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SUMMARY REPORT ON MINERAL RESOURCE STUDIES IN THE JAMES BAY REGION

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Énergie et Ressources naturelles
Québec

SUMMARY REPORT

on

MINERAL RESOURCE STUDIES IN THE JAMES BAY REGION

to

THE JAMES BAY DEVELOPMENT CORPORATION

March, 1974

by

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PREFACE

From the beginning, it has been recognized that minerals are one of the principal resources upon which any future growth of the region could be based. Accordingly, a comprehensive survey of the known mineral resources of the region was carried out by Services Géotechniques Shickshocks (S.G.S.) as a foundation for this study. Except for several large, low grade iron deposits, these survey revealed only a few scattered mineral deposits which, given their size and grade, could be considered as likely candidates for development within the next ten years. Other than these, any major mining developments in the region will have to come from new discoveries or from favorable results in proving up inadequately developed "showings" now known.

This basic finding was confirmed by Battelle scientists in conversations with private company executives and government minerals experts.

The responsibility of Sores-Battelle in its second repport is to provide the James Bay Development Corporation with a preview of the development possibilities for the region. For forestry, this preview could be developed based upon the quantification of a known measurable resource. For minerals, this is only partially true in that it was necessary to attempt to appraise resources largely presumed to exist, but which are associated with unknown locations, unknown quantities and unknown qualities. In order to impute some meaning and logic to such a set of virtual unknowns, there are three analytic elements to the second phase study of mineals. First, it was necessary to assess the likelihood of new mineral discoveries of commercial value. Second, market analyses were carried out to determine whether the demand for the minerals would be sufficient to make new deposits marketable. Third, a set of hypothetical projects was developed to analyze the cost factors involved in developing the various ore types expected to be found in the region. A combination of the above could then provide a reference point and a set of guidelines to the James Bay Development Corporation for its future planning of mineral exploration and exploitation.

ASSUMPTIONS CONCERNING EXISTING OPERATIONS

Based upon the judgments of S.G.S. and conversations with mining company and government officials, the development scenario for this study was based on the assumption that mining and milling in the Matagami and Chibougamau areas would continue at approximately their same level for the duration of this planning period (to the year 2000). Some existing operations will be mined out before that time, but it is expected that new "replacement" deposits in the area will soon be opened up. These deposits will provide feed stocks for the existing mills and employment for the same number of miners.

ASSUMPTIONS CONCERNING THE POSSIBILITY FOR NEW DISCOVERIES OF COMMERCIAL VALUE

The geological consultants (S.G.S.) prepared a geologic map of the region and an inventory of mineral deposits and showings within the region, as recorded in the files of the Federal and Provincial Governments. The basic use for this information is to assist the James Bay Development Corporation in the development of an exploration program. Sorès-Battelle also used S.G.S.'s summary judgments as an input to its regional planning analyses. In the Sorès-Battelle Phase I report, it was concluded that, for the known deposits which could be defined by tonnage and grade, few showed significant promise for development within the next ten years. Therefore, any new development would have to be based upon discovery of new deposits. For this reason, S.G.S. undertook a special study to provide estimates of the likelihood of finds of basic metals and uranium of specified tonnage and grades within the Superior province of the Canadian Shield. These estimates were used to provide an assessment of the mineral potential of the James Bay region.* The S.G.S. special study had three major components:

1. First, S.G.S. developed estimates of undiscovered mineral wealth for base metals and uranium in the James Bay region. These estimates were calculated independent of market demand and independent of the economics of extraction. Based on the average mineral wealth per square mile of the earth's land areas, these estimates would justify an expectation of an average mineral wealth of $200,000 of exploitable minerals per square mile in the James Bay region. Multiplying this by the area of the region, 130,000 square miles approximately, one arrives at a gross estimate of mineral wealth in the region of $26 billion.

   As the value of the known deposits in the James Bay region approaches $10 billion, approximately $16 billion worth of minerals remains to be found.

2. Second, S.G.S. disaggregated the total $16 billion into ten basic rock types, for which estimates of the distribution of deposit size and grade were generated. For example, for volcanogenic Cu-Zn deposits, S.G.S. estimates that 50 percent of deposits likely to be found have a 50 percent probability of containing at least 2 million tons of ore. Similarly, in terms of grade, it is estimated that 67 percent of all deposits will have a value per ton of between $20 to $40 per ton, and so on. These estimates were then used to identify apparent threshold combinations of tonnage and grade for the principal rock types.

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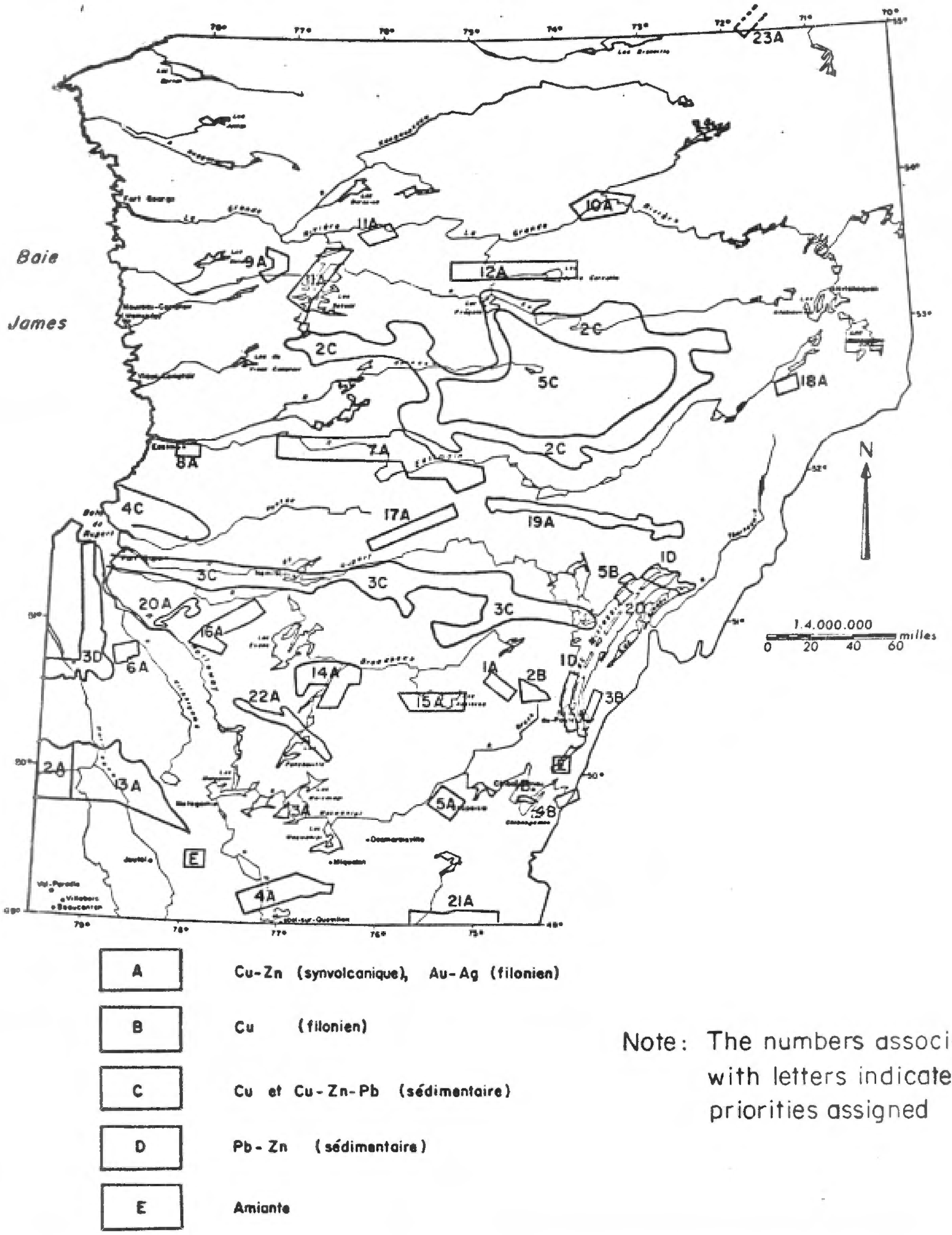
* Services Géotechniques Shickshocks, Etude de la Géologie et du Potential Minéral du Territoire de la Baie James. Dorval, Québec, Ch. 7.

3. The third component was to identify the most promising zones for further exploration and assign an expected value of finds and priorities to these zones. The latter are illustrated in Figure 1 and Figure 2. Letters are used to identify zones in which certain types of rock are found and, by geologic association, certain minerals. The numbers accompanying the letter identifications establish a rough order of priority for future exploration activity. The criteria for priorities included: (1) an assigned apportionment of the "undiscovered mineral potential" of the region for the host rock type; (2) the surface area of the zones; and (3) professional geologic judgment that commercializable deposits could be discovered in the zone by exploration. Sorès-Battelle selected certain of the zones as sites for the hypothetical projects on the basis of priority and location within the region. It is not possible at this time to make any distinction between the zones concerning the probability of a find, its grade or tonnage. Similarly, within the zones, there is an equal chance of finding a deposit in any given square mile of the zone. Additionally, there are no defined directional tendencies associated with the probabilities with the exception that belts of rock with high potential generally get thinner as one moves north.

MARKET STUDIES

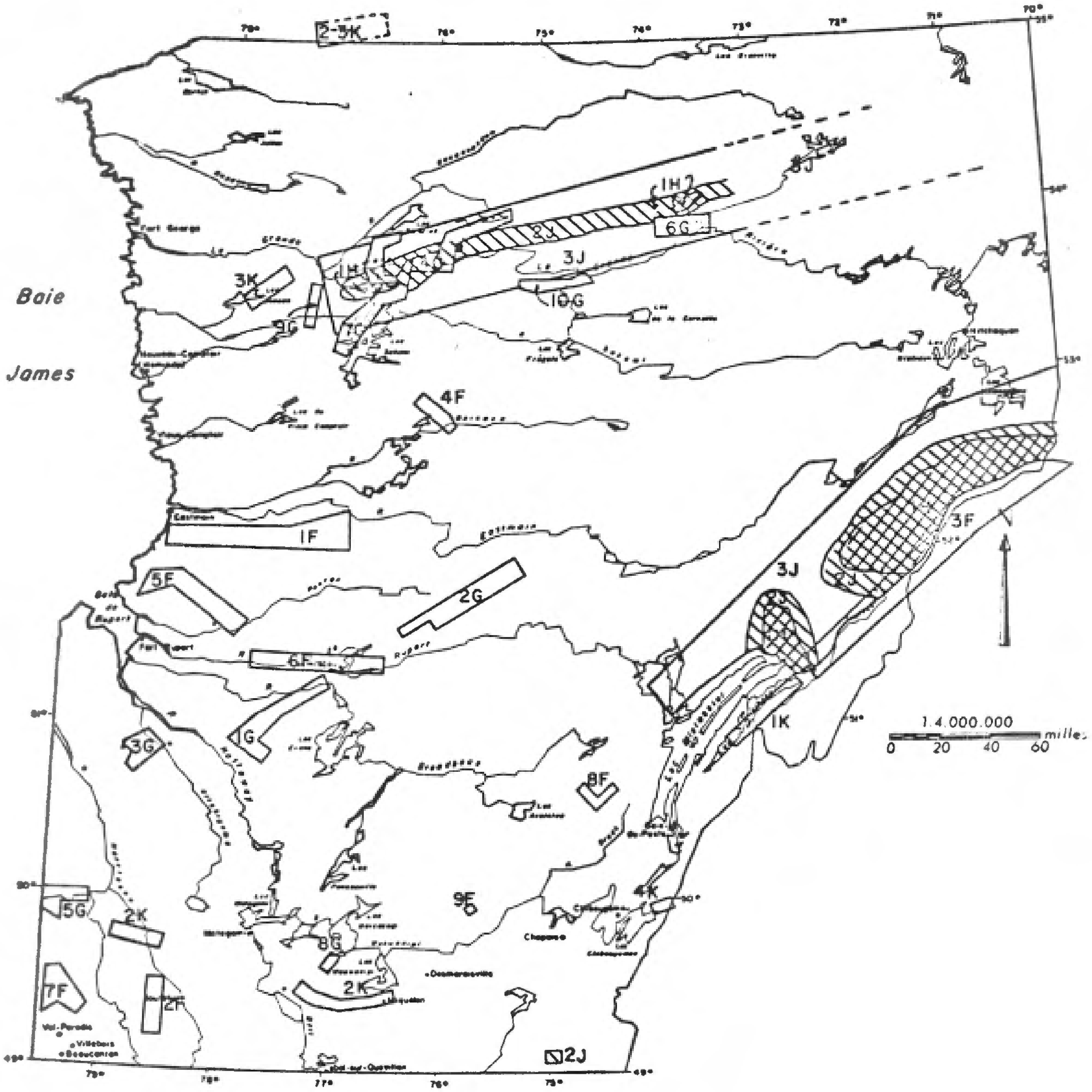
Based upon the work of S.G.S. and knowledge of existing mining operations in the southern part of the region, the James Bay Development Corporation can expect that an exploration program would identify commercially valuable deposits of nickel, uranium, iron, copper, zinc, and asbestos, with at least accessory values of lead, gold, and silver.

For each of the above, market studies were carried out to provide the James Bay Development Corporation with projections of future changes in world and North American markets and how these might influence the possibilities for exploitation of known deposits or deposits expected to be found in the James Bay region. In particular, knowledge of market trends (and problems) provides the necessary background for incorporating a timing dimension into proposed projects.



Note: The numbers associated with letters indicate the priorities assigned

FIGURE I. FAVORABLE ZONES FOR EXPLORING FOR COPPER, ZINC, LEAD, GOLD, SILVER, AND ASBESTOS IN THE JAMES BAY REGION



F	Ni	(roches mafiques)
G	Ni	(roches ultramafiques)
▨ H	U	(sédimentaire)
▨ J	U	(filonien ou pegmatitique)
K	Fe	

Note: The numbers associated with letters indicate the priorities assigned

FIGURE 2. FAVORABLE ZONES FOR EXPLORING FOR NICKEL, URANIUM AND IRON IN THE JAMES BAY REGION

HYPOTHETICAL PROJECTS

The second part of the Phase II Sorès-Battelle mineral studies has been to develop hypothetical projects which illustrate the key economic and logistic factors that will influence the exploitation of the most likely kinds of ore finds. Illustrative projects have been developed for the mining and milling of ores of uranium, nickel-copper, copper-zinc, asbestos and iron. In the cases of iron and asbestos, the illustrative projects described are based upon known, but presently unexploited ore bodies. In the cases of the other ores, locations were selected based upon the mineral potential maps developed by S.G.S. For all projects descriptions, however, the purposes for their inclusion are the same: to provide a preliminary point of reference for evaluating undiscovered deposits; and to provide useful guidance for an exploration strategy.

In developing the hypothetical projects, an attempt was made to be on the conservative side when choices were available concerning cost factors, geologic situations and tonnages and grades. The purpose of this was to develop an example that would be close to the margin of feasibility. It was not viewed as useful to develop the hypothetical projects around assumed rich deposits. If a $5 billion deposit is found, it will not present any very troublesome questions on the feasibility of development. It will be developed. The concern for this study was to offer guidance to the James Bay Development Corporation on the thresholds of feasibility for exploiting the more common, medium-sized mineral deposits in terms of tonnage and grade.

In developing these hypothetical projects, Sorès-Battelle has made assumptions on the tonnage, grade, geological setting, mining and milling procedures, capital, and operating costs connected with the project. Assumptions have also been made on their location and their relationship to infrastructure and townsites in the region. When actual deposits are located, they can be compared to the assumptions of the hypothetical project. With poor finds or more distant locations, one could expect an operation less profitable than the one set up as the archetype. Richer finds could be expected to generate a higher rate of return, as well as more certain and quicker development into commercial operations.

Obviously no one would proceed to develop a mine or make a decision on the exploitation of a deposit based on these hypothetical studies. They provide guidelines only. When actual deposits are located, they must be evaluated in terms of the specific costs associated with their development and the specific marketing outlook for the firm owning the rights to the deposit.

Each of the projects illustrates the development of a combined mine and mill. In some cases, two locations have been selected to illustrate two major ore types for the same metal. In no case is processing carried beyond milling as, e.g., one Noranda smelter would require the output of at least ten mines of the kind hypothesized for copper and zinc. To develop a smelter within the James Bay region would require a combination of circumstances:

1. The discovery and development of several ore bodies in the 60 to 100 million ton range
2. Expanding markets which would result in regular, full utilization of existing smelter capacity in Eastern Canada
3. Favorable costs within the region, particularly in the case of labor cost and reductant cost.

This combination cannot be foreseen at the present time. However, developments such as the location of natural gas trunk pipelines through the region, or a major, rich ore find could bring about an alternation of this judgment.

Finally, each project has been subjected to examination of the sensitivity of selected cost or revenue elements to develop a "profitability profile". These calculations are based on a technique known as Probabilistic Discounted Cash Flow Analysis. This analytic technique is described more fully in the attachment at the end of this document.

CHAPTER 1. URANIUM

CHAPTER 1. URANIUM

SUPPLY/DEMAND SITUATION TO THE YEAR 2000

Occurrence and Beneficiation

Occurrence

Uranium is one of the less common elements, with a concentration of four parts per million in the earth's crust. It is more abundant than gold or silver but slightly less abundant than nickel, copper, lead, and zinc. Uranium is widely distributed throughout the world, but most of the large deposits have a uranium content of only about 0.1 percent. It is always found in chemical combination with other elements, forming more than 100 known minerals. The most commonly mined uranium minerals are pitchblende or uraninite, UO_2; brannerite, $(U,Ca,Fe,Th,Y)_3\ Ti_5O_{16}$; uranothorite, $(U.Th)O_2$; carnotite, $K_2(UO_2)_2(VO_4)_2 \cdot 3H_2O$, and uranophane, $CaO \cdot 2UO_3 \cdot 2SiO_2 \cdot 6H_2O$. These minerals occur as separate grains or in veins in different types of rocks.

In Canada, all deposits of the commercial kind have been found in Precambrian rocks in the Canadian Shield. These deposits are of three distinct types: conglomeratic, vein, and pegmatitic. Table 1-1 gives details of the geological characteristics of important Canadian uranium deposits.

Of the deposits listed in Table 1-1, the most productive type comprises pyritiferous quartz-pebble conglomerates that occur in the Elliot Lake-Blind River area of Ontario.

Pitchblende-bearing vein and replacement deposits are also important. Deposits of this type are located at Uranium City (including the Beaverlodge and Gunnar deposits), Rayrock, and Rabbit Lake.

Less productive have been vein and replacement deposits of complex mineralogy of which the Eldorado mine at Port Radium was the only large producer.

Significant production has also been obtained from pegmatitic deposits in the Bancroft area of Ontario.

Beneficiation

Uranium ore is usually beneficiated at or near the mine site to yield U_3O_8 concentrates or yellowcake. The usual steps in this process are:

(1) Crushing, grinding, classifying of ore

(2) Separation of ore from gangue by lump ore sorting or gravity concentration and flotation

TABLE 1-1. GEOLOGICAL CHARACTERISTICS OF IMPORTANT CANADIAN URANIUM DEPOSITS

DEPOSIT OR DISTRICT	Elliot Lake Blind River	Agnew Lake	Eldorado (Port Radium) and No. 2 vein Echo Bay mine	Rayrock (Marian River)	Uranium City (Beaverlodge area)	Gunnar (south of Uranium City)	Rabbit Lake	Makkovik area	Bancroft area
CLASSIFICATION OF DEPOSIT(S)	placer (quartz-pebble conglomerate)	placer (quartz-pebble conglomerate)	vein and replacement	vein	vein	replacement	replacement	vein and replacement	pegmatite
URANIUM MINERALS	brannerite uraninite uranoan monazite minor thucholite coffinite uranothorite gummite uranothorianite	uranothorite monazite	pitchblende	pitchblende	pitchblende	pitchblende uranophane	Massive and sooty pitchblende uranophane	pitchblende minor soddyite kasolite	uranothorite uraninite minor uranophane cyrolite thorite
ASSOCIATED MINERALS Metallic	pyrite minor hematite magnetite zircon ilmenite sphene anatase rutile	pyrite allanite sphene	Co–Ni arsenides hematite pyrite sphalerite tetrahedrite bornite chalcopyrite galena silver minerals bismuth	hematite minor pyrite and chalcopyrite	hematite minor sulphides selenides nolanite	hematite minor sulphides	pyrite some hematite minor galena and sphalerite	hematite pyrite	magnetite pyrite allanite
Gangue	apatite muscovite chlorite	apatite chlorite	quartz carbonates	quartz epidote	calcite chlorite quartz oligoclase	calcite dolomite quartz chlorite kaolin	dolomite quartz kaolin sericite	quartz calcite feldspar apatite	quartz feldspar pyroxene locally gypsum or anhydrite
HOST ROCKS	Basal Huronian (Early Proterozoic): quartz-pebble conglomerate feldspathic quartzite	Basal Huronian (Early Proterozoic): quartz-pebble conglomerate feldspathic quartizite	Echo Bay Group (Archean or Proterozoic): quartzite tuff chert argillite limestone conglomerate	granodiorite (Proterozoic): quartz 'epidote rock'	Tazin Group (Archean, possibly Proterozoic): quartzite amphibolite gneiss metasomatic granitic rocks	Tazin Group (Archean possibly Proterozoic): syenite granite gneiss paragneiss	(Archean or Early Proterozoic): gneiss quartzite marble arkose argillite calc-silicate rocks	Aillik Group (Early Proterozoic): argillite tuffaceous beds porphyroblastic quartzite	Grenville Group (Late Proterozoic) amphibolite gneiss quartzite marble granulite
HYDROTHERMAL ALTERATION	local chloritization and albitization	chloritization silicification	hematitization carbonatization chloritization sericitization argillization	hematitization silicification chloritization	hematitization carbonatization silicification feldspathization	hematitization carbonatization silicification chloritization albitization	carbonatization silicification argillization sericitization chloritization little hematization	hematite carbonatization silicification feldspathization	hematite albitization
STRUCTURAL RELATIONSHIPS	Conglomerates truncated by unconformity at edges of basins Conglomerates are progressively younger northward	Conglomerate resting unconformably upon granite, and dipping vertically	Vein deposits with wall rock replacement in main, northeasterly-trending fault and branching faults	Stockwork of veins in northeasterly-trending shear zone	Ore deposits are in and adjacent to strong northeasterly-trending faults, and often concentrated at fault intersections	pipe-like orebody near intersection of northeasterly- and easterly-trending faults	irregular lenticular body in wedge of brecciated rocks between northeasterly-trending and intersecting faults	vein deposits with wall rock replacement along northeasterly-trending faults	pegmatite bodies in metamorphic rocks, structural control variable
AGE (million years)	2250 to 2400	Not determined	1450	Not determined	1780, 1110, 270 and 50	Not determined	Not determined	600 (one determination)	950 to 1070 (one 600)

(3) Extraction by either an acid leach or an alkaline leach

(4) Concentration and purification by ion exchange, solvent extraction, or by extraction direct from the leach pulp

(5) Chemical precipitation of the U_3O_8

(6) Drying and packing.

Manufacture of Final Product

The ultimate use of the yellowcake is the production of reactor fuels--usually uranium metal for natural uranium and heavy water reactors, or enriched UO_2 for light water reactors.

To obtain uranium metal, the U_3O_8 is converted to UO_2 which is fluoridized to obtain UF_4. The UF_4 is blended with magnesium and bomb-reduced to form uranium metal, which is then vacuum melted and cast into ingots.

In the case where enriched UO_2 is desired, the U_3O_8 is converted to UO_3, the UO_3 is fluoridized in two different steps to UF_6. To enrich the uranium, that is to increase the uranium - 235 content, the UF_6 is processed in a gaseous diffusion plant. After enrichment of the UF_6, the product is converted to UO_2 by a two-step process.

Uses and Substitutes for Uranium

Uses for Uranium

Though uranium and its compounds are used in non-energy applications, by far its largest use is to fuel nuclear reactors. Uranium is unique as it is the only naturally occurring element which is readily fissionable on exposure to low energy neutrons. Actually, the uranium isotope, uranium-235, is the component of uranium which is fissionable. Natural uranium contains only 0.71 percent uranium-235, the remainder being mainly uranium-238. Most of the growth in uranium's use as a reactor fuel will be as a fuel for enriched uranium reactors of the light water type. In these fuels, uranium-235 content is 2 to 4 percent of the total uranium present.

Substitutes for Uranium

Although uranium is the only naturally occurring element which is fissionable, other man-made isotopes may substitute or partially substitute for uranium-235 as a reactor fuel. These are plutonium-239 and uranium-233. Both are produced in nuclear reactors and result from the irradiation of uranium-238 and thorium-232, respectively. The source of these substitute fissionable materials is the spent fuel of nuclear reactors. Currently, the reclamation of plutonium-239 from spent reactor fuels for reuse in fueling commercial nuclear reactors is in its infancy. However, in the mid-seventies and beyond, plutonium recycle is expected to lower uranium demand by about 4 percent.

Currently, only several reactors are in use which produce uranium-233. These reactors (high temperature, gas-cooled or HTGR types) utilize thorium as part of their fuel. There are no commercial size facilities for reprocessing the uranium-233 produced, and because of the lack of significant current orders for HTGR reactors it is unlikely that uranium-233 will seriously compete with mined uranium for many years.

The most serious competition for uranium markets is expected from the development of the fast breeder reactor (FBR). This reactor is designed to be fueled with plutonium and depleted* uranium. The FBR is designed to produce more plutonium than it consumes. If the FBR is a commercial success, in time these reactors will be producing fuel which will compete with mined uranium in many reactor applications. More importantly, the FBRs are fueled with plutonium and their use for power production will stymie the growth of uranium reactors and the need for uranium. Thus, late in this century, it is expected that the growth in uranium demand will subside and at some point demand may actually decline.

Free World Sources of Uranium

The resources of uranium, current production rates, and current and projected capacities are reviewed in this section for the Free World. Major supply countries are reviewed in detail.

Much of the data obtained are given in the ENEA/IAEA report issued in September, 1970**, and is current through April, 1970. However, new data available from open literature sources, were utilized to update the findings.

Free World Summary

Resources. Table 1-2 presents the Free World's U_3O_8 resources as described by the four different categories reported by the ENEA/IAEA: (1) reasonably assured resources recoverable at less than \$10 per pound (i.e. reserves), (2) estimated additional resources recoverable at less than \$10 per pound, (3) reasonably assured resources recoverable at \$10 to \$15 per pound, and (4) estimated additional resources recoverable at \$10 to \$15 per pound. With the exception of South Africa, uranium reserves that may be recoverable as byproducts are included. In the case of South Africa, uranium is currently recovered as a byproduct of gold mining, and therefore, these reserves are included.

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*Uranium that has only about 0.25 percent Uranium-235 content. Depleted uranium is obtained from uranium enrichment tails and spent uranium fuel. The supply of depleted uranium is likely to be sufficient to fuel FBR's well into the 21st century.

**"Uranium-Resources, Production, and Demand", September, 1970, a joint report by the European Nuclear Energy Agency and the International Atomic Energy Agency.

TABLE 1-2. FREE WORLD URANIUM RESOURCES(1)
(Short Tons of U_3O_8)

Country	Recoverable at or below $10 per Pound of U_3O_8		Recoverable at $10-$15 per Pound of U_3O_8	
	Reasonably Assured Resources*	Estimated Additional Resources**	Reasonably Assured Resources*	Estimated Additional Resources**
United States	390,000(2)	680,000	140,000	300,000
South Africa	300,000(3)	15,000	65,000	35,000
Canada	232,000	230,000	130,000	170,000
Australia	156,000	6,700	9,200	6,600
France	45,000	25,000	9,000	15,500
Other	96,400	96,000	456,400	153,300
	1,219,400	1,052,700	809,600	680,400

SOURCE: Compiled by Battelle's Columbus Laboratories (BCL)

Notes:

(1) In addition to the reserves listed, U_3O_8 in government stockpiles is estimated as follows: United States - 50,000 tons; Canada - 8,500; South Africa - 8,000; France - 2,000; United Kingdom - 21,000; Australia - 2,000 for a total of 91,500 tons.

(2) Includes 90,000 tons recoverable by 2000 AD as by-product from copper and phosphoric acid production.

(3) Includes approximately 240,000 tons recoverable as a by-product from gold mining and approximately 60,000 tons of uranium reserves in South-West Africa.

* The term Reasonably Assured Resources refers to uranium which occurs in known ore deposits of such grade, quantity and configuration that it can, within the given price range, be profitably recovered with currently proven mining and processing technology. Estimates of tonnage and grade are based on specific sample data and measurements of the deposits and on knowledge of ore-body habit. Reasonably assured resources in the price category below $10 per pound are equivalent to Reserves in the mining sense.

** The term Estimated Additional Resources refers to uranium surmised to occur in unexplored extensions of known deposits or in undiscovered deposits in known uranium districts, and which is expected to be discoverable and economically exploitable in the given price range. The tonnage and grade of Estimated Additional Resources are based primarily on knowledge of the characteristics of deposits within the same districts.

The resources reported are essentially those recognized by the ENEA/IAEA report, with three major exceptions. These are as follows:

(1) The latest available U.S. Atomic Energy Commission (USAEC) data were included

(2) South African reserves were upgraded as per official announcements, to include the South-West Africa uranium find and increased uranium in gold ore reserves, and

(3) Australian reserves were upgraded based on analyses of individual company data on recent important uranium finds.

Table 1-2 shows Free World reserves of 1,219,400 tons of U_3O_8 recoverable at less than \$10 per pound. Market prices for U_3O_8 have been significantly below \$10 per pound, in fact current prices are roughly \$6 per pound. Because of this, there has been very little commercial interest in evaluating uranium deposits likely to be exploitable at \$8 to \$10 per pound. Consequently, it is felt that the reserve figure is conservative.

Production and Future Potential

The 1970 production of U_3O_8, current production capability, and estimated future production capability are given in Table 1-3 by country. Attainable production capability is estimated to 1980. Production capability beyond that period will depend on many factors and is difficult to estimate. Further, it is presumed that additional production capacity will appear if markets and resources permit.

In 1970, Free World production of U_3O_8 totaled 23,707 tons or about two-thirds of the production capacity then active. This reflects the current soft market situation. This situation is the result of maintaining U_3O_8 facilities which were originally producing to meet military demand until civilian nuclear energy demand took over. Though many uranium mines and mills closed as a result of the lack of demand in this period of change, many mines and mills survived as a result of government purchases which were stockpiled for later release. This is particularly true in the United States and Canada.

The attainable Free World U_3O_8 production capacity for 1972 through 1980 is given in Table 1-3. It is possible that the 1972 capacity of over 35,000 tons could be based on known plans and available reserves. Potential production capability beyond 1980 will depend greatly on uranium reserves discovered during the period 1972 to 1980.

Canada. Canada is currently the second largest U_3O_8 producer in the Free World. In terms of reasonably assured reserves recoverable at less than \$10 per pound, Canada is third after the United States and South Africa. However,

TABLE 1-3. FREE WORLD U_3O_8 PRODUCTION, CURRENT AND ATTAINABLE, 1970-1980

Country	Actual Production 1970	Attainable Production Capacity, Short Tons of U_3O_8 1972	1974	1976	1978	1980
United States	12,768	19,000	19,000	23,000	30,000	37,000
South Africa	4,119	4,900	6,000	8,000	8,000	13,000
Canada	4,011	7,300	7,300	14,130	14,130	17,500
France	1,627	1,750	1,750	2,690	2,340	2,340
Australia	330	--	4,000	8,500	10,500	12,500
Other	852	2,171	3,381	4,498	4,560	4,560
Totals	23,707	35,121	41,431	60,818	69,530	86,900

SOURCE: U. S. Bureau of Mines IAEA/ENA Report and BCL Estimates.

since much of South Africa's reserves are available only as by-product of gold mining, Canada is likely to maintain its number two production status for a number of years.

Resources. Canadian U_3O_8 resources were last reported on a national basis in 1970 by the Department of Energy, Mines and Resources, as follows:

Reasonably assured resources recoverable at less than \$10 per pound (reserves)	232,000 tons
Estimated additional resources recoverable at less than \$10 per pound	230,000 tons
Reasonably assured resources recoverable at \$10-\$15 per pound	130,000 tons
Estimated additional resources recoverable at \$10-\$15 per pound	170,000 tons

Of the reasonably assured resources recoverable at less than \$10 per pound (reserves), 80 percent are located in quartz-pebble conglomerates of Huronian age in the Elliot Lake and Agnew Lake areas of Ontario. Most of the remainder are pitchblende-bearing vein-type deposits which occur primarily in Saskatchewan, British, Columbia, the Northwest Territories and Labrador. Small amounts of low-cost reasonably assured resources occur in pegmatites and are limited to the Bancroft area of Ontario.

Production and future potential. The status of Canadian uranium mines and the mills which serve them is given in Table 1-4. Currently, production at the Denison Mines and Rio Algom mines at Elliot Lake accounts for over 80 percent of Canada's production. El Dorado Nuclear operates mines and a mill in the Beaverlodge area of Saskatchewan and accounts for the remaining production. In 1970, U_3O_8 production was 4,011 tons; in 1971 it was approximately 4,100 tons. A portion of the production was stockpiled for future sale.

Two mining and milling projects may be considered as firmly planned. It is likely that the first to come on stream will be the Gulf Mineral Corporation and Uranerz-Bonn project at the Wallaston Lake area in Saskatchewan. First production is planned for 1975 and construction at the mine site is in progress. It is estimated the mill will be capable of a throughput of 2,000 tons of U_3O_8 per year. Another project, in the Agnew Lake area of Ontario, was under development by Agnew Mines, Ltd. This project was delayed due to lack of current markets. It will be activated when uranium demand picks up. It is estimated that the total output of the planned mill would have been 1,200 tons of U_3O_8 per year.

Table 1-4 lists inactive mines and mills which have not been dismantled, and which could again be reactivated should markets permit. In addition to those listed, a property owned by Stanrock Uranium Mines in the Elliot Lake Region

TABLE 1-4. CURRENT STATUS OF CANADIAN URANIUM MINES AND MILLS

Location	Company	Mine	Mill	Milling Capacity, Short Tons of Ore/day	Estimated U_3O_8 Production Capabilities Short Tons U_3O_8/year	Approximate Average Grade of Ore, pounds of U_3O_8 per Short Ton	Remarks
Operating							
Elliot Lake Region	Denison Mines Ltd.	Denison Mine	----	6,000(3)	3,000(3)	2.8	Operating at about 2/3 of capacity
	Rio Algon Mines Ltd.	New Quirke (1)	Quirke	6,200(4)	3,100(4)	2.9	
Beaverlodge Area, Saskatckewan	El Dorado Nuclear Ltd.	Fay & Hab Mines	----	2,000	1,200	2.8	Operating at about 1/2 of capacity
				Sub Total	7,300		
Planned							
Rabbit Lake Wallaston Lake Area, Saskatckewan	Gulf Minerals Corp. and Uranerz-Bonn	----	----	2,000	2,000	7.0	First production planned for 1975
Agnew Lake Area, Ontario	Agnew Mines Ltd.	Faraday Mine	----	3,000	1,200	3.0	Construction plans for mill delayed
				Sub Total	3,200		
Inactive(2)							
Elliot Lake	Rio Algom Mines Ltd.	Nordic Mine	Nordic	3,700	1,000	2.1	Can reopen if markets develop
		Panel Mine	Panel	3,000	820	---	
	Preston Mines Ltd.	Stanleigh Mine	----	3,000	820	---	
Bancroft, Ontario	Can-Fed Resources Ltd.	----	----	1,500	400	2.0	
				Sub Total	3,040		
				Grand Total	13,540		

Source: Canadian Minerals Yearbooks, 1970 IAEA/ENEA Report.

(1) Rio Algom also owns other mines in the Elliot Lake area which could be activated if markets develop.

(2) Capacities listed refer to capabilities at time of closure, other past producing mills have been dismantled.

(3) Design work completed for expansion to 7,500 tons of ore per day equivalent to 3,750 tons of U_3O_8 per year, expansion expected after 1975.

(4) Expansion planned to a capacity of 6,500 tons of ore per day expected after 1975. This is equivalent to 3,350 tons of U_3O_8 per year.

ceased production in 1970. Stanrock's total production was from leaching of an abandoned uranium property. Production rates were very low.

The current production capacity of Canadian mines in operation is estimated to be 7,300 tons of U_3O_8. Attainable annual production by 1976 is 14,300 tons of U_3O_8 if inactive mines are re-opened. Production rates could probably be expanded by 1980. However, unless new uranium resources are found, it is not likely that a 17,500 ton per year rate would be exceeded.

United States. The United States is the World's largest producer of U_3O_8 and vast reserves are available. A major factor in the maintenance of this number one production position has been the policy of the United States Atomic Energy Commission (USAEC). By restricting the enrichment of foreign uranium to use in foreign reactors, the AEC has essentially prevented foreign competition with the United States uranium industry.

Resources. United States U_3O_8 resources reported by the USAEC in May, 1971, were as follows:

Reasonably assured resources, recoverable at less than $10 per pound (reserves)	300,000 tons
Estimated additional resources, recoverable at less than $10 per pound	680,000 tons
Reasonably assured resources, recoverable at $10-$15 per pound	140,000 tons
Estimated additional resources, recoverable at $10-$15 per pound	300,000 tons.

By the year 2000, about 90,000 tons of U_3O_8 could be recovered at less than $10 per pound as a by-product of copper and phosphoric acid production. Thus, this by-product production potential can be considered as additional reserves, although recovery is not currently being practiced.

Production and future potential. Table 1-5 presents the current status of United States uranium mills. Many of these mills are served by more than one mine. In 1970, a total of 263 mines were operating and U_3O_8 production was 12,768 tons, while capacity stood at 19,000 tons of U_3O_8 per year. This capacity could be expanded to an estimated 23,000 tons by 1975, if needed. Capacity attainable by 1978 and 1980 is roughly estimated to be 30,000 and 37,000 tons of U_3O_8 per year, respectively.

South Africa*. South Africa is the Free World's third largest producer of uranium. Current U_3O_8 production is almost entirely derived as a by-product of gold production. Primarily, U_3O_8 deposits exist in South-West Africa and are a likely source of uranium in the late 1970's.

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*Including the territory of South-West Africa.

TABLE 1-5. STATUS AND CAPACITY OF UNITED STATES URANIUM MILLS (1)

Company	Plant Location	Capacity, Short Tons of Ore per Day	Estimated 1972 Capacity, Short Tons of U_3O_8 per year
The Anaconda Co.	Bluewater, N. Mex.	3,000	
Atlas Corp.	Moab, Utah	1,500	
Continental Oil Co. Pioneer Nuclear, Inc.	Karnes County, Tex.	1,750 (2)	
Cotter Corp.	Canon City, Colo.	450	
Dawn Mining Co.	Ford, Wash.	500	
Federal-American Partners	Gas Hills, Wyo.	950	
Humble Oil and Refining Co.	Powder River Basin, Wyo.	2,000 (2)	
Kerr-McGee Corp.	Grants, N. Mex.	7,000	
Mines Development, Inc.	Edgemont, S. Dak.	650	
Petrotomics Co.	Shirley Basin, Wyo.	1,500	
Rio Algom Mines, Ltd.	Moab, Utah	500 (2)	
Susquehanna Corp.	Falls City, Tex.	1,000	
	Ray Point, Tex.	1,000	
Union Carbide Corp.	Uravan, Colo.	2,000	
	Gas Hills, Wyo.	1,000	
United Nuclear-Homestake Partners	Grants, N. Mex.	3,500	
Utah Construction and Mining	Gas Hills, Wyo.	1,200	
	Shirley Basin, Wyo.	1,200	
Western Nuclear, Inc.	Jeffrey City, Wyo.	1,200	
Total		31,900	19,000

Source: United States Atomic Energy Commission.

(1) Does not include mill of American Metal Climax, Inc., at Grand Junction, Colo., and mill of Union Carbide Corp., at Rifle, Colo., which were closed in 1970-1972.

(2) Under construction; planned completion in 1972.

Resources. In 1971, the South African Atomic Energy Board announced that U_3O_8 reserves available at less than $10 per pound had increased to 300,000 tons. The increased reserve position resulted from new discoveries in South African and South-West African, as well as from achievement of higher yields from improved extraction processes. U_3O_8 reserves are listed as follows:

Reasonably assured resources, recoverable at less than $10 per pound (reserves)	300,000 tons
(It is estimated that 60,000 tons is available from uranium deposits in South-West Africa and that 240,000 tons is available from by-product production.)	
Estimated additional resources recoverable at less than $10 per pound	15,000 tons
Reasonably assured resources, recoverable at $10-$15 per pound	65,000 tons
Estimated additional resources, recoverable at $10-$15 per pound	35,000 tons

South Africa has extensive low-grade deposits of uraninite associated with gold, pyrites, and small quantities of othermetallic sulphides in the Republic of South Africa. They are found in conglomerates of four contiguous Precambrian systems: Dominion Reef, Witwatersrand, Ventersdorp, and Transvaal Systems. The conglomerates at several horizons in the Witwatersrand System are the principal sources of uranium that is recovered as a by-product of mining operations in the Witwatersrand, Kherksdorp, and Orange Free State gold fields. These ore bodies account for upwards of 90 percent of the known South African uranium resources. The balance consists of uranothorianite in carbonatite. The South-West African deposits have not been characterized. Currently, nearly all South Africa's uranium is recovered as a by-product of gold production. A small amount of uranium is also recovered from tailings generated from copper mining. Some 11 gold producers and one copper producer recover uranium. Production of U_3O_8 totaled 4,119 tons in 1970. Current annual capacity is rated at 4,900 tons of U_3O_8. If needed, by-product capacity of 6,000 tons of U_3O_8 per year can be attained by 1974 and 8,000 tons of U_3O_8 by 1980.

In South-West Africa, a uranium deposit at Rossing is under development. This deposit is massive, but uranium content is low, about 2 pounds per ton of ore. Rio Tinto Zinc Corporation, Ltd., and the Industrial Development Corporation of South Africa, Ltd., will mine this deposit by open-pit technique. Data regarding this project are not available. However, contracts for uranium delivery beginning in 1976 have been reported, and contractors for the mine and mill have been chosen. Preliminary reports set the mill capacity at a minimum of 1,000 tons of U_3O_8 per year, with some sources quoting a 5,000-ton mill. For the purposes of this report, it is assumed that the Rossing project will be capable of producing 2,000 tons of U_3O_8 per year by 1976, and 5,000 tons per year by 1980.

Total U_3O_8 production capability attainable by South Africa will likely expand as follows:

1972 - 4,900 tons (all by-product)
1974 - 6,000 tons (all by-product)
1976 - 8,000 tons (6,000 tons by-product and 2,000 tons primary)
1978 - no change
1980 - 13,000 tons (8,000 by-product and 5,000 tons primary).

France. France is the smallest of the top four Free World uranium producers. Uranium reserves are small and it is unlikely that significant increases in U_3O_8 production from French mines will occur. Outside France, CEA (the French Atomic Energy Commission) holds an interest in uranium deposits in Gabon and Niger. Uranex, a firm owned by several private companies and the CEA was formed to market U_3O_8 not needed by France.

Resources. France's uranium situation was last evaluated in 1970. Resources are as follows:

Reasonably assured resources, recoverable at less than \$10 per pound (reserves)	45,000 tons
Estimated additional resources, recoverable at less than \$10 per pound	25,000 tons
Reasonably assured resources, recoverable at \$10-\$15 per pound	9,000 tons
Estimated additional resources, recoverable at \$10-\$15 per pound	15,500 tons.

France's low-cost reserves occur in either vein-type deposits in crystalline rock or as stratiform deposits in vein-type deposits. About 80 percent of the low-cost reserves are in vein-type deposits.

Production and future potential. In 1970, France's production of U_3O_8 from domestic mines totaled 1,627 tons. Uranium mining and milling is performed by the CEA in mainly three districts:

LaCrouzille Division, Limousin
LaForey Division, near Roanne
Vendee Division, near Mantes.

Current and achievable mill capacity is given in Table 1-6 by mill location. Current capacity is 1,750 tons of U_3O_8 per year, with expansion to 2,690 tons possible by 1975. By 1978, the mines in the Forey sector will be depleted and it is not likely that the mill will be used. Thus, 1978 mill capacity will be 2,340 tons.

Australia. Australia currently is not a significant producer of uranium; however, the addition to reserves attained in recent years assures Australia a future position as a major world producer.

Resources. Official estimates of Australian resources are out of date. Since the 1970 report to the IAEA/ENA, large discoveries have been made. In 1907, Australia reported reasonably assured resources, recoverable at less than $10 per pound, of 21,700 tons of U_3O_8. Since that time, the following discoveries have been made and reserves can be estimated with some degree of accuracy. Table 1-7 gives the reported reserves estimated recoverable at less than $10 per pound of U_3O_8 by location and ownership. Small uranium finds or finds not well documented are not included. Using only the tentative data given in Table 1-7, Australia's current reserve position is as follows:

Reasonably assured resources, recoverable at less than $10 per pound (reserves).	173,600 tons
Estimated additional resources, recoverable at less than $10 per pound	22,000 tons*
Reasonably assured resources, recoverable at $10-$15 per pound	9,200 tons*
Estimated additional resources, recoverable at $10-$15 per pound	6,600 tons*.

The geological potential for additional discoveries of uranium in Australia is excellent.

Production and future potential. The only recently operated uranium mill in Australia (operated by the government at Rum Jungle, Northern Territory) was scheduled for closing in 1971. It produced U_3O_8 concentrates from previously mined ore. Though at least five other uranium mills operated at one time in Australia, only one is scheduled for reopening. That one, located at the Mary Kathleen mine, is expected to be in operation by 1974. Further details of likely openings of new mines and mills are given in Table 1-7. If markets permit, mill capacity is estimated to be attainable as follows: 1974 - 4,000 tons of U_3O_8, 1976 - 8,500 tons; and 1978 - 10,500 tons of U_3O_8. Reserves appear sufficient to permit capacity to reach 12,500 tons per year by 1980.

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*The same as reported by the IAEA/ENA in 1970, not updated because of lack of definative information.

TABLE 1-6. U_3O_8 PRODUCTION CAPABILITIES OF FRANCE

Sector	Present Production Capacity, Short Ton of U_3O_8 per Year	Possible Production Capacity, Short Tons of U_3O_8 per Year(1)	
		1975	1978
Crouzille	1,050	1,750	1,750
Forey	350	350	Deposit exhausted in 1978
Vandee	350	590	590
Total	1,750	2,690	2,340

Source: Nuclear Engineering International, August, 1972, p 693.

Note: (1) Estimated to be achieveable if markets are available.

Table 1-7. CURRENT STATUS OF THE URANIUM INDUSTRY IN AUSTRALIA

Location	Company	Mine	Milling Capacity, Short Tons of U_3O_8/Year	Estimated Reserves of U_3O_8 Recoverable at less than $10 per pound, Short Tons	Average Grade of Ore, Pounds of U_3O_8 per Short Ton	Remarks
			Planned Operations			
Near Mt. Isa	Mary Kathleen Uranium	Mary Kathleen	2,000	10,000	2.66	Will be brought back into production in late 1974. Mill is on stand-by status.
Alligator River District, Northern Territory	Peko-Wallsend Ltd. and Electrolytic Zinc Industries Ltd.	Ranger I	3,000	70,000	7.0(1)	Production scheduled to start July, 1976.
	Queensland Mines	Narbarlek	1,500(2)	10,500	47.0	Production to start in 1975. Ore may be milled at another Allegator River Mill.
	Noranda Australia	Jim Jim(2)	2,000(2)	20,000 est.	---	Mill in 1974 is a possibility. Size is estimated based on reserves.
Western Australia	Western Mining	Yeelirrie	2,000(2)	50,600	3.3(4)	Mining to start in late 1970's.
Lake Frome District, South Australia	Petromin N.L., Exoil N.L. and Transoil N.L.	Beverly	---	12,500 est.	9.0(1)	Property is not fully evaluated.
		Totals	10,500	173,600		
			In Recent Operation			
Rum Jungle, Northern Territory	Commonwealth Government	Rum Jungle	2,000	nil	---	Mining ceased in 1963, Mill was operated through 1971 or 1972.

Source: Compiled by BCL.

Notes: (1) Preliminary.

(2) Estimated by BCL.

(3) To be known in the future as Koongarra.

(4) Half of ore grades 8 pounds of U_3O_8 per short ton.

Other Free World Countries. As mentioned earlier, uranium occurrences are wide spread.

Uranium resources. Uranium resources of countries not previously mentioned are given in Table 1-8. Unless otherwise stated, the reserves are those given in the 1970 report by the ENEA/IAEA as follows:

Reasonably assured resources, recoverable at less than \$10 per pound (reserves)	96,400 tons
Estimated additional resources, recoverable at less than \$10 per pound	96,000 tons
Reasonably assured resources, recoverable at \$10-\$15 per pound	456,400 tons
Estimated additional resources, recoverable at \$10-\$15 per pound	153,300 tons.

The bulk of the reserves are in the African nations of Niger, Gabon, and the Central African Republic. These nations account for about half of the 96,400 tons listed above. The uranium projects in these countries are controlled by French interests.

The production of U_3O_8, current production capacity, and attainable production of the other Free World Countries are listed in Table 1-9. Production in 1970 totaled 852 tons of U_3O_8. Current capacity is 2,170 tons of U_3O_8, but this could expand rapidly and more than double by 1976. Capacity attainable by 1978-1980 is estimated at 4,560 tons of U_3O_8. It is not likely that this figure will be exceeded significantly unless major uranium deposits are found. Sweden could perhaps expand its uranium production significantly, if for reasons of self-sufficiency it is decided to exploit resources recoverable at \$10-\$15 per pound.

Free World Demand for Uranium

Free World demand for U_3O_8 to the year 2000 was derived from various short-term and long-term estimates for the United States and the rest of the Free World. These estimates were derived from forecasts of expected nuclear electrical generation capacity.

Summary of Free World Uranium Demand

The annual Free World demand for U_3O_8 to fuel power reactors is given in Figure 1-1. The annual demand at ten-year intervals is estimated to be as follows:

TABLE 1-8. URANIUM RESERVES OF OTHER FREE WORLD COUNTRIES

(Short Tons of U_3O_8)

Country	Reasonably Assured* Resources Recoverable at less than $10 per Pound of U_3O_8	Estimated Additional Resources Reasonable at less than $10 per Pound of U_3O_8	Reasonably Assured Resources Recoverable at $10-$15 per Pound of U_3O_8	Estimated Additional Recoverable at $10-$15 per Pound of U_3O_8	Remarks
Argentina	11,600	18,400	32,300	50,200	Reserves reported in 1972
Brazil	1,000	1,000	---	---	
Central African Republic	10,400	10,400	---	---	
Denmark	---	---	5,000	---	Available in Greenland
Gabon	13,500	6,500	---	6,500	
India	---	---	3,000	1,000	
Italy	1,500	---	10,000	---	
Japan	2,700	---	4,500	---	
Mexico	1,300	---	1,200	---	
Niger	26,000	39,000	13,000	13,000	
Portugal	9,600	7,700	---	30,000(1)	
Spain	10,000	---	34,000	---	Reserves updated in late 1971.
Sweden	---	---	350,000	50,000	
Yugoslavia	5,200	2,000	1,900	2,600	Reserves reported in 1972.
Others	3,600 (2)	11,000	1,500	---	
Totals	96,400	96,000	456,400	153,300	

Source: "Uranium--Resources, Production, and Demand", September, 1970, a joint report by the European Nuclear Energy Agency and the International Atomic Energy Agency.

Notes: (1) Includes 15,000 tons in Angola.
(2) Does not include 25,000 short tons potentially available from as a by-product of phosphate production in Israel.

* Reserves.

TABLE 1-9. URANIUM PRODUCTION, PRODUCTION CAPACITY AND ATTAINABLE PRODUCTION CAPACITY OF OTHER COUNTRIES

(Short Tons of U_3O_8)

Country	U_3O_8 Production in 1970	Attainable Production Capacity					Remarks
		1972	1974	1976	1978	1980	
Argentina	55	88	88	88	150	150	
Central African Republic	--	--	780	780	780	780	Mill output is shipped to France for further refining. Mill expansion not likely unless new reserves are found.
Gabon	463	780	780	780	780	780	
Italy	--	--	120	120	120	120	
Japan	--	40	40	40	40	40	
Mexico	--	40	200	200	200	200	
Niger	59	825	825	1,650	1,650	1,650	
Portugal	105	150	300	300	300	300	
Spain	90	108	108	400	400	400	
Sweden(1)	80	140	140	140	140	140	A plant of a capacity of 1,400 short tons is under consideration; however, because Sweden's reserves are high cost its construction is not viewed as likely by 1980.
Totals	852	2,171	3,381	4,498	4,560	4,560	

Source: United States Bureau of Mines Data, ENEA/IAEA report and BCL estimates.

Note: (1) BCL estimate based on evaluation of reserves, country policy regarding home production of nuclear fuel and other factors.

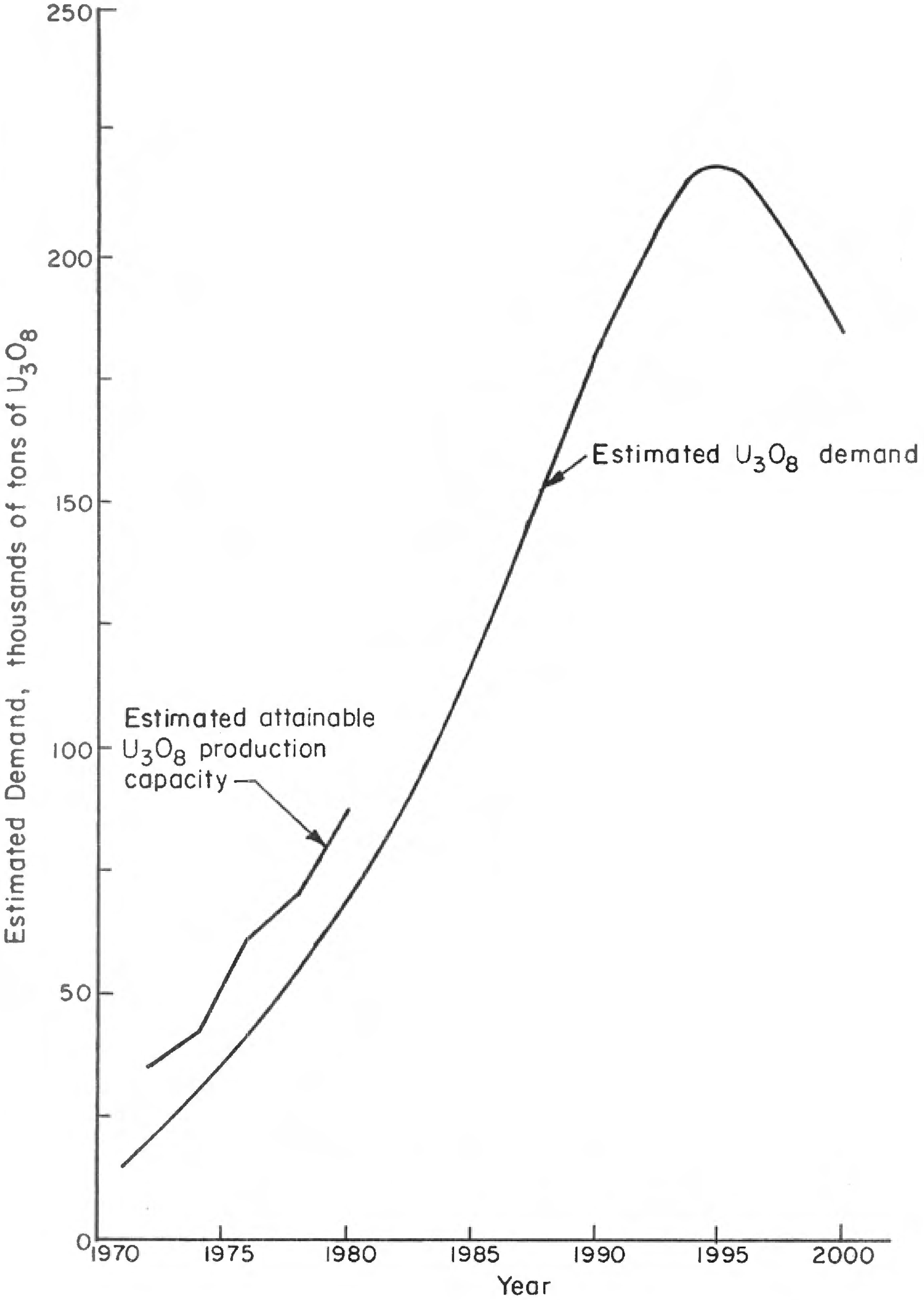

FIGURE I-I. ESTIMATED ANNUAL FREE WORLD URANIUM DEMAND AND ATTAINABLE PRODUCTION CAPACITY

Year	U_3O_8Demand, Tons
1980	69,000
1990	180,000
2000	186,000

In the years through 1985 light water and natural uranium reactors will account for nearly all of the U_3O_8 demand. Beyond 1985, nuclear power needs become more difficult to forecast. In addition, the mix of reactors which will provide nuclear power beyond 1985 is difficult to forecast. The most important factor in the reactor mix is the timing of the commercialization of the fast breeder reactor (FBR). For the purposes of this report, FBR introduction is assumed to become significant in 1990. Although the time of FBR introduction affects annual U_3O_8 demand, its introduction as early as 1986 would have relatively little effect on cumulative U_3O_8 requirements through 2000.

United States

Demand for U_3O_8 in the United States are periodically forecast by the USAEC. The most recent forecast* gives annual U_3O_8 demand through 1985 without regard to the possible development of fast breeder reactors. A second USAEC forecast gives U_3O_8 demand through the year 2000 for three separate assumptions regarding future mix of the LWR, HTGR and FBR reactor types.** The demand for U_3O_8 to 1985 was based on operating, planned, and estimated future nuclear power generation capacity in the United States, allowing proper lead times for procurement of the U_3O_8, and its processing into fuel elements.

The construction of a nuclear power plant in the United States may take 7 to 8 years from announcement to completion; this forecast is believed to be accurate to about 1980. Table 1-10 gives the forecast for U_3O_8 demand to 1985. The dates shown assume that plutonium recycle will be employed--that is, that the plutonium generated will be used as a partial substitute for uranium-235 in the makeup of new fuel. The demand shown does not include effects of fast breeder reactor development on U_3O_8 demand, it shows only the demand created by LWR's (both the boiling water and pressurized water types). Examination of Table 1-10 shows that from 1971 to 1985, U_3O_8 demand will grow from about 7,000 tons per year to almost 60,000 tons per year. Cumulative demand through 1985 is estimated to be over 452,000 tons of U_3O_8.

To forecast U_3O_8 demand to the year 2000 requires additional assumptions regarding reactor types, because the advent of the FBR will greatly affect U_3O_8 demand. Thus, future U_3O_8 demand was forecast by the AEC using various assumptions

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*"Forecast of Growth of Nuclear Power" USAEC report WASH-1139, January, 1971.

**"Updated (1970) Cost-Benefit Analysis of the United States Breeder Reactor Program", USAEC Report WASH-1184, January, 1972.

TABLE 1-10. DETAILED FORECAST OF U.S. U_3O_8 DEMAND TO 1985(1)

(Short Tons of U_3O_8)

Year	Annual Demand	Cumulative Demand
1971	6,900	6,900
1972	10,200	17,100
1973	14,000	31,100
1974	16,700	47,800
1975	18,400	66,200
1976	21,100	87,300
1977	24,400	111,700
1978	28,600	140,300
1979	31,700	172,000
1980	34,200	206,200
1981	39,300	245,500
1982	44,300	289,800
1983	49,100	338,900
1984	53,900	392,800
1985	59,300	452,100

Source: USAEC Report WASH-1139 January, 1971.

Note: (1) Assumes plutonium recycle is employed.

with regard to reactor mix. Table 1-11 shows the assumptions used. For the three cases shown, a plot of U_3O_8 demand to the year 2000 was given, and is presented in Figure 1-2. This figure shows that although the introduction of the FBR will ultimately decrease the need for U_3O_8, in the years immediately preceding its introduction and for a period afterward, U_3O_8 demand actually increases. This strange phenomenon occurs because of two reasons. First, an increased nuclear market is expected when the FBR becomes available (see Table 1-11) and secondly, the FBR's will divert a significant fraction of the plutonium supply, thus, LWR fueling would be more dependent on uranium-235. Figure 1-2 indicates that yearly U_3O_8 demand in 1990 could vary from 70,000 to 90,000 tons, depending on reactor mix; by 2000, demand could vary between 65,000 and 110,000 tons per year.

For the purpose of presenting only one forecast for U_3O_8 demand to the year 2000, only one assumed reactor mix will be chosen. It is BCL's opinion that the assumption of FBR introduction in 1990 with four gigawatts of installed capacity is closer to current expectations than that of FBR installations totaling 40 gigawatts in 1986. Therefore, assumption 3 in Table 1-11 is adopted. Combining the two aforementioned USAEC forecasts yield the U_3O_8 demand, by year, to the year 2000. These data are given in Figure 1-2 and essentially coincides with the curve labeled "With FBR by 1990".

Expected United States annual demand at 10-year intervals is given below:

Year	U_3O_8 Demand, Tons
1980	34,200
1990	90,000
2000	93,000

Cumulative United States demand for U_3O_8 is estimated to be as follows:

Year	U_3O_8 Demand, Tons
1980	206,000
1990	900,000
2000	1,900,000

Of interest is the fact that cumulative demand to the year 2000 is relatively unaffected by the reactor mix assumed in the three cases given in Table 1-11.

Other Free World Countries

The demand for U_3O_8 in Free World countries other than the United States has been forecast by the USAEC to the year 1985. The most recent forecast

TABLE 1-11. PROJECTED TOTAL AND NUCLEAR U.S. ELECTRICAL CAPACITY

Demand and Capacities	1980	1990	2000
Total Energy Demand (10^{12}KWh/yr.)	3.2	6.0	10.0
Total non-peaking capacity (GWe)	580	1,050	1,800
Total Nuclear/FBR capacity (GWe)			
LWR+HTGR (Assumption 1)	150/0	520/0	1,350/0
LWR+HTGR/+FBR by 1986 (Assumption 2)	158/0	600/40	1,500/670
LWR+HTGR/+FBR by 1990 (Assumption 3)	154/0	560/4	1,400/360

Source: USAEC Report WASH-1184, January, 1972.

Legend: KWh/yr = Kilowatt hours per year.

GWe = Gigawatts electrical.

Gigawatt = 10^9 watts = 1,000 megawatts.

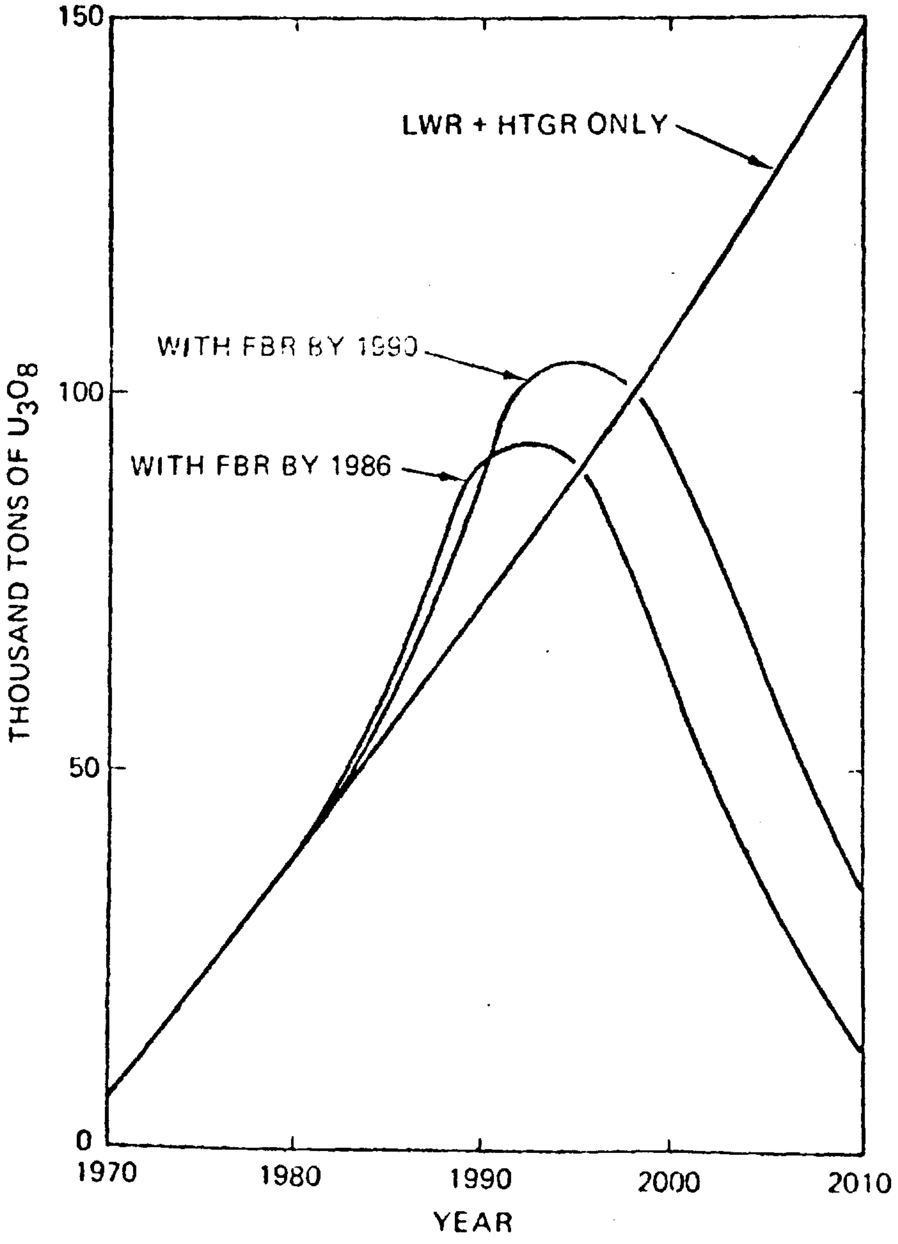

FIGURE 1-2. ANNUAL U_3O_8 DEMAND FOR THE U.S.

Source: USAEC Report WASH-1184

was made in 1971*. The results are given in Table 1-12. Annual demand for U_3O_8 is expected to rise from 7,400 tons in 1971 to 60,400 tons in 1985. Cumulative demand to 1985 totals 443,000 tons, and is approximately equivalent to United States demand for the same period. Demand forecasts beyond 1985 have not been performed recently, and those available show a high degree of uncertainty beyond 1985. This uncertainty is related to reactor mix, which is complicated by the various types of breeder reactors under development and uncertainty regarding the time of commercialization. Available forecasts for Free World uranium demand beyond 1985 indicate that United States and other Free World demand will be approximately equivalent if it is further assumed that the effect of FBR development on uranium demand will be similar to that predicted for the United States. Thus, annual U_3O_8 demand (Free World excluding United States) is estimated to be as follows:

Year	U_3O_8 Demand, Tons
1980	34,500
1990	90,000
2000	93,000

and cumulative U_3O_8 demand is estimated to be as follows:

Year	U_3O_8 Demand, Tons
1980	191,000
1990	900,000
2000	1,900,000

The major portion of U_3O_8 needs will come from Japan and Western Europe.

Supply/Demand Relationships

Figure 1-1 (page 1-20) presents a plot of estimated Free World U_3O_8 demand to the year 2000 and estimated attainable U_3O_8 production capacity to 1980; the demand curve shows rapidly increasing U_3O_8 demand to a peak in 1995. The increase in estimated annual demand is phenomenal and grows from 14,300 tons in 1971 to a peak of 218,000 tons in 1995, an average annual increase of 12 percent per year.

Production Capacity Versus Demand

Using the data plotted in Figure 1-1, several conclusions can be made regarding the need for U_3O_8 production capacity.

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*"Forecast of Growth of Nuclear Power", USAEC report WASH-1139, January, 1971.

TABLE 1-12. FREE WORLD (OTHER THAN U.S.) DEMAND FOR U_3O_8

(Short tons of U_3O_8)

Year	U_3O_8 Demand			
	For Enriched Uranium Reactors[a]	For Natural Uranium Reactors	Total	Cumulative Demand From 1971
1971	3,800	3,600	7,400	7,400
1972	4,300	4,000	8,300	15,700
1973	7,200	3,700	10,900	26,600
1974	10,000	3,700	13,700	40,300
1975	12,500	3,800	16,300	56,600
1976	16,700	4,000	20,700	77,300
1977	20,000	4,300	24,300	101,600
1978	21,800	4,500	26,300	127,900
1979	23,800	4,800	28,600	156,500
1980	29,400	5,100	34,500	191,000
1981	34,300	5,400	39,700	230,700
1982	39,300	5,700	45,000	275,700
1983	43,700	6,000	49,700	325,400
1984	50,900	6,300	57,200	382,600
1985	53,700	6,700	60,400	443,000

Source: USAEC Report WASH-1139.

Note: (1) Assumes plutonium recycle and, therefore, is slightly conservative.

Through the year 1980, it is apparent that the attainable U_3O_8 production capacity will exceed demand by substantial margins. Production attainable from low cost, proven reserves from mills in operation or likely to be in operation will total an estimated 86,900 tons of U_3O_8 while projected demand will reach only the 69,000 ton mark. Therefore, it is likely that U_3O_8 supply will outstrip demand through 1980.

In considering the 1980 to 1990 decade, the supply/demand relationships become more hazy. By 1983, U_3O_8 demand will outstrip the production capability estimated for 1980. Where will the needed new production come from? It is possible that mines and mills in operation in 1980 could expand output; however, output expansion of this sort will be severely limited by reserves at the mine sites. Optimistically, perhaps mines and mills in operation could expand from an attainable 1980 capacity of 86,900 tons to 110,000 tons. Such an expansion would only be sufficient to handle the need for U_3O_8 through the year 1984. From this exercise it becomes obvious that by about 1985 U_3O_8 demand will exceed attainable production capacity by a large margin. Thus, new mine and mill development at currently marginal locations or at locations not yet discovered must take place in the late 1970's and early 1980's to provide for the demand circa 1985. The expansion in U_3O_8 capacity will be large. From the mid-1980's to 1990 expansion needs will total a minimum of 75,000 tons of annual capacity or approximately twenty-five 3,000 ton mine-mill complexes.

Uranium Demand Versus Resources

Estimated cumulative U_3O_8 demand is presented in Fgiure 1-3 (page 1-29) to the year 2000. This figure also presents the current status of U_3O_8 resources and government stocks. If all of the uranium resources now known to be recoverable at $15 per pound of U_3O_8 or less, can in fact be recovered by the year 2000, then the currently known resources are just sufficient to last to the end of this century. If, however, only reasonably assured resources (reserves) recoverably at $15 per pound or less are considered, then these reserves which total slightly over 2,000,000 tons will last only until about 1991. In order to maintain an orderly expansion of nuclear power, forward reserves must be available in order to justify building a power plant fueled with uranium. The amount of forward reserve needed is a matter of conjecture, with an 8-year supply being the minimum indicated. Though various conclusions may be drawn by analyses of the data presented with regard to the needed discovery rate for new uranium reserves, the conclusion reached in the previous section is sufficient--namely, that by the mid-1980's, about 75,000-ton per year is new mining and milling capacity will be needed. If a 15 to 20 year mine life is assumed, then minable reserves totaling 1.125 million to 1.50 million tons of U_3O_8 must be found. If one assumes a five year mine-mill development program then these reserves must be discovered before about 1980.

Between 1985 and 1990 an additional 60,000 tons of U_3O_8 capacity will be needed. Again assuming a 5-year development period for a 15 to 20 year mine life, minable reserves of 0.9 to 1.2 million tons will be needed.

Capacity needs beyond 1990 are definitely not clear and depend greatly on the time of introduction of the FBR; however, capacity expansion will most likely drop in the 1990-1995 period as the future needs for U_3O_8 will peak out

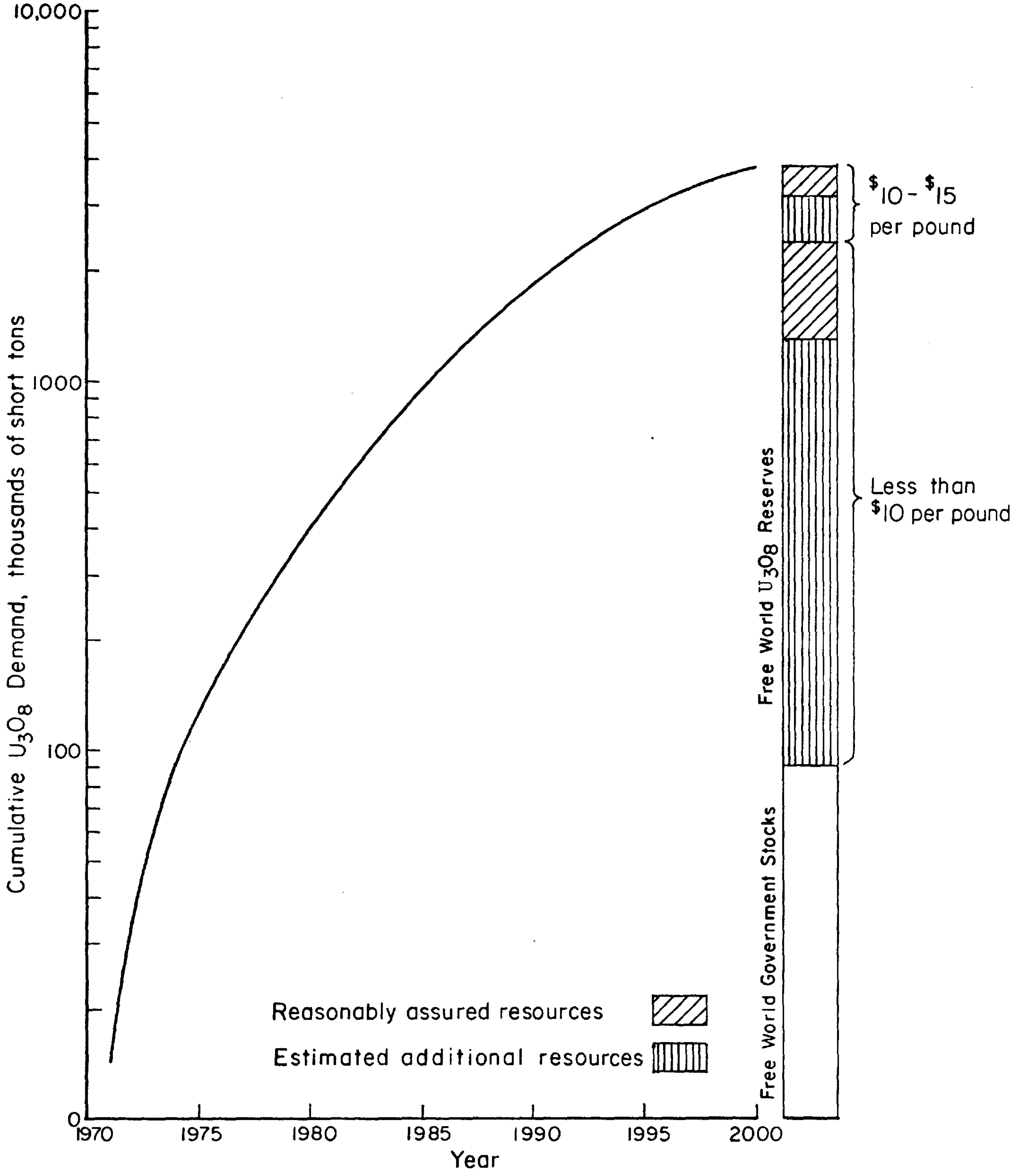

FIGURE 1-3. CUMULATIVE FREE WORLD U_3O_8 DEMAND AND RESERVES

and begin to drop. Only comparatively minimal U_3O_8 capacity expansion will be needed, perhaps 35,000 tons per year. Actual capacity needs are likely to be higher as uranium mines operating in the 1970's become exhausted. The extent of replacement capacity needs is difficult to assess but could amount to 25,000 tons. Thus, for the period 1990-1995 approximately 60,000 tons of new U_3O_8 capacity will be needed; and concomitant reserve finds in the 1985-1990 period would have to be 0.9 to 1.2 million tons.

Beyond 1995, U_3O_8 needs will diminish as the new nuclear power plants are expected to be FBR's fueled with plutonium rather than uranium. Thus, the need for uranium will diminish to a level needed to refuel existing LWR's. If plutonium supplies become abundant, plutonium could partially replace enriched uranium in LWR's. If such proves to be the case, then U_3O_8 demand could dip below the level required to maintain existing LWR's.

Table 1-13 reviews U_3O_8 capacity needs that are viewed as having to come from reserves not yet discovered; from these needs, the need for reserve discoveries in the previous 5-year period are calculated. The table shows that by 2000, U_3O_8 capacity needs (from reserves not yet discovered) total about 200,000 tons, the reserve additions needed to support such a capacity expansion total 4.7 to 5.7 million tons of U_3O_8.

Probable Price Trends

The current price for uranium is a somewhat elusive figure because most U_3O_8 sales being consummated currently are for future delivery, and prices even when available are subject to escalation clauses. Prices on the non-United States market are in the range of \$5.50 to \$6.50 per pound of U_3O_8 for delivery in the mid-seventies. In cases where delivery before 1975 is desired, prices may be as low as \$5 per pound or slightly lower. In the United States, where the uranium industry is protected by AEC regulations, uranium prices are generally higher by \$0.50 to \$1.00 per pound. In 1972, the United States Bureau of Mines reported that the average price of U_3O_8 sold by private industry was \$6.20 per pound.

Prices for United States delivery of U_3O_8 in 1976 are to be \$8.00 per pound. It appears that sufficient uranium reserves recoverable at less than \$8 per pound* exist in the United States, Canada, Australia, France, and South Africa to maintain a U_3O_8 price of roughly \$8 per pound to about 1980. Circa 1980 prices will likely jump to the \$8 to \$10 per pound range and remain there for a considerable period of time. If only reasonably assured resources available today at less than \$10 per pound are considered, then by about 1987, uranium prices would go above \$10 per pound of U_3O_8 (see Figure 1-3). However, between now and 1987, a huge exploration effort will have been undertaken. It is considered likely that reserves recoverable at \$10 per pound of U_3O_8 will be found in sufficient quantity to insure a price of about \$10 per pound until about 1990. For the period 1990-2000, U_3O_8 prices of \$10-\$15 per pound are likely.

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*"Forecast of Growth of Nuclear Power USAEC report WASH-1139, January, 1971.

TABLE 1-13. ESTIMATED FREE WORLD U_3O_8 CAPACITY NEEDS FROM UNDISCOVERED RESERVES(1)

Period		Approximate Capacity Addition Requiring New Reserves, Short Tons U_3O_8 per Year	Approximate Reserve Additions Needed, Millions of Tons of U_3O_8
Date-1980		(2)	1.125 - 1.50
1980-85		75,000	0.90 - 1.20
1985-90		60,000	0.90 - 1.20
1990-1995		60,000	1.00(3)
1995-2000		small	0.80(3)
	TOTAL	195,000	4.7 - 5.7

Source: Estimated by BCL.

Notes: (1) Assumes reserves are discovered 5 years before mine development, with a 15 to 20 year mine life.

(2) It is assumed that demand can be met from currently known uranium resources.

(3) Estimated to be equivalent to U_3O_8 demand for the next 5 year period.

Need for a New Producer in the James Bay Region

It is obvious from the supply/demand analysis, that additional new reserves of uranium will need to be discovered if the projected demand is to be realized. This conclusion has been reached by a number of companies engaged in the mining and processing of uranium. Conforming to historical mining industry trends, exploration activity will likely be concentrated in areas that have yielded commercial discoveries in the past with less effort being expanded in totally unknown areas.

In the above context, the James Bay region represents one of the lesser explored areas in which uranium might be found. However, the inferences drawn by the S.G.S. report regarding undiscovered potential are encouraging and suggest an attempt by the James Bay Development Corporation to promote exploration, especially in the priority zones identified. To best fit the uranium market situation, exploration should be started as soon as possible in order to maximize the probability of a commercial discovery before 1980. If such a discovery is made, the ensuing feasibility study should include the condition of profitable operation at a sales price of about \$8 to \$15 per pound of U_3O_8.

HYPOTHETICAL MINE/MILL PROJECT FOR THE JAMES BAY REGION

Introduction

The following hypothetical uranium mine-mill project is described based on geological considerations, assumed deposit characteristics and likely minimum mill size. It is expected that project construction could begin in 1983 with first output produced in 1984.

Project Summary

The envisioned project consists of an open-pit uranium mine capable of producing 3.5 million tons of ore per year at an average grade of 0.25 percent U_3O_8, for a 10-year period. The associated mill will have a capacity for treating 1,000 tons per day of ore to a concentrate of 75 percent U_3O_8. Project costs, exclusive of road building requirements other than in the immediate vicinity, and exclusive of any townsite costs are as follows:

Mine:	Mine Development	$ 7.00 million
	Equipment, Building and Other	1.25 million
Mill:	Construction and Equipment	8.75 million
	Total	$17.00 million

Provision for townsite and road requirements would possibly double the above figure. However, as both facilities are in the immediate vicinity of LG-2, it is likely that such facilities would not be necessary.

The output over the 10-year period will total 10,850 tons of concentrates containing 8,140 tons of U_3O_8. Value of the output over the 10-year period starting in the early mid-80's could be expected to be about $163 million at an assumed price of $10 per pound of U_3O_8.

Preliminary estimates of profitability suggest that there is a 64 percent chance of earning at least a 33 percent return on investment.

The Deposit

Based on geological considerations, the LaGrande region was identified by S.G.S as offering one of the best prospects for discovery of a commercially viable uranium deposit. They estimate that the undiscovered potential for uranium in the LaGrande area could approach $525 million at a price of $10 per pound of U_3O_8. Much of this uranium is likely to occur in sandstone at two locations: 1H LaGrande-East and 1H LaGrande-West (see Figure 2 in Introduction). As the latter is situated close to existing and planned infrastructure, it is reasonable to assume that 1-H West will, in fact, receive earlier exploration attention. Thus, 1-H West is assumed to be the location of the hypothetical deposit on which this project is based.

Assuming only a slightly above average grade of ore, 0.25 percent U_3O_8, such a mine would indicate a milling operation of about 1,000 tons per day as the minimum size for an economically profitable operation. If a 10-year mine life is envisioned, then the hypothetical deposit required will need to contain 8,750 tons of U_3O_8 or about one-third of the uranium assigned to the LaGrande region. The uranium occurrence probably is characterized as uranium in sandstone. Such deposits are usually irregular, nearly tabular masses varying from small pockets to a few large bodies several thousand feet across, and up to 20 feet in thickness. Uranium sandstone deposits often occur in the presence of carbonaceous matter and accompanied by sulfide and carbonate minerals. Most important ore bodies in sandstone are carbonaceous at depth and contain typically black, unoxidized uranium minerals.

It is assumed that the hypothetical uranium occurs in one or several closely located deposits, which can be mined with an ore to waste ratio of about 1 to 3 after removal of an overburden averaging about 20 feet in depth.

The Mine

Mine Size and Materials Handling Requirements

The open pit mine(s) will yield 3.5 million tons of uranium ore over the life of the project. For a 1 to 3 ore/waste ratio, the waste which will have to be removed with the ore would be about 10.5 million tons. The volume occupied by this mass of ore and waste would be about 10 million cubic yards.

Overburden volume will depend on the geometry of the pit used for claiming the ore. If the entire deposit is in one location and can be mined by using a pit shaped like an inverted cone, with a depth equivalent to half the radius, then the mine pit would be 450 feet deep with a surface diameter of 1,620 feet. The overburden would occupy about 1.35 million cubic yards at an average depth of about 18 feet. Under these conditions, the total pit volume would be 11.35 million cubic yards.

To support a 1,000 ton per day mill for ten years, the materials handling capability of the mine is as follows:

	Approximate Capacity in Tons		
Material	10-year Total	Annual	Daily
Ore	3.5 million	350,000	1,000
Waste	10.5 million	1.05 million	3,000

Removal of overburden would be approximately 1.35 million cubic yards, or about 1 million tons during the mine development phase.

Mine Equipment Needs

The mine must feed 7,000 tons of ore per week to the mill, which will be operating on a three-shift basis, seven days per week. With an ore-to-waste ratio of 1 to 3, weekly movement of material will aggregate 28,000 tons.

The mine equipment needed is estimated on the basis of two eight-hour shifts working five days per week. Thus, the equipment needed must handle 2,800 tons of material per shift. The trucks used will haul waste to a nearby waste pile, or the mill which is likely farther away. If it is assumed that the average haul is about four to five miles, the trucks can average 14 trips per shift. Use of a 36-ton truck enables moving 500 tons of material per truck per shift. Thus, to move the required 2,800 tons per shift, a fleet of six trucks is needed as well as two 6-cubic yard loaders.

Other excavation equipment needs include a road grader for maintaining haulage roads, a bulldozer and a front-end loader for pit-floor clean-up, as well as an additional bull dozer for waste pile excavation.

Auxiliary vehicle needs include a small tank truck for fuel delivery and three small trucks for transporting supplies and personnel.

A self-contained, self-propelled drill will be needed. This unit will be used to drill holes for placement of explosives used in breaking up the ore and waste rock.

Table 1-14 summarizes mine equipment needs.

Mine Support Facilities

The mine will require at least one building to provide office space, a vehicle maintenance area and storage for certain supplies. Also needed, are a fuel storage tank and an explosive storage shack. Electrical needs could be supplied from the LG-2 power plant or by a diesel generator if necessary, while heat should be provided by fuel oil.

Mine Manpower List

The manpower requirements for the mine can be estimated by calculating what is required for full time operation of all equipment and by adding the necessary supervision and support personnel. Table 1-15 presents the list of personnel.

TABLE 1-14. URANIUM MINE EQUIPMENT LIST

6 - Trucks, 36 ton

2 - Loaders or power shovels, 6 cubic yards

1 - Self-propelled drilling unit

2 - Bulldozers

1 - Front-end loader

1 - Tank truck

3 - Small trucks for supplies and personnel transport

Source: Estimated by BCL.

TABLE 1-15. URANIUM MINE MANPOWER LIST

Designation		Number
Skilled equipment operators		6
Semiskilled equipment operators		20
Skilled maintenance personnel		2
Semiskilled maintenance personnel		4
Unskilled labor		3
	Total labor	35
Management		6
Office and technical		3
	Total Office	9
Total		44

Source: Estimated by BCL.

Operating Supplies

The major operating supplies required are estimated on a daily and yearly basis.

Diesel Fuel. The major operating mine vehicles are estimated to consume 80 gallons of diesel fuel per two-shift day. For the 13 major pieces of equipment, an estimated 1,040 gallons of diesel fuel will be needed per day. When diesel fuel needs for heating of mine buildings and generation of electrical power are added, the total estimated daily consumption will be 1,200 gallons or roughly 312,000 gallons per year.

Other Vehicle Supply Requirements. Gasoline will be needed for fueling the small trucks. In addition, lubricating oil, grease, hydraulic fluid, and spare parts will be needed. These requirements are not estimated here. In terms of transport weight, the weight of these items is within the margin of error of the estimated diesel fuel needs.

Explosives. The requirements for explosives can vary greatly with the nature of the deposit. If material removal is difficult, power requirements could total one pound per cubic yard, or roughly 3,000 pounds per day, equivalent to 390 tons per year.

Mine supply needs are summarized as follows:

	Daily	Annually(1)
Diesel fuel	4.6 tons	1,200 tons
Other vehicle supplied (included above)	Nil	Nil
Explosives	1.5 tons	390 tons
Approximate Total	6.1 tons	1,600 tons

---

(1) Based on a five-day, two-shift week.

Mining Effluents

Aside from the solid waste generated in the mining operation (detailed in the interium report), there are really only two types of effluents to be con-concerned with, liquid effluents and gaseous effluents.

Liquid Effluents. Although the uranium content of the tailing piles will be low compared to that of the ore, some radioactivity will be present in the runoff water. By proper tailing site selection, this runoff can be minimized. However, any water runoff found should be evaluated for radioactivity, acid content, etc., and the results should be analyzed for possible environmental effects on local water resources.

Gaseous Effluents. The major gaseous effluents from the mining operation will be airborne dust which can be generated either by blasting, at materials tailing piles, or by roads. Radiation monitors can be utilized to determine the extent of airborne radioactivity. Wherever dusting is a problem, it can be minimized by wetting.

The Mill

Flowsheet

The ore to be treated is expected to be amenable to acid leaching by virtue of its occurrance in sandstone. Processing beyond the leading stage depends greatly on other characteristics of the ore that can be determined only after the ore is found. For this hypothetical mill, it has been assumed that the ion exchange method of recovery will be appropriate.

A flowsheet for the 1,000 tons per day mill is presented in Figure 1-4. Ore from the mine is crushed in two stages then wet ground to minus 325 mesh in rod and ball mills, the latter operated in closed circuit witha classifier. After thickening, the pulp is leached at atmospheric presure in strong sulfuric acid, excess acid is neutralized with lime and the barren waste solids are removed by filtration. The pregnant solution is trickled through columns containing ion exchange resins which absorb the uranyl sulfate. The uranium is desorbed from the resin by a weak acid solution which is treated with an alkali to precipitate U_3O_8. The U_3O_8, familiarly called yellowcake from its color, is filtered, dried, and packaged for shipment to a plant making uranium florides. Barren waste solids and barren waste solutions are neutralized and pumped to a tailing's disposal area.

Mill Equipment Needs

The mill equipment list presented below is based on information obtained from the literature.(*) This list illustrates major equipment needs for the acid leach-ion exchange milling of uranium ore. The equipment needs are as follows:

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[*] "Uranium Industry-History, Technology and Prospects" by J. W. Griffith, Mineral Report 12, Mineral Resources Division, Department of Energy, Mines and Resources (Canada), 1967.

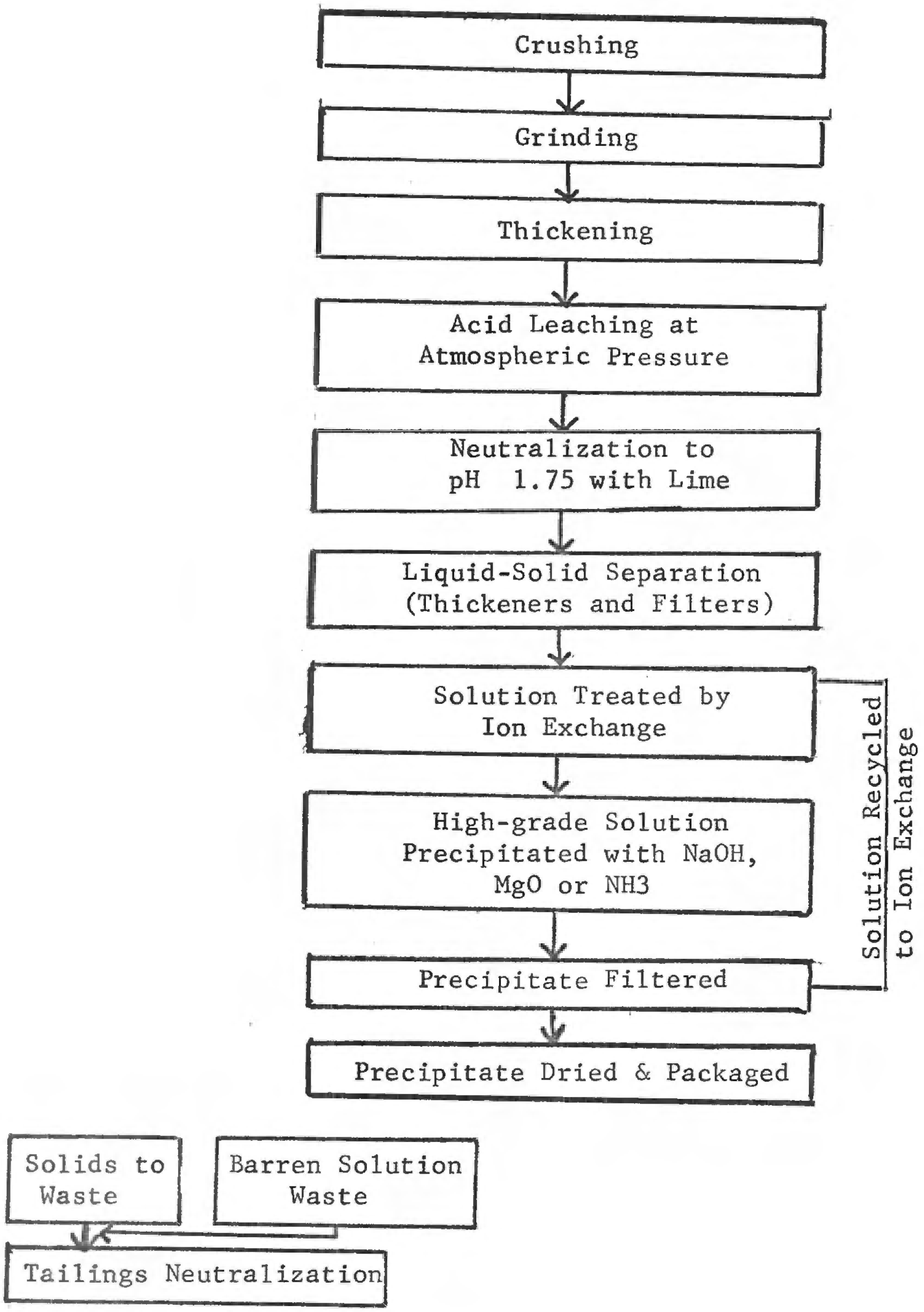

FIGURE 1-4. TYPICAL URANIUM ACID-LEACH/ION-EXCHANGE FLOWSHEET

Source: Mineral Report 12, EMR (Canada), 1967.

Crushing Circuit

Primary and secondary crushers
Surge, waste and storage bins
Feeders for discharge of bins
Grizzlies and rod deck screens
Conveyor system

Grinding Circuit

Rod and ball mills
Classifier
Slurry pump

Thickening Circuit

Thickener tanks
Thickener overflow tanks
Pumps

Precipitation Circuit

Precipitation tank
Make-up tank
Filter presses

Leaching Circuit

Leaching tanks
Pumps

Filtration Circuit

Primary and secondary filters
Surge tank
Pumps

Clarification Circuit

Clarithickener
Clarifiers
Pumps

Ion Exchange Circuit

Brine and recycle tanks
Ion exchange columns
Pumps

Drying and Packaging

Receiving hopper and chute
Dryer
Drum packer

Mill Services

Water storage tanks and pumps
Vacuum (for mill operations) - 3 vacuum pumps driven by 100 hp motors
Low pressure air, 27 pounds per square inch - compressor driven by 125 hp motor.

Mill Manpower List

Mill manpower requirements can vary considerably. If, for instance, the mill manufacturers its own sulfuric acid, greater than normal employment will result. Similarly, the extent of chemical analysis needs can affect the employment of people in the analytical laboratory. Table 1-16 lists likely manpower requirements based on 7-day operation, 3 shifts per day, with sulfuric acid needs being purchased.

Mill Operating Supplies

The chemical supplies, grinding steel requirements, and utilities for milling uranium can vary with the composition of the ore and the method used. Table 1-17 presents the range of needs found for a typical uranium mill. From this, needs for the mill were estimated. The results are given in Table 1-18. This table shows that approximately 27,400 tons of supplies will be needed annually to maintain the facility.

Mill Effluents

The major effluent from the mill is tailings. As the uranium content of the ore is only 0.25 percent, the tailings produced are essentially equivalent to the amount of ore processed. Thus, about 1,000 tons per day of tailings will be discharged. These tailings will consist of ore waste, spent chemicals and minute amounts of uranium which are suspended in an aqueous solution. Usual practice is to keep the tailings in tailings impoundments on impermeable soil. After the mill is shut down, the tailings impoundment will eventually dry out and become a potential dusting problem. Stabilization of the tailings can sometimes be accomplished by planting of various grasses, shrubs, etc.

Liquids used in the milling process are either recycled or discharged with the tailings. Liquid effluents are, therefore, limited. However, possible seepage from open storage of ore or from the tailings impoundment can occur. Prevention of such situations can be controlled by periodic monitoring of the impoundments as well as for stream or lakes in the immediate area.

Like the mine, the major potential airborne effluent is dust. To a large degree, dusting can be controlled by use of available dust control equipment, especially in the U_3O_8 drying and packaging area. Other sources of dust are from mill roads and ore piles. Again, when dry conditions exist, the dust can be minimized by wetting down the source. As with the liquid effluents, the air in the vicinity of the mill can be monitored readily with radiation detection devices.

TABLE 1-16. URANIUM MILL MANPOWER LIST

Designation		Number
Skilled operators		12
Semiskilled operators		12
Skilled maintenance personnel		6
Semiskilled maintenance personnel		8
Unskilled labor		16
	Total Labor	54
Management		4
Office and laboratory		6
	Total Office	10
Total		64

Source: Estimated by BCL.

TABLE 1-17. CONSUMPTION RANGE OF MAIN CHEMICALS, OPERATING SUPPLIES AND UTILITIES IN MILLING URANIUM ORE BY THE ACID PROCESS

Chemicals	Pounds/Ton Milled(1)
Sulfuric acid	70-140
Sodium chlorate	0-3
Nitric acid	0-5
Sodium chloride	0-6
Ammonia	0.2-2
Lime	0-50
Ion exchange resin	0-2%/yr.
Miscellaneous supplies, glue, flocculating agents, etc.	0.2
Grinding Steel	
Rods	0.5-1.5
Balls	2.0-3.5
Utilities	
Power, kilowatt hours	15-35
Water, gallons	200-600
Steam, pounds	150-300

Source: "United States Uranium, Economics and Technology - 1969", Paul F. Shutt, Jr., Nuclear Assistance Corporation.

(1) At 100 percent concentration.

TABLE 1-18. URANIUM MILL SUPPLIES AND UTILITIES REQUIREMENTS

Chemicals	Daily Pounds/ton	Daily, Pounds	Annual, Tons
Sulfuric acid	100	100,000	18,250
Sodium chlorate	2	2,000	365
Sodium chloride	5	5,000	912.5
Lime	40	40,000	7,300
Ion exchange resin	--	--	0.2
Miscellaneous, glue, flocculating agents, etc.	0.2	200	36.5
Grinding steel			
Rods	1	1,000	182
Balls	2	2,000	365
Approximate total	150	150,000	27,400
Utilities			
Power, kilowatt hours	20	20,000	7,300,000
Water, gallon	400	400,000	146,000,000
Steam, pounds	225	225,000	82,125,000

Source: Estimated by BCL.

Mill Output and Value

At a recovery rate of 93% of total ore milled, output of 75 percent U_3O_8 concentrates would be:

Daily	Yearly	Life of Mine
6,198 lbs	1,085 tons	10,850 tons

If production of U_3O_8 commences in the early to mid-80's, prices for U_3O_8 are expected to be in the $8 to $10 price range in current dollars per pound of contained uranium.

At $10 per pound of contained uranium, revenues for the project would be:

Daily	Yearly	Life of Mine (10 years)
$46,500	$16.3 million	$163 million

Capital Requirements

Mine Cost

Uranium mine development costs vary widely. A typical figure for open pit mining of a deep-lying ore body (250 feet of overburden) is $7.70 per ton of ore. For an assumed ore body of 3.5 million tons at this depth, development costs would be about $27 million. However, for the 1-H deposits, it is reasonable to assume ore body depth of 20 feet, thus, overburden removal cost would be considerably less. For the shallower depth, development costs in a readily accessible area would likely be $3 to $4 million dollars. In the remote location considered here, development costs could easily double. Thus, mine development is expected to cost about $7 million, exclusive of road building outside the immediate mine area, and townsite costs.

Other costs, such as equipment and buildings for a typical open-pit operation average about $720,000 for a 1,000 ton per day ore operation. Again, because of the remote location involved and because of the need for special cold weather operation equipment on the mine vehicle, costs will be higher. A figure of $1.25 million is estimated for equipment, buildings and other costs. Thus, total capital costs for putting the mine into operation are expected to be about $8.25 million.

Mill Costs

The costs associated with constructing uranium mills of various capacities are summarized in Figure 1-5. Both a "high range" and a "low range" are given.

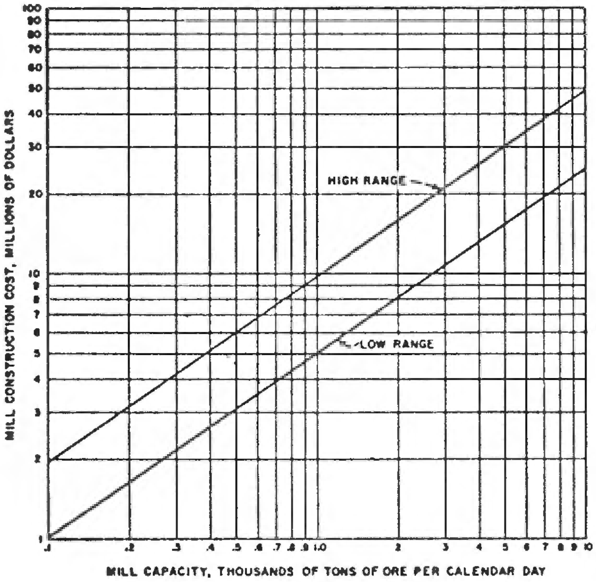

FIGURE 1-5. URANIUM MILL CONSTRUCTION COST
(adjusted to January 1, 1970).

Source: "The Extractive Metallurgy of Uranium" by Robert C. Merritt, Colorado School of Mines Institute (1971).

For the 1,000 ton per day mill hypothesized, costs may vary from a low of $5 million to a high of $10 million, with an average" cost of $7.5 million. This range is a reflection of several factors which, in different combinations, can affect the ultimate total mill cost. Such factors include:

Ore Factors - Complexity of ore
- Hardness of ore
- Variation in ore, either from one or from multiple sources

Locational Factors - Road cost
- Water, gas and power availability
- Employee housing, cafeterias, or townsite costs

Climatic Factors - Building insulation, heating and equipment protection costs.

The mill construction costs associated with deviations in ore factors can be assumed to be average. However, those construction costs associated with locational and climatic factors can be expected to be higher than usual for the remote locations being considered. Neglecting construction costs for roads (except within the mine/mill complex) and neglecting construction costs for townsite facilities, it is likely that the mill complex would cost somewhere between the "average" figure of $7.5 million and the "high range" figure of $10 million or about $8.75 million.

Project Total

Capital requirements for the project thus total $17.00 million (in current dollars).

Evaluation of Potential Worth

Having identified some of the basic characteristics concerning the size, volume of output, capital cost structure, etc., of the hypothetical uranium mine/mill complex, this final section attemps to provide some indication of the potential feasibility of the project. Two key points concerning the following analysis should be emphasized:

(1) The evaluation is, of course, for a hypothetical deposit which remains to be found. Thus, throughout previous sections, a variety of assumptions had to be made which will influence feasibility evaluations. It is felt that the assumptions made are reasonable in that they are based on the best judgements available at this time. However, there is no guarantee that analytic results will be totally reflective of reality when reality is ultimately defined.

(2) Given the degrees of uncertainty associated with such hypothetical projects, it may be argued that the following feasibility estimates are premature at this time. In part, this is true in that even <u>after</u> discovery of an ore body, proving of reserves, and doing prefeasibility studies based on engineering cost estimates for plant, equipment and operating expenses, there are still unknowns and uncertainties which must be assumed by risk takers. However, it is noted that the above is undertaken only after enough is known about the deposit to indicate promise, i.e., some <u>a priori</u> notion that the potential for profit "looks good". For the hypothetical deposits currently being analyzed by Sores-Battelle, even "looking good" has not yet been firmly established. That is, some notions of development costs and expected revenues, by themselves, are not always sufficient to demonstrate promise. In some cases, high operating costs (especially transport charges), may change a seemingly good project into a marginal one. Thus, it is important, in the context of the total study mission, to demonstrate the chances for <u>potential</u> success of the projects identified and for the assumptions made. At the very least, this approach provides a yardstick against which to measure the relative merits of other developmental projects.

A third important point relates to the particular analytic procedure utilized in the preliminary feasibility calculations.* The model used permits varying certain cost and expense elements according to user-specified probabilities. The key advantage derived from the approach is that profitability estimates are presented as ranges, where the ranges are a function of the uncertainty ascribed to those cost, expense, or revenue elements which cannot be estimated exactly. Output is generated in the form of multiple solutions of the same problem which, in total, yield an investment profile of all possible combinations. To see the effects of value changes for particular parameters, the user need only select the solution which was derived from that particular combination of values.

<u>Data Inputs and Assumptions</u>

<u>Capital Outlays</u>. There are three capital outlays associated with the complex:

(1) Mine development costs are estimated at $7.0 million

(2) Building construction and equipment costs associated with mine operation are estimated at $1.3 million

(3) Mill construction and equipment costs are estimated at $8.75 million.

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*Refer to the attachment to this report for a more detailed discussion of the probabilistic cash flow model.

Fifty percent of capital requirements are borrowed on a 10-year loan at 8 percent annual interest. The depreciable life of all buildings and equipment is 10 years, with no salvage value.

Annual Operating Expenses. There are five annual expenses associated with the project.

(1) Mine operating costs are selected from the following distribution:

Annual Cost (Millions of $)	Median per ton Cost ($/ton of ore mined)	Probability* (Percent)
3.9 - 4.2	11.50	10
4.3 - 4.6	12.50	80
4.7 - 4.9	13.50	10

(2) Mill operating costs are sampled from the following distribution:

Annual Cost (Millions of $)	Median per ton Cost ($/ton of ore milled)	Probability* (Percent)
2.1 - 2.4	6.50	80
2.5 - 2.8	7.50	10
2.9 - 3.2	8.50	10

(3) Transportation costs are comprised of two elements:

(a) Ore hauling costs from open pit to mill are estimated at $1.10 per ton for up to 15-mile haul at .065¢ per wet ton mile, or $401,500 per year

(b) Transport costs for 1,085 tons of concentrate (to Montreal) at 10¢ per ton mile, or $108,500 per year.

Thus, annual transport costs would be about $500,000 per year.

(4) Interest costs for 50 percent of the total capital outlay ($8.5 million) on a 10-year loan at 8 percent interest rate are calculated internally within the cash flow program.

(5) Depreciation costs are internally calculated as straight line of the total capital outlay over a 10-year period.

Annual Revenues from the sale of U_3O_8 concentrates are selected from the following distribution:

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* Probabilities assigned to each of the cost ranges suggest the likelihood of occurrence of each as estimated by BCL.

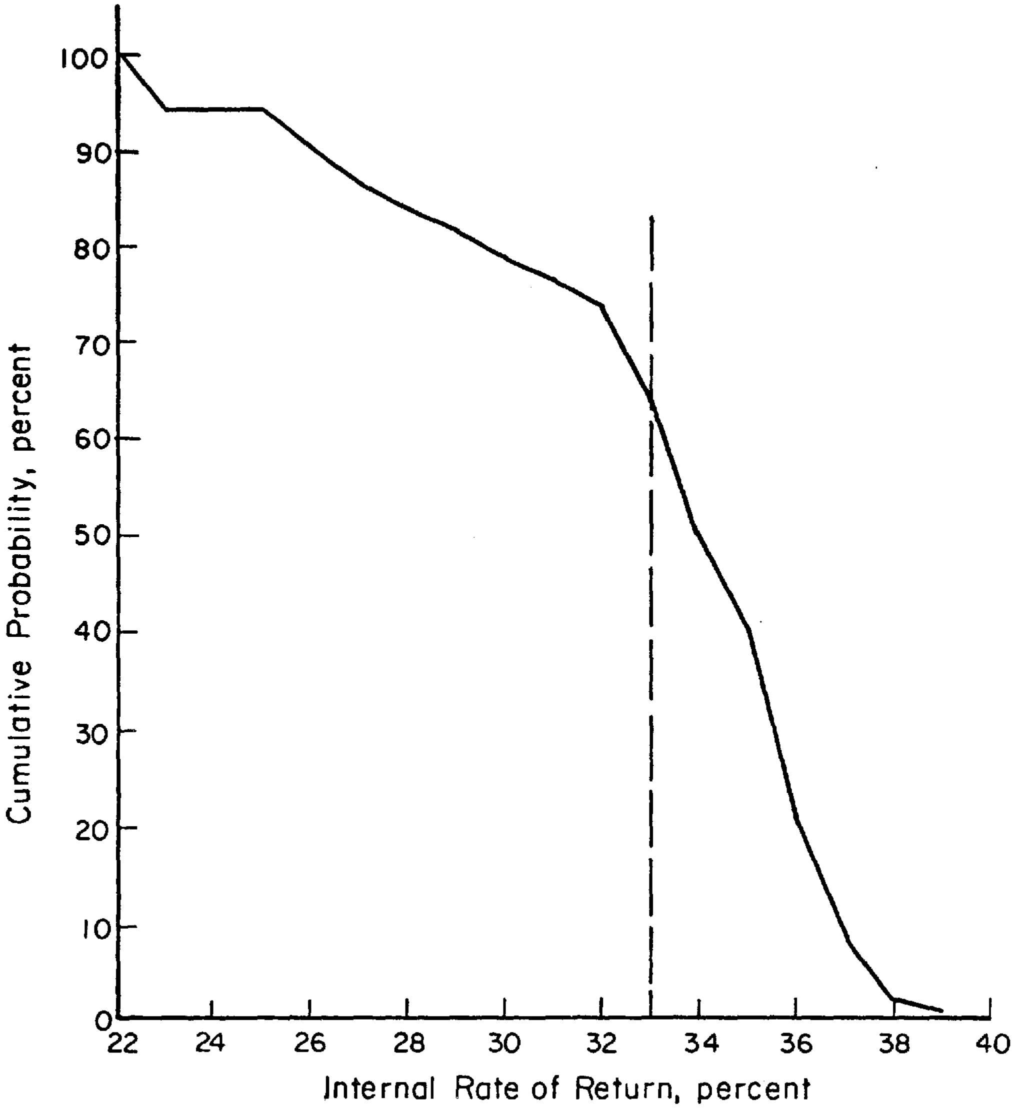

FIGURE I-6. URANIUM PROJECT INTERNAL RATES OF RETURN VERSUS PROBABILITY

TABLE 1-19. URANIUM MINE / MILL COMPLEX AT L6-2
ITERATION NUMBER 3

YEAR	1	2	3	4	5	6	7	8	9	10	11
CAPITAL OUTLAYS											
MINE DEVELOPMENT											
-----INTERNAL FUNDS	3.8	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	.3	.3	.3	.3	.4	.4	.4	.4	.5	.5	0.0
BLDG.,EQUIPMENT											
-----INTERNAL FUNDS	.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	.0	.0	.1	.1	.1	.1	.1	.1	.1	.1	0.0
MILL CONSTRUCTION											
-----INTERNAL FUNDS	3.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	.2	.2	.3	.3	.3	.3	.4	.4	.4	.5	0.0
MILL EQUIPMENT											
-----INTERNAL FUNDS	1.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	.1	.1	.1	.1	.1	.1	.1	.1	.1	.2	0.0
TOTALS	9.4	.7	.7	.8	.8	.9	1.0	1.0	1.1	1.2	0.0
ANNUAL EXPENSES											
MINING COSTS	0.0	4.5	4.5	4.5	4.5	4.5	4.5	4.5	4.5	4.5	4.5
MILL OPERATION	0.0	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2	2.2
TRASPORTATION COSTS	0.0	.5	.5	.5	.5	.5	.5	.5	.5	.5	.5
INTEREST ON LOANS	.7	.7	.6	.5	.5	.4	.3	.3	.2	.1	0.0
TOTAL DEPRECIATION	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6
TOTALS	2.3	9.5	9.4	9.4	9.3	9.2	9.2	9.1	9.0	8.9	8.8
ANNUAL REVENUES											
SALES OF 75% U3O8	0.0	14.6	14.6	14.6	14.6	14.6	14.6	14.6	14.6	14.6	14.6
TOTALS	0.0	14.6	14.6	14.6	14.6	14.6	14.6	14.6	14.6	14.6	14.6
TAXABLE INCOME	0.0	2.8	5.2	5.2	5.3	5.4	5.4	5.5	5.6	5.7	5.8
INCOME TAX	0.0	1.5	2.8	2.9	2.9	2.9	3.0	3.0	3.1	3.1	3.2
NET INCOME	0.0	1.3	2.3	2.4	2.4	2.4	2.5	2.5	2.5	2.6	2.6
CASH FLOW	-10.1	4.5	3.2	3.2	3.2	3.1	3.1	3.1	3.0	3.0	4.2

TERMINAL WORTH = 0.000

YIELD(IN PERCENT) = 28.537
PRESENT WORTH AT 0 PERCENT = 23.509
PRESENT WORTH AT 5 PERCENT = 15.063
PRESENT WORTH AT 10 PERCENT = 9.378

```
TABLE 1-20. URANIUM MINE / MILL COMPLEX AT L6-2
                                    ITERATION NUMBER      8

            YEAR                1      2      3      4      5      6      7      8      9     10     11
CAPITAL OUTLAYS

MINE DEVELOPMENT
-----INTERNAL FUNDS           3.8    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0
-----LOAN PRINCIPAL            .3     .3     .3     .3     .4     .4     .4     .4     .5     .5    0.0
BLDG.,EQUIPMENT
-----INTERNAL FUNDS            .6    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0
-----LOAN PRINCIPAL            .0     .0     .1     .1     .1     .1     .1     .1     .1     .1    0.0
MILL CONSTRUCTION
-----INTERNAL FUNDS           3.3    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0
-----LOAN PRINCIPAL            .2     .2     .3     .3     .3     .3     .4     .4     .4     .5    0.0
MILL EQUIPMENT
-----INTERNAL FUNDS           1.1    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0
-----LOAN PRINCIPAL            .1     .1     .1     .1     .1     .1     .1     .1     .1     .2    0.0

              TOTALS          9.4     .7     .7     .8     .8     .9    1.0    1.0    1.1    1.2    0.0

ANNUAL EXPENSES

MINING COSTS                  0.0    4.0    4.0    4.0    4.0    4.0    4.0    4.0    4.0    4.0    4.0
MILL OPERATION                0.0    2.2    2.2    2.2    2.2    2.2    2.2    2.2    2.2    2.2    2.2
TRASPORTATION COSTS           0.0     .5     .5     .5     .5     .5     .5     .5     .5     .5     .5
INTEREST ON LOANS              .7     .7     .6     .5     .5     .4     .3     .3     .2     .1    0.0
TOTAL DEPRECIATION            1.6    1.6    1.6    1.6    1.6    1.6    1.6    1.6    1.6    1.6    1.6

              TOTALS          2.3    9.0    9.0    8.9    8.9    8.8    8.7    8.7    8.6    8.5    8.4

ANNUAL REVENUES

SALES OF 75% U3O8             0.0   16.6   16.6   16.6   16.6   16.6   16.6   16.6   16.6   16.6   16.6

              TOTALS          0.0   16.6   16.6   16.6   16.6   16.6   16.6   16.6   16.6   16.6   16.6


TAXABLE INCOME                0.0    5.3    7.6    7.7    7.7    7.8    7.9    7.9    8.0    8.1    8.2

INCOME TAX                    0.0    2.9    4.2    4.2    4.2    4.3    4.3    4.3    4.4    4.4    4.5

NET INCOME                    0.0    2.4    3.4    3.5    3.5    3.5    3.6    3.6    3.6    3.7    3.7

CASH FLOW                   -10.1    5.6    4.3    4.3    4.3    4.2    4.2    4.2    4.1    4.1    5.3

TERMINAL WORTH                =      0.000

YIELD(IN PERCENT)             =     37.553
PRESENT WORTH AT  0 PERCENT   =     34.495
PRESENT WORTH AT  5 PERCENT   =     23.082
PRESENT WORTH AT 10 PERCENT   =     15.352
```

TABLE 1-21. URANIUM MINE / MILL COMPLEX AT L6-2
ITERATION NUMBER 23

YEAR	1	2	3	4	5	6	7	8	9	10	11
CAPITAL OUTLAYS											
MINE DEVELOPMENT											
-----INTERNAL FUNDS	3.8	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	.3	.3	.3	.3	.4	.4	.4	.4	.5	.5	0.0
BLDG..EQUIPMENT											
-----INTERNAL FUNDS	.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	.0	.0	.1	.1	.1	.1	.1	.1	.1	.1	0.0
MILL CONSTRUCTION											
-----INTERNAL FUNDS	3.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	.2	.2	.3	.3	.3	.3	.4	.4	.4	.5	0.0
MILL EQUIPMENT											
-----INTERNAL FUNDS	1.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	.1	.1	.1	.1	.1	.1	.1	.1	.1	.2	0.0
TOTALS	9.4	.7	.7	.8	.8	.9	1.0	1.0	1.1	1.2	0.0
ANNUAL EXPENSES											
MINING COSTS	0.0	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6
MILL OPERATION	0.0	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3	2.3
TRASPORTATION COSTS	0.0	.5	.5	.5	.5	.5	.5	.5	.5	.5	.5
INTEREST ON LOANS	.7	.7	.6	.5	.5	.4	.3	.3	.2	.1	0.0
TOTAL DEPRECIATION	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6	1.6
TOTALS	2.3	9.7	9.7	9.6	9.5	9.5	9.4	9.3	9.2	9.2	9.1
ANNUAL REVENUES											
SALES OF 75% U3O8	0.0	13.3	13.3	13.3	13.3	13.3	13.3	13.3	13.3	13.3	13.3
TOTALS	0.0	13.3	13.3	13.3	13.3	13.3	13.3	13.3	13.3	13.3	13.3
TAXABLE INCOME	0.0	1.3	3.7	3.7	3.8	3.8	3.9	4.0	4.1	4.2	4.3
INCOME TAX	0.0	.7	2.0	2.0	2.1	2.1	2.1	2.2	2.2	2.3	2.3
NET INCOME	0.0	.6	1.7	1.7	1.7	1.7	1.8	1.8	1.8	1.9	1.9
CASH FLOW	-10.1	3.8	2.5	2.5	2.5	2.4	2.4	2.4	2.3	2.3	3.5

TERMINAL WORTH = 0.000

YIELD(IN PERCENT) = 22.039
PRESENT WORTH AT 0 PERCENT = 16.629
PRESENT WORTH AT 5 PERCENT = 10.040
PRESENT WORTH AT 10 PERCENT = 5.636

Annual Revenues (Millions of $)	Median Price ($ per pound of U_3O_8)	Probability* (percent)
13.0 - 13.8	8.00	10
13.9 - 14.7	8.50	10
14.8 - 15.5	9.00	10
15.6 - 16.2	9.50	10
16.3 - 16.3	10.00	55
16.4 - 17.0	10.50	5

Output

Based on the above inputs, 100 iterations were run, yielding internal rates of return (yields) between 22 and 38 percent. The yields are calculated by successive approximations leading to the rate which, when used to discount all of the annual cash flows, gives a present worth of zero. The range of returns is illustrated in Figure 1-6 as a cumulative frequency distribution which indicates that there is a 64 percent chance of earning at least a 33 percent return on investment.

Sample output iterations indicating high, low and medium rates of return are presented as Tables 1-19 to 1-21. Variation in the extreme rates of return can be attributed to high operating costs and low revenue figures and vice versa. The 28 percent yield is derived from median values on all three measures.

While the resultant investment profile indicates significant yields for the project, based on reasonable cost and revenue projections, it is noted that yields would be much lower should the deposits be found at much greater depths. Present judgments, however, do not indicate the latter.

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* Probabilities assigned to each of the revenue ranges suggest the likelihood of occurrence as estimated by BCL.

HYPOTHETICAL ENRICHMENT PLANT FOR THE JAMES BAY REGION

Introduction

One of the most promising resources of the James Bay region is the potential for large quantities of reasonably low-cost electrical power. As the enrichment of uranium by the gaseous diffusion method is one of the most power intensive industries known, the following project analysis was performed to acquaint the James Bay Development Corporation with the supply-demand situation, the costs, anticipated revenues, and actions needed to initiate a gaseous diffusion plant in the region. Also included are ideas for several vertical integration projects if the enrichment plant project eventually is undertaken.

The project analysis is made for an 8.75 million SWU plant and includes notes on a 17.5 million SWU plant.* It is possible that the project could consist of building the smaller plant with later expansion to a greater capacity.

Supply-Demand Analysis

Current Free World enrichment capacity totals 17.75 million SWU, with the United States accounting for 96 percent of the total. By 1980, two expansions will come on line, the USAEC with an additional 9.7 million SWU, and URENCO, the European consortium, with about 0.28 million SWU. Thus, by 1980, enrichment capacity will total 27.7 million SWU. The demand for enrichment thereafter will increase at an even more rapid rate. Additional Free World enrichment capacity needs have been estimated by the Atomic Industrial Forum to be as follows:

Year	Additional Capacity Needed (millions SWU)
1981	8.75
1982	8.75
1983-85	17.5

Beyond 1985, enrichment requirements are difficult to assess. Although further increases in enrichment capacity will be needed, the effect of Fast Breeder Reactor (FBR) development will likely diminish the need for new capacity additions. Any delay in FBR development would extend the need for new enrichment capacity.

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*SWU - kilogram separative work unit. The separative work unit is a measure of the work performed in the isotope separation process. To produce 1 kg of 3 percent enriched uranium from natural uranium requires 4.3 units of separative work.

In terms of foreseeable market needs, the most appropriate time for bringing a new enrichment plant on stream would be in the years 1981 through 1985. Since construction of such a project may take up to 8 years,* it should be started in the period 1973-1977.

Project Summary

An 8.75 million SWU per year plant is envisioned as a potential project for the James Bay region. For such a project to proceed, long-term, low-cost commitment of 2,400 megawatts of electrical power is needed along with long-term financing at reasonable rates. Since the cost of such a project will exceed $1 billion, the formation of a consortium of companies and/or government agencies is suggested. Further, to obtain access to U.S. diffusion technology, it is recommended that a U.S. company be a member of such a consortium.

Table 1-22 presents estimates of power needs, feed requirements, product value, manpower needs, and other details for typical diffusion plants. The potential profitability of a gaseous diffusion project is greatly dependent on power rates and the cost of capital. The subject of potentioal feasibility is dealt with in detail in a subsequent section.

A Gaseous Diffusion Plant

In principle, gaseous diffusion enrichment of uranium is a simple operation. The feed material, natural UF_6 containing about 0.7 percent U-235 is pressurized against a permeable metal membrane. This membrane or barrier permits the U-235 isotope to pass through with slightly greater ease than the U-238 isotope. Thus, the gas passing through the barrier is slightly enriched and that not passing through the barrier is slightly depleted. The enriched stream of UF_6 gas is pumped through successive barriers until the U-235 content desired is obtained. The depleted UF_6 stream is also pumped through successive barriers until the U-235 content is about 0.20 to 0.25 percent, at which point it is withdrawn.

The gaseous diffusion plant is made up of a series of stages, each stage consisting of a gas diffuser (or barrier), gas compressor, gas cooler, compressor drive motor and control valve. A 8.75 million-SWU plant would contain about 1,180 stages. The process cell (which is a group of stages operated as a unit) consists of 16 stages interconnected by process piping and valves. Figure 1-7 shows the layout of a typical 16-stage cell.

The major equipment needs for a gaseous diffusion plant are as follows:

Gas Diffusers or Barriers
Gas Compressors - 3 sizes
Compressor Drive Motors - 3 sizes
Process Piping and Valves
Process Building and Enclosures

---

*Two years for design, site acquisition, etc. and 6-years for actual construction.

TABLE 1-22. NEW GASEOUS DIFFUSION PLANT ESTIMATES(1)

	Technology		
	1970	1970	Late 1970's
Separative Capacity, Millions of SWU/yr.	8.75	17.5	8.75
Specific Investment, $ /SWU/yr.	137	109	120
Power, MW	2,430	4,730	2,050
Specific Power, kW/SWU/yr.	0.278	0.270	0.234
Operating Cost (excluding Power Costs) $ Million/yr.	14	16	16
Natural UF_6 Feed, short tons of UF_6	17,700	35,400	17,700
Depleted UF_6 Tails, short tons of UF_6	14,500	29,000	14,500
Enriched UF_6 Product, short tons of UF_6	3,200	6,400	3,200
Value of Product $ Millions per yr.(2)	340	680	340
Manpower Requirements	900	925	900

Source: USAEC Estimates.

Notes: (1) Assumes (a) U.S. location; (b) Product assay of 3-4 percent U-235; (c) Tails assay of 0.25 percent U-235. In 1971 Dollars.

(2) At current AEC prices.

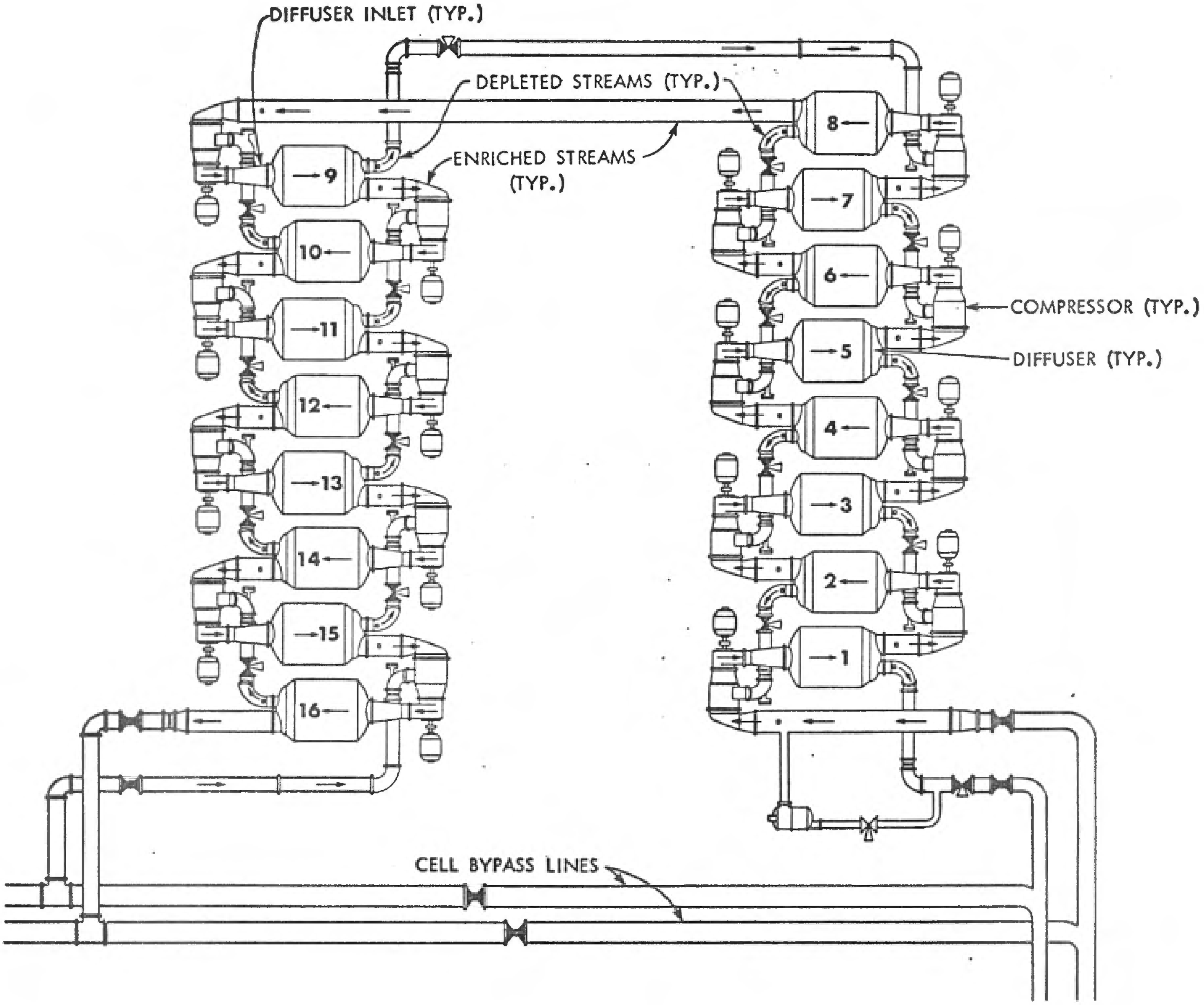

FIGURE 1-7. 16-STAGE PROCESS CELL FOR URANIUM ENRICHMENT

Instrumentation
Process Support Facilities
Electrical System
Heat Removal System.

Construction Phase

The factors important in site location are reviewed below along with approximate construction costs and requirements of labor, land, and supplies for construction of a gaseous diffusion enrichment plant.

Site Choice. Because of the quantity of electric power required by the gaseous diffusion process, the primary siting consideration is the availability of low cost electricity. In fact, to minimize power transmission costs, the diffusion plant should be located "next door" to the power plant. Also, to keep construction costs low, the plant should be located near existing infrastructure. Similarly, a site near a sizeable construction labor pool would aid in minimizing construction costs. A final factor relates to the need for a location near a significant body of water to aid in wasteheat disposal. In view of these factors, it would appear reasonable to locate the facility near LG-2.

Costs. Costs for construction and equipping gaseous diffusion enrichment plans have been estimated by the USAEC and are given below:

Date of Construction Start	1970	1970	Projected Advanced Technology
Separative Work Capacity, Millions SWU/yr.	8.75	17.5	8.75
Costs, Billions of Dollars	1.2	1.9	1.05

These costs are in 1971 dollars, and are for an "average" U.S. location. The construction cost envisioned for the 875 million SWU plant with technology advances forecast for the late 1970's is estimated at $1.05 billion as compared to $1.2 billion estimated for one started in the early 1970's. The larger 17.5 million, it is noted that economies of scale occur in the construction phase, as doubling plant capacity results in an increase of construction costs of only 60 percent. Market conditions would not appear to warrant consideration of the larger plant in the very near future, but it is also noted that such economies can still be realized in plant expansion.

Land, Labor, Material, and Utility Requirements. The land requirements, construction materials, and utility capacities needed for a 8.75 million-SWU plant are shown in Table 1-23. Construction labor needs will, of course, vary. Peak

TABLE 1-23. KEY REQUIREMENTS FOR AN 8.75 MILLION-SWU/YR. PLANT (utilizing current technology)

Requirement	Item	Quantity
Physical	Plant Ground Coverage	300 acres
	Process Buildings Ground Coverage	50 acres
	Process Support Facilities	25 acres
	Feed, Product, and Waste Storage	7 acres
	Soil Bearing Requirement	2500 psf on spread-type footings
Construction Materials	Concrete	300,000 cu yd
	Reinforcing Steel	10,000 tons
	Structural Steel	60,000 tons
	Process Steel Pipe	15,000 tons
	Auxiliary Systems Pipe	10,000 tons
	Organic Coolant	1,500 tons
Utilities Capacities	Electric Power	2,430 Mw
	Water Supply	20,000,000 gpd
	Dry Air	12,000 scfm
	Steam	350,000 lb/hr

Source: USAEC.

labor needs will occur 3 to 4 years after the start of the project and total about 4,000 workers. An estimated 140,000 man-months of construction labor will be required for the entire project.

Total construction time is expected to be 6 years. However, production can actually begin after installation of a minimum number of stages. Thus, although the plant will be completed 6 years after the start of construction, the equivalent of a full production start can possibly be achieved in 4 years and 8 months.

Production Phase

Key factors concerning production requirements and output of a diffusion plant are reviewed below. These include operational needs in terms of inputs of feed and labor, output, effluents, and operating costs.

Feed Material, Product, and Tails. An 8.75 million SWU/yr. plant, operating solely on natural UF_6 would, at full capacity, have the following feed requirements and product and tails yield.

		Annual Rate		
Material	Uranium, kg.	UF_6, kg.	Uranium Short Tons	UF_6, Short Tons
Natural UF_6 Feed	11.1 million	16.1 million	12,200	17,700
UF_6 Tails, 0.25 percent U-235	9.1 million	13.2 million	10,000	14,500
UF_6 Product, 3 percent U-235	2.0 million	2.9 million	2,200	3,200

The feed requirements and product and tails yield for a 17.5 million SWU/yr. plant are simply double those stated above.

If the degree of enrichment of the product changes, the feed requirements would change too. For example, if the desired product was less than 3 percent enriched, more enriched UF_6 would be produced per SWU, and more feed would be required.

Labor Requirements. The permanent labor force required for an 8.75 million SWU plant is estimated to be about 900. Table 1-24 gives the employment breakdown for such a plant.

Effluents. The major effluent of the gaseous diffusion process is the depleted UF_6 tails. This gas is normally stored in steel cylinders designed for this purpose. Aside from possible uses of depleted uranium in fast breeder reactors, it is likely that there will be little need for the depleted UF_6. Therefore, perpetual storage should be planned.

TABLE 1-24. PERSONNEL BREAKDOWN FOR AN 8.75 MILLION-SWU/YR PLANT

Plant Superintendent and Staff	6
Operations	
Central Control Room	8
Shift Superintendent	5
Process Operations	164
Process Engineering	12
Laboratory	27
Fire and Guards	44
Utilities	60
Plant Engineering	28
Finance and Materials	113
Industrial Relations	22
Maintenance	
Services	104
Field Crews	207
Maintenance Engineering	12
Shops	88
	900

Source: USAEC

The waste heat of gaseous diffusion plants now in operation in the U.S. is rejected by means of cooling towers. Cooling water requirements naturally vary with the ambient temperature. The water flow rate for a typical 8.75 million SWU/yr. plant located in the U.S. will have a winter-to-summer range of 160,000 to 285,000 gpm. The makeup water required is about 13,500 gpm.

Operating Costs. Direct annual operating costs for a U.S. location (at 1970 prices), exclusive of power costs, are estimated at $13.9 million and $15.8 million for an 8.75 million-SWU plant and a 17.5 million-SWU plant, respectively. The components of these costs are shown in Table 1-25. Power costs, however, constitute the major operating expense, in that, in most cases, they are at least six times the cost of all other operating expenses. For example, if power costs are assumed to be 4.5 mills/Kwh, the cost of power becomes almost $100 million a year for the 8.75 million-SWU plant and $190 million a year for the 17.5 million-SWU plant.

For the 4.5 mill rate, operating costs per SWU for the 8.75 million-SWU plant would then be about $13.00 exclusive of amortization, and other financial costs.

In view of the current SWU price of $38.50,* it would appear that sufficient margin for capital costs and profit exists at the 4.5 mill power rate. In order to more fully evaluate the effect of power rates and capital costs on a gaseous diffusion plant project, the following section analyzes their impact in light of expected SWU prices.

Evaluation of Potential Worth

Given the above-mentioned specifications and additional qualifying assumptions, estimates of potential worth, cash flow, and profits are calculated below to demonstrate the potential feasibility of locating a gaseous diffusion uranium enrichment plant in the James Bay region. The probabilistic cash flow model used to evaluate other projects is again used here, but with slightly different emphasis. This is due to the fact that there are a variety of policy decisions which ave to be made concerning the project which will have substantial influence on levels of profitability (i.e., ownership, tax status, power pricing policies). Lacking these, the following analysis is intended to demonstrate two key factors which have strong policy implications:

(1) Payback Period: As such a large financial commitment is involved (both public and private), one important factor to consider is the time period involved in paying off debt. From the public viewpoint, it would be desirable to demonstrate that payback can be achieved in a reasonably short period of time. Thus, while the expected life of such a plant may be fifty years, in terms of analyzing the project for feasibility it is more desirable to use a twenty-year life with the expectation that payback can occur

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* August, 1973, price as recently proposed by the USAEC. The USAEC has announced that the above prices will be escalated 2 percent per year in the future.

TABLE 1-25. OPERATING COSTS, EXCLUDING POWER, FOR DIFFUSION PLANTS

Item	At 1970 Technology	
Plant Separative Capacity, Millions of SWU/yr.	8.75	17.5
Labor	Millions of Dollars/Year	
Operations	3.7	3.8
Maintenance	4.0	4.1
Overhead	2.4	2.4
Material (1)	2.1	2.2
Carrying Cost for Uranium Inventory (2)	1.7	3.3
Total	13.9	15.8

Source: USAEC

Note: (1) Excludes such spare equipment items as compressors, motors, converters (fully equipped), transformers, etc., which are included in the capital cost.

(2) In-cascade inventory only. No external stockpiles on hand or any other kind of working capital is included.

somewhere within the period. That is, if debts are amortized over a twenty-year project life (with fourteen years of production), any rate of return in excess of zero means that payback is complete. This would then permit a second look at the project, which would have much higher returns as no interest or loan principal payments would be involved.

(2) Power Pricing Policies: In previous sections, continued mention has been made of the need for reasonably low power costs. The exact contract price for power would, of course, have to be negotiated. However, to demonstrate the sensitivity of profits to power rate structures, power costs are permitted to vary widely.

Data Inputs and Assumptions

Capital Outlays. There are six capital outlays, each of which is associated with one of the six construction phases. As noted previously, the 1970 cost for the 8.75 million SWU facility is $1.2 billion, with the possibility of being as low as $1.05 billion by the late 1970's. Again, these are for U.S. locations, assumedly in developed areas. Costs for the same facility in James Bay could be expected to be higher. Total capital costs are sampled from the following distribution for each of the construction phases:

Annual Construction Cost (millions of $)	Median Total Construction Cost (billions of $)	Probability* (percent)
170 - 189	1.08	5
190 - 209	1.20	30
210 - 229	1.32	50
230 - 250	1.44	15

Each construction phase is sampled independently and each is financed by 100 percent debt financing by six separate 14-20 year loans at 9 percent interest. In all cases, principal payback begins in the first year of actual production (year 7). Thus, to finance Phase 1 construction, capital is borrowed in year 1 for 20 years; for Phase 2 construction, capital is borrowed in year 2 for 19 years, etc.

So as not to permit depreciation and terminal worth calculations to affect the yield calculations (which they would for an assumed 20-year project that, in reality, has a much longer life), it is assumed that the diffusion plant is initially operated as a Crown Corporation which pays no taxes. In this way, depreciation has no effect, as in normal cash flow calculations this item is treated as an annual expense to arrive at taxable income, and then added back to annualized cash flow (as cash reserve).

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* See footnote, page 1-49.

Annual Operating Expenses. Only two annual expenses are considered, power costs and all other production costs. Of particular interest are power costs, which comprise about 85 percent of total annual operating costs. Power costs are calculated as follows:

.278 kilowatts[*] per year are required to produce one (1) SWU.

In terms of kilowatt hours, .278 Kw x 365 days x 24 hours x .995 efficiency = 2423.1 Kwh/per SWU per year are required.

On a unit cost basis for mill rates between 4 and 8 mills per Kwh, the following costs result:

Mill Rate	Power Cost per SWU
4	$ 9.69
5	12.12
6	14.54
7	16.96
8	19.38

On an annual basis for 8.75 million SWU's, costs are selected from the following distribution:

Annual Power Cost (millions of $)	Median Mill Rate (per Kwh)	Probability[**] (percent)
84.8 - 105.9	4.5	25
106.0 - 127.1	5.5	25
127.2 - 148.3	6.5	25
148.4 - 169.6	7.5	25

It is noted that the probabilities associated with each cost range are equal. Unlike previous examples, this should not be interpreted to mean that they are reflective of what power costs are really expected to be. In this particular case, equal probabilities are assigned in order to permit the widest possible selection of annual power costs. In this way, analytic sensitivity is increased as more possible combinations of power costs and capital costs can be evaluated.

"Other production costs" comprise all other annual operating expenses, including purchase of feed material. As note above, operating costs exclusive of power costs are minimal in comparison to power costs themselves. Annual "other production costs" are computed from escalating the 1970 figure ($14.0 million per year) by 2 percent annually. Thus, for an assumed first production beginning in 1982, total other production costs would be $20.5 million per year in 1982, and $27.1 million in 1996.

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[*] Note: Specific power requirements of 0.234 Kw/SWU/yr are projected for 1980, based on power recovery schemes which are expected to markedly improve overall power utilization efficiency. As these technological advances are still in the thinking stages, the existing specific power requirements are more suitable for this analysis.

[**] See footnote, page 1-49.

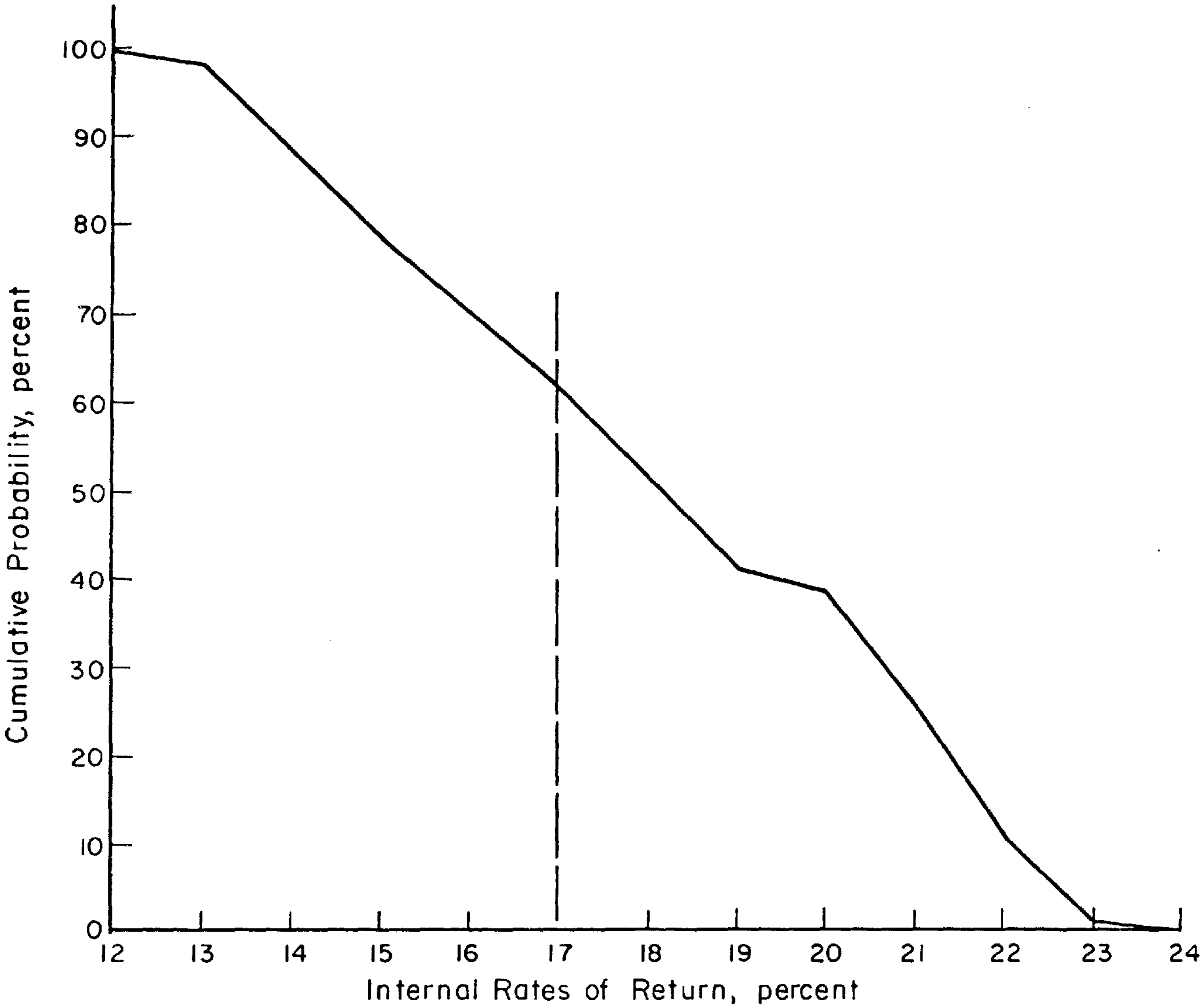

FIGURE I-8. URANIUM DIFFUSION PLANT PROJECT INTERNAL RATES OF RETURN VERSUS PROBABILITY

TABLE 1-26. 8.75 MILLION SWU GASEOUS DIFFUSION URANIUM ENRICHMENT PLANT
ITERATION NUMBER 10

YEAR	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
CAPITAL OUTLAYS																				
PHASE 1 CONSTR.																				
-----INTERNAL FUNDS	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	0.0	0.0	0.0	0.0	0.0	7.9	8.6	9.3	10.2	11.1	12.1	13.2	14.4	15.7	17.1	18.6	20.3	22.1	24.1
PHASE 2 CONSTR.																				
-----INTERNAL FUNDS	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	0.0	0.0	0.0	0.0	0.0	6.7	7.3	7.9	8.6	9.4	10.2	11.2	12.2	13.3	14.5	15.8	17.2	18.7	20.4
PHASE 3 CONSTR.																				
-----INTERNAL FUNDS	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	0.0	0.0	0.0	0.0	0.0	7.7	8.4	9.1	10.0	10.9	11.8	12.9	14.1	15.3	16.7	18.2	19.9	21.6	23.6
PHASE 4 CONSTR.																				
-----INTERNAL FUNDS	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	0.0	0.0	0.0	0.0	0.0	8.7	9.4	10.3	11.2	12.2	13.3	14.5	15.8	17.2	18.8	20.5	22.3	24.3	26.5
PHASE 5 CONSTR.																				
-----INTERNAL FUNDS	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	0.0	0.0	0.0	0.0	0.0	8.2	8.9	9.8	10.6	11.6	12.6	13.8	15.0	16.4	17.8	19.4	21.2	23.1	25.2
PHASE 6 CONSTR.																				
-----INTERNAL FUNDS	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	0.0	0.0	0.0	0.0	0.0	7.0	7.6	8.3	9.0	9.8	10.7	11.7	12.7	13.9	15.1	16.5	18.0	19.6	21.3
TOTALS	0.0	0.0	0.0	0.0	0.0	0.0	46.0	50.2	54.7	59.6	65.0	70.8	77.2	84.2	91.7	100.0	109.0	118.8	129.5	141.1
ANNUAL EXPENSES																				
POWER COSTS	0.0	0.0	0.0	0.0	0.0	0.0	86.6	86.6	86.6	86.6	86.6	86.6	86.6	86.6	86.6	86.6	86.6	86.6	86.6	86.6
OTHER PROD. COSTS	0.0	0.0	0.0	0.0	0.0	0.0	20.5	20.9	21.3	21.7	22.2	22.6	23.1	23.5	24.0	24.4	25.0	25.5	26.0	26.5
INTEREST ON LOANS	18.4	34.0	52.0	72.3	91.5	107.8	107.8	103.7	99.1	94.2	88.9	83.0	76.6	69.7	62.1	53.9	44.9	35.0	24.4	12.7
TOTAL DEPRECIATION	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
TOTALS	18.4	34.0	52.0	72.3	91.5	107.8	214.9	211.1	207.0	202.5	197.6	192.2	186.3	179.8	172.7	164.8	156.4	147.1	136.9	125.8
ANNUAL REVENUES																				
SALES OF U-235	0.0	0.0	0.0	0.0	0.0	0.0	394.7	402.6	410.6	418.9	427.2	435.8	444.5	453.4	462.4	471.7	481.4	490.8	500.6	510.6
TOTALS	0.0	0.0	0.0	0.0	0.0	0.0	394.7	402.6	410.6	418.9	427.2	435.8	444.5	453.4	462.4	471.7	481.4	490.8	500.6	510.6
TAXABLE INCOME	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	144.9	216.4	229.6	243.6	258.2	273.6	289.7	306.9	325.0	343.7	363.7	384.8
INCOME TAX	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
NET INCOME	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	144.9	216.4	229.6	243.6	258.2	273.6	289.7	306.9	325.0	343.7	363.7	384.8
CASH FLOW	-18.4	-34.0	-52.0	-72.3	-91.5	-107.8	133.8	141.3	148.9	156.8	164.6	172.8	181.0	189.5	198.0	206.9	216.0	224.9	234.2	243.7

TERMINAL WORTH = .000

YIELD(IN PERCENT) = 23.207
PRESENT WORTH AT 0 PERCENT = 2236.058
PRESENT WORTH AT 5 PERCENT = 1004.409
PRESENT WORTH AT 10 PERCENT = 435.420

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        TABLE I-27.   8.75 MILLION SWU GASEOUS DIFFUSION URANIUM ENRICHMENT PLANT
                                    ITERATION NUMBER     15

      YEAR                  1      2      3      4      5      6      7      8      9     10     11     12     13     14     15     16     17     18     19     20
CAPITAL OUTLAYS

PHASE 1 CONSTR.
-----INTERNAL FUNDS       0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0
-----LOAN PRINCIPAL       0.0    0.0    0.0    0.0    0.0    0.0    9.3   10.2   11.1   12.1   13.2   14.3   15.6   17.0   18.6   20.2   22.1   24.0   26.2   28.6
PHASE 2 CONSTR.
-----INTERNAL FUNDS       0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0
-----LOAN PRINCIPAL       0.0    0.0    0.0    0.0    0.0    0.0    6.9    7.6    8.2    9.0    9.8   10.7   11.6   12.7   13.8   15.1   16.4   17.9   19.5   21.2
PHASE 3 CONSTR.
-----INTERNAL FUNDS       0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0
-----LOAN PRINCIPAL       0.0    0.0    0.0    0.0    0.0    0.0    8.8    9.6   10.5   11.4   12.4   13.5   14.8   16.1   17.5   19.1   20.8   22.7   24.7   27.0
PHASE 4 CONSTR.
-----INTERNAL FUNDS       0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0
-----LOAN PRINCIPAL       0.0    0.0    0.0    0.0    0.0    0.0    8.9    9.7   10.6   11.5   12.6   13.7   14.9   16.3   17.7   19.3   21.1   23.0   25.0   27.3
PHASE 5 CONSTR.
-----INTERNAL FUNDS       0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0
-----LOAN PRINCIPAL       0.0    0.0    0.0    0.0    0.0    0.0    7.9    8.6    9.4   10.3   11.2   12.2   13.3   14.5   15.8   17.2   18.7   20.4   22.3   24.3
PHASE 6 CONSTR.
-----INTERNAL FUNDS       0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0
-----LOAN PRINCIPAL       0.0    0.0    0.0    0.0    0.0    0.0    9.4   10.2   11.2   12.2   13.3   14.5   15.8   17.2   18.7   20.4   22.2   24.3   26.4   28.8

            TOTALS        0.0    0.0    0.0    0.0    0.0    0.0   51.3   55.9   60.9   66.4   72.3   78.9   86.0   93.7  102.1  111.3  121.3  132.2  144.2  157.1

ANNUAL EXPENSES

POWER COSTS               0.0    0.0    0.0    0.0    0.0    0.0  158.5  158.5  158.5  158.5  158.5  158.5  158.5  158.5  158.5  158.5  158.5  158.5  158.5  158.5
OTHER PROD. COSTS         0.0    0.0    0.0    0.0    0.0    0.0   20.5   20.9   21.3   21.7   22.2   22.6   23.1   23.5   24.0   24.4   25.0   25.5   26.0   26.5
INTEREST ON LOANS        21.8   38.0   58.6   79.5   98.0  120.0  120.0  115.4  110.4  104.9   98.9   92.4   85.3   77.6   69.1   60.0   49.9   39.0   27.1   14.1
TOTAL DEPRECIATION        0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0

            TOTALS       21.8   38.0   58.6   79.5   98.0  120.0  299.0  294.8  290.1  285.1  279.6  273.5  266.9  259.5  251.6  242.8  233.4  223.0  211.6  199.1

ANNUAL REVENUES

SALES OF U-235            0.0    0.0    0.0    0.0    0.0    0.0  394.7  402.6  410.6  418.9  427.2  435.8  444.5  453.4  462.4  471.7  481.4  490.8  500.6  510.6

            TOTALS        0.0    0.0    0.0    0.0    0.0    0.0  394.7  402.6  410.6  418.9  427.2  435.8  444.5  453.4  462.4  471.7  481.4  490.8  500.6  510.6


TAXABLE INCOME            0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0  131.8  162.3  177.6  193.9  210.8  228.9  248.0  267.8  289.0  311.5

INCOME TAX                0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0

NET INCOME                0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0    0.0  131.8  162.3  177.6  193.9  210.8  228.9  248.0  267.8  289.0  311.5

CASH FLOW               -21.8  -38.0  -58.6  -79.5  -98.0 -120.0   44.5   52.0   59.6   67.5   75.3   83.5   91.7  100.2  108.7  117.6  126.7  135.6  144.9  154.4

TERMINAL WORTH                =        .000

YIELD(IN PERCENT)             =      12.088
PRESENT WORTH AT  0 PERCENT   =     945.751
PRESENT WORTH AT  5 PERCENT   =     322.179
PRESENT WORTH AT 10 PERCENT   =      57.526
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           TABLE 1-28.  8.75 MILLION SWU GASEOUS DIFFUSION URANIUM ENRICHMENT PLANT
                                  ITERATION NUMBER     35

       YEAR                 1       2       3       4       5       6       7       8       9      10      11      12      13      14      15      16      17      18      19      20
CAPITAL OUTLAYS

PHASE 1 CONSTR.
-----INTERNAL FUNDS       0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0
-----LOAN PRINCIPAL       0.0     0.0     0.0     0.0     0.0     0.0     8.5     9.3    10.1    11.1    12.1    13.1    14.3    15.6    17.0    18.5    20.2    22.0    24.0    26.2
PHASE 2 CONSTR.
-----INTERNAL FUNDS       0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0
-----LOAN PRINCIPAL       0.0     0.0     0.0     0.0     0.0     0.0     7.7     8.4     9.2    10.0    10.9    11.9    13.0    14.1    15.4    16.8    18.3    19.9    21.7    23.7
PHASE 3 CONSTR.
-----INTERNAL FUNDS       0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0
-----LOAN PRINCIPAL       0.0     0.0     0.0     0.0     0.0     0.0     7.6     8.3     9.1     9.9    10.8    11.8    12.8    14.0    15.2    16.6    18.1    19.7    21.5    23.4
PHASE 4 CONSTR.
-----INTERNAL FUNDS       0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0
-----LOAN PRINCIPAL       0.0     0.0     0.0     0.0     0.0     0.0     6.7     7.3     8.0     8.7     9.5    10.3    11.3    12.3    13.4    14.6    15.9    17.3    18.9    20.6
PHASE 5 CONSTR.
-----INTERNAL FUNDS       0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0
-----LOAN PRINCIPAL       0.0     0.0     0.0     0.0     0.0     0.0     7.9     8.6     9.4    10.2    11.2    12.2    13.3    14.5    15.8    17.2    18.7    20.4    22.3    24.3
PHASE 6 CONSTR.
-----INTERNAL FUNDS       0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0
-----LOAN PRINCIPAL       0.0     0.0     0.0     0.0     0.0     0.0     7.9     8.6     9.4    10.3    11.2    12.2    13.3    14.5    15.8    17.2    18.8    20.4    22.3    24.3

             TOTALS       0.0     0.0     0.0     0.0     0.0     0.0    46.5    50.6    55.2    60.2    65.6    71.5    77.9    84.9    92.6   100.9   110.0   119.9   130.7   142.4

ANNUAL EXPENSES

POWER COSTS               0.0     0.0     0.0     0.0     0.0     0.0   159.0   159.0   159.0   159.0   159.0   159.0   159.0   159.0   159.0   159.0   159.0   159.0   159.0   159.0
OTHER PROD. COSTS         0.0     0.0     0.0     0.0     0.0     0.0    20.5    20.9    21.3    21.7    22.2    22.6    23.1    23.5    24.0    24.4    25.0    25.5    26.0    26.5
INTEREST ON LOANS        20.0    38.1    56.0    71.7    90.3   108.8   108.8   104.6   100.1    95.1    89.7    83.8    77.3    70.3    62.7    54.4    45.3    35.4    24.6    12.8
TOTAL DEPRECIATION        0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0

             TOTALS      20.0    38.1    56.0    71.7    90.3   108.8   288.3   284.5   280.3   275.8   270.9   265.3   259.4   252.8   245.7   237.7   229.2   219.8   209.6   198.3

ANNUAL REVENUES

SALES OF U-235            0.0     0.0     0.0     0.0     0.0     0.0   394.7   402.6   410.6   418.9   427.2   435.8   444.5   453.4   462.4   471.7   481.4   490.8   500.6   510.6

             TOTALS       0.0     0.0     0.0     0.0     0.0     0.0   394.7   402.6   410.6   418.9   427.2   435.8   444.5   453.4   462.4   471.7   481.4   490.8   500.5   510.6


TAXABLE INCOME            0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0    77.0   156.3   170.5   185.1   200.6   216.7   234.0   252.2   271.0   291.0   312.3

INCOME TAX                0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0

NET INCOME                0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0    77.0   156.3   170.5   185.1   200.6   216.7   234.0   252.2   271.0   291.0   312.3

CASH FLOW               -20.0   -38.1   -56.0   -71.7   -90.3  -108.8    60.0    67.5    75.1    83.0    90.8    99.0   107.2   115.7   124.2   133.1   142.2   151.1   160.4   169.9

TERMINAL WORTH           =         .000

YIELD(IN PERCENT)        =       14.956
PRESENT WORTH AT  0 PERCENT =  1193.797
PRESENT WORTH AT  5 PERCENT =   459.528
PRESENT WORTH AT 10 PERCENT =   138.050
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Annual Revenues. Annual revenues from sales of enriched uranium are generated by multiplying USAEC fixed prices per SWU times annual production. The August, 1973, price of \$38.50 per SWU x 8.75 million SWU yields an annual revenue of \$394.7 million for the first year of production. For subsequent years, revenues are escalated 2 percent per year, again according to US AEC pricing policies. Thus, for the final production year, 1966, revenues are estimated at \$520.8 million.

Outputs

Multiple iterations of the cash flow model were run to estimate the projected rates of return possible. For the 20-year project, rates of return range from 12 percent to 23 percent (Figure 1-8), where rates of return were 17 percent or better 62 percent of the time. The evenness of the distribution is, of course, a result of assigning equal probabilities to the ranges between the five power costs, which is somewhat unrealistic. The effects of varying power rates and construction costs on the sensitivity of yields are well illustrated in the sample output (Tables 1-26 to 1-28). For example, the lowest yield of 12 percent results from reasonably high construction costs (\$1.35 billion) and high power costs (\$158.5 million, which corresponds to a 7.5 mill rate). Conversely, the highest yield of 23 percent results from low construction costs (\$1.19 billion) and low power costs (\$86.6 million, which corresponds to a mill rate of 4.1). Intermediate yields of between 16 and 18 percent can result from: (1) Average construction and power costs; (2) high power, but low construction costs; or, (3) low power costs associated with high construction costs.

A more useful sensitivity analysis appears as Figure 1-9 which illustrates the possible construction cost and power cost combinations which yield the same rate of return. For example, a 7.5 mill power rate combined with a \$1.32 billion construction cost yields a 12 percent rate of return. A 5.0 mill rate and a 1.43 billion construction cost would also yield about a 12 percent return.

The most important finding of the analysis is that there is a payback within the 20-year period for any of the ranges of power or construction cost assumptions. In most cases, it would appear that 100 percent payback could be achieved in less than 15 years. Once payback is complete, a second series of analyses, where interest payments and debt amortization are excluded, would generate rather high rates of return. This is due to the fact that revenues from sales are in excess of 2-1/2 times total production costs.

In terms of financing such a venture, one other point is worthy of mention. There is a possibility that for long term, guaranteed supply contracts, the USAEC may suggest that buyers, in consideration for a lower pegged rate per SWU, make partial prepayments for total contracted supply. The suggested prepayment is as follows:

(1) Payment in three equal installments

(2) First installment is made in year agreement is signed, (i.e., 1974 for 1982 delivery) and equal to one-third of the value of the <u>total</u> contract

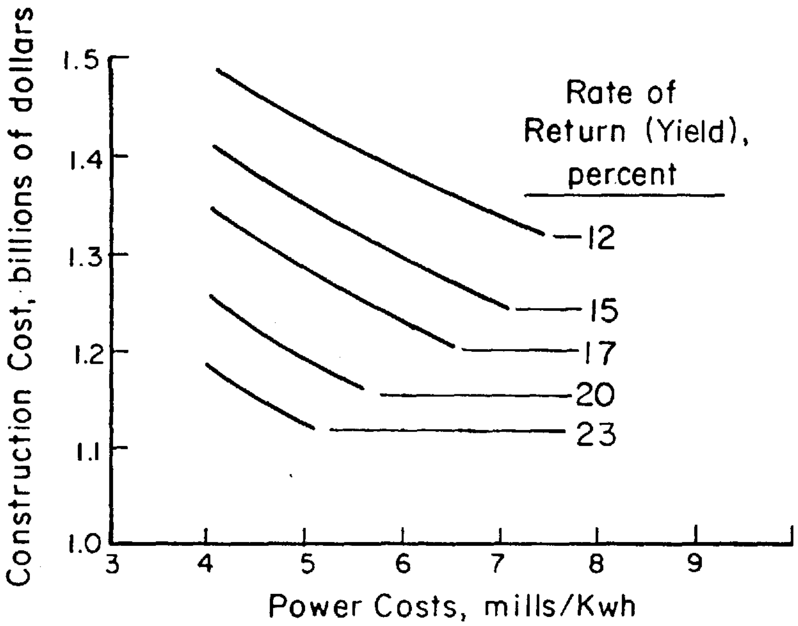

FIGURE I-9. URANIUM DIFFUSION PLANT YIELD VERSUS COSTS RELATIONSHIPS

(3) This payment procedure entitles the buyer to a price of \$36.50 per SWU (as opposed to \$38.50), but still escalates at 2 percent per year over the life of the contract.

For a 10-year contract, worth about \$4.5 billion, such a prepayment procedure would require a first installment of about \$1.5 billion. Under this type of arrangement, the net result is the availability of interest-free capital for the entire construction of the facility. Certainly, this particular method of financing should be looked into further.

Access to U.S. Technology

A critical point to consider in promoting the establishment of uranium enrichment plant in the James Bay region is access to United States technology, which currently is classified. To date, the USAEC has no official policy regarding release of United States technology for multinational projects. Policy, to date, has been to consider release of enrichment technology on a case by case basis.

Admittedly, it would be easier to obtain the needed technology if an American partner were involved in the project. This would facilitate access to more preliminary information, for example, to permit detailed engineering cost estimates by cleared American nationals. However, the ultimate release of all the technology needed to Canadian interests would be a matter to be decided.

To date, there has been little or not historical information on USAEC's willingness to release information for enrichment projects involving foreign countries. It is believed that the release of such information would likely follow the apparent policy taken thus far by the USAEC in connection with planned enrichment projects by United States companies. In these cases, companies or groups of companies that have demonstrated financial capability and serious intent have gained access to enough technology to permit preliminary evaluation. It appears that further technology will be released in more and more detail to the company or companies still demonstrating interest in continuing with the project. Thus, the entire technology would only be released to those companies or groups of companies that ultimately build an enrichment plant. It is felt that Canadian enrichment project plans would likely be handled in a similar fashion.

The current reluctance of the USAEC to release technology is a stance which many think will have to change. The pressing need for new enrichment plant capacity in the early 1980's is cited as a major driving force behind this possible relaxation of current policy by the USAEC.

The USAEC is not the only source of gaseous diffusion uranium enrichment technology. The British, French, and Russians also have the capability. In fact, the French are currently interested in finding additional partners for a gaseous diffusion plant project.

Competing Technologies

Uranium enrichment may be performed using technologies other than the gaseous diffusion method. One such technique, the gas centrifuge, has aroused considerable interest in recent years and is now under development in several countries. The URENCO consortium noted earlier is one of the more publicized projects* (a joint project of the U.K., West Germany, and the Netherlands). Generally, the potential advantages of centrifuge technology are cost related in that:

(1) It is alleged that smaller sized plants could be built without compromising total cost per unit output, thus requiring a substantially lower total capital investment. In contrast, the smallest efficient diffusion plant is the 8.75 million kg SWU size** which requires a minimum $1 billion investment (without power plant).

(2) Electric power requirements for the centrifuge method are only 14-16 percent of gaseous diffusion electricity requirements. This is especially advantageous to the Europeans where power costs are generally much higher.

However, there are also strong political considerations related to expansion of enrichment facilities of any kind. These center on the increased desire of European governments to avoid dependence on the U.S. for such a strategic service as enrichment and the possibility that U.S. suppliers will acquire a near monopoly over enrichment facilities which would further reinforce their competitive lead.

What must be underlined, however, is that the potential cost savings of the centrifuge method are still largely speculation in that only small scale pilot plants are in operation. The Dutch plant (25 metric tons of separative work capacity) was completed this spring and is now undergoing systems testing, while the German (25 metric tons) and Britich (15 metric tons) plants are scheduled to be operational by the end of the year. Current URENCO plans call for the construction of a 300 metric ton facility for

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* See, for example: A. J. Surrey, "The Future Growth of Nuclear Power", Part 1. Supply & Demand, Energy Policy, (Sept. 1973); Uranium Enrichment. Atomic Industrial Forum: A Report of an Ad Hoc Forum Policy Committee, (Jan. 1973); The Nuclear Power Industry in Europe, European Atomic Forum, (May 1972); K. Pavitt, "Technology in Europe's Future", Research Policy, (July 1972); E. V. Anderson, "Enriched Uranium Gets Lift from Energy Crunch", Chemical & Engineering News, (August 20, 1973); J. Trotter, "Centrifuge Process", Financial Times, (October 18, 1972); D. Fishlock, "Prospects Bright for Uranium Centrifuge Investment", Financial Times, (March 9, 1973); T. Schoeters, "Uranium Enrichment Study by GE", Financial Times, (January 22, 1973); L. Beaton, The Reform of Power, London: Chatto & Windus, (1972); A. Tucker, "Blow to French Nuclear System", The Guardian, (May 29, 1973).

** Smaller plants would have much higher unit investment costs because a plant requires approximately the same number of process stages to span the same product assay regardless of whether the plant is of large or smaller capacity.

operation by late 1975. As two to four years of pilot plant operation will probably be required to demonstrate centrifuge lifetimes, it is not expected that the economics of the European centrifuge program will be firmly established until 1976 or 1977.

A key cost consideration in commercially successful centrifuge enrichment relates to the unit cost of the centrifuges themselves. It is estimated that 100,000 centrifuge machines would be required for the 300 metric ton facility, or about 40-50 percent of the total facility cost. As larger facilities will require proportionately more centrifuges, mass production methods for making the components will be needed. At present, this is far from being the case.

An approximate economic comparison of the gaseous diffusion and gas centrifuge processes under typical U.S. conditions circa 1980 is presented below*. The comparison assumes that the technical and economic goals of the European centrifuge program are actually achieved (which is by no means assured).

COMPARATIVE PROCESS ECONOMICS FOR TYPICAL PLANTS
(1972 U.S. $)

	Gaseous Diffusion	Gas Centrifuge
Specific investment ($/SWU/yr)	120-130	120-150
Annual capital charge (%)	15	20
Power consumption (kwh/SWU)	2,050-2,400	300-400
Cost of power (mills/kwh)	8-9	8-9
Separative work cost ($/SWU)		
Capital	18.00-19.50	24.00-30.00
Power	16.40-21.60	2.40-3.60
Operation & maintenance	1.50-2.00	4.00-6.00
Total	35.90-43.10	30.40-39.60

For specific investment, the initial Dutch pilot plant is reported to have cost between $400-500/SWU/year, though estimates of expected costs for the 300 ton facility to be completed by 1976 indicate per unit costs of $200 or less. This is still not competitive with gaseous diffusion, but the target for capacity coming into operation in the 1978-80 period is

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* Based on the Report of the Ad Hoc Forum Policy Committee, Op. cit.

$120-150/SWU/year (and as low as $100/SWU/year in the early 1980's. Escalation of capital costs is not included, but the power cost of 8-9 mills/kwh is the minimum projected for nuclear power plants coming on line in 1980*. The annual capital charge is higher for the centrifuge facilities to allow for centrifuge replacement. If centrifuge lifetimes are adequate (5-10 years), in order to meet a specific investment target of $120-150/SWU/year, centrifuges must cost no more than $500-750 each. If this set of costs is achievable, the centrifuge process would appear to be about 10-15 percent more economical than gaseous diffusion during the 1980's, especially if power costs climb above 8-9 mills/kwh. However, this 10-15 percent "savings" must be weighed against the probability of realizing the cost minimums noted above. Additionally, where current technology gaseous diffusion plants can utilize cheager power sources, the gaseous diffusion process is clearly more economical.

Plan of Action

If the James Bay Development Corporation is to pursue an uranium enrichment project, a serious effort must be undertaken under the leadership of a responsible well-known individual.

The following major tasks should be undertaken in approximate order of presentation:

(1) Obtain a long-term commitment for "low" cost electric power. Such a commitment is necessary to assure production of enriched uranium at "competitive" prices. Since the enrichment plant envisioned would likely require 2,400 megawatts at a 100 percent load, price concessions should be obtainable

(2) Form a consortium of companies interested in being partners in an enrichment project. Because a uranium enrichment project is over a billion dollar venture, it is likely that a group of companies and/or governments would be involved from the standpoint of raising capital. In addition, participation by a United States company or companies is desirable to aid in obtaining USAEC technology. Such a consortium could include utility companies of the United States, Western Europe, and Japan that are concerned with obtaining future supplies of enriched uranium. Uranium mining companies, and the current diffusion plant operators, Union Carbide and Goodyear may also have an interest in participating.

One consortium that has already declared interest in a gaseous diffusion project consists of Westinghouse, Union Carbide, and Bechtel.

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* European costs would probably be higher.

(3) Obtain the approval and blessing of AECL and the Canadian government.

(4) Obtain financial commitment for the project. In addition to the capital that may be available from the consortium, loans will surely be needed. The cost of capital, if considered as part of the operating cost of an enrichment facility, can be as important as power costs. Therefore, a long-term line of credit at reasonably "low" rates should be established.

(5) After the consortium is formed and has the needed financial backing, the USAEC can be approached for serious discussions regarding release of technology needed for an engineering design study. At this point, it would be wise to utilize the capabilities of a United States based engineer-architect to facilitate the release of data from the USAEC.

(6) Further plans can be made if the project looks favorable after the engineering design study.

Vertical Integration Potential

The establishment of a gaseous diffusion plant in the James Bay region might spawn other directly related business opportunities though not all could be located in the region. They include: production of UF_6 from yellowcake (U_3O_8); production of enriched UO_2 from enriched UF_6; and the production of steel cylinders for the storage and transportation of UF_6.

Production of Uranium Hexafluoride

Enrichment of uranium by the gaseous diffusion process depends on the availability of an uranium compound that can be maintained in gaseous form over long periods of time under safe conditions of temperature and pressure. Uranium hexafluoride (UF_6) is such a compound.

The production of UF_6 from U_3O_8 is accomplished by a series of chemical reactions and physical separation and purification steps, all conducted under scrupulous conditions of control and safety. In addition to avoiding personnel exposure to radiation hazards, the processing has been designed to permit strict accountability for the uranium that enters the system. One system for the conversion is described briefly.

Yellowcake and recycled fuel fabrication scraps are dissolved in nitric acid forming uranyl nitrate which is extracted and purified by solvent extraction procedures. Purified and concentrated uranyl nitrate is then thermally decomposed to uranium trioxide (UO_3). Pellets of UO_3 are reduced to UO_2 by hydrogen derived from cracked ammonia and then reacted with anhydrous HF to form UF_4. Subsequently, UF_4 and F_2 are exposed to a high temperature flame and the UF_6 produced is liquefied by refrigeration and transferred to a shipping container where it regasifies on warming to ambient temperature.

As suggested in Chapter 1., the production of UF_6 from yellowcake can be completely independent of the mining and milling of uranium. However, it will require adequate supplies from extraneous sources of ammonia, nitric acid, sulfuric acid, fluorspar, and other processing chemicals and materials as well as skilled chemical operators and an experienced and competent management and laboratory staff. At 1972 prices, a new plant to treat 5,000 tons per year of uranium could cost up to $25 million.

Production of Enriched Uranium Dioxide

After passing through the gaseous diffusion enrichment process, the enriched UF_6 is converted to uranium dioxide (UO_2) and fabricated into fuel elements for nuclear reactors. Again, chemical reactions and physical manipulations are involved although the scale of operations tends to be smaller than for the production of UF_6, largely because of nuclear safety considerations. The processing steps required to produce the basic UO_2 are given in a subsequent paragraph. The fabrication of fuel elements is not described because these procedures depend on the reactor core design selected.

The enriched UF_6 is mixed with water to form a solution of uranium oxyfluoride which is reacted with ammonia to produce ammonium diuranate which precipitates. The precipitate is filtered, dried, and then thermally decomposed to uranium trioxide (UO_3). The UO_3 is subsequently reduced with hydrogen to uranium dioxide (UO_2). The powdered UO_2 serves as the raw material for fuel elements fabricated by ceramic processing or by reduction to elemental uranium.

Estimates of the cost of facilities to produce enriched UO_2 are not available in the open literature. Eldorado Nuclear Ltd. has such a facility already at Port Hope, Ontario, as a part of an extensive installation that can handle natural or enriched uranium. However, many purchasers of enriched uranium may prefer to purchase the material as UF_6 and have the fuel elements fabricated independent of the enrichment entity.

Production of Cylinders for Uranium Hexafluoride

When dry, uranium hexafluoride is essentially noncorrosive and may be stored and shipped in mild steel cylinders. This applies to natural uranium as well as to the enriched and depleted tailings forms. If UF_6 is to be produced in Quebec, there could be an opportunity to manufacture the equivalent of 1335 14-ton cylinders per year, having an annual sales value of about $1 million. As noted in Chapter 1., the cylinders are used for shipments of natural and enriched UF_6 and for the indeterminate storage of the depleted UF_6 tails from the enrichment plant.

Manufacture of cylinders for this service follows standard metallurgical practice for fail-safe welding. Any well equipped welding shop having X-ray inspection and annealing equipment should be capable of handling an order for the cylinders, exclusive of valve fittings.

Further, specifications for the 14-ton or 20-ton sizes that commonly are used should be readily available after the decision to install an enrichment facility has been made. No major difficulty is anticipated in the procurement of cylinders if the opportunity to make them is presented.

TABLE 1-29. CURRENT FREE WORLD CAPACITY FOR CONVERSION OF U_3O_8 TO UF_6

Country	Company	Capacity Short Tons of Uranium	
		Current	Planned Additional
United States(1)	Allied Chemical Corp.	10,000	
	Kerr-McGee Corp.	5,000	5,000(2)
Canada	Eldorado Nuclear, Ltd.	2,750	2,250
United Kingdom	Government Plant at Springfields	3,300	---
France	Government Plant at Pierrelatte	3,800	---
	Total	24,850	7,250
	Total plus planned	32,100	

Source: Compiled by BCL from various published references.

(1) The USAEC maintains U_3O_8 to UF_6 conversion facilities but does not compete with private industry for non-government business.

(2) Expansion when markets permit.

CHAPTER 2. NICKEL

CHAPTER 2. NICKEL

SUPPLY/DEMAND SITUATION TO THE YEAR 2000

Occurrence, Beneficiation and Metal Recovery

Occurrence

Compared to many other economic metals, nickel is relatively abundant in the earth's crust, reaching 1,000 - 2,000 p.p.m. in some large bodies of ultra-basic rocks. The primary abundance of comparably priced metals such as copper and tin, is only 30 p.p.m. and 3 p.p.m. respectively. Consequently, nickel is very expensive in relation to its abundance in the earth. This is basically caused by nickel's chemical properties which inhibit its concentration into rich ore-bodies. Nickel readily enters into the structure of the magnesium silicate mineral, olivine, and has a lesser affinity for sulphur than many other metals.

Sulfide Ores. Ores in which nickel is in the sulfide form are found in, or close to, bodies of basic to ultrabasic rocks. These rocks are rich in iron and magnesium and relatively low in silica. They are dark, instrusive, deep-seated, igneous rocks. The common and close association of sulfide nickel ores with rocks of these types leads to the conclusion that the nickel was a constituent of the original molten rock or magma, the nickel sulfide being derived from it either during or after the cooling period. The sulfide minerals may be present in massive or disseminated form, usually close to geologie faults.

Pentlandite, $(NiFe)_9S_8$, is the only sulfide mineral containing nickel which is of major economic significance. It is similar to brass or bronze in color and, ideally, has a nickel content of 34.2 percent. Pentlandite is generally found in association with pyrrhotites, which are iron sulfides that are very similar in appearance to pentlandite. Small amounts of nickel may substitute for iron in these sulphides, making them nickeliferous. However, the pentlandite which occurs with pyrrhotites generally contains much greater quantities of nickel than the pyrrhotites themselves. Another sulfide mineral commonly associated with pentlandite is the copper ore, chalcopyrite, $CuFeS_2$. It is present in variable relationship to the nickel content depending on the character of the host rock. In basic environments (host rock approximately norite) the nickel: copper ratio approaches 1:1 while in ultrabasic associations (host rock like peridotite) the ratio ranges from 4:1 to 6:1.

Cobalt invariably accompanies nickel in its sulfide ores, nickel: cobalt ratios being in the range of 50:1 to 25:1. Significant amounts of the platinoid metals, mainly platinum and palladium, are also present in most sulfide deposits of nickel.

The worlds largest sulfide ore occurrence is the famous Sudbury District deposits in Canada. Other significant nickel sulfide deposits are found in Australia, the U.S.S.R., and Manitoba, Canada.

Oxide Ores. Nickel oxide ores are lateritic materials formed by tropical weathering of surface exposed peridotite or its alternation product, serpentine. Peridotite is an ultrabasic rock composed mainly of olivine, a silicate of iron and magnesium which typically contains as much as 0.3 percent nickel. Weathering action forms limonitic nickel ore $(Fe,Ni)O(OH)$ and garnierite, $(Ni,Mg)_6Si_4O_{10}(OH)_8$, a silicate type nickel ore.

As in the case of the sulphide ores, cobalt is always present in oxide ores. Nickel: cobalt ratios are in the range 100:1 to less than 10:1, with the limonites being notably rich in cobalt.

By far the greatest known reserves of nickel are in the oxide ores, most of which occur between the Tropics of Cancer and Capricorn. Major laterite deposits are found in New Caledonia, Indonesia, the Philippines, and Cuba.

Beneficiation and Metal Recovery

Nickel sulfide ores lend themselves to standard practice in minerals beneficiation, whereas the lateritic ores are difficult to beneficiate and methods are still being developed to recover the metallic values economically.

Sulfide Ores. Beneficiation and recovery of metallic values from sulfide ores depends in part on the grade of ore, the associated impurities, and the opportunity to blend ores from various sources to assure a reasonably uniform feed for pyrometallurgical processes. However, the primary beneficiation of virtually any sulfide ore can be effected by standard concentrating techniques.

In the Sudbury District, mixed ore is crushed and passed over electromagnets to separate a lump product rich in nickel for direct smelting. The balance of the ore is ground to liberate the remaining sulfides which are successively floated to recover nickel, copper, and pyrrhotite concentrates. These concentrates and lump ore are treated separately by a variety of processes that yield nickel, nickel oxide, a nickeliferous iron, pelletized iron ore, or copper.

Sulfide ore produced outside the Sudbury District or not directly controlled by the major Sudbury operators usually is crushed, ground, floated by a bulk separation process whereby the nickel and copper minerals report in one stream and the pyrrhotite in a waste stream that is discarded. The nickel/copper concentrates may be sold to the major Sudbury operators and blended into their smelting scheme, or they may be treated hydrometallurgically (a la Sherritt Gordon) for recovery of nickel and copper separately with associated cobalt and precious metals in a concentrate form.

Laterite Ores. Since the nickel content of oxide and silicate nickel ores is chemically disseminated throughout the parent material, lateritic ores are not amenable to physical concentration as practiced on sulphide ores. In order to protect the extraction plant from serious fluctuations in feed characteristics, selective and carefully controlled multi-face mining operations must be combined with large-scale storage and blending facilities.

The extraction of nickel from the lower grade lateritic ores containing 1.0 to 2.5 percent nickel requires different processing techniques from those used on sulphide ores. Until recently, the lower grade lateritic ores were restricted in general use to the production of ferronickel additives used in the manufacture of stainless steels. These ferronickels are produced by several different pyrometallurgical processes.

Hydrometallurgical and vapometallurgical techniques can also be used to obtain nickel oxide or powder and nickel pellets, respectively.

Product Form

Most of the nickel produced and delivered to markets is one of three types: electrolytic nickel, nickel oxide sinter, or ferronickel. Other products are produced and marketed in relatively small quantities, primarily to the foundry or chemical industries. These are nickel pellets, shot, briquettes, rondelles and powder.

Uses and Substitutes for Nickel

Uses for Nickel

Nickel is used for many reasons. It is added to steel to impart strength and corrosion resistance. It is used as electroplated surfaces on automobile bumpers and other items as a foundation for a highly lustrous chromium finish. Nickel is used in high temperature materials and superalloys because it is highly resistant to oxidation at elevated temperatures. There are also other reasons why nickel is used, e.g., resistance to corrosion, magnetic properties, electrical properties, and thermal properties.

Main Applications. Free World consumption of primary nickel in 1970 is distributed to major intermediate products in percentage terms as follows:

Product	Percent of Total Consumption
Stainless steel	41
High nickel alloys	14
Nickel plating	13
Constructional alloy steels	11
Iron and steel castings	9
Copper and brass alloys	3
Other	9
TOTAL	100

Thus, stainless steels represent the major end-use market for nickel accounting for slightly over 40 percent of demand. The next four important end-use areas each account for only 9-14 percent of the nickel consumed. Details of major applications are given below:

Stainless steel. Stainless steels offer the user a good combination of corrosion resistance, strength, fabricability, and aesthetic characteristics at low cost when compared to substitutes. They are very versatile materials, and are used practically everywhere. Some of the most important of these applications are:

- Household and institutional utensils, cutlery and flatware
- Automotive hubcaps, trim, and antipollution devices
- Chemical process equipment
- Aircraft frame and engine parts
- Architectural and other construction work
- Dairy and other food processing equipment.

High nickel alloys. These alloys can be further separated into two classes, superalloys and other high-nickel materials. Superalloys are generally jet engine materials and are those that are alloyed and processed in such a way so as to achieve maximum structural integrity and fatigue resistance at elevated temperatures. This generally means relatively expensive processing in a vaccum induction furnace, followed by cacuum-arc-remelting. Unlike superalloys, other high nickel materials are used in a wide variety of different applications. Some of the most important are:

- Burner element sheathing for stoves and other electric appliances
- Heat exchanger tubing in nuclear power plants

- Chemical process equipment and parts
- Industrial heat treating
- Electronic components.

Nickel plating. Nickel coatings, generally with a thin layer of chromium over the nickel, are used for decorative surfaces and for building up worn parts. The automotive industry uses over 50 percent of the nickel consumed in electroplating. Major applications are bumpers, door handles and windshield wiper parts. The consumer appliances industry uses a major portion of the remainder.

Constructional alloy steels. These alloys contain only small amounts of nickel--generally less than 3 percent. These alloys have superior strength and ductility over common steels and are used for forged and machined components in the automotive, farm implements, electrical generating equipment, and aircraft and hydro-space equipment industries. Other important markets are for 9 percent nickel steel cryogenic vessels used in bulk storage of liquid nitrogen and oxygen.

Iron and steel castings. The main application for primary nickel in the iron and steel castings area is to make ductile iron. Ductile iron is used principally for automotive, farm and industrial equipment applications, e.g., crankshafts, steering knuckels, and gears. Furthermore, a large amount of cast iron pipe is now made of ductile iron. Other nickel containing cast products are Ni-Resist and Ni-Hard irons, two specialty irons developed by INCO with unique properties. Markets for the latter irons are small, however.

Copper and brass alloys. Major applications for nickel containing copper-base alloys are coinage, condenser tubes, electrical resistance alloys, nickel silver, and various cast brasses and bronzes. These materials are employed in a wide variety of applications, but due to relatively high prices for the alloys, these alloys are being replaced with cheaper substitutes.

Other Uses. Nickel is used for a variety of other applications such as catalysts, chemicals, iron-nickel alloys, magnets, welding rods, etc.

Substitutes for Nickel

Alternate materials may substitute for nickel in many applications. However, with few exceptions, use of substitutes would mean increased cost or some sacrifice in product performance.

Stainless steels containing chromium, manganese, and relatively little nickel can be used in place of the conventional 300 series steels for some applications. Columbium, molybdenum, chromium, and vanadium can be substituted for nickel in some of the carbon steel alloys and cobalt, chromium, and columbium-base alloys can be used in place of some of the nickel and superalloys. Manganese, molybdenum, and copper can be used in place of nickel in some types of iron castings, and the modified stainless steels described above also can be used in some cast forms. Platinum, cobalt, and copper can take the place of nickel in some types of catalysts.

The biggest field for substituting other materials for nickel is in its end uses where nickel-bearing material is used for its corrosion resistance, high strength, or special magnetic and electronic properties. For example, carbon steel clad with titanium could perform satisfactorily in many applications now filled by the stainless steels and high-nickel alloys. Many plastics have equal or superior corrosion resistance compared with the nickel-bearing corrosion-resistance materials and plastic coatings on high-strength steels or other material are comparatively inexpensive. Paint, enamel, or other attractive metallic or non-metallic finishes and aluminum can be used in place of nickel-chromium used in trim.

World Sources of Nickel

Current Production

World mine production of nickel, by country, is presented in Table 2-1 for 1960, 1965, and 1970. Canada has been the mining leader, followed by New Caledonia and the U.S.S.R. In 1970, Canada supplied about 42 percent of the 735,000 short tons total, while New Caledonia and the U.S.S.R. contributed 21 and 16 percent respectively. Production of nickel as metal, nickel oxide sinter, and ferronickel amounted to 654,000 short tons. Unless otherwise identified, further references to tons in this chapter will refer to short tons.

Reserves

The reserves of nickel as estimated by the U.S. Bureau of Mines in 1969 are presented in Table 2-2. These data indicate that at the current price of nickel (approximately $1.50 per pound) world reserves increase about 60 percent to 100 million tons. In terms of low-cost reserves, New Caledonia has the greatest amount with 15.5 million tons of nickel recoverable at $1.50 per pound or less. If nickel recoverable up to $2.00 per pound is considered, the Philippines have the largest reserves, with 30.0 million tons available, and Cuba is second with 18.0 million tons available.

Current and Future Capacity

Estimated world mine capacity is given in Table 2-3 by country. Current capacity is near 800,000 tons of contained nickel. Canadian mine capacity is nearly 50 percent of world capacity, while New Caledonia and the U.S.S.R. each have roughly 17 percent of the total.

By 1980 world mining capacity has been estimated by BCL at 1,230,000 tons per year. Canada will still have the largest mining capacity. However, Canada's share will have dropped to 36 percent. Details of planned expansions are given in Table 2-4 and in subsequent sections on Canada and New Caledonia. Not all of the planned expansions are expected to be completed by 1980. Therefore, the estimated 1980 mine capacity of 1,230,000 tons of contained nickel is less than current plans would indicate.

Canada. Canada is the world's leading miner of nickel, its 1970 mine production totaled 305,900 tons of contained nickel or about 42 percent of the world's total mine production. Current production capacity based on mill capacity is estimated to be 365,500 tons. Nickel resources consist mainly of sulfide ore.

Reserves. International Nickel Company of Canada, Ltd. (INCO) has Canadian reserves of 6.35 million tons of contained nickel. The second largest reserve position is held by Falconbridge Nickel Mines, Ltd., and it is estimated that their Sudbury and Wabowden properties alone contain 1.3 million tons of contained nickel. Total Canadian reserves recoverable at about $1.50 per pound or less are estimated to be 8 million tons.

Current and Future Capacity. Table 2-5 summarizes the current status of Canadian nickel mines and mills and give 1970 production statistics. It is obvious that INCO is by far the leading producer, accounting for 83 percent of Canadian production. In fact, INCO in 1970 accounted for 38 percent of the world's mine production of nickel.

In 1972, Canadian capacity for producing nickel (in concentrate form) amounted to 365,500 tons. Planned expansion of production capacity, mainly from INCO and Falconbridge mines and mills, was slowed by slack demand in 1971-1972. It is expected, however, that by 1975 mine production capacity will be slightly over 420,000 tons of contained nickel and that by 1980 the figure will be nearly 450,000 tons.

New Caledonia. French-controlled New Caledonia is the world's second largest miner of nickel. In 1970, mine production totaled 152,700 tons of contained nickel or about 21 percent of total world production. Production capacity is undergoing a major expansion.

TABLE 2-1. WORLD MINE PRODUCTION OF NICKEL (thousands of short tons of nickel content)

Country	1960	1965	1970
Canada	215	259	306
New Caledonia	59	67	153
U.S.S.R.	68	88	122
Cuba	16	32	44
Australia	--	--	33
Other	19	34	77
TOTAL	377	480	735

Source: Metallgeschellschaft

TABLE 2-2. WORLD NICKEL RESOURCES RECOVERABLE AT VARIOUS PRICES (millions of short tons of nickel)

Country	Price (constant 1969 dollars per pound)		
	$ 1.50	$ 1.75	$ 2.00
New Caledonia	15.5	16.0	16.5
Cuba	10.0	16.0	18.0
U.S.S.R.	10.0	10.0	10.0
Canada	8.0	10.0	12.5
Phillipines	9.0	15.0	30.0
Indonesia	5.0	6.5	8.0
Other	3.0	4.4	4.6
TOTAL	60.5	77.9	99.6

Source: U.S. Bureau of Mines

TABLE 2-3. ESTIMATED CURRENT AND FUTURE WORLD MINE CAPACITY FOR NICKEL
(thousands of short tons)

Country	Year		
	1970	1975	1980
Canada	353	424	445
New Caledonia	138	176	205
U.S.S.R.	123	160	190
Australia	49	61	98
Cuba	39	39	39
United States	15	15	15
Dominican Republic	--	35	35
Indonesia	11	25	100
Phillipines	--	8	38
Other	32	57	65
TOTAL	760	1,000	1,230

Source: Estimated by Battelle Columbus Laboratories.

TABLE 2-4. PLANNED EXPANSIONS IN NICKEL MINING CAPACITY (1)

Country/Company	Annual Capacity, (tons of contained nickel)	Estimated Date of Production	Destination of Concentrates	Remarks
Australia				
Freeport Minerals Company & Metals Exploration N.L., Greenvale deposit, Queensland	27,000	1975-76	Own smelter	Smelter to be built at Townsville.
Freeport Minerals Company, Australian Consolidated Minerals, N.L., & Metals Exploration N.L., Mt. Keith, Western Australia	27,000	1975-76	Own smelter	May begin production in 1980's.
Posedon N.L., Union Oil Company of California, Homestake Mining Company and The Hanna Mining Co., Windarra, Western Australia	30,000	1974	Own smelter	Initial production expected to be 12,000 ton/year. Mine production may begin in 1974.
Selection Trust Limited, Spargoville, Western Australia	6,000	1972	Kalgoorlie	Production from No. 2 location to begin in 1972; from No. 3 in 1973.
Botswana				
Bamangwato Concessions, Ltd. Selebi-Pikwe	18,700	Late 1973	Port Nickel, Louisiana, U.S.A.	
Colombia				
The Hanna Mining Company, Compania Niquel Chevron and Industrial Development Institute of Colombia, Cerro Matoso, Cordoba	18,750	1975-76	Own smelter	Re-evaluation of plans in progress.
Dominican Republic				
Falconbridge Dominicana C. por A.	31,500	1971	Own smelter	On stream in mid 1972.
Finland				
Outokumpu Oy, Vuonos mine	2,200	1972	Own smelter	Operates several nickel mines and a smelter in Finland.
Greece				
Intercontinental Mining and Abrasives, Inc., and Southland Mining Co. Evia mine, Lake Lonina	9,000	1974	Own smelter	Proposal not yet approved by government.
Guatemala				
Exploraciones y Explotaciones Mineral Izabal, S,A. (EXMIBAL), Lake Izabal	30,000	1974	Own refinery	To produce fire-refined nickel. Production possible 3 years after construction begins.
Indonesia				
P.T. International Nickel Indonesia, Malili, Sulawesi	25,000	1979	Own refinery	Plans to expand to 50,000 tons/year. Financing incomplete. May begin production in mid-70's.
P.T. Pacific Nikkel Indonesia, Waigeo area, Irian Barat	25,000	1979	Own smelter	12,000 tons ore shipped to Sherritt Gordon Mines Ltd. for test treatment.
Sulawesi Nickel Development Cooperation Company (SUNIDECO), Pomalea, Sulawesi	13,200	1979	Own smelter	To produce ferronickel for export to Japan.
Republic of the Philippines				
Marinduque Mining & Industrial Corporation Nonoc Island	37,500	1975	Own refinery	Constructing Sherritt Gordon type refinery.
Infanta Mineral and Industrial Corporation Palawan Island	8,000	1972	Japan	
Rhodesia				
Johannesburg Consolidated Investment Company, Shangani mine	4,500		Own smelter	Plans to expand to 7,500 tons nickel a year.
Yugoslavia				
Government company	13,000		Own smelter	Will produce 48,500 tons of ferronickel. Could be in production in 1976.

Source: Annual reports, technical press, and estimates by BCL.

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(1) Mining expansions by Canada and New Caledonia are given in subsequent sections.

TABLE 2-5. CURRENT STATUS OF CANADIAN NICKEL MINES AND MILLS

Company	Location	Mine	Mill	Milling Capacity tons per day	Estimated Average Nickel Content of Ore, Percent	Estimated Capacity for Nickel Production Concentrates in Short Tons per Year	1970 Production Short Tons of Contained Nickel	Remarks
Operating								
International Nickel Company of Canada, Ltd.	Sudbury, Ontario	10 different mines	Copper cliff	76,600	---	---	---	
	Shebandowan, Ontario		Shebandowan	2,500	---	---	---	
	Thompson, Manitoba	4 different mines	Thompson	18,400	2.8	300,000	259,435	
Falconbridge Nickel Mines, Ltd.	Falconbridge, Ontario	8 different mines	Falconbridge, Strathcona Fecunis Lake & Hardy	14,100	1.3	45,000	42,071	
	Wabowden, Manitoba	Manibridge	Manibridge	1,000	2.55	6,800	---	
Sherritt Gordon Mines Ltd.	Lynn Lake, Manitoba	---	Lynn Lake	3,500	0.66	8,400	6,438	
Giant Mascot Mines, Ltd.	Hope, British Columbia	---	Hope	1,750	0.77	3,300	1,435	
Texmont Mines, Ltd.	Timmins, Ontario	Timmins	No mill	600	---	---	---	
Dumbarton Mines, Ltd.	Bird River, Manitoba	Bird River	Use Canadian Consolidated Faraday's Mill at Gordon Lake, Ontario	800	.86	2,000	1,170	
Societe Miniere d' Exploration Somex Ltée	Lac Edouard Quebec	---	Lac Edouard	240	---	---	---	Opened in 1972
Inactive								
Consolidated Canadian Faraday, Ltd.	Gordon Lake Ontario	---	---	---	---	---	627	Mine closed in 1972. Mill is still operating.
Renzy Mines, Ltd.	Haunault Township, Quebec	---	Renzy	1,000	0.43	1,000	957	Mine closed in 1972.
Hudson-Yukon Mining Co., Ltd.	Kluane Lake, Yukon Territory	Wellgreen	Wellgreen	600	2.04	3,600	---	
Prospective Producers								
Noranda Mines	Timmins, Ontario	Langmiur	---	700	1.87	2,000	---	Production expected in 1973
			TOTAL			372,100	312,133	

Source: Compiled by BCL.

Reserves. Almost all of New Caledonia's nickel resources are in the form of laterite ore. Reserves are estimated to be 15.5 million tons of nickel recoverable at about $1.50 per pound or less. New Caledonia has the world's largest low-cost nickel reserves.

Capacity and Future Capacity. Nickel is mined in New Caledonia by approximately 15 companies. The French company Societe Le Nickel (SLN) controls two-thirds of them. The New Caledonian Nickel Company, a producer of ferronickel, is a cooperative venture between SLN and Kaiser Aluminum and Chemical Company. This company has recently opened new mine capacity. Current mine production capacity is estimated to be as follows:

Company	Capacity (tons of contained nickel)
SLN	50,000
New Caledonian Nickel Co.	32,500
Small Mines	93,000
	175,500

Many companies had been planning nickel mining projects on New Caledonia circa 1969-70; however, demand for nickel was not as strong as predicted, and many of these planned projects have been delayed. The following are projects that will likely be started when market conditions permit.

The Goro deposit in New Caledonia at one time was to have been exploited by COFIMPAC, a joint venture between Inco and a group of French companies. Although final French government approval has not been received, it appears that Inco may develop the Goro deposit alone. Current plans call for mining to begin in 1977 at the rate of 22,500 tons of contained nickel per year. Inco is planning further expansions to 55,000 and eventually to 110,000 tons per year. Dates for these last two projects have not been announced, but obviously these latter expansions will not take place until the 1980's or 1990's.

A second expansion in New Caledonia is planned by PENAMAX, a joint venture between Societe Miniere et Metallurgique de Penarroya, S.A. and American Metal Climax, Inc. Originally, a 25,000 ton per year mine-mill complex was planned for start-up in 1975. These plans have been delayed and now it appears that the planned capacity will not be available until after 1980.

Another joint venture team of Societe Nationale des Petroles d'Aquitaine (SNPA) and Freeport Mineral Company is seeking French government approval to exploit New Caledonian nickel. A 25,000 ton per year nickel smelter is under consideration for production by 1978.

Still another project is planned by Patino-Pechiney-Ugine Kuhlmann (PUK)-Graenges. The project is expected to lead to the production of 40,000 tons of nickel per year in the form of ferronickel. Project feasibility studies and site location are underway. The nickel to be exploited is in the Poum deposit in northern New Caledonia, initial production is not likely to begin until the late 70's or early 80's.

As is evident from the above many new mining projects have been planned on the island of New Caledonia. They have all been delayed beyond initially targeted start-up dates. These delays can be attributed to the slack demand for nickel in the early seventies and to the slowness of the French government in granting mining concessions.

Nickel mining capacity by 1975 is expected to be approximately 176,000 tons, and 205,000 tons by 1980. Capacity increases beyond 1980 are expected, but such a forecast would be highly speculative. If all planned expansions take place New Caledonia's nickel mining capacity will eventually reach 285,000 tons per year.

World Demand for Nickel

Current Consumption

Nickel containing steels and other products are used in a wide variety of end-use products. World consumption of primary nickel in 1970 totaled 647,000 tons. Free World consumption by end-use markets was as follows:

End-Use Market	Percent of Total Consumption
Consumer Products	16 percent
Machinery and Transportation	14 percent
Automotive	12 percent
Electronic	9 percent
Chemical	8 percent
Petroleum	8 percent
Process	7 percent
Aircraft	6 percent
Energy Conversion	4 percent
Marine	3 percent
Architecture	3 percent
Coinage	2 percent
All Others	8 percent

It is likely that world consumption patterns are similar. The most striking feature about the end-use markets for nickel is the fact that they are very diverse. Also, the major end-use market, consumer products, accounts for only 16 percent of consumption.

In the 1960's nickel demand nearly doubled and was especially strong in the latter years of the decade. In fact, demand was outstripping production capacity to the point that the industry was scheduling expansions as rapidly as possible while outsiders frantically searched for new sources and programmed big new installations. Just as expanded capacity caught up with consumption, in 1970, industrial stagnation set in for North America and Western Europe, and later for Japan. By early 1972, International Nickel Company (INCO)--responsible for nearly 38 percent of the world's supply--had cut back on operations by 33 percent in an attempt to reduce swollen inventories. Thus, neither 1970 nor the immediately subsequent years confirm statistically the 7 percent annual growth rate claimed by producers to be the long-term potential for nickel.

Future Demand

Growth in the demand for nickel in all of the end uses cited previously is expected. Certain end-use markets will grow at higher rates than in the past. For instance, the use of stainless steels should expand more rapidly in new applications such as auto emission control devices, air and water pollution control equipment, and nuclear reactors. The use of nickel in superalloys is expected to accelerate as gas turbine applications grow, especially in non-aircraft applications such as ship propulsion, automotive propulsion, electrical power generation, and other stationary power uses.

Battelle's estimated demand for primary nickel for the years 1980, 1990, and 2000 is shown in Table 2-6 for the world and by geographical area. This table shows that total demand is estimated to increase from 647,000 tons in 1970 to 2,120,000 tons in 2000, an increase of 3.3 times or at an average rate of growth of 4.0 percent annually. Estimates for demand in 1980 were based on approximately current growth rates. Beyond 1980, the following growth rates were used:

Western Europe & U.S.	3.0 percent
Japan	5.0 percent
Canada	5.0 percent
Rest of World	3.8 percent

The estimates shown in Table 2-6 are for primary nickel only and does not include demand for scrap or secondary nickel. The demand for mined nickel is, however, slightly higher than the demand for scrap or secondary nickel. On average, about 90 percent of the nickel contained in mined ore is actually recovered. Therefore, the estimated demand for primary nickel must be multiplied by 1.11 to obtain the demand for nickel in the form of nickel ore.

TABLE 2-6. WORLD DEMAND ESTIMATES FOR PRIMARY NICKEL

Geographical Area	Estimated Demand, Thousands of Short Tons			
	1970	1980	1990	2000
Western Europe	193	300	400	540
United States	156	250	335	450
Japan	108	200	325	530
Canada	15	20	35	50
Rest of World	175	260	375	550
TOTAL	647	1,030	1,470	2,120

Source: Metal Statistics and Battelle Columbus Laboratories' Estimates.

Supply/Demand Relationships

Figure 2-1 presents the estimated world supply/demand situation for nickel to the year 2000. Demand for nickel ore is plotted to reflect the estimates for primary nickel derived in the previous section. Annual mine capacity is shown through 1980, and is above the demand curve through that year. Though the mine capacity curve is shown only through 1980, there is a surfeit of intended projects which will likely be sufficient to handle demand through 1990. New capacity beyond 1990 will have to come from unannounced projects.

Also shown in Fibure 2-1, the cumulative demand for nickel ore from 1970 to 2000 will total over 43 million tons of ore against which nickel resources recoverable at $2.00 per pound or less are expected to total 99 million tons. Therefore, more than twice the nickel needed between 1970 and 2000 occurs in reserves known now. Without a doubt, additional reserves will be discovered between now and 2000.

Probable Price Trends

The current price of nickel cathodes is $1.53. Since the reserves recoverable at $2.00 per pound or less far exceeds expected demand until 2000, no scarcity of resources is expected and barring possible temporary undersupply situations it is likely that nickel prices will not exceed $2.00 per pound in 1970 dollars.

On the shorter term, between now and 1990, it appears that the current market price is sufficient to justify the many projects planned for the late 1970's and early 1980's. Because of the temporary lack of demand and other difficulties, these projects have been delayed; but they will eventually come to fruition. It is likely that projects already announced and expansions in current mines will add sufficient nickel supplies to prevent large price increases until after 1990. The price of nickel is expected to be as follows (in current dollars):

Year	Price per pound
Current Price	$ 1.53
1980	1.55
1990	1.60
2000	1.90

It should be remembered that these estimates do not include the effects of inflation which can be considerable. For instance, INCO's labor rates increased 35 percent in 1969, and by about 20 percent in 1972, these increases yielded producer price rises of 24 and 15 percent respectively.

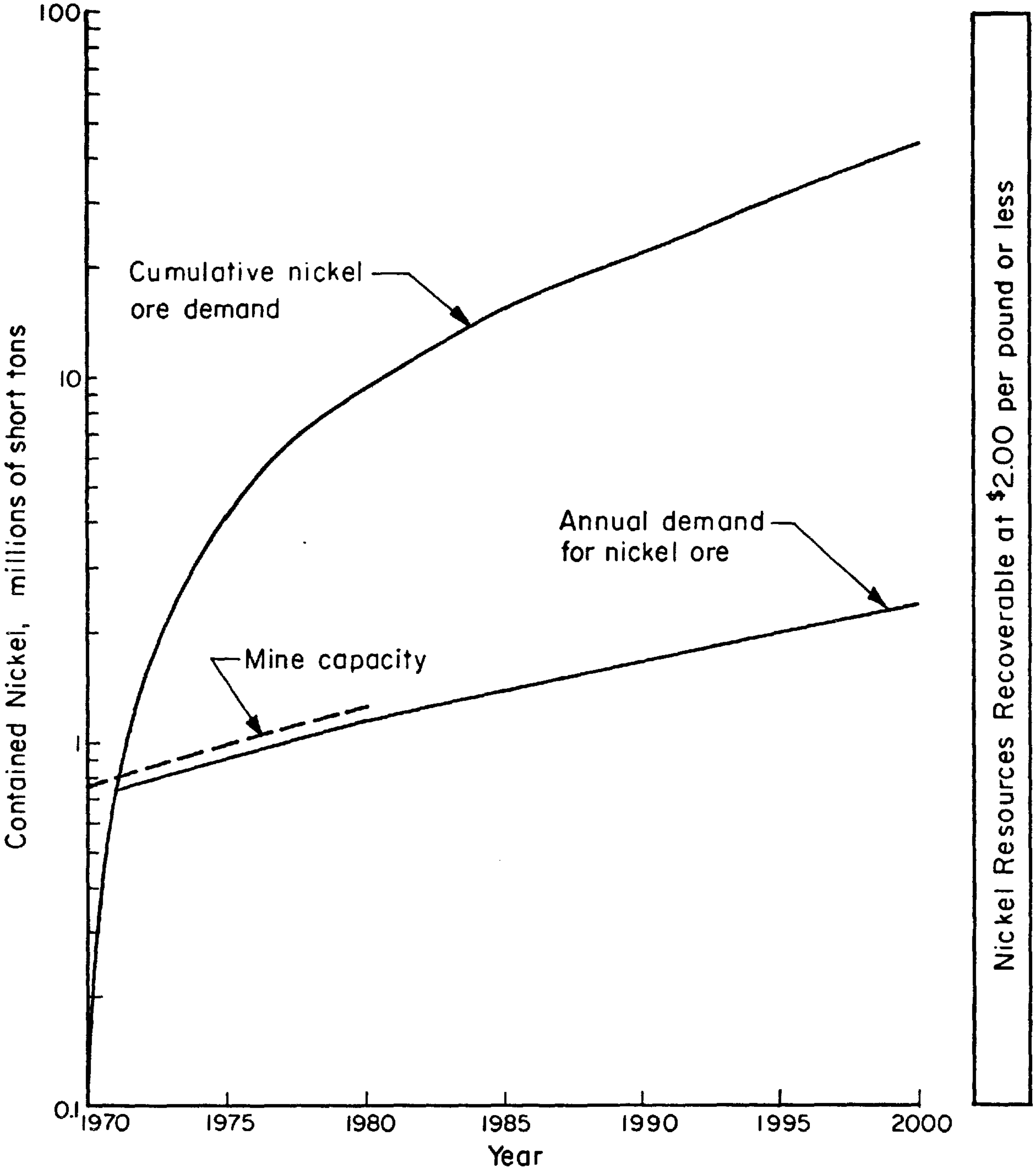

FIGURE 2-1. ANNUAL AND CUMULATIVE DEMAND FOR NICKEL ORE AND RESERVES

Need for a New Producer in the James Bay Region

Based on the foregoing analysis, there is no apparent need for a new nickel mine in the James Bay region before 1990. Even new discoveries made by the present major producers are likely to remain unexploited for several years unless they are very high grade. However, this does not preclude the possibility that a modest new mine in the region might be opened shortly after discovery if the discoverer happened to be an aggresive marketer, as well as technically competent to produce nickel products. Both of these qualifications suggest involving one of the smaller producers--e.g., Sherritt Gordon or Hanna Mining Company--in future exploration in high priority zones. Should such activity start in the near future, it is conveivable that a mine/mill project in the James Bay region could be initiated by about 1985.

HYPOTHETICAL MINE/MILL PROJECT FOR THE JAMES BAY REGION

Introduction

The two hypothetical mine-mill projects which follow are based on nickel deposits in both basic and ultrabasic rock.

The minimum economic deposit is described in terms of deposit geology and likely size and location. Development costs associated with each of the projects as well as operating costs associated with mining and beneficiation of the ore are estimated.

Project Summary

Two potential nickel projects are envisioned. Each project consists of developing an underground nickel deposit containing 2.7 million tons of ore and a supporting 750 ton per day mill, with an overall employment of 225 people. Project costs, exclusive of road building requirements other than in the immediate vicinity, and exclusive of any townsite are as follows:

Mine Development Cost.	$6.00 million
Mill Construction and Equipment Cost	$3.25 million
Total	$9.25 million

One project is visualized as a basic nickel deposit in Zone 7F (see Figure 2 in Preface). The ore is assumed to have a value in the ground of $81 million and to contain 0.83 percent nickel and 0.83 percent copper. The ten-year output will be worth $52.9 million as concentrates. No townsite for mine and mill workers would be required in Zone 7F.

The second project is the development of an ultrabasic nickel deposit in Zone 1G (Figure 2). The ore is assumed to have a value in the ground of $100 million and to contain 1.3 percent nickel and 0.3 percent copper. Value of the concentrates produced is expected to total $66.5 million over the ten-year life of the project. The mine-mill project in Zone 1G will likely require a townsite devement.

The Deposits

Based on geological considerations reported by S.G.S., $3.5 billion worth of nickel-copper ore may be discovered in the James Bay region. The major portion of this potential is concentrated in six zones designated high priority for future exploration. These zones, the potential value of nickel-copper ore in each, and the type of host rock present are given in Table 2-7. Estimates of the minimum value of an exploitable deposit in each zone shown in the table are, of course, very approximate. As the need for infrastructural development increases, and operational and transport expenses rise, the minimum exploitable deposit becomes larger. Zone 7F is near an established town and roughly only 10 to 30 miles of new roads would be required; therefore, the minimum value deposit is relatively low, about $81 million. In order to be exploitable, a deposit in Zone 3F would need both a townsite and up to 130 miles of new road as well as more ore of higher grade to justify the risks involved. Therefore, the minimum value of nickel-copper deposits in Zone 3F would have to be about $126 million. Table 2-1 also shows the hypothetical deposit parameters for the minimum valued exploitable deposit in each zone.

The Orebody in Basic Rocks

Of the three zones of basic rock formation, only Zone 7F offers outstanding potential for initial development. In terms of geological potential, it has more value per unit area than Zones 1F and 3F. More importantly, Zone 7F is located near an established town, Val Paradis. Therefore, no townsite would be needed for a mine-mill complex, and a minimum of road building would be required. The minimum economic project for the basic nickel deposit in 7F is estimated to be of the order of $81 million. This "value in-the-ground" was obtained by assuming a geologically likely deposit size of 2.7 million tons and an average or slightly above average ore grade of $30 per ton.

A basic deposit usually contains nickel and copper in about a one-to-one ratio. At a value of $30 per ton, such ore would contain 0.83 percent nickel and 0.83 percent copper.* The deposit is likely to be geologically similar to two other nickel-copper deposits found in the Belleterre area of Quebec. One is a deposit owned by Consolidated Regcourt Mines and is reported to contain 2.2 million tons of ore grading 0.67 percent nickel and 0.73 percent copper. This deposit has an in-the-ground value of $24.50 per ton or about $54 million total, and has not been mined. The second deposit was mined by the Lorraine Mining Company between 1965 and 1968. Ore grade was reported as 1.60 percent copper and 0.6 percent nickel for a value of approximately $31.60 per ton.

The 7F ore body will contain massive and disseminated sulfides within basic lava. The deposit will likely be tabular in character and extend to a depth of about 1,000 feet.

The Orebody in Ultrabasic Rocks

Zones 1G, 2G, and 3G have good potential for nickel discovery. All three locations (see Figure 2) would require townsite development; however, the 1G location might require little or no road development. Thus, the minimum economic nickel project in ultrabasic rocks will be examined in terms of the 1G location.

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*In-the-ground values were calculated using $1.30 per pound for nickel and $0.50 per pound for copper.

TABLE 2-7. ESTIMATED MINIMUM VALUE EXPLOITABLE DEPOSIT IN SIX ZONES OF HIGH GEOLOGICAL POTENTIAL FOR NICKEL-COPPER ORE

Zone & Type of Host Rock	Geological Potential for Nickel-Copper Ore, (millions of dollars)(1)	Infrastructure Needs		Estimated Minimum Value of Exploitable Deposit (millions of dollars)	Hypothetical Deposit Parameters(3)			
		Townsite	New Road (miles)		Deposit size, (millions of tons)	Value of Ore (dollars/ton)(2)	Nickel Content (percent)	Copper Content (percent)
1F Basic	420	Yes	0 to 50	108	2.7	40	1.11	1.11
3F Basic	280	Yes	30 to 130	126	3.0	42	1.17	1.17
7F Basic	350	No	10 to 30	81	2.7	30	0.83	0.83
1G Ultrabasic	350	Yes	0 to 20	100	2.7	37	1.30	0.30
2G Ultrabasic	280	Yes	45 to 80	108	2.7	40	1.42	0.30
3G Ultrabasic	280	Yes	30 to 100	108	2.7	40	1.42	0.30

Source: Compiled by BCL.

Notes:
(1) S.G.S. Report to Sores-Battelle.
(2) Value "in-the-ground" based on $1.30 per pound for nickel and $0.50 per pound for copper.
(3) An underground deposit at a maximum depth of 1,000 to 1,200 feet is assumed.

The value of the nickel deposit at 1G will have to be greater than the deposit at 7F to justify the needed townsite and higher operational and transportation costs. Thus, the 1G deposit is assumed to contain nickel-copper ore with a value in-the-ground of $37 per ton. Typically, in ultrabasic deposits the nickel content is much higher than the copper content. For the 1G deposit, a nickel content of 1.3 percent and a copper content of 0.3 percent would yield the value per ton needed. A total deposit of 2.7 million tons of ore is geologically probable which would have a value in-the-ground of $100 million.

The deposit is likely to be similar to but not as rich as the No. 1 orebody mined by Marbridge Mines in LaMotte Township in western Quebec. Mineralzation of that orebody consisted of massive sulfides--pyrite, pentalandite and small amounts of pyrrhotite and chalcopyrite. The ore body was about 400 feet long and dipped at a 50 degree angle. The core of massive sulfides was up to 7 feet thick and was surrounded by a mantle of disseminated sulfides. The deposit extended to a depth of about 1,200 feet.

The Mine

Mine Size and Materials Handling Requirements

Both the basic and ultrabasic hypothetical nickel projects have common mine parameters. The underground mine envisioned will consist of a main shaft, which is expected to go to a depth of 1,000 to 1,200 feet. Initially, three different working levels would be developed starting at about the 300 feet.

It is expected that the mine will have an ore-to-waste ratio of about 5 to 1. Based on a ten-year mine life, a nominal daily mining capacity of 1,100 tons of ore is obtained for 5 days per week operation. The following material removal rates are needed:

	Approximate Capacity in Tons		
Material	Daily	Annual	10-Year Mine Life
Ore	1,100	270,000	2.70 million
Waste	220	54,000	0.54 million
Total	1,320	324,000	3.24 million

Mine Equipment Needs

The need for underground mine equipment will vary greatly with the type of mining method employed. The most obvious need is for hoisting capacity capable of handling 1,300 tons of ore and waste per day and for conveying miners and supplies. Major equipment needs are summarized in Table 2-8, not including specialized equipment dictated by a particular mining method.

TABLE 2-8. NICKEL MINE EQUIPMENT LIST[1] (2 shift, 5 day/week)

ITEM
Underground
Skip hoist
Jumbo drill rigs
Loading equipment
Ore cars
Portable air tools
Ventilation system
Pumps (ground water control)
Primary crusher
Above Ground
Conveyor (mine to mill)
Air compressor
Ventilation heating system
Maintenance and storage buildings
Office

Source: Compiled by BCL.

Note: (1) Only a generalized listing is possible until a deposit has been discovered and delineated by surface drilling and underground development.

Mine Manpower List

The mine will operate on a 5-day week, 2 shifts per day basis. To mine the 550 tons per shift required will take approximately 60 underground workers, for the three different levels. In addition, 15 surface and supervisory employees will be needed per shift. Thus, a total of 75 employees will be required per shift for a grand total of 150 employees. Table 2-9 presents these requirements by skill level.

Operating Supplies

Assuming that the mine operates with electric powered vehicles rather than diesel fueled ones, little or no diesel fuel will be needed, and the major supply item in terms of weight would be explosives. It is estimated about 1 pound of explosives will be needed per ton of ore mined. Other needed supplies include drill bits, rock bolts, lumber, hoses, replacement parts for various pieces of equipment, lubricating oil, etc. If these supplies weigh as much as the explosives needed, the weight of daily inbound freight would be 1 ton or 250 tons per year.

Electrical needs for the mine were estimated under the assumption that all major equipment is electrical, i.e., the hoist, air compressor, ventilation fans, crusher, hauling vehicles, and the various pumps. They approximate 10,000 kilowatt hours per day or 2,500,000 kilowatt hours per year.

Mining Effluents

The solid waste generated in an underground mining project is much less than in an open pit mine because there is no overburden removal, and the mine is selectively developed to follow the ore body. Thus, in this hypothetical mine, for an ore removal rate of 1,100 tons per day, only 220 tons of waste per day are expected, or about one-half million tons over the life of the mine. Presumably, this material will end up in a tailings pile.

The underground mining of sulfide ore usually leads to acid mine water drainage. In addition, water runoff from the tailings pile could also be acidic. Thus, mine planning should include methods for minimizing the effects of these drainages, for example, pumping them to the disposal system for mill tailings.

The only airborne pollutant of consequence from the mining operation will be dust. This can arise from drilling and blasting, or at any ore or waste transfer point. Essentially, all of these operations are underground and represent an internal problem which can be solved by proper filtering or by wetting of the dust source. Above ground, mine roads and tailings piles also represent possible dust sources.

TABLE 2-9. NICKEL MINE MANPOWER LIST
(2 shift, 5 days/week)

Designation	Number
Skilled equipment operators	21
Semiskilled equipment operators	70
Skilled maintenance personnel	4
Semiskilled maintenance personnel	14
Unskilled labor	32
Total Labor	141
Management	4
Office and engineering	5
Total Office	9
Total	150

Source: Estimated by BCL.

TABLE 2-10. NICKEL MILL EQUIPMENT NEEDS(1)
(3 shift, 7 day/week)

ITEM
Secondary crusher, gyratory
Vibrating screen
Ball mills
Classifier
Cyclones
Flotation cells
Thickener
Filter
Dryer

Source: Compiled by BCL.

Note: (1) Only a generalized listing is possible until the ore has been characterized and a specific treatment procedure developed.

The Mill

The mill will have a 750 tons-per-day capacity based on a 3 shift, 7 days-per-week operation. Ore treatment begins with the crushed ore received from the mine, the final product being a concentrate containing nickel and copper.

Flowsheet

A flowsheet for a typical 750 tons-per-day nickel-copper mill is shown in Figure 2-2. Ore from the mine is recrushed and screened for reduction to minus 3/4 inch size. Thus, fine ore is then wet ground in ball mills to -250 mesh, the secondary mill being operated in closed circuit with a classifier. The -250 mesh material is further ground in a tertiary ball mill to -325 or -425 mesh (or other size appropriate to liberate the nickel and copper minerals) in closed circuit with cyclones. The fully ground pulp is conditioned with frothers and collectors and circulated through the flotation circuit. A bulk nickel-copper concentrate is floated off the rougher cells while the sink material is cleaned and recleaned for ultimate disposal through scavenger cells to tailings disposal with neutralization. The nickel-copper floats are thickened, filtered, dried, and packaged for shipment.

Mill Equipment Needs

The major equipment needs for a mill which treats approximately 750 tons of nickel-copper ore per day are shown in Table 2-10. Sizing of equipment and a floor plan layout of the mill must await the discovery of a mineable deposit and the development of treatment procedures appropriate for the characteristics of the ore.

Mill Manpower List

The mill will be operating 24 hours per day, 7 days per week, therefore, four crews of workers will be needed for jobs which are manned continuously. Employee needs are estimated in Table 2-11.

Mill Operating Supplies

The chemical supplies, grinding steel requirements, and utilities for milling nickel-copper ores can vary with the physical and chemical properties of the ore and the beneficiation method used. Table 2-12 presents the range of supply needs that are typical of several nickel mills operating in Canada. From this table, the probable needs for this hypothetical mill were estimated. The results are given in Table 2-13. It shows that only about 550 tons of chemicals and supplies are needed annually. Of this amount, about half will be steel balls for ore grinding.

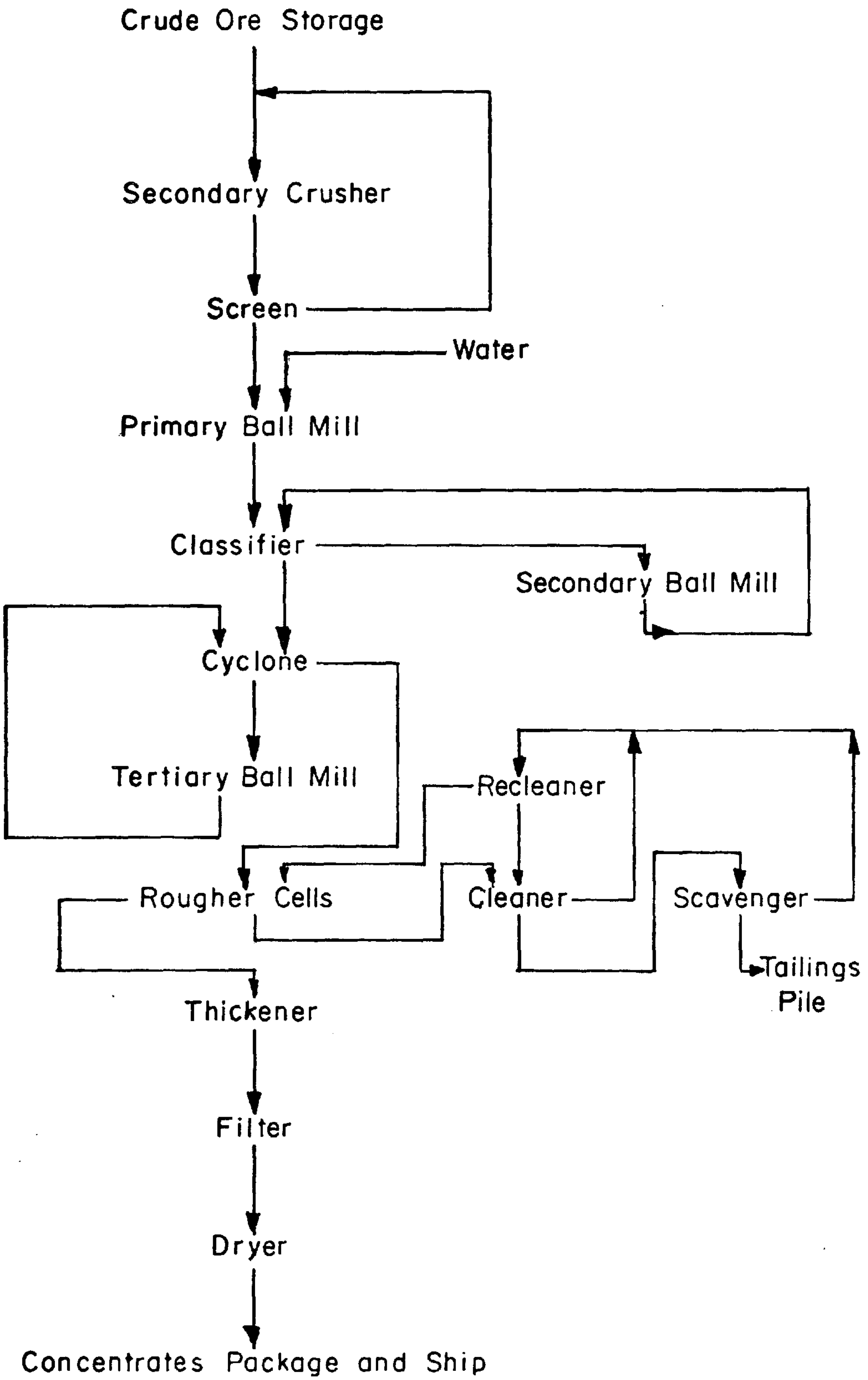

FIGURE 2-2. TYPICAL NICKEL-COPPER MILL FLOWSHEET

TABLE 2-11. NICKEL MILL MANPOWER LIST
(3 shifts, 7 day/week)

Designation	Number
Skilled equipment operators	12
Semiskilled equipment operators	32
Skilled maintenance personnel	7
Semiskilled maintenance personnel	8
Unskilled labor	8
Total labor	67
Management	4
Office and laboratory	4
Total office	8
Total	75

Source: Estimated by BCL.

TABLE 2-12. CONSUMPTION RANGE OF OPERATING SUPPLIES FOR NICKEL MILLS(1)

Chemicals	Pounds/Ton Milled
Sodium Cyanide	0 - 0.02
Copper Sulfate	0 - 0.27
Soda Ash	0 - 1.05
Sodium Hydroxide	0 - 0.03
Sodium Silicate	0 - 0.77
Sulfuric Acid	0 - 0.91
Lime	0.04 - 1.5
Frothers (such as flotanol, Dowfroth 250)	0.04 - 0.23
Collectors (such as Xanthate, PAX, etc.)	0.14 - 0.29
Grinding Steel	
Rods & Balls	1.7 - 2.4
Utilities	
Fresh Water, gal/ton	233 - 513
Power, Kilowatt-hrs/ton	24 - 32

Source: Compiled by BCL from various trade publications.

Note: (1) Based on operations of several nickel mills in Canada.

TABLE 2-13. NICKEL MILL SUPPLIES AND UTILITIES REQUIREMENTS

Chemicals	Daily Pounds/Ton Milled	Daily Pounds	Annual Tons
Copper Sulfate	0.20	150	27
Soda Ash	0.10	75	14
Lime	1.00	750	135
Frothers (Dowforth 250)	0.20	150	27
Collectors (Potassium amyl xanthate)	0.20	150	27
Other	0.35	265	48
Grinding Steel			
Balls	2.0	1,500	270
TOTAL	4.05	3,040	550
Utilities			
Fresh Water	250 gal/ton	187,500 gal	67.5 $\times 10^6$ gal
Power	28 KWH/ton	21,000 KWH	7.6 $\times 10^6$ KWH

Source: Estimated by BCL.

Mill Effluents

The major effluent from the hypothetical mill will be the tailings, which will approximate 675 tons per day. These tailings will consist of ore waste and spent chemicals in slurry form. Usual practice is to keep the tailings in ponds or impoundments after neutralization. In some instances, the tailings are treated and used for mine back-fill.

Liquids used in the milling process are either recycled or discharged with the tailings. Liquid effluents of concern are those which may occur by seepage from ore piles and tailings impoundments.

The major air effluent likely is dust. Dust comes from sources such as roads, ore piles, or tailings. These dust sources can be minimized by wetting when dry conditions occur.

Mill Output and Value

The output of concentrates from either the 7F and 1G project will be as follows:

Daily	Yearly	Life of Mine
75 tons	27,000 tons	270,000 tons

The recovery of nickel and copper in both projects is estimated to be 90 percent and 80 percent, respectively, of the assumed content of the ore.

The concentrates from the 7F project are expected to contain 7.5 percent nickel and 6.6 percent copper with a value of $196 per ton.* Value of the output will, therefore, be as follows:

Daily	Yearly	Life of Mine
$14,700	$5.29 million	$52.9 million

The concentrates from the 1G project are expected to contain 11.7 percent nickel and 2.4 percent copper and have a value of $246 per ton.* Value of the output will be as follows:

Daily	Yearly	Life of Mine
$18,460	$6.65 million	$66.5 million

---

*The value of the concentrate is estimated to be three-quarters of the metal value, assuming an average nickel price of $1.30 per pound and an average copper price of $0.50 per pound.

Capital Requirements

Mine Costs

Mine development costs vary widely. One reference cites typical development costs of \$1.5 million for an underground mine at a depth of 600 feet and capable of handling 1,100 tons of ore per day.* Underground nickel mines now being developed in Canada by Inco at Sudbury and Shebandowan are believed to cost about \$8.4 million and \$5.9 million respectively, when capacities are normalized to 1,100 tons per day.** On the basis of these numbers, a development cost of \$6.0 million is assigned to both the basic deposit in Zone 7F and the ultrabasic deposit in Zone 1G. This estimate does not include townsite costs at 1G, nor costs for roads leading to either mine-mill complex.

Mill Cost

Recent capital cost figures for nickel mills with a capacity near 750 tons per day are unavailable. Extrapolation of capital costs from available data on much larger mills indicates a cost of \$3 million exclusive of services. Because both of the hypothetical locations under consideration will require little or no road construction, road costs are not included in the above. Other needed services such as water, electricity, natural gas, etc., exclusive of possible townsite cost will add perhaps another \$0.25 million to capital costs. Thus, a capital expenditure of \$3.25 million is estimated for the mill.

Project Total

The capital requirements for either project are summarized as follows:

Mine	\$6.00 million
Mill (including services)	3.25 million
Total	\$9.25 million

Evaluation of Potential Worth

As in the case of the uranium mine/mill complex, preliminary estimates of project feasibility for the nickel-copper mine/mill project are presented as guidelines. Again, it is noted that such estimates are merely illustrative of the types of returns which can result if a deposit is found similar to the characteristics and assumptions described previously. Location 7F (at Val Paradis) has been selected as the prototype. A 2-year construction period beginning in 1982 is anticipated with product sales commencing in 1984.

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* "U.S. Uranium, Economics & Technology", P.F. Schutt, Jr., Nuclear Assurance Corporation, 1969.

** Engineering and Mining Journal, January, 1970.

Data Inputs and Assumptions

Capital Outlays. There are two capital outlays associated with the project.

(1) Mine development costs are estimated at $6.0 million

(2) Mill construction and equipment costs are estimated at $3.25 million.

Fifty percent of total capital requirements are borrowed. Building and equipment life is 10 years, with no salvage value.

Operating Expenses. Five annual operating expenses are considered.

(1) Costs associated with mining 1,000 tons per day (270,000 tons per year) are selected from the following distribution:

Annual Cost (thousands of $)	Median per ton Cost ($/ton of ore mined)	Probability* (percent)
1485 - 1619	5.75	5
1620 - 1754	6.25	15
1755 - 1889	6.75	30
1890 - 2024	7.25	35
2025 - 2160	7.75	15

(2) Operating costs associated with milling 750 tons per day (270,000 tons per year) are sampled from the following distribution:

Annual Cost (thousands of $)	Median per ton Cost ($/ton of ore mined)	Probability* (percent)
783 - 810	2.95	10
811 - 836	3.05	35
837 - 863	3.15	40
864 - 890	3.25	10
891 - 918	3.35	5

(3) Annual transport costs are comprised of two elements

(a) Costs associated with hauling concentrates to railhead (La Sarre) from the mill site are estimated at $162,000 (27,000 tons of concentrates for 60 miles at 10 cents per ton mile)

(b) Rail costs from La Sarre to a smelter (Sudbury was chosen as the most likely destination) are estimated at $5.00 per ton of concentrate, or $135,000 per year.

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* See footnote, page 1-49.

Total annual transport charges are $297,000 per year.

(4) Depreciation is calculated on a straight line basis for the life of the project and is calculated internally within the cash flow program

(5) Interest charges are also calculated within the program on the basis of a $4,625,000 loan over a 12-year period at 8 percent annual interest. Payback on loan principal begins in the first year of mill operation (the third year of the 12-year project).

Revenues. Revenues are generated from the sale of concentrates and are calculated according to the following example.

One ton of concentrates (7.5 percent Ni and 6.6 percent Cu) contains 150 lbs of nickel and 132 lbs of copper combined. The value of the concentrate is 75 percent of the total metal content value. For a nickel price of $1.30/lb and a copper price of $0.50/lb, a ton of concentrate would be equal to

For Ni - 150 lbs at .75 x $1.30 = $146.25/ton
For Cu - 132 lbs at .75 x $0.50 = $ 49.50/ton

Combined value = $195.75/ton,
or $196.00/ton.

For an annual output of 27,000 tons of concentrates, annual revenues for the above would be $5,292,000.

Revenues are sampled from the distribution

Annual Revenues (thousands of $)	Median Sales Price ($ per ton)	Probability* (percent)
5157 - 5291	193.50	5
5292 - 5292	196.00	75
5293 - 5426	198.50	10
5427 - 5562	203.50	10

Output

Based on the above combinations of inputs, 50 iterations of the model were run. Results indicate that potential rates of return vary between 8 percent and 15 percent. A cumulative probability distribution (Figure 2-3) of the above results indicates that there is a 64 percent chance of earning 12 percent or better on the proposed project. Sample outputs are included in Tables 2-14 to 2-16 which demonstrate the sensitivity of certain cost and revenue values.

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* See footnote, page 1-54

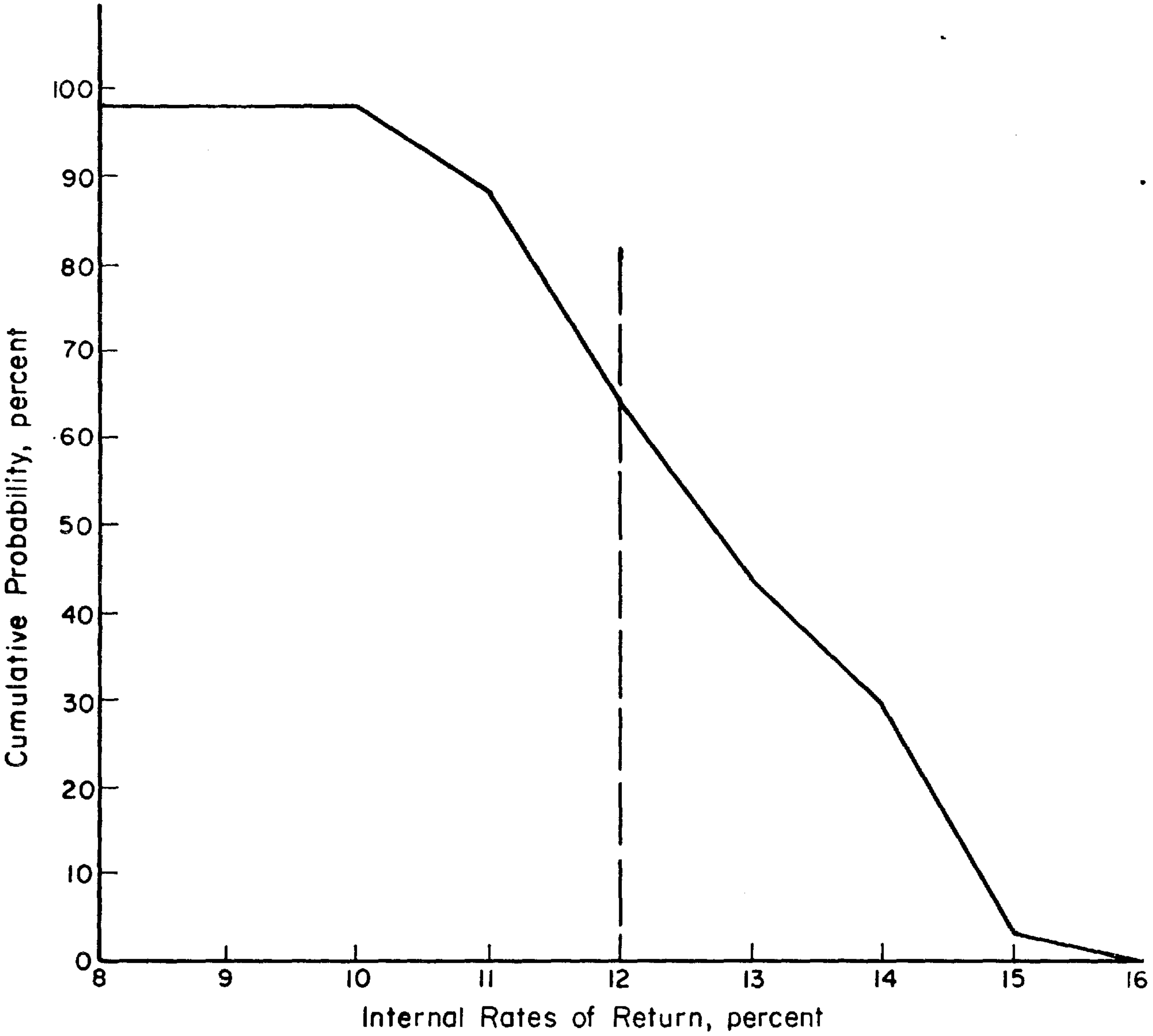

FIGURE 2-3. NICKEL PROJECT INTERNAL RATES OF RETURN VERSUS PROBABILITY

TABLE 2-14. NICKEL / COPPER MINE-MILL COMPLEX AT VAL PARADIS (7F)
ITERATION NUMBER 8

YEAR	1	2	3	4	5	6	7	8	9	10	11	12
CAPITAL OUTLAYS												
MINE DEVELOPMENT												
-----INTERNAL FUNDS	3000.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	0.0	207.1	223.7	241.5	260.9	281.7	304.3	328.6	354.9	383.3	414.0
MILL CONSTR./EQUIP.												
-----INTERNAL FUNDS	1625.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	0.0	112.2	121.1	130.8	141.3	152.6	164.8	178.0	192.2	207.6	224.2
TOTALS	4625.0	0.0	319.3	344.8	372.4	402.2	434.4	469.1	506.6	547.2	590.9	638.2
ANNUAL EXPENSES												
MINING COSTS	0.0	0.0	1548.4	1548.4	1548.4	1548.4	1548.4	1548.4	1548.4	1548.4	1548.4	1548.4
MILLING COSTS	0.0	0.0	820.1	820.1	820.1	820.1	820.1	820.1	820.1	820.1	820.1	820.1
TRANSPORT COSTS	0.0	0.0	297.0	297.0	297.0	297.0	297.0	297.0	297.0	297.0	297.0	297.0
INTEREST ON LOANS	370.0	370.0	370.0	344.5	316.9	287.1	254.9	220.2	182.6	142.1	98.3	51.1
TOTAL DEPRECIATION	770.8	770.8	770.8	770.8	770.8	770.8	770.8	770.8	770.8	770.8	770.8	770.8
TOTALS	1140.8	1140.8	3806.4	3780.9	3753.3	3723.5	3691.3	3656.6	3619.1	3578.5	3534.8	3487.5
ANNUAL REVENUES												
SALE OF NI/CU CONC.	0.0	0.0	5292.0	5292.0	5292.0	5292.0	5292.0	5292.0	5292.0	5292.0	5292.0	5292.0
TOTALS	0.0	0.0	5292.0	5292.0	5292.0	5292.0	5292.0	5292.0	5292.0	5292.0	5292.0	5292.0
TAXABLE INCOME	0.0	0.0	0.0	715.0	1538.7	1568.5	1600.7	1635.4	1672.9	1713.5	1757.2	1804.5
INCOME TAX	0.0	0.0	0.0	391.5	842.4	858.8	876.4	895.4	915.9	938.1	962.1	988.0
NET INCOME	0.0	0.0	0.0	323.6	696.3	709.7	724.3	740.0	757.0	775.3	795.2	816.5
CASH FLOW	-4995.0	-370.0	1937.2	1545.7	1094.7	1078.4	1060.8	1041.8	1021.2	999.0	975.1	949.2

TERMINAL WORTH = .000

YIELD(IN PERCENT) = 14.917
PRESENT WORTH AT 0 PERCENT = 6337.945
PRESENT WORTH AT 5 PERCENT = 3284.493
PRESENT WORTH AT 10 PERCENT = 1292.423

TABLE 2-15. NICKEL / COPPER MINE-MILL COMPLEX AT VAL PARADIS (7F)
ITERATION NUMBER 26

YEAR	1	2	3	4	5	6	7	8	9	10	11	12
CAPITAL OUTLAYS												
MINE DEVELOPMENT												
-----INTERNAL FUNDS	3000.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	0.0	207.1	223.7	241.5	260.9	281.7	304.3	328.6	354.9	383.3	414.0
MILL CONSTR./EQUIP.												
-----INTERNAL FUNDS	1625.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	0.0	112.2	121.1	130.8	141.3	152.6	164.8	178.0	192.2	207.6	224.2
TOTALS	4625.0	0.0	319.3	344.8	372.4	402.2	434.4	469.1	506.6	547.2	590.9	638.2
ANNUAL EXPENSES												
MINING COSTS	0.0	0.0	2094.6	2094.6	2094.6	2094.6	2094.6	2094.6	2094.6	2094.6	2094.6	2094.6
MILLING COSTS	0.0	0.0	824.8	824.8	824.8	824.8	824.8	824.8	824.8	824.8	824.8	824.8
TRANSPORT COSTS	0.0	0.0	297.0	297.0	297.0	297.0	297.0	297.0	297.0	297.0	297.0	297.0
INTEREST ON LOANS	370.0	370.0	370.0	344.5	316.9	287.1	254.9	220.2	182.6	142.1	98.3	51.1
TOTAL DEPRECIATION	770.8	770.8	770.8	770.8	770.8	770.8	770.8	770.8	770.8	770.8	770.8	770.8
TOTALS	1140.8	1140.8	4357.2	4331.7	4304.1	4274.3	4242.1	4207.4	4169.9	4129.3	4085.6	4038.3
ANNUAL REVENUES												
SALE OF NI/CU CONC.	0.0	0.0	5292.0	5292.0	5292.0	5292.0	5292.0	5292.0	5292.0	5292.0	5292.0	5292.0
TOTALS	0.0	0.0	5292.0	5292.0	5292.0	5292.0	5292.0	5292.0	5292.0	5292.0	5292.0	5292.0
TAXABLE INCOME	0.0	0.0	0.0	0.0	601.3	1017.7	1049.9	1084.6	1122.1	1162.7	1206.4	1253.7
INCOME TAX	0.0	0.0	0.0	0.0	329.2	557.2	574.8	593.8	614.4	636.6	660.5	686.4
NET INCOME	0.0	0.0	0.0	0.0	272.1	460.5	475.1	490.8	507.8	526.1	545.9	567.3
CASH FLOW	-4995.0	-370.0	1386.3	1386.3	1057.1	829.2	811.5	792.5	772.0	749.8	725.8	699.9

TERMINAL WORTH = .000

YIELD(IN PERCENT) = 10.130
PRESENT WORTH AT 0 PERCENT = 3845.527
PRESENT WORTH AT 5 PERCENT = 1532.651
PRESENT WORTH AT 10 PERCENT = 31.165

TABLE 2-16. NICKEL / COPPER MINE-MILL COMPLEX AT VAL PARADIS (7F)
ITERATION NUMBER 31

YEAR	1	2	3	4	5	6	7	8	9	10	11	12
CAPITAL OUTLAYS												
MINE DEVELOPMENT												
-----INTERNAL FUNDS	3000.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	0.0	207.1	223.7	241.5	260.9	281.7	304.3	328.6	354.9	383.3	414.0
MILL CONSTR./EQUIP.												
-----INTERNAL FUNDS	1625.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	0.0	112.2	121.1	130.8	141.3	152.6	164.8	178.0	192.2	207.6	224.2
TOTALS	4625.0	0.0	319.3	344.8	372.4	402.2	434.4	469.1	506.6	547.2	590.9	638.2
ANNUAL EXPENSES												
MINING COSTS	0.0	0.0	1865.1	1865.1	1865.1	1865.1	1865.1	1865.1	1865.1	1865.1	1865.1	1865.1
MILLING COSTS	0.0	0.0	834.3	834.3	834.3	834.3	834.3	834.3	834.3	834.3	834.3	834.3
TRANSPORT COSTS	0.0	0.0	297.0	297.0	297.0	297.0	297.0	297.0	297.0	297.0	297.0	297.0
INTEREST ON LOANS	370.0	370.0	370.0	344.5	316.9	287.1	254.9	220.2	182.6	142.1	98.3	51.1
TOTAL DEPRECIATION	770.8	770.8	770.8	770.8	770.8	770.8	770.8	770.8	770.8	770.8	770.8	770.8
TOTALS	1140.8	1140.8	4137.3	4111.7	4084.1	4054.3	4022.2	3987.4	3949.9	3909.4	3865.6	3818.3
ANNUAL REVENUES												
SALE OF NI/CU CONC.	0.0	0.0	5220.6	5220.6	5220.6	5220.6	5220.6	5220.6	5220.6	5220.6	5220.6	5220.6
TOTALS	0.0	0.0	5220.6	5220.6	5220.6	5220.6	5220.6	5220.6	5220.6	5220.6	5220.6	5220.6
TAXABLE INCOME	0.0	0.0	0.0	0.0	1047.0	1166.2	1198.4	1233.2	1270.7	1311.2	1355.0	1402.3
INCOME TAX	0.0	0.0	0.0	0.0	573.2	638.5	656.1	675.2	695.7	717.9	741.9	767.7
NET INCOME	0.0	0.0	0.0	0.0	473.8	527.7	542.3	558.0	575.0	593.3	613.1	634.5
CASH FLOW	-4995.0	-370.0	1534.9	1534.9	961.7	896.4	878.8	859.7	839.2	817.0	793.0	767.2

TERMINAL WORTH = .000

YIELD(IN PERCENT) = 11.541
PRESENT WORTH AT 0 PERCENT = 4517.758
PRESENT WORTH AT 5 PERCENT = 2009.357
PRESENT WORTH AT 10 PERCENT = 378.075

It is expected that returns for the 1G ultrabasic orebody would be roughly similar (though slightly higher) in that, while $13.5 million in additional revenues are generated over the life of the project, townsite development and higher operating costs would more or less negate most of the additional revenues. In both of the above projects, potential rates of return could be higher if ore grades were higher. At the present time, however, the probability for finding higher grade ores remains virtually unknown.

CHAPTER 3. COPPER/ZINC

CHAPTER 3. COPPER/ZINC

COPPER SUPPLY/DEMAND SITUATION TO THE YEAR 2000

Copper has been used continuously by man since the Stone Age. The metal is widely distributed and occurs in the earth's crust at a level of about 60 parts per million. Despite its wide distribution there are relatively few large copper producing areas in the world. The important copper producing areas are in the Western United States, Canada, Peru, Chile, Zambia, and Zaire. The USSR has substantial deposits in the Ural mountains and the Kazakstan region.

Occurrence and Beneficiation

Occurrence

Mineralogically, copper ores can be divided into several categories--sulphide, native, secondary oxidized, and complex. The sulphide ores are the most important. For example, chalcopyrite (copper-iron sulphide) accounts for about half of the world copper output. The secondary copper minerals, such as chalcocite and covellite, are formed by the action of ground water on primary copper sulphides. The oxidized ores occur above the zone of secondary enrichment and contain oxides, carbonates and silicates. The complex ores contain lead, zinc, molybdenum, gold and silver minerals in addition to copper.

Copper ores occur in many types of deposits in various host rocks. A large proportion of the world's copper production now comes from low-grade but extensive porphyry deposits. Such deposits contain most of the estimated world copper reserves. Generally, porphyry copper deposits are large, low-grade deposits containing chalcopyrite, pyrite, and traces of gold, silver, and molybdenite. These deposits are amenable to open pit or block caving mining methods. With a few exceptions, porphyry copper deposits seldom run over 0.80 percent copper, and many run in the range of 0.40 percent.

The Precambrain deposits in central Canada occur as complex sulfides in volcanic or sedimentary rocks with ultrabasic intrusive associations. Chalcopyrite, sphalerite, pyrite, and pyrrbotite are the major sulfides although traces of silver and gold are found in certain areas and represent significant secondary values in vein-type replacement deposits of chalco-pyrite. Copper or zinc may prodominate as the primary base metal value with both being present in the environment described by S.G.S. as "volcanogene". Copper concentration in the ore ranges from less than 0.2 percent to over 3.0 percent as copper while zinc ranges from about 0.5 percent to as high as 11.0 percent as zinc. In the "filonien" (vein-type) copper occurrences, zinc seldom appears and the copper content of 1.0 to 3.5 percent as copper is supplemented by up to 2 ounces of silver and 0.2 ounces of gold per ton of ore.

Beneficiation

At most porphyry copper concentrating plants, the procedure usually includes crushing, grinding, classification, flotation and filtration. Copper sulphide minerals are recovered by the flotation process where the froth contains the copper sulphide minerals and the underflow a tailing product. The copper content of the ore entering the flotation process may be one percent or less. The final copper concentrate may contain from 11 to 38 percent copper. Leaching techniques are becoming increasingly important in the prouction of primary copper, mainly because of the continued decline in the average grade of copper ore mined and the frequent availability of oxide ores that are readily segregated in mining. In all leaching methods, the copper-containing solids are leached with a dilute solution of sulphuric acid which may also contain ferric sulphate. Leach solutions contain copper sulphate, sulphuric acid, iron sulfate, and other salts. Copper is precipitated from solution by contact and cementation with scrap iron. The solution can be reused directly for further dump leaching.

For the underground copper/zinc ores in Canada the usual treatment involves crushing, grinding, classifying, flotation of a copper concentrate, reconditioning and flotation of a zinc concentrate from the tailings which contain the pyrite and pyrrlotite. Since these deposits seldom contain oxidized copper mineralizations leaching is not an important treatment procedure. Copper concentrates usually range from 15 to 35 percent copper on a dry basis.

Product Form

Copper concentrates are smelted in reverbratory furnaces and rotary converters to blister copper--an impure form that contains the precious metals from the ore. Electrolytic refining removes the undesirable impurities and yields various grades of pure copper. For selected ores, including native coppers, fire refining produces acceptable grades of pure copper without the need to undergo the electrolytic process.

Copper is sold as concentrates, as copper matte or blister copper, as well as refined copper or in copper alloys.

Uses and Substitutes for Copper

Uses for Copper

Electrolytic refined and fire refined copper are sold in standard grades--tough pitch, oxygen free, and deoxidized as well as a number of shapes--wire bars, cathodes, ingots, cakes, slabs and billets. The shapes are largely determined by the requirements of the fabricator's equipment, the grades by the end use intended.

Copper, having high conductivity, is used primarily in the electrical industry in the form of cables and wire. The electrical industry uses more than half the world output of copper.

The balance of world output is used to take advantage of properties such as heat conductivity, corrosion resistance, mechanical strength, easy formability and fabrication, lack of magnetic properties, and pleasing appearance in copper and copper-base alloys. The principal families of alloys are called brasses, tin bronzes, aluminum bronzes, cupro-nickels, nickel silvers, and chromium coppers. Copper and its alloys are fabricated into components for end-use industries by casting, rolling, extruding, drawing, machining, welding, brazing, and soldering of semi-fabricated shapes such as ingots, billets, rods, bars, cakes, and slabs.

Although end-use distribution patterns vary from country to country, the 1968 pattern for copper and copper alloys in the United States is presented below as an illustration of the major categories recognized in world statistical publications:

End-Use Category	United States Consumption 1968 (percentage)
Electrical equipment and supplies	48.9
Construction	15.8
Industrial machinery, except electrical	10.0
Transportation	11.9
Ordnance	6.1
Other, including jewelry, chemical, pigments, and coinage	7.3
Total	100.0

Substitutes for Copper

For many applications, the properties of copper make it nearly irreplaceable, while in other areas aluminum, steel, glass, plastics, and other materials are competing with copper. For example, aluminum is now used for virtually all high-voltage, overhead, transmission lines but copper is still dominant for underground transmission. Steel has somewhat replaced brass for shell cases, and printed circuits have replaced some copper for electrical circuits.

Both aluminum and stainless steels have reduced the use of copper in many areas of building construction. Additionally, plastic tubing has been substituted for copper piping in some applications in the building, automotive, and appliance industries.

World Sources of Copper

Copper is truly an international commodity, much more so than other nonferrous metals. The only major producing area that uses the bulk of its mine output is the United States which is also the largest producing and consuming nation. The United States accounts for about one-third of world production and consumption.

Europe is a major importing area, obtaining over 90 percent of its copper requirements primarily from Africa and Chile. The United States imports copper mainly from Western Hemisphere nations, including Canada, Peru, and Chile, with about 10 percent of imports coming from South Africa. Imports from Chile have dropped appreciably since the copper industry in that country was nationalized.

The third largest producer, Chile, consumes about 5 percent of its copper output domestically. In the past, exports went roughly equally to the United States and to Europe. This ratio has changed during the past 2 years--more going to Europe, less to the United States, and other countries purchasing some Chilean copper.

The major African producers are Zambia and Zaire, ranking second and sixth in world copper production. African producers export over 95 percent of their copper output, with nearly all of it going to Europe. Canada, the fifth ranked producer, exports about half its total supply.

Current Production

International trade in copper involves concentrates for smelting, matte for converting, blister for refining, scrap and skimmings for recovery, and refined copper and copper alloys in semifabricated shapes, finished components, and assembled end-use products. Accordingly, it is important to detail production in terms of mine, smelter, and refinery outputs in order to understand copper marketing.

Mine Production. The largest copper producing countries in the world are the United States, the U.S.S.R., and the four C.I.P.E.C. countries (Chile, Peru, Zambia, and Zaire). In 1970, these six countries produced 70 percent of the world's copper output.

World copper mine production for 1960, 1965, and 1970 is given in Table 3-1. Total mine production increased by 2.3 million tons over that period from 4.67 million short tons in 1960 to 6.96 million short tons in 1970. This represents an annual compound growth rate of 4.0 percent over the period.

Smelter Production. Copper smelting is an intermediate stage in the recovery of copper from ore and subsequent processing to metal. The largest copper producing countries also tend to have the major share of smelter production. The United States as the principal copper smelter, treated about 24 percent of the world's production in 1970, the U.S.S.R. smelted about 15 percent, and Zambia about 11 percent. Tabel 3-2 shows world copper smelter production for 1960, 1965, and 1970. Total production increased from 4.73 million short tons in 1960 to 6.92 million tons in 1970, an annual growth rate of just under 4 percent.

TABLE 3-1. WORLD MINE PRODUCTION OF COPPER
(thousand short tons of copper content)

Country	1960	1965	1970
Canada	439.3	507.9	676.1
United States	1080.2	1351.8	1706.0
Total	1519.5	1859.7	2382.1
Latin America	879.9	936.5	1085.4
Europe	143.2	164.9	231.3
Africa	1083.1	1236.6	1414.0
Asia	235.8	272.4	388.1
Australia	122.5	101.2	160.5
Total	3984.0	4571.3	5661.4
Soviet Sphere	689.0	1011.2	1300.2
Total World	4673.0	5582.5	6961.6

Source: World Metal Statistics

TABLE 3-2. WORLD SMELTER PRODUCTION OF COPPER
(thousand short tons of copper content)

Country	1960	1965	1970
Canada	397.9	424.4	497.3
United States	1233.7	1434.0	1641.2
Total	1631.6	1858.4	2138.5
Latin America	795.1	844.8	977.3
Europe	233.1	291.1	365.2
Africa	1038.6	1217.9	1403.5
Asia	249.6	330.6	596.3
Australia	79.6	82.2	132.7
Total	4027.6	4625.0	5613.5
Soviet Sphere	698.8	1019.1	1308.5
Total World	4726.4	5644.1	6922.0

Source: Metallgesellschaft Metal Statistics.

Refinery Production. Table 3-3 presents production of refined copper on a worldwide basis for 1960, 1965, and 1970. The total increased from 5.5 million short tons in 1960 to 8.3 million short tons in 1970, a growth rate slightly above 5 percent per year.

The United States is the largest copper refiner, accounting for 27 percent of the output, the U.S.S.R. contributed 14 percent, and Europe produced 17 percent of the total. Japan achieved almost a threefold increase in refined copper production during this period.

Further, it should be noted that production of refined copper in the Free World from recycled scrap increased from 16.0 percent in 1960 to 19.0 percent in 1970.

Table 3-4 present world trade in unrefined copper by region for the period 1966-1970. The major markets for copper ores are Europe and Japan. The United States, Europe, and Japan import large quantities of blister, while Europe is the major market for African copper. World trade in refined copper is shown in Table 3-5. The dominant exporters in the American region are Chile and Canada, while Zambia is the leading exporter of refined copper in Africa. In 1970, America exported over 40 percent of world refined copper and Africa about 34 percent.

Reserves

The world's copper reserves can be grouped broadly into two categories (1) large porphyry orebodies and non-porphyry type deposits. It has been estimated that perhaps 70 percent of mineable copper ore reserves are contained in two porphyry provinces, one around the Pacific Rim and the other bordering the Russian Steppes. Other important copper deposits are located in eastern Canada, South Africa, Australia, and in the central African copperbelt.

In 1970, the U.S. Bureau of Mines updated previous statments of world reserves of copper and estimated a total of 307.9 million short tons. Although not explicitly stated, this quantity is believed to be recoverable as long as the price for copper remains about $0.50 per pound and does include proven and inferred reserves. Area-wise, the largest segment of these resources are on the west slopes of the Andes in South America, followed very closely by the United States. The central African copper belt and the U.S.S.R. are other major resource areas followed by Poland and Canada. Table 3-6 presents the revised compilation, of which the Free World accounts for 260 million tons.

Current and Future Capacity

In looking at the future, it can be assumed that the geographic distribution of copper sources will not change significantly from the present pattern. Of course, exploration will continue to turn up new deposits in new areas--for example, Ertsberg, Bougainville, or Cerro Colorado (Panama)--but it is unlikely that such finds will require a major realignment of copper trade channels. More

TABLE 3-3. WORLD REFINERY PRODUCTION OF COPPER
(thousand short tons)

Country	1960	1965	1970
Canada	417.1	434.1	543.1
United States	1810.8	2157.1	2242.1
Total	2227.9	2591.2	2785.2
Latin America	316.6	417.6	611.1
Europe	1073.4	1275.5	1439.4
Africa	617.7	778.0	958.8
Asia	300.2	425.4	813.2
Australia	92.2	102.2	157.9
Total	4628.0	5589.9	6765.6
of which, secondary	741.2	970.6	1285.0
of which, primary	3886.4	4619.3	5480.6
Soviet Sphere	881.4	1224.0	1539.6
Total World	5509.5	6813.9	8305.2

Source: World Metal Statistics.

TABLE 3-4. WORLD UNREFINED COPPER TRADE BY REGION
(thousand short tons, copper content)

Unrefined Copper	1966	1967	1968	1969	1970
Exports					
Ores and Concentrates					
Africa	6.3	2.5	23.5	37.1	9.5
America	154.8	234.5	291.0	236.0	328.6
Asia	110.2	121.3	155.2	182.0	210.9
Australia	11.5	13.4	11.5	11.8	30.0
Europe	9.4	6.3	5.5	10.0	9.3
TOTAL	292.2	378.0	486.7	476.9	588.3
Blister					
Africa	434.3	442.7	416.2	442.2	453.4
America	395.1	429.2	433.5	379.6	372.1
Asia	22.0	16.9	16.7	7.2	5.0
Australia	7.3	7.7	7.8	9.4	7.7
Europe	26.2	24.6	24.9	20.4	22.6
TOTAL	884.9	921.1	899.1	858.8	860.8
Imports					
Ores and Concentrates					
America	43.0	32.9	27.6	39.0	32.7
Asia	171.6	210.2	287.5	298.4	431.1
Europe	99.8	109.9	143.1	144.8	130.7
TOTAL	314.4	353.0	458.2	482.2	594.5
Blister					
America	357.8	269.4	270.7	238.0	224.3
Asia	69.6	127.7	140.3	165.0	150.5
Europe	493.0	523.7	495.0	432.1	461.4
TOTAL	920.4	920.8	906.0	835.1	836.2

Source: World Metal Statistics

TABLE 3-5. WORLD REFINED COPPER TRADE BY REGION
(thousand short tons)

Refined Copper	1966	1967	1968	1969	1970
Exports					
Africa	738.6	771.6	823.6	909.1	866.8
America	855.8	868.1	980.6	934.5	1035.4
Asia	1.7	1.6	13.1	16.6	52.0
Australia	8.4	10.5	18.5	36.1	34.6
Europe	614.7	678.2	678.3	576.0	583.1
TOTAL	2219.2	2330.0	2514.1	2472.3	2571.9
Imports					
Africa	43.1	36.1	6.9	3.6	2.6
America	222.1	376.0	460.9	204.2	205.0
Asia	115.6	122.0	207.6	257.6	212.2
Europe	1667.8	1524.9	1686.6	1924.8	2010.3
TOTAL	2048.6	2059.0	2362.0	2390.2	2430.1

Source: World Metal Statistics

TABLE 3-6. WORLD RESERVES OF COPPER

Country	Copper Content (millions of short tons)
Canada	10.0
Chile	59.3
Congo (Kinshasa)	20.0
Peru	24.6
U.S.S.R.	38.5
United States	85.5
Zambia	30.0
Other	40.0
Total	307.9

Source: Bulletin 650, U.S. Bureau of Mines.

and more copper smelting may be done in the countries where concentrates are produced because of increasing environmental pressure in the highly industrialized countries and the desire of the less developed countries to industrialize. However, it is anticipated that refining will continue to be done in those countries that are the leading refiners now.

Mine Capacity. The estimated mine capacity for producing copper is shown in Table 3-7 for the years 1970, 1971, and 1972. North America, Africa, and South America are the leading copper mining areas, and based on the estimated reserves shown previously it can be inferred that these areas will continue to be the leaders in copper mining for the coming three decades.

It is extremely difficult to estimate what mining capacity will be in the future. Based on present knowledge it is expected that mine capacity will have to be increased by 1980 from its present level. This means commitments must be made soon as it takes several years to plan, open up, and put into operation a copper mine. For 1990 and 2000 it is assumed that mining capacity will be expanded in order to satisfy the market demand for copper.

Smelter Capacity. Table 3-7 also presents the copper smelting capacity of the Free World for 1970, 1971, and 1972. Presently North America, South America, and Africa are the leading smelting areas.

This situation may change in the future to one where more copper may be smelted in the countries producing copper concentrates. This could mean increased smelting in the developing countries and less expansion of smelter capacity in the highly industrialized nations. There are two major reasons for this possible change. First, the developing nations want to upgrade their raw materials at home rather than exporting ores or concentrates in order to provide more employment and increase the value of the product to be exported. The second reasons involves the desire of the industrial nations to clean up pollution and improve environmental quality. This means that all smelters would have to be fully equipped to minimize water and air pollutants--a costly investment. Thus, it might be less expensive to increase smelting capacity in the developing countries where anti-pollution measures are not as stringent as in established nations.

BCL beleives that smelter capacity will have to be increased before 1980 to satisfy demand and that capacity for 1990 and 2000 can be expanded sufficiently to meet demands.

Refining Capacity. Free World copper refining capacity for 1970, 1971, and 1972 is shown in Table 3-8. It appears that capacity was increased by about 7 percent over that time span. It is anticipated that another 10 percent might be added to capacity for 1980.

In the longer term, 1990-2000, it can be expencted that refinery capacity will be expanded to meet market demands for copper. It is further anticipated that this additional capacity will be located where refinery capacity now exists, primarily in the United States, Western Europe, and Asia.

TABLE 3-7. FREE WORLD COPPER MINE AND SMELTER CAPACITIES
(thousands of short tons of copper content)

Continent	1970	1971	1972	1970	1971	1976
North America						
Canada	683	772	804	526	526	571
United States	1,832	1,886	1,926	1,758	1,814	1,837
Mexico	82	82	96	101	101	101
TOTAL	2,597	2,740	2,826	2,385	2,441	2,509
South America						
Chile	833	1,018	1,221	893	893	893
Peru	242	249	249	232	232	232
TOTAL	1,075	1,267	1,470	1,125	1,125	1,125
Europe	261	302	325	709	725	780
Africa						
Zambia	777	729	898	806	834	862
Zaire	430	441	496	436	436	496
Other	246	269	310	250	250	250
TOTAL	1,453	1,439	1,704	1,492	1,520	1,608
Asia	442	521	650	930	930	1,018
Australasia						
Australia	157	157	167	157	157	157
Bougainville	--	--	196	--	--	--
TOTAL	157	157	363	157	157	157
GRAND TOTAL	5,985	6,426	7,338	6,798	6,898	7,197

Source: Assembled from various publications by BCL.

TABLE 3-8. FREE WORLD COPPER REFINING CAPACITY
(thousands of short tons)

Continent	Annual Capacity		
	1970	1971	1972
North America			
Canada	540	540	594
United States			
Electrolytic) Lake and Fire Refining)	2,676	2,800	3,024
TOTAL	3,216	3,340	3,618
Latin America	1,007	1,007	1,007
Europe	1,586	1,586	1,620
Africa	1,268	1,268	1,313
Asia	964	964	1,090
Australia	246	246	246
TOTAL	8,287	8,411	8,894

Source: Assembled from various publications by BCL.

Free World Demand for Copper

Current Consumption

About 80 percent of the world's refined copper consumption is concentrated in the developed countries of the Free World. Table 3-9 shows the consumption for the years 1966-1970. Canada, the United States, France, West Germany, Italy, the United Kingdom, and Japan are the leading consuming nations. The table also shows that Free World consumption in 1970 was slightly more than 500,000 tons higher than it was in 1966. Consumption decreased about 300,000 tons in the United States over the period, but increases in Western Europe and Japan provided an overall increase in consumption in 1970 as compared to 1966.

Historical Copper Demand. From about 1900 to the end of World War I, world copper consumption increased at an average annual rate of 5 to 6 percent. The growth rate subsequently decreased to less than 2 percent per year during the interwar period, increased markedly during World War II and dropped sharpely in 1946. Since 1947, world copper consumption has shwon an average annual growth rate of about 4 percent. Even though there have been several years when demand dropped appreciably, the long-term trend has always been up.

Future Demand

Most copper is consumed as refined metal. The principal user--wire mills--accounts for about 60 percent of consumption. These mills produce bare wire, insulated wire, and insulated communication wire. Brass mills consume some 35 percent of total copper in the production of sheet, rod, wire, and tubing. The remainder is used in foundries, chemical plants, and miscellaneous other uses.

The following discussion of markets for copper relates primarily to the United States, but is considered reasonably descriptive of the rest of the Free World.

Electrical Equipment and Supplies. This category is the largest consumer of copper. The future for electrical apparatus and electrical transmission appears bright, as do the communications and appliance sectors. Increased emphasis on safety, comfort, recreation, and a pollution-free environment should result in greater demand for copper in electrical applications. The trend to underground power distribution systems is likely to boost the use of copper over competitive materials for technological reasons, especially in Western Europe.

There are factors that could inhibit the demand for copper. These include the substitution of aluminum for copper, copper clad aluminum, new power generation systems not requiring generators, cryogenic techniques in power transmission, and microminiaturization of communication circuitry. The relative price advantages of substitute materials and technological developments that reduce unit material requirements will tend to restrict growth rates for copper in the future.

TABLE 3-9. FREE WORLD CONSUMPTION OF REFINED COPPER (thousands of short tons)

Continent	1966	1967	1968	1969	1970
North America					
Canada	273	230	256	259	252
United States	2,360	1,936	1,880	2,142	2,031
Mexico	33	38	62	72	73
TOTAL	2,666	2,204	2,198	2,473	2,356
South America	127	89	111	125	124
Western Europe					
Belgium	119	108	132	124	124
France	321	299	323	369	365
West Germany	506	553	671	723	764
Italy	215	245	249	262	300
Spain	72	77	84	106	87
Sweden	92	95	94	97	90
United Kingdom	653	567	594	603	604
Other	253	253	261	282	313
TOTAL	2,231	2,197	2,408	2,566	2,647
Asia					
Japan	532	679	766	890	917
Other	54	70	63	76	67
TOTAL	586	749	829	966	984
Africa	46	44	44	52	54
Australia	121	100	114	110	123
GRAND TOTAL	5,777	5,383	5,704	6,292	6,288

Source: World Bureau of Metal Statistics.

Construction. A plus factor in the use of copper for construction is the image often desired by an affluent society in its housing. The technical advances in copper cladding many materials should enhance this decorative use. Another factor could be the need for superior performance in materials to combat future high maintenance costs.

Curtailing future use of copper in construction is the trend to multiple housing units that reduces material needs per unit, and the use of alternate materials to replace higher priced copper.

Industrial Machinery-Except Electrical. Any industrial society, and developing nations, will use products from this category. Therefore, as overall economic conditions continue at a high level, the use of copper in industrial machinery should increase.

The principal deterrent to use of copper for industrial machinery is its high cost relative to alternate choice materials.

Transportation. Growth of copper consumption in transportation is dependent on increasing production of automobiles and trucks, installation of rapid transit systems, and continued growth in aircraft passenger miles per year.

On the other hand, environmental regulations that would restrict use of the automobile, plus replacement of copper by aluminum and plastics could reduce copper demand.

Other uses. Other uses such as jewelry, chemical, pigments, etc., are usually related to population and economic conditions. Overall this is a small part of the total copper demand and will not have too much influence in the future.

Demand by Geographic Area. Battelle's estimated demand for refined copper in the Free World for the years 1980, 1990, 2000 is shown in Table 3-10. This table shows that demand is estimated to increase about 11 million tons, from 6.3 million to 16.7 million tons during that time span. This would be an increase of nearly threefold at an average annual rate of 3.3 percent.

Battelle used the following annual growth rates to arrive at the demand shown in Table 3-10.

Canada--3.5 percent to 1980, 2.5 percent to 1990, 2000
United States--2.5 percent to 1980, 1990, 2000
Western Europe--3.7 percent to 1980, 3.4 percent to 1990, 3.0 percent to 2000
Japan--5 percent to 1980 and 4 percent to 1990, 2000
Rest of World--4 percent to 1980, 1990, 2000

TABLE 3-10. PROJECTED FREE WORLD DEMAND FOR REFINED COPPER
(thousands of short tons)

Geographic Area	1970	1980	1990	2000
Canada	252	356	456	584
United States	2,031	2,600	3,328	4,260
Western Europe	2,647	3,807	5,318	7,147
Japan	917	1,494	2,211	3,273
Rest of Free World	441	653	967	1,431
TOTAL	6,288	8,910	12,280	16,695

Source: Estimated by BCL.

Supply/Demand Relationships

Previous tables and discussions have indicated that Free World demand for refined copper is expected to increase from 6.3 million tons in 1970 to 8.9 million tons in 1980, to 12.3 million tons in 1990, and 16.7 million tons in 2000. If these estimates are reasonably correct, it is obvious that mining, smelting and refining capacity will have to be increased appreciably in order to satisfy the demands.

It should be recognized that not all of this projected demand for refined copper will have to be supplied from primary sources. The reuse of scrap in refineries accounted for 19 percent of refined copper output in 1970. By 1980 this could be at 20 percent and quite stable at that level. Based on this assumption, Free World demand for primary refined copper is projected to be 7.1 million tons by 1980, about 9.85 million tons by 1990, and 13.4 million tons in 2000.

Mine production of copper should exceed refinery output of primary metal by at least a factor of 1.1 to compensate for inevitable product losses during smelting and refining. At the same time mine capacity greater than actual production is desirable to provide a cushion for inadvertent outages. Assuming a 90 percent utilization of mining facilities, mine capacity should thus be about 1.2 times as large as primary refined metal production. This indicates that from a mine capacity level of 7.3 million tons in 1972 (see Table 3-7, page 3-13), an addition of 1.2 million tons will be needed by 1980. Similarly, the 1990 refinery output of 9.85 million tons of copper could support a mine capacity of nearly 12 million tons, or an increase of 3.5 million tons during the decade. Finally, by 2000, mine capacity will need to grow again by 8.0 million tons to 20 million tons to sustain an annual production rate of 13.4 million tons of primary refined copper.

Figure 3-1 presents BCL's projections of Free World demand for copper ores on an annual and cumulative basis and for comparison, the estimates of reserves that will be available at prices between $0.50 and $0.65 per pound. By 1997, the presently inferred reserves will be exhausted unless exploration in the interim has added to this stockpile. While the current pace of exploration activity has subsided because of major discoveries in the 1968 to 1971 period that are not yet in production, it well could be reactivated on a major scale, world-wide during the 1975-1980 period in order to prove up the inferred portion of estimated reserves, as well as to make new discoveries.

Probable Price Trends

The price of copper is basically determined by the interaction between supply and demand. Many factors influence this interaction and minor factors such as the possibility of a strike or a political upheaval can have a dramatic short term influence on copper price. For example, the sharp increase in price in 1970 was partially attributed to anticipated loss of production due to labor problems in the Chilean mines. Natural causes such as flooding of the Mufulera mine in Africa in 1968, and the drop in production from the White Fine mine in Michigan, apparently due to a fault zone in the underground workings, caused price rises because of lowered supply.

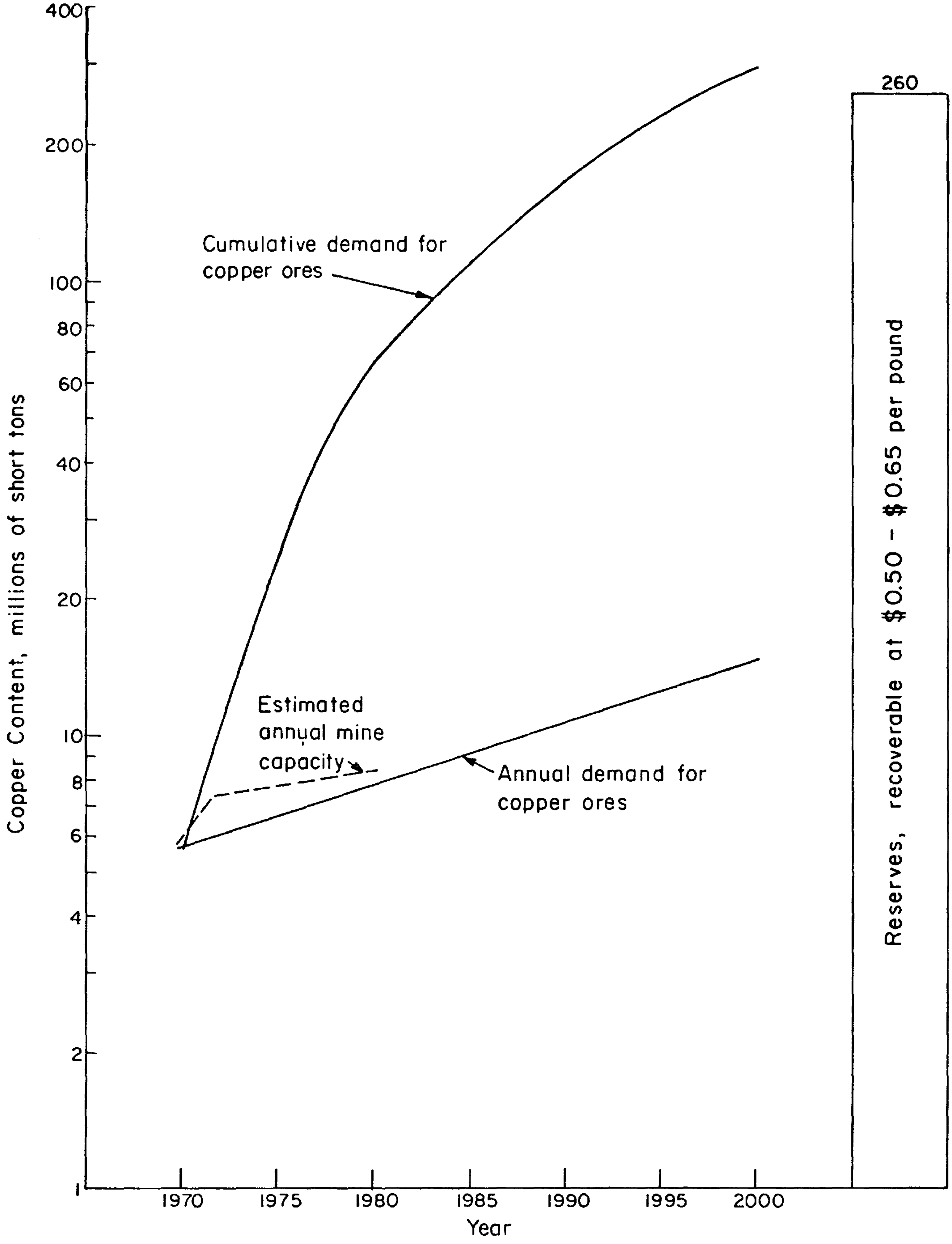

FIGURE 3-1. ANNUAL AND CUMULATIVE FREE WORLD DEMAND FOR COPPER ORES AND RESERVES

Technical problems such as failure of important smelting or refining equipment at any of the large producers, or transportation problems such as experienced by the African producers in recent years can be expected to keep production below the rated capacity.

Any one of these factors or happenstances can have a short term effect on copper price. However, the long term trend in copper price will be influenced more by surplus or inadequate production capacity, increasing production costs, substitutional trends, and inflation.

Overall, a surplus or deficiency in capacity can be considered a short-term situation. If there is a surplus, production will be geared to demand and some facilities shut down. When there is a deficiency in capacity, more will be built.

Increasing production costs appear to be a fact of life in the mining and smelting business. However, technological advances and productivity increases have, and will continue to exert a downward pressure on production costs.

The subject of substitution is complex. Basically, there is a continuing process of change in both price relationships between copper and various competitive materials. In times of short copper supply and higher prices, substitutions are sought and, once a market is lost to an alternate material, it is difficult for copper to recapture that market. The threat of substitution acts as a strong deterrent to higher copper prices, and producers are well aware of the need to minimize the inroads of other materials, especially aluminum.

The recent price rise in the United States from about 53 cents per pound to 60 cents per pound is typical of a short term situation in copper marketing. First, there is an increased demand in both the United States and abroad. Second, there have been port strikes and strikes at the mines in Chile. Third, Zambia is having trouble moving its copper to the ocean for exporting to foreign markets. Fourth, Phase II controls have been lifted in the United States. All of these situations serve to exert upward pressure on copper prices.

In summary, it is anticipated that quoted copper prices will rise over the coming decade, but in terms of constant dollars, will remain reasonably level. Given the normal swings in supply and demand, a price of $0.75 per pound (1972 $'s) should not be exceeded up to about 1985. Predicting prices beyond that time would be a useless exercise because of all the intangibles that are inherent in the situation, but impending exhaustion of currently identified reserves is likely to exert strong upward price pressures.

Need for a New Producer in the James Bay Region

The above discussion indicates that there will be need for expanded copper mine and smelter capacity before 1980, and a larger need before 1990 and 2000. Certainly, some of the present operators of mines and smelters will expand their facilities when they become convinced that the market demand will continue to grow.

Yet, some currently operating properties will be exhausting reserves and closing down. Thus, there should be an opportunity for a new producer to compete for a share of the market. The requirements would be a good grade of ore with sufficient reserves to ensure a long life, operating costs that are competitive with others, sufficient capitalization to weather startup problems, good management, and a strong marketing group.

What is the potential for James Bay? It is assumed that the first operations in the area would be mining and concentrating, and the selling of concentrates. This, of course, is dependent upon finding and developing an ore body. In spite of the world-wide stagnation of exploration, it should be, and is being, continued in the James Bay region to have reserves to tap as the Matagami and Chibougamau districts play out: witness Selco's recent find in the Lake Frotet area. Similar activities should be encouraged in the other zones identified by S.G.S. as priority targets, especially Zone 2A,to provide, hopefully, additional employment opportunities for the residents of Val Paradis. Promotion of exploration activity by the James Bay Development Corporation should be initiated as soon as it can be arranged with any competent exploration company. When an apparent commercial discovery is made, companies having competence in mining and milling will start to express interest and negotiations can then be conducted for an early exploration schedule if feasibility studies turn out favorably.

Markets for the concentrates may be found in eastern Canada, the United States, Western Europe, or Japan, depending on conditions at the time of appropriate exploitation. If several medium to large deposits are located within a few years, it could be possible to consider the establishment of at least a smelter, if not a smelter and refinery, in the James Bay region. An assured minimum of about 50,000 to 60,000 tons per year of recoverable copper would justify a serious study of feasibility for a smelter.

ZINC SUPPLY/DEMAND SITUATION TO THE YEAR 2000

Occurrence and Beneficiation

Zinc was used as an alloy long before it became known as a metal. The oldest known object containing zinc has been found in prehistoric ruins and was an idol in which zinc was the principal metal. In the pre-christian era, the Romans manufactured brass by treating copper with zinc carbonate. It is doubtful if they knew the true significance of what they were doing, but they did recognize the advantages derived. The smelting of zinc was first perfected in China and crude zinc was an import into the western world in the 17th and 18th centuries. The technology for producing zinc metal was imported into England about 1830.

Occurrence

Both zinc and copper are relatively abundant in the earth's crust, accounting for 75 parts per million out of the overall composition. Zinc is less expensive than copper even though it is a difficult metal to smelt because of the relatively common occurrence of large concentrations of zinc containing up to one thousand times the natural abundance.

The principal zinc ore is the sulfide, sphalerite, frequently called "blende". Deposits containing sphalerite are known but usually this zinc sulfide is found in conjunction with other sulfides such as lead, iron, and copper sulfides. Both silver and cadmium are found frequently in association with zinc while germanium may show up as a trace element.

Commercial deposits of zinc have been found worldwide. In Canada's Atlantic Provinces, lead and zinc occur together as sulfides, while throughout the Canadian shield, zinc appears to be more readily associated with copper. In the far west, for example British Columbia, lead and zinc again occur together. East of the Mississippi River, in the United States, zinc is the principal metallic value in commercial deposits found in New York and Tennessee. West of the Mississippi River, lead and zinc again occur, usually in association with pyrite. Throughout the balance of the western hemisphere--Mexico, Peru, Bolivia, and Argentina in particular--zinc and lead sulfides constitute the principal metallic values of the minerals found in commercial deposits.

Beneficiation

The treatment accorded sulfide ores containing zinc depends on the composition and the relative value of the metals that are to be recovered from that particular ore.

The generalized procedure for treating sulfide ores consists of crushing and grinding the ore to a size such that the mineral constituents are liberated, which is followed by selective flotation of the mineral values present and disposal of the waste gangue. In those ores that contain lead as well as zine, the lead is the first mineral floated, followed by zinc and if present, copper is the last mineral recovered with or without pyrite.

For ores containing zinc and copper as the principal metallics of value, the usual crushing and grinding is followed by selective flotation of copper followed by the selective flotation of the zinc. The usually associated pyrite appears in the tails from the zinc flotation. Copper and zinc concentrates are forwarded separately to smelters for subsequent treatment.

Product Form

Historically, the zinc sulfide concentrates were roasted to zinc oxide which was then reduced in the presence of carbon in either horizontal or vertical retort kilns. The metallic zinc collected at the base of the furnace and was drawn off into molds to form pigs. The usual product from such operations was known as "prime western" grade zinc, and contained a small amount of lead. It was used principally as a galvanizing material or, if necessary, refined to higher grades of zinc.

An alternative procedure involves roasting the zinc sulfide concentrates to zinc oxide, with recovery of the off gases containing sulphur to make sulfuric acid. This sulfuric is used in turn to dissolve the zinc oxide from the roaster to form zinc sulphate from which zinc metal is then recovered in an electrolgtic cell. The product resulting from this type of treatment is a pure zinc identified as "Special High Grade" (SHG) that is required for the manufacture of die-casting alloys.

Traditionally, zinc prices have been quoted on the basis of prime western grade with SHS commanding a one-cent-per-pound premium. Currentlv, all production of zinc in Canada is by the electrolytic process that yields SHG material directly.

Uses and Substitutes for Zinc

Uses for Zinc

Zinc has many uses that result from its physical and chemical characteristics. One of its largest uses is as a protective coating for iron and steel. Applied directly to the steel in a thin coating, zinc forms a weather resistant film that resists rusting of the base metal. The zinc may be applied electrolytically or by dipping the part in molten zinc. Both processes are called galvanizing. In addition to direct protection, the zinc affords protection for base metal exposed by scratches or cuts because of the sacrificial nature of zinc relative to iron.

A second major use for zinc is as the base for die casting alloys. These are relatively low melting alloys with excellent molding properties and sufficient strength to serve as decorative and functional parts for automobiles, appliances, business machines, tools, building hardware, toys, and novelties.

A third major use for zinc is as an alloying constituent in brass with copper. Zinc improves the strength and workability characteristics of brass and may be present in amounts ranging from 5 to 45 percent of the total composition weight. On the average, brass contains about 30 percent zinc. However, this is not a major market for primary zinc in that large quantities of brass are reclaimed annually from both zinc base and copper base scrap. Minor percentages of slab zinc (primary zinc) are consumed annually in preparing primary brass alloys.

Zinc also is used to make photo-engraving plates, galvanic protection plates and rods, cases for battery dry cells, shields for radio condensors and tubes, weather stripping, roofing sheet, and for the manufacture of zinc oxide pigments and zinc chemicals.

The consumption pattern of zinc varies markedly from country to country, but on a worldwide basis, galvanizing accounts for 35 to 40 percent of total world zinc consumption. Brass and die casting alloys take about 20 percent each, and rolled zinc sheet accounts for roughly 10 percent. The balance gets into zinc oxide, zinc chemicals, and other small miscellaneous uses.

Substitutes for Zinc

For galvanizing iron and steel products, there is no adequate subsitute for zinc. Ceramic and plastic coatings, cadmium, and aluminum have been substituted at times, but usually with loss of properties and frequently at considerably higher cost. In certain forms, sheet aluminum may substitute or at least replace galvanized steel sheet, but again, the cost comparision distinctly favors the galvanized steel product.

Brass has been replaced to some extent by alloys of aluminum or stainless steel. The aluminum alloys are favored where casting techniques are the preferred fabrication method and light weight is a desirable property. Stainless steel is favored whenever the corrosion resistance requirements becomes severe, the strength to cross-sectional area ratio is high, and the silvery appearance is desirable. In recent years, these substitutions have tended to limit the growth of zinc consumption in brass.

With respect to zinc die casting applications, alternate materials include die cast aluminum alloys, stainless steel in thin sections, and the so called "engineering plastics". Size, shape and finish of the part all play a major role in determining which material will be used, but the controlling factor in many instances is not economics or availability, but the personal preference of the design engineer who conceived the part and its function in the overall product.

Among the zinc pigments (lithapone and zinc oxide) titanium dioxide has virtually replaced lithapone, but complements rather than displaces zinc oxide. In rubber compounding zinc oxide has never found a suitable replacement and a similar statement could be made for many of the zinc chemicals such as zinc chloride, zinc sulfate, and zinc chromate.

The displacement of zinc by alternate materials is a constant threat that, in part, is being met by an international association of producers and users which seeks to develop better zinc-based products, less expensive methods of application, and better understanding of zinc's advantages and limitations on the part of designers and engineers charged with materials selection.

World Sources of Zinc

Zinc is an international commodity in spite of its relatively low price per pound in comparison to nickel or copper. Zinc concentrates and refined zinc metal both are important items in international trade. However, unlike copper, zinc scrap is an insignificant item of commercial between countries.

Measured by imports, roughly one-third of world mine output of zinc is shipped to another country for recovery of the metallic content. Western Europe, the United States, and Japan account for substantially all of these imports. The principal exporting countries in order are: Canada, Peru, Australia, Ireland, and Mexico. It might be noted that Ireland is a quite recent addition to this list since mining on a substantial scale started in this country only in 1968.

About 18 percent of world production of slab zinc enters international trade and the principal importers are the United States, the United Kingdom, and West Germany. The principal countries furnishing exports include in order: Canada, Australia, Belgium-Luxembourg, the U.S.S.R., and Poland. It should be noted further that the Soviet sphere both imports and exports slab zinc; probably in differing grades and the net export balance accounts for about 10 percent of total world trade in this commodity.

Current Production

Mine Production. Currently, the leading miners of zinc are: Canada, the U.S.S.R., the United States, Australia, Peru, Japan, Mexico, and Poland, all of whom produce over 200,000 tons per year of zinc contained in concentrates. In 1970, these eight countries produced more than 71 percent of total world mine output. Table 3-11 presents world mine production of zinc by country or region for 1960, 1965, and 1970. Total mine production increased from 3,692,000 tons of zinc content in 1960 to 6,079,000 in 1970, for an average annual rate of growth of 5.1 percent. Canada had the largest gain, vaulting from third place in 1960 to a commanding lead in 1970.

Smelter Production. The ultimate disposition of the zinc content of mine concentrates is difficult to follow. The bulk of the concentrates go to smelters where various grades of zinc metal are produced for use or sale. In this context, the smelter may refer to either pyro-metallurgical processes (horizontal or vertical retort kilns) or an electrolytic refinery. Quite minor quantities of concentrates or ore are used in the direct production of zinc oxide and certain zinc chemicals and, thus, would not be reported in smelter production of zinc. International statistical publication agencies (such as the United Nations International Lead and Zinc Study Group or World Bureau of Metal Statistics)

TABLE 3-11. WORLD MINE PRODUCTION OF ZINC
(thousands of short tons of zinc content)

Country/Region	1960	1965	1970
Canada	406.7	821.8	1380.9
United States	435.3	610.9	586.8
Latin America	544.3	597.2	785.6
Western Europe	613.3	615.4	794.7
Africa	294.4	328.1	284.9
Asia	209.7	288.9	439.9
Oceania	355.4	391.0	536.9
Soviet sphere	833.1	1013.6	1269.7
World Total	3692.2	4666.9	6079.4

Source: World Metal Statistics.

ignore the diversion of zinc ore or concentrates into products other than slab zinc and a similar procedure is followed in this report. The principal countries producing slab zinc include: the United States, Japan, the U.S.S.R., Canada, West Germany, Australia, Belgium, France, and Poland. Each of these nine countries produced more than 200,000 tons of slab zinc in 1970 and collectively, they accounted for 69 percent of total world production. The smaller countries of Western Europe contributed an additional 12 percent of world production while the balance was widely distributed. Table 3-12 presents world production of slab zinc by country or region for 1960, 1965, and 1970. Over the decade of the sixties production increased from 3,472,000 tons of zinc to 5,584,000 tons, an average annual compound growth rate of 4.8 percent.

On a worldwide basis secondary zinc has very little impact on the production of slab zinc from primary sources such as ore or concentrates. This results from the fact that for the most part the uses of zinc tend to be consumptive in nature, aside from die casts parts rolled zinc, or brass alloys. In the United States the recovery of zinc from old scrap such as die castings, battery cases, and even discarded brass items amounts to less then 5 percent of total supply. In other countries it will be even less because of the absence of a significant quantity of die casting parts being discarded because of wear or obsolescence. New scrap from brass mills provides most of the recycled zinc on a worldwide basis as well as in the United States. By virtue of the importance of this use in countries other then the United States it is highly probable that total zinc metal consumption should be increased about 10 percent above the level of primary zinc consumption in order to account for the zinc that is continuously recycled in new brass scrap. In this report it is assumed that slab zinc is derived from primary sources such as mined ores and concentrates throughout the period under consideration. It is possible that this situation could be changing after 1990. But any attempt to analyze what may happen in the final decade of this century undoubtedly will be covered up by any margin of error inherent in projecting demand and supply over a thirty year period.

Reserves

The frequent occurrence of zinc with lead or copper has no direct bearing on the calculation of reserves but does complicate any attempt to match reserves with future production in assessing supply/demand relationships. Deposits of sulfides containing predominantly zinc as the economic metal value are known but, almost invariably, they have been discovered in the course of searching for other metals such as copper or iron. The most recent statement of world reserves by the U.S. Bureau of Mines includes information that was available to 1969 thus counting the large formation uncovered in the late 1960's by New Jersey Zinc Company in northern Tennessee. Table 3-13 presents world reserves by country or region and shows an estimate of 124 million tons of zinc in deposits scattered over 51 individual countries. Because of the co-production of zinc with either lead or copper in large segments of these reserves, it is impossible to indicate an economic price at which zinc alone could be produced from these deposits. However, it can be anticipated that improvement in the techniques of exploration that have occurred in the past decade or so will enable companies to discover more deposits containing only zinc as the economic metal value. This would be illustrated, of course, by the most recent find in northern Tennessee in which a magnetic anomaly attributed to sulfide minerals was explored primarily in the hope of discovering zinc.

TABLE 3-12. WORLD PRODUCTION OF SLAB ZINC(1)
(thousands of short tons)

Country/Region	1960	1965	1970
Canada	260.8	358.4	460.5
United States	867.4	1077.8	954.7
Latin America	111.6	162.8	211.5
Western Europe	1017.8	1107.7	1517.5
Africa	92.2	115.0	158.8
Asia	205.6	413.9	773.6
Oceania	131.6	219.0	283.8
Soviet Sphere	785.2	1008.6	1223.3
World Total	3472.2	4463.2	5583.7

Source: World Metal Statistics

Note: (1) Total production of slab zinc by smelters and refineries.

TABLE 3-13. WORLD ZINC RESERVES

Country/Region	Zinc Content (millions of tons)
Canada	25
United States	34
Latin America	12
Western Europe	14
Africa	6
Asia	10
Oceania	9
Soviet Sphere	14
World Total	124

Source: U.S. Bureau of Mines

Current and Future Capacity

In assessing the record of mine production of zinc in the past decade, it is obvious that the pattern of regional distribution has changed because of the varying rates of growth encountered. In particular, Canadian production represents outstanding growth based on a number of new large mines that have been commercialized with financial help from outside the country. At the opposite end of this scale would be Africa which actually registered a very slight decline in production from 1960 to 1970, although the mid-point 1965 record did represent a slight increase.

It is doubtful that Canada's record of 1960 decade will be repeated in the 1970 decade or in the 30 year period to the end of this century. In fact, it is much more likely that 1980, 1990 and the year 2000 will in general show much the same pattern of distribution geographically that 1970 does. The foregoing, of course, refers to mine production but there is little evidence to indicate that the pattern of smelter production will change significnatly either. The United States, Western Europe, and Japan presently account for the bulk of zinc smelting and they are likely to retain this leadership in the years to come. The U.S.S.R. constitutes a growing challenge for inclusion as an industrialized nation, but this will not seriously affect the role of the already developed countries. On the whole, no big disruption of the current pattern of either mining or smelting of zinc is anticipated for the Free World.

Again, because of the complex nature of zinc production a clear statement of mine capacity for zinc is difficult. An analysis by BCL of published announcements regarding new zinc mines and expansions at the present mines suggests that 1970 production could be exceeded by 15 percent by 1975 but beyond this any projection at this time is highly questionnable. If the rate of growth outside of Canada for the 1960 decade is accepted as a reasonable rate for mine production in the 1975-1980 period, this would suggest a mine production in 1980 of 8.3 million tons of contained zinc.

According to the Department of Energy, Mines, and Resources, world capacity to produce zinc metal totalled 5.4 million tons in 1970 exclusive of the Soviet sphere countries. Balancing announced closings with new plants and expansions, the same source projects that smelter capacity will rise only to 5.5 million tons by 1975. Between 1969 and 1972 roughly 400,000 tons of zinc metal production capacity was closed in the United States for a variety of reasons, including environmental considerations as well as the economics of importing zinc concentrates. Only about 100,000 tons of this is expected to be replaced in the near future, leaving the United States with an effective zinc smelting or refining capacity of only 900,000 short tons. Also during 1971 two plants were closed in the United Kingdom with a combined capacity of 100,000 tons and no anticipation that they will be replaced in the near future. Increases in zinc smelter capacity have been announced for the 1973 to 1976 period in South Africa, Italy, Mexico, Japan, Netherlands, Spain, Peru, Belgium, India, Algeria, Turkey, Yugoslavia, Canada, and Australia. A more rapid increase in the second half of the 1970 decade apparently will be needed if the projected demand is to be met.

World Demand for Zinc

Current Consumption

Over 86 percent of Free World slab zinc consumption is concentrated in the developed countries. Table 3-14 contains reported data on zinc consumption by country or region for the world from 1966 through 1970. Consumption increased from nearly 4.7 million tons to close to 5.4 million tons, an average annual rate of growth of about 3.5 percent. This is a lower rate than the ten year rate for the 1960 decade of 4.7 percent and results from the minor recession of 1967 and the abrupt slow down starting in late 1970. In the United States, 1970 consumption suffers in comparison with 1966 in virtually every end-use category but, unquestionably, die casting suffered the most. A somewhat similar situation existed in Canada also. Taking a longer range view, all the countries or regions shown in Table 3-14 registered increases in consumption between 1960 and 1970 ranging from the 1.9 percent average annual growth rate in Australia to an 11.6 percent average annual growth rate in Asia, primarily from Japan. Canada bettered the world average at 6.6 percent while the United States fell behind to the extent of a 3.1 percent annual rate.

Future Demand

The variable pattern of zinc usage, even in the developed countries, makes analysis of end uses for projection purposes difficult. Accordingly, major consuming areas will be discussed in order to attempt to predict annual rates of growth for the last three decades of this century.

Canada. Canadian consumption has been heavily oriented toward galvanizing applications. This results in part from the fact that many of the products that could have been die cast in zinc or fabricated from brass in Canada have actually been produced in the United States and imported into Canada for the assembly of automobiles, appliances, and similar articles. Without question, the decade did witness some reversal of this trend as intermediate products manufacturing facilities began to be built up in Canada to support the assembly operations already in place. This type of activity should continue at a rather high rate at least through the first half of the 1970 decade and perhpas then tend to level off in a pattern somewhat similar to that of the United States. Under these assumptions the average annual rate of growth between 1970 and 1980 should approximate 6 percent, dropping off then so that the 1980 to 1990 period averages roughly 4 percent, and finally, the 1990 to 2000 period averages 3.0 percent.

United States. Bolstered by vigorous support for die casting applications on the part of zinc producers and recovery in the general tone of economic development, it is unlikely that consumption of zinc in the United States will decline in the 1970's as it did in the latter half of the 1960 decade. However, it may be difficult to achieve average annual rates of growth comparable to the 1960's because of the tendency toward saturation of the domestic markets and some loss of export markets to Canada for intermediate products based on zinc or zinc

TABLE 3-14. WORLD SLAB ZINC CONSUMPTION
(thousands of short tons)

Country/Region	1966	1967	1968	1969	1970
Canada	107.7	107.8	115.9	118.7	105.7
United States	1402.4	1230.8	1333.3	1362.3	1183.9
Latin America	136.5	128.3	145.0	164.5	172.9
Western Europe	1409.5	1376.9	1526.7	1718.3	1674.4
Africa	48.5	62.8	59.8	60.9	73.7
Asia	556.2	696.2	782.1	848.4	896.9
Oceania	100.4	119.2	120.3	127.9	124.2
Soviet Sphere	921.6	971.5	1027.9	1055.5	1130.7
World Total	4682.8	4693.5	5111.0	5456.5	5362.4

Source: World Metal Statistics

containing materials. Overall, 1970 to 1980 should show an average annual growth rate of 2.5 percent which will then drop off to 1.5 percent for the 1980 to 2000 period.

Western Europe. Galvanizing and brass are the two principal markets for zinc in Western Europe. Die casting applications, although running a poor third, are growing as the automotive and appliance industries in Europe approach the sophistication of design that has characterized American products. However, traditionally Europe has exported a considerable percentage of its manufactured goods including items that are galvanized and also fabricated from brass. On balance it is likely that Western Europe will maintain the 1960 to 1970 pace of 3.0 percent average annual growth rate through 1980 and then tend to taper off to a 2.0 percent average annual growth rate to the year 2000.

Asia. The phenomenal growth of industrial production in Japan in the post World War II era continued right through 1969 before showing evidences of plateauing in 1970 concurrent with the world wide period of difficulty. Undoubtedly, Japan will serve as the industrial manufacturing complex for a good share of Asia for many years to come but India, Indonesia, Taiwan, and Korea also are building and should retard Japan's growth somewhat. The Free World Asian countries have been assigned a 6.0 percent average annual growth rate to 1980 before falling to a 4 percent rate for the balance of the period.

Soviet Sphere. Without question, consumer goods are likely to get a significantly larger share of attention in the Soviet sphere of influence in the closing decades of this century. However, the present record suggests that an average annual growth rate higher then 4.5 percent is unlikely unless a complete turn around is experienced.

World Total. Overall, as is shown in Table 3-15, world slab zinc demand is projected to increase from 5.4 million tons in 1970 to 7.9 million tons in 1980, to 10.8 million tons in 1990, and to 14.8 million tons in the year 2000.

Supply/Demand Relationships

The previous sections relating to zinc have discussed supply and demand in relation to total world situations and needs. For the assessment of specific supply/demand relationships in the future it appears more appropriate to concentrate the balance of this discussion on the Free World rather then on the total world picture.

In brief retrospect, Free World demand for slab zinc is expected to rise from about 4.2 million tons in 1970, to 6.2 million tons in 1980, to 8.1 million tons in 1990, and to 10.6 million tons in the year 2000. Because of losses inherent in the smelting and refining of zinc, the metallic content of mined ore should be at least 10 percent higher then the demand level of slab

TABLE 3-15. PROJECTED WORLD DEMAND FOR SLAB ZINC
(thousands of short tons)

Country/Region	1970	1980	1990	2000
Canada	106	190	280	380
United States	1184	1500	1750	2000
Latin America	173	310	500	740
Western Europe	1674	2250	2750	3350
Africa	74	150	250	410
Asia	897	1600	2350	3500
Oceania	124	150	180	220
Soviet Sphere	1131	1750	2700	4200
World Total	5363	7900	10,760	14,800

Source: Estimated by BCL

zinc. Assuming that scrap utilization balances the demand for ores and concentrates for zinc oxide, slab zinc demand can be equated to the demand for zinc and mined concentrates by applying a factor of 1.1. In 1970 actual mine output was approximately 4.8 million tons, which will need to grow to 6.9 million tons by 1980, 9.0 million tons by 1990, and 11.8 million tons by 2000. Figure 3-2 presents this curve for ores and estimates of actual mine capacity that may be installed by 1975 and 1980 which should provide the necessary production level. The cumulative affect of zinc ore production between 1970 and 2000 also is shown in relation to currently estimated reserves. Unless major new occurrences are discovered, the Free World may run out of zinc by about 1987. This is not a likely result because in the meantime exploration for copper and nickel will be going on during the course of which it can be anticipated tha additional new sources of zinc, not included in the above reserves estimates, will be located. In affect, this reinforces the conclusion that exploration in the high priority zones identified by S.G.S. for copper-zinc mineralizations is desirable and should be promoted promptly by the James Bay Development Corporation.

Probable Price Trends

Historically, zinc prices tend to respond rather rapidly to quite minor changes in the position of producers' stocks. This results from the fact that actual consumption of zinc may fluctuate rather rapidly from month to month but the smelter and mine operator are more rigidly constrained in terms of monthly production levels. As with other non-ferrous metals, a build up of producers' stocks usually results first in some discounting from quoted prices until metal begins to move again or an eventual drop in quoted prices if the inventory build up continues. In the opposite direction a rapid pull down of producers' stocks tends first to eliminate any discounting, secondly to a rise in quoted prices of about one-half cent per pound, and several successive price jumps if adjusted production schedules cannot keep up with demand. Again, it should be emphasized that a substantial portion of zinc production is tied rather directly to the output levels of lead on the one hand and copper on the other. Thus, while it is possible to stockpile zinc concentrates at the mine or at the smelter and refined zinc at the smelter or in consumers' warehouses, the elapsed time between mining of zinc and its sale to consumers as refined metal is seldom longer then one or two months. Moreover, the local market appears to be the determining factor since alternate sources of supply, such as imports, also have an inherent time lag that increases the pressure on prices. In this instance, the local market really means either a national situation, such as currently in force in the United States by virtue of price controls, or a regional situation such as the Western European area.

Historically, two series of prices are quoted with reference to world markets for zinc. They are the U.S. producer price and the London Metal Exchange price. The U.S. producer price reflects conditions in what is the largest single cohesive market in the world. The London Metal Exchange price (LME) on the other hand represents primarily spot purchases of small quantities of material from a London warehouse inventory, these transactions accounting for only a very small percentage of the total European market. Most zinc metal transfers directly from producer to consumer by way of regularly negotiated contracts. With the United States representing a substantial market even for European produced zinc, LME prices usually are anywhere from 1-1/2 to 3 cents a pound less than prices in the

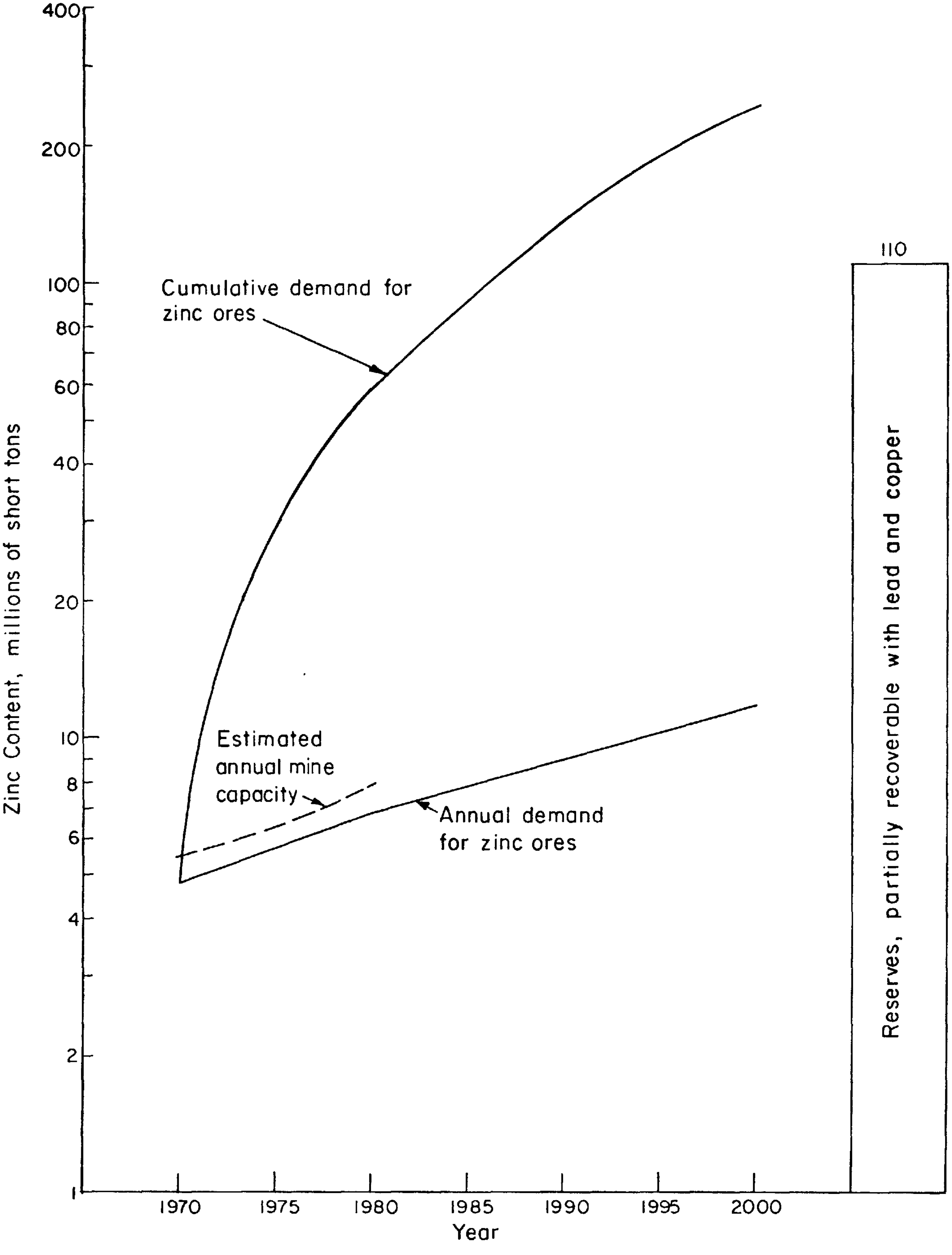

FIGURE 3-2. ANNUAL AND CUMULATIVE FREE WORLD DEMAND FOR ZINC ORES AND RESERVES

United States. The difference represents the probable actual cost of transportation and insurance from Europe to the U.S. and import duties into the U.S. that would be charged against European material, thus allowing it to be competitive with the U.S. producer price. In times of shortage in Western Europe or if an unusually heavy demand appears to be developing in the U.S., LME prices may approach or even exceed U.S. producer prices for some period of time. However, even with LME prices on top there is very little incentive for U.S. producers to try to sell in Europe rather than in the U.S. in order to realize higher sales values.

During the 1960's in the United States, the producer price of prime western zinc varied between 12 and 14 cents per pound in the early part of the decade and finally rose to 16 cents per pound in the latter years of the decade, ending up at 15 cents per pound in 1970. On the other hand, LME prices were as low as 8 cents per pound equivalent in 1963 and then rose rapidly to 16 cents per pound before settling again underneath the U.S. producer price, winding up the decade at about 13 cents per pound. During 1971, as industrial activity picked up in Western Europe, the United States, and Japan, the price of zinc in the U.S. rose from 15 to 17 cents per pound and then was frozen by price controls. In the meantime, LME prices continued to rise, eventually reaching a level of 22 cents per pound in 1972 when consumers in Europe and the U.S. were having difficulty finding material in the face of the closure of approximately 400,000 tons refining capacity in the United States because of environmental pollution and unfavorable economic conditions. Inflationary pressures as well as actual real increases in costs are likely to apply to zinc prices for some period of time yet as well as a slight market imbalance until additional refining capacities can be built. A return to 15 cent per pound zinc in the United States appears unreasonable, especially in 1972 constant dollars. A range of 17 to 20 cents per pound seems to provide a better setting for the next several years. If several major new finds--on the order of Pine Point, Anvil, Mattabi, or Ruttan--are located anywhere in the Free World between now and 1980, the 17 to 20 cents per pound range in constant 1972 dollars probably will hold through the 1970 decade. Even with such finds it is probable that zinc prices will increase in the 1980's, but the extent of the rise cannot be projected from information available at this point in time. Production of zinc in connection with copper--with copper forecasted to remain in the 50 to 75 cents a pound range--will tend to restrict any rise in the price of zinc. But the real key in the 1980's, particularly about 1985 to 1987, will be the status of supply/demand, and the proven reserves that exist at that time.

The Need for a New Producer in the James Bay Region

Because of the occurrence of zinc in connection with copper so frequently in the Canadian shield, the suggestion made in the preceding section on copper that additional producers in the James Bay Region would be desirable can automatically be extended to zinc. Obviously, the supply/demand situation with respect to zinc worldwide will provide a favorable climate for the sale of refined zinc or zinc concentrates from Canada in the major market areas. The only implication to be derived from this zinc marketing section would be that the zinc content of any future ore discoveries may produce a higher proportion of the revenues to be derived from the ore than they have in the past. Thus, this gives added impetus to search for both copper and zinc containing ores.

HYPOTHETICAL MINE/MILL PROJECTS FOR THE JAMES BAY REGION

Introduction

The history of mining in the Canadian shield strongly suggests that the probabilities are quite high that exploration in the James Bay Region will lead to the discovery and development of commercial deposits containing copper and zinc with associated trace elements. Present operations in the Matagami and Chibougamau Districts confirm this judgment. Accordingly, it appears appropriate to develop hypothetical mine/mill projects for zones that have been identified by S.G.S. as having high discovery potential. Again, the intent is not to choose already identified deposits of commercial size in order to make pre-engineering feasibility studies. Rather, the intent is to devise for each zone selected an integrated mine/mill project that would be considered to have threshold economic viability by virtue of the size and grade of deposit, apparent ease of mining, and apparent ease or difficulty in supplying manpower and operating supplies due to the location of the hypothetical deposit. In other words, these projects are not intended to limit in any fashion the exploitation of any deposit discovered in the future in James Bay Region that some operating company determines to represent a worthwhile opportunity for investment. Instead, they should be treated as guidelines by which the James Bay Development Corporation can readily evaluate whether or not a reported discovery in a given zone stands a reasonable chance of becoming a commercial operation prior to the time that a complete feasibility study has been performed on the specific deposit located.

Project Summary

Two possible copper-zinc projects are visualized. Each project consists of developing an underground deposit having a value of $80 million and supporting a 750 ton per day mill with an overall employment of 225 people. Project costs exclusive of road building requirements other then in the immediate vicinity are as follows:

Mine Development Cost	$5.00 million
Mill Construction & Equipment	3.00 million
Total	$8.00 million

One project is visualized as a copper-zinc deposit with minimal silver and gold in Zone 2A (See Figure 1 in Preface). The ore is assumed to have a value in the ground of $28.30 per ton based on 1.75 percent copper and 3.60 percent zinc contents and the deposit is expected to contain 2.83 million tons of ore. The 10-year output will be worth $52 million as concentrates. No town site would be required for workers in this particular zone because of proximity to Val Paradis.

The second project is considered to be a copper-zinc deposit having both silver and gold in recoverable quantities in Zone 1A (Figure 1), the Lake Frotet area. The ore is assumed to have a value in the ground of $30 per ton with 1.90 percent copper, 3.00 percent zinc, 1.0 oz. silver, and 0.02 oz. gold per ton, and the deposit contains 2.66 million tons of ore. The 10-year output will be worth $57.8 million as concentrates, with credit being given for silver and gold contained in the copper concentrates. This project will require a town site at an estimated capital investment cost of $2 million.

The Deposits

Based on geological considerations reported by S.G.S., the undiscovered potential in volcanogenic host rocks for copper, zinc, gold, silver, and possibly lead amounts to $5.8 billion. About 60 percent of this potential has been assigned to 6 zones in which this host rock type is found. These zones are identified and characterized in Table 3-16 together with estimates of the minimum value of an exploitable deposit in each zone and the assumed deposit characteristics that represent one possible solution for such a deposit. Zones 4A and 6A both are well within the infrastructural development that has already taken place in the southerly part of the James Region. In neither instance would it be required to have a town site or a significant road development program.

In such a reasonably developed sector the minimum value in the ground that represents a feasible mine/mill project has been established by S.G.S. by means of a statistical analysis of similar deposits in the Canadian Shield. The value assigned is $60 million, and, considering the fact that 2 million tons of ore represents the average size of deposit found, it follows that the value per ton will have to be on the order of $30 from combined lead, copper, and zinc in Zone 4, for which an archetype would be Orchan Mines, Ltd. It can be anticipated that a mine in the 6A zone would have slightly different characteristics with respect to the metallic values, although the size and grade would still work out to a $60 million value in the ground. In view of known deposits in the area an archetype for this mine would be Opemiska Mines, Ltd., where a copper content of 2.8 percent, a silver value of 0.5 oz. per ton, and a gold value of 0.03 oz. per ton may be expected.

As one moves away from established infrastructure, the development of an economic deposit usually involves additional costs. These may be compensated for by realizing higher revenues from the mine, which may be achieved by treating an ore with better grade value or by decreasing unit costs through handling larger daily volumes of material, or a combination of both. For Zone 2A, for example, while no town site would be required in consideration of the proximity to Val Paradis, operation will depend upon the construction of anywhere from 45 to 75 miles of road to get to the deposit site with attendant higher transportation charges. Under these conditions, BCL estimates that the minimum value in the ground of a deposit in this zone should be of the order of $80 million. It it is assumed that Mines de Poirier is the archetype of deposit likely to be found in the zone, it can then be established that the deposit would have a value per ton of $28.30, based on 1.75 percent copper and 3.60 percent zinc, and would contain 2.83 million tons of ore, which could support a 750 ton per day milling plant. Similarly, in Zone 1A the distance from Chibougamau suggests that a town site close to the mine-mill area would be needed for at least a portion of the miners and mill operators. With the announced construction of a highway into the area by the Department of Natural Resources of Quebec, the project itself might have

TABLE 3-16. ESTIMATED MINIMUM VALUE EXPLOITABLE DEPOSIT IN SIX ZONES OF HIGH GEOLOGICAL POTENTIAL FOR ZINC ORE COPPER

Zone and Type of Host Rock	Geological Potential for Copper-Nickel Ore (millions of dollars)(1)	Infrastructure Needs		Estimated Minimum Exploitable Deposit (millions of dollars)	Hypothetical Deposit Parameters(3)					
		Townsite	New Road (miles)		Deposit Size (millions of tons)	Value of Ore (dollars/ton)(2)	Copper Content (percent)	Zinc Content (percent)	Silver Content (oz/ton)	Cold Content (oz/ton)
1A	696	yes	0 to 15	80	2.66	30.00	1.90	3.00	1.0	0.02
2A	870	no	45 to 75	80	2.83	28.30	1.75	3.60	nil	---
4A	696	no	0	60	1.95	30.80	0.70	7.60	nil	---
5A	348	yes	40 to 60	96	3.00	32.00	2.00	4.00	nil	---
6A	464	no	0	60	2.00	30.00	2.80	---	0.5	0.03
8A	406	yes	40 to 50	110	3.30	35.50	2.20	4.50	nil	---

Source: Compiled by BCL.

Notes: (1) S.G.S. Report to Sores-Battelle
(2) Value "in-the-ground" based on $0.50 per pound for copper, $0.15 per pound for zinc, $1.00 per ounce for silver, and $50.00 per ounce for gold
(3) An underground deposit at a maximum depth of 1,200 to 1,500 1,500 feet is assumed.

to support only up to 15 miles of road building to get to the site. The added capital for a town site and transportation charges from and to railhead at Chibougamau mean a more expensive operation than in Zone 4A. Although differing in the makeup of the added capital required to exploit such a deposit from the 2A site, BCL estimates that the minimum value in the ground for this project also would be about $80 million. Recent announcements of a find being investigated by Selco Mining Corporation in Zone 1A leads to the suggestion that an archetype deposit might contain 1.90 percent copper, 3.00 percent zinc, 1 oz. per ton of silver, and 0.02 oz. per ton of gold for a total value of $40 per ton, thus resulting in a deposit of 2.66 million tons which could support a 750 ton per day mill. Lake Dufault also could be an archetype deposit.

Moving even farther away from infrastructure, a deposit in Zone 5A, between the Nottaway and Harricana Rivers, might require between 40 to 60 miles of road building from the Matagami to LG-2 Road, as well as a town site at which miners and mill operators with families could live. Considering that the junction of the access road with the Matagami to LG-2 Road would be roughly 100 miles from Matagami, it is considered likely that the deposit would need to have a higher grade of ore and slightly larger tonnage in order to be economic. Thus, a value of $96 million in the ground has been assigned to this particular hypothetical deposit, with Mines de Poirier again serving as the archetype. For Zone 8A, an access road of 40 to 50 miles in length from the Matagami to LG-2 Road also would be required as well as a town site for mill and mine personnel. The access road would take off at a point somewhat in the vicinity of 200 to 220 miles north of Matagami. Still using Mines de Poirier as the archetype, the added costs and risks involved suggest that the grade of ore should at least be equal to $35.50 per ton and that the deposit should contain no less than 3.3 million tons of ore which could support a mill treating 1,000 tons of ore per day. Total value in the ground of such a deposit would be approximately $110 million.

Of course, if a deposit is found in any of these zones that exceeds the minimums cited above, either in grade or tonnage or both, the probability of establishing a viable mining and milling project should be greater. The information assumed above should be used only as a rough guideline to the threshold of feasibility in various parts of the James Bay Region. Any future discoveries may be assessed rather readily with these guidelines prior to the preparation of specific feasibility studies based on proven deposits of ore tested for mineability and susceptibility to beneficiation.

The Mine

Mine Size and Material Handling Requirements

The mine in either Zone 1A or Zone 2A will consist of a single shaft which will eventually extend to a depth of 1,200 to 1,500 feet. Initially, three separate working levels would be developed ranging downward from about the 300 foot level.

It is expected that the mine would have an ore to waste ratio of approximately 5 to 1, after the initial adits have been opened up. Based on an anticipated 10 year mine life, the planned mill can be supported by the extraction of 1,100 tons of ore daily, working 5 days out of each of 51 weeks of the year. The following material removal rates would thus apply:

	Approximate Capacity in Tons		
Material	Daily	Annual	10-Year Mine Life
Ore	1,100	280,000	2.80 million
Waste	220	56,000	0.56 million
Total	1,320	336,000	3.36 million

Mine Equipment Needs

Before either of the deposits can be mined, the extent and position of the orebody must be determined by development drilling and a mining plan evolved. The development drilling also should indicate the character of the host rock which, together with the outline of the orebody, should suggest the mining approach to be used. Since underground mining equipment needs will vary rather widely depending upon the type of mining method employed, only the most general statement of such needs can be made prior to actual discovery and delineation of a deposit. Beyond the obvious need for hoisting capacity capable of handling 1,300 tons of ore and waste per day and for conveying miners and supplies, major equipment needs are summarized in Table 3-17, not including specialized equipment dictated by a particular mining method. It is assumed that 2 shifts of operation per day for the five day week will be the preferred schedule.

Mine Manpower List

As noted in the previous section, the mine is expected to operate 2 shifts per day for 5 days per week. To mine the 550 tons of ore per shift and 110 tons of gangue will take approximately 60 underground workers for the 3 different levels operated simultaneously. In addition, 15 surface and supervisory employees will be needed per shift. Thus, a total of 75 employees will be required per shift for a grand total of 150 employees. Table 3-18 present these requirements by skill levels.

Operating Supplies

Assuming that the mine can be equipped for electric powered vehicles rather then diesel-fueled ones, the major supply item in terms of weight would be explosives. Based on the experience of other mines in the area it can be assumed that 1 pound of explosives will be needed for each ton of ore and waste mined. Other supplies would include drill bits, rock bolts, lumber, hoses, replacement parts for various pieces of equipment, lubricating oil, etc. If the supplies weigh as much as the explosives required daily, inbound freight would amount to approximately 1 ton or 255 tons per year.

TABLE 3-17. COPPER-ZINC MINE EQUIPMENT LIST(1)
(2 shift, 5 days/week)

Item
Underground
Skip hoist
Jump drill rigs
Loading equipment
Ore cars
Portable air tools
Ventilation system
Pumps (ground water control)
Primary crusher
Above Ground
Conveyor (mine to mill)
Air compressor
Ventilation heating system
Maintenance and storage buildings
Office

Source: Compiled by BCL.

Note: (1) Only a generalized listing is possible until a deposit has been discovered and delineated by surface drilling and underground development.

TABLE 3-18. COPPER-ZINC MINE MANPOWER LIST
(2 shift, 5 days/week)

Designation	Number
Skilled Equipment Operators	21
Semiskilled Equipment Operators	70
Skilled Maintenance Personnel	4
Semiskilled Maintenance Personnel	14
Unskilled Labor	32
Total Labor	141
Management	4
Office and Engineering	5
Total Office	9
Total	150

Source: Estimated by BCL

Mining Effluents

Underground mining generates a small volume of waste in comparison to the ore removed primarily because selective mining usually follows the orebody. Adit development and the removal of other inadvertently placed waste should average only about 220 tons per day. Depending on the mining method, some part of this could be used as backfill in order to prevent collapse, or, it may be necessary to remove this material completely from the mine, in which case it would end up in a tailings pile.

Underground seepage in mines of the Canadian shield is virtually a foregone conclusion. In the presence of sulfide ores this seepage is likely to become acidic and require above ground treatment for disposal. In addition, run off from the tailings pile also could be acidic, thus provision to pump both of these effluents to the mill tailings treatment plant should be made when starting the mine.

The only airborne pollutant of consequence from the mining operation is likely to be dust, which can come from drilling and blasting, from underground crushing, or from any point at which there is a transfer of ore or waste materials. As, all of these operations are essentially underground, dust represents an internal problem that can be solved by proper filtering of the ventilation system or by wetting of the dust source. Above ground mine roads and tailings piles also represent potential dust sources.

The Mill

The mill will have a 750 tons per day capacity based on a 3 shift, 7 days per week operation. Crushed ore is received from the mine and the final output consists of a copper concentrate, a zinc concentrate, and tailings for disposal.

Flowsheet

Based primarily on the Mines de Poirier type of orebody, a flowsheet for a typical copper-zinc mill is given in Figure 3-3. Crude ore from the mine is passed through a secondary crusher and then wet-ground in a rod mill operating in closed circuit with a classifier, which passes approximately 100 mesh material onto the cyclones and ball mill circuit. The cyclones are operated to pass a minus 250 product onto the flotation circuits and recyle the plus 250 mesh fraction to the ball mill. After conditioning with a frother and collectors, and copper sulfate to supress zinc, the minus 250 mesh material is floated in copper rougher cells with the float material going forward to the copper cleaner cells, to the copper thickener, filter, drier, and then to storage for shipment. The underflow from the copper rougher cells passes through the copper scavenger cells, with the zinc material going forward to the zinc flotation conditioner. Here a promoter and soda ash are added as conditioning materials, and the pulp is sent to zinc rougher cells. The sink material from the rougher cells is sent to scavenger cells, then rescavenger cells and, finally, the underflow is sent to a tailings treatment tank

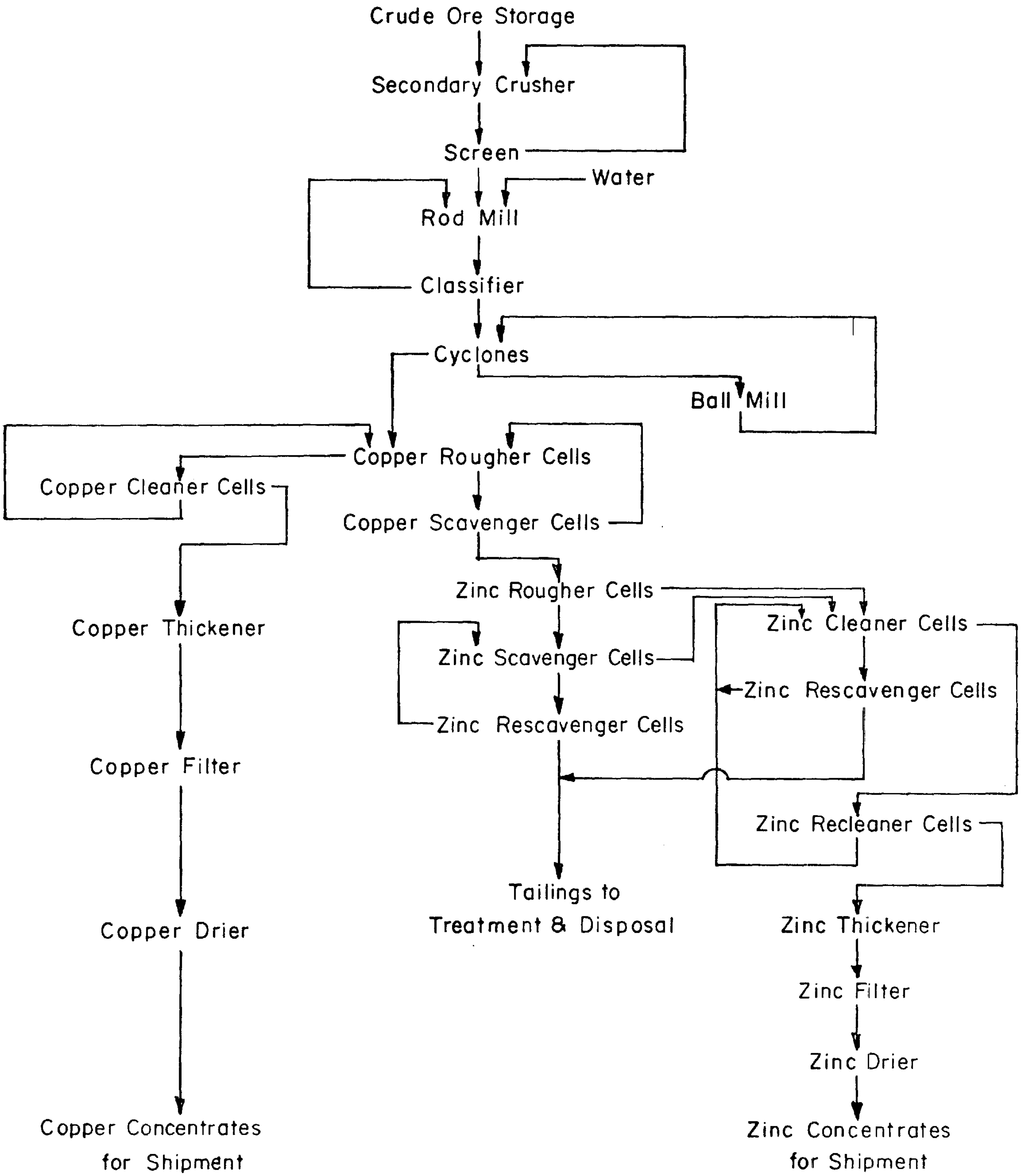

FIGURE 3-3. TYPICAL COPPER-ZINC MILL FLOWSHEET

where lime is added for pumping to disposal areas. The float material from scavenger cells and rescavenger cells is recycled as shown in the flowsheet. The float from the rougher cells proceeds to the zinc cleaner cells and recleaner cells, with appropriate internal recycling, and the ultimate float from the recleaner cells goes to the zinc thickener, filter, drier, and then to storage for shipment.

Mill Equipment Needs

The major equipment requirements for the flowsheet presented above are shown in Table 3-19. Sizing of equipment or a floor plan layout of the mill must await the discovery of a mineable deposit and the determination of specific treatment conditions for the characteristics of the ore found. It is anticipated that a 750 tons per day mill will require only one secondary crusher, one rod mill and one ball mill in order to liberate the minerals in the ore at a grind of about 250 mesh. Multiple units will be required for the cyclones and for flotation cells.

Mill Manpower List

Operation of the mill is scheduled for 3 shifts per day, 7 days per week, necessitating 4 crews of workers to stay within a 40 hour work week. Manpower requirements are given in Table 3-20 according to skill levels needed. Automated control for such a small installation is not anticipated but each operation will be well instrumented.

Mill Operating Supplies

The grinding steel requirements, chemical supplies, and water and electricity needed to operate this hypothetical mill can vary widely depending upon the actual characteristics of the ore discovered. Using the Mines de Poirier orebody as an archetype, the hypothetical mill in Zone 2A will use approximately 8 pounds of chemical per ton of ore milled and 2.23 pounds of grinding rods and balls. Fresh water intake is estimated at 360 gallons per ton of ore milled, thus total daily water requirements are 270,000 gallons. Total electric power needs are estimated at 28 Kilowatt hours per ton, a total of 21,000 Kilowatt hours per day. A possible breakdown of individual chemicals which may be appropriate are given in Table 3-21.

In Zone 1A, based on Lake Dufault as the archetype, a little less than 6.75 pounds of chemicals per ton of ore milled are required. Since the ore appears to be harder, roughly 2.50 pounds per ton of grinding rods and balls would be consumed. Again, the fresh water requirements are estimated at 360 gallons per ton with electricity at 30 Kilowatt hours per ton, or a total daily requirement of 22,500 Kilowatt hours. Possible chemical requirements are given in Table 3-21.

Exclusive of diesel fuel required to heat the building and to dry both copper and zinc concentrates, an approximate annual tonnage of supplies would be 1,431 tons for the Zone 2A site and 1,290 tons for the Zone 1A site.

TABLE 3-19. COPPER-ZINC MLL EQUIPMENT NEEDS
(3 shift, 7 days/week)

Item
Secondary crusher, gyratory
Vibrating screen
Rod mill
Classifier
Ball mills
Cyclones
Flotation cells (rougher cells, scavenger calls, cleaner cells, recleaner cells)
Thickeners
Filters
Driers

Source: Compiled by BCL

Note: (1) Only a generalized listing is possible until the ore has been characterized and a specific treatment procedure developed.

TABLE 3-20. COPPER-ZINC MILL MANPOWER LIST
(3 shift, 7 days/week)

Designation	Number
Skilled equipment operators	12
Semiskilled equipment operators	32
Skilled maintenance personnel	7
Semiskilled maitenance personnel	8
Unskilled labor	8
TOTAL LABOR	67
Management	4
Office and laboratory	4
TOTAL OFFICE	8
TOTAL	75

Source: Estimated by BCL

TABLE 3-21. COPPER-ZINC MILL SUPPLIES AND UTILITIES REQUIREMENTS

Item	Zone 1A		Zone 2A	
	Lbs/ton milled	Annual tons	Lbs/ton milled	Annual tons
Chemicals				
Collectors and promoters (xanthates)	0.10	14	0.16	22
Frothers (MIBC)	0.05	7	0.12	17
Soda ash	1.95	273	1.00	140
Copper sulfate	1.30	182	2.25	315
Guar gum	0.01	2	0.01	2
Lime	3.30	462	4.45	623
Subtotal	6.71		7.99	
Grinding Steel				
Rods and balls	2.50	350	2.23	312
TOTALS		1,290		1,431
Utilities		Daily		Daily
Fresh water	360 gal/ton	270,000 gal	360 gal/ton	270,000 gal
Electricity	30 Kwh/ton	22,500 Kwh	28 Kwh/ton	21,000 Kwh

Source: Estimated by BCL

Mill Effluents

The major effluent from the hypothetical mills will, of course, be the tailings that represent the waste and gangue separated from the concentrates. These tailings will consist of ore waste and spent chemical in slurry form which are sent to a thickener and treated with lime to neutralize any acid forming materials present in the waste. The thickened, neutralized tailings are pumped to a tailings disposal pond for storage and eventual reclamation as landfill. Under certain conditions, the tailings could be dewatered and pumped back into the mine to be used for backfilling mined-out areas. This would depend on the type of mining operation that is used.

Liquids used in the milling process are either recycled or discharged with the tailings. Liquid effluents of concern are those which may occur by seepage from the ore piles, from mine drainage, or from the tailings inpoundment area.

The major airborne effluent likely is dust, which could originate from sources such as roads, ore piles, the secondary crushing section, as well as from the product driers. An exhaust fan equipped with filters probably will be adequate for the crusher section and product drying area. Other possible dust sources inside or outside the plant may be controlled by wetting.

Mill Output and Value

For each of the hypothetical mills, it has been assumed that the recovery of copper will average 90 percent of the content of the ore while the recovery of zinc will be roughly 85 percent of the ore content. Further, it is expected that the copper concentrates from the Zone 1A site will contain recoverable quantities of silver and gold for which credit to the mill will be allowed.

The output at the Zone 1A site will consist of concentrates containing:

Daily	Yearly	Life of the Mine
12.9 tons of copper	4,600 tons of copper in concentrates	46,000 tons of copper
19.1 tons of zinc	6,820 tons of zinc in concentrates	68,200 tons of zinc

Taking 40 cents per pound as a value of copper in concentrates (representing roughly 3/4 of the sale price of refined metal), plus a credit of 5 cents per pound for the silver and gold content, and taking zinc at 12 cents per pound in concentrate form (again, 3/4 of the value of the sale price of the metal) the revenue for the mill should be approximately as follows:

Daily	Yearly	Life of the Mine
$16,065	$5,735,000	$57,352,000

The output of the hypothetical plant in Zone 2A will consist of concentrates containing:

	Yearly	Life of the Mine
11.7 tons of copper	4,180 tons of copper in concentrates	41,800 tons of copper
23.0 tons of zinc	8,210 tons of zinc in concentrates	82,100 tons of zinc

Again, using three fourths of the refined metal sale price as value of the material in concentrates, the revenue should be approximately:

Daily	Yearly	Life of the Mine
$14,880	$5,312,000	$53,122,000

Capital Requirements

Mine Costs

The cost of discovering and developing a mine can vary widely. Exploration and preliminary drilling, plus underground development to prove a commercially workable deposit, can range up to several million dollars, especially in areas where no ready access to a site is available by road. Headframe and service building construction, plus mining equipment costs and labor to drive the passage ways for the mine will add several million dollars more. Based on published costs for opening a number of mines in the Joutel, Matagami, and Chapais districts, and adjusted to the scale of 1,100 tons of ore per day and current prices, mine development and equipment costs are expected to be approximately $5 million for either the Zone 1A or the Zone 2A sites.

Mill Costs

No directly applicable mill cost figures were found in recent literature for a facility with a capacity of 750 tons per day. Based on the extrapolation of the hypothetical nickel mill costs, it can be estimated that the copper-zinc mill building and equipment, plus tailings treatment and disposal area, will cost about $3 million installed. This should be reasonably consistent for either the Zone 1A or the Zone 2A sites.

Project Total

The capital requirements for either project are summarized as follows:

Mine	$5.00 million
Mill	$3.00 million
Total	$8.00 million

In addition, it is expected that the town site required at the Zone 1A location would have a cost to the company of approximately $2 million.

Evaluation of Potential Worth

To illustrate the types of returns which can result if deposits are found similar to the characteristics and assumptions described previously, two sets of cash flow analyses were performed. The projects selected for analysis are:

(1) Mine/mill in Zone 2A (Mines de Poirier archetype)

(2) Mine/mill in Zone 1A, Lake Frotet (Lake Dufault archetype)

Data Inputs and Assumptions - Zone 2A Project

Capital Outlays. Two capital outlays are associated with the project:

(1) Mine development costs are estimated at $5.0 million

(2) Mill construction and equipment costs are estimated at $3.0 million

Fifty percent of total capital requirements are borrowed on an 11 year loan at 8 percent interest. Building and equipment life is 10 years with no salvage value.

Operating Expenses. Five annual operating expenses are considered:

(1) Mining costs associated with mining 1100 tons per day (at depths between 300-1500 feet) are selected from the following distribution:

Annual Costs (thousands of $)	Median Cost ($ per ton)	Probability * (percent)
1728-1835	6.60	5
1836-1944	7.00	10
1945-2052	7.40	65
2053-2160	7.80	20

(2) Milling costs are sampled from the distribution below:

Annual Costs (thousands of $)	Median Cost ($ per ton)	Probability * (percent)
810- 877	3.12	10
878- 944	3.37	30
945-1012	3.62	50
1013-1080	3.87	10

(3) Annual transport costs are calculated as:

For truck shipment of 16,450 tons/year of copper concentrates from the mill site to Noranda @ 10¢ per ton mile for a distance of 150 miles = $246.7 thousand

For shipment of 12,880 tons/year of zinc concentrates, costs associated with the mill site to railhead haul (LaSarre) are $90.2 thousand. The rail shipment costs from LaSarre to Valleyfield (at $8.50/ton) are $109,480. Thus total costs for shipping zinc concentrates are $199.7 thousand. For both concentrates, total annual costs are $446.4 thousand.

(4) Interest charges are computed within the program for total debt of $4.0 million for an 11 year loan at 8 percent interest. Payback on principal begins in the first year of production.

(5) Straight line depreciation is also computed within the program.

Revenues. Revenues from the project are computed at 75 percent of refined metal sales prices anticipated over the life of the project.

(1) Thus, for a refined copper price of $.50/lb., value of the concentrate would be $.375/lb. Total annual revenues from copper concentrate sales are sampled from the distribution below:

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* See footnote, page 1-49.

Annual Revenues (thousands of $)	Median Refined Metal Price(¢ per lb.)	Probability * (percent)
3071-3377	52.5	30
3378-3685	57.5	50
3686-3992	62.5	15
3993-4300	67.5	5

(2) A refined zinc price of $.16/lb. accordingly would suggest a mill revenue of $.12/lb. in concentrate form. Annual revenues are sampled from the following:

Annual Revenues (thousands of $)	Median Refined Metal Price (SHG) (¢ per lb.)	Probability * (percent)
1691-1931	15.0	5
1932-2173	17.0	35
2174-2414	19.0	50
2415-2657	21.0	10

Output - Zone 2A Project

For the above input ranges, 50 iterations of the cash flow model were run. Potential rates of return for the 2A project vary between 11 and 29 percent. Figure 3-4 expresses the above yields as a cumulative probability distribution and indicates that there is a 65 percent chance of earning a rate of return of 19 percent or better. Tables 3-22 to 3-24 are sample outputs which demonstrate the sensitivity of variable cost and revenue values.

Data Inputs and Assumptions - Zone 1A Project

Capital Outlays. As expected, mining and milling output volumes for the 1A project are the same as for 2A, thus mine development and mill development costs are the same, $5.0 and $3.0 million respectively. An additional capital outlay for the Frotet deposit relates to the probable need for a townsite, as the location of the hypothetical deposit is at least 80 miles from the nearest town. Townsite development costs are estimated at $2.0 million. Fifty percent of total capital requirements are borrowed at 8 percent interest for an 11 year loan. Principal payback begins in year 2, the first year of production. Depreciable life of buildings and equipment are 10 years with no salvage value.

Operating Expenses. Five annual operating expenses are again considered.

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* See footnote, page 1-54.

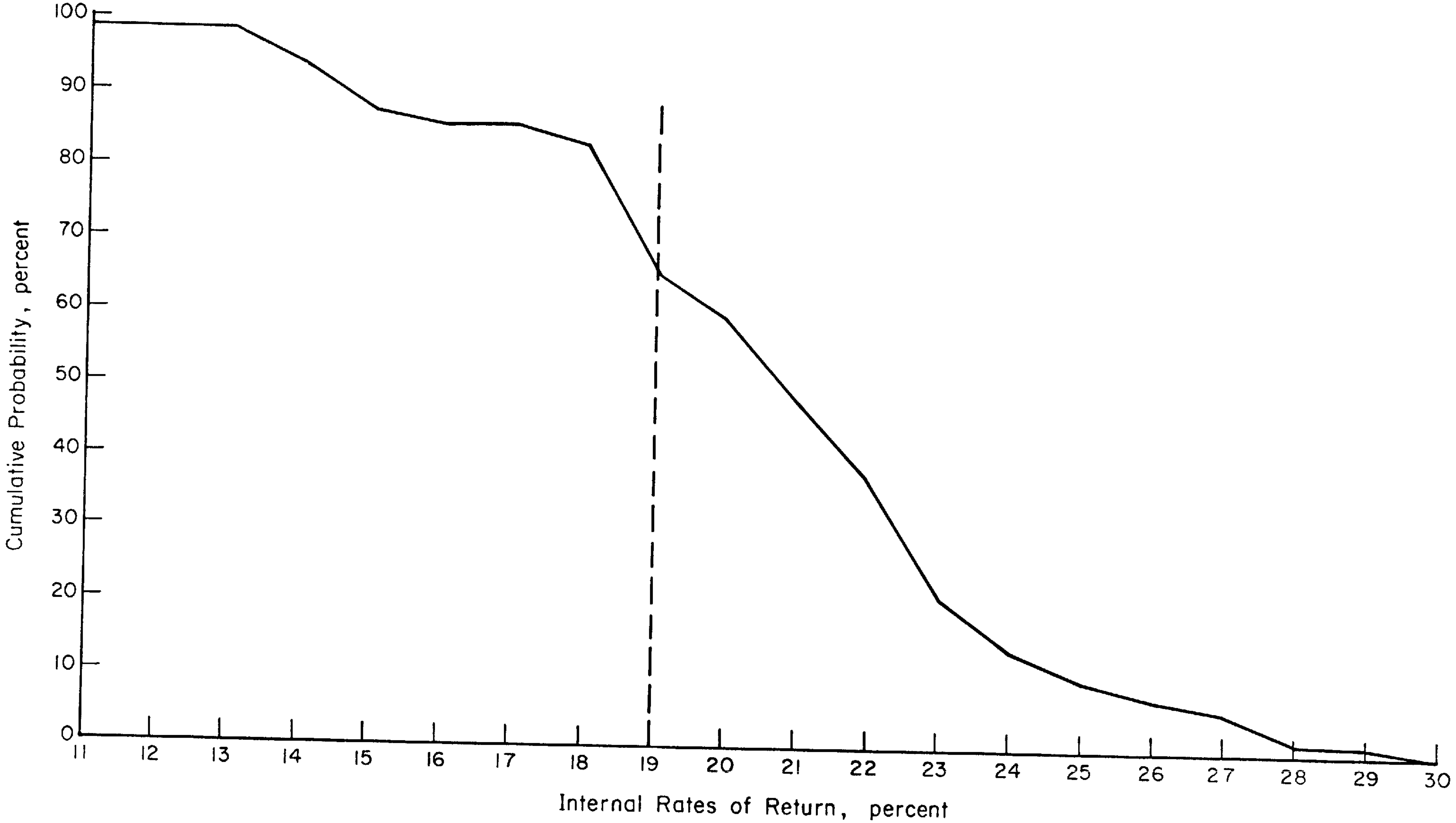

FIGURE 3-4. INTERNAL RATES OF RETURN VERSUS PROBABILITY, ZONE 2-A, COPPER-ZINC PROJECT

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                    TABLE 3-22.  COPPER-ZINC MINE/MILL AT 2-A
                                         ITERATION NUMBER     13
             YEAR                1        2        3        4        5        6        7        8        9       10       11
 CAPITAL OUTLAYS  -

 MINE DEVELOPMENT
 -----INTERNAL FUNDS        2500.0      0.0      0.0      0.0      0.0      0.0      0.0      0.0      0.0      0.0      0.0
 -----LOAN PRINCIPAL           0.0    172.6    186.4    201.3    217.4    234.8    253.5    273.9    295.8    319.4    345.0
 MILL CONSTR.,EQUIP.
 -----INTERNAL FUNDS        1500.0      0.0      0.0      0.0      0.0      0.0      0.0      0.0      0.0      0.0      0.0
 -----LOAN PRINCIPAL           0.0    103.5    111.8    120.8    130.4    140.9    152.1    164.3    177.5    191.7    207.0

                 TOTALS     4000.0    276.1    298.2    322.1    347.8    375.7    405.7    438.2    473.2    511.1    552.0

 ANNUAL EXPENSES

 MINING COSTS                  0.0   2053.0   2053.0   2053.0   2053.0   2053.0   2053.0   2053.0   2053.0   2053.0   2053.0
 MILLING COSTS                 0.0   1073.9   1073.9   1073.9   1073.9   1073.9   1073.9   1073.9   1073.9   1073.9   1073.9
 TRANSPORT COSTS              -0.0    444.0    444.0    444.0    444.0    444.0    444.0    444.0    444.0    444.0    444.0
 INTEREST ON LOANS           320.0    320.0    297.9    274.1    248.3    220.5    190.4    158.0    122.9     85.0     44.2
 TOTAL DEPRECIATION          727.3    727.3    727.3    727.3    727.3    727.3    727.3    727.3    727.3    727.3    727.3

                 TOTALS     1047.3   4618.1   4596.1   4572.2   4546.4   4518.6   4488.6   4456.1   4421.0   4383.2   4342.3

 ANNUAL REVENUES

 SALE OF CU CONCEN.            0.0   3485.5   3485.5   3485.5   3485.5   3485.5   3485.5   3485.5   3485.5   3485.5   3485.5
 SALE OF ZN CONCEN.            0.0   1746.6   1746.6   1746.6   1746.6   1746.6   1746.6   1746.6   1746.6   1746.6   1746.6

                 TOTALS        0.0   5232.1   5232.1   5232.1   5232.1   5232.1   5232.1   5232.1   5232.1   5232.1   5232.1


 TAXABLE INCOME                0.0      0.0    292.7    659.9    685.7    713.5    743.6    776.0    811.1    848.9    889.8

 INCOME TAX                    0.0      0.0    111.0    361.3    375.4    390.6    407.1    424.9    444.1    464.8    487.2

 NET INCOME                    0.0      0.0     91.7    298.6    310.3    322.9    336.5    351.1    367.0    384.1    402.6

 CASH FLOW                 -4320.0   1065.1    954.1    703.8    689.7    674.5    658.0    640.3    621.1    600.3    577.9

 TERMINAL WORTH                   =        0.000

 YIELD(IN PERCENT)                =       11.276
 PRESENT WORTH AT  0 PERCENT =          2854.961
 PRESENT WORTH AT  5 PERCENT =          1275.163
 PRESENT WORTH AT 10 PERCENT =           210.436
```

TABLE 3-23. COPPER-ZINC MINE/MILL AT 2-A

```
                                     ITERATION NUMBER     38
         YEAR                    1         2        3        4        5        6        7        8        9       10       11
CAPITAL OUTLAYS

MINE DEVELOPMENT
-----INTERNAL FUNDS         2500.0       0.0      0.0      0.0      0.0      0.0      0.0      0.0      0.0      0.0      0.0
-----LOAN PRINCIPAL            0.0     172.6    186.4    201.3    217.4    234.8    253.6    273.9    295.8    319.4    345.0
MILL CONSTR.,EQUIP.
-----INTERNAL FUNDS         1500.0       0.0      0.0      0.0      0.0      0.0      0.0      0.0      0.0      0.0      0.0
-----LOAN PRINCIPAL            0.0     103.5    111.8    120.8    130.4    140.9    152.1    164.3    177.5    191.7    207.0

           TOTALS           4000.0     276.1    298.2    322.1    347.8    375.7    405.7    438.2    473.2    511.1    552.0

ANNUAL EXPENSES

MINING COSTS                   0.0    2061.4   2061.4   2061.4   2061.4   2061.4   2061.4   2061.4   2061.4   2061.4   2061.4
MILLING COSTS                  0.0    1028.0   1028.0   1028.0   1028.0   1028.0   1028.0   1028.0   1028.0   1028.0   1028.0
TRANSPORT COSTS               -0.0     444.0    444.0    444.0    444.0    444.0    444.0    444.0    444.0    444.0    444.0
INTEREST ON LOANS            320.0     320.0    297.9    274.1    248.3    220.5    190.4    158.0    122.9     85.0     44.2
TOTAL DEPRECIATION           727.3     727.3    727.3    727.3    727.3    727.3    727.3    727.3    727.3    727.3    727.3

           TOTALS           1047.3    4580.7   4558.6   4534.7   4509.0   4481.2   4451.1   4418.6   4383.6   4345.7   4304.9

ANNUAL REVENUES

SALE OF CU CONCEN.             0.0    3519.7   3519.7   3519.7   3519.7   3519.7   3519.7   3519.7   3519.7   3519.7   3519.7
SALE OF ZN CONCEN.             0.0    2329.4   2329.4   2329.4   2329.4   2329.4   2329.4   2329.4   2329.4   2329.4   2329.4

           TOTALS              0.0    5849.1   5849.1   5849.1   5849.1   5849.1   5849.1   5849.1   5849.1   5849.1   5849.1


TAXABLE INCOME                 0.0     221.1   1290.5   1314.3   1340.1   1367.9   1398.0   1430.4   1465.5   1503.3   1544.2

INCOME TAX                     0.0     121.1    706.5    719.6    733.7    748.9    765.4    783.2    802.4    823.1    845.5

NET INCOME                     0.0     100.1    583.9    594.7    606.4    619.0    632.6    647.3    663.1    680.3    698.8

CASH FLOW                  -4320.0    1598.5   1013.1    999.9    985.8    970.6    954.1    936.4    917.2    895.5    874.1

TERMINAL WORTH                 =        0.000

YIELD(IN PERCENT)              =       19.912
PRESENT WORTH AT  0 PERCENT =        5826.132
PRESENT WORTH AT  5 PERCENT =        3447.364
PRESENT WORTH AT 10 PERCENT =        1879.391
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                         TABLE 3-24.  COPPER-ZINC MINE/MILL AT 2-A
                                                ITERATION NUMBER      3

            YEAR                   1       2       3       4       5       6       7       8       9      10      11
CAPITAL OUTLAYS

MINE DEVELOPMENT
-----INTERNAL FUNDS           2500.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0
-----LOAN PRINCIPAL              0.0   172.6   186.4   201.3   217.4   234.8   253.5   273.9   295.8   319.4   345.0
MILL CONSTR.,EQUIP.
-----INTERNAL FUNDS           1500.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0
-----LOAN PRINCIPAL              0.0   103.5   111.8   120.8   130.4   140.9   152.1   164.3   177.5   191.7   207.0

               TOTALS         4000.0   276.1   298.2   322.1   347.8   375.7   405.7   438.2   473.2   511.1   552.0

ANNUAL EXPENSES

MINING COSTS                     0.0  1731.2  1731.2  1731.2  1731.2  1731.2  1731.2  1731.2  1731.2  1731.2  1731.2
MILLING COSTS                    0.0   948.0   948.0   948.0   948.0   948.0   948.0   948.0   948.0   948.0   948.0
TRANSPORT COSTS                 -0.0   444.0   444.0   444.0   444.0   444.0   444.0   444.0   444.0   444.0   444.0
INTEREST ON LOANS              320.0   320.0   297.9   274.1   248.3   221.5   190.4   158.0   122.9    85.0    44.2
TOTAL DEPRECIATION             727.3   727.3   727.3   727.3   727.3   727.3   727.3   727.3   727.3   727.3   727.3

               TOTALS         1047.3  4170.4  4148.4  4124.5  4098.7  4070.9  4040.9  4008.4  3973.3  3935.5  3894.6

ANNUAL REVENUES

SALE OF CU CONCEN.               0.0  3965.2  3965.2  3965.2  3965.2  3965.2  3965.2  3965.2  3965.2  3965.2  3965.2
SALE OF ZN CONCEN.               0.0  2328.8  2328.8  2328.8  2328.8  2328.8  2328.8  2328.8  2328.8  2328.8  2328.8

               TOTALS            0.0  6294.0  6294.0  6294.0  6294.0  6294.0  6294.0  6294.0  6294.0  6294.0  6294.0

TAXABLE INCOME                   0.0  1076.3  2145.7  2169.5  2195.3  2223.1  2253.2  2285.6  2320.7  2358.5  2399.4

INCOME TAX                       0.0   589.3  1174.7  1187.8  1201.9  1217.2  1233.6  1251.4  1270.6  1291.3  1313.7

NET INCOME                       0.0   487.0   970.9   981.7   993.4  1006.0  1019.6  1034.2  1050.1  1067.2  1085.7

CASH FLOW                    -4320.0  1985.5  1400.0  1386.9  1372.8  1357.6  1341.1  1323.4  1304.2  1283.4  1261.0

TERMINAL WORTH             =         0.000

YIELD(IN PERCENT)          =        28.854
PRESENT WORTH AT  0 PERCENT =     9695.830
PRESENT WORTH AT  5 PERCENT =     6272.254
PRESENT WORTH AT 10 PERCENT =     3943.910
```

(1) Mining costs associated with producing 1100 tons per day (at depths between 300-1800 feet) are sampled from the distribution below:

Annual Costs (thousands of $)	Median Cost ($ per ton)	Probability * (percent)
1836-1943	7.00	5
1944-2051	7.40	15
2052-2159	7.80	55
2160-2268	8.20	25

(2) Milling costs are selected from the following:

Annual Costs (thousands of $)	Median Cost ($ per ton)	Probability * (percent)
810- 877	3.12	5
878- 944	3.37	35
945-1012	3.62	45
1013-1080	3.87	15

(3) Transport costs are calculated as:

For trucking copper concentrates (18,350 tons/year) to Chibougamau (80 miles @ 10¢ per ton mile), annual costs are $146.8 thousand; for rail shipment to Noranda @ $4.50/ton, costs are $82,575 annually. Total shipping costs for copper concentrates = $219.3 thousand.

For trucking zinc concentrates (10,700 tons/year) to railhead, costs @ 10¢ per ton mile for 80 miles are $85.6 thousand; for rail shipment to Valleyfield at $8.50/ton, costs are 90.9 thousand. Total shipping costs for zinc concentrates = $176.5 thousand.

Total annual transport costs are estimated as $396.0 thousand.

(4, 5) Interest and depreciation are calculated as in the 2A project.

Revenues. Sales prices of copper concentrates from the 1A deposit are slightly higher than for 2A due to associated values of gold and silver (0.02 oz./ton and 1.0 oz./ton respectively) which report primarily with the copper.

(1) Annual revenues are sampled from:

---

* See footnote, page 1-49.

Annual Revenues (thousands of $)	Median Price (¢/lb. of concentrate)	Probability* (percent)
3902-4268	44.5	30
4269-4589	48.5	50
4590-4910	52.5	15
4911-5278	55.5	5

(2) Annual revenues from sale of zinc concentrates are selected from:

Annual Revenues (thousands of $)	Median Refined Zinc Price (¢/lb. of concentrate)	Probability* (percent)
1446-1651	11.2	5
1652-1858	12.7	35
1859-2065	14.2	50
2066-2272	15.7	10

Output

For the above combinations of inputs, 50 iterations of the cash flow model indicated a range of rates of return between 13 and 26 percent. A cumulative percentage distribution of the yields (Figure 3-5) indicates that there is a 64 percent chance of obtaining a yield of 18 percent or better. Tables 3-25 to 3-27 are illustrative of which types of combinations can result in high, medium, and low rates of return.

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* See footnote, page 1-54

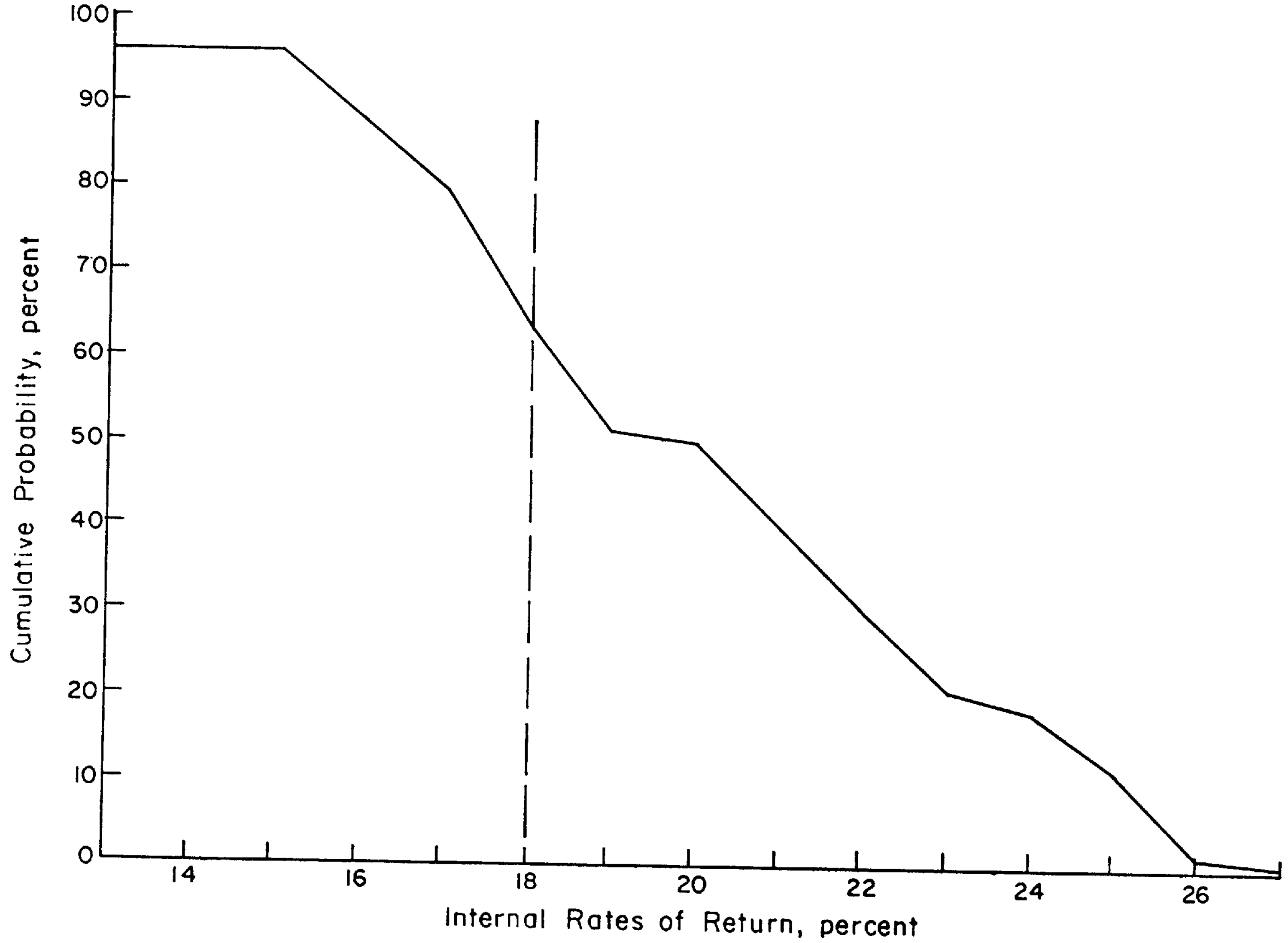

FIGURE 3-5. INTERNAL RATES OF RETURN VERSUS PROBABILITY, ZONE I-A, COPPER-ZINC PROJECT

TABLE 3-25. LAKE FROTET COPPER-ZINC MINE / MILL (1A)
ITERATION NUMBER 22

YEAR	1	2	3	4	5	6	7	8	9	10	11
CAPITAL OUTLAYS											
MINE DEVELOPMENT											
-----INTERNAL FUNDS	2500.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	172.6	186.4	201.3	217.4	234.8	253.6	273.9	295.8	319.4	345.0
MILL CONSTR.,EQUIP.											
-----INTERNAL FUNDS	1500.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	103.5	111.8	120.8	130.4	140.9	152.1	164.3	177.5	191.7	207.0
TOWNSITE											
-----INTERNAL FUNDS	1000.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	69.0	74.6	80.5	87.0	93.9	101.4	109.5	118.3	127.8	138.0
TOTALS	5000.0	345.1	372.8	402.6	434.8	469.6	507.1	547.7	591.5	638.8	690.0
ANNUAL EXPENSES											
MINING COSTS	0.0	2228.3	2228.3	2228.3	2228.3	2228.3	2228.3	2228.3	2228.3	2228.3	2228.3
MILLING COSTS	0.0	996.7	996.7	996.7	996.7	996.7	996.7	996.7	996.7	996.7	996.7
TRANSPORT COSTS	-0.0	389.0	389.0	389.0	389.0	389.0	389.0	389.0	389.0	389.0	389.0
INTEREST ON LOANS	400.0	400.0	372.4	342.6	310.4	275.6	238.0	197.4	153.6	106.3	55.2
TOTAL DEPRECIATION	909.1	909.1	909.1	909.1	909.1	909.1	909.1	909.1	909.1	909.1	909.1
TOTALS	1309.1	4923.1	4895.5	4865.7	4833.4	4798.7	4761.1	4720.5	4676.7	4629.4	4578.3
ANNUAL REVENUES											
SALES OF CU CONCEN.	0.0	3950.8	3950.8	3950.8	3950.8	3950.8	3950.8	3950.8	3950.8	3950.8	3950.8
SALES OF ZN CONCEN.	0.0	1850.4	1850.4	1850.4	1850.4	1850.4	1850.4	1850.4	1850.4	1850.4	1850.4
TOTALS	0.0	5801.2	5801.2	5801.2	5801.2	5801.2	5801.2	5801.2	5801.2	5801.2	5801.2
TAXABLE INCOME	0.0	0.0	474.8	935.6	967.8	1002.6	1040.1	1080.7	1124.5	1171.8	1222.9
INCOME TAX	0.0	0.0	259.9	512.2	529.9	548.9	569.5	591.7	615.7	641.6	669.6
NET INCOME	0.0	0.0	214.9	423.3	437.9	453.7	470.7	489.0	508.8	530.3	553.4
CASH FLOW	-5400.0	1442.1	1182.1	929.9	912.2	893.2	872.6	850.4	826.4	800.5	772.5

TERMINAL WORTH = 0.000

YIELD(IN PERCENT) = 12.573
PRESENT WORTH AT 0 PERCENT = 4081.952
PRESENT WORTH AT 5 PERCENT = 1962.268
PRESENT WORTH AT 10 PERCENT = 540.165

TABLE 3-26. LAKE FROTET COPPER-ZINC MINE / MILL (1A)
ITERATION NUMBER 4

YEAR	1	2	3	4	5	6	7	8	9	10	11
CAPITAL OUTLAYS											
MINE DEVELOPMENT											
-----INTERNAL FUNDS	2500.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	172.6	186.4	201.3	217.4	234.8	253.6	273.9	295.8	319.4	345.0
MILL CONSTR.,EQUIP.											
-----INTERNAL FUNDS	1500.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	103.5	111.8	120.8	130.4	140.9	152.1	164.3	177.5	191.7	207.0
TOWNSITE											
-----INTERNAL FUNDS	1000.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	69.0	74.6	80.5	87.0	93.9	101.4	109.5	118.3	127.8	138.0
TOTALS	5000.0	345.1	372.8	402.6	434.8	469.6	507.1	547.7	591.5	638.8	690.0
ANNUAL EXPENSES											
MINING COSTS	0.0	2152.5	2152.5	2152.5	2152.5	2152.5	2152.5	2152.5	2152.5	2152.5	2152.5
MILLING COSTS	0.0	1042.9	1042.9	1042.9	1042.9	1042.9	1042.9	1042.9	1042.9	1042.9	1042.9
TRANSPORT COSTS	-0.0	389.0	389.0	389.0	389.0	389.0	389.0	389.0	389.0	389.0	389.0
INTEREST ON LOANS	400.0	400.0	372.4	342.6	310.4	275.6	238.0	197.4	153.6	106.3	55.2
TOTAL DEPRECIATION	909.1	909.1	909.1	909.1	909.1	909.1	909.1	909.1	909.1	909.1	909.1
TOTALS	1309.1	4893.5	4865.9	4836.1	4803.9	4769.1	4731.5	4691.0	4647.2	4599.8	4548.7
ANNUAL REVENUES											
SALES OF CU CONCEN.	0.0	4452.0	4452.0	4452.0	4452.0	4452.0	4452.0	4452.0	4452.0	4452.0	4452.0
SALES OF ZN CONCEN.	0.0	1838.3	1838.3	1838.3	1838.3	1838.3	1838.3	1838.3	1838.3	1838.3	1838.3
TOTALS	0.0	6290.3	6290.3	6290.3	6290.3	6290.3	6290.3	6290.3	6290.3	6290.3	6290.3
TAXABLE INCOME	0.0	87.7	1424.4	1454.2	1486.4	1521.2	1558.7	1599.3	1643.1	1690.4	1741.5
INCOME TAX	0.0	48.0	779.8	796.2	813.8	832.8	853.4	875.6	899.6	925.5	953.5
NET INCOME	0.0	39.7	644.5	658.0	672.6	688.3	705.3	723.7	743.5	764.9	788.0
CASH FLOW	-5400.0	1912.7	1180.9	1164.5	1146.9	1127.8	1107.3	1085.1	1061.1	1035.2	1007.2

TERMINAL WORTH = 0.000

YIELD(IN PERCENT) = 18.119
PRESENT WORTH AT 0 PERCENT = 6428.600
PRESENT WORTH AT 5 PERCENT = 3685.736
PRESENT WORTH AT 10 PERCENT = 1834.760

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              TABLE 3-27.  LAKE FOREST COPPER-ZINC MINE / MILL (1A)
                                     ITERATION NUMBER    19

           YEAR                      1       2       3       4       5       6       7       8       9      10      11
   CAPITAL OUTLAYS

   MINE DEVELOPMENT
   -----INTERNAL FUNDS          2500.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0
   -----LOAN PRINCIPAL             0.0   172.6   186.4   201.7   217.4   234.8   253.6   273.9   295.8   319.4   345.0
   MILL CONSTR.,EQUIP.
   -----INTERNAL FUNDS          1500.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0
   -----LOAN PRINCIPAL             0.0   103.5   111.8   120.8   130.4   140.9   152.1   164.3   177.5   191.7   207.0
   TOWNSITE
   -----INTERNAL FUNDS          1000.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0     0.0
   -----LOAN PRINCIPAL             0.0    69.0    74.6    80.5    87.0    93.9   101.4   109.5   118.3   127.8   138.0

               TOTALS           5000.0   345.1   372.8   402.6   434.8   469.6   507.1   547.7   591.5   638.8   690.0

   ANNUAL EXPENSES

   MINING COSTS                    0.0  1982.6  1982.6  1982.6  1982.6  1982.6  1982.6  1982.6  1982.6  1982.6  1982.6
   MILLING COSTS                   0.0   904.4   904.4   904.4   904.4   904.4   904.4   904.4   904.4   904.4   904.4
   TRANSPORT COSTS                -0.0   389.0   389.0   389.0   389.0   389.0   389.0   389.0   389.0   389.0   389.0
   INTEREST ON LOANS             400.0   400.1   372.4   342.6   310.4   275.6   238.0   197.4   153.6   106.3    55.2
   TOTAL DEPRECIATION            909.1   909.1   909.1   909.1   909.1   909.1   909.1   909.1   909.1   909.1   909.1

               TOTALS           1309.1  4585.0  4557.4  4527.6  4495.4  4460.6  4423.0  4382.5  4338.7  4291.3  4240.2

   ANNUAL REVENUES

   SALES OF CU CONCEN.             0.0  4668.8  4668.8  4668.8  4668.8  4668.8  4668.8  4668.8  4668.8  4668.8  4668.8
   SALES OF ZN CONCEN.             0.0  2172.2  2172.2  2172.2  2172.2  2172.2  2172.2  2172.2  2172.2  2172.2  2172.2

               TOTALS              0.0  6840.9  6840.9  6840.9  6840.9  6840.9  6840.9  6840.9  6840.9  6840.9  6840.9


   TAXABLE INCOME                  0.0   946.8  2283.5  2313.3  2345.5  2380.3  2417.9  2458.5  2502.3  2549.6  2600.7

   INCOME TAX                      0.0   518.4  1250.2  1266.6  1284.2  1303.2  1323.8  1346.0  1370.0  1395.9  1423.9

   NET INCOME                      0.0   428.4  1033.3  1046.8  1061.4  1077.1  1094.1  1112.5  1132.3  1153.7  1176.8

   CASH FLOW                   -5400.0  2301.5  1569.6  1553.3  1535.7  1516.6  1496.1  1473.8  1449.9  1423.9  1396.0

   TERMINAL WORTH                  =         0.000

   YIELD(IN PERCENT)               =        25.719
   PRESENT WORTH AT  0 PERCENT =         10316.341
   PRESENT WORTH AT  5 PERCENT =          6523.791
   PRESENT WORTH AT 10 PERCENT =          3949.185
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CHAPTER 4. ASBESTOS

CHAPTER 4. ASBESTOS

SUPPLY/DEMAND SITUATION TO THE YEAR 2000

Occurrence and Beneficiation

Occurrence

Asbestos is a designation applied to a variety of fibrous mineral silicates that are found in nature. Although differing in chemical composition and certain other properties they have common characteristics of incombustibility, thermal insulation, some resistance to chemicals, and fibrosity. Six varieties of asbestos are generally recognized. The first is the finely fibrous form of serpentine known as chrysotile. The other five are minerals of the amphibole group and include amosite, anthophyllite, crocidolite, tremolite, and actinolite.

More then 90 percent of the current world production of asbestos consists of the chrysotile variety. All but a fraction of a percent of this is derived from deposits whose host rocks are ultrabasic in composition. Predominantly, chrysotile asbestos is the result of hydrothermal reactions under certain conditions within the serpentine rock, whether it be peridotite, dunite, or pyroxenite. Variability in the host rock as well as in the mechanism of chrysotile formation leads to imperfections in the crystal structure of asbestos, as well as inclusions of impurities between fiber bundles which are difficult to remove by mere mechanical means. Therefore, the value of a given deposit may be assessed only after fibers have been separated from the host rock and then subjected to various mechanical, chemical, thermal, and electrical tests.

The bulk of chrysotile production comes from three principal areas: the Eastern Townships of Quebec in Canada, the Bajenova District in the Urals of the U.S.S.R., and from Southern Rhodesia and the Transvaal in Africa. The Quebec deposits are reported to be mid Paleozoic in age, those in the Urals are believed to be either late Paleozoic or Triassic while those in Africa are Precambrian in age.

Chrysotile is a hydrated magnesium silicate. Its fibers are fine and flexible, the better qualities having a silky luster and high tensile strength. They are short, usually from 1/16 to 1/2 inch long and rarely exceed 6 inches in length. The longer grades can be spun and woven into cloth. Chrysotile resists heat up to about 400° C where it begins to deteriorate, but its resistance even to weak acids and sea water is low.

In brief, ground waters contacting hot ultrabasic intrusive rocks along geologic faults are believed to be responsible for the formation of fibrous chrysotile. Almost invariably, it is surrounded by host rock of similar chemical composition.

in massive form. Commercial deposits may contain from 4 to about 9 percent of chrysotile in rock considered ore, and will range upward from 500,000 tons to many millions of tons of ore. Both underground and open pit mining have been used to exploit deposits in Canada and Africa. The choice of mining method depends primarily on the size, position, and depth of the deposit, as well as the grade of ore that can be extracted for subsequent treatment.

Beneficiation

When the possibility exists for separating fibers that are 2 inches or longer, crude ore from a mine may be hand sorted and the initial fiber bundles separated manually. Such a procedure seldom applies to chrysotile, which tends to be much shorter then 2 inches at best. A majority of the asbestos produced in Canada is known as mill fiber, indicating that the fiber bundles have been released by a milling action and the bundles fluffed or "teased" to liberate individual fibers.

The treatment of asbestos ore may be separated into several sections: (1) crushing, (2) drying, (3) primary liberation, (4) fiber opening, (5) screening and grading, and (6) packaging. All of these operations are done in the dry state to facilitate handling of the fiber bundles. Similarly, the individual fibers are moved by air currents to minimize breakage. The flowsheet for a typical asbestos mill is quite complicated and may involve as many as four fiber liberation stages, with all of the attendent screening, fiber opening, rescreening, grading, separating, and cleaning operations. Further details would be inappropriate at this time except to say that the geologic conditions under which the deposit was formed establish the inherent fiber length for the asbestos that can be recovered. At best, the mill designed for a given deposit will recover a maximum percentage of the longest fiber present and minimize fiber breakage. At worst, recovery of inherent fiber length will be minimized and the shorter lengths will become the predominant output.

Product Form

The Quebec Screen Test, a procedure developed and supervised by the Quebec Asbestos Mining Association is a generally recognized test for classifying chrysotile asbestos grades. It establishes specifications for 7 Groups of fibers and 2 Groups of so-called asbestos sands and gravels. Individual Groups may contain from 1 to 9 different variations based on fiber length. Group 1 consists of 1 crude, all cross-vein fibers having 3/4 inch staple or longer. Group 2 has three variants starting with 2 crude which has cross-vein fibers having 3/8 inch to 3/4 inch staple. The other two variants also consist of crude materials, predominantly fiber bundles before fluffing or teasing. Group 3 is commonly referred to as textile or spinning fiber, which is the first grade subjected to the Quebec Standard Testing Machine Routine. The machine consists of a nest of three sieve boxes with 1/2 inch, 4 mesh, and 10 mesh screens respectively, and a bottom box serving as a pan. A test is made on 16 ounces of asbestos with the apparatus being mechanically shaken for 600 revolutions at 328 rpm. Individual subgrades in Groups 3, 4, 5, 6, and 7 depend upon the amounts of the original 16 ounce sample retained on each of the 3 screens or eventually passed through into the pan. Group 4 is commonly referred to as shingle or asbestos cement fiber, Group 5 is frequently called paper stock grade, while Groups 6 and 7 are both considered as short fibers.

As an indication of the grade distribution of Canadian asbestos fibers, in a recent year less than half of one percent of all shipments made could qualify as crudes. About 2 percent qualified as Group 3, nearly 27 percent were Group 4, about 15.5 percent were Group 5, some 20.5 percent were Group 6, and 34.5 percent were classified as Group 7 fibers.

Uses and Substitutes for Asbestos

Uses for Asbestos

Asbestos has a multitude of uses based on its fibrous nature, its resistance to heat, and its non-flammability. The longer fibers, down through Group 3, are used for making textile products such as cloth, yarn, tape, and rovings which frequently are used for packing materials. These products in turn become brake band linings, clutch facings, safety clothing, packings, gaskets, and various other appliances. Fiber with extreme low electrical conductivity is used in cable insulations. Groups 4, 5, and 6 fibers, singly or in combination, are used widely for asbestos cement products such as roofing shingles, flat and corrugated siding, and pipe. Group 5 fibers are processed into paper and board that find application in heat and flame resistant coverings for pipe and industrial equipment. The Group 6 and 7 fibers are used for boiler and roofing cements, and as fillers in asphalt and vinyl floor tile, as well as some caulking compounds, undercoatings, and paints.

Asbestos textiles and paper products are essentially all asbestos, but the asbestos content of other products ranges to a low of about 5 percent in certain caulking compounds. In major uses such as cement products, the asbestos content varies from 14 to 20 percent, while in floor tile it may be close to 15 percent.

Substitutes for Asbestos

Glass fibers have been used as substitutes for asbestos in a number of textile products where fire resistance and some thermal insulation are appropriate. However, glass fibers are not a satisfactory substitute in friction materials or asbestos cement types of products. Other inorganic fibrous materials have found limited application for certain uses of asbestos, for the most part limited to applications that are not sensitive to heat. Aluminum sheet and aluminum foil-backed paper have been substituted for asbestos as radiant heat insulators but these products do not have the strength and rigidity characteristics of asbestos and usually are more costly. New forms of heat resistant plastics presently are being tried and used in place of asbestos in electrical insulation. But, there is no satisfactory substitute at present for asbestos for many of its uses.

World Sources of Asbestos

Current Production

As noted earlier three specific areas, Canada, the U.S.S.R., and South Africa have supplied the bulk of the world's asbestos production in the past. Other countries do have some resources and, on occasion, have produced reportable quantities of asbestos as shown in Table 4-1, which presents world asbestos production for 1960, 1965, and 1970. It is obvious that Canada occupies a very significant role in the supply of world asbestos fibers. In spite of more rapid growth on the part of the Soviet sphere countries over the decade, Canada supplied better then 43 percent of world production in 1970. The second largest producer is the U.S.S.R. which in 1970 accounted for 30 percent of world fiber output, while the South African area contributed over 10.5 percent. These three areas accounted for 85 percent of world production in 1970.

These data do not agree precisely with reports of production by industry experts in Canada. The most recent of these, an article by N. W. Hendry, Vice President of the Canadian Johns-Manville Company, Ltd.* suggests that the U.S.S.R. has become the leading producer of asbestos in the world with an estimated 1970 output of 2.2 million tons. Careful study of U.S.S.R. production data by commodity experts at the U.S. Bureau of Mines and in England suggests that Russian statistical reports tend to be inflated. In 1972, for example, a previous estimate of 2.1 million tons for U.S.S.R. production was reduced to 1.2 million tons to represent a better approximation of fiber production--as opposed to total shipments classified as asbestos but believed to contain material that was used as "railway ballast". Although Russian exports of actual fiber undoubtedly are increasing, it is difficult to justify a domestic consumption that could be of the order of 2 million tons of fiber in the Russian economy today.

Reserves

Data relating to specific reserves of proven or probable asbestos ore are difficult to find. Some information has been published by individual companies operating in Canada, and it is readily apparent that operations probably can continue for 15 to 25 years at present rates of exploitation even if new deposits are not discovered. The situation in South Africa and Rhodesia is less clear, predominantly because the currently known deposits are intensely folded and not amenable to extrapolation from observable information. Finally, the U.S.S.R. does not publish reserve estimates.

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* "The Outlook for Asbestos in Canada", The Canadian Mining and Metalurgical Bulletin, 65 (724), 40-4 (August 1972).

TABLE 4-1. WORLD PRODUCTION OF ASBESTOS FIBRE (thousands of short tons)

Country/Region	1960	1965	1970
Canada(1)	1,118	1,388	1,662
United States(2)	45	118	126
South Africa-Rhodesia	310	413	405
Italy	61	79	93
Cyprus	23	17	28
Other Free World	133	126	73
Free World subtotal	1,690	2,141	2,486
U.S.S.R.(3)	660	865	1,150
China(3)	90	140	190
Soviet sphere subtotal	750	1,005	1,340
World Total	2,440	3,146	3,826

Source: U.S. Bureau of Mines

Notes:
(1) Shipments
(2) Sold or used by producers.
(3) Estimated by U.S.B.M.

The latest estimate prepared by the U.S. Bureau of Mines suggests only that apparent reserves exist to supply world requirements beyond the year 2000 at current or considerably enlarged rates of output.

Current and Future Capacity

World capacity to mine and treat asbestos ore is less difinitive than that given for many metals, and output can vary rather drastically with changes in the fiber content of the ore mined.

In Canada in 1970, 13 asbestos mills were in operation with a nominal capacity to treat 88,900 tons per day of asbestos rock. In the course of that year nearly 26.3 million tons of asbestos rock were milled which resulted in the production of 1,666,199 tons of asbestos fiber. The largest individual mill was that of Canadian Johns-Manville at Asbestos, Quebec, with a capacity of 32,000 tons of ore per day. The smallest is that of Headman Mines, Ltd. at Matheson, Ontario, rated at 300 tons of ore per day. Overall recovery of asbestos in Canada in that year was 6.34 percent of the ore treated.

Capacity in other countries is less clearly defined. Information filtering out from the U.S.S.R. indicates that there are 3 primary centers for asbestos production, each of which is, or will be on completion, capable of producing over 600,000 tons of asbestos fiber per year. Whether or not these numbers include the production of extremely short-fibered material used as railway track ballast is uncertain. However, it is likely that they do because an initial production of 2.2 million tons in 1970 was later reduced to 1.2 million tons of asbestos fiber. The industry in South Africa appears to be holding its own with the bulk of output consisting of crocidolite and amosite, varieties not produced in Canada. The situation in Rhodesia is obscure because of the isolation of that country since 1965 when the present government severed relations with the British Commonwealth.

With respect to increases in production, it is known that Canadian mill capacity has been increased by approximately 21,000 tons per day milling rate by the end of 1972, an increase of over 23 percent since 1970. This increase does not automatically assure that the output of asbestos also will increase by an equivalent amount since preliminary production data for 1972 suggest that output was 1,692,000 tons (an increase of only 1.6 percent above the 1970 level). In spite of this underemployment of capacity, other increases may be expected before 1975. Canadian fiber capacity should reach a level of 1.85 million tons per year by 1974, while production in other areas of the Free World (Australia, Mexico, Colombia, and Greece in particular) should be expanding by roughly 200,000 tons per year of fiber. Free World capacity should approach 3 million tons of fiber per year by 1975. Beyond 1975, demand will play a large role in determining new or expansion projects to be undertaken especially if there is any shift in the end use markets that would increase the need for short fiber production.

Canada. Despite the slowdown in asbestos shipments in the last 2 years the industry continues to expand. The new Asbestos Hill mine in Ungava got started in 1972 but full capacity was not reached because of severe weather during the shipping season. It is expected that the planned output of 100,000 tons per year of asbestos fiber (300,000 tons of concentrates will be shipped to Germany for treatment) will be achieved in 1973. Further, Canadian Johns-Manville, after completing a modernization program in 1973, is expecting to boost annual fiber output from 600,000 tons to 700,000 tons per year by 1975. Beyond these two, several prospects are known to have identified reserves that are large enough and may be of adequate grade to support a mill. In Quebec, Abitibi Asbestos Mining Company has a property north of Amos with proven ore reserves of about 100 million tons of a 4 percent fiber content in Groups 4, 5, and 6. A mill to recover approximately 155,000 tons of fiber a year has been planned but recent announcements suggest no possible construction date.

About 20 miles east of Chibougamau, a series of 4 deposits have been located in the vicinity of Lake Roberge. Exploration by McAdam Mining Corporation suggests that the deposits contain more then 165 million tons of asbestos bearing ore from which Grade 4, 5, and 6 fibers can be recovered. Under working option to Rio Tinto Canadian Exploration, Ltd., an underground work program has been initiated on the largest of the 4 deposits from which bulk samples will be taken to the Quebec Government pilot plant for processing tests. Again, no date for a mine or mill has been announced.

In the Eastern Townships, Pathfinder Resources, Ltd. has located a deposit with indications of over 25 million tons of ore, a part of which may grade as high as 4.68 percent fiber content. Nothing definite has yet been planned for this property.

In Ontario, south of Timmins, Allied Mining Corporation owns a property on which 31 million tons of ore, averaging 9 percent fiber content in Grades 5, 6, and 7, has been located by drilling. In conjunction with United Asbestos Corporation, Ltd., Allied had an initial production objective of 100,000 tons of fiber a year. Since 1972 no further word has been received about plans for this property.

Australia. During 1972 production was initiated at a new mine/mill complex by Woodsreef Mines, Ltd. in New South Wales. Annual production of 70,000 tons of fiber per year is expected under sales contracts to Japanese buyers for fibers in the Groups 4, 5, 6, and 7 category. Financing is already lined up to permit an expansion of this operation to 140,000 tons per year during 1974 or 1975.

Mexico. Freeport Minerals Company and Industrias Penoles, S.A. have located a deposit containing over 110 million tons of ore in Oaxaca State. Currently, announced plans suggest a rather modest operation around 1975 that might require a total investment of about $16 million.

Greece. Agreement seems to have been reached between Cerro Corporation of New York and the Hellenic Industrial Development Bank to proceed with the exploitation of an asbestos deposit near Kozani in Northern Greece. Production may be as much as 45,000 tons of fiber by 1974.

Colombia. Asbestos Colombianos, S.A. hopes to have a 28,000 tons per year fiber plant in operation in early 1974 at Cambamento. In addition to satisfying Colombian needs, this operation could provide nearly 14,000 tons per year for export to other South American countries.

U.S.S.R. Details are lacking about future plans for asbestos production in the U.S.S.R. However, the steady increase in exports to both Eastern and Western Europe suggests that the expansions carried out in the late 1960's are being continued and that additional reserves to support such expansions are available. A Canadian Asbestos Industry spokesman has predicted that the U.S.S.R. may export as much as 500,000 tons of fiber in the Grades 4, 5, and 6 categories by 1975.

Demand for Asbestos

Current Consumption

Consumption data for asbestos are reported on a systematic basis only in the United States. Even here, a total is reported for apparent consumption but no attempt is made to relate this to the thousands of end uses except for the qualitative statement that asbestos cement building products constitute the principal application. A recent European publication* in surveying the world asbestos scene estimated that 70 percent of the total world fiber was consumed annually in asbestos cement building products. This was as far as they were willing to go in distributing asbestos fiber consumption to end uses.

In the United States the U.S. Bureau of Mines in 1970 suggested that in that year consumption of asbestos shorts in asphalt tile was second only to asbestos cement products as a consumer of fiber. Some 10 percent of a 728,000 tons consumption total was believed to be used in this category. It becomes obvious from an examination of import data that asbestos textile and other spinning fiber products will in total account for less than 5 percent of total consumption. If the 70 percent figure for asbestos cement products is accepted, this leaves over 100,000 tons of medium to short fibers going into brake lining, clutch facings, asbestos paper, asbestos insulations, and as a filler for plastics, putties, caulking compounds, and paint.

From a variety of sources and using several simplifying assumptions, Table 4-2 has been assembled to approximate Free World consumption of asbestos fiber in 1960, 1965, and 1970. The basic information came primarily from annual yearbooks of the U.S. Bureau of Mines (USBM) or the Canadian Department of Energy, Mines, and Resources (CDEMR). N. W. Hendry, Vice President of Canadian Johns-Manville,

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* "Asbestos", Industrial Minerals, (28), 9-29 (January 1970).

TABLE 4-2. FREE WORLD CONSUMPTION OF ASBESTOS FIBER (thousands of short tons)

Country/Region	1960	1965(1)	1970(1)
Canada(2)	50	55	51
United States(3)	709	794	728
Rest of Free World(4)	1,041	1,555	2,133
Free World Total	1,800	2,404	2,912

Source: Compiled by BCL.

Notes: (1) Free World total and Canada reported by N. W. Hendry, Canadian Johns-Manville.
(2) For 1960, Department of Energy, Mines and Resources; industry shipments less exports.
(3) Apparent consumption reported by U.S. Bureau of Mines.
(4) By difference.

provided part of the interpretation in his review article* on production, trade, and consumption. In 1965 and 1970, he equates production with consumption on a world basis that excludes only U.S.S.R. and mainland China. Although this tends to ignore exports from the U.S.S.R. as a part of world consumption this omission is counterbalanced to some degree by production of small quantities in the Eastern European countries. For 1960, Free World production according to USBM, has been increased by an estimated 110,000 tons of exports from the U.S.S.R. into Western European countries in order to arrive at Free World consumption. Generally, the principal objection to equating production with consumption is that this technique ignores possible changes in producers' or consumers' inventories of the commodity under consideration. Although Table 4-2 may not be statistically correct, it is believed to represent order-of-magnitude accuracy and well illustrates the fact that consumption of asbestos fiber in countries other than North America is growing at a much more rapid rate. Actually, in the 1960 decade the peak year for consumption in the United States was 1968 when over 817,000 tons of asbestos was reported as apparent consumption. For that 8 year period the average annual growth rate in the U.S. was about 1.8 percent. Canadian domestic consumption appears to have done not much better in the 1960 decade whereas the rest of the Free World, led by Western Europe and Japan, had an average annual growth rate of over 7 percent for the decade.

As another aspect of where asbestos is consumed, Table 4-3 presents some of the major components of international trade in asbestos fiber in 1970. Canada, the U.S.S.R., and South Africa are the big exporters followed by Italy and the United States. Nearly 2.4 million tons of asbestos fiber was shipped from these major producing countries. This presentation avoids the complication of re-export trade among the Western European countries. The major consuming areas as judged by imports are Western Europe, the United States, Japan, followed by Latin America, Australia, India, and Canada. The latter appears as an importer because of the receipt of small quantities of crocidolite and amosite from South Africa. The difference between imports at 2.1 million and exports at 2.4 million may be accounted for by consumption in Eastern Europe, the rest of Africa, and in the other countries of Asia, excluding Mainland China which is assumed to be self sufficient.

Future Demand

From the foregoing discussion it appears that future demand is likely to be determined by what happens in the asbestos cement building materials sector. Although these products have excellent physical properties and are relatively inexpensive in comparison with other building materials, their growth in North America appears to have been blunted in recent years by the incursion of improved thermal plastic materials for piping applications particularly in the drain, waste, and vent field. It seems reasonably assured that this competition will continue in North America and hold down the future growth of asbestos consumption. In other applications, for example, brake linings and clutch facings, the Environmental Protection Agency in the United States has started to do some test work with respect to the level of airborne asbestos in metropolitan areas resulting from normal wear on these asbestos-containing surfaces. No decision with regard

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*"The Outlook for Asbestos in Canada", The Canadian Mining and Metallurgical Bulletin, Volume 65, No. 724, pp. 40 - 44 (August 1972).

TABLE 4-3. MAJOR COMPONENTS OF TRADE IN ASBESTOS FIBER
(thousands of short tons)

Country/Region	1970
Exports[1]	
Canada	1,562
U.S.S.R.	425
South Africa	320
Italy	53
U.S.A.	38
Total of above	2,398
Imports[2]	
Western Europe	925
U.S.A.	649
Japan	329
Latin America	98
Australia	76
India	44
Canada	6
Total of above	2,127

Source: Statistical Summary of the Mineral Industry, London.

Note: (1) Major producing countries only.
(2) May include some intra-Western European re-export trade.

to the severity of this potential hazard has been reached at this time. On the other hand, there are essentially no suitable substitutes now known for asbestos in these applications. Without question, substitutes will be sought and may include such things as a recently reported development in England of an alkali resistant glass fiber. The point is that aside from the construction field, particularly asbestos cement products, a reasonable growth rate for asbestos fiber consumption appears possible even in the United States and Canada. Overall, increases in the North American market for asbestos fiber are unlikely to exceed 2 percent per year through 1980 or, for that matter, for the rest of this century. Beyond 1980, one of the determining factors may be the development of a suitable substitute for long fiber asbestos in textile and woven products applications. This may be required by environmental considerations. However, a firm base in asbestos cement products, especially large diameter pipe, should maintain the minimal rate of growth assigned to the end of the century.

There is evidence in the last several years that the substitution of plastic pipe for asbestos cement pipe in Western Europe and Japan has taken place in diameters up to about 8 inches. Accordingly, the rapid rates of growth experienced over the past 20 years are expected to decline radically in these areas. The less developed countries will still be increasing at fairly rapid rates because of the economy and ease of fabrication of asbestos cement products and the unavailability of the sophisticated plastics. Overall, the balance of the Free World probably will expand at an average annual growth rate of about 4 percent through 1980 declining toward a 3.0 percent rate by the year 2000.

As shown in Table 4-4, Free World demand for asbestos fiber is expected to rise from 2.9 million tons in 1970 to 4.1 million in 1980, 5.6 million in 1990, and 7.7 million in the year 2000.

Supply/Demand Relationships

Whereas, asbestos supply and demand appeared to be roughly in balance in 1970, the expansions that have occurred in Canada, South Africa, and Australia suggest that production capability presently outpaces demand. Allowing for an additional 300,000 tons per year of production coming onstream before 1975, the current thinking is that demand may not catch up until 1975 or a bit later. Still, beyond 1976 new capacity will be required if demand follows the projections. Figure 4-1 includes projected annual Free World consumption rates at 10 year intervals and the curve of cumulative withdrawals from reserves occasioned by yearly demand through the year 2000. A representation of presently known reserves is included, which only can indicate that, at projected rates of increase, known resources of asbestos fiber will not be exhausted before the year 2000. It has been estimated by industry experts that if the rate of growth for future consumption were to average 6 percent annually on a world basis, all of the currently known resources of asbestos would be exhausted before the year 2000. This suggests that as of 1970, known and inferred reserves were more than 200 million tons and less than 300 million tons. However, it is anticipated that exploration, if only for other minerals, is also likely to turn up occasional discoveries of asbestos before the end of the century. Moreover, the emphasis on asbestos exploration is likely to come in countries other than Canada and the Southern African region as various political entities attempt to become self-sufficient in a number of mineral products.

TABLE 4-4. PROJECTED FREE WORLD DEMAND FOR ASBESTOS FIBER (thousands of short tons)

Country/Region	1970	1980	1990	2000
Canada	51	62	75	90
United States	728	888	1,075	1,310
Rest of Free World	2,133	3,150	4,450	6,300
Free World Total	2,912	4,100	5,600	7,700

Source: Estimated by BCL.

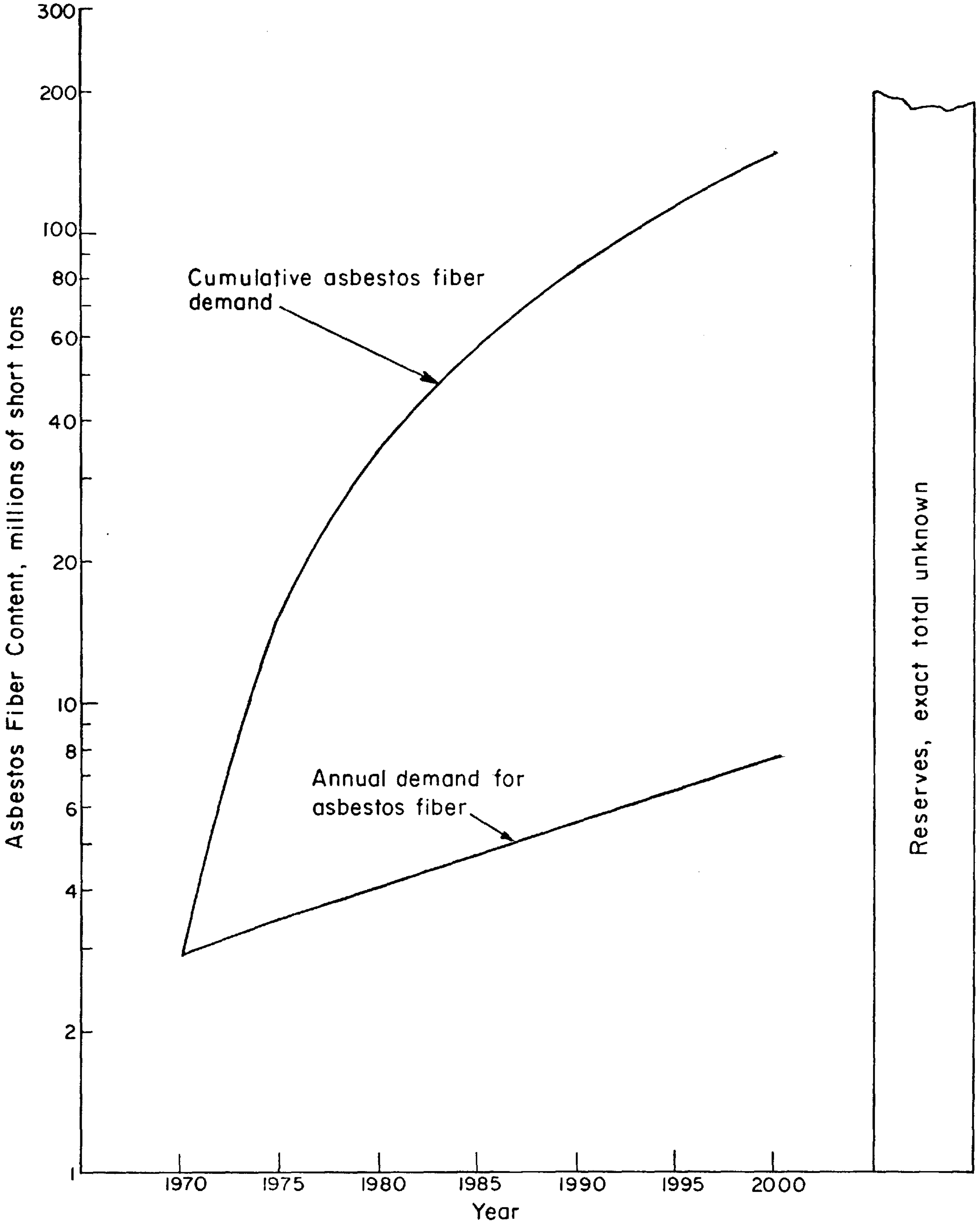

FIGURE 4-1. ANNUAL AND CUMULATIVE FREE WORLD DEMAND FOR ASBESTOS FIBER AND RESERVES

Probable Price Trends

Just as there are many grades of asbestos, so there are many prices for asbestos. Although grading and pricing vary from country to country, Quebec chrysotile tends to set the tone for world markets. For 1972, Quebec asbestos prices ranged from $1,615 per ton for crude #1 to $52 a ton for the shortest fibers in Group 7. Since the bulk of world production appears to fall in the categories that would correspond to Quebec Group 4 through Group 6, it is appropriate to note that these prices range from $120 for the 6D fiber to a high of $383 per ton for the 4A class fiber. The preliminary report for Canadian shipments in 1972 indicates that the average price received by producers was nearly $130 per ton, an increase of $5 per ton over the figure received in 1970. For the most part, recent increases have been justified on the basis of increasing costs of mining and milling, primarily for labor and supplies. Coming in the face of surplus production capacity, 1971 price rises suggest that they will be continued in the future as labor contracts are renegotiated. Productivity in the Canadian mines and mills is at fairly high rates presently and significant increases may be difficult to achieve. These factors would indicate a continuation of upward price pressures, especially in the Group 4 through Group 6 categories where increases have been highest recently. On the other hand, a slackening of these markets, that is the markets for asbestos cement grades, could tend to maintain current price levels in 1972 constant dollars through 1975, at least. Any adjustments are more likely to be on the longer fiber grades like Groups 1 through 3. Beyond 1975 and into the 1980 period, the capital expenditures necessary to bring in new mining and milling operations will help to establish the trend for prices. It is likely that on the basis of the current $130 a ton average for Canadian production, after 1980 there will be strong pressures to raise prices unless environmental developments and substitutional possibilities seriously restrict growth. Uncertainties about these factors precludes a price forecast beyond 1980.

Need for a New Producer in the James Bay Region

Without question, Quebec already has proven reserves capable of sustaining many years of operation at the existing mines and mills. Additionally, there are at least 3 undeveloped deposits now known and, undoubtedly, more will be discovered both in the Ungava District as well as in the James Bay region. The current producers in Canada have either captive or well established market outlets for their output and will be in the best position to retain those markets in the future. Competition will come from increasing U.S.S.R. exports and local production brought into being by other than economic considerations. Canadian industry experts already are predicting difficult marketing times ahead for the present Canadian producers. On this basis, it is difficult to foresee how a new producer--without marketing experience or contacts--could possibly justify an operation much before 1990 when the world situation is likely to favor new production anywhere. At that time, if arrangements cannot be made for marketing through one of the established producers, it will be necessary to search for overseas markets in order to assure the annual revenues that will be required to repay a fairly high cost venture.

In fact, it is rather obvious that any new venture in Canada before 1975 is a physical impossibility from the viewpoint of planning and construction that would be required to initiate production. Also, the ease with which current Canadian producers could expand beyond the planned 100,000 tons per year addition, should be considered in any feasibility study by a potential new producer. Thus, the timing for any new production in Canada or the James Bay region seems to depend on the marketing arrangements that can be made.

POTENTIAL MINE/MILL PROJECT FOR THE JAMES BAY REGION

Introduction

The following example of an asbestos mining and milling operation has been developed around a known property east of Chibougamau at Lake Roberge. However, as was the case with the hypothetical projects, the purpose of this analysis is not to encourage the development of this particular property, but rather to provide an illustration of the cost and other factors involved in the development of this type of mine within the James Bay region. It is considered that the Lake Roberge asbestos property might not come into production until after 1990 because of the modest growth projected for the markets for asbestos and the known reserves contained in existing operations of major producers elsewhere in Canada.

Project Summary

The potential project is concerned with an operation that may be minimally economic for the grade of ore as presently known and the location. It is designed to develop one of four deposits found in the claim area, containing 73 million tons of fiber-bearing rock from which enough ore would be mined to support an 8,000 tons per day mill with an overall employment of about 550 people. Project costs, exclusive of road building requirements other then in the open pit mine and the immediate vicinity, are estimated as:

Mine Development (preproduction).	$10 million
Mine Equipment and Facilities	25 million
Mill Construction and Equipment	35 million
Shipping and Warehouse Facilities	5 million
Total	$75 million.

The value reported for this particular deposit is $6 per ton of ore based on the recovery of Group 4 and Group 6 fibers. At an average f.o.b. price of $120 per ton for Group 6 fibers and of $250 per ton for Group 4 fibers, the 16 year output from this one mine should be approximately $275 million. No townsite for the mine/mill workers would be required because Chibougamau is less than 15 miles from the site of this potential project.

The Deposits

According to information compiled by S.G.S., this deposit located about latitude 49° 54' north and longitude 73° 58' west was discovered in 1955 by prospectors. Intermittent activity was carried on between 1956 and 1958 with McAdam Mining acquiring exploration rights to the property in 1961. Subsequently, investigations culminated in 1967 in the sinking of a shaft and lateral access tunnels on one of four mineralized zones for the procurement of a 500 tons sample which was tested in the Quebec Government pilot plant. The four zones are estimated to contain 165 million tons of ore averaging about 4 percent fiber. The test work at the pilot plant indicates that elimination of a friable fraction after crushing results in a "beneficiated" ore with 5 percent fiber content, the fibers falling primarily in Group 4 and Group 6. This beneficiated ore has a calculated value of $6 per ton.

The largest of the four mineralized zones, identified as C-D Zone, is located partially under one arm of Lake Roberge with a length of about 4,200 feet and a width of about 830 feet. Depth of the zone is not indicated but 73 million tons of fibrated material is reported from which 49 million tons of beneficiated ore could be recovered for treatment in the mill.

The mineralized zones lie on the south flank of a regional syncline plunging weakly from the east to the west. It is located a little over a mile west of the Grenville Front which traverses the region in an NNE direction. It occurs in serpentine rocks that are altered dunite, perhaps 2.5 billion years old, out of the Kenoran metallogenic epoch. Serpentinization of the dunite appears to be complete with inclusions of carbonate and talc gangue.

The Mine

Mine Size and Materials Handling Requirements

In the absence of specific information on the cross sectional plan of the deposit at specific intervals, it is assumed that the mine could be opened for initial production by the removal of approximately 0.5 million cubic yards of overburden. Primarily, this would be used to build a dike across the arm of Lake Roberge behind which mining could proceed. Hypothetically, waste rock from the mine could be used behind the dike as fill materials leading to the eventual exposure of the total deposit under the lake by successive movements of the dike across the lake, with water removal by pumping.

Based on a performance slightly better than Canadian industry average, for each ton of ore milled, a little over 2.2 tons of waste rock will have to be blasted and removed. To support the 8,000 ton-a-day mill, which works 7 days a week, the mine operating 6 days a week on 3 shifts would have the following materials handling requirements:

	Approximate Capacity in Tons		
Material	Daily	Weekly	Annual
Ore (to mill)	9,300	56,000	2.8 million
Ore (to crusher)	14,000	84,000	4.2 million
Total Rock Mined	30,000	180,000	9.0 million
Mine Waste Rock	20,700	124,000	6.2 million.

Mine Equipment Needs

A generalized mining plan would assign one shovel to removal of overburden and three to the loading of rock and ore which has been blasted from benches within the mine area itself. Self-propelled drills prepare the benches for loading of explosives. The shovels load dump trucks which transport waste rock to the dike for distribution by a bulldozer and ore to the primary crusher station. Both are assumed to be located within 2 miles of the loading site. A flat-bed truck, pick-up trucks, and a tank truck supply the rotary drills and diesel fueled vehicles in the mine and dike area. It is assumed that the shovels are electrically powered and a panel-truck supplies explosives for the blaster and his helper.

At the crusher station, ore from the mine is dumped into a feed hopper for a 54"x74" gyratory crusher set to pass minus 6 inch chunks of rock. Crushed ore passes over a grizzly having 1-1/2" openings to feed bins for two short head cone crushers. These secondary crushers are set at 1-1/2" discharge opening and the product passes over a vibrating screen which rejects material smaller than 3/4". The undersize from both screenings is considered to be barren rock and is loaded to dump trucks for disposal at the dike site. The plus 3/4" material is conveyed to crude ore storage at the mill site.

Table 4-5 presents a summary of the major equipment items believed necessary to operate this size of mine on a three shift, 6 days per week basis.

Mine Manpower List

The schedule calls for 6 day-a-week operation around the clock which will necessitate a minimum of three complete crews, except for the explosive man and his helper. The critical point will be for the mine to supply a minimum of 56,000 tons of plus 3/4" ore to the mill weekly and whether or not a 6th day for each of the three crews is necessary will depend on productivity throughout the first 5 days of the week. The manning requirements, based on three crews, are given in Table 4-6 with a rough breakdown of skill levels.

Operating Supplies

The list of mine supplies includes diesel fuel, gasoline, explosives, and electricity. The diesel fuel is used for rotary drills, dump trucks, the front-end loaders, and bulldozers and amounts to approximately 13,000 gallons per day at the scheduled rate of production. Gasoline requirements for the pickup, flat-bed, and tank trucks is rather nominal at 300 gallons per day. In the absence of any specific data about the rock to be fractured, it is assumed that approximately 2/3 of a pound of AN/FO explosive will be required per ton of rock removed. This adds up to about 20,000 pounds per day or 10 tons. Electricity for the four shovels and the crusher installation, with associated conveyors and screens, should amount to no more than 6,000 Kilowatt hours per day.

TABLE 4-5. ASBESTOS MINE EQUIPMENT LIST
(3 shift, 6 days per week)

Item	Number
Rotary drills, self powered	3
Electric shovels, 10 cubic yard buckets	4
Front-end loaders	2
Bulldozers	4
Dump trucks, 100 ton capacity	14
Flat-bed trucks, 3 ton capacity	1
Pick-up trucks, 3/4 ton capacity	2
Panel truck, 1 ton capacity	1
Tank truck, 1,000 gallon capacity	1
Gyratory crusher, 54" x 74"	1
Cone crushers, 2' short head	2

Source: Compiled by BCL.

TABLE 4-6. ASBESTOS MINE MANPOWER LIST
(3 shift, 6 days per week)

Designation	Number
Skilled equipment operators	33
Semiskilled equipment operators	84
Skilled maintenance personnel	7
Semiskilled maintenance personnel	20
Unskilled labor	40
Total Labor	184
Management	11
Office and engineering staff	5
Total Office	16
Total	200

Source: Estimated by BCL.

Mining Effluents

As indicated earlier, waste rock from the mine as well as from the crusher station will be used initially to reinforce and build up the dike holding back Lake Roberge from the mine operation. At some point in the mine life, it will become more appropriate to dispose of the waste rock and screenings in a tailings pile other than in Lake Roberge. Meanwhile, overburden removal continues to move the dike to expose fresh ore for mining. The waste rock apparently contains little, if any, minerals that on exposure to air or rain are likely to produce acid drainage from a waste disposal area.

Open pit mining, of course, does produce dust from explosives used to shatter the rock, as well as from material transfer operations and mining roads. When these problems become serious, transfer points and the road can, of course, be wetted. At the crusher station, it would be appropriate to install a dust collection system consisting of exhaust fans and cyclone collectors with the dust eventually being hauled to the waste disposal area.

The Mill

The mill has been sized to handle 2.8 million tons of asbestos-bearing rock per year for 50 weeks on a three-shift operation 7 days per week. This results in a nominal operating capacity of 8,000 tons per day of feed to the mill. Ore treatment begins with a drying operation, and the final product will be predominantly Group 4 and Group 6 fibers, with some Number Seven floats as well as sand and gravel wastes.

Flowsheet

A composite flowsheet for a typical Canadian asbestos mill is presented in Figure 4-2. After drying, the ore is sorted on classification screens whose openings range from 3/8 of an inch to 1-1/4". From two to four screen sizes will be provided depending on the liberation characteristics of the ore and the range of fiber grades desired. Assuming three screens, this classification operation provides long-, medium-,and short-fiber concentrates. Each concentrate is subjected to impact milling followed by removal of liberated fibers by aspiration and remilling of material that passes over a 20-mesh screen. The aspirated fibers are collected by cyclones and separated into appropriate grade designations by trommels. Dust passing through the 20-mesh screening is collected as waste product for disposal at the waste-rock pile.

Short fiber concentrates may be subjected to two successive liberation stages before the residual rock is small enough to pass the 20-mesh opening in the screens. Medium fiber concentrates may go through three successive milling stages before essentially all of the fiber has been liberated. The long-fiber concentrates may be treated four or more times before final discard of waste or

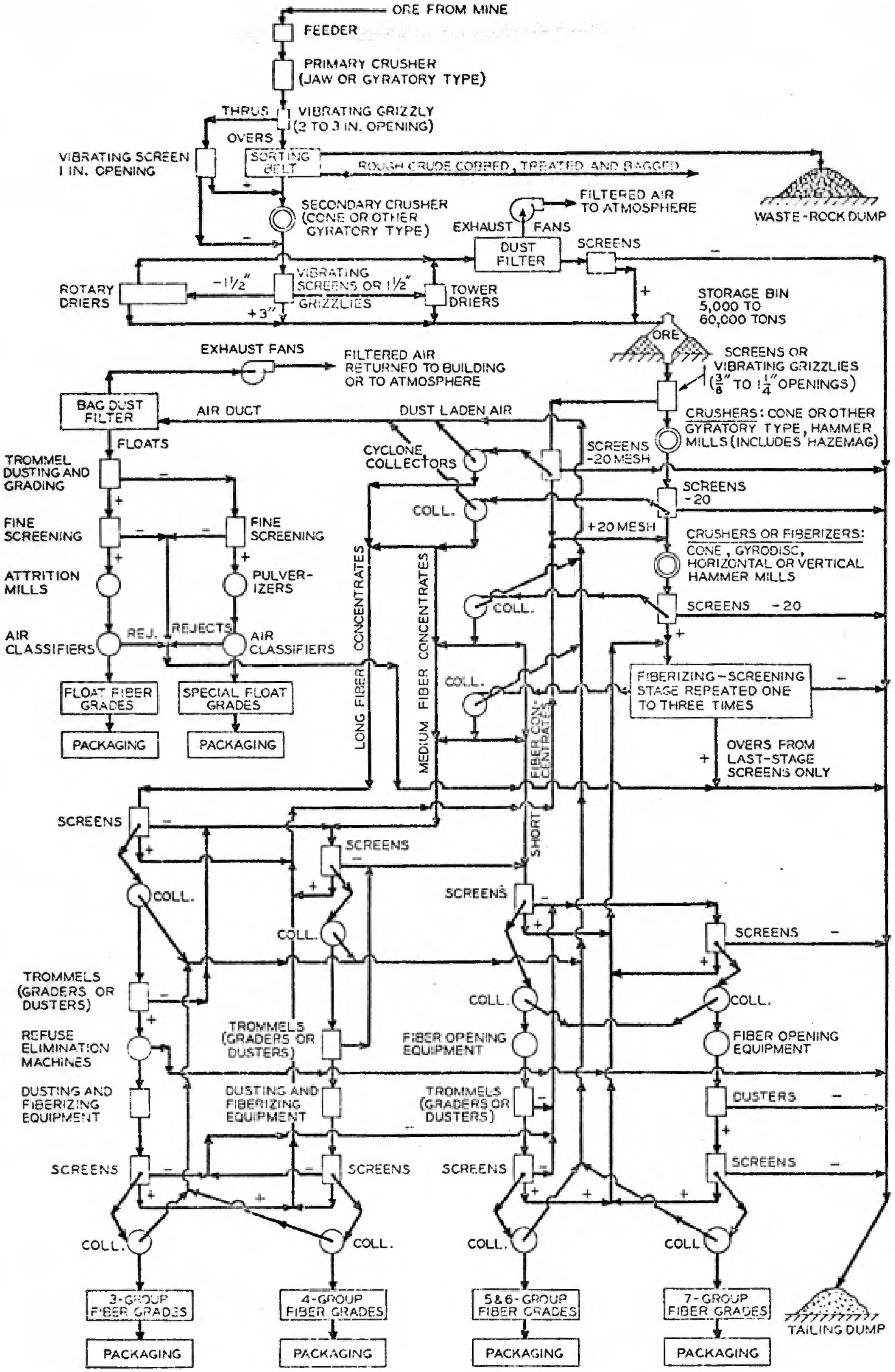

FIGURE 4-2. TYPICAL CANADIAN ASBESTOS MILL FLOWSHEET

barren rock. Each screening stage following a milling operation is designed to remove waste dust while passing fiber bearing rocks on to the next milling stage and capturing the liberated fiber near the end of the screen with air currents that carry the fibers to cyclone separators for recovery. Exhaust from the cyclone usually contains some extremely short fibers known as floats which are captured in a bag house for further grading and packing for shipment. Fibers collected by the cyclone separators are graded for length, stored by grade, and finally packaged under pressure in 100-pound bags for shipment.

Mill Equipment Needs

Mill building design and specific equipment requirements are determinable only after extensive testing of bulk samples of the ore to determine its liberation characteristics. However, it should be noted that the probable installation will be a sizeable building with multiple sets of equipment for each operation. One of the critical design points appears to be the air system for removal and collection of the fibers as they are liberated. Recent installations in Eastern Canada have included steel framed mill structures with poured cement floors for nine major operating levels. Enclosures for rotary or tower dryers, warehouse and shipping dock, change house for mill personnel, and office and laboratory space are also provided. Table 4-7 lists the identification of a number of the major pieces of equipment that are likely to be needed for the mill. Sizes of equipment and the number of units to be installed can be cited only after a complete processing sequence has been designed on the basis of bulk sample testing.

Mine Manpower List

On a schedule of three-shift operation for 7 days per week, the mill will require four complete crews of operators to stay within a 40-hour work week. The types of activity that are going on and the multiplicity of equipment installations throughout the plant with little opportunity for automation dictate that the labor requirements for an asbestos mill will be high. Table 4-8 presents an estimate of the manpower needed to run this 8,000 tons-per-day mill.

Mill Operating Supplies

Aside from a little distillate fuel oil to operate the dryers and sundry mechanical maintenance parts, the principal operating supplies will consist of shipping containers and electricity to operate the motors throughout the installation.

Shipping containers, whose cost is charged to the customers, have averaged about $5 per ton of product in recent years in Eastern Canadian mills. Predominantly, they are laminated paper bags specifically designed for pressure packaging and holding 100 pounds of asbestos fiber. With palletizing and shrink-wrap coverings, a package that is durable and easy to handle with mobile equipment is provided.

TABLE 4-7. ASBESTOS MILL EQUIPMENT LIST(1)
(3 shift, 7 days per week)

Item	Number
Crude ore dryers	
Crushers, cone or gyratory	
Vibrating screens	
Cyclone collectors	
Fiberizers	
Vibrating screens	
Cyclone collectors	
Fiberizers	
Vibrating screens	
Cyclone collectors	
Grading Trommels	
Cyclone collectors	
Air system fans	
Bag house	
Pressure packaging equipment	

Source: Compiled by BCL.

Note: (1) Only a generalized listing can be given until a complete processing sequence has been determined by bulk sample testing.

TABLE 4-8. ASBESTOS MILL MANPOWER LIST
(3 shift, 7 days per week)

Designation	Number
Skilled equipment operators	48
Semiskilled equipment operators	180
Skilled maintenance personnel	10
Semiskilled maintenance personnel	32
Unskilled labor	60
Total Labor	330
Management	11
Office and laboratory staff	9
Total Office	20
Total	350

Source: Estimated by BCL.

Pending the detailed design of the mill, it is estimated that the electrical requirements will be about 17,000 Kilowatt hours per day.

Mill Effluents

The principal effluent from this hypothetical mill will be the waste rock, all of which has passed through a 20-mesh screen. Since recovery of fiber represents less than 4 percent of the mill feed on a dry basis, most of the input leaves as a relatively fine powder (the sand and gravel). Actually, 7,700 tons of waste rock must be disposed of to the waste pit daily. In order to prevent windage losses in transit, hopper type trucks are recommended for the transport of this waste rock to the disposal site.

The suppression of airborne dust is one of the key design criteria for modern asbestos mills. This results from the possible health hazard that such dust presents to workers in the factory itself as well as any other inhabitants of the immediate area. The specific details of dust collection systems are beyond the scope of this pre-engineering analysis, but the problem needs to be recognized and provided for in the ultimate mill design.

Mill Output and Value

To approximate the $6-per-ton value reported in connection with this deposit, it is estimated that the output of the mill will consist of 100 tons per day of Group 4 fibers at an average value of $250 per ton and 200 tons per day of Group 6 fibers at a value of $120 per ton. Output of the mill, thus, would be as follows:

Daily	Yearly	Life of Mine
300 tons	105,000 tons	1,740,000 tons.

This is based on the recovery of approximately 75 percent of the fiber content of the beneficiated ore brought into the mill.

Based on the above values, the revenues to the mill for the asbestos alone will be as follows:

Daily	Yearly	Life of Mine
$ 49,000	$17,150,000	$274,400,000.

Capital Requirements

Mine Costs

The investigation of a potential asbestos deposit and the subsequent evaluation of drill cores and bulk samples is an expensive undertaking. Not only must the deposit be outlined fully by diamond drilling, but a bulk sample needs to be extracted from underground workings that will provide a representative material on which to conduct milling tests. These tests are necessary in order to determine grade of fiber that can be recovered and devise a method for treating the material. Beyond this, pre-production costs will also involve the stripping of overburden and in this instance, the initial construction of a dike across the end of the lake. At current prices, it is estimated that mine development costs might be about $10 million.

Equipment for use within the mine and buildings for housing the crushers, office, and repair shops may be expected to add an additional $25 million in order to initiate production. Neither of the above figures includes any allowance for a townsite or for any road building other than those necessary for the mine operation and disposal pits.

Mill Costs

The published literature on the construction cost of asbestos mills is rather meager. Extrapolation of data for mills in the 2,000 to 5,000 tons-per-day class built within the past 10 years suggests that a first approximation would reserve $35 million in current dollars for the 8,000 tons-per-day mill proposed.

At the same time, it is recognized that the output of this mill will have to compete on a price basis with mills located primarily in the Eastern Townships of Quebec. In order for this to be feasible, it is predicated that a part of the project costs would involve the construction of a sizeable warehouse at a tidewater port and the construction of a railroad spur from Chibougamau to the mill. An allowance of $5 million has been provided for the warehousing and shipping facilities.

Project Total

The capital requirements for this potential project are summarized as follows:

Mine Development Cost	$10 million
Mine Equipment and Building Construction	25 million
Mill Construction and Equipment	35 million
Warehousing and Shipment	5 million
Total	$75 million.

Evaluation of Potential Worth

In order to provide some measure of the potential feasibility of putting the Lake Roberge asbestos property into production, a cash flow analysis was performed according to the characteristics and assumptions specified above. The approximate life of the first pit is at least 15 years and, allowing 2 years for construction, a total project life of 17 years is proposed for the financial considerations. From an operational viewpoint, the mill could, in fact, continue to process ore from three other pits after exhaustion of the C-D Zone chosen. However, for the purposes of these calculations, it appears more realistic to amortize the mill over the 17-year life of the project.

Data Inputs and Assumptions

Capital Outlays. There are four capital outlays associated with this project as indicated above, totalling $75 million. Seventy-five percent of total capital requirements are financed by debt over a 17-year period at 8 percent interest. Principal payback begins in the first years of production (assumed to be 1997 from the marketing report). The depreciable life of all plant and equipment is 17 years with no salvage value.

Operating Expenses. There are seven annual operating expenses associated with the project.

(1) Cost of shipping containers are estimated at $25,000 per year.
(2) Operating and maintenance costs are estimated at $2,100,000 per year.
(3) Wages and salaries are estimated at $4,000,000 per year.
(4) Fuel and electricity costs are estimated at $785,000 per year.
(5) Transport costs are estimated at $3.50 per ton to tidewater or about $400,000 per year. This is essentially a price equalization to f.o.b. Thetford mine prices.
(6 & 7) Annual interest and depreciation are calculated internally according to the specifications noted under capital outlays.

Revenues. Revenue streams are generated from sales of Group 4 and Group 6 fibers. Total annual production of both fiber grades is expected to be 105,000 tons comprised of 35,000 tons of Group 4 and 70,000 tons of Group 6. At prices of $250 and $120 per ton respectively, total annual revenues would be $17.15 million or about $163 per ton of combined fiber grades. To take into account the effects of variability in possible fiber grade combinations, annual revenues are sampled from the following distribution:

Annual Revenues (millions of $)	Median Sale Price ($ per ton)	Probability* (percent)
16.2 - 16.6	157.00	20
16.7 - 17.1	161.00	30
17.2 - 17.5	165.00	40
17.6 - 18.0	169.00	10

Output

For the above inputs, 50 iterations of the cash flow model were performed. Results indicated that rates of return vary between minus 4 and plus 4 percent. The range is presented as a cumulative probability distribution in Figure 4-3, which suggests that there is only a 24-percent chance of earning a yield of 2 percent or better and no chance of earning more than a 4 percent yield. Sample output iterations are included as Tables 4-9 to 4-11. These results are not encouraging at the present time as it would appear that current prices would have to experience "real" increases of 15 to 20 percent to make the venture more financially promising. Such "real" price increases would have to result from the exhaustion of known resources which are exploitable at current prices.

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* See footnote, page 1-54.

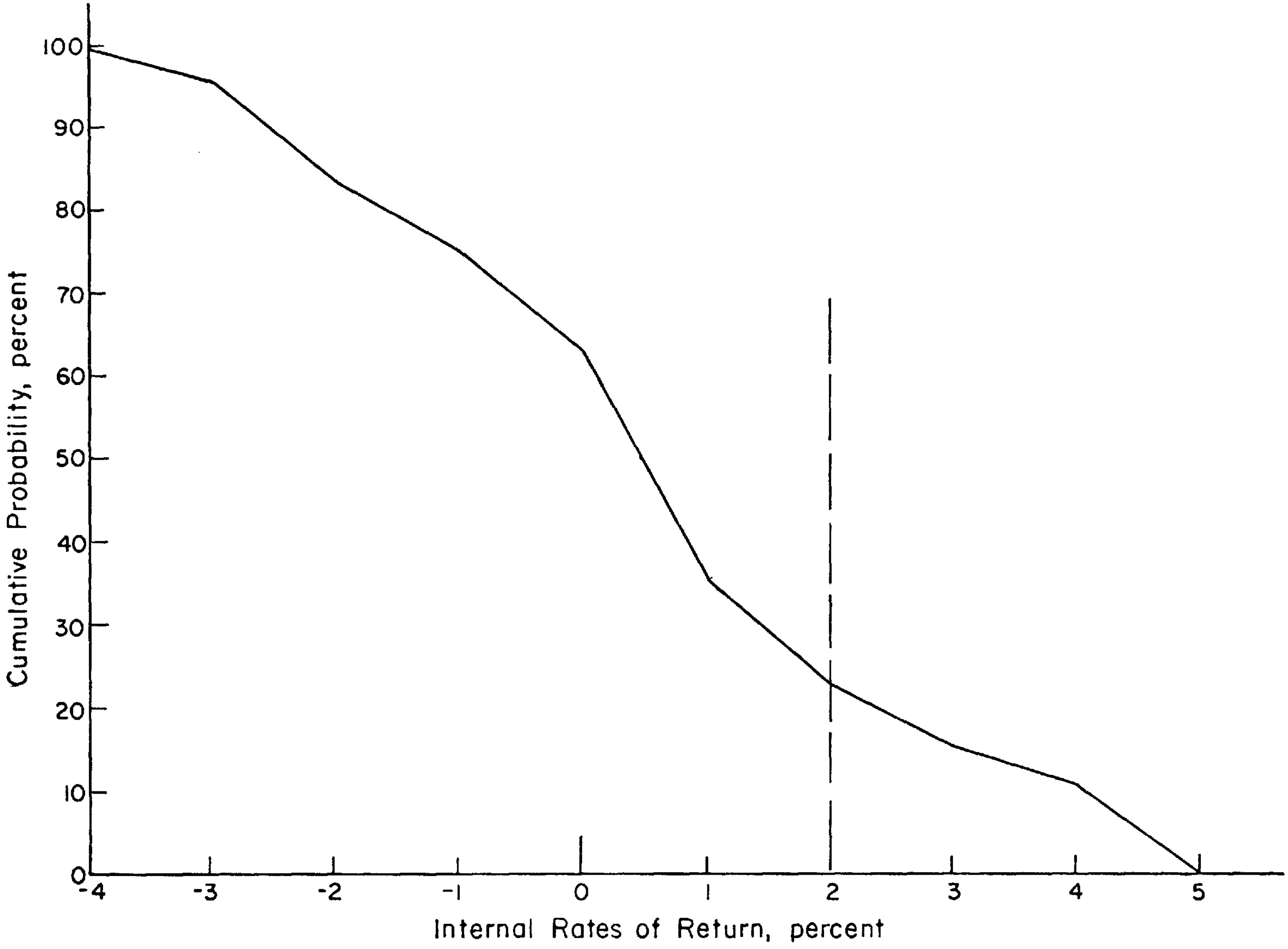

FIGURE 4-3. ASBESTOS PROJECT INTERNAL RATES OF RETURN VERSUS PROBABILITY

TABLE 4-9. LAKE ROBERGE ASBESTOS MINE / MILL COMPLEX
ITERATION NUMBER 11

YEAR	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004	2005	2006	2007	2008	2009	2010	2011
CAPITAL OUTLAYS																	
PREPRODUCTION COSTS																	
-----INTERNAL FUNDS	2.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	0.0	0.3	0.3	0.3	0.3	0.4	0.4	0.4	0.5	0.5	0.6	0.6	0.6	0.7	0.8	0.8
MINE EQUIP.																	
-----INTERNAL FUNDS	6.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	0.0	0.7	0.7	0.8	0.9	0.9	1.0	1.1	1.2	1.3	1.4	1.5	1.6	1.7	1.9	2.0
MILL EQUIP.																	
-----INTERNAL FUNDS	0.0	8.8	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	0.0	1.0	1.0	1.1	1.2	1.3	1.4	1.5	1.7	1.8	1.9	2.1	2.3	2.4	2.6	2.8
TRANSPORT INFRASTR.																	
-----INTERNAL FUNDS	0.0	1.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	0.0	0.1	0.1	0.2	0.2	0.2	0.2	0.2	0.2	0.3	0.3	0.3	0.3	0.3	0.4	0.4
TOTALS	8.8	10.0	2.1	2.2	2.4	2.6	2.8	3.0	3.3	3.6	3.8	4.1	4.5	4.8	5.2	5.6	6.1
ANNUAL EXPENSES																	
CONTAINERS	-0.0	-0.0	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5
OPER.,MAINTENANCE	-0.0	-0.0	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1
WAGES, SALARIES	-0.0	-0.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0
FUEL + ELECTRICITY	-0.0	-0.0	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8
TRANSPORT COSTS	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4
INTEREST ON LOANS	2.1	4.5	4.5	4.3	4.2	4.0	3.8	3.5	3.3	3.0	2.7	2.4	2.1	1.7	1.4	0.9	0.5
TOTAL DEPRECIATION	2.1	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.5
TOTALS	4.6	9.5	16.9	16.7	16.5	16.3	16.1	15.9	15.6	15.4	15.1	14.8	14.5	14.1	13.7	13.3	12.8
ANNUAL REVENUES																	
REVENUE FROM SALES	0.0	0.0	17.0	17.0	17.0	17.0	17.0	17.0	17.0	17.0	17.0	17.0	17.0	17.0	17.0	17.0	17.0
TOTALS	0.0	0.0	17.0	17.0	17.0	17.0	17.0	17.0	17.0	17.0	17.0	17.0	17.0	17.0	17.0	17.0	17.0
TAXABLE INCOME	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.9	1.4	1.6	1.9	2.2	2.6	2.9	3.3	3.7	4.2
INCOME TAX	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.5	0.8	0.9	1.1	1.2	1.4	1.6	1.8	2.0	2.3
NET INCOME	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.4	0.6	0.7	0.9	1.0	1.2	1.3	1.5	1.7	1.9
CASH FLOW	-11.2	-14.9	2.7	2.7	2.7	2.7	2.7	2.1	1.9	1.8	1.6	1.4	1.2	1.1	0.8	0.6	0.4

TERMINAL WORTH = 0.000

YIELD(IN PERCENT) = 0.350
PRESENT WORTH AT 0 PERCENT = 0.386
PRESENT WORTH AT 5 PERCENT = -6.386
PRESENT WORTH AT 10 PERCENT = -9.917

TABLE 4-10. LAKE ROBERGE ASBESTOS MINE / MILL COMPLEX
ITERATION NUMBER 21

YEAR	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004	2005	2006	2007	2008	2009	2010	2011
CAPITAL OUTLAYS																	
PREPRODUCTION COSTS																	
-----INTERNAL FUNDS	2.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	0.0	0.3	0.3	0.3	0.3	0.4	0.4	0.4	0.5	0.5	0.6	0.6	0.6	0.7	0.8	0.9
MINE EQUIP.																	
-----INTERNAL FUNDS	6.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	0.0	0.7	0.7	0.8	0.9	0.9	1.0	1.1	1.2	1.3	1.4	1.5	1.6	1.7	1.9	2.0
MILL EQUIP.																	
-----INTERNAL FUNDS	0.0	8.8	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	0.0	1.0	1.0	1.1	1.2	1.3	1.4	1.5	1.7	1.8	1.9	2.1	2.3	2.4	2.6	2.8
TRANSPORT INFRASTR.																	
-----INTERNAL FUNDS	0.0	1.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	0.0	0.1	0.1	0.2	0.2	0.2	0.2	0.2	0.2	0.3	0.3	0.3	0.3	0.3	0.4	0.4
TOTALS	8.8	10.0	2.1	2.2	2.4	2.6	2.8	3.0	3.3	3.6	3.8	4.1	4.5	4.8	5.2	5.6	6.1
ANNUAL EXPENSES																	
CONTAINERS	-0.0	-0.0	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5
OPER.,MAINTENANCE	-0.0	-0.0	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1
WAGES, SALARIES	-0.0	-0.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0
FUEL + ELECTRICITY	-0.0	-0.0	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8
TRANSPORT COSTS	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4
INTEREST ON LOANS	2.1	4.5	4.5	4.3	4.2	4.0	3.8	3.5	3.3	3.0	2.7	2.4	2.1	1.7	1.4	0.9	0.5
TOTAL DEPRECIATION	2.1	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.5	4.6	4.5
TOTALS	4.6	9.5	16.9	16.7	16.5	16.3	16.1	15.9	15.6	15.4	15.1	14.8	14.5	14.1	13.7	13.3	12.9
ANNUAL REVENUES																	
REVENUE FROM SALES	0.0	0.0	16.3	16.3	16.3	16.3	16.3	16.3	16.3	16.3	16.3	16.3	16.3	16.3	16.3	16.3	16.3
TOTALS	0.0	0.0	16.3	16.3	16.3	16.3	16.3	16.3	16.3	16.3	16.3	16.3	16.3	16.3	16.3	16.3	16.3
TAXABLE INCOME	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.1	0.7	1.2	1.5	1.9	2.2	2.6	3.0	3.5
INCOME TAX	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.1	0.4	0.7	0.8	1.0	1.2	1.4	1.7	1.9
NET INCOME	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.3	0.6	0.7	0.8	1.0	1.2	1.4	1.6
CASH FLOW	-11.2	-14.9	1.9	1.9	1.9	1.9	1.9	1.9	1.9	1.5	1.3	1.1	0.9	0.7	0.5	0.3	0.3

TERMINAL WORTH = 0.000

YIELD(IN PERCENT) = -4.011
PRESENT WORTH AT 0 PERCENT = -6.119
PRESENT WORTH AT 5 PERCENT = -10.539
PRESENT WORTH AT 10 PERCENT = -12.822

TABLE 4-11. LAKE ROBERGE ASBESTOS MINE / MILL COMPLEX
ITERATION NUMBER 25

YEAR	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004	2005	2006	2007	2008	2009	2010	2011
CAPITAL OUTLAYS																	
PREPRODUCTION COSTS																	
-----INTERNAL FUNDS	2.5	[illegible]	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	0.3	0.3	0.3	0.3	0.3	0.4	0.4	0.4	0.5	0.5	0.6	0.6	0.6	0.7	0.8	0.9
MINE EQUIP.																	
-----INTERNAL FUNDS	6.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	0.0	0.7	0.7	0.8	0.9	0.9	1.0	1.1	1.2	1.3	1.4	1.5	1.6	1.7	1.9	2.0
MILL EQUIP.																	
-----INTERNAL FUNDS	0.0	8.8	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	0.0	1.0	1.0	1.1	1.2	1.3	1.4	1.5	1.7	1.8	1.9	2.1	2.3	2.4	2.6	2.9
TRANSPORT INFRASTR.																	
-----INTERNAL FUNDS	0.0	1.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
-----LOAN PRINCIPAL	0.0	0.0	0.1	0.1	0.2	0.2	0.2	0.2	0.2	0.2	0.3	0.3	0.3	0.3	0.3	0.4	0.4
TOTALS	8.8	10.0	2.1	2.2	2.4	2.6	2.8	3.0	3.3	3.6	3.8	4.1	4.5	4.8	5.2	5.6	6.1
ANNUAL EXPENSES																	
CONTAINERS	-0.0	-0.0	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5
OPER., MAINTENANCE	-0.0	-0.0	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1
WAGES, SALARIES	-0.0	-0.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0
FUEL + ELECTRICITY	-0.0	-0.0	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8
TRANSPORT COSTS	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4
INTEREST ON LOANS	2.1	4.5	4.5	4.3	4.2	4.0	3.8	3.5	3.3	3.0	2.7	2.4	2.1	1.7	1.4	0.9	0.5
TOTAL DEPRECIATION	2.1	4.6	4.6	4.6	4.5	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6
TOTALS	4.6	9.5	16.9	16.7	16.5	16.3	16.1	15.9	15.6	15.4	15.1	14.8	14.5	14.1	13.7	13.3	12.9
ANNUAL REVENUES																	
REVENUE FROM SALES	0.0	0.0	17.8	17.8	17.8	17.8	17.8	17.8	17.8	17.8	17.8	17.8	17.8	17.8	17.8	17.8	17.8
TOTALS	0.0	0.0	17.8	17.8	17.8	17.8	17.8	17.8	17.8	17.8	17.8	17.8	17.8	17.8	17.8	17.8	17.8
TAXABLE INCOME	0.0	0.0	0.0	0.0	0.0	0.0	0.0	1.5	2.1	2.4	2.7	3.0	3.3	3.7	4.1	4.5	4.9
INCOME TAX	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.8	1.2	1.3	1.5	1.6	1.8	2.0	2.2	2.4	2.7
NET INCOME	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.7	1.0	1.1	1.2	1.3	1.5	1.7	1.8	2.0	2.2
CASH FLOW	-11.2	-14.9	3.4	3.4	3.4	3.4	3.4	2.6	2.2	2.1	1.9	1.8	1.6	1.4	1.2	0.9	0.7

TERMINAL WORTH = 0.000

YIELD(IN PERCENT) = 3.856
PRESENT WORTH AT 0 PERCENT = 7.256
PRESENT WORTH AT 5 PERCENT = -1.605
PRESENT WORTH AT 10 PERCENT = -6.597

CHAPTER 5. IRON ORE

CHAPTER 5. IRON ORE

SUPPLY/DEMAND SITUATION TO THE YEAR 2000

Occurrence and Beneficiation

Occurrence

Iron occupies 5 percent by weight of the earth's crust and is the fourth most abundant element. The earliest known artifacts made of iron have been dated as approximately 3000 B.C., but the very rare occurrence of metallic iron in nature suggests that these articles may have been made from meteoric iron. Chemically combined iron occurs to varying extents in a wide range of minerals. Some of the more important of these minerals are given in the following tabulation which also indicates the elemental iron content (in percent) of the pure mineral:

Mineral	Iron Content (percent)
Magnetite	72.4
Haematite	70.0
Goethite	62.9
Pyrrhotite	61.5 (approximately)
Limonite	60.0
Siderite	48.3
Pyrite	46.6
Illmenite	36.8.

For these minerals to be in a form amenable to commercial exploitation, they must have undergone some geological process of concentration. In general, these concentrations can be classified by the processes responsible for their existence. Thus, iron ores may be identified as igneous ores, contact-metamorphic ores, hydrothermal ores, sedimentary ores, or residual ores. Another means of classification was one prepared for the 1954 United Nations' survey of world iron ore resources which, dispite its manifest imperfections, has been cited extensively and appears satisfactory as a working classification. Five major divisions are recognized in this approach. They are: (A) bedded iron deposits, (B) massive iron deposits, (C) residual deposits, (D) by-product sources, and (E) other types of deposits. Class A is further subdivided into three types known as A-1 Iron Formations (Lake Superior or Algoma type), A-2 Ironstone Formations (Minette or Clinton type), A-3 Other Iron Sediments (Black-band Ores, Ferruginous Sandstone or Shales, Ferruginous Sands and Gravels, Ferruginous Carbonate Beds). Class B has four subclasses which are B-1 Bilbao type, B-2 Magnitnaya type, B-3 Kiruna type, and B-4 Taberg type. Class C has three subclasses, as C-1 Laterites, C-2 River Bed Deposits and Bulk Iron, and C-3 Other Residual Deposits.

Iron resources occur in virtually all parts of the world although it is important to recognize that a clear distinction needs to be made between iron ore reserves, potential ore, and iron ore resources when discussing this subject. For example, the Expert Panel which updated the 1954 resources study made a first recommendation that this distinction between reserves and resources is still valid in 1970. Potential ores are distinguished by a combination of factors that may range from low-grade ores whose utilization poses difficult problems of metallurgy, all the way to premium quality ores far-removed from adequate transportation. Another of the Panel's conclusions was that there is a distinct pattern in geological distribution of types of iron ore deposits: (1) The ores associated with cherty and/or finely metamorphosed iron formations of the Lake Superior and Algoma types occur almost entirely in the Precambrian Shield areas of the world; (2) Bedded, oolitic iron ores are largely distributed in gently folded sedimentary rocks of a period later than Precambrian; (3) The massive types of deposits most commonly occur in tectonically deformed belts of the world; (4) The laterite deposits are generally located within 20^o of the equator.

Other observations of the Panel pertinent to a discussion of the occurrence of iron ores would include the following:

(a) The very large demand for iron and its low unit value have led to the development of low cost, mass mining techniques for its recovery and within limits, the economies so afforded increase with the scale of production. Therefore, small iron mines and underground iron mines will become less significant in the industry;

(b) As iron ore deposits that are economically exploitable increase in size, they require an extensive and expanding infrastructure which may consist of port development, railway construction, the creation of new towns and ancillary small industry. In such cases, these developments are likely to have a pronounced and durable salutary effect on the economy of the producing country;

(c) With increasing use of prepared blast furnace feed, the possibility arises for some of the ore preparation to be done at the producing center rather than at the point of consumption. There will be a desire on the part of non-consumers to do as much preparation as possible of export ore in their own countries in order to retain maximum economic benefits;

(d) As an extension to the foregoing, the Panel notes the marked interest in developing countries in creating an iron and steel industry based on indigenous raw materials. The Panel recognizes that there are other factors in addition to resources affecting such a decision and has no wish to comment thereon. Attention is drawn, however, to the increasing mobility of iron ore in consequence of low cost ocean freight. Any country possessing ready access to marine shipping may give equal consideration to the establishment of a domestic iron industry based on imported raw materials.

Beneficiation

Prior to World War I, virtually all mining of iron ore was conducted on the basis of extracting a crude material from the earth, screening out fine particles, and shipping lump ore to the consuming centers. This hardly could be considered beneficiation except to the extent that the screened-out fines frequently did contain a lower percentage of iron than the lump ore from which they had been separated. The iron content of such "direct shipping" ores as they were called, ranged from around 40 percent in Western Europe to over 65 percent in some of the high grade deposits, for example, in Brazil. Just prior to World War II, a number of rather simple methods for concentrating an iron ore came into practice where the high grade direct shipping ores were starting to become scarce. Simple washing with water frequently removes sand, clay, and other non-iron bearing minerals that may be associated with an otherwise satisfactory ore. Additionally, certain ores contain significant quantities of magnetically responsive iron minerals and by simple procedures, the non-magnetic fraction may be removed. Until nearly 1960, these two procedures, or variations thereon, constituted the bulk of iron ore beneficiation as practiced throughout the world.

During the late 1950's, an end to the direct shipping ores of the Upper Great Lakes states in the United States could be foreseen. However, it was known that in virtually the same general vicinity there were vast reserves of low-grade iron formations locally called taconites. Some of them contained magnetite which does respond to magnetism and some contained haematite which is non-magnetic. After extensive experimentation and legal revisions regarding minerals exploitation, American companies learned how to liberate the fine grained iron minerals in taconite, convert the haematite to magnetite by roasting, and upgrade a crude ore containing 20 to 45 percent iron to one that would run 62 to 67 percent iron. However, this beneficiated product frequently was an extremely fine powder, whose shipment to consuming centers in the Lower Great Lakes states would be virtually impossible or extremely expensive. The solution for this problem turned out to be a technique known as pelletization which transforms this fine powder into reasonably uniform and structurally hard spheres ranging from 3/8" up through to 1" in diameter. Further, when this product became available to the steel mills, they soon discovered that the uniformity of pellets permitted a more rapid reduction of the iron oxide to metallic iron in the blast furnace. Thus, the throughput of the blast furnace was increased significantly, with consequent savings in the cost of the metal produced. From meager beginnings in 1955, the production of pellets on a worldwide basis gradually increased to about 16 million tons in 1960, and then grew rapidly to over 106 million tons in 1970, accounting for roughly one seventh of the iron ore concentrates shipped in that year.

There are still extensive deposits of high-grade ore from which direct shipping products can be obtained. However, an important segment of world iron ore production receives beneficiation by way of gravity separations, magnetic separations, and combinations of both, with or without the need to produce pellets in order to have a saleable product. Beneficiation is here to stay in the iron ore industry and it plays an increasing role whenever a major consuming area has to depend on low-grade minerals for at least a part of its requirements.

Uses and Substitutes for Iron Ore

Uses for Iron Ore

The overwhelmingly predominant use for iron ore is the production of iron and steel. Minor quantities are consumed in the manufacture of pigments, cement, basic refractories, and as flux in nonferrous metal smelting. Statistically, the minor uses of iron ore can be ignored.

The preface of a recently published document* in Canada contains a one sentence summary about iron and its primary derivative, steel. The statement is, "Steel is unquestionably the most important single metal used in an industrial society and iron ore, the principal source of the metal, is therefore, a most important ore". The uses for iron and steel are legion. Construction, transportation, machinery, appliances, home and office furnishings, sports equipment and recreational vehicles, household utensils, food preservation, even personal care products depend on iron or steel in one form or another. In fact, it is difficult to name an aspect of life in any of the industrialized societies that does not contain one or more products based on iron and steel.

Steel is so basic that many projections of future demand are predicted on relationships with macroeconomic indicators rather than the end-use breakdown of steel consumption. However, it may be useful to note that in the United States the major categories of end-uses of iron in 1968 had the following relationships:

End Use	Demand for Iron or Steel (percent)
Construction products	26.6
Transportation	25.1
Machinery and equipment	17.8
Containers	6.6
Pipes, tubes, and equipment (oil and gas)	4.7
Home appliances and equipment	5.1
Other	14.1
Total	100.0

Substitutes for Iron

Just as the uses for iron and steel are legion, so the substitutes for iron and steel also are legion. Ever since iron and steel came into common usage many people have been looking for alternate materials that could replace them in one or more applications. From the viewpoint of tonnage, perhaps the major substitute for iron and steel is concrete but the actual displacement of steel by concrete has in no way seriously retarded the growth of steel itself. In like fashion, aluminum, copper, zinc, and titanium have captured some share of their present markets at the expense of steel but again, they have barely dented steel's

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* "Canadian Iron Ore Industry in 1970", Mineral Bulletin MR120, Department of Energy, Mines and Resources, Ottawa, 1971.

potential. In more recent years, plastics including glass-fiber reinforced plastics have won places for themselves, in some instances at the expense of steel and iron. Further, it should be recognized that each of the so-called substitutes for iron and steel also frequently combine with them to create totally new concepts that expand the use of both.

In many of its applications involving flectural and load-bearing characteristics, steel is irreplaceable. The basic iron, of course, has been modified in a wide variety of ways to expand its usefulness in these areas. From iron to low-carbon steel, to high-alloy steels, and to low-alloy high-tensile steels, it would be difficult to find adequate replacements at comparable cost levels. As long as countries strive to improve their standards of living, the markets for iron and steel will continue to grow.

World Sources of Iron Ore

Current Production

World production of iron ore by broad geographic region is presented in Table 5-1 for 1960, 1965, and 1970. Overall, world growth has occurred at an average annual rate of 3.9 percent in the 1960 decade. This is made up of an average rate of 3.7 in the Free World and 4.4 in the Soviet sphere area. The outstanding feature of this table is the shift in origin of iron ore just within a ten year period. Western Europe declines and the United States is stagnant. Canada spurts ahead, Africa more than doubles, and Australia suddenly vaults into the upper ranks. Growth is in the less developed countries, as well as in Canada and the U.S.S.R.

Even before 1960 the U.S.S.R. had replaced the United States as the largest producer of iron ore. In 1970 they still rank first and second although the U.S.S.R. produces more than twice as much as the U.S.A. France hangs on as third largest producing country in terms of total tonnage of ore, although the grade of these ores continues to decline and is approaching the range of 30 percent iron content. Canada ranks fourth as a producer in 1970, followed very closely by Australia whose growth since 1965 has been phenominal. Both of these could pass France in the very near future in total tonnage as they have long since passed her in iron content. Sweden and India rank sixth and seventh with Venezuela, Brazil, Liberia, and the United Kingdom bunched in the 15 to 20 million tons per year class. At 192.5 million tons in 1970, the U.S.S.R. supplied 26 percent of total world production. North America's share was 18 percent, a decrease from the 21 percent it held in 1960.

TABLE 5-1. WORLD MINE PRODUCTION OF IRON ORE
(thousands of long tons)

Country/Region	1960	1965	1970
North America	106,842	121,399	134,373
Canada	19,236	33,656	46,984
United States	87,606	87,743	87,389
Latin America	41,784	54,531	70,787
Western Europe	140,895	134,518	126,662
Africa	15,224	38,993	46,885
Asia	25,442	35,400	55,405
Oceania	4,602	6,993	46,275
Australia	4,330	6,697	45,175
Free World subtotal	334,792	391,834	480,387
Soviet sphere	170,186	216,585	261,026
U.S.S.R.	103,900	150,500	192,511
World Total	504,978	608,419	741,413

Source: United Nations statistics.

Reserves

The most comprehensive statement of world reserves of iron ore is that published by the United Nations in 1970*. It resulted from the work of a panel of international experts working wherever possible with national groups. As in the pioneering study done in 1954, the panel has retained the distinction of reserves and resources with full recognition that technological advances in the two decades have converted quite sizeable amounts of resources into reserves. The panel's primary conclusion is that as of the date of the present report, world reserves of iron ore are more than 250 billion metric tons while over 500 billion metric tons qualify as resources. Table 5-2 presents a brief summary by major geographic region.

Again, the U.S.S.R. leads the way both in reserves as well as potential ore accounting for 39 percent of the total quoted. Canada and the West Indies, with 125.7 billion tons, ranks second with about 16 percent of the total. Undoubtedly, Australia, New Zealand, and New Caledonia would be credited with considerably more than 2 percent of the total resources if some figure were inserted for potential ore. Since the last meeting of the panel that prepared this report in 1967, exploration in the Western Australian mountain ranges has increased the reserves statement to more than 18 billion tons and added at least 12 billion tons of potential ore to what previously had been called merely "vast".

Current and Future Capacity

Unlike some other commodities, iron ore is a difficult material to which to assign a mine capacity for any given country--let alone the world. World production of more than 741 million tons in 1970 represented a 5.5 percent increase over 1969 and a record year as far as production and shipments were concerned. Obviously, capacity was somewhat higher than this since few installations are capable of sustaining capacity rate production over extended periods of time. At the same time it could be noted that in the one country where capacity and production both are reported, namely in Canada, an operating rate of 99 percent was experienced in 1970. Throughout the world, except for the United States, steel production in 1970 was on the upswing and the production of iron ore was being pushed to keep up with demand. Ignoring the Soviet sphere countries for a moment, Free World production at 480 million tons probably represented a production capacity of at least 500 million tons, a part of which would be accounted for by unused capacity in Western Europe because of the shift from domestic to imported ores. As presented in Table 5-3, 1972, 1973, and 1974 were expected to be pretty good years for increasing iron ore mining capacity. In 1972, for example, two rather large projects in Australia, both by Hamersley, were expected to add nearly 32.5 million tons of iron ore per year. For 1973, the big projects were Iron Ore of Canada (12.0 million), Robe River Consortium (9.0 million), Orinoco Mining (6.0 million), and Goldsworthy Mining (5.5 million at two locations). For 1974,

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* "Survey of World Iron Ore Reserves", United Nations, New York, 1970.

TABLE 5-2. WORLD IRON ORE RESOURCES
(millions of metric tons)

Region	Reserves	Potential Ore	Total Resources
Africa	6,800	24,500	31,300
Middle East, Asia and the Far East	17,300	54,200	71,500
Australia, New Zealand and New Caledonia	16,800	Vast	16,800 plus
Canada and West Indies	36,300	89,400	125,700
Europe	21,300	12,800	34,100
South America	34,100	58,400	92,500
U.S.S.R.	110,500	193,800	304,300
United States, Puerto Rico, Mexico and Central America	8,200	98,100	106,300
Totals	251,300	531,200	782,500

Source: "Survey of World Iron Ore Resources", United Nations, New York, 1970, p 5.

TABLE 5-3. PLANNED EXPANSIONS IN IRON ORE MINE CAPACITY
(millions of tons per year)

Country/Company	Location	1971	1972	1973	1974	1975	1976
Algeria							
Sonarem	Ouenza			1.7			
Argentina							
Hierro Patagonics	Rio Negro				3.5		
Australia							
Goldsworthy Mining	Kennedy Gap			1.5			
Goldsworthy Mining	Shay Gap			4.0			
Hamersley Holdings	Mount Tom Price		22.5				
Hamersley Holdings	Paraburdoo		10.0		5.0		
Mt. Newman Mining	Mt. Newman				10.0		
Robe River Consortium	Robe River			9.0			
Brazil							
Amazonia Mineracao	Para						10.0
Consortium	Aguas Claras					10.0	
Vale do Rio Doce	Caue				19.0		
Canada							
Iron Ore Company	Labrador City			12.0			
Quebec Cartier	Mt. Wright					16.0	
Chile							
Bethlehem Chile	Romeral		1.5				
Consortium	Caldera						7.0
India							
National Mineral Dev. Co.	Sandur		3.0				
National Mineral Dev. Co.	Madhya Pradesh			2.0			
Liberia							
Lamco	Tokadeh			1.5			
Mauritama							
Societe des Mines de Fer	Kedia d'Idjil			2.2			
Mexico							
Altos Hornos	Chiha				0.6		
Government	Micho					0.5	
New Zealand							
New Zealand Steel						0.3	
Spain							
Altos Hornos Vizcaya	Bodavalle			2.2			
Sweden							
Luossavaaia Kiirunavaara	Malimberget			2.5			
United States							
Cleveland Cliffs	Ishpeming				10.0		
Venezuela							
Orinoco Mining	Orinoco			6.0			
Totals		-	37.0	44.6	48.1	26.8	17.0
Cumulative		-	37.0	81.6	129.7	156.5	173.5

Source: Compiled by BCL from various trade publications.

Vale do Rio Doce at 19 million, Mt. Newman Mining at 10 million, and Cleveland-Cliffs at 10 million constitute the major projects. For 1975, Quebec Cartier at 16 million and a Consortium in Brazil at 10 million are the major projects announced to date, but more may be coming along in the not too distant future. Beyond 1975, only 17 million tons of added capacity, 10 in Brazil, and 7 in Chile seem to be in the works. However, through 1975 at least 156 million tons of additional capacity may be added to the Free World total of 500 million in 1970. This could amount to roughly a 30 percent increase in capacity, not suprisingly, in countries that are already major exporters of iron ore.

Between 1975 and 1980 some additional capacity may be needed but where and when it will be installed is questionable. The exception is Canada, where commodity experts at the Department of Energy, Mines and Resources already are predicting that capacity will be 106 million tons in 1980, from which approximately 90 million tons is expected to be shipped for sale or use. The only indication of where this expansion will be located comes from consideration of the reserves picture within Canada. The Labrador Geosyncline has by far the largest reserves as well as resources. The Southeast and Southwest portions of the Canadian Shield constitute the next largest areas followed by the Appalachian region and finally, the Western Plains region. The Grenville region receives only minor attention because, although the resources are quite high at 10 billion tons estimated, over 9 billion of these have an abnormally high content of titanium that prevents their immediate and direct use. Increases in Canadian capacity during the 1960 decade and those already planned and obviously expected in the 1970 decade strongly suggest that Canada expects to remain in a leading position as a supplier of world iron ore.

World Demand for Iron Ore

Current Consumption

Consumption of iron ore depends, of course, predominantly on the production of "crude steel", the United Nations designation for the basic product from which semifabricated steel shapes are made. In addition to iron produced directly from iron ore, steel contains recycled scrap and minor quantities of alloying agents. If one is concerned primarily with iron ore it must be recognized that the chain from iron ore to crude steel has a substantial number of alternate routes, each of which gets involved with factors such as beneficiation of iron ores, prereduction of lump ores or pellets, preparation of sinter, manufacture of steel by various processes (open hearth, electric, or basic oxygen furnace), and casting of steel into ingot molds or by continuous methods. As a point of reference, a recent study by the United Nations* suggested that the ratio of pig iron production to crude steel production is in the range of 72 to 75 percent with the balance being scrap and ferro alloy additions. In addition to this, the recovery

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* "The World Market for Iron Ore", Economic Commission for Europe, United Nations, New York, 1968.

of the iron content of iron ore in the manufacture of pig iron varies from country to country, but may be taken as an overall average of 90 percent. In effect, this says that to make one ton of steel requires only about 2/3 of a ton of iron in iron ore. To further complicate things, iron ore from various parts of the world also varies widely in iron content, ranging from the low mentioned earlier in connection with Europe of around 30 percent up to better than 65 percent in certain high-grade direct shipping ores as well as in certain pellets. The trend toward concentration, and especially pelletization, has been responsible for an overall increase in the iron content of ores from 52.2 percent in 1965 to an estimated 54.4 percent in 1970. If the 1970 average grade of iron ore is accepted it can then be said that every ton of crude steel produced requires 1.23 tons of iron ore.

Table 5-4 summarizes these relationships for some of the important steel making areas of the world showing crude steel production for 1969 and 1970 and iron ore consumption for the same years. According to this summary the world average actually is 1.245 tons of iron ore per ton of crude steel production, both for 1969 and 1970. It is immediately obvious from the table that certain areas are using a poorer grade of iron ore than others. For example, both the European Economic Community and the U.S.S.R. are using between 1.3 and 1.4 tons of iron ore for each ton of crude steel. Japan, on the other hand, is less than 1.2 in 1970 and was 1.01 in 1969. Variability from year to year may not be as important as would appear from the two years presented here. In many instances, the percentage of iron content of the ore has to be estimated because analytical results are not available. Moreover, it also is possible that a shift in the sources of ore could be responsible for a part of the variation from year to year.

World Trade in Iron Ore

Aside from the U.S.S.R. the three big steelmaking areas of the world--Western Europe, United States, and Japan---are becoming increasingly dependent on imports of iron ores. Between 1969 and 1970, imports as a percentage of total supply increased from 56 percent to 58.1 percent. Overall, world trade amounted to 282.4 million tons in 1969 and 316.1 million tons in 1970 including 16.7 million tons that went into consumers' inventories.

Figure 5-1, based on iron content of ores or concentrates, gives a representation of world trade in 1970 showing major export/import relationships. Canada and the U.S.S.R. each exported about 21 million tons of iron in ores, followed by Australia at 12.3 million tons. Other major exporters not identified on the figure included: Sweden, Brazil, Liberia, Venezuela, and India.

Future Demand

In the early 1970's, experts of the steel industry from around the world appear reasonably well agreed that crude steel demand during the 1970's will increase at an average rate that may fall between 4.8 and 5.2 percent annually. This represents a rather narrow range from about 930 million tons to approximately 960 million tons from which the figure of 951.2, corresponding to a rate of 5.0 percent, has been chosen. Canada, at an average annual rate of 4.3 percent, is

TABLE 5-4. WORLD CRUDE STEEL PRODUCTION AND IRON ORE CONSUMPTION
(millions of long tons)

Country/Region	Crude Steel Production		Iron Ore Consumption	
	1969	1970	1969	1970
Canada	9.2	11.0	9.2	11.5
United States	128.7	117.9	128.6	123.3
European Economic Community	105.6	107.3	141.7	143.7
United Kingdom	26.5	27.5	30.5	32.0
Japan	80.8	91.8	81.3	95.5
U.S.S.R.	108.5	113.1	150.5	157.9
Others	105.5	114.0	160.6	160.5
World Total	564.8	582.6	702.4	724.4

Source: Adapted by BCL from Canada Department of Energy, Mines and Resources.

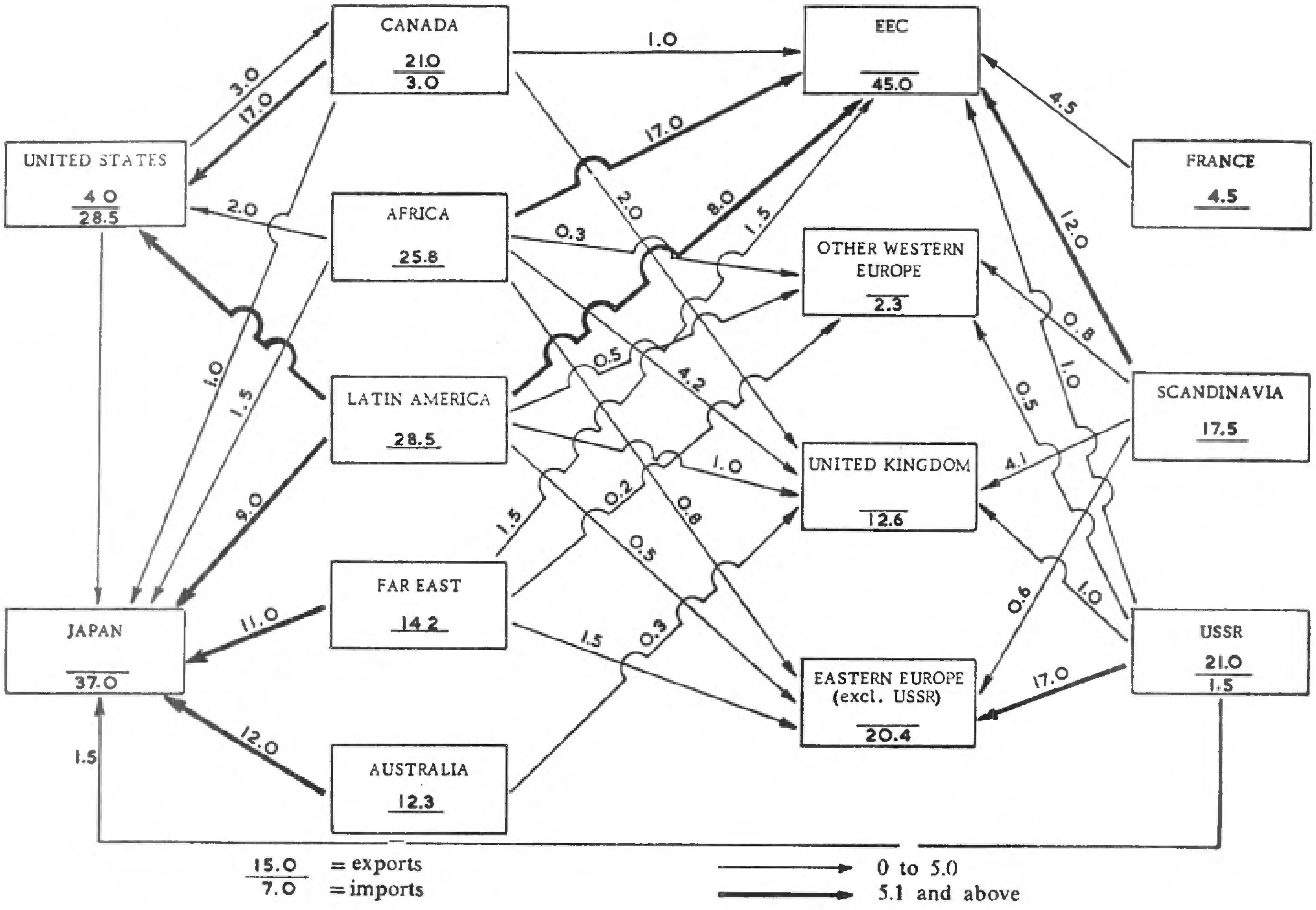

FIGURE 5-1. PATTERN OF WORLD TRADE IN IRON ORE, 1970 (millions of tons of iron content)

slightly below the world average as befits one of the more developed countries. The United States slows up to a rate of growth even less than population increase at about 1.7 percent. The Economic Community in Europe, now including the United Kingdom, has about a 3.1 percent rate, while Japan at 8.6 percent still falls substantially below rates that it achieved in the last decade. The U.S.S.R. and Eastern Europe also fall below the world average, and it is the developing countries that will be growing--some, almost explosively.

The ratio of iron ore input needed per ton of steel produced is expected to change slightly by 1980. First, the premise is used that continuous casting of steel is not likely to seriously alter the ratio of scrap to pig iron required in making crude steel. Then, there should be a continuing increase, worldwide in the grade level of iron ore that will be consumed. Some experts have suggested that it could rise as high as 57.2 percent, an increase of 2.8 percent over 1970. BCL believes it will be difficult to achieve this as long as both Western Europe and the U.S.S.R. continue to depend on the readily available but low-grade ores they have used for their sinter production. Considering all of the variables that are possible, BCL anticipates that world iron ore demand will fall short of 1.2 billion tons of ore by 1980, at a figure of 1.13 billion tons.

Crude steel demand increases at a rate of about 5 percent in the 1970 to 1980 decade according to the projection above. This is a decline from the 6 percent and better rates that prevailed in prior decades since World War II. Between 1980 and 2000, the overall rate should be further declining as the industrialized areas approach rates corresponding to population increases. There remains a considerable differential between the developed and the developing countries in terms of steel consumption/per-capita GNP ratios. Therefore, it is expected that the developing countries will have higher rates of growth although they too will be on declining curves.

The only recognized prediction of steel demand in the year 2000 is that made by the U.S. Bureau of Mines for U.S. demand, at 171.4 million long tons on the median projection. This corresponds to an average annual growth rate between 1.2 and 1.3 percent for the 30 year period from 1970 to 2000. The other developed countries, except for the U.S.S.R., should be at or below an annual rate of about 2 percent by the year 2000, while the U.S.S.R. may still be at around 2.5 percent. Overall, BCL projects that world crude steel demand will increase at a rate averaging between 3.0 and 3.5 percent between 1970 and 2000. This would suggest that crude steel production in the year 2000 could fall between 1.40 and 1.64 billion tons.

By this time the "inventory" of steel products in use and approaching obsolesence will be such that a higher proportion of recycled scrap will be used on a worldwide basis. This tends to reduce the amount of iron ore needed for the production of primary metal, and with the increasing application of prereduction to iron ore concentrates near the site of mining, the ratio of iron concentrates to crude steel is expected to approach 1.0 in contrast to the 1.245 in 1970. Using a median figure of 1.52 billion tons for crude steel, it would be anticipated that 1.52 billion tons of iron concentrates would be produced in the year 2000. Table 5-5 summarizes these projections.

TABLE 5-5. PROJECTED WORLD DEMAND FOR CRUDE STEEL AND IRON ORE
(millions of long tons)

Country/Region	Crude Steel Demand			Iron Ore Demand		
	1970	1980	2000	1970	1980	2000
Canada	11.0	17.9	(1)	11.5	18.3	(1)
United States	117.9	141.8	171.4	123.3	148.5	173.2
European Economic Community	107.3	186.3	(1)	143.7	230.0	(1)
United Kingdom	27.5			32.0		
Japan	91.8	214.5	(1)	95.5	223.1	(1)
U.S.S.R.	113.1	175.6	(1)	157.9	231.4	(1)
Others	114.0	215.1	(1)	160.5	280.0	(1)
World Total	582.6	951.2	1520.0	724.4	1131.3	1520.0

Source: Estimated by BCL

Note: (1) Not available from published data.

Supply/Demand Relationships

The prior discussions of supply indicate that there is no reasonable probability of running out of reported iron ore reserves before the end of the century. As illustrated in Figure 5-2, the cumulative drawdown of iron ore in the 30 year period to the year 2000 is expected to be less than 40 billion tons, as opposed to the 250 billion tons of reserves already counted. Further, there appears to be little need to reorient the sources of supply as they have developed through 1970. In other words, in view of reserves, and political and financial climates, the leading producer in the year 2000 still will be the U.S.S.R., with Canada and Australia vying for second place. The United States, then South America and, Finally, Africa will follow somewhat distantly as major producing centers. However, it will be necessary to add new capacity almost continuously beyond 1975 which may eventually start to put a strain on the financial capabilities of the developed countries.

Probable Price Trends

Iron ore has various values assigned to it by producers and consumers in the markets of the world. The pattern of end uses for steel influence the processes that a given company employs to make it, which carries back to the iron ore desired. Thus, a range of ores is available on world markets with price adjusted for the markets served. This may be illustrated by the prices posted for grades of iron ores delivered to Lake Erie ports for Canadian and U.S. steel mills. These range from a grade known as Mesabi Nonbessemer ore, averaging roughly 54 percent iron, to pellets averaging about 64 percent iron content. In 1970, the Mesabi Nonbessemer more was selling for $10.80 per ton delivered at rail of ship, lower lake ports, while the pellets commanded prices as high as $17.29 per ton delivered. Each company chooses from among the ores and concentrates offered to best fit their equipment and processing, at minimum cost for the metallic iron, Similar situations exist in the other large markets for iron ore, which creates opportunity for more than one supplier to survive in a competitive market.

From the viewpoint of the iron ore producer, the implication is that, regardless of his location, the sale of his product in any given market will require that he meet the price competition of delivered ore in that market. Since 1970, Lake Erie prices for pellets have risen from $.266 per unit of iron to $0.291 per unit in 1973. Obviously, this puts a premium on the utilization of the most efficient and economical transportation possible since the producer will be absorbing this cost. By way of illustration, a Canadian producer in the Labrador Trough region might be able to realize 25 cents per unit of iron for pellets at his loading port, if the delivery is to be made in the Great Lakes area. If, however, the market is Western Europe his realized sale price at the Canadian port is likely to be less then 24 cents per unit. And, if he has to ship to Japan, the value of his pellets at portside in Canada may fall as low as 22 cents per unit. Comparable price differences would apply for other varieties of iron ores or concentrates in the different markets.

In view of the adequate or more then adequate supplies of iron ore available, future prices are likely to depend more on the cost of the capital required to exploit them, rather than on a supply/demand relationship. It can

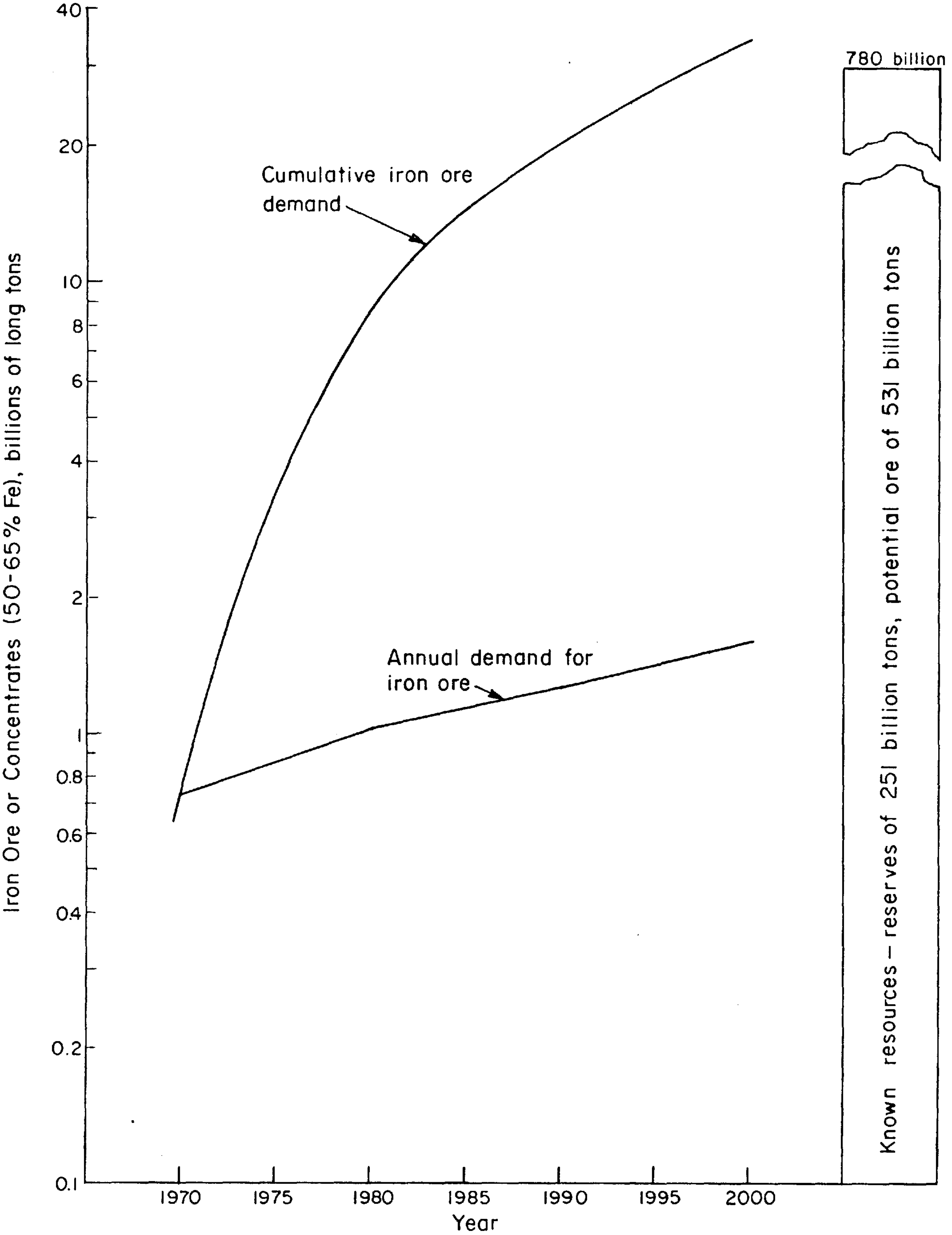

FIGURE 5-2. ANNUAL AND CUMULATIVE WORLD DEMAND FOR IRON ORE AND RESERVES

be anticpated that labor costs, transportation charges, and other expenses connected with the exploitation and processing of iron ores will continue to rise. It is also expected that these may be almost entirely offset by increases in productivity and thus, prices will follow inflationary trends. The prices of delivered iron ore to various major markets are expected to remain relatively constant through the 1980 period as measured in constant dollars. Beyond 1980, it becomes extremely difficult to try to predict what interest rates for new capital will do or how iron ore prices will be affected.

Need for a New Producer in the James Bay Region

In 1970, Canada exported approximately 83 percent of the iron ore it produced. In view of the fact that the mining of iron ore in Canada is increasing at a substantially higher rate than the consumption of iron ore in the domestic steel industry, it is obvious that exports will play an even more important role in Canada's iron ore industry in the future. Moreover, the projected growth rate for the iron ore industry in Canada suggests that new producers are not particularly required until sometime in the future, probably after 1980. Even then, a new producer might be better off to exploit high-grade ores that still are available in the Labrador Trough. But, regardless of where and when a new producer does enter the scene, the primary requisite will be assured markets for at least a 15 to 20 year period in the future. This is the present trend for new iron ore projects and major expansions. Potential customers either advance capital funds in return for a proportionate share of the output, or negotiate long-term, firm contracts at prices dependent on future production costs.

In particular in the James Bay Region, low-grade of the deposits available and their relative inaccessibility to deepwater ports strongly suggests that the involvement of one of the major international iron ore marketing organizations should be secured at the start of any project evaluation. Moreover, it would be appropriate to involve one of the larger companies that specialize in the management of iron ore properties, in order to assure the utilization of the best and latest technology in processing, as well as in transportation, in order to assure the lowest cost operation possible. It then would be the role of the marketer and the management company to determine when any iron ore project is feasible in the James Bay Region.

POTENTIAL MINE/MILL PROJECT FOR THE JAMES BAY REGION

Introduction

Currently, exploration in the James Bay region has revealed three massive formations of relatively low-grade iron ore. They have been identified as the Great Whale, Duncan Range, and Temiscamie formations, each containing billions of tons of ore averaging less than 30 percent iron.

The Temiscamie formation has received the most attention to date because of its location--near the southeastern border of the region and rather ready accessibility from established settlements. Based on these factors, it has been selected to illustrate the magnitude of a minimum economically viable project in the mining and processing of iron ore. For convenience, this paper study has been called the Lake Albanel Iron Ore Project.

This study presupposes that prior to the commitment of funds for the execution of the project, the following conditions could be satisfied:

(1) Development drilling and sampling to prove the existence of exploitable ore for a minimum of 15 years of operation and selection of the first two or three deposits to be mined.

(2) Acquisition by the project manager of control of sufficient individual deposits in the formation for an assured 15-year life for the project--control may be via participation in financing or by optioning of properties.

(3) Preparation of an economic feasibility study, including engineering cost estimates for all facilities, annual operating cost estimates, and market volume and sales price projections.

(4) Assurance of at least a "break-even" operation by negotiating with customers for (a) financial participation to be repaid by future deliveries of ore, or (b) firm future sales contracts.

Project Summary

According to Audette,* the management of Wabush Mines, Ltd., feels that the presently installed capacity of 6 million tpy of iron ore pellets is only marginally economic from an operating viewpoint. The infrastructure (especially the 277 mile railroad and remote area townsite) needed to service the location of the deposits suggests an annual production rate of at least 12 million tons of pellets.

Confirmation of this scale of operation as minimally economic for the Labrador Trough appeared in Canadian Javelin, Ltd.'s announcement** for their

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* Audette, Paul M., "Notes on Canada's Great Iron Ore Potential", Department of Industry, Trade and Commerce, Septemter 20, 1971.

** Anonymous, "Canadian firm to ship 12 million tpy of pellets", Engineering and Mining Journal, 171(8), 109 (August,1970).

Julian Lake and Star O'Keefe deposits in the Wabash-Mt. Wright district. In each instance, pellets would be produced at a port site on the St. Lawrence River. By analogy, the Lake Albanel deposits also should be exploited at a level of 10 to 12 million tpy of pellets giving them a minimum expected life of 17 to 20 years based on presently inferred reserves.

In broad outline, the project would consist initially of three open pit mines and a concentrating plant close to Lake Albanel, a slurry pipeline of 250 to 275 miles in length to the vicinity of Port Alfred, Quebec, and a concentrator and pelletizing plant adjacent to the port facilities. Annually, 45 million tons of ore containing an average of 30 percent iron would be mined and processed to 12 million tons of pellets containing 66 percent iron and 4 percent silica.

Pre-engineering estimates of the overall cost of the project, including a townsite near Lake Albanel, are about $380 million in 1973 dollars.

In addition, the Chibougamau-Fort Mistassini road needs to be extended to the project site and up-graded for year-round use, including extra heavy loads during the mine and mill construction period.

The Deposit

The Lake Albanel iron ore deposits consist of a series of occurrences extending for about 36 miles along the southerly shore of the lake, northeast of Chibougamau. Ferruginous mineralization has been identified in Morisset, St. Lusson, #1330, #1429, #1530, #1531, and #1631 Townships and classified as later Precambrian sediments of the Lake Superior type. Geologically, two parallel bands of cherty iron-bearing formations lie between Lake Albanel and the Temiscamie River from which the formation takes its name. The westerly band along Lake Albanel, up to a mile in width at places, is composed of nearly horizontal beds of interlayered iron-rich and cherty members about 450 feet thick, containing the bulk of the mineral value of the formation. The easterly band, much narrower in width, also is more steeply inclined and more severely faulted, probably from the movement of the Grenville rocks against the Mistassini Group that includes the Temiscamie formation. The contact zone, known as the Grenville front, forms the eastern boundary of Temiscamie River valley and the eastern edge of the James Bay region.

Several geological authorities (Neilson, Quirke) have inferred from their studies of the area that the Temiscamie iron formation contains over 6 billion tons of ore averaging a little less than 30 percent of iron from which at least 200 million tons of a 66 percent iron concentrate could be readily recovered. This establishes the formation as one of the major reserves of potential iron ore in Canada.

Mineralogically, the iron-rich members contain magnetite, hematite, and siderite in varying proportions. The member of greatest economic interest is described as a magnetic chart that is about 150 feet thick in the westerly band and ranges from 24 to 38 percent iron of which more than half is magnetite. This member has a pronounced oolitic and granular texture that yields good liberation on grinding to 100 mesh size. Davis magnetic tube tests indicate that a 66 percent iron concentrate is readily achievable at a recovery of more than 65 percent of the iron values. It is assumed that grinding to 325 mesh will liberate much of the residual silica for flotation removal to yield a concentrate of over 68 percent iron and about 4 percent of silica suitable for pelletizing. Preliminary chemical analyses suggest that impurities of phosphorous, sulfur, titanium, vanadium, and manganese will be quite minor, thus assuring a broad market acceptance for the 66 percent iron pellets.

Much of the foregoing information has been developed by or inferred from work performed by Albanel Minerals, Ltd., a subsidiary of Cleveland Cliffs Iron Company.

The Mines

Claims already staked in the Lake Albanel area suggest that operating efficiency and control of concentrator plant feed would be optimized by initiating the project with three open-pit mines located within a radius of about 3 miles from the plant site. In the absence of specific information regarding the deposits to be mined and their topographical features, it is assumed that the three pits will require the removal of about 600,000 cubic yards of overburden and the construction of at least 15 miles of haulage roads prior to actual mining production.

Mine Size and Materials Handling Requirements

Annual mines operations will involve the blasting and extraction of an average of about 49 million tpy of material, consisting of 45 million tpy of ore and 4 million tpy of waste. For a ten-year operating life, the waste pit should be capable of holding 30 million cubic yards of material. Initially, at least one third of this should be provided on the assumption that mined-out pits become available for waste disposal in the later years of the project. A 10 million cubic yard initial disposal area would cover 1000 acres if the terrain permits a 20 yard average depth when full.

Based on a 350 days per year operating schedule, the daily materials handling task involves 140,000 tons over the 24 hour period. Primary crushing is handled by directly dumping the ore trucks to either a 60" x 89" or a 54" x 74" gyratory crusher, set at - 8" discharge. A spare 54" x 74" crusher is provided in the crusher house to assure the flow of 129,000 tons per day of ore to the mill. The crushed ore is conveyed to open storage and distributed over reclaim conveyer tunnels by a rotary stacker.

Mine Equipment Needs

Each of the three pits represents a large installation with big, heavy equipment. Table 5-6 presents a summary of the major equipment items and some comments on sizes that may be appropriate.

TABLE 5-6. IRON ORE MINE EQUIPMENT LIST
(3 shift, 7 days per week)

Item	Number	Notes
Rotary drills, diesel powered, self propelled	12	GD 120 or equivalent
Electric shovels, 11 cu. yd. buckets	15	P&H 2100 BL or equivalent
Front end loaders, 2-1/2 cu. yd. scoop	9	
Bull dozers	9	
Dump trucks, 100 ton capacity	40	
Dump trucks, 45 ton capacity	6	
Flat-bed trucks, 3 ton capacity	3	
Pick-up trucks, 3/4 ton capacity	6	
Panel truck, 3 ton capacity	1	Fitted for explosives work
Tank truck, 1500 gallon capacity	1	
Gyratory crusher, 60" x 89"	1	700 HP motor
Gyratory crusher, 54" x 74"	2	500 HP motor

Source: Compiled by BCL.

Mine Manpower List

The rotary drill, shovel, crusher station, operators and blasters will be highly skilled personnel. Dump truck, front-end loader, and bulldozer operators would be considered semiskilled. For the schedule planned, four complete crews would be required to remain within a 40 hour work week. Table 5-7 gives a list of manpower needs by skill levels.

Operating Supplies

Daily operating supplies, exclusive of mechanical parts for equipment maintenance, will involve:

(1) 20,000 gallons of diesel fuel
(2) 500 gallons of gasoline
(3) 42 tons of solids for on-site mixing of a slurry explosive
(4) 100,000 KWH of electricity (at 11 MVA demand).

Mine Effluents

Open-pit mining involves blasting, materials transfers, and haulage by large equipment over unimproved roads. Dust and noise are likely to be the major objectionable features.

Dust from material transfer points and mine roads may be partially controlled by wetting appropriate surfaces in dry periods. On the more permanent haulage roads, a light surface dressing of an asphaltic compound will help to allay dust and preserve contours. At best, neither dust nor noise can be abated completely.

The Mill

Flowsheet

The mill flow diagram shown in Figure 5-3 is designed for autogenous grinding of the ore and magnetic separation to a grade of 65 percent iron concentrate which is to be pumped via a slurry pipeline to a deep-water port near Port Alfred, Quebec, on the Saguenay River.

Autogenous grinding (the ore itself serves as the grinding medium) was demonstrated with large scale equipment on the taconite ores of the Mesabi Range in Minnesota in the late 1950's. Since these early installations, the size of the mills has increased substantially with essentially little loss in grinding efficiency or recovery of the desired mineral in subsequent treatment. Primarily, the autogenous mill provides a material that has been reduced to the inherent crystal sizes of the minerals present. This tends to improve concentration efficiencies for physical

TABLE 5-7. IRON ORE MINE MANPOWER LIST
(3 shifts, 7 days per week)

Designation	Number
Skilled equipment operators (drills, shovels, blasters)	120
Semiskilled equipment operators (loaders, bulldozers, truckers)	350
Skilled maintenance men	100
Semiskilled maintenance men	40
Unskilled labor	90
Mine Labor	700
Management	18
Office and engineering staff	32
Office	50
Total	750

Source: Estimated by BCL.

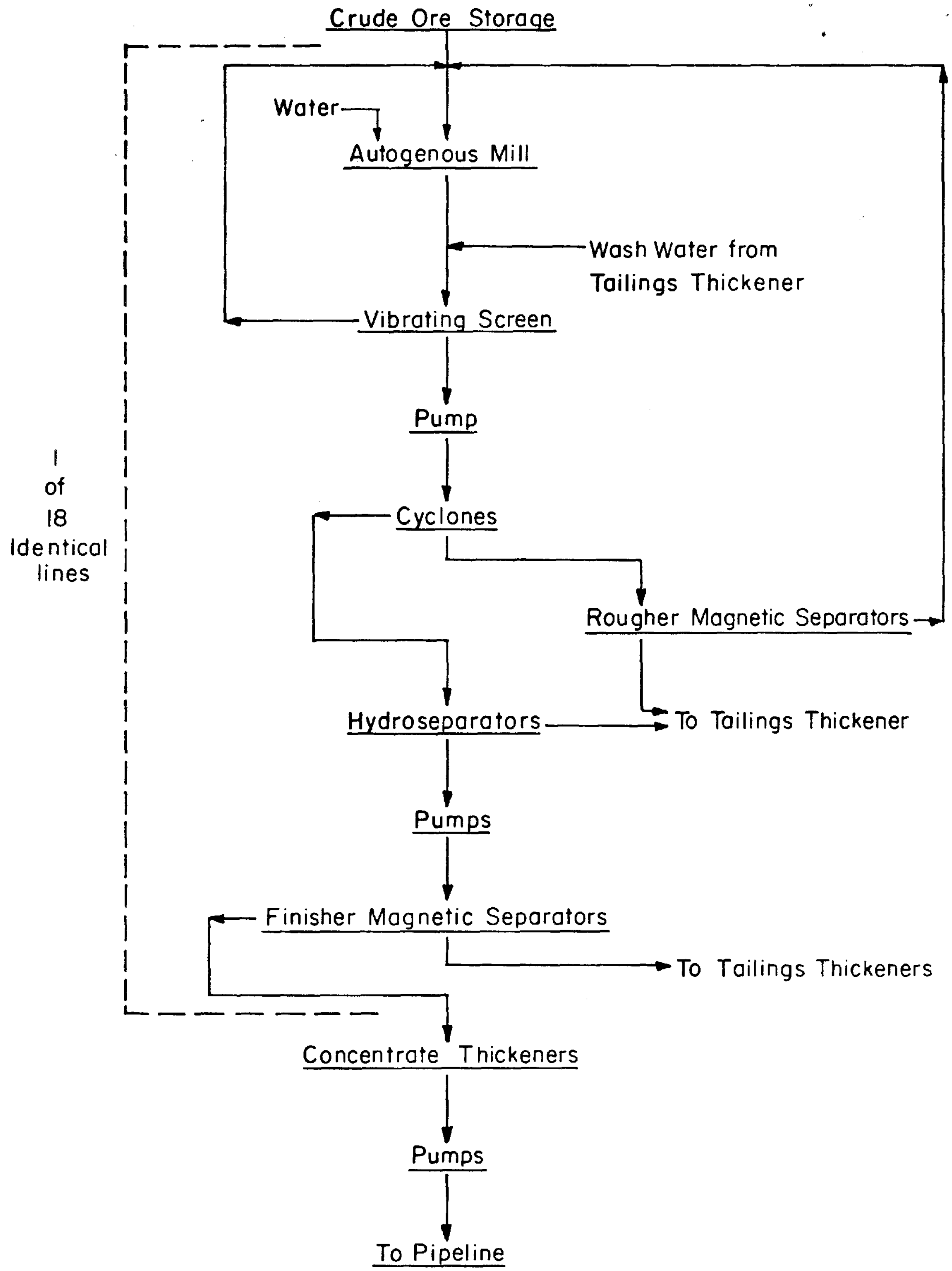


FIGURE 5-3. MATERIALS FLOW DIAGRAM FOR LAKE ALBANEL MILL

separation precedures based on gravity (spirals) or magnetism. The mills selected for this project are large (32 feet in diameter by 11 feet in length) and require drive motors capable of delivering 6000 horsepower to the mill. Wet autogenous grinding has been selected to minimize the dusting problem as well as to provide a little more flexibility in throughput rate without overgrinding.

The mill overflow is screened at 14 mesh, with the oversize being returned to the mill while the underflow is pumped to set cyclones. The cyclones provide a coarse separation at about 100 mesh, the underflow (plus 100 mesh) returning to the mills via rougher magnetic separators, while the overflow (about 20 percent of the solids fed to the cyclones) proceeds to hydroseparators and finisher magnetic separators. A concentrate thickener yields a pumpable 55 percent solids slurry that is pipelined to the pellet plant at Port Alfred on the Saguenay River.

Mill Equipment Needs

Major mill equipment requirements have been adapted to the 31 percent iron content of the ore on the basis of other iron ore operations in Canada. The major assumptions used were:

(1) Grindability slightly better than Griffith Mine ore.
(2) Magnetic separation at 100 mesh yields a 65 percent Fe, 7 percent SiO_2 concentrate.
(3) Recovery in concentrates of 65 percent of Fe content of ore.
(4) Waste rock and tailings will be neutral to slightly basic, no chemical treatment necessary for tailings pond disposal.

Table 5-8 presents the major equipment, its size, and horsepower requirements and the number of units involved.

Mill Manpower List

For the three-shift 7 days-per-week schedule, four complete crews of operating personnel will be required to stay within a 40-hour work week. Table 5-9 shows the level of skills expected in a highly instrumented, if not automated, mill. Employment will be 150 people.

Operating Supplies

Aside from 22,500 gallons per minute of make-up water, taken from a nearby lake without chemical treatment, the principal operating supply is electricity, supplied to the site at 115 KV, 3 phase, 60 cycle current. Daily requirements will be about 2.15 million Kwh at a demand level of 112 MVA. The power line to the site should be designed to handle 150 MVA minimum to cover both mill and mine requirements, exclusive of needs for the townsite.

TABLE 5-8. IRON ORE MILL EQUIPMENT LIST
(3 shifts, 7 days per week)

Item	Number	Motors (Number x HP)
Autogenous mills, 32' x 11'	18	6000 HP/mill (2 x 3000 HP)
Vibrating screens, 9' x 23'	18	60 HP/screen (2 x 30 HP)
Cyclone feed pumps, 18" x 16"	18	500 HP each
Cyclones, 12"	954	None
Rougher magnetic separators, 36" double drums x 10'	108	10 HP/separator (2 x 5 HP)
Hydroseparators, 65' diameter	36	4 HP/separator (2 at 3, 1 HP)
Finish magnetic separator feed pumps, 8" x 6"	36	50 HP each
Finish magnetic separators, 36" triple drum x 10'	72	15 HP/separator (3 x 5 HP)
Concentrate thickeners, 50'	4	3 HP/thickener (2 at 2, 1 HP)
Concentrate pumps, 14" x 12"	4	500 HP each
Tailings thickeners, 400'	4	50 HP/thickener (2 at 40, 10 HP)
Tailings pumps, 20" x 18"	4	650 HP each

Source: Adapted by BCL from "The Influence of Large Unit Equipment on Mill Plant Design", CIM Bulletin, 61(2), 166-173 (February, 1968).

TABLE 5-9. IRON ORE MILL MANPOWER LIST
(3 shifts, 7 days per week)

Designation	Number
Skilled equipment operators	20
Semiskilled equipment operators	52
Skilled maintenance men	16
Semiskilled maintenance men	24
Unskilled labor	8
Plant labor	120
Management	18
Office and laboratory staff	12
Office	30
Total	150

Source: Estimated by BCL.

Mill Effluents

The principal effluent from the mill will be waste rock, separated by the magnetic drums. Of the 129,000 tons of daily feed, nearly 92,000 tons will be pumped to a tailings pond for settling and disposal. No chemical treatment of the tailings should be needed.

The Pipeline

Although a 53-mile slurry pipeline has been operating successfully in Tasmania since October, 1967, transporting a 60 percent solids iron ore concentrate, a commercial installation of 250 miles has not yet been attempted. However, as an alternative to constructing 112 miles of new railroad and still being faced with unit train freight charges, a slurry pipeline deserves consideration. The following presentation is strictly conceptual and subject to detailed route and engineering studies to determine the feasibility of a 250 to 275 mile slurry pipeline from Lake Albanel to Port Alfred on the Saguenay River.

The mill site is expected to be somewhat less than 20 miles from the Grenville Front uplift that forms the water shed between James Bay and the St. Lawrence River. It may be anticipated that a crossing of the watershed will be at an elevation between 1,700 and 1,800 feet above sea level while the mill will be at an elevation between 1,500 and 1,600 feet, leaving a pumping head of 100 to 200 feet. Beyond the water shed, an above-ground run of about 15 miles should lead to the Mistassini River in whose bed the pipeline would be anchored until Lake St. John is reached. Between Lake St. John and Port Alfred, there are at least two power dams that will have to be circumvented, but a continuous downhill flow from the water shed to the pellet plant site is expected.

The pipe will be 18 inches inside diameter, probably gunnite coated for corrosion protection in the rivers and insulated for -30 F in the above-ground sections. The distance to be covered has not been measured precisely, but may be approximated to be between 250 and 275 miles.

A similar pipeline project was considered by Shelpac Research and Development, Ltd., in 1970 for Canadian Javelin, Ltd., in connection with its Julian Lake and Star O'Keefe iron ore deposits, north of Wabush. This project has been suspended because assured markets for the pellets could not be secured immediately.

The Pellet Plant

Concentrate from the Lake Albanel mill is expected to be 65 percent iron and 7 percent silica, with about 60 percent passing a 250 mesh screen size. A silica level of 4 percent in the pellets would be desirable in order to assure the widest acceptance of them, but this will require further grinding and a flotation operation. The grinding also will provide a size distribution suitable for the formation of pellets having good green (uncured) strength.

Flowsheet

The materials flow diagram given in Figure 5-4 and the major equipment listed in Table 5-10 have been developed using several assumptions that need to be confirmed by tests on bulk samples of representative concentrates. These assumptions are:

(1) Oversize from the cyclones, largely +250 mesh -100 mesh, can be completely ground to -250 mesh in approximately two cycles through the ball mills.
(2) The resulting slurry will be approximately 70 percent -325 mesh, a grind adequate to liberate more than half of the silica content for removal by flotation.
(3) Further, this grind provides a particle size distribution suitable to the formation of strong green pellets and sound fired pellets.
(4) The grate-kiln system can be engineered to produce satisfactory pellets with a heat input of about 750,000 BTU per ton.

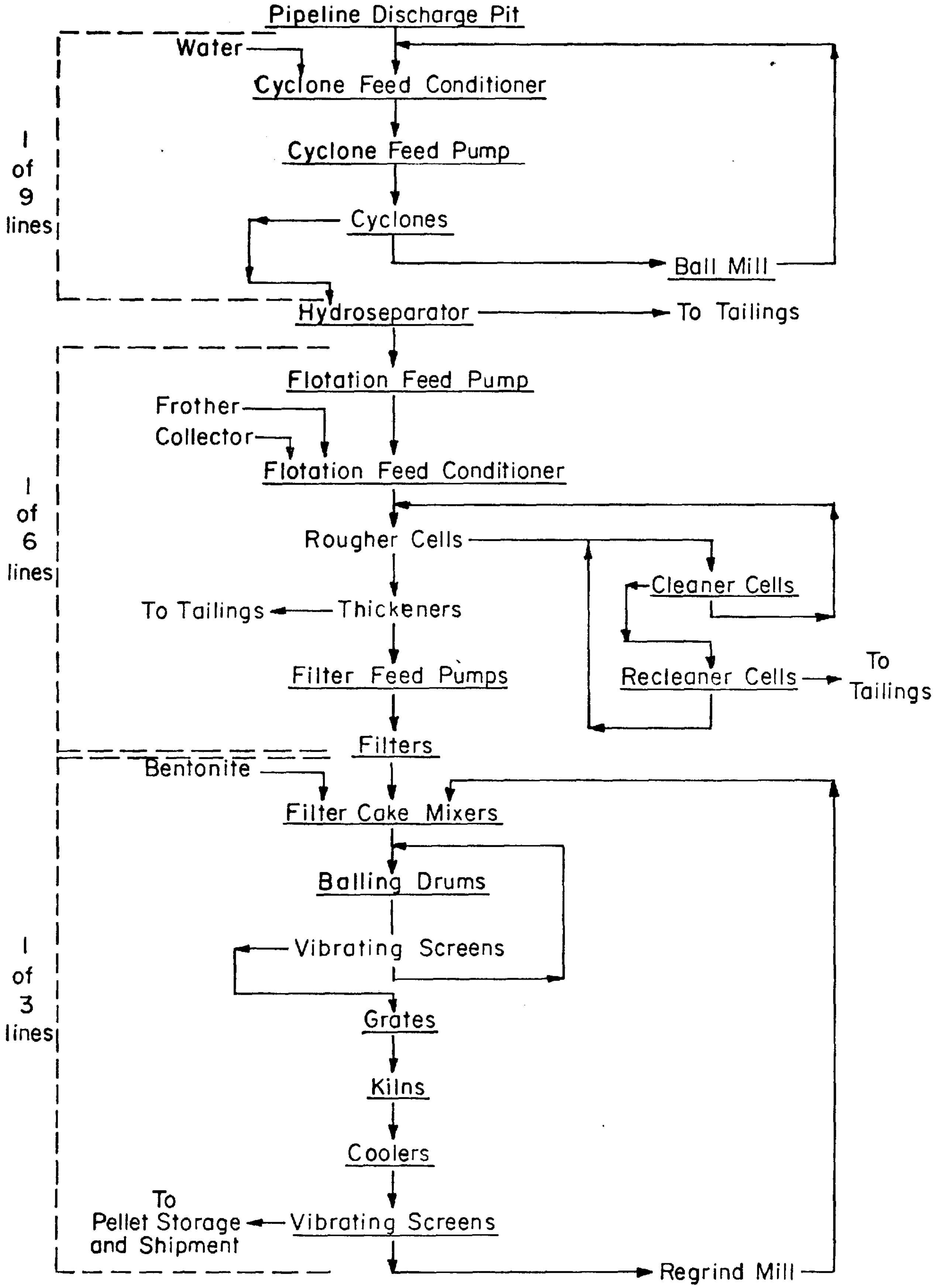
Until test results provide data for definitive engineering calculations, the plant has been sized to contain nine grinding lines, six flotation and filtration lines, and three induration lines.

The incoming concentrate slurry, thus, is diluted and sized by cyclones at 250 mesh with the oversize being ground in ball mills operated in closed circuit with the cyclones. The -250 mesh fraction is dewatered to 30 percent solids by hydroseparators and conditioned by the additon of a frother (Ucon 130 or equivalent) and a silica collector (MG 83 or equivalent) before flotation. The flotation circuits are based on 300 cubic feet cells, operating with six rougher cells, two cleaner cells, and four recleaner cells in closed circuit so that the silica tailings exit from the system in the recleaner cell froth, while the beneficiated ore appears as the rougher cell underflow. This is further dewatered in hydroseparators to a 65-70 percent slurry which is filtered on 10 leaf filters.

Filtercake at 9.5-9.8 percent moisture content is conveyed to muller mixers with 14-15 pounds of bentonite added per ton on the conveyor belts. After mixing with reground chips and undersize from the final product screening, the cake is fed to inclined balling drums along with recycled (under 3/8") seed pellets. Pellets larger than 3/8" pass on to the indurating section for drying and preheating on grates, hardening in kilns fired with low-sulfur distillate fuel, and cooling in annular coolers. Load-out facilities should be provided for rail or ship, the latter capable of servicing vessels up to 80,000 tons.

Pellet Plant Manpower List

Including load-out facilities, a total of 255 people are needed to man this plant, as shown in Table 5-11 by skill levels.



FIBURE 5-4. MATERIALS FLOW DIAGRAM FOR PORT ALFRED PELLET PLANT

TABLE 5-10. IRON ORE PELLET PLANT EQUIPMENT LIST
(3 shifts, 7 days per week)

Item	Number	Motors (Number x HP)
Cyclone feed conditioner, 8'D x 10'	9	100 HP/each (1 x 100)
Cyclone feed pump, 14" x 12"	9	250 HP/each (1 x 250)
Cyclones, 20"	54	None
Ball mills, 13' x 28'	9	3000 HP/each (1 x 3000)
Hydroseparators, 65'D	3	4 HP/each (2 x 3, 1)
Flotation feed pump, 10" x 8"	6	150 HP/each (1 x 150)
Flotation conditioner, 10'D x 5'	6	150 HP/each (1 x 150)
Flotation cells, 120" x 90" x 53"	72	30 HP/each (1 x 30)
Thickeners, 35'D	6	3 HP/each (2 x 2, 1)
Filter feed pumps, 12" x 10"	6	300 HP/each (1 x 300)
Regrind ball mill, 13' x 28'	1	2000 HP (1 x 2000)
Filters, 6'9"D x 10 leaf	18	10 HP/each (1 x 10)
Filter vacuum pumps	6	65 HP/each (1 x 65)
Filter cake mixers	21	100 HP/each (1 x 100)
Balling drums, 12'D x 33'	21	100 HP/each (1 x 100)
Vibrating screens, 9' x 20' x 3/8"	21	25 HP/each (1 x 25)
Grates, 18' x 150'	3	4500 HP/each (1 x 500, 1 x 1000, 2 x 1500)
Kilns, 25'D x 150'	3	1400 HP/each (2 x 700)
Coolers, 66'D x 10'	3	2030 HP/each (2 x 15, 2 x 1000)
Vibrating screens, 10' x 15' x 3/8"	3	15 HP/each (1 x 15)

Source: Estimated by BCL.

TABLE 5-11. IRON ORE PELLET PLANT MANPOWER LIST
(3 shifts, 7 days per week)

Designation	Number
Skilled equipment operators	40
Semiskilled equipment operators	80
Skilled maintenance men	28
Semiskilled maintenance men	32
Unskilled labor	40
Plant labor	220
Management	20
Office and laboratory staff	15
Office	35
Total	255

Source: Estimated by BCL.

Operating Supplies

Daily operating supplies and services will include:

1.13 million Kwh electricity, at a demand level of 60 MVA
2,000 gallons of low-sulfur #6 fuel oil
11,000 gallons of fresh process water
213 short tons of bentonite
1 short ton of Ucon 130
3 short tons of MG 38.

Pellet Plant Effluents

The only effluent of consequence should be the tailings from the flotation circuit which will contain silica, some low-grade iron-bearing gangue, and the residue of the chemicals used. This may require treatment by flocculation agents prior to thickening to recover water and concentrate the tailings to about 40 percent solids before pumping to a disposal pond.

Pellet Plant Output

The pellet plant is designed to produce 34,300 tons per day of 3/8" to 3/4" pellets containing a minimum of 66 percent iron and 4 percent or less of silica. This rate yields 12 million tons of pellets annually and the deposits should permit a total production of something over 200 million tons of pellets. At 1973 Great Lakes' prices, this production should bring in gross revenues of at least $3.0 billion. allowing for some freight absorption to sell in overseas foreign markets.

Capital Requirements

The capital requirements for an unengineered project can, at best, be rough approximations only. However, several announcements of potentially impending projects and major expansions at existing facilities in the Labrador Trough area indicate "order of magnitude" costs in current dollars that may be considered reasonable for preliminary planning purposes. In particular, Canadian Javelin's Julian Lake-Star O'Keefe project (announced in 1970) and Iron Ore of Canada's expansions (announced in 1971) appear to be pertinent.

The three major elements of the Lake Albanel Iron Ore Project are estimated to cost:

(1) Lake Albanel region	- $180 million
(2) Pipeline	- 80 million
(3) Port Alfred region	- 120 million
Total	$380 million

The Lake Albanel region includes the mines, magnetic concentration plant, tailings disposal areas and ponds, and a townsite for about 5,000 people. The pipeline cost does not include any allowance for right-of-the way purchase other than the out-of-river segments at both ends. The Port Alfred region includes the pellet plant, a tailings disposal pond, product storage area, a deep-water dock and dredging at the mouth of the Saguenay River to a depth of 55 feet.

Evaluation of Potential Worth

Calculation of the potential worth of this project is decidedly more complex than for the previous project descriptions given the rather larger number of unsupported assumptions that had to be made. After additional information has been received and processed, a more detailed evaluation of project worth will be undertaken. The key point to be stressed concerning the potential viability of the project is solution of the marketing problem.

CHAPTER 6. OTHER POSSIBLE PROJECTS

CHAPTER 6. OTHER POSSIBLE PROJECTS

INTRODUCTION

The implementation of several of the hypothetical or potential projects discussed earlier may provide an opportunity to consider either ancillary or integrative projects in connection with them. Additionally, there are other known mineral resources in the James Bay region that could serve as bases for future projects.

This chapter introduces such possible projects and discusses them briefly. The intent is to assure that appropriate opportunities in the future are not overlooked for lack of attention. Thus, wherever appropriate, the technical or economic difficulties that constrain ready implementation of these projects are identified. Should these constraints be removed by research by others or a change in the economic environment, they may then be revived and reexamined in the light of the new knowledge.

TONNAGE OXYGEN

One of the major resources of the James Bay region is the hydropower potential that will be exploited by Hydro-Quebec by way of a series of projects. At some point in this hydropower development, it might be appropriate to divert a small block of power to the production of tonnage oxygen in the James Bay region.

This suggestion stems from the fact that electric power represents one of the more important cost elements in this manufacturing operation which requires only readily available air as the raw material. Further, if the required power can be supplied at a cost that is lower than power costs in heavily industrialized centers, the production and utilization of oxygen in the James Bay region might provide the economic advantage needed to implement other industrial manufacturing opportunities.

A brief review of the major uses for tonnage oxygen revealed that only two immediately appropriate opportunities appeared to exist. They were: (1) Crude steel production via Basic Oxygen Furnace; and (2) Smelting of copper concentrates in an oxygen-enriched atmosphere. Neither of these opportunities has been incorporated in the projects discussed earlier for copper mining and milling or iron ore mining and milling.

Basic Oxygen Steel

In the case of iron ore, the progression sequence beyond pelletization has several rather major intermediate steps that would have to be taken before oxygen would become an item of necessity. At the least, iron ore pellets would have to be pre-reduced to a partially metallized product in order to furnish a suitable feed material for a Basic Oxygen Furnace (BOF). Unfortunately, the James Bay region has little to offer by way of resources that could serve as the reductant needed in the prereduction process. Economic sources of carbon--from coal, lignite, petroleum, or natural gas--would be desirable.

Even if a cheap reductant were available and BOF steel were produced in the James Bay region, the resulting product would have difficulty competing for markets in Canada with steel produced in established centers that are closer to markets. One of the difficulties encountered here is that the freight rate for semifinished steel (the category that would apply to BOF ingots or cast forms) is substantially higher than the rate applicable to iron ore and the rate for ore presently precludes its shipment from the region if a railroad were already installed. Moreover, this same freight situation would make it extremely difficult for James Bay steel to compete in any overseas markets.

It is difficult to visualize a period of time in future Canadian industrial development when the economics of crude steel production would favor the location of a facility in the James Bay region unless or until exploitable high-grade iron ore and coal or natural gas are discovered in the same vicinity.

Possibilities for Copper Conversion

The possibility for developing a base metal smelter in the James Bay region is contingent upon satisfying three basic conditions:

1) growth in copper markets to the point where demand exceeds the capacity of existing smelters

2) discovery and development of new ore deposits required to support the smelter

3) proven development of new chemical conversion techniques which require less feedstock.

With reference to 1), it is noted that existing smelter capacity in Eastern Canada is underutilized. Additionally, a substantial portion of the output of existing mining operations is committed to the Rouyn-Noranda complex, where additional smelting capacity of 50,000 tons per year is being installed.

With reference to 2), it is noted that feedstock requirements for the additional capacity at Noranda, according to conventional techniques, would consume the output of 8-10 of the archetypical mine/mill complexes identified earlier (750 tons per day of 25 percent concentrates).

With respect to 3), it is mentioned that most of the newer chemical conversion and continuous smelting techniques are as yet unproved. That is, most are still in the experimental stages, although some prototype pilot plant operations have been announcing success stories at that level of operation. Three key points concerning the new techniques are worthy of mention. First, it must be noted that most of the new processes being tested are the result of a dramatic thrust in research and development efforts by the copper industry to cope with anti-pollution requirements. Second, when technically proven in commercial size plants, feedstock requirements will be significantly lower, i.e., 40-50 percent of conventional operations or only 4-5 mine/mill complexes of the threshold type described earlier in the report. Third, if claims made to date are substantiated, significant cost savings, in terms of both capital investment and operating costs, will be realizable. For example, it is estimated that the Mitsubishi continuous smelting process will require only 70 percent of the investment needed for a conventional smelter and much lower operating costs due to increased automation and consequent reduced work force. Generally, all three of the above points must be considered as advantages to potential James Bay operations, but to no greater degree than potential operations anywhere else at this point in time. Again, it must be reiterated that none of the newer processes have been proven at commercial size. Thus, while there may be some hope for the future, the initial step toward its realization suggests that present emphasis be directed toward locating and developing new copper sources in the region. Without the latter, any thoughts concerning potential smelting operations must be regarded as frivolous.

HEAVY WATER

Hydrogen is now known to exist in three isotopic forms, identified as protium, deuterium, and tritium. Protium has an atomic weight of 1 and accounts for about 99.98 percent of the earth's hydrogen. Deuterium, with an atomic weight of 2, and tritium, with an atomic weight of 3, make up the remaining 0.02 percent of hydrogen and deuterium is the predominant isotope. Water (H_2O) contains approximately proportional ratios of the hydrogen isotopes and the deuterium oxide (D_2O) form, when isolated in the relatively pure state, is known as "heavy water".

In peaceful applications, the primary use for heavy water is to serve as the moderator - heat transfer medium in nuclear power reactors, such as the CANDU-PHW series. The first three commercial installations of this pressurized heavy water (PHW) concept -- at Pickering, Ontario -- have performed exceptionally well almost from the initial startup. Because of this, Atomic Energy of Canada Limited (AECL) has expressed increasing optimism for the PHW concept in recent months and has suggested the desirability of expanding Canada's production capability for heavy water.

Currently, there are four installations in Canada, in operation or under construction, that utilize the Spevack process to concentrate D_2O from the naturally - occurring level of 120-150 parts per million to a 25 to 40 percent solution suitable for fractional distillation. Also known as the Girdler/Sulfide process, the procedure involves repeated intimate contacts between liquid and gas at two different temperatures, wherein the deuterium ions migrate between the phases in the course of enrichment. Although the concept is simple, an installation for commercial production is massive and complex. Each new Bruce unit (400 tons per year of D_2O) will cost over $75 million and handle 40,000 tons of water hourly with an annual consumption of 85 million watts of energy (60 percent as steam, 40 percent as electricity). These, plus Canadian General Electric's Port Hawkesburg unit and Deuterium of Canada's Glace Bay plant being rebuilt, will provide more than 1400 tons of D_2O per year, enough to build and outfit two 1000 megawatt reactors or three of a 750 megawatt size.

Early in May, 1973, AECL publically acknowledged a probable need for at least two additional heavy water plants at 400 tons per year each. Simultaneously, it was indicated that six provinces -- New Brunswick, Quebec, Ontario, Manitoba, Saskatchewan, and British Columbia -- had expressed interest in siting one or more of the facilities.

Prior to the submission of the Interim Report to the James Bay Development Corporation, the Sores-Battelle team had considered briefly proposing a heavy water unit as a specific project. The preliminary analysis suggested that such a project should be assigned a low priority as long as the James Bay region remains a fuel-deficient area. The Phase 2 effort of Sores-Battelle has failed to develop a firm probability of extensive fuel resources in the James Bay region unless or until the trunk pipeline for natural gas from the Arctic islands is located on the east side of Hudson and James Bays. Pending a decision on this routing, a cursory study could be appropriate of the approximate cost of importing residual fuel oil to a probable location for a heavy water plant in the James Bay region. Aside from political considerations, an appropriate location must include adequate water for cooling and ready access to both fuel and electricity, hence probably along the Matagami - LG2 road at one of the major rivers.

FERRO-VANADIUM MINING AND MILLING

In 1966, the Quebec Department of Natural Resources (QDNR) acquired control of a series of magnetite-ilmenite deposits southeast of Lake Chibougamau. As reported in a leading mining journal,* "This followed the discovery by Dr. Gilles Allard, who mapped the area for the Mineral Deposits Branch of Q.D.N.R., of important vanadium tenors in that ore." Prior exploration had outlined three separate zones of iron mineralization extending over a distance of 9 miles along a NE - SW axis, containing an estimated 70 million tons of ore with a combined iron and titanium content ranging from 12 to 45 percent. Tests run by Q.D.N.R. suggest that the vanadium would report primarily with the magnetite which is readily liberated by grinding to minus 100 mesh. Magnetic separation yields a magnetite concentrate (13 percent of the ore) that analyzes 65 percent Fe, 8 percent TiO_2, 0.34 percent SiO_2, and 1.4 percent V_2O_5. Further, an ilmenite concentrate containing 40 - 42 percent TiO_2 and 35 percent Fe can be recovered from the nonmagnetic fraction left from the separation of the magnetite. The procedure for obtaining the ilmenite concentrate is not detailed.

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* Robert Bergeron, "Quebec 1968", Canadian Mining Journal, 89(4), 79 (April 1968)

Additional work obviously needs to be done before a possible project is evaluated for these deposits. Further core drilling to assess tonnage and grade of ore available in each zone and acquisition of bulk samples for development of appropriate treatment processes appear to be the next steps. However, the information already assembled suggests that technologies presently exist by which iron and a vanadium-rich product could be won from the magnetite concentrate and iron and a titanium-rich product could be recovered from the ilmenite concentrate.

In brief, the magnetite fraction might be amenable to treatment similar to that developed by Highveld Steel and Vanadium Corporation for a titaniferous magnetite of the northeastern Transvaal in South Africa. The magnetite concentrate is pre-reduced with powdered coal in a rotary kiln and then smelted in an electric arc furance. Most of the vanadium in the feed material can be held in the molten pig iron while nearby all of the titanium concentrates in a slag which is separated off. The molten pig iron is then transferred to specially designed shaking ladles and blown with oxygen. The resulting slag contains 20 to 25 percent V_2O_5 with a recovery of more than 90 percent of the vanadium content of the magnetite. The slag is separated, solidified, crushed, and treated chemically to recover the V_2O_5. The blown iron is recarburized and transferred to the steel plant for treatment in a basic oxygen furnace.

The economics of the Highveld process depend on the efficient production of a magnetite concentrate from the raw ore and the effective utilization of the hot metal (pig iron) in the production of steel. If the recovery of magnetite from the Chibougamau deposits would average only 13 percent, as indicated by the initial tests and if the pig iron product had to be sold on the open market, the economic application of the Highveld process would be highly questionable.

Some improvement in economics might be realized if the ilmenite fraction of the ore were to be treated by a process similar to that used by Quebec Iron and Titanium (QI & T) for their Allard Lake concentrates. Again, pre-reduction with coal precedes an electric furance smelting operation that yields a titanium-rich slag (70 percent TiO_2) and a pig iron suitable for use in foundries. The titanium-bearing slag has been readily marketable in Canada, the United States, and Western Europe to producers of titanium dioxide via the sulfate process. Conceivably, a new producer of titanium-rich slag could compete with QI & T for these expanding markets. However, the profitability of QI & T's operation is believed to be heavily dependent on the sale of essentially all of their pig iron at premium prices in regional foundry markets which barely sustain QI & T at present. A second supplier probably would result in a nonprofitable status for both companies.

Exploitation of the Chibougamau deposits now held by Q.D.N.R. thus is questionable unless suitable arrangements can be made to transfer pig iron--preferably molten--to a steel-making operation in the immediate vicinity. This in turn appears to hinge on an expansion of the steel industry in Quebec that could be oriented toward a Chibougamau location.

ALUMINUM FROM CLAY

Over the years, the exhaustion of local sources of readily available high grade aluminum-containing materials has led chemists to search for economic means of recovering aluminum from low-grade sources. Bauxite and diaspore--minerals usually containing more than 35 percent of alumina (Al_2O_3)--are considered to be high-grade sources. Bentonite, cryolite, feldspar, fuller's earth, kaolin, kyanite, and dawsonite--minerals usually containing less than 20 percent of alumina--are considered to be low-grade sources. Alunite, nepheline syenite, and sillimanite--minerals ranging from 20 to 35 percent of alumina--are used in special situations as substituting for bauxite.

Many processes have been developed, and in some instances patented, that are capable of separating an aluminum-enriched fraction from one or more of the low-grade sources. Universally, these processes are incapable of competing economically with bauxite for established markets at current bauxite prices. Further, bauxite prices would have to triple before the major consumers would consider low-grade domestic materials for any substantial portion of their needs.

Without question, the existence of massive deposits of aluminum-bearing minerals--such as alunite in Mexico, dawsonite associated with shale oil in the western United States, or the andalusite occurences in Wyoming--stimulates periodic re-evaluation of existing processing schemes or research to find significant economic improvements. The alunite is attractive to Mexico because it could free that country of bauxite imports. Dawsonite is one of three co-products that may have to be recovered and used to commercialize shale oil as a hydrocarbon source. Andalusite may represent a potential source of aluminum if procedures other than the Bayer process can be developed to separate alumina and silica.

In general, the clay minerals--kaolin, attapulgite, fuller's earth, and bentonite--contain 15 percent or less of alumina. Thus, they are at a distinct disadvantage in comparison with bauxite (at about 45 percent alumina). Clay-derived alumina in the James Bay region would need to cost less than about $40 per ton--including the cost of disposing of unsaleable by-products--in order for aluminum production to be economically viable.

ATTACHMENT

MATHEMATICAL APPENDIX

Introduction

This section provides selected mathematical details of the probabilistic cash flow model developed by Battelle's Columbus Laboratories.

The probabilistic cash flow model recognizes that future outlays or returns cannot be estimated with certainty. The resulting uncertainty is introduced into the model by associating with each outlay or return an estimate of the probability of its occurrence. Thus, output is generated in the form of a range of rates of return on the investment.* Given the assumptions made about various outlays and revenues, the rate of return might be as low as x_1 percent or as high as x_2 percent. In fact, it might fall anywhere in between these limits, each value occurring with a certain probability. The decision maker must then compare the likelihood of the occurrence of the rate of return and the variance or spread of its distribution with other alternatives in order to select the one in which he will invest.

Figure 1 illustrates the above ideas. Figure 1(a) depicts the deterministic case--where the yield is a specific value--because only point estimates of inputs are utilized. In this case, the rate of return is estimated at 5 percent. In Figure 1(b) the most likely rate of return is also 5 percent but it could be as low as 0 or as high as 10 percent. Figure 1(c) indicates the most likely or average rate of return of 5 percent, but the variance about this average value is such that the venture can be expected to lose 32 percent of the time. In this latter case, even though the most likely rate of return is 5 percent as it was in cases (a) and (b), the decision maker may feel that his chance of losing money is too great and, thus, forego this investment for a more attractive one. Figures 1 (b) and (c) illustrate the kind of results that the probabilistic model provides. It does this by taking explicit account of uncertainties in forecasting future outlays and revenues by associating them with probability distributions specified by the user.

The model operates by repeated random sampling from the distributions associated with the outlays and revenues. This sampling procedure continues until distributions on the rates of return, such as those in Figures 1(b) and (c), are well defined.

This approach to cash flow analysis is to be preferred to the more popular sensitivity analysis of deterministic cash flows.** Sensitivity analysis refers to the process of making different runs using "best", "most pessimistic" and "most optimistic" estimates of the various elements of the problem. This provides a range of yields (the worst case and the best case) with no indication as to the likelihood of achieving these results. The probabilistic cash flow analysis provides an indication

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* This discussion is couched in terms of rate of return on investment (yield). With no loss in generality, the discussion could also be couched in terms present value of the venture.

** The deterministic cash flow uses point estimates of the inputs to generate a single estimate of the rate of return as in Figure 1(a).

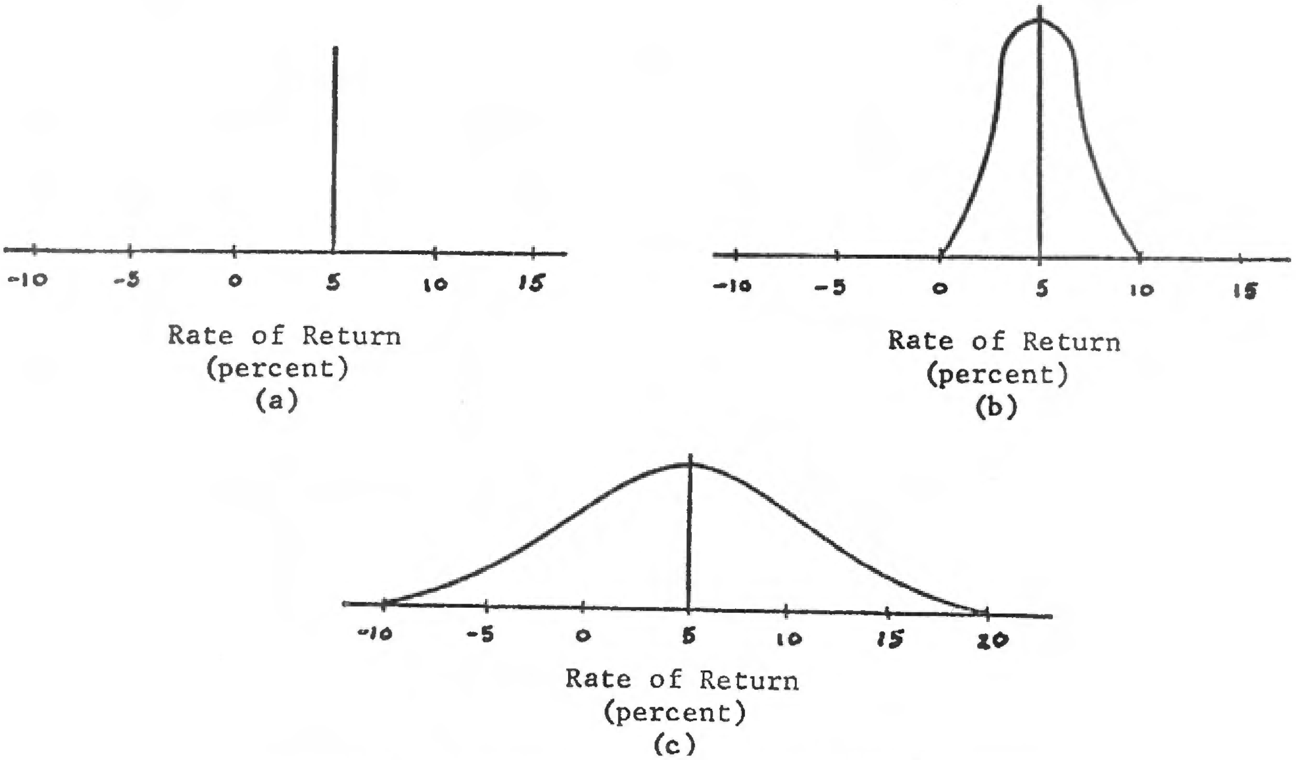

FIGURE 1. THEORETICAL PROBABILITY DISTRIBUTIONS

of the likelihood of achieving various returns over all relevant intervals. The analysis therefore, obtains a clearer picture of the various possibilities that might occur continuing the likelihood of obtaining these various outcomes.

The information output for individual alternatives is reported by associating ranges of rates of return with their likelihood of occurrence (as in Figures 1(b) and (c). This information may also be expressed as an investment profile which indicates the frequency with which a particular rate of return will be equalled or exceeded. Figure 2 indicates the investment profile associated with the distribution of rates of return in Figure 1(c).

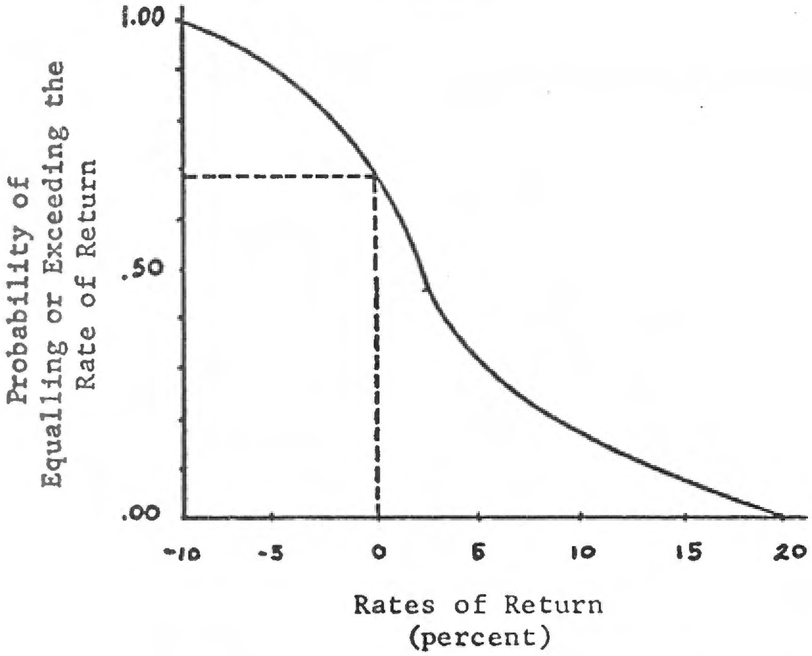

FIGURE 2. THEORETICAL INVESTMENT PROFILE

Program Inputs

Consider a planning horizon of n years. The following items are to be supplied for each year of the period: (1) major capital investments, (2) operating expenses, and (3) revenues. The above items can be supplied by item. For instance, there are typically many different kinds of costs incurred in any year, thus it is sometimes more meaningful to consider each of these components separately. Other inputs required are (a) life of various kinds of investments in (1) above; (b) the number of replications for each venture analysis (i.e., the number of trial runs).

Each of the three principal inputs (i.e., (1), (2), and (3) above) can be read in terms of point estimates (i.e., single values) or frequency functions. As a matter of interest, it is not necessary to treat all inputs alike. That is revenues can be read in as point estimates while costs can be read in terms of frequency functions. Also, it is possible for a particular type of input to be read in as a point estimate in one year and in terms of a frequency function for another year. This is done by operating the input in the probabilistic mode and assigning a probability of 1.0 to the desired single value. Such might be the case in the early years of the venture when it is felt that more is known about prevailing conditions.

The frequency functions can be of any particular "shape" the user might want to specify. This is because the program is designed to incorporate empirical rather than mathematical distributions. For each cost and revenue item treated probabilistically, the user specifies the information as shown in Figure 1. This figure is an example based on a revenue item but the procedure applies to costs as well.

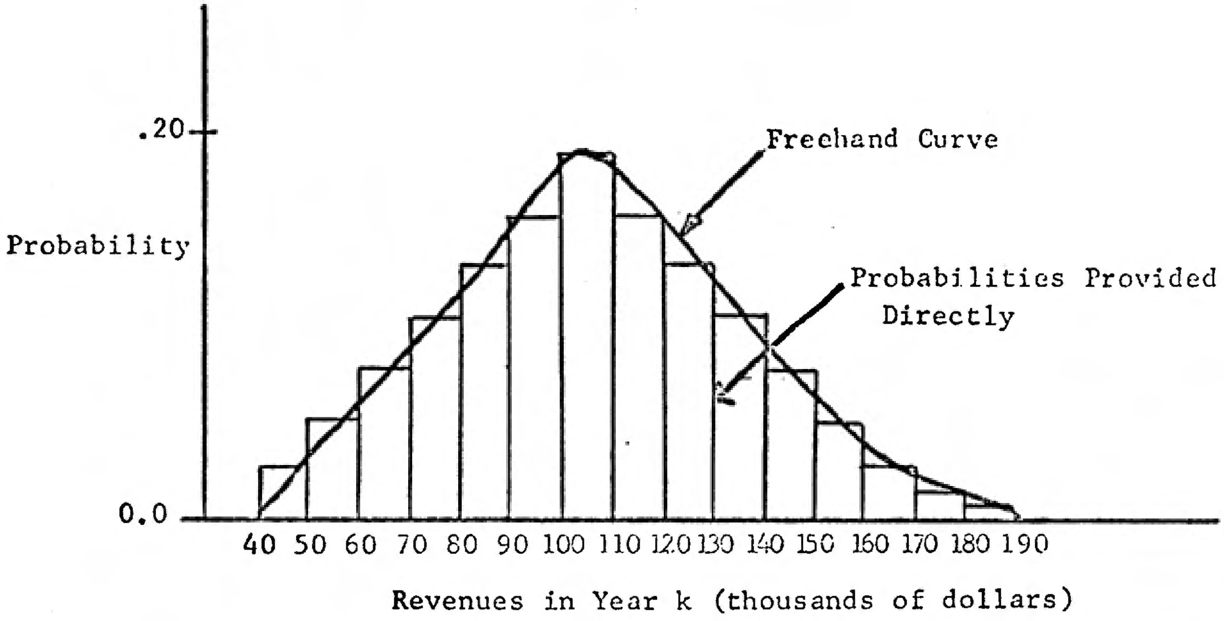

FIGURE 1. EMPIRICAL FREQUENCY FUNCTION

The diagram indicates that revenues in year k (some arbitrary year) are estimated to be between \$40,000 and \$190,000.* However, it is most probable that revenues will be between \$100,000 and \$110,000. The user either provides the probabilities of attaining each interval directly or he provides a freehad curve representing a continuous probability distribution from which the frequency function will be derived.

In summary, the user supplies the following cost and revenue inputs, some of which could be zero in any given year.

Year	Capital Investment	Revenues	Expenses
0	C_0 (or a distribution)	0	0
1	C_1 "	R_1 (or a distribution)	E_1 (or a distribution)
2	C_2 "	R_2 "	E_2 "
3	C_3 "	R_3 "	E_3 "
.	.	.	.
.	.	.	.
.	.	.	.
n	C_n "	R_n "	E_n "

Given the above inputs, the program procedure for each iteration is to sample randomly from the capital investment, revenue, and expense distributions. The sampled values are then combined to calculate the annual cash flows which, in turn, are used to calculate the rate of return and present worth according to the following procedures.

Present Worth

The present worth of the venture is calculated as

$$P = \sum_{i=0}^{n} \left(A_i \, e^{-ri} + T_i \, (1+r)^{-i}\right) \tag{1}$$

under continuous discounting of the net cash flow and

$$P = \sum_{i=0}^{n} \left(A_i \, (1+r)^{-i} + T_i \, (1+r)^{-i}\right) \tag{2}$$

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* To be more accurate, this diagram deals in intervals. That is \$60,000 is the midpoint of the interval \$55,000 to \$64,999+. Thus, in discussing 60,000, it is assumed that this represents the above interval.

under annual discounting. In the above equations,

P = Present worth
A_i = Net cash flow in year i
T_i = Value of assets salvaged in year i
r = Discount rate

The net cash flow in any year, A_i, is calculated as the sum of positive and negative flows. That is

$$A_i = I_i + O_i \text{ ,} \tag{3}$$

where

I_i = Positive cash flows
O_i = Negative cash flows

The positive cash flows, I_i, are the revenues in year i. Negative cash flows, O_i, are taken to be the negative sum of income taxes, operating expenses, and capital investments in that year. In computing income taxes, an appropriate depreciation allowance is taken using either straight line, declining balance sum-of-the-years digits methods.

Rate of Return (Yield)

The discount rate required to reduce the present worth of the venture to zero (i.e., set P=0 in (1) and (2) and solve for r) is calculated by the Newton-Raphson iterative technique for solving nonlinear equations.

It is desired to obtain an approximate real solution of a real equation of the form:

$$f\ (x) = 0 \tag{4}$$

An initial approximation Z_o to a desired root $x = \alpha$ is obtained and a recurrence relation is used to generate a sequence of successive approximations Z_1, Z_2 . . ., Z_n, . . . which converges to the limit α.

The recurrence relation can be expressed in the simple form,

$$Z_{k+1} = F\ (Z_k) . \tag{5}$$

In the Newton-Raphson method, the equivalent of (5) is

$$Z_{k+1} = Z_k - \frac{f\ (Z_k)}{f\ '\ (Z_k)} . \tag{6}$$

The necessary conditions for convergence are that f ' (x) and f" (x) do not change sign in the interval between Z_o and the root of f (x).

In the model, assuming continuous discounting is employed, we have the following equation for present worth:

$$P = \sum_{i=1}^{n} \left(A_i\, e^{-ri} + T_i\, (1+r)^{-i}\right) \qquad (7)$$

where

P = Present worth

A_i = Net cash flow in year i

T_i = Salvage in year i

r = Interest rate

We wish to find r such that P = 0. From (6),

$$f(r) = \sum_{i=1}^{n} \left(A_i\, e^{-ri} + T_i\, (1+r)^{-i}\right) \qquad (8)$$

and

$$f'(r) = \sum_{i=1}^{n} \left((A_i)(-i)e^{-ri} + (T_i)(-i)(1+r)^{-i-1}\right) \qquad (9)$$

$$r_{k+1} = r_k - \frac{\sum_{i=1}^{n} \left(A_i e^{-r_k i} + T_i\, (1+r_k)^{-i}\right)}{\sum_{i=1}^{n} \left(-A_i i e^{-r_k i} - T_i i (1+r_k)^{-i-1}\right)} \,. \qquad (10)$$

The approximation is carried out until the desired accuracy is obtained.

The Council must appoint one or more advisory committees to assist it in carrying out its duties. A majority of the membership of committees created to assist in site and corridor location shall be comprised of members of the public (Sec. 9).

Any site or transmission line route designated as suitable by the Council may be purchased or condemned for such use by the utility requesting such site or route. The issuance of a certificate of site or route compatibility, and the subsequent purchase and use of such site or route locations, are the only governmental permits, licenses, or actions required to be obtained by the utility. The certificate supersedes and pre-empts all other state, regional, county, or local laws, ordinances or regulations (Sec. 11(1)).

However, the utilities must obtain any other state permits required to construct and operate power plants and transmission lines; but any such state agency, in processing a utilities permit application, is bound by the decision of the Council with regard to all matters within the jurisdiction of the Council (Sec. 11(2)). The state agencies involved are to participate in and present their respective views and recommendations at the Council's hearings (Sec. 11(3)).

The decisions of the Council are subject to appeal to the district court (Sec. 15). The Act contains rather standard enforcement provisions, and criminal penalties are specified for violations of the Act. (Sec. 18).

A portion of the funding for the Council's activities is derived from application fees and by the assessment of a tax on the kilowatt-hours of power generated by the various utilities in the State. The remaining revenue is provided by general fund appropriations (Sec. 19).