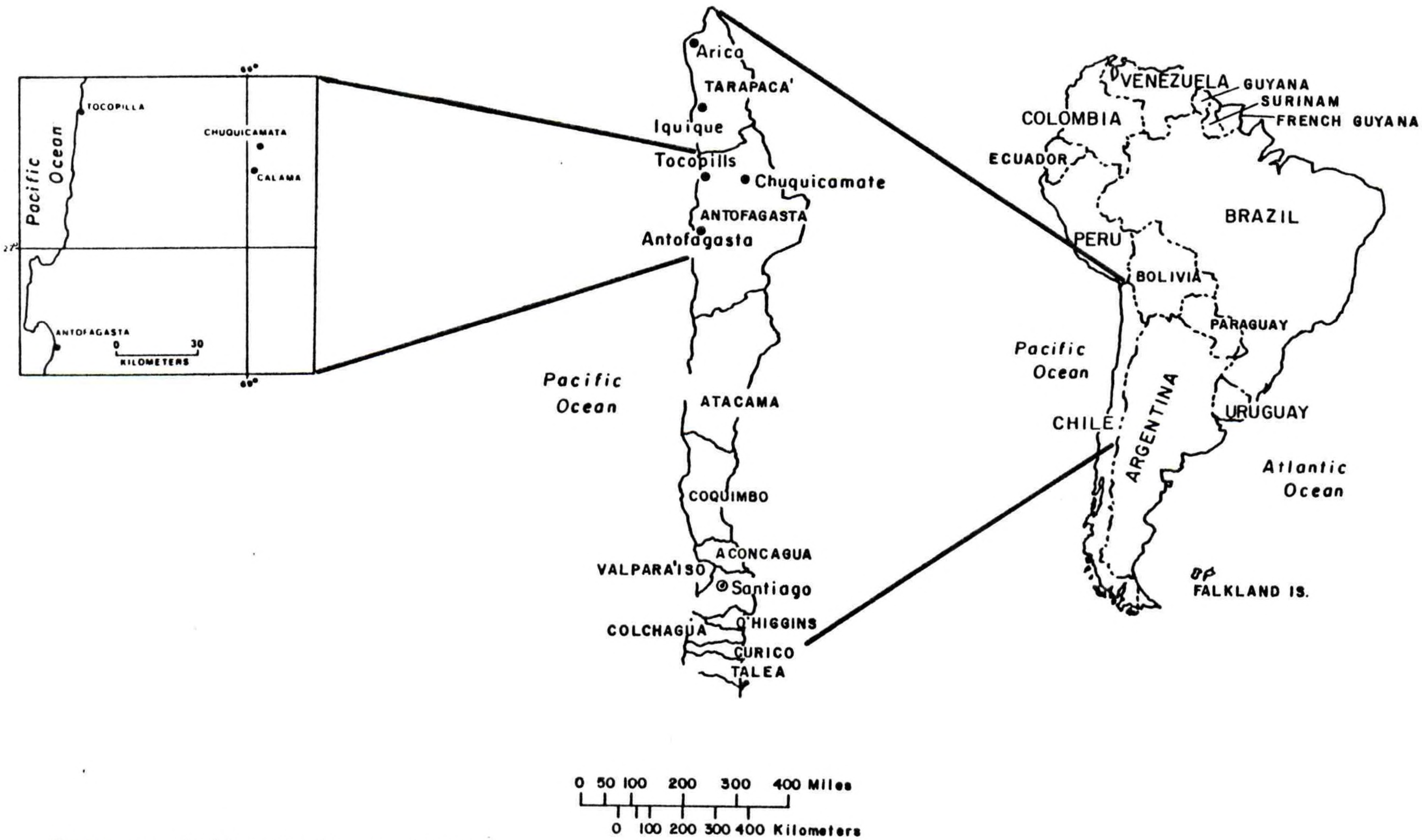
The construction and operation of any new or modified facilities would necessarily involve the acquisition of both state and federal approval in the form of construction and operating permits. For facilities of the type described here (mine and smelter), this would involve a demonstration of compliance with both the regulations governing the Prevention of Significant Deterioration (PSD) and the National Ambient Air Quality Standards (NAAQS). Compliance with these requlations would be necessary to ensure that the general air quality in the area would not significantly deteriorate from existing levels and that public health would not be threatened as a result of the operation of any new or expanded facility. Thus, regulatory limitations on air quality will impose some local and regional controls on copper mining and smelting.

3.1.2 Chuquicamata, Chile

Chuquicamata is a surface mine in northern Chile 90 miles inland from the small port of Tocopilla, 150 miles northeast of the port city of Antofagasta in the Province of Antofagasta in the Atacama Desert (Figure 3-4). The mine is at an elevation of about 9300 feet on the edge of a low hill on the Pacific slope of the Andes (Smith, 1967).

3.1.2.1 General Setting

Chile stretches 2600 miles along the west coast of South America from the edge of the Pacific to the crest of the Andes. It averages 100 miles in width, with the widest point some 220 miles across. The country is readily divisible into three areas both north to south and east to west. The northern arid deserts are inhospitable and are sparsely populated. The central area of Chile has a Mediterranean climate, level terrain, rich soils, and supports most of the population. The southern part of the country varies greatly in topography and climate - the forested lands resembling the Pacific Northwest and the Tierra del Fuego being comparable to that of the Alaskan panhandle. Chile can also be divided into three geographic areas from west to east, with a coastal range, a central valley, and the Andes.



Sources: Weil and others, 1969.
Cook, 1978.

Figure 3-4. General Location of Chuquicamata, Chile.

Chuquicamata is in the so called "Great North." This is an extremely arid region ecompassing the provinces of Antofagasta and Tarapaca in the northern Atacama Desert (Figure 3-4). The coastal reange is between 2000 and 3000 feet above sea level, but some peaks rise to 7000 feet. Cliffs abutting the ocean are common, and the few towns occupy the localized coastal plain areas. The central valley comprises a series of high plateau basins with sands, clays, and salts; it was in this area that the mining of nitrates was carried on extensively prior to World War I. The Andes form the eastern geographic division, and the crest marks the boundary with Bolivia and Argentina. In this area passes are all above 10,000 feet. The Loa River rises in the Andes and flows 270 miles to the Pacific at Tocopilla. Numerous other streams rise in the Andes, but sink into the desert basins without reaching the sea.

The climate of Chile is extremely varied - influenced not only by the great changes in latitude, but also by the extremes of altitude - from sea level to 10,000 feet and more in distances of 100 miles. Also influencing the climate is the Humboldt or Peru Current which carries cold waters and accompanying air masses from the Antarctic along the coast. At the northern border with Peru at Arica (Figure 3-4) the rainfall is 0.03 inch/year, and the average annual temperature is 66°F. Rains are so rare in parts of the interior deserts that some sites have been reported to have had no rain for 400 years (Weil and others, 1969). Rain falls occasionally in the Andes at Elevations greater than 8000 feet. In the town of Calama just south of Chuquicamata humidity is 48 percent, and in some parts of the desert the coefficient of evaporation is higher than in the Sahara. At Iquique on the coast the average humidity if 81 percent. In the Great North, heavy winter mists, camanchaca, provide moisture along the coast for some seasonal desert flora. In the interior deserts however there is almost no rain and no native vegetation; the soil has extremely high salinity. On the slopes of the Andes scattered cacti and desert shrubs occur.

Wildlife in the northern Andes include the guanau, llama, alpaca, and vicuna (all members of the camel family). Also present are a large deer (the huemul), the Andean wolf, puma, and two kinds of wildcat. The chinchilla is a native rodent, now nearly extinct. Birds include the condor, which occurs occasionally in the Andes.

There are few population centers in the Great North. Most of the urban population lives in coastal cities and towns which are primarily fishing villages or ports providing for export of minerals mined in the interior. Population centers in the interior have grown up primarily around mines in the mountains and the desert. The desert centers have declined following the decline in nitrate mining. The only other significant far northern population centers are scattered desert oases which support small farming communities. Some date back to the days of the Inca Empire when they served as way stations between Cuzco and the Inca settlements farther to the south. The largest of these, Calama on the Loa River, is a service center for the Chuquicamata copper mine and has important air and rail facilities. Many of the smaller oasis villages survive because of their service to nearby mining operations (Weil and others, 1969).

The provinces of Tarapaca and Antofagasta were acquired by duels following a war with Peru and Bolivia in 1883. Acquisition of the port of Arica however was not finalized until 1929. The government for most of this century has been centered in the central, populated part of the country so that the northern provinces have been considered more as colonies.

Surface mining is typically not labor intensive, so the number of jobs created by the integrated Chuquicamata mining/processing facilities is not large. Mining companies have, however, historically paid the highest wages in Chile. In 1967 mining, including quarrying, employed 83,400 people, or 3 percent of the labor force. In 1969, while accounting for only 4 percent of the labor force, the mining industry accounted for 90 percent of Chile's export; copper represented 8 percent of the gross national product, and 60 percent of the nation's exports.

3.1.2.2 Mine and Processing Facilities

Production of copper from oxide ores began at Chuquicamata in 1915. Mining of sulfides began in 1955. The Chuquicamata deposit is inone of the greatest zones of porphyry copper ever discovered, so large in fact that the limits of the ore zone have never been determined. Reserves are estimated at 17.6 million metric tons (mt) of copper (grading more than 1.01 percent). These figures do not include reserves at Mina Sur (170 million mt of 1.36 percent copper) 6 km south of the mine, nor those at Chuqui Norte (2.42 million mt of 0.7 percent copper) 6 km to the north. Ore grades taken from the pit in 1979 at Chuquicamata averaged 2.07 percent copper. This is expected to decline to 1.55 percent in 1981; 1.27, in 1989; and 1.02, by the year 2000. If declines in ore grade are not accompanied by an increase in the size of the processing facilities, output would drop to 50 percent of the 1978 level of 500,635 metric tons of commercial copper. Studies indicate that an investment of 56 million dollars could maintain production at 75 percent of 1978 levels (Dayton, 1979b).

The mine, owned by Corporcaion Nacional del Cobre del Chile (CODELCO, Chile) is an open pit, 3000 meters by 1600 meters by 450 meters deep. Preproduction stripping removed more than 60 million metric tons of material; the present stripping ratio is 3:1.

The Chuquicamata porphyry deposit consists predominantly of altered and fractured intrusives characteristic of the many plutons which form the core of the Andes. The ore body contains more or less evenly disseminated sulfide mineral grains and mineralized veins. Zones of enrichment have been formed by leaching and redeposition of copper from downward percolating surface water.

The sulfide mineral suite is dominated by chalcocite (45 percent). As mining continues, the importance of chalcocite will decrease, and chalcopyrite, covellite, and pyrite content of the ores should increase. This will be accompanied by a decline in feed grade requiring an increase in the grade, of copper in the concentrate and thus additional smelting energy use at the smelter.

The ore is concentrated, smelted, and refined at the mine and the electrolytic metal and blister bars are transported by rail (the Antofagasta and Bolivian Railroad) to the city of Antofagasta, where tthey are exported by cargo vessel. There is also an airline from various cities to Calama and there are paved highways to major cities.

Although oxides were originally mined from Chuquicamata, a sulfide concentrator became operative in 1952; this plant was sized at 25,000 mtpd. Flotation capacity was enlarged in 1964 to give the plant a capacity of 55,000 mtpd. Further modification, including a secondary-tertiary crusher, new grinding capacity, and flotation lines, have increased the capacity to 70,000 mtpd (Dayton, 1979b). In 1978, production was 450,000 net of product. Of this about 355,000 tons was new copper, and the remainder was from scrap.

3.1.2.3 Environmental Impacts

As discussed in the previous section, the Chuquicamata copper mine is a large open pit surface mine with an ultimate pit size of approximately 10,000 x 5000 ft. Approximately 330,000 metric tons/day of material is handled at this mine, of which 75 percent is rejected as waste rock. The remaining 25 percent undergoes further processing at the mine (mainly crushing and grinding) and at the smelter (mineral reduction blister copper). Approximately half of Chile's copper production is attributable to the Chuquicamata mine (Weil and others, 1969).

Because of the importance of copper to the Chilean economy and the isolated barren nature of the mine site there is apparently little concern for environmental protection. In general, many environmental impacts are similar to those discussed in Chapters 1.0 and 2.0. There are no competitive land uses in the area. There is almost no vegetation; wildlife is mostly confined to the higher slopes of the Andes where there is some vegetation.

Applying the impact factors of Chapter 2.0 to present production levels at Chuquicamata yields the following estimate of annual resource use: land for mining and processing, 1000 acres; energy, 4.5 x 10^{13} Btus; water, 2.3 x 10^{10} gallons; employment, about 9000. However, it should be noted that, at a grade of about 2 percent, less than one-third as much ore must be mined at Chuquicamata to yield the same amount of copper as in our prototype mine. Resource use at Chuquicamata mine is probably correspondingly lower, though it will approach that of the prototype by 2010.

The major impact on air quality from the mining operation is from fugitive dust. This dust originates from typical open pit mining activities such as overburden removal, ore body excavation and ore transportation and processing, as described in Section 1.2.3.

Fugitive dust emissions from this mine can be expected to be aggravated by the fact that the area is extremely arid with almost no rainfall in the surrounding desert region. Under this type of climatological influence fugitive dust is very difficult to suppress. Although the magnitude of the impact on air quality from fugitive dust is expected to be significantly greater than at the Pinto Valley mine, the characteristics of the impacts can be expected to be quite similar. Both operations are in very dry environments and both mine a similar type of copper deposit. As with the Pinto Valley mine, the major impacts of the fugitive dust generated at the Chuquicamata mine will occur on a local scale (on the order of 10 miles).

A relatively small quantity of particulate and SO_2 emissions result from diesel engine exhaust from trucks, shovels and generators interspersed throughout the mine site. No significant impact on air quality should be expected to result from the operation of these sources.

The most significant air quality impact usually associated with the production of copper results from the emissions of sulfur dioxide (SO_2) at copper smelters. The annual production of copper at the Chuquicamata smelter is about 450,000 metric tons, 355,000 metric tons of which can be attributed to new copper production (Dayton, 1979b). This should be compared with an annual copper production rate of about 120,000 tons/year at the Inspiration Consolidated smelter near Miami, Arizona (Section 3.1.1). It would be expected that, since the deposits for each of these mines are of a similar type, their emissions would be proportional to their production of copper. By this reasoning, the SO_2 emissions at the Chuquicamata smelter should be approximately three times those of the Independence smelter in Arizona if the same degree of SO_2 control measures are applied. If the level of SO_2 control at Chuquicamata is significantly lower than at the Independence smelter (as is thought to be the case), substantially more SO_2 could be released. There is apparently little concern for environmental protection in the area since the area is isolated and relatively barren. Furthermore, the production of copper is extremely important to the Chilean economy and it is not likely that expensive control measures would be imposed at Chuquicamata.

Since both the Chuquicamata and Miami, Arizona, areas have similar climates, one would expect similar types of impacts on air quality can be expected to occur as a result of the smelting operations in these areas, although the magnitude of the impacts at Chuquicamata are expected to be significantly greater than at the Pinto Valley site (no emissions or air quality data are available). For the magnitude of the impact of SO_2 emissions in the Chuquicamata area, the reader should refer to the discussion on the observed impacts of SO_2 emission resulting from the operation of the Pinto Valley mine (Section 3.1.1), recognizing that emissions from Chuquicamata are probably at least three times greater. It is likely that U.S. ambient air quality standards (if they applied) would be exceeded quite frequently.

3.1.3 San Manuel, Arizona

San Manuel is an example of a large domestic underground porphyry copper mine. It is located in the southeastern corner of Pinal County, Arizona, near the intersection of the Pinal-Graham-Cochise-Pima County lines, and is about 35 miles northeast of Tucson. Mining began in 1956, and San Manuel has since become the largest underground mine in the United States. Since over two-thirds of the ore reserves remain to be extracted, continued large-scale production is considered likely.

3.1.3.1 General Setting

The San Manuel mine is approximately 8 miles north-northwest of the town of San Manuel (population 4670) in T 8 and 9 S, R 16 E (Gila and Salt River Base Line). The area is characterized by dissected hills, alluvial-filled valleys, and numerous intermittent streams which ultimately drain into the San Pedro River.

The rocks occurring in the area include Precambrian granite; Cretaceous granodiorite; Cretaceous and Tertiary fanglomerate, latite flows, and intrusives; Tertiary to Quaternary fanglomerate; and Quaternary gravel. The area is highly faulted, most of the faults trend NW-SE or NE-SW.

Throughout the area there are numerous mines and prospects. The area is enriched in copper, gold, silver, lead, zinc, molybdenum, tungsten, vanadium, iron, bismuth, tellurium, and selenium.

The approximate elevation of the San Manuel mine is 3500 feet. The predominant vegetative type is Encinal Oak Series (Brown and others, 1979), a rocky habitat dominated by several species of evergreen oaks, alligator juniper, manzanita, and a number of chaparral shrubs. The following is a description of the Encinal Oak Series.

Terrain: Foothills and mountain canyons, 3500-7000 feet elevation.

Climate: Semiarid, 12-20 inches of precipitation annually, bimodally distributed, only a small part as snow. Warm summers, with highs often above 100. Mild winters days with freezing temperatures at night fairly common.

Soils: Udorthentic haplustalls, lithic torriorthids and haplargids, lithic camborthids and haplustralls.

Vegetation: Dense shrubby growth, of farily uniform height from 3 to 7 feet. Dominants are sclerophyllous evergreen shrubs like scrub oak, manzanita, sugar sumac, squawbush, mountain-mohagony, Emory oak, Arizona White oak, and gray oak.

Fauna: Common bushtit, rufous-sided towhee, blue-gray gnatcatcher, brush mouse, and mule deer.

Important biological features of the area include the occurrence of game species, such as mule deer and quail, and the fairly limited distribution of this vegetative community in Arizona and New Mexico.

The San Manuel concentrator, smelter, and refinery are all located in the company town of the same name; the mine is some 8 miles to the north-northwest of the town (Figure 3-5). Total 1975 employment was 4151, including both the labor-intensive underground mining activities, and the processing facilities. Of the four counties within a 30-mile radius of the site, only two, Pinal and Pima, have population centers likely to be socioeconomically affected. Current population and unemployment rate estimates are as follows:

Current Population and Unemployment Rate Estimates
San Manuel, Arizona

County	Projected Population (7/1/80)	Population Density (per sq. mile)	Unemployment Rate (12/31/79)
Pinal	92,500	17	6.7
Pima	539,800	58	4.9

3.1.3.2 Mine and Processing Facilities

The San Manuel mine is an integrated mining, processing, smelting and refining facility. Most of the workers live in the company-owned

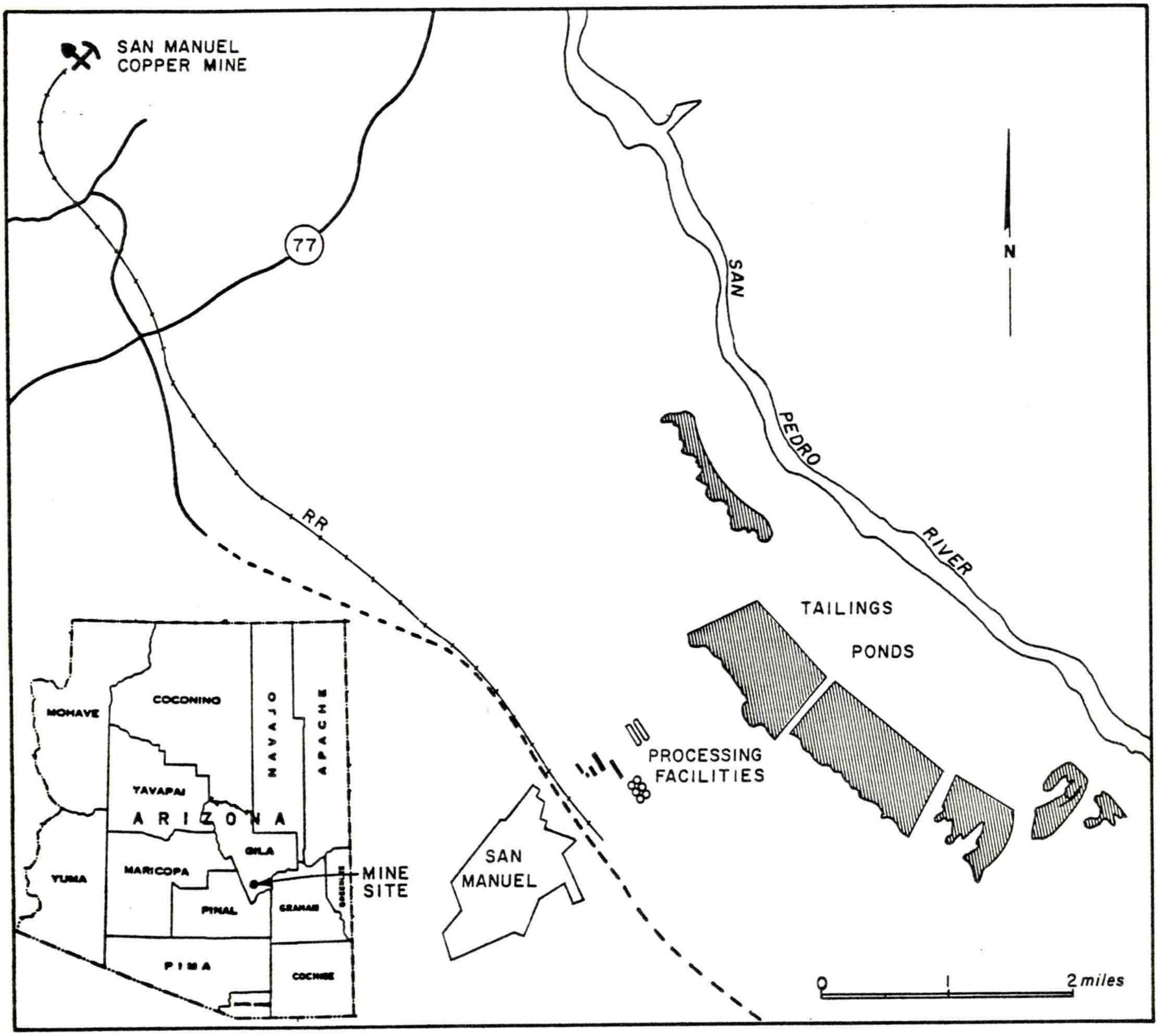


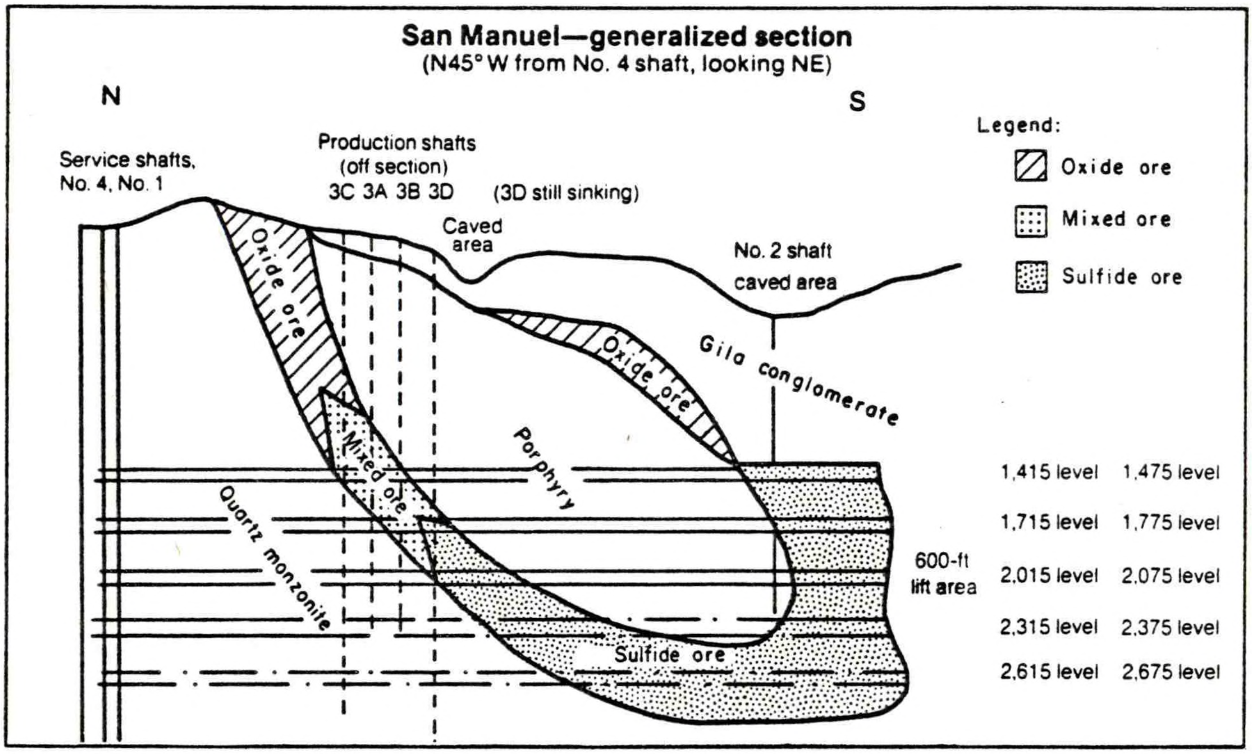
Figure 3-5. Location of San Manuel Mine

town of San Manuel. The mine, which is owned by the Magma Copper Company, was opened in 1956 with a daily capacity of 30,000 tons of ore; this has since been increased to an average of 52,000 tpd. Estimates are that production of copper over the last 20-plus years has removed only about one-third of the reserves (Jackson, 1978).

The ore body is a low grade (0.7 percent copper) disseminated porphyry deposit. Mineralization is about equally distributed between coarse-grained quartz monzonite (Precambrian) and monzonite porphyry (Mesozoic). The Precambrian deposits are of slightly higher grade. The elliptical shell which forms the ore deposit (Figure 3-6) is from 100 to 1000 feet thick. The entire mineralized area is about 8000 feet long with cross sectional axes of 5000 and 2500 feet. Primary ore minerals are pyrite, chalcopyrite, molybdenite, and some rare bornite; in the oxide zone (Figure 3-6) the primary mineral is chrysocolla (Jackson, 1978).

Mining was initiated at the 1475 foot level and has progressed downward to 2675 feet, expansion is planned down to 3740 feet. The mine has been able to remain competitive with nearby surface mining facilities because of the use of the block caving method of mining (see Part II of this report, and Jackson, 1978). To date some 115 acres of undercut area have been excavated. The ore is stored in rises and transported by trolly-operated railroad to the hoists. At the surface the ore is placed in storage bins for transfer to gyratory crushers. From the rushers the ore is again removed to storage bins from which it is transferred by rail 6 miles to the concentrator and from there to the smelting and refining facilities.

Power demands exceed 90 MWE (which is purchased from Arizona Public Service Company Company). Power is used for the hoists, air compressors, ventilation systems, dewatering pumps, and cooling towers. The facility also maintains a standby boiler which can produce emergency power if necessary.



Source: Jackson, 1978.

Figure 3-6. The San Manuel Ore Body.

The smelter is at the townsite; annual production is 200,000 tons, primarily of anode copper. Tailings are dumped behind levees more than 200 feet upslope from the San Pedro River Valley. In 1972 some 15 years after the facilities were placed in operation these tailings sites covered approximately 1200 acres.

Proven reserves total 270 million tons of ore grading 0.7 percent copper; probable reserves total 671 million tons of 0.7 percent copper ore, and possible reserves are estimated at more than 1 billion tons of 0.7 percent ore (Jackson, 1978).

3.1.3.3 Environmental Impacts

The San Manual mine is an underground mine and, consequently, has less direct effect on land resources than the surface mines previously discussed. Impacts to the biological resources of the area involve loss of habitat in the immediate area due to spoils dumps, buildings, facilities, and appurtenances. Spoils and effluent are contained at the mine, and risks of contaminating any nearby streams are low.

The several smelter tailings ponds occupied approximately 1200 acres in 1972 and were retained behind levees upslope from the main drainage (San Pedro River, normally no surface flow) in the area (Figure 3-5). Should any overflow occur from these areas as a result of heavy rains, seepage, or levee failure contamination of surface water and of water from the many springs in the area could occur.

Given the relative nearness of the city of Tucson, many workers commute to San Manuel. Also, the town of San Manuel was built by Magma and could be enlarged if necessary. This tends to minimize the potential for adverse socioeconomic impacts associated with future mining operations.

Applying the unit parameters for underground porphyry mines in Chapter 2.0 to San Manuel, and assuming that all the probable reserves are mined (4.7 million tons of copper), resource use would be as follows: land for mining (subsidence) and waste rock, 2300 acres; land for processing and tailings, 5800 acres; energy for mining and processing, 0.48 quad; water, 23.5 x 10^{10} gallons. Direct use of land would be more than 12 square miles, an indication of the very large commitment required to mine such quantities of porphyry copper, even from an underground mine.

Since the San Manuel mine is underground, its impact on ambient levels of total suspended particulate matter (TSP) is not expected to be very great. Those impacts that do occur are usually in the immediate vicinity of discharge air vents and at ore processing facilities (crushers, grinders, etc.) located at the surface. The emissions of fugitive dust from this type of mine can be expected to be several orders of magnitude less than for a comparably sized open-pit surface mine, such as the Pinto Valley mine.

Significant impacts on ambient air quality can be expected to result from operation of the San Manuel copper smelter. Virtually all of the SO_2 emissions from this smelter are associated with the San Manuel copper mine.

According to conversations with the Arizona Department Health Services (Chelgrin, 1980), significant emissions of SO_2 result from the operation of seven copper smelters in Arizona, four of which are located in the Pinal-Gila County air quality control district. According to the state, the average daily SO_2 emission from the San Manuel Smelter for the years 1976-1978 were as follows:

Year	Average Daily SO_2 Emissions (tons)
1976	514
1977	558
1978	600

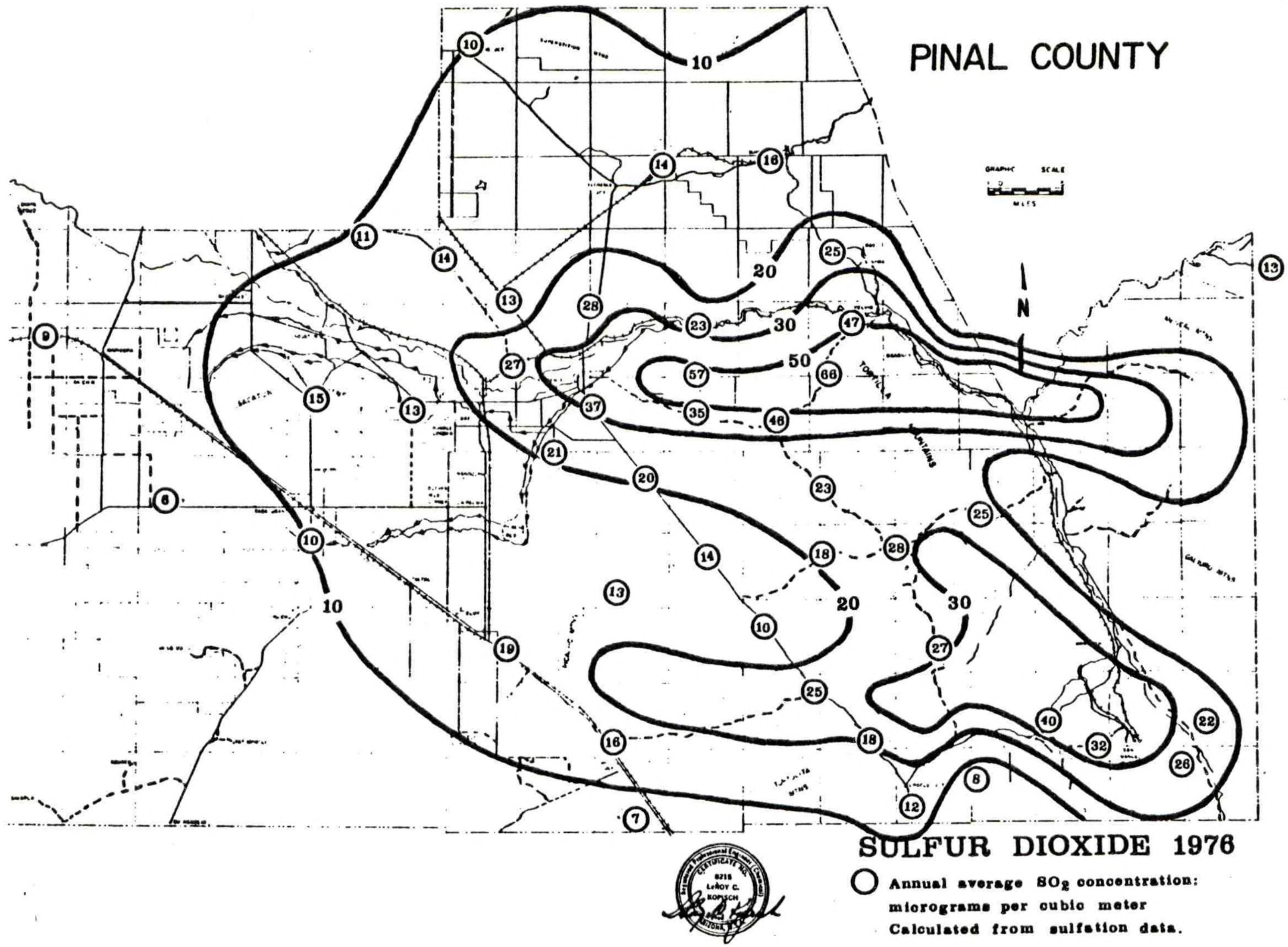
As can be seen, the emissions from this facility are relatively constant and are approximately 400 percent of those from the Inspiration smelter discussed previously for the Pinto Valley mine. The current level of SO_2 removal at the San Manuel smelter is about 50 percent (compared with 80 percent removal at Pinto Valley), the primary method being the production of sulfuric acid at an onsite acid plant.

Since the San Manuel mine is located in the same general area as the Pinto Valley mine (Section 3.1.1) there should be notable similarities in the general meteorological and ambient air quality conditions between the two sites. For a general discussion of the existing air quality in the area, the reader can refer to Section 3.1.1 (Pinto Valley mine).

The results of annual average SO_2 monitoring in the area of the San Manuel smelter as performed by the Pinal-Gila Air Quality Control District Agency during 1976 are shown in Figure 3-7. The figure shows that in the San Manuel area (that is, the southeast section of Pinal County), there is a distinct node or area of locally high concentrations. Within approximately 20 miles of the San Manual mine, SO_2 concentrations are seen to be below 10 $\mu g/m^3$, except for an area to the north where the influence of another smelter is evident (the ASARCO smelter near Hayden).

As with the Pinto Valley mine, high annual and short-term concentrations (those which come close to the ambient air quality standards) are typically observed only in the proximity of the actual smelting activities, usually within 2 to 3 miles of each smelter. According to the state agency, short-term violations of the air quality standards in the vicinity of the San Manuel smelter are usually, but not always, associated with equipment failure (such as an inoperative sulfuric acid plant).

On the basis of these arguments it appears that the general area surrounding both the San Manuel and Pinto Valley copper mines could conceivably support additional mining and smelting activities, depending on their location. The construction and operation of any new or modified facilities would necessarily involve the acquisition of both state and Federal approval in the form of construction and operating



Source: Pinal-Gila Counties Air Quality Control District, 1977.

Figure 3-7. Observed Annual Average SO_2 Concentrations in Pinal County for the Year 1976.

permits. For facilities of the type described here (i.e., mine and smelter), this would involve a demonstration of compliance with both the regulations governing the Prevention of Significant Deterioration (PSD) and the National Ambient Air Quality Standards (NAAQS). Compliance with these regulations would be necessary to ensure that the general air quality in the area would not significantly deteriorate from existing levels and that public health would not be threatened as a result of the operation of any new or expanded faility.

If it is assumed that the emissions from the San Manuel smelter alone could result in ambient short-term SO_2 concentrations that approach or equal the ambient standards for SO_2, then any potential expansion of smelting activity would necessarily have to be associated with an increased efficiency of SO_2 removal. This would almost certainly be necessary to ensure compliance with the regulations governing PSD and the NAAQS as discussed here. It also appears that a greater level of SO_2 reduction is possible at San Manuel than at Pinto Valley, primarily because the current level of control at San Manuel is relatively low (i.e., approximately 50 percent).

3.2 NICKEL

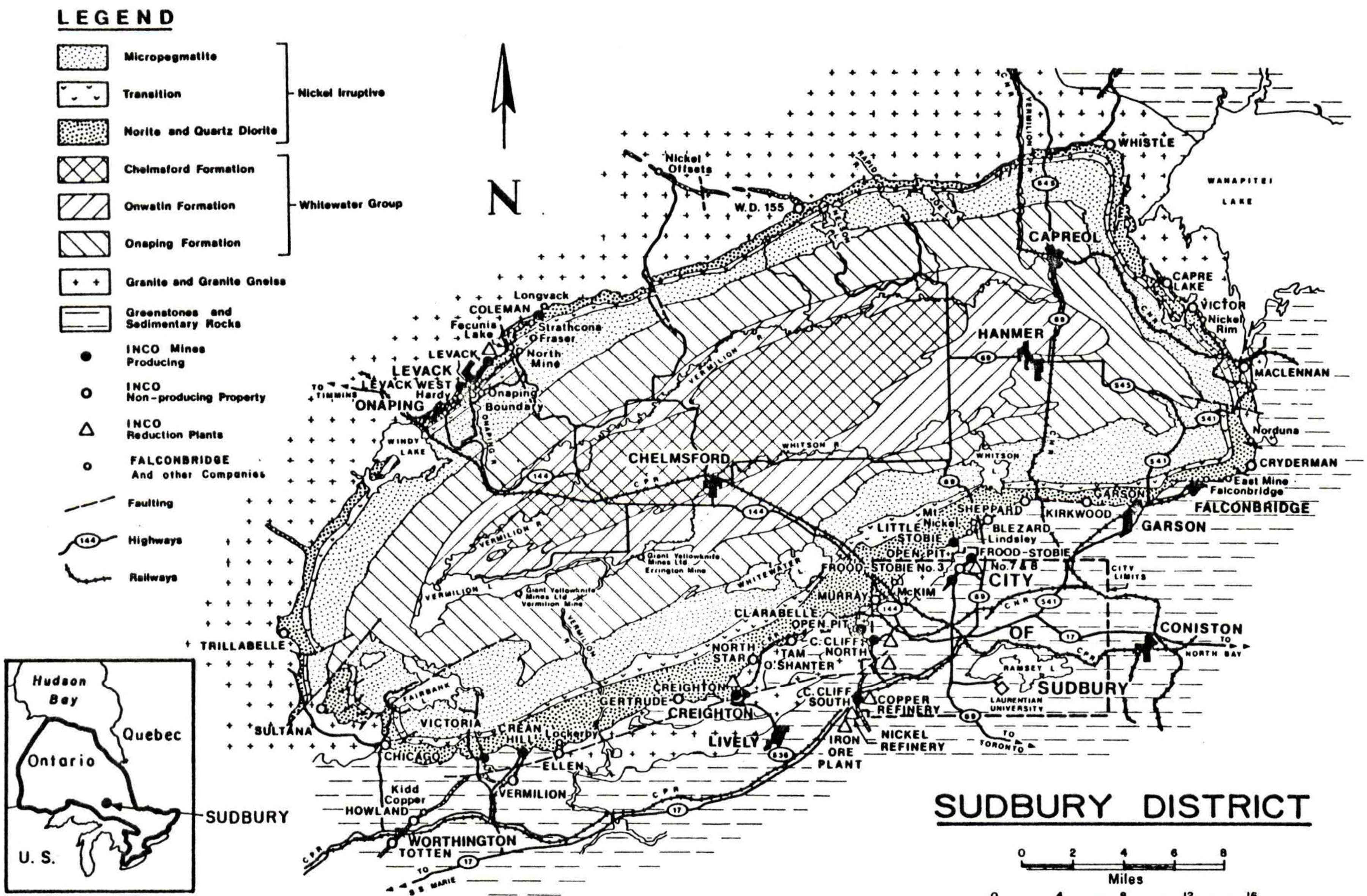
Nickel ores are of two primary types: sulfide deposits and nickel laterites (see Part I and Part II). Most sulfide deposits are mined by underground methods, while laterites are produced from surface mines. Sulfide ores are usually associated with mafic igneous intrusions such as the Precambrian Sudbury Complex at Sudbury, Ontario. Laterites are derived from weathering of various ultramafic rocks, primarily in tropical and subtropical climates. Historically, sulfide ores provided most of the world's nickel - mines at Sudbury, Ontario, produced 80 percent of the world supply in 1964. The large laterite deposits in such countries as Cuba, the Philippines, Indonesia, New Caledonia, and Australia are providing increasingly larger portions of world supply. Because of the importance of both types of ores to world supply, one of each type was selected for discussion: The Creighton mine at Sudbury, Ontario, and the Greenvale mine, in the northeast sector of Queensland, Australia.

3.2.1 Creighton Mine, Sudbury, Ontario

The Sudbury region of southern Ontario (Figure 3-8) has been a famous mining area since the late 19th century. The mines at Sudbury produce copper, nickel, and cobalt from sulfide ores. As of 1978, this area had twelve underground mines either operating or on a standby basis.

3.2.1.1 General Setting

The Sudbury district of Ontario is in a glaciated region of moderate relief. The ore bearing rocks occur in an elliptical area 37 miles long by 17 miles, wide with the longer axis trending north-northeast (Figure 3-8). The central portion of the ellipse is occupied by late Huronian (Precambrian) rocks which have been eroded to a peneplain. Surrounding this area is a hilly belt of eruptive rocks. The geologic sequence consists of Keewatin volcanics (oldest); graywacke, quartzite, and conglomerate; and highly metamorphosed granites and gneisses. These rocks have been intruded by somewhat younger Precambrian rocks,



Source: Mohide and others, 1977.

Figure 3-8. Geologic and Location Map of Sudbury District.

which have been differentiated into a lower, noritic portion and an upper granite micropegmatite; the rocks grade into each other. The ore deposits are associated with the root intrusion (Emmons, 1940). The intrusion is shaped like the bowl of a spoon; the outcrop area around the rim of the spoon has a hilly surface.

The area is in the Canadian Shield region which is underlain almost entirely by mixed crystalline rocks of Precambrian age with only small areas of younger sedimentary rocks. Surficial materials consisting primarily of glacial materials are distributed very irregularly, with little or none in the upland areas and thick deposits largely confined to the valleys. The topography is rugged, but relief is seldom more than a few hundred feet. The climate is humid continental, and total annual precipitation (much as snow) varies from 22 inches in the west near the Interior Plains to 44 inches in the east near the Gulf of St. Lawrence. About one third of the mean annual total precipitation falls as snow in the west and about one half in the eastern part.

Good surface water is abundant, and ground water is not widely used. Only some 20 percent of the total population, including rural, uses ground water. "Approximately 150 million gpd of water (not including industrial supplies) are used by municipalities and of this an estimated 11 million gallons are ground water" (Brown, 1968). The best sources are the coarser grained surficial materials deposited by glacial meltwaters.

In general, both surface and ground water quality are good. Both the crystalline rocks and the surficial materials are composed of nearly insoluble minerals, and ground water quality is similar to the acidic surface waters and has a similar low pH. Poor quality water may be encountered near some mineral deposits. In most base metal mines the ore body contains sulphides and the ground water is strongly acidic and corrosive (Brown, 1968).

3.2.1.2 The Mine and Processing Facilities

At present, INCO has nine underground mines and one open-pit mine in operation in the Sudbury district. In addition to this production capacity, INCO also maintains four mines in the Sudbury district on a standby basis. To increase its ore reserves, INCO has initiated a deep drilling program at all of its properties in the Sudbury Basin. According to INCO sources, there are plans to produce approximately 135,000 to 150,000 tons of nickel from its Sudbury operations. This is approximately 65 percent of INCO's total Ontario production capacity.

The principal copper-nickel mineralization consists of pyrrhotite, pentlandite, and chalcopyrite. Mineralization is localized and does not occur uniformly throughout the district at the norite-micropegmatite contact. Through exploration, the most economically significant mineralization has been found.

The Creighton ore body is an irregular tabular sheet - as much as 1000 feet long and 180 feet wide (Smith, 1967). The ore deposits occur from the surface to a depth of over 3800 feet in a depression at the base of the Sudbury intrusive in a zone of intense shearing and brecciation. The mineralization consists of massive and disseminated sulfides at the intersection of shears in the Footwall Rocks. According to Boldt (1967, p. 35), the typical Creighton orebody "is made up essentially of an upper part consisting of disseminated sulfides in contact phase norite and a lower part consisting of massive or breccia sulfide in the footwall rocks. Sheets and pipes of sulfide also occur along and at the intersection of shears in the disseminated norite."

The Creighton mine is one of the largest underground mines in the Sudbury district with a daily ore production capacity of approximately 8000 tons/day. Ore milling and concentrate preparation is performed at the mine site and the resulting concentrates are piped as a slurry approximately 7.5 miles to a smelting operation operated by the International Nickel Company of Canada (INCO) in nearby Copper Cliff.

3.2.1.3 Environmental Impacts

The unit impact parameters of Chapter 2.0 could be used to estimate the approximate resource use during the life of the Creighton mine. Assuming a production of 150,000 tons of nickel, land use would be 165 acres; energy use, 2.5×10^{13} Btus, and water use, 0.35×10^{10} gallons during the life of the project. Comparing these figures with those for the porphyry mines in Section 3.1, it is obvious that the smaller copper-nickel sulfide mines are much less demanding of resources. Impacts per unit of production are not too dissimilar, however.

Since the Creighton mine is underground, its impact on ambient levels of total suspended particulate matter (TSP) is not very great. Those impacts that do occur are usually in the immediate vicinity of discharge air vents and at ore processing facilities located at the surface. The emissions of fugitive dust from this type of mine can be expected to be several orders of magnitude less than for a comparably sized open pit surface mine. Some fine particle emissions may be associated with storage areas for tailings; however the magnitude of these is expected to be relatively insignificnt.

The most significant source of emissions associated with the operation of this mine are SO_2 emissions from the smelter complex located in nearby Copper Cliff. Emissions of SO_2 from this particular smelter represent the largest single point source of SO_2 emissions in the world, with daily emissions of up to 3600 tons. It has been estimated by Likens and others (1979) that some 1 percent of the total annual emissions of sulfur dioxide throughout the world (from both natural sources and human activity) come from this one smelter.

The INCO smelter at Copper Cliff is a rather unusual case, not only because it emits such a massive amount of sulfur dioxide into the atmosphere, but also because these emissions are released from the world's tallest stack (1250 ft.). The result is that the impacts of the emissions from this smelter extend over a very large area. Its plume is in fact visible at times for many hundreds of kilometers downwind. This situation can give rise to many impacts which would probab ly not occur with lower heights of emissions (such as those at Pinto Valley and San Manuel). Among them is acid precipitation (see Section 1.2.3). Since the SO_2 in the plume can remain airborne for very large distances and hence very long times, the probability of some form of acid precipitation is increased considerably. Furthermore, the impact on ground-level concentrations at large distances could become very significant.

Under regulations imposed by the Province of Ontario's Ministry of Environment, SO_2 emissions from this facility are limited to a maximum of 3600 tons/day. At present, daily SO_2 emissions from the facility are very close to the 3600 ton/day emission limitation, with the average daily emissions for the past 6 years as follows (Fitz, 1980):

Year	Average Daily SO_2 Emissions (tons)
1973	3100
1974	3100
1975	3300
1976	3200
1977	3000
1978	1500 (labor strike)

As can be seen, the emissions from the smelter are relatively constant with the exception of the year 1978 during which a labor strike curtailed production. For a given production rate, these values are not expected to vary significantly, since, unlike many smelters, only 10 percent of the original SO_2 in the exhaust gases is removed for the production of sulfuric acid. The remaining 90 percent is emitted directly into the atmosphere. (In contrast, an 80 percent SO_2 removal efficiency is achieved at the Inspiration Consolidated Copper smelter in Miami, Arizona, with the removed SO_2 being used for onsite sulfuric acid production, Section 3.1.1.3).

It should be noted that it would be very difficult to quantitatively determine the impact of this source on ground-level concentrations at distant locations. Conventional dispersion models lack sufficient accuracy to predict plume behavior at these large distances. It would also be difficult to determine the contribution from a single source in ambient air quality monitoring data obtained at large distances, since more than one source can contribute to the apparent background concentrations. For distances very close to the stack (within several miles) high concentrations in excess of the Provincial air quality standards normally occur only under unstable atmospheric conditions in conjunction with plume looping (that is, a condition whereby the plume body is abruptly displaced downward to ground level by thermal-induced currents in the atmosphere).

It is estimated that approximately 15 percent of the production at the Copper Cliff smelter is attributable to the activity at the Creighton mine. This information is based on the daily mine capacity of all of the INCO mines in the Sudbury district (Worobec, 1977). This would indicate that for a maximum daily emission rate of 3600 tons/day, approximately 540 tons/day would result from the Creighton mine activities. (The SO_2 emissions associated with the Creighton Mine are four times those of the Pinto Valley mine and equal to those at San Manuel, discussed in Sections 3.1.1 and 3.1.3.)

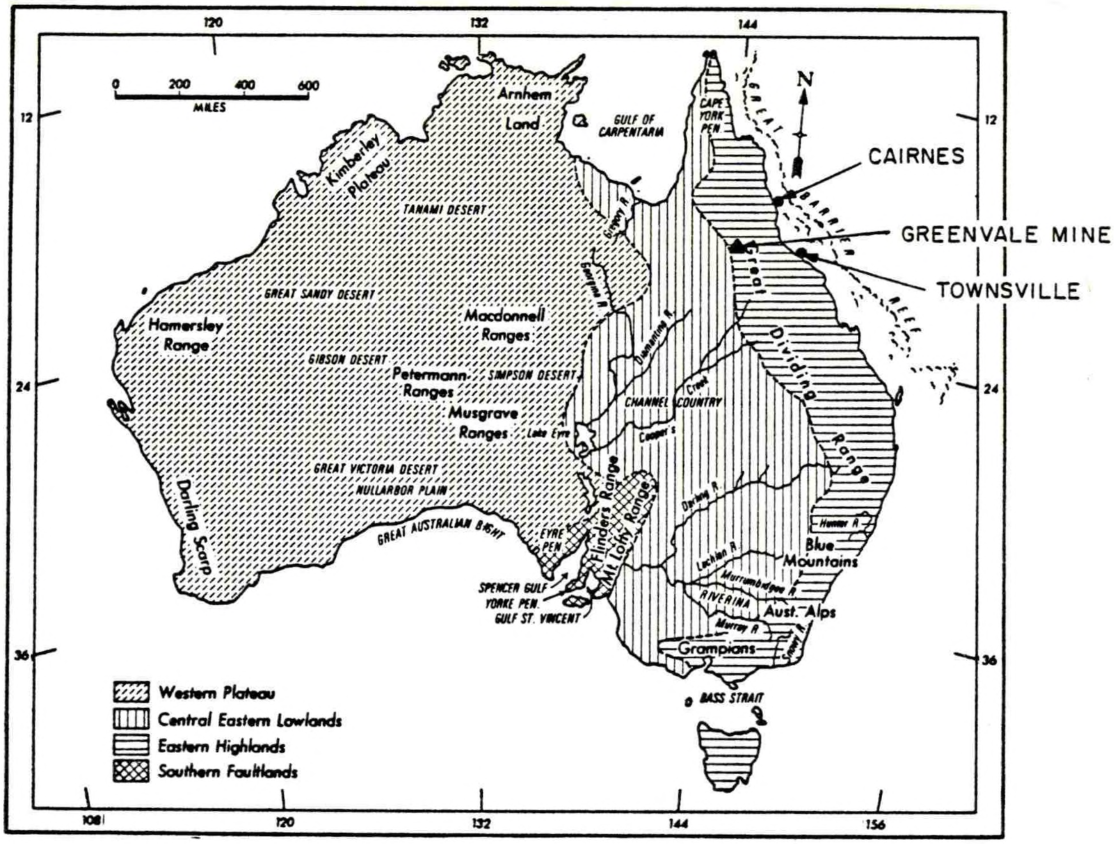
3.2.2 Greenvale, Australia

Australia is the fourth largest source of nickel in the world (Mohide and others, 1977). Greenvale in the state of Queensland near the northeast coast (Figure 3-9) produces from laterites; all other deposits are sulfides from the state of Western Australia. Mine production in 1976 reached 75,400 metric tons (mt) of contained nickel, and refinery production was 46,000 mt.

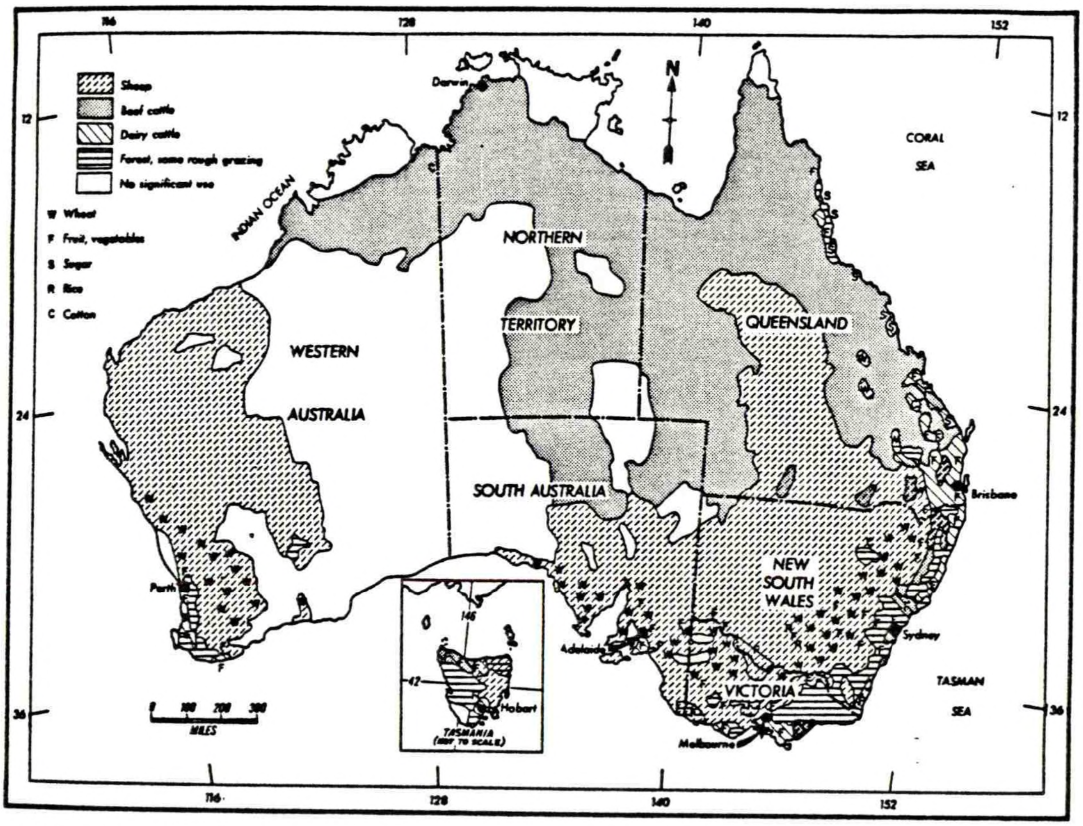
3.2.2.1 General Setting

The state of Queensland can be divided into two major physiographic regions: the Eastern Highlands or Great Dividing Range and the Central Eastern Lowlands (Figure 3-9). The Greenvale mine is within the Great Dividing Range area, with drainage flowing to the east to the

PHYSICAL FEATURES



LAND USE



Source: Whitaker and others, 1974.

Figure 3-9. Physical Features and Land Use - Australia.

coast. The Great Barrier Reef lies offshore of the coast of Queensland (Whitaker and others, 1974).

The Great Dividing Range has an average elevation of less than 3000 feet. Near Cairnes the highest peaks exceed 5000 feet. The Coastal Plains, which vary greatly in width and border the range on the east contain many small streams which originate in the highlands.

In general the climate is warm and temperate. The east is an area of abundant rainfall usually well distributed throughout the year. In Queensland, the northern portion is tropical, with only slight temperature differences throughout the year and with higher rainfall amounts in the summer months. Farther to the south the climate changes, with greater seasonal differences and a change to wet winter months. To the west of the humid coastal zone is an intermediate zone of 30 to 50 inches rainfall and more pronounced temperature changes including hot summer months.

Subhumid soils include black earths which occur on either side of the Great Dividing Range and are fertile, supporting the best agricultural development. In some areas of Queensland as well as in the southwestern part of the country lateritic soils have developed; these have low fertility, with a hard ironstone pan which inhibits most growth (Whitaker and others, 1974).

Vegetation types vary greatly across the continent. Along the East Coast there are relatively narrow strips of dense forest. Inland from this zone in Queensland is an area of more open forests. In the Central Lowlands of Queensland, savanna type vegetation predominates. In the coastal, dense, tropical rain forests, a great variety of trees, primarily of Malaysian origin, predominate. In the more temperate rain forests, beach and indigenous conifers dominate. In the northeast, open forest is widespread and the understory of grasses provides for good grazing.

Because of its isolation from other biotic communities Australia developed many unusual life forms. Of about 230 species of mammals native to the continent nearly half are marsupials. There are 520 endemic species of birds, numerous reptiles, and more than 2200 species of fish. Many native species are threatened or endangered by interference from man, introduction of exotic species from other continents, and conflicts over land use.

Concern has been growing through the 1970s with regard to environmental health and environmental protection. A national water program was established prior to 1975. A subsequent change in government has modified this program - but the general policy is to evaluate water management, assess supplies, conduct research, and educate (Cameron and Klaassen, 1977). Increasing environmental interest has been reflected in concerns regarding air quality as well (Whitaker and others, 1974).

3.2.2.2 The Mine and Processing Facilities

The Greenvale mine is 140 miles inland from the refinery which is at Yabula near the port of Townsville (see Figure 3-8). Ore reserves are estimated to be 40 million metric tons averaging 1.57 percent nickel and 0.12 percent cobalt. The refinery at Yabula has a yearly capacity of 24,000 metric tons of nickel (21,000 tons nickel in 90 percent nickel oxide and 3,000 tons in nickel cobalt sulfides (Mohide and others, 1977).

The mine is a joint venture of the Australian Company Metals Exploration Limited and the Freeport Minerals Company. The deposit was discovered in 1967; initial processing of ore at the Yabula refinery began in 1974, but full production was not achieved until 1976 (Mohide and others, 1977).

The Greenvale ore is a laterite derived from oxidation of an ultramafic Precambrian serpentinized peridotite by percolating ground water. The highly fractured nature of the peridotite allowed easy access for the ground water. The laterites have developed in a terraced pattern as a result of progressive lowering of the water table. The older profiles have been partially eroded and the metal concentrations have been reworked. The laterites are overlain with 15 meters of alluvial clay and lateritic detritus. The deposit occupys an area of 3.3 square kilometers and has an average thickness of 8 meters; in some localities the ore body reaches a thickness of as much as 25 meters.

Mining is by an open cast operation. Several characteristics of the ore body require special consideration: the top and bottom surface are undulating; there is an inverse correlation between rock compentency and nickel content; and the ease of handling varies with the moisture content. The overburden is removed by scrapers, the ore is blasted, and is excavated by two 6 cubic meter draglines with 30 meter booms and is removed by a fleet of six 35 ton trucks for rail transportation to the smelter (Reid, 1979).

At the smelter the ore is processed using the Caron process (see Part II for a discussion). The refinery requires 30,000,000 liter/day of water. Supplies are obtained from 22 wells in gravel beds in the Black River Basin about 50 km from the plant.

3.2.2.3 Environmental Impacts

As noted in Section 3.2.2.1, much of Australia's wildlife is unique and many species would be considered rare or endangered in more environmentally concerned areas such as the United States. Within this area of Queensland, the duckbilled platypus, which occupies the eastern and southern water courses (Whitaker and others, 1974) is one of three known species of primitive egg-laying mammals. Destruction of species, such as kangaroos, has been considered as necessary for protection of range for cattle and sheep, and regulation has met with lack of support. Regulation of land use has also met with general opposition. Clearing, overgrazing, and some mining operations have also affected stability of soils. There are indications that wind erosion, silting of streams, and increased flooding are problems which are of increasing concern.

Applying the unit parameters for nickel-laterite surface mines in Chapter 2.0 to the Greenvale mine and assuming that all the probable reserves are mined (40 million tons of ore averaging 1.57 percent nickel and 0.12 percent cobalt), resource use would be as follows: land for mining and processing facilities, 1500 acres (70 percent for mining and 30 percent for processing facilities); energy for mining and processing, 0.4 quad (97 percent for processing); water, 1.1×10^{10} gallons for processing. Employment over the life of the mine for both mine and processing facilities would total 42,000 man years.

Fugitive dust emissions resulting from open pit mining at Greenvale are not expected to be very great when compared to other mines of similar capacity. Initially the overburden is removed with large scrapers and some dust is expected to result from this part of the process. Once the overburden is removed, the ore is blasted loose and removed by draglines. The ore is then hauled from the pit to a stockpile where it is crushed, screened, and loaded onto railcars for eventual transportation to the Yabula facility. Fugitive dust from these operations is expected to be significantly less than for many other similarly sized mines due to the high water content (20 to 45 percent by weight) in the ore.

At the Yabula processing facility, the moist ore is passed through a series of rotary kilns at a rate of up to 300 tons/hr. Normally two kilns are used for this purpose; however a third kiln is used for ore with an unusually high moisture content. The kilns are fired by diesel fuel, the combustion of which will result in a nominal quantity of SO_2 emissions. The magnitude of these emissions should not be unlike those from a similarly sized industrial furnace and no significant adverse effects on air quality are expected from them. The exhaust gases from these kilns are passed through a set of cyclones and a series of four parallel electrostatic precipitators. Therefore, particulate emissions from the roasting operation should be minimal. After the ore is dried it undergoes further crushing and grinding in preparation for final processing (Reid, 1979).

Ore reduction is accomplished using multiple hearth roasters where the oxide ore is reduced to its metallic state. Heavy fuel oil is mixed with the ore mixture to increase the efficiency of this process.

As with the roasting kilns, SO_2 emission from ore reduction processing will occur primarily due to the combustion of fuel oil. The emissions should not give rise to any major adverse effects on air quality.

Once the ore is reduced to its metallic state, it undergoes "hydrometallurgical" processing, which is basically a chemical process. Conversations with company officials (Collins, 1980) indicate that SO_2 emissions associated with hydrometallurgical ore processing should be almost non-existent.

As of the mid-1970s in Australia, the concept of environmental conservation was still looked upon generally as at least a questionable interference with progress. Mining laws were archaic; conflicting land uses lacked any priority over mineral extraction rights; and concerns about water and air quality and reclamation were minimal (Springell, 1975). Future trends toward inceased regulation are uncertain.

3.3 COBALT

Cobalt is recovered primarily as a by-product of mining and processing other metals, especially copper and nickel. Considered here are the cobalt deposits associated with the stratabound copper of Zaire, the nickel sulfides of Sudbury, Ontario, and the nickel laterites of Greenvale, Australia. Information on the economic geology of cobalt has been presented in Part I and on general mining and processing methods in Part II.

3.3.1 Kolwezi Area, Zaire

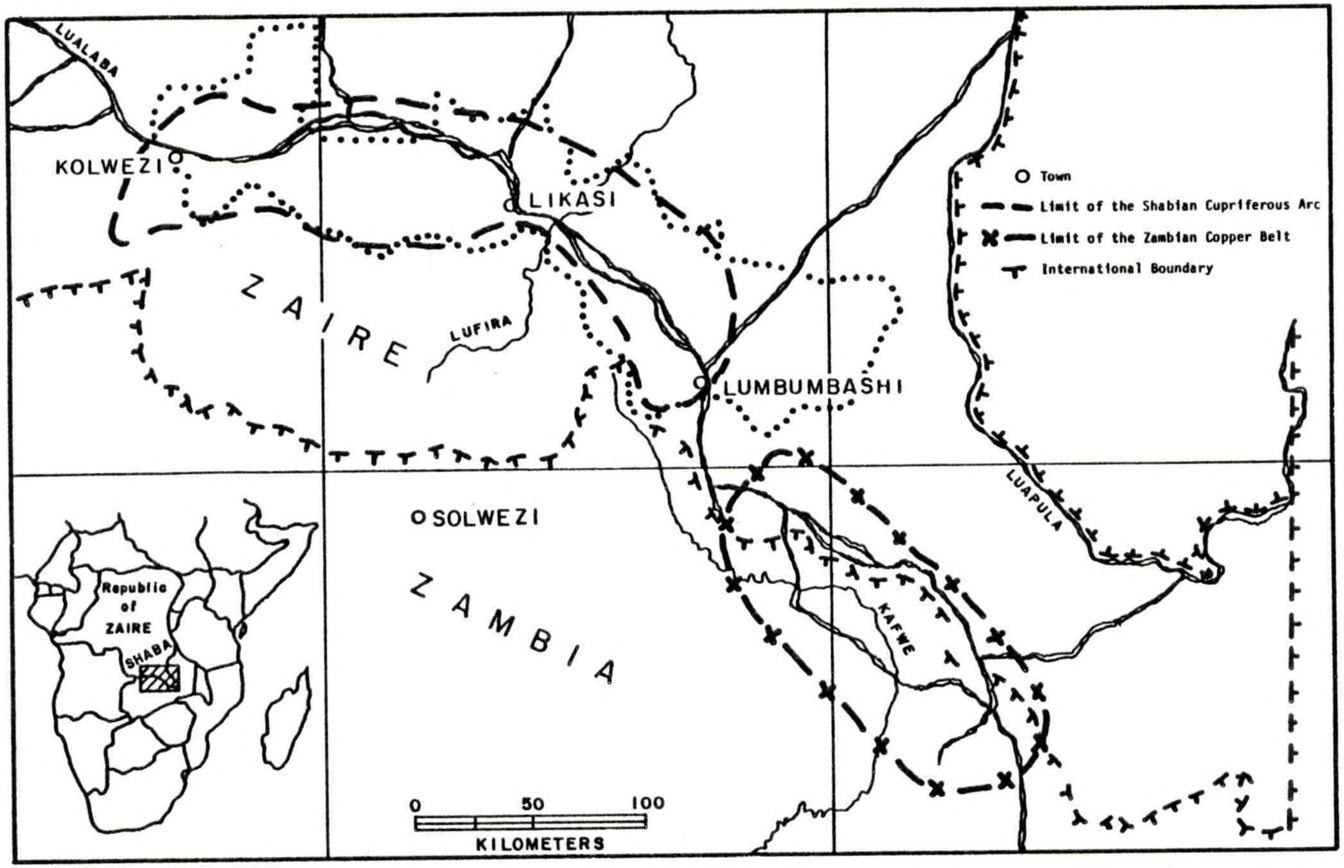
The Kolwezi Area is in the West Sector of the Zairian Copper belt (see Figure 3-10). Within just a few miles of Kolwezi there are six open pit mines and one underground mine which currently produce 11 million metric tons per year of ore. At the open pit mines copper grades average 4.3 percent, and at the underground mine 4.2 percent. Cobalt grades vary in the open pits, but average 0.35 percent at the underground mine. The mines and concentrators are operated by the state company Generale de Carrieres et des Mines (Gecamines) in the State of Shaba.

3.3.1.1 General Setting

The Republic of Zaire in south-central Africa includes the greater part of the Zaire River Basin (formerly the Congo). The country's only connection to the Atlantic Ocean is a narrow strip of land on the north bank of the Zaire River along the south Atlantic coast.

The vast, low-lying central area is a basin-shaped plateau sloping toward the west and covered by tropical rain forest. This area is surrounded by mountainous terraces in the west, plateaus merging into savannas in the south and southwest, and dense grasslands extending toward the Zaire River in the northwest. High mountains are found in the extreme eastern region.

Zaire lies on the Equator, with one-third of the country to the north and two-thirds to the south. The area is hot and humid. In the



Source: Kruszona and others, 1978.

Figure 3-10. The Copperbelt of Zaire and Zambia.

south, the rainy season lasts from October to May, and in the north it lasts from April to November. In the central region rainfall is more or less regular throughout the year. In the south, in the October-May wet season the storms often are violent but seldom last more than a few hours. The average annual rainfall for the entire country is about 42 inches.

The main copper-cobalt mining area is in the south part of the country in the "boot" which projects into Zambia. Ore deposits can be differentiated into three sectors - West, Middle, and East Sectors - which together stretch for 300 km in a curve from Kolwezi in the northwest to south of Lubumbashi (Figure 3-10).

The West Sector covers an area 12 km wide and 25 km long, trending east-northeast. The rocks are part of Serie des Mines sequence of the Roan rocks of Precambrian age. They are predominantly carbonate and fine clastics (Kruszona and others, 1978). The ore minerals occur as impregnations of the Precambrian carbonates and clastics and probably originated from basic magmas along fracture zones. In this area the Serie des Mines has been severely deformed, consisting of flanks of sheared folds. The copper-cobalt ores in the Kolwezi area are in two complexly folded and faulted sedimentary units. The important copper and cobalt minerals, are chalcopyrite, bornite, linneaite and/or carrollite. In general the ore minerals are evenly distributed in the ore bearing beds. In oxidation zones, heterogenite and asbolane are secondary cobalt minerals (Kruszona and others, 1978).

3.3.1.2 Mines and Processing Facilities

The six surface mines and one underground mine in the vicinity of Kolwezi account for 75 percent of Gecamines production. Of the 11 million metric tons of ore produced in the West Sector, about 8 million tons was from the surface mines; this ore yielded nearly 11,000 tons of cobalt. In addition to the mines, there are several processing facilities in the area around Kolwezi. There is a concentrator at Kolwezi, and a new one is scheduled to come on stream in 1980 at Dima. There is also a concentrator at Kamato and a wash plant at Mutoshi (White, 1979b).

Surface Mines

Two of the surface mines, Dikuluwe and Mashamba, as well as the concentrator at Dima, have been developed as part of a project "P2" which is designed to increase Zairian production. The other surface mines are Kamoto East, Kamoto North, Musonoi, and Mutoshi.

Development of the surface mines has resulted in the removal of some 24 million metric tons of overburden. The ore is either hauled directly to the concentrators or stockpiled for later haulage to a concentrator.

In general, the surface mines have been able to meet production schedules, but removal of overburden has been a problem since 1975, primarily as a result of fuel shortages, lack of parts, and lack of trained personnel (White, 1979b).

The pits are below the water table in some areas, and considerable pumping is required, especially during the rainy season from November to April. At times during the rainy season, the surface mines are forced to close because of flooding.

Underground Mine

The underground mine at Kamoto is highly mechanized and modern, perhaps one of the cleanest mines in the world. Most of the production is from sublevel caving, with the remainder from longitudinal and transverse cut-and-fill stoping.

The ore occurs at two horizons 10 to 12 meters thick, separated by a non-productive zone 12 to 14 meters thick. Sublevels are currently developed at 10-meter vertical intervals. From the bottom of the pit at 165 meters to a depth of 425 to 450 meters, the ore zones strike east-west, and dips vary from moderate to steep. Studies indicate that below 450 meters the ore beds feather out and that room and pillar methods may be more effective.

The underground mine employes 1800, of whom 90 are supervisory personnel; of these trained personnel 70 are from Zaire and 20 are from other countries.

Processing Facilities

As noted previously, the ores from the seven mines are concentrated at the wash plant at Mutoshi and at the flotation plants at Kolwezi and Kamoto. The new concentration plant at Dima is scheduled to begin operation in 1980 (White, 1979b).

At Mutoshi, the ore is crushed and washed and passed through screening, jigging, and heavy media separation processes. The plant has a capacity of 2.5 million metric tons per year of ore; the concentrate averages 25 to 27 percent copper. The concentrates are sent to Lubumbashi (a pyrometallurgical plant) or to Panda (an electric funace).

The concentrator at Kolwezi has been in operation since 1941; three processing lines can handle 2.8 million metric tons of oxide ores and two can process 1.5 million metric tons of oxide ores or 2 million metric tons of oxide-sulfide ores. The principal steps at Kolwezi, and at Kamoto and Dima are crushing, wet grinding through rod and ball mills, thickening, and filtation. At the newer plants, processing steps have been simplified and more efficient equipment is in use. High copper concentrates grade to 63 percent copper and 2 percent cobalt, and low copper concentrates grade 38 percent copper and up to 3 percent cobalt (White, 1979b).

Concentrates from the Kamoto concentrator are piped as slurry to the Luilu metallurgical plants (a distance of about 5 km).

Transportation

The mines in the Kolwezi area are about 1800 km by rail from the port of Lobito, Angola, on the Atlantic. This railway was closed in 1975 requiring that ores be transported over the much longer route via Zambia, Rhodesia, and Botswana to the ports of East London and Duban in

South Africa. As part of the repeated disputes with Angola, Kolwezi itself has been captured at times by troops from Angola, leading to a disruption not only of transportation but of mining and smelting activities (Mining Annual Review, 1978).

3.3.1.3 Environmental Impacts

The copper-cobalt belt of Zaire stretches from the area of Kolwezi to Lubumbashi and from there in an arc into neighboring Zambia. Zaire has not been as stable as Zambia and has been less concerned about environmental effects than has Zambia. Problems in the two countries can, however, be expected to be similar.

In Zambia, growth of the mining industry has been accompanied by development of secondary industries and of new population centers scattered along the rail line which is the transportation line for the processed ores (Reilly, 1975). This development has included heavy industry, refining, automobile assembly, and extensive manufacturing. The added population, primarily of unskilled workers, has not been absorbed into the mining and other industries and has caused development of migrant worker camps.

The mining industry accounts for 90 percent of Zambia's revenue. Copper companies have provided funds for education, medical services, industrial and agricultural develoment, transportation, and defense.

When mining began in Zambia much of the area was unsettled and there was little concern for the environment; overburden and tailings were dumped across the countryside. Dumps and waste ponds were constructed without concern for water pollution. By the early 1970s concern for the environment was rising and at the same time profits from mining were decreasing. At the UN conference on the Human Environment at Stockholm in 1972, Zambia reported on its steps toward environmental protection - steps which, however, diverted needed capital from other areas into non-productive pollution control and environmental preservation (Reilly, 1975).

In the copper belt of Zaire and Zambia the main concerns are the sulfur dioxide emissions from smelters and the waste disposal from the surface mines and mills. During the wet season, torrential rains can cause dams to break and a slurry of quicksand-like waste to flow out onto farmlands. In the dry season, even slight winds produce dust storms from the waste piles causing pollution problems for nearby residents.

In Zambia it was, until recently, more economical to import sulfuric acid from outside the country than to recover it from the sulfur dioxide from the smelters (Reilly, 1975). New equipment will recover 90 percent of the SO_2. It is uncertain whether Zaire will also install sulfuric acid plants.

Sludges of unrecovered heavy metals, including cobalt and cadmium from electolytic furnaces, are highly toxic and in Zambia the outflow of sludge ponds is carefully monitored.

At Kolwezi fugitive dust emissions are expected to be similar to those from similar size open pit mines, with the possible exception of the Greenvale mine in Australia where the ore has a relatively high moisture content. Dust emissions associated with crushing, grinding, and other mine-related activities should be typical of those described in Section 1.2.3.

The primary method of obtaining cobalt from the concentrated mineral is through hydrometallurgical processing. Oxide concentrates are leached directly with sulfuric acid. No significant sulfur dioxide emissions occur during this process. Sulfide concentrates are roasted to convert the sulfides to soluble sulfates and then leached. SO_2 emissions occur during the processing of sulfides although these are only part of the total ore processed and thus SO_2 emissions should be significantly less than for a comparably sized system of smelters processing only sulfide ores.

3.3.2 Sudbury, Ontario

Cobalt is recovered as by-product of mining nickel ores at Sudbury, Ontario. The ores there average 0.07 percent cobalt. Nickel content of these ores is 1.5 percent, and the nickel-to-cobalt ratio varies from 20:1 to 33:1. In 1976, Canada produced 1373 metric tons of cobalt in conjunction with nickel production. Baseline conditions and impacts of nickel mining and processing are described for Sudbury in Section 3.2.1. Only a small amount of additional land, energy, SO_2 emissions, and employment occur as a result of cobalt mining and processing.

3.3.3 Greenvale, Australia

As noted in Section 3.2.2, the Greenvale ores contain 1.57 percent nickel and 0.12 percent cobalt. The metals are mined together and are processed at Yabula to a black compacted powder of nickel-cobalt sulfide, containing 13 to 16 percent cobalt, 38 to 40 percent nickel, and 35 to 37 percent sulfur. The nickel-cobalt sulfide is sold to other facilities in Japan for refining. Baseline conditions and environmental impacts are described for nickel in Section 3.2.2. Since cobalt is refined elsewhere, only the small amount of shipping activity has potential for environmental impact in Australia.

3.4 MANGANESE

Most of the world's reserves and proven resources of manganese are found in sedimentary deposits - as strataform oxide deposits (Part I). These occur in several areas of the world. The largest deposits are in Russia, with other important resources located in the Kalahari Desert of South Africa, and in the Urucum area of Brazil. Weathered carbonate deposits are important souces in Ghana, Brazil, Zaire, and Gabon; and in Hidalgo, Mexico, unweathered carbonate deposits form an important deposit (see Part I). In the U.S., production of manganese has been spotty. There is at present no commercial producton of manganese in the U.S., although some is produced as ferromanganese. For the present study the new large underground mine at Middelplaats, South Africa, and mines at Hidalgo, Mexico, and Urucum, Brazil, are described.

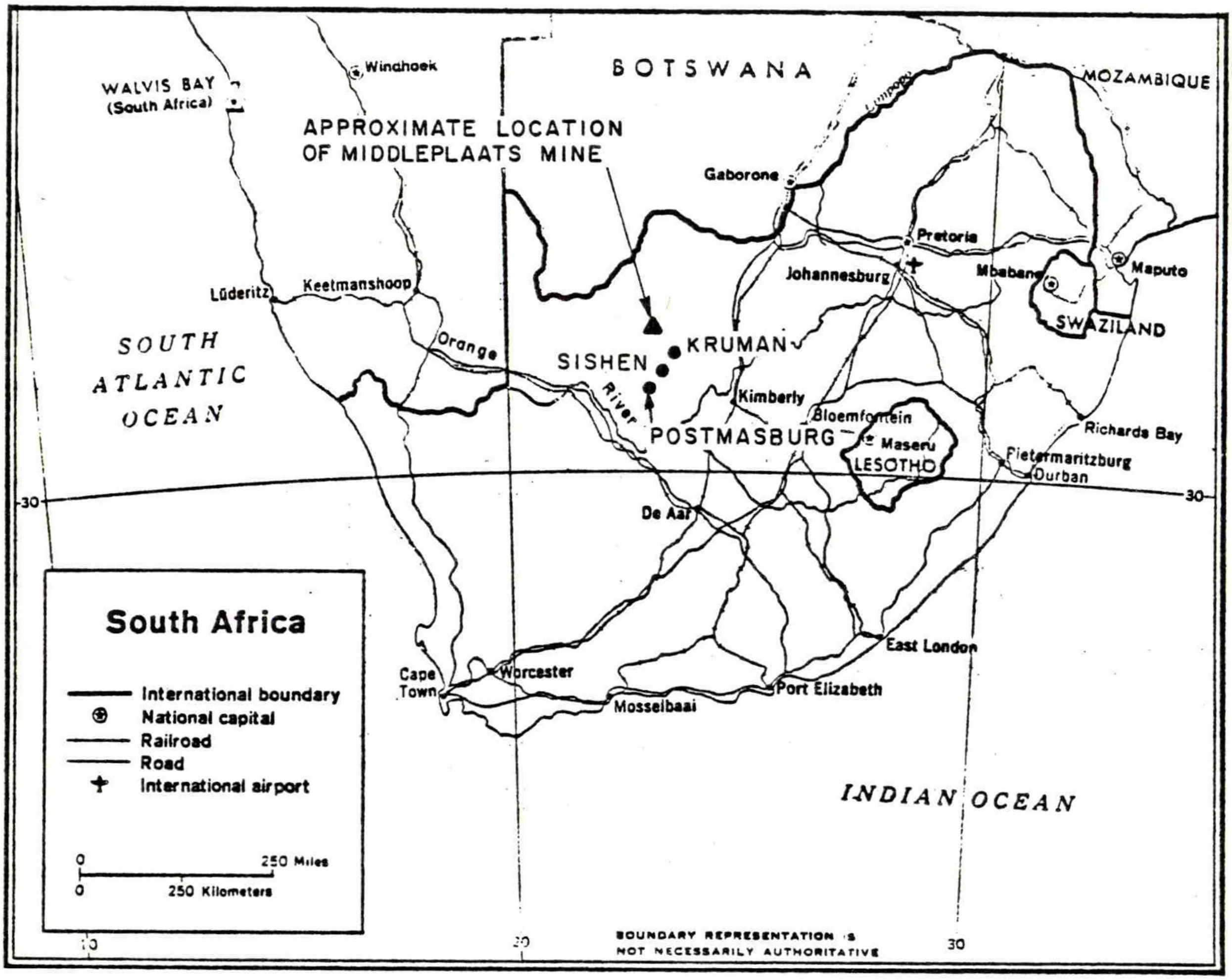
3.4.1 Middelplaats, South Africa

The Middelplaats mine is a large underground mine in the Kalahari manganese field of Cape Province, South Africa (Figure 3-11). The mine, which is operated by the Anglo American Corporation of South Africa Ltd., was opened in January 1979 and is expected to add one million tons to South Africa's already large annual manganese production capacity.

3.4.1.1 General Setting

South Africa ranks second only to the USSR in manganese production. Occurrences are widespread, but three areas have the most important reserves: the Postmasburg Field in northern Cape Province, the Kalahari Field, which is an extension of the Postmasburg Field, and deposits in the Dolomite Series near the Witwatersrand gold fields near Johannesburg.

The Kalahari Field is north of Sishen and west-northwest of Kuruman (Figure 3-11). Ore occurs in a banded ironstone formation in a sedimentary basin with a north-south axis about 25 miles in length and an east-west axis 9 miles wide. The area is in the southern portion of the Kalahari Desert. The climate is semiarid, annual rainfall averages



Source: U.S. Dept. State, 1977.

Figure 3-11. South Africa and Middelplaats Mine Area

15 inches per year. There is no surface water and the two main rivers, tributaries to the Kuruman, are dry most of the year. The area is about 3000 feet above sea level (Wilson and Dunn, 1978).

3.4.1.2 The Mine and Processing Facilities

The ore-bearing unit is the Gamagara Formation of the Upper Griquatown banded ironstones which are at the top of the Pretoria Series of the Precambrian Transvaal System. Manganese bearing beds are interbedded with banded ironstone, and both rock types are cut by diabase-like sills and dikes. The ore bodies are strataform, continuous or lenticular, distinctly layered, and sometimes folded conformably with the enclosing ironstone units. Strata immediately above and below the main ore zones are calcareous and, in places, laminated (Wilson and Dunn, 1978).

The deposits are believed to have formed contemporaneously with deposition of enclosing rocks and to have been subjected to supergene enrichment and replacement at a later date. The mineable zones are confined primarily to near-surface beds. Reserves in the field have been estimated at 9 billion tons of 30 percent manganese; some ore bodies have as much as 60 to 62 percent manganese. Many ore minerals are present, different suites predominating in different layers. Within the Kalahari field the principal ore minerals are braunite, bixbyite, rhodochrosite, hausmammite, and jacobsite (Wilson and Dunn, 1978).

The ore body is at a depth of 985 to 1640 feet; it dips at about 10° to the northwest. The ore, a mixture of manganese oxides and silicates, is generally competent. The shaft is 1600 feet in depth; a ramp which also provides access to the mine is 8200 feet long, inclined at 10° to 12°. The main haulage level is 1312 feet below ground surface. The ore is extracted to widths up to 46 feet, in 2 or 3 cuts, mining is by room and pillar method (World Mining, 1979).

The hoist in the shaft is capable of lifting 1.2 million tons of ore/year. The ore is crushed in the primary crusher and is then lifted to the surface, transferred to a 600-ton capacity bin and from there by conveyor belt to a secondary crushing mill (to a maximum size of 75 mm). After secondary crushing, the ore is washed and screened to two sizes (more than 6 mm; and between 6 mm and 2 mm). Further processing produces different sized fractions which are transported to stockpiles from which the ore can be loaded on rail cars to be sent to Postmasburg and Kimberly and on to Port Elizabeth (Figure 3-11).

3.4.1.3 Environmental Impacts

In general, the government has been anxious to foster economic growth and has taken a largely regulatory role (with regard to health and safety considerations, labor, and legislative matters). Because of the concerns over economic independence and the lack of competing land uses, there is apparently little environmental concern.

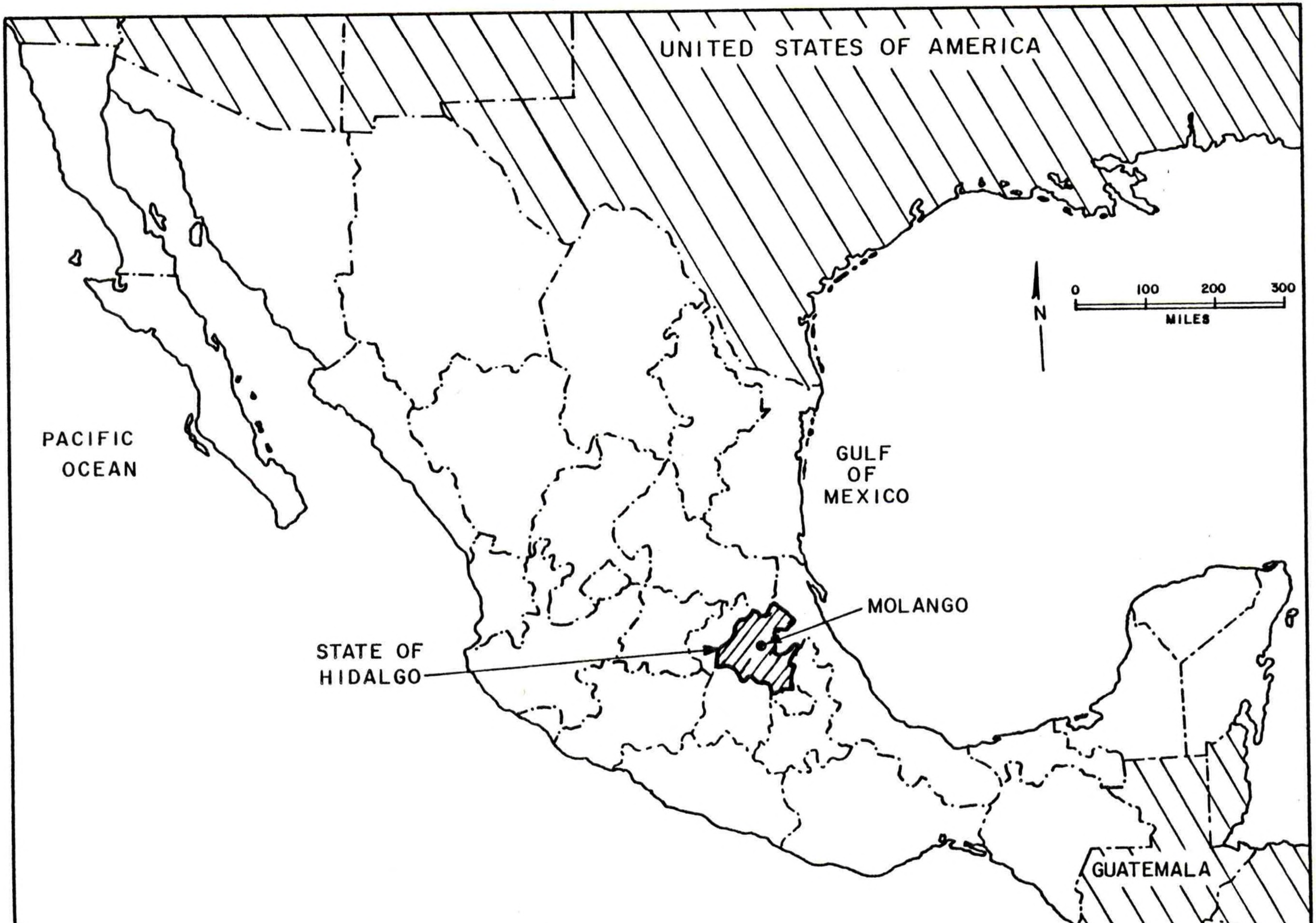
Though the quantities of ore to be removed are potentially huge, there is little potential for subsidence. There is some energy and water usage required to crush, wash, and screen the ore. Wastes are very low in volume because no chemical concentration is required. There is no SO_2 emission, though some dust is generated at the processing plant. The very small resource usage involved in manganese mining and processing is a factor of the high concentration of commercial grade ore in the rock as indicated in information presented in Chapter 2.0.

3.4.2 Molango, Mexico

Manganese ore averaging 27 percent manganese is obtained from a combination surface/underground mine in the state of Hidalgo, Mexico, north of the town of Molango. The site is some 100 miles north-northeast of Mexico City and 120 miles southwest of Tampico (Figure 3-12).

3.4.2.1 General Setting

The region of Hidalgo is mountainous and semitropical. The area is in the physiographic province of the Sierra Madre Oriental. Average elevation is about 3000 feet above sea level. The nearest town is Molango which is about 30 km south of the mine site. The population of



Source: Weil and others, 1975a.

Figure 3-12. States of Mexico Showing General Location of Molango.

Hidalgo is under 10,000. The site is connected by highways to both Mexico City and Tampico.

In the period 1969-1978 rainfall in the area ranged from 10.5 cm to 40.6 cm per year and the median temperature from 18°C to 28.8°C.

3.4.2.2 The Mine and Processing Facilities

The mine is owned by Copania Minera Autlan S. A. de C. V. and produces from unweathered carbonate deposits (see Part I). Production is from both surface and underground facilities. The surface mine produces 384,000 metric tons of ore per year, and the underground mine 391,000 metric tons. The surface mine currently occupies 185 acres, the plant some 62 acres, and housing an additional 37 acres.

The open pit mine has been developed by a truck-shovel system with a stripping ratio of 12:1. The underground mine utilizes sublevel stoping.

The ore deposit crops out over a distance of 30 miles and is 200 feet in average thickness. Proven reserves are 21.3 million metric tons; probable reserves are 200 million metric tons; and possible reserves are 1 billion metric tons. The proven reserves should last for another 15 or more years at current production rates.

3.4.2.3 Environmental Impacts

Lumbering activities have tended to be destructive and in certain states, including Hidalgo, no timber at all may be cut. Any clearing of land, therefore, for expanded surface mining would increase problems associated with loss of vegetative cover.

Water requirements are minimal and little adverse environmental impact is anticipated.

3.4.3 Urucum, Brazil

Manganese deposits have been known for more than 100 years in the area of Urucum, Brazil, in the state of Matto Grosso in southwestern

Brazil (Figure 3-13). Difficulties of transportation and changing economic and political environments have interrupted mining of the deposit in the past.

3.4.3.1 General Setting

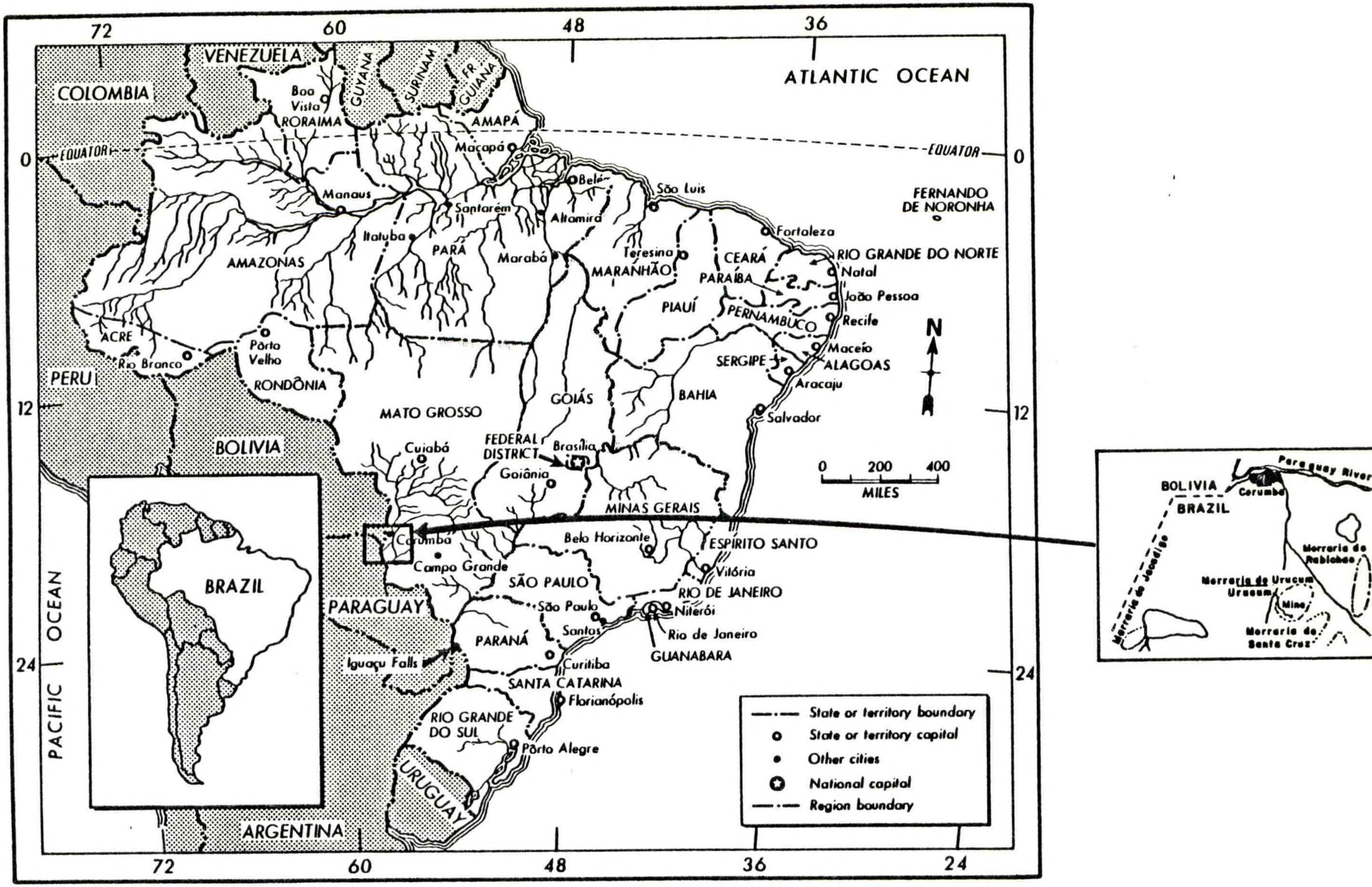
The Urucum manganese deposit is 18 miles south of the town of Corumba in the state of Matto Grosso near the Brazil/Bolivia border (Figure 3-13). This area is in the Central-West area of Brazil which includes the states of Matto Grosso and Goias and the Federal District.

The area surrounding Corumba, the Pantanal, is a swampy lowland. Most of the Central-West, however, is in the Central Highland, with the extreme northern portion in the Amazon Basin.

The Pantanal is largely swamp and marsh with an average elevation of 500 feet. Abundant rainfall produces many streams which form the headwaters of the Paraguay River. The Paraguay flows to the south to join the Parana River which flows into the Atlantic at Buenos Aires. Flooding from the many streams in the Pantanal has produced a rich soil which provides good agricultural land. The area is too wet for forest growth except for patches on higher ground. There is a long dry season with lower temperatures than those of the Amazon lowland to the north and grasslands are abundant, providing excellent forage for cattle (Weil and others, 1975b).

The region around the Pantanal is part of the Central Highlands. The Urucum ores crop out on the western face of one of four large mesalike mountains of sedimentary rocks in the area (Argall, 1979). The ores are found in sediments overlying Precambrian rocks which are exposed west of the mine.

The Rio de la Plata basin comprises three main streams - the Aruguay, the Paraguay and the Parana. The mine site is near the Paraguay River which rises near Cuiaba in Mato Grosso and flows south into Paraguay and Argentina. The Parana River rises in Goias, flows south across the Central Highlands to form part of the boundary with Paraguay



Sources: Weil and others, 1975b; Argall, 1979.

Figure 3-13. Location of Urucum Manganese Deposits.

and Argentina. The two rivers join near the city of Corrientes, Argentina, and flow through Argentina to Buenos Aires where they enter the Rio de la Plata estuary.

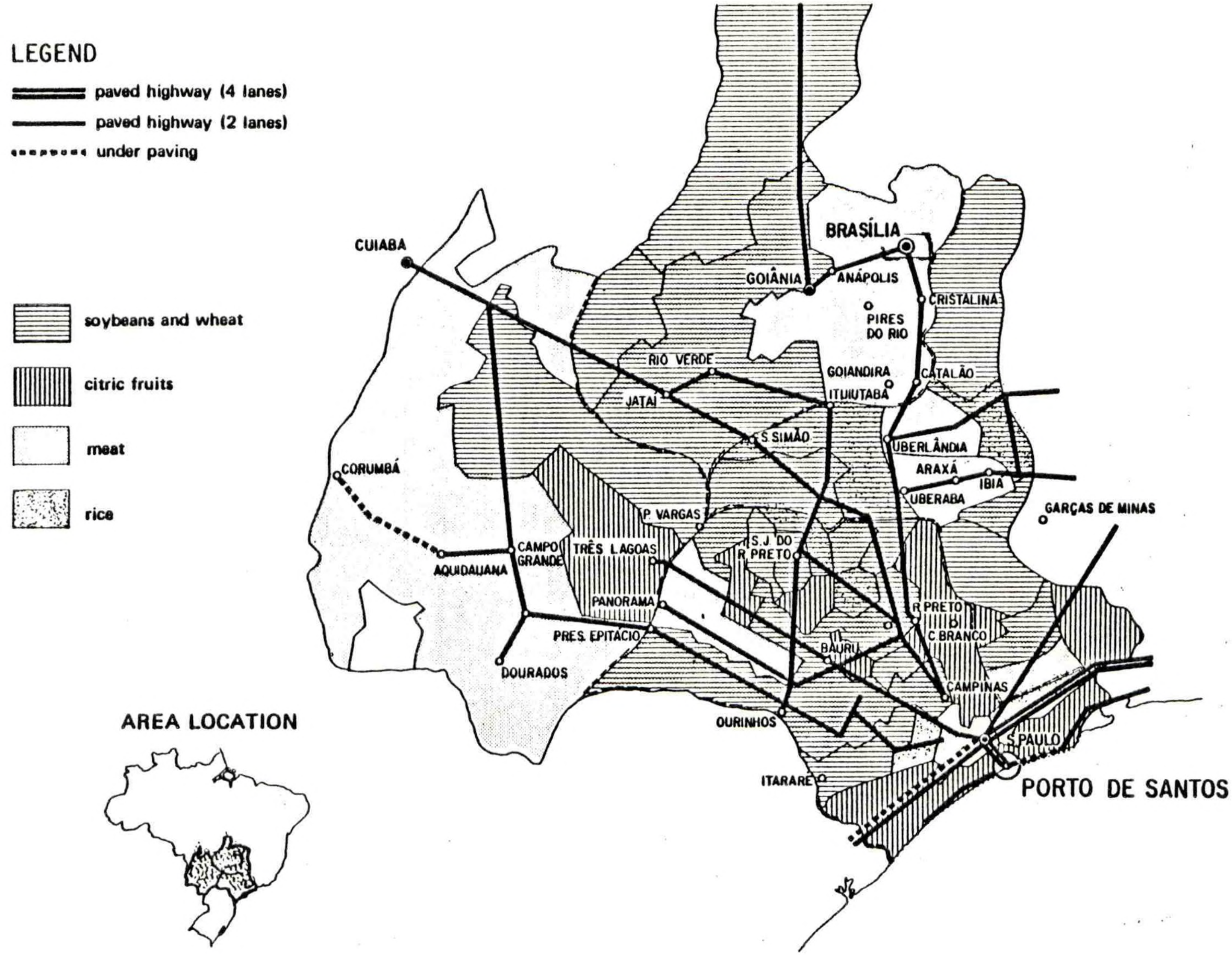
Wildlife is abundant and varied throughout much of the sparsely inhabited Amazon jungles and in the interior. In the Central-West there are jaguars, puma, wild boars, ocelots, and armadillos.

The Brazilian government embarked on a program of highway development in the early 1970s. Corumba is connected by gravel road to Aquidauana and from there by paved highway to the Port of Santos (Figure 3-14). In addition a rail line connects the two cities. The rivers also provide transportation for ores and other products.

3.4.3.2 The Mine and Processing Facilities

The mine at Urucum is underground. It is being developed by room and pillar method back from the face of the outcrop on the western face of the Morrarra do Urucum Mesa. The ores are found at the contact of two sedimentary units - the Camada No. 1 manganese bed at an elevation of 700 meters and within the upper unit - the Camada No. 2 manganese bed which crops out 40 meters above the Camada No. 1. The No. 1 bed is the ore being mined in Brazil. The No. 2 bed has been mined in Bolivia. The ore bodies strike east and dip 12°N. The average thickness of the Camada No. 1 is 3 meters. The mine is developed through parallel adits driven 1400 meters east along strike. Rooms 60 meters long are turned off the adits at right angles every 15 meters. The ore is blasted, loaded on 1.7 ton cars and hauled to the surface where it is dumped into truck bins. With present equipment maximum annual capacity is 250,000 tons.

The ore is transported by truck to Lodario just downstream from Corumba on the Paraguay River. There it is loaded on steel barges and floated down the Paraguay-Parana Rivers to the coast where it is loaded on ocean-going vessels. Transportation is one of the major problems in development of the ore, and one reason given for interruptions of mining has been low water in times of drought.



Source: Resende, 1973.

Figure 3-14. Highway System and Land Use, Southern Brazil.

The ore is 46 percent manganese; reserves were originally estimated at 32 million tons, but additional exploration in the area (see Figure 3-13) to include the Rabichao site to the northeast and other prospects has increased resource estimates to 100 million tons (Argall, 1979).

3.4.3.3 Environmental Impacts

The mine is developed primarily as an underground mine so there is little impact to the surrounding area. The nearby land uses are primarily for cattle grazing, the area is sparsely populated. Possible spills of ore from barges into the Paraguay-Parana River system could have local adverse impacts on fish life and water quality.

3.5 IMPACTS DUE TO SHORTAGES OF METAL PRODUCTION

Under business as usual conditions, present worldwide reserves of copper, nickel, cobalt, and manganese are sufficient to meet the most likely cumulative demand through the year 2010. However, in certain cases, such as those for cobalt and manganese, mine capacity may not grow at a rate sufficient to meet projected demand without substantial planning and capital investment. In the case of copper and cobalt, presently known world reserves would not be sufficient to meet the high demand projected in Part I of this report. Mine capacity shortfall would be even greater under this scenario. It is to be expected that marginal worldwide supply conditions would place market access constraints on potential U.S. supply sources. However, since the U.S. has sufficient reserves at present to meet even the projected high rate of domestic copper demand, potentially severe supply problems could be expected only in the case of cobalt even at the high demand projections. As prices rise, world and U.S. reserve quantities will also grow, perhaps to a level sufficient to meet demand under all projected scenarios.

4.0 EXAMPLES OF SITES EXPECTED TO SUPPLY U. S. DEMAND ASSUMING NEED FOR NATIONAL SELF-SUFFICIENCY

Should foreign supplies of copper, nickel, cobalt and/or manganese be unavailable, the U.S. would be forced to rely entirely on its own resources or reserves. For copper, this would require extending the life of some existing mines by utilizing lower grade ores, leaching previously stockpiled ore, and developing new mines. For nickel, cobalt, and manganese, resources may have to be developed even if they would not normally be profitable. Characteristics of the mines and processing areas selected as examples for discussion are indicated on Table 3-1. The possibility of shortages is discussed for each metal. Recycling and substitution are reviewed in Section 4.5.

4.1 COPPER

Of the four minerals under discussion, the U.S. is most capable of self-sufficiency with copper. The U.S. has historically been the world leader in copper production and has resources totaling an estimated 427 million short tons (Part I). For the period 1973-1975 the U.S. used an average of 1.65 million short tons per year. Between 1980 and 2010, cumulative U.S. demand is estimated to range from a low of 75.4 million short tons to a high of 122.3 million tons. Present reserves of 107 million tons could theoretically supply the most likely cumulative demand during that period (95.3 million tons). Though U.S. mines presently supply only about 84 percent of national demand (Schroder, 1977), this is more a result of world economics and over-production than a lack of potential mine capacity.

It is assumed here that, since 81 million short tons of U.S. reserves are in porphyry copper and vein and replacement deposits, this will be the major domestic source through the period. The two U.S. mines described in Sections 3.1.1 and 3.1.2, Pinto Valley and San Manuel, Arizona, are considered typical of the mines which would supply the bulk of this copper.

4.1.1 Pinto Valley, Arizona

As discussed in Section 3.1.1, the Pinto Valley mine is an open pit mine in the semiarid southwest in eastern Arizona (see Figure 3-2). Reserve/production estimates indicate that this mine will be depleted by the year 2000. However, the life of the mine might be extended by leaching of lean ore stockpiles, or by mining adjacent lower grade ores. Other mines of similar size and with generally similar baseline conditions and environmental impacts might be developed in the vicinity. It is probable that the smelting facilities at Miami would continue to be used for several reasons, not the least of which is the expense and regulatory difficulty of constructing new smelting capacity.

As discussed in Section 3.1.1, the most significant air quality related problem is associated with SO_2 emissions from smelting. Concern is also rising over water pollution in the Globe-Miami mining area, and land for tailings and waste rock disposal will be increasingly distant and more costly to utilize. If copper production should increase in the area to meet increased U. S. demand (more than doubled by 2010), air, water, and land resource constraints will be even more limiting. Water supply could also be a significant problem, particularly if parallel growth occurs in state population and agricultural production.

As far as air quality is concerned, it is apparent that a doubling of copper production in the Pinto Valley Mine area would not be possible without a significant reduction in SO_2 emissions from the smelting operations. In the event that the same smelter(s) would be utilized to handle the increased production, the most likely course of action would be to remove additional SO_2 from the exhaust gas stream by onsite sulfuric acid production or other means.

4.1.2 San Manuel, Arizona

The San Manuel mine was described in some detail in Section 3.1.3. This is a large underground mine in eastern Arizona (Figure 3-5). Should the U.S. be forced to depend on its own reserves for total copper needs through 2010, production would probably be expanded at this or similar mine/processing facilities elsewhere. As with the Pinto Valley mine, it is expected that air quality and water supply constraints will be the most limiting on any possible expanded mining/smelting activities. Since the San Manuel mine is located in the same general area as the Pinto Valley mine, the same limitations would apply.

4.1.3 Other Copper Supplies

Obviously, other copper porphyries in the southwestern U.S. could be developed if a sufficiently strong demand for copper were present. Capacity growth has been very slow recently because of excessive world production of copper and consequent low prices. If economic conditions warrant, other deposit types could be developed in the U.S. One of the most promising is the Duluth Complex in northern Minnesota. Reserves in the Duluth Complex total an estimated 29 million tons of copper in ore grades greater than 0.5 percent; 8.8 million tons of nickel; and 440,000 tons of cobalt.

Potential environmental impacts of large scale mining and processing of the Duluth Complex are summarized in Section 4.2.1. Only the potential for meeting a significant fraction of the U. S. copper demand is discussed here.

According to recent studies (Minnesota Environmental Quality Board, 1979), a typical 20 million tons (ore) per year open pit development in the Duluth Complex would produce about 85,000 tons/year of copper metal. This constitutes less than 2 percent of the projected annual U.S. demand in 2010. Obviously a substantial number of such mines would be required to supply a large portion of U.S. demand. The long-term potential of this reserve is significant, however.

4.2 NICKEL

As discussed in Part I, nickel is found in three areas in the U.S.: in laterite deposits in California and Oregon; in association with the Mississippi Valley type lead-zinc deposits of Missouri, and in the sulfides of the Duluth Complex in northern Minnesota.

The only nickel mine currently operating in the United States is at Riddle, Oregon. This mine produces some 12,000 to 13,000 tons of ferronickel per year. Other, similar deposits have been identified in Oregon and northern California, including that at Gasquet Mountain, described in Section 4.2.2.

Concentrations of nickel (0.02 percent) and cobalt (0.015 percent) occur with the Mississippi Valley type lead-zinc deposits in Missouri and adjoining states. These deposits are not presently economically recoverable because of the low price of nickel (see Part I).

Copper-nickel-cobalt deposits in the Duluth Complex in north Minnesota contain large quantities of these minerals at average grades of 0.66 percent copper, 0.20 percent nickel, and 0.01 percent cobalt. Environmental and market conditions have to date made development of these deposits infeasible.

U.S. reserves of nickel total an estimated 200,000 short tons; resources are about 15.1 million tons. Cumulative domestic demand projections made in Part I of this report range from a low of 8.4 million tons to a high of 12 million tons, with a most likely demand of 11 million tons. In 1980, the U.S. will produce about 6 percent of its demand; by 2010, production is expected to grow to 59,000 tons, still only about 11 percent of demand. Thus, it is considered virtually impossible for the U.S. to become self-sufficient in nickel during this period.

As examples of potential mining developments should the U.S. need to develop its own reserves of nickel, the Duluth and Gasquet Mountain deposits are discussed in the following sections. Of the total 59,000 tons per year (tpy) production capacity projected for 2010, 15,000 tpy are projected to come from Gasquet Mountain and 30,000 tpy from the Duluth Complex.

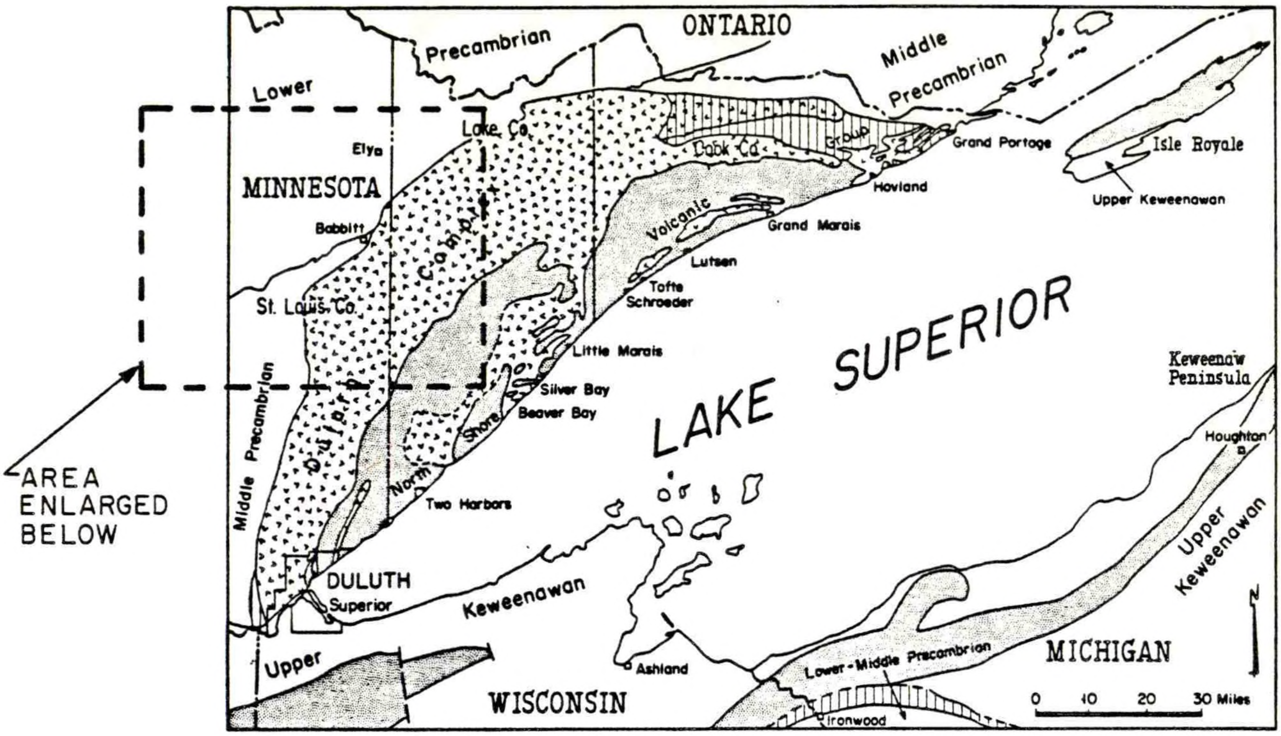
4.2.1 Duluth Complex, Minnesota

The Duluth Complex is a large complex intrusive of Keewenawan (Upper Precambrian) age in northeastern Minnesota. It occupies an area of roughly 2500 square miles stretching as an arc some 150 miles in length from Duluth almost to the Canadian border at Lake Superior (Figure 4-1). Anorthosites, gabbros, and granitic rocks have been intruded along a major unconformity separating the Lower and Middle Precambrian rocks.

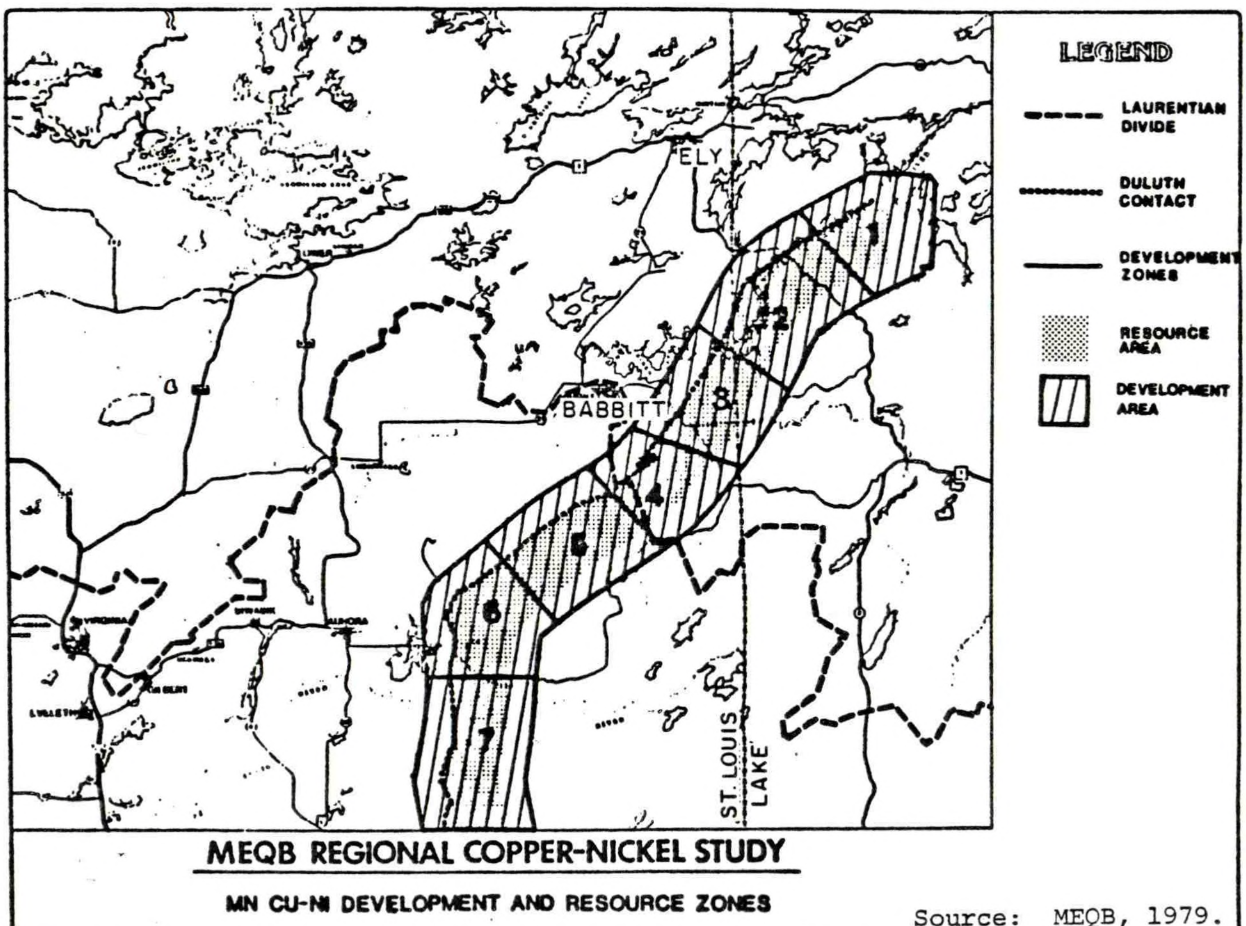
Exploration for copper and nickel began in the early 1950s (Phinney, 1972). The major position of all known copper-nickel mineralization in the complex occurs in the lowermost several hundred feet in what is known as the "basal zone." The copper-nickel mineralization exists as either disseminated or massive sulfides, with the disseminated sulfides of greater significance. Associated with the copper-nickel deposits are cobalt deposits which could be developed in conjunction with any copper-nickel production.

A detailed study of the area was initiated in the middle 1970s by several cooperating state agencies, and a series of reports are now available through the State Planning Agency in Minneapolis (Minnesota Environmental Quality Board, 1979). These studies present the environmental conditions (natural and socioeconomic) in the 2100 square mile regional study area in St. Louis and Lake Counties (Figure 4-1) and impacts which could be expected from developing surface and/or underground mines and processing facilities in the area. Of special concern are any impacts to the Boundary Waters Canoe Area (BWCA) and to other sensitive areas in the Superior National Forest.

The studies indicate rather large, but low grade deposits of copper, nickel, and cobalt: copper 29 million metric tons (MMmt); nickel 9 MMmt; and cobalt 80,000 to 90,000 metric tons.



Source: Green, 1972.



Source: MEQB, 1979.

Figure 4-1. Possible Development Areas of Duluth Complex.

4.2.1.1 General Setting

Minnesota has been glaciated several times in the latest (Quaternary) geologic epoch. Continental ice sheets have scoured the bedrock, deposited glacial till and moraines up to 500 feet in thickness, and altered drainage. The crystalline Precambrian rocks of the northeastern part of the state have also exerted a strong influence on the topography, forming highlands and hills. The climate also affects the landscape. The average annual temperature ranges from 35°F in the extreme northern part of the study area to 40°F in the southern part, and rainfall ranges from 27 to 30 inches per year in the northeastern part of the state.

Land use in the 2100 square mile study region is diversified; 77 percent is forested (57 percent aspen-birch, 30 percent spruce-fir, and 10 percent white, red, and jackpine); 8 percent water; 3 percent mining; 4 percent bogs/swamp; 4 percent pasture, 2 percent cultivated agricultural; and 2 percent residential/commercial. Of these lands, more than 52 percent is owned by Federal, state, or county governments, including 30 percent in the Superior National Forest. Competing land uses include taconite mining, timber, and tourist industries (MEQB, 1979).

The 1977 population of the study area was 51,200, about 7300 of whom were employed directly by the iron (taconite) industry. Employment in this industry is expected to reach 8600 by 1985, and the population is projected to peak at 55,000 in the late 80s before falling back to 49,000 by 1995 as production in the iron industry falls. Major towns are Virginia, with a population of nearly 12,000, and Ely, with a population of 5200. Virginia is a supply/service center for the taconite industry while Ely is a tourist center for the Boundary Waters Canoe Area (BWCA).

Nearly 75 percent of the study area is north of the Laurentian Divide. Here drainage is to the north to the Boundary Waters Canoe

Area. Drainage in the remaining area flows to the south and eventually into the Mississippi River.

Ground water supplies are generally poor. The bedrock has low permeability and the glacial deposits vary in thickness and extent and do not regularly provide a continuous aquifer.

4.2.1.2 Mine and Smelter Models

Several sizes of open pit, underground and combination open pit/ underground mines were modeled (Veith and others, 1979), including two underground mines with annual capacities of 5.35 and 12.35 MMmt of ore and two open pit models with capacities of 11.33 MMmt and 20 MMmt per year. In addition, a combination mine was modeled for the 5.35 MMmt underground and 11.33 MMmt open pit mines (total capacity 16.68 MMmt/year). These mine models were correlated with the smelter/ refinery model to produce 100,800 mt of copper plus nickel. Information on requirements for the 20 MMmt model is summarized on Tables 4-1 and 4-2.

4.2.1.3 Environmental Impacts

Environmental impacts were developed on the basis of the integrated mine-mill-smelter-refinery models. Table 4-2 summarizes some of the projected requirements of a facility having a capacity of 20 MMmtpy of ore, or about 85,000 mtpy copper, 15,000 mtpy nickel, and 200 mtpy cobalt. Substantial requirements for land (10,241 acres), labor (2000 men), energy (16.2×10^{12} Btu/yr), and water (2.86×10^{9} gal/yr) are projected. The land requirements are further detailed for three different mine models in Figure 4-2. Tailings disposal is the single largest land use impact.

If 30,000 mtpy of the U. S. nickel demand are to be provided by mining from the Duluth Complex, two of these 20 MMmtpy mines would be required. This committment of approximately 20,000 acres of land (about 70 percent directly impacted) would provide about 5 percent of the U.S. demand in 2010. It should be noted that the amount of land impacted per unit metal production in this model is about twice that

TABLE 4-1

20MMmt OPERATING MODEL FOR DULUTH COMPLEX

Mining Variable

Operating personnel		964
Energy requirements		
Electricity,	10^6 kwh/yr	32.2
	10^9 Btu/yr	338.1
Diesel fuel,	10^6 gal/yr	11.1
	10^9 Btu/yr	1539.5
Propane,	10^6 gal/yr	0
	10^9 Btu/yr	0
Gasoline,	10^6 gal/yr	0.4
	10^9 Btu/yr	50.0
Total,	10^9 Btu/yr	1927.6
Water requirements		
Discharged only, 10^9 gal/yr[a]		0.23
Surface area, acres		
Waste rock-lean ore piles		1988
Overburden piles		173
Miscellaneous mine		40
Open pit mine		523
Underground mine		N.A.
Total, direct		2724
Undisturbed area		1090
(40% of direct total)		

Beneficiation Variable

Energy requirements		
Electricity,	10^6 kwh/yr	463.0
	10^9 Btu/yr	4861.5
Thermal,	10^9 Btu/yr	124.1
Total,	10^9 Btu/ur	4985.6
Water requirements[c]		
Total requirement[d], 10^9 gal/yr		13.00
(total system flow)		
Recycle, 10^9 gal/yr		12.53
Make-up (loss)[e], 10^9 gal/yr		0.47
Operating area requirements		
Plant, acres		400
Tailing basin, acres		4016
Total, acres		4416
Undisturbed area, acres		1766
(40% of total)		

Smelting Variable

Metal Recoveries,	% Cu	96.30
	% Ni	91.67
	% Co	50
	% Precious metals	100.00
Operating personnel		621
Energy requirements		
Electricity,	10^6 kwy/yr	580
	10^9 Btu/yr	6090
Propane,	10^6 gal/yr	1
	10^9 Btu/ur	92
Coal,	10^6 mt/yr	40
	10^9 Btu/yr	1060
Natural gas,	10^6 ft^3/yr	222
	10^9 Btu/yr	222
Thermal,	10^9 Bty/yr	1631
Total,	10^9 Btu/yr	9095
Operating surface area, acres		
Smelter		50
Refineries		100
Slag disposal		25
Total		175
Udisturbed area, acres		70
Water requirements[f], 10^9 gal/yr		
Total[g]		22.87
Make-up (losses)[h]		1.87
Internal recycle		21.06

[a] 24 hr/day 365 day/yr.

[b] N.A. = not applicable.

[c] Water balance includes tailing pond contribution and losses.

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

[e] Includes water in ore fed to plant.

Source: Veith and others, 1979.

[f] Maximum internal recycle assumed.

[g] Includes 0.1 x 10^9 gpy water in concentrate not included in makeup water.

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

TABLE 4-2

DATA SUMMARY FOR COMMON SITE MINE,
PROCESSING PLANT, SMELTER, AND REFINERY
(Rated Annual Capacity 20 MMmt)

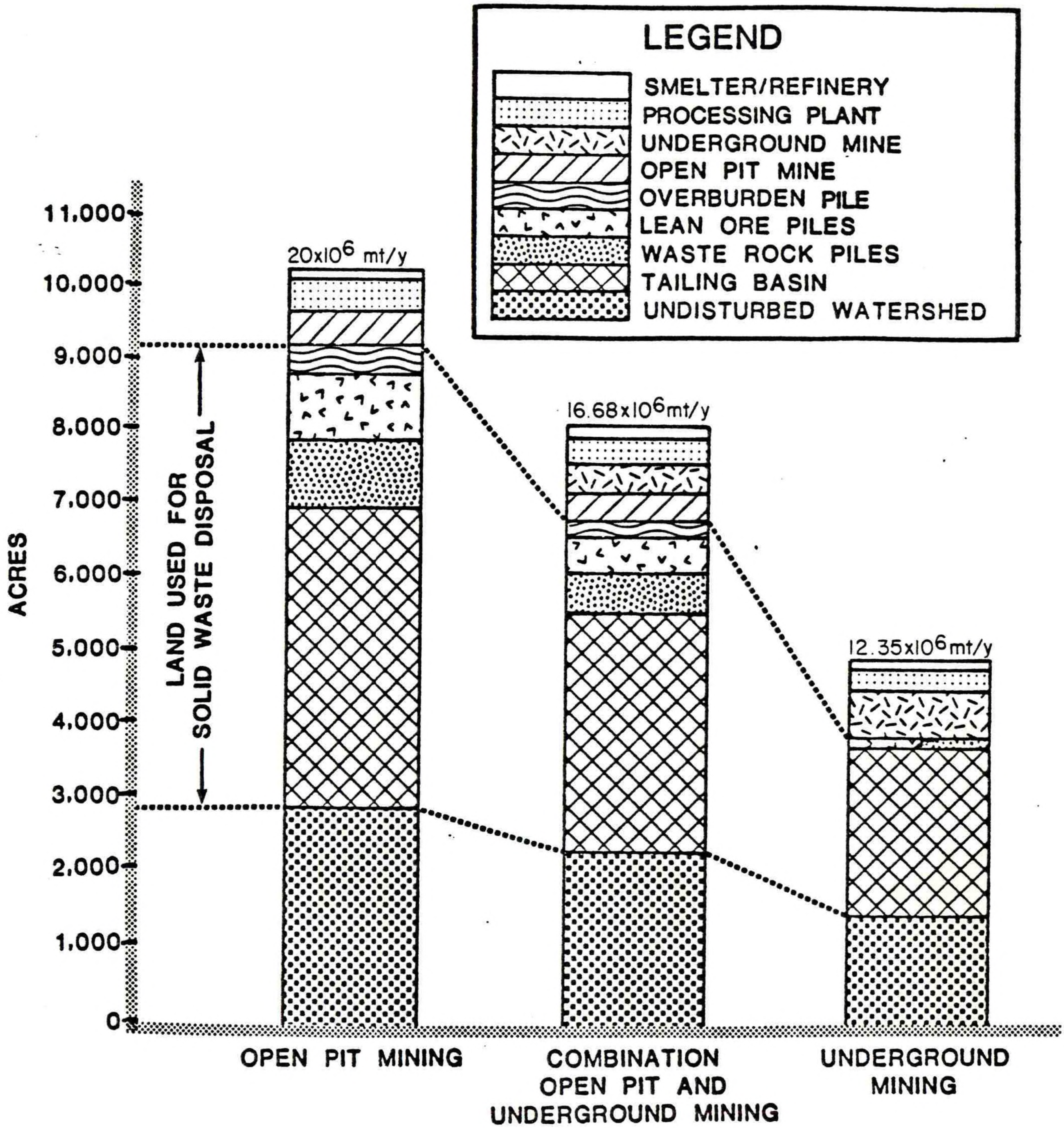
Item	Requirement
Capital cost, $\$10^6$ total	761.03
, $/annual mt ore	38.05
Operating cost, $\$10^6$/yr	119.47
$/mt ore	5.97
Construction manpower, peak	2818
Operating manpower, at full production	1999
Energy requirement, 10^{12} Btu/yr	16.21
Electricity, Kwh/mt ore	54.0
Fossil fuel, 10^3 Btu/mt ore	243.5
Water requirement	
Process, 10^9 gal/yr[a]	37.73
, 10^3 gal/mt ore	1.89
Make-up water, 10^9 gal/yr[b]	0.76
Potable, 10^9 gal/yr	2.1
, gpm	180
Total life of operation, yr	30
Actual production life, yr	27
Effective full production life, yr	25
Area requirement, acres[c]	10241

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

[b]Excludes water in ore, mine discharge water, and precipitation on tailing basin as detailed in Table 4-1.

[c]Direct area plus undisturbed area.

Source: Veith and others, 1979.



Source: Veith and others, 1979.

Figure 4-2. Land Requirements for Three Mine Models.

predicted in Chapter 2.0 for present, more conventional sources of copper and nickel. This reflects the generally lower grade ores which are available in the Duluth Complex.

Air quality impacts would also be potentially significant. In combination with planned development of coal-fired power plants and conversion of the taconite industry to coal from natural gas and oil, SO_2 emissions from copper-nickel smelters, even with 95 percent SO_2 control, could cause exceedances of Class I PSD requirements in portions of the BWCA. In addition, northeast Minnesota's pine and aspen forests and poorly buffered soils and lakes are susceptible to damage from acid rain. Emissions of fugitive dust and particulates could also be limited by the present nonattainment status of portions of the area.

Another major concern is the release of heavy metals from the huge volumes of waste rock and lean ore stockpiles, possibly by leaching as a result of acid rain. Other possible sources of pollution are tailing basin discharges and mine dewatering and smelter waste waters (assuming zero discharge cannot be achieved). Local creeks already contain elevated levels of copper and nickel and exceed EPA guidelines for mercury in edible fish.

Additional adverse impacts which were identified in the study are the removal of water from tributaries of the BWCA, potential health effects due to release of asbestos-like fibers into public drinking water supplies, elevated local noise levels, and exposure to accidents and respiratory irritants among workers and residents near smelters.

Increased employment levels will raise local population by an estimated 12,000 people for each integrated complex, creating many problems such as housing shortages, traffic, need for additional public services, displacement of local residents, and possibly conflicts in life styles. More jobs will be available from mining activities than from smelters. However, the net tax benefits are considerably higher from smelters; these might have to be located outside the immediate area, however, because of PSD restrictions in the BWCA. On the whole, the socioeconomic effects of a well-planned mining/processing opera tion are likely to be diffused to a large extent by the prior existance of a well-developed mining industry, with its associated infrastructure, and the profusion of nearby communities.

4.2.2 Gasquet Mountain, California

Nickel laterite deposits in the Pacific Northwest have been known for many years. The Hanna Mining Company operates an open pit mine at Nickel Mountain near Riddle, Oregon. This mine produces 12,000 to 13,000 tons of ferronickel a year (see Part I) with identified resources estimated at 10 million tons of ore containing 1.5 percent nickel (Cornwall, 1973). Recently the California Nickel Company announced finding a new nickel-cobalt layer in the Gasquet Mountain Area (Alm, 1979). The company plans to develop a mine and processing plant in northern California at Gasquet Mountain in Del Norte County, 14 miles northeast of Crescent City (Figure 4-3).

4.2.2.1 General Setting

Gasquet Mountain is within the Klamath Mountains Geologic Province. The proposed mining area is north of the Smith River, about 6 miles south of the Oregon border. The area consists of rounded hilltops and ridges with steep canyons along the major streams. Elevations range from 750 to 2600 feet.

The bedrock consists of metemorphosed sediments and volcanics which have been intruded by granite and by sheetlike ultramafic rocks. The rocks have been folded, faulted, and sheared as a result of crystal deformation over long periods of geologic time. Within the area of interest ultramafic serpentinite is the predominant rock type. Soils in the proposed project area are deep, reddish-brown laterites, overlying yellow-brown saprolite which has developed on the ultramafic rocks. Soils average 30 feet in thickness and are permeable and highly erodible. The serpentine-derived soils characteristically support only poor vegetation.

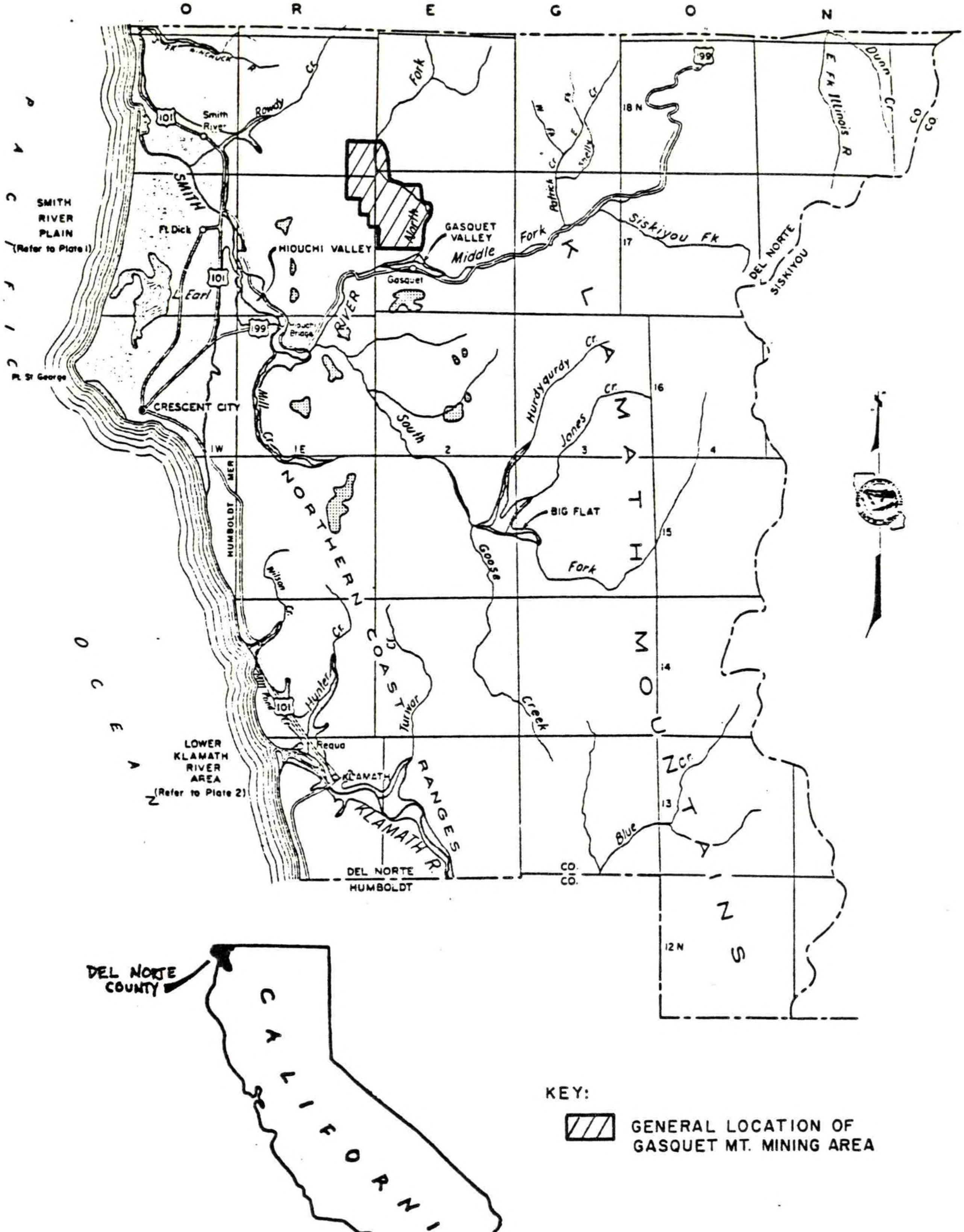


Figure 4-3. Location of Gasquet Mountain Development Area.

The Smith River drainage basin encompasses approximately 650 square miles, the basin is elliptical, with the longer axis north-south. At the eastern margin, elevations range from 5000 or 6000 feet to sea level where the river enters the ocean near the Oregon border. Parts of the Smith River drainage are being considered for inclusion in the Wild and Scenic Rivers System (National Geographic, 1977).

The Gasquet Mountain site is drained by Hardscrabble Creek and several unnamed creeks which join the Smith River or one of its main tributaries, the North Fork. The California Nickel Company has conducted continuous water monitoring for a year on Hardscrabble Creek (Alm, 1979).

The USGS maintains a monitoring station near Crescent City; data from this station indicate that the waters are alkaline, with a pH of 7.3 to 8.2.

Ground water is obtained from several aquifers in the Smith River ground water basin. Most ground water supplies are developed along the coast and in the floodplain of the major streams.

Sharp climatic variations over short geographic distances are common in northern California because of the rapid changes in topography. The coastal maritime climate gives way rapidly to a continental climate inland over the coastal mountains. Rainfall averages 50 to 70 inches per year, and locally exceeds 100 inches; most of the rain occurs from October to April. The summers are hot and dry, especially in the higher elevations.

The region is rich in timber. Redwoods cover the coastal areas, and Douglas fir and other conifers grow on the inland areas. This characteristic vegetation, however, is lacking on the soils derived from serpentine. In the site area, the vegetation consists of a patchy forest shrub community, with open stands of Douglas fir. Several sensitive plants (those that may at some time be classified as threatened or endangered) may occur in the site area.

The area contains widely differing habitats, including stands of Douglas fir, extensive brush-covered areas, marshy habitats, and aquatic and riparian habitats. These provide food, shelter, and breeding sites for a wide range of terrestrial species. The U.S. Forest Service (1972) lists more than 200 species of birds in the Six Rivers National Forest which covers much of the study area.

The nearest large town is the county seat, Crescent City (population 2600), some 25 miles to the west of the mine site. Del Norte County had a 1977 population of just under 16,000, with nearly 80 percent of that total residing in the vicinity of Crescent City. The labor force at that time was estimated to be around 7000, with an average unemployment rate of nearly 11 percent.

Over 90 percent of Del Norte County is forested. In view of this natural endowment, the forest industry historically has been the dominant sector of the local economy, but is expected to decline in importance after 1985. The next largest activity is the fish canning and processing industry, which is only one-tenth the size of the logging industry.

4.2.2.2 The Mine and Process Plant

The California Nickel Company owns leases on Gasquet Mountain and in other areas of Del Norte County. The deposits to be developed are in a saprolite layer underlying the laterites. This new deposit is higher in nickel content (1.0 percent) than the upper beds and lower in cobalt content (0.05 percent). Exploratory data indicate that the ore bearing layer is at least 13 feet thick and extends for 2 miles along the eastern side of the mountain (Alm, 1979). Reserves are estimated at 500,000 tons of nickel and 25,000 tons of cobalt.

Only small areas (40 to 50 acres) would be mined each year, and only to a depth of 20 to 30 feet. Production capacity is planned for up to 15,000 tons of nickel and 750 tons of cobalt annually. Once the metal is recovered the remaining material will be returned to the pit. The slopes to be mined are reported to be gentle, and the mine sites will be separated, leaving green belts between the excavated areas to retain runoff and to provide seed sources for reestablishment of biotic communities.

The onsite processing facilities are intended to be generally self contained and thus would produce few emissions or effluents. The process "uses chemicals and water at relatively low temperatures compared to old-fashioned high-temperature smelting" (Alm, 1979).

It is anticipated that construction will require about 1200 workers and that the mining/processing operation will require about 400 to 500 workers, of whom most will come from the area.

4.2.2.3 Environmental Impacts

As indicated in the Del Norte Triplicate report (Alm, 1979), the California Nickel Company intends to do everything possible to minimize environmental impacts. Areas of concern however, include displacement of any sensitive plant species which may occur in this area and problems related to runoff and water quality degradation considering the possible inclusion of the Smith River in the Wild and Scenic River system.

The introduction of mining to the area will undoubtedly cause certain socioeconomic impacts within the county as a whole, and in several of the local communities. Some of this impact will probably be absorbed by the labor surplus in the area, but any significant influx of new workers (and their dependents) will have a noticeable effect on housing, public services, retail trade, and traffic levels.

There are expected to be no significant impacts on the ambient air quality in the area as a result of the operation of this facility. The processing methodology will result in little or no SO_2 emissions. There will be some fugitive dust emissions but these would be noticeable only in the immediate vicinity of the mine.

Should the Gasquet Mountain deposit be developed as planned, it could provide approximately 15,000 tpy of the estimated 59,000 tpy U.S.

production of nickel by 2010. This is less than 3 percent of the projected U.S. demand for that year. Though of importance, development of Gasquet Mountain is not a significant factor in attainment of U. S. self-sufficiency in nickel supply.

4.3 COBALT

As noted in Section 3.3, most cobalt is obtained as a by-product of mining copper and/or nickel. The United States receives most of its present supplies from Zaire (Section 3.3.1). In the U.S., cobalt occurs in several areas, but these have to date been uneconomical to produce. None of these deposits are considered to be reserves. Estimated U.S. resources total 842,000 tons, occurring primarily in the Duluth Complex in Minnesota, the Blackbird District of Idaho, the laterite deposits of northwestern California (Gasquet Mountain), and in the Missouri lead belt. U.S. demand over the period 1980 to 2010 is projected at a low of 421,000 tons, a most likely of 579,000 tons, and a high of 661,000 tons. The annual demand in 2010 will be about 30,000 tons. U.S. domestic supplies are not expected to exceed 4200 tons annually during this period and thus self-sufficiency is not likely to be achieved.

4.3.1 Duluth Complex, Minnesota

The Duluth Complex in northeastern Minnesota contains large amounts of copper, nickel, and cobalt. The potential for development of these resources was discussed in Section 4.2.1 with regard to development of mines to provide nickel supply should the U.S. become dependent entirely on its own resources. Since the cobalt would be obtained as a by-product of nickel processing, no substantial additional impacts would be created. See Section 4.2.1 for a discussion of the setting, the proposed mine development, and the potential impacts.

Cobalt resources from the Duluth Complex are estimated at 80,000 to 90,000 metric tons. A single integrated complex is expected to produce about 200 tons per year, which is less than 1 percent of the projected U.S. demand in 2010. It is evident that only a massive commitment of land, water, and other resources could provide a substantial portion of U.S. demand from the Duluth Complex.

4.3.2 Blackbird District, Idaho

The Blackbird Mining District is in Lemhi County in east-central Idaho in the Panther Creek Planning Unit of the U.S. Forest Service (Figure 4-4). The unit is in the Salmon River Mountains; the Salmon River marks the northern boundary of the planning unit (Bennett, 1977).

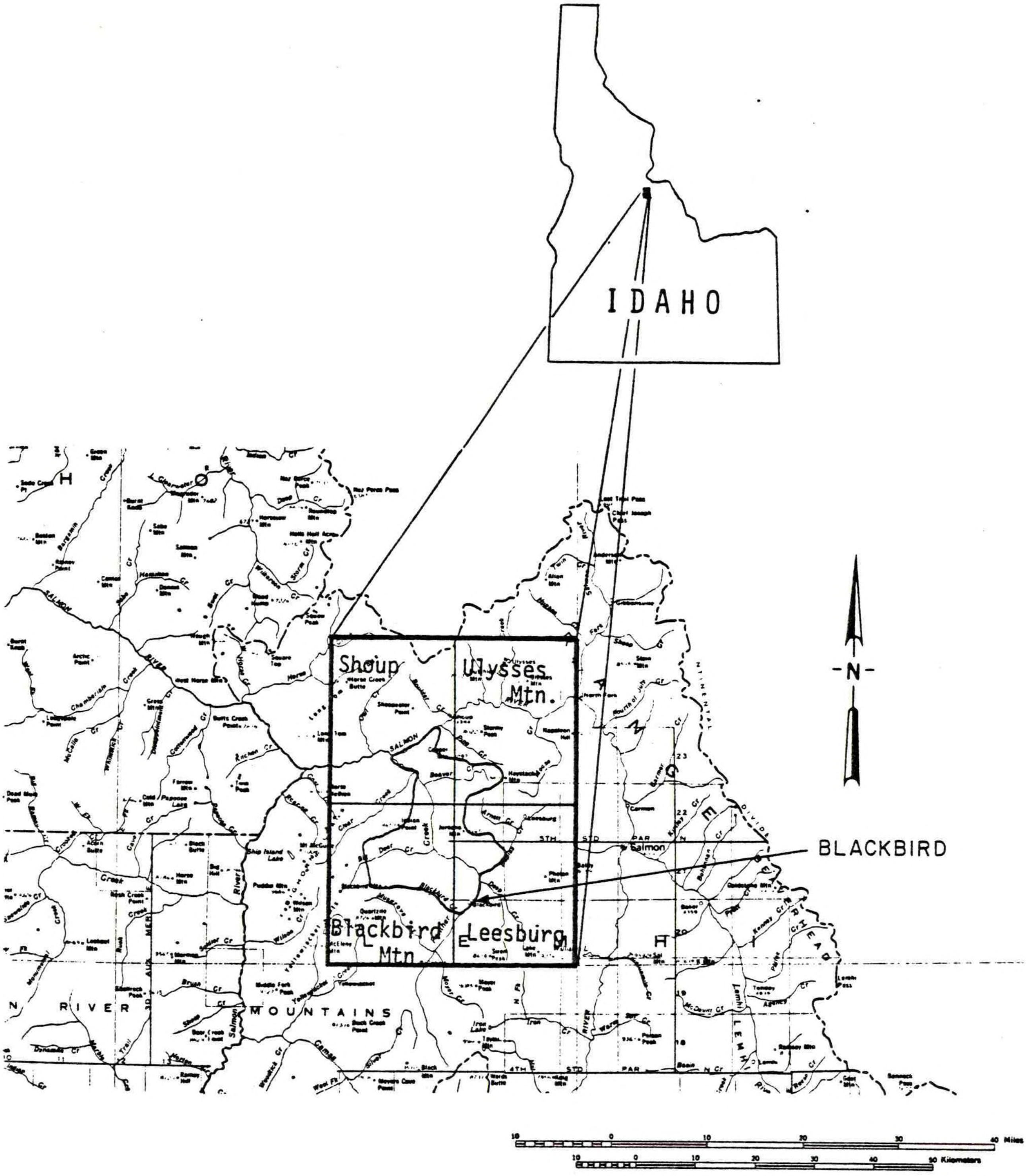
Cobalt was reported from the district as early as 1899, although at that time interest was primarily in copper and gold. In 1915 cobalt prices rose and mining began in the area and continued until 1920. Underground mining was started in 1951 and continued until 1958. From 1958 to 1960 the open pit mine continued to operate but falling copper prices and importation of cobalt from South Africa forced closing of the underground mine. The mine was reopened briefly in 1966. The Idaho Mining Company (Hanna Mining Company) operated the mine from 1967 until 1969. The mine has been inoperative since that date (Bennett, 1977). The mine was purchased by Noranda in 1979.

4.3.2.1 General Setting

The Blackbird District of east-central Idaho is located in the eastern half of Lemhi County. The area is intersected by the Panther River, and is part of the Salmon National Forest. The district was included in the River of No Return Wilderness Area in July 1980 (Environmental Reporter, 1980b).

Lemhi County, with a population of only 5566, has a population density of just over 1 person per square mile. Within a 30-mile radius, the only "large" population center is the town of Salmon (population 3000). The few other communities in the area all have populations of 200 or less. The nearest cities of any size (e.g., Boise, Butte, Missoula) are each more than 100 miles distant.

The town of Cobalt, owned by the Hanna Mining Company was built to house the miners at the Blackbird Mine. It is now occupied by the caretakers left to supervise the property. The town is reached by a dirt road from Shoup, a small center containing a general store and gas



Source: Bennett, 1977.

Figure 4-4. Location of the Blackbird Area in the Panther Creek Planning Unit, Lemhi County, Idaho.

station, just north of the Panther Creek Planning Unit (Bennett, 1977).

Elevations range from 3200 feet in the Salmon River canyon on the north to more than 9000 feet in the western part of the study area. The main drainage is the Salmon River which flows to the west; Panther Creek is a north-flowing tributary of the Salmon. To the west and north of the area portions of the Salmon River have been designated as part of the National Wild and Scenic River system (Environmental Reporter, 1980b).

Idaho lies to the east of the Cascade Mountains, some 300 to 500 miles inland from the Pacific and therefore has many characteristics of a continental climate. These characterics are, however, modified by the influence of the prevailing westerlies. The topographic relief in much of the state is also striking, causing considerable contrast in climate within short geographic distances (Ross and Savage, 1967). Within the study area mean maximum temperatures vary from between 28°F and 32°F in January to 80° to 84°F in July. Precipitation in the planning unit varies from 10 to 20 inches in the valley bottoms to 25 to 30 inches on the higher ranges (Bennett, 1977).

Vegetation in the area also follows patterns related to the topography and corresponding rainfall and temperature variations. Sagebrush and grasses are dominant in the canyons, and subalpine fir, lodgepole pine, and Douglas fir are common above 6000 feet.

The rocks of the planning unit are Precambrian in age. They include metasediments of the Yellowjacket Formation which have been intruded by an augen gneiss. The copper-cobalt mineralization occurs in the upper quartzite/schist unit of the Yellowjacket Formation in the Blackbird structural block. Primary sulfide ores are cobaltite and chalcopyrite mixed with pyorrhotite and pyrite (Bennett, 1977). A conservative estimate places the total tonnage of ore at 6 million tons having an average grade of 0.5 percent cobalt and 1.0 percent nickel.

Until recently, economic activity in the county was largely centered around logging. Declining demand for lumber and other wood products, however, has caused a major reduction in operations in the area's sawmills and logging camps. Unemployment in Lemhi County is now greater than 10 percent.

4.3.2.2 Mine and Processing Facilities

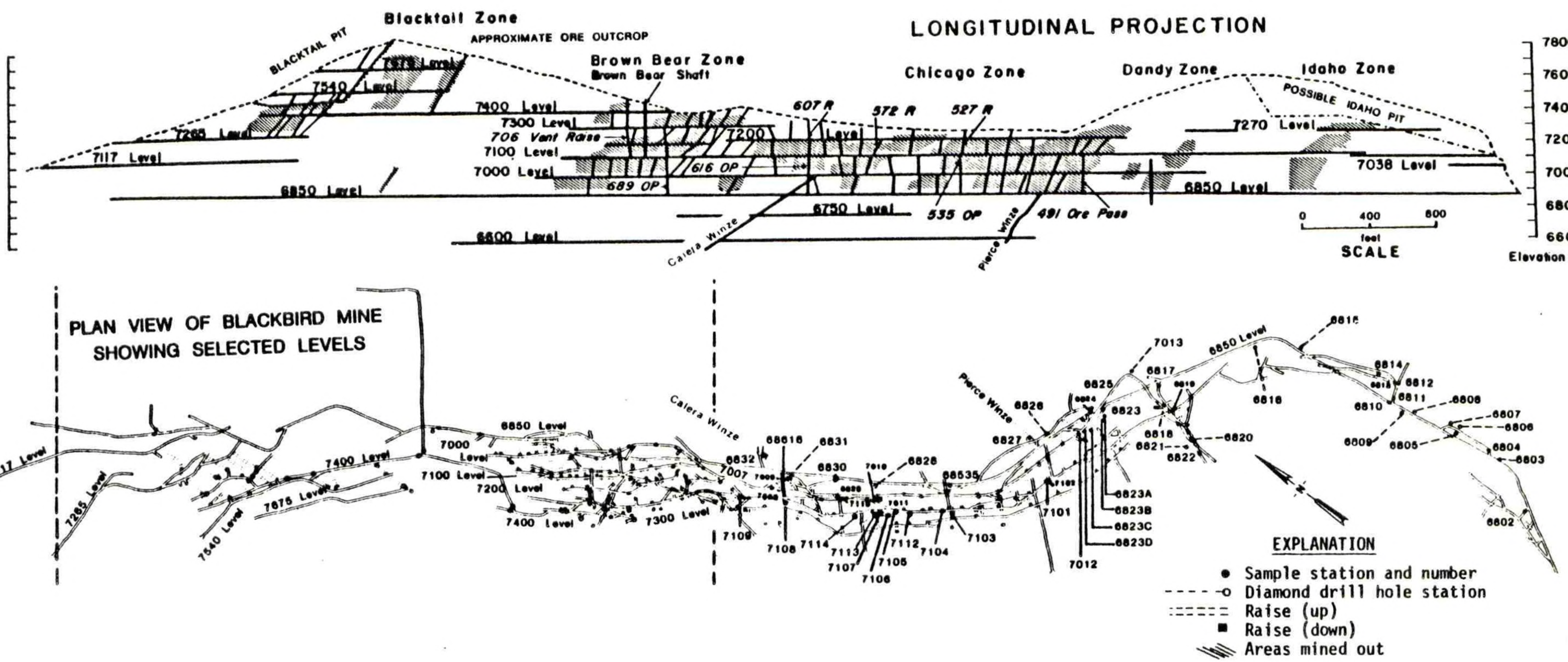
There are five main ore bodies in the mine, they dip approximately 50° to 60° NE and strike N 40°W. Most underground production is from two of these zones, and open pit mining is from a third (Figure 4-5). The entrance of the Brown Bear Shaft (Figure 4-5) is about 7500 feet above mean sea level (MSL); production is from several levels from 6850 feet to 7675 feet above MSL.

The Blackbird ore body could contain as much as 60,000 tons of nickel and 30,000 tons of cobalt. Targeted annual production of cobalt is 2200 tons or about 7 percent of U.S. demand in 2010. However, the ore body is expected to have a lifetime of only about 10 years unless other reserves are discovered.

In past operations, the ore was crushed at a mill on site and separated into cobalt-rich and copper-rich concentrates. The cobalt concentrate contained up to 80 percent cobalt, and the copper rich concentrate 90 percent copper. The ore contains considerable quantities of arsenic and should the mine be reopened, special processing would be required to meet regulatory requirements.

4.3.2.3 Environmental Impacts

Since the main mine is underground and, particularly, since it has apparently been well maintained, few impacts to the environment should occur from reopening. If the mine were reopened, there would be increased traffic on the roads, which would cause dust and perhaps some erosion during wet weather. There would be little need for construction as the mining town is already in existence. Increase of the output of the mine might require some additional work force and expansion of the town of Cobalt.



Source: Bennett, 1977.

Figure 4-5. Plan View and Longitudinal Projection of the Blackbird Mine.

The Middle Fork of the Salmon River is just west of the planning area. This stream is part of the Wild and Scenic River system; it is on the west side of a drainage divide, however, and should not be impacted by any mining activity. The main Salmon River which forms the northern boundary of the planning district (Figure 4-4) was included in the Wild and Scenic Rivers system in July 1980 (Environmental Reporter, 1980). The Blackbird Mine site is, however, several miles upstream on Panther Creek and any impacts from mine drainage (with a well run operation) should be minor. As noted previously the district was included in the River of No Return Wilderness Area. It was, however, set aside as "a special management zone in which mining is the dominant use" (Environmental Reporter, 1980b, p 446).

With mining plans reportedly calling for only the construction of bunkhouses at the mine site, socioeconomic effects are likely to be concentrated in the town of Salmon. Locally high unemployment will tend to deflate the demand for outside labor and thereby reduce some of the potential mining-induced growth effects. To the extent that some outside hiring may be necessary, socioeconomic effects may occur in the housing, retail trade, and public and private services sectors.

In the event that a large scale smelting operation would be required at or near this mine site, significant environmental impacts could occur as a result of SO_2 emissions from the smelter. These impacts have previously been discussed in Section 1.2.3. If these impacts were determined to be excessive (i.e., resulting in violations of the ambient air quality regulation), alternative control measures such as SO_2 removal from exhaust gas streams by sulfuric acid production, would have to be utilized. Also, the relatively high content of arsenic in the ore at this time may necessitate special processing methods in order to reduce the arsenic content in the processed ore so as to comply with regulatory requirements. In the years to come it is very likely that trace metals (such as arsenic) will be subject to stringent environmental control measures at facilities of the type discussed here.

4.4 MANGANESE

At present, the U.S. is almost completely reliant on foreign sources of manganese ores. There are no economically recoverable domestic reserves of manganese. Resources, which total 8 million tons in contained manganese, are virtually all manganiferous iron ores with less than 10 percent manganese content; these are not substitutable for manganese ores directly, and metallurgical or other problems exclude most from beneficiation to higher grade products in the near future.

Cumulative U.S. demand for manganese ores is predicted to range from a low of 52 million tons to a high of 68 million tons during the period 1980 to 2010. Annual demand is estimated at about 2.5 million tons by 2010.

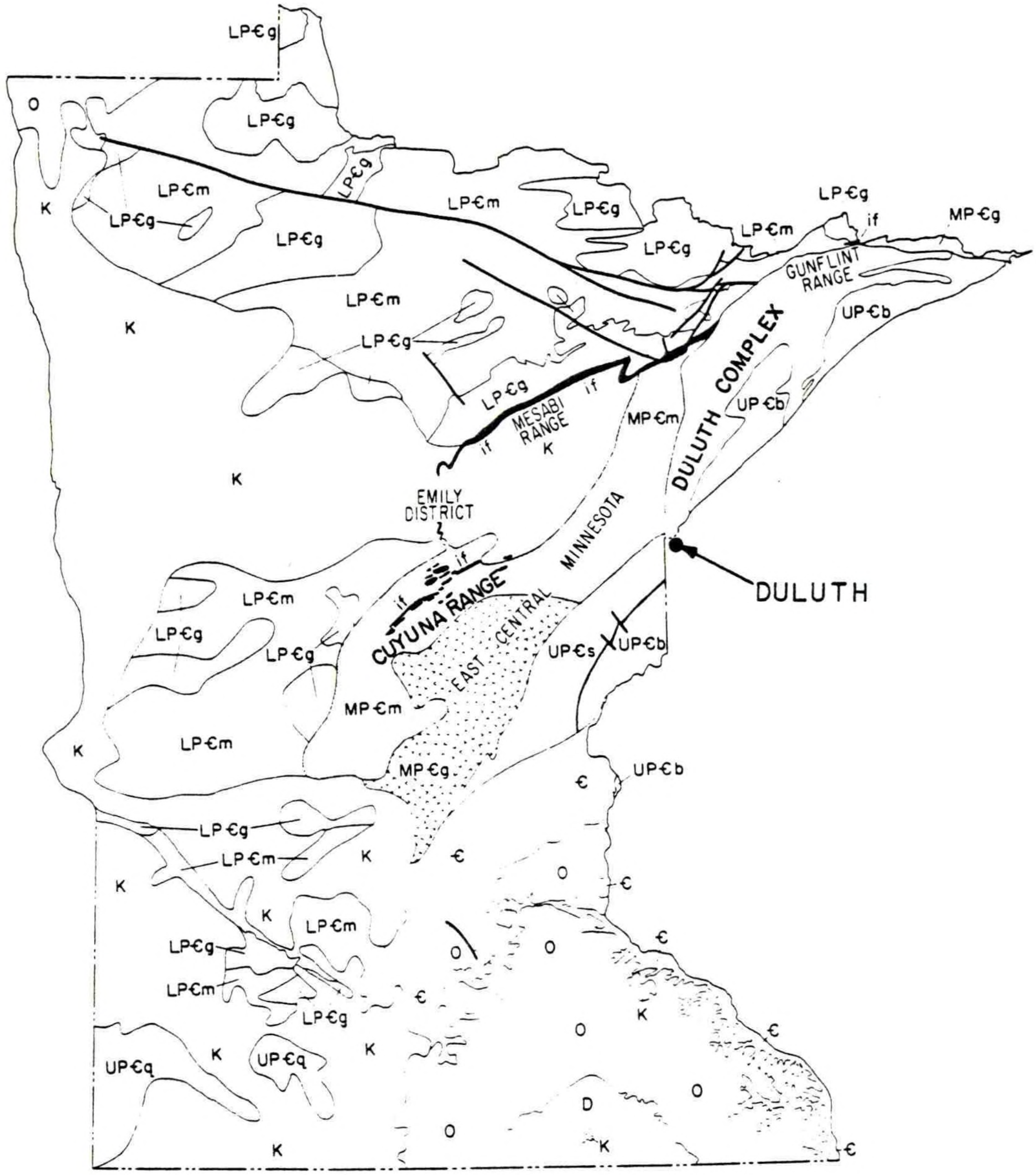
Of the potential sources of manganese in the U.S., two have been selected for review: the Cuyuna Range in central Minnesota and Artillary Mountains in west-central Arizona.

4.4.1 Cuyuna District, Minnesota

The Cuyuna Mining District is in Central Minnesota about 100 miles west-southwest of Duluth (Figure 4-6). Mining for iron began in 1911 (Grout and Wolff, 1955). The district is part of the extensive Precambrian iron ore province of the Lake Superior region (which includes other famous iron mining "ranges" - the Mesabi, Vermillion, Geogebic, and Marquette). The Cuyuna ores differ from the other iron bearing formations in that they contain 2 to 6 percent manganese. Iron reserves in the region have been greatly diminished and production had decreased to 474,000 gross tons in 1970 from 3.6 million gross tons in 1953 (Marsden, 1972).

4.4.1.1 General Setting

The area is about 120 miles southwest of the Duluth Complex Study Area described in Section 4.2.1 and general environmental conditions are similar.



Source: Morey, 1972.

Figure 4-6. Generalized Geologic Map of Minnesota Showing Cuyuna Range and Duluth Complex.

Populations of the four counties in which the deposits occur are given in the following tabulation from the Minnesota Office of Economic Development.

	Population	Population Density	Unemployment Rate (12/79)
Aitkin	13,600	7.4	12.8
Crow Wing	41,300	41.5	9.6
Morrison	28,400	25.2	9.6
Todd	26,700	28.3	7.6

The major economic activity in the region is related to tourism and recreation, drawn by the area's numerous lakes and forests. If it became necessary to mine the range for its manganese content, the socioeconomic impacts would be potentially significant.

4.4.1.2 The Mining District

The Cuyuna District extends southwest from Aitkin to Todd Counties, a distance of about 75 miles. The most extensive deposits are in Crow Wing County. Three iron ranges are recognized - The Emily, North, and South Ranges. The ore-bearing rocks are of Middle Precambrian Age. Their geologic relationships are hard to determine because of the cover of glacial deposits and Cretaceous age rocks which together reach thicknesses ranging from 20 to 450 feet (Marsden, 1972).

Topographically, the Cuyuna District is not a "range," but a low, swampy plain covered with a mantle of glacial drift 25 to 300 feet thick, making the deposit amenable to open-pit mining. The manganiferous iron ores were probably formed by oxidation of the primarily iron-bearing formation. Both the enclosing rocks and the iron-bearing beds have been strongly folded and faulted, and dip steeply.

The primary ores of the Cuyuna Range are of two types: 1) green carbonate slate, and 2) cherty iron formation. The green carbonate slate (an approximately 50-foot bed) is stratigraphically located some 100 to 150 feet beneath the cherty iron formation. It contains 3 to 8 percent manganese, 20 to 30 percent iron, and 25 to 40 percent silica. Upon oxidation, the slate has yielded the so-called "brown ores" which contain about 9 percent manganese, 44 percent iron, 6 percent silica, and 0.27 percent phosphorous. The manganese minerals are manganite, pyrolusite, psilomelene, and manganosiderite (Grout and Wolff, 1955). The cherty iron formation is apparently much thicker, and often contains more manganese and less iron. Oxidized zones yield the so-called "black ores" which contain about 15 percent manganese, 37 percent iron, 15 percent silica, and 0.09 percent phosphorous.

As discussed in Part I of this report the total manganese resources of the Cuyuna Range are estimated to be 272 million tons of ore, averaging 8 percent manganese and 32 percent iron. They may represent the most promising domestic land resource for manganese in the U.S. Also, because the range is part of an area in which iron ores have been mined for three-quarters of a century, a large infrastructure base is already in place.

In the late 1950s, as noted in Part II, the government sponsored a semicommercial pilot operation for treatment of Cuyuna Range manganiferous ores. Using an ammonium carbonate leach process, the plant could process 200 tons of ore per day (equivalent to an annual capacity of about 7000 tons of manganese). The plant was closed in 1962, apparently for economic and technical reasons. Difficulties were encountered due to the intimate interlocking of manganese minerals with gangue minerals. In addition, ammonia recovery was apparently poor, resulting in high operating costs. The metallurgical problems encountered in the early test plant remain unsolved. Reports that ammonia requirements would be staggering in a full-size operation indicate that an appropriate extraction technique remains to be found.

4.4.1.3 Environmental Impacts

Large scale open pit mining of manganiferous ores at Cuyuna would require extensive disruption of the land surface. Production sufficient to meet the U.S. demand for 1980 of 1.54 million tons (contained metal) would require removal of about 19 million tons of ore (excluding overburden), or open pit mining of about 100 acres assuming an ore body thickness of 50 feet. The substantial quantities of waste might possibly be returned to the pit if processing did not produce materials harmful to the environment. If not, additional land would be needed for tailings or waste piles. Disruption of such large acreages of potentially productive lowlands could have substantial adverse effects on atmospheric (from fugitive dust) and on biological resources.

Processing of the ores by either ammonia carbonate leaching or roasting requires substantial inputs of water and energy. Though sulfur dioxide would not be emitted, process wastes and runoff could be high in iron content and contribute to low pH conditions in receiving waters.

Socioeconomic impacts would stem primarily from two causes: the influx of labor and the potential physical changes to the environment. If mining were to proceed at a rate sufficient to meet projected domestic demand in 1980, an estimated 700 men would be employed in mining operations alone. With a typical mine employment multiplier of 2.5, that translates into a total of 1750 new jobs. Combined with a reasonable dependency ratio of two- or three-to-one this would cause a potential population increase of more than 3500 people (exclusive of those persons in employment related to processing). Given the large number of small communities in the area, the impact of this population increase would likely be diffused to some degree, but significant pressure on the basic socioeconomic structure of the area would still exist.

Socioeconomic impacts resulting from changes in the environment are much more difficult to quantify. Any reduction in environmental quality could have a major effect on tourism and recreation opportunities in the area. Likewise, a degradation of environmental quality would be certain to affect resident lifestyles and livelihoods.

4.4.2 Artillery Mountains, Arizona

The Artillery Mountains manganese deposits are located in the southeast corner of Mohave County, Arizona, near both the Yuma and Yavapai County lines (Figure 4-7). There is no active manganese mining, although several small mines have been developed in the area. No large communities are within a 30-mile radius of the area; the nearest major trade centers are Prescott (60 miles to the east), Phoenix (105 miles southeast), and Flagstaff (105 miles northeast). Bagdad, a mining community owned by the Cyprus Bagdad Copper Company, is a town with a population of 2300, approximately 30 miles to the northeast of the mine area.

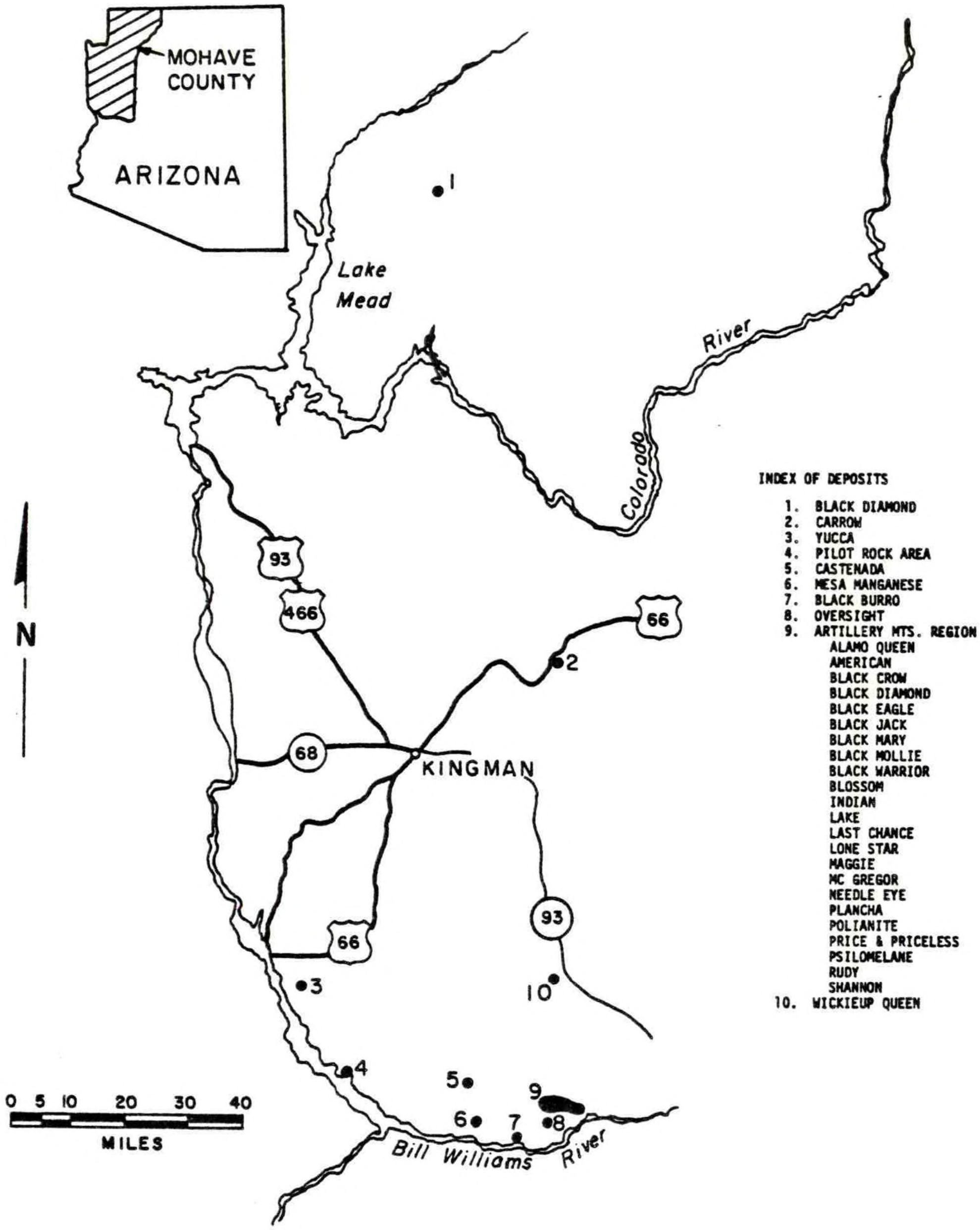
4.4.2.1 General Setting

The Artillery Peaks Mining area (manganese) includes numerous claims, primarily in T 11 & 12 N, R 12 & 13W (Gila and Salt River Base Line). Most of these claims are in a U-shaped area 3 miles wide by 8 miles long, with the base of the U along the Bill Williams River. The area is characterized by dissected mountains, alluvial-filled valleys, and numerous intermittent streams which empty into the Big Sandy River and Bill Williams River.

Two basic types of manganiferous deposits have been described in the area:

- Stratified oxide deposits in the clays, standstones, conglomerates, and other sediments of the Artillery, Chapin Wash, and Sand Trap formations (of Tertiary age).
- Fracture fillings.

Of the two, only the stratified oxide deposits of the Chapin Wash Formation are thought to be of any potential significance.



Source: Farnham and Stewart, 1958.

Figure 4-7. Manganese Deposits of Mohave County.

The approximate elevation of the Artillery Peaks area is 2600 feet. The predominant vegetation is Paloverde-Mixed Cacti Series (Brown and others, 1979). The following is a description of this desert habitat.

Terrain: Bajada, foothills, steep slopes of desert mountains; 800-3500 ft. elevation.

Climate: Hot-arid with bimodal annual precipitation distribution; diurnal temperature variation often large. Temperatures below 20° F occur only a few days in winter. Average precipitation 5-13 inches annually. Frost-free period 220-300 days.

Soils: Lythic haplargids and torriorthents, typic durorthids, calciorthids, and haplargids.

Vegetation: Upland desert scrubland formation, highly diverse species and life forms (tree, shrub, cactus, vine, grass); large saguaro and trees 30-60 ft., perennial dominants include little-leaf paloverde, saguaro, ironwood, triangle bursage, ocotillo, jumping cholla, barrel cactus, creosotebush, bursage.

Fauna: Abundant and diverse; cacti provide nesting sites for many birds such as Gila woodpecker, cactus wren, several species of owls, gilded woodpecker, phainopepla, Gambel's quail; reptiles include numerous species of snakes and lizards; mammal species include black-tailed jackrabbit, peccary, coyote, and rodents such as pocket mouse, kangaroo rat, and cactus mouse.

The Paloverde-Mixed Cacti Series is a highly diverse community, rich in both species composition and diversity of life forms. This community has been called the most diverse of any major plant community in the United States (Lowe, 1964). Additional importance is attached to this community in that it is the "hallmark" for the State of Arizona. The State Tree is the Paloverde, the State Flower is the bloom on the saguaro cactus, and the State Bird is the cactus wren. All three of these species are dominant in this desert community.

Several plant species of the Mohave Desert also occur in the general area interspersed with Paloverde-Mixed Cacti vegetation. Joshua Tree and Bladder-Sage are both interspersed with Sonoran vegetation.

The rocks occurring in the area include Precambrian granite and gneiss; Cretaceous to Tertiary shale, sandstone, conglomerate, limestone, and intrusives; Tertiary sandstone, siltstone, and conglomerate; and Tertiary to Quaternary gravel, sand, silt, and basalt. There are several faults in the area which trend either N-S or NW-SE.

Throughout the area there are numerous mines and prospects. The area is rich in manganese, iron, copper, gold, silver, uranium, selenium, and tellurium.

Surface water absoption and runoff is nearly complete, leaving few bodies of surface water in the area. Water for human use is obtained from deep wells and a few springs. Well water is generally of good quality, ranging in dissolved solids from about 60 to 1200 milligrams per liter. The water has a natural fluoride content of about 1.0-1.5 milligrams per liter, which is slightly higher than the optimum level.

Archaeological artifacts have been found in numerous areas or sites. Pottery dating to about 1100 AD has been identified as belonging to the Prescott Branch of the Patayan cultural tradition.

Air quality is generally very good. Few sources of air pollution exist in the area; and those pollutants emitted are readily dispersed by a high frequency of strong morning winds. Airborne particulate matter is generated by winds which frequently exceed 12 miles per hour, and by human sources such as vehicle emissions, local fires, chimneys, traffic on unpaved roads, and mining. The estimated average concentration of suspended particulates is approximately 40 micrograms per cubic meter of ambient air. Sulfur dioxide is present in the ambient air, generally in trace quantities, usually less than 0.01 parts per million (26 gm/m^3).

4.4.2.2 Manganese Resources

The Chapin Wash Formation underlies most of the valley between the Artillery and Rawhide Mountains in the southern part of Mojave County and has two essentially parallel manganiferous zones, which have a stratigraphic separation of approximately 900 feet. The upper bed crops out along the Artillery Mountains on the northwest side of the valley, and is found at or near the top of the Chapin Wash Formation. The lower zone of mineralization crops out mainly along the Rawhide Mountains on the southwest side of the valley. Although it can be traced along strike for nearly 4 miles, and is as much as 350 feet thick, mineralization consists only of widely separated manganiferous lenses scattered among thicker and more frequently barren zones. The entire formation is interbedded with fanglomerates, clays, and tuffs. In addition to interfingering with barren beds, the deposits grade laterally into barren rock. The manganese is said to range from zero to 30 percent, with only small quantities containing more than 20 percent manganese. The largest part of the deposits contain less than 5 percent manganese, along with 3 percent iron, 0.08 percent phosphorous, 1.1 percent barium, and trace amounts of copper, lead, and zinc. Most of the deposit could be developed only by underground mining. A review of mining and processing activities through the mid-1950s may be found in Farnham and Stewart (1958).

Total manganese resources are estimated at 175 million tons of ore grading 4 percent manganese and 1 to 3 percent iron. The USBM has conducted extensive pilot-plant work on Artillery Peak ore in the past. Two flotation methods, fatty-acid and oil emulsion, have been successfully demonstrated (from a technical standpoint) on select high grade (10.6 percent manganese) samples. The former yielded a concentrate assaying 36 percent manganese and a middling assaying 15 percent (with 85 percent recovery), while the latter yielded a concentrate assaying 36 percent manganese (with no middling product) at 80 percent recovery.

Nonetheless, the Artillery Mountain deposits are not regarded as promising prospects for development. The scattered nature of the deposits, remote location, and low grade will tend to prohibit or at least severely restrict the development of the district, barring a major national shortage of manganese.

4.4.2.3 Environmental Impacts

Given the remoteness of the site, a major mining operation would, in all likelihood, require the construction of a "company town" with all the requisite private and public services and facilities. Large-scale mining would also necessitate underground mining, which as was noted earlier, is typically more labor intensive than surface mining. Transportation lines and a source of power would also have to be brought into the area. Aside from the impacts associated with the creation of a new town, other socioeconomic impacts would be limited to the few small communities located nearby. The magnitude of the project would determine the actual severity of these localized socioeconomic effects.

Impacts to the biological resources of the area would involve loss of habitat in the immediate area due to spoils dumps, buildings, facilities and appurtenances. The need for minimizing makeup water requirements would also reduce potential effluents as reuse and treatment would be preferable to discharge.

Air quality in the region of a new mine would be adversely affected only to the extent that additional dust will be generated. Since the mining would be primarily underground, these impacts are not expected to be nearly as great as for an open pit mine. The reader should refer to Section 1.2.3 for a general discussion of impacts of mining. SO_2 emissions are not expected to occur from processing the manganese ores.

4.5 IMPACTS DUE TO SHORTAGES OF METAL PRODUCTION

If the U.S. is forced to rely solely on domestic sources of copper, nickel, cobalt, and manganese, only the demand for copper could reasonably be met during the period 1980 to 2010. As discussed in Part I and in Sections 4.2, 4.3, and 4.4 of Part III, domestic resources of nickel, cobalt, and manganese exceed cumulative projected demand during this period, but many obstacles exist to their development. Among the more important obstacles are economics, processing technology, environmental impacts, and lead time required to get sufficient production capacity in place. In a national emergency, two other options are available: recycling and substitution. Application of those options was discussed in the demand projections of Part I.

A reduction in the supply of nickel would have extensive impact on the availability and quality of stainless steels and various metal alloys, certain chemical products, and batteries. Substitute materials exist for many nickel uses, including the use of plastics and other materials in place of stainless steel where nickel functions as a corrosion inhibitor. To the extent nickel is replaced by other alloying agents, such as aluminum, cobalt, and molybdenum, those materials will also have to be mined and processed, probably at a higher cost than for nickel under present market conditions, and with their own environmental impacts. Secondary recovery of nickel (recycled scrap) provided about 17 percent of the U. S. nickel supply in 1977, and this is expected to increase to 28 percent by 2000. A sharp reduction in nickel supply and use would certainly be very disruptive to the U.S. economy, through probably not catastrophic.

A reduction of cobalt availability in the U. S. would have a severe effect on the manufacture of jet engines and high-speed cutting tools, and a substantial impact on the production of high strength cobalt-alloy magnets. The superalloys used in jet engines, particularly the high performance power plants used in military applications, commonly have a cobalt base, although nickel is substitutable to some extent. This use gives cobalt a strategic value. Substitutes have reduced the demand for cobalt-alloy magnets in some consumer products. No acceptable substitute exists for cobalt in high-speed cutting tools. Very little cobalt is, or can be, recycled because of the extremely small proportion of cobalt in most alloys. In summary, though cobalt demand could be somewhat reduced if necessary, certain critical and strategic needs do exist.

A reduction in the supply of manganese in the U. S. could have a critical impact on the production of steel and iron, as well as on certain aluminum and magnesium alloys. Another important (though small) use of manganese is in the construction of dry cell batteries. The primary use for manganese is in metallurgical applications where it functions as a very necessary desulfurizer. Substitution of other materials for this use is highly limited by availability, cost, and effectiveness considerations. Use of low-grade manganese substitutes also have several disadvantages, including increased waste, higher net manganese consumption, reduced quality steel, and higher energy use. The recycling of manganese-containing slag is no longer considered viable because of continued dilution by "lower grade" slags. In summary, a significant reduction in manganese availability would have a crippling impact upon the quality, quantity and cost of steel produced in the U. S. and perhaps upon the economy as a whole.

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