Information Circular 8814

Valuation of Potash Occurrences Within the Nuclear Waste Isolation Pilot Plant Site in Southeastern New Mexico

By Robert C. Weisner, Jim F. Lemons, Jr.,
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Cecil D. Andrus, Secretary

BUREAU OF MINES
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Library of Congress Cataloging in Publication Data

Weisner, Robert C

Valuation of potash occurrences within the nuclear waste isolation pilot plant site in southeastern New Mexico.

(Information circular—Bureau of Mines; 8814)

Bibliography: p. 80

Supt. of Doc. no.: I 28.27:

1. Potash deposits—New Mexico—Carlsbad region. I. Lemons, Jim F., joint author. II. Coppa, Luis V., joint author. III. Title. IV. Series: United States. Bureau of Mines. Information circular; 8814.

TN295.U4 [TN919] 622'.08s [553'.636'0978942]

79–607990

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PREFACE

This report was prepared during the period 1976 through 1977 to serve a specific need of the Energy Research and Development Administration (ERDA), now a part of the Department of Energy. Subsequent broader interest in the report has demonstrated the need for publication. Some of the information presented on potash is now dated. The latest available statistics are published regularly by the Bureau of Mines in Mineral Industry Surveys, Mineral Commodity Summaries, and Mineral Commodity Profiles. Readers needing current potash information can contact the Bureau of Mines commodity specialist:

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TERMS AND ABBREVIATIONS

WIPP—Waste Isolation Pilot Plant.

The WIPP site is the area within the boundaries of land proposed for withdrawal for the waste isolation facility. The site is subdivided into Zones I through IV. Zone I is the area of the proposed location of the isolation facility; Zones II through IV are located concentrically at approximately 1-mile intervals from center of the site.

Ore zones is a term used to identify salt beds of minerals that locally contain potash minerals in commercial amounts.

The Bureau of Mines study area includes lands in the WIPP site, and adjacent lands extending out for up to 2 miles (3 kilometers) from the site boundary.

Mining Unit is a reasonable sized operating unit in terms of daily operating capacity and in lateral extent of workings for optimum ore handling, disregarding any limits imposed by the current proposed WIPP site. Size limits also considered were (1) present and projected market size and (2) quantity of process water available. The unit values for the potash, determined for a Mining Unit, were then used to determine the total potash values within the WIPP site.

K_2O—by usage and for convenience, the concentrations of various commercial potassium minerals in ores, concentrates, and products are often stated as equivalent concentrations of potassium oxide (K_2O).

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VALUATION OF POTASH OCCURRENCES WITHIN THE NUCLEAR WASTE ISOLATION PILOT PLANT SITE IN SOUTHEASTERN NEW MEXICO

by

Robert C. Weisner,[1] Jim F. Lemons, Jr.,[2] and Luis V. Coppa[3]

ABSTRACT

Current production costs and market conditions in the potash industry of the Carlsbad area were studied to determine the potential values of the potash mineral resource that would be lost or foregone if the Waste Isolation Pilot Plant (WIPP) facility is constructed on the proposed site in that area. The purpose of the WIPP project is to investigate the possibility of developing a nuclear waste disposal plant in the salt formations at the site. Analyses were made of all potash deposits determined to be in the site. Mining and processing under the most favorable recovery systems were considered. Value determinations were based upon estimated operating and capital costs of current mine-mill operations in the Carlsbad area. This study was made for the Energy Research and Development Administration (ERDA) by members of the Federal Bureau of Mines Minerals Availability System staff.

INTRODUCTION

The use of nuclear fuels produces radioactive waste that must be stored or disposed of in an acceptable and safe manner. A safe storage location has been determined to be in thick salt beds. ERDA requested the U.S. Geological Survey to indicate the location of known salt beds so that a study could be made to determine those most suitable for disposal or storage sites. Geological Survey recommendations included the Los Medanos area of Eddy County, N. Mex.

In September 1976 ERDA asked the Bureau of Mines to quantify and evaluate commercial potash mineralization in a proposed WIPP site in southeastern New Mexico. This information was necessary as one element in an environmental impact assessment by ERDA of the proposed WIPP project investigating the possibility of developing a nuclear waste disposal plant in the salt formations at the site.

The Bureau of Mines therefore conducted a study first to determine the amount and likely methods of recovery of existing potash deposits within the

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WIPP site and immediate vicinity, and second to determine the value of commercial potash mineralization, present and future, within the site and study area. As part of the study, an analysis was made to determine if any of the potash mineral occurrences are commercially recoverable by existing mining and processing techniques. This task was assigned to the Bureau's Minerals Availability System (MAS) personnel, since this group routinely conducts studies to determine the availability and cost to the United States of minerals and metals.

This report presents the details and results of the study prepared for ERDA. Included in the report are discussions of the geology and geography of the study area, its potash mining history, and current and projected market conditions in the potash industry. Prerequisite to the economic analysis, the Bureau of Mines obtained resource evaluation data from the Conservation Division of the U.S. Geological Survey (USGS) on measured and indicated categories of potash deposits in the WIPP site; their grades and tonnages were then reevaluated and modified to ascertain grades and tonnages that could be recovered in mining and processing operations.

The Bureau of Mines used criteria consistent with industry practice in preparing its economic feasibility studies; it employed a method of potash ore reserve calculations using engineering design and economic analytical procedures, including discounted cash flow, to determine the tonnage of minable potash ore that will yield an assumed, commercially acceptable (15 percent) rate of return on total capital investment. The mining and beneficiation systems evaluated were based on current extraction and processing technology in the Carlsbad district and were the least costly systems amenable to the ore to be mined.

This analysis isolated and estimated the values that exist in the unmined potash mineralization in the WIPP site and therefore determined a cost chargeable to the WIPP facility as losses that would result from constructing the nuclear waste disposal plant. This cost is the sum of lost taxes, royalties, and bonus bid amounts that would otherwise be generated by development of the potash mineralization to the maximum commercial extent—potash values that would be foregone due to the closing of the site to future mining operations.

ACKNOWLEDGMENTS

The authors were assisted in this evaluation by the following personnel of the System Operations Group and the Domestic Evaluation Group, Minerals Availability System, Intermountain Field Operations Center, Denver, Colorado: R. J. Minarik, W. L. Rice, R. L. Baer, L. B. Burgin, R. L. Davidoff, A. G. Hite, J. D. Lewis, C. M. Palencia, R. A. Salisbury, and R. C. Steckley. Data and assistance, especially helpful in preparing tables on worldwide and U.S. potash production, were given by R. H. Singleton, potassium commodity specialist of the Bureau of Mines Washington, D.C., headquarters office. Metallurgical testing was performed by personnel of the Bureau's Salt Lake City Research Center under the direction of Jerry L. Huiatt.

The U.S. Geological Survey provided information on the amount, occurrence, distribution, mineralization, and grade of potash in the study area, and data interpretation, based on available information and data from ERDA exploration drill holes. Conversations with Charles L. Jones, who supervised the USGS core drilling within the WIPP site and authored the USGS report summarizing potash mineralization in the study area, were helpful in Bureau of Mines analyses. His geologic work, together with other USGS studies, provided the foundations for the geologic section of this report.

Invaluable assistance was received from the potash industry in the Carlsbad area, utility companies, and State and local government offices.

SURVEY OF THE DOMESTIC POTASH INDUSTRY

HISTORY

From early colonial times until 1860, the manufacture of potash from wood ashes constituted a significant chemical industry that met the needs of the United States and also provided an important commodity for export. This industry was seriously curtailed by the development of the Le Blanc process that provided a cheap source of sodium carbonate, which could be used instead of potassium carbonate for many industrial purposes. The end of this first potash industry came in 1861 when commercial muriate of potash was produced from deposits near Stassfurt in northern Germany.

From 1861 to 1914, the potash industry was virtually nonexistent, and agriculture in the United States was almost totally dependent upon imported German potash (*7*).[4] However, in January 1915, shortly after the onset of World War I, Germany placed an embargo on the export of potash salts and the total U.S. supply was cut off. As a result, potash salts which previously had sold at a normal price of $35.00 to $40.00 per ton ($38.58 to $44.09 per metric ton) were quoted at prices ranging from $350.00 to $425.00 per ton ($385.80 to $468.47 per metric ton) (*6*). These prices resulted in a burst of activity in the domestic potash industry. Methods of producing potassium compounds from kelp, wood ashes, lake brines, alunite, cement dust, sugar beet waste, blast furnace dust, and other sources were developed. By 1918 there were 128 producers of potash compounds with a total annual production of approximately 55,000 tons (49,896 metric tons) of K_2O(*19*).

With the resumption of German imports following the end of World War I, the domestic potash industry virtually collapsed again (*20*). By 1920 the American Trona Corp. (subsequently American Potash and Chemical Corp. and now Kerr-McGee Chemical Corp.) was the only significant producer of potash salts in the United States. Between 1920 and 1930 the United States was again dependent upon foreign imports for more than 80 percent of its potash requirements.

[4] Underlined numbers in parentheses refer to items in the list of references preceding the appendixes.

The Federal Government in 1924 authorized the Bureau of Mines and the U.S. Geological Survey "to determine location and extent of potash deposits in the United States" (*3*). By 1931, these agencies had identified, by core drilling, saline beds in the Permian Basin of Texas and New Mexico ranging in thickness from 1-½ to more than 8 feet (½ to more than 2 meters) and containing 9 to 14 percent K_2O (*18*).

Interest in the Permian Basin area had been generated by potash showings from oil exploration. The most significant of these showings was in the McNutt No. 1 oil test, drilled by the Snowden and McSweeney Co. in February 1925. Additional exploration by the Snowden and McSweeney Co. generated further interest and eventually resulted in the formation of the American Potash Co. This company continued the exploration efforts initiated by the Snowden and McSweeney Co. The minerals sylvite, polyhalite, langbeinite, and carnallite were found in many of the core tests, and the continuity of the bedded deposits became reasonably well established. In December 1929, the No. 1 shaft of the American Potash Co. was begun, and the first commercial production of potash from an underground mining operation began on March 7, 1931.

The name of American Potash Co. was changed to the U.S. Potash Co. in 1930. This company completed construction of its first potash refinery, a crystallization plant, and began production of muriate of potash on September 17, 1932. Its second mine shaft was completed in June 1933, which increased mining capacity to more than 2,000 tons (1,814 metric tons) of ore per day. U.S. Potash Co. was merged and became U.S. Borax and Chemical Co. in 1956. The Carlsbad potash properties were than sold to U.S. Potash and Chemical Co., a subsidiary of Continental American Royalty Co., in 1968. Subsequent sales of this property were to Teledyne, Inc., in 1972 and to Mississippi Chemical Co. in 1974 (*5*).

Potash Co. of America (now a division of Ideal Basic Industries, Inc.) began potash production in the Carlsbad area in 1934. This company used the same room-and-pillar mining methods as U.S. Potash Co., but its refinery consisted of a

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halite flotation process to separate the sylvite and halite minerals. This company has been in continuous production since 1934. Its refining process has been converted from halite flotation to the more economical potash flotation method now in almost universal use.

Union Potash and Chemical Co. (now International Minerals and Chemical Corp.) began mining both sylvite and langbeinite in 1940 (*9*). This refinery was the first commercial application of the potash flotation process (rather than halite flotation) for muriate of potash production. The langbeinite was refined by a leaching process. Commercial production of agricultural-grade potassium sulfate began shortly thereafter. Since the only known commercial deposits of langbeinite ore are in the Carlsbad area, this was the first commercial production of this mineral worldwide.

During World War II, the production capabilities of the three Carlsbad potash companies, plus the production from American Potash and Chemical Co. at Trona, Calif., and Bonneville Potash, Ltd. (now Kaiser Aluminum and Chemical Corp.) at Wendover, Utah, prevented a serious shortage of potash in the United States. Potash production was classified as a defense industry, and appropriate priorities were provided for personnel, materials, and equipment so that maximum production was maintained during the war years.

Postwar expansion of the potash industry began with the completion of the Duval Corp. (now a division of Pennzoil Corp.) plant near Carlsbad in 1951. This project was followed by the Southwest Potash Co. (a wholly owned subsidiary of AMAX, Inc.) construction in 1952, and the National Potash Co. (a division of Freeport Minerals Corp.) construction in 1957. All three of these operations used a potash flotation process to refine their sylvite ores. In 1964 Duval opened a new mine (Nash Draw) to produce langbeinite ore and increase its sylvite reserves. The Duval flotation plant was expanded at that time to include a langbeinite leaching process and to provide facilities for the production of potassium sulfate.

Development of the first domestic underground potash operation outside the Carlsbad area was begun in 1961 in the Paradox Basin near Moab, Utah, by Texas Gulf Sulphur Co. (now Texasgulf, Inc.). Production of muriate of potash from this operation began in 1965, but mining problems made this operation uneconomic. In 1971 Texasgulf removed all equipment from the mine and started using another technique for potash recovery. Water is pumped into the mine from the nearby Colorado River and becomes saturated with potassium and sodium chlorides; it is then pumped out of the "mine" to solar evaporation ponds. On evaporation, a mixture of sylvite and halite is deposited in the ponds. These salts are then harvested and refined in a conventional potash flotation plant to produce muriate of potash.

The most recent domestic potash operation is Kermac Potash Co. (a division of Kerr-McGee Corp.), which began production of muriate of potash in 1965. Kermac is located a few miles east of the other six producers in the Carlsbad area. Because the characteristics of the Kermac ore made it very difficult to refine by flotation, a crystallization process was developed. This process makes a higher quality (K_2O content) muriate product.

During the 1960's a major expansion in potash production capacity occurred in Saskatchewan, Canada. As these large new operations came onstream, the world supply of muriate of potash increased substantially and prices dropped. Investigation by the U.S. Government concluded that there was evidence of "dumping," and economic sanctions were proposed against some Canadian producers. Under the threat of these proposed economic sanctions, the Government of Saskatchewan in 1970 imposed production limitations and established minimum prices for Canadian potash. These actions by the Saskatchewan Government returned a degree of stability to the domestic potash industry (*22*).

ECONOMIC BACKGROUND

The world potash industry is regarded by some as an oligopoly (*17*). They contend there is a recognized interdependence in the market and high cross elasticities of demand. That is, from the buyer's viewpoint, potash is an undifferentiated product, equal in specifications from all suppliers, making market price virtually the only consideration in purchasing. Because potash is a comparatively high-bulk, low-value commodity, its marketing is affected significantly by transportation costs that constitute a substantial part of the market price. Almost 50 percent of U.S. consumption is in Illinois, Iowa, Indiana, Minnesota, Ohio, and Wisconsin. Because of the transportation cost, Saskatchewan producers maintain an advantage over the New Mexico producers in these markets (*17*).

Prior to World War II, price leadership was the coordinating mechanism for determining price and market share in the industry. Since World War II, structural changes in both the selling and buying markets have progressively limited the feasibility of collusion among potash-producing firms. In fact, during the late 1960's, price cutting became widespread and the entire price structure collapsed, resulting in severe financial losses. The aggregate demand for potash is generally regarded as relatively inelastic, mainly due to the minor position of fertilizer in total farming costs.

About 1960 "bulk blending" of fertilizers began. Natural gas is used in petrochemical complexes as the key ingredient for producing ammonia that is combined directly with phosphoric acid to manufacture nitrogen-phosphate compounds. These nitrogen-phosphate compounds are then blended with potash prior to being sold as fertilizers. The implementation of this process had a threefold impact on the potash industry. First, some petroleum companies in the fertilizer business, because of their natural gas product, integrated vertically and horizontally by acquiring financial interests in the potash industry. Second, long-term contractual agreements for potash purchases were negotiated; and third, the number of potential buyers of straight potash fertilizer was greatly reduced.

Entry into the potash industry by a new producer is difficult because of the lack of exploitable deposits, the requirement of technological expertise, large capital requirements, and a likely cost disadvantage to a new firm as compared to an established company.

MARKET ANALYSIS AND MARKET PROJECTIONS

During 1975, United States consumption of potash (K_2O) decreased 17 percent to 5 million short tons (4.5 million metric tons)(table 1). This decrease, the first since 1961, was mainly due to general resistance by the agriculture industry to high fertilizer prices. Of total U.S. potash production, 95 percent is consumed by the fertilizer industry.

This reversal proved short-lived, however, as 1976 (estimated) consumption was up to 6 million short tons (5 million metric tons) representing a 26-percent increase over the 1975 figure and exceeding the previous (1974) high by more than 300,000 short tons (272,000 metric tons). Lower prices and supplier discounting during the summer largely accounted for this increase.

In 1976, domestic production of potash was approximately 2.4 million short tons (about 2.2 million metric tons), about 40 percent of domestic demand. By 2000, U.S. production is projected to decline significantly, supplying less than 10 percent of the total U.S. consumption in that year (*22*). Domestic consumption is projected to double to 12 million short tons (11 million metric tons) of K_2O. This increased production represents a growth rate of 2.9 percent per year (*16*), a decline from the previous 6.3-percent annual growth rate. The decline is based on the assumption of increasing physical limitations on potash consumption. Examples of these limitations include the decline of available land for farming, more efficient use of fertilizer, new technology in the use of fertilizer, and a possible decline in the foreign market for U.S. food products, particularly in the developing nations. Production and consumption rates projected for 2000 could be changed by such developments as higher foreign prices for potash, thereby making the deeply buried beds in Mich-

TABLE 1.—Potash supply-demand relationships, 1965–76

(Thousand short tons of K_2O)

	1965	1966	1967	1968	1969	1970	1971	1972	1973	1974	1975	1976[e]
World production:												
United States	3,140	3,320	3,299	2,722	2,804	2,729	2,587	2,659	2,603	2,552	2,501	2,390
Rest of world	12,060	12,739	14,054	15,145	16,394	17,284	19,358	19,401	21,695	23,516	24,922	25,110
Total	15,200	16,059	17,353	17,867	19,198	20,013	21,945	22,060	24,298	26,068	27,423	27,500
Components of U.S. supply:												
Domestic mines	3,140	3,320	3,299	2,722	2,804	2,729	2,587	2,659	2,603	2,552	2,501	2,390
Imports	1,108	1,491	1,708	2,166	2,332	2,605	2,766	2,961	3,587	4,326	3,736	4,741
Industry stocks, Jan. 1	295	504	690	863	676	392	454	428	468	206	211	619
Total U.S. supply	4,543	5,315	5,697	5,751	5,812	5,726	5,807	6,048	6,658	7,084	6,448	7,750
Distribution of U.S. supply:												
Industry stocks, Dec. 31	504	690	863	676	392	454	428	468	206	211	619	500
Exports	648	621	693	735	700	544	564	764	889	787	769	891
Demand	3,391	4,004	4,141	4,340	4,720	4,728	4,815	4,816	5,563	6,086	5,061	6,359
U.S. demand pattern:												
Agriculture	3,174	3,771	3,913	4,101	4,490	4,516	4,566	4,538	5,261	5,792	—	—
Chemicals	217	233	228	239	230	212	249	278	302	294	—	—
Total U.S. primary demand	3,391	4,004	4,141	4,340	4,720	4,728	4,815	4,816	5,563	6,086	5,061	6,359
Price: Cents per short ton unit (20 lb) of K_2O, standard 60 percent muriate, f.o.b. Carlsbad (average, quoted)	39	39	34	29	25	33	34	34	35	49	73	72

[e]Estimate.

igan and North Dakota economic, or the failure of the domestic agriculture market to develop as anticipated.

Estimated U.S. exports in 1976 increased about 15 percent from the 1975 figure to nearly 900,000 short tons (810,000 metric tons) of K_2O. North America produced about 25 percent of the world's potash in 1976. Estimated annual production capacity of potash in North America totals 10 million tons (9.1 million metric tons) of K_2O, nearly three-quarters of which is in Saskatchewan (table 2).

About 82 percent of the 1976 domestic production was from the Carlsbad region in New Mexico; the balance was from Utah and California. Ten companies comprise the domestic industry. Nine companies have only one operation, while one (Kerr-McGee) has a mine in New Mexico and a plant in California (table 3).

About 15 percent of U.S. products are potassium sulfate or potassium magnesium sulfate. In 1975, U.S. production of these sulfates totaled nearly 400,000 tons (360,000 metric tons). Sulfate compounds are produced both from langbeinite ore by two Carlsbad companies and from brines in Utah and California. However, the Carlsbad district is the only known source of commercial langbeinite mineralization in the United States. About one-third of the sulfate production is exported.

Net potash imports have grown steadily to meet increasing U.S. consumption. In 1974, 96 percent of the imports were from Canada, 1 percent from Israel, the balance from the U.S.S.R., West Germany, and other countries (fig. 1)(*15*).

Domestic potash reserves are estimated to be about 200 million short tons (about 181 million metric tons) of K_2O recoverable at 1973 prices. These reserves include about 100 million short tons (about 90.7 million metric tons) in bedded deposits in New Mexico, 70 million short tons (64 million metric tons) in brines, and an estimated 30 million short tons (27 million metric tons) in Utah bedded deposits (*14*).

In addition to the reserves, domestic potash resources include perhaps an additional billion or more tons of K_2O, mostly in the extension of the Williston Basin southward into the Montana-North Dakota area and revised estimates of deposits in the Paradox Basin in Utah. Large deposits in Montana and North Dakota are now being studied by Kalium Chemicals, a subsidiary of PPG Industries, and others for possible recovery by solution mining. Exploratory drilling was begun in 1976 in the Montana-North Dakota area near the Canadian (Saskatchewan) border by two operators, Kalium Chemicals and

TABLE 2.—World mine production and reserves

(Thousand short tons of K_2O)

World mine production and reserves	Production 1975	Production 1976[e]	Reserves
United States:			
WIPP Site	0	0	6,000
Carlsbad District	W	W	100,000
Rest of United States	W	W	100,000
Total	2,501	2,390	206,000
Canada	5,992	5,400	5,000,000
Congo (Brazzaville)	309	300	20,000
France	2,298	2,300	100,000
Germany, West	2,450	2,200	1,800,000
Israel and Jordan (Dead Sea)	789	700	240,000
Italy	160	160	10,000
Spain	506	500	80,000
Other market economy countries	50	250	250,000
Central economy countries	12,368	13,300	3,500,000
World total	27,423	27,500	11,206,000

[e]Estimate.
W Withheld to avoid disclosing company proprietary data.

TABLE 3.—Domestic potash producers, yearend 1976

Company	Location
AMAX Chemical Corp.	New Mexico.
Duval Corp. (subsidiary of Pennzoil)	New Mexico.
International Minerals & Chemical Corp. (IMC)	New Mexico.
Great Salt Lake Minerals & Chemical Corp.	Utah.
Kaiser Aluminum & Chemical Corp.	Utah.
Kerr-McGee Corp.	New Mexico. California.
Mississippi Chemical Corp.	New Mexico.
National Potash Co. (subsidiary of Freeport Minerals Co.)	New Mexico.
Potash Company of America (subsidiary of Ideal Basic Industries)	New Mexico.
Texasgulf	Utah.

jointly by the Burlington Northern Railroad and C. F. Industries. It is reported both parties are planning large investments in solution mining, contingent upon the outcome of current feasibility studies. These deposits are about 7,000 to 9,000 feet (about 2,134 to 2,743 meters) deep.

During the first half of the 1970's (through 1975) the price in constant dollars for North American potash approximately doubled. In the same period, constant-dollar prices for phosphate and ammonia nearly quadrupled and tripled, respectively. The rate of these price rises escalated during this period and was particularly significant in 1974 and 1975. The average 1975 quoted selling price for U.S. standard muriate grade potash was \$77 per ton (\$85 per metric ton) of K_2O f.o.b. producers' plant site. The price increase trend for potash produced in the United States was reversed in June 1976 when the quoted price was lowered by the Potash Co. of America to \$63 per ton (\$69 per metric ton) of contained K_2O f.o.b. Carlsbad, N. Mex. (fig. 2).

In 1962, the United States became a net importer of potash. This was due largely to the tremendous expansion in the Canadian potash

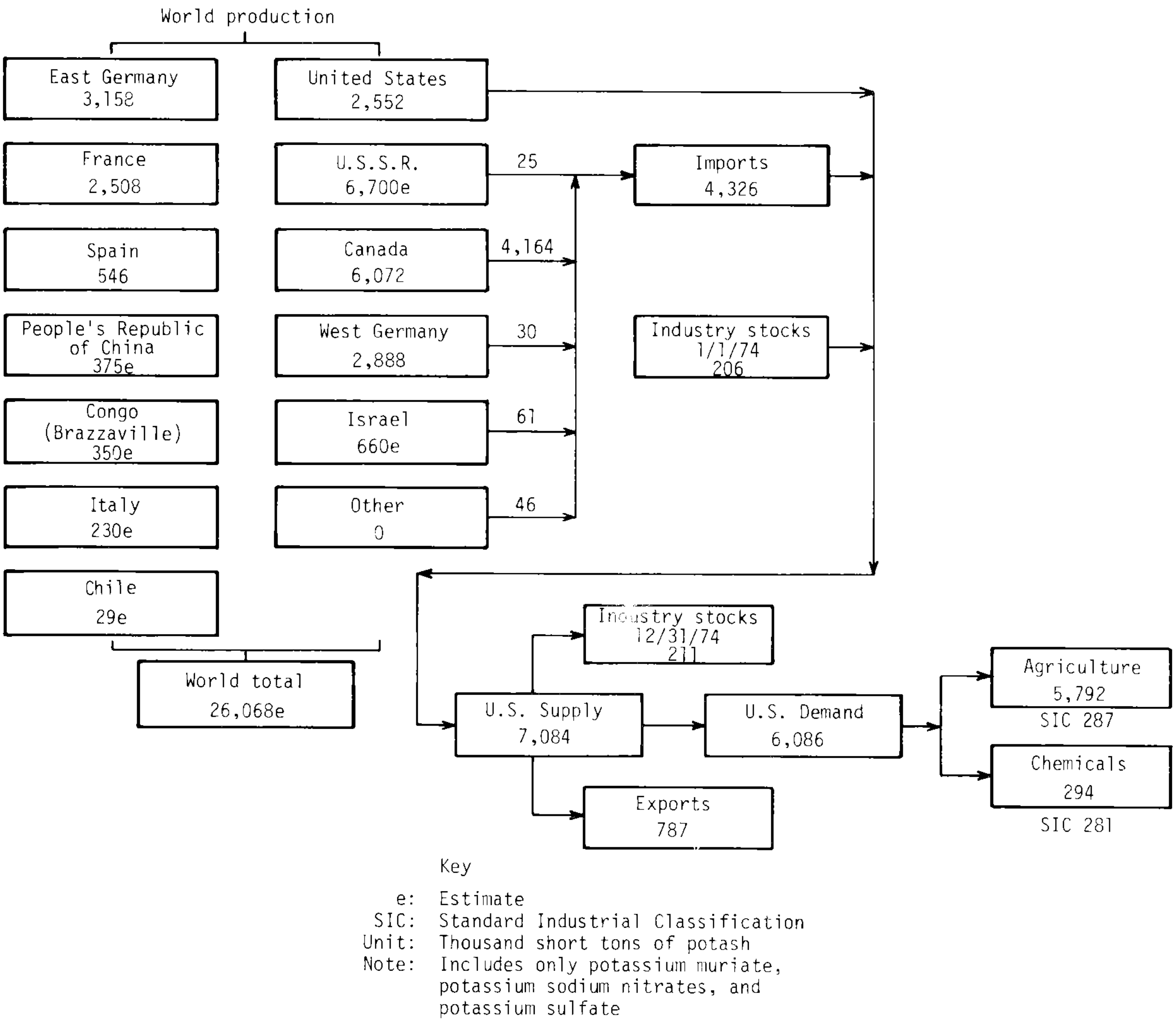

FIGURE 1.—Potash supply-demand relationships—1974.

industry. By 1968, the Canadians moved from a position of net importation and had captured 45 percent of the U.S. market (*24*). On December 19, 1969, the Treasury Department issued a finding that Canadian, West German, and French firms were selling muriate of potash (KCl) on the U.S. market at less than fair prices, in violation of the 1921 Antidumping Act. In 1970, authority was given to the Tariff Commission (now called the International Trade Commission) to collect a dumping duty. However, no duties have been collected because the Saskatchewan Government imposed restrictions on production, and prices of Canadian potash exported to the United States were increased. Nevertheless, firms selling Canadian potash on the U.S. market have been subject to extensive reporting requirements, so that price differentials could be evaluated by U.S. Customs. Subsequent heavy demand and resulting price increases allowed the Saskatchewan Provincial Government to lift these controls in 1974. Most of the importers have been removed from the reporting list; several were dropped in 1974 and more on August 5, 1976.

A complicated reserves tax was imposed by the Saskatchewan Government in October 1974, which, according to the Canadian Potash Association, amounts to over 80 percent of gross profit after adding Federal, Provincial, and local taxes and levies. Members of the industry filed suit in Provincial court on the constitutionality of the reserves tax, and a lengthy litigation was promised by the Provincial Government through to the Canadian Supreme Court, if necessary.

Legislation enabling acquisition of 50 percent or more of the Provincial potash industry was enacted by the Saskatchewan Government in

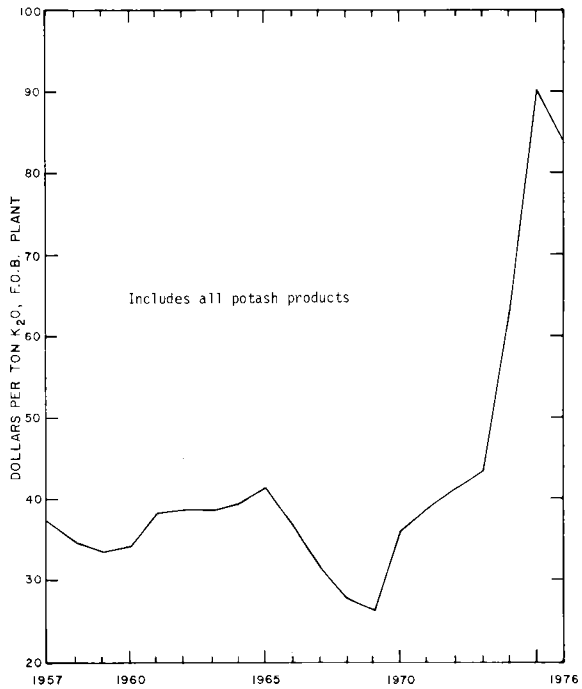

FIGURE 2.—Annual trend for 20-year span of U.S. domestic potash prices.

January 1976. The first unit, owned by Duval Corp., was acquired in October 1976 for $128.6 million, and the second, owned by Sylvite of Canada, Ltd., was acquired in April 1977 for $144 million. These facilities represent about 18 percent of the production capacity of the Saskatchewan potash industry (*14*).

The Antitrust Division of the U.S. Department of Justice met in early 1975 in Chicago to begin an investigation of the fertilizer industry. In June 1976, five U.S. producers were indicted and charged with restricting production and controlling prices in the United States and also with conspiring to coordinate United States and Canadian production for the control of prices of potash; they were also indicted on a charge of coordinating export of potash from the United States and the import of potash from outside North America into the United States. The potential fine is $50,000 per company, but more serious is the threat of customers bringing legal actions asking payment for damages sustained. Court trial began in Chicago in January 1977; all five producers were acquitted in May 1977. However, many class action civil suits seeking damages for overcharging are still pending.

LOCATION AND DESCRIPTION OF STUDY AREA

The proposed site considered by ERDA for a waste isolation pilot plant is 25 miles (40 kilometers) east of Carlsbad, N. Mex., and occupies about 29.6 square miles (76.7 square kilometers), with its center at the intersection of sections 20, 21, 28, and 29, T 22 S, R 31 E (fig. 3). A study area of roughly 64 square miles (roughly 166 square kilometers), including 34.4 square miles (89.1 square kilometers) outside the WIPP site, was examined to better understand the geology and potash mineralization within the site. The study area consists of sections 1–2, 11–14, 23–26, and 35–36 in T 22 S, R 30 E; all of T 22 S, R 31 E; sections 1, 2, 11, and 12 in T 23 S, R 30 E; and sections 1–12 in T 23 S, R 31 E.

Carlsbad is the nearest population center. The

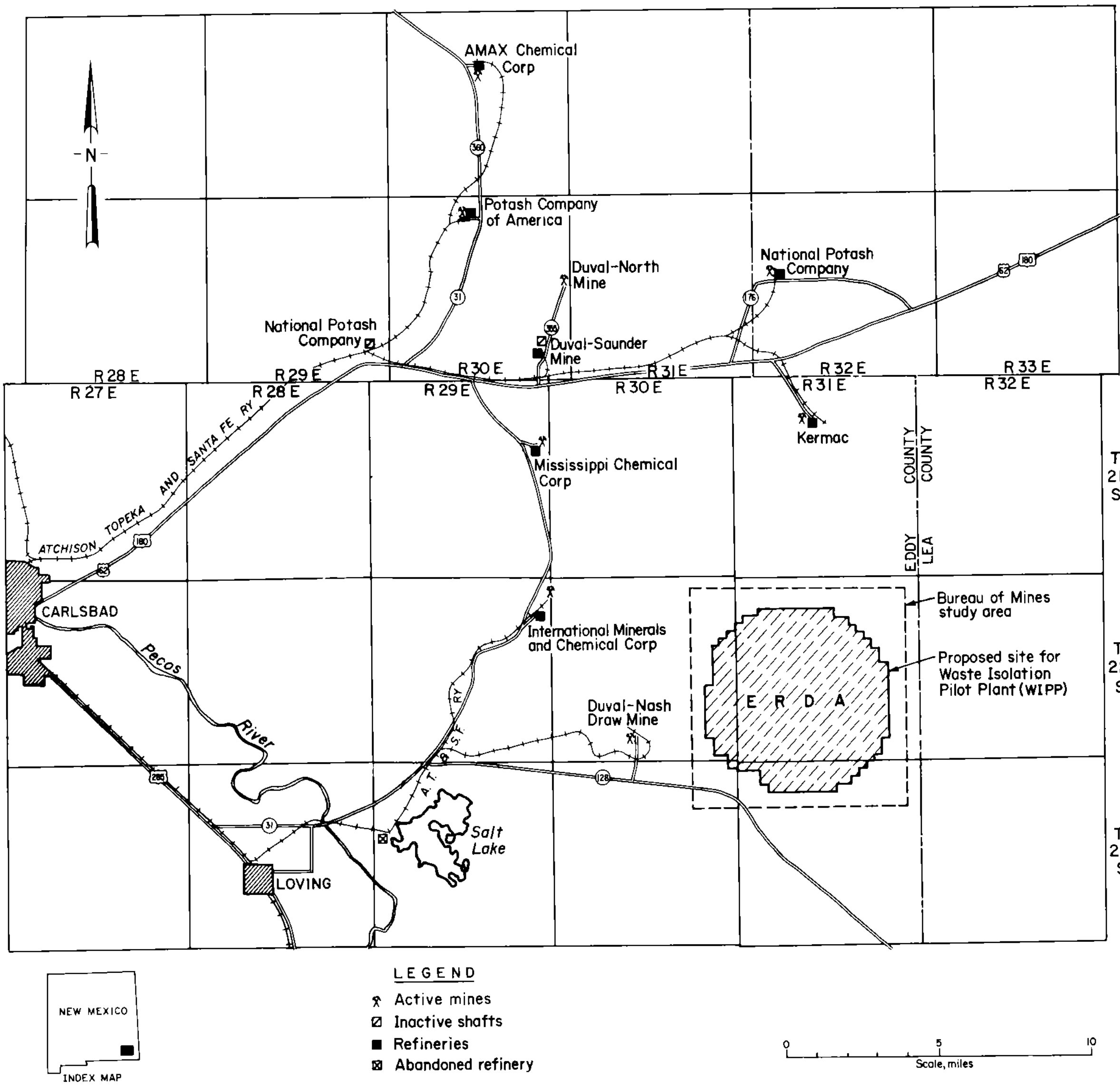

FIGURE 3.—Location map of the WIPP site.

nearest potash operation is Duval Corporation's Nash Draw mine, about 2 miles (about 3 kilometers) west of the WIPP site boundary. Records indicate that four holes were drilled within the site for oil or gas; all were dry holes, and they have since been plugged and abandoned. Producing wells would be detrimental to ERDA objectives in that they could act as conductors of hydrocarbon gas or fluid and thus provide access of such hazardous materials to storage areas constructed in the site. Oil and gas producing formations are stratigraphically below the proposed site of the underground waste storage facility. The top of the salt beds is 750 to 1,500 feet (229 to 457 meters) below the surface. The beds range from 1,500 to 2,000 feet (457 to 610 meters) thick and dip gently to the southeast. The area is stable tectonically with no known active faults.

The Federal Government owns most of the land surface and mineral rights in the area, but there are some State and privately owned interests. The ownership and areas of land involved are listed in table 4.

Within the study area, all State lands and part of the Federal potash mineral lands are either leased or under lease applications. Figure 4 indicates the leaseholders' names and lease numbers along with the names of the prospecting permit applicants and their application serial numbers. The size and location of the WIPP site with zones within the study area are depicted on figure 5.

Access is afforded by State Highway 128 and from the north by U.S. Highway 180. If an all-weather road were constructed into the WIPP site, it probably would be built south from Highway 180. This highway is better suited to heavy haulage than Highway 128 and provides more convenient access to Carlsbad and other centers.

A rail spur built into the area would probably be serviced by the Santa Fe Railroad as an extension of its lines. A spur could be built as an extension of the Nash Draw mine or the Kermac

TABLE 4.—Surface and mineral ownership

	In the study area		In the WIPP site	
	Acres	Percentage	Acres	Percentage
Federally owned surface and mineral rights.	35,440	86.5	17,201.58	90.7
State owned surface and mineral rights.	5,120	12.5	1,759.49	9.3
Privately owned surface and mineral rights.	80	.2	.00	—
Privately owned surface and federally owned mineral rights.	40	.1	.00	—
Privately owned surface and mineral rights except oil and gas federally owned.	280	.7	.00	—
Total	40,960	100.0	18,961.07	100.0

spurs (fig. 3). In either case, about 7 miles (about 11 kilometers) of standard-gage rail line would be required.

The water supply for a new potash-producing mine-mill complex is a critical item. Possible sources of water considered for potash refining are the Pecos River and aquifers in the Capitan Reef, Rustler, and Ogallala Formations. The best quality water found in the region is produced from the Caprock area of the Ogallala Formation, 25 to 30 miles (40 to 48 kilometers) to the northeast of the study area. This source presently supplies the potash industry with about two-thirds of its water and is being used increasingly by the industry. The water quality is good, ranging from 500 to 600 parts per million total dissolved solids.

This study assumes that Caprock area-Ogallala Formation water will be available for a new refinery. Adequate water rights are available and well site leases are obtainable on most of the Caprock field. The producing area is owned largely by the State of New Mexico. Present wells are 200 to 250 feet (61 to 76 meters) deep and are spaced on about ¼-mile (0.4-kilometer) centers. Each well, if properly designed and developed, can produce 200 to 250 gallons per minute (756 to 945 liters per minute) for 30 to 40 years. It is estimated that a pipeline from the Caprock field to a new refinery in the study area would be 30 miles (48 kilometers) long.

Natural gas for a potash refinery could be supplied by the Gas Co. of New Mexico. This company presently supplies the Kermac refinery using a 6-inch (15-centimeter) line, which has sufficient capacity to supply an additional refinery and could be extended into the study area.

Electric power is supplied by the Southwestern Public Service Co. A typical load for a potash mine and refinery is estimated to be about 6,000 kilowatts, with usage at approximately 4 million kilowatt-hours per month. All power supplied to the potash industry comes over 69-kilovolt powerlines. An existing 69-kilovolt line to the Kermac operation, capable of handling the added capacity required for a new mine and mill complex, could be extended into the study area.

Loving and Carlsbad are the towns nearest to the study area, located about 20 and 28 miles (32 and 45 kilometers) respectively, from its center. Total population of the two towns is about 28,000, with about 20,000 people living in Carlsbad.

Unskilled and semiskilled labor generally is available in the area. Trained miners and other skilled workers are presently employed in the

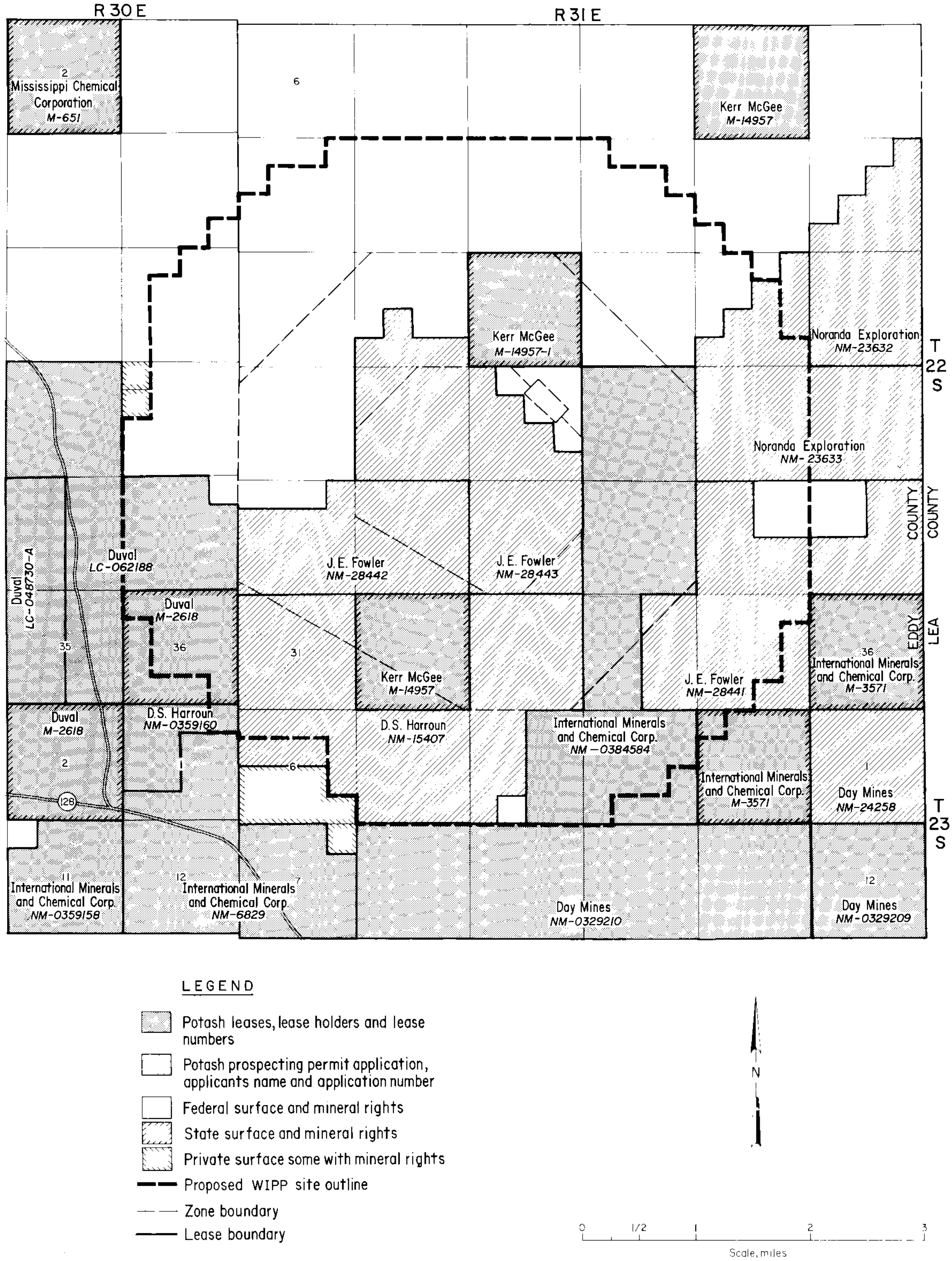

FIGURE 4.—Map of surface and mineral ownership, leases, and lease applications in the study area.

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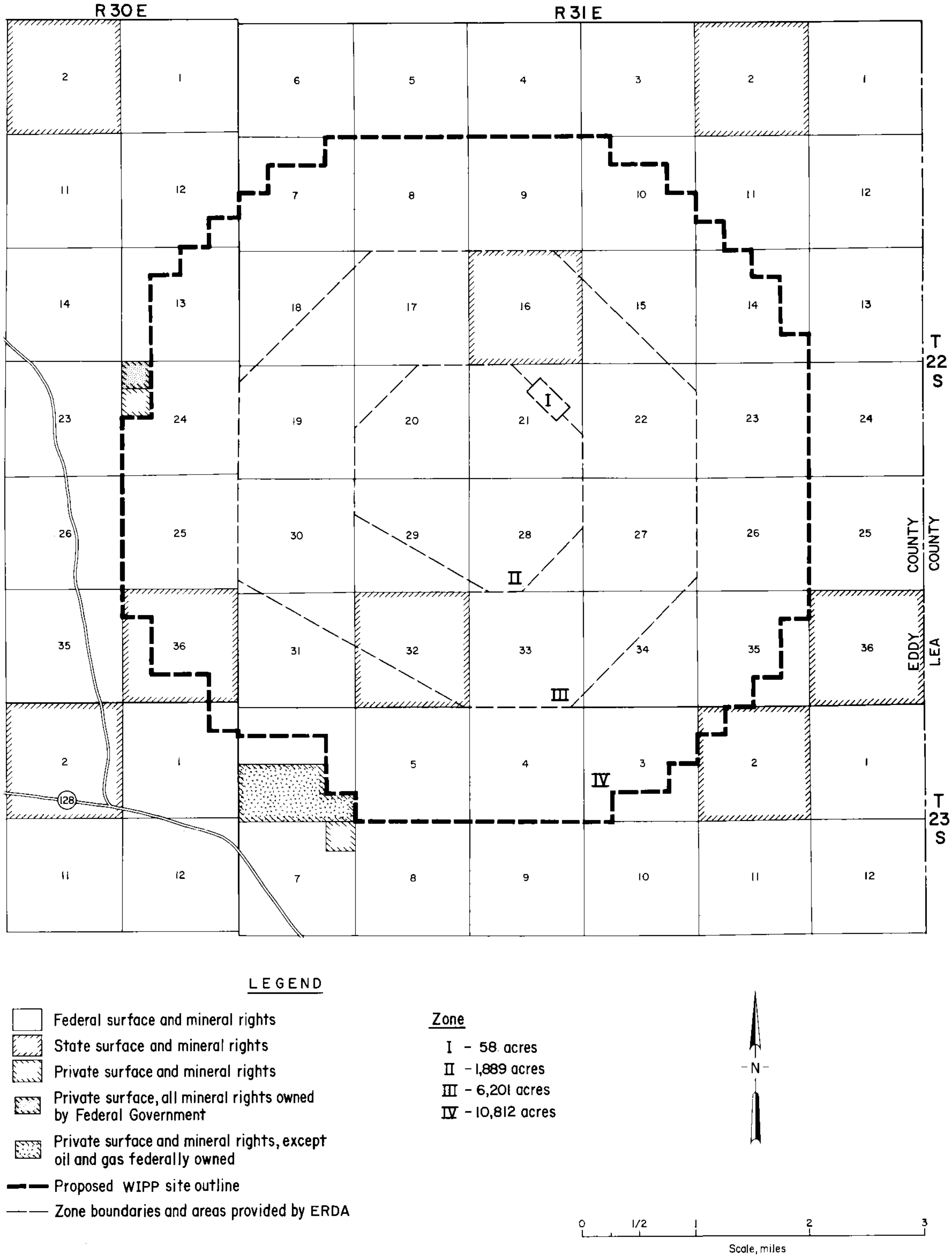

FIGURE 5.—Approximate location and size of WIPP zones within the study area.

potash industry; therefore, a new operation would require training programs for new employees.

Present Federal procedures require that archeological surveys be made along all pipelines, powerlines, access roads, and railroad rights-of-way granted by the Federal Government; also, all construction sites and pond sites must have archeological surveys. Once surveys have been conducted, salvage operations of the important sites must be conducted by a certified archeologist. These archeological surveys and excavations will add to our knowledge of the cultural heritage of the area.

GEOLOGY OF THE WIPP AREA

REGIONAL GEOLOGY

The WIPP site is near Carlsbad, N. Mex., in the western half of two adjoining structural basins of Permian age. The two basins, the Delaware Basin on the west and the Midland Basin to the east, together comprise the much larger regional Permian Basin (*21*). The Delaware Basin occupies an area in southeastern New Mexico and west Texas roughly 135 miles (roughly 217 kilometers) long by 75 miles (121 kilometers) wide (fig. 6). The basin is nearly surrounded by the large horseshoe-shaped Capitan Limestone that opens to the south. This reef grew on shallow platforms to the east and west and on the shelf area north of the Delaware Basin. Growth of the reef probably contributed to formation of the embayment in which potash salts were deposited. Capitan Limestone extends in subsurface eastward from Carlsbad to Hobbs and thence southeastward along the Central Basin Platform into Texas. To the south and west of Carlsbad, the reef is exposed as El Capitan Peak and forms a portion of the Guadalupe Mountains.

Before Permian time the basin area was submerged, and a thick section of sedimentary rocks was deposited. Toward the end of Permian time, the reef growth was halted due to an influx of high-salinity seawater. The evaporite-bearing Castile, Salado, and Rustler Formations were then deposited on top of several thousand feet of Lower Paleozoic sedimentary rocks (fig. 7). The evaporite sequence was followed by the deposition of terrestrial sedimentary rocks known as the Dewey Lake Redbeds. Later terrestrial sand, clay, and sandstone were deposited, and in part eroded, into Quaternary time. Today, much of the land surface is covered by caliche and low-lying sane dunes (*1*).

Several economic minerals are found in the area. Significant gas and oil are produced from the Pennsylvanian age Strawn, Atoka, and Morrow Formations. Less important hydrocarbon production is derived from scattered reservoirs in the Middle Paleozoic section in the Delaware Basin. Some sulfur has been produced from anhydrite caprock material in the Rustler and Salado Formations in Culbertson County, Tex., and potash minerals are mined from several mineral-bearing zones in the Salado Formation.

LOCAL GEOLOGY

Geologic information on the WIPP area has been compiled largely by Charles Jones (*10–12*) and other personnel of the U.S. Geological Survey (*2, 4, 8*). Lower and Middle Paleozoic rocks do not play an important role in the WIPP site geology and are not pertinent to this report; only the Permian and younger rocks are briefly discussed for this economic analysis.

Thick sections of salt at the top of the Salado Formation and anhydrite within the Rustler Formation have been dissolved and removed by ground water action, resulting in the creation of many sinkholes and a general lowering of the land surface over much of southeastern New Mexico. The most pronounced topographic feature near the WIPP area is Nash Draw, a depression that contains several sink areas within its boundaries (*25*).The most pronounced depression within Nash Draw is a salt lake called Laguna Grande de la Sal. The Salado Formation does not crop out in the area.

The Rustler Formation crops out in several places west and northwest of, but not within, the WIPP area. The Dewey Lake Redbeds are exposed in Nash Draw and along the western perimeter of the area. Younger rocks in the area include alluvial bolson deposits and windblown sand (dunes).

Structural deformation in the Permian rocks is limited to a gently eastward-dipping monocline with some minor flexures as illustrated in cross sections (figs. 8–12). A few collapse structures can be found in the evaporite sections. Many pre-Permian structures are reflected in the overlying beds. Such a feature is a localized structural trough opening southeastward from the southeast corner of Sec 8, T 22 S, R 31 E. This trough shows up in the 11th and lower ore zones and extends for more than 4 miles (6 kilometers). The cause of such a structure is unknown, but it is hypothesized that the trough could be the expression of an ancient drainageway or collapse feature. The trough appears in east-west cross section B–B' through the north-

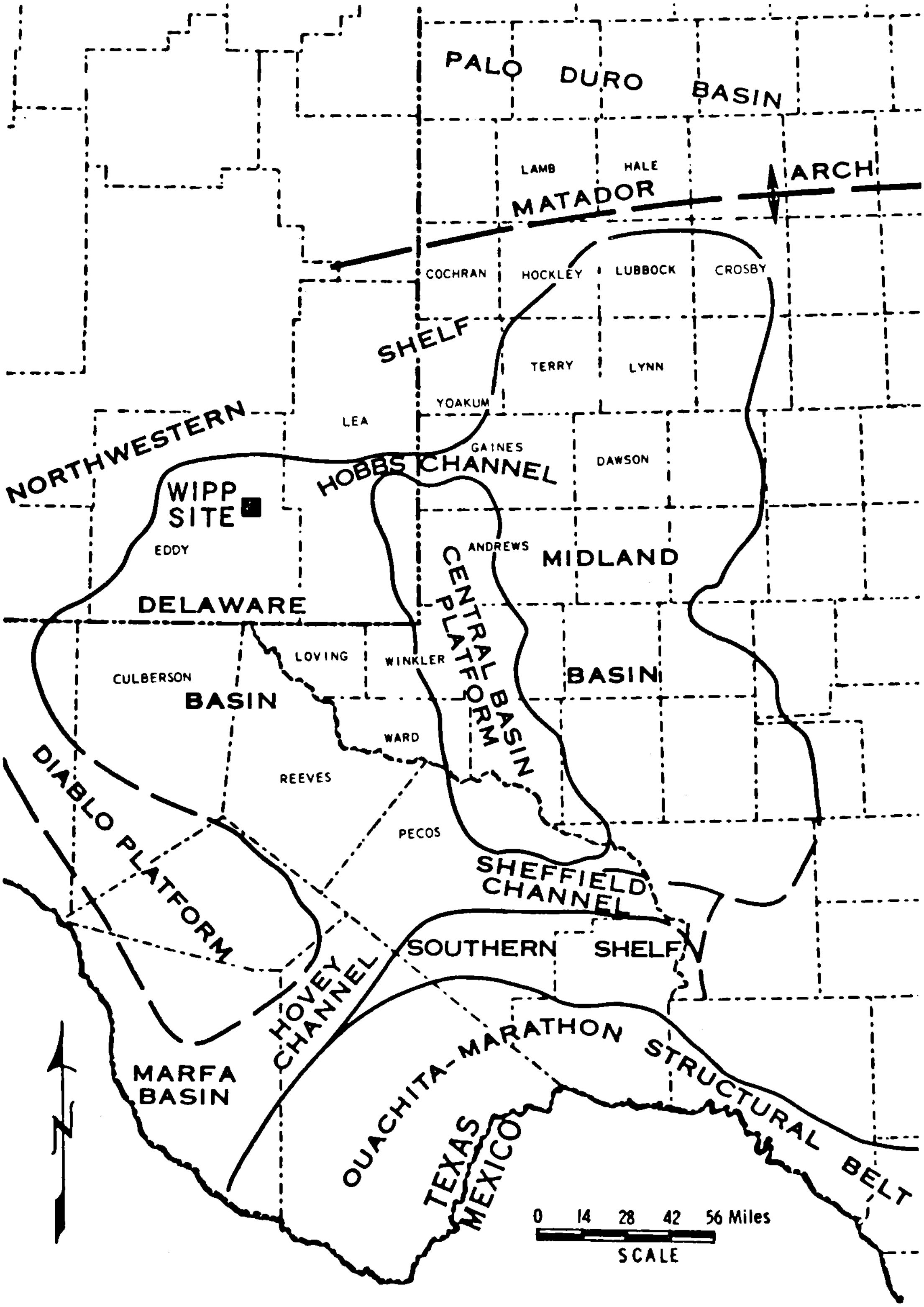

FIGURE 6.—Major Permian geologic features in the region. Source: (*22*).

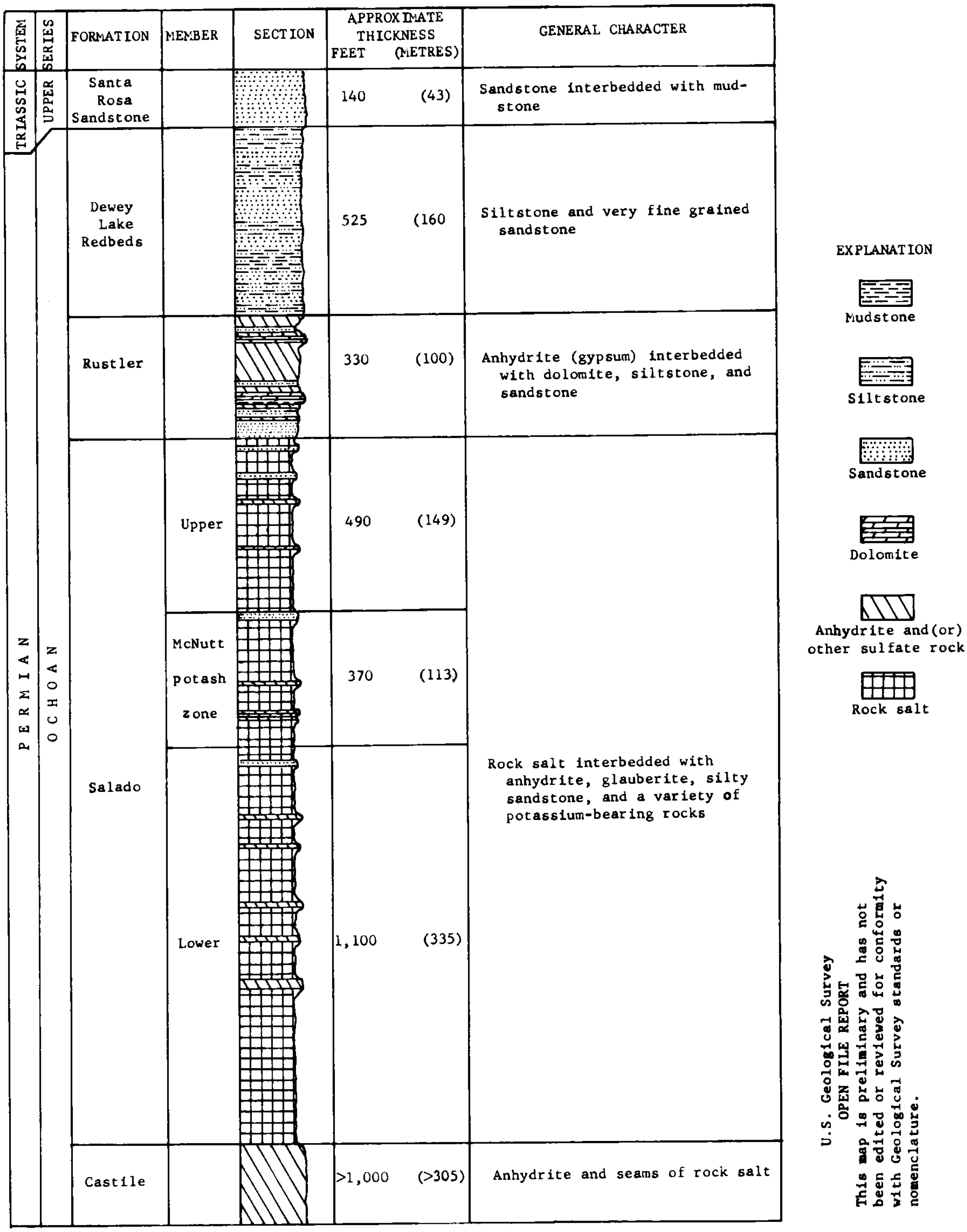

FIGURE 7.—Stratigraphic column of consolidated rocks penetrated by site evaluation borings sunk in Los Medanos area.

U.S. Geological Survey
OPEN FILE REPORT
This map is preliminary and has not been edited or reviewed for conformity with Geological Survey standards or nomenclature.

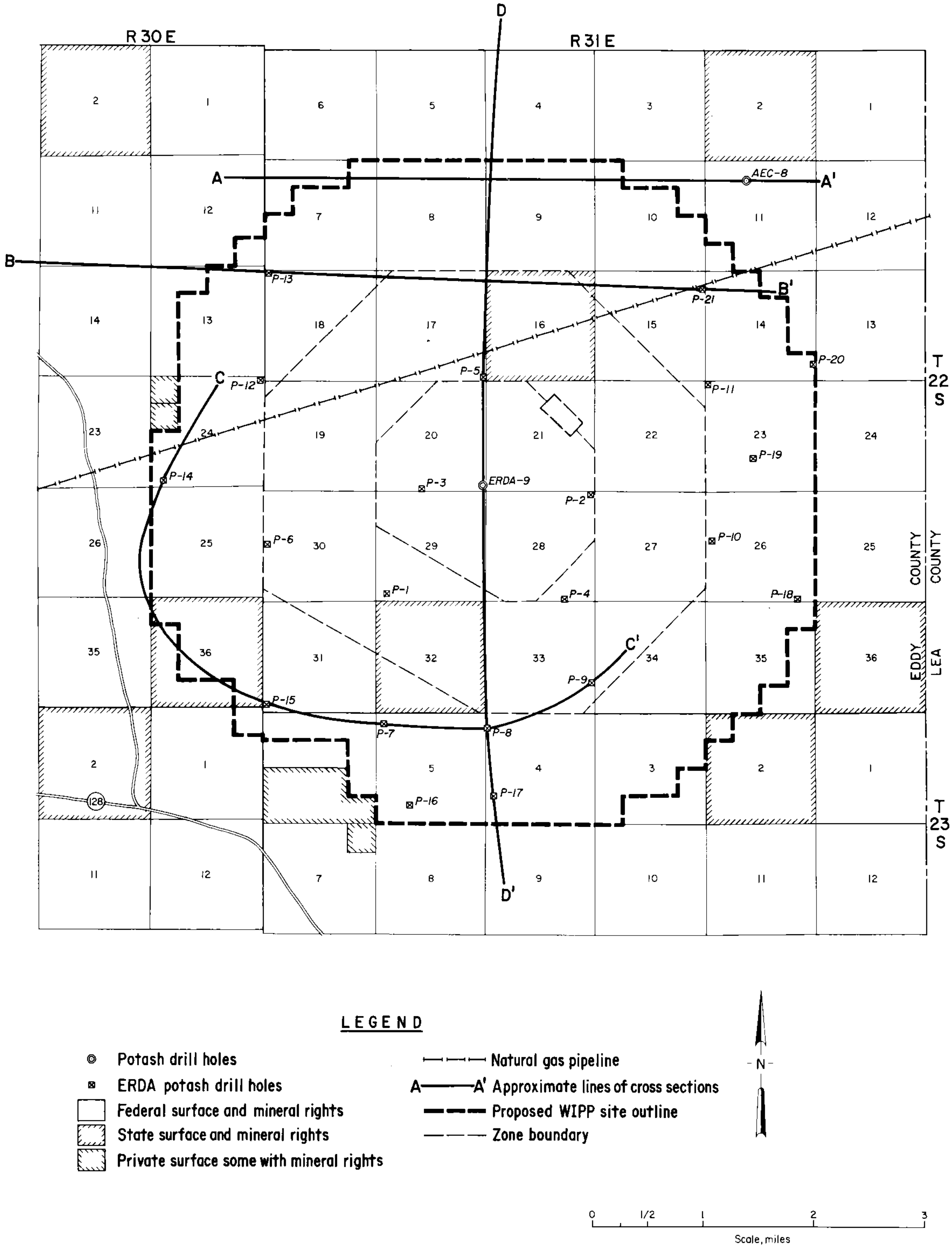

FIGURE 8.—Approximate location of cross sections at the WIPP site.

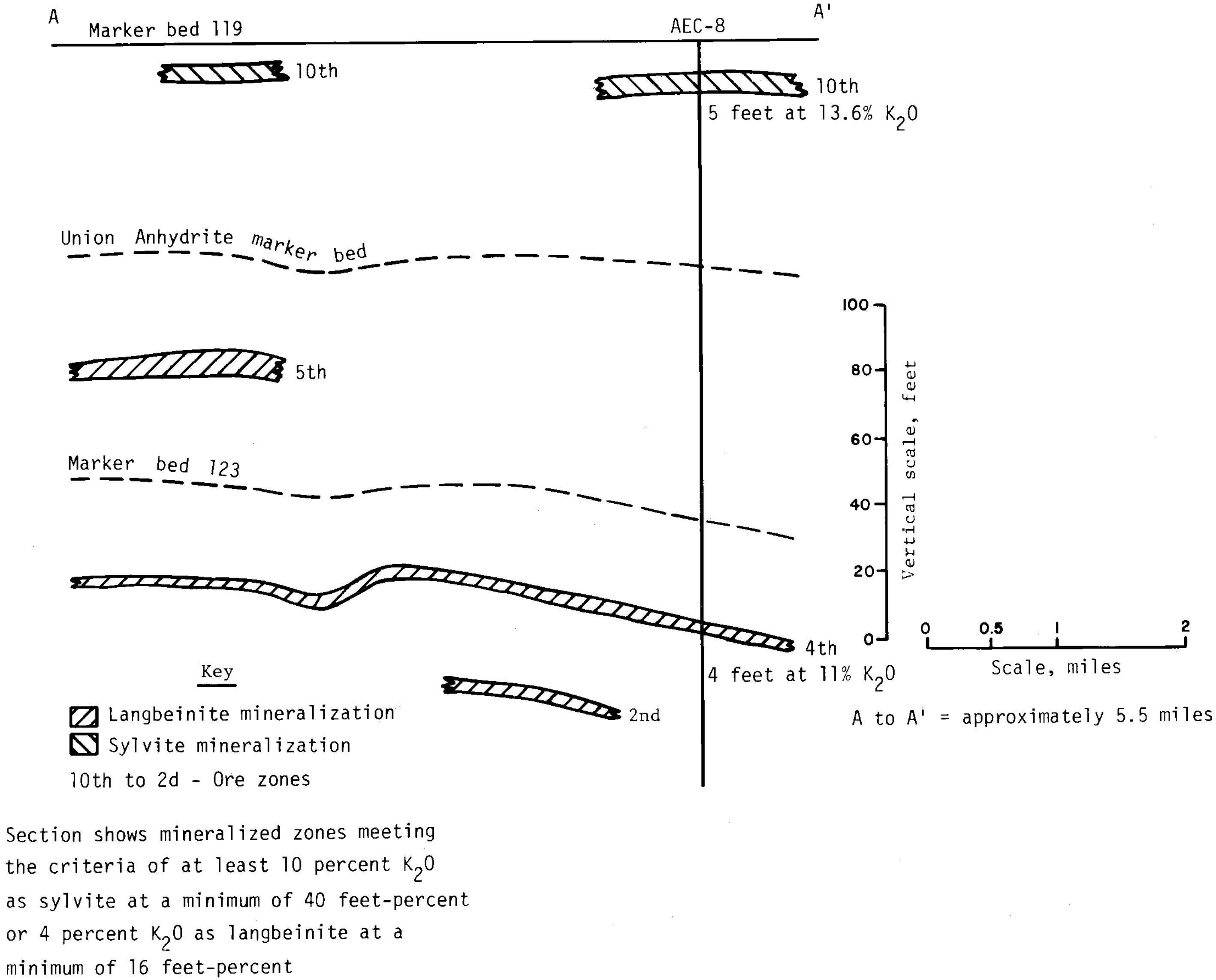

FIGURE 9.—Approximate cross section through A–A'.

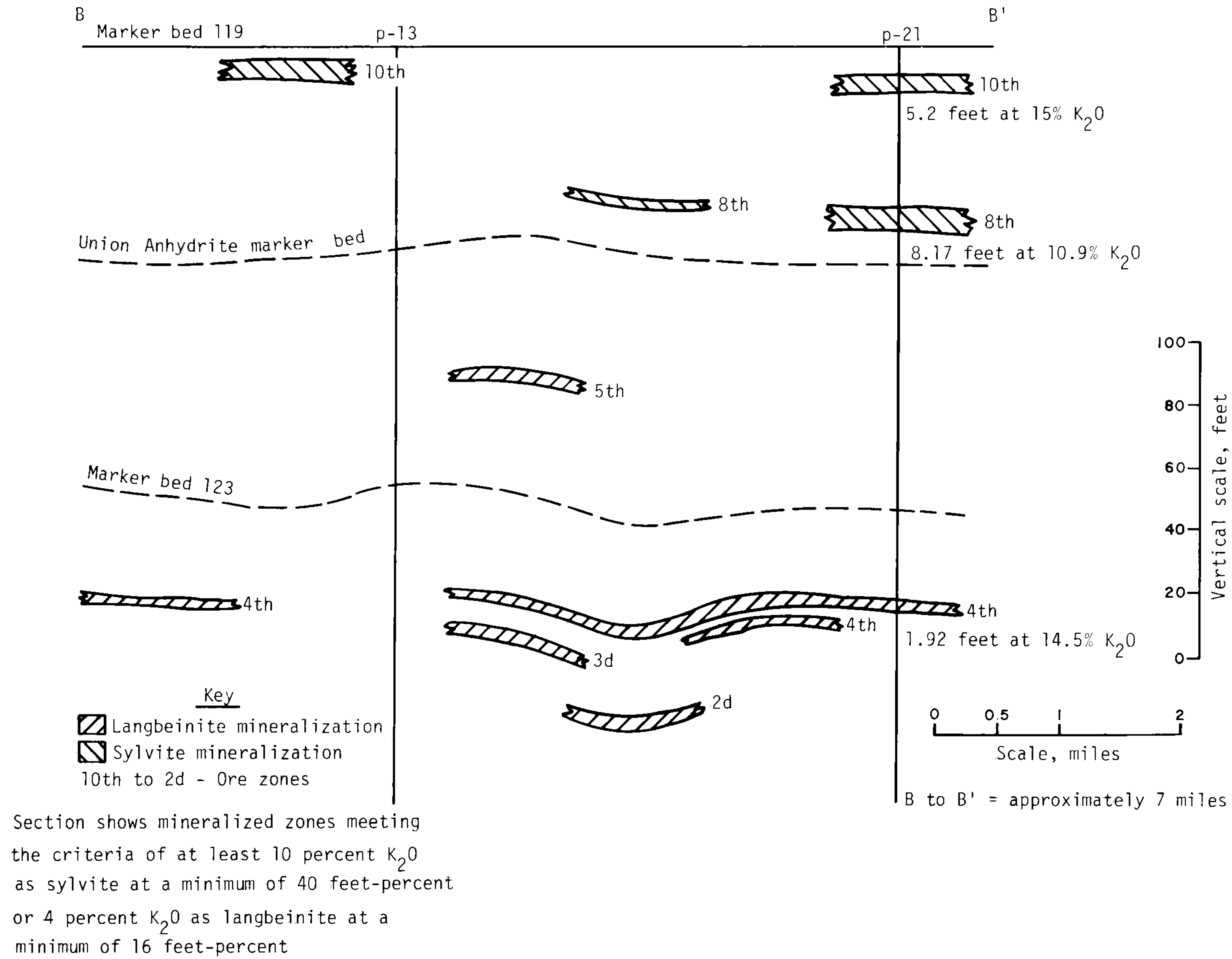

FIGURE 10.—Approximate cross section through B–B′.

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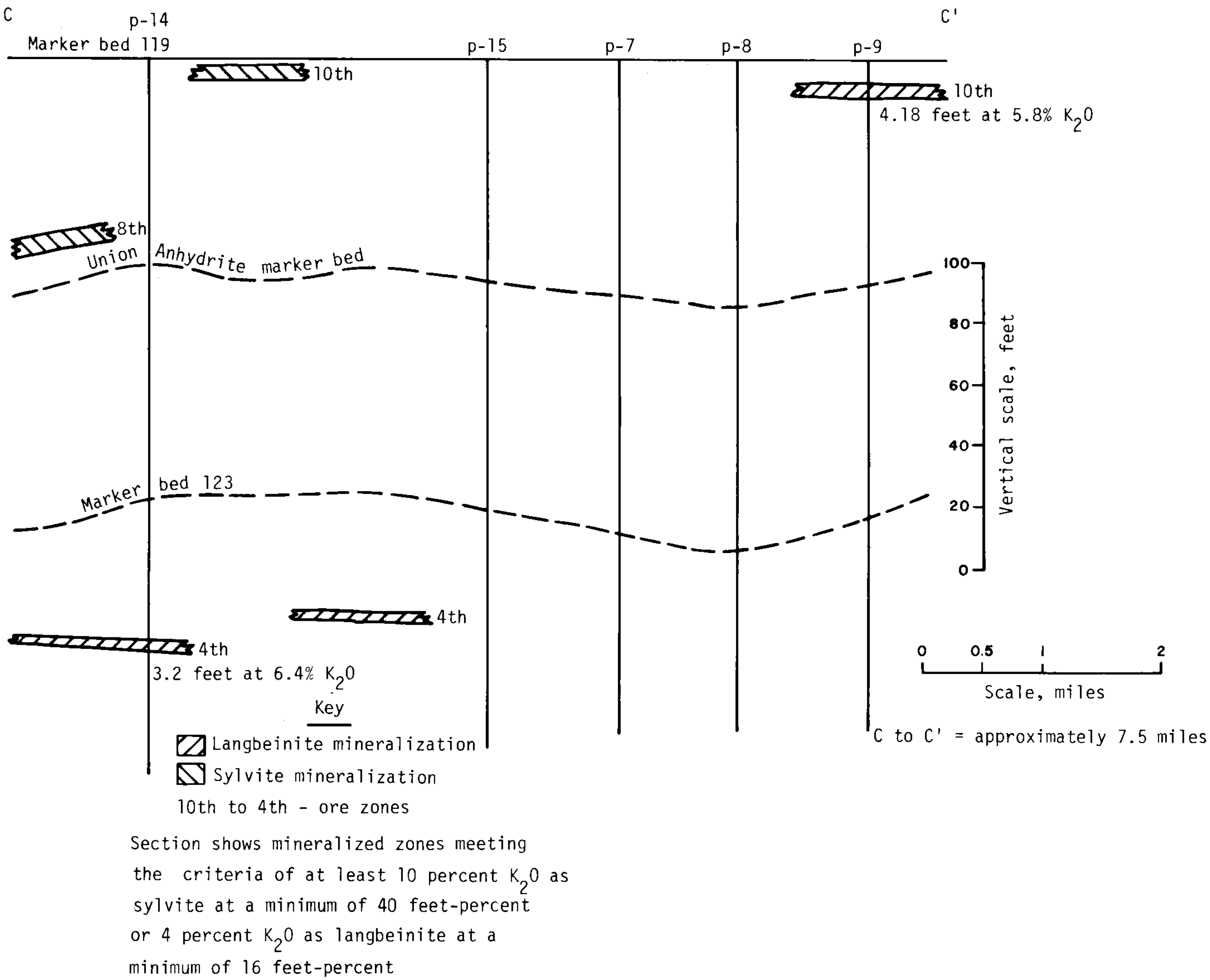


FIGURE 11.—Approximate cross section through C–C′.

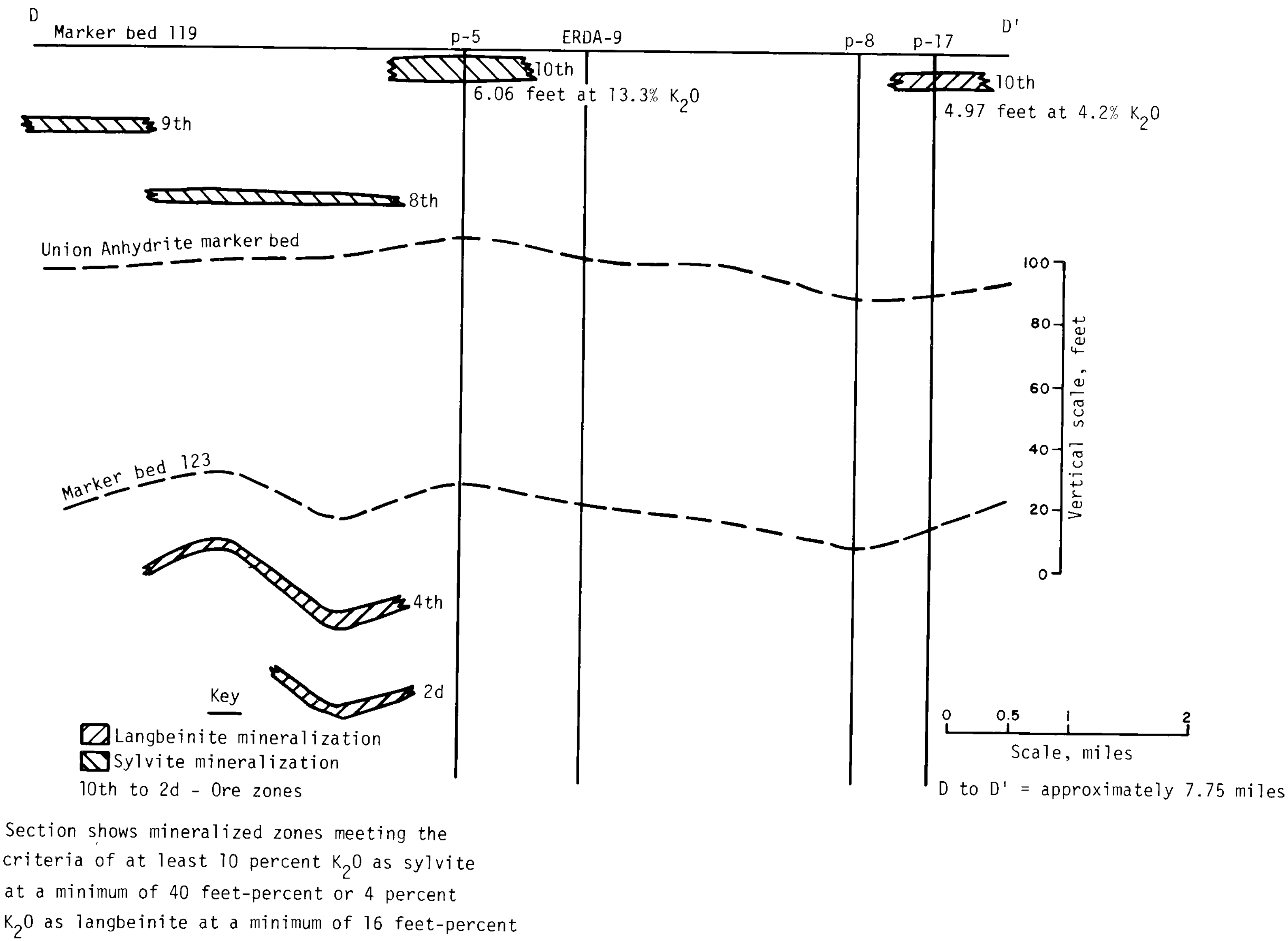


FIGURE 12.—Approximate cross section through D–D′.

ern area. Structural features related to a doming effect from the possible hydration of gypsum are found several places outside the area. There are no known active faults within the WIPP site or study area. Rocks penetrated by drilling in the study area are from Late Permian to Late Triassic in age.

STRATIGRAPHY

The Upper Permian System in the WIPP area includes the Ochoan Series, which consists of four rock salt, anhydrite, and siltstone units; the Castile, Salado, and Rustler Formations; and the Dewey Lake Redbeds. The Salado Formation contains all the potash mineralization of economic importance (fig. 13, in pocket). The Ochoan Series overlies shaly siltstone and associated limestone of the Guadalupian Series (Lower and Upper Permian) and is unconformably overlain by rocks ranging in age from Late Triassic to Middle Pleistocene.

The Castile Formation is the deepest and oldest stratigraphic unit penetrated by test hole drilling in the WIPP site. In the ERDA No. 9 borehole at the center of the area (fig. 8), the top of the formation was penetrated at a depth of 2,836 feet (864.4 meters). Thickness of the formation is estimated to be between 1,280 and 1,740 feet (390 and 530 meters). Anhydrite is the main constituent along with a few interbeds of rock salt and a small amount of calcitic limestone. The upper contact of the Castile Formation is conformable with the overlying Salado Formation where there is a lateral and vertical gradation from anhydrite to rock salt.

The top of the Salado Formation in test hole ERDA No. 9 is at a depth of 860 feet (262 meters). No solution residue is at the top of the bed, indicating that it has not been affected by ground water as it has in other areas to the west. The formation thickness at this location is 1,976 feet (602.3 meters).

The Salado Formation is divided into three members: lower; middle or McNutt potash zone; and upper. The members are similar in lithology but differ in potash mineral content. As it exists at test hole ERDA No. 9, the top of the lower member is at a depth of 1,741 feet (530.7 meters), and the member is 1,095 feet (333.8 meters) thick. Information available shows that this member consists mainly of rock salt with minor amounts of anhydrite, polyhalite, and glauberite.

The middle member, the McNutt potash zone, contains the currently economic potash minerals, langbeinite and sylvite. Mineralization in commercial concentrations is not present in test hole ERDA No. 9. The top of the McNutt potash zone is at a depth of 1,362 feet (415.1 meters), and the zone is 379 feet (115.5 meters) thick. It is comprised of rock salt with interbedded polyhalite, minor anhydrite, and clay. The normal potash-bearing zone, barren at this point, is identified by a marker bed of anhydrite near the base and by a thin seam of silty sandstone near the top.

The upper member is 520 feet (158 meters) thick; the top is at a depth of 860 feet (262 meters). This member is mainly rock salt with a few interbeds of polyhalite, anhydrite, and brown sandstone. The upper boundary of the Salado Formation is a sharp but conformable contact between rock salt and siltstone of the overlying Rustler Formation.

The top of the Rustler Formation is 550 feet (168 meters) below the surface, and the formation is 310 feet (94.5 meters) thick. The unit is composed of interbedded anhydrite, dolomite, salt, and fine-grained sandstone (fig. 13). The permeability of four beds in the Rustler Formation indicates that they may function as aquifers. These zones are a 24-foot (7.3-meter) dolomite section located at a depth of 608 feet (185 meters), a 15-foot (4.6-meter) anhydrite bed at a depth of 680 feet (207 meters), a 25-foot (7.6-meter) section of Culebra Dolomite at a depth of 714 feet (218 meters), and a 12-foot (3.7-meter) thickness of clay and minor silt 758 feet (231 meters) below the surface.

A distinct reddish-brown mudstone marks the sharp, unconformable contact of the Rustler Formation with the overlying Dewey Lake Redbeds. The Dewey Lake Redbeds comprise a red-colored silty unit located 63 feet (19 meters) below the surface. It is 487 feet (148 meters) thick and is present throughout the WIPP site. The Dewey Lake Redbeds have been eroded in post-Permian time and vary greatly in thickness; in the WIPP site these beds occur as a repeated sequence of siltstone and very fine-grained sandstone.

The Triassic Santa Rosa Formation, consisting of interbedded sandstone and mudstone, unconformably overlies the Dewey Lake Redbeds. Nine feet (3 meters) of medium-grained, friable sandstone is present. The top of the formation is at a depth of 54 feet (16 meters).

Overlying the Santa Rosa Formation is the Gatuna Formation, which consists of 27 feet (8.2 meters) of silty, calcitic sandstone.

The Gatuna Formation is overlain by 5 feet (1.5 meters) of Pleistocene Mescalero Formation caliche, which is covered by dune sand. The lateral extent of the Gatuna and Mescalero Formations is uncertain in the WIPP area because of erosion and mantling by dune sands.

DATA COLLECTION AND INTERPRETATION

The U.S. Geological Survey recommended, as one of several sites suitable for a waste storage location, the use of thick salt beds in the lower member of the Salado Formation (fig. 13). Exploratory holes were drilled to confirm the lithology, structure, mineralogy of the salt-bed sequence, and grades and amounts of potash minerals in the site area. For this purpose, 21 holes were drilled on about 1-mile (about 1.6-kilometer) centers, and the cores were analyzed. Conventional rotary drilling methods were used to penetrate to the top of the McNutt potash zone. When this horizon was reached, the drilling fluid was changed to saturated brine to prevent dissolution of the minerals and to allow for good core recovery. The interval was cored through the McNutt potash zone into the underlying lower member of the Salado Formation. The cores were visually logged by a geologist and splits were sent to laboratories for chemical analyses. The analytical data were used for preliminary estimates of tonnage and grade of potash occurring in the WIPP site.

ORE ZONES AND MARKER BEDS IN THE SALADO FORMATION

During the late 1920's, the U.S. Geological Survey and the Bureau of Mines drilled the Carlsbad potash area. At that time, there was no geologic type section or nomenclature to delineate stratigraphic units or mineralized zones.

As the drill samples were analyzed, chemists recognized various stratigraphic intervals containing high K_2O (potash) values. The potash usually occurred in argillaceous salt beds separated by calcium sulfate beds composed of anhydrite and/or polyhalite and other barren rock salt beds. About 40 individual beds within the Salado Formation that contained some percentage of K_2O were numbered from the top of the formation downward in an effort to trace stratigraphic units and mineralized zones. Each potash company working in the area used a different type of nomenclature varying from letters to numbers or combinations of both to identify beds and mineralized zones.

In the late 1950's, the U.S. Geological Survey devised a system for identifying the stratigraphic units in the Carlsbad district that segregated the potash-bearing rock units into economic and noneconomic types. The calcium sulfate rocks carried no economic values, whereas the mineralized chloride or rock salt beds did. A system of marker beds was devised to number and identify the noneconomic calcium sulfate beds within the Salado Formation, beginning with marker bed 100 near the top and ending with marker bed 143 near the base.

The salt beds of economic importance were found to occur in the middle member of the Salado Formation now known as the McNutt potash zone. Eleven mineralized salt zones were identified which were numbered from ore zone number 1 near the base of the member to ore zone number 11 near the top (fig. 13). A 1960 open-file report by Jones, Bowles, and Bell proposed these new numbering systems for potash well logging and stratigraphic correlation purposes (*13*). The method was accepted by industry and is currently being used.

POTASSIUM-BEARING DEPOSIT ESTIMATION

A triangular method to calculate potash tonnages, using standards of a minimum bed thickness of 4 feet (1.2 meters) of 4 percent K_2O as langbeinite [4 feet (1.2 meters) × 4 percent = 16 feet-percent) (4.8-meter-percent)], or 4 feet (1.2 meters) of 10 percent K_2O as sylvite [4 feet (1.2 meters) × 4 percent = 40 feet-percent (12 meter-percent)], or the equivalent minimum feet-percentage product if thickness was less than 4 feet (1.2 meters). In other words, the thickness could be reduced below 4 feet (1.2 meters) if the grade were high enough to meet the above minimum feet-percentage criteria. Triangles drawn to connect adjacent test holes peripheral to the drilling network were projected outward for distances considered prudent based on geologic interpretation. The rule of linear distribution was used to determine cutoff points between holes having insufficient thickness and/or grade and holes having mineral concentrations at or above cutoff grade. In a few instances, a circle of influence with a ½-mile (0.8-kilometer) radius was established around an outlying hole of significant grade when no other holes existed within a distance of 1-½ to 2 miles (2.4 to 3.2 kilometers).

The areas thus established were weight-averaged by mineralized bed thickness and grade to determine an average grade-thickness value. The areas were then measured by polar planimeter, and the results multiplied by the average bed thickness to determine volumes that were converted to short tons. The average thickness and grade within each triangle were multiplied by the area of the triangle, and these numbers were divided by the sum of the areas to achieve a weight-averaged grade and thickness. These data were then used to determine recoverable tonnage based on data supplied by the U.S. Geological Survey. The weight-averaged grade and thickness, tonnages, values, or zone elevations, and potash-bearing areas were plotted on base maps. This method of ore reserve estimation is suitable for uniformly layered sedimentary deposits.

EVALUATION OF POTASH MINERALIZATION

The following is a discussion of current flotation and leaching technology, the expected effect of impurities, and a discussion of the tests made by the Bureau of Mines.

CURRENT PROCESSING OF POTASSIUM-BEARING ORES

Currently, ores containing sylvite (KCl), langbeinite ($K_2SO_4 \cdot 2MgSO_4$), and mixed-sylvite-and-langbeinite minerals are being mined and processed in the Carlsbad area. The ores are upgraded by heavy media, flotation, leaching, and crystallization techniques that separate the desired potash minerals from halite, clays, slimes, and other mineral impurities.

Sylvite

Sylvite ores can be beneficiated by either flotation or solution-crystallization techniques. Both methods are being employed in the Carlsbad area. In a flotation process, sylvite ores containing between 13 and 23 percent K_2O equivalent are crushed, deslimed, and floated in a series of pneumatic flotation cells after conditioning with selective collectors and depressive reagents. Figure 14 is a generalized schematic diagram of this process.

The conditioners often include an amine collector, which makes the potassium chloride hydrophobic; a blinder which depresses slime flotation; and an alcohol which acts as a frothing agent. The floated sylvite mineral is then dried, sized, and stored for market consumption under the product name muriate of potash (60 percent K_2O). Fines separated in the sizing operation are either compacted and sold or are used as feed to a potassium sulfate operation that will be described later. Brine and tailings from the flotation operation are separated, and tailings are discarded. The brine may then be directly returned to the wash desliming circuit, or it may first go to a crystallization circuit where more of the dissolved sylvite is removed. Typical KCl recoveries of 80 to 87 percent are achieved with such a sylvite flotation process.

When the clay and slimes impurities in sylvite ores increase to the range of more than 3.5 to 4 percent, extensive mechanical desliming is required, or the sylvite flotation recovery decreases significantly. To avoid these difficulties and other liberation problems, one company is currently processing high-clay sylvite ore by solution and crystallization methods. A schematic diagram of this process is shown in figure 15. The sylvite in the crushed and mechanically deslimed ore is leached in hot [185° to 200° F (85° to 93.3° C)] unsaturated brine. The sylvite product, muriate, is crystallized from the pregnant leach liquor by vacuum cooling, then dried and stored for market. Typical KCl recoveries of 80 to 85 percent are achieved.

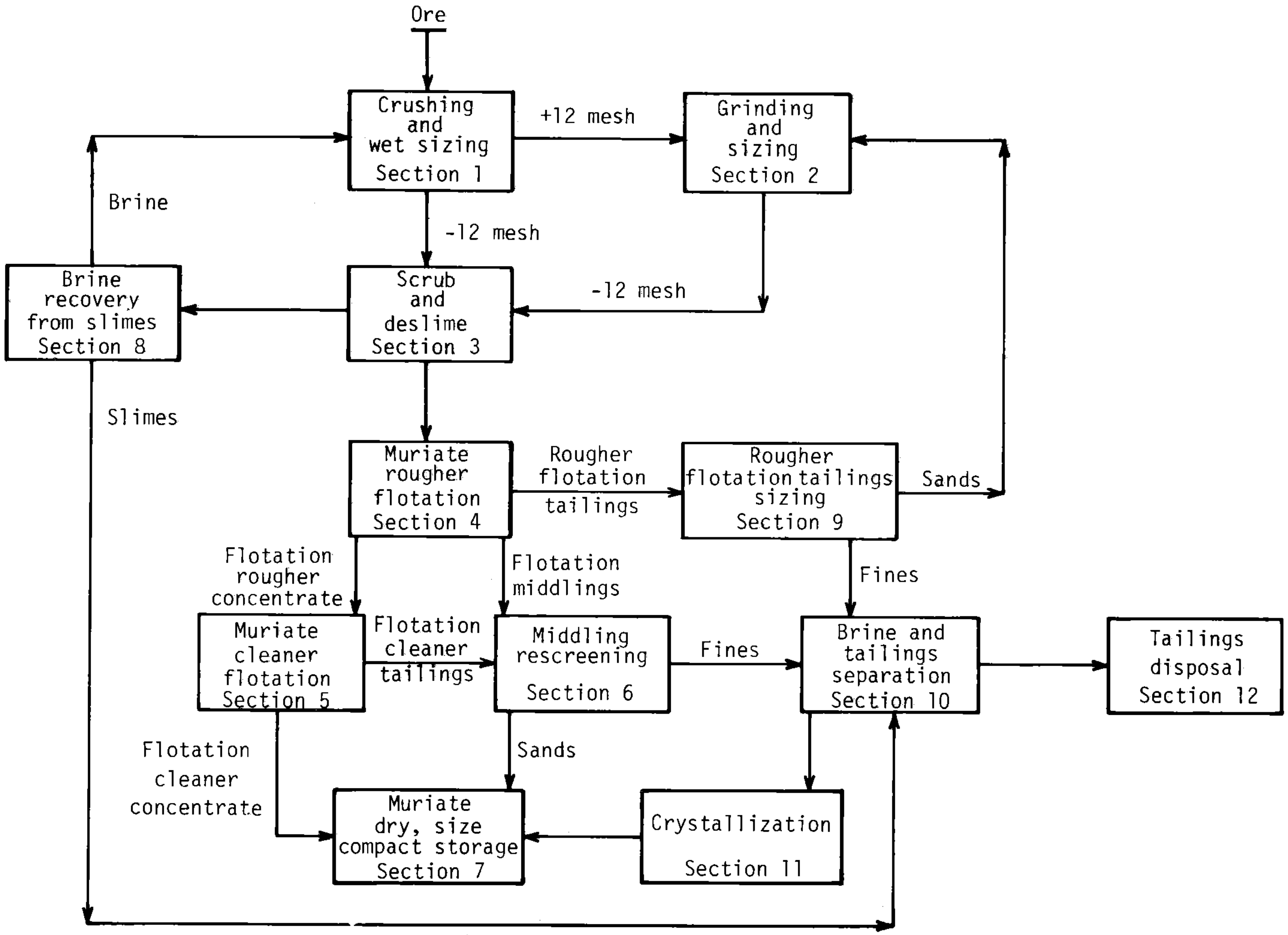
Langbeinite

Figures 16 and 17 illustrate a generalized beneficiation method for langbeinite ores. These ores, typically containing 7.5 to 10 percent K_2O, are crushed and sized. The impurities are then water-leached from the less soluble langbeinite. Then, the langbeinite is dried, sized, and stored for market consumption under some form of the name sulfate of potash-magnesia (22 percent K_2O). Typical langbeinite mill recoveries are 85 to 90 percent. Fines from the sizing operation are used as feed to a potassium sulfate operation.

Mixed Ore

Langbeinite may occur associated with sylvite in amounts such that both minerals can be recovered. Currently, a mixed ore of sylvite (8 to 10 percent K_2O) and langbeinite (2 to 3 percent K_2O) is being beneficiated in the Carlsbad area. A schematic diagram of this operation is shown in figure 16. After crushing, sizing, and mechanical desliming of the ore, the two potassium minerals in the coarse (plus 20 mesh) ore are separated by a two-stage, magnetite-heavy-media separation process.

The denser langbeinite is separated as the sink product from the lighter sylvite and halite minerals in the first heavy-media stage. In the second stage, the media specific gravity is readjusted and sylvite is separated as the float product from the halite. The separated sylvite joins the mill



Key
Section identifications refer
to equipment lists for
specific tonnage throughput

FIGURE 14.—Diagram of sylvite flotation section.

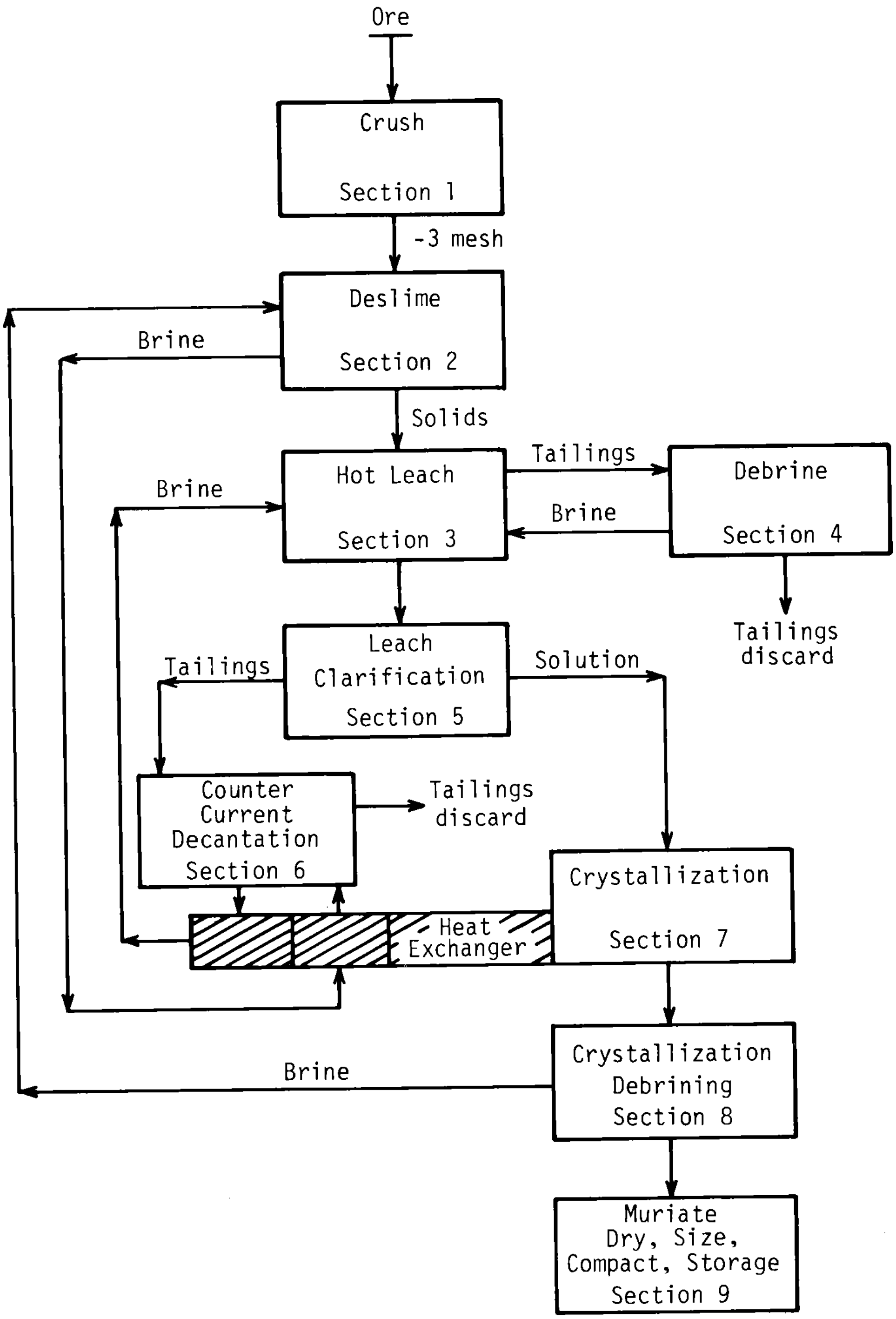


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FIGURE 15.—Diagram of sylvite solution-crystallization section.

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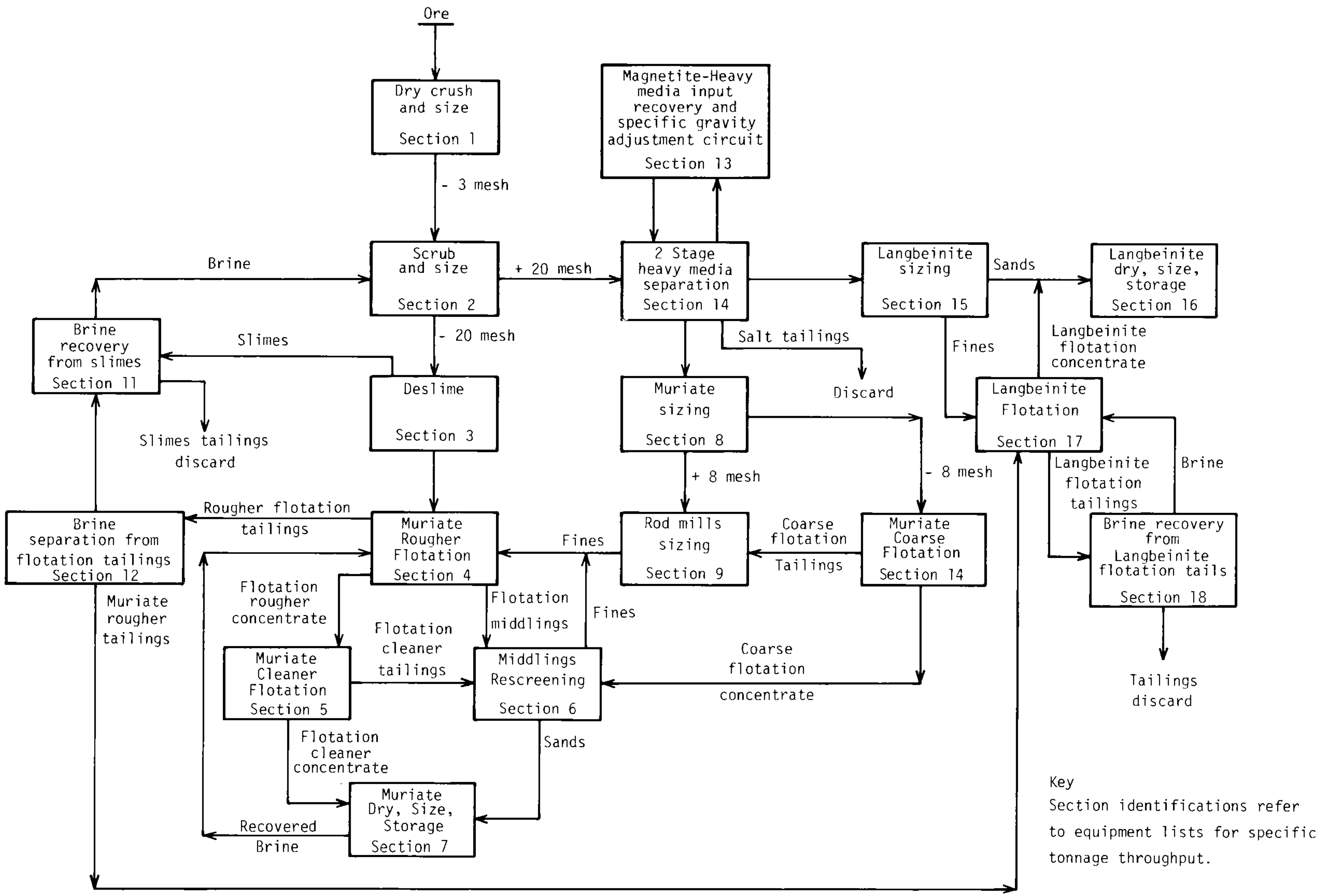
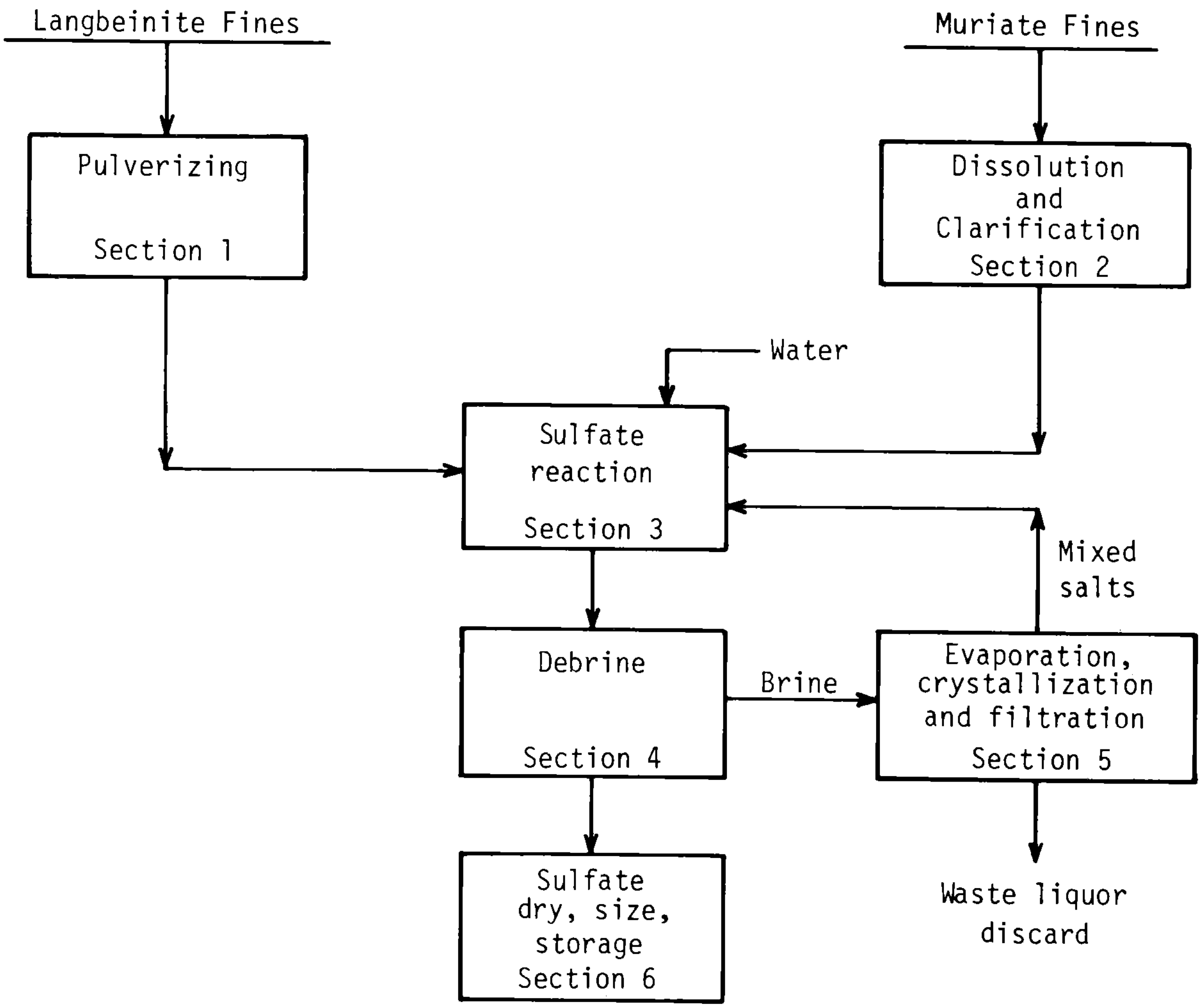


FIGURE 16.—Diagram of mixed ore section.

feed ore fines (minus 20 mesh) in a standard sylvite flotation circuit. The floated sylvite is dried, sized, compacted when necessary, and stored for market consumption as muriate of potash. The flotation tails are separated from the brine and combined with the langbeinite fines recovered from the heavy-media operation. These langbeinite fines are then floated, dried, and stored for use as feed to a sulfate operation. The langbeinite sands recovered from the heavy-media operation are dried, sized, and stored for market. Typical recoveries of 60 to 70 percent langbeinite and 70 to 80 percent sylvite are achieved.

Potassium Sulfate

Langbeinite product fines are typically combined with the necessary amount of sylvite product fines to produce the marketable product



Section identifications refer to attached equipment lists for specific tonnage throughput

Langbeinite and muriate fines are concentrate products from the langbeinite leach and sylvite flotation circuits.

FIGURE 17.—Diagram of a sulfate section.

potassium sulfate; figure 17 is a schematic diagram of this operation. In this base exchange reaction, dissolved muriate reacts with pulverized langbeinite to form soluble potassium sulfate and magnesium chloride. The potassium sulfate is then crystallized from the solution under controlled conditions as the temperature decreases. The potassium sulfate product (50 percent K_2O) is then separated from reaction liquor, dried, and stored for market. Mixed potassium salts remaining in the reaction liquor, including potassium chloride, are then recovered by vacuum crystallization and are returned to the feed input. Then the brine, which still contains the magnesium chloride and other noncrystallized salts, is discarded. Typical recoveries of potassium sulfate range between 70 and 80 percent.

Current Impurities Treatment

Other potassium-containing minerals and clays associated with the sylvite and langbeinite ores can drastically affect recovery and concentrate grades. The impurities, if not removed, can make the product not marketable, and their removal can greatly increase the capital investments needed to recover a marketable product.

The effect of clays and slimes in sylvite recovery has already been discussed as one of the major processing difficulties. Companies processing high-clay ores by flotation have found it necessary to increase desliming and brine clarification circuits and to increase the addition of flocculant and blinder reagents. Even with these modifications, as much as 7 to 8 percent greater loss of K_2O has been noted in processing some high-clay ore. New processing techniques aimed at increasing potash recoveries from high-clay ores have been studied by the Carlsbad companies and by the Bureau of Mines, but none of these studies has progressed to the commercial stage.

Sylvite ore recovery can also be affected by the presence of kieserite ($MgSO_4{\cdot}H_2O$). If the kieserite is fine grained and interlocked with the potassium chloride, it may be impossible to liberate the two minerals except at very fine sizes. This fine size will affect the product grade and the K_2O recovery that can be achieved in the flotation circuit. The presence of kieserite in a crystallization circuit may cause the formation of glaserite [$K_3Na(SO_4)_2$], which will precipitate in the lines and hamper recovery of sylvite. Maximum SO_4 concentration in ore being processed in a dissolution circuit should be 3.5 to 4 percent.

Carnallite ($KCl{\cdot}MgCl_2{\cdot}6H_2O$)impurities can also affect sylvite recovery. In the flotation processes, carnallite affects the viscosity of the brine and may cause the formation of fine-grained potassium chloride. This fine-grained KCl can be lost during flotation without careful control of the flotation circuit. One company experiencing high carnallite impurities (5 to 6 percent) has modified its circuit to include additional cleaner steps in fines flotation in order to eliminate this problem. In a crystallization circuit, the presence of carnallite can affect the Mg^{++} ion concentration in the brine leach, causing significant KCl losses. To avoid this, a preleach circuit for carnallite is necessary, followed by a recrystallization stage to recover the KCl that dissolves in the preleach.

Kainite ($MgSO_4{\cdot}KCl{\cdot}3H_2O$) is an impurity in sylvite ore that can affect the sylvite product concentrate grade. In flotation, the kainite will also float and be recovered as an impurity in the concentrate.

In langbeinite ores, polyhalite ($K_2SO_4{\cdot}MgSO_4{\cdot}2CaSO_4{\cdot}2H_2O$) is an impurity that cannot be leached from insoluble langbeinite because of similar solubility characteristics. The recovery of polyhalite in the langbeinite concentrate will thus be comparable with langbeinite recovery. Required market grade for langbeinite is 22 percent K_2O with a maximum of 4 percent impurities. It is difficult to make the required grade if polyhalite impurities are high.

Clay and slimes can also affect langbeinite ores if sufficient desliming is not included in the process. It is not unusual to expect 20 percent of clay and slimes in the feed to report to concentrate. Therefore, control must be maintained to lower this amount to meet market grade.

Theoretically, kieserite should be water leachable and thus not affect the recovery of langbeinite. Plant experience, however, seems to indicate that the kieserite does not completely leach and may report to the concentrate, thus lowering the grade.

BENCH-SCALE METALLURGICAL TESTS

As noted previously and shown in table 5, many of the potash impurities that affect langbeinite and sylvite recovery are present in the WIPP site ores. To understand the effect of the impurities on recovery, the Bureau of Mines Salt Lake City Research Center conducted preliminary sylvite flotation and langbeinite leach tests on two core samples taken from drill sites near the WIPP area. The primary purpose of these tests was to determine if any ore characteristics would preclude beneficiation by current flotation or leaching technology. The tests were designed to estimate the range of potash grade and recovery expected for the ores tested and to determine concentrate grades and impurities. No attempts were made to reclean concentrates or to include secondary ore-desliming steps in the metallurgical tests. The addition of such procedures could make the concentrates higher in grade, with a possible decrease in recoveries, but such tests are beyond the scope of this preliminary study. The tests performed, however, could indicate where such recleaning would be necessary. The results of these metallurgy tests, in conjunction with previously discussed current technology and known impurity problems, were used to estimate the expected metallurgical characteristics of the WIPP site ore.

The core samples used were AEC–7 and AEC–8; these cores were obtained from a previous drilling investigation in the area, which was reported in a U.S. Geological Survey open-file report (*10*). These core samples were used because they provided a larger amount of ore for brine makeup and metallurgical tests than would have been available by using the 2-inch ERDA drill cores from this investigation. Table 5 summarizes the chemical analyses made by the Salt Lake City Research Center on portions of the core used in the tests. The table also shows a calculated chemical analysis based on the USGS report for these same core portions. Variations between the two analyses are due to slight differences between core sample splits. Because of the close comparison between the Bureau of Mines sample analyses and those made by the U.S. Geological Survey, the mineralogy expected for the Bureau of Mines samples should be comparable with that determined in the USGS analysis. The author of the USGS open-file report (*10*) on the two core samples, C. L. Jones, also supervised the core drilling within the WIPP site for this evaluation. Discussion with C. L. Jones confirmed that the mineralogy shown in the USGS report could be assumed to be the mineralogy of the Salt Lake City test ores. Further, he indicated that the test ores were representative of ores found in the WIPP area. Table 6 shows the assumed mineralogy.

Sylvite Flotation Tests

Sylvite flotation tests were performed by the Bureau of Mines Salt Lake City Research Center on two ore samples, shown as AEC 7–5 and AEC 8–10 in tables 5 and 6. The laboratory procedure included the following steps:

1. The ore at minus 10 mesh was scrubbed for 15 minutes in a 50-percent solids saturate brine.
2. The scrubbed ore slurry was diluted to 30 percent solids and then decanted over a 150-mesh screen. This desliming operation is similar to the laboratory procedure used in the Carlsbad potash industry. The oversize (plus 150-mesh) fraction was diluted (22 to 33 percent solids) and deslimed again. The undersize (minus 150 mesh) was discarded. Commercially, the undersize would be thickened and discarded and the separated brine recycled.

TABLE 5.—Chemical analysis (percent) of core samples used in Bureau of Mines metallurgical test (approximate chemical analysis based on U.S. Geological Survey Open-File Report 75–407 included for comparison)

	AEC 7–5	AEC 8–10	AEC 8–4A	AEC 8–4B	AEC 8–4C[2]
Ore used	5th ore zone	10th ore zone	4th ore zone	4th ore zone	4th ore zone
Core	AEC–7	AEC–8	AEC–8	AEC–8	AEC–8
Depth, feet	1726.5–1741.5	1589–1594.5	1753–1756.5	1752.0–1756.5	1753.0–1756.7; 1752–1753
K_2O:					
USBM	13.4	13.1	12.6	9.50	3.70
USGS	15.75	13.4	12.8	10.74	4.56
Calcium:					
USBM	.42	.20	<.20	<.20	<.20
USGS	.42	.21	.05	.06	.07
Magnesium:					
USBM	2.10	3.40	6.60	5.27	1.92
USGS	2.80	3.70	6.36	5.00	2.20
Chlorine:					
USBM	44.70	40.60	24.38	31.90	45.60
USGS	44.92	41.56	26.41	33.53	48.30
SO_4:					
USBM	12.50	14.90	39.50	29.70	12.15
USGS	12.51	13.70	38.122	29.94	13.05
Water-insoluble matter:					
USBM	.80	4.50	.70	1.20	2.50
USGS	.81	5.53	1.104	1.18	1.33
Water loss, 60°–200° C:					
USBM	([3])	([3])	([3])	([3])	([3])
USGS	1.34	1.73	.64	.53	.299

[1] Tables used in calculation from USGS open-file report (9).
[2] Sample AEC 8–4C is a synthetic sample consisting of 1,500 grams of sample AEC 8–4A (depth 1752.7–1756.7) mixed with 3,200 grams of ore from core AEC–8, 4th ore zone (depth 1752–1753).
[3] Water loss analysis not conducted by the Bureau of Mines (USBM).

TABLE 6.—Approximate mineral content (weight-percent) of core samples used in Bureau tests. Mineralogical analysis calculated from data reported in U.S. Geological Survey Open-File Report 75–407[1]

Mineral	Chemical composition	Test AEC 7–5	Test AEC 8–10	Test AEC 8–4A	Test AEC 8–4B	Test AEC 8–4C[2]
Sylvite	KCl	21.65	20.0	0.05	0.10	0.29
Langbeinite	$K_2SO_4 \cdot 2MgSO_4$	.28	0	51.77	43.14	16.79
Kainite	$MgSO_4 \cdot KCL \cdot 3H_2O$	7.91	0	1.04	1.00	1.29
Leonite	$K_2SO_4 \cdot MgSO_4 \cdot 4H_2O$	0	0	2.31	1.93	.74
Bloedite	$Na_2SO_4MgSO_4 \cdot 4H_2O$	0	0	.34	.29	.11
Kieserite	$MgSO_4 \cdot H_2O$	10.52	18.87	0	0	0
Carnallite	$KCl \cdot MgCl_2 \cdot 6H_2O$	.18	4.20	0	.11	.33
Polyhalite	$K_2SO_4 \cdot MgSO_4 2CaSO_2 2H_2O$	3.09	.31	.21	.31	.53
Halite	$NACl$	55.11	50.2	43.23	54.88	78.91
Anhydrite	$CaSO_4$	.06	.60	.09	.07	.03
Water-insoluble matter	----	.83	5.52	1.08	1.16	1.31

[1] Tables used in calculation from USGS open-file report (9).
[2] Sample AEC 8–4C is a synthetic sample and does not represent a particular ore matrix of the area tests; it does, however, give an indication of leaching characteristics of very low langbeinite.

3. The deslimed sample is then conditioned prior to flotation. The plus 150-mesh solids were diluted to 23 percent solids, and then the reagents were added in two steps. Commercially, conditioning could be done in a tumbler at a higher percentage of solids, and such conditioning would then use less reagent than laboratory tests because of the smaller dilution and possibly greater contact.
   a. In the first stage, a blinder reagent is added to the slurry which is then conditioned for 2 minutes. The blinder keeps insoluble slimes from absorbing amine and floating. The portion of feed and reagent used are as follows:
      0.2 pound (91 grams) MRL 201[5] for tests with core AEC 7–5
      0.3 pound (136 grams) MRL 201 for tests with core AEC 8–10
   b. In the second stage, additional flotation reagents were added to the slurry, which is then conditioned for 2 minutes. Both core samples used identical amounts of the following reagents per ton of feed:
      0.2 pound (91 grams) of Armeen T. D., an amine collector
      0.1 pound (45 grams) Barretts oil 634
      0.018 pound (8.17 grams) Hexanol frother
4. The conditioned sample is floated for 2 minutes to produce a rougher concentrate.
5. The rougher concentrate is then reconditioned prior to any cleaner flotation. This additional conditioning step was found necessary in the laboratory, possibly due to dilution of the sample, or due to impurities present in the ore. Commercially, only initial conditioning (step 3) is needed. In conditioning, the rougher concentrate is diluted to 23 percent solids (for AEC 7–5 sample) and 24 percent solids (for AEC 8–10 sample), and is conditioned for 2 minutes with the addition of 0.1 pound (45 grams) of MRL 201 blinder per ton of original feed.
6. The conditioned rougher concentrate is then floated for 2.5 minutes to produce a cleaner concentrate.
7. The first cleaner concentrate is then conditioned for 2 minutes by adding the following reagents per ton of original feed:
   0.05 pound (23 grams) Armeen T. D.
   0.024 pound (11 grams) Barretts oil 634
   0.01 pound (4.5 grams) Hexanol
8. The conditioned cleaner concentrate is then floated for 1.5 minutes in a slurry of 10 percent solids. The concentrates and tailings are then analyzed. A discussion of the test results for each ore sample follows.

[5] Use of specific brands of reagents does not in any way mean an endorsement of these reagents in preference to similar reagents sold by other companies.

Sample AEC 7–5—Sylvite (USBM analysis): 13.4 weight-percent K_2O equivalent

This sylvite ore sample analysis, as shown in tables 5 and 6, contains both kainite and kieserite. In addition, the K_2O content in this sample is slightly lower than that in Carlsbad ores currently used for sylvite recovery by flotation. The kainite in the sample would be expected to float, thus lowering the recovery and concentrate grade of the sylvite. The kieserite can also affect the concentrate grade, since it is often interlocked with sylvite and cannot be liberated. In this sample, however, microscopic examination of this sample indicated that kieserite is not entrained in the sylvite, and thus the two might be separated. The flotation results are shown in table 7. A recovery of 81.1 percent of the potash value was achieved, but the K_2O concentrate grade was only 55.26 percent rather than the market grade of 60 percent. Analysis of the concentrate, as expected, did indicate the presence of kainite. Because of this kainite, which could not be mechanically separated from the ore, recovery of market grade sylvite by flotation may not be possible. Kainite may not affect a solution crystallization circuit as detrimentally, and thus sylvite may be recoverable from this material by solution-crystallization methods. Extensive solution and crystallization tests were not conducted due to the limited scope of these preliminary tests.

Sample AEC 8–10—Sylvite sample: 13.1 weight-percent K_2O equivalent

This sylvite sample, as shown in tables 5 and 6, contains kieserite and, in addition, the K_2O content is slightly lower than those currently used for sylvite flotation recovery. The kieserite can affect recovery and the concentrate grade of the sylvite since it is often interlocked with sylvite and cannot be liberated. However, microscopic analysis indicates that kieserite is not entrained with the sylvite and should not affect flotation. Sample AEC 8–10 has no kainite impurities but has a very large insoluble content. The flotation results are shown in table 8. A recovery of 73 percent of the potash value was achieved, but the K_2O concentrate grade was only 54.14 percent rather than the market grade of 60 percent. Analysis of the concentrate indicated the presence of schoenite, with occluded halite, fine halite, and insoluble slimes. It might be possible to further upgrade this ore by recleaning to meet market specifications, although such recleaning would reduce potash recovery. Sylvite from this type of ore may also be recoverable by solution and crystallization methods, where the insolubles can be more effectively removed.

Langbeinite Leach Tests

Langbeinite leach tests were performed by the Bureau of Mines Salt Lake City Research Center on three samples, shown as AEC 8–4A, AEC 8–4B, and a composite sample AEC 8–4C in tables 5 and 6. The laboratory procedures were based on similar laboratory leach methods used by the Carlsbad potash industry. The tests included the following steps:

1. The sample is separated into a plus 14-mesh and minus 14-mesh fraction.
2. The coarse material (minus 4 to plus 14 mesh) is leached with cold water in an agitator for 2 minutes. Leach water requirements were calculated to give a 12 percent chloride brine if 100 percent of contained halite dissolved during leaching (steps 2 and 4).
3. The leach liquor is decanted and filtered from the insoluble coarse product.
4. The fine material (minus 4 mesh) is leached

TABLE 7.—Chemical assay mass balance of sylvite flotation: test on sample AEC 7–5

(Weight-percent)

Product	Percent of total input tonnage	K_2O	Insolubles	Magnesium	SO_4	Sodium	Chlorine	Calcium
Feed:								
Coarse	85.9	14.06	0.23	2.02	11.90	22.41	45.58	0.38
Slime (discarded)	14.0	9.34	4.28	2.61	16.16	17.36	39.27	.66
Total feed	100.0	13.4	.80	2.1	12.5	21.7	44.7	.42
Rougher concentrate	20.5	53.01	.18	.85	5.17	2.51	42.52	.08
Cleaner concentrate	19.66	55.26	.14	.76	4.76	1.55	42.64	.07
Tailings:								
Rougher	65.5	1.87	.25	2.38	14.01	28.64	46.54	.47
Cleaner	.90	4.22	.89	2.9	14.0	23.3	39.77	.44
Total tailings	66.4	1.90	.26	2.39	14.01	28.57	46.45	.47

TABLE 8.—Chemical assay mass balance of sylvite flotation: test on sample AEC 8–10

(Weight-percent)

Product	Percent of total input tonnage	K_2O	Insolubles	Magnesium	SO_4	Sodium	Chlorine	Calcium
Feed:								
Coarse	80.9	13.38	1.06	3.06	13.23	20.40	43.94	0.12
Slime (discarded)	19.1	11.92	19.06	4.8	21.96	13.61	26.67	.10
Total feed	100.0	13.1	4.5	3.4	14.9	19.1	40.6	.12
Rougher concentrate	19.8	49.79	1.12	.88	3.70	4.75	44.77	.06
Cleaner concentrate	17.7	54.14	.98	.61	2.50	2.76	45.04	.04
Tailings:								
Rougher	61.1	1.58	1.04	3.77	16.32	25.47	43.67	.14
Cleaner	2.1	13.13	2.37	3.18	13.94	21.56	42.46	.19
Total tailings	63.3	1.96	1.08	3.74	16.22	25.30	43.56	.14

for 1 minute with the unsaturated leach liquor separated in step 3.
5. The leach liquor is then decanted and filtered from the insoluble fine product.

The leach liquor and products were analyzed. A discussion of the test results for each sample follows.

Sample AEC 8–4A—Langbeinite sample: 12.6 weight-percent K_2O equivalent

This langbeinite sample analysis, shown in tables 5 and 6, has a higher K_2O content than Carlsbad ores currently used for langbeinite leach recovery. The impurities that affect leaching, polyhalite and other insolubles, occur in small quantities. The leach results are shown in table 9. A recovery of 91 percent of the potash value was achieved, but only a 21.27 percent K_2O grade was made. This grade is close to minimum market requirements of 22 percent K_2O. Because analysis of the concentrate indicates that most of the impurities are insoluble slimes, a recleaning of the concentrate should increase the concentrate grade. In plant practice, ores similar to AEC 8–4A should be processable with recoveries in the range of 90 to 91 percent.

Sample AEC 8–4B—Langbeinite sample: 9.5 weight-percent K_2O equivalent

This langbeinite sample analysis, as shown in tables 5 and 6, has a K_2O content comparable or slightly higher in grade than Carlsbad ores currently used for langbeinite leach recovery. The impurities that affect leaching, polyhalite and insolubles, occurred in small quantities. The leach results are shown in table 10. A recovery of 89.8 percent of the potash value was achieved, but only a 21.5 percent K_2O grade was made. Because analysis of the concentrate indicated that most impurities are insoluble slimes, a recleaning of the concentrate should increase the concentrate grade. In plant practice, ores similar to AEC 8–4B should be processable with recoveries in the range of 88 to 90 percent.

Sample AEC 8–4C—Langbeinite sample: 3.7 weight-percent K_2O equivalent

This sample analysis, listed in tables 5 and 6, is a composite and does not represent a specific deposit found in the area. It does, however, give some indication of the possibility of leaching 4-percent K_2O-equivalent langbeinite. The impurities, polyhalite and insolubles, are in low percentage, but the low grade would make any

TABLE 9.—Chemical assay mass balance of langbeinite leach: test on sample AEC 8–4A

(Weight-percent)

Product	Percent of total input tonnage	K_2O	Insolubles	Magnesium	SO_4	Sodium	Chlorine
Feed:							
Coarse	61.1	13.4	0.43	6.84	42.41	14.88	22.93
Fine	38.9	11.3	1.11	6.22	34.93	16.47	26.66
Total	100.0	12.6	.70	6.60	39.50	15.50	24.38
Concentrate:							
Coarse	35.81	21.4	.21	11.1	69.0	.12	.21
Fine	18.16	21.0	.97	11.2	68.8	0	0
Total concentrate	53.97	21.27	.47	11.13	68.93	.08	.14
Brine solution	46.03	2.43	.97	1.29	4.99	33.58	52.80

TABLE 10.—Chemical assay mass balance of langbeinite leach test on sample AEC 8–4B

(Weight-percent)

Product	Percent of total input tonnage	K_2O	Insolubles	Magnesium	SO_4	Sodium	Chlorine
Feed:							
Coarse	55.0	9.7	0.82	5.39	30.48	21.05	31.29
Fine	45.0	9.25	1.65	5.11	28.74	21.16	32.64
Total	100.0	9.5	1.2	5.27	29.7	21.1	31.9
Concentrate:							
Coarse	23.24	21.5	.61	12.2	69.3	0	0
Fine	16.42	21.5	1.2	11.9	69.7	0	0
Total concentrate	39.66	21.5	.85	12.08	69.47	0	0
Brine solution	60.34	1.61	1.43	.79	3.57	34.97	52.87

impurities in the concentrate detrimental to making market grade. The leach test results are shown in table 11. A recovery of 87 percent of the potash value was achieved, but only a 19.48 percent K_2O grade was achieved. An analysis of the concentrate indicated the presence of impurities; however, more than 80 percent of the impurities were already leached in the test. It is doubtful if deposits of this grade could be beneficiated unless it contained virtually no polyhalite or insolubles as impurities or unless these impurities were concentrated in certain size fractions that could be discarded.

Summary of Metallurgical Tests

The tests conducted by the Bureau of Mines Salt Lake City Research Center confirmed the problems expected in ores found in the WIPP area. The WIPP ore types seem to behave similarly to ores currently processed in the Carlsbad area. High percentages of polyhalite and insolubles can lower the concentrate grade of langbeinite ores. If the langbeinite ore has a very low K_2O content, impurities may be large enough to require recleaning processes, or the impurities may not be removable, resulting in an unmarketable product. High impurities of kainite, kieserite, and insolubles in sylvite ores can affect the K_2O recovery and grade of concentrate and, in the extreme, can make it difficult to reclean concentrates to meet market specifications.

MINERALIZATION IN THE WIPP SITE FROM U.S. GEOLOGICAL SURVEY DATA

The principal ore zones in the study area are the 10th and 4th; subordinate ore zones in the area include the 8th, 3d, and 2d. The ore zones dip gently to the east-southeast at approximately 100 feet per mile (about 19 meters per kilometer).

The mineralized zones comprise lenticular bodies lying between anhydrite and/or thick salt beds. Normally, seams of clay and salt occur at the top and bottom of these lenses. In several places, however, potash mineralization occurs above or below the designated ore zone, and experience dictates that these occurrences are localized and not laterally persistent. This situation may exist within the WIPP area; where these spotty occurrences are present, they have not been included in the ore-reserve estimate (figs. 18, 19, and 20).

The designated ore zones occur as laterally persistent beds of halite and argillaceous halite that locally contain complex potash minerals in various concentrations. The zones are persistent, but commercial grades within the zones occur at irregular intervals. Thus, these persistent beds which are, by usage, called ore zones occur over large areas, including areas where potash mineralization is not present in commercial concentrations.

The occurrence of potash minerals in the ore zones is typically massive, coarsely crystalline, fairly even-grained, and mineralogically complex. The mineralized bodies contain halite and one or more potassium minerals as major constituents, at least one magnesium mineral, and small amounts of clay, silt, polyhalite, and/or anhydrite. An example of K_2O occurrence is illustrated in a list of minerals and grade by ore zone for test hole AEC–8 in table 12.

Grades and tonnages of potash mineralization supplied by the U.S. Geological Survey were divided into three basic grade categories, which include both measured and indicated amounts. The criteria for the three categories were (1) highest grade, a minimum thickness of 4 feet with a minimum grade of 8 percent K_2O as langbeinite, 14 percent K_2O as sylvite, or mixed ore of comparable grade; (2) medium grade, a minimum thickness of 4 feet (1.2 meters) with a minimum grade of 4 percent K_2O as langbeinite, 10 percent K_2O as sylvite, or mixed ore of comparable grade; (3) lowest grade, a minimum thickness of 4 feet (1.2 meters)with a minimum grade of 3 percent K_2O as langbeinite, 8 percent

TABLE 11.—Chemical assay mass balance of langbeinite leach test on sample AEC 8–4C

(Weight-percent)

Product	Percent of total input tonnage	K_2O	Insolubles	Magnesium	SO_4	Sodium	Chlorine
Feed:							
Coarse	74.8	2.88	2.47	1.49	9.31	32.62	50.06
Fine	25.2	6.18	2.60	3.21	20.59	22.2	32.37
Total	100.0	3.7	2.5	1.92	12.15	30.0	45.6
Concentrate:							
Coarse	9.98	19.2	2.09	10.2	66.1	1.08	1.37
Fine	6.53	19.9	2.28	10.1	67.1	.5	.33
Total concentrate	16.51	19.48	2.17	10.16	66.50	.85	.96
Brine solution	83.49	.58	2.57	.29	1.40	35.76	54.43

TABLE 12.—Ore zone thickness and grade in test hole AEC–8

(After C. Jones, 1975)

Ore zone	Depth (feet)	Thickness (feet)	K_2O distribution by minerals[1]				
			Syl	Lan	Leo	Kai	Car
11th	1,521.8–1,523.1	1.3	1.5	----	----	----	0.5
10th	1,589.7–1,594.7	5.0	13.6	----	----	----	1.1
9th	1,604.3–1,607.7	3.4	3.9	----	----	----	3.4
8th	1,636.6–1,638.1	1.5	11.9	----	----	----	.9
7th	1,666.5–1,671.0	0	----	----	----	----	----
6th	1,681.9–1,683.3	0	----	----	----	----	----
5th	1,688.7–1,697.0	0	----	----	----	----	----
4th	1,753.4–1,757.4	4.0	----	11.0	0.6	0.5	----
3d	1,766.0–1,767.0	1.0	----	3.4	.6	.5	----
2d	1,781.9–1,782.6	.7	----	----	----	----	----
1st	1,796.0–1,810.5	0	----	----	----	----	----

[1] Syl = Sylvite; Lan = Langbeinite; Leo = Leonite; Kai = Kainite; Car = Carnallite.

K_2O as sylvite, or mixed ore of comparable grade. Mineralized areas, as determined by the U.S. Geological Survey and fitting the preceding criteria, are shown in figures 18–20. Exploration drill hole sample analyses, thicknesses, and depths are listed in table 13.

TABLE 13. - Calculated mineral content of selected samples from potassium-bearing intervals with summation of percent K_2O as ore mineral

Drillhole No.: Drillhole designations; P, Energy Research and Development Administration; FC, Farm Chemical Res. Dev. Corp; IMC, International Minerals and Chemical Corp; NFU, Farmers Edu. and Coop. Union of America; D, Duval Sulphur and Potash Co.; U, U.S. Potash Co., Inc.

Calculated Minerals Present and other headings: Arc, arcanite; Bs, bischofite; Bl, bloedite; C, carnallite; Gl, glaserite; Gu, glauberite; Ka, kainite; Ki, kieserite; L, langbeinite; Le, leonite; Lo, loeweite; S, sylvite, Va, vanthoffite

Drillhole No.	Ore Zone	Sample No.	Depths of interval (feet)	Thickness (feet)	Calculated minerals present (weight percent) Polyhalite	Halite	Sylvite	Langbeinite	Other minerals	K_2O as ore minerals (percent)	Weighted average K_2O as ore mineral for intervals preceding by ore zone (feet and percent)
P-1	5	9	1440.47-1441.35	0.88	8	76	---	15.0	Tr/Ka[1]	3.4/L	
	5	10	1441.35-1442.30	0.95	9	86	---	4.4		1.0/L	
	5	11	1442.30-1443.50	1.20	---	79	---	---	14.0/Ka, Tr/Bl[1]	---	
	5	12	1443.50-1444.35	0.85	---	95	---	2.34	---	0.53/L	
	5	13	1444.35-1445.30	0.95	1	79	---	17.07	---	3.87/L	4.83-1.67/L
P-2	10	14	1627.15-1628.35	1.20	1	39	---	39.9	1.9/Ka 5.0/Le	9.05/L	
	10	15	1628.35-1629.52	1.17	1	46	---	43.1	2.6/Ka 0.5/Le	9.78/L	2.37-9.41/L
	4	5	1802.70-1804.00	1.30	1	39	---	38.0	12.9/Ka	8.6/L	
	4	6	1804.00-1805.00	1.00	6	80	---	---	---	---	
	4	7	1805.00-1805.85	0.85	5	89	---	---	---	---	
	4	8	1805.85-1806.30	0.45	6	60	---	20.0	4.0/Ka 3.7/Le	4.54/L	3.60-3.67/L
	2	1	1833.08-1834.00	0.92	2	79	---	4.4	1.9/Ka	3.14/L	
	1	2	1834.00-1834.50	0.50	---	38	---	52.9	0.5/Ka	12.0/L	1.42-6.26/L
P-3	4	3	1596.30-1597.60	1.30	2	72	---	24.7	0.5/Ka	5.6/L	
	4	4	1597.60-1598.70	1.10	2	62	---	18.6	0.5/Ka	4.22/L	
	4	5	1598.70-1599.53	0.83	34	38	---	23.4	---	5.31/L	3.23-5.06/L
P-4	10	2	1572.60-1574.97	2.37	1	56	46.0	---	---	29.40/S	
	10	3	1574.97-1576.17	1.20	4	64	27.6	---	---	17.48/S	
	10	4	1576.17-1577.77	1.60	3	86	7.0	---	---	4.32/S	
	10	5	1577.77-1578.69	0.92	4	64	28.0	---	---	17.46/S	6.09-18.41/S
P-5	10	6	1546.69-1548.65	1.96	6	65	28.0	---	---	18.28/S	
	10	7	1548.65-1549.66	1.01	4	76	13.3	---	---	8.42/S	
	10	8	1549.66-1551.40	1.74	3	60	21.0	---	---	13.41/S	
	10	9	1551.40-1552.75	1.35	2	65	14.5	---	---	9.14/S	6.06-13.2/S
P-6	4	12	1476.00-1477.45	1.45	6	74	---	22.0	1.0/Ki	5.01/L	
	4	13	1477.45-1478.37	0.92	1	83	2.0	8.0	6.0/Ki	1.90/L 1.26/S	
	4	14	1478.37-1480.00	1.63	3	84	---	12.0	---	2.78/L	4.00-3.39/L
	2	18	1510.50-1511.32	0.82	1	37	2.0	54.0	6.0/Ki	12.31/L	
	2	19	1511.32-1512.10	0.78	---	53	---	42.0	---	9.65/L	
	2	20	1512.10-1513.05	0.95	7	79	3.0	1.0	8.6/Ka	0.19/L	2.55-6.98/L
P-7	4	2	1479.73-1481.20	1.47	2	65	1.0	29.0	---	6.53/L	
	4	3	1481.20-1483.00	1.80	---	89	2.0	3.0[4]	---	0.75/L[4]	
	4	4	1483.00-1483.48	0.48	3	69	2.0	3.8	5.7/Ka 1.0/Le 1.0/Bl	3.8/L	3.75-3.41/L
P-8	10	2	1363.70-1365.00	1.30	---	52	29.0	25.0	---	18.45/S 5.62/L	
	10	3	1365.00-1366.72	1.72	1	46	3.0	45.0	4.0/Ka 2.0/Le	10.3/L 1.65/S	
	10	4	1366.72-1368.05	1.33	1	86	---	14.0	---	3.23/L	4.35-6.74/L 4.35-6.17/S mixed ore equivalent: 4.35-9.2/L

Data provided by U.S. Geological Survey.

TABLE 13. - Calculated mineral content of selected samples from potassium-bearing intervals with summation of percent K_2O as ore mineral

Drillhole No.: Drillhole designations; P, Energy Research and Development Administration; FC, Farm Chemical Res. Dev. Corp; IMC, International Minerals and Chemical Corp; NFU, Farmers Edu. and Coop. Union of America; D, Duval Sulphur and Potash Co.; U, U.S. Potash Co., Inc.

Calculated Minerals Present and other headings: Arc, arcanite; Bs, bischofite; Bl, bloedite; C, carnallite, Gl, glaserite; Gu, glauberite; Ka, kainite; Ki, kieserite; L, langbeinite; Le, leonite; Lo, loeweite; S, sylvite, Va, vanthoffite

Drillhole No.	Ore Zone	Sample No.	Depths of intervals (feet)	Thickness (feet)	Calculated minerals present (weight percent) Polyhalite	Halite	Sylvite	Langbeinite	Other minerals	K_2O as ore minerals (percent)	Weighted average K_2O as ore mineral for intervals preceding by ore zone (feet and percent)
P-9	10	8	1522.56-1523.41	0.85	1	42	7.0	48.0	0.67/Ka	4.73/S 10.8/L	
	10	9	1523.41-1524.07	0.66	1	93	2.0	---	3.0/Ki	0.96/S	
	10	10	1524.07-1524.70	0.63	2	17	7.0	63.37	---	4.42/S 14.37/L	
	10	11	1524.70-1526.04	1.34	3	95	---	1.10	---	0.24/L	
	10	12	1526.04-1526.74	0.70	2	65	---	31.31	---	7.1/L	4.18-5.63/L 2.14-3.48/S mixed ore equivalent: 4.18-6.34/L
	4	17	1703.65-1705.23	1.58	2	70	---	28.70	---	6.51/L	
	4	18	1705.23-1705.65	0.42	1	89	---	6.80	3.0/Ka	1.55/L	2.00-5.47/L
P-10	11	1	1650.38-1651.22	0.84	6	89	1.8	---	1.0/Ka	1.13/S	
	11	2	1651.22-1652.03	0.81	7	81	6.1	---	2.0/Ki	3.83/S	
	11	3	1562.03-1653.83	1.80	5	85	7.7	---	1.0/Ki	4.85/S	3.45-3.70/S
	11	4	1653.83-1654.58	0.75	6	94	---	---	---	---	
P-11	11	2	1601.90-1603.56	1.66	---	56	5.0	---	3.60/Ki 1.0/C	3.34/S	
	11	3	1603.56-1604.64	1.08	3	61	2.0	---	31.0/Ka 3.0/C	1.48/S	
	11	4	1604.64-1605.38	0.74	4	84	3.0	---	7.0/Ka 2.0/C	2.20/S	3.48-2.52/S
	10	7	1670.70-1671.84	1.14	1	71	19.0	8.0	1.0/Ki	12.0/S 1.82/L	
	10	8	1671.84-1673.42	1.58	2	66	10.0	18.0	1.0/Ki	6.32/S 3.98/L	
	10	9	1673.42-1674.70	1.28	3	71	16.7	---	1.0/Ki	10.53/S	4.00-9.29/S 2.72-3.07/L mixed ore equivalent: 4.00-14.51/S
	9	14	1688.72-1689.60	0.88	3	76	22.0	---	---	13.74/S	
	9	15	1689.60-1690.89	1.29	1	64	36.9	---	---	23.30/S	
	9	16	1690.89-1691.95	1.06	1	71	21.3	---	2.0/C	13.43/S	
	9	17	1691.95-1693.28	1.33	2	92	2.0	---	1.0/C	1.24/S	4.56-12.73/S
	4	19	1840.60-1842.35	1.75	1	58	---	40.0	5.0/Ki	9.14/L	
	4	20	1842.35-1843.40	1.05	4	76	5.0	4.9	---	1.11/L 2.98/S	2.80-6.0/L
	2	22	1868.67-1870.28	1.61	1	38	2.0	60.0	2.0/Ki	13.65/L	
	2	23	1870.28-1871.10	0.82	---	54	5.0	45.0	1.0/Ki	13.53/L	
	2	24	1871.10-1872.30	1.20	---	27	---	58.3	8.0/Ka Tr/Bl&Lo[1]	13.24/L	3.63-13.49/L

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TABLE 13. - Calculated mineral content of selected samples from potassium-bearing intervals with summation of percent K_2O as ore mineral

Drillhole No.: Drillhole designations; P, Energy Research and Development Administration; FC, Farm Chemical Res. Dev. Corp; IMC, International Minerals and Chemical Corp; NFU, Farmers Edu. and Coop. Union of America; D, Duval Sulphur and Potash Co.; U, U.S. Potash Co., Inc.

Calculated Minerals Present and other headings: Arc, arcanite; Bs, bischofite; Bl, bloedite; C, carnallite, Gl, glaserite; Gu, glauberite; Ka, kainite; Ki, kieserite; L, langbeinite; Le, leonite; Lo, loeweite; S, sylvite, Va, vanthoffite

Drillhole No.	Ore Zone	Sample No.	Depths of interval (feet)	Thickness (feet)	Calculated minerals present (weight percent): Polyhalite	Halite	Sylvite	Langbeinite	Other minerals	K_2O as ore minerals (percent)	Weighted average K_2O as ore mineral for intervals preceding by ore zone (feet and percent)
P-12	10	2a	1344.97-1345.27	0.3	4	36	62.3	---	1.0/Ki	39.38/S	
	10	2b[4/]	1345.27-1345.90	0.63	---	10	24.0[2/]	---	59.0/Ki	15.08/S	
	10	3	1345.90-1346.95	1.05	1	59	18.0	---	18.0/Ki	11.39/S	
	10	4	1346.95-1348.90	1.95	3	64	18.0	---	19.0/Ki	11.36/S	
	10	5	1348.90-1349.91	1.01	7	44	22.0	---	17.0/Ki	14.06/S	
	10	6	1349.91-1350.80	0.89	3	67	17.0[3/]	---	1.0/Ki	10.69/S	5.83-13.57/S
	8	14	1390.19-1390.97	0.78	4	80	15.5	---	---	9.77/S	
	8	15	1390.97-1392.66	1.69	7	88	7.0	---	---	4.12/S	
	8	16	1392.66-1394.29	1.63	4	88	10.3	---	---	6.47/S	
	8	17	1394.29-1394.90	0.61	8	51	35.5	---	---	22.42/S	4.71-8.24/S
	4	21	1520.00-1521.55	1.55	1	22	23.0	45.0	9.0/Ki	14.47/S 10.13/L	2.39-7.08/L
	4	22	1521.55-1522.39	0.84	10	48	7.0	6.0	16.0/Ka 10.0/Le	1.44/L 4.42/S	2.39-10.94/S mixed ore equivalent: 4.00-8.23/L
	3	26	1533.50-1535.05	1.55	---	40	11.0	49.0	---	7.0/S 8.59/L	
	3	27	1535.05-1535.59	0.54	34	86	1.0	8.0	1.0/Ki	1.80/L 0.62/S	3.51-5.98/L
	3	28	1535.59-1537.01	1.42	2	47	10.0	21.0	---	4.72/L 6.32/S	3.51-5.74/S mixed ore equivalent: 3.51-8.27/L
	2	34	1549.79-1550.65	0.86	3	85	---[4/]	11.0	2.42/L	0.54/L	
	2	35a	1550.65-1551.29	0.64	---	20	5.0	70.0	1.0/Ka Tr/Le[1/]	15.93/L	
	2	35b	1551.29-1551.61	0.32	---	21	5.0	30.0	18.0/Ka 19.0/Le 17.0/Bl	12.48/L	1.82-8.05/L
P-13	10	29	1318.02-1319.00	0.98	1	46	49.5	---	---	31.32/S	
	10	30	1319.00-1320.22	1.22	1	62	17.0	2.0	15.0/Ki	10.92/S	
	10	31	1320.22-1320.88	0.66	3	97	---	---	---	---	
	10	32[4/]	1320.88-1321.87	0.99	1	30	20.0	10.0	30.0/Ki 10.0/Ka	12.6/S 2.27/L	3.85-14.67/S
	4	2	1480.20-1481.73	1.53	2	82	2.0	9.0	3.0/Ki	2.03/L	
	4	3	1481.73-1482.78	1.05	1	63	---	33.2	0.4/Le	7.53/L	3.53-4.06/L
	4	4	1482.78-1483.73	0.95	1	64	---	7.0	7.0/Bl	1.59/L	
	3	8	1493.23-1493.88	0.65	2	86	---	7.0	---	1.59/L	
	3	9	1493.88-1494.58	0.70	---	15	---	53.0	---	12.06/L	
	3	10	1494.58-1495.50	0.92	3	86	---	10.0	---	2.21/L	2.27-5.07/L 4.00-2.88/L[5/]
	8	21	1359.65-1360.63	0.98	2	67	30.2	---	---	19.13/S	
	8	22	1360.63-1361.70	1.07	8	80	9.14	---	---	4.97/S	2.05-11.74/S

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TABLE 13. - Calculated mineral content of selected samples from potassium-bearing intervals with summation of percent K_2O as ore mineral

Drillhole No.: Drillhole designations; P, Energy Research and Development Administration; FC, Farm Chemical Res. Dev. Corp; IMC, International Minerals and Chemical Corp; NFU, Farmers Edu. and Coop. Union of America; D, Duval Sulphur and Potash Co.; U, U.S. Potash Co., Inc.

Calculated Minerals Present and other headings: Arc, arcanite; Bs, bischofite; Bl, bloedite; C, carnallite; Gl, glaserite; Gu, glauberite; Ka, kainite; Ki, kieserite; L, langbeinite; Le, leonite; Lo, loeweite; S, sylvite, Va, vanthoffite

Drillhole No.	Ore Zone	Sample No.	Depths of interval (feet)	Thickness (feet)	Calculated minerals present (weight percent) Polyhalite	Halite	Sylvite	Langbeinite	Other minerals	K_2O as ore minerals (percent)	Weighted average K_2O as ore mineral for intervals preceding by ore zone (feet and percent)
	9	36	1334.82-1336.38	1.56	2	94	2.0	---	---	1.12/S	
	9	37	1336.38-1337.32	0.94	2	62	34.8	---	---	21.99/S	
	9	38	1337.32-1338.64	1.32	---	90	3.8	---	---	2.41/S	3.82-6.7/S
P-14	10	2	1255.24-1255.64	0.40	---	41	57.0	---	---	36.03/S	
	10	3	1255.64-1257.56	1.92	---	41	57.0	---	---	36.06/S	
	10	4	1257.56-1259.07	1.51	3	64	14.0	18.0	Tr/Ka[1/]	8.84/S 3.99/L	
	10	5	1259.07-1260.15	1.08	4	74	7.0	22.0	---	4.36/S 4.88/L	
	10	6	1260.15-1261.05	0.90	2	55	12.0[6/]	---	16.0/Ka	6.21/S	5.81-18.47/S 2.59-4.36/L mixed ore equivalent: 5.81-23.33/S
	4	18	1440.79-1441.98	1.19	4	46	6.0	38.5	---	8.74/L 3.99/S	
	4	19	1441.98-1442.84	0.86	5	95	---	---	---	---	
	4	20	1442.84-1443.98	1.14	16	56	4.0	23.0	Tr/Le 2.0/Ka	5.22/L 2.53/S	
	4	21	1443.98-1444.61	0.63	3	88	---	11.0	---	2.50/L	3.82-4.69/L 3.19-2.39/S
	5	10	1364.44-1366.11	1.67	5	74	5.0	15.0	1.0/Ka	3.35/L 3.00/S	
	5	11	1366.11-1367.86	1.75	3	83	---	19.0	Tr/Ka[1/]	4.3/L	
	5	12[6/]	1367.86-1369.26	1.40	1	80	---	13.0	Tr/Ka[1/]	3.03/L	4.82-3.60/L 1.67-3.0/S mixed ore equivalent: 4.82-4.02/L
P-15	4	7	1371.94-1372.81	0.87	6	76	---	25.0	---	5.59/L	
	4	8	1372.81-1374.77	1.96	8	89	---	4.0	---	1.00/L	
	4	9	1374.77-1375.80	1.03	9	64	---	28.0	---	6.30/L	3.86-3.45/L
	2	13	1399.66-1400.38	0.72	3	64	---	32.0	4.0/Ka	7.18/L	
	2	14	1400.38-1401.51	1.13	---	78	---	17.0	3.0/Ka 3.0/Le	4.28/L	1.85-5.41/L
P-16	10	4	1301.94-1302.57	0.63	3	93	0.7	4.0	---	0.47/S 0.89/L	
	10	5	1302.57-1303.91	1.34	3	79	---	21.0	---	4.77/L	
	10	6	1303.91-1304.39	0.48	---	61	---	34.0	8.1/Ka 8.0/Le Tr/Ki[1/]	7.77/L	2.45-4.63/L
	4	10	1476.76-1478.40	1.64	---	36	2.0	53.0	Tr/Ka&Le[1/]	12.01/L 1.26/S	
	4	11	1478.40-1478.95	0.55	2	86	---	8.7	---	1.98/L	
	4	12	1490.12-1491.00	0.88	1	83	1.0	14.1	---	4.94/L	
	4	13	1491.00-1491.56	0.56	0	39	7.0	19.7[7/]	---	4.40/L 4.12/S	
	4	14	1491.56-1492.64	1.08	7	79	---	12.0	---	2.75/L	
	4	15	1492.64-1493.89	1.25	2	56	3.0	31.0	6.0/Le 2.0/Ka	7.05/L 2.09/S	5.96-6.61/S

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TABLE 13. - Calculated mineral content of selected samples from potassium-bearing intervals with summation of percent K_2O as ore mineral

Drillhole No.: Drillhole designations; P, Energy Research and Development Administration; FC, Farm Chemical Res. Dev. Corp; IMC, International Minerals and Chemical Corp; NFU, Farmers Edu. and Coop. Union of America; D, Duval Sulphur and Potash Co.; U, U.S. Potash Co., Inc.

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Drillhole No.	Ore Zone	Sample No.	Depths of interval (feet)	Thickness (feet)	Calculated minerals present (weight percent) Polyhalite	Halite	Sylvite	Langbeinite	Other minerals	K_2O as ore minerals (percent)	Weighted average K_2O as ore mineral for intervals preceding by ore zone (feet and percent)
P-16	2	19	1526.90-1528.00	1.10	1	51	1.0	42.0	2.0/Ka	9.05/L	1.10-9.60/L
P-17	10	2	1365.60-1367.20	1.6	1	27	---	68.8	---	15.61/L	
	10	3	1367.20-1368.45	1.25	3	76	---	12.6	2.0/Ka	2.87/L	
	10	4	1368.45-1369.70	1.25	3	44	---	6.64	30.0/Le 9.0/Ka	1.51/L	4.10-7.43/L
	4	8	1542.90-1543.68	0.78	---	48	---	38.0	13.0/Le Tr/Ka[1]	8.62/L	
	4	9	1543.68-1544.46	0.78	---	43	---	53.0	3.0/Le	12.03/L	1.56-10.33/L
	2	18	1591.39-1592.71	1.32	1	44	---	46.0	1.0/Ka	10.46/L	
	2	19	1592.71-1594.21	1.50	4	83	---	2.4	2.0/Ka	0.53/L	2.82-5.18/L
P-18	10	3	1728.40-1728.78	0.38	1	41	33.0	35.0	---	22.83/S 7.94/L	
	10	4	1728.78-1730.45	1.67	3	86	9.0	---	---	5.61/S	
	10	5	1730.45-1731.49	1.04	6	80	0.72	---	---	0.45/S	
	10	6	1731.49-1732.29	0.80	4	74	0.81	---	---	0.51/S	3.89-4.86/S 0.38-7.94/L mixed ore equivalent: 3.89-6.8/S
P-19	10	7	1741.80-1742.35	0.55	1	55	6.70	36.0	---	4.23/S 8.24/L	
	10	8	1742.35-1743.72	1.37	1	28	2.0	59.0	---	1.20/S 13.39/L	
	10	9	1743.72-1745.09	1.37	4	81	1.0	15.0	---	0.62/S 3.40/L	
	10	10	1745.09-1746.00	0.91	2	53	---	30.0	8.4/Le 2.0/Ka	9.21/L	4.20-8.03/L
	4	21	1925.20-1925.90	0.70	1	74	---	11.0	12.0/Ki	2.50/L	
	4	22	1925.90-1926.70	0.80	1	27	---	57.0	2.0/Ka 9.0/Ki	12.93/L	
	4	23	1926.70-1927.94	1.24	2	62	---	3.6	5.0/Ka Tr/Le[1]	0.82/L	2.74-4.79/L
	2	26	1956.40-1957.36	0.96	6	64	1.0	16.0	15.0/Ki	3.74/L	
	2	27	1957.36-1958.71	1.35	1	29	---	65.0	Tr/Ka[1]	14.83/L	
	2	28	1958.71-1959.21	0.50	0.5	61	---	25.0	Tr/Ka[1]	5.72/L	2.81-9.42/L
P-20	10	2	1725.00-1726.15	1.15	2	72	21.2	Tr/L[4]	---	13.43/S	
	10	3	1726.15-1728.10	1.95	4	64	7.0	35.0	---	4.42/S 8.01/L	
	10	4	1728.10-1729.62	1.52	3	60	28.0[6]	---	6.0/Ka	17.69/S	1.95-8.01/L
	10	5	1729.62-1731.48	1.86	3	60	34.0	---	---	21.48/S	6.51-14.03/S mixed ore equivalent: 6.51-20.03/S
	4	10	1898.80-1900.45	1.65	5	53	1.0	37.0	5.0/Ka	8.35/L	
	4	11	1900.45-1901.77	1.32	16	85	---	1.3	---	0.30/L	
	4	12	1901.77-1903.35	1.58	10	74	---	17.7	---	4.0/L	4.55-4.5/L

Data provided by U.S. Geological Survey.

TABLE 13. - Calculated mineral content of selected samples from potassium-bearing intervals with summation of percent K_2O as ore mineral

Drillhole No.: Drillhole designations; P, Energy Research and Development Administration; FC, Farm Chemical Res. Dev. Corp; IMC, International Minerals and Chemical Corp; NFU, Farmers Edu. and Coop. Union of America; D, Duval Sulphur and Potash Co.; U, U.S. Potash Co., Inc.

Calculated Minerals Present and other headings: Arc, arcanite; Bs, bischofite; Bl, bloedite; C, carnallite; Gl, glaserite; Gu, glauberite; Ka, kainite; Ki, kieserite; L, langbeinite; Le, leonite; Lo, loeweite; S, sylvite, Va, vanthoffite; An, anhydrite

Drillhole No.	Ore Zone	Sample No.	Depths of interval (feet)	Thickness (feet)	Calculated minerals present (weight percent) Polyhalite	Halite	Sylvite	Langbeinite	Other minerals	K_2O as ore minerals (percent)	Weighted average K_2O as ore mineral for intervals preceding by ore zone (feet and percent)
P-20	2	14[4/]	1925.08-1926.30	1.22	3	95	---	0.88	---	0.2/L	
	2	15	1926.30-1927.75	1.45	1	61	---	35.5	---	8.05/L	2.67-4.46/L
P-21	10	2	1644.03-1644.84	0.81	4	80	17.0	---	---	10.82/S	
	10	3	1644.84-1646.00	1.16	4	81	11.8	---	2.0/Ki	7.46/S	
	10	4	1646.00-1646.33	0.33	1	28	20.2[6/]	---	42.0/Ki	12.76/S	
	10	5	1646.33-1647.20	0.87	1	41	25.5[6/]	---	18.0/Ki	16.14/S	
	10	6	1647.20-1648.22	1.02	1	62	29.5[2/]	---	1.0/Ki	18.66/S	
	10	7	1648.22-1649.23	1.01	3	64	31.3	---	---	19.81/S	5.20-14.37/S
	8	15	1685.17-1686.48	1.31	0	55	45.0	---	---	28.4/S	
	8	16	1686.48-1687.20	0.72	4	83	6.0[6/]	---	---	3.72/S	
	8	17	1687.20-1688.24	1.04	5	92	3.0	---	---	1.95/S	
	8	18	1688.24-1688.77	0.53	4	95	0.8	---	---	0.52/S	
	8	19	1688.77-1690.19	1.42	6	86	9.9	---	---	6.24/S	
	8	20	1690.19-1691.26	1.07	8	90	1.0	---	---	0.75/S	
	8	21	1691.26-1692.40	1.14	3	65	33.0	---	---	21.16/S	
	8	22	1692.40-1693.34	0.94	5	65	13.0	---	---	8.18/S	8.17-10.24/S
	4	24	1809.90-1811.50	1.60	0	31.0	3.0	64.0	2.5/Ka 3.9/Le	14.60/L	1.92-14.69/L[5/]
	4	25	1811.50-1811.82	0.32	0	18	5.0[2/]	54.66	9.0/Ka 10.60/Le	15.12/L	4.00-7.05/L[2/]
	4	30	1815.51-1816.10	0.59	11	51	5	27.0	9.0/Ki	6.08/L	
	4	31	1816.10-1817.25	1.15	3	42	5	44.0	8.0/Ka	9.95/L	1.74-8.64/L
AEC-8	10	10	1589.10-1589.70	0.60	1	85	5	---	5/Ki 2/An	2.97/S	
	10	11	1589.70-1591.70	2.00	---	43	16	---	40/Ki	10.32/S	
	10	12[6/]	1591.70-1592.20	0.50	---	37	24	---	6/C 30/Ki 0 1/An	15.16/S	
	10	13[6/]	1592.20-1594.50	2.30	---	49	32	---	7/C 9/Ka 0.8/An	20.31/S	
	10	14	1594.50-1594.70	0.20	---	38	4	---	51/C 9/Ka 1/Ki 0.1/An	2.34/S	
	10	15[6/]	1594.70-1595.50	0.80	2	85	3	---	4/C 2/Ki 9/Ka 0.9/An	2.18/S	6.4-12.33/S
	4	2	1752.70-1753.40	0.70	---	95	---	4	---	0.86/L	
	4	3	1753.40-1754.00	0.60	1	69	2	33	3/Ka	7.39/L	
	4	4	1754.00-1755.00	1.00	---	33	2	68	2/Ka	13.90/L	
	4	5	1755.00-1756.70	1.70	---	24	---	69	3/Le	15.38/L	4-11.27/L

TABLE 13. - Calculated mineral content of selected samples from potassium-bearing intervals with summation of percent K_2O as ore mineral

Drillhole No.: Drillhole designations; P, Energy Research and Development Administration; FC, Farm Chemical Res. Dev. Corp; IMC, International Minerals and Chemical Corp; NFU, Farmers Edu. and Coop. Union of America; D, Duval Sulphur and Potash Co.; U, U.S. Potash Co., Inc.

Calculated Minerals Present and other headings: Arc, arcanite; Bs, bischofite; Bl, bloedite; C, carnallite; Gl, glaserite; Gu, glauberite; Ka, kainite; Ki, kieserite; L, langbeinite; Le, leonite; Lo, loeweite; S, sylvite, Va, vanthoffite

Drillhole No.	Ore Zone	Sample No.	Depths of interval (feet)	Thickness (feet)	Calculated minerals present (weight percent): Polyhalite	Halite	Sylvite	Langbeinite	Other minerals	K_2O as ore minerals (percent)	Weighted average K_2O as ore mineral for intervals preceding by ore zone (feet and percent)
FC-70	10	2	1377.67-1379.00	1.33	2	65	11.2	---	20.0/Ki	7.11/S	
	10	3	1379.00-1381.00	2.00	2	51	39.1[6/]	---	1.0/Ki	24.72/S	
	10	4	1381.00-1382.00	1.00	2	64	19.4[6/]	---	3.0/C 3.0/Ki	12.29/S	4.33-16.44/S
	9	6	1391.50-1393.00	1.50	1	53	28.00	---	---	17.74	1.5-17.74/S
	8	7	1415.08-1415.75	0.67	2	70	25	---	Tr/Ki[1/]	16.10/S	
	8	8	1415.75-1416.50	0.75	5	89	2	---	Tr/Ki[1/]	1.35/S	
	8	9	1416.50-1418.00	1.50	2	97	1	---	1.0/Ka Tr/Ki[1/]	0.57/S	
	8	10	1418.00-1419.50	1.50	3	91	3	---	---	1.62/S	
	8	11	1419.50-1420.42	0.92	2	80	16	---	1.0/Ka Tr/Ki[1/]	10.04/S	
	8	12	1420.42-1421.50	1.08	1	71	16	---	1.0/Ka	10.21/S	6.42-5.5/S
	5	16	1466.00-1467.58	1.58	---	72	---	19.6	---	4.43/L	
	5	17	1467.58-1469.00	1.42	1	53	---	40.7	---	9.22/L	
	5	18	1469.00-1469.67	0.67	1	70	---	24.2	---	5.49/L	3.67-6.48/L
	4	19	1529.92-1531.42	1.50	1	36	---	52.6	---	11.93/L	
	4	20	1531.42-1532.42	1.00	---	45	---	33.0[6/]	---	7.42/L	2.5-10.13/L
FC-81	8	7	1564.17-1564.92[8/]	0.75	2	58	36.6	---	2.0/C	23.16/S	
	8	8	1564.92-1566.13	1.21	2	79	15.3	---	4.0/C	9.71/S	
	8	9	1566.13-1567.38	1.25	5	66	8.9	6.1	10.0/C	5.62/S 1.38/L	3.21-11.26/S 3.21-0.54/L mixed ore equivalent: 3.21-12.61/S
	4	10	1687.00-1688.21	1.21	---	20	---	74.0	4.0/Ka Tr/Va[1/]	16.71	
	4	11	1688.21-1689.46	1.25	1	25	---	67.0	Tr/Va[1/]	15.31/L	2.46-16.0/L
	2	15	1712.88-1714.46	1.58	1	40	---	56.0	2.2/Le	12.7/L	
	2	16	1714.46-1715.46	1.00	---	65	---	30.3	3.0/Ka Tr/Le[1/]	6.9/L	
	2	17	1715.63-1716.00	0.38	4	45	---	42.8	Tr/Le[1/]	9.71/L	3.96-10.4/L
FC-82	5	1	1541.42-1541.79	0.38	---	17	---	72.2	2.0/Le 6.0/C	16.34/L	
	5	2	1541.79-1542.29	0.42	---	85	7.0	7.2	---	1.63/L	
	5	3	1542.29-1543.96	1.67	2	5	---	49.8	4.0/Lo 20.9/C	11.3/L	
	5	4	1543.96-1544.50	0.54	1	59	5.0	20.0	Tr/Le[1/]	4.54/L	3.01-9.37/L
	4	5	1613.42-1615.08	1.67	---	17	---	63.0	9.3/Ki Tr/Lo[1/]	14.3/L	
	4	6	1615.08-1615.92	0.83	1	48	0.7	12.8	17.7/Le 12.0/Ki	2.9/L	2.5-10.52/L

Data provided by U.S. Geological Survey.

TABLE 13. - Calculated mineral content of selected samples from potassium-bearing intervals with summation of percent K_2O as ore mineral

Drillhole No.: Drillhole designations; P, Energy Research and Development Administration; FC, Farm Chemical Res. Dev. Corp; IMC, International Minerals and Chemical Corp; NFU, Farmers Edu. and Coop. Union of America; D, Duval Sulphur and Potash Co.; U, U.S. Potash Co., Inc.

Calculated Minerals Present and other headings: Arc, arcanite; Bs, bischofite; Bl, bloedite; C, carnallite; Gl, glaserite; Gu, glauberite; Ka, kainite; Ki, kieserite; L, langbeinite; Le, leonite; Lo, loeweite; S, sylvite; Va, vanthoffite

Drillhole No.	Ore Zone	Sample No.	Depths of interval (feet)	Thickness (feet)	Calculated minerals present (weight percent) Polyhalite	Halite	Sylvite	Langbeinite	Other minerals	K_2O as ore minerals (percent)	Weighted average K_2O as ore mineral for intervals preceding by ore zone (feet and percent)
FC-82	3	7	1624.33-1625.58	1.25	2	57	---	31.1	3.5/Bl 3.7/Ka	7.06/L	
	3	8	1625.58-1627.00	1.42	2	67	---	29.9	0.6/C[1]	6.78/L	
	3	9	1627.00-1627.75	0.75	1	28	---	62.2	Tr/Bl[1] 5.0/Gl	14.11/L	3.42-8.49/L
FC-91	4	34	1712.25-1712.66	0.42	1	18	---	50.0	11.0/Ka 7.5/Le 6.0/Ki	11.35/L	
	4	35	1712.66-1713.08	0.42	---	62	---	---	5.0/Ka 3.4/Ki	---	
	4	36	1713.08-1714.17	1.08	1	25	---	64.6	5.0/Ki	14.66/L	
	4	37	1714.17-1714.75	0.58	2	32	---	58.7	5.0/Ka 1.3/Ki	13.32	
	4	38	1715.75-1715.75	1.00	20.0	38	---	47.0	5.0/Ka Tr/Gu[1]	10.66	4.08-11.3/L
	4	39	1715.75-1716.33	0.58	6	31	---	54.0	6.2/Ka 4.0/Gl 1.2/Ki	12.25/L	
	4	40	1718.66-1719.42	0.75	3	81	---	8.0	3.0/Ka 9.0/Bs	1.82/L	
	4	41	1719.42-1720.75	1.33	2	37	---	59.5	---	13.5/L	
	4	42	1720.75-1722.00	1.25	---	60	---	37.08	Tr/Gu 3.4/Bs	8.41/L	
	4	43	1722.00-1722.58	0.58	---	71	---	13.5	0.4/Ka 6.0/Bs Tr/Gu[1]	3.06/L	3.91-8.08/L
	2	44	1742.75-1743.25	0.50	---	15	---	62.0	8.8/Ka Tr/Va[1]	14.07/L	
	2	45	1743.25-1744.25	1.00	---	47	---	30.5	10.0/Ki 10.0/Va	6.92/L	
	2	46	1744.25-1745.00	0.75	---	61	---	1.0	3.2/Ka 10.1/Ki 24.0/Va Tr/Gu[1]	0.73/L	2.25-6.44/L
FC-92[9]	8	5	1604.92-1606.04	1.12	1	51	47	Tr[1]	Tr/C[1]	29.0/S	
	8	6	1606.04-1606.50	0.46	---	50	49	Tr[1]	Tr/C[1]	31.0/S	
	8	7	1606.50-1606.75	0.25	---	60	29	1	5.7/C	18.5/S	1.83-28/S
FC-92	4[9]	57	1740.66-1741.58	0.92	1	51	---	46.7	---	10.6/L	
	4	58	1741.58-1742.25	0.66	1	36	---	61.0	---	13.85/L	
	4	59	1742.25-1743.25	1.00	1	36	---	61.2	0.1/Ki	13.89/L	
	4	60	1743.25-1744.08	0.83	1	34	---	61.3	---	13.91/L	
	4	61	1744.08-1745.00	0.92	2	38	---	48.8	1.8/Ki	11.07/L	4.33-12.6/L
	2[9]	62	1769.42-1770.08	0.66	1	42	---	55.4	3.0/Va	12.57/L	
	2	63	1770.08-1771.42	1.33	1	51	---	45.7	2.0/Va 1.0/Arc	10.37/L	
	2	64	1777.42-1772.08	0.66	---	39	---	33.0	17.0/Va 8.0/Ki 1.0/Gu	7.49/L	
	2	65	1772.08-1772.58	0.5	---	73	---	1.5	14.0/Va 1.0/Gu	0.34/L	3.15-8.64/L

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TABLE 13. - Calculated mineral content of selected samples from potassium-bearing intervals with summation of percent K_2O as ore mineral

Drillhole No.: Drillhole designations; P, Energy Research and Development Administration; FC, Farm Chemical Res. Dev. Corp; IMC, International Minerals and Chemical Corp; NFU, Farmers Edu. and Coop. Union of America; D, Duval Sulphur and Potash Co.; U, U.S. Potash Co., Inc.

Calculated Minerals Present and other headings: Arc, arcanite; Bs, bischofite; Bl, bloedite; C, carnallite, Gl, glaserite; Gu, glauberite; Ka, kainite; Ki, kieserite; L, langbeinite; Le, leonite; Lo, loeweite; S, sylvite, Va, vanthoffite

Drillhole No.	Ore Zone	Sample No.	Depths of interval (feet)	Thickness (feet)	Calculated minerals present (weight percent) Polyhalite	Halite	Sylvite	Langbeinite	Other minerals	K_2O as ore minerals (percent)	Weighted average K_2O as ore mineral for intervals preceding by ore zone (feet and percent)
IMC-374	4	8	1430.70-1432.10	1.40	6	86	0.7	2.0	0.7/Ki 0.3/Ka	0.46/L 0.44/S	
	4	9	1432.10-1433.10	1.00	13	68	1.0	6.4	1.35/Le	1.46/L 0.63/S	2.4-0.88/L 2.4-0.51/S
IMC-375	4	7	1627.90-1629.00	1.10	2	78	---	6.6	2.0/Ka	1.50/L	
	4	8	1629.00-1630.50	1.50	4	70	2.0	13.2	2.0/Le 1.0/Ka	3.0/L 1.3/S	2.5-2.46/L 1.5-1.3/S
IMC-376	10	2	1427.00-1427.70	0.7	---	60	38.4	---	---	24.3/S	
	10	3	1427.70-1428.30	0.6	---	98	0.2	---	---	0.1/S	
	10	4	1428.30-1429.20	0.9	1	96	0.4	---	---	0.26/S	
	10	5	1429.20-1431.00	1.8	2	86	11.2	---	---	7.1/S	4.0-7.52/S
	5	13	1528.00-1528.90	0.9	2	89	1.0	2.8	---	0.63/L	
	5	14	1528.90-1530.90	2.0	0	45	---	51.3	1.0/Le	11.65/L	3.7-6.5/L
	5	15	1530.90-1531.70	0.8	2	93	1.0	1.05	---	0.25/L	(4.0-6.03/L)5/
NFU-1	8	10/	1536.25-1540.17	3.92	---	---	---	---	---	19.67/S	3.92-19.67/S
	4	10/	1646.75-1649.08	2.33	---	---	---	---	---	11.96/L	2.33-11.96/L
	8	10/	1479.80-1483.80	4.00	---	---	---	---	---	9.1/S	4.0-9.1/S
	4	10/	1598.50-1602.70	4.20	---	---	---	---	---	13.1/L	4.2-13.1/L
	10	10/	1441.08-1445.17	4.08	---	---	---	---	---	0.8/S	4.08-0.8/S
D-120	10	10/	1248.30-1249.80	1.50	---	---	---	---	---	31.63/S	
	10	10/	1249.80-1251.10	1.30	---	---	---	---	---	10.4/S	
	10	10/	1251.10-1252.50	1.40	---	---	---	---	---	7.34/S	4.2-15.3/S
	4	10/	1419.60-1421.30	1.70	---	---	---	---	---	13.8/S	2.2-10.7/S
	4	10/	1421.30-1421.80	0.50	---	---	---	---	---	0.5/S	2.2-7.5/L mixed ore equivalent: 2.2-29.45/S
D-48	10	10/	1236.60-1240.60	4.00	---	---	---	---	---	---	4.0-11/S 4.0-2.1/L (visual estimate)
	8	10/	1280.52-1289.78	9.26	---	---	---	---	---	---	9.26-17.37/S
	4	10/	1414.20-1416.20	2.00	---	---	---	---	---	---	2.0-8.8/L (visual estimate)

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Drillhole No.: Drillhole designations; P, Energy Research and Development Administration; FC, Farm Chemical Res. Dev. Corp; IMC, International Minerals and Chemical Corp; NFU, Farmers Edu. and Coop. Union of America; D, Duval Sulphur and Potash Co.; U, U.S. Potash Co., Inc.

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Drillhole No.	Ore Zone	Sample No.	Depths of interval (feet)	Thickness (feet)	Calculated minerals present (weight percent): Polyhalite	Halite	Sylvite	Langbeinite	Other minerals	K_2O as ore minerals (percent)	Weighted average K_2O as ore mineral for intervals preceding by ore zone (feet and percent)
D-207	4	33	1533.10-1533.50	0.40	2	53	36	7	---	1.51/L S	22.74
	4	34	1533.50-1533.80	0.30	4	86	2	11	---	2.43/L	
	4	35	1533.80-1534.40	0.60	2	85	1	2	---	0.50/L	
	4	36	1534.40-1534.70	0.30	3	95	---	4	---	0.85/L	
	4	37	1534.70-1535.00	0.30	2	74	4	1	3.0/Ki	0.14/L	
	4	38	1535.00-1539.80	4.80	6	87	2	4	---	0.85/L	6.7-0.90/L 0.4-22.74/S mixed ore equivalent: 6.7-1.44/L
	10	24	1358.60-1359.60	1.00	4	87	5	---	2/Ka	3.34/S	
	10	25	1359.60-1360.20	0.60	4	89	5	1	2/Ka	3.34/S 0.25/L	
	10	26 4/	1360.20-1360.60	0.40	1	75	4	1	6/Ka	0.31/L	
	10	27 4/	1360.60-1361.20	0.60	1	47	6	16	5/Ka	3.79/S 3.70/L	
	10	28	1361.20-1361.80	0.60	2	69	2	9	2/Ka	2.04/L	
	10	29	1361.80-1362.30	0.50	7	62	5	10	7/Ka	3.34/S 2.31/L	
	10	30	1362.30-1363.10	0.80	6	83	4	---	1/C 3/Ka	2.66/S	4.5-2.54/S 2.7-1.8/L mixed ore equivalent: 4.5-5.24/S
	3	44	1545.90-1546.30	0.40	1	66	---	32.0	0.74/Le	7.2/L	
	3	45	1546.30-1546.66	0.30	1	11	---	73.2	9.0/Ki	16.6/L	
	3	46	1546.66-1548.00	1.40	2	62	---	30.3	3.8/Le	6.89/L	
	3	47	1548.00-1548.60	0.60	1	43	---	54.7	---	12.4/L	2.7-9.24/L
	2	52	1552.20-1553.00	0.80	11	83	---	5.0	---	1.05/L	
	2	53	1553.00-1553.20	0.20	11	45	---	41.5	2.0/Le	9.41/L	
	2	54	1553.20-1553.60	0.40	4	81	---	15.8	---	3.59/L	
	2	55	1553.60-1554.00	0.40	1	47	---	49.4	1.32/Le	11.19/L	
	2	56	1554.00-1554.80	0.80	18	86	1.4	8.8	---	2.0/L 0.9/S	2.6-3.94/L 4/ Tr/S
D-104	10 10/		1340.40-1341.70	1.30	---	---	---	---	---	---	
	10		1341.70-1343.20	1.50	---	---	---	---	---	---	
	10		1343.20-1344.00	0.80	---	---	---	---	---	---	
	10		1344.00-1344.70	0.70	---	---	---	---	---	---	6.0-5.5/S
	10		1344.70-1346.40	1.70	---	---	---	---	---	---	3.2-4.71/L mixed ore equivalent: 6.0-11.78/S
	8 10/		1391.10-1392.00	0.90	---	---	---	---	---	---	
	8		1392.00-1395.30	3.30	---	---	---	---	---	---	
	8		1395.30-1396.30	1.00	---	---	---	---	---	---	5.2-10.8/S
	4 10/		1519.99-1522.88	2.90	---	---	---	---	---	---	2.9-9.4/L

Data provided by U.S. Geological Survey.

TABLE 13. - Calculated mineral content of selected samples from potassium-bearing intervals with summation of percent K_2O as ore mineral

Drillhole No.: Drillhole designations; P, Energy Research and Development Administration; FC, Farm Chemical Res. Dev. Corp; IMC, International Minerals and Chemical Corp; NFU, Farmers Edu. and Coop. Union of America; D, Duval Sulphur and Potash Co.; U, U.S. Potash Co., Inc.

Calculated Minerals Present and other headings: Arc, arcanite, Bs, bischofite; Bl, bloedite; C, carnallite; Gl, glaserite: Gu, glauberite; Ka, kainite; Ki, kieserite; L, langbeinite; Le, leonite; Lo, loeweite; S, sylvite; Va, vanthoffite

Drillhole No.	Ore Zone	Sample No.	Depths of interval (feet)	Thickness (feet)	Calculated minerals present (weight percent) Polyhalite	Halite	Sylvite	Langbeinite	Other minerals	K_2O as ore minerals (percent)	Weighted average K_2O as ore mineral for intervals preceding by ore zone (feet and percent)
D-104	3[10/]		1527.50-1528.90	1.40	---	---	---	---	---	---	1.4-5.0/L
	3		1528.90-1530.50	1.60	---	---	---	---	---	---	2.9-11.4/S mixed ore equivalent: 2.9-6.66/L
	2[10/]		1539.50-1540.20	1.20	---	---	---	---	---	---	1.2-15.0/L
U-134	10		1319.58-1321.25	1.69	3	---	6.0	---	2.0/Ka 0.7/C	3.79/S	
			1321.25-1322.83	1.58	2	---	---	---	12.5/Ka	17.22/S	3.27-10.28/S
	8[10/]		1361.10-1362.18	1.08	---	---	---	---	---	---	
	8		1362.17-1364.50	2.33	---	---	---	---	---	---	
	8		1364.50-1366.50	2.00	---	---	---	---	---	---	3.41-11.6/S
	8		1366.50-1367.42	0.92	---	---	---	---	---	---	(4.0-10.0/S)[5/]
	5		1406.75-1409.42	2.66	---	---	---	---	---	14.44/L	
	5		1409.42-1410.00	0.58	---	---	---	---	---	8.03/L	
	5		1410.00-1411.66	1.66	---	---	---	---	---	7.05/L	4.9-11.2/L
	4		1471.66-1474.00	2.33	---	---	---	---	---	8.54/L	2.33-8.54/L
	3		1484.91-1487.33	2.42	---	---	---	---	---	3.6/L	
	3		1487.33-1490.30	2.92	---	---	---	---	---	1.86/L	
	3		1490.25-1491.33	1.08	---	---	---	---	---	8.7/L	6.42-3.7/L[3/]

Data provided by U.S. Geological Survey.

- FOOTNOTES -
Table

1/ Trace amount; equals 0 to 2.0%

2/ Incomplete dissolution of sample

3/ 5.9% Insolubles, by weight

4/ Incomplete or unreliable assay

5/ Grade adjusted to 4 foot interval

6/ High insoluble content

7/ 7.1% Potassium Assay used

8/ Outside of the ERDA area by 300 feet, included due to influence

9/ Raw data unavailable; these are company figures

10/ Company interval data; raw data unavailable

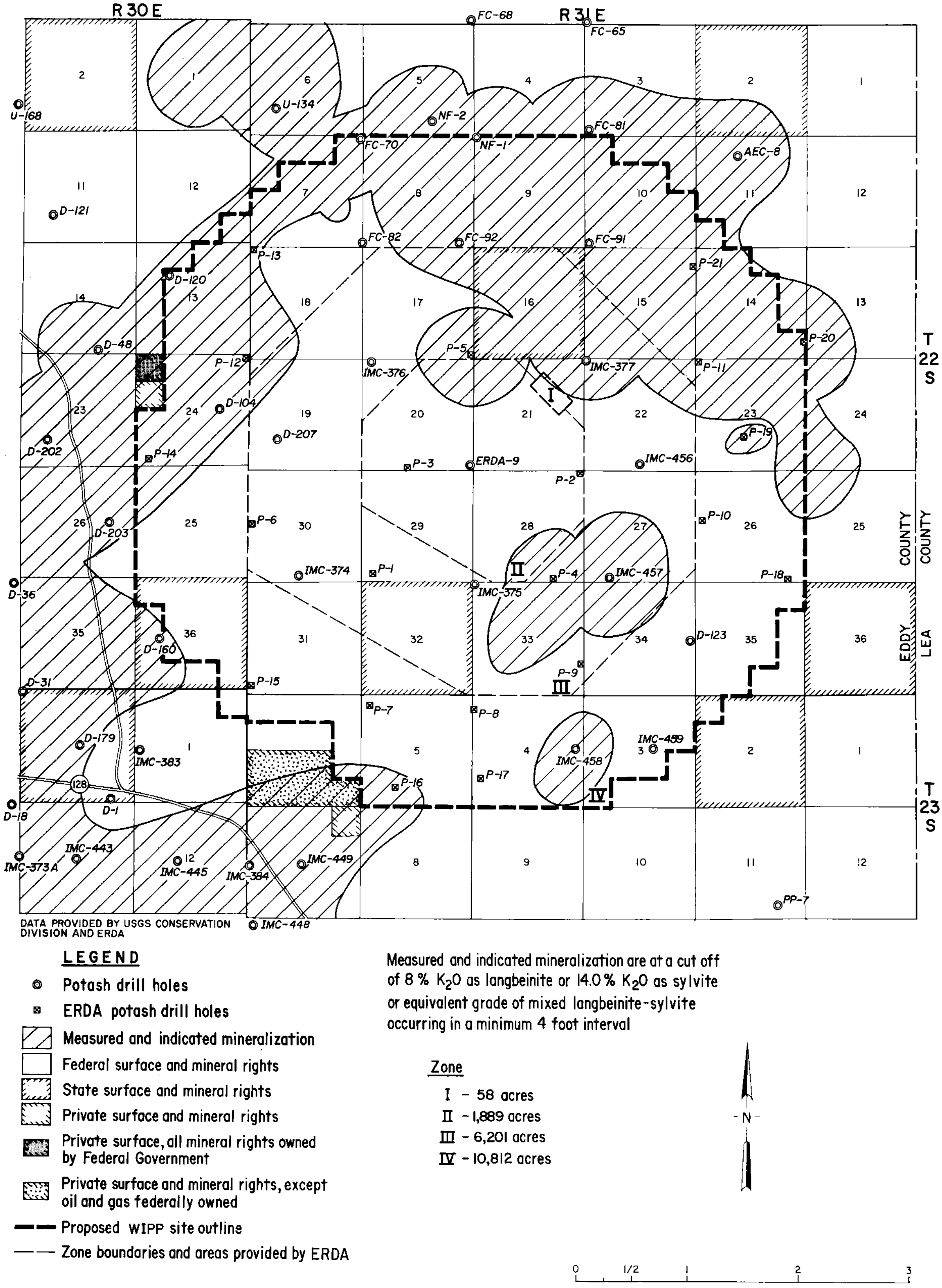

FIGURE 18.—Composite map of mineralization in various ore zones at 8 and 14 percent K_2O as langbeinite and sylvite, respectively.

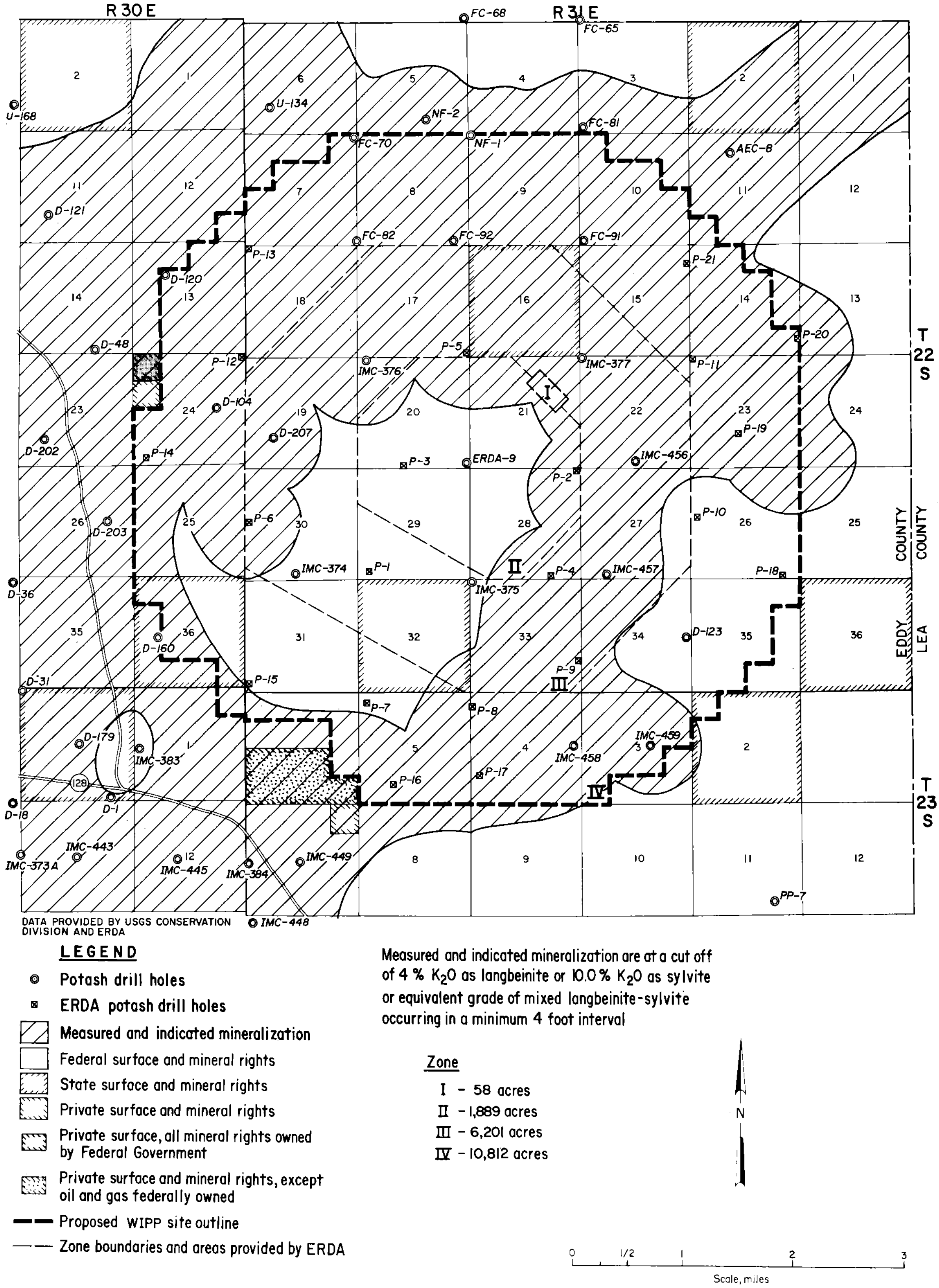

FIGURE 19.—Composite map of mineralization in various ore zones at 4 and 10 percent K_2O as langbeinite and sylvite, respectively.

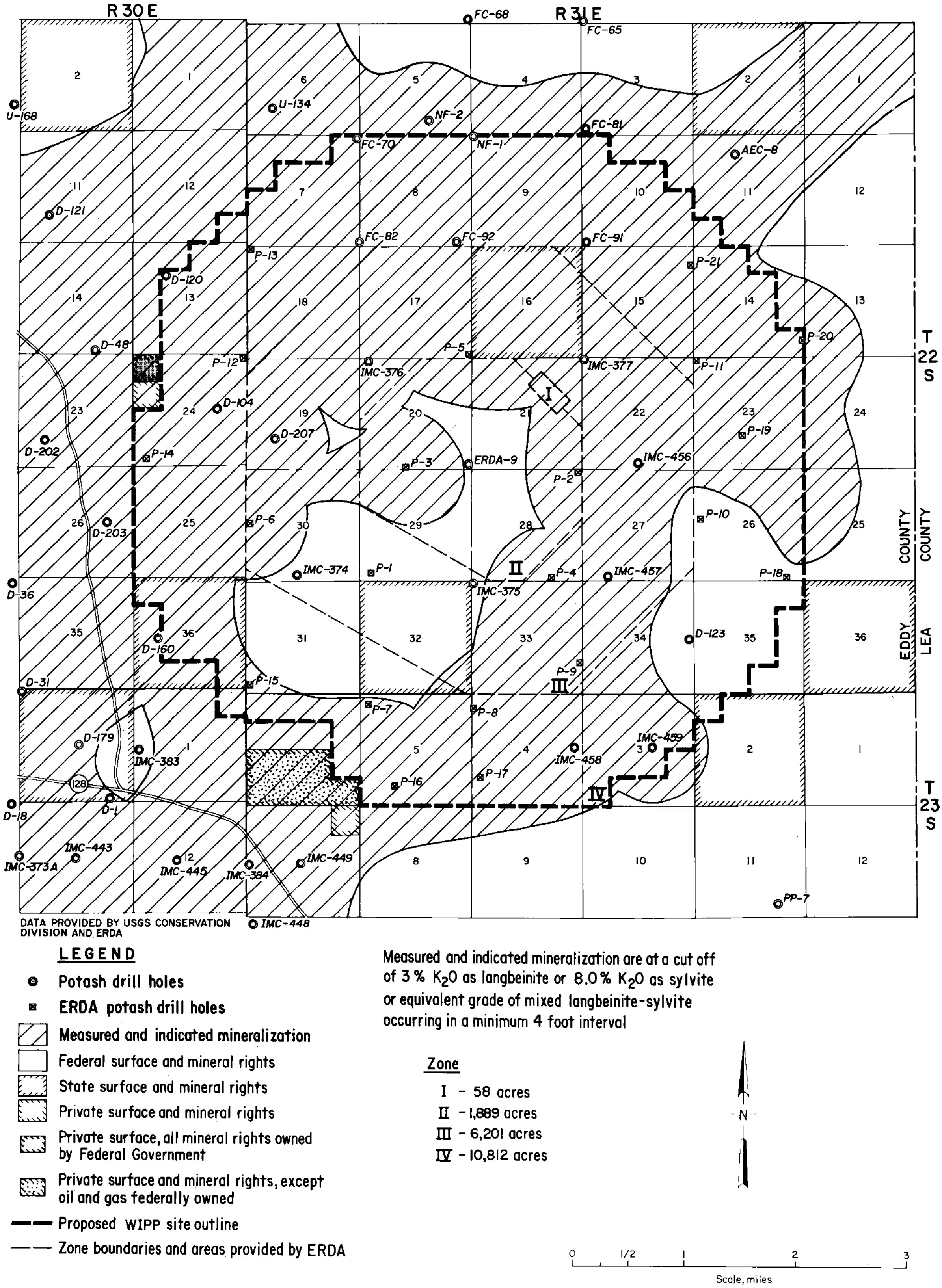

FIGURE 20.—Composite map of mineralization in various ore zones at 3 and 8 percent K_2O as langbeinite and sylvite, respectively.

FINANCIAL ANALYSIS

METHODS USED

Present value analyses were used to determine the worth of the potash mineralization in the WIPP site. These are the values that would be chargeable to the WIPP facility if it were built and the potash products were not recovered. All reserves (presently commercial) and presently subeconomic resources of potassium mineral deposits were examined. Incomes, costs, and investments are assumed to occur as discrete amounts over the life of the project. For economic evaluation purposes and to compare equivalent values that have been adjusted for the time value of money, these amounts are converted (discounted at an appropriate discount rate) to equivalent values (1977 dollars) at one point in time (project initiation date). Present value of a mineral deposit is defined for these analyses as the present worth of the cash flows from a hypothetical potash operation minus the present worth of the investments (*23*). The potash values are taxes, royalties, and bonus bid amounts that would be generated from potash product sales and paid to the several levels of government. Such taxes, royalties, and bonus bid amounts (or their equivalent value) will not accrue to the governments if the potash deposits are not developed. All other values of production and investments originate when projects are initiated, or when capital is invested, and therefore are not a loss if the unmined, commercial potash is not developed.

Cash Flow Estimates

The cash flows are the gross revenues minus all direct, indirect, overhead, and front office production costs. These do not include bookkeeping charges for depreciation, depletion, amortization and extraordinary charges to capital reserves that are not actually paid out.

Cash flows of the project are the gross revenues minus (1) mine operating costs; (2) mill operating costs; (3) all overhead costs, including front office charges; (4) all Federal and State royalties; and (5) all Federal and State taxes.

Cash flows are the project's total self-generated cash after costs and are the amounts that provide for the return of the original investment plus the interest on that investment. The cash flows are a production-cost element, in dollar amounts generated by the project, that return the investment plus sufficient interest on the investment to induce investors to provide the capital for the project.

Investment Estimates

The total investments include (1) exploration and engineering study; (2) acquisition costs (including any lease bonus payment); (3) mine preparation; (4) mine plant investment; (5) mine equipment; (6) mine reinvestments in capital items during project life; (7) mill plant investment; (8) mill capital reinvestments during project life; (9) all working capital; and (10) startup and break-in costs.

The cash flows and investments are discounted by continuous discounting (versus discrete discounting) at an interest rate (15 percent) suitable to investors in the potash industry. A lower interest rate (return on investment) is assumed to be unacceptable and would divert the capital to more attractive investment opportunities.

The values in the commercial potash deposits that would be foregone in favor of the installation of the Waste Isolation Pilot Plant include the amounts of bonus bids that could be paid by potash investors to the Federal and State governments for potash leases. In addition to these amounts are the present values of Federal and State royalty payments and taxes that would have been generated in the production of the potash products. To estimate the present value of the taxes and royalty payments, these amounts were discounted at 8 percent interest from the year in which they would have occurred to 1977 dollars. The total values that would be foregone are (1) bonus bid amounts, (2) present value of tax amounts, and (3) present value of royalties.

In the event the WIPP installation was built, and if the government-owned potash in the site were under lease, these values would be foregone to the government entities. In addition, the lessee would lose any amounts, including interest, that have been paid out for acquisition of the property, for exploration, for development engineering studies, for legal fees, etc. If presently commercial potash deposits are leased, the lessee will also lose a potash investment opportunity in addition to the aforementioned out-of-pocket amounts spent.

Determination of Commercial and Subeconomic Mineralization

When the sum of cash flows, discounted at 15 percent interest, equals or exceeds the sum of the investment discounted at 15 percent interest, both to 1977 dollars, the project is considered to be commercial; and the potash deposit is classified as a reserve or ore. When the sum of the discounted cash flows is less than the sum of the discounted investments, the project is not commercial, and the potash deposit is considered to be economically submarginal and is classified as a resource. These potash resources may become ore at some future time when potash becomes more valuable relative to costs of production.

When the cash flows will return the original investment in a mine-mill complex and more than the acceptable 15-percent interest rate, the potash deposits being examined have a present value or worth in excess of all other investments. In this case, the owners of the potash (in the WIPP site, the Federal and State governments) could require payment of the present value or worth in the form of a bonus bid amount for a lease to produce the potash (designated an acquisition cost in this report).

To examine the potash resources (presently subeconomic deposits), the market prices of the products were increased, without increasing the costs of production, until the deposits became commercial. This analysis provides a guide to the potential value of the potash resources within the WIPP site. An estimate of the time at which these increased potash values might occur is so speculative it is not made in these analyses.

An analysis of the impact of the loss of the potash reserves and resources, such as the effect on the gross national product and balance of payments, was not made because it was beyond the scope of this study.

Taxes and Royalties

Taxes are a major cost item, and a discussion of applicable New Mexico State taxes is included. New Mexico potash operations are subject to the following taxes: Federal corporation income tax; New Mexico corporation income tax; resource, processors, and service tax; severence tax; property tax; and in lesser amounts—franchise tax, corporate organization and qualification fees; motor vehicle registration fees; motor carrier fees; and other indirect taxes.

The State Corporation Commission administers and collects the corporation annual report filing fees, the corporation and qualification fees, the franchise tax, miscellaneous motor carrier fees, and the pipeline companies tax. The Bureau of Revenue administers the resource excise tax, the severance tax, and the gross receipts tax; the Property Tax Department administers and collects the mining property tax.

Federal Corporation Income Tax

Federal income tax for corporations is based on gross income minus certain deductions. These deductions in general are costs of operations and include compensation of officers, salaries and wages, repairs, bad debts, rents, taxes, interest, contributions, amortization, depreciation, depletion, advertising, pension, profit-sharing, and employee benefit programs. Federal income tax rates for corporations are 20 percent on the first $25,000 of taxable income, 22 percent on the next $25,000, and 48 percent on all taxable income over $50,000.

New Mexico Corporation Income Tax

The New Mexico income tax is based on the Federal corporation income tax. The State income tax is 5 percent of the net taxable income within the State of New Mexico.

If the corporation operates exclusively in New Mexico, interest on U.S. obligations, to the extent that they are included in Federal taxable income, and interest on amounts taxed as income to another member of an affiliated group of corporations are listed as nontaxable income and are deductible.

If the corporation operates both within and outside the State, the taxable income for the State of New Mexico is estimated by dividing the nonbusiness and business income between New Mexico and elsewhere.

Nonbusiness income includes dividends, interest, rents, royalties, profit or loss on the sale of nonbusiness assets, and partnership income. The nonbusiness income is allocated to New Mexico in three ways: (1) to the extent that it is generated in the State; (2) if the nonbusiness income was earned from the sale of property, if the property is located in New Mexico at the time of sale; and (3) if New Mexico is considered the commercial domicile of the corporation.

The division of all business income is apportioned to New Mexico by a three-factor formula: (1) the property factor is the average value of real and tangible personal property owned or rented and used in New Mexico divided by the total real and tangible property owned, rented, or used by the corporation. The average value of the property owned is determined by averaging property values at the beginning and the end of the year; the value of property rented is eight times the net annual rental rate; (2) the payroll factor is the amount paid as compensation by a corporation to its employees in New Mexico divided by the total compensation paid by the corporation in the tax period. Compensation means wages, salaries, commissions, and other forms of remuneration for personal services except wages paid to independent contractors; and (3) the sales factor is determined by dividing all gross receipts of the corporation from transactions and activity in the regular course of business in New Mexico during the tax period by the total sales of the corporation in the same period. Sales in this context exclude nonbusiness income. The three factors are then averaged to determine the percentage of the corporation's business income that is applicable to New Mexico taxes.

Finally, that portion of the nonbusiness income allocated to New Mexico is added to the apportioned business income to determine the net taxable income for a corporation operating in the State.

Resources, Processors, and Service Tax

Resources, processors, and service tax are mutually exclusive, that is, only one of the taxes is imposed on the potash produced.

The resource tax applies to unprocessed products sold. The amount taxable is determined by subtracting from the gross value of the products sold (1) gross sales to any government entity, (2) government royalty payments, and (3) any service charge on which a service tax is payable. This taxable value is then multiplied by one-half of 1 percent to arrive at the resource tax payment.

The processors tax is based on all processed products sold. The taxable value is determined by subtracting from the gross value of the processed products sold, the same deductions applicable to the resource tax. This value is then multiplied by one-eighth of 1 percent to arrive at the processors tax payment.

The service tax is imposed on a corporation that either mines or processes New Mexico potash owned by another company or not otherwise taxed with a resource or processors tax. The rate applied is the same as the rate used to calculate the resource or processors tax.

Severance Tax

The New Mexico severance tax is an excise tax based on an adjusted gross value of potash at the first marketable point.

The adjusted gross value is 33-⅓ percent of the proceeds realized from the sale of muriate of potash and sulfate of potash magnesia, in terms of standard grades, and 33-⅓ percent of products consumed in production of other potash products. The taxable value is equal to the adjusted value minus all royalties and up to 50 percent of expenses incurred.

In the case of mine-run salt that has a posted market price at the point of production, the taxable value is 40 percent of gross value minus all royalties and expenses incurred in hoisting, crushing, and loading.

The tax rate is 2-½ percent of the taxable value.

Property Tax

Property tax is imposed on all property used in connection with the production of potash. The tax is based on the property value of improvements and production. The property value of improvements, for tax purposes, is determined to be equal to the gross value of the products sold minus any royalties paid. The property value of the orebody, for tax purposes, is determined to be one-half the gross value of the products sold minus any royalties paid. The sum of the property value of improvements and production is then multiplied by a 33-⅓ percent assessment ratio to determine the taxable value. The taxable value is multiplied by the mill levy of the district in which the property is located to determine the property tax due the county. The mill levy assessment for the Carlsbad School District, Eddy County, was 19.8 mills in 1976.

Rents and Royalties on Federal Leases

Royalties on Federal land leases are negotiated on an individual case basis prior to issuance of the lease. Royalty rates range from 2 to 7-½ percent of the gross value of production. Current lease agreements require a payment of royalty on a minimum annual production beginning the sixth full calendar lease year.

Rental for potassium leases is 25 cents per acre (62 cents per hectare) for the first calendar year; 50 cents for the second, third, fourth, and fifth year; 1 dollar for the sixth and succeeding years. The rent is credited against the first royalties paid during the year for which rent was paid. In any one State, a lease may not exceed 2,560 acres (1,036 hectares). Also, holdings in leases may not exceed 25,600 acres (10,360 hectares) in one or more mining units.

If conditions warrant, the Secretary of the Interior can reduce the rental or minimum royalty.

Rents and Royalties on State Leases

Royalties on State lands are payable on a negotiated basis but not less than 5 percent of the sale value of the minerals produced then delivered to the nearest shipping point. Rental charge is negotiated but not less than 10 cents per acre (25 cents per hectare), with a minimum first year rental of $100.

ESTIMATION OF CURRENT MINE AND MILL CAPITAL AND OPERATING COSTS

In the financial evaluation of mineralized langbeinite and sylvite zones found within the WIPP site boundaries, the costs for three types of currently operating potash mining properties were estimated. The capital and operating costs estimated were then extrapolated to estimate costs of mines and processing plants of a size and design best suited to recover the deposits determined to be in the WIPP site. Personnel of the System Operations Group (SOG) and the Domestic Evaluation Group of the Minerals Availability System (MAS) toured seven beneficiation plants and underground potash operations in the Carlsbad area to evaluate potash mine and mill operations.

From the information gathered, operating and capital replacement costs of current potash mining and milling properties were estimated for operations applicable to WIPP site ores. Consideration was given to the economics of (1) using existing mill complexes with a new mine operation and (2) building a new mine-mill complex.

Operations for WIPP Study

Cost estimates were developed for a 13,000-ton-per-day (11,794-metric-ton-per-day) operation, a 7,000-ton-per-day (6,350-metric-ton-per-day) operation, and an 8,500-ton-per-day (7,-711-metric-ton-per-day) operation. A brief discussion of these three operations follows.

13,000-ton-per-day Mine-Mill

Estimates were made of a mining operation with a design capacity of about 13,000 tons per day (11,794 metric tons per day) to extract mixed ore (sylvite-langbeinite) and langbeinite ore (fig. 21) using a room-and-pillar mining system (figs. 22 and 23). Conventional mining (figs. 24 and 25) is used because of hardness of langbeinite and polyhalite minerals.

Entry is by vertical shafts with connecting drifts. Mobile trackless equipment is used to drill, load, and haul the ore to a conveyor system near the working areas. The blasting agent used is ANFO. Ore is transferred by conveyor to the production shaft and hoisted to the surface.

This operation includes a beneficiation system that recovers sylvite and langbeinite from 10,000 tons per day (9,072 metric tons per day) of mixed ore by heavy media separation and sylvite flotation. The mill is designed for ore having a clay content of 1 to 1.5 percent. In addition, the operation includes recovery of langbeinite from a 3,000-ton-ore-per-day (2,722-metric-ton-ore-per-day) langbeinite wash circuit. Sylvite and langbeinite fines are then used as feed material in the production of potassium sulfate (fig. 17). The sulfate plant was designed for an output capacity of 500 tons per day (454 metric tons per day) of product (fig. 26).

7,000-ton-per-day Mine-Mill

The 7,000-ton-per-day (6,350-metric-ton-per-day) room-and-pillar system includes the mining of two separate mineralized zones, one sylvite and the other langbeinite (fig. 27). The sylvite mining operation has a capacity of 4,000 tons per day (3,629 metric tons per day), while the langbeinite operation has a 3,000-ton-per-day (6,350-metric-ton-per-day) capacity.

Vertical shafts and connecting drifts provide access to each mineralized zone. Both mines utilize conventional room-and-pillar mining methods; drilling, blasting, and loading are done by mobile trackless equipment. A conveyor-belt system transports the ore to the production shaft. The ore is then hoisted to the surface and transported to the mill. Sylvite is recovered from the low-clay (1 to 1.5 percent) ore by flotation, and langbeinite is beneficiated from its ore by water leaching of the impurities. Sylvite and langbeinite fines are then used as feed material in the production of potassium sulfate (fig. 17). This sulfate plant has an output design of 225 tons per day (204 metric tons per day) of product.

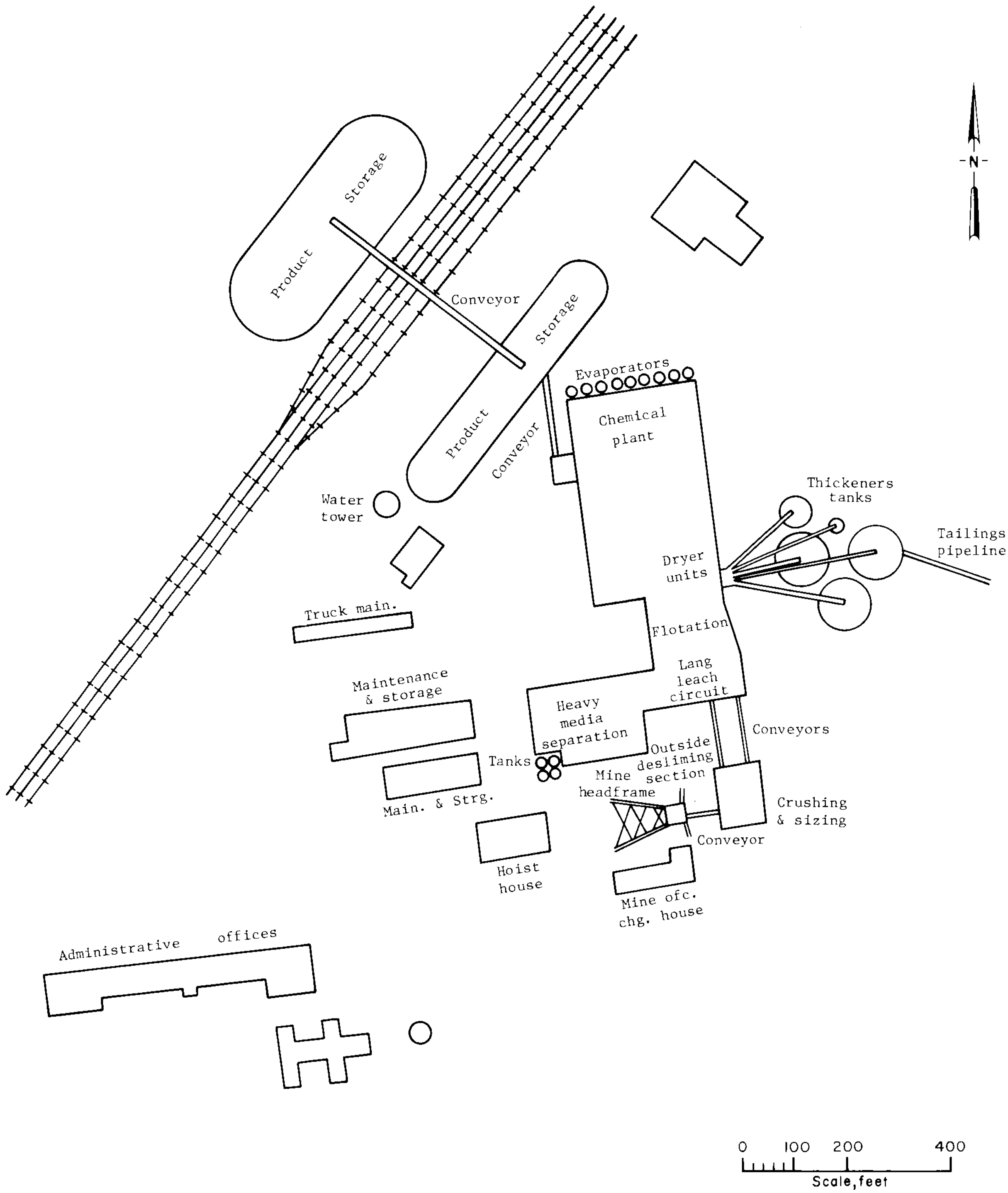

FIGURE 21.—Generalized layout of 13,000-tpd surface plant.

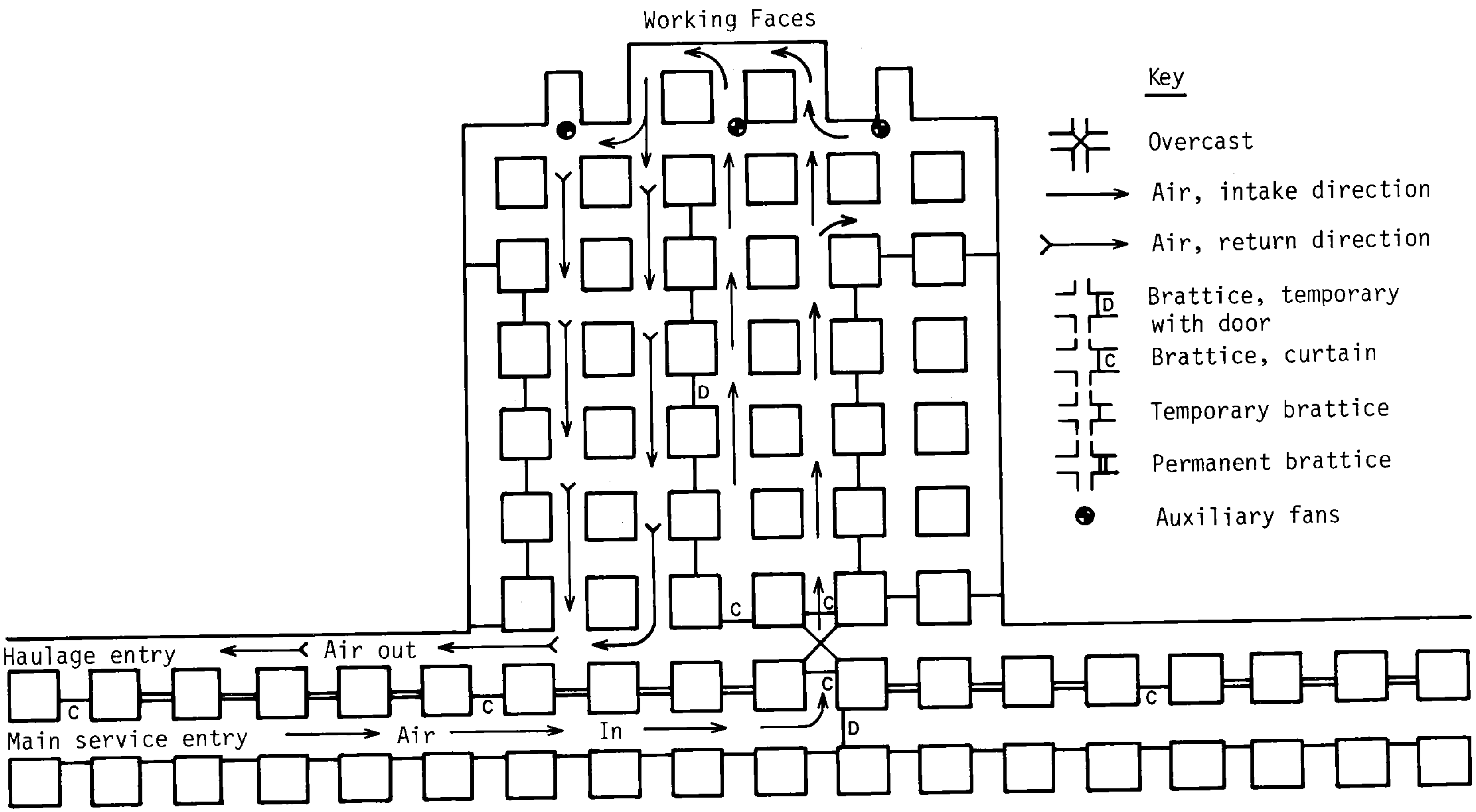

FIGURE 22.—Generalized plan of room-and-pillar mining system showing typical ventilation flow.

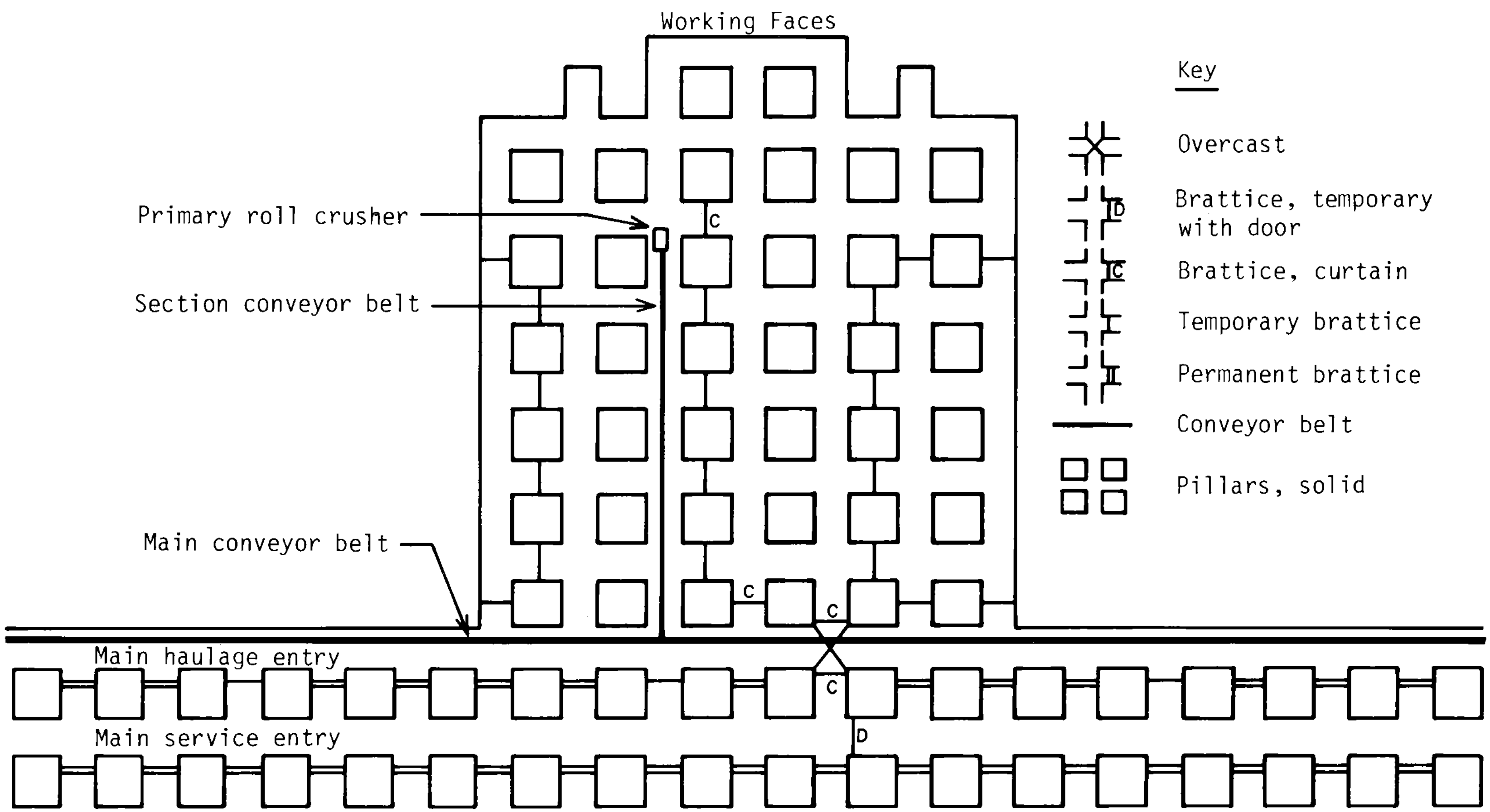

FIGURE 23.—Generalized plan of room-and-pillar mining system showing typical conveyor layout.

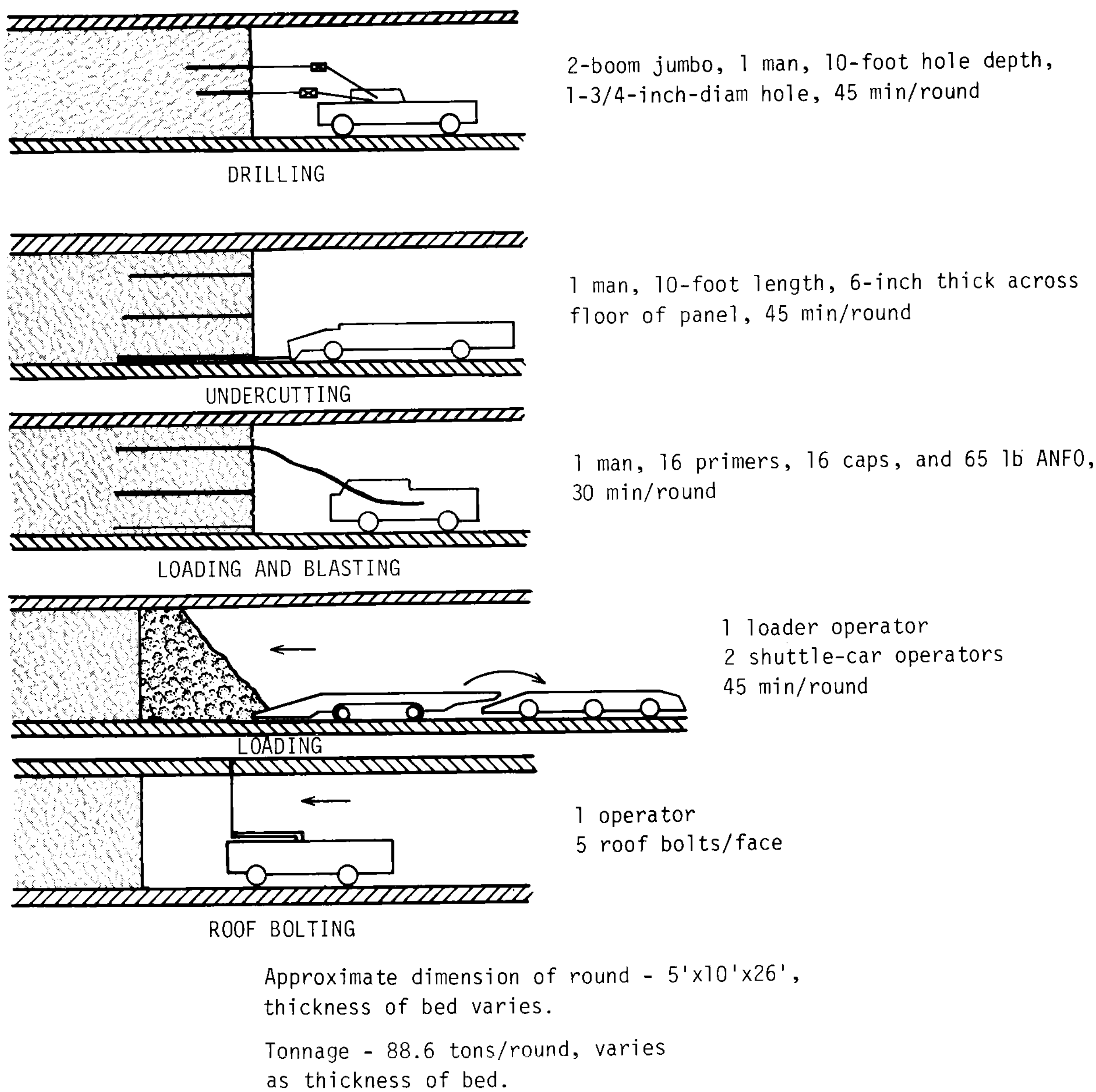

FIGURE 24.—Diagram of conventional mining sequence.

8,500-ton-per-day Mine-Mill

The third operation is an 8,500-ton-per-day (7,700-metric-ton-per-day) room-and-pillar mine (fig. 27) using, in part, a continuous mining system (fig. 28).The processing system is amenable to sylvite ore with a low langbeinite and polyhalite content. Approximately 75 percent of the mine output is extracted by continuous mining methods. In the continuous mining method, boring machines are used to extract and load the ore into shuttle cars which transport the mined ore to a conveyor system. The conveyor system transfers the mined ore to the shaft area to be hoisted to the surface.

The remaining 25 percent of the ore is mined by conventional mining method (fig. 24) much in the same manner as described previously.

The beneficiation system will recover sylvite from the ore by solution-crystallization methods (fig. 15). The ore is assumed to contain approximately 4 percent clays.

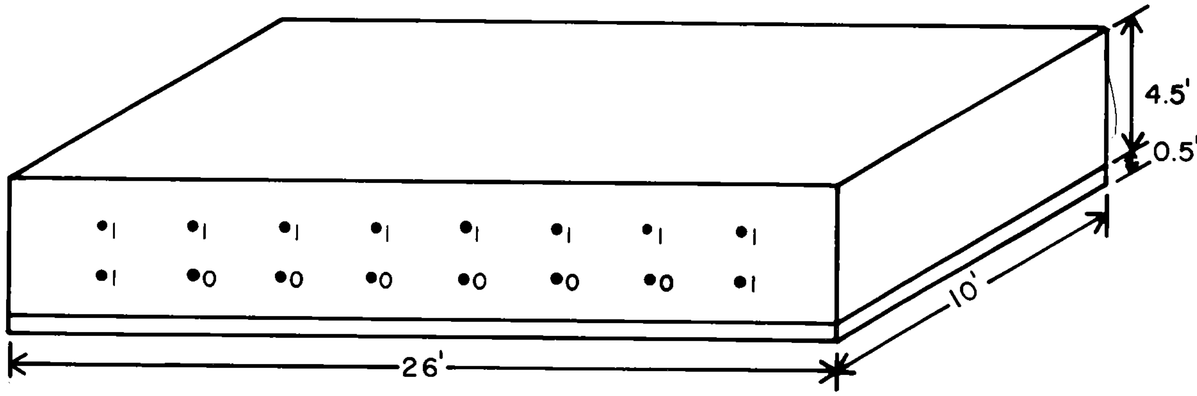

FIGURE 25.—Typical round dimensions for conventional mining. Approximate dimensions of broken round—5 by 26 by 9.5 ft (hole length 10 ft, breaking length 9.5 ft); volume, about 1,235 cu ft; 16 holes per round; hole diameter, 1¾ in; about 7 ft of ANFO per hole; 2 delays—0 and 1 millisec; tonnage 90 tons/round, assuming a powder factor of 0.7 lb explosive per ton of ore; 0.82 lb of ANFO per foot of drill hole = 90 lb of ANFO per round; primer—1- by 8-in 70-percent gelatin.

Mine Capital Costs

Mine capital investments are those costs associated with bringing an operation into production. The estimate is based on 1977 dollar values and is the cost for new (replacement) mine-mill complexes. Mine capital investments include exploration, development, mine plant, mine equipment, and working capital.

Exploration investments include the cost to define the mineralized zone in size, tonnage, and grade. This is done by estimating the cost of the drilling program and of a geologic and engineering analysis of the data obtained in the study area.

The development costs include all investments involved in preparing the operation for production. This includes the cost of constructing roads and railroads and providing utilities to the study area. The development costs also include all investments related to sinking of shafts, development of main haulage drifts, and preparing the working areas for full production.

Mine plant costs include all investments associated with permanent structures such as buildings, shops, and permanent systems such as ventilation and hoisting. These costs are considered to last the life of the operation.

Mine equipment costs include all costs associated with mobile underground equipment as well as support equipment on the surface. Other items such as tools, personnel-carrying vehicles, and an inventory of mining supplies are also included in this investment.

Working capital is the amount of money needed to account for all operating expenses between the time of first production and the time of initial revenue from the products. This cost was estimated to be equal to 6 months' operating cost for each mining system.

A summary of the estimated capital investments of the three potash operations is given in table 14.

Mine Operating Costs

Mine operating costs are the expenses incurred in extracting the ore and waste rock from the deposit. Operating costs are subdivided into three basic subcategories: (1) direct, (2) indirect, and (3) fixed costs. Factors that influence operating costs include hardness of rock, mining thickness, and variability of mineralization.

Direct costs are the investments related to the actual mining output. These costs include direct

TABLE 14.—Summary of estimated mine capital investments

Mine	Plant cost	Equipment cost	Exploration and development cost	Working capital	Total investment
13,000 tpd	$6,093,500	$ 8,114,400	$13,248,300	$3,906,200	$32,362,400
7,000 tpd	9,015,200	9,817,900	15,079,400	4,330,600	38,243,100
8,500 tpd	5,712,500	11,691,300	10,278,100	4,659,800	32,341,700

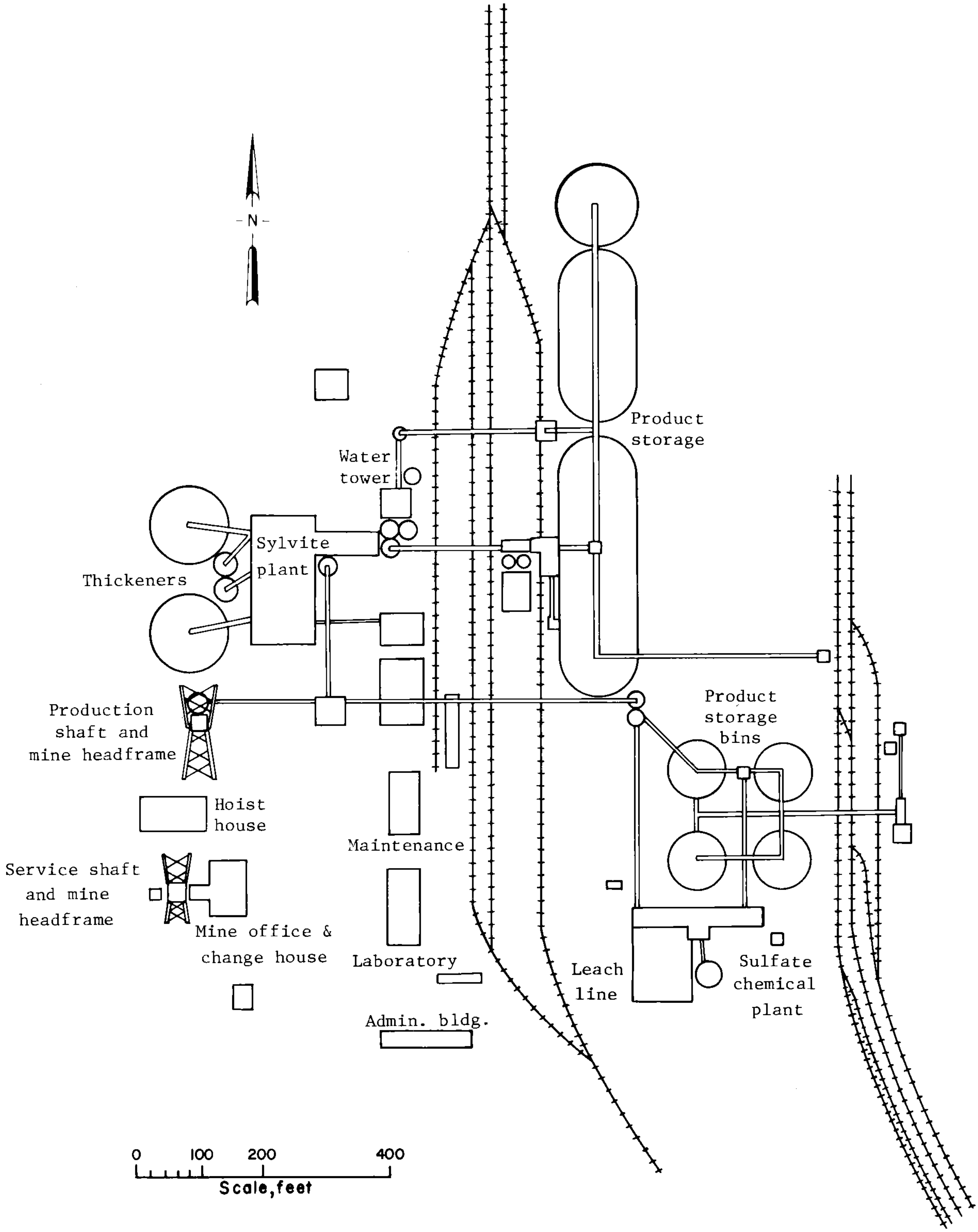

FIGURE 26.—Generalized layout of 7,000-tpd surface plant.

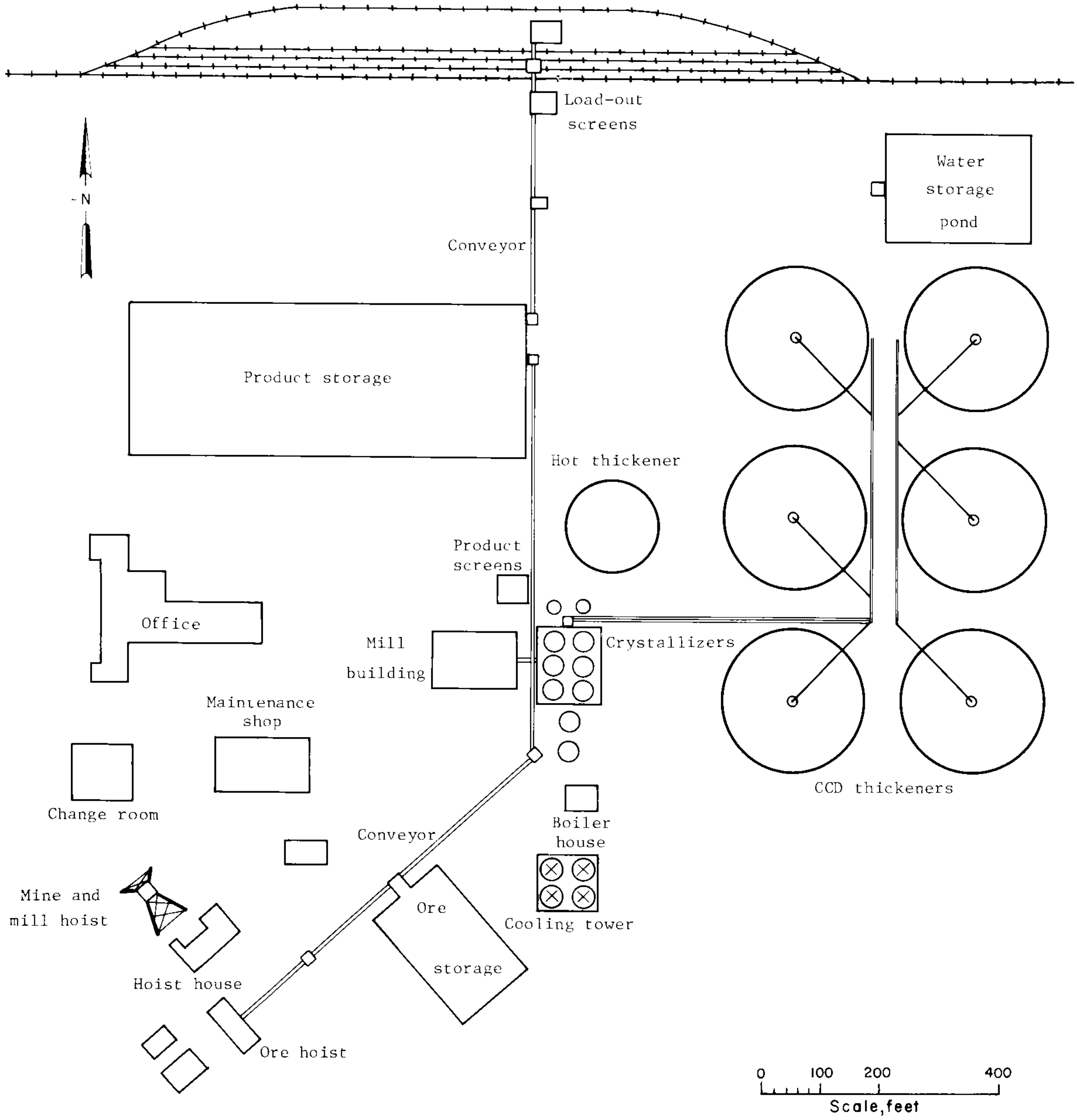

FIGURE 27.—Generalized layout of 8,500-tpd surface plant.

mine labor and supervision, maintenance labor and supervision, maintenance supplies, and material and utilities costs.

Indirect costs are related to support functions such as administrative technical labor, maintenance of the surface facilities, and general overhead. The direct and indirect costs would be affected by fluctuations in mine output, variability in rock and mineralized composition, and external economic factors.

Local taxes and insurance make up the fixed costs. These costs remain relatively constant and are not affected by physical or economic changes to the mining system.

A summary of the estimated operating costs for the three typical potash operations is shown in table 15.

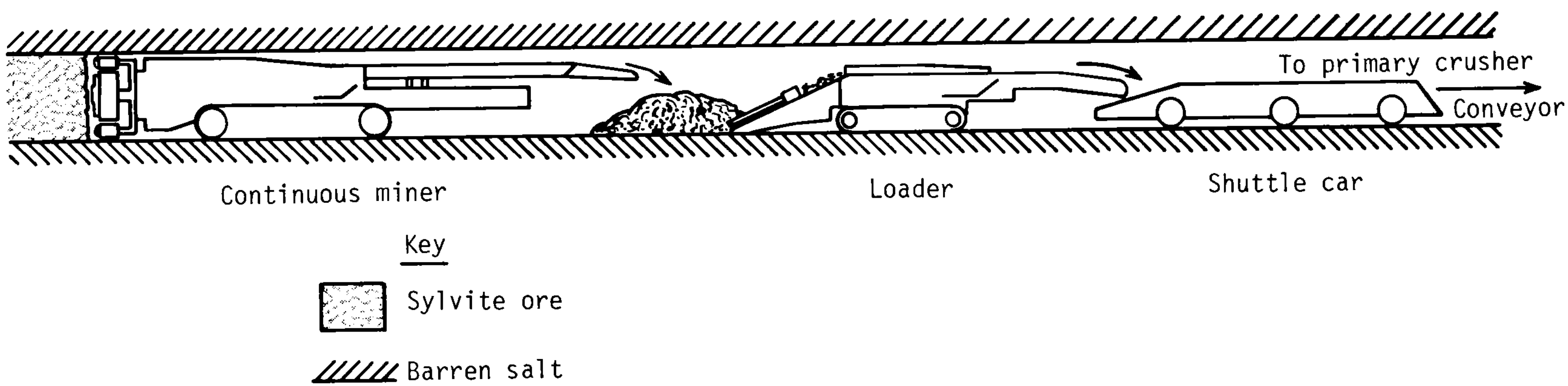

FIGURE 28.—Diagram of mining sequence using continuous miners. Continuous miner 4-man crew (1 operator, 1 loader operator, 2 shuttlecar operators). Continuous miner dumps ore onto floor; the ore is then loaded into shuttlecars by a loading machine. The shuttlecars then transfer the ore to primary crushing and conveyor belt system.

TABLE 15.—Summary of estimated mine operating costs

(Dollars per short ton of materials mined)

Mine	Direct	Indirect	Fixed	Total
13,000 tpd	1.80	0.31	0.05	2.16
7,000 tpd	3.00	.79	.10	3.89
8,500 tpd	2.97	.35	.10	3.42

Mill Capital Costs

The mill capital investments for the three operations include the cost for the plant, equipment, and working capital. The plant and equipment investment includes all costs associated with the permanent structures such as buildings and labs, processing equipment, and support systems such as piping, insulation, etc. The working capital is the amount of money needed to account for all operating expenses between the time of first production and the time of initial revenue from the products. These costs were estimated to be equal to 6 months' operating cost for each mill system.

Mixed Ore Plant

The mixed ore plant has a design capacity of about 10,000 tons per day (9,072 metric tons per day) of a mixed (sylvite-langbeinite) low clay (1 to 1.5 percent) ore. This ore is separated into sylvite and langbeinite products and fines are fed into a sulfate plant designed to produce 500 tons per day (464 metric tons per day) of product. Langbeinite ore is processed in a 3,000-ton-per-day (2,722-metric-ton-per-day) langbeinite leach circuit.

The three mill sections of this plant were evaluated separately so that any portion of the plant could be used where variations from the models were needed for WIPP ore. In addition, the 10,000-ton-per-day (9,072-metric-ton-per-day) mixed-ore circuit was costed in two parts: a sylvite flotation circuit and a langbeinite float-heavy media circuit, allowing each circuit to be used separately if the WIPP site ore was a langbeinite or sylvite ore rather than a mixed ore.

Sylvite Ore Flotation Plant

The sylvite flotation plant has a design capacity of about 4,000 tons per day (3,629 metric tons per day) of a low-clay (1 to 1.5 percent) sylvite ore beneficiated by flotation. The concentrate fines from this operation and the fines from a 3,000-ton-per-day (2,722-metric-ton-per-day) langbeinite leach circuit concentrate are fed to a sulfate plant designed to produce 225 tons per day (204 metric tons per day). As with the mixed ore plant, the three mill sections of this plant were evaluated separately so that any portion of the plant could be used if variations from the models were needed to evaluate a hypothetical plant for WIPP deposits.

TABLE 16.—Summary of estimated plant capital investments

Plant description	*Costs*
13,000-tpd mill complex:	
10,000-tpd mixed (sylvite-langbeinite) ore circuit employing heavy media separation flotation	$26,789,400
3,000-tpd langbeinite leach circuit	4,748,100
Sulfate plant—500 tpd product	24,161,700
Total	$55,708,200
7,000-tpd mill complex:	
4,000-tpd sylvite flotation circuit	$20,325,500
3,000-tpd langbeinite leach circuit	5,407,200
Sulfate plant—225 tpd product	15,242,800
Total	$40,975,500
8,500-tpd mill complex:	
8,500-tpd sylvite crystallization plant	$52,899,500

Sylvite Ore Crystallization Plant

A sylvite crystallization plant was designed to beneficiate 8,500 tons per day (7,711 metric tons per day) of a sylvite ore containing approximately 4 percent clays by dissolution and recrystallization of the sylvite.

The three mills have separate mill circuits for different ore treatment in the 10,000- and 7,000-ton-per-day operations and for potassium sulfate production. Each circuit was evaluated separately so that any portion of the plant could be used for estimating capital and operating costs of mine-mill complexes best suited for recovery of the WIPP deposits.

A summary of the estimated capital investments is shown in table 16.

Mill Operating Costs

Mill operating costs are the expenses incurred in recovering the potash from the mined ore. Operating costs are subdivided into three basic subcategories: (1) direct, (2) indirect, and (3) fixed costs. Direct costs are those related to the actual production such as labor, water, electricity, and reagent requirements. Indirect costs are expenses related to support functions such as administration, technical labor, maintenance of the facilities, and general overhead. The direct and indirect costs are affected by mill input, variability of the rock, mineral composition, and external economic factors. Local taxes and insurance comprise the fixed costs; these costs remain relatively constant. A summary of the estimated operating costs for the three mill complexes is shown in table 17. A separate cost is shown for the different ore treatment circuits within the mill complex.

TABLE 17.—Summary of estimated annual mill operating costs

Plant description	Annual costs
13,000-tpd mill complex:	
10,000-tpd mixed (sylvite-langbeinite) ore circuit employing heavy-media separation flotation	$9,755,900
3,000-tpd langbeinite leach circuit	1,550,400
Sulfate plant—500 tpd product	5,563,000
Total	$16,869,300
7,000-tpd mill complex:	
4,000-tpd sylvite flotation circuit	$5,038,600
3,000-tpd langbeinite leach circuit	1,498,200
Sulfate plant—225 tpd product	2,267,500
Total	$8,804,300
8,500-tpd mill complex:	
8,500-tpd sylvite crystallization plant	$8,294,000

Utility (Infrastructure) Costs

Utility costs include estimated costs of required roads, railroad spurs, water supply systems, electric utility lines, and natural gas supply lines.

Water

The water supply for a new potash-producing mine-mill complex is a critical item. Possible sources of water considered for potash refining are the Pecos River and aquifers in the Capitan Reef, Rustler, and Ogallala Formations. The best quality water found in the region is produced from the Caprock area of the Ogallala Formation, 25 to 30 miles (40 to 48 kilometers) to the northeast of the study area. This source presently supplies the potash industry with about two-thirds of its water and is being used increasingly by the industry. The water quality is good, ranging from 500 to 600 parts per million total dissolved solids.

This study assumes that Caprock area-Ogallala Formation water will be available for a new refinery. Adequate water rights are available and well site leases are obtainable on most of the Caprock field. The State of New Mexico owns most of the producing area. Present wells are 200 to 250 feet (61 to 76 meters) deep and are spaced on about ¼-mile (0.4-kilometer) centers. Each well, if properly designed and developed, can produce 200 to 250 gallons per minute (756 to 945 liters per minute) for 30 to 40 years. It is estimated that a pipeline from the Caprock field to a new refinery in the study area would be 30 miles (48 kilometers) long. A 600-gallon-per-minute (2,268-liter-per-minute) water system is estimated to cost about $3.5 million installed, a 3,600-gallon-per-minute (13,608-liter-per-minute) water system about $6.9 million.

Electric Power

Electric power is supplied in the area by the Southwestern Public Service Co. A typical load for a potash mine and refinery is estimated to be about 6,000 kilowatts, with usage at approximately 4 million kilowatt-hours per month. All power supplied to the potash industry is by 69-kilovolt powerlines. These lines are estimated to cost approximately $35,000 per mile. The voltage is reduced at a substation near the mine or refinery. The present rate for power is estimated to be approximately 2 cents per kilowatt-hour. A 69-kilovolt line to the Kermac operation, capable of handling the added capacity required for a new mine and mill complex, could be extended into the study area.

Natural gas for a refinery could be supplied by the Gas Company of New Mexico. A 6-inch (15.2-centimeter) line presently supplies the Kermac refinery and has sufficient capacity to supply an additional refinery. The normal gas usage by existing refineries ranges from 15,000 to 100,000 million Btu per month. The present intrastate commercial rate for gas in New Mexico is estimated to range from $1.15 to $1.35 per million Btu. To supply a new refinery, a 4-inch (10.2-centimeter) pipeline from the Kermac operation south to the study area would cost approximately $27,000 per mile.

All commercial natural gas customers except Kermac have alternate emergency supplies of burner fuel to use in case of gas curtailment. Kermac stores propane for an alternate emergency fuel. Burner fuels are stored in 20,000- to 30,000-gallon (75,600- to 113,400-liter) storage tanks at the potash refineries. Estimated costs for alternate fuels are $2.36 per million Btu for No. 2 burner fuel and $2.72 per million Btu for propane.

Coal could be used instead of natural gas in the refining of potash. Steam could be produced in coal-fired boilers and its heat used for the leaching, drying, and granulating steps of the refining process. It is estimated that the extra capital cost to build a coal-fired refinery over the cost of building a natural-gas-fired refinery is 15 to 25 percent. Coal for use in the Carlsbad area probably could be supplied from a coal mine near Gallup, N. Mex. It is estimated that this 12,000 Btu per pound (26,400 Btu per kilogram) of coal would cost about $0.78 per million Btu delivered. Table 18 shows a comparison of fuel costs between coal and natural gas for a langbeinite wash plant.

Table 19 shows a comparison of capital investment necessary to accommodate natural gas and coal as fuel for a langbeinite wash plant.

TABLE 18.—Comparison of fuel cost for a 3,000-tpd langbeinite wash plant

Fuel	Cost/MM Btu	Total MM Btu/year	Total cost/year
Natural gas	$1.15	160,313	$184,360
Coal	.78	160,313	125,040

TABLE 19.—Comparison of capital investment: coal versus natural gas in a 3,000-tpd langbeinite wash plant

Fuel utilized	Percent markup	Base price	Total cost	Additional cost
Natural gas	0	$4,748,100	$4,748,100	$0
Coal	15	4,748,100	5,460,300	712,000
Coal	20	4,748,100	5,697,700	949,600
Coal	25	4,748,100	5,935,100	1,187,000

The savings that would accrue by using coal instead of natural gas is $59,320 per year. The dollars of fuel saving, as simple interest on the additional investment necessary to use coal, provide 8.3 percent interest on the additional investment for the best use and 5 percent for the most costly use.

Access to the Area

Access into the area would be north from State Highway 128 or south from U.S. Highway 180. If an all-weather road were to be constructed into the WIPP site, it probably would be built south from Highway 180, which is better suited to heavy haulage than Highway 128 and provides a more convenient route to Carlsbad and other centers. A 24-foot (7.3-meter) wide road with a 4- to 6-inch (10- to 15-centimeter) crushed stone base, covered by 2 inches (5 centimeters) of hot-mixed asphalt without structures, is estimated to cost about $40,000 per mile ($24,860 per kilometer).

A rail spur built into the area would probably be serviced by the Santa Fe Railroad as an extension of its lines. A spur could be built as an extension of the Nash Draw mine or Kermac spurs (fig. 3). In either case, about 7 miles (11.3 kilometers) of standard-gage rail line would be required. Where no major structures are needed, such as bridges, culverts, etc., costs would range from $400,000 to $500,000 per mile ($248,600 to $310,800 per kilometer) of track.

ESTIMATES OF HYPOTHETICAL MINING UNITS IN THE WIPP SITE

The WIPP potash deposits valuations are based on the economic analysis of hypothetical mine-mill complexes by extrapolation, where necessary, of the capital and operating costs of existing operations estimated in the previous section. These operations would be located where it is determined to be most economic and convenient for mining recovery and processing of the deposits delineated by USGS (fig. 29).

For the purposes of this study, a deposit or group of deposits, suitably located for mine recovery from one operation, is designated as a Mining Unit. Four such groups are in the site and were labeled A, B, C, or D. Each deposit size (tonnage), thickness, and lateral extent were estimated by USGS at three potash cutoff grades: high, medium, and low. The medium and low grade deposits include all material in the higher grade deposits. Each grade cutoff is considered a subunit, making 12 subunits—one each with (1) the high grade, (2) the medium grade, and (3) the low grade for each of the groups A–D. These were then designated as hypothetical operations (Mining Unit A–1, A–2, A–3, B–1, B–2, etc.) to evaluate the economics of potash recovery in the deposit or group of deposits at various cutoff grades and tonnages. The boundaries of the WIPP site were disregarded when evaluating the deposits in a Mining Unit. The value of only that portion of the deposit or deposits within the WIPP site was then determined to be a loss chargeable to the WIPP facility provided its removal did not affect the commercial viability of the remainder of the deposit.

U.S. Geological Survey tonnage and grade data were adjusted to a minimum mining height of 4.5 feet (1.4 meters) and 85-percent expected mining extraction of the ore. Where the thickness of the mineralized zone is less than 4.5 feet (1.4 meters), the ore tonnage was increased by assuming the mineralized material would be recovered in a minimum 4.5-foot (1.4-meter) mine heading. Increasing the tonnage in this way necessitated grade adjustments since the additional material is assumed to be barren of potash values. Langbeinite grades were also adjusted due to an assumed dilution by 6 inches (15 centimeters) of barren rock, because the undercut used in the mining recovery would be made below the comparatively hard langbeinite zone. Measured and indicated ore tonnages were summed and included as one value.

Hypothetical mine-mill complexes suitable to the type of mineralization and desired capacity were developed for each Mining Unit. Results from the metallurgical testing were analyzed to determine which mineralized zones were amenable to beneficiation for each subunit. Other criteria, such as current and anticipated market demand, energy requirements and usage, and available water supplies, were also considered.

The least costly method of mining and processing was then applied to each Mining Unit. This required the examination of alternative methods of extracting and processing each Mining Unit, with more than one system being tried. Consideration was given both to the economics of constructing new mill complexes and to utilizing portions of existing plants in the Carlsbad area where sulfate was one of the products. The capacity, using either or both of the existing sulfate plants, was examined. A brief description of the methods, assumptions, and considerations used in evaluating each Mining Unit at the three grade cutoffs follows.

MINING UNIT A

Mining Unit A deposits consist of the western lobe of the 10th and 8th sylvite ore zones and the langbeinite of the western lobe of the 2d and 3d ore zones. Grade and tonnage information for the three subunits are given in table 20.

Mining Unit A–1

Mining Unit A–1 consists of the western lobe of the 10th and 8th sylvite ore zones. No langbeinite mineralization containing the high grade (8 percent K_2O) langbeinite is in this subunit. Grade and impurity information is shown in table 21.

As shown in table 21, the resource contains comparatively high grade sylvite, which could be beneficiated by either flotation or solution-crystallization methods. Kainite and carnallite are not present in sufficient quantity to signif-

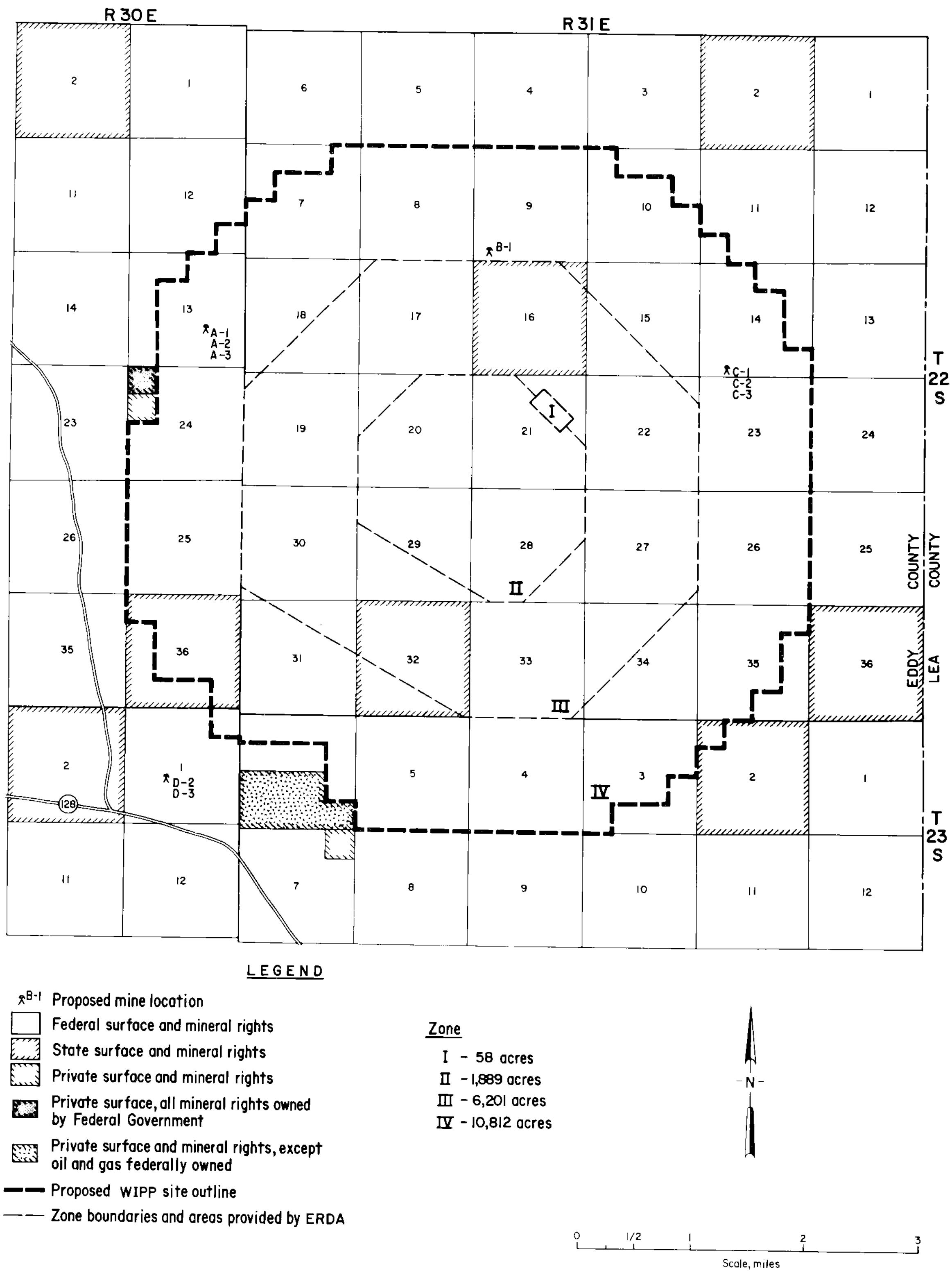

FIGURE 29.—Approximate locations of Mining Units.

TABLE 20.—Average K_2O grade of sylvite and langbeinite deposits and total tonnage, Mining Unit A

Ore zone	Weighted average grade (percent)	Total tonnage (MM)	Percent inside WIPP site	Tonnage inside WIPP site (MM)
Mining Unit A-1:				
Sylvite:				
10th, west lobe	13.18	47.74	43	20.53
8th	13.78	9.86	70	6.90
Total	13.33	57.60	47.6	27.43
Mining Unit A-2:				
Sylvite:				
10th, west lobe	11.32	57.26	46	26.34
8th	11.24	41.06	62	25.46
Total	11.28	98.32	52.7	51.80
Langbeinite:				
2d, west lobe	4.60	14.18	100.0	14.18
3d	4.76	26.81	90.0	24.13
Total	4.70	40.99	93.5	38.31
Mining Unit A-3:				
Sylvite:				
10th, west lobe	10.65	69.91	45	31.46
8th	9.56	65.11	65	42.31
Total	10.03	135.02	54.6	73.77
Langbeinite:				
2d, west lobe	4.04	34.40	50	17.20
3d	4.15	40.85	80	32.68
Total	4.12	75.25	66.3	49.88

icantly affect flotation recovery. Kieserite, while present, should not affect flotation. Microscopic tests and studies by the Salt Lake City Research Center indicate that liberation of the sylvite from the Kieserite for similar ores is possible—insolubles are high and would lower recovery and grade in a flotation circuit. To increase the recovery and grade, additional slimes treatment facilities would probably be needed to process this resource in the flotation circuit. In a solution process the insolubles would not be a major problem and the kieserite, at about 2.3 percent SO_4, is within limits that can be processed commercially.

Based on these resource and mineralization studies, three hypothetical mine-mill circuits were evaluated.

1. The deposit mined at a rate of 9,484 tons per day (8,600 metric tons per day) using a room-and-pillar mining method, then shipped to a currently operating mill for processing. Mining costs adjusted from the 13,000-ton-per-day (11,800-metric-ton-per-day) operation to correct for the reduced mining height. Processing estimated by sylvite flotation recovery in an existing plant. No additional slimes treatment circuits added; K_2O recovery of 77 percent estimated.
2. The same conditions as in evaluation 1 but with the addition of circuits for desliming and brine recovery; K_2O recovery of 81 percent assumed.
3. Analysis was made of a mine having a 7,500-ton-per-day (6,800-metric-ton-per-day) capacity with the material treated for recovery of sylvite by solution and crystallization in an existing plant and with an estimated K_2O recovery of 81.2 percent.

Of the three mine and mill circuits considered, the first option, with the use of a flotation circuit and without additional slimes treatment, was found to be the most economic.

Mining Unit A–2

This subunit includes sylvite deposits of the northwest lobe of the 10th and 8th ore zones and langbeinite deposits from the west lobe of the 2d and 3d ore zones. Descriptions of resource, grade, and impurities are shown in tables 20 and 21.

The langbeinite resource, because of its low grade, would be difficult to process to market grade, as indicated by the Salt Lake City Research Center tests. It contains more polyhalite and insolubles than the Salt Lake City test samples (table 2). Calculations, assuming that the recovery of langbeinite and polyhalite will be comparable (83 percent) and with 80 percent of insolubles removed, show that the maximum concentrate will be 20.8 percent K_2O (below the market standard of 22 percent). They further show that such a concentrate would contain 10.1 percent impurities, which is above the maximum allowable market amount of 4 percent. This langbeinite resource does not appear amenable to processing. For this reason, the langbeinite mineralized zones were not evaluated.

TABLE 21.—Mineralogical compositions of sylvite and langbeinite deposits, Mining Unit A

(Weight-percent)

Mineral	Mining Unit A-1	Mining Unit A-2		Mining Unit A-3	
	Sylvite	Sylvite	Langbeinite	Sylvite	Langbeinite
Sylvite	13.33	11.28	----	10.03	----
Langbeinite	----	----	4.70	----	3.78
Insolubles	3.48	5.19	1.52	5.54	1.67
Polyhalite	.99	2.14	1.46	2.29	1.80
Kainite	.91	.64	.85	.57	.80
Kieserite	3.04	2.14	0	1.90	0
Leonite	.62	.44	1.43	.39	1.43
Carnallite	1.19	3.27	0	3.75	0

The sylvite resource, while of lower grade than that in Mining Unit A–1, could be concentrated by either flotation or solution-crystallization methods. Kainite is not present in sufficient quantity to significantly affect flotation, and microscopic tests indicate the sylvite can be liberated from the kieserite. The amount of slimes would require modification of the previously discussed mill operations for extra slimes treatment. The amount of carnallite is only slightly greater than 2 percent and could be tolerated in a solution-crystallization circuit with an estimated K_2O recovery slightly lower than that expected for Unit A–1; carnallite may also affect the recovery in a flotation circuit.

Based on these assumptions, sylvite resources in Mining Unit A–2 were evaluated for two mining and milling circuits.

1. The deposit would be mined at a rate of 9,484 tons per day (8,600 metric tons per day) using a room-and-pillar mining method. The material is then shipped to an existing mill based on flotation recovery. The existing plant would be modified for extra desliming and brine recovery circuits. A K_2O recovery of 80 percent is estimated.
2. The deposit would be mined at a rate of 7,500 tons per day (6,800 metric tons per day). The material would be recovered by solution-crystallization methods from an existing plant. The plant would be modified to include a carnallite circuit. A recovery of 80.0 percent is assumed.

The mine and flotation complex of evaluation 1, as modified with additional desliming and brine recovery, was the more economic.

Mining Unit A–3

Mining Unit A–3 is a combination of sylvite deposits in the 8th ore zone and the western lobe of the 10th ore zone and langbeinite deposits occurring in the western lobe of the 2d and 3d ore zones. These mineralized zones include the zones mentioned previously in Mining Units A–1 and A–2, plus additional lower grade material. Resource and impurities within these mineralized zones are shown in tables 20 and 21.

The langbeinite resource is low grade and contains significant amounts of polyhalite and insolubles. Calculations, assuming a recovery of langbeinite and polyhalite of 83 percent and 90 percent removal of insolubles, respectively, indicate that the concentrate would have a grade of 19.9 percent K_2O equivalent as langbeinite and 12 percent impurities. This amount of impurities is above the maximum allowable market value of 4 percent. Thus, the langbeinite resource does not appear amenable to processing. The sylvite resource, while of lower grade than that in Mining Unit A–1, could be concentrated by either flotation or solution-crystallization methods. Kainite is not present in sufficient quantity to significantly affect flotation, and kieserite should not affect flotation appreciably since microscopic tests indicate the sylvite can be liberated. The amount of slimes is high enough to necessitate modification of mill models for extra slimes treatment. The carnallite may affect recovery in a flotation or solution-crystallization plant.

Two mine and mill operations were proposed. The proposed mine for the extraction of the sylvite mineralized zones would be identical to that in the previous section on Mining Unit A–1 but with extended project life because of the larger resource amount.

The related mill circuits would be:

1. Sylvite would be recovered by flotation through the flotation circuit of an existing plant, modified with additional desliming and brine recovery circuits, and would provide an estimated K_2O recovery of 80 percent.
2. Sylvite would be recovered by a solution and crystallization method in an existing operation and would provide an estimated K_2O recovery of 80 percent. The plant would be modified to include a carnallite circuit.

A flotation circuit, modified with additional desliming and brine recovery, was determined to be the more economic milling circuit.

MINING UNIT B

The hypothetical Mining Unit B was designed to extract and process material from the 4th and 5th langbeinite-mineralized ore zones. Following is a brief description of the methods and assumptions used in the evaluation of hypothetical Mining Unit B.

Mining Unit B–1

Mining Unit B–1 (highest grade) includes langbeinite deposits of the 4th and 5th ore zones. Average mining grade of langbeinite (K_2O equivalent) and reserve within the WIPP site are given in table 22.

A hypothetical 6,000-ton-per-day (5,400-metric-ton-per-day) capacity langbeinite leach plant, constructed at the site, was assumed. In addition to a langbeinite product, this plant would produce enough langbeinite fines to supply sulfate sections at two existing plants. A conventional

TABLE 22.—Average K_2O grade of langbeinite and total tonnage, Mining Unit B

Ore zone	Weighted average grade (percent)	Total tonnage (MM)	Percent inside WIPP site	Tonnage inside WIPP site (MM)
Mining Unit B-1:				
4th	9.18	72.23	65	46.95
5th	7.20	7.55	20	1.51
Total	9.11	79.78	60.8	48.46
Mining Unit B-2:				
4th north	7.69	126.33	57	72.01
5th	4.59	27.85	85	23.67
Total	6.92	154.18	62.1	95.68
Mining Unit B-3:				
4th north	6.87	164.83	60	98.90
5th	4.02	30.23	82	24.89
Total	6.30	196.07	63.1	123.79

mining system with adjusted costs to mine a 4.5-foot (1.4-meter) bed of langbeinite was used (see appendix for further detail). All costs of mining sylvite needed for sulfate production were added to the costs of sulfate production.

The amount of polyhalite, insolubles, and kieserite present in the resource (table 23) are within current processing tolerances. Calculations indicate that a concentrate grade of 21.6 percent K_2O equivalent or better could be made including 4 to 4.5 percent impurities. These calculations assume a comparable recovery of polyhalite and langbeinite and 80 percent removal of insolubles. These grade and impurity content calculations indicate a concentrate meeting market specifications could be recovered. This resource is leachable and the concentrate fines could then be combined with purchased sylvite fines to make a sulfate product.

Mining Units B–2 and B–3

Mining Units B–2 (medium grade) and B–3 (lowest grade) include langbeinite deposits from the 5th ore zone and the north portion of the 4th ore zone. Average mining grades of langbeinite (K_2O equivalent) and total resources in the WIPP site are given in table 22.

Langbeinite resources in both Units B–2 and B–3 (table 22) are low grade and contain amounts of high impurities so that, by calculation, neither

TABLE 23.—Mineralogical compositions of langbeinite deposits, Mining Unit B

(Weight-percent)

Mineral, average comp.	Mining Unit B-1	Mining Unit B-2	Mining Unit B-3
Langbeinite	9.11	6.92	6.30
Insolubles	2.07	3.15	3.21
Polyhalite	1.38	1.41	1.42
Kainite	1.10	.98	1.01
Kieserite	.47	.66	.68
Leonite	.93	1.05	1.00
Carnallite	1.17	2.23	1.91

would meet market grades through current commercial processing. The calculation assumes comparable recoveries of langbeinite and polyhalite in the concentrate and 80 percent of the insolubles removed. For Unit B–2, calculations indicate the best product grade that could be expected would be about 21 percent K_2O with 7 percent impurities, and for Unit B–3 about 20.9 percent K_2O with 7.8 percent impurities.

These calculations indicate the langbeinite resources in both Units B–2 and B–3 are not amenable to commercial processing, and no economic evaluations were performed in either case.

MINING UNIT C

Mining Unit C deposits consist of the eastern lobe of the 10th ore zone, the 9th sylvite-mineralized ore zone, and the east lobe of the 2d and 10th ore zones. Grade and tonnage information on resources for the three subunits are given in table 24.

Mining Unit C–1

Mining Unit C–1 includes the 10th and 9th mineralized zones, which contain sylvite miner-

TABLE 24.—Average K_2O grade of sylvite and langbeinite deposits and total tonnage, Mining Unit C

(Weight-percent)

Ore zone	Weighted average grade (percent)	Total tonnage (MM)	Percent inside WIPP site	Tonnage inside WIPP site (MM)
Mining Unit C-1:				
Sylvite:				
10th	14.51	31.20	50	15.60
9th	12.65	.64	100	.64
Total	14.44	31.84	51	16.24
Langbeinite:				
10th	8.12	6.90	100	6.90
2d	8.06	11.60	78	9.05
Total	8.08	18.50	86.2	15.95
Mining Unit C-2:				
Sylvite:				
10th east lobe	11.57	51.75	60	31.05
9th	10.16	5.44	100	5.44
Total	11.36	57.19	63.8	36.49
Langbeinite:				
10th east lobe	4.91	50.74	86	43.64
2d east lobe	5.49	41.81	90	37.63
Total	5.18	92.55	87.8	81.27
Mining Unit C-3:				
Sylvite:				
10th east lobe	10.59	61.28	71	43.51
9th	9.56	9.36	100	9.36
Total	10.41	70.64	74.8	52.87
Langbeinite:				
10th east lobe	4.38	52.03	95	49.43
2d east lobe	5.13	52.53	95	49.90
Total	4.76	104.56	95	99.34

TABLE 25.—Mineralogical compositions of sylvite and langbeinite deposits, Mining Unit C

(Weight-percent)

Mineral, average composition	Mining Unit C-1		Mining Unit C-2		Mining Unit C-3	
	Sylvite	Langbeinite	Sylvite	Langbeinite	Sylvite	Langbeinite
Sylvite	14.44	----	11.36	----	10.41	----
Langbeinite	----	8.08	----	5.18	----	4.76
Insolubles	4.7	2.59	3.98	2.40	3.82	2.40
Polyhalite	2.06	1.64	1.9	1.55	1.81	1.52
Kainite	----	1.51	----	1.71	----	1.61
Kieserite	9.02	----	6.52	.55	6.25	.49
Leonite	----	.88	----	1.45	----	1.45
Carnallite	.39	----	----	----	----	----

alization. In addition, this subunit contains langbeinite mineralization in the 10th and 2d ore zones.

Analysis of the langbeinite resource in Unit C–1 indicates that the deposits are not amenable to current commercial processing to market standards. The optimum product would contain 20.5 percent K_2O and 9.6 percent impurities. For these reasons, the resource is not amenable to commercial processing.

The sylvite from Unit C–1 would be recoverable by flotation. The amount of kieserite in the resource (table 25) contains enough SO_4 (6 percent) to cause difficulties in a crystallization circuit but would not affect a flotation circuit. Bureau of Mines tests indicate the sylvite would be liberated. The insolubles content of the resource is sufficiently high to require extra desliming and brine recovery circuiting.

The mining system used for this hypothetical subunit has a design capacity of 3,720 tons per day (3,400 metric tons per day) resulting in a project life of 15 years. The costs estimated for this mining system were adjusted from the sylvite-mine portion of the 7,000-ton-per-day (6,398-metric-ton-per-day) operation discussed in a previous section. The mined material would be shipped to an existing mill for processing to a muriate product.

Mining Unit C–2

This subunit includes sylvite products in the 9th ore zone and the eastern portion of the 10th ore zone, and langbeinite deposits in the eastern lobe of the 2d and 9th ore zones. Average mining grade (K_2O equivalent) is 11.36 percent of sylvite and 5.18 percent for langbeinite (table 24). Analysis of the impurities in the langbeinite (table 25) indicate the potash product would not meet market standards. A 20.6 percent K_2O grade product containing 8.8 percent impurities is estimated. For these reasons, the langbeinite mineralized zones were not evaluated.

The sylvite deposit in Unit C–2 is estimated to be processable by flotation. The amount of kieserite is at the high (2.99 percent SO_4) limits for a standard solution circuit, but microscopic tests indicate that the sylvite can be liberated from the kieserite, which is necessary for flotation separation. The insolubles content is high enough to require extra desliming and brine recovery circuiting over that designed in the operations described previously.

In evaluating Mining Units C–2 and C–3, a larger mining capacity [9,486-ton-per-day (8,604-metric-ton-per-day) operation] was estimated for the sylvite ore zones because of the larger resource tonnage at the medium and low grade levels. Evaluation at this increased capacity resulted in lower cost per unit. A sylvite flotation mill was modified to include brine recovery and extra desliming, resulting in a K_2O recovery of 80 percent.

Mining Unit C–2 was found to be slightly more profitable than Mining Unit C–1 (the higher grade) because economy of scale in the mine and the mill in this case results from the capacity increase. For this reason, total resource revenue on subunits C–1 and C–2 is equal to the revenue from the medium grade deposit unit (Mining Unit C–2).

Mining Unit C–3

Mining Unit C–3 (lowest grade) includes sylvite deposits in the 9th ore zone and the eastern lobe of the 10th ore zone and langbeinite deposits in the eastern lobes of the 2d and 3d ore zones (table 24). Average mining grade (K_2O equivalent) was 10.41 percent for sylvite and 4.76 percent for langbeinite. Analysis of the impurities in the langbeinite deposits indicates that it would yield a product containing 20.5 percent K_2O grade with 9.6 percent impurities that does not meet market standards (table 25). For this reason, the langbeinite mineralization was not evaluated.

The sylvite resource from this unit could be beneficiated by flotation. Kieserite should not preclude flotation recovery, since the sylvite can be liberated, but the SO_4 content is too high (2.99 percent) for recovery by solution-crystallization. The insolubles are in sufficiently high concentrations to require extra desliming and brine recovery circuiting in the flotation plant.

Based on these assumptions, the resource was evaluated for sylvite flotation, modified to include extra desliming and brine recovery circuits, with estimated K_2O recovery of 78 percent.

A conventional room-and-pillar mining system, using the same mining capacity as Mining Unit C–2, was used to extract this low-grade sylvite mineralization.

MINING UNIT D

This Mining Unit was used to evaluate the southern portion of the 4th langbeinite ore zone. No evaluation was performed on the highest grade resource (Mining Unit D–1) because sufficient tonnage to be commercial is not present. Evaluations were made on the medium and low grade resources. Resource data are shown in table 26.

Mining Unit D–2

Mining Unit D–2 contains langbeinite mineralization from the southern portion of the 4th ore zone. As shown in table 26, the medium grade resource has an average mining grade of 5.00 percent (K_2O equivalent).

The resource of Mining Unit D–2 contains only a small amount of impurities, as illustrated in table 27. Because the percentage of impurities and insolubles is below that of synthetic low-grade samples analyzed by the Salt Lake City Research Center, the deposits could be processed commercially to yield a marketable product.

TABLE 26.—Average K_2O grade of langbeinite and total tonnage, Mining Unit D

(Weight-percent)

Ore zone	Weighted average grade (pecent)	Total tonnage (MM)	Percent inside WIPP site	Tonnage inside WIPP site (MM)
Mining Unit D–2:				
4th	5.00	87.93	26.8	23.57
Mining Unit D–3:				
4th	4.30	140.27	30.3	42.45

TABLE 27.—Mineralogical compositions of langbeinite deposits, Mining Unit D

(Weight-percent)

Mineral, average composition	Mining Unit D–2	Mining Unit D–3
Langbeinite	5.00	4.30
Insolubles	1.57	1.57
Polyhalite	.04	.03
Kainite	.09	.08
Leonite	.004	.03

A conventional room-and-pillar mining system with a capacity of 10,000 tons per day (9,070 metric tons per day) was used in this evaluation. The mined material would be sent to a langbeinite wash plant built at the site. A langbeinite product would be produced, and the resulting fines would be shipped and processed at two existing sulfate plants.

All costs of mining a sufficient amount of sylvite to produce the necessary amount of fines used with the langbeinite fines in the sulfate process were charged to the mining cost of this subunit.

Mining Unit D–3

This subunit consists of the lowest grade resource found in the 4th ore zone. The average mining grade and resource data are shown in table 26. The impurities are low, indicating that the langbeinite could be processed to a marketable product.

The same type of mine-mill complex described in D–2 was used in the evaluation of Mining Unit D–3 but at a reduced capacity of 7,000 tons per day (6,350 metric tons per day) in order to provide sufficient langbeinite fines for only one sulfate plant. A langbeinite wash plant would be built at the site. The langbeinite fines from this wash plant would be shipped to an existing sulfate plant for further processing. All costs associated with the mining and milling of sylvite to produce the necessary fines required in the production of a sulfate product were charged to the D–3 mine and mill complex.

ESTIMATED MINING UNITS CAPITAL INVESTMENTS AND OPERATING COSTS

Table 28 shows operating costs and capital investments for 8 of the 12 hypothetical Mining Units. These Units were selected for analysis to determine if they were commercial at 1977 market product prices and estimated costs. No deposits corresponding to Mining Unit D–1 (highest grade) are in the site; Mining Units B–2 and B–3 were eliminated because they contain material that could not be treated by current technology to recover marketable products; Mining Unit C–1 mineralization was included with that for Unit C–2 (because of better economics, it would be mined and processed as that unit). Table 29 shows estimates of product recovery as sylvite (or muriate), langbeinite, and sulfate.

TABLE 28.—Summary of operating costs and capital investments for hypothetical Mining Units

Mining Unit	Operating costs[1], $/st			Capital investment, $1,000		
	Mine	Mill	Freight	Mine	Mill	Total
A-1	3.55	2.67	[2] 0.54	$35,652	$ 3,530	$39,182
A-2	3.55	2.67	[2] .54	35,652	3,530	39,182
A-3	3.55	2.67	[2] .54	35,652	3,530	39,182
B-1	5.64	7.24	[3] .04	56,042	8,447	64,489
C-2	3.55	2.67	[2] .54	35,652	3,530	39,182
C-3	3.55	2.67	[2] .54	35,652	3,530	39,182
D-2	4.58	4.34	[3] .02	37,439	11,826	49,265
D-3	4.89	4.56	[3] .02	31,256	10,213	41,469

[1] Costs of obtaining sufficient sylvite (where sylvite is not mined at the operation) to process langbeinite fines for sulfate production are added to the costs for langbeinite mining and milling.
[2] Added to mine operating costs.
[3] Added to mill operating costs.

TABLE 29.—Potash mineral resource evaluation data in ERDA's WIPP site

Mining Unit	Deposits included [1]	Adjusted potash grade percent K_2O	Deleterious minerals weight percent [2]	Total adjusted in deposits (short tons (MM))	Total used in min. eval. MU [3] (short tons (MM))	Percent inside WIPP site	Inside WIPP site (MM) short tons	Mining unit capacity /day short tons [4]	Life of opr. yrs.	Mine opr. cost [5] $/short tons	Mill opr. cost [5] $/short tons	Freight cost $/short tons	Mine capital investment $	Mill capital investment $	Total capital investment $	Total short tons of product recovered in the WIPP site [5]	Additional short tons of product within WIPP site [6]	Product to be commercial ($/short ton)	Gross revenue of additional products 1977 $ [7]
A-1	10th west lobe, 8th	13.33 S	I 3.48, P 0.99, Ki 3.04, C 1.19, Ka 0.91, Le 0.62	57.60	55.39	48	27.43	9,484	16	3.55	2.67	0.54 add to mine op. cost	35,652,700	3,529,500	39,182,200	4,935,900 S	4,935,900 S	52.04 S	256,864,200
A-2	10th west lobe, 8th	11.28 S	I 5.19, P 2.14, Ka 0.64, Ki 2.14, Le 0.44, C 3.27	98.32	96.93	47	51.80	9,484	28	3.55	2.67	0.54 add to mine op. cost	35,652,700	3,529,500	39,182,200	7,790,500 S	2,854,600 S	61.73 S	176,214,500
	2d west lobe, 3d	4.70 L	I 1.52, P 1.46, Ka 0.85, Le 1.43	40.99	--	93	38.31	Langbeinite mineralization not amenable to commercial processing											
A-3	10th west lobe, 8th	10.03 S	I 5.54, P 2.29, Ki 1.90, Ka 0.57, Le 0.39, C 3.75	135.02	103.85	55	73.77	9,484	30	3.55	2.67	0.54 add to mine op. cost	35,652,700	3,529,500	39,182,200	9,742,200 S	1,951,700 S	70.28 S	137,165,500
	2d west lobe, 3d	4.12 L	I 1.67, P 1.80, Ka 0.80, Le 1.43	75.25	--	66	49.88	Langbeinite mineralization not amenable to commercial processing											
B-1	4th, 5th	9.11 L	I 2.07, P 1.38, Ka 1.10, Ki 0.47, Le 0.93, C 1.17	79.78	65.25	61	48.46	5,959	30	5.64	7.24	0.04 add to mill op. cost	56,042,000	8,447,100	64,489,100	14,174,800 L 5,998,300 Sul	14,174,800 L 5,998,300 Sul	35.00 L 88.50 Sul	496,118,000 530,850,000
B-2	4th north portion, 5th	6.92 L	I 3.15, P 1.41, Ka 0.98, Ki 0.66, Le 1.05, C 2.23	154.18	--	62	95.68	Langbeinite mineralization not amenable to commercial processing											
B-3	4th north portion, 5th	6.30 L	I 3.21, P 1.42, Ka 1.01, Ki 0.68, Le 1.00, C 1.91	196.07	--	63	123.79	Langbeinite mineralization not amenable to commercial processing											
C-1 [8]	10th, 9th	14.44 S	I 4.7, P 2.06, Ki 9.02, C 0.39	31.84	31.23	61	16.24												
	10th, 2d east lobe	8.08 L	I 2.59, P 1.64, Ka 1.51, Le 0.88	18.50	--	86	15.95	Langbeinite mineralization not amenable to commercial processing											
C-2	10th east lobe, 9th	11.36 S	I 3.98, P 1.9, Ki 6.52	57.19	58.85	64	36.49	9,484	17	3.55	2.67	0.54 add to mine op. cost	35,452,700	3,529,500	39,182,200	5,526,900 S	5,526,900 S	61.74 S	341,231,000
	10th east lobe, 2d east lobe	5.18 L	I 2.40, P 1.55, Ka 1.71, Ki 0.55, Le 1.45	92.55	--	87	81.27	Langbeinite mineralization not amenable to commercial processing											
C-3	10th east lobe, 9th	10.41 S	I 3.82, P 1.82, Ki 6.25	70.64	69.23	75	52.87	9,484	20	3.55	2.67	0.54 add to mine op. cost	35,652,700	3,529,500	39,182,200	7,292,800 S	1,765,900 S	67.52 S	119,234,000
	10th east lobe, 2d east lobe	4.75 L	I 2.40, P 1.52, Ka 1.61, Ki 0.49, Le 1.45	104.56	--	88	99.34	Langbeinite mineralization not amenable to commercial processing											
D-1	Deposits of this grade are not present.																		
D-2	4th south portion	5.00 L	I 1.57, P 0.04, Ka 0.09, Le 0.04	87.93	80.30	27	23.57	10,000	22	4.58	4.34	0.02 add to mill op. cost	37,439,200	11,826,200	49,265,400	3,768,900 L 1,598,400 Sul	3,768,900 L 1,598,400 Sul	36.51 L 92.31 Sul	137,603,000 147,548,000
D-3	4th south portion	4.30 L	I 1.57, P 0.03, Ka 0.08, Le 0.03	140.27	87.38	30	42.45	7,980	30	4.89	4.56	0.02 add to mill op. cost	31,256,000	10,212,800	41,468,800	5,724,200 L 2,767,400 Sul	1,955,300 L 1,169,000 Sul	42.26 L 106.86 Sul	82,631,000 124,919,000

1/ USGS determined tonnages of mineralization present at three cutoff grades: 3 and 8 percent; 4 and 10 percent; 8 and 14 percent K_2O as langbeinite and sylvite.
2/ B-bloedite; C-carnallite; I-insolubles; Ka-kainite; Ki-kieserite; Le-leonite; P-polyhalite.
3/ USGS tonnage adjusted to 4.5 minimum mining height and 85 percent recovery.
4/ Target capacity estimated for operation 365 days/year.
5/ The operating costs of sufficient sylvite to process langbeinite fines (where sylvite is not mined at the operation) for sulphate production are added to the langbeinite mining and milling costs.
6/ S-sylvite or muriate product; L-langbeinite; Sul-sulphate.
7/ Product amounts that would be recovered in higher grade Mining Units are subtracted from product amounts that would be recovered in lower grade Mining Units.
8/ Because of better economics, MU C-1 mineralization is included in MU C-2 and would be mined and processed as MU C-2.

SUMMARY AND CONCLUSIONS

The Bureau of Mines evaluation of potash deposits in the WIPP site and subsequent economic and financial analyses of hypothetical mining and processing operations show that one of the 12 Mining Units—Unit B–1, parts of which are in the site—is commercial at current mining and processing costs and at present market prices. These deposits are thus ore and are classified as reserves.

Potash deposits included in these Mining Units occur in various ore zones accessible from proposed shaft locations. The mining and beneficiation systems evaluated are based on current extraction and processing technology in the district and are the least costly systems amenable to the ore mined. An economic evaluation of the mineralization determined to be in the site near its western boundary was made considering mining recovery through an existing shaft in addition to recovery through a new shaft. The grade of the mineralization and deleterious mineral content determined from drill core samples indicate that this mineralization is not commercial using current Carlsbad technology.

This analysis isolates and estimates the values that exist in the unmined potash mineralization in the WIPP site and, therefore, determines a cost chargeable to the WIPP facility. These values are items of costs and capital investments (taxes, royalties, and bonus bid amounts) that would be generated from potash product sales and would be paid to the several levels of government. They are losses because the tax and royalty revenues and bonus bid amounts, or their equivalent value, will not accrue to the governments if the potash deposits are not mined and produced. Unlike taxes, royalties, and bonus bid amounts, all other items of production costs, investments, and values are created when the projects are initiated (when capital is invested) and, therefore, are not a loss if the unmined, commercial potash is not developed.

Loss to the owners (other than government) would be royalty payments and a value determined for the unmined potash (equivalent to the bonus bid amount due the governments). The taxes that would have been generated if the potash were produced from privately owned potash would be a loss to the governments. These values (taxes and royalties—because they are estimated to occur in a series of lump sums over the life of the project—and the bonus bid amounts) are converted (discounted at an appropriate discount rate) to equivalent values at a single point in time (usually project initiation). In this analysis, it results in an amount in current dollars or the present value. The present or current value (as commonly used in economic analyses) of the unmined potash (bonus bid amount) is the dollar amount that the gross revenues exceed the sum of the (1) costs of production, (2) return of the investment, and (3) interest on that investment; all items discounted at a future worth factor of money. The present value is the amount that an investor would be able to pay for the unmined potash and still receive (1) the return of his total investment and (2) an acceptable rate of interest on his total investment including the amount he pays for the unmined potash. In this analysis, for those government-owned deposits that are commercial, the present value amount is added to the investments as a lease bonus bid (a dollar amount paid as a bonus to obtain a lease).

Items such as the return of the original investment and interest on that investment as well as all direct and indirect costs of production must be provided from gross revenue in a profitable operation. This can be expressed as follows: The gross revenue from the sale of all potash products equals all direct and indirect costs including (1) materials, (2) labor, (3) supplies, (4) utilities, (5) taxes, (6) royalties, and (7) other costs plus the interest on the investment plus return of the investment for (1) exploration, (2) development, (3) mine plant, (4) mill plant, (5) bonus bid amount, and (6) all other capital required.

The major portion of the dollar amounts of gross revenues is used to purchase goods and services required for potash production.

One group of the highest grade deposits, designated as Mining Unit B–1 and occurring partially in the site, was determined to be commercial. About 20.2 million tons (about 18.3 million metric tons) of potash products could be produced

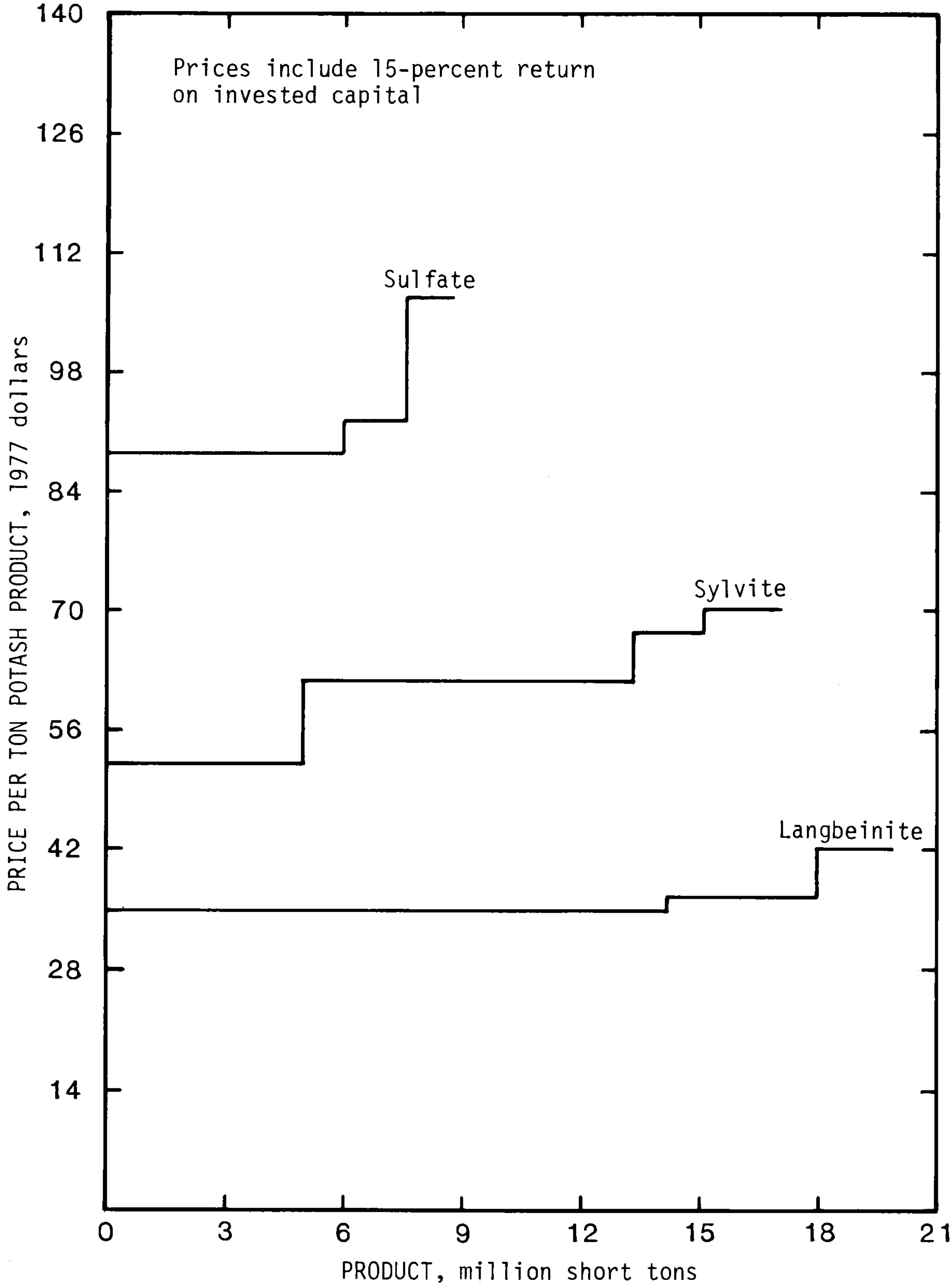

FIGURE 30.—Product availability and market prices at which potash tonnages become commercial assuming fixed production costs.

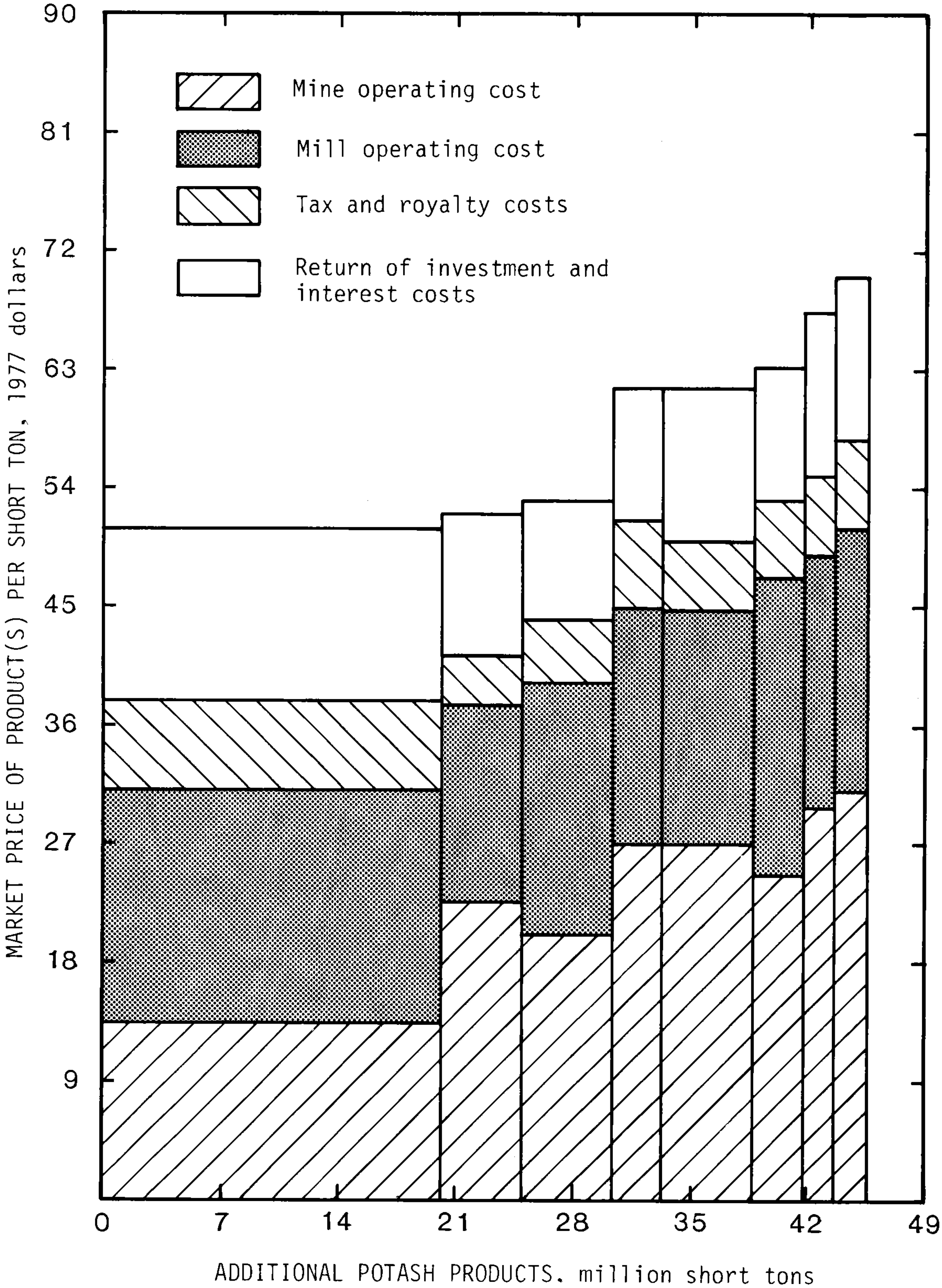

FIGURE 31.—Approximate cost analysis of increased market prices at which potash tonnages become commercial.

from the 48.46 million tons (43.95 million metric tons) of ore in these reserves. The value lost (dollars that would not be generated from potash production) in terms of current dollars for an acquisition cost, for taxes payable on estimated revenues, and for royalties payable to the owners of the potash, is about $51.8 million. The gross market value of the products is about $1.03 billion which, less operating costs other than taxes and royalties, less the return of the investment other than the acquisition cost, and less interest on the investment limited to 15 percent, is the $51.8 million value chargeable to the WIPP project. The procedure used to calculate an estimate of the values that will be foregone if the WIPP installation goes forward is shown in table 30.

The amounts of potash products that could be recovered from potash reserves and resources in the WIPP site, at specific market prices and associated costs, are illustrated in figures 30 and 31 and in tables 31, 32, and 33. Evaluation data are summarized in table 29.

An additional 169.1 million tons (153.4 million metric tons) of lower grade, paramarginal potash resources in the site contain 25.5 million tons (23.1 million metric tons) of products if today's operating costs did not increase and market prices were increased until the deposits become commercial. The dollar values of these subeconomic resources are speculative; however, if the relationship of actual losses to gross market values exists as in the commercial deposits, the value lost at some future time when they become commercial is about $75.4 million from a gross market value of products of about $1.5 billion. Recent market prices are shown in table 34.

Products that meet market standards cannot be produced, using current Carlsbad processing techniques, from the remainder of the mineralization determined to be in the site. This mineralization is considered to be a submarginal resource.

An option is being considered that would allow commercial recovery of potash products in WIPP Zone IV. Zone IV is a comparatively large area—10,812 acres (4,376 hectares) that contains much of the mineralization in the WIPP site. For this reason, the estimated amounts and values of potash product that would be foregone if mining operations were prohibited only in Zones I, II, and III are 13.3 million tons (12.1 million metric tons) of ore (commercial mineralization) containing 5.5 million tons (5.0 million metric tons) of potash products (table 31). The value lost in terms of current dollars is about $14.3 million. The gross market value of the products is about $282.4 million. The gross market value of the products is about $282.4 million. Paramarginal resources in 51.37 million tons (46.59 million metric tons) of potash mineralization contain 7,640,500 tons (6,931,400 metric tons) of potash products. The ratio (the same as for the commercial resources) of value lost to gross product market value is about $17.5 million from a gross market value of $346.3 million.

TABLE 30.—Mining Unit B–1 estimated potash values within the WIPP site that would be foregone

Tonnage:	*1,000 short tons of ore*
Total in deposit	79,780
Total in Mining Unit B–1 operation[1]	65,250
Part of Mining Unit B–1 in WIPP site	48,460
Total of B–1 in WIPP site on State lands (all in Sec 16, T 22 S, R 31 E)	7,753.6
Total of B–1 in WIPP site on Federal lands	40,706.4

Item	Mining Unit present value, at 8 percent over 30 years ($1,000)	Value, $/st	WIPP tonnage factor	Revenues lost ($1,000) State Govt.	Revenues lost ($1,000) Federal Govt.
Royalties	19,021	0.29	----	----	----
Federal[2]	----	.29	40,706.4	5,902	5,902
State	----	.29	7,753.6	2,249	----
State income taxes	2,383	.04	48,460	1,938	----
Severance taxes[3]	5,060	.08	48,600	3,877	----
Federal income taxes	19,632	.30	48,460	----	14,538
Acquisition	23,325	.36	----	----	----
Federal	----	.36	40,706.4	----	14,654
State	----	.36	7,753.6	2,791	----
Total	69,421	----	----	16,757	35,094
Grand total				$51,851	

[1] For estimating purposes, this tonnage was calculated by multiplying annual production rate times life of proposed operation. The tonnage was then used to derive the unit production costs, which were assigned to total tonnage in the deposit.
[2] Federal royalties are divided 50–50 between the Federal and State governments.
[3] Severance taxes include severance, property, and processors taxes.

TABLE 31.—Summary of amounts and values of potash mineralization in WIPP site

Mining Unit	Products	1,000 short tons: Recoverable ore in Mining Unit	Recoverable ore in WIPP site	Product tonnage in WIPP site	Value, $1,000: Price[1] that provides a 15 percent ROR[2]	Gross revenue of products inside the WIPP site	Gross[3] revenue of additional products	Present value[4] foregone in WIPP site
B–1	Langbeinite	79,780	48,460	14,175	35.00	496,125	1,026,948	51,851
	Sulfate	----	----	5,998	88.50	530,823	----	----
A–1	Muriate	57,600	27,410	4,936	52.04	256,869	256,869	----
D–2	Langbeinite	87,930	23,570	3,769	36.51	----	137,606	----
	Sulfate	----	----	1,598	92.31	----	147,511	----
A–2	Muriate	98,320	51,800	7,791	61.73	----	176,239	----
C–2	Muriate	57,190	36,490	5,527	61.74	----	341,237	----
D–3	Langbeinite	140,270	42,450	5,724	42.26	----	82,618	----
	Sulfate	----	----	2,767	106.86	----	124,919	----
C–3	Muriate	70,640	52,870	7,293	67.52	----	119,240	----
A–3	Muriate	135,020	73,770	9,743	70.28	----	137,165	----

[1] Market price assuming no increase in production costs. Estimated weighted average annual price per ton of product f.o.b., Carlsbad, N. Mex., used for evaluation.
[2] Rate of return.
[3] Product amounts that would be recovered in higher grade Mining Units (A–1, B–1, C–1) are subtracted from product amounts that would be recovered in lower grade Mining Units (A–2, etc.).
[4] Present values at increased potash products market prices of resources foregone in WIPP site were not estimated because the time and conditions at which they could become commercial are too speculative. For this study, the 1975–1976 weighted average prices received for potash products in the Carlsbad area were used to estimate the values of commercial potash deposits (ore).

TABLE 32.—Mining Unit product data and required market prices at which potash tonnages in the WIPP site become commercial at fixed production costs

(1,000 short tons)

Mining Unit	Products	Price[1] that provides a 15 percent ROR[2]	Total Mining Unit product tonnage	Total product tonnage in WIPP	Net additional product tonnage	Cumulative product tonnage
B–1[3]	Langbeinite	$35.00	19,086	14,175	14,175	14,175
	Sulfate	88.50	8,077	5,998	5,998	20,173
A–1	Muriate	52.04	9,967	4,936	4,936	25,109
D–2	Langbeinite	36.51	12,840	3,769	3,769	28,878
	Sulfate	92.31	5,446	1,598	1,598	30,476
A–2	Muriate	61.73	14,578	7,791	2,855	33,331
C–2	Muriate	61.74	8,914	5,527	5,527	38,858
D–3	Langbeinite	42.26	11,783	5,724	1,955	40,813
	Sulfate	106.86	5,697	2,767	1,169	41,982
C–3	Muriate	67.52	9,550	7,293	1,766	43,748
A–3	Muriate	70.28	13,715	9,742	1,952	45,700

[1] Market price assuming no increase in production costs. Estimated weighted average annual price per ton of product, f.o.b., Carlsbad, New Mex., used forevaluation. For this study the 1975–1976 weighted-average prices received for potash products in the Carlsbad area were used to estimate the values of commercial potash deposits.
[2] Rate of return.
[3] Commercial at recent weighted-average annual prices.

TABLE 33.—Mining Unit price components per ton of weighted-average product price assuming fixed production costs and a 15-percent discounted cash flow rate of return

Mining Unit	B–1	A–1	D–2	A–2	C–2	D–3	C–3	A–3
Product, short tons:								
Annual	905,400	622,900	831,100	520,600	524,300	582,700	477,500	457,200
Additional within WIPP	20,173,100	4,935,900	5,367,300	2,854,600	5,526,900	3,124,300	1,765,900	1,951,700
Cost per ton product:								
Mine operating cost	$13.55	$22.71	$20.11	$27.18	$26.98	$24.44	$29.63	$30.94
Mill operating cost	17.48	14.84	19.15	17.75	17.63	22.89	19.36	20.22
Taxes and royalties	6.80	3.60	4.65	16.56	5.15	5.65	5.85	6.46
Interest and amortization of investment	13.08	10.89	9.22	10.24	11.98	10.33	12.68	12.66
Total	52.04	52.04	53.13	61.73	61.74	63.31	67.52	70.28

TABLE 34.—U.S. average market prices for potash products—f.o.b. plant[1]

(Dollars per short ton)

Product	1973	1974	1975: 1st 6 mo.	1975: 2d 6 mo.	1976: 1st 6 mo.	1976: 2d 6 mo.
Muriate, all grades, at 61 percent K_2O:						
Per ton K_2O	36.39	52.50	75.19	76.56	70.91	59.81
Per ton of product	22.20	32.03	45.87	46.70	43.26	36.48
Sulfate, at 51 percent K_2O:						
Per ton of K_2O	82.68	110.79	163.39	181.63	177.00	224.02
Per ton of product	42.17	56.50	83.33	92.63	90.27	114.25
Langbeinite, at 22 percent K_2O:						
Per ton of K_2O	88.41	119.41	149.73	155.73	149.27	154.77
Per ton of product	19.45	26.27	32.94	34.26	32.84	34.05

[1] Product market prices vary considerably from month to month and from company to company. The potash market and product prices have been seasonal and, in addition to the usual market supply-demand fluctuations, have been subject to discounting, inventories, buildup pressures, and product grade variations.

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APPENDIX A.—Assumptions and Estimations Used for the Economic Analysis of the Hypothetical Mining Unit B–1 (MU B–1)

The operation is treated (for economic analysis) as a separate company, not a subsidiary where losses could possibly be used for tax credits. A system diagram, showing approximate materials flow and gross revenue, and a computer printout of the investments and revenues analysis are included.

It is assumed that two operating, competing companies in the Carlsbad area would form a third company to mine and process (through a mine complex and langbeinite wash plant) the ore determined to be in the area, provided a reasonable return on this investment could be expected from the project. A portion of this ore is located under the WIPP site. Sufficient ore exists outside the WIPP site to support a commercial potash operation, so that only the ore within the WIPP boundary is charged to the WIPP project.

A langbeinite wash plant could be constructed at the mine site, and langbeinite product sold at the plant site; langbeinite fines produced would be shipped to two existing sulfate plants.

Sylvite required at one of the sulfate plants (Sulfate Plant No. 2, fig. A–1) would be provided from its ongoing sylvite processing operation. MU B–1 is charged for mining and processing costs for sylvite fines produced in the normal processing operation. Sulfate processing costs are charged to and the sulfate revenues are credited to MU B–1.

Sylvite for the other sulfate plant (Sulfate Plant No. 1, fig. A–1) is provided from a mixed ore. The amounts of sylvite and langbeinite from the ore, plus the langbeinite fines from MU B–1, are estimated to provide capacity operation of this sulfate plant. This amount of sylvite required results in a small excess amount of langbeinite from the mixed ore. Mining and processing costs for the mixed ore are charged to and the sulfate and excess langbeinite revenues are credited to MU B–1.

Capital requirements include construction of (1) a langbeinite wash plant at the mine site; (2) the necessary storage and loading facilities to handle the langbeinite product and to ship the langbeinite fines produced to two existing sulfate plants; (3) a mine including two shafts, underground facilities, and surface structures; and (4) reinvestment of some shorter life equipment in the sylvite ore, mixed ore, and sulfate plants. The capacity of Mining Unit B–1 is estimated to produce langbeinite fines in an amount approximately equal to the present langbeinite fines feed to the existing sulfate plants.

The ore deposits within this study area are estimated to contain a total of 79.78 million short tons (72.37 metric tons). In addition, an unknown amount of recoverable ore is estimated to occur outside the study area, contiguous to the deposits in the area. Of those deposits within the study area, 65.25 million short tons (59.19 metric tons) would be required for MU B–1, of which 48.46 million short tons (43.96 metric tons) are within the WIPP site. The 79.78 million short tons (72.37 metric tons) of recoverable ore are in a minimum thickness of 4.5 feet, contain langbeinite in an average grade of 9.11 percent K_2O equivalent, and exist within reasonable mining distance of the approximate location of a production shaft. The 65.25 million short tons (59.19 metric tons) of ore deposits were used for the MU B–1 economic analysis.

It is assumed the construction of the WIPP, with the consequent unavailability of the 48.46 million short tons (43.96 million metric tons) that exist within the WIPP site, would not make the recoverable ore (outside the WIPP site and within the study area) uneconomic. Discounting the cash flows yields a rate of return on the estimated investment in excess of 15 percent. For this reason, a lease bonus bid amount (acquisition cost) was added to the other total initial investments. The amount of the lease bonus bid was determined so that, when added to all other investments, the project will provide a return of 15 percent on the total investment. The amount of the lease bonus bid is also termed a present value. In other words, the producers could make an acceptable profit, arbitrarily assumed to be 15 percent on their total investment, when that investment includes a sizable bonus bid amount for leases in addition to all other investments required for MU B–1.

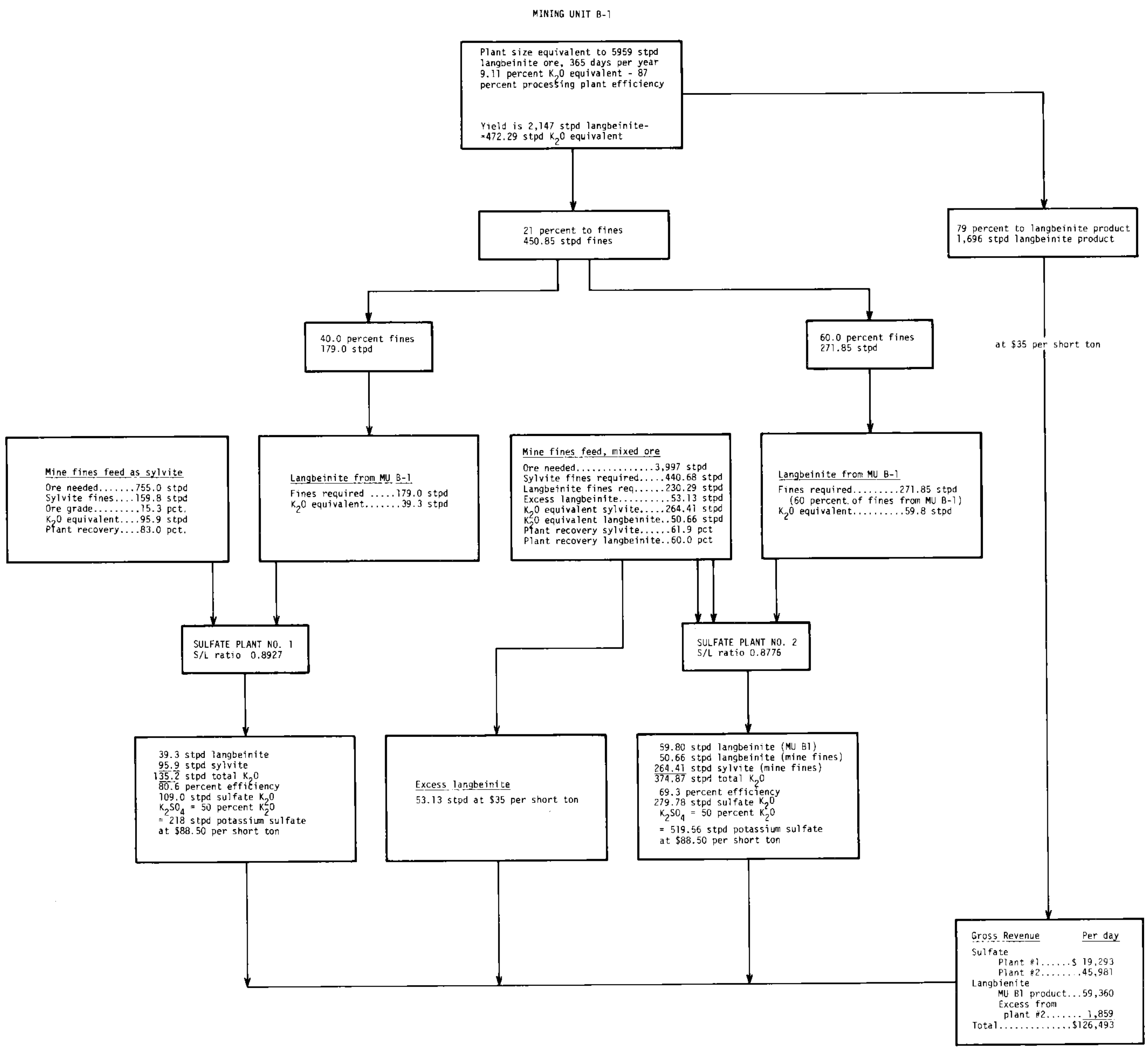

FIGURE A–1.—System diagram of hypothetical Mining Unit B–1 operation, with approximate minerals flow through the plants and estimated gross revenues.

The hypothetical MU B–1 plant is designed to handle a capacity of ore equivalent to 5,959 short tons per day (5,406 metric tons per day) 365 days per year. The average grade of the ore is 9.11 percent K_2O as langbeinite. The wash plant has an overall recovery of 87 percent. The plant produces about 21 percent fines; the balance (89 percent) provides about 1,696 short tons per day (1,539 metric tons per day) of langbeinite marketable product which, at $35/short ton, is $59,360 gross revenue per day. The hypothetical operation then produces 450.85 short tons (409.00 metric tons) of langbeinite fines for processing in the two sulfate plants.

It is further assumed that about 179.0 short tons (162.4 metric tons), or 40 percent of the 450.8 short tons (409.0 metric tons) of langbeinite fines, is shipped to Sulfate Plant No. 1, where the sylvite-langbeinite ratio required for sulfate production is about 0.8927, indicating that 159.8 short tons (145.0 metric tons) of sylvite is required to combine with the langbeinite for sulfate production. This sylvite is provided, at mining and processing costs, from the current mining operation at the sulfate plant. With a sulfate plant efficiency of about 80.6 percent, this provides about 218 short tons per day (198 metric tons per day) of sulfate. At $88.50/short ton, this provides $19,293 gross revenue per day.

The balance of the langbeinite fines from the MU B–1 plant are shipped to Sulfate Plant No. 2, where the sylvite fines are provided from a mixed ore with a grade of about 9.2 percent K_2O as sylvite and 2.6 percent K_2O as langbeinite. Mining Unit B–1 is charged mining and processing costs for sufficient mixed sylvite and langbeinite ore (the sylvite is combined with the langbeinite fines from MU B–1) for capacity operation of that sulfate plant. This results in a small excess of langbeinite, the gross revenue of which is credited to MU B–1.

The 3,997 short tons per day (3,626 metric tons per day) of mixed fines normally produced in processing 9,608 short tons per day (8,716 metric tons per day) of ore at 9.2 percent K_2O sylvite grade with a sylvite recovery of 71.9 percent, would provide 440.68 short tons per day (399.78 metric tons per day) of sylvite fines. The 3,997 short tons (3,636 metric tons) of ore is considered to be sylvite fines only, for the purpose of determining the costs to be charged to MU B–1. At 2.6 percent grade, 283.43 short tons (257.12 metric tons) of langbeinite is produced with a langbeinite processing recovery of about 60 percent; 230.29 short tons (209.91 metric tons) of this langbeinite is combined with the 271.85 short tons (246.62 metric tons) langbeinite fines from MU B–1. These combined tonnages result in the production of about 519.56 short tons (471.33 metric tons) of sulfate per day from this plant, which has an overall recovery of 69.3 percent. At $88.50/short ton, this provides a gross revenue of $45,981 per day.

In addition, the 53.13 short tons (48.20 metric tons) of langbeinite from mixed ore not needed in sulfate production are considered a credit, the "cost" charged to sylvite fines production. At $35/short ton, this provides a gross revenue of $1,859 per day.

APPENDIX B.—Financial Evaluation of MU B-1A

FINANCIAL EVALUATION

MU B 1A-LANG. MINE WITH WASH PLANT, SHIP FINES TO 2 MILLS, 5959 TPD

BEGINNING OPERATION IN 1977
AND ENDING OPERATION IN 2010
WITH 4 PREPRODUCTION YEARS
USING 1 SIMULATION(S) AND
A TARGET RATE OF RETURN OF
15.0 PERCENT

LAST YEAR ADDITIONS TO CASH FLOW	
CUMULATIVE WORKING CAPITAL	5666000.
CUMULATIVE SALVAGE VALUE	0.
OTHER CUMULATIVE VALUES	
REVENUES	1382788350.
ROYALTIES	69139417.
PROPERTY TAXES	0.
SEVERENCE TAXES	18391085.
STATE INCOME TAXES	10043848.
FEDERAL INCOME TAXES	87181903.
CASH FLOW	254880979.
LAST YEARS ANALYSIS FIGURES	
CONTINUOUS RATE OF RETURN	15.001
15.00 PCT PRESENT VALUE	1927.

FINANCIAL EVALUATION

MU B 1A-LANG. MINE WITH WASH PLANT, SHIP FINES TO 2 MILLS, 5959 TPD

BEGINNING OPERATION IN 1977
AND ENDING OPERATION IN 2010
WITH 4 PREPRODUCTION YEARS
USING 1 SIMULATION(S) AND
A TARGET RATE OF RETURN OF
15.0 PERCENT

	1977	1978	1979	1980	1981	1982	1983	1984	1985
GENERAL DATA * * * * * * * * * *									
INVESTMENTS									
EXPLORATION - - - - - - - - -	600000.	0.	0.	0.	0.	0.	0.	0.	0.
LAND ACQUISITION- - - - - -	23325000.	0.	0.	0.	0.	0.	0.	0.	0.
MINING PREPARATION- - - - -	0.	4079500.	4079500.	4079500.	0.	0.	0.	0.	0.
INVEST. NUMBER 1- MINE- - -	0.	0.	3046750.	3046750.	0.	0.	0.	0.	0.
LIFE - - - - - - - - - - - -	0	0	15	15	0	0	0	0	0
DEPRECIATION - - - - - - -	0.	0.	203117.	406233.	406233.	406233.	406233.	406233.	406233.
SALVAGE VALUE- - - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 2- MINE- - -	0.	0.	4059500.	4059500.	0.	0.	0.	0.	0.
LIFE - - - - - - - - - - - -	0	0	8	8	0	0	0	0	0
DEPRECIATION - - - - - - -	0.	0.	507438.	1014875.	1014875.	1014875.	1014875.	1014875.	1014875.
SALVAGE VALUE- - - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 3- MINE- - -	0.	0.	734805.	734805.	0.	0.	0.	0.	0.
LIFE - - - - - - - - - - - -	0	0	8	8	0	0	0	0	0
DEPRECIATION - - - - - - -	0.	0.	91851.	183701.	183701.	183701.	183701.	183701.	183701.
SALVAGE VALUE- - - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 4- PROC- - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
LIFE - - - - - - - - - - - -	0	0	0	0	0	0	0	0	0
DEPRECIATION - - - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
SALVAGE VALUE- - - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 5- PROC- - -	0.	0.	0.	0.	8447074.	0.	0.	0.	0.
LIFE - - - - - - - - - - - -	0	0	0	0	15	0	0	0	0
DEPRECIATION - - - - - - -	0.	0.	0.	0.	563138.	563138.	563138.	563138.	563138.
SALVAGE VALUE- - - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 6- PROC- - -	0.	0.	0.	0.	670438.	0.	0.	0.	0.
LIFE - - - - - - - - - - - -	0	0	0	0	6	0	0	0	0
DEPRECIATION - - - - - - -	0.	0.	0.	0.	111740.	111740.	111740.	111740.	111740.

SALVAGE VALUE- - - - - -									
INVEST. NUMBER 7- MISC- - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
LOAN NUMBER 1 - - - - - - - -	0.	4000000.	10500000.	10500000.	0.	0.	0.	0.	0.
INTEREST RATE- - - - - -	0.000	0.090	0.090	0.090	0.000	0.000	0.000	0.000	0.000
INTEREST PAID- - - - - -	0.	0.	0.	0.	2619015.	2446631.	2258733.	2053924.	1830682.
PRINCIPLE PAID - - - - -	0.	0.	0.	0.	1915375.	2087759.	2275657.	2480466.	2703708.
LOAN NUMBER 2 - - - - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
INTEREST RATE- - - - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
INTEREST PAID- - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
PRINCIPLE PAID - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
LOAN NUMBER 3 - - - - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
INTEREST RATE- - - - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
INTEREST PAID- - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
PRINCIPLE PAID - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
EQUITY CAPITAL- - - - - - -	23925000.	79500.	1420555.	1420555.	16698887.	2087759.	2275657.	2480466.	2703708.
WORKING CAPITAL - - - - - -	0.	0.	0.	0.	5666000.	0.	0.	0.	0.
OPERATING COSTS PER UNIT									
MINE- - - - - - - - - - - - -	0.00000	0.00000	0.00000	0.00000	5.64000	5.64000	5.64000	5.64000	5.64000
MILL- - - - - - - - - - - - -	0.00000	0.00000	0.00000	0.00000	7.27800	7.27800	7.27800	7.27800	7.27800
TOTAL OVERHEAD- - - - - - -	0.00000	0.00000	0.00000	0.00000	0.00000	0.00000	0.00000	0.00000	0.00000
UNITS TREATED- - - - - - - - -	0.	0.	0.	0.	2175035.	2175035.	2175035.	2175035.	2175035.
MISC. OPERATING EXPENSES - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
COMMODITY DATA * * * * * * * *									
KMAG									
ORE GRADE - - - - - - - - - -	0.000000	0.000000	0.000000	0.000000	0.075000	0.075000	0.075000	0.075000	0.075000
MILL RECOVERY - - - - - - - -	0.000000	0.000000	0.000000	0.000000	0.858000	0.858000	0.858000	0.858000	0.858000
MILL CONCENTRATE GRADE- - -	0.0000	0.0000	0.0000	0.0000	0.2200	0.2200	0.2200	0.2200	0.2200
SMELTER RECOVERY- - - - - -	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000
SMELTER CONCENTRATE GRADE -	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
REFINER RECOVERY- - - - - -	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000
OP. COSTS/UNIT PROCESSED									
SMELTER- - - - - - - - -	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
REFINER- - - - - - - - -	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
TRANS. COSTS/UNIT PROCESSED									
MILL TO SMELTER- - - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
SMELTER TO REFINER - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
REFINER TO MARKET- - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PRICE/UNIT RECOVERED- - - -	0.00	0.00	0.00	0.00	35.00	35.00	35.00	35.00	35.00
UNITS RECOVERED - - - - - -	0.	0.	0.	0.	636198.	636198.	636198.	636198.	636198.
REVENUES- - - - - - - - - -	0.	0.	0.	0.	22266921.	22266921.	22266921.	22266921.	22266921.
DEPLETION - - - - - - - - -	0.	0.	0.	0.	2961500.	2961500.	2961500.	2961500.	2961500.
SULFATE									
ORE GRADE - - - - - - - - - -	0.000000	0.000000	0.000000	0.000000	0.085600	0.085600	0.085600	0.085600	0.085600
MILL RECOVERY - - - - - - - -	0.000000	0.000000	0.000000	0.000000	0.723000	0.723000	0.723000	0.723000	0.723000
MILL CONCENTRATE GRADE- - -	0.0000	0.0000	0.0000	0.0000	0.5000	0.5000	0.5000	0.5000	0.5000
SMELTER RECOVERY- - - - - -	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000
SMELTER CONCENTRATE GRADE -	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
REFINER RECOVERY- - - - - -	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000
OP. COSTS/UNIT PROCESSED									
SMELTER- - - - - - - - -	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
REFINER- - - - - - - - -	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
TRANS. COSTS/UNIT PROCESSED									
MILL TO SMELTER- - - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
SMELTER TO REFINER - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
REFINER TO MARKET- - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PRICE/UNIT RECOVERED- - - -	0.00	0.00	0.00	0.00	88.50	88.50	88.50	88.50	88.50
UNITS RECOVERED - - - - - -	0.	0.	0.	0.	269221.	269221.	269221.	269221.	269221.
REVENUES- - - - - - - - - -	0.	0.	0.	0.	23826024.	23826024.	23826024.	23826024.	23826024.
DEPLETION - - - - - - - - -	0.	0.	0.	0.	3168861.	3168861.	3168861.	3168861.	3168861.

FINANCIAL SUMMARY * * * * * *

TAXES									
PROPERTY- - - - - - - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
SEVERANCE - - - - - - - - - -	0.	0.	0.	0.	613036.	613036.	613036.	613036.	613036.
INVEST. TAX CREDITS USED- -	0.	0.	0.	0.	504471.	1072967.	880176.	0.	0.
STATE INCOME- - - - - - - -	0.	0.	0.	0.	108601.	233271.	237969.	243089.	248670.
FEDERAL INCOME- - - - - - - -	0.	0.	0.	0.	479471.	1047967.	1283599.	2210472.	2261371.
CASH FLOW									
EXPENSED EXPLORATION +									
TOTAL REVENUES- - - - - - - -	0.	0.	0.	0.	46092945.	46092945.	46092945.	46092945.	46092945.
LESS OPERATING COSTS- - - -	0.	0.	0.	0.	28505052.	28505052.	28505052.	28505052.	28505052.
LESS LOAN INT. PAYMENTS - -	0.	0.	0.	0.	2619015.	2446631.	2258733.	2053924.	1830682.
LESS DEPRECIATION - - - - -	0.	0.	802405.	1604810.	2279688.	2279688.	2279688.	2279688.	2279688.
LESS ROYALTY PAYMENTS - - -	0.	0.	0.	0.	2304647.	2304647.	2304647.	2304647.	2304647.
LESS PROPERTY TAXES - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
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EQUALS INCOME BEFORE TAXES-	0.	0.	-802405.	-1604810.	10384543.	10556927.	10744825.	10949634.	11172876.
LESS TOTAL DEPLETION- - - -	0.	0.	0.	0.	5192272.	5278463.	5372413.	5474817.	5586438.
LESS SEVERANCE TAXES- - - -	0.	0.	0.	0.	613036.	613036.	613036.	613036.	613036.
LESS TAX LOSS CARRY - - - -	0.	0.	0.	0.	2407214.	0.	0.	0.	0.
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EQUALS TAXABLE INCOME - - -	0.	0.	-802405.	-1604810.	2172021.	4665427.	4759376.	4861781.	4973402.
LESS STATE INCOME TAX - - -	0.	0.	0.	0.	108601.	233271.	237969.	243089.	248670.
LESS FEDERAL INCOME TAX - -	0.	0.	0.	0.	479471.	1047967.	1283599.	2210472.	2261371.
PLUS TAX ADJUSTMENTS- - - -	0.	0.	0.	0.	2407214.	0.	0.	0.	0.
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EQUALS NET INCOME - - - - -	0.	0.	-802405.	-1604810.	3991164.	3384188.	3237808.	2408220.	2463361.
PLUS DEPRECIATION - - - - -	0.	0.	802405.	1604810.	2279688.	2279688.	2279688.	2279688.	2279688.
PLUS DEPLETION- - - - - - -	0.	0.	0.	0.	5192272.	5278463.	5372413.	5474817.	5586438.
PLUS DEFFERED DEDUCTIONS- -	0.	0.	0.	0.	407950.	407950.	407950.	407950.	407950.
LESS EQUITY INVESTMENT- - -	23925000.	79500.	1420555.	1420555.	16698887.	2087759.	2275657.	2480466.	2703708.
	======	======	======	======	======	======	======	======	======
EQUALS CASH FLOW- - - - - -	-23925000.	-79500.	-1420555.	-1420555.	-4827814.	9262531.	9022201.	8090208.	8033728.
ANALYSIS FIGURES									
CONTINUOUS RATE OF RETURN -	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	1.478
NOTE - * INDICATES THAT A DUAL RATE OF RETURN EXISTED IN THAT YEAR									
15.00 PCT PRESENT VALUE- -	-22217077.	-22280618.	-23257866.	-24098991.	-26559409.	-22496437.	-19090141.	-16461172.	-14214194.
	1986	1987	1988	1989	1990	1991	1992	1993	1994

GENERAL DATA * * * * * * * * * *

INVESTMENTS									
EXPLORATION - - - - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
LAND ACQUISITION- - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
MINING PREPARATION- - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 1- MINE- - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
LIFE - - - - - - - - - - -	0	0	0	0	0	0	0	0	0
DEPRECIATION - - - - - - -	406233.	406233.	406233.	406233.	406233.	406233.	406233.	406233.	203117.
SALVAGE VALUE- - - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 2- MINE- - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
LIFE - - - - - - - - - - -	0	0	0	0	0	0	0	0	0
DEPRECIATION - - - - - - -	1014875.	507438.	0.	0.	0.	0.	0.	0.	0.
SALVAGE VALUE- - - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 3- MINE- - -	0.	3445510.	3445510.	0.	0.	0.	0.	0.	0.
LIFE - - - - - - - - - - -	0	8	8	0	0	0	0	0	0
DEPRECIATION - - - - - - -	183701.	522539.	861378.	861378.	861378.	861378.	861378.	861378.	861378.
SALVAGE VALUE- - - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 4- PROC- - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
LIFE - - - - - - - - - - -	0	0	0	0	0	0	0	0	0
DEPRECIATION - - - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.

SALVAGE VALUE	0.	0.	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 5- PROC	0.	0.	0.	0.	0.	0.	0.	0.	0.
LIFE	0	0	0	0	0	0	0	0	0
DEPRECIATION	563138.	563138.	563138.	563138.	563138.	563138.	563138.	563138.	563138.
SALVAGE VALUE	0.	0.	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 6- PROC	0.	957951.	0.	0.	0.	0.	0.	957951.	0.
LIFE	0	6	0	0	0	0	0	6	0
DEPRECIATION	111740.	159659.	159659.	159659.	159659.	59659.	159659.	159659.	159659.
SALVAGE VALUE	0.	0.	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 7- MISC	0.	0.	0.	0.	0.	0.	0.	0.	0.
LOAN NUMBER 1	0.	0.	0.	0.	0.	0.	0.	0.	0.
INTEREST RATE	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
INTEREST PAID	1587348.	1322114.	1033009.	717885.	374400.	0.	0.	0.	0.
PRINCIPLE PAID	2947042.	3212276.	3501381.	3816505.	4159998.	0.	0.	0.	0.
LOAN NUMBER 2	0.	0.	0.	0.	0.	0.	0.	0.	0.
INTEREST RATE	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
INTEREST PAID	0.	0.	0.	0.	0.	0.	0.	0.	0.
PRINCIPLE PAID	0.	0.	0.	0.	0.	0.	0.	0.	0.
LOAN NUMBER 3	0.	0.	0.	0.	0.	0.	0.	0.	0.
INTEREST RATE	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
INTEREST PAID	0.	0.	0.	0.	0.	0.	0.	0.	0.
PRINCIPLE PAID	0.	0.	0.	0.	0.	0.	0.	0.	0.
EQUITY CAPITAL	2947042.	7615737.	6946891.	3816505.	4159998.	0.	0.	957951.	0.
WORKING CAPITAL	0.	0.	0.	0.	0.	0.	0.	0.	0.
OPERATING COSTS PER UNIT									
MINE	5.64000	5.64000	5.64000	5.64000	5.64000	5.64000	5.64000	5.64000	5.64000
MILL	7.27800	7.27800	7.27800	7.27800	7.27800	7.27800	7.27800	7.27800	7.27800
TOTAL OVERHEAD	0.00000	0.00000	0.00000	0.00000	0.00000	0.00000	0.00000	0.00000	0.00000
UNITS TREATED	2175035.	2175035.	2175035.	2175035.	2175035.	2175035.	2175035.	2175035.	2175035.
MISC. OPERATING EXPENSES	0.	0.	0.	0.	0.	0.	0.	0.	0.
COMMODITY DATA * * * * * * * *									
KMAG									
ORE GRADE	0.075000	0.075000	0.075000	0.075000	0.075000	0.075000	0.075000	0.075000	0.075000
MILL RECOVERY	0.858000	0.858000	0.858000	0.858000	0.858000	0.858000	0.858000	0.858000	0.858000
MILL CONCENTRATE GRADE	0.2200	0.2200	0.2200	0.2200	0.2200	0.2200	0.2200	0.2200	0.2200
SMELTER RECOVERY	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000
SMELTER CONCENTRATE GRADE	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
REFINER RECOVERY	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000
OP. COSTS/UNIT PROCESSED									
SMELTER	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
REFINER	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
TRANS. COSTS/UNIT PROCESSED									
MILL TO SMELTER	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
SMELTER TO REFINER	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
REFINER TO MARKET	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PRICE/UNIT RECOVERED	35.00	35.00	35.00	35.00	35.00	35.00	35.00	35.00	35.00
UNITS RECOVERED	636198.	636198.	636198.	636198.	636198.	636198.	636198.	636198.	636198.
REVENUES	22266921.	22266921.	22266921.	22266921.	22266921.	22266921.	22266921.	22266921.	22266921.
DEPLETION	2961500.	2961500.	2961500.	2961500.	2961500.	2961500.	2961500.	2961500.	2961500.
SULFATE									
ORE GRADE	0.085600	0.085600	0.085600	0.085600	0.085600	0.085600	0.085600	0.085600	0.085600
MILL RECOVERY	0.723000	0.723000	0.723000	0.723000	0.723000	0.723000	0.723000	0.723000	0.723000
MILL CONCENTRATE GRADE	0.5000	0.5000	0.5000	0.5000	0.5000	0.5000	0.5000	0.5000	0.5000
SMELTER RECOVERY	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000
SMELTER CONCENTRATE GRADE	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
REFINER RECOVERY	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000
OP. COSTS/UNIT PROCESSED									
SMELTER	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
REFINER	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000

TRANS. COSTS/UNIT PROCESSED									
MILL TO SMELTER- - - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
SMELTER TO REFINER - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
REFINER TO MARKET- - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PRICE/UNIT RECOVERED- - - -	88.50	88.50	88.50	88.50	88.50	88.50	88.50	88.50	88.50
UNITS RECOVERED - - - - - -	269221.	269221.	269221.	269221.	269221.	269221.	269221.	269221.	269221.
REVENUES- - - - - - - - - -	23826024.	23826024.	23826024.	23826024.	23826024.	23826024.	23826024.	23826024.	23826024.
DEPLETION - - - - - - - - -	3168861.	3168861.	3168861.	3168861.	3168861.	3168861.	3168861.	3168861.	3168861.
FINANCIAL SUMMARY * * * * * *									
TAXES									
PROPERTY- - - - - - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
SEVERANCE - - - - - - - - -	613036.	613036.	613036.	613036.	613036.	613036.	613036.	613036.	613036.
INVEST. TAX CREDITS USED- -	0.	408415.	344551.	0.	0.	0.	0.	63864.	0.
STATE INCOME- - - - - - - -	254753.	264401.	275844.	291578.	308752.	327472.	327472.	327472.	337628.
FEDERAL INCOME- - - - - - -	2316851.	1996425.	2164645.	2652689.	2809318.	2980045.	2980045.	2916181.	3072666.
CASH FLOW									
EXPENSED EXPLORATION +									
TOTAL REVENUES- - - - - - -	46092945.	46092945.	46092945.	46092945.	46092945.	46092945.	46092945.	46092945.	46092945.
LESS OPERATING COSTS- - - -	28505052.	28505052.	28505052.	28505052.	28505052.	28505052.	28505052.	28505052.	28505052.
LESS LOAN INT. PAYMENTS - -	1587348.	1322114.	1033009.	717885.	374400.	0.	0.	0.	0.
LESS DEPRECIATION - - - - -	2279688.	2159007.	1990408.	1990408.	1990408.	1990408.	1990408.	1990408.	1787291.
LESS ROYALTY PAYMENTS - - -	2304647.	2304647.	2304647.	2304647.	2304647.	2304647.	2304647.	2304647.	2304647.
LESS PROPERTY TAXES - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
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EQUALS INCOME BEFORE TAXES-	11416210.	11802124.	12259829.	12574953.	12918438.	13292838.	13292838.	13292838.	13495055.
LESS TOTAL DEPLETION- - - -	5708105.	5901062.	6129914.	6130362.	6130362.	6130362.	6130362.	6130362.	6130362.
LESS SEVERANCE TAXES- - - -	613036.	613036.	613036.	613036.	613036.	613036.	613036.	613036.	613036.
LESS TAX LOSS CARRY - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
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EQUALS TAXABLE INCOME - - -	5095069.	5288026.	5516878.	5831555.	6175040.	6549440.	6549440.	6549440.	6752557.
LESS STATE INCOME TAX - - -	254753.	264401.	275844.	291578.	308752.	327472.	327472.	327472.	337628.
LESS FEDERAL INCOME TAX - -	2316851.	1996425.	2164645.	2652689.	2809318.	2980045.	2980045.	2916181.	3072666.
PLUS TAX ADJUSTMENTS- - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
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EQUALS NET INCOME - - - - -	2523464.	3027200.	3076389.	2887288.	3056970.	3241923.	3241923.	3305787.	3342263.
PLUS DEPRECIATION - - - - -	2279688.	2159007.	1990408.	1990408.	1990408.	1990408.	1990408.	1990408.	1787291.
PLUS DEPLETION- - - - - - -	5708105.	5901062.	6129914.	6130362.	6130362.	6130362.	6130362.	6130362.	6130362.
PLUS DEFFERED DEDUCTIONS- -	407950.	407950.	407950.	407950.	407950.	407950.	407950.	407950.	407950.
LESS EQUITY INVESTMENT- - -	2947042.	7615737.	6946891.	3816505.	4159998.	0.	0.	957951.	0.
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EQUALS CASH FLOW- - - - - -	7972165.	3879482.	4657770.	7599503.	7425692.	11770643.	11770643.	10876555.	11667866.
ANALYSIS FIGURES									
CONTINUOUS RATE OF RETURN -	4.786	5.986	7.148	8.600	9.663	10.909	11.820	12.459	12.992
NOTE - * INDICATES THAT A DUAL RATE OF RETURN EXISTED IN THAT YEAR									
15.00 PCT PRESENT VALUE- -	-12295023.	-11491188.	-10660520.	-9494006.	-8512941.	-7174446.	-6022393.	-5106131.	-4260121.
	1995	1996	1997	1998	1999	2000	2001	2002	2003
GENERAL DATA * * * * * * * * *									
INVESTMENTS									
EXPLORATION - - - - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
LAND ACQUISITION- - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
MINING PREPARATION- - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 1- MINE- - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
LIFE - - - - - - - - - -	0	0	0	0	0	0	0	0.	0.
DEPRECIATION - - - - - -	0.	0.	0.	0.	0.	0.	0.	0	0
SALVAGE VALUE- - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 2- MINE- - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
LIFE - - - - - - - - - -	0	0	0	0	0	0	0.	0.	0.
DEPRECIATION - - - - - -	0.	0.	0.	0.	0.	0.	0	0	0
							0.	0.	0.

SALVAGE VALUE- - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 3- MINE- - -	2052050.	2052050.	0.	0.	0.	0.	0.	0.	2052050.
LIFE - - - - - - - - - - -	8	8	0	0	0	0	0	0	8
DEPRECIATION - - - - - -	687195.	513013.	513013.	513013.	513013.	513013.	513013.	513013.	513013.
SALVAGE VALUE- - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 4- PROC- - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
LIFE - - - - - - - - - - -	0	0	0	0	0	0	0	0	0
DEPRECIATION - - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
SALVAGE VALUE- - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 5- PROC- - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
LIFE - - - - - - - - - - -	0	0	0	0	0	0	0	0	0
DEPRECIATION - - - - - -	563138.	0.	0.	0.	0.	0.	0.	0.	0.
SALVAGE VALUE- - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 6- PROC- - -	0.	0.	0.	0.	957951.	0.	0.	0.	0.
LIFE - - - - - - - - - - -	0	0	0	0	6	0	0	0	0
DEPRECIATION - - - - - -	159659.	159659.	159659.	159659.	159659.	159659.	159659.	159659.	159659.
SALVAGE VALUE- - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 7- MISC- - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
LOAN NUMBER 1 - - - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
INTEREST RATE- - - - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
INTEREST PAID- - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
PRINCIPLE PAID - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
LOAN NUMBER 2 - - - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
INTEREST RATE- - - - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
INTEREST PAID- - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
PRINCIPLE PAID - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
LOAN NUMBER 3 - - - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
INTEREST RATE- - - - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
INTEREST PAID- - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
PRINCIPLE PAID - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
EQUITY CAPITAL- - - - - - -	2052050.	2052050.	0.	0.	957951.	0.	0.	0.	2052050.
WORKING CAPITAL - - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
OPERATING COSTS PER UNIT									
MINE- - - - - - - - - - - - -	5.64000	5.64000	5.64000	5.64000	5.64000	5.64000	5.64000	5.64000	5.64000
MILL- - - - - - - - - - - - -	7.27800	7.27800	7.27800	7.27800	7.27800	7.27800	7.27800	7.27800	7.27800
TOTAL OVERHEAD- - - - - - -	0.00000	0.00000	0.00000	0.00000	0.00000	0.00000	0.00000	0.00000	0.00000
UNITS TREATED- - - - - - - -	2175035.	2175035.	2175035.	2175035.	2175035.	2175035.	2175035.	2175035.	2175035.
MISC. OPERATING EXPENSES - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
COMMODITY DATA * * * * * * * *									
KMAG									
ORE GRADE - - - - - - - - -	0.075000	0.075000	0.075000	0.075000	0.075000	0.075000	0.075000	0.075000	0.075000
MILL RECOVERY - - - - - - -	0.858000	0.858000	0.858000	0.858000	0.858000	0.858000	0.858000	0.858000	0.858000
MILL CONCENTRATE GRADE- - -	0.2200	0.2200	0.2200	0.2200	0.2200	0.2200	0.2200	0.2200	0.2200
SMELTER RECOVERY- - - - - -	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000
SMELTER CONCENTRATE GRADE -	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
REFINER RECOVERY- - - - - -	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000
OP. COSTS/UNIT PROCESSED									
SMELTER- - - - - - - - -	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
REFINER- - - - - - - - -	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
TRANS. COSTS/UNIT PROCESSED									
MILL TO SMELTER- - - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
SMELTER TO REFINER - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
REFINER TO MARKET- - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PRICE/UNIT RECOVERED- - - -	35.00	35.00	35.00	35.00	35.00	35.00	35.00	35.00	35.00
UNITS RECOVERED - - - - - -	636198.	636198.	636198.	636198.	636198.	636198.	636198.	636198.	636198.
REVENUES- - - - - - - - - -	22266921.	22266921.	22266921.	22266921.	22266921.	22266921.	22266921.	22266921.	22266921.
DEPLETION - - - - - - - - -	2961500.	2961500.	2961500.	2961500.	2961500.	2961500.	2961500.	2961500.	2961500.
SULFATE									
ORE GRADE - - - - - - - - -	0.085600	0.085600	0.085600	0.085600	0.085600	0.085600	0.085600	0.085600	0.085600

MILL RECOVERY - - - - - - - -	0.723000	0.723000	0.723000	0.723000	0.723000	0.723000	0.723000	0.723000	0.723000
MILL CONCENTRATE GRADE- - -	0.5000	0.5000	0.5000	0.5000	0.5000	0.5000	0.5000	0.5000	0.5000
SMELTER RECOVERY- - - - - -	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000
SMELTER CONCENTRATE GRADE -	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
REFINER RECOVERY- - - - - -	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000
OP. COSTS/UNIT PROCESSED									
SMELTER- - - - - - - - -	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
REFINER- - - - - - - - -	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
TRANS. COSTS/UNIT PROCESSED									
MILL TO SMELTER- - - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
SMELTER TO REFINER - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
REFINER TO MARKET- - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PRICE/UNIT RECOVERED- - - -	88.50	88.50	88.50	88.50	88.50	88.50	88.50	88.50	88.50
UNITS RECOVERED - - - - - -	269221.	269221.	269221.	269221.	269221.	269221.	269221.	269221.	269221.
REVENUES- - - - - - - - -	23826024.	23826024.	23826024.	23826024.	23826024.	23826024.	23826024.	23826024.	23826024.
DEPLETION - - - - - - - - -	3168861.	3168861.	3168861.	3168861.	3168861.	3168861.	3168861.	3168861.	3168861.
FINANCIAL SUMMARY * * * * * *									
TAXES									
PROPERTY- - - - - - - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
SEVERANCE - - - - - - - - -	613036.	613036.	613036.	613036.	613036.	613036.	613036.	613036.	613036.
INVEST. TAX CREDITS USED- -	205205.	205205.	0.	0.	63864.	0.	0.	0.	205205.
STATE INCOME- - - - - - - -	356493.	393359.	393359.	393359.	393359.	393359.	393359.	393359.	393359.
FEDERAL INCOME- - - - - - -	3039509.	3375728.	3580933.	3580933.	3517069.	3580933.	3580933.	3580933.	3375728.
CASH FLOW									
EXPENSED EXPLORATION +									
TOTAL REVENUES- - - - - - -	46092945.	46092945.	46092945.	46092945.	46092945.	46092945.	46092945.	46092945.	46092945.
LESS OPERATING COSTS- - - -	28505052.	28505052.	28505052.	28505052.	28505052.	28505052.	28505052.	28505052.	28505052.
LESS LOAN INT. PAYMENTS - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
LESS DEPRECIATION - - - - -	1409992.	672671.	672671.	672671.	672671.	672671.	672671.	672671.	672671.
LESS ROYALTY PAYMENTS - - -	2304647.	2304647.	2304647.	2304647.	2304647.	2304647.	2304647.	2304647.	2304647.
LESS PROPERTY TAXES - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
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EQUALS INCOME BEFORE TAXES-	13873254.	14610575.	14610575.	14610575.	14610575.	14610575.	14610575.	14610575.	14610575.
LESS TOTAL DEPLETION- - - -	6130362.	6130362.	6130362.	6130362.	6130362.	6130362.	6130362.	6130362.	6130362.
LESS SEVERANCE TAXES- - - -	613036.	613036.	613036.	613036.	613036.	613036.	613036.	613036.	613036.
LESS TAX LOSS CARRY - - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
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EQUALS TAXABLE INCOME - - -	7129856.	7867177.	7867177.	7867177.	7867177.	7867177.	7867177.	7867177.	7867177.
LESS STATE INCOME TAX - - -	356493.	393359.	393359.	393359.	393359.	393359.	393359.	393359.	393359.
LESS FEDERAL INCOME TAX - -	3039509.	3375728.	3580933.	3580933.	3517069.	3580933.	3580933.	3580933.	3375728.
PLUS TAX ADJUSTMENTS- - - -	0.	0.	0.	0.	0.	0.	0.	0.	0.
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EQUALS NET INCOME - - - - -	3733854.	4098090.	3892885.	3892885.	3956749.	3892885.	3892885.	3892885.	4098090.
PLUS DEPRECIATION - - - - -	1409992.	672671.	672671.	672671.	672671.	672671.	672671.	672671.	672671.
PLUS DEPLETION- - - - - - -	6130362.	6130362.	6130362.	6130362.	6130362.	6130362.	6130362.	6130362.	6130362.
PLUS DEFFERED DEDUCTIONS- -	407950.	407950.	407950.	407950.	407950.	407950.	407950.	407950.	407950.
LESS EQUITY INVESTMENT- - -	2052050.	2052050.	0.	0.	957951.	0.	0.	0.	2052050.
	======	======	======	======	======	======	======	======	======
EQUALS CASH FLOW- - - - - -	9630107.	9257023.	11103868.	11103868.	10209781.	11103868.	11103868.	11103868.	9257023.
ANALYSIS FIGURES									
CONTINUOUS RATE OF RETURN -	13.340	13.611	13.875	14.090	14.252	14.398	14.519	14.619	14.689
NOTE - * INDICATES THAT A DUAL RATE OF RETURN EXISTED IN THAT YEAR									
15.00 PCT PRESENT VALUE- -	-3659126.	-3161835.	-2648520.	-2206663.	-1856976.	-1529641.	-1247900.	-1005404.	-831401.
	2004	2005	2006	2007	2008	2009	2010		
GENERAL DATA * * * * * * * * * *									
INVESTMENTS									
EXPLORATION - - - - - - - -	0.	0.	0.	0.	0.	0.	0.		
LAND ACQUISITION- - - - - -	0.	0.	0.	0.	0.	0.	0.		

MINING PREPARATION- - - - -	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 1- MINE- - -	0.	0.	0.	0.	0.	0.	0.
LIFE - - - - - - - - - - -	0	0	0	0	0	0	0
DEPRECIATION - - - - - -	0.	0.	0.	0.	0.	0.	0.
SALVAGE VALUE- - - - - -	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 2- MINE- - -	0.	0.	0.	0.	0.	0.	0.
LIFE - - - - - - - - - - -	0	0	0	0	0	0	0
DEPRECIATION - - - - - -	0.	0.	0.	0.	0.	0.	0.
SALVAGE VALUE- - - - - -	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 3- MINE- - -	2052050.	0.	0.	0.	0.	0.	0.
LIFE - - - - - - - - - - -	8	0	0	0	0	0	0
DEPRECIATION - - - - - -	513013.	513013.	513013.	513013.	513013.	513013.	513013.
SALVAGE VALUE- - - - - -	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 4- PROC- - -	0.	0.	0.	0.	0.	0.	0.
LIFE - - - - - - - - - - -	0	0	0	0	0	0	0
DEPRECIATION - - - - - -	0.	0.	0.	0.	0.	0.	0.
SALVAGE VALUE- - - - - -	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 5- PROC- - -	0.	0.	0.	0.	0.	0.	0.
LIFE - - - - - - - - - - -	0	0	0	0	0	0	0
DEPRECIATION - - - - - -	0.	0.	0.	0.	0.	0.	0.
SALVAGE VALUE- - - - - -	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 6- PROC- - -	0.	957951.	0.	0.	0.	0.	0.
LIFE - - - - - - - - - - -	0	6	0	0	0	0	0
DEPRECIATION - - - - - -	159659.	159659.	159659.	159659.	159659.	159659.	159659.
SALVAGE VALUE- - - - - -	0.	0.	0.	0.	0.	0.	0.
INVEST. NUMBER 7- MISC- - -	0.	0.	0.	0.	0.	0.	0.
LOAN NUMBER 1 - - - - - - - -	0.	0.	0.	0.	0.	0.	0.
INTEREST RATE- - - - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000
INTEREST PAID- - - - - -	0.	0.	0.	0.	0.	0.	0.
PRINCIPLE PAID - - - - -	0.	0.	0.	0.	0.	0.	0.
LOAN NUMBER 2 - - - - - - - -	0.	0.	0.	0.	0.	0.	0.
INTEREST RATE- - - - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000
INTEREST PAID- - - - - -	0.	0.	0.	0.	0.	0.	0.
PRINCIPLE PAID - - - - -	0.	0.	0.	0.	0.	0.	0.
LOAN NUMBER 3 - - - - - - - -	0.	0.	0.	0.	0.	0.	0.
INTEREST RATE- - - - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000
INTEREST PAID- - - - - -	0.	0.	0.	0.	0.	0.	0.
PRINCIPLE PAID - - - - -	0.	0.	0.	0.	0.	0.	0.
EQUITY CAPITAL- - - - - - -	2052050.	957951.	0.	0.	0.	0.	0.
WORKING CAPITAL - - - - - -	0.	0.	0.	0.	0.	0.	0.
OPERATING COSTS PER UNIT							
MINE- - - - - - - - - - - - -	5.64000	5.64000	5.64000	5.64000	5.64000	5.64000	5.64000
MILL- - - - - - - - - - - - -	7.27800	7.27800	7.27800	7.27800	7.27800	7.27800	7.27800
TOTAL OVERHEAD- - - - - - -	0.00000	0.00000	0.00000	0.00000	0.00000	0.00000	0.00000
UNITS TREATED- - - - - - - - -	2175035.	2175035.	2175035.	2175035.	2175035.	2175035.	2175035.
MISC. OPERATING EXPENSES - - -	0.	0.	0.	0.	0.	0.	0.
COMMODITY DATA * * * * * * * *							
KMAG							
ORE GRADE - - - - - - - - -	0.075000	0.075000	0.075000	0.075000	0.075000	0.075000	0.075000
MILL RECOVERY - - - - - - -	0.858000	0.858000	0.858000	0.858000	0.858000	0.858000	0.858000
MILL CONCENTRATE GRADE- - -	0.2200	0.2200	0.2200	0.2200	0.2200	0.2200	0.2200
SMELTER RECOVERY- - - - - -	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000
SMELTER CONCENTRATE GRADE -	0.000	0.000	0.000	0.000	0.000	0.000	0.000
REFINER RECOVERY- - - - - -	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000
OP. COSTS/UNIT PROCESSED							
SMELTER- - - - - - - - - -	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
REFINER- - - - - - - - - -	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
TRANS. COSTS/UNIT PROCESSED							
MILL TO SMELTER- - - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000

SMELTER TO REFINER - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000
REFINER TO MARKET- - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PRICE/UNIT RECOVERED- - - -	35.00	35.00	35.00	35.00	35.00	35.00	35.00
UNITS RECOVERED - - - - - -	636198.	636198.	636198.	636198.	636198.	636198.	636198.
REVENUES- - - - - - - - - -	22266921.	22266921.	22266921.	22266921.	22266921.	22266921.	22266921.
DEPLETION - - - - - - - - -	2961500.	2961500.	2961500.	2961500.	2961500.	2961500.	2961500.
SULFATE							
ORE GRADE - - - - - - - - -	0.085600	0.085600	0.085600	0.085600	0.085600	0.085600	0.085600
MILL RECOVERY - - - - - - -	0.723000	0.723000	0.723000	0.723000	0.723000	0.723000	0.723000
MILL CONCENTRATE GRADE- - -	0.5000	0.5000	0.5000	0.5000	0.5000	0.5000	0.5000
SMELTER RECOVERY- - - - - -	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000
SMELTER CONCENTRATE GRADE -	0.000	0.000	0.000	0.000	0.000	0.000	0.000
REFINER RECOVERY- - - - - -	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000	0.000000
OP. COSTS/UNIT PROCESSED							
SMELTER- - - - - - - - -	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
REFINER- - - - - - - - -	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000
TRANS. COSTS/UNIT PROCESSED							
MILL TO SMELTER- - - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000
SMELTER TO REFINER - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000
REFINER TO MARKET- - - -	0.000	0.000	0.000	0.000	0.000	0.000	0.000
PRICE/UNIT RECOVERED- - - -	88.50	88.50	88.50	88.50	88.50	88.50	88.50
UNITS RECOVERED - - - - - -	269221.	269221.	269221.	269221.	269221.	269221.	269221.
REVENUES- - - - - - - - - -	23826024.	23826024.	23826024.	23826024.	23826024.	23826024.	23826024.
DEPLETION - - - - - - - - -	3168861.	3168861.	3168861.	3168861.	3168861.	3168861.	3168861.
FINANCIAL SUMMARY * * * * * *							
TAXES							
PROPERTY- - - - - - - - - -	0.	0.	0.	0.	0.	0.	0.
SEVERANCE - - - - - - - - -	613036.	613036.	613036.	613036.	613036.	613036.	613036.
INVEST. TAX CREDITS USED- -	205205.	63864.	0.	0.	0.	0.	0.
STATE INCOME- - - - - - - -	393359.	393359.	393359.	393359.	393359.	393359.	393359.
FEDERAL INCOME- - - - - - -	3375728.	3517069.	3580933.	3580933.	3580933.	3580933.	3580933.
CASH FLOW							
EXPENSED EXPLORATION +							
TOTAL REVENUES- - - - - - -	46092945.	46092945.	46092945.	46092945.	46092945.	46092945.	46092945.
LESS OPERATING COSTS- - - -	28505052.	28505052.	28505052.	28505052.	28505052.	28505052.	28505052.
LESS LOAN INT. PAYMENTS - -	0.	0.	0.	0.	0.	0.	0.
LESS DEPRECIATION - - - - -	672671.	672671.	672671.	672671.	672671.	672671.	672671.
LESS ROYALTY PAYMENTS - - -	2304647.	2304647.	2304647.	2304647.	2304647.	2304647.	2304647.
LESS PROPERTY TAXES - - - -	0.	0.	0.	0.	0.	0.	0.
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EQUALS INCOME BEFORE TAXES-	14610575.	14610575.	14610575.	14610575.	14610575.	14610575.	14610575.
LESS TOTAL DEPLETION- - - -	6130362.	6130362.	6130362.	6130362.	6130362.	6130362.	6130362.
LESS SEVERANCE TAXES- - - -	613036.	613036.	613036.	613036.	613036.	613036.	613036.
LESS TAX LOSS CARRY - - - -	0.	0.	0.	0.	0.	0.	0.
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EQUALS TAXABLE INCOME - - -	7867177.	7867177.	7867177.	7867177.	7867177.	7867177.	7867177.
LESS STATE INCOME TAX - - -	393359.	393359.	393359.	393359.	393359.	393359.	393359.
LESS FEDERAL INCOME TAX - -	3375728.	3517069.	3580933.	3580933.	3580933.	3580933.	3580933.
PLUS TAX ADJUSTMENTS- - - -	0.	0.	0.	0.	0.	0.	0.
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EQUALS NET INCOME - - - - -	4098090.	3956749.	3892885.	3892885.	3892885.	3892885.	3892885.
PLUS DEPRECIATION - - - - -	672671.	672671.	672671.	672671.	672671.	672671.	672671.
PLUS DEPLETION- - - - - - -	6130362.	6130362.	6130362.	6130362.	6130362.	6130362.	6130362.
PLUS DEFFERED DEDUCTIONS- -	407950.	407950.	407950.	407950.	407950.	407950.	407950.
LESS EQUITY INVESTMENT- - -	2052050.	957951.	0.	0.	0.	0.	0.
	======	======	======	======	======	======	======
EQUALS CASH FLOW- - - - - -	9257023.	10209781.	11103868.	11103868.	11103868.	11103868.	16769868.
ANALYSIS FIGURES							
CONTINUOUS RATE OF RETURN -	14.748	14.803	14.853	14.895	14.931	14.962	15.001

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  NOTE - * INDICATES THAT A DUAL RATE OF RETURN EXISTED IN THAT YEAR
   15.00 PCT PRESENT VALUE- -     -681634.   -539462.   -406377.   -291830.   -193239.   -108380.      1927.
LAST YEAR ADDITIONS TO CASH FLOW
  CUMULATIVE WORKING CAPITAL          5666000.
  CUMULATIVE SALVAGE VALUE                  0.
OTHER CUMULATIVE VALUES
  REVENUES                         1382788350.
  ROYALTIES                          69139417.
  PROPERTY TAXES                            0.
  SEVERENCE TAXES                    18391085.
  STATE INCOME TAXES                 10043848.
  FEDERAL INCOME TAXES               87181903.
  CASH FLOW                         254880979.

#EOT 6178  SYSTEM/BACKUP on SYSTEM
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U. S. GOVERNMENT PRINTING OFFICE : 1980 0 - 314-720