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FEASIBILITY BRUSSILOF MAGNESITE PROJECT

ACRES

<u>BRUSSILOF MAGNESITE PROJECT</u>

<u>FEASIBILITY STUDY</u>

February, 1971

Acres Western Limited
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Telephone: 683-9141

ACRES

March 3, 1971.

Mr. Orhan Baykal
Baykal Minerals Limited
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Calgary, Alberta

Dear Mr. Baykal:

We are pleased to submit our report on the feasibility of bringing your Brussilof Magnesite property to production.

Our studies indicate that a project to produce 150,000 tons per year of high quality dead burned magnesite is technically and economically feasible at the present time. It is expected that marketing contracts for approximately this amount of product can be negotiated at prices perhaps as high as $90.00 per ton.

At the expected price and production levels the gross capital investment of $20,609,000 is justified. It is expected that Government of Canada grants will be available to assist the project and that the cash flows available for servicing and repayment of debt and for provision of return on equity will be sufficient to attract the required capital.

Our studies have been as complete as possible within the contraints of time and information available. The findings are conservatively based and may be refined by the further work that we have recommended.

We will be pleased to amplify our comments or to discuss any aspect of our work with you at any time.

Yours very truly,

J. Trevor Martin

J. Trevor Martin
Vice President

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TABLE OF CONTENTS

MOUNT BRUSSILOF MAGNESITE PROJECT

CONTENTS

		Page
I	INTRODUCTION	1
II	SUMMARY	3
III	CONCLUSIONS AND RECOMMENDATIONS	7
IV	DEFINITION OF TERMS	11
V	DEAD BURNED MAGNESITE QUALITY FACTORS	21
VI	MINE AND PLANT SITES	27
VII	GEOLOGY AND ORE RESERVES	37
VIII	MINING	79
IX	LABORATORY TESTS AND INTERPRETATIONS	83
X	PROCESSING	99
XI	TRANSPORTATION	107
XII	MARKETING	111
XIII	CAPITAL COSTS	139
XIV	OPERATING COSTS	151
XV	PROJECT SCHEDULE	159
XVI	ECONOMIC FEASIBILITY	163
XVII	PLATES	199

I INTRODUCTION

I INTRODUCTION

As part of their ongoing program of mapping and exploration, the Geological Survey of Canada reported that field work undertaken in 1965 revealed the occurrence of high quality magnesite in apparently major quantities on the west flank of Mount Brussilof, in the Kootenay region of the Rocky Mountains, in south-eastern British Columbia. On the basis of this information, a substantial number of claims were staked in the area. The general and regional locations of the deposit are shown in Plates 1 and 2.

In 1969 and 1970, Baykal Minerals, in association with others, acquired the major interest in the claims staked in the Mount Brussilof area and initiated a program of exploration and sampling. Initial results were sufficiently encouraging that a production feasibility report by an independent consulting firm was justified.

Accordingly, in July 1970, Baykal Minerals Limited retained Acres Western Limited to undertake a feasibility study of the Mount Brussilof Magnesite Project. Acres were instructed to make a preliminary investigation of the engineering and economic feasibility of developing the mine and processing and selling the product. This was to be done in sufficient depth to provide the information necessary to evaluate the project as a potential investment.

Although there are many uses for magnesite, the major use is in the production of dead burned magnesite for refractory purposes. It was agreed that, for the purpose of this study, investigations should centre around the production and marketing of dead burned magnesite.

In examining viability, the main elements were to review ownership and legal agreements, to establish the quantity and quality of the reserves, to supervise the analysis of samples and laboratory scale testing of processing methods, to determine the most suitable locations for the access road and processing plant, to perform the preliminary engineering for mining and processing necessary to estimate all capital and operating costs, to examine the market for the products, to determine an appropriate production capacity, and to examine the economic feasibility of the development.

The report is the end result of several months of extensive study. Acres' representatives have visited the mine site and have satisfied themselves that the material presented in each section of the report is factual. Individual components, designs and cost estimates will of course be subject to change as more detailed information becomes available. At this stage, however, all cost allowances and assumptions are considered conservative, and subsequent changes should therefore tend to enhance project feasibility.

II SUMMARY

II SUMMARY

Over the last two years, Baykal Minerals Limited have acquired majority control of 344 mineral claims located in the Mount Brussilof region of south-eastern British Columbia. To ascertain the extent and grade of this magnesite occurrence, sixteen holes were diamond drilled in 1968 and 1970 in the Mount Eon section of the deposit. This exploration demonstrated magnesite reserves of:

Proven	14,800,000 tons
Probable	11,800,000 tons
TOTAL	26,600,000 tons

The average unbeneficiated grade of these reserves, on an uncorrected, ignited product basis, is:

MgO	94.66 percent	
CaO	2.07 percent	3.28:1.
SiO_2	0.63 percent	
Fe_2O_3	1.36 percent	
Al_2O_3	0.32 percent	
HF Insolubles	0.23 percent	

Laboratory tests conducted by the University of British Columbia, the Department of Energy, Mines and Resources and Lakefield Research on Brussilof magnesite samples have demonstrated that:

(a) This material can be single stage calcined to produce a dead burned magnesite pellet having a bulk density of 3.35 to 3.37 gm per cm^3, a hydration tendency of less than 0.5 percent and a total porosity of approximately 5.5 percent. All of these properties correspond to high quality dead burned magnesite specifications.

(b) The majority of the iron in the deposit occurs as pyrite and is amenable to extraction by flotation. The anticipated analysis, on a burnt product basis, of average ore grade material after flotation and adjusting the silica content to provide a dicalcium silicate ratio, is:

silica source?.. Ample on Kootenay.

95.30 percent MgO
2.07 percent CaO
1.10 percent SiO_2 > 1.88%
0.80 percent Fe_2O_3
0.32 percent Al_2O_3
0.23 percent HF Insolubles

The low overburden to ore ratio, the attitude of the magnesite bed and the topography of the Mount Eon deposit area lend themselves to open pit mining. Run of mine ore averaging more than 96 percent MgO can be selectively extracted if exceptionally high grade magnesite products are required.

Five topographically possible road access routes exist between the mine property and railhead. In determining the most suitable process plant site, three of these routes were rejected because they passed through national parks. The two remaining routes from the mine terminated at Invermere and Canal Flats. Considering capital and operating cost variables and unquantifiable factors concerned with attracting and maintaining a steady work force, the most suitable process plant location is Canal Flats.

Estimates prepared over a range in plant capacities indicated that the current project capital and operating costs will be:

	Total Capital Cost $	Net Capital Cost after Government Grant $	Operating Cost per Ton Product $
75,000 TPY single kiln	13,609,000	11,629,000	39.18
150,000 TPY double kiln	20,609,000	17,849,000	33.24
225,000 TPY triple kiln	27,812,000	24,392,000	31.06
150,000 TPY single kiln	18,858,000	16,488,000	31.51

The market investigation and the complete feasibility study were confined to the sale and production of bulk dead burned magnesite grain, as opposed to bagged specialty products, caustic calcined magnesia, refractory brick, or related magnesium products. Three approaches were employed to estimate the share of the world market which could be captured. First, published and quoted statistics were analyzed to determine the penetration which could be expected in North American and world markets. Second, responses from major consumers and agents were evaluated. Third, comparisons were made with the growth of comparable high quality magnesite producers.

These studies indicate that with a suitable sales program, a market for at least 90,000 tons of dead burned magnesite could be developed in the first year of operation. The sales price of this product is estimated to be between $70 and $90 f.o.b. Vancouver.

The project viability was examined over a range of conditions. The variables considered were product selling price, initial production level and staging of production capacity, capital and operating costs at various production levels, and two ratios of debt to equity financing. The assumptions made were that money could be borrowed at 10 percent interest (1) and repaid in 10 equal payments, (2) that the taxation modifications outlined in the federal government's White Paper would be implemented, (3) and that (4) government financial assistance would be forthcoming under the Regional Development Incentives Act. In order to evaluate the resultant combinations of assumptions and variables, a computer program was developed to perform income and cash flow analyses over a period of 20 years. The program also calculated payback periods and rates of return on investment.

The computer analyses indicate that, based on a selling price of $80 per ton of product and the installation of a 150,000 ton per year double kiln plant operating at capacity, the payback period on net capital cost will range between 2.7 and 3.1 years depending upon the debt to equity ratio. Similarly, the average rate of return on net capital cost will range between 23.8 and 27.0 percent. The rate of return to equity, based on 30 percent equity financing will be 75.4 percent.

Two major factors determine the plant start-up date. The first is the date on which a decision is taken to proceed with the necessary process development work and preliminary engineering. The second factor is that senior financing must be forthcoming as soon as major financial commitments are required. Assuming these requirements are met, the project can be designed and constructed within 20 months of a firm decision to proceed.

III CONCLUSIONS AND RECOMMENDATIONS

CONTENTS

		Page
1.	CONCLUSIONS	7
2.	RECOMMENDATIONS	8

III CONCLUSIONS AND RECOMMENDATIONS

1. CONCLUSIONS

This study has demonstrated that Baykal Minerals Limited control one of the most significant high quality magnesite ore bodies known and that:

(a) The Brussilof Magnesite project is technically and economically feasible.

(b) Sufficient ore in the proven and probable categories exist in the Mount Eon portion of the deposit to sustain an operation producing 150,000 tons per year of dead burned magnesite for 65 years. Additional ore in the possible category also exists in the Mount Eon, Mount Brussilof and Cross River sections of the deposit.

(c) The ore can be beneficiated and processed to produce a dead burned magnesite which will meet high quality specifications.

(d) Based on the marketing and other data presently available and a comparison of the five production levels and kiln alternatives, a 150,000 ton per year capacity plant with two kilns is the most suitable development.

(e) Prior to final plant design, additional studies and preliminary engineering are required to optimize the plant process and capacity parameters and to enable the plant to be placed in production within 20 months.

2. RECOMMENDATIONS

The overall feasibility of the Brussilof Magnesite Project has been demonstrated in this study and it is recommended that the property be brought into production. However, additional investigations are required to optimize the plant flowsheet and design and to assure the earliest possible start up date, namely:

(a) Preliminary engineering studies are required to:

i Determine the most suitable specific plant site location taking into consideration costs, services and transportation facilities.

ii Ascertain sub-surface soil conditions and permit preparation of civil tender documents.

iii Determine major process equipment requirements and prepare specifications on long delivery equipment items.

iv Develop preliminary plant layouts.

(b) Laboratory testing should be continued in order to optimize the plant flowsheet and to determine whether a process can be developed to extract dolomite.

(c) Bulk sampling should be undertaken to provide a substantial quantity of magnesite for briquetting, grinding, calcining and other tests which will be required. This material might also be required for market development purposes.

(d) Continuing market investigations are required to confirm or adjust the product selling price range and market volume. This information will confirm the most suitable initial plant capacity.

(e) Negotiations should be initiated with the Federal Government regarding assistance grants available for the project.

(f) As additional information and data become available, these should be processed on the project computer program to determine the effect on the overall economics of the development.

(g) Consideration should also be given to producing caustic calcined magnesia, magnesite brick and other magnesium based products.

mineralogy? Dolomite?

IV DEFINITION OF TERMS

CONTENTS

		Page
1.	INTRODUCTION	11
2.	GEOLOGICAL AND MINING	11
	2.1 Ore	
	2.2 Possible or Inferred Ore	
	2.3 Probable or Indicated Ore	
	2.4 Proven Ore	
3.	PROCESSING AND GENERAL	12
	3.1 Briquetting	
	3.2 Brucite	
	3.3 Burning	
	3.4 Calcining	
	3.5 Caustic Calcined Magnesia	
	3.6 Clinker	
	3.7 Concentrate	
	3.8 Dead Burned Magnesite	
	3.9 Densities	
	3.10 Dolomite	
	3.11 Flotation	
	3.12 Green Density	
	3.13 Hydration Tendency	
	3.14 Loss on Ignition	
	3.15 Magnesia	
	3.16 Magnesite	
	3.17 Periclase	
	3.18 Pyncnometer	
	3.19 Refractories	
	3.20 Rotary Kiln	
	3.21 Sensible Heat	
	3.22 Shaft Kiln	
	3.23 Sintering	
	3.24 Specific Gravity	
	3.25 Tons	
4.	FINANCIAL	18
	4.1 Cash Flow	
	4.2 Net Capital Cost	
	4.3 Payback on Net Capital Cost	
	4.4 Payback Period	
	4.5 Total Capital Cost	

IV DEFINITION OF TERMS

1. INTRODUCTION

This section provides definitions of selected geological, mining, processing and financial terms. It does include many of the more commonly employed expressions. Generally, the terminology employed in the magnesite and ceramics industries is similar and, if there is some doubt as to the interpretation of a word or phrase, the meaning accepted in the ceramics industry should be used. The definitions of several financial terms have been shown, where it was felt this was necessary to clarify their specific usage in this report.

2. GEOLOGICAL AND MINING

2.1 Ore

Ore is defined as a natural aggregate of one or more minerals which, at a specified time and place, may be mined and sold at a profit, or from which some part may be profitably separated.

2.2 Possible or Inferred Ore

Possible ore or inferred ore is that material for which quantitative estimates are based largely on broad knowledge of the geologic character of the deposit and for which there are few, if any, samples or measurements. The estimates are based on an assumed continuity or repetition for which there are reasonable geological indications; these indications may include comparison with deposits of similar type. Bodies that are completely concealed may be included if there is specific evidence of their presence.

2.3 Probable or Indicated Ore

Probable ore or indicated ore is that material for which tonnage and grade are computed partly from specific measurements, samples, or production data, and partly from projection for a reasonable distance on geologic evidence. The sites available for inspection, measurement and sampling are too widely or otherwise inappropriately spaced to outline the material completely or to establish its grade throughout.

2.4 Proven Ore

Proven ore or measured ore is that material for which tonnage is computed from dimensions revealed in outcrops or trenches or underground workings or drill holes and for which the grade is computed from the results of adequate sampling. The sites for inspection, sampling and measurement are so spaced and the geological character so well defined that the size, shape and mineral content are established. The computed tonnage and grade are judged to be accurate within limits which are stated. It is stated whether the tonnage and grade of proven or measured ore is in situ or extractable, with dilution factors shown, and the reasons for the use of these dilution factors are explained.

3. PROCESSING AND GENERAL

3.1 Briquetting

Briquetting is that process in which finely divided solids are compacted at pressure into pellets. Roll or piston presses are frequently employed for this operation. Organic compounds, sulphuric acid and water may be used as binders to produce a more cohesive product.

3.2 Brucite

Brucite, or $Mg(OH)_2$, is a magnesium hydroxide mineral that theoretically contains 69 percent MgO and may be converted to magnesia by calcination. In nature, however, it is usually associated with limestone making the extraction of magnesia relatively expensive.

3.3 Burning

For purposes of this report, the terms burning and calcining are synonymous.

3.4 Calcining

Calcining is the process of heating the ore to a high temperature to eliminate volatiles and chemically combined constituents such as carbon dioxide or sulphur dioxide. The term is also applied to the process of firing magnesite in shaft or rotary kilns to produce dead burned magnesite. In this context, a more appropriate term would be "high temperature calcining", because the temperature employed is much higher than that required to simply volatilize off the carbonates.

Single stage calcining or burning refers to the burning of raw magnesite directly to a dead burned product.

In two stage calcining or burning, magnesite is first burned to MgO at a temperature of approximately 1000°C, then compressed into pellets or briquettes, and fired a second time at temperatures of approximately 1650°C. Two stage burning usually produces a more dense, dead burned product than single stage burning.

3.5 Caustic Calcined Magnesia

This is a reactive magnesium oxide made by calcining magnesium carbonate or magnesium hydroxide at 900°C or lower. In appearance it is a loose, bulky mass, its specific gravity increasing as the firing temperature is raised. Its use in the preparation of oxychloride cement, rubber, fertilizer, rayon, paper, etc. is dependent on density, chemical purity, reactivity and grain size.

3.6 Clinker

Clinker is the term applied to dead burned magnesite as it emerges from the kiln. The term infers that some sintering and build up of particle size has occurred in the firing operation and the larger pieces are sometimes referred to specifically as clinker. Usually the larger pieces of clinker are crushed as soon as they are discharged from the kiln.

3.7 Concentrate

A product, containing the valuable mineral or metal portion of the ore, from which most of the waste material has been eliminated.

3.8 Dead Burned Magnesite

Dead burned magnesite is the granular product obtained by calcining magnesite, or other minerals convertible to magnesia, upon heating above 1450°C long enough to form dense, weather-stable granules suitable for use as a refractory or in refractory products.

The decomposition rate for magnesite is a function not only of the temperature, but also of the nature of the impurities, structure and density of the material.

3.9 Densities

The terms true density and specific gravity in ceramic work are synonymous. The various density terms employed in this report are defined below.

Bulk volume $= V_b$	Bulk density $= d_b$
Apparent volume $= V_a$	Apparent density $= d_a$
True volume $= V_t$	True density $= d_t$
Volume of open pores $= V_o$	Wet weight $= W$
Volume of closed pores $= V_c$	Dry weight $= D$
Volume of total pores $= V_v$	Suspended weight $= S$

Defined:

Bulk volume, $V_b = V_o + V_c + V_t = \frac{D}{d_b}$

Apparent volume, $V_a = V_c + V_t = \frac{D}{d_a}$

Volume of closed pores, $V_c = V_a - V_t$

Volume of open pores, $V_o = V_b - V_a$

Volume of total pores, $V_v = V_c + V_o = V_b - V_t$

Bulk density, $d_b = \frac{D}{V_b}$

Apparent density, $d_a = \frac{D}{D - S} = \frac{D}{V_t}$

True density, $d_t = \frac{D}{V_t}$

Volume of closed pores, $V_c = \frac{D}{d_a} - \frac{D}{d_t}$

3.10 Dolomite

Dolomite is a mineral consisting of a calcium magnesium carbonate found in crystals and in extensive beds as a compact limestone. The rock contains up to about 22 percent magnesia. Calcined dolomite is also used as a refractory.

3.11 Flotation

Flotation is a processing technique in which some mineral particles are induced to float to the surface of an agitated and aerated slurry, whereas other particles sink. In this way the valuable minerals can be separated from the waste.

3.12 Green Density

Green density is a term used to define the degree to compaction obtained in briquetting. It is equivalent to apparent density, as defined under Section 3.8.

3.13 Hydration Tendency

The tendency of a material to chemically combine with water is termed hydration tendency. Quantitatively, it is the percentage weight increase of a dry sample attributable to chemical association with water after exposure to steam for 5 hours.

3.14 Loss on Ignition

This term, commonly abbreviated L.O.I., is, for purposes of this report, taken to be the weight loss on calcining raw magnesite for 2 hours at 1000 to 1100^{o}C.

3.15 Magnesia

Magnesia, in the strictest sense, is pure MgO. However, the term is commonly applied to all compounds containing a high proportion of MgO. These include dead burned magnesite or magnesia, caustic calcined magnesia, synthetically produced magnesia and periclase. Crystallographically, there is no difference between magnesia and periclase. However, the term periclase is usually applied to dead burned magnesite grains of relatively high purity. Magnesia is a white, highly infusible, magnesium oxide used in refractories, in cements, insulation, fertilizers and rubber, and in medicine as an antacid and mild laxative.

3.16 Magnesite

Magnesite is one of the calcite group of carbonate minerals consisting of magnesium carbonate and usually containing varying amounts of calcium carbonate and oxides of aluminium, iron, silica and boron. The color of pure magnesite is white.

Magnesite, or $MgCO_3$, appears in natural deposits in two modifications:

(a) As a crypocrystalline (often called amorphous), earthy, hard and compact mineral, which probably is a hardened colloidal precipitate. It is often concretionary and has a conchoidal fracture like that of unglazed porcelain. In this form it is an alteration product of serpentine or allied magnesian rocks.

(b) As a crystalline mineral, isomorphous with calcite and usually holo-crystalline granular. In this form it is generally a replacement of dolomite produced by magnesian solutions in connection with intrusions. Brucite is sometimes present.

Natural magnesites consist of variously sized grains reaching submicroscopic dimensions. Crystalline magnesite, with well-formed crystals, is usually formed when limestones or dolomites are acted upon by solutions containing magnesium bicarbonate. The amorphous magnesites, in which the crystals can only be made out with X-ray or electronic spectra, are products of the breakdown of rocks consisting primarily of magnesium silicates subjected to the action of carbon dioxide in the presence of water.

The manner in which the natural magnesites were formed is responsible for the presence of impurities.

The word magnesite in the strict sense refers only to the natural mineral magnesium carbonate but is commonly applied also to material consisting largely of magnesia obtained by calcination of the natural mineral.

3.17 Periclase

Periclase, or MgO, occurs in limited quantities in nature as a metamorphic mineral. The name is also applied to relatively pure dead burned magnesite.

When magnesite sinters, there is virtually a complete formation of new crystalline phases, depending on the temperature and duration of the firing. The sintered product consists chiefly of periclase crystals, which are the most stable crystalline phase in the fired magnesite. The periclase imparts hardness, chemical stability and other desirable characteristics of refractory materials. Hence, the grade of the sintered product is determined to a considerable extent by the amount and degree of crystallization of the periclase.

3.18 Pyncnometer

A pyncnometer is a flask or vessel whose weight and volume have been accurately determined, and which is used in specific gravity and density determinations.

3.19 Refractories

Refractories are materials used for constructing various types of industrial furnaces, fire boxes and apparatus which operate at high temperatures ranging in modern industrial furnaces and fire boxes anywhere between 1000 and 1800°C. Hence, refractoriness is required, i.e. the ability to withstand structural loads at certain temperatures.

When subjected to high temperatures, most refractory materials decrease in volume on account of additional sintering and densification. Major volume reduction usually takes place on the first firing to a given temperature. Minor volume changes can be expected on subsequent cycles. Variation in the volume of a refractory may cause damage and even destruction of the furnace brickwork. For this reason, refractory materials must also exhibit virtually constant volume at service temperatures.

The ability of refractory materials to withstand temperature variation without cracking is called their spalling resistance. Insufficient spalling resistance is one of the most important factors reducing the service life of the lining of industrial furnaces. The refractory lining of industrial furnaces and fire boxes is damaged most extensively through chemical reaction with the ash from the fuel being burnt, or with the materials that are being melted or fired in the furnaces. The degree of damage, or slag erosion, depends on the chemical composition of the material affecting the lining, on the temperature of the reaction and on the chemical composition and porosity of the refractory.

In practice, the isolated effect of a single destructive factor is rarely encountered. Sometimes the action of the slag causes softening of the refractory and at the same time a loss of structural strength. Considerable additional shrinkage of refractories at high temperatures reduces their spalling resistance.

At the present time, there are no refractories combining in equal measure all the working properties required for stable service under all conditions.

3.20 Rotary Kiln

A rotary kiln is a rotating open-ended cylindrical shell with its ends projecting into fixed feed and discharge structures and sloping at approximately 1/2 to 3/4 inch per foot towards discharge end. The shell diameters generally vary between 5 and 15 feet and the length between 60 and 500 feet. Heat is applied at the discharge end to accomplish the chemical or physical transformation required of the feed material.

3.21 Sensible Heat

Sensible heat is that heat the addition or removal of which results in a change of temperature, as opposed to latent heat.

3.22 Shaft Kiln

A shaft kiln consists essentially of a vertical, stationary, refractory-lined stack with provision for introducing heating fuel. Feed material is introduced at the top of the kiln. The calcined or sintered product is removed from the bottom of the kiln by means of one of a variety of discharge arrangements. As material progresses through the shaft kiln, it passes through three zones - preheating, burning and cooling.

3.23 Sintering

Sintering is that process which results in the agglomeration of particles by application of sufficient heat to cause materials to soften and adhere together, but not to cause melting.

3.24 Specific Gravity

See definition for Densities, Section 3.8.

3.25 Tons

Unless otherwise specified, all tonnages refer to short tons.

4. FINANCIAL

4.1 Cash Flow

Cash flow is the difference between cash income and cash costs and is a measure of project profitability.

For purposes of this study, cash flow is equal to net operating income less interest on borrowed money, less loan repayments, less income tax.

Two basic methods of financing the project are considered. One method assumes financing with 100 percent equity, and in this case, as there is no borrowed money, the cash flow equals net operating income less income taxes.

The other method assumes financing with 30 percent equity money. In this case the cash flow represents cash available to pay to equity shareholders and interest on borrowed money and income taxes are deducted from the net operating income.

4.2 Net Capital Cost

Net capital cost is equal to total capital cost less government capital assistance grants.

4.3 Payback on Net Capital Cost

This term applies to that cash flow available to repay the net capital cost. It is equivalent to the cash flow to equity plus loan repayments. It is most significant in those analyses in which the project is partially debt financed.

Payback on net capital cost takes into consideration the interest on borrowed money, the debt-equity ratio, and is a measure of the soundness of the project under the proposed financing arrangement.

4.4 Payback Period

The payback period is the number of years required to repay the net capital cost of the project out of undiscounted payback on net capital cost.

4.5 Total Capital Cost

Total capital cost takes into consideration all capital expenses for real property, equipment and for engineering and management services involved in bringing this property into production. It does not make any allowance for federal government capital assistance grants.

V DEAD BURNED MAGNESITE QUALITY FACTORS

CONTENTS

		Page
1.	INTRODUCTION	21
2.	CHEMICAL PROPERTIES	21
	2.1 Magnesia, or MgO	
	2.2 Lime, or CaO	
	2.3 Iron	
	2.4 Silica, or SiO_2	
	2.5 Alumina, or Al_2O_3	
	2.6 Boron, or B_2O_3	
	2.7 Ash	
	2.8 Calcium Silica Ratio	
3.	PHYSICAL PROPERTIES	24
	3.1 Specific Gravity	
	3.2 Bulk Density	
	3.3 Crystal Size	
	3.4 Grain Size	
	3.5 Shape of Grains	
	3.6 Hydration Tendency	
	3.7 Firing Shrinkage	
	3.8 Miscellaneous Tests	

V DEAD BURNED MAGNESITE QUALITY FACTORS

1. INTRODUCTION

The most important properties determining the quality of refractories are high temperature structural strength, volumetric stability at high temperature, spalling resistance and resistivity to attack from slag. There is no single group of refractories which optimizes all of these properties. For example, carborundum refractories show the greatest thermal conductivity and spalling resistance, silica refractories exhibit the greatest structural strength and magnesite refractories the greatest resistance to a variety of slags. In selecting a refractory for a given application it is important, therefore, to make a general determination of the refractory group best suited for the specific application. The performance of a specific refractory within a general group will vary depending upon chemical and mineralogical composition, and the methods used to process the refractory material.

Dead burned magnesite produced from a natural deposit will have mineralogical and chemical properties which are unique to its source and the combination of these properties, together with the techniques used in producing the refractory grain, will determine its acceptability. Although the ultimate acceptability of a refractory will be determined by its performance in service, a relatively reliable comparison can be made on the basis of chemical analyses and physical properties. The balance of this section is given to a discussion of some of the more significant criteria used in evaluating dead burned magnesite.

2. CHEMICAL PROPERTIES

2.1 Magnesia, or MgO

Magnesite refractories are materials consisting primarily of the mineral periclase, or MgO. Dead burned magnesite produced from sea water may have MgO contents as high as 97 percent. The most commonly used high quality grades of dead burned magnesite range from 93 percent to 96 percent MgO. The melting point of magnesite refractories is generally above 2000^{o}C. Pure periclase melts at approximately 2800^{o}C.

The presence of impurities reduces the melting temperature and consequently increases the sintering tendency during firing. However, large amounts of impurities reduce the refractoriness of the material.

2.2 Lime, or CaO

This is a particularly harmful impurity which, present in the free state, tends to hydrate or combine chemically with water. If it is combined as a silicate, it is susceptible to crumbling because of phase transformations at various termperatures.

Good quality, dead burned magnesite should contain no more than 2 to 3 percent CaO. Some sea water magnesia contains as low as 0.8 percent CaO. In order to lower the sintering temperature, however, CaO is usually adjusted to 1.5 to 2.5 percent.

2.3 Iron

The presence of iron can greatly facilitate sintering and recrystallization. If iron in the form of pyrite is present in the ore deposit and is not removed, the calcining process results in the discharge of sulphur dioxide into the atmosphere.

Historically, magnesites with iron contents of 0.3 to 1.0 percent have been the most acceptable, but refractory magnesites with iron contents as high as 7 percent are presently in use.

2.4 Silica, or SiO_2

Magnesite refractories usually contain 1 to 2 percent SiO_2. If the silica content of magnesite brick appreciably exceeds 2 percent, the heat from the furnace causes the silica fraction to migrate away from the hot brick face. Thus, a zone of mechanical weakness is created at the hot side of the brick which results in gradual deterioration of the brick.

The lower the silica concentration, the less likelihood there is of a weak zone and crack formation. The SiO_2 content in sintered magnesite should generally not exceed 3 to 3.5 percent.

2.5 Alumina, or Al_2O_3

The alumina content in magnesite refractories is usually in the order of 1 to 2 percent, although in some special refractories alumina concentrations as high as 6 percent have been used to increase the thermal shock resistance of the brick.

2.6 Boron, or B_2O_3

Boron combines with silica to form low melting point silicates. Boron is sometimes added to prevent dusting in refractories in which the calcium and silica concentrations are high and exist in a dicalcium silicate ratio. The B_2O_3 content should not exceed 0.05 percent when used as a dicalcium silicate stabilizer. Otherwise the inter granular hot-strength of the refractory product is adversely affected.

2.7 Ash

In the process of dead burning magnesite, if the kilns are fired with coal dust, about 25 to 30 percent of the entire fuel ash remains in the magnesite, leading to about a 1 percent increase in the amount of SiO_2 in the sintered product. Hence, kilns for dead burning magnesite are best run on ash-free liquid or gaseous fuel.

2.8 Calcium Silica Ratio

The melting temperature curves for CaO and SiO_2 mixtures indicate that, if the CaO to SiO_2 molecular ratio falls below 2 to 1, equivalent to a weight ratio of 1.87 to 1, the melting point drops rapidly to about 1500°C. This adversely affects the refractoriness of the dead burned magnesite in which these two materials are present. On the other hand, if there is insufficient SiO_2 to chemically bond with the CaO, the refractory will have a high tendency to hydrate because of the presence of free CaO. For these reasons, the CaO to SiO_2 weight ratio should usually be maintained between 1.7 and 2.6 to 1. 1.87 to 1 corresponds to a theoretical dicalcium silicate, or $2CaO.SiO_2$, ratio.

3. PHYSICAL PROPERTIES

3.1 Specific Gravity

The specific gravity of completely dead burned magnesite is determined by the impurities, and generally ranges between 3.56 and 3.65. The specific gravity of pure periclase is 3.58.

3.2 Bulk Density

This is the weight of a sample divided by the volume of solids plus open and closed pores. The bulk density is a function of the specific gravity, grain size and the degree of sintering of the product. Bulk density of the dead burned magnesites presently in use varies between 3.15 and 3.40 gm/cm. A high bulk density is desirable.

3.3 Crystal Size

Examination of the crystal size supplies confirming evidence of the degree to which the material is dead burned, well burned material being usually 0.03 mm or over. The grade of the sintered product is determined to a considerable extent by the amount and degree of crystallization of the periclase. The periclase crystals of a natural dead burned product are generally considerably larger than those of sea water magnesia and result in a high degree of solid to solid bond producing an extremely dense grain, the pores occurring largely within the periclase crystals themselves.

3.4 Grain Size

Various users have varying grain size specifications. The maximum grain size is usually in the order of 10 mm. Frequently the grains smaller than 0.8 mm are screened out as they may have a greater tendency to hydrate than the coarser grains. Good gradation of grains results in a high density brick.

3.5 Shape of Grains

In magnesite produced from sea water, the grain shape has a tendency to be round whereas, in most natural magnesites, the grains are irregular in shape and produce a more dense brick.

3.6 Hydration Tendency

Excessive hydration of dead burned magnesite results in dimensional instability, cracking and reduced bulk density. Dead burned magnesite should exhibit a very low hydration tendency.

3.7 Firing Shrinkage

If the firing shrinkage is too great, cracking is likely to occur in the kiln and the brick, if made in standard moulds, will be under size. This is a comparative test and magnesite is a desirable refractory material because of low shrinkage tendency.

3.8 Miscellaneous Tests

A number of other specialized tests are used to judge the quality of magnesite from a specific source. These tests are generally carried out on the manufactured brick and hence depend on the process used in the manufacturing of the brick as well as the properties of the dead burned magnesite. Examples of such tests are: thermal shock resistance, refractoriness under load, cold crushing strength, slag resistance and bonding qualities.

VI MINE AND PLANT SITES

CONTENTS

	Page
1. MINE SITE	27
1.1 Regional Setting	
1.2 Physiography	
1.3 Climate	
1.4 Current Access	
1.5 History of Claims and Exploration	
2. PLANT SITE SELECTION	31
2.1 Factors Affecting Selection	
2.1.1 Access Routes to the Mine	
2.1.2 Alternative Plant Locations Tested	
2.1.3 Calcining Fuel Costs	
2.1.4 Other Considerations	
2.2 Method of Comparison	
2.3 Findings and Recommendations	

VI MINE AND PLANT SITES

1. MINE SITE

1.1 Regional Setting

Plate 1 shows the general location of the ore deposit in the southeastern section of the Province of British Columbia in a region known as the Kootenays. The site lies within the Rocky Mountain chain, about 600 road miles west of Vancouver, at longitude 115°39'W, latitude 50°49'N, approximately 20 miles northeast of Radium Hot Springs, British Columbia, and 27 miles south and slightly west of Banff, Alberta. The mine location is shown in more detail in Plate 2, which also shows that the area is bounded to the north and east by Banff National Park and to the west by Kootenay National Park.

The economy of the Kootenays is heavily dependent upon resource-based industries, the main industries being mining, forestry and agriculture. The Kootenay region is the chief mining area of British Columbia. Lead and zinc ores are by far the most abundant, although many other minerals, including silver, gold, copper coal, magnesite, tungsten, barite and gymsum are also present in commercially attractive grades and quantities. Cominco's Sullivan Concentrator at Kimberley is one of the world's largest lead, silver and zinc complexes and Kaiser Coal maintains a large coal mining operation in Fernie.

The local wood products industry is supported by the medium to heavy density forest cover which is typical in most areas of the Kootenays up to the 5 or 6,000 foot level. Agriculture has also been a relatively important industry. Of the 560,000 acres of potentially arable land available in the larger valleys, about 300,000 acres are presently farmed.

A gradual decline in the early mining boom, extraction of the more easily accessible forests, destruction of timber through fire, and lagging interest in agriculture produced a loss of population in the years just prior to World War II. Since that time, however, mining and mineral-processing have expanded and both agriculture and forestry have been stimulated by improvements in access and technology. Today, the recreational opportunities afforded by the Kootenays are attracting increasing numbers of hunters, fishermen, campers, photographers, etc., who are also helping to stimulate the economy.

1.2 Physiography

The proposed mine site is located at an altitude of approximately 5,000 feet on the lower southwest slope of Eon Mountain. The valleys surrounding the mountain form part of the Rocky Mountain Trench which parallels the western flank of the Rocky Mountains. The British Columbia/Alberta border, about 7 miles to the east, coincides with the Continental Divide which in Canada separates waters draining into the Pacific Ocean from those flowing toward the Arctic Ocean or Hudson Bay.

The topography consists of a series of north-south mountain ranges separated by deep valleys containing large rivers and lakes. The floors of these valleys generally lie between 1,300 and 3,000 feet in elevation while the mountain peaks and ridges frequently rise above 7,500 feet. The mountains are a rich source of timber, minerals, and wildlife, but are also major obstacles to rail and highway transportation.

The southern slope of Mount Eon is flanked by the valleys of the Mitchell River to the west and the Cross River to the east. At the confluence of the rivers, about 2 miles south of the mine site, the elevation of the valley floor is about 4,500 feet. The Cross River joins the Kootenay River at the southeastern tip of Kootenay National Park, a distance of approximately 14 miles from the site. The Kootenay River has its source near the north end of Kootenay National Park, flows at a gentle gradient southward between ranges of the Rocky Mountains and swings westward at Canal Flats, into the Columbia Valley. From the confluence with the Cross River to Canal Flats, a distance of approximately 55 miles, the Kootenay River drops from an elevation of 3,900 feet to an elevation of approximately 2,650 feet.

Comparatively flat flood plains fringe the rivers and are up to a mile in width on the Kootenay. The flood plains are bordered by rolling benches of glacial till in which coarse, excessively drained silts, sands and gravel predominate. The area is designated as Zone 2, moderate damage for earthquake intensity. From patches of swamp and sedge meadow fringing the rivers, vegetation grades upward into a mixed covering of native grasses and trees. The forest cover is a light open mixture of Douglas fir, white spruce, yellow pine, larch, lodge-pole pine and aspen in varying proportions.

In the past, large areas were heavily logged, and this, combined with the effects of forest fires and overgrazing, modified the original ecological balance of these areas.

1.3 Climate

Although the Kootenays are more than 200 miles inland from the Pacific Ocean, they benefit from the residual moisture and mildness of eastward circulating maritime air-masses. Seasonal influences of cold, dry air from Northern Canada or of hot, dry air from the southwestern United States complicate the climatic patterns.

In broad terms, the area experiences warm, dry summers, comparatively cold, moist winters, and low annual precipitation. July temperatures average 62° to 64° whereas January means are 13° to 19°F. The frost-free period averages less than 100 days and the extreme temperatures range from a high of 95°F. to a low of 40°F.

The annual precipitation is generally less than 15 inches, but this rises sharply on exposed upland slopes. March, April and May are months of low rainfall, while June is a month of peak or mean-peak precipitation for the year. However, total rainfall during June, July and August averages only 3½ to 4½ inches.

1.4 Current Access

At present, there are two vehicular routes leading into the mine site. The shortest route starts at Radium Hot Springs and initially follows Provincial Highway 93 through Kootenay National Park. Access through the park is restricted to light vehicles and this road could not be used by ore trucks or other heavy equipment.

At a point about 11 miles northeast of Radium, a gravel road known as Settler's Road leaves Highway 93 and leads south-southeast along the valley of the Kootenay River. After 8 miles, the road crosses a wooden bridge over the Kootenay River and turns northeast to follow the valley of the Cross River. From this point to the mine site, a distance of approximately 14 miles, an existing logging road has been up-graded by bulldozer sufficiently to allow vehicular traffic. The total distance from the property to the Canadian Pacific Railway station at Radium is 36 miles.

More recently, the Kootenay valley has been opened up by a network of logging roads which extends from Canal Flats up to Settler's Road at the southeast boundary of Kootenay National Park. The quality of these roads varies from single lane dirt trail to relatively high speed two lane gravel surface road. About 20 miles north of Canal Flats a bridge with a design loading of 30 tons crosses the Kootenay River. The distance by road from Canal Flats to the mine site is approximately 60 miles.

1.5 History of Claims and Exploration

As a result of field work during the summer of 1965, Mr. G.B. Leech of the Geological Survey of Canada reported that magnesite occurred in apparently major quantities on the west flank of Mount Brussilof. Grab samples were collected and analyzed with an X-ray diffractometer. This initial testing indicated that the bulk of the material was of high quality.

New Jersey Zinc Exploration Company (Canada) Limited subsequently initiated a substantial claim-staking program. This was followed by a diamond drill core program in 1968, which was restricted to a relatively small area on the south slope of Mount Eon. In time, the company allowed most of the claims to lapse, retaining only six contiguous claims in the area of their drilling program.

In the summer of 1968, Mr. P. Roy Swainson and associates of Calgary staked thirty-six claims which surrounded the New Jersey Zinc claims and which also covered the more accessible magnesite beds on the lower western slopes of Mount Brussilof. Baykal Minerals Limited have subsequently staked or acquired an additional 308 claims in the area.

In May 1969, an agreement was signed between Baykal Minerals Limited and Swainson and associates. Mr. Orhan Baykal, of Baykal Minerals Limited, made a geological interpretation of the area from aerial photographs and commissioned a preliminary geological field investigation and report under the supervision of Dr. John D. Godfrey, P. Geol., of Edmonton. This helicopter supported investigation covered one zone of magnesite-bearing strata over a strike length of approximately a mile on the west flank of Mount Brussilof. Based on the chip sampling carried out and detailed examination of the area, the report concludes that the "visible" indicated tonnage of good to high grade magnesite rock could be expected to be in the order of 15.6 million tons.

In October 1969, a staff geologist from Imperial Oil Enterprises Limited visited Mount Brussilof and expressed satisfaction with the tonnage and grade possibilities as indicated at that time. Subsequent investigations into the marketing situation for the magnesite apparently did not yield sufficient encouragement and further interest in developing the property by Imperial Oil was terminated.

An agreement was signed in March 1970, which gave Baykal Minerals Limited the right to explore and develop the six claims held by New Jersey Zinc in exchange for a royalty to be paid on the product extracted. The geological and chemical data from the New Jersey drilling and testing program was turned over to Baykal Minerals Limited.

The present access road to the site was cut by bulldozer in the late fall of 1969 and improved in the spring of 1970. In late spring of 1970 a campsite was established for the extensive drilling and mapping program directed by Acres.

In addition to the six New Jersey Zinc claims, 338 claims have been registered and are in good standing to date in the name of Baykal Minerals Limited. The known magnesite occurrences are considered to be well protected and the likeliest regional extensions have been covered. The waters of the Mitchell and Cross Rivers are included within the boundaries of the staked ground. Local access to the property from the west via the valleys of the Cross and Mitchell Rivers appears to be adequately assured by the property boundaries.

2. PLANT SITE SELECTION

2.1 Factors Affecting Selection

The number of potential processing plant sites is limited by the availability of access routes to the area. The first task, therefore, was to identify those routes which could be used and then to evaluate those factors for which the cost varied with location. By comparing the variable capital and operating costs for each alternative, the location resulting in the lowest long-term overall cost can then be identified.

The economics of plant location are also affected by the plant size. Accordingly, economic analyses were performed for plant capacities of 75,000, 150,000 and 225,000 tons of product annually. Initial production will be somewhere within this range and by examining the economics of location over a range of capacities a more objective decision on the best location can be made.

2.1.1 Access Routes to the Mine

The access route must be such that transportation of men and materials to the mine and of product to the market can be accomplished as reliably and economically as possible.

The nature of the product and the location of the property in relation to the potential markets indicates that the bulk of the material will travel by rail either to Vancouver, British Columbia, for shipping worldwide, or direct to North American markets. The most direct access to a railway loading point is important. Extension of existing railway lines to the vicinity of the mine site is limited by topography to a possible route from Canal Flats up the valleys of the Kootenay and Cross Rivers, a distance of some 60 miles. Because of the high costs associated with a spur line of this length over difficult terrain, no further consideration is given to this alternative.

Examination of the topography surrounding the mine site reveals five topographically possible road routes from the mine site to railhead. The railheads in question were Sawback and Canmore in Alberta to the northeast, and Radium Hot Springs, Invermere and Canal Flats in British Columbia to the southwest. Roads to Sawback and Canmore would have to pass through Banff National Park and traverse mountain passes up to 6,900 feet. The road to Radium Hot Springs would pass through Kootenay National Park and over a mountain pass at an elevation of approximately 4,500 feet.

The Federal Department of Indian Affairs and Northern Development, when approached regarding the possibility of constructing an access route through one of the National Parks, made it clear that such access would be contrary to National Parks policy because of the detrimental effect on natural park values. A letter from the Acting Regional Director of the Department of Indian Affairs and Northern Development states:

"Our branch headquarters in Ottawa has now informed us that they would not be prepared to entertain any request for road or pipeline access through a National Park to facilitate the operation of a commercial enterprise located outside the Park boundary."

Because of the shorter road distances involved (28 miles to Canmore versus 62 miles to Canal Flats) and lower fuel costs in Canmore, considerable cost savings might be realized if permission was obtained to construct a route through a National Park. However, as this permission was not given and in view of current public and government concern with protection of the environment, no further consideration is given in the feasibility study to the use of routes through National Parks. Prior to project implementation, however, discussions can be reopened with the Federal Government to determine whether or not policy changes might permit access to the mine through one of the National Parks.

Because of these restrictions, the analysis is confined to a comparison between road routes terminating at Invermere and Canal Flats. These two possibilities are included as variables in the larger problem of selecting the most suitable processing plant site.

2.1.2 Alternative Plant Locations Tested

In dead burning magnesite, a weight reduction of approximately 50 percent occurs when the volatile constituents are driven off. Processing at the mine site would thus appreciably reduce the amount of material to be transported from the site. However, the cost of supplying fuel at the mine could more than offset this saving, and for this reason the study of alternative sites for the processing plant is not limited to the mine site.

To minimize materials handling and storage facilities and associated costs, the processing plant should be located either at the mine site or at the railhead. This means that there are two possible routes and three possible plant locations to be examined:

	Railhead	Plant Location
Alternative 1	Canal Flats	Canal Flats
Alternative 2	Canal Flats	Mine Site
Alternative 3	Invermere	Invermere
Alternative 4	Invermere	Mine Site

2.1.3 Calcining Fuel Costs

Five possible types of fuel were considered for each location, namely: oil; natural gas; coal; propane; butane. For each plant location, the fuel with the lowest annual delivered cost was selected.

2.1.4 Other Considerations

A number of intangible factors, although unquantifiable in the cost comparison, must be taken into account when making the final decision as to plant location.

The costs of attracting and keeping a skilled and experienced labour force at the relatively remote mine site rather than in an established community cannot readily be assessed. There is no doubt, however, that amenities possessed by existing communities, such as schools, churches, and shopping and entertainment facilities, could only be duplicated in a new community at very high cost. From this point of view, therefore, a plant located at Invermere or Canal Flats would be preferable to a plant at the mine site.

Canal Flats is a slightly larger community than Invermere, and it would probably be easier and more economic to attract and accommodate the required labour force in Canal Flats. The existing labour force is primarily engaged in the forest industry, both in logging and in a large sawmill complex.

Apart from differences in the cost of building and maintaining a suitable road to the mine site from Canal Flats and Invermere, there is some doubt that the Teggart Pass section of the Invermere route, which rises to 6,300 feet, could be kept open at all times in the winter. This is another intangible factor which favours the Canal Flats railhead.

2.2 Method of Comparison

In determining the most suitable route and plant location, consideration must be given to both capital and operating cost factors which vary with location.

Capital costs of the various alternatives can be compared on a total investment basis, and can also be converted to annual equivalents by amortizing them at a suitable interest rate over ten years. In this way it is possible to derive the cumulative variable amortized capital and annual operating costs for each alternative.

Certain capital cost items such as initial mine preparation, mining equipment and mine access roads will not vary significantly with the plant and route location and are not included in the comparison. Some of the variable capital cost items estimated and totalled for each alternative are: plant shop and mine site; roads to mine site; mobile equipment; processing plant and equipment; land at the railhead; handling and storage facilities at the railhead; power supply; accommodation; offices; services.

Some operating cost items such as materials, labour and maintenance of equipment for mining operations, materials labour and maintenance of plant and equipment for processing operations, and royalties are not affected by the route and plant location and therefore do not affect the choice. Those operating cost items which could vary with location include: fuel for the kiln operation; transportation to and from the railhead; product storage and loading at the railhead; rail transportation; power; mine overhead; office and camp staff and operations; road maintenance and personnel transportation.

2.3 Findings and Recommendations

For the purposes of this feasibility study, certain conservative assumptions must be made. The individual assumptions, locations, designs and costs cannot all be optimized within the scope of such a study. If feasibility is demonstrated under these conditions, then further design and study can only improve the picture. The recommendations in this section, therefore, are for the purpose of the feasibility study and should not be taken as the final plant site selection.

The design of the processing plant is such that the production capacity may be increased in increments to meet the demand. This being the case, more weight was placed on the economics of location for the larger capacity plant than on the first stage which is expected to operate for only a short time.

For a plant with a capacity in the order of 75,000 tons of product per year, the Canal Flats route would result in savings of between 2 and 3 dollars per ton of product, depending upon the location of the plant. For this scale of development, a plant located at the mine site would have a slight cost advantage over the Canal Flats location. However, consideration of some of the unquantifiable factors mentioned in Section 2.1.4 would put the Canal Flats plant location in a favoured position.

For plant capacities of 150,000 tons of product per year and greater, the plant should definitely be located at the railhead. This would produce cost savings in the order of 2.50 dollars per ton of product. Total costs for the two routes are almost identical, and at this scale, quantifiable cost factors show no difference between a plant located at Canal Flats or at Invermere. However, at the Invermere location coal was the least expensive kilning fuel, while at Canal Flats natural gas was the most economic. Because of the superiority of natural gas over coal as the kilning fuel and because of the other advantages of Canal Flats over Invermere listed in Section 2.1.4, the Canal Flats route with the plant at the railhead is considered to be the most satisfactory alternative for a larger capacity plant.

On the basis of the above findings, Acres recommend that the route to the mine site be from Canal Flats and that the processing plant be located at a railhead in the vicinity of Canal Flats. Preliminary investigations indicate that suitable land is available in the area.

With only one kiln constructed and in operation, the most economical fuel at Canal Flats is Bunker C oil, unless the gas company can be persuaded to construct the pipeline on the basis of the proposed staging of fuel requirements. To provide this natural gas at Canal Flats involves the construction of a pipeline from Skookumchuk, a distance of approximately 18 miles.

After the addition of the second kiln, the most economic fuel is natural gas. This should be used in a dual system which allows combustion of natural gas and/or Bunker C oil. This permits maximum operational flexibility and takes advantage of improved burning characteristics inherent in a 90 percent natural gas/10 percent oil mix.

VII GEOLOGY AND ORE RESERVES

CONTENTS

		Page
1.	GEOLOGY	39
	1.1 Introduction	
	1.2 Stratigraphy	
	1.3 Structural Geology	
	1.4 Detailed Geology	
	1.4.1 Mount Eon	
	1.4.2 Mount Brussilof	
2.	ORE RESERVES	43
	2.1 Diamond Drilling Program	
	2.1.1 Diamond Drilling and Mapping	
	2.1.2 Assay Procedure	
	2.1.3 Drilling Results	
	2.1.4 Surveys	
	2.2 Proven and Probable Ore	
	2.2.1 Definition of Ore	
	2.2.2 Tonnage Calculations	
	2.3 Possible Ore	
	2.3.1 Mount Eon	
	2.3.2 Mount Brussilof	
	2.3.3 Cross River	
	2.4 Summary	
	2.5 Bibliography	
	2.6 Diamond Drill Core Logs	

VII GEOLOGY AND ORE RESERVES

TABLES

		Page
VII - 1	Geological Formations in Claim Area	40
VII - 2	Spectrographic Analyses of Two Mount Eon Samples	45
VII - 3	Mount Eon Deposit - Weighted Average Drill Hole Grades before Iron Beneficiation	47
VII - 4	Mount Eon Deposit - Summary of Material in Diamond Drill Holes	48
VII - 5	Mount Eon Deposit - Ore Reserves	52
VII - 6	Proven and Probable Tonnage and Average Assays after Iron Beneficiation at Specified Cut-off Grades	53

FIGURES

VII - 1	Iron Distribution - Mount Eon Deposit	50

VII GEOLOGY AND ORE RESERVES

1. GEOLOGY

1.1 Introduction

This section is based on interpretation of published and private reports on the project area, and on visits made to the property during the 1970 field season by both an Acres geologist and mining engineer. During these visits examinations were made of the stratigraphic sections on Mount Eon and Mount Brussilof and of diamond drill core from the Mount Eon deposit and arrangements were made for additional drilling to be carried out. A bibliography of geological reports referred to is included as sub-section 2.5.

In 1966 attention was drawn to the high grade magnesite on the west side of Mount Brussilof by G. B. Leech of the Geological Survey of Canada. New Jersey Zinc staked claims, mapped and performed limited drilling on Mount Eon. All but six New Jersey Zinc claims were subsequently allowed to lapse and 338 mineral claims were located over and around the New Jersey claims by the present title holders. These claims have been checked at the Mine Recorders Office and found to be in good standing.

During 1969 John D. Godfrey wrote a report based on detailed mapping and sampling of magnesite occurrences outcropping on the west side of Mount Brussilof. In 1970 an access road was constructed and a program of topographic and geologic mapping, drilling and sampling was carried out on the Mount Eon deposit to establish grade and tonnage on that portion of the property. Mount Eon was chosen as the exploration target because of the relative ease of access for drilling.

There is some question as to the geological origin of this magnesite deposit, but whether it is hydrothermal or sedimentary has no immediate bearing on the feasibility of developing a commercial ore body to production. For the purpose of this study and based on the work carried out in the field, it is sufficient to note that, although the magnesite mass conforms to bedding planes and varies in thickness similarly to other strata which reflect cycles of sedimentary deposition, veins and veinlets of magnesite forming solution breccias in grey limestone are also present on Mount Brussilof. The present stratigraphic position of the magnesite mass may therefore be the result of preferential bedding plane replacement.

In summary, there is evidence to support a hydrothermal replacement derivation despite the stratified nature of the magnesite.

1.2 Stratigraphy

The magnesite occurs within the Cathedral Formation of Middle Cambrian age, and this formation underlies most of the claimed area. The following table summarizes the significant stratigraphy in this area.

TABLE VII - 1

GEOLOGICAL FORMATIONS IN CLAIM AREA

Age	Lithology	Thickness feet
RECENT	Coarse-to-fine detrital	0-200
MIDDLE CAMBRIAN		
Stephen Formation	Siliceous shales, and thin-bedded limestone	90-350
Cathedral Formation	Massive dolomite and limestone (magnesite locally)	800-1,900
Mount Whyte Formation	Shales and limestone; some sandstone	0-600
MIDDLE AND LOWER CAMBRIAN		
St. Piran Formation	Slate, phyllite and orthoquartzite	1,200-2,000

Source: D. G. Cook
Acres

The preceding formations are believed to have conformable relations one to the other and represent portions of the five depositional cycles which took place during Cambrian time. The St. Piran Formation has been included in Middle

Cambrian as well as in Lower Cambrian as this formation is now thought to represent, in part, a lateral facies equivalent of the Middle Cambrian east of Mitchell River.

(a) St. Piran Formation

In the lower part of this formation, hard 1/2 to 2 foot beds of brown-weathered orthoquartzite contain fine grained white quartz set in a siliceous matrix. Above the orthoquartzites, varied coloured slates are thin-bedded with phyllite. On the east face of Mount Docking, across the Mitchell River from Mount Brussilof, exposures of the St. Piran Formation are present.

(b) Mount Whyte Formation

The Mount Whyte Formation consists of a thin-bedded, dark grey siliceous shale and fine grained, dark grey, argillaceous limestone which changes laterally to fine grained sand dolomite or calcareous sandstone.

(c) Cathedral Formation

The formation consists of massive, thick sections of fine to coarse grained, white to dark grey, argillaceous to siliceous dolomite, and usually coarse grained, cream to buff, interlayered magnesite zones.

The magnesite is usually coarse grained and weathers to a coarse, granular, sand talus or to a rough, granular, etched surface where the rock is found in place.

(d) Mount Stephen Formation

The Mount Stephen Formation is composed of a series of greenish, siliceous shales and thinly bedded, flaggy, grey limestone with minor grey shale partings. The formation is exposed on Mount Eon and overlies the Cathedral Formation.

(e) Recent

The diamond drill hole logs indicate that the overburden on Mount Eon is from 0 to 35 feet thick. The Mitchell River valley floor shows no outcrop and the overburden is believed to range from 0 to 200 feet thick.

1.3 Structural Geology

Cook has traced unbroken strata from one facies to the other at four localities within the Middle-Lower Cambrian formations, and has questioned the validity of the Stephen-Dennis fault as being a continuous structural break. Some thrust faults occur in the area of the facies change, and transverse faults cut the eastern facies and trend into the zone of thrusts and folds. The concept of a fault along Mitchell Creek, therefore, would have to be substantiated by detailed lithological and structural mapping before being accepted. Strata in the area of interest are folded along a northwest-southeast axis with related cross folding and faulting present on Mount Brussilof. The magnesite mass on Mount Eon dips approximately 45 degrees to the southwest and is relatively undisturbed structurally. The faults on Mount Brussilof, as mapped by Godfrey, do not represent a significant mining problem.

1.4 Detailed Geology

The ensuing description is concerned with the magnesite sections within the Cathedral Formation of Middle Cambrian Age.

1.4.1 Mount Eon

On Mount Eon the magnesite mass has a strike of N.30°W. and an average dip of 40 degrees to the southwest. The magnesite is relatively undisturbed structurally although some jointing is evident. A generalized section from the drill logs would be as follows:

ROCK	DESCRIPTION	TRUE THICKNESS feet
Overburden	Unconsolidated brown gravel usually with much clay or shale in the matrix.	0-35
*Limestone	Grey, occasionally siliceous, finely banded.	
Magnesite	Usually coarse grained, elongated crystals, grey to white with some pink sections; fine cubes of pyrite are common as irregular 1 inch to 2 inch patches and as fine bands or along	0-350+

ROCK	DESCRIPTION	TRUE THICKNESS feet
Magnesite (contd.)	fractures; impurities in the form of dolomite and/or siliceous sections are present within the magnesite as occasional laterally discontinuous sections 5 feet or slightly more in thickness.	
Dolomite, Argillite	The pure magnesite usually terminates in dolomite or argillite with varying percentages of siliceous and calcereous mixtures. There are no obvious marker beds in this section although to the down dip side sharply contacted quartzite has been logged in the drill core beneath the magnesite mass.	110

* The magnesite is exposed over a large area and the grey St. Stephen limestone only covers the magnesite up the slope of Mount Eon away from the immediate area of interest.

1.4.2 Mount Brussilof

Extensive sampling carried out along the west face of Mount Brussilof has shown that the stratigraphic zone of interest is a 250 foot section of which 150 feet of the section is composed of magnesite and dolomite with two average 50 foot dolomite interbeds. Rocks along Bear Creek underlying the magnesite consist of limestone, argillite, orthoquartzite and phyllite and these strata dip 20 degrees to the west away from the mountain. Evidence of faulting can be seen along Bear Creek. The magnesite is fine to coarse grained, white to buff, massive in nature, and has been tested by sampling and assaying along a mile of strike length.

2. ORE RESERVES

2.1 Diamond Drilling Program

2.1.1 Diamond Drilling and Mapping

Five holes were drilled by New Jersey Zinc in the Mount Eon deposit in 1968. These are designated by prefix NJ in subsequent tables and Plate 3. Eleven holes, denoted by the prefix B, were drilled in August 1970 on approximately 200 foot centres. This diamond drilling and topographic mapping was supervised and logged by Eugene Stary, P.Eng., of Baykal Minerals Limited in discussion with, and with the agreement of, Acres. The 1970 logs are presented in sub-section 2.6. Prior to assaying, the drill core was composited in 20 foot sections unless there was a change in mineralization, in which case shorter lengths were composited for assay.

2.1.2 Assay Procedure

All assays were done by Bondar-Clegg and Company, North Vancouver, B.C. The first drill cores were assayed on both a natural and a calcined basis. The calcined assays were more consistent and indicative of final product and a decision was therefore made to run all assays on this basis. The natural assay may be approximated by multiplying the reported result by the factor $\frac{100 - L.O.I.}{100}$

The following assay procedures were used:

L.O.I.	- Weight loss on calcining for 2 hours at 1000 to 1100^{o}C.
MgO	- Titration with ethylene diamine tetra acetate after comparison of results obtained using phosphate precipitation procedure.
CaO	- Atomic absorption spectrophotometry based on comparison with matrix standards.
Al_2O_3	- Atomic absorption spectrophotometry based on comparison with matrix standards.
Fe_2O_3	- Atomic absorption spectrophotometry based on comparison with matrix standards.
SiO_2	- Standard gravimetric chemical analysis.
HF Insolubles	- Standard gravimetric chemical analysis.

The possibility of additional elements being present in significant quantities was investigated by performing spectrographic analyses on two typical samples. These results, shown in Table VII - 2, indicate that significant

TABLE VII - 2

SPECTROGRAPHIC ANALYSES OF TWO MOUNT EON SAMPLES

Sample	#1	#2
Aluminium	G	G
Antimony	ND	ND
Arsenic	ND	ND
Barium	Internal	Standard
Beryllium	L 0.001	L 0.001
Bismuth	ND	ND
Boron	L 0.001	0.001
Cadmium	ND	ND
Calcium	G	G
Chromium	L 0.001	L 0.001
Cobalt	ND	ND
Copper	0.001	0.003
Gallium	ND	ND
Gold	ND	ND
Iron	0.05	G
Lead	L 0.001	L 0.001
Magnesium	Major	Major
Manganese	0.007	0.01
Molybdenum	L 0.001	L 0.001
Columbium	ND	ND
Nickel	L 0.001	0.001
Silicon	G	G
Silver	L 0.001	L 0.001
Strontium	0.01	0.005
Sodium	Trace	Trace
Tin	ND	ND
Titanium	0.007	0.01
Tungsten	ND	ND
Vanadium	0.01	0.01
Zinc	ND	ND

All results are expressed as percent by weight.
Major: above normal spectrographic range
G : greater than 0.1 percent
ND : not detected
L : less than

quantities of other elements are not present. The fact that the sum of the weighted average assays totals 99.3 percent rather than 100 percent is attributable primarily to MgO assay variation, the presence of sulphates and minute quantities of other elements not assayed for in the calcined product.

For the following reasons it is estimated that the true average MgO assay of the logged drill core is closer to 95.1 percent than 94.66 percent:

(a) The sum of the recorded assays totals 99.3 percent, indicating that the assays should be adjusted upward.

(b) The spectrographic analysis shown in Table VII-2 indicates that the weight of components present, other than those shown in Table VII - 3, is insignificant.

(c) The assay procedures used for Fe_2O_3, Al_2O_3, CaO and SiO_2 are considered to be more accurate than that used for MgO.

(d) As MgO predominates in this material, most of the assay corrections should be applied to this component.

2.1.3 Drilling Results

Table VII - 4 summarizes the drilling results obtained on the Mount Eon deposit. Table VII - 3 illustrates the average grade of the ore in the drill holes. It was evident from both the drill core and Table VII - 3 assays that there is considerable variation in the iron content of the magnesite. Logging of the drill core and mineralogical examination indicated that most of the iron occurs as pyrite.

2.1.4 Surveys

Surface topography was determined by air photo interpretation and by ground survey using a pocket altimeter with readings every 100 feet along grid lines. The two surveys agree closely and for tonnage calculations the contours obtained by altimeter have been used. Correlation of aerial photo and altimeter contours indicate the actual tonnage may be slightly higher than calculated, due to the fact that the cliff surface on Plate 3, sections 5, 6 and 7 is higher than plotted.

TABLE VII - 3

MOUNT EON DEPOSIT - WEIGHTED AVERAGE DRILL HOLE GRADES* BEFORE IRON BENEFICIATION

Hole No.	Total length of ore ft.	MgO %	CaO %	Fe_2O_3 %	Al_2O_3 %	SiO_2 %	L.O.I. %	HF Insol. %
NJ-1	149	96.12	1.97	0.47	0.44	0.63	51.82	0.32
NJ-2	200	95.74	2.16	0.53	0.48	0.70	51.49	0.36
NJ-3	200	95.85	2.32	0.65	0.36	0.52	51.48	0.21
B-1	50	96.66	1.72	0.54	0.26	0.61	51.58	0.22
B-2	68	93.12	2.26	3.11	0.45	1.00	51.36	0.26
B-3	40	96.00	2.50	0.78	0.18	1.38	51.10	0.21
B-5	95	93.32	2.18	1.98	0.42	1.73	51.12	0.53
B-6	283	95.37	2.05	2.00	0.23	0.43	51.38	0.15
B-7	323	96.12	2.10	0.50	0.16	0.24	51.74	0.09
B-8	249	93.06	1.88	3.94	0.41	0.47	51.30	0.48
B-9	432	96.26	1.97	1.01	0.17	0.33	51.48	0.16
B-10	357	93.54	2.13	1.78	0.30	0.95	50.77	0.15
B-11	219	93.96	2.06	1.08	0.59	1.30	50.70	0.28
	2,665							
	Weighted Average	94.66	2.07	1.36	0.32	0.63	51.32	0.23

* Weighted average grade of diamond drill cores calcined at 1100-1200°C. for two hours.

TABLE VII - 4

MOUNT EON DEPOSIT - SUMMARY OF MATERIAL IN DIAMOND DRILL HOLES*

Hole No.	Drilled footage	Over-burden	Waste on surface	Ore*	Included waste	Footwall contamination	Footwall waste
NJ-1	178	0	0	149	0	7	22
NJ-2	291	0	0	200	50	15	26
NJ-3	288	2	0	200	20	40	26
B-1	201	27	10	50	0	26	88
B-2	163	35	0	68	0	34	26
B-3	135	25	0	40	0	20	50
B-4	110	24	0	0	0	0	86
B-5	239	30	0	95	64	21	29
B-6	338	22	0	283	5	10	18
B-7	434	5	20	323	0	20	66
B-8	411	6	40	249	66	6	44
B-9	485	13	0	432	20	10	10
B-10	465	4	20	357	51	21	12
B-11	348	5	0	219	90	0	34
	4,086	198	90	2,665	366	230	537

* Ore is defined in Appendix D, Section 2.2.1.

2.2 Proven and Probable Ore

2.2.1 Definition of Ore

For feasibility purposes, ore is considered to be material containing less than 6 percent effective contaminants, excluding Fe_2O_3, on an ignited product basis. Effective contaminants are taken to be equivalent to the sum of:

1½ x CaO assay
1 x Al_2O_3 assay
1 x HF insolubles assay

Laboratory tests have demonstrated that the iron and silica fractions of the ore can be extracted by flotation, and they are therefore not classified as effective contaminants.

The weighted averages of the individual and combined diamond drill holes are shown in Table VII - 3. A summary of ore and waste material based on the preceding definition is presented in Table VII - 4. A curve illustrating the relationship between the diamond drill iron assays and the percentage of drill footages is presented in Figure VII - 1.

Laboratory tests have demonstrated that material assaying 8 percent Fe_2O_3 on an ignited product basis can be beneficiated to 0.8 percent Fe_2O_3 by flotation. As the latter figure is well within the tolerances of a high grade product, it is apparent that less than 5 percent of material drilled would have to be rejected on the basis of iron content.

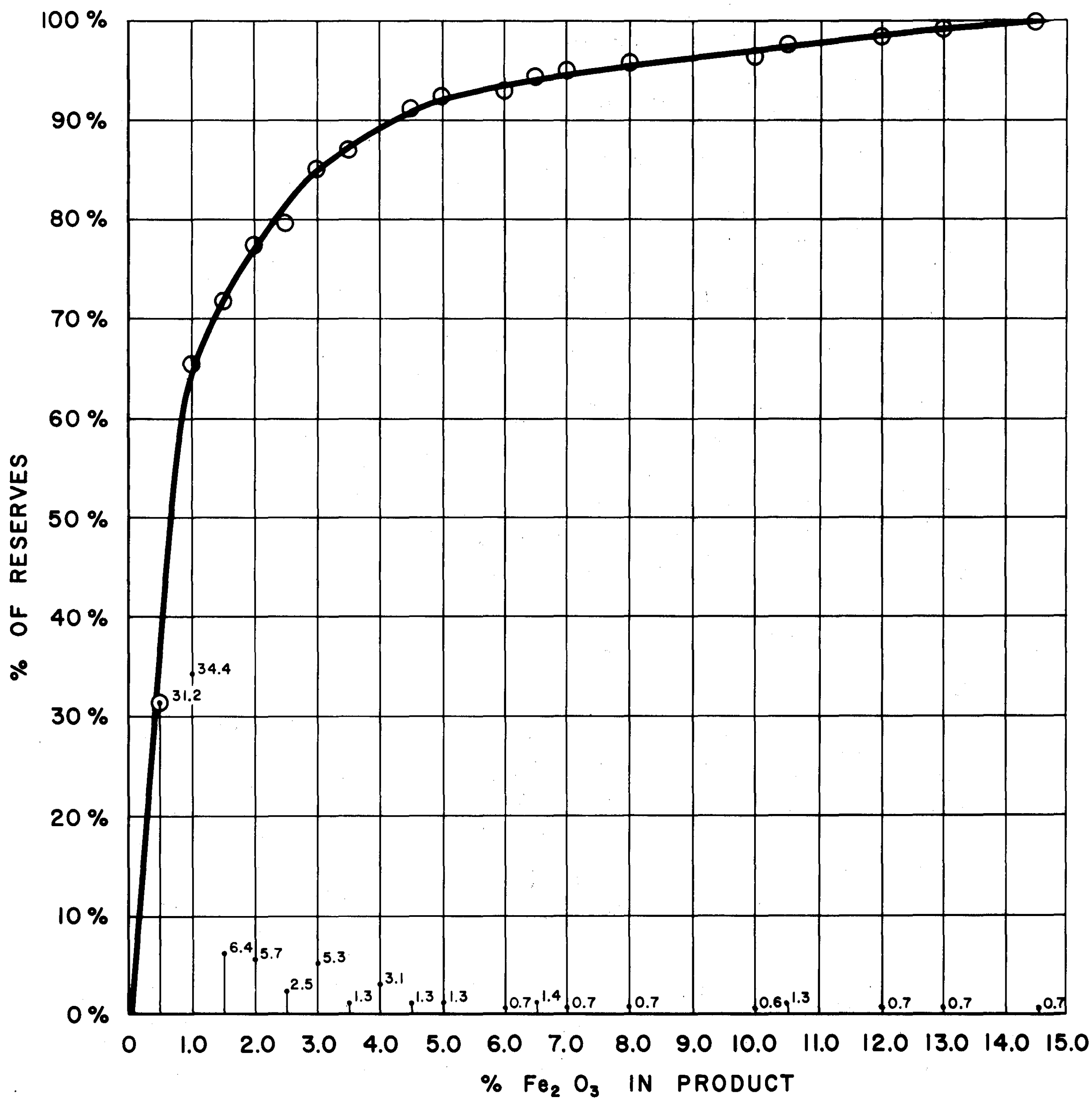
2.2.2 Tonnage Calculations

Proven ore is taken to be that material in the Mount Eon deposit within the perimeter of the drill holes noted in Plate 2 and 200 feet horizontally beyond this perimeter, up and down dip and on strike, excluding 20 percent of the total as included waste and non-recovered ore.

Probable ore is that material within 200 horizontal feet of the proven ore outline, less 20 percent as above.

The proven and probable ore outlines were plotted in plan and then in section in Plate 3. The sections were planimetered and the volume between sections calculated by multiplying the average area at two adjacent sections by the distance

IRON DISTRIBUTION
MOUNT EON DEPOSIT



NOTES

1. CURVE SHOWS % OF UNBENEFICIATED RESERVES CONTAINING LESS THAN INDICATED % Fe_2O_3
2. % Fe_2O_3 IS EXPRESSED ON IGNITED PRODUCT BASIS

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FIGURE VII-1

between them. The end volumes were calculated as half cylinders. Volume was divided by a tonnage factor of 12 cubic feet per ton. Table VII - 5 summarizes the proven and probable Mount Eon reserves established by this drilling program.

Mr. G. Stary has examined the proven and probable reserves at various cut-off grades and his findings are summarized in Table VII - 6. The effect of iron beneficiation has been considered in determining the reserves and grades shown in Table VII - 5, whereas the preceding tables in this section are based on actual drill core assays. For this reason Table VII - 5 should not be compared directly against preceding data. However, it demonstrates that the proven and probable reserves presently identified on Mount Eon are sufficient to sustain a 150,000 tons per year operation, producing:-

Better than 90 percent MgO product for 65 years or
Better than 95 percent MgO product for 48 years or
Better than 97 percent MgO product for 13 years.

2.3 Possible Ore

2.3.1 Mount Eon

Possible tonnages of high grade magnesite on Mount Eon, subject to confirmation of lateral and down dip continuity, based on surface mapping and down dip projections, are conservatively estimated to be 15 million short tons.

2.3.2 Mount Brussilof

Sampling and assaying during 1969 on Mount Brussilof indicated the presence of a magnesite deposit having an aggregate 90 foot thickness, 5.200 foot strike length and 400 dip length. On this basis the estimated possible reserves are estimated to be 15.6 million short tons of high grade magnesite.

2.3.3 Cross River

Several outcrops of coarse grained magnesite have been examined north of Cross River and south of the area on Mount Brussilof described by Godfrey. No detailed work has been carried out in this area but, despite extensive overburden, it appears that approximately 15 million tons of high grade magnesite are present which would be available for future mining.

TABLE VII - 5

MOUNT EON DEPOSIT - ORE* RESERVES

	Section block	Tons proven	Tons probable
N.W. of section	1	2,782,000	6,358,000
Between	1 - 2	3,425,000	2,375,000
Between	2 - 3	3,691,000	2,275,000
Between	3 - 4	3,608,000	1,833,000
Between	4 - 5	2,900,000	1,292,000
Between	5 - 6	1,375,000	616,000
Between	6 - 7	338,000	-
S.E. of section	7	383,000	-
SUB TOTALS		18,502,000	14,749,000
Less 20% (included waste, non-recovery)		3,700,000	2,950,000
TOTALS		14,802,000	11,799,000

* Ore is defined in Section VII, 2.2.1.

TABLE VII - 6

PROVEN AND PROBABLE TONNAGE AND AVERAGE ASSAYS AFTER IRON BENEFICIATION AT SPECIFIED CUT-OFF GRADES

Cut-off MgO grade	+90%	+94%	+95%	+96%	+97%	+98%
Proven and probable ore tonnage (1,000 tons)	26,601	23,084	19,791	12,819	5,323	399
Weighted average assays:						
MgO %	95.51	96.08	96.35	96.84	97.41	98.10
CaO %	2.19	2.01	1.94	1.78	1.54	
Fe_2O_3 %	0.70	0.69	0.64	0.55	0.49	
Al_2O_3 %	0.54	0.38	0.33	0.24	0.20	
SiO_2 %	0.52	0.42	0.37	0.30	0.23	
Duration in years of present reserves at mining rate of 412,000 TPY (150,000 TPY burnt product)	65	53	48	31	13	

Source: G. Stary, Baykal Minerals

2.4 Summary

Diamond drilling programs conducted on one of the three known deposits within the claim boundary have defined 14,802,000 proven tons and 11,799,000 tons of probable ore.

Additional tonnages of high grade magnesite in the possible ore category are conservatively estimated to be:

Mount Eon	15,000,000
Mount Brussilof	15,000,000
Cross River	15,000,000
Combined Possible Tonnage	45,000,000 tons

Considering that an annual mining rate of 412,000 tons would be required to produce 150,000 tons of dead burned magnesite annually, it is apparent that the presently proven and probable reserves would support an operation of this magnitude for 65 years. For the present feasibility study purposes it is therefore unnecessary to speculate on the extent of the possible reserves.

2.5 Bibliography

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2.6 Diamond Drill Core Logs

Pages 57 - 77 contain the core logs of the diamond drilling programs previously referred to.

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DIAMOND DRILL CORE LOG

DIAMOND DRILL HOLE NJ-1, -90° STARTED ____________

GRID LOCATION ____________ COMPLETED ____________

STARTING ELEVATION 4,700 FOOT LOGGED BY ____________

DEPTH 178 FOOT DATE LOGGED ____________

DRILLED BY ____________

CORE SECTION FROM	TO	DESCRIPTION	Est. Mg CO_3
0	2	CASING, Hi purity magnesite.	
2	101	Hi purity magnesite, coarse-grained, locally some li gysh blotches in pure whi magnesite due to fine py aggregs or other impurities. 30' & 89', 1/4" thick magnesite vnlts @ 30-35°. Vnlt walls bounded by thin fine py lining presumably as thin crusts along walls of pre-existing fracs later filled with magnesite. Some thin & irreg fine py vnlts @ 35° appear to occur in tight fracs along x'tal faces. These fracs are jts, opposite walls match.	0 - 130 --- >90%
101	120	Magnesite, li gysh blotches increasing in amount or higher py & CaO content.	101 - 120 --- ~85%
120	130	Magnesite, shows crude banding of li gysh pyritic bands and less pure magnesite especially between 128 & 130.	120 - 130 --- ~80%
130	140	Magnesite, increasing crude banding structure @ 45°, rendering a more gysh appearance of the rk.	130 - 140 --- ~75%
140	150	Ditto, sli more gysh.	140 - 150 --- ~70%
150	157	Ditto, increasing CaCO content - as finer grained unreplaced (?) bands (45°) in m g gysn whi magnesite.	150 - 157 --- ~50%

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DIAMOND DRILL CORE LOG

DIAMOND DRILL HOLE NJ-1 (Cont'd.) STARTED ______

GRID LOCATION ______ COMPLETED ______

STARTING ELEVATION ______ FOOT LOGGED BY ______

DEPTH ______ FOOT DATE LOGGED ______

DRILLED BY ______

CORE SECTION FROM	TO	DESCRIPTION
		157 - Approx. contact, magnesite & ls. Gradation of magnesite to dk gy dolomitic ls within 2-3 inches.
157	178	Magnesitic ls, grading to dolomitic ls. Gen rk is impure ls composed of dk gy and li gy alternating bands, with dk gy component tending to be thicker bands @ 45°. Thin lenticular vnlts of magnesite in tight fracs almost perpendicular to appar bedding struct cut thru the ls, although apparently more numerous near the contact.
		END

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DIAMOND DRILL CORE LOG

DIAMOND DRILL HOLE NJ-2, -90° STARTED ______

GRID LOCATION ______ COMPLETED ______

STARTING ELEVATION 4,730 FOOT LOGGED BY ______

DEPTH 291 FOOT DATE LOGGED ______

DRILLED BY ______

CORE SECTION FROM	TO	DESCRIPTION	Est. Mg CO_3
0	220	Hi purity magnesite, c g, loc some li gysh blotches. Thin irreg vnlts fine py along opposite walls of healed fracs: @ 19' 1" @ 30°, 73' 1/2" @ 10°, & 82' 1" @ 10°	0 - 220 --- >90%
220	230	Magnesite, more gysh blotches	220 - 230 --- 90%
230	240	Ditto	230 - 240 --- 90%
240	250	Ditto, increasing	240 - 250 --- 85%
250	258	Dolomitic magnesite	250 - 258 --- 60%
258	265	Magnesitic dolomite, banded ls of alternating thicker dk gy f g massive ls @ li gy f g ls with crosscutting magnesite/pyrite vnlts @ 45° perpendicular, bedding attit @ 45°.	258 - 265 --- 50%
265	291	Dolomitic limestone, decreasing amount of magnesitic or dolomitic component to purer limestone.	265 - 291 --- 10%
		END	

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DIAMOND DRILL CORE LOG

DIAMOND DRILL HOLE NJ-3, -90° STARTED ______

GRID LOCATION ______ COMPLETED ______

STARTING ELEVATION 4,800 FOOT LOGGED BY ______

DEPTH 288 FOOT DATE LOGGED ______

DRILLED BY ______

CORE SECTION FROM	TO	DESCRIPTION	Est. Mg CO_3
0	2	OVBDN	
2	141	Hi puritiy magnesite, locally li gysh patches having some fine py specks. Sli buff color of rk along joint walls @ 76, 85, 100, 120 - due to weathering. 2 - 141 ---	90%
141	153	Magnesite with gradual increase of impurities towards 153. Loc present are irregular bands of li-gysh component: gy ls or other impurities in the rock @ 150. 141 - 153 ---	90%
153	230	Magnesite, some gysn masses but otherwise better grade than the previous section. 153 - 230 ---	90%
230	230.2	2-1/2" thick tabular inclus (?) @ 60° brnsh gy soapstone.	
230.2	234	Hi grade magnesite 230.2 - 234 ---	90%
234	234.3	4" thick tabular inclus (,) dk gy ls @ 60°.	
234.3	250	Low-grade magnesite 234.3 - 250 ---	80%
250	257	Dolomite: Apparent compositi nal gradation from magnesitic dolomite to dolomite. 250 - 257 ---	40%

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DIAMOND DRILL CORE LOG

DIAMOND DRILL HOLE NJ-3 (Cont'd.) STARTED ______

GRID LOCATION ______ COMPLETED ______

STARTING ELEVATION ______ FOOT LOGGED BY ______

DEPTH ______ FOOT DATE LOGGED ______

DRILLED BY ______

CORE SECTION		DESCRIPTION	Est. $MgCO_3$
FROM	TO		
257	260	Gradational contact dolomite and dolomitic ls. Irregular crude wavy li gy bands in dk gy ls.	257 - 260 --- 40%
260	270	Dolomitic ls. Irregular thin lenticular vnlts of magnesite in banded ls, same type as @ DH #1? Apparent bedding @ 45° cross-cutting vnlts nearly perpendicular to bedding.	260 - 270 --- 10%
270	288	Impure ls, some vnlts of magnesite.	270 - 288 --- 3%
		END	

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DIAMOND DRILL CORE LOG

DIAMOND DRILL HOLE B-1, -90° STARTED July 29/70

GRID LOCATION 0+27S, 2+00E COMPLETED Aug. 1/70

STARTING ELEVATION 4,740 FOOT LOGGED BY E. Stary

DEPTH 201 FOOT DATE LOGGED Aug. 14/70

DRILLED BY L. Hemmelgarn

CORE SECTION FROM	TO	DESCRIPTION
0	27	- Overburden.
27	36	- Argillaceous Dolomite - Fine grained. Dark grey. Brecciated.
36	41.5	- Dolomitic Magnesite - Coarsely brecciated. Mottled grey.
41.5	112.8	- Magnesite - Very coarse grained, white, massive.
		99.9 - 100.3 - Fracture filled with fine-grained, cream colored limestone.
		100.3 - 100.5 - Fine irregular fractures filled with $CaCO_3$.
		107 - 112.8 - Progressively finer grained.
		110.6 - Open seam 2", mud filled.
112.8	117	- Dolomite - Medium-grey. Vuggy, brecciated.
117	120	- No core.
120	201	- Limestone, argillaceous - very fine grained. Dark grey. Well banded @ 40° to core axis.
		- 120 - 130 - Occasional fine irregular fracture filled with limonite.
		- 127 - 138 - 5' core.
		- 138 - 142 - 2-1/2' core.
201		- End of hole.

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DIAMOND DRILL CORE LOG

DIAMOND DRILL HOLE B-3, -90° STARTED Aug. 4/70

GRID LOCATION 0+95.7N, 2+91.9E COMPLETED Aug. 5/70

STARTING ELEVATION 4,780 FOOT LOGGED BY E. Stary

DEPTH 135 FOOT DATE LOGGED Aug. 14/70

DRILLED BY L. Hemmelgarn

CORE SECTION FROM	TO	DESCRIPTION
0	25	- Overburden.
25	84.6	- Magnesite - white, very coarse grained, massive. - 25 - 65 - Traces partly oxidized pyrite in very fine fractures. - 73 - 76.5 - Coarsely brecciated.
84.6	89	- Dolomite - white, fine grained, massive.
89	100	- Limestone, dolomitic - mottled grey, massive. Lower 1' vuggy.
100	135	- Limestone, argillaceous. Fine grained, medium to dark grey. Well banded @ 40° to core axis.
135		- End of hole.

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DIAMOND DRILL CORE LOG

DIAMOND DRILL HOLE B-4, -90° STARTED Aug. 5/70

GRID LOCATION 0+95.5N, 3+88E COMPLETED Aug. 6/70

STARTING ELEVATION 4,740 FOOT LOGGED BY E. Stary

DEPTH 110 FOOT DATE LOGGED Aug. 14/70

DRILLED BY L. Hemmelgarn

CORE SECTION FROM	TO	DESCRIPTION
0	24	- Overburden.
24	38	- Limestone, argillaceous. Medium grey. Upper 3' brecciated. Banding at 45°. Lower 1' rust streaked.
38	44	- Transition zone. Limestone to argillite. Grading from medium grey to pale green. Well banded at 45°.
44	98	- Argillite, calcic. Very fine grained, pale greenish grey. Well banded at 50° to core axis.
		- 60 - 98 - Frequent dark greenish blebs to 1/4 x 1/8 inch.
98	110	- Limestone - pale grey, fine grained. Banded at 60° to core axis.
110		- End of hole.

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DIAMOND DRILL CORE LOG

DIAMOND DRILL HOLE B-5, -90° STARTED Aug. 7/70

GRID LOCATION 1+97N, 0+04E COMPLETED Aug. 8/70

STARTING ELEVATION 4,780 FOOT LOGGED BY E. Stary

DEPTH 239 FOOT DATE LOGGED Aug. 15/70

DRILLED BY L. Hemmelgarn

CORE SECTION		DESCRIPTION
FROM	TO	
0	30	- Overburden.
30	98.4	- Magnesite - white, coarsely crystalline.
		- 31.5 - 33 - Pyrite 5-7% fine grained in irregular veinlets. Partly oxidized.
		37.4 - 39.5 - Pyrite 3-5%, as above.
		58.4 - 59.6 - Pyrite 5%, as above.
		63 - 75 - 5.5' core
		74 - 75 - Pyrite 3-5%.
		73 - 84 - 4' core.
		80 - 83.3 - Pyrite 3-5%.
		84 - 95 - 5.7' core
98.4	145	- Dolomite, argillaceous, calcic. Fine grained, medium grey, massive. Mildly effervesces with HCl.
145	159	- Magnesite, as above.
159	169.5	- Dolomite - calcic. Coarse grained. Medium grey massive. Mildly effervesces. Trace pyrite in irregular veinlet at 167.5
169.5	210.4	- Magnesite, as above.
210.4	239	- Argillite - upper 5' medium grey grading to pale greenish grey. Very fine grained. Banded at 70° to core axis.
239		- End of hole.

ACRES WESTERN LIMITED

DIAMOND DRILL CORE LOG

DIAMOND DRILL HOLE B-6, -90° STARTED Aug. 9/70

GRID LOCATION 3+82N, 2+48W COMPLETED Aug. 10/70

STARTING ELEVATION 4,720 FOOT LOGGED BY E. Stary

DEPTH 338 FOOT DATE LOGGED Aug. 15/70

DRILLED BY L. Hemmlegarn

CORE SECTION FROM	TO	DESCRIPTION
0	22	- Overburden.
22	319.5	- Magnesite - White, coarse grained massive.
		- 23 - 6" with 5% pyrite in irregular fractures. Partly oxidized.
		- 35 - 36 - Pyrite 5-8%. Fine crystals in irregular fractures.
		- 66 - 68 - Pyrite 3%, as above.
		- 71 - 75.5 - Pyrite 3%, as above.
		- 121.5 - 122.2 - Pyrite 5%, as above.
		- 131 - 141 - Pyrite 7-10%, as above.
		- 152 - 153.5 - Pyrite 5%, as above.
		- 156 - 156.7 - Pyrite 7%
		- 161.5 - 162.2 - Pyrite 5%.
		- 172 - 176 - Pyrite 3%.
		- 198 - 207 - Pyrite 3-5%.
		- 209.5 - 214 - Salmon colored section. Calcarious.
		- 214 - 310 - Very pure magnesite. Rare thin veinlet of pyrite in upper 20'.
319.5	327.5	- Dolomite - Light grey grading to medium grey, fine grained somewhat fractured.
327.5	338	- Argillite - Dark grey, very fine grained. Brecciated contact at 20°. Odd blebs pyrite to 1/2" long.
338		- End of hole.

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DIAMOND DRILL CORE LOG

DIAMOND DRILL HOLE B-7, -90° STARTED Aug. 11/70

GRID LOCATION 1+48N, 2+45W COMPLETED Aug. 13/70

STARTING ELEVATION 4,660 FOOT LOGGED BY E. Stary

DEPTH 434 FOOT DATE LOGGED Aug. 15/70

DRILLED BY L. Hemmelgarn

CORE SECTION FROM	CORE SECTION TO	DESCRIPTION
0	5	- Overburden.
5	368	- Magnesite - White, very coarse grained. Massive. Upper 20' somewhat fractured with vuggy sections.
		- <u>88.1 - 88.7</u> - Limestone inclusion. Pale salmon colored. Coarse grained. Effervesces readily. Whole section very consistant and pure. Practically no pyrite except for rare fine crystals.
		- <u>274</u> - Faint banding at 60° to core axis.
		- <u>333.6</u> - 2" shear @ 45°.
		- <u>354</u> - Carbonate filled fracture 1/4" wide at 30°.
368	393	- Dolomite - White to pale grey, very fine grained. Effervesces mildly.
393	404	- Quartzite. Pale grey, very fine grained, massive. Very hard.
404	410	- Silicious limestone. Light grey, fine grained. Effervesces readily with HCl.
410	428	- Limestone, silicious. Dark to medium grey.
		- <u>416 - 428</u> - Many carbonate filled fractures to 1/4" thick.
428	434	- Quartzite - as above.
434		- End of hole.

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DIAMOND DRILL CORE LOG

DIAMOND DRILL HOLE B-8, -90° STARTED Aug. 13/70

GRID LOCATION 5+94N, 4+13W COMPLETED Aug. 14/70

STARTING ELEVATION 4,670 FOOT LOGGED BY E. Stary

DEPTH 411 FOOT DATE LOGGED Aug. 19/70

DRILLED BY L. Hemmelgarn

CORE SECTION FROM	TO	DESCRIPTION
0	6	- Overburden.
6	273.5	- Magnesite - Coarse grained, massive, white with local light blueish grey sections indicating impurities (Ca?).
		- <u>13 - 21</u> - Greyish color mentioned above.
		- <u>22.5 - 23.5</u> - Pyrite 7-10% in irregular fractures.
		- <u>34 - 34.5</u> - Pyrite 5% as above.
		- <u>36.3 - 38</u> - Pyrite 5%.
		- <u>42 - 43</u> - Pyrite 3-5%.
		- <u>44 - 47</u> - Pale yellowish colored section. Minor disseminated pyrite.
		- <u>55.5 - 58.2</u> - Pyrite 3%.
		- <u>53 - 75</u> - Faint blueish grey, somewhat mottled.
		- <u>61.5 - 63.7</u> - Pyrite 3%.
		- <u>65 - 67</u> - Pyrite 3%.
		- <u>71.7 - 73.2</u> - Pyrite 5%.
		- <u>74.7 - 76.4</u> - Pyrite 3-5%.
		- <u>79 - 108</u> - Pyrite 1-2%.
		- <u>108 - 116</u> - Pyrite 5-7% - Pale mottled blueish grey color.
		- <u>116 - 121</u> - Pyrite 2-3%.

ACRES WESTERN LIMITED

DIAMOND DRILL CORE LOG

DIAMOND DRILL HOLE B-8 (Cont'd.) STARTED ______

GRID LOCATION ______ COMPLETED ______

STARTING ELEVATION ______ FOOT LOGGED BY ______

DEPTH ______ FOOT DATE LOGGED ______

DRILLED BY ______

CORE SECTION FROM	TO	DESCRIPTION
		- 128.7 - 131 - Pyrite 3%.
		- 131 - 147 - Pyrite 1-2%.
		- 147 - 149 - Pyrite 3-5% - Faintly blueish grey.
		- 155 - 173 - Pyrite 7%.
		- 173 - 188 - Pyrite 4-5%.
		- 192 - 201 - Pyrite 3%.
		- 203.6 - 212 - Pyrite 3%.
		- 213.5 - 219.5 - Pyrite 1-2%.
		- 240 - 272.6 - More greyish colored (Ca?).
		- 272.6 - 273.5 - Shear (?) filled with very soft light brownish grey rock very fine grained. Upper contact 25°, lower 45° to core.
273.5	301	- Impure magnesite. Consists of 60% medium grey crystals (?) up to 1" x 1/4" with white auriols smaller than 1/8" wide. Remainder coarse, crystalline, white magnesite. No pyrite.
		- 281.7 - 283.7 - Core pulverized. Light grey, carbonate. Effervesces readily.
301	366.8	- Magnesite. Coarse grained, massive, generally with local light greyish sections separated from above with pure white 3/4" magnesite band at 80° to core.

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DIAMOND DRILL CORE LOG

DIAMOND DRILL HOLE B-8 (Cont'd.) STARTED ____________

GRID LOCATION ____________ COMPLETED ____________

STARTING ELEVATION ________ FOOT LOGGED BY ____________

DEPTH ________ FOOT DATE LOGGED ____________

DRILLED BY ____________

CORE SECTION FROM	TO	DESCRIPTION
		- 302.5 - 303 - 7-10% pyrite.
		- 310 - 317 - Light greyish colored.
		- 311.5 - 313 - Pyrite 5%.
		- 340 - 350 - Blueish grey colored.
		- 366.2 - 1" Argillite (?) band at 60°.
366.8	411	- Argillite - Fine grained. Greenish grey. Well banded with dark grey bands smaller than 1/4".
		- 380 - Banding 60°.
		- 395 - " 40°.
		- 408 - " 30°.
411		- End of hole.

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DIAMOND DRILL CORE LOG

DIAMOND DRILL HOLE B-9, -90° STARTED Aug. 15/70

GRID LOCATION 4+30N, 4+24.5W COMPLETED Aug. 17/70

STARTING ELEVATION 4,640 FOOT LOGGED BY E. Stary

DEPTH 485 FOOT DATE LOGGED Aug. 20/70

DRILLED BY L. Hemmelgarn

CORE SECTION FROM	TO	DESCRIPTION
0	13	- Overburden.
13	465	- Magnesite - Very coarse grained, massive. Predominantly white with local blueish grey sections caused by impurities (Ca?). Some pyrite in thin irregular stringers and disseminated crystals.
		- 27 - 57 - Pyrite 1-2% in thin stringers. Core has very faint blueish tint.
		- 58 - 81.5 - White, very pure magnesite.
		- 81.5 - 88 - Pyrite 3-4%.
		- 102 - 105.5 - Pyrite 5%.
		- 107 - 129 - Pyrite 3-5%.
		- 129 - 138 - Pyrite 1-2%.
		- 138 - 201.5 - Pure white magnesite.
		- 201.5 - 213 - Blueish grey colored.
		- 204 - 205 - 20% pyrite.
		- 226 - 239 - Some blueish grey streaks.
		- 239 - 306 - Pure, white, coarse magnesite.
		- 306 - 342.5 - Somewhat finer grained. Pale grey tint.
		- 342.5 - 403 - Pure magnesite, coarse grained.

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DIAMOND DRILL CORE LOG

DIAMOND DRILL HOLE B-9 (Cont'd.) STARTED ____________

GRID LOCATION ____________ COMPLETED ____________

STARTING ELEVATION ________ FOOT LOGGED BY ____________

DEPTH ________ FOOT DATE LOGGED ____________

DRILLED BY ____________

CORE SECTION FROM	TO	DESCRIPTION
		- 403 - 465 - Somewhat mottled looking due to blueish grey colored crystals. Finer grained. - 414.5 - 416, 419.5 - 421 - Silicious sections. Milky quartz.
465	475.3	- Impure argillaceous magnesite - elongated crystals magnesite to 1/2" x 1/4", 60% in dark grey fine grained ground mass. (graphitic? shale) Lower 2-1/2' interbanded with irregular medium grey impure quartzite bands to 4" wide. Several small slips at 45-60° with shaly slickensides.
475.3	478.5	- Quartzite - Impure, medium grey, fine grained. Lower contact sharp at 45° to core axis.
478.5	485	- Shale - Limy, soft. Fine grained dark grey to black. Well banded with lighter grey bands at 45-60° to core axis.
485		- End of hole.

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DIAMOND DRILL CORE LOG

DIAMOND DRILL HOLE B-10, -90° STARTED Aug. 18/70

GRID LOCATION 5+84.5N, 6+25W COMPLETED Aug. 19/70

STARTING ELEVATION 4,600 FOOT LOGGED BY E. Stary

DEPTH 465 FOOT DATE LOGGED Aug. 23/70

DRILLED BY L. Hemmelgarn

CORE SECTION FROM	TO	DESCRIPTION
0	4	- Overburden.
4	421.8	- Magnesite - White, coarse grained, massive. Some blueish grey sections due to impurities. Locally pyrite occurs in irregular veinlets.
		- 9 - 12 - Pyrite 1-2%. 4 - 86 - Blueish grey in color.
		- 37 - 47 - Pyrite 3-5%.
		- 57 - 58 - Pyrite 3-5%.
		- 63 - 67 - Pyrite 3%.
		- 67 - 73 - Pyrite 5-7%.
		- 83 - 84.5 - Pyrite 5%
		- 120 - 132 - Pyrite 3-5%.
		- 135 - 138 - Pyrite 2-3%.
		- 139 - 144 - Pyrite 3%.
		- 144 - 147 - Rusty fracture parallel to core.
		- 152 - 162 - Pyrite 3%.
		- 173.6 - 175 - Core friable, appears to be sheared.
		- 174.5 - 1/2" Irregular clay seam.
		- 181 - 184.5 - Pyrite 3%.
		- 188 - 190 - Pyrite 5-7%.

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DIAMOND DRILL CORE LOG

DIAMOND DRILL HOLE B-10 (Cont'd.) STARTED ______

GRID LOCATION ______ COMPLETED ______

STARTING ELEVATION ______ FOOT LOGGED BY ______

DEPTH ______ FOOT DATE LOGGED ______

DRILLED BY ______

CORE SECTION		DESCRIPTION
FROM	TO	
		- 202 - 206 - Pyrite 7-10%.
		- 207 - 219 - Pyrite 3-5%. Occurs in wavey veinlets less than 1/4" thick at 60-80° to core axis.
		- 236 - 241 - Pyrite 2%.
		- 246 - 248.5 - Pyrite 5-7%.
		- 249 - 341 - Pure white magnesite. Rare thin veinlets of pyrite.
		- 341 - 346 - Somewhat finer grained. Pale blueish grey mottled colored.
		- 346 - 431.8 - Faint blueish colored magnesite.
431.8	453	- Impure magnesite. Magnesite in elongated blebs to 1-1/2" x 1/4" - 60% randomly oriented in dark grey soft groundmass (graphitic shale ?).
453	465	- Argillite - Medium greenish grey, fine grained, soft. Well banded at 30° to core axis.
465		- End of hole.

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DIAMOND DRILL CORE LOG

DIAMOND DRILL HOLE B-11, -52° STARTED Aug. 20/70

GRID LOCATION 4+30N, 4+24.5W COMPLETED Aug. 21/70

STARTING ELEVATION 4,640 FOOT LOGGED BY E. Stary

DEPTH 348 FOOT DATE LOGGED Aug. 24/70

DRILLED BY L. Hemmelgarn

CORE SECTION FROM	TO	DESCRIPTION
0	5	- Overburden
5	245	- Magnesite - white, very coarse grained, massive. Some blueish grey sections due to impurities (Ca ?). Pyrite occurs in thin irregular veinlets.
		- 5 - 8 - 1' core.
		- 8 - 16 - 4.5' core.
		- 16 - 28 - 9' core.
		- 21 - 24 - Fracture parallel to core axis.
		Note: Incorrect footage. 8-1/2' after 58' marker, footage marked as 56' and continues from this point. Therefore 56' should read 66'.
		- 64 - 65 - Pyrite 3%.
		- 65 - 129 - Relatively pure magnesite. Very minor pyrite and rare short blueish sections.
		- 129 - 132 - Pyrite 3%.
		- 130.5 - 135.6 - Mottled blueish colored. Pyrite 2-3%.
		- 145.5 - 148 - Pyrite 10%.
		- 151 - 190 - Pyrite 7-10%.
		- 190 - 206 - Pyrite 5%.
		- 220 - 223 - Pyrite 2%.

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DIAMOND DRILL CORE LOG

DIAMOND DRILL HOLE B-11 (Cont'd.) STARTED ____________

GRID LOCATION ____________ COMPLETED ____________

STARTING ELEVATION ________ FOOT LOGGED BY ____________

DEPTH ________ FOOT DATE LOGGED ____________

DRILLED BY ____________

CORE SECTION FROM	TO	DESCRIPTION
		- 223 - 245 - Blueish grey. Minor pyrite.
245	251	- Shale. Dark grey, fine grained, soft.
251	254.5	- Impure magnesite. White magnesite crystals 70% in dark grey fine groundmass.
254.5	309.5	- Magnesite.
		- 254.5 - 262.5 - Blueish grey.
		- 271 - 273 - Pyrite 3%.
		- 263 - 281 - Core badly broken.
309.5	313	- Transition zone. Magnesite to impure limestone.
313	348	- Limestone, argillaceous - medium to dark grey, fine grained. Well banded at 60° to core axis.
348		- End of hole.

VIII MINING

CONTENTS

		Page
1.	INTRODUCTION	79
2.	SELECTED MINING AREA	79
3.	PIT PLAN	80
4.	SAMPLING	81
5.	ANCILLARY FACILITIES	81
6.	CREW	81

VIII MINING

1. INTRODUCTION

The Mount Brussilof magnesite property consists of 344 mineral claims located approximately 20 miles northeast of Radium Hot Springs in southeastern British Columbia. Proven and probable ore reserves totalling 26,601,000 short tons have been established in the Mount Eon deposit by diamond drill programs undertaken in 1968 and 1970 and are described in Section VI. The magnesite deposit lies along the southwest side of a mountain located between Assiniboine Creek and Mitchell River and the elevation of the mining area ranges between 4,500 feet and 5,200 feet. Plates 2 and 3 illustrate the regional location and general arrangement of the deposit.

thickness ?

The average vertical depth of the magnesite bed is conservatively estimated to be 120 feet, with an unconsolidated overburden depth averaging 20 feet or less. The magnesite deposit trends approximately N.30°W. and dips 35° to 40° SW. Because of the relatively low overburden to ore ratio and the favorable attitude of the magnesite bed, open pit mining will be employed.

2. SELECTED MINING AREA

For feasibility purposes the initial mining operation will be undertaken in the proven and probable ore areas shown in Plate 3. Further exploration and drilling carried out prior to mining may indicate that the initial pit could be advantageously relocated.

The ore at the northwest end of the area, as shown in Plate 3, sections 1, 2 and 3, has the highest iron content. In order to minimize this impurity in the early operation, the initial pit will be laid out southeast from section 3 to the end of the deposit beyond section 7, and from the 4,600 foot to the 4,900 foot elevations. This initial area contains approximately 500,000 tons of ore and is 400-700 feet above the valley floor. In order to maintain access to the waste dump, the cliff areas of sections 5, 6, 7 and beyond 7 will be mined off as the pit retreats downward.

Any provision for grade change & storage?

3. PIT PLAN

The pit is planned to facilitate selective mining of the deposit with a minimum of capital expense in the early stage of the project. Some degree of selective mining combined with beneficiation at the process plant will be required to ensure that the dead burned magnesite product meets high grade market specifications.

As mining progresses, the pit will be expanded towards the footwall and along strike until the haulage cost to the crusher becomes high. The crusher will then be relocated at a lower level and another ore pass system developed.

Storage will be maintained at the plant for the various ore grades required. High pyrite ore will be stockpiled at the mine loading area.

The recommended pit mining conditions are:

Benches	- 30 feet high with berms 60 feet wide every 2nd bench.
Drilling	- 6 ½ inch rotary drill holes on a 16.9 foot pattern drilled 3 feet below grade. Holes may be sloped at 70^{o} to improve safety of loading.
Blasting	- 80 percent Ammex or ANFO explosive, 20 percent high explosive. Wet holes loaded using a plastic liner.
Loading	- 6 yard rubber tired front end loader, tramming ore to an ore pass.
Ore Pass	- 200 foot ore pass along pit wall, the top mined off as the pit retreats downward.
Haulage Adit	- 16 feet by 14 feet by 200 feet long on the 4580 horizon, which is the initial crusher elevation.
Crushing	- Jaw crusher located on surface at 4580 level.
Conveying	- Inclined conveyor from the crusher to the truck loading bin at the 4350 foot elevation.

4. SAMPLING

Blast hole cuttings will be sampled to determine grade and locate included waste and high pyrite zones.

5. ANCILLARY FACILITIES

The mining area will be serviced by access roads, at maximum grade of 10 percent. A repair shop, spare parts warehouse, office, change house and power plant will be located in the loading bin area. Pit equipment will be serviced by a mechanic with a service vehicle. Personnel will be transported on the job site by pickup truck.

emergency only

The operating cost allowance includes leasing a suitable kitchen, dining room and bunkhouse facility for the operating crew. Capital has been provided for site preparation and servicing these facilities. Water for the mine camp will be obtained from Mitchell River.

why lease for 20 years a stove? & bed. Sewage

6. CREW

Based on mining 1,320 tons per day of magnesite ore, which is equivalent to producing 150,000 tons of product per year, the mine crew is estimated to be:

Geologist	1
Mine Foreman	1
Front End Loader and Mobile Equipment Operators	2
Crusher Operator	1
Serviceman	1
Drillers and Blasters	2
Mechanics and Welders	2
TOTAL	10

one shift.

The drillers, blasters, loader operators and service men will be capable of operating tractors, dump truck etc. as required. The men will operate 5 days per week, 250 days per year.

IX LABORATORY TESTS AND INTERPRETATION

CONTENTS

		Page
1.	INTRODUCTION	85
2.	TEST WORK	85
	2.1 University of British Columbia	
	2.2 Department of Energy, Mines and Resources	
	2.2.1 Single Stage Burning	
	2.2.2 Two Stage Burning	
	2.2.3 Hydration Stability and Density	
	2.2.4 Discussion of Results	
	2.3 Lakefield Research	
	2.3.1 Iron Extraction Tests	
	2.3.2 Calcium Extraction Tests	
3.	PROCESS CONSIDERATIONS	95

IX LABORATORY TESTS AND INTERPRETATION

TABLES

		Page
IX - 1	Summary of Firing Tests Conducted at the University of British Columbia on Minus 35 Mesh Samples	87
IX - 2	Firing Tests on Diamond Drill Core Particles	88
IX - 3	Single Stage Firing Tests	89
IX - 4	Two Stage Firing Tests	90
IX - 5	Hydration Stability Tests	91
IX - 6	Summary of Lakefield Iron Flotation Tests	94

FIGURES

IX - 1	Bulk Density of Pelletized Raw Magnesite after Single Stage Burning	92

IX LABORATORY TESTS AND INTERPRETATION

1. INTRODUCTION

Laboratory tests have been conducted to establish data which could be used in determining a suitable method of processing Mount Brussilof magnesite. The primary objectives were:

(a) To determine the specific gravity and hydration tendency of dead burned magnesite produced from Mount Brussilof magnesite and to establish the degree of product decrepitation or breakdown which could be expected if the crushed ore was fired in a kiln without briquetting.

(b) To investigate the requirement for single or two stage burning in kilns and to measure the bulk density and porosity of briquettes fired under a variety of conditions.

(c) To determine the degree to which iron and calcium could be extracted from the ore by flotation, or gravity separation.

The initial firing tests were performed at the University of British Columbia because of the proximity to and ease of communications with Acres Vancouver office. Subsequent firing investigations were conducted at the Department of Energy, Mines and Resources in Ottawa because of the availability of more complete briquetting and laboratory furnace facilities. The mineral beneficiation testwork was performed by Lakefield Research of Canada Limited, Lakefield, Ontario, because of Lakefield's background in this type of work.

2. TEST WORK

2.1 University of British Columbia

In October 1970 arrangements were made with the Department of Metallurgy, University of British Columbia, to conduct dead burning tests on Mount Brussilof magnesite samples. This work was conducted primarily to determine the hydration tendency and specific gravity of products formed over a range of firing times and temperatures.

In these tests the samples were crushed to minus 35 mesh and calcined for durations ranging between 1 and 5 hours at 1650°C. and ½ and 2 hours at 1700°C. in a natural gas fired laboratory furnace. Longer firing times were not investigated at the 1700°C. temperature because of problems with the furnace. The specific gravities were determined by displacement in pyncnometer flasks and by X-ray diffraction. The hydration tendency was measured by exposing minus 120 mesh samples to steam for 5 hours.

The maximum product specific gravity and minimum hydration tendency was obtained after firing the samples for 5 hours at 1650°C. The chemical analyses of the samples tested and the results obtained are summarized in Table IX - 1.

These results demonstrate that a burned magnesite product with a hydration tendency of less than 1 percent, which is a requirement in high grade dead burned magnesite, can be produced from Mount Brussilof ore. The close correlation between the specific gravities as determined by displacement and X-ray diffraction indicates that the material was completely burnt to MgO. However, no appreciable sintering or agglomeration of the product was noted in these tests. It was therefore decided to calcine some relatively coarse magnesite ore particles to establish whether a larger, more cohesive dead burned grain could be produced. Samples were fired for 5 hours at 1650°C. The test results are contained in Table IX - 2.

The apparent density of sample 217, measured after breaking to minus 10 mesh, was 3.52 gm per cm^3.

one is c.gn anyway

The most significant result of these tests is the high degree of particle breakdown which occurred on firing and the crumbly nature and low compressive strength of the product.

It was apparent that, although the calcination was complete, the firing temperatures and retention times were insufficient to promote particle agglomeration and sintering. Therefore, arrangements were made to have the Mines Branch, Department of Energy, Mines and Resources conduct additional tests to determine the conditions necessary to produce a dead burned product of satisfactory bulk density and compressive strength.

TABLE IX - 1

SUMMARY OF FIRING TESTS CONDUCTED AT THE UNIVERSITY OF BRITISH COLUMBIA ON MINUS 35 MESH SAMPLES

Sample Number	Chemical Analysis after Ignition at $1000^{o}C$. (%)								Physical Properties after Firing for 5 hrs.@ $1650^{o}C$.		
	LOI	Residual CO_2	MgO	CaO	Al_2O_3	Fe_2O_3	SiO_2	Acid Insols.	Specific Gravity by Displacement	Specific Gravity by X-ray Diffraction	Hydration Tendency %
5076A	51.3	.10	93.1	2.00	.64	.58	2.32	.64	3.50	3.58	.82
5076X	51.8	.69	96.1	1.78	.16	.56	.30	.18	3.51	3.58	.91
5077 Composite	51.5	.31	97.4	1.52	.16	.51	.07	.04	3.47	3.57	1.01

TABLE IX - 2

FIRING TESTS ON DIAMOND DRILL CORE PARTICLES

Sample Number	Weight gm.		Volume cm^3		Average Particle Size Before Burning inches	Estimated % Passing 10 mesh	
	Before Burning	After Burning	Before Burning	After Burning		Before Burning	After Burning
194	30.50	14.64	10.2	--	½	3	20
217	43.94	21.03	14.5	5.97	1	2	20
205	45.42	21.78	15.2	--	1½	0	20

2.2 Department of Energy, Mines and Resources

Single and two stage burning tests were conducted in November, 1970 by the Ceramics Section of the Mines Branch.

X-ray analysis of the sample submitted indicated that it consisted primarily of magnesite with minor amounts of dolomite, quartz and pyrite.

2.2.1 Single Stage Burning

Samples were ground to an average particle size of approximately 5 microns and the product was compressed at 3 percent moisture into 1 inch diameter by 3/4 inch briquettes having a green density of 2.42 gm per cm^3. Table IX - 3 summarizes the results of these tests.

TABLE IX - 3

SINGLE STAGE FIRING TESTS

Firing		Bulk Density gm/cm^3	Apparent Porosity %	Total Porosity %
Temperature ^oC.	Time at Temperature hrs.			
1650	0	3.37	.04	5.35
1700	0	3.35	1.04	5.9
1700	1	3.37	.47	5.47
1700	3	3.37	.67	5.31

2.2.2 Two Stage Burning

These tests were conducted to determine whether significant improvements in product densities and porosities could be achieved by burning the Brussilof magnesite at a relatively low temperature, grinding and compressing the primary calcined product and sintering the compressed briquettes at an elevated temperature. The results are shown in Table IX - 4.

TABLE IX - 4

TWO STAGE FIRING TESTS

Primary Burning		Green Density of Briquettes gm/cm^3	Secondary Firing			
			Firing		Bulk Density gm/cm^3	Total Porosity %
Temperature ^oC.	Time hrs.		Temperature ^oC.	Time hrs.		
900	3	2.08	1650	3	3.41	4.11
			1700	3	3.41	4.30
1000	2	2.24	1650	3	3.42	4.06
			1700	3	3.41	4.26
1100	2	2.40	1650	3	3.48	2.18
			1700	3	3.48	2.19

2.2.3 Hydration Stability and Density

Additional tests were conducted to determine the hydration tendency and the densification of both single and double burned briquettes, or pellets, formed at 25,000 psi. Briquettes were withdrawn at progressively higher temperatures and the hydration stability was measured after exposing the pellets to pressurized steam for 5 hours. The final pellet density was also measured. The hydration stability data is contained in Table IX - 5, the densification data in Figure IX - 1.

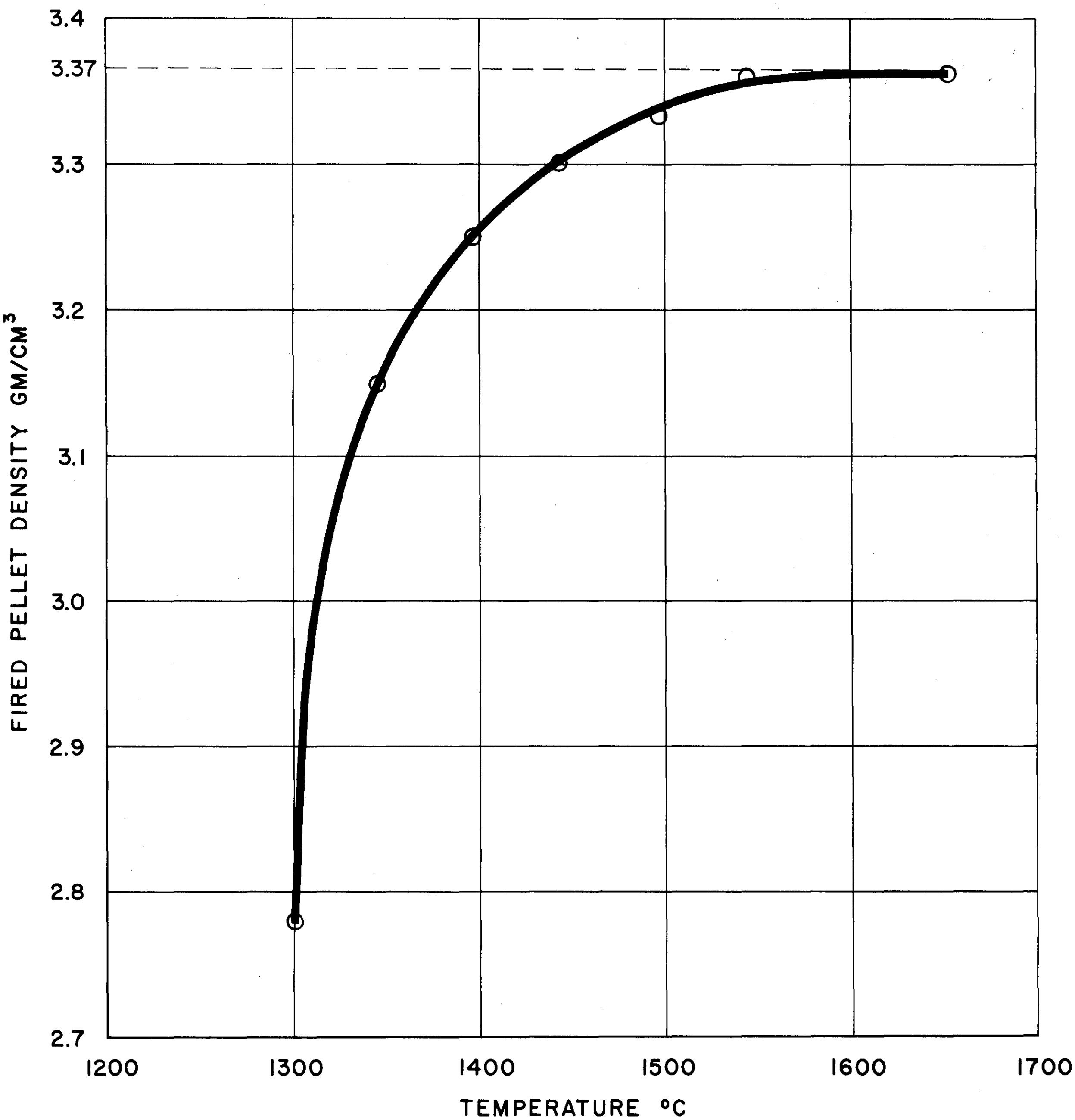
TABLE IX - 5

HYDRATION STABILITY TESTS

First Stage		Second Stage		Hydration Stability LOI %
Calcination Temperature °C.	Calcination Time hrs.	Firing Temperature °C.	Soak Period hrs.	
Uncalcined	Uncalcined	1300	-	14,903
Uncalcined	Uncalcined	1350	-	7.824
Uncalcined	Uncalcined	1400	-	1.491
Uncalcined	Uncalcined	1450	-	0.777
Uncalcined	Uncalcined	1500	-	0.069
Uncalcined	Uncalcined	1550	-	0.041
Uncalcined	Uncalcined	1650	-	0.010
Uncalcined	Uncalcined	1650	3	0.013
900	3	1650	3	0.016
1000	2	1650	3	0.020
1100	2	1650	3	0.051
900	3	1700	3	0.010
1000	2	1700	3	0.010
1100	2	1700	3	0.006

Source: Department of Energy, Mines and Resources Tests on Brussilof Magnesite.

BULK DENSITY OF PELLETIZED MAGNESITE AFTER SINGLE STAGE BURNING



SOURCE:
DEPARTMENT OF ENERGY, MINES AND RESOURCES TEST WORK ON BRUSSILOF MAGNESITE SAMPLES

ACRES

FIGURE IX-1

2.2.4 Discussion of Results

The results of these investigations demonstrated that:

(a) A dead-burned magnesite briquette having a bulk density of 3.35 to 3.37 gm per cm^3, a total porosity of approximately 5.5 percent and a hydration tendency of less than 0.5 percent can be produced by single stage burning.

(b) Some relatively small improvements in physical qualities can be achieved by two stage burning. Bulk densities of 3.41 to 3.48 gm per cm^3, total porosities of 2.2 to 4.3 percent and less than 0.5 percent hydration tendency should be expected.

(c) With either single or two stage burning techniques, high density, dead burned products can be produced with firing temperatures of 1650-1700°C. and short retention times relative to those provided in a rotary kiln. Bulk densities of 3.35 to 3.37 gm per cm^3 were obtained in these tests, which compares favourably with normal bulk density specification requirements of 3.33 gm per cm^3, minimum. The measured hydration tendencies of 0.01 to 0.07 percent also compared favourably with industrial specifications.

2.3 Lakefield Research

In November, 1970 samples of Brussilof magnesite were tested by Lakefield Research of Canada Limited to determine the degree to which iron could be extracted by flotation or gravity separation.

The feasibility of extracting dolomite by flotation was also cursorily investigated.

2.3.1 Iron Extraction Tests

Mineralogical examination of the samples tested indicated that the majority of the iron was present as pyrite. The remainder of the iron occurred as ferroan-dolomite.

As pyrite is usually amenable to flotation, this technique was selected as the most appropriate procedure for beneficiating the magnesite samples. Three samples of varying iron content were submitted to Lakefield Research for testing and the results of these tests are summarized in Table IX - 6.

TABLE IX - 6

SUMMARY OF LAKEFIELD IRON FLOTATION TESTS

Sample Number	% Fe_2O_3 in Sample	% Fe Present as Pyrite	% Fe Extracted	% Fe_2O_3 in Magnesite Concentrate	Weight Magnesite Concentrate/ Sample Weight
Composite 1	2.41	73.3	76.7	.60	.94
Composite 2	3.99	77.3	84.4	.69	.90
Composite 3	.66	36.8	42.7	.40	.95

These results were obtained using an amyl zanthate collector and Dow DF-250 frother at a pH of 8.3 after grinding to approximately 68 percent minus 200 mesh. The effect of increased collector additions was investigated on composite 2 and it was determined that a limit of just under 0.4 percent Fe_2O_3 could be approached using the flotation technique. In achieving this 0.29 percent reduction in Fe_2O_3 content, the magnesite concentrate weight recovery decreased by approximately 10 percent from that shown in Table IX - 6.

10% wht. loss

It is concluded from these tests that a magnesite concentrate containing approximately 0.4 percent Fe_2O_3 can be produced by the flotation process provided the raw magnesite does not contain more than 4 percent Fe_2O_3. Such a material can be calcined to a dead burned product containing 0.8 percent Fe_2O_3. The weight recovery of magnesite concentrate will vary between 80 percent and 95 percent with raw product Fe_2O_3 analyses of 4 percent and 0.7 percent respectively.

2.3.2 Calcium Extraction Tests

Microscopic examination of a sample of minus 10 mesh material indicated the carbonate portion of the sample was composed of distinct magnesite and dolomite grains. In view of the favourable mineralogical separation, flotation tests were undertaken to separate the dolomite and magnesite fractions using established processes.

achieve. separation

A series of 9 flotation tests were conducted, but a significant elimination of dolomite was not achieved. It is considered that flotation test work should be continued as there is a good possibility a suitable technique for extracting dolomite from this ore can be developed.

3. PROCESS CONSIDERATIONS

On the basis of laboratory tests described in this section, the feasibility capital and operating cost allowances upon which feasibility will be tested have been based on the following process considerations:

(a) Beneficiation - Pyrite can be effectively extracted by flotation from Mount Brussilof ore after grinding to 68 percent minus 200 mesh. The process plant will therefore incorporate a grinding circuit to reduce the crushed ore to minus 65 mesh and flotation conditioners and cells to reduce the pyrite fraction of the ore to an acceptable level.

Although laboratory tests conducted to date have not demonstrated that dolomite can be effectively extracted by flotation, the capital cost estimate includes equipment for this operation.

% dolomite?

Heavy media tests were not performed as these separations are conducted most successfully on plus 1/4 inch material. However, if dolomite flotation tests continue to give negative results, the possibility of heavy media separation should be investigated.

(b) Briquetting - In the calcining tests conducted at the University of British Columbia on plus 1/2 inch lumps of magnesite, appreciable decrepitation occurred and the burnt product had a low compressive strength. These results indicated that dust losses could be excessive if Brussilof ore was calcined in lump form.

On the other hand, the Department of Energy, Mines and Resources tests demonstrated that a dense, durable product could be produced by briquetting and firing at 1650°C. Therefore, for feasibility purposes a decision was taken to incorporate briquetting equipment in the plant layout. Before plant design commences, additional investigations should be undertaken to determine conclusively whether briquetting is essential and to establish the most appropriate type of press and optimum particle size for briquetting.

(c) Grinding - Laboratory tests should be conducted to establish power requirements and media consumption rates for both the primary and secondary grinding operations. The secondary grinding tests should be conducted after the most suitable particle size has been established for the briquetting operation. For feasibility purposes it has been assumed that the briquetting installation will require an average particle size of 5 microns.

(d) Calcining - Prior to plant design, and after the question of briquetting has been settled, bench scale firing tests should be conducted on samples of the projected kiln feed. Some calcined product samples may be useful in market development. If so, this will partially determine the scale of the calcining tests.

For feasibility study purposes it has been assumed that rotary kilns will be employed in the firing operation. However, if the preceding investigations confirm the necessity of briquetting prior to calcining, then shaft type kilns might also be applicable and should be considered because of their higher thermal efficiency relative to the rotary kiln used in operating cost estimate.

X PROCESSING

CONTENTS

		Page
1.	PROCESS SUMMARY	99
2.	PLANT CAPACITY AND OPERATING SCHEDULE	99
3.	FLOWSHEET	100
	3.1 Crushing	
	3.2 Grinding - General	
	3.2.1 Primary Grinding	
	3.2.2 Secondary Grinding	
	3.3 Pyrite and Dolomite Extraction	
	3.4 Additive Blending	
	3.5 Briquetting	
	3.6 Calcining	
	3.7 Grain Sizing and Storage	

TABLES

X - 1	Plant Operating Schedules	101
X - 2	Anticipated Chemical Analyses of Average Grade Brussilof Magnesite Proven and Probable Reserves at Various Stages of Processing	105

X PROCESSING

1. PROCESS SUMMARY

Laboratory tests have been conducted by the University of British Columbia, the Department of Energy, Mines and Resources, and by Lakefield Research to establish process criteria for the Brussilof Magnesite Project. This work indicates that the material can be calcined in one stage to produce dead burned magnesite having a bulk density of 3.37 and a hydration tendency of less than 1 percent. These investigations are described in Section IX.

To obtain this product density and a suitable refractory grain size, the ore will be finely ground and briquetted prior to firing. Flotation circuits will be installed to assure product quality and uniformity and to decrease the selectivity required in mining. Provision will also be made to blend in silica and other additives prior to briquetting to control dicalcium-silicate and other critical product ratios. Grain sizing, storage, bagging and loading facilities will be incorporated in the plant.

For the reasons indicated in Section VI, the processing and calcining plant will be located at Canal Flats. The run of mine ore will be crushed to minus 6 inches prior to trucking to the processing plant.

2. PLANT CAPACITY AND OPERATING SCHEDULE

The feasibility evaluation is based on plants ranging in capacity from 75,000 to 225,000 tons per year of finished product.

In establishing plant capacity requirements, the following assumptions have been made:

(a) Five percent of run of mine ore will be rejected to waste in the flotation operation.

(b) The dust recirculated from the kiln exhaust gases to the briquetting process will approximate 5 percent of the kiln product.

(c) The average weight loss on ignition will be 51 percent.

On these bases, in order to produce 75,000 tons per year of dead burned magnesite the plant design capacities will be:

(a)	Grain sizing and storage plant	75,000 TPY	dead burned magnesite
(b)	Briquetting and calcining plant	157,000 TPY	magnesite ore
(c)	Ore crushing and iron extraction plants	165,000 TPY	magnesite ore

Table X - 1 summarizes the operating schedules of the production components. Plant operating and maintenance considerations require that the rotary kiln run 24 hours per day. The grinding, flotation and briquetting facilities will function on the same schedule as the kiln operation. The ore crushing and screening plant will operate 5 days per week and the Stage I, II and III capacity requirements will be satisfied by operating 1, 2 and 3 shifts per day respectively.

For the beneficiating, briquetting and calcining facilities, however, the incremental capacity requirements of Stages I, II and III will require physical additions of plant and facilities.

3. FLOWSHEET

The plant flowsheet is illustrated in Plate 5 and the individual production components for the Stage I operation are described under separate headings below.

3.1 Crushing

Provision for stock-piling if roads bad & [illegible]

A 42 inchby 48 inch primary jaw crusher will be installed at the mine site to reduce the ore to minus 6 inches. The crushed magnesite will be transported by a 30 inch belt conveyor to a 420 ton storage bin located at the 4,350 foot elevation. Tractor trailer trucks carrying 70 ton payloads will transport the magnesite to Canal Flats where it will be screened to remove the minus 1/2 inch fraction and either crushed directly or accumulated in a stockpile providing three weeks' storage. The minus 1/2 inch fraction will be conveyed directly to the fine ore bin.

TABLE X - 1

PLANT OPERATING SCHEDULES

PLANT SECTION	Scheduled Operating Days per Year	Availability of Scheduled Time/%	Tons per Year Stage I	Magnesite Ore	Dead Burned Magnesite	OPERATING DATA AT SPECIFIED DEVELOPMENT STAGES CAPACITIES EXPRESSED IN TONS											
						Stage I 75,000 Tons/Year Product				Stage II 150,000 Tons/Year Product				Stage III 225,000 Tons/Year Product			
						Days per Week	Shifts per Day	Average Daily Capacity/Tons	Average Hourly Capacity/Tons	Days per Week	Shifts per Day	Average Daily Capacity/Tons	Average Hourly Capacity/Tons	Days per Week	Shifts per Day	Average Daily Capacity/Tons	Average Hourly Capacity/Tons
Primary Crushing	250	-	165,000	x		5	1	660	83	5		1,320		5		1,980	
Fine Crushing and Screening	250	95	165,000	x		5	1	660	83	5	2	1,320	83	5	2	1,980	83
Grinding, Iron and Sulfur Extraction	365	95	165,000	x		7	3	530	22	7	3	900	38	7	3	1,350	57
Briquetting and Burning	365	85	157,000	x		7	3	510	21	7	3	860	36	7	3	1,290	54
Grain Sizing and Storage	250	95	75,000		x	5	1	300	38	5	2	600	38	5	2	900	38

Secondary and tertiary crushing and screening facilities to reduce the ore to minus 1/2 inch will be adjacent to the process plant. Both crushing stages will be performed in 4 foot cone crushers operated in closed circuit with a double deck vibrating screen. The minus 1/2 inch crushed magnesite will be accumulated on week days in a 1,500 ton storage bin to allow the process plant to operate 7 days per week.

3.2 Grinding - General

what ?
why ?
10μ.

There will be two stages of grinding. Primary grinding will reduce the crusher plant product to a size suitable for flotation. Secondary grinding of the beneficiated magnesite to minus 10 microns will also be required to produce briquettes having a high green density and the appropriate particle size needed to produce a dense dead burned product.

For feasibility purposes the grindability of Brussilof magnesite was calculated from published operating data for similar ores. Laboratory and, if necessary, pilot plant grinding tests will be conducted prior to plant design and selection of grinding systems.

3.2.1 Primary Grinding

The mill will reduce crusher plant product to minus 65 mesh for flotation. Based on a grindability index of 12.0 as defined in F. Bond's "Third Theory of Comminution", the energy requirements will approximate 10.4 kilowatt hours per ton of ground product. Making allowances for equipment availability and plant shutdowns, a 300 horsepower grinding mill will be required.

The feasibility cost allowance is based on a 9 foot diameter by 12 foot pebble mill. The initial cost of this mill will be somewhat higher than that of a conventional mill using steel media. However, the possibility of introducing iron contamination to the product makes it inadvisable to use steel grinding balls. Autogenous and two stage grinding should also be investigated prior to final plant design. The mill will operate in closed circuit with liquid cyclones overflowing a minus 65 mesh product to the flotation circuit.

3.2.2 Secondary Grinding

Following flotation, the beneficiated magnesite ore will be thickened and ground to minus 10 microns in a 1,000 horsepower 12 foot by 18 foot pebble mill operating in open circuit. As the consumption of mill media and liners varies directly with power requirements, non-metallic grinding media will be used in this mill also.

3.3 Pyrite and Dolomite Extraction

Pyrite and dolomite flotation systems will be provided to improve the uniformity and quality of the kiln feed. Laboratory testwork has established that pyrite can be effectively extracted but, to date, has not demonstrated that dolomite can be effectively removed by flotation. However, as the dolomite is present as distinct mineral grains at 10 mesh size and smaller, and as successful flotation techniques have been developed elsewhere for the separation of magnesite and dolomite, it is considered that, with continued testwork, an effective separation process will probably be developed. The capital cost allowance includes both flotation circuits. Since the ore must be ground prior to briquetting, the relatively small additional capital cost of the flotation circuit is justified because:

(a) The elimination of erratic pyrite and dolomite contaminants will produce a more uniform and higher grade product.

(b) The extraction of pyrite from the rotary kiln feed will decrease the sulphur content of the kiln gases.

(c) The extraction of pyrite appreciably extends and upgrades the ore reserves.

The major equipment items required in this system consist of two conditioning tanks, two banks of flotation cells providing 600 cubic feet volume, and ancillary equipment.

Incremental expansion to Stage II and Stage III capacities will be accomplished by duplicating the Stage I installation.

3.4 Additive Blending

Some of the magnesite users contacted have indicated a preference for a product having a 2 to 1, CaO to SiO_2 ratio. In order to satisfy this requirement provision will be made in the plant design to blend in silica or other materials ahead of the fine grinding circuit.

Silica flour.

The average ore, as defined in Section VII, 2.1.2 and Table VII - 3, contains 94.66 percent MgO before and 95.1 percent MgO after assay correction. Iron extraction upgrades the magnesite further. The anticipated analyses of average ore grade material at various stages of processing are illustrated in Table X - 2.

3.5 Briquetting

The secondary grinding product will be dewatered and dried before briquetting in roll presses. Waste heat from the kiln exhaust gases will be used in the drying operation.

Alternative briquetting systems will be investigated prior to final plant design to determine the most appropriate system and operating conditions.

3.6 Calcining

The firing operations will be performed in approximately 10 foot diameter by 265 foot long rotary kilns, each having a capacity of 75,000 tons per year of product. Each system includes an air cooler to extract sensible heat from the kiln product and high efficiency collectors to recover dust from the kiln and cooler exhausts.

Tests indicate that firing temperatures between 1650°C and 1800°C will be required to produce the required product density. Fuel requirements are estimated to be 10 million BTU per ton of product. Since Bunker C fuel oil provides better luminosity and heat transfer characteristics than natural gas, and as the latter is a less expensive source of heat at Canal Flats, the kiln will be designed to operate on either Bunker C fuel oil, natural gas, or a combination of the two. This arrangement also provides flexibility in the event of an interruption in supply of either fuel. The kiln firing system will be automated to achieve optimum fuel economy and uniformity of operation.

TABLE X - 2

ANTICIPATED CHEMICAL ANALYSES OF AVERAGE GRADE BRUSSILOF MAGNESITE PROVEN AND PROBABLE RESERVES AT VARIOUS STAGES OF PROCESSING

(%)

	MgO	CaO	Fe_2O_3	Al_2O_3	SiO_2	L.O.I.	HF Insol.
Average Run-of-mine Ore, after Assay Correction	95.1	2.07	1.36	0.32	0.63	51.32	0.23
Analysis after Iron Extraction	95.7	2.08	0.80	0.32	0.63	51.30	0.23
Analysis after Adjustment to Dicalcium Silicate Ratio	95.3	2.07	0.80	0.32	1.10	51.30	0.23

silica flour?
where from

Pilot plant kiln tests should be conducted on a substantial bulk sample prior to kiln selection to establish design parameters and to produce material for market development.

3.7 Grain Sizing and Storage

The kiln clinker will be crushed, sized and stored in five grain sizes. The screen undersize material from the segregation plant will be marketed together with the dust collected in the screening plant and from the kiln product cooler.

Product bagging, blending and weighing systems will be provided to satisfy a variety of marketing requirements. Loading facilities for rail or truck shipments in bulk or in bags will be adjacent to this section of the plant.

XI TRANSPORTATION

CONTENTS

		Page
1.	INTRODUCTION	107
2.	MINE TO PROCESS PLANT	107
	2.1 Mine Access Road	
	2.2 Method and Cost of Hauling	
3.	PROCESS PLANT TO MARKETS	108
	3.1 Rail Transport	
	3.2 Port Services and Shipping	

XI TRANSPORTATION

1. INTRODUCTION

Once the existence of a sufficient quantity of high quality ore has been demonstrated and the methods of mining and processing have been resolved, the next major item to be considered is that of transportation. This can be divided into the method and cost of moving the ore to the processing plant and subsequently of moving the finished product from the plant to the various markets or distribution centres.

2. MINE TO PROCESS PLANT

2.1 Mine Access Road

In Section VI, the most suitable access route from the mine site to the railhead was found to be along the Kootenay valley to Canal Flats. A railway along this route would be too costly and the ore will therefore be transported the 60 miles from the mine to Canal Flats by truck.

Ore haulage will be on a Forest Service road along the Mitchell, Cross and Kootenay Rivers. The road has been built to its present state by Baykal Minerals Limited, Revelstoke Lumber Company Limited and Crestbrook Industries Limited, all under permits from the B.C. Forest Service.

The road and bridges must be improved and strengthened for year-round heavy duty hauling. On such Forest Service roads, the construction and maintenance is performed and paid for by the users. In this instance, cost sharing agreements will be necessary with the other companies involved.

During spring breakup, the Forest Service restricts loads or bans trucking altogether on Service roads. This means that ore haulage from the mine to the Canal Flats processing plant could be interrupted some years for a period of up to one month during breakup. Adequate coarse ore storage capacity has been provided at the process plant to cover this contingency.

The road will be gravel surfaced, and for the weight and frequency of ore haul trucking contemplated, a high level of road maintenance will be essential. Allowance is made in the cost estimates for the necessary labour, material and equipment. The road gravelling itself might best be done by contract.

2.2 Method and Cost of Hauling

For the purpose of estimating hauling cycle times and costs, the ore hauler selected is a tractor trailer unit with a 70 ton payload.

At 150,000 tons per year of product, 1,310 tons of raw ore must be hauled per day on a five day week basis.

It is estimated that an average speed of 30 mph can be expected. Allowing for loading and unloading time, this means that each unit will make two round trips in a 10 hour shift. By working two shifts per day, the ore can be handled by using five units with one additional unit as a spare. For higher or lower tonnages these requirements change accordingly.

Haulage costs were estimated on the basis that the tractor trailer units would be purchased and operated by the mine. Allowances for this were made in the capital cost estimates. Operating costs of haulage, including the cost of operating the units, of road maintenance and of supervision, were estimated to be $3.18, $2.80 and $2.65 per ton of product for production capacities of 75,000, 150,000 and 225,000 tons per year respectively.

3. PROCESS PLANT TO MARKETS

3.1 Rail Transport

After processing, the product will be shipped by rail to eastern Canada and the United States or to the port of Vancouver. Freight rates to eastern Canada were assumed to be $15 per ton for an annual volume of 25,000 tons and $18 for a volume of 10,000 tons. These rates would apply to such areas as Toronto, Hamilton and Montreal, which are all within the same rate grouping. Delivery costs to the eastern United States would be increased somewhat by any applicable switching and freight charges from those areas served by the Canadian Pacific Railway. For volumes of less than 50,000 tons per year, rail transport would be more flexible and more economic than shipping via the Panama Canal.

For the quantities proposed, rail is also the most economic mode of transport from the plant to the port of Vancouver. The distance is 608 road miles and 642 miles by the Canadian Pacific Railway. Canadian Pacific Railway quoted a rate of

$6 per short ton assuming the use of 100 ton open hopper cars moving in 20 car lots twice a week. This could move 4,000 tons per week and provide an adequate safety margin of capacity for shipping up to 150,000 tons of product per year through the port.

3.2 Port Services and Shipping

Canadian Pacific Railways have access to all the bulk loading facilities in the Vancouver area. No additional switching time or direct charges are involved in transport to Port Moody, Pacific Coast Terminals, the Roberts Bank installation or Westview Terminals. The most suitable, general purpose bulk commodity terminal in the area is Pacific Coast Terminals. An allowance of $1.55 per ton was made for unloading the rail cars to the ship and for handling and storage. This was based on an open storage capacity for up to one ship load of 25,000 tons. If the rail cars were to contain frozen product, the dock would assess an additional surcharge for thawing before unloading.

The costs of shipping from Vancouver were estimated by adding the costs of ship charter, port charges, Panama Canal charges and bunkering. To ship 150,000 tons per year to Europe would cost roughly $14 per ton or $12 per ton to the east coast of the United States or to Japan. For quantities in the order of 50,000 tons per year, these rates would increase to around $23 and $18 respectively.

XII MARKETING

CONTENTS

		Page
1.	INTRODUCTION	113
2.	USES OF MAGNESIA	113
3.	PRODUCTION	115
	3.1 Existing Sources	
	3.1.1 Natural Deposits	
	3.1.2 Sea Water Plants	
	3.2 World Production	
	3.3 Canadian Production	
4.	TRADING PATTERNS	122
5.	QUALITY AND PRICE PERSPECTIVE	127
	5.1 Quality Comparison	
	5.2 Price Review	
6.	MOUNT BRUSSILOF MAGNESITE	133
	6.1 Potential Market	
	6.2 Anticipated Selling Price	

XII MARKETING

TABLES

		Page
XII - 1	World Magnesite Reserves, by Country.	117
XII - 2	World Sea Water Magnesia Plants.	119
XII - 3	World Production of Crude Magnesite, by Country.	120
XII - 4	Import of Magnesite, by Country.	123
XII - 5	Export of Magnesite, by Country.	124
XII - 6	Canadian Magnesia Imports.	126
XII - 7	Chemical Quality Comparison of Dead Burned Magnesites.	128, 129
XII - 8	Estimated First Year Market Potential.	135

XII MARKETING

1. INTRODUCTION

The purpose of this Marketing Section is to briefly summarize the various uses for magnesia, to discuss the relative merits of natural and synthetically produced magnesites and to review world magnesia trading patterns and prices. On the basis of this background information, the relative value and potential market for the Brussilof product will be placed in perspective. This will permit an estimate of the quantity of dead burned product which could be sold and the price which could be expected.

Magnesite can be used as a raw material for producing magnesium metal and a number of magnesium compounds, but more commonly it is valued for its use as a basic refractory. Although the following section, "Uses of Magnesia," briefly covers the range of possible uses of magnesia, the market analysis deals only with the market for dead burned magnesite as a refractory material.

Throughout the course of this market analysis, difficulty was experienced in obtaining up to date and complete statistical data on reserves, plant capacities and national production, consumption, imports and exports. It should be remembered, therefore, when reading this section that the data presented are not necessarily complete. However, this does not mean that they are without value. The available statistics give a valid indication of the relative importance of the magnesite industry in various countries and of the roles of those countries in the world magnesia market.

2. USES OF MAGNESIA

By far the greatest use of magnesia is as a refractory material. The magnesite is calcined to a point where it is no longer chemically active and the product is generally referred to as dead burned magnesite or dead burned magnesia. As a refractory material it has a melting point of over 2800°C and excellent physical and chemical stability.

The major demand for magnesia refractories is for use in the steelmaking industry, since these refractories have no ready reaction with open hearth slags and have the unique ability to absorb large quantities of iron oxide without serious loss of refractoriness. The changing pattern of steelmaking has speeded the decline in the use of other types of refractories, particularly those made of silica and fireclay, and brought a swing to magnesite and magnesite chrome. Therefore, recent developments in refractories have favoured magnesite but have made more exacting demands on the suppliers of raw materials and on the refractories industry.

Some indication of the growth of the demand for dead burned magnesite can be obtained by relating refractory demand to steel production.

Although both national and worldwide annual steel production figures are affected by major economic factors, in the long term, steel production has exhibited a steady growth pattern of more than 4 percent annually in the United States for the last 10 years.

A convenient index of the relationship between dead burned magnesite consumption and steel production is the ratio of total pounds of magnesite consumed to tons of steel produced. While world steel production has been steadily growing, the magnesite to steel ratio has also increased. In 1930 the average ratio was 4 pounds per ton of steel. This increased to 14.5 pounds per ton of steel by 1964. Recent technological improvements in the steel industry are lowering this ratio. In large part, this reduction has been due to the increased use of basic oxygen furnaces, which require a very high grade refractory material. Refractory linings for the Siemens-Martin type of furnace are replaced at the rate of about 16 pounds per ton of steel, while basic oxygen furnaces consume about 4 pounds of lining per ton of steel. World statistics are not available, but in the United States between 1964 and 1968 the overall ratio declined from 13.8 to 10.8. Despite this partial reduction in ratios, the steel industry has required and is expected to continue to require increasing quantities of high purity magnesia refractories.

The second most common use of magnesia is in the production of caustic calcined magnesia, also referred to as calcined magnesite or calcined magnesia. In the United States, from 15 to 20 percent of current magnesia production is used in manufacturing calcined magnesia. In the process, high quality magnesite is lightly calcined to produce the chemically active product.

Caustic calcined magnesia is used in the manufacture of pulp and paper, rayon, rubber, flooring cements, specialty fertilizers, insulation and in a wide variety of chemical products and processes. These uses generally require a very high quality magnesia and the major part of the market is serviced by sea water plants where the chemical process can be carefully controlled. Beneficiation of materials from natural deposits to produce this high quality magnesia is generally not considered economical, but acceptable grades of calcined magnesia can be produced from some high grade natural deposits without the need for extensive beneficiation.

Other magnesium compounds which can be produced from magnesite include magnesium hydroxide, used in the manufacture of pulp and paper, pharmaceuticals, and sugar processing; magnesium chloride, used to produce magnesium metal, and in the manufacture of cements, ceramics and various chemicals; and precipitated magnesium carbonate which can also be used in many of the foregoing manufacturing processes.

In summary, the main market for magnesite is in the production of dead burned magnesite for refractory purposes. In addition there are a wide range of secondary products and uses. In order to be conservative, the examination of the feasibility of putting the Mount Brussilof magnesite deposit into production was based solely upon the market for dead burned magnesite. If the mine proves feasible for this basic product, then consideration can be given to producing other materials for which there appear to be good marketing possibilities but which in themselves could not justify initial mine production. The remainder of this Marketing Section will therefore be confined to a discussion of dead burned magnesite.

3. PRODUCTION

3.1 Existing Sources

Dead burned magnesite can be produced in two main ways. One is by processing ore from natural deposits of magnesite, dolomite, brucite or olivine. The other approach is to produce the material synthetically by combining salt water - either sea water or well brines - with calcined dolomite or lime.

The history of magnesia production has seen synthetic production capturing an increasing share of the magnesia market, which at one time was met entirely by production from natural deposits. The higher quality deposits now being developed are reversing this trend somewhat.

Initially, all magnesia refractories were produced from natural deposits. However, in the 1940's some highly industrialized countries, such as Great Britain and the United States, with insufficient natural deposits to meet their own demand began to construct sea water plants to ensure their magnesia supply. Sea water plants produced a more expensive magnesia but one which was of higher quality and which could be produced within close physical and chemical tolerances. As a result, the synthetic process, with its subsequent improvements, was gradually adopted by many other countries which had previously relied upon relatively low quality dead burned magnesite and dolomite.

In the early 1960's, new higher quality natural magnesite deposits were developed in Greece. Recent experience has shown that, aside from cost savings, refractories produced from high quality natural deposits have several important advantages over synthetically produced refractories. The relative qualities of the two types of product are discussed in more detail in Sub-section 5.1 of this section. Recent improvements in dead burned magnesite quality from natural deposits has resulted in a resurgence of the use of natural deposits as a source of magnesia refractories.

3.1.1 Natural Deposits

There is little published information on the known world reserves of magnesite. Table XII - 1, published in 1965 by the United States Bureau of Mines, indicates that at that time the known world reserves of magnesite for all countries with deposits of more than 1 million tons were 8.3 billion tons.[1]

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1 Unless stated otherwise, all tonnages given in the Marketing Section refer to short tons, i.e. 2,000 pounds.

TABLE XII - 1

WORLD MAGNESITE RESERVES, BY COUNTRY

Country	Million Tons	Percentage of Total
China	5,000	60
Korea, North	2,000	24
New Zealand	600	7
U.S.S.R	400	5
Czechoslovakia	100	1
India	100	1
United States **	65	1
Brazil	7	*
Venezuela	3	*
Greece	1	*
Turkey	1	*
Canada	1	*

* Less than 1 percent of the total.

** 27 million tons of the indicated U.S. reserves contain less than 5 percent MgO.

Source: U.S. Bureau of Mines, "Mineral Facts and Problems", 1965.

Approximately 90 percent of the world's reserves are in the communist countries. China alone is credited with 60 percent of the total. Subsequent to publication of Table XII - 1, a 2 billion ton deposit of crystalline magnesite, claimed to be the world's largest, has been discovered in eastern Siberia. Another deposit which is not reflected in Table XII - 1 is a band of crystalline magnesite which runs through Austria and which for many years has been the main source of supply for the European steel industry.

It can be seen from Table XII - 1 that in 1965 Canada was estimated to have only 1 million tons of raw magnesite. The discovery of the Mount Brussilof deposit, which already has proven reserves of about 15 million tons and for which the proven, probable and possible reserves are estimated to be in excess of 70 million tons, clearly puts Canada in a very strong position as a future source of world magnesite.

3.1.2 Sea Water Plants

Sea water has an average magnesium content of 0.13 percent by weight, and constitutes an inexhaustible source. Table XII - 2 lists the sea water magnesia plants of the world by country, company and capacity. This table is incomplete as it does not provide information on all plants.

The relative share of total magnesia production which is produced by sea water plants varies considerably from country to country. Statistics are always a few years behind, but as recently as 1968, roughly 20 percent of the total world magnesia production capacity was in sea water plants. Of the world's total synthetic magnesia production, the bulk was produced in the United States, Britain and Japan.

3.2 World Production

Available statistics on the recent annual production of magnesite from natural deposits are summarized for the major producing countries in Table XII - 3.

The most recent available worldwide statistics on the production of magnesia are for 1968. In that year, the total world production of crude natural magnesite is estimated to have been roughly 11,700,000 tons. In the same year, the estimated world production of synthetic magnesia from sea water plants was more than 1,200,000 tons.

TABLE XII - 2

WORLD SEA WATER MAGNESIA PLANTS

Country	Location	Company	Annual Capacity (Short Tons MgO)
Canada	Aguathuna, Newfoundland	Sea Mining Corp. Ltd.	30,000
Ireland	Dungarvan	Quigley Magnesite Ltd.	75,000
Italy	Sardinia	Steetley Magnesite Co. Ltd.	55,000
Mexico	Tampico, Vera Cruz	Quimica del Mar, S.A.	50,000
Norway	Heroya	Norsk Hydro-Elektrisk	60,000
United Kingdom	Hartlepool, England	Steetley Magnesite Co. Ltd.	250,000
U.S.S.R.	NA	-	100,000
United States	Cape May, N.J.	Northwest Magnesite Co.	50,000
	Port St. Joe, Fla.	Basic, Inc.	60,000
	Pascagoula, Miss.	H. K. Porter Co. Inc.	50,000
	Freeport, Tex.	The Dow Chemical Co.	250,000
	Moss Landing, Calif.	Kaiser Aluminum & Chemical Corp.	150,000
Japan	NA	NA	650,000
		TOTAL	1,830,000

NA Not Available
* Acres Estimate

Source: U.S. Bureau of Mines, Minerals Yearbook, 1968, and Acres

TABLE XII - 3

WORLD PRODUCTION OF CRUDE MAGNESITE, BY COUNTRY*
(Short Tons)

Country	1964	1965	1966	1967	1968
North America: United States	W	W	W	W	660,000
South America: Brazil	103,331	137,394	140,071	120,430	120,000
Columbia	243	209	210	210	200
Europe: Austria	1,826,058	2,001,363	1,779,829	1,692,386	1,704,923*
Czechoslovakia	1,858,047	2,029,154	2,095,221	2,322,331	2,000,000
Greece	397,054	347,453	413,366	524,476	550,000
Italy	6,954	3,898	2,867	5,445	-
Poland	41,888	46,297	46,000	46,000	46,000
Spain	102,874	111,944	110,000	110,000	110,000
U.S.S.R	3,090,000	3,200,000	3,200,000	3,300,000	3,300,000
Yugoslavia	548,311	579,750	580,570	468,219	441,272
Africa: Kenya	187	74	747	465	NA
Rhodesia, South	42,410	39,242	33,000	NA	NA
South Africa	94,443	95,789	102,847	88,199	65,915
Sudan	-	-	3,307	3,307	NA
Tanzania	546	1,260	5,270	2,246	NA
Asia: China, Mainland	1,100,000	1,100,000	1,100,000	880,000	990,000
India	229,210	264,346	255,650	270,893	278,264
Iran	6,033	9,259	6,790	6,600	7,000
Korea, North	990,000	990,000	1,100,000	1,375,000	1,275,000
Pakistan	680	577	812	2,240	2,200
Turkey	43,065	83,320	106,934	93,651	129,742
Oceania: Australia	35,001	29,525	21,903	26,492	25,000
New Zealand	676	937	624	636	NA
TOTALS	10,516,001	11,071,791	11,106,018	11,339,226	11,705,516

W Withheld to avoid disclosing individual company confidential data

NA Not Available

* Does not include sea water plants production

Source: U.S. Bureau of Mines, Minerals Yearbook, 1968.

The annual growth of world magnesia production has been slow, but relatively constant. In recent years, the annual growth rate has been between 1 and 2 percent. Some countries, such as Greece, Korea, Turkey, India and Brazil, have had growth rates considerably above the world average while Yugoslavia, China, Austria, South Africa and Australia all have declined in production.

From the available statistics, it appears that the communist countries presently account for almost 70 percent of the world's production. We know, however, that these countries do not enjoy such a large share of the world's refractory operations. It is concluded, therefore, that production figures for some of the communist countries include low grade products and large amounts used for purposes other than refractories, such as for fertilizer. Of the communist block countries, the largest producer was Russia, followed by Czechoslovakia, North Korea and then China.

Of the remaining world production, amounting to 3.6 million tons* in 1968, Austria produced 1.7 million tons. The first commercial refractory magnesite in the world was mined in Austria in 1880 and since that time Austria has supplied a substantial share of the free world's requirements.

Greek magnesite has gained wide acceptance in the last few years, and as can be seen in Table XII - 3, in the four years prior to 1968, production increased 39 percent to 550,000 tons per year.

3.3 Canadian Production

At the present time, there is no company in Canada which is mining and producing high quality refractory magnesia. Because of the previous lack of high quality magnesite deposits, Canada's magnesia requirements to date have been met by imports, details of which are given in Sub-section 4, and by lower quality dolomite refractories.

Total Canadian magnesia consumption at present is in the order of 90,000 to 100,000 tons per year; however, the bulk of this is not dead burned magnesite and is used for purposes other than as a refractory. The majority of the refractory

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* This does not include Japanese production for which statistics were not available.

magnesia is supplied by Canadian Refractories, who import from the United States, Yugoslavia and Greece. Canadian Refractories Limited also produce some material from their own mine. This product is a relatively low grade magnesite, but because of unique qualities has proven particularly suitable for the lining of cement kilns, and has received wide acceptance for this purpose in the North American market. General Refractories, in Smithville, Ontario, also manufactures refractory materials, primarily from imported Greek magnesite.

Most Canadian steel companies buy their refractory materials from either Canadian Refractories or General Refractories. One exception is Dofasco, who are producing their own material from a nearby dolomite deposit. Dolomite bricks tend to hydrate, and therefore cannot be stored for any length of time. The proximity of the Dofasco dolomite deposit permits the manufacture of bricks with a minimum of storage afterwards, and Dofasco claim that their tar-bonded dolomite bricks are meeting their needs.

Canadian Magnesite Mines are operating a mine near Timmins, Ontario. This deposit is about 92 percent MgO and 6.5 percent Fe_2O_3. A process has been developed in which the high iron content is leached out. This produces a very high quality magnesia at high cost. The product is used in the manufacture of certain drugs and pharmaceutical materials and is not competitive as a commercial refractory.

Canada's only sea water plant, at Aquathuna in Newfoundland, commenced producing magnesium hydroxide and magnesium oxide from sea water in September 1968. Plant production of 30,000 tons per year of caustic calcined magnesia was geared towards products for the pulp and paper industry. The initial managers, Sea Mining Corporation Limited, experienced difficulties and in 1970 the plant was bought by Lundrigans Limited. A five year supply contract has been signed with the Cohart refractories division of Corning Glass Works and production is expected to start again shortly.

4. TRADING PATTERNS

Tables XII - 4 and XII - 5 present world import and export data by country. From these tables certain conclusions may be drawn as to trading patterns of the various countries.

TABLE XII - 4

IMPORT OF MAGNESITE, BY COUNTRY
(Short Tons)

Country	1966	1967
North America: Canada	NA	39,240
U.S.A.	118,459	119,852
Western Europe: Belgium, Luxembourg	3,731	3,600
France	42,154	39,620
Germany W: Crude	1,142	1,508
Caustic calcined, sintered, fired	313,996	302,259
Refractories	37,389	24,616
Italy	45,439	52,979
Netherland	36,907	22,347
Norway	3,753	2,348
Spain	7,212	6,420
Sweden	15,924	7,249
Switzerland	4,114	3,362
Austria: Crude	32,314	3,988
Sintered, calcined	23,417	23,786
Other products	1,824	1,667
Finland	2,038	2,059
Eastern Europe: Hungary	78,180	75,777
Poland	154,671	104,147
Rumania	62,400	-
Asia: Japan: Magnesia clinker	22,100	34,000
Calcined and dead burned	4,410	4,420
Middle East: Israel	1,174	1,134
South America: Argentina	2,188	2,718
Chile	4,872	2,727
Mexico	39,865	31,762
Peru	1,273	3,234
Africa: South Africa	80,598	55,133
Australia	22,378	18,127
TOTALS	1,163,922	989,079

Source: U.S. Bureau of Mines

TABLE XII - 5

EXPORT OF MAGNESITE, BY COUNTRY
(Short Tons)

Country	1966	1967
North America: U.S.A: Dead burned	-	64,369
Other	-	7,788
Western Europe: Austria: Crude	412	386
Sintered	223,969	183,310
Caustic calcined	91,530	79,623
Bricks and plates	136,749	134,801
Other not burned	97,011	82,070
Greece	145,294	148,224
Italy	265	87
Spain	16,047	10,428
Eastern Europe: Czechoslovakia	207	217
U.S.S.R: Magnesite powder	15,500	-
Other	80,600	118,200
Yugoslavia: Crude	2,855	3,580
Calcined	17,068	12,378
Sintered	83,088	68,515
Middle East: Turkey: Crude	41,643	14,515
Calcined	24,146	22,310
Asia: China	-	-
India	19,916	21,920
Japan	51,100	10,900
Korea	-	-
Africa: South Africa	3,899	6,378
Australia	2,306	1,879
South America: Brazil	4,653	4,700
TOTALS	1,058,258	1,004,578

Source: U.S. Bureau of Mines

In 1967 the total world magnesite production, including sea water production, was roughly 12.6 million tons. Of this, about 980,000 tons, or 8 percent, was exported from its country of origin. The largest exporting countries in order of magnitude are Austria, Greece and the U.S.S.R. Although the Austrian magnesite is high in iron and relatively low in MgO content, it continues to be the main source of European magnesite. The higher quality Greek magnesite is making steady inroads into not only the European market, but also the American market. In the non-communist world, exports amounted to 780,000 tons, or 22 percent of non-communist production. Total exports from the non-communist countries were more than three times as great as those from the communist world.

The largest magnesite importers were West Germany, the U.S.A., Poland and Hungary. As would be expected, the major importers are also major steel producers.

Two countries which play a minor role in world magnesia import and export markets are Great Britain and Japan. Both countries are major steel producers and both have no appreciable natural magnesia deposits. The apparent discrepancy is explained by the fact that both countries are largely self-sufficient and supply their refractory requirements from their considerable sea water plant capacity.

To date, there has been little trading of magnesite between communist and non-communist countries. Lately, however, the newly started magnesite industry in Hungary has developed markets in West Germany and Italy. In addition, both Yugoslavia and China have started to export limited quantities to non-communist countries. In view of the apparently vast reserves of magnesite in the communist countries, this trend is likely to continue. There is some danger that, if the communist countries started to export on a large scale, it could have a depressing effect on world market magnesite prices.

A breakdown of Canadian imports for the last three years is shown in Table XII - 6. It can be seen that the United States has been the major source of supply, followed by Yugoslavia, Greece and West Germany. Imports have been increasing steadily each year.

TABLE XII - 6

CANADIAN MAGNESIA IMPORTS*

Country of Origin	1967 Quantity Short Tons	1967 Value $	1968 Quantity Short Tons	1968 Value $	1969 Quantity Short Tons	1969 Value $
Greece	2,200	167,000	1,430	108,000	3,360	302,000
Yugoslavia	4,140	275,000	5,280	370,000	5,040	415,000
Japan	770	59,000	5	-	-	-
United States	31,130	2,523,000	35,700	2,913,000	32,500	3,046,000
West Germany	-	-	-	-	4,840	189,000
Spain	-	-	-	-	15	51,000
Czechoslovakia	-	-	-	-	1,530	55,000
TOTAL	38,240	3,024,000	42,415	3,390,000	47,285	4,057,000

* Dead burned and sintered magnesia

Source: D.B.S.

5. QUALITY AND PRICE PERSPECTIVE

5.1 Quality Comparison

The most conclusive determination of the quality of a dead burned magnesite is obtained by measuring its useful life as a refractory under actual working conditions. Short of making exhaustive and time consuming empirical comparisons, it is possible to determine the product quality obtainable from a deposit by conducting laboratory firing tests and by comparison of chemical analyses.

The chemical analyses of dead burned magnesites vary widely as can be seen in Table XII - 7. This table is a sampling of published and company supplied chemical specifications. The MgO contents shown are the minimum for the products and the percentages for the impurities are intended to be a maximum. Generally speaking, the higher the MgO content, the better the quality of the material. This is not always true, however, as pointed out in Section V, since the amount of each impurity and in certain cases the ratio of one impurity to another can be the final factor in determining quality.

For comparative purposes, the chemical analysis of dead burned magnesite from the Brussilof Mount Eon deposit is included in Table XII - 7. The Mount Eon analysis shown represents the product which could be expected when processing average ore grade. The proven ore reserves approximate 15 million tons of magnesite, large sections of which contain higher than average grade material. Indeed, the drilling program indicates that by selective mining at slight additional cost a higher quality magnesite than shown in Table XII - 7 can be extracted. For this reason, and because the obviously large quantities of ore at Mount Brussilof and at Cross River have not been fully explored, it is considered very conservative to base the product comparison on the premise that the processing plant handles average ore grade.

The extraction of magnesia from sea water is a chemical process and theoretically the chemical make-up of the final product can be carefully controlled to produce the best possible magnesite refractory. In practice, however, this is not usually the case. The production of sea water plants can be virtually doubled by the addition of dolomite to the

TABLE XII - 7

CHEMICAL QUALITY COMPARISON OF DEAD BURNED MAGNESITES

	MgO %	CaO %	SiO_2 %	Fe_2O_3 %	Al_2O_3 %	B_2O_3 %
CANADA						
Brussilof Magnesite:						
Beneficiated Average Grade	95.3	2.1	1.1	0.8	0.3	Traces
UNITED STATES						
Kaiser Refractories:						
Magnesite 90	88.5	5.5-7.5	2.5-4.4	0.7	0.5	-
Magnesite 95	94.0	2.5-3.5	1.25-1.75	0.5	0.5	Nil
Periclase 96	95.7	*	1.75-2.25	0.5	0.5	0.25
Periclase 98	97.7	*	0.5 Max.	0.3	0.3	0.25
Northwest Magnesite Co. (Chewala)**	85.0	4.5-6.0	6.0-8.0	*	*	*
Bethlehem Steel Corporation	83.3-85.0	3.0-6.0	4.4-6.0	7.2	1.2	*
Continental Ore Company	95.5	2.5	1.6	0.12	0.12	*
Dow Chemical Company (sea water):						
Synthetic C-1	96.0	0.5	0.4	*	*	*
Synthetic L-2	95.0	1.25	*	*	*	*
GREECE						
Financial Corporation of Greece:						
E21A	95.0	2.8	1.4	0.5	*	*
E21B	91.85	4.5	2.65	0.75	*	*
E21S	96.0	2.0	1.0	0.5	*	*
N11A	95.0	2.0	2.5	0.4	*	*
Scalistiri Natural:						
Greek IA	94.0	1.8	2.8	0.2	0.2	*
IB	93.5	1.8	3.8	0.2	0.2	*
IC	90.0	2.5	5.5	0.2	0.2	*
ISL	95.5	2.5	1.6	0.2	0.2	*
Greek IIA1	94.6	3.2	1.6	0.4	0.1	*
A2	92.8	3.6	2.5	0.7	0.1	*

/Contd.

TABLE XII - 7 (Contd.)

CHEMICAL QUALITY COMPARISON OF DEAD BURNED MAGNESITES

	MgO %	CaO %	SiO_2 %	Fe_2O_3 %	Al_2O_3 %	B_2O_3 %
TURKEY						
Natural 1	95.3	2.7	1.3	0.3	0.1	*
2	94.6	1.3	3.4	0.1	0.1	*
ENGLAND						
Steetley Sea Water Magnesia:						
Britmag 112P	96.0-97.0	0.8-1.0	0.7-0.9	1.3-1.5	0.3-0.5	*
Britmag 212SP	95.0-96.0	1.4-1.8	0.7-0.9	1.3-1.5	0.3-0.5	*
Britmag 114CP	85.0-86.0	0.8	1.0	4.0	2.5	*
Britmag 132	93.0-95.0	0.8-1.0	3.0-3.5	1.3-1.5	*	*
Britmag 222	93.0-94.0	2.0-2.4	1.8-2.2	1.3-1.5	0.5-0.7	*
Britmag 215	91.0-93.0	1.5-1.7	0.8-1.0	4.5-5.5	0.3-0.5	*
Britmag 525	87.0-89.0	4.0-4.5	2.0-2.5	4.5-5.5	0.5-0.8	*
GERMANY Martin and Pagenstecher	95.1	2.5	1.4	0.5	0.5	Traces
AUSTRIA	87.0-89.0	2.5	1.7-2.1	4.0-8.2	0.9-2.4	*
CZECHOSLOVAKIA	85.2	3.7	1.3	7.1	1.6	*
NORTH KOREA	91.0	2.0	3.5	2.0	1.5	*
RUSSIA Satka Deposit	91.6-92.8	1.4-1.7	2.0-2.6	*	*	*

* Information not available
** Plant closed 1968

Source: Acres

process and this is generally done in order to reduce costs. However, with the addition of dolomite to the process, many of the impurities associated with the dolomite are incorporated in the magnesia. These impurities, such as boron, are difficult to extract and even small quantities of them can be detrimental to the useful life of the final product.

The firing temperature and retention time required to sinter sea water magnesia increase with the MgO content. These requirements increase rapidly when the MgO content exceeds 95 to 96 percent. To lower the sintering temperature and reduce costs, impurities are frequently added to the synthetic magnesia prior to firing. Thus, as can be seen from Table XII - 7, the chemical composition of most sea water magnesites varies from an MgO content of 85 percent to a high of 97 percent, as compared to the average Brussilof product with an MgO content of 95.3 percent.

Under working conditions, it has been found that magnesites from natural deposits are generally superior to sea water magnesites despite the fact that chemically they may not always appear to be as high grade. The reason for this anomaly is found in the shape of the crystals and the effect of trace elements on the interstices of the dead burned product.

In recent years, the Grecian dead burned magnesites have become the standard for comparison purposes and the demand for such high grade dicalcium silicate natural dead burned magnesites continues to exceed the supply. As can be seen from Table XII - 7, the average grade of Brussilof magnesite compares very favourably with the Grecian products. An examination of the drill hole analyses presented in Section VI shows that, with selective mining, the Mount Eon deposit could supply material with an acceptable ratio of impurities and which is higher in MgO content and lower in impurities than any of the Grecian products shown.

Spectrographic analyses of drill core material presented in Section VII shows that Mount Eon magnesite contains only minute quantities, if any, of potentially detrimental elements. The boron content is well within the limits set by specifications for refractory materials.

Bench scale tests for bulk density of the finished product have shown that the material can be pelletized and single burned at an economical temperature to produce a bulk density of 3.37, which corresponds to high quality dead burned magnesite products.

In conclusion, the site exploration and sample testing described in this report indicate that the Brussilof magnesite deposit is one of the largest deposits of high quality natural magnesite known. With the beneficiation facilities provided for in the capital cost estimates, the Brussilof dead burned product will meet high quality refractory specifications.

5.2 Price Review

As is the case with most industrial minerals, magnesite products are susceptible to price fluctuations reflecting variations in the supply and demand of the product, import and export restrictions and the location of the market. In addition, since the quality and properties of dead burned magnesites vary widely, the price is directly related to the quality of the specific product. For these reasons, it is hazardous to generalize about long-term price trends. However, because of the fact that dead burned magnesite demand is primarily related to the steel-making industry, which is steadily growing, and because the mining and processing costs for dead burned magnesite will probably increase with time, it is reasonable to suppose that for high quality products there will not be any long-term price reductions. An examination of price trends in recent years shows a gradual increase which supports this conclusion.

Data were gathered from several different sources for the analysis of prices. Some came from existing manufacturers, some from companies specializing in the trading of industrial minerals, and some of the information was published in trade journals and governmental reports. Understandably, the price data covered a wide range of product qualities and points of sale. In order to bring the data into perspective, it was necessary to convert all prices to their equivalent at a common location. This was accomplished by reducing all prices to their equivalent f.o.b. Vancouver, as if Vancouver were the port of supply. This approach has the added benefit that one can readily compare the cost of Brussilof dead burned magnesite f.o.b. Vancouver with the equivalent sales price of competitive products at the same location.

The unreduced dead burned magnesite price data ranged from $112 per long ton f.o.b. German ports for high quality material to $38 per long ton c.i.f. a European port for low quality Czechoslovakian material.

The price data were reduced by subtracting allowances for freight and handling, and insurance costs where specified.

The first cost factor to be subtracted was the freight cost incurred in moving the product to the market. Freight costs will vary with distance, the quantity being transported and the frequency of shipping. For the purpose of this analysis, the following sea freight rates were assumed: for shipping 150,000 tons per year or more, $14 Canadian per ton between European ports and Vancouver and $12 per ton between the eastern coast of the United States and Vancouver. If the quantity shipped per year is less, say in the order of 50,000 tons, the above prices would increase to somewhere between the $21 to $24 and $15 to $20 range respectively. Rail freight costs in North America are difficult to establish on a preliminary basis for an uncommon material like magnesite. The total rail costs could be in the range of $17 to $23 per ton, depending upon the quantity shipped and the point of delivery. For comparative purposes, we know that magnesite shipped from Luning, Nevada, to Bronte, Ontario, is presently charged at the rate of U.S. $24.40 per ton if 90 ton rail hoppers are used, and $28.60 per ton for 40 ton hoppers.

Allowance is made in the operating cost estimates for the cost of sales. Accordingly, for the purpose of this selling price comparison, sales costs were not deducted. An additional cost factor which did not enter into the calculation, but which bears mention, is the cost of import duties in the country of purchase. These custom tariffs can amount to a large share of the selling price, but since all original cost data were in terms of f.o.b. or c.i.f. the port of delivery, the tariffs had not been included and would have been paid by the buyer subsequent to unloading.

After conversion to their equivalent prices f.o.b. Vancouver, as described above, there was still a considerable range in dead burned magnesite prices. This range was directly related to the quality of the various products. Product quality, more than any other factor, has a marked effect on the sales price. The discussion of quality factors in

Section V illustrated how fractional changes in chemical and physical properties can have a major effect on the serviceability and value of the product. Thus, two products which at first glance might appear to have similar characteristics could well command widely different prices.

The reduced price data were therefore divided into three broad categories - high, medium and low quality products. The prices within each of these categories showed a high degree of correlation.

The equivalent price in Canadian dollars of high quality dead burned magnesite f.o.b. Vancouver ranged between \$65 and \$90 per short ton, the average being \$75. Medium quality magnesite ranged between \$45 and \$60 per short ton, with an average price of \$54. The lower quality grades ranged between \$15 and \$41 per short ton, with the average in the region of \$26 per short ton.

6. MOUNT BRUSSILOF MAGNESITE

The previous sections in this market analysis have examined the world market for dead burned magnesite. From the above information, conclusions may be drawn as to the potential market and price for Brussilof dead burned magnesite.

6.1 Potential Market

The Brussilof deposit is one of the highest quality natural deposits in the world. The dead burned product will be a high grade natural magnesite with high MgO content, correct lime-silica ratio, low iron content, and high bulk density. The Brussilof product would be comparable to the best presently produced from natural deposits and would have operating characteristics which would also compare favourably with most synthetically produced refractories. The high quality and relatively low mining and processing costs will be a decided advantage in securing markets for the product.

In estimating the initial market potential, three separate approaches will be used:

(a) By estimating the possible share of each individual market.

(b) By considering the response and offers from major consumers, agents or distributors.

(c) By making comparisons with the growth in production from similar deposits.

From a combination of the results of these three independent approaches, an estimate of the potential share of world markets can be made.

At present, the main potential markets for Canadian refractories are in Canada, the United States and in continental Europe. (Japan and Britain import very little, producing their own synthetically, and the communist countries have very large natural reserves and production capacity.) Once the mine is in operation and production of a consistently high quality product is demonstrated, there will no doubt be other potential market areas, such as Australia, to be developed.

In the meantime, it is necessary to concentrate on the three major areas mentioned above. Table XII - 8 summarizes the import data for these three market areas and shows the assumed share of present imports which we feel could reasonably be supplied by Brussilof in the first year of production. Of the total estimated demand of about 90,000 tons, some 9,000 would be in Canada, 42,000 in the United States and 39,000 in Europe.

It appears reasonable to assume that eventually the Brussilof deposit could supply most of what is presently imported into Canada after having allowed for special products or quotas due to trading agreements. Initially, however, we must assume a fairly low share of this market, since it is relatively small and contracts held by one or two companies can substantially reduce the present market. Accordingly, Acres estimated that 20 percent of present Canadian imports, or 9,000 tons, could be supplied in the first year.

Most American imports are from Europe. Magnesite produced in Canada would, therefore, have a price advantage because of slightly lower transportation costs and lower tariffs. We estimate that perhaps 35 percent of current American imports, or 42,000 tons, could be replaced by Mount Brussilof material.

TABLE XII - 8

ESTIMATED FIRST YEAR MARKET POTENTIAL
(Short Tons)

Market Area	Imports	Assumed Share of Market
Canada	47,285	9,000
U.S.A.	119,852	42,000
Europe	498,258	39,000
TOTALS	665,395	90,000

Import data are for 1967, with the exception of Canada which is for 1969.

In Europe, a Canadian product would be at a price disadvantage because of additional freight costs, therefore it has been assumed that only 8 percent of present imports, or roughly 39,000 tons, could be captured.

The second indicator of demand is from expressions of interest from potential buyers or sellers. Two companies have already expressed a desire to act as agents or distributors. Continential Ore Corporation, of New York, have said in a recent letter that "if you can meet our quality requirements at competitive prices ... we would be interested in concluding a contract for a quantity in the range of 75,000 to 100,000 tons annually of dead burned magnesite." It is worth noting here that European representatives of Continental Ore Corporation state that a product which is comparable in quality to the Greek Scalistiri grades ISL and IIA1 as shown in Table XII - 7 would presently command a price of U.S. $100 per metric ton f.o.b. Vancouver. All indications are that the quality requirements can be satisfied.

In addition, in an ongoing correspondence with N.V. International Ertshandel Wambesco, a wholly owned subsidiary of one of the largest trading firms in the Netherlands, Wambesco stated that, based on today's market situation, they would be capable of selling the production of one kilm (say 60/70,000 tons of product per year) with a maximum of the production of two kilms (120/140,000 tons per year) from the start of production.

Finally, we must consider the recent spectacular growth of the Greek magnesite industry. In four years, production increased 40 percent from 397,000 tons to 550,000 tons per year and demand continues to exceed production. The average quality of the Greek products is very similar to the expected Brussilof product, which indicates that there will be a good potential demand for Brussilof dead burned magnesite.

In summary, all the indications are that there exists a strong demand for high quality dead burned magnesite from the Brussilof deposit. It is felt that, assuming a suitable sales program is developed, a market for at least 90,000 tons of dead burned magnesite could be found in the first year of operation.

The growth of the market after the first year will depend very much upon a good initial production and quality record, and an aggressive sales effort. Additional production capacity can be installed by increments as the demand increases.

6.2 Anticipated Selling Price

In Sub-section 5.2, those factors affecting the price commanded by a specific dead burned magnesite product were outlined in some detail. One factor which was not eleaborated upon was that of the effect of competition from other suppliers. Clearly, in those markets where existing suppliers have large profit margins, they will probably be prepared to take a considerable reduction in profits in order to prevent their being replaced. Accordingly, prices for the same product will vary from place to place.

Specific markets and the corresponding sales prices for Brussilof dead burned magnesite will not be known until after an aggressive sales effort has been in effect for some time. In the meantime, it is necessary to make certain assumptions which can be shown to be conservative.

It is reasonable to assume that in any particular market the Brussilof product can command at least the same price as that presently paid for an inferior quality material. The quality and price perspective provided by Sub-section 5 showed that the product would be of high quality. For comparative purposes, existing sources were divided into the broad categories of high, medium and low quality products. The corresponding prices were expressed in terms of the equivalent price f.o.b. Vancouver. The average prices were as follows: $75 per ton for the high quality group, $54 for the medium and $26 for the low quality category. The range of prices for the high quality group is from $65 to $90.

The Brussilof product would be vastly superior to those in the medium and low quality categories and would be better than or comparable to most in the high quality category. It is reasonable to expect, therefore, that a price in the order of $70 to $90 per ton f.o.b. Vancouver could be obtained.

To demonstrate the effect of variations in selling price on the project, feasibility is examined for a range of prices, namely \$60, \$80, \$100 and \$120 per ton f.o.b. Vancouver. The actual selling price will fall somewhere within this range.

XIII CAPITAL COSTS

CONTENTS

		Page
1.	INTRODUCTION	139
2.	ESTIMATE	141

XIII CAPITAL COSTS

1. INTRODUCTION

This section includes estimates for the following levels of production:

Stage I	-	75,000 short tons product per year
Stage II	-	150,000 short tons product per year
Stage III	-	225,000 short tons product per year.

The stage II and III estimates provide incremental costs over stages I and II respectively. These production levels cover the capacity range that can be expected over the first few years of operation, but are not necessarily the optimum production rates for the project. The assumptions regarding operating schedules and plant hourly capacities are specified in Section X, Table X - 1.

The bases on which various sections of this estimate were prepared are defined below:

(a) Civil - General plant arrangements were prepared to establish building areas and volumes. Unit prices were applied on the basis of recent construction experience assuming that construction commences in 1971-72. Preliminary designs for the kiln foundations and exhaust stacks were prepared for quantity take-off purposes and application of unit prices. As sub-surface soil investigations and site selections were not undertaken, some allowance has been made for possible adverse foundation conditions.

(b) Mechanical - Estimating quotations were obtained from suppliers for all major equipment items. The estimate is based on the use of new equipment. Allowances for fabricated steel, piping, instrumentation, installation etc. are based on recent experience on similar projects. Provision has been made for freight, taxes and duty, when applicable.

The necessity for and size of certain processing equipment has not yet been firmly established. The additional work required to establish these requirements is described in Section IX. The equipment cost allowances should be reviewed once this work has been completed.

(c) Electrical - A preliminary motor list and power requirement estimate was prepared and unit prices applied to establish the cost of motors, controls and lighting. The high voltage system, ball mill and blower drives and mine generators were priced on a component basis.

This estimate is considered to be accurate to within 15 percent if financing and authority to proceed is arranged in 1971. An escalation allowance should be made if these arrangements are delayed.

One additional capital cost estimate was undertaken to determine the economy-of-scale savings that could be achieved by constructing a plant having an annual capacity of 150,000 tons and exploying larger capacity equipment when applicable. The capital cost of such a plant is estimated to be approximately 1.4 million dollars less than the combined cost of stages I and II. Some operating cost savings would also be achieved because of the reduced number of production components in the larger plant.

2. CAPITAL COST ESTIMATE

SUMMARY

	Stage I $	Stage II $	Stage III $
1. MINE			
1.1 Preproduction	257,000	-	-
1.2 Mine Equipment	373,000	50,000	50,000
1.3 Crushing, Conveying, Storage	404,000	6,000	6,000
1.4 Mine Camp	146,000	12,000	13,000
1.5 Electrical Equipment	127,000	-	-
Sub Total	1,307,000	68,000	69,000
2. PROCESS PLANT			
2.1 Crushing	508,000	3,000	103,000
2.2 Beneficiating and Briquetting	2,251,000	2,209,000	2,209,000
2.3 Kiln and Dryer	2,232,000	2,232,000	2,232,000
2.4 Product Sizing and Storage	625,000	26,000	26,000
2.5 Land and Services	423,000	56,000	46,000
2.6 Electrical Equipment	721,000	578,000	578,000
Sub Total	6,760,000	5,104,000	5,194,000
3. TRANSPORTATION			
3.1 Road	1,234,000	270,000	440,000
3.2 Equipment	358,000	358,000	300,000
Sub Total	1,592,000	628,000	740,000
SUB TOTAL	9,659,000	5,800,000	6,003,000
4. WORKING CAPITAL	1,450,000	-	-
5. PROJECT MANAGEMENT	1,000,000	200,000	200,000
6. CONTINGENCY	1,500,000	1,000,000	1,000,000
TOTAL	13,609,000	7,000,000	7,203,000

CAPITAL COST ESTIMATE

	Stage I $	Stage II $	Stage III $
1. MINE			
1.1 PREPRODUCTION			
Roads	50,000	-	-
Clearing	4,000	-	-
Overburden Removal	50,000	-	-
Waste Removal	63,000	-	-
Adit	37,500	-	-
Ore Pass	20,000	-	-
Reclamation Studies and Guarantees	32,500	-	-
TOTAL PREPRODUCTION	257,000	-	-
1.2 MINE EQUIPMENT			
Drilling Equipment	160,000	-	-
Front End Loader	105,000	-	-
Tractor	78,000	-	-
Dump Truck	30,000	-	-
Miscellaneous		50,000	50,000
TOTAL MINE EQUIPMENT	373,000	50,000	50,000
1.3 CRUSHING, CONVEYING, STORAGE			
1.3.1 Civil			
Crushing Plant	54,000	-	-
Conveyor Foundations and Bins	37,000	-	-
Sub Total Civil	91,000	-	-
1.3.2 Mechanical			
Maintenance Crane	8,000	-	-
Grizzly Feeder	17,000	-	-
Jaw Crusher	110,000	-	-
Conveyor System and Storage Bin	83,000	2,000	2,000
Chutes and Fabrications	10,000	1,000	1,000
Sub Total	228,000	3,000	3,000
Tax and Freight	20,000	2,000	2,000
Installation	65,000	1,000	1,000
Sub Total Mechanical	313,000	6,000	6,000
TOTAL CRUSHING, CONVEYING, STORAGE	404,000	6,000	6,000

CAPITAL COST ESTIMATE

	Stage I $	Stage II $	Stage III $
1.4 MINE CAMP			
1.4.1 Civil			
Camp Site	8,000	-	-
Mine Service Buildings	55,000	-	-
Powerhouse Building	10,000	-	-
Sub Total Civil	73,000	-	-
1.4.2 Mechanical			
Workshop Equipment and Crane	20,000	3,000	3,000
Water Pumps and Chlorinator	8,000	-	-
Water Reservoir	3,000	-	-
Sewage Treatment Plant	4,000	-	-
Fuel Storage Facilities	5,000	3,000	3,000
Sub Total	40,000	6,000	6,000
Tax and Transportation	4,000	1,000	1,000
Installation	10,000	3,000	3,000
Heating	5,000	-	-
External Water and Sewage Piping	8,000	1,000	1,000
Water and Sanitary	3,000	1,000	2,000
Fire Protection	3,000	-	-
Sub Total Mechanical	73,000	12,000	13,000
TOTAL MINE CAMP	146,000	12,000	13,000
1.5 ELECTRICAL EQUIPMENT			
Electrical Generation	91,000	-	-
Motors, Motor Control, Lighting	36,000	-	-
TOTAL ELECTRICAL EQUIPMENT	127,000	-	-
TOTAL MINE	1,307,000	68,000	69,000

CAPITAL COST ESTIMATE

	Stage I $	Stage II $	Stage III #
2. **PROCESS PLANT**			
2.1 CRUSHING			
2.1.1 Civil			
Coarse Ore	40,000	-	30,000
Crushing Plant	100,000	-	-
Transfer Tower Building	10,000	-	-
Conveyor Foundations	16,000	-	-
Coarse Ore Reclaim	5,000	-	60,000
Piling	9,000	-	-
Sub Total Civil	180,000	-	90,000
2.1.2 Mechanical			
Truck Dump Discharge Feeders	12,000	-	-
Coarse Ore Conveyors	51,000	-	4,000
Fine Ore Conveyors	13,000	2,000	3,000
Screen	12,000	-	-
Coarse Crusher	50,000	-	-
Fine Crusher	53,000	-	-
Chutes, Pan, Hopper and Miscellaneous Fabrications	7,000	-	1,000
Dust Collector	10,000	-	-
Crane and Lifting Gear	9,000	-	-
Sub Total	217,000	2,000	8,000
Tax and Freight	24,000	-	1,000
Installation	61,000	1,000	4,000
Heating	8,000	-	-
Dust Collector Ducting	8,000	-	-
Water and Air Distribution	4,000	-	-
Fire Protection	6,000	-	-
Sub Total Mechanical	328,000	3,000	13,000
TOTAL CRUSHING	508,000	3,000	103,000

CAPITAL COST ESTIMATE

	Stage I $	Stage II $	Stage III $
2.2 BENEFICIATING AND BRIQUETTING			
2.2.1 Civil			
Fine Ore Bins	123,000	123,000	123,000
Grinding, Flotation and Briquetting Foundations	589,000	589,000	589,000
Pilings	36,000	36,000	36,000
Sub Total Civil	748,000	748,000	748,000
2.2.2 Mechanical			
Cranes and Lifting Gear	17,000	17,000	17,000
Feed Conveyor	4,000	4,000	4,000
Belt Scale and Controls	8,000	8,000	8,000
Primary Grinding Mill	93,000	93,000	93,000
Secondary Grinding Mill	207,000	207,000	207,000
Flotation Cells	25,000	25,000	25,000
Samplers	10,000	10,000	10,000
Slurry Pumps	13,000	13,000	13,000
Conditioners	6,000	6,000	6,000
Reagent System	10,000	10,000	10,000
Thickener Mechanism and Tank	36,000	36,000	36,000
Thickener Pumps	3,000	3,000	3,000
Pneumatic Conveyor System	25,000	25,000	25,000
Briquetting Surge Bins	12,000	12,000	12,000
Briquetting Presses	426,000	426,000	426,000
Air Compressors	11,000	11,000	11,000
Pump Boxes, Launders and Miscellaneous Fabrications	12,000	12,000	12,000
Sub Total	918,000	918,000	918,000
Tax and Freight	130,000	130,000	130,000
Installation	173,000	173,000	173,000
Process, Air and Water Piping	160,000	160,000	160,000
Heating and Ventilation	20,000	20,000	20,000
Dust Collection Ducting	10,000	10,000	10,000
Fire Protection	50,000	50,000	50,000
Front End Loader	35,000	-	-
Fork Life	7,000	-	-
Sub Total Mechanical	1,503,000	1,461,000	1,461,000
TOTAL BENEFICIATING AND BRIQUETTING	2,251,000	2,209,000	2,209,000

CAPITAL COST ESTIMATE

	Stage I $	Stage II $	Stage III $
2.3 KILN DRYER			
2.3.1 Civil			
Kiln Foundations	235,000	235,000	235,000
Dust Collector Foundation	21,000	21,000	21,000
Main Exhaust Chimney	120,000	120,000	120,000
Oil Storage Base	10,000	10,000	10,000
Provision for Pilings	20,000	20,000	20,000
Sub Total Civil	406,000	406,000	406,000
2.3.2 Mechanical			
Kiln and Auxiliary Equipment	394,000	394,000	394,000
Cooler	140,000	140,000	140,000
Oil Storage Tank	30,000	30,000	30,000
Cooler Dust Collector	15,000	15,000	15,000
Feeder and Controls	10,000	10,000	10,000
Discharge Dragchain	7,000	7,000	7,000
Laboratory Equipment	30,000	30,000	30,000
Brickhoists	5,000	5,000	5,000
Slurry Dryer, Dust Collection and I.D. Fan	600,000	600,000	600,000
Ducting and Chutes	20,000	20,000	20,000
Sub Total	1,251,000	1,251,000	1,251,000
Tax and Freight	130,000	130,000	130,000
Installation	295,000	295,000	295,000
Refractories Kiln	65,000	65,000	65,000
Refractories Cooler	20,000	20,000	20,000
Refractories Feed and Discharge	20,000	20,000	20,000
Air and Water Piping	20,000	20,000	20,000
Heating and Ventilation	15,000	15,000	15,000
Fire Protection	10,000	10,000	10,000
Sub Total Mechanical	1,826,000	1,826,000	1,826,000
TOTAL KILN AND DRYER	2,232,000	2,232,000	2,232,000

CAPITAL COST ESTIMATE

	Stage I $	Stage II $	Stage III $
2.4 PRODUCT SIZING AND STORAGE			
2.4.1 Civil			
Foundations, Building Shell	136,000	-	-
Bagging Warehouse	170,000	-	-
Provision for Pilings	18,000	-	-
Sub Total Civil	324,000	-	-
2.4.2 Mechanical			
Conveyors and Magnet	40,000	10,000	10,000
Screens	30,000	-	-
Hammermill	10,000	-	-
Dust Collector	20,000	-	-
Airslides Including Blower	8,000	-	-
Surge Tanks	10,000	-	-
Bagging and Bag Handling	15,000	2,000	2,000
Bucket Elevators	16,000	-	-
Silo Gates and Controls	7,000	6,000	6,000
Pans and Chutes	20,000	-	-
Sub Total	176,000	18,000	18,000
Tax and Freight	20,000	2,000	2,000
Installation	67,000	6,000	6,000
Dust Collector Ducting	10,000	-	-
Air and Water Piping	6,000	-	-
Heating and Ventilation	12,000	-	-
Fire Protection	10,000	-	-
Sub Total Mechanical	301,000	26,000	26,000
TOTAL PRODUCT SIZING AND STORAGE	625,000	26,000	26,000

CAPITAL COST ESTIMATE

	Stage I $	Stage II $	Stage III $
2.5 LAND AND SERVICES			
2.5.1 Civil			
Land	20,000	-	-
Grading, Fencing, Landscaping	80,000	10,000	-
Site Roads and Parking Lot	13,000	-	-
Water Supply Pumphouse	25,000	-	-
Tailings Disposal	40,000	10,000	10,000
Administration Building	50,000	10,000	10,000
Sub Total Civil	228,000	30,000	20,000
2.5.2 Mechanical			
Workshop Equipment and Crane	30,000	3,000	3,000
Water Pumps and Controls	22,000	6,000	6,000
Water Reservoir and Head Tanks	5,000	2,000	2,000
Water and Sewage Treatment	15,000	-	-
Sub Total	72,000	11,000	11,000
Tax and Freight	7,000	1,000	1,000
Installation	18,000	3,000	3,000
Piping Pumphouse	45,000	-	-
Plant Water Distribution	15,000	3,000	3,000
Effluent Lines	8,000	2,000	2,000
Dom and Sanit for Administration	12,000	4,000	4,000
Heating and Ventilation	18,000	2,000	2,000
Sub Total Mechanical	195,000	26,000	26,000
TOTAL LAND AND SERVICES	423,000	56,000	46,000
2.6 ELECTRICAL EQUIPMENT			
Electrical Distribution, Motors, Motor Control Lighting	70,000	50,000	50,000
(a) Storage and Crushing	85,000	-	-
(b) Beneficiation, Briquetting and Drying	373,000	373,000	373,000
(c) Calcination	193,000	155,000	155,000
TOTAL ELECTRICAL EQUIPMENT	721,000	578,000	578,000
TOTAL PROCESS PLANT	6,760,000	5,104,000	5,194,000

CAPITAL COST ESTIMATE

		Stage I $	Stage II $	Stage III $
3.	TRANSPORTATION			
3.1	ROAD			
	Road and Bridges, Mine Camp to Cross Kootenay Junction	675,000	-	-
	Road and Bridges, Cross Kootenay Junction to Palliser Bridge	314,000	-	-
	Road and Bridges, Palliser Bridge to Lower Kootenay Bridge	30,000	270,000	-
	Road and Bridges, Lower Kootenay Bridge to Canal Flats	155,000	-	440,000
	Palliser Bridge Rebuilding	60,000	-	-
	TOTAL ROAD	1,234,000	270,000	440,000
3.2	EQUIPMENT			
	Trucks	300,000	300,000	300,000
	Graders	58,000	58,000	-
	TOTAL EQUIPMENT	358,000	358,000	300,000
	TOTAL TRANSPORTATION	1,592,000	628,000	740,000

		Stage I $	Stage II $	Stage III $
4.	WORKING CAPITAL			
	Working Capital	750,000	-	-
	Spare Parts	200,000	-	-
	Interest During Construction	500,000	-	-
	TOTAL WORKING CAPITAL	1,450,000	-	-

XIV OPERATING COSTS

CONTENTS

		Page
1.	INTRODUCTION	151
2.	ESTIMATE	153
3.	DISCUSSION	157

TABLES

XIV - 1	Operating Staff	158

XIV OPERATING COSTS

1. INTRODUCTION

This section includes operating cost estimates for the following production levels:

Stage I - 75,000 short tons product per year
Stage II - 150,000 short tons product per year
Stage III - 225,000 short tons product per year

The estimates cover the total operating costs for each stage, as opposed to the incremental cost approach adopted in the capital cost estimates. The bases used in establishing these costs are:

(a) Labour costs are based on recently negotiated 1971 skilled and unskilled labour rates in the East Kootenay area of British Columbia.

(b) Operating supply allowances for processing consumables such as grinding media, liners and flotation reagents are based on recent data for similar projects. Allowances for replacement of mechanical and wearing parts are generally based on a percentage of original purchase price, or on a price per ton of material processed.

(c) Fuel prices for the natural gas and oil required in the firing operation are based on preliminary quotations from suppliers. Alterations to the process system, as described in Section IX, which may be required as further testing is carried out, may decrease or increase the projected operating costs. For this reason, no contingency factor has been applied to the operating cost estimate. However, an escalation allowance should be made if project financing is not arranged in 1971.

In order to attract and maintain a capable and stable work force, most new mining ventures are finding it necessary to institute a housing subsidy program. On the basis of a recent study by Acres for a mining operation in the East Kootenay area, a housing allowance of 75 dollars per month per employee is used in the operating cost estimate.

An allowance of 15 dollars per man per day is made to cover the cost of board and providing bunkhouse, kitchen and dining room facilities for employees working and living at the mine site. The capital cost estimate covers site preparation and servicing of these facilities.

Electric power costs are based on quoted rates received from the B.C. Hydro and Power Authority company which services the Canal Flats area. The energy and demand charges are based on a compilation of installed motor horsepowers.

2. OPERATING COST ESTIMATE

SUMMARY

	Stage I $		Stage II $		Stage III $	
	Annual Cost	Per Ton Product	Annual Cost	Per Ton Product	Annual Cost	Per Ton Product
1. MINE	179,000	2.40	257,740	1.72	353,550	1.57
2. PROCESS PLANT	775,070	10.33	1,222,840	8.15	1,636,310	7.27
3. SERVICES	594,900	7.93	1,163,300	7.76	1,732,900	7.70
4. TRANSPORTATION	673,200	8.98	1,272,800	8.49	1,871,900	8.32
5. ADMINISTRATION AND SELLING EXPENSE	528,000	7.04	693,700	4.62	831,900	3,70
6. ROYALTIES	187,700	2.50	375,000	2.50	562,500	2.50
TOTAL OPERATING COST	$2,938,960	39.18	4,985,380	33.24	6,989,060	31.06

OPERATING COST ESTIMATE

		Stage I $	Stage II $	Stage III $
1.	MINE			
1.1	Salaried Staff	33,800	46,000	63,600
	Hourly Workers	77,700	103,700	141,600
	Total Mining Labour	111,500	149,700	205,200
1.2	Drilling Supplies	5,800	11,600	17,400
	Blasting Supplies	8,240	16,480	24,720
	Load and Haul Supplies	15,700	31,400	47,100
	Crushing Supplies	1,980	3,960	5,940
	Conveying Supplies	2,920	4,000	5,840
	Equipment Supplies	33,850	40,600	47,350
	Total Mining Supplies	68,490	108,040	148,350
	TOTAL MINE	179,990	257,740	353,550

OPERATING COST ESTIMATE

		Stage I $	Stage II $	Stage III $
2.	PROCESS PLANT			
2.1	Salaried Staff	118,800	128,400	138,000
	Crushing Plant Labour	21,500	43,000	64,500
	Grinding, Beneficiating and Briquetting Labour	130,800	172,000	258,000
	Calcining Labour	86,000	127,200	127,200
	Utility Labour	11,400	11,400	11,400
	Product Plant	31,800	53,300	53,300
	Services Labour	101,900	157,700	194,600
	Total Processing Labour	502,200	693,000	847,000
2.2	Crushing Supplies	22,400	44,800	67,200
	Grinding, Beneficiating and Briquetting Supplies	145,950	287,900	429,350
	Calcining Plant Supplies	104,520	197,140	292,760
	Total Processing Supplies	272,870	529,840	789,310
	TOTAL PROCESS PLANT	775,070	1,222,840	1,636,310

OPERATING COST ESTIMATE

		Stage I $	Stage II $	Stage III $
3.	SERVICES			
	Kiln Fuel	396,000	792,000	1,188,000
	Electric Power	198,900	371,300	544,900
	TOTAL SERVICES	594,900	1,163,300	1,732,900
4.	TRANSPORTATION			
	Mine to Plant Site Labour	87,600	131,600	175,100
	Mine to Plant Site Supplies	135,600	241,200	346,800
	Rail Shipment to Vancouver	450,000	900,000	1,350,000
	TOTAL TRANSPORTATION	673,200	1,272,800	1,871,900
5.	ADMINISTRATION AND SELLING EXPENSE			
	Corporate Administration	63,500	64,500	64,500
	Selling Expense	150,000	200,000	250,000
	Plant Administration	270,900	369,000	435,000
	Minesite Administration	43,800	60,200	82,100
	TOTAL ADMINISTRATION AND SELLING EXPENSE	528,200	693,700	831,900
6.	ROYALTIES	187,700	375,000	562,500

3. DISCUSSION

The operating costs per ton of product for various production rates are:

$39.18 at 75,000 tons per year with one kiln or
$33.24 at 150,000 tons per year with two kilns or
$31.06 at 225,000 tons per year with three kilns.

It is apparent that appreciable economies can be achieved by operating at as high a production level as the market will support.

Additional estimates were prepared to determine costs for a 150,000 ton per year, single kiln plant operated at full capacity and at a rate of 100,000 tons per year. The unit operating costs were $31.51 and $35.02 respectively. These savings are attributable primarily to the reduced labour costs of a single kiln plant. The cost of operating supplies per unit of product will be similar with single or multiple kiln operations. The profitability and payback periods of this single line plant are illustrated graphically and compared with other cases in Section XVI.

Two major components of the operating costs are the processing plant and services. Significant economies-of-scale can be achieved in the processing section because the labour costs decrease from 71 percent to 52 percent of the total process operating costs as the capacity increases from 75,000 to 225,000 tons per year of product. With a 150,000 ton per year single line plant as compared to a double line plant, the labour costs could be reduced even further. On the other hand, the cost of services is not significantly affected by the level of production. Unit electrical energy costs decrease slightly with increased power consumption. The heat requirements for the kiln operation are based on 10 million BTU per ton of product and using kilns rated at 75,000 tons per year of product. This requirement will not be affected by the number of kilns installed; however, if larger kilns are employed, the thermal efficiency will be slightly higher.

TABLE XIV - 1

OPERATING STAFF

	TOTAL NUMBER ON PAYROLL		
	Stage I	Stage II	Stage III
MINE			
Salaried	2	2	4
Labour	6	8	11
Sub Total	8	10	15
PROCESS PLANT			
Salaried	9	10	11
Labour	35	52	65
Sub Total	44	62	76
TRANSPORTATION			
Labour	8	12	16
RESIDENT ADMINISTRATION	6	7	9
TOTAL OPERATING STAFF	66	91	116

XV PROJECT SCHEDULE

CONTENTS

		Page
1.	SUMMARY	159
2.	PROJECT SCHEDULE	161

FIGURES

XV - 1	Project Schedule	161

XV PROJECT SCHEDULE

1. SUMMARY

The project schedule presented in Figure XV - 1 illustrates that plant start-up can be achieved within 20 months of making a firm decision to proceed with the project, provided that:

(a) Laboratory and, wherever necessary, pilot plant investigations are undertaken to establish optimum processing conditions and long-delivery equipment requirements for the beneficiating, briquetting and calcining operations. This information is required to complete preliminary plant layouts and engineering prior to the preparation of construction tender documents.

(b) Senior financing is obtained at least 15 months before the planned production date.

(c) Plant site selection, sub-surface foundation investigations and preliminary planning are completed within 5 months of a positive decision to place this property into production. This will make it possible to effectively complete detailed plant design and have construction drawings ready when required.

(d) The civil portion of the construction contract can be started before the onset of adverse winter weather conditions.

The tentative start-up date is determined primarily by two factors. The first is the elapsed time between ordering and completing the installation of the long-delivery equipment items. Two of the longest delivery items are the rotary kiln and the secondary grinding mill. The current delivery times on this equipment are 9 to 11 months and 11 to 13 months respectively. Assuming installation times of 4 and 3 months respectively, the overall elapsed time between ordering and start-up is approximately 15 months.

The second major factor affecting the plant start-up is the date of senior financing. Several investigations are required before final plant design commences and major financial commitments are made. These investigations include market confirmation, definition of initial plant capacity requirements, process testwork referred to in Sections IX and X, site selection, foundation investigations preliminary engineering and project planning. It is estimated that 5 months will be required to complete this work and that major financial commitments can be deferred for this period without delaying the plant start-up date.

No serious obstacles are foreseen to constructing the plant within 15 months of senior financing, provided construction commences prior to the coldest winter months and provided sufficient preliminary engineering has been completed by the time senior financing is arranged.

2. PROJECT SCHEDULE

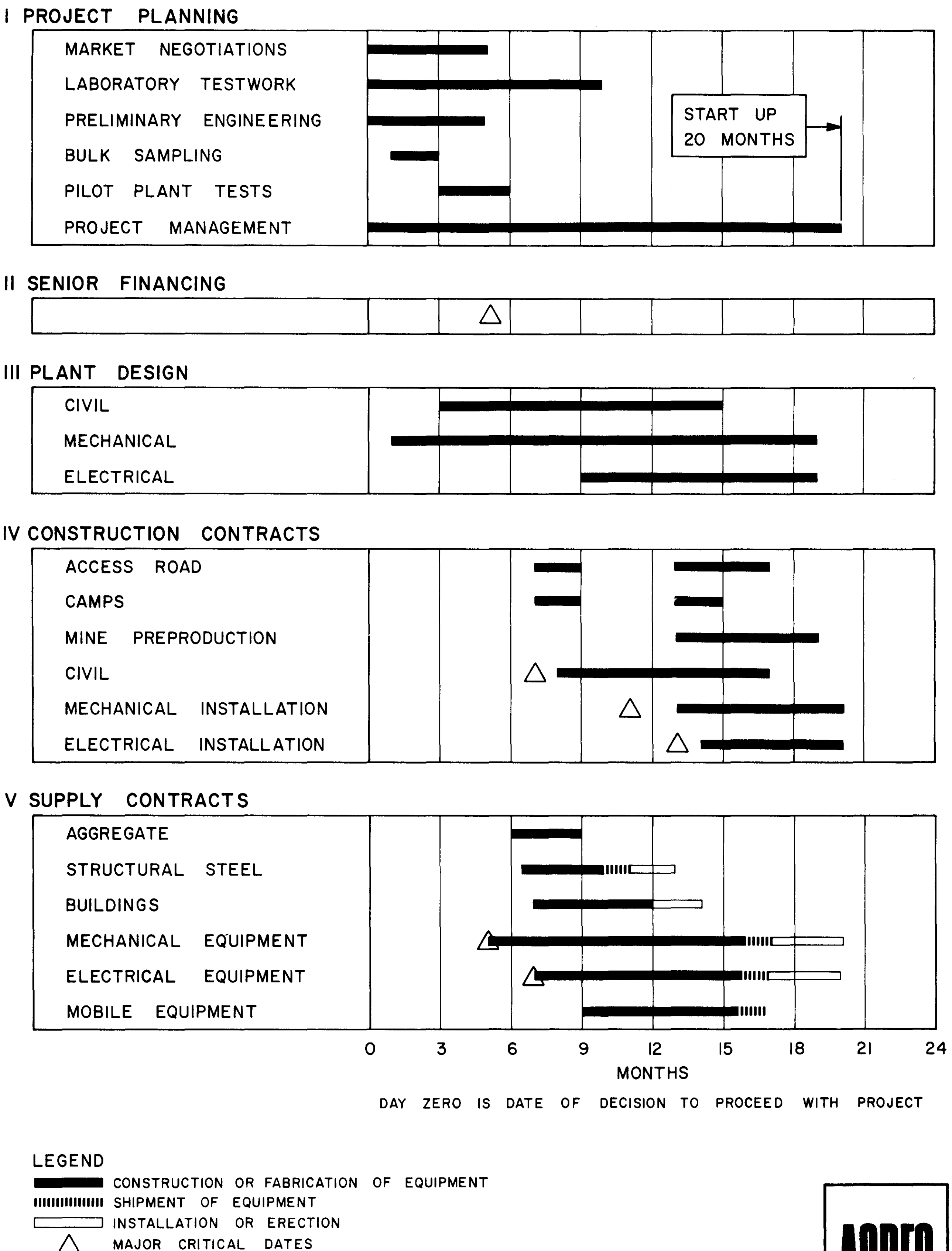

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FIGURE XV-I

XVI ECONOMIC FEASIBILITY

CONTENTS

		Page
1.	INTRODUCTION	165
2.	ECONOMIC VARIABLES AND ASSUMPTIONS	165
	2.1 Selling Price	
	2.2 Production Level	
	2.3 Taxation and Government Assistance	
	2.3.1 Taxation	
	2.3.2 Government Assistance	
	2.4 Financing	
3.	ECONOMIC ANALYSES	169
	3.1 Methodology	
	3.2 Results	
	3.2.1 Computer Cash Flow Analyses	
	3.2.2 Tabular Summary	
	3.2.3 Graphical Summary	
4.	DISCUSSION OF RESULTS	188
5.	CONCLUSIONS	196

XVI ECONOMIC FEASIBILITY

TABLES

		Page
XVI - 1	Alternatives Examined.	170
XVI - 2	Summary of Results of Economic Analyses.	189

FIGURES

XVI - 1	Payback Period vs. Selling Price at 30% Equity.	192
XVI - 2	Payback Period vs. Selling Price at 100% Equity.	193
XVI - 3	Rate of Return vs. Selling Price at 30% Equity.	194
XVI - 4	Rate of Return vs. Selling Price at 100% Equity.	195

XVI ECONOMIC FEASIBILITY

1. INTRODUCTION

The objective of this section is to combine the data from the foregoing sections in order to evaluate overall project viability over a range of conditions. Such an analysis will permit a decision by both owners and potential investors as to whether or not the project is an attractive investment.

Where assumptions have been made in this study, they have been conservative. It is expected that, as more detailed information becomes available, the changes will tend to increase the profitability of the project.

If the feasibility analysis indicates project viability, further study and investigations will be necessary prior to and during project design in order to optimize the economics of the operation.

2. ECONOMIC VARIABLES AND ASSUMPTIONS

A number of items which directly affect project viability cannot be finalized at this stage. Analyses are therefore performed on a variety of cases to illustrate the effect of variations in some of these factors on project viability. Understandably, the more variables that are incorporated, the larger the number of alternatives to be examined. For the sake of simplicity, certain assumptions were made on a rather arbitrary basis and further analysis will be required when more reliable information is available.

2.1 Selling Price

The product selling price is one of the major variables and one of the most difficult to predict. Selling price is directly related to the quality and serviceability of the product but there are virtually no published data relating price to dead burned magnesite quality.

As has been explained in the foregoing sections of this report, the mine will have the ability to produce several grades of product by practising selective mining techniques. In addition, the selling price which could be realized will depend upon the degree of success of the sales force in their negotiations with prospective buyers. For these reasons, it was necessary to perform the analyses for a range of product prices in order to relate the viability to changes in selling price. The prices used were $60, $80, $100 and $120 per ton f.o.b. Vancouver. It is considered that the prices commanded by Brussilof dead burned magnesites will fall within this range.

2.2 Production Level

Another major variable is the initial production level and the staging of increases in production capacity with time. This will depend upon a number of factors, particularly the quality of the product, the world market demand for that type of product and the relative economic performance of the various production capacity and staging alternatives.

Five separate cases are examined in this section, namely:

Case 1 - A single kiln operation producing 75,000 tons per year, and remaining at that level for the period of analysis.

Case 2 - A two kiln operation producing 150,000 tons per year for the period of analysis.

Case 3 - An initial single kiln operation producing 75,000 tons per year, followed by the addition of similar kilns in the fourth and seventh years of operation to produce 150,000 and finally 225,000 tons per year for the balance of the analysis period.

Case 4 - A single kiln operation producing 150,000 tons per year for the period of analysis.

Case 5 - A single kiln operation with a production capacity of 150,000 tons per year, but producing only 100,000 tons per year for the first 3 years and then expanding to full capacity for the balance of the analysis period.

2.3 Taxation and Government Assistance

2.3.1 Taxation

For the purpose of the study, it is assumed that the recommendations of the White Paper on Tax Reform will be in effect by the time production starts. In fact, if construction were completed in 1972, it is possible that the mine would have one year of operation before the Tax Reforms are implemented. For the sake of simplicity, however, the same conditions were assumed throughout.

Based on the White Paper recommendations, no three year tax exemption is allowed. All capital costs are written off against income as fast as they can be absorbed. Depletion allowances are restricted to one-third of expenditures on exploration and development.

To simplify the calculations, Federal and Provincial Income Taxes are combined. Assuming a Provincial Income Tax of 15 percent before depletion allowances and 50 percent Federal Income Tax after, a combined approximate Income Tax rate of 57 percent is used. This is about 10 percent higher than the equivalent Total Income Tax applicable at present.

An allowance for B.C. Property Taxes is made in the operating cost estimates.

2.3.2 Government Assistance

There is a reasonable expectation that some assistance will be forthcoming from the Federal Government to help establish this new industry. The most likely source is the Department of Regional Economic Expansion which administers the Regional Development Incentives Act, passed in 1969 for the purpose of creating new job opportunities in regions of slow economic growth.

Assistance is in the form of capital grants to industrial manufacturers for establishing, expanding or modernizing plants in designated regions of Canada. The mine site and the proposed location of the processing plant for this project are within such a designated region.

The Act provides that assistance may be given to the building of a plant for the "processing by roasting, leaching or smelting of mineral concentrates to produce metals", and the "processing, other than oil refining, of a product resulting in a significant chemical change in the principal material used". "Processing" as defined does not include costs attributable to such items as transportation and the extraction of minerals. The incentives for a new plant are up to 25 percent of capital costs plus up to $5,000 for each job created in the operation, with the maximum benefit established at the lesser of $12 million or $30,000 for each job created. For the purpose of this study, the $30,000 per job allowance was used.

To simplify the economic analyses, it was assumed that this grant would be available during construction, whereas in fact, 80 percent of the incentive is paid when commercial production begins, with the balance being paid within 3½ years. Thus, no allowance has been made in the study for the cost of borrowing short term money to cover the delay in grant payments. The cost of this borrowing is within the accuracy of the cost estimates.

An application for assistance must be made when the project is in the planning stage before any commitments are undertaken. Canadian manufacturers must be given reasonable opportunity to supply machinery and equipment and the operation must employ residents of the designated region to the maximum extent practicable. The operation must come into production by December 31, 1976.

A second possible source of assistance is through the Industrial Research and Development Incentives Act (IRDIA) administered by the Department of Industry, Trade and Commerce. No allowance has been made in the study for assistance from this source; however, all possible sources of assistance should be investigated prior to project implementation.

2.4 Financing

The method of raising the net capital cost, that is the total capital cost minus government grants, has not yet been decided. Variable factors in the consideration of financing at this stage, therefore, include the ratio of equity to debt capital, the interest rate paid on debt capital and the period of loan repayment.

Two conditions are examined in the feasibility analyses: one assumes 30 percent equity and 70 percent debt capital; the other assumes that the entire net capital cost will be raised by equity.

Where debt capital is used, it is assumed that the principal would be repaid in 10 equal annual repayments and that 10 percent interest would be paid on the declining balance.

3. ECONOMIC ANALYSES

3.1 Methodology

Based on the above assumptions, the total number of combinations of product capacity and staging, debt to equity ratios and product selling price resulted in 34 separate alternatives to be examined. The alternatives are detailed in Table XVI - 1.

To assist in the economic analyses, Acres developed a computer program to perform an income and cash flow analysis for each alternative. The program also calculated payback periods, and rates of return on investment and representative computer printout sheets are included in Section 3.2.1.

For the sake of uniformity, and to permit comparisons between the economic performance of various alternatives, each alternative is examined over a period of 20 years, and all figures are in 1971 dollars.

In the income and cash flow analyses, the operating income for each year is calculated by deducting operating costs from the annual sales revenue. The annual interest payment on debt capital is then deducted, if applicable, and the net capital cost written off against the remaining income as fast as possible until totally recovered. At that point in time, the accumulated depletion allowance assumed is deducted and depletion deductions continue for the period of analysis. As was explained in Section 2.3, the income tax is calculated by taking 57 percent of the remaining taxable income and the balance represents the net earnings for that year.

TABLE XVI - 1

ALTERNATIVES EXAMINED

Alternative No.	Production Case	Production Capacity Tons/Year (000's)	No. of Kilns	Equity Assumed %	Selling Price $/Ton	Page No. in Report
1	CASE 1	75	1	30	60	172-173
2		75	1	30	70	
3		75	1	30	80	
4		75	1	30	100	
5		75	1	30	120	
6		75	1	100	60	174-175
7		75	1	100	70	
8		75	1	100	80	
9		75	1	100	100	
10		75	1	100	120	
11	CASE 2	150	2	30	60	176-177
12		150	2	30	80	
13		150	2	30	100	
14		150	2	30	120	
15		150	2	100	60	178-179
16		150	2	100	80	
17		150	2	100	100	
18		150	2	100	120	
19	CASE 3	75,150,225	1,2,3	30	60	180-181
20		75,150,225	1,2,3	30	80	
21		75,150,225	1,2,3	30	100	
22		75,150,225	1,2,3	30	120	
23		75,150,225	1,2,3	100	60	182-183
24		75,150,225	1,2,3	100	80	
25		75,150,225	1,2,3	100	100	
26		75,150,225	1,2,3	100	120	
27	CASE 4	150	1	100	60	184-185
28		150	1	100	80	
29		150	1	100	100	
30		150	1	100	120	
31	CASE 5	100,150	1	100	60	186-187
32		100,150	1	100	80	
33		100,150	1	100	100	
34		100,150	1	100	120	

Two cash flow streams are projected, the cash flow to equity and the total payback to net capital cost. Cash flow to equity, being operating income minus interest, income taxes and loan repayments, takes into consideration the project financing arrangements and represents the return on the equity invested by the shareholders. The total payback to net capital cost adds loan repayments to the cash flow to equity and is an indicator of the economic performance of the project as a whole. In those cases where the equity is assumed to be 100 percent, the payback to net capital cost represents the total cash flow which is available for distribution to shareholders.

The values of the two revenue streams over the period of analysis for each alternative are summarized by calculating the resultant average annual rates of return on capital invested. Thus, the cash flow to equity stream is converted to the equivalent average annual rate of return on equity while the payback to net capital cost stream yields the average annual rate of return on net capital cost. These rates of return are calculated by the present worth method. Where capital expenditures are incurred in several stages, the capital costs are converted back to their present worth at year 1 by discounting them at 15 percent interest.

Two additional indicators of economic viability are used. One is the payback period on net capital cost, which represents the number of years required for the cumulative undiscounted payback to net capital cost revenue stream to equal the net capital cost. Finally, the payback to net capital cost streams are discounted to present worth at 10, 15 and 20 percent and these totals are divided by the present worth of the net capital cost to yield ratios of discounted profitability at those interest rates.

3.2 Results

3.2.1 Computer Cash Flow Analyses

Cash flow analyses were performed for each of the 34 alternatives listed in Table XVI - 1. Rather than present all the computer output sheets, representative outputs are shown for each of the 5 production cases, assuming 30 and 100 percent equities and a selling price of $80 per ton f.o.b. Vancouver.

MOUNT BRUSSILOF MAGNESITE PROJECT ECONOMIC PERFORMANCE ANALYSIS

ALTERNATIVE 3 75 TONS PER YEAR AT $ 80 PER TON

NOTE: ALL DOLLAR AND TON FIGURES IN THOUSANDS

TOTAL CAPITAL COST	= $	13609.
GOVERNMENT GRANT	= $	1980.
NET CAPITAL COST	= $	11629.
EQUITY PERCENTAGE	=	30.
EQUITY	= $	3489.
DEBT CAPITAL	= $	8140.

TOTAL PAYBACK ON NET CAPITAL COST DISCOUNTED TO CURRENT VALUE AT 10 PERCENT = $ 14999.
TOTAL PAYBACK ON NET CAPITAL COST DISCOUNTED TO CURRENT VALUE AT 15 PERCENT = $ 11603.
TOTAL PAYBACK ON NET CAPITAL COST DISCOUNTED TO CURRENT VALUE AT 20 PERCENT = $ 9418.

YEAR	CUMULATIVE NET CAPITAL COST	EQUITY INPUT	DEBT CAPITAL INPUT
1	$ 11629.	$ 3489.	$ 8140.

MOUNT BRUSSILOF MAGNESITE PROJECT ECONOMIC PERFORMANCE ANALYSIS

ALTERNATIVE 3 75 TONS PER YEAR AT $ 80 PER TON

NOTE: ALL DOLLAR AND TON FIGURES IN THOUSANDS

YEAR	SALES TONS	SALES REV. $	OP. COSTS $	OP. INCOME $	INTEREST $	DEPR. $	DEPL. $	TAX. INC. $	INC. TAX $	NET EARNS. $	LOAN REPT. $	CASH FLOW $	PAYBACK /NCC $
1	75.	6000.	2938.	3062.	814.	2248.	0.	0.	0.	-0.	814.	1434.	2248.
2	75.	6000.	2938.	3062.	733.	2329.	0.	0.	0.	-0.	814.	1515.	2329.
3	75.	6000.	2938.	3062.	651.	2411.	0.	0.	0.	-0.	814.	1597.	2411.
4	75.	6000.	2938.	3062.	570.	2492.	0.	0.	0.	-0.	814.	1678.	2492.
5	75.	6000.	2938.	3062.	488.	2148.	170.	255.	146.	110.	814.	1614.	2428.
6	75.	6000.	2938.	3062.	407.	0.	20.	2635.	1502.	1133.	814.	339.	1153.
7	75.	6000.	2938.	3062.	326.	0.	20.	2716.	1548.	1168.	814.	374.	1188.
8	75.	6000.	2938.	3062.	244.	0.	20.	2798.	1595.	1203.	814.	409.	1223.
9	75.	6000.	2938.	3062.	163.	0.	20.	2879.	1641.	1238.	814.	444.	1258.
10	75.	6000.	2938.	3062.	81.	0.	20.	2961.	1688.	1273.	814.	479.	1293.
11	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.
12	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.
13	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.
14	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.
15	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.
16	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.
17	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.
18	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.
19	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.
20	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.

PAYBACK PERIOD ON NET CAPITAL COST = 4.9 YEARS TOTAL UNDISCOUNTED PAYBACK TO NET CAPITAL COST = $ 31304.
AVE. ANNUAL RATE OF RETURN ON EQUITY = 38.5 % AVE. ANNUAL RATE OF RETURN ON NET CAPITAL COST = 15.0 %

MOUNT BRUSSILOF MAGNESITE PROJECT ECONOMIC PERFORMANCE ANALYSIS

ALTERNATIVE 8 75 TONS PER YEAR AT $ 80 PER TON

NOTE: ALL DOLLAR AND TON FIGURES IN THOUSANDS

TOTAL CAPITAL COST	=	$	13609.
GOVERNMENT GRANT	=	$	1980.
NET CAPITAL COST	=	$	11629.
EQUITY PERCENTAGE	=		100.
EQUITY	=	$	11629.
DEBT CAPITAL	=	$	0.

TOTAL PAYBACK ON NET CAPITAL COST DISCOUNTED TO CURRENT VALUE AT 10 PERCENT = $ 16620.
TOTAL PAYBACK ON NET CAPITAL COST DISCOUNTED TO CURRENT VALUE AT 15 PERCENT = $ 13110.
TOTAL PAYBACK ON NET CAPITAL COST DISCOUNTED TO CURRENT VALUE AT 20 PERCENT = $ 10827.

YEAR	CUMULATIVE NET CAPITAL COST	EQUITY INPUT	DEBT CAPITAL INPUT
1	$ 11629.	$ 11629.	$ 0.

MOUNT BRUSSILOF MAGNESITE PROJECT ECONOMIC PERFORMANCE ANALYSIS

ALTERNATIVE 8   75 TONS PER YEAR AT $ 80 PER TON

NOTE: ALL DOLLAR AND TON FIGURES IN THOUSANDS

YEAR	SALES TONS	SALES REV. $	OP. COSTS $	OP. INCOME $	INTEREST $	DEPR. $	DEPL. $	TAX. INC. $	INC. TAX $	NET EARNS. $	LOAN REPT. $	CASH FLOW $	PAYBACK /NCC $
1	75.	6000.	2938.	3062.	0.	3062.	0.	-0.	-0.	0.	0.	3062.	3062.
2	75.	6000.	2938.	3062.	0.	3062.	0.	-0.	-0.	0.	0.	3062.	3062.
3	75.	6000.	2938.	3062.	0.	3062.	0.	-0.	-0.	0.	0.	3062.	3062.
4	75.	6000.	2938.	3062.	0.	2443.	150.	469.	267.	202.	0.	2795.	2795.
5	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.
6	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.
7	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.
8	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.
9	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.
10	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.
11	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.
12	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.
13	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.
14	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.
15	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.
16	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.
17	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.
18	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.
19	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.
20	75.	6000.	2938.	3062.	0.	0.	20.	3042.	1734.	1308.	0.	1328.	1328.

PAYBACK PERIOD ON NET CAPITAL COST = 3.9 YEARS   TOTAL UNDISCOUNTED PAYBACK TO NET CAPITAL COST = $ 33230.

AVE. ANNUAL RATE OF RETURN ON EQUITY = 18.1 %   AVE. ANNUAL RATE OF RETURN ON NET CAPITAL COST = 18.1 %

MOUNT BRUSSILOF MAGNESITE PROJECT ECONOMIC PERFORMANCE ANALYSIS

ALTERNATIVE 12 150 TONS PER YEAR AT $ 80 PER TON

NOTE: ALL DOLLAR AND TON FIGURES IN THOUSANDS

TOTAL CAPITAL COST = $ 20609.

GOVERNMENT GRANT = $ 2760.

NET CAPITAL COST = $ 17849.

EQUITY PERCENTAGE = 30.

EQUITY = $ 5355.

DEBT CAPITAL = $ 12494.

TOTAL PAYBACK ON NET CAPITAL COST DISCOUNTED TO CURRENT VALUE AT 10 PERCENT = $ 32142.
TOTAL PAYBACK ON NET CAPITAL COST DISCOUNTED TO CURRENT VALUE AT 15 PERCENT = $ 24897.
TOTAL PAYBACK ON NET CAPITAL COST DISCOUNTED TO CURRENT VALUE AT 20 PERCENT = $ 20302.

YEAR	CUMULATIVE NET CAPITAL COST	EQUITY INPUT	DEBT CAPITAL INPUT
1	$ 17849.	$ 5355.	$ 12494.

MOUNT BRUSSILOF MAGNESITE PROJECT ECONOMIC PERFORMANCE ANALYSIS

ALTERNATIVE 12   150 TONS PER YEAR AT $ 80 PER TON

NOTE: ALL DOLLAR AND TON FIGURES IN THOUSANDS

YEAR	SALES TONS	SALES REV. $	OP. COSTS $	OP. INCOME $	INTEREST $	DEPR. $	DEPL. $	TAX. INC. $	INC. TAX $	NET EARNS. $	LOAN REPT. $	CASH FLOW $	PAYBACK /NCC $
1	150.	12000.	4986.	7014.	1249.	5765.	0.	0.	0.	-0.	1249.	4515.	5765.
2	150.	12000.	4986.	7014.	1124.	5890.	0.	0.	0.	-0.	1249.	4640.	5890.
3	150.	12000.	4986.	7014.	1000.	6014.	0.	0.	0.	-0.	1249.	4765.	6014.
4	150.	12000.	4986.	7014.	875.	180.	158.	5801.	3307.	2495.	1249.	1583.	2833.
5	150.	12000.	4986.	7014.	750.	0.	22.	6242.	3558.	2684.	1249.	1457.	2706.
6	150.	12000.	4986.	7014.	625.	0.	22.	6367.	3629.	2738.	1249.	1511.	2760.
7	150.	12000.	4986.	7014.	500.	0.	22.	6492.	3701.	2792.	1249.	1564.	2814.
8	150.	12000.	4986.	7014.	375.	0.	22.	6617.	3772.	2845.	1249.	1618.	2867.
9	150.	12000.	4986.	7014.	250.	0.	22.	6742.	3843.	2899.	1249.	1672.	2921.
10	150.	12000.	4986.	7014.	125.	0.	22.	6867.	3914.	2953.	1249.	1725.	2975.
11	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.
12	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.
13	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.
14	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.
15	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.
16	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.
17	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.
18	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.
19	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.
20	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.

PAYBACK PERIOD ON NET CAPITAL COST = 3.1 YEARS   TOTAL UNDISCOUNTED PAYBACK TO NET CAPITAL COST = $ 67830.
AVE. ANNUAL RATE OF RETURN ON EQUITY = 75.4 %   AVE. ANNUAL RATE OF RETURN ON NET CAPITAL COST = 23.8 %

MOUNT BRUSSILOF MAGNESITE PROJECT ECONOMIC PERFORMANCE ANALYSIS

ALTERNATIVE 16 150 TONS PER YEAR AT $ 80 PER TON

NOTE: ALL DOLLAR AND TON FIGURES IN THOUSANDS

TOTAL CAPITAL COST = $ 20609.

GOVERNMENT GRANT = $ 2760.

NET CAPITAL COST = $ 17849.

EQUITY PERCENTAGE = 100.

EQUITY = $ 17849.

DEBT CAPITAL = $ 0.

TOTAL PAYBACK ON NET CAPITAL COST DISCOUNTED TO CURRENT VALUE AT 10 PERCENT = $ 34386.
TOTAL PAYBACK ON NET CAPITAL COST DISCOUNTED TO CURRENT VALUE AT 15 PERCENT = $ 26911.
TOTAL PAYBACK ON NET CAPITAL COST DISCOUNTED TO CURRENT VALUE AT 20 PERCENT = $ 22135.

YEAR	CUMULATIVE NET CAPITAL COST	EQUITY INPUT	DEBT CAPITAL INPUT
1	$ 17849.	$ 17849.	$ 0.

MOUNT BRUSSILOF MAGNESITE PROJECT ECONOMIC PERFORMANCE ANALYSIS

ALTERNATIVE 16 150 TONS PER YEAR AT $ 80 PER TON

NOTE: ALL DOLLAR AND TON FIGURES IN THOUSANDS

YEAR	SALES TONS	SALES REV. $	OP. COSTS $	OP. INCOME $	INTEREST $	DEPR. $	DEPL. $	TAX. INC. $	INC. TAX $	NET EARNS. $	LOAN REPT. $	CASH FLOW $	PAYBACK /NCC $
1	150.	12000.	4986.	7014.	0.	7014.	0.	-0.	-0.	0.	0.	7014.	7014.
2	150.	12000.	4986.	7014.	0.	7014.	0.	-0.	-0.	0.	0.	7014.	7014.
3	150.	12000.	4986.	7014.	0.	3821.	136.	3057.	1742.	1315.	0.	5272.	5272.
4	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.
5	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.
6	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.
7	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.
8	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.
9	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.
10	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.
11	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.
12	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.
13	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.
14	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.
15	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.
16	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.
17	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.
18	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.
19	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.
20	150.	12000.	4986.	7014.	0.	0.	22.	6992.	3985.	3007.	0.	3029.	3029.

PAYBACK PERIOD ON NET CAPITAL COST = 2.7 YEARS   TOTAL UNDISCOUNTED PAYBACK TO NET CAPITAL COST = $ 70785.
AVE. ANNUAL RATE OF RETURN ON EQUITY = 27.0 %   AVE. ANNUAL RATE OF RETURN ON NET CAPITAL COST = 27.0 %

MOUNT BRUSSILOF MAGNESITE PROJECT ECONOMIC PERFORMANCE ANALYSIS

ALTERNATIVE 20 75-150-225 TONS PER YEAR AT $ 80 PER TON

NOTE: ALL DOLLAR AND TON FIGURES IN THOUSANDS

TOTAL CAPITAL COST	= $	27811.
GOVERNMENT GRANT	= $	3420.
NET CAPITAL COST	= $	24391.
EQUITY PERCENTAGE	=	30.
EQUITY	= $	7317.
DEBT CAPITAL	= $	17074.

TOTAL PAYBACK ON NET CAPITAL COST DISCOUNTED TO CURRENT VALUE AT 10 PERCENT = $ 35869.
TOTAL PAYBACK ON NET CAPITAL COST DISCOUNTED TO CURRENT VALUE AT 15 PERCENT = $ 25455.
TOTAL PAYBACK ON NET CAPITAL COST DISCOUNTED TO CURRENT VALUE AT 20 PERCENT = $ 19098.

YEAR	CUMULATIVE NET CAPITAL COST	EQUITY INPUT	DEBT CAPITAL INPUT
1	$ 11629.	$ 3489.	$ 8140.
4	$ 17849.	$ 1866.	$ 4354.
7	$ 24391.	$ 1963.	$ 4580.

MOUNT BRUSSILOF MAGNESITE PROJECT ECONOMIC PERFORMANCE ANALYSIS

ALTERNATIVE 20   75-150-225 TONS PER YEAR AT $ 80 PER TON

NOTE: ALL DOLLAR AND TON FIGURES IN THOUSANDS

YEAR	SALES TONS	SALES REV. $	OP. COSTS $	OP. INCOME $	INTEREST $	DEPR. $	DEPL. $	TAX. INC. $	INC. TAX $	NET EARNS. $	LOAN REPT. $	CASH FLOW $	PAYBACK /NCC $
1	75.	6000.	2938.	3062.	814.	2248.	0.	0.	0.	-0.	814.	1434.	2248.
2	75.	6000.	2938.	3062.	733.	2329.	0.	0.	0.	-0.	814.	1515.	2329.
3	75.	6000.	2938.	3062.	651.	2411.	0.	0.	0.	-0.	814.	1597.	2411.
4	150.	12000.	4986.	7014.	1005.	6009.	0.	0.	0.	-0.	1249.	4759.	6009.
5	150.	12000.	4986.	7014.	880.	4852.	174.	1108.	632.	477.	1249.	4253.	5502.
6	150.	12000.	4986.	7014.	755.	6259.	0.	0.	0.	-0.	1249.	5009.	6259.
7	225.	18000.	6989.	11011.	1088.	284.	54.	9584.	5463.	4121.	1707.	2752.	4459.
8	225.	18000.	6989.	11011.	918.	-0.	32.	10061.	5735.	4326.	1707.	2651.	4358.
9	225.	18000.	6989.	11011.	747.	0.	32.	10232.	5832.	4400.	1707.	2724.	4432.
10	225.	18000.	6989.	11011.	576.	-0.	32.	10403.	5930.	4473.	1707.	2798.	4505.
11	225.	18000.	6989.	11011.	405.	0.	32.	10574.	6027.	4547.	893.	3685.	4579.
12	225.	18000.	6989.	11011.	316.	0.	32.	10663.	6078.	4585.	893.	3724.	4617.
13	225.	18000.	6989.	11011.	227.	-0.	32.	10752.	6129.	4623.	893.	3762.	4655.
14	225.	18000.	6989.	11011.	137.	0.	32.	10842.	6180.	4662.	458.	4236.	4694.
15	225.	18000.	6989.	11011.	92.	0.	32.	10887.	6206.	4682.	458.	4256.	4714.
16	225.	18000.	6989.	11011.	46.	-0.	32.	10933.	6232.	4701.	458.	4275.	4733.
17	225.	18000.	6989.	11011.	0.	0.	32.	10979.	6258.	4721.	0.	4753.	4753.
18	225.	18000.	6989.	11011.	0.	0.	32.	10979.	6258.	4721.	0.	4753.	4753.
19	225.	18000.	6989.	11011.	0.	0.	32.	10979.	6258.	4721.	0.	4753.	4753.
20	225.	18000.	6989.	11011.	0.	0.	32.	10979.	6258.	4721.	0.	4753.	4753.

PAYBACK PERIOD ON NET CAPITAL COST = 5.9 YEARS   TOTAL UNDISCOUNTED PAYBACK TO NET CAPITAL COST = $ 83516.

AVE. ANNUAL RATE OF RETURN ON EQUITY = 42.9 %   AVE. ANNUAL RATE OF RETURN ON NET CAPITAL COST = 20.6 %

MOUNT BRUSSILOF MAGNESITE PROJECT ECONOMIC PERFORMANCE ANALYSIS

ALTERNATIVE 24 75-150-225 TONS PER YEAR AT $ 80 PER TON

NOTE: ALL DOLLAR AND TON FIGURES IN THOUSANDS

TOTAL CAPITAL COST = $ 27812.

GOVERNMENT GRANT = $ 3420.

NET CAPITAL COST = $ 24392.

EQUITY PERCENTAGE = 100.

EQUITY = $ 24392.

DEBT CAPITAL = $ 0.

TOTAL PAYBACK ON NET CAPITAL COST DISCOUNTED TO CURRENT VALUE AT 10 PERCENT = $ 38502.
TOTAL PAYBACK ON NET CAPITAL COST DISCOUNTED TO CURRENT VALUE AT 15 PERCENT = $ 27702.
TOTAL PAYBACK ON NET CAPITAL COST DISCOUNTED TO CURRENT VALUE AT 20 PERCENT = $ 21064.

YEAR	CUMULATIVE NET CAPITAL COST	EQUITY INPUT	DEBT CAPITAL INPUT
1	$ 11629.	$ 11629.	$ 0.
4	$ 17849.	$ 6220.	$ 0.
7	24392.	$ 6543.	$ 0.

MOUNT BRUSSILOF MAGNESITE PROJECT ECONOMIC PERFORMANCE ANALYSIS

ALTERNATIVE 24 75-150-225 TONS PER YEAR AT $ 80 PER TON

NOTE: ALL DOLLAR AND TON FIGURES IN THOUSANDS

YEAR	SALES TONS	SALES REV. $	OP. COSTS $	OP. INCOME $	INTEREST $	DEPR. $	DEPL. $	TAX. INC. $	INC. TAX $	NET EARNS. $	LOAN REPT. $	CASH FLOW $	PAYBACK /NCC $
1	75.	6000.	2938.	3062.	0.	3062.	0.	-0.	-0.	0.	0.	3062.	3062.
2	75.	6000.	2938.	3062.	0.	3062.	0.	-0.	-0.	0.	0.	3062.	3062.
3	75.	6000.	2938.	3062.	0.	3062.	0.	-0.	-0.	0.	0.	3062.	3062.
4	150.	12000.	4986.	7014.	0.	7014.	0.	-0.	-0.	0.	0.	7014.	7014.
5	150.	12000.	4986.	7014.	0.	1649.	174.	5191.	2959.	2232.	0.	4055.	4055.
6	150.	12000.	4986.	7014.	0.	6543.	22.	449.	256.	193.	0.	6758.	6758.
7	225.	18000.	6989.	11011.	0.	0.	32.	10979.	6258.	4721.	0.	4753.	4753.
8	225.	18000.	6989.	11011.	0.	0.	32.	10979.	6258.	4721.	0.	4753.	4753.
9	225.	18000.	6989.	11011.	0.	0.	32.	10979.	6258.	4721.	0.	4753.	4753.
10	225.	18000.	6989.	11011.	0.	0.	32.	10979.	6258.	4721.	0.	4753.	4753.
11	225.	18000.	6989.	11011.	0.	0.	32.	10979.	6258.	4721.	0.	4753.	4753.
12	225.	18000.	6989.	11011.	0.	0.	32.	10979.	6258.	4721.	0.	4753.	4753.
13	225.	18000.	6989.	11011.	0.	0.	32.	10979.	6258.	4721.	0.	4753.	4753.
14	225.	18000.	6989.	11011.	0.	0.	32.	10979.	6258.	4721.	0.	4753.	4753.
15	225.	18000.	6989.	11011.	0.	0.	32.	10979.	6258.	4721.	0.	4753.	4753.
16	225.	18000.	6989.	11011.	0.	0.	32.	10979.	6258.	4721.	0.	4753.	4753.
17	225.	18000.	6989.	11011.	0.	0.	32.	10979.	6258.	4721.	0.	4753.	4753.
18	225.	18000.	6989.	11011.	0.	0.	32.	10979.	6258.	4721.	0.	4753.	4753.
19	225.	18000.	6989.	11011.	0.	0.	32.	10979.	6258.	4721.	0.	4753.	4753.
20	225.	18000.	6989.	11011.	0.	0.	32.	10979.	6258.	4721.	0.	4753.	4753.

PAYBACK PERIOD ON NET CAPITAL COST = 5.6 YEARS TOTAL UNDISCOUNTED PAYBACK TO NET CAPITAL COST = $ 93555.

AVE. ANNUAL RATE OF RETURN ON EQUITY = 22.7 % AVE. ANNUAL RATE OF RETURN ON NET CAPITAL COST = 22.7 %

MOUNT BRUSSILOF MAGNESITE PROJECT ECONOMIC PERFORMANCE ANALYSIS

ALTERNATIVE 28 150 TONS PER YEAR AT $ 80 PER TON

NOTE: ALL DOLLAR AND TON FIGURES IN THOUSANDS

TOTAL CAPITAL COST = $ 18858.

GOVERNMENT GRANT = $ 2370.

NET CAPITAL COST = $ 16488.

EQUITY PERCENTAGE = 100.

EQUITY = $ 16488.

DEBT CAPITAL = $ 0.

TOTAL PAYBACK ON NET CAPITAL COST DISCOUNTED TO CURRENT VALUE AT 10 PERCENT = $ 34774.
TOTAL PAYBACK ON NET CAPITAL COST DISCOUNTED TO CURRENT VALUE AT 15 PERCENT = $ 27135.
TOTAL PAYBACK ON NET CAPITAL COST DISCOUNTED TO CURRENT VALUE AT 20 PERCENT = $ 22276.

YEAR	CUMULATIVE NET CAPITAL COST	EQUITY INPUT	DEBT CAPITAL INPUT
1	$ 16488.	$ 16488.	$ 0.

MOUNT BRUSSILOF MAGNESITE PROJECT ECONOMIC PERFORMANCE ANALYSIS

ALTERNATIVE 28 150 TONS PER YEAR AT $ 80 PER TON

NOTE: ALL DOLLAR AND TON FIGURES IN THOUSANDS

YEAR	SALES TONS	SALES REV. $	OP. COSTS $	OP. INCOME $	INTEREST $	DEPR. $	DEPL. $	TAX. INC. $	INC. TAX $	NET EARNS. $	LOAN REPT. $	CASH FLOW $	PAYBACK /NCC $
1	150.	12000.	4730.	7270.	0.	7270.	0.	-0.	-0.	0.	0.	7270.	7270.
2	150.	12000.	4730.	7270.	0.	7270.	0.	-0.	-0.	0.	0.	7270.	7270.
3	150.	12000.	4730.	7270.	0.	1948.	136.	5186.	2956.	2230.	0.	4314.	4314.
4	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
5	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
6	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
7	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
8	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
9	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
10	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
11	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
12	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
13	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
14	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
15	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
16	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
17	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
18	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
19	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
20	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.

PAYBACK PERIOD ON NET CAPITAL COST = 2.5 YEARS TOTAL UNDISCOUNTED PAYBACK TO NET CAPITAL COST = $ 72211.
AVE. ANNUAL RATE OF RETURN ON EQUITY = 30.2 % AVE. ANNUAL RATE OF RETURN ON NET CAPITAL COST = 30.2 %

MOUNT BRUSSILOF MAGNESITE PROJECT ECONOMIC PERFORMANCE ANALYSIS

ALTERNATIVE 32 100-150 TONS PER YEAR AT $ 80 PER TON

NOTE: ALL DOLLAR AND TON FIGURES IN THOUSANDS

TOTAL CAPITAL COST	=	$	19248.
GOVERNMENT GRANT	=	$	2760.
NET CAPITAL COST	=	$	16488.
EQUITY PERCENTAGE	=		100.
EQUITY	=	$	16488.
DEBT CAPITAL	=	$	0.

TOTAL PAYBACK ON NET CAPITAL COST DISCOUNTED TO CURRENT VALUE AT 10 PERCENT = $ 31320.
TOTAL PAYBACK ON NET CAPITAL COST DISCOUNTED TO CURRENT VALUE AT 15 PERCENT = $ 23770.
TOTAL PAYBACK ON NET CAPITAL COST DISCOUNTED TO CURRENT VALUE AT 20 PERCENT = $ 19008.

YEAR	CUMULATIVE NET CAPITAL COST	EQUITY INPUT	DEBT CAPITAL INPUT
1	$ 16488.	$ 16488.	$ 0.

MOUNT BRUSSILOF MAGNESITE PROJECT ECONOMIC PERFORMANCE ANALYSIS

ALTERNATIVE 32 100-150 TONS PER YEAR AT $ 80 PER TON

NOTE: ALL DOLLAR AND TON FIGURES IN THOUSANDS

YEAR	SALES TONS	SALES REV. $	OP. COSTS $	OP. INCOME $	INTEREST $	DEPR. $	DEPL. $	TAX. INC. $	INC. TAX $	NET EARNS. $	LOAN REPT. $	CASH FLOW $	PAYBACK /NCC $
1	100.	8000.	3502.	4498.	0.	4498.	0.	-0.	-0.	0.	0.	4498.	4498.
2	100.	8000.	3502.	4498.	0.	4498.	0.	-0.	-0.	0.	0.	4498.	4498.
3	100.	8000.	3502.	4498.	0.	4498.	0.	-0.	-0.	0.	0.	4498.	4498.
4	150.	12000.	4730.	7270.	0.	2994.	158.	4118.	2347.	1771.	0.	4923.	4923.
5	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
6	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
7	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
8	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
9	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
10	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
11	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
12	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
13	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
14	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
15	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
16	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
17	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
18	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
19	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.
20	150.	12000.	4730.	7270.	0.	0.	22.	7248.	4131.	3117.	0.	3139.	3139.

PAYBACK PERIOD ON NET CAPITAL COST = 3.6 YEARS TOTAL UNDISCOUNTED PAYBACK TO NET CAPITAL COST = $ 68635.
AVE. ANNUAL RATE OF RETURN ON EQUITY = 23.8 % AVE. ANNUAL RATE OF RETURN ON NET CAPITAL COST = 23.8 %

3.2.2 Tabular Summary

The complete results of the computer analyses are summarized in tabular form in Table XVI - 2.

3.2.3 Graphical Summary

To further illustrate the effect of changes in the selling price on the viability of the various cases, Figures XVI - 1 to 4 are attached. Using these figures, it is possible to enter the curves at a particular selling price and determine immediately the economic performance for any of the cases investigated. In addition, comparisons may be made more readily between the relative performance of cases.

4. DISCUSSION OF RESULTS

From an examination of the computer printout sheets, summarized in Table XVI - 2, and of Figures XVI - 1 to 4, the following points arise:

(a) As would be expected, higher selling prices mean shorter payback periods and higher rates of return.

(b) For any particular alternative, the rate of return on net capital cost, which is a reliable indicator of project profitability regardless of the debt-equity financing arrangements, is always higher at 100 percent equity than at 30 percent. This is clearly because the total equity arrangement requires no interest payments.

(c) When the equity is 100 percent, the rate of return to equity equals the rate of return to net capital cost. When debt capital is introduced, the rate of return to equity is increased,because of the leverage effect of the equity when the rate of return on borrowed money exceeds the interest rate paid on it.

(d) At a product selling price of $80 per ton, the payback period to net capital cost at 100 percent equity ranges between 2.5 and 5.9 years while the rate of return to net capital cost ranges from 18.1 to 30.2 percent. The corresponding range for the rate of return to equity at 30 percent equity is 38.5 to 75.4 percent.

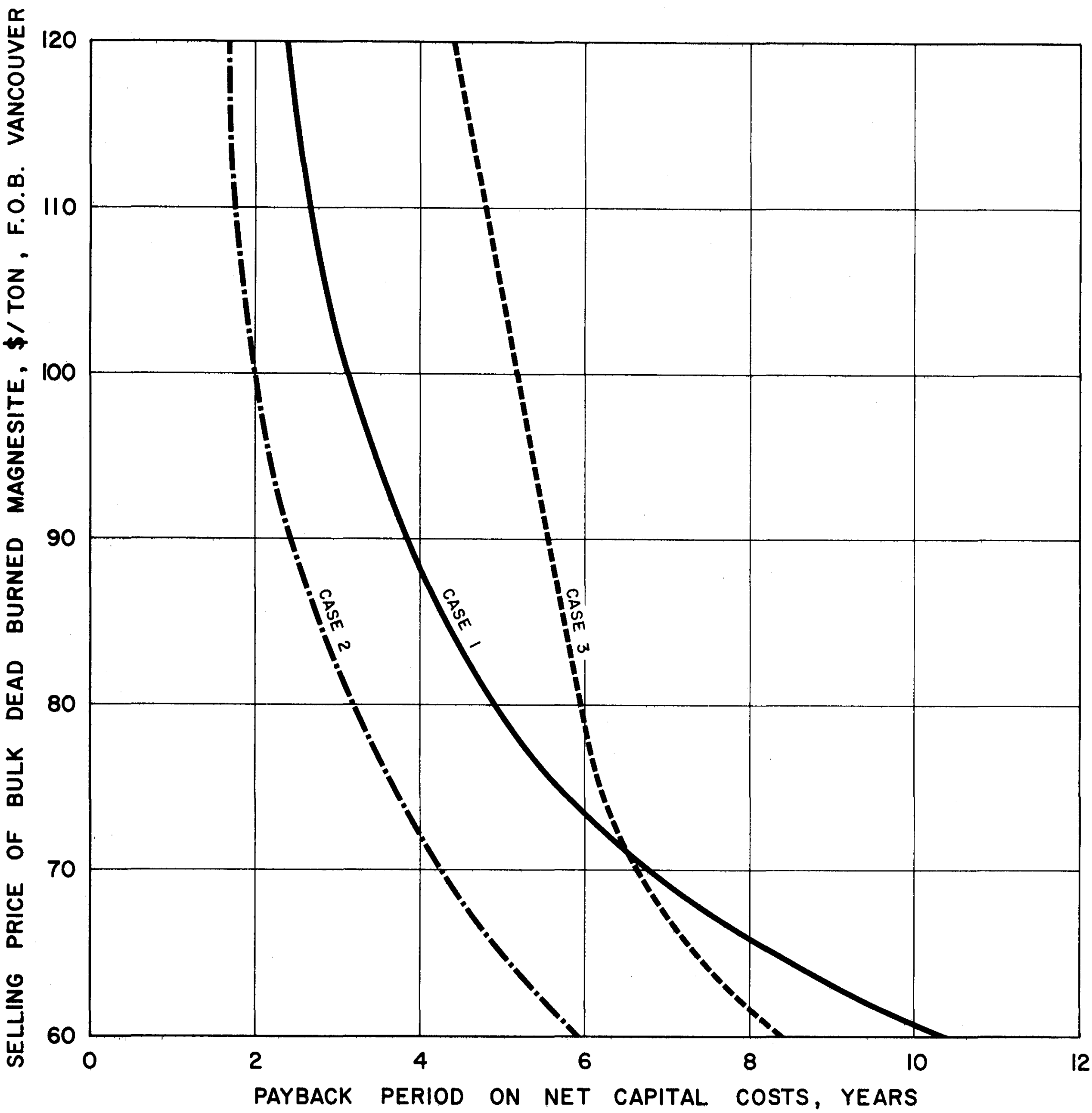
TABLE XVI - 2

SUMMARY OF RESULT OF ECONOMIC ANALYSES

Alter-native No.	Production Case	Production Capacity Tons/Year (000's)	No. of Kilns	Selling Price $/Ton fob Van.	Net Capital Cost $(000's)	Equity Assumed %	Payback Period on N.C.C.* Years	Ave. Annual Rate of Return on Equity %	Ave. Annual Rate of Return on N.C.C. %	Total Payback on N.C.C. Discounted to Present Worth, at			Ratio of: Total Discounted Payback/ Net Capital Cost		
										10%	15%	20%	10%	15%	20%
1	1	75	1	60	11,629	30	10.4	9.3	5.3	8,229	6,076	4,705	0.708	0.523	0.405
2	1	75	1	70	11,629	30	6.8	22.2	10.3	11,801	9,033	7,243	1.015	0.777	0.623
3	1	75	1	80	11,629	30	4.9	38.5	15.0	14,999	11,603	9,418	1.290	0.998	0.810
4	1	75	1	100	11,629	30	3.1	75.3	23.8	20,946	16,224	13,228	1.801	1.395	1.138
5	1	75	1	120	11,629	30	2.4	111.5	32.2	26,647	20,540	16,700	2.291	1.766	1.436
6	1	75	1	60	11,629	100	7.6	8.0	8.0	10,335	8,101	6,614	0.889	0.697	0.569
7	1	75	1	70	11,629	100	5.1	13.2	13.2	13,590	10,732	8,849	1.169	0.923	0.761
8	1	75	1	80	11,629	100	3.9	18.1	18.1	16,620	13,110	10,827	1.430	1.127	0.931
9	1	75	1	100	11,629	100	2.7	27.0	27.0	22,411	17,539	14,426	1.927	1.508	1.241
10	1	75	1	120	11,629	100	2.0	35.7	35.7	28,085	21,820	17,858	2.415	1.876	1.536
11	2	150	2	60	17,849	30	5.9	28.3	12.2	20,096	15,478	12,495	1.126	0.867	0.700
12	2	150	2	80	17,849	30	3.1	75.4	23.8	32,142	24,897	20,302	1.801	1.395	1.137
13	2	150	2	100	17,849	30	2.0	123.2	34.7	43,535	33,518	27,234	2.439	1.878	1.526
14	2	150	2	120	17,849	30	1.7	165.4	44.9	54,672	41,804	33,774	3.063	2.342	1.892
15	2	150	2	60	17,849	100	4.6	15.2	15.2	22,705	17,931	14,803	1.272	1.005	0.829
16	2	150	2	80	17,849	100	2.7	27.0	27.0	34,386	26,911	22,135	1.927	1.508	1.240
17	2	150	2	100	17,849	100	1.9	38.0	38.0	45,678	35,400	28,914	2.559	1.983	1.620
18	2	150	2	120	17,849	100	1.6	48.5	48.5	56,802	43,669	35,433	3.182	2.447	1.985
19	3	75,150,225	1,2,3	60	24,392	30	8.4	21.6	12.5	22,268	15,460	11,286	0.913	0.634	0.463
20	3	75,150,225	1,2,3	80	24,392	30	5.9	42.9	20.6	35,869	25,455	19,098	1.471	1.044	0.783
21	3	75,150,225	1,2,3	100	24,392	30	5.2	62.7	27.2	48,490	34,361	25,820	1.988	1.409	1.059
22	3	75,150,225	1,2,3	120	24,392	30	4.4	83.3	33.2	60,925	43,045	32,303	2.498	1.765	1.324
23	3	75,150,225	1,2,3	60	24,392	100	7.3	14.8	14.8	25,365	18,212	13,748	1.040	0.747	0.564
24	3	75,150,225	1,2,3	80	24,392	100	5.6	22.7	22.7	38,502	27,702	21,064	1.579	1.136	0.864
25	3	75,150,225	1,2,3	100	24,392	100	4.9	29.2	29.2	51,001	36,463	27,630	2.091	1.495	1.133
26	3	75,150,225	1,2,3	120	24,392	100	4.1	35.3	35.3	63,404	45,106	34,066	2.599	1.849	1.397
27	4	150	1	60	16,488	100	3.9	17.7	17.7	23,245	18,343	15,151	1.410	1.113	0.919
28	4	150	1	80	16,488	100	2.5	30.2	30.2	34,774	27,135	22,276	2.109	1.646	1.351
29	4	150	1	100	16,488	100	-	-	-	-	-	-	-	-	-
30	4	150	1	120	16,488	100	1.4	53.2	53.2	57,110	43,788	35,450	3.464	2.656	2.150
31	5	100,150	1	60	16,488	100	5.2	14.2	14.2	20,715	15,798	12,632	1.256	0.958	0.766
32	5	100,150	1	80	16,488	100	3.6	23.8	23.8	31,320	23,770	19,008	1.900	1.442	1.153
33	5	100,150	1	100	16,488	100	2.7	32.6	32.6	41,621	31,370	24,978	2.524	1.903	1.515
34	5	100,150	1	120	16,488	100	2.0	41.0	41.0	51,782	38,794	30,750	3.141	2.353	1.865

* N.C.C. = Net Capital Cost
Net Capital Cost = Total Capital Cost less government grant

PAYBACK PERIOD VS SELLING PRICE AT 30 % EQUITY



LEGEND

CASE 1 ▬▬ 75,000 TONS PER YEAR, SINGLE KILN PLANT

CASE 2 ▬·▬ 150,000 TONS PER YEAR, DOUBLE KILN PLANT

CASE 3 ▬ ▬ ▬ 75,000 - 150,000 - 225,000 TONS PER YEAR, STAGED PLANT

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FIGURE XVI-1

PAYBACK PERIOD VS SELLING PRICE AT 100 % EQUITY

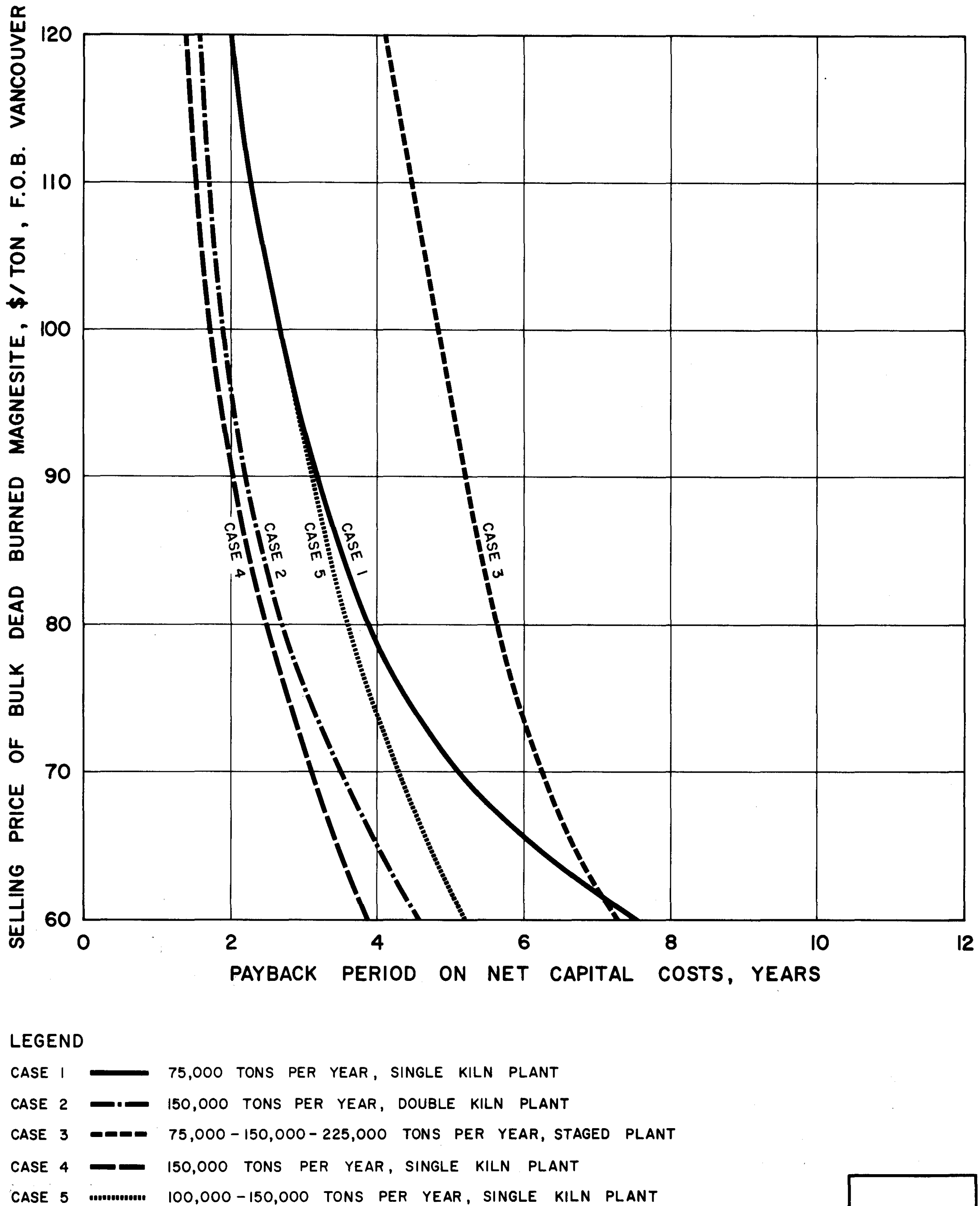

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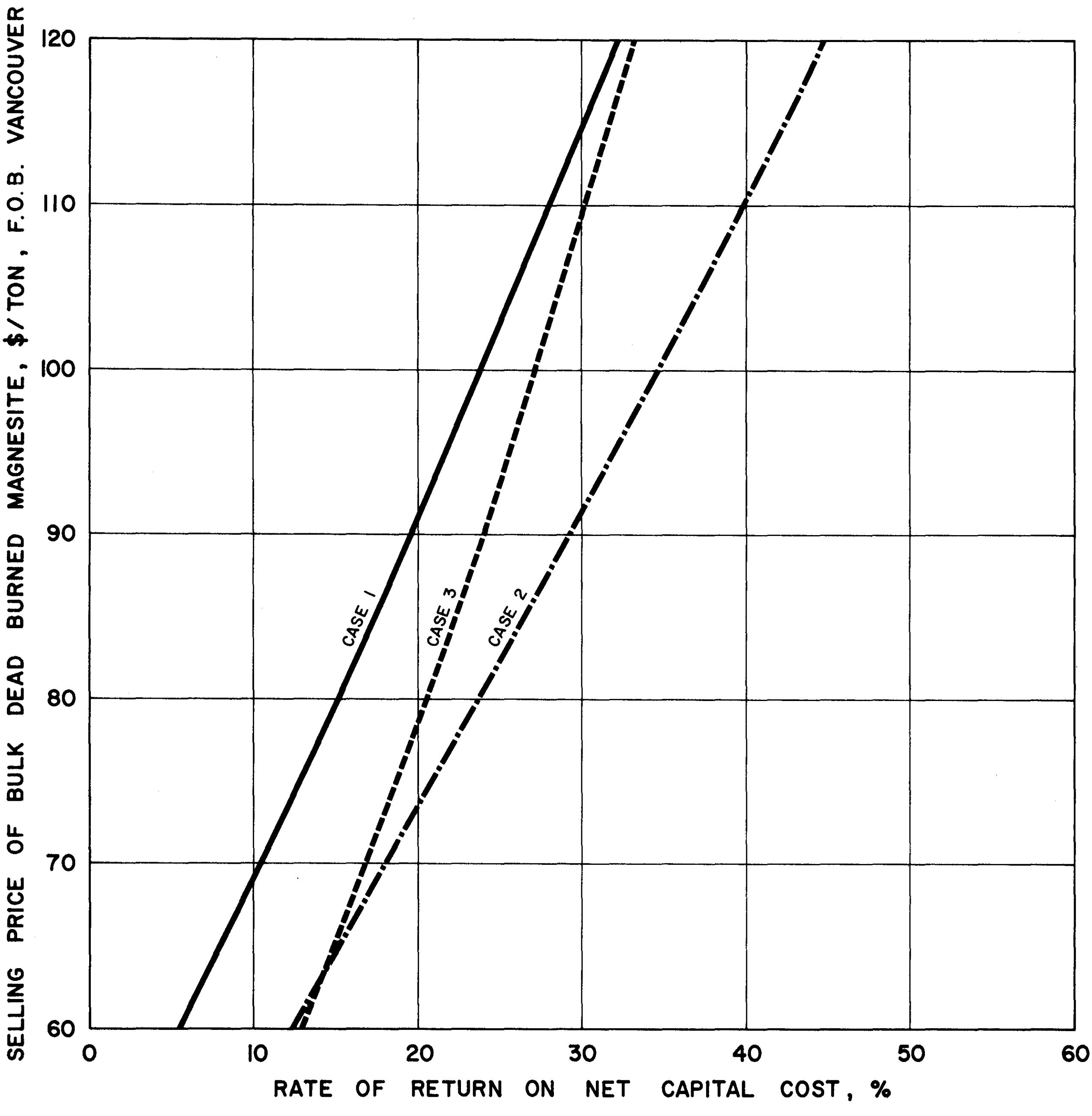
FIGURE XVI-2

RATE OF RETURN VS SELLING PRICE AT 30 % EQUITY



LEGEND

CASE 1 —— 75,000 TONS PER YEAR, SINGLE KILN PLANT

CASE 2 —·— 150,000 TONS PER YEAR, DOUBLE KILN PLANT

CASE 3 - - - 75,000 - 150,000 - 225,000 TONS PER YEAR, STAGED PLANT

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FIGURE XVI-3

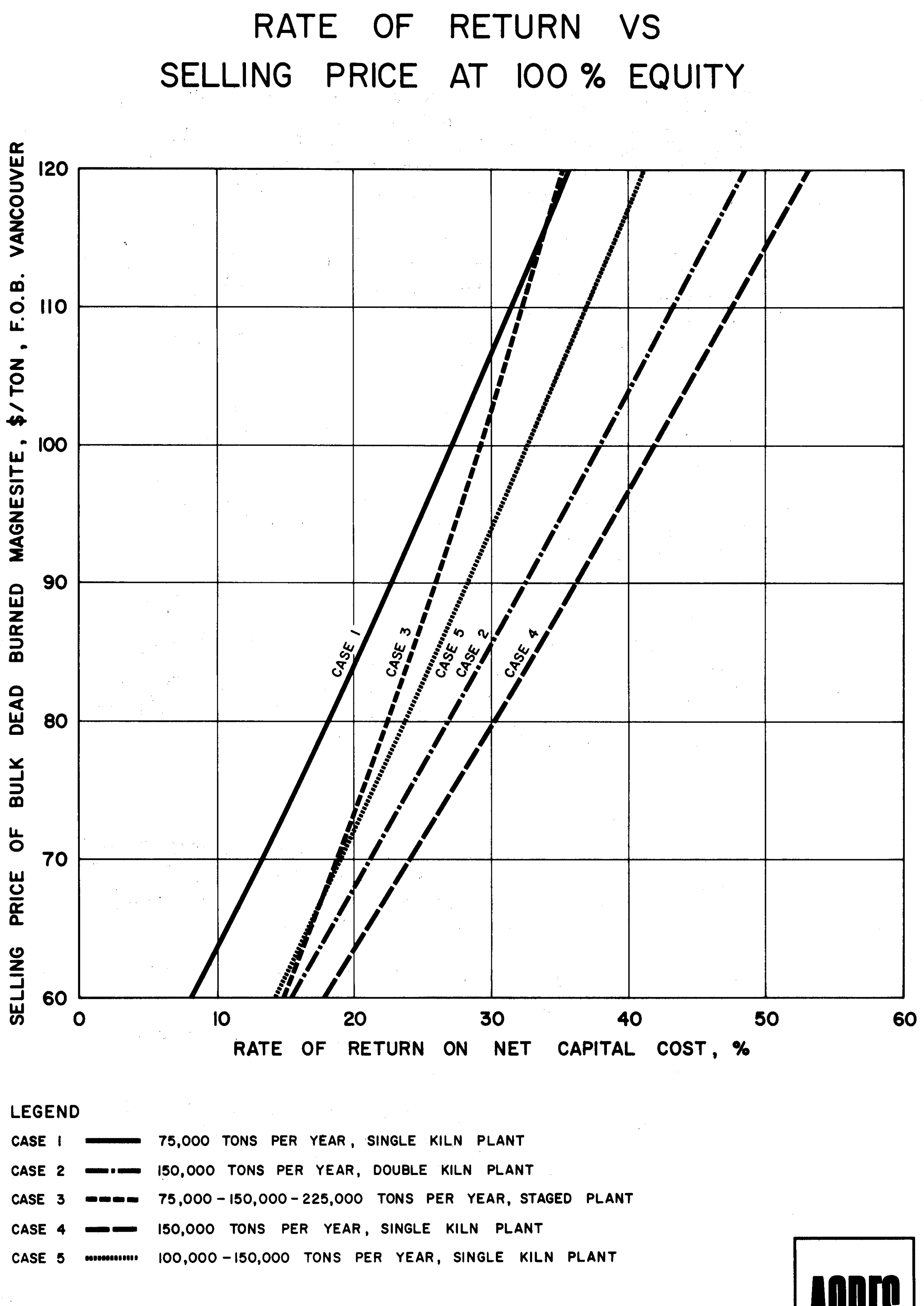

RATE OF RETURN VS
SELLING PRICE AT 100 % EQUITY
SELLING PRICE OF BULK DEAD BURNED MAGNESITE, $/TON, F.O.B. VANCOUVER
60
70
80
90
100
110
120
0
10
20
30
40
50
60
RATE OF RETURN ON NET CAPITAL COST, %
CASE 1
CASE 3
CASE 5
CASE 2
CASE 4
LEGEND
CASE 1 75,000 TONS PER YEAR, SINGLE KILN PLANT
CASE 2 150,000 TONS PER YEAR, DOUBLE KILN PLANT
CASE 3 75,000 - 150,000 - 225,000 TONS PER YEAR, STAGED PLANT
CASE 4 150,000 TONS PER YEAR, SINGLE KILN PLANT
CASE 5 100,000 - 150,000 TONS PER YEAR, SINGLE KILN PLANT
ACRES

FIGURE XVI-4

This means that, at an average selling price of $80 per ton, the choice of initial production level and staging could mean a difference of more than 10 percent on the rate of return to net capital cost and more than 3 years to the payback period.

(e) For Case 3, which represents the fully staged development, the payback period will be longer than for the single kiln Stage I, because of the higher capital costs, but the rates of return for Case 3 are higher than for Case 1 because of the much higher revenues in later years.

(f) From the point of view of relative payback periods, the best production case is Case 4, followed by Cases 2, 5, 1 and 3.

(g) From the point of view of rate of return to net capital cost, the most desirable is Case 4, followed by 2, 5, 3 and 1.

(h) While Case 4 is more economically attractive than Cases 2 and 5, the difference is not great. If factors other than straight economic performance favoured Cases 2 or 5, then consideration must be given to the advantages of accommodating those other factors at the expense of a partial reduction in the theoretical profitability of the project.

5. CONCLUSIONS

(a) The results of the economic feasibility analyses show that, aside from marketing considerations and other limiting factors, the most suitable production cases in decreasing order of preference are:

Case 4 - 150,000 tons per year initial capacity single kiln plant.

Case 2 - 150,000 tons per year initial capacity double kiln plant.

Case 5 - 150,000 tons per year initial capacity single kiln plant, but operated at 100,000 tons for the first 3 years.

(b) The final selection between these production alternatives involves a consideration of marketing limitations and operational flexibility.

Section XII indicated that a market for at least 90,000 tons of product could be found for the first year of production. Sales efforts subsequent to this feasibility study could possibly increase this quantity, in which case the staging plans might require modification. At this stage, however, conclusions must be based on the available information, which indicates that an initial production level of 150,000 tons per year is too high. At the same time, Cases 4 and 5, being single kiln operations, suffer from the disadvantage of being inflexible. Case 2, on the other hand, with two kilns, provides increased operating flexibility in terms of plant shutdowns and maintenance, and the fact that two product types may be produced at the same time.

(c) For the reasons outlined in (b) above, it is considered that the most suitable development is Case 2. This provides increased flexibility for a slight decrease in profitability. If the initial demand is in the order of 100,000 tons, one of the kilns may be operated for part of the year and then shut down without affecting the whole labour force.

(d) At a sales price of $80 per ton, Case 2 will yield a payback period on net capital cost in the order of 2.7 to 3.1 years, depending upon the debt to equity ratio. Similarly, the average annual rate of return to net capital cost will be in the order of 23.8 to 27.0 percent.

(e) It is stressed that the results of the economic feasibility analysis are based upon the assumption that a stringent new taxation policy will be in effect. Accordingly, a direct comparison between the above results and the corresponding performance for existing mines is not reasonable. Nevertheless, the analyses indicate that the project is not only viable, but economically attractive.

In addition, the proven and probable reserves total 25 million tons of high quality ore. For the production capacities proposed, this means a very long project life. This in turn reduces the requirement for an extremely short payback period and high rate of return on capital. The fact that the Mount Brussilof Magnesite Project combines a long project life with a relatively short payback period and high return to capital indicates that this project is a good risk and a desirable investment.

(f) Before senior financing is arranged, consideration should be given to the desirability of producing additional product types, such as bagged dead burned magnesite, caustic calcined magnesite, dead burned magnesite bricks or other magnesium products.

XVII PLATES

1. LOCATION MAP

2. REGIONAL LOCATION MAP

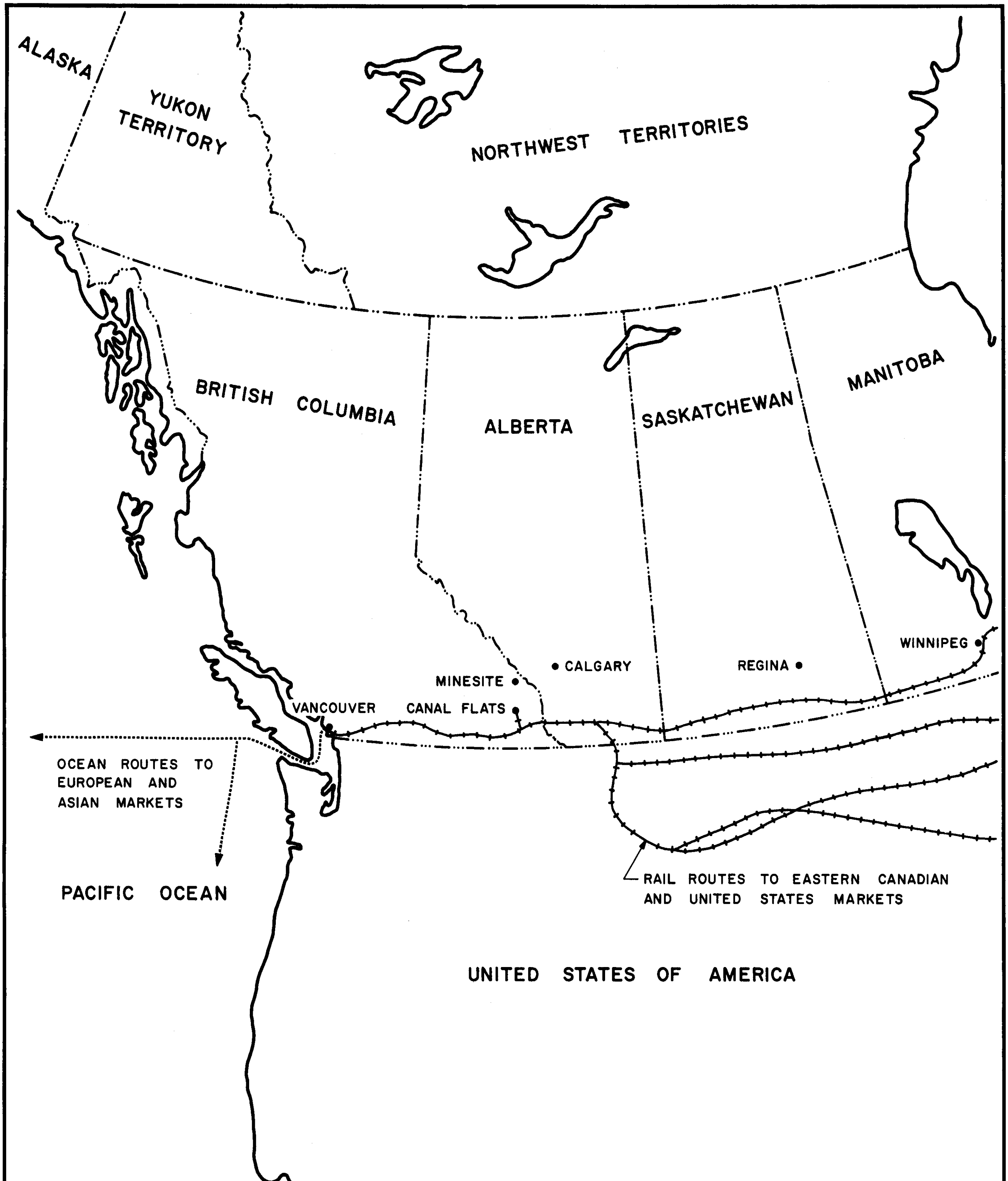
3. ORE BODY PLAN AND SECTIONS

4. MINE SURFACE FACILITIES GENERAL ARRANGEMENTS

5. FLOWSHEET

6. PROCESS PLANT LAYOUT

7. PROCESS PLANT CROSS SECTIONS



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BRUSSILOF MAGNESITE PROJECT

LOCATION MAP

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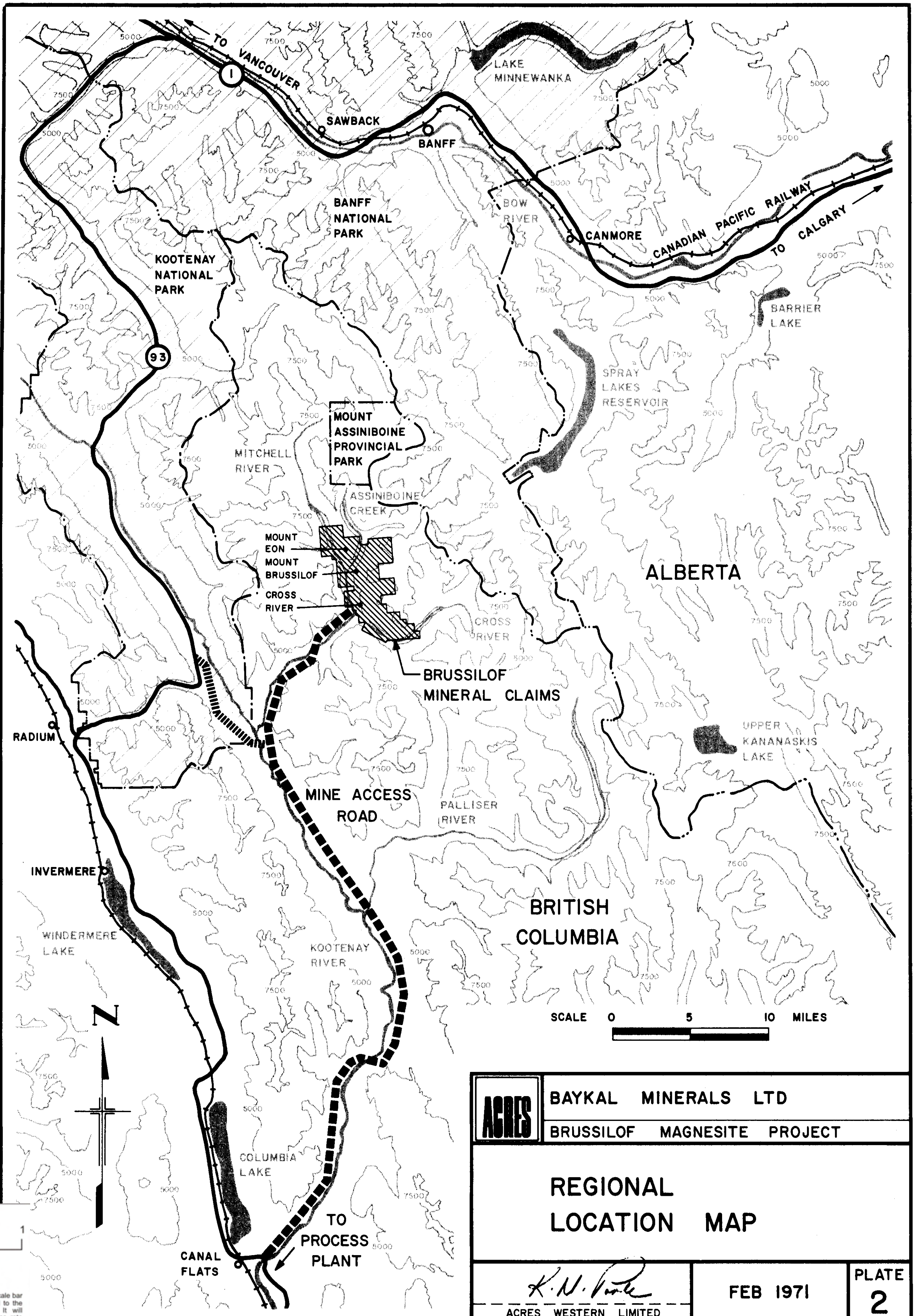
PLATE 1



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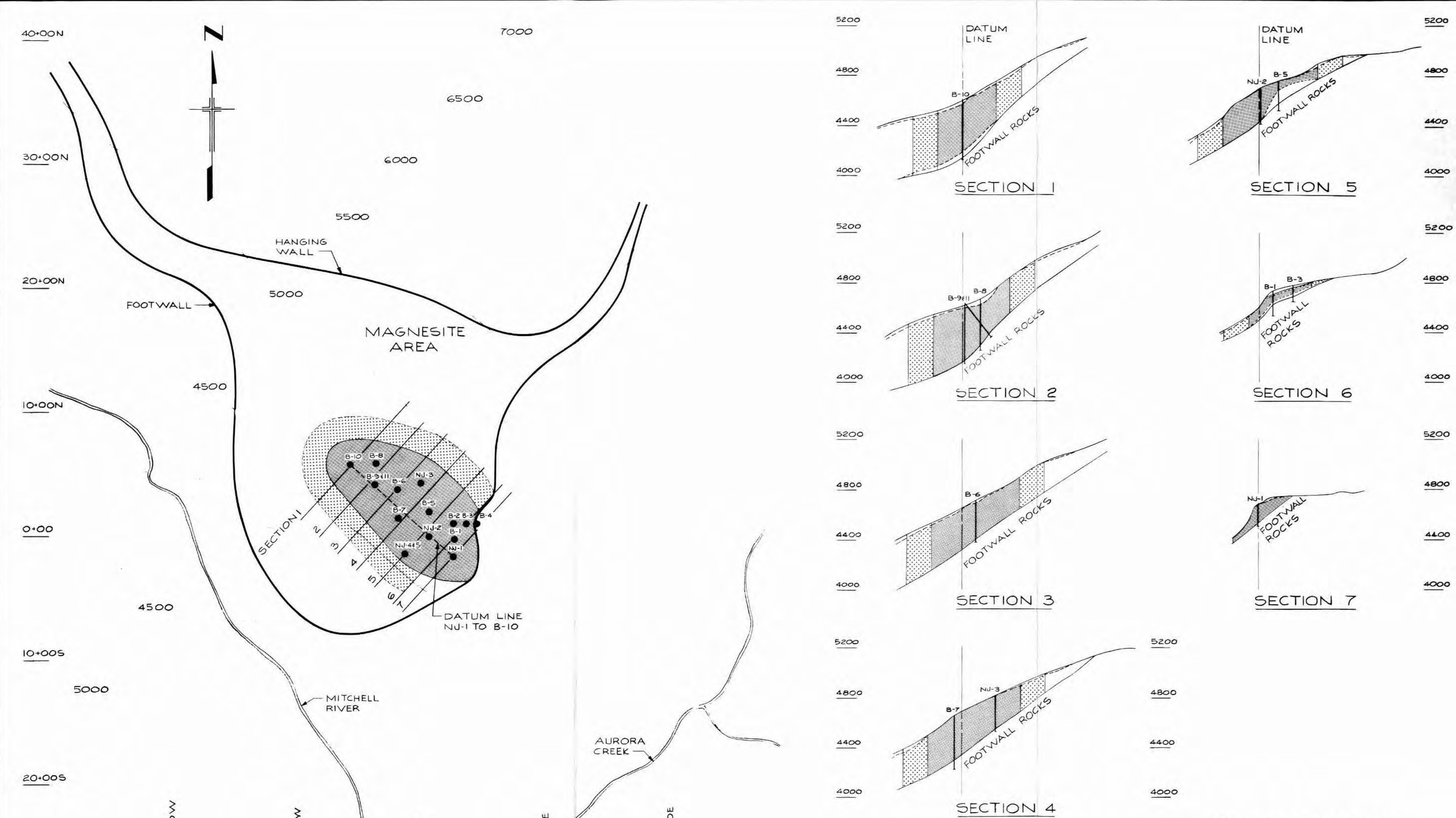
BRITISH COLUMBIA GEOLOGICAL SURVEY



LEGEND

PROVEN ORE

PROBABLE ORE

OVERBURDEN AND CONTAMINATED ROCK

GEOLOGY BY E. STARY P.ENG

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BRUSSILOF MAGNESITE PROJECT

ORE BODY PLAN AND SECTIONS

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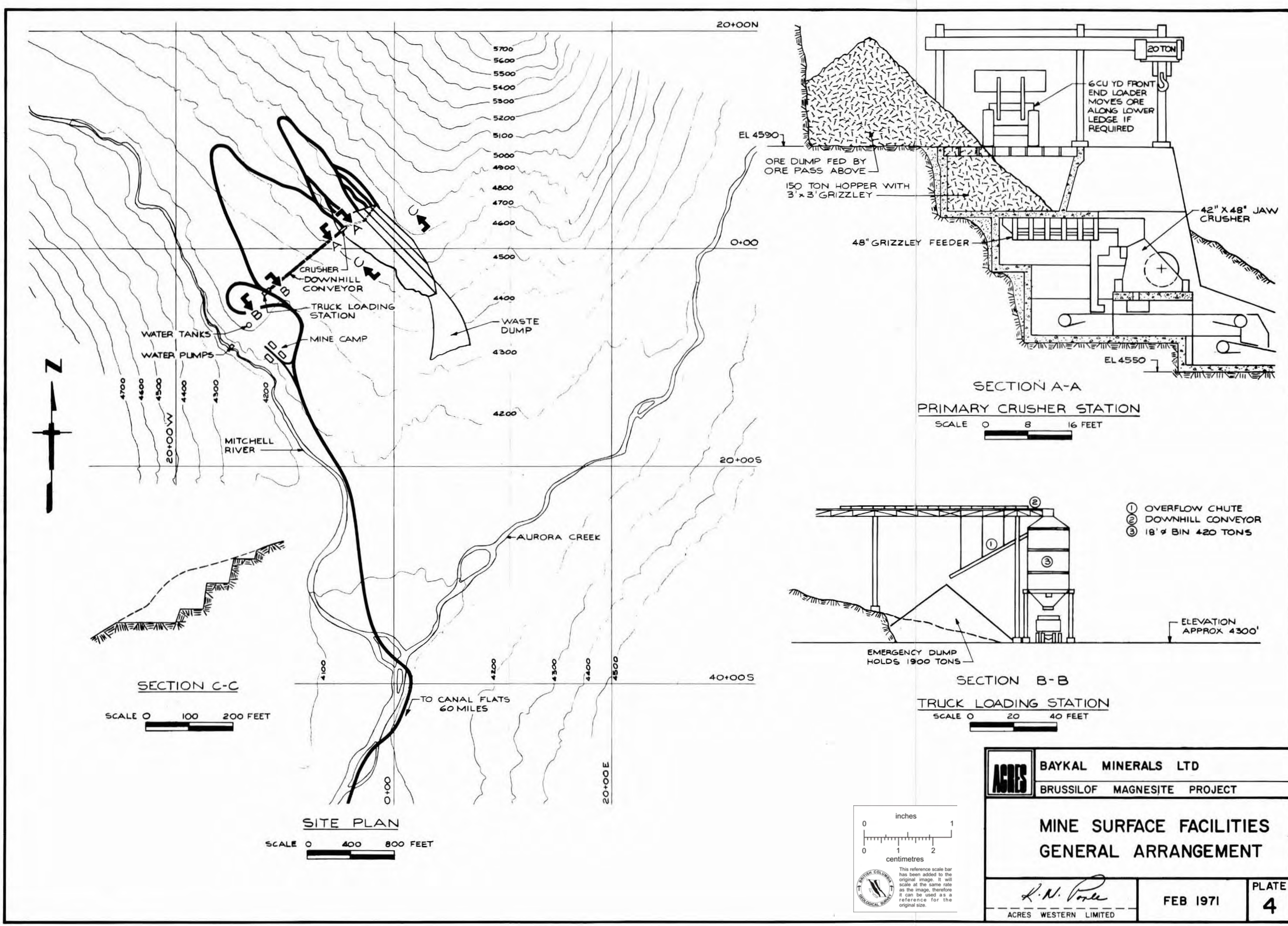
PLATE 3


20+00N
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5000
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4300
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0+00
20+00S
40+00S
20+00W
20+00E
0+00
4700
4600
4500
4400
4300
4200
4100
4200
4300
4400
4500
N
CRUSHER
DOWNHILL CONVEYOR
TRUCK LOADING STATION
WATER TANKS
WATER PUMPS
MINE CAMP
WASTE DUMP
MITCHELL RIVER
AURORA CREEK
TO CANAL FLATS 60 MILES
SITE PLAN
SCALE 0 400 800 FEET
SECTION C-C
SCALE 0 100 200 FEET
EL 4590
20 TON
6CU YD FRONT END LOADER MOVES ORE ALONG LOWER LEDGE IF REQUIRED
ORE DUMP FED BY ORE PASS ABOVE
150 TON HOPPER WITH 3'x3' GRIZZLEY
42" X 48" JAW CRUSHER
48" GRIZZLEY FEEDER
EL 4550
SECTION A-A
PRIMARY CRUSHER STATION
SCALE 0 8 16 FEET
① OVERFLOW CHUTE
② DOWNHILL CONVEYOR
③ 18' ⌀ BIN 420 TONS
ELEVATION APPROX 4300'
EMERGENCY DUMP HOLDS 1900 TONS
SECTION B-B
TRUCK LOADING STATION
SCALE 0 20 40 FEET
inches
0 1
0 1 2
centimetres
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BRITISH COLUMBIA GEOLOGICAL SURVEY
ACRES
BAYKAL MINERALS LTD
BRUSSILOF MAGNESITE PROJECT
MINE SURFACE FACILITIES GENERAL ARRANGEMENT
K.N. Poole
ACRES WESTERN LIMITED
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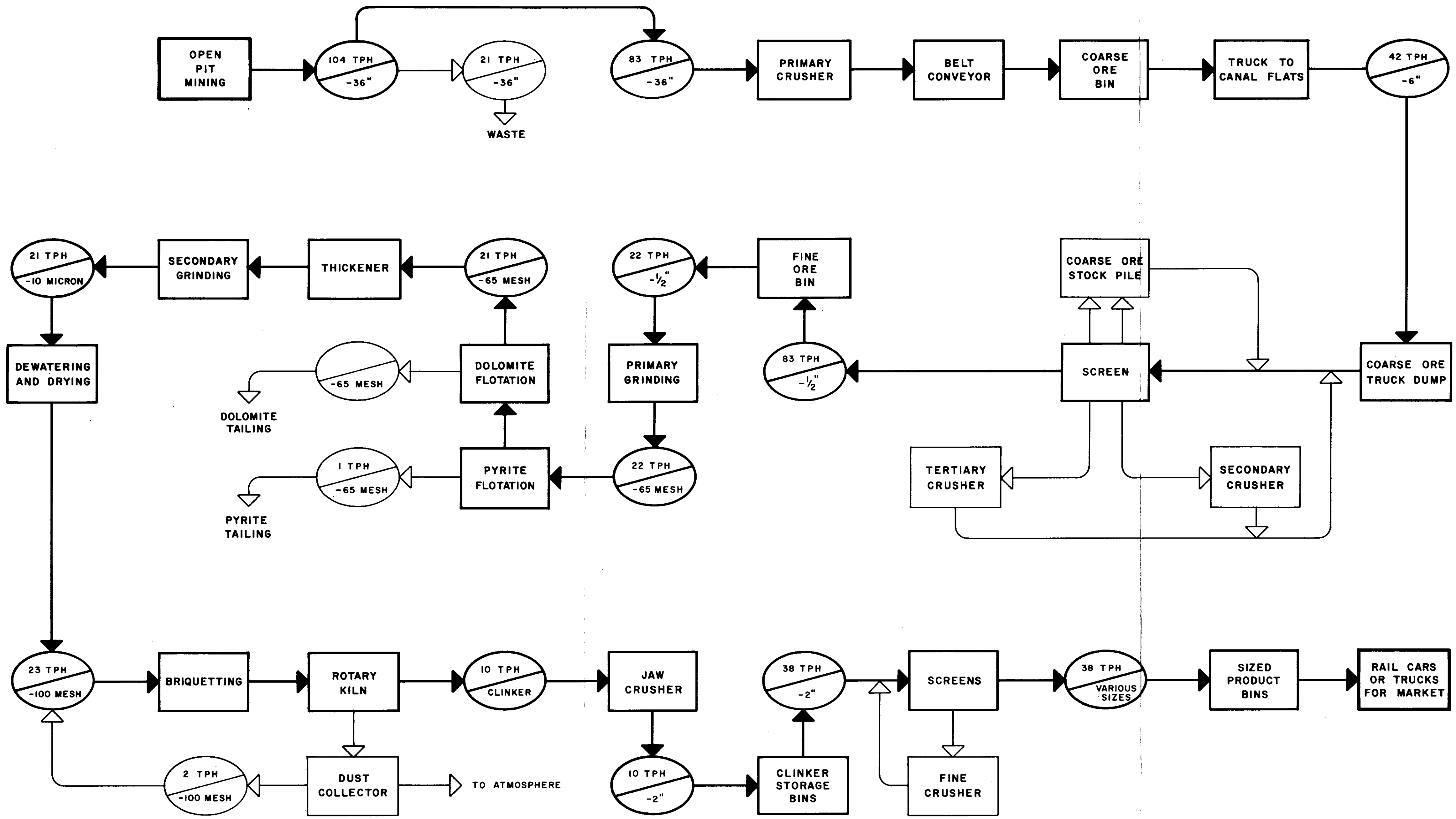
PLATE 4

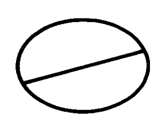


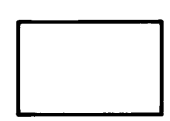
NOTES

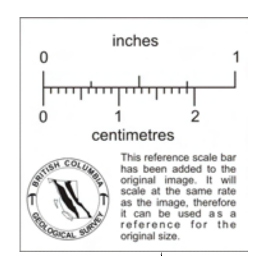
1. QUANTITIES SHOWN ARE BASED ON STAGE I OPERATION HAVING AN ANNUAL CAPACITY OF 75,000 TONS.
2. OPERATING SCHEDULES AT STAGE I PRODUCTION LEVEL —

MINING	1 SHIFT PER DAY, 5 DAYS PER WEEK
TRUCK TRANSPORT	2 SHIFTS PER DAY, 5 DAYS PER WEEK
CRUSHING	1 SHIFT PER DAY, 5 DAYS PER WEEK
BENEFICATION AND CALCINING	3 SHIFTS PER DAY, 7 DAYS PER WEEK
PRODUCT SIZING AND STORAGE	1 SHIFT PER DAY, 5 DAYS PER WEEK

LEGEND


HOURLY CAPACITY SHORT TONS
MATERIAL SIZE


OPERATION OR EQUIPMENT



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FLOW SHEET

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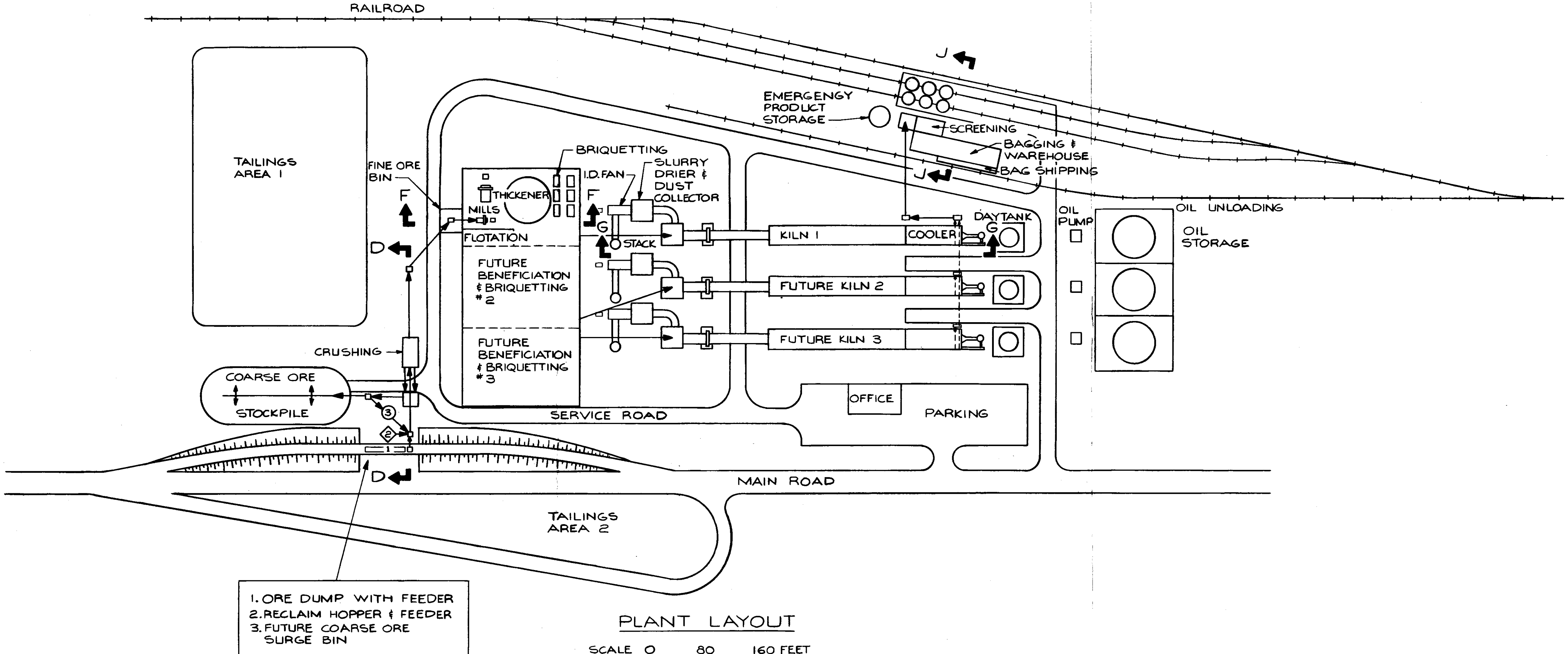
PLATE 5



PLANT LAYOUT

SCALE 0 80 160 FEET

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BAYKAL MINERALS LTD

BRUSSILOF MAGNESITE PROJECT

PROCESS PLANT LAYOUT

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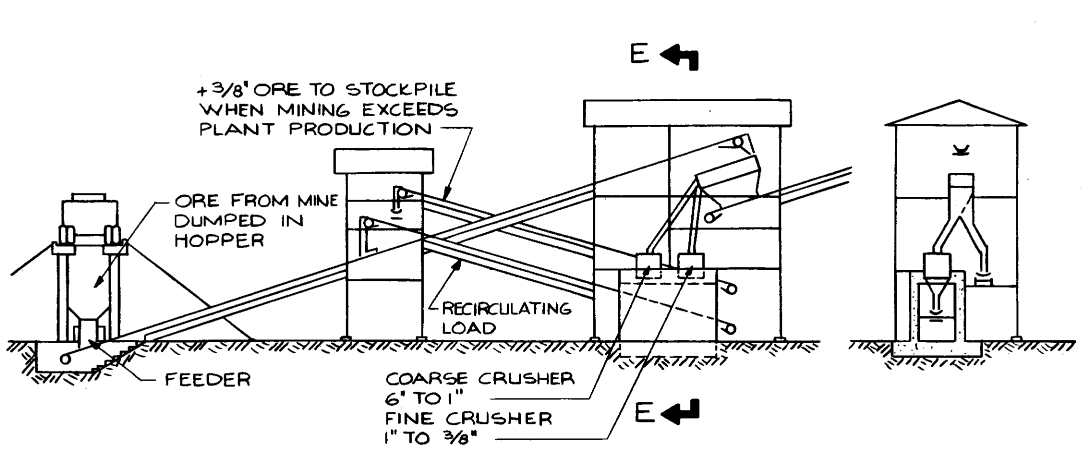
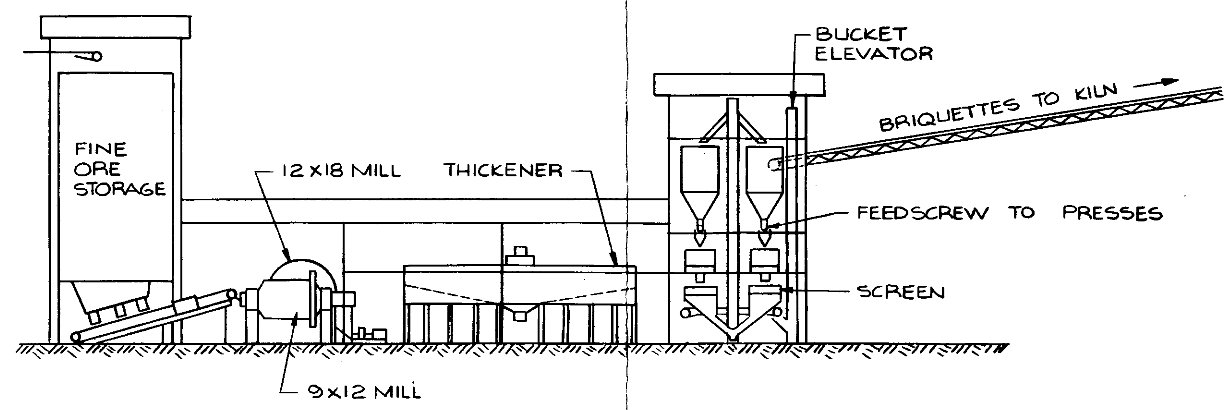
PLATE 6



ELEVATION D-D SECTION E-E

CRUSHING ARRANGEMENT

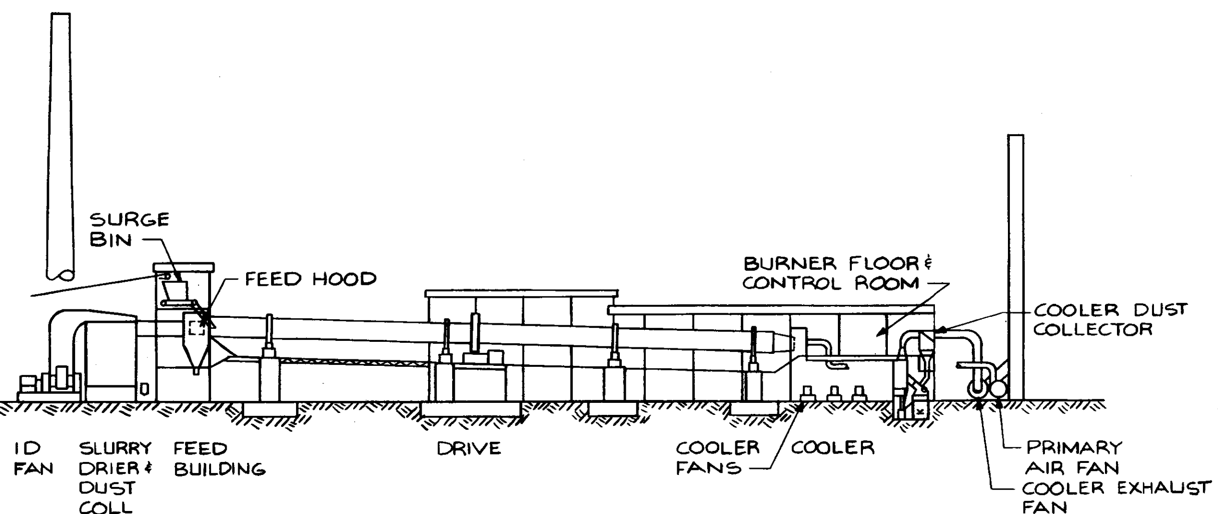
SCALE 0 20 40 FEET



ELEVATION F-F

BENEFICIATION AND BRIQUETTING PLANT

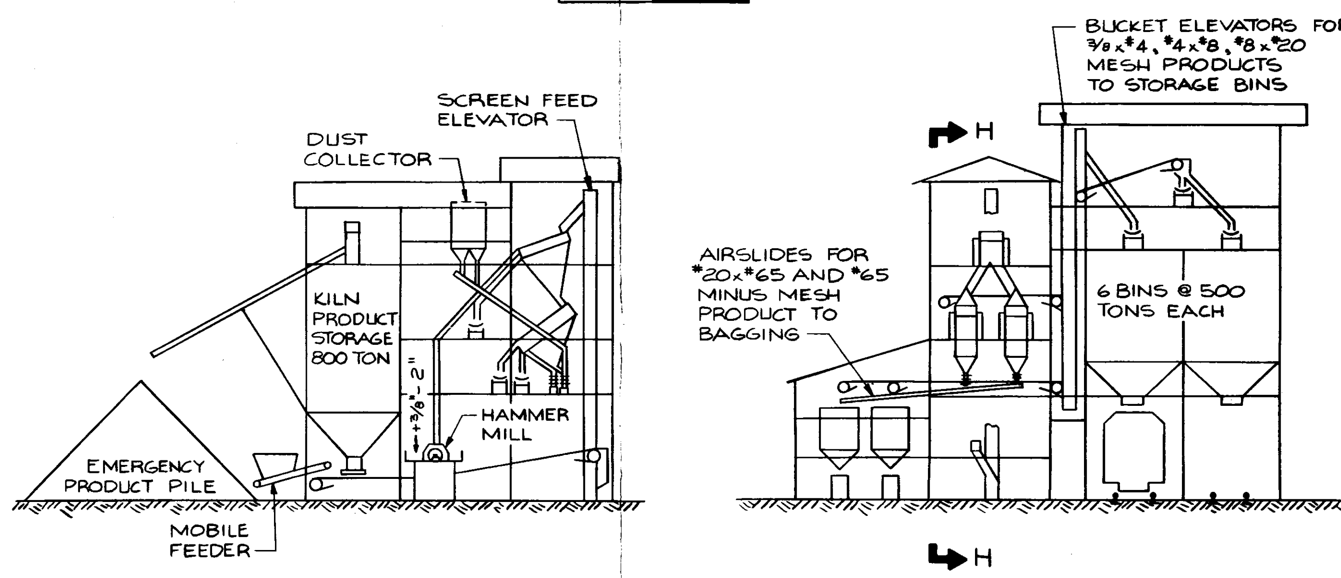
SCALE 0 25 50 FEET



ELEVATION G-G

KILN AND COOLER

SCALE 0 50 100 FEET



ELEVATION H-H ELEVATION J-J

GRAIN SIZING PLANT

SCALE 0 20 40 FEET

ACRES	BAYKAL MINERALS LTD
	BRUSSILOF MAGNESITE PROJECT
	PROCESS PLANT CROSS SECTIONS

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PLATE 7