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Benefit Cost Analysis of the Coal Extraction R&D Program

J. ANTIZZO
A. COHEN
J. HEIZER

MAY 1976

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ABSTRACT

This document is the final report of the benefit/cost analysis of the Bureau of Mines' Coal Extraction R&D Program. The analysis is designed to generate the various major benefits that will flow from a successful Coal Extraction R&D Program and relate these benefits to the R&D costs.

ACKNOWLEDGEMENTS

This document is based, in part, on the preliminary work done by:

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who were members of the Engineering and Economic Studies Group of MITRE's Energy, Resources and Environmental Systems Engineering Department.

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1.0 INTRODUCTION

This document presents the results of the benefit/cost analysis performed by The MITRE Corporation on the Bureau of Mines' Coal Extraction R&D Program. This analysis was designed to generate the various major benefits that would flow from a successful Coal Extraction R&D Program, and to relate these benefits to the associated R&D costs.

1.1 Background

A description of the Department of Interior's planned research effort for fossil fuels for a six-year period from FY 75 through FY 80 is given in Chapter 5 of Energy Research Program of the U.S. Department of Interior, which was published in June 1974 by the Office of Research and Development of DOI. This June plan contained only a preliminary benefit/cost analysis which was based on estimates of the impact the coal extraction research efforts would have on the cost of energy derived from coal. Only two estimates were made and used in the analysis (i.e., the change in cost of surface-mined coal and the change in cost of underground-mined coal).

On 5 August 1974, the Bureau of Mines contracted with the MITRE Corporation to conduct an in-depth benefit/cost analysis of the Coal Extraction R&D Program. The effort was separated into three phases. The first phase, a one-month effort, consisted of developing a methodology and performing a preliminary analysis. The phase-one effort culminated in a report entitled Coal Extraction R&D Program--Preliminary Benefit/Cost Analysis, dated September 1974.

The second phase of the study effort resulted in a report dated December 1974 and titled Scenarios for Benefit/Cost Analysis of the Coal Extraction R&D Program. The second-phase effort consisted of a description of the current status of mining, and an evaluation of selected production scenarios.

The third phase was the actual detailed analysis of the benefit/cost ratios associated with the Cost Extraction R&D Program, which resulted in this document.

1.2 Summary

Our purpose is to determine the benefits and costs associated with the Coal Extraction R&D Program of the Bureau of Mines. These benefits and costs are dependent on the specific technological and management projects of the R&D program, the measures of effectiveness used and the anticipated scenario of coal mine development and utilization anticipated. Each of these issues is addressed briefly here and in more detail in later sections.

The Coal Extraction R&D Program is an integrated, on-going program whose purpose is to develop improved equipment and techniques that can be combined in various ways by coal mine operators into mining systems that will meet their specific needs. The improved techniques and equipment that will result from this R&D program will be designed to help the coal mine operator:

a) increase productivity

b) increase safety

c) reduce production costs

d) reduce capital costs

e) reduce environmental damage

f) increase resource recovery

g) add to recoverable reserves

The Coal Extraction R&D Program is discussed in further detail in Section 3.2.

The R&D projects that make up this program are grouped into technological and management areas called subprograms which are identified in Table 1-1.

The benefit/cost analysis was performed for the funding period 1975 through 1980 and the benefits were calculated for the period 1980 through 2020. Thus, the equipment and technologies generated by the R&D program during the period 1975-1980 were examined in terms of total benefits to society over the period 1980-2020 for various coal production projections. Specific hypothetical new systems were postulated for this analysis which represented those systems that the R&D program, if successful, will produce in the early 1980's. These hypothetical new systems are described more fully in Section 3.3 and individually in later sections of the report. They were then compared to the systems the coal mine operators would use if there had been no Coal Extraction R&D Program. This comparison was done in terms of the following measures of effectiveness:

a) Capital cost savings

b) Total cost savings

c) Manpower savings due to increased productivity

TABLE 1-1

SUBPROGRAMS

1. High-Speed Coal Mine Development
2. Automated Continuous Miner
3. Automated Longwall Mining
4. Continuous Haulage
5. Automated Roof Support
6. Thick and Multiple Seam Mining
7. Environmental Protection
8. Methane Removal and Recovery
9. Surface Mining - Extraction and Reclamation
10. Surface Mining - Equipment
11. Advanced Mining Systems

d) Manpower savings due to accident reduction

e) Reduction in production costs

f) Reduction in environmental damage

g) Improved resource recovery

h) Additions to recoverable reserves

Total benefits from a hypothetical new system depend on the extent to which industry utilizes the system. This, in turn, is a function of the production scenario assumed for coal production. Two production scenarios were used: Scenario 1 is a low production scenario which represents meeting only our most urgent coal needs. Scenario 2 is a high production scenario which represents meeting all United States coal needs over the period 1980-2020. The scenarios are further described in Section 3.5.

A summary of the benefit/cost analysis for the hypothetical new systems developed under Scenario 1 is presented in Table 1-2. A similar summary of the benefit/cost analysis for Scenario 2 is given by Table 1-3. Entries in the two tables with "--" are either not relevant or are smaller than can be expressed by the number of digits used for the entry.

Total costs for each hypothetical system are the total R&D costs associated with the development of the system. The benefits of the systems, expressed as benefit/cost ratios, are total cost savings, capital cost savings, man days saved and additions to recoverable reserves. The benefit/cost ratios for the total R&D program were calculated by summing the benefits associated with the individual hypothetical new systems divided by the total discounted R&D program funding. The tables

TABLE 1-2

SUMMARY OF BENEFIT/COST ANALYSIS--SCENARIO 1

BENEFITS-BENEFIT/COST RATIOS	HYPOTHETICAL NEW SYSTEMS					R&D PROGRAM (TOTAL)
	HIGH SPEED MINE DEVELOPMENT	AUTOMATED LONGWALL MINING	ADVANCED ROOM & PILLAR MINING	THICK & MULTIPLE SEAM MINING	SURFACE MINING	
DISCOUNTED R&D FUNDING ($MILLIONS)	36.55	73.78	92.36	20.29	62.80	285.78
BENEFIT/COST RATIOS						
TOTAL COST SAVINGS ($/R&D$)	13.34	16.88	9.69	5.68	50.30	20.65
CAPITAL COST SAVINGS ($/R&D$)	1.34	8.72	0.89	0.58	19.10	6.95
MANDAYS SAVED (MANDAYS/R&D$)						
- DUE TO ACCIDENT REDUCTION	--	0.03	--	--	0.16	0.04
- DUE TO INCREASED PRODUCTIVITY	--	1.47	3.20	--	--	1.41
ADDITIONS TO RECOVERABLE RESERVES (TONS/R&D$)	--	1.39	2.07	1079.79	--	77.69
BENEFITS						
REDUCTION IN PRODUCTION COSTS ($/TON)	--	4.07	1.87	5.69	1.78	2.41
INCREASE IN RESOURCE RECOVERY	--	LARGE	LARGE	LARGE	SMALL	LARGE
REDUCTION IN ENVIRONMENTAL DAMAGE	--	MODERATE	MODERATE	SMALL	MODERATE	MODERATE

TABLE 1-3

SUMMARY OF BENEFIT/COST ANALYSIS--SCENARIO 2

BENEFITS-BENEFIT/COST RATIOS	HYPOTHETICAL NEW SYSTEMS					R&D PROGRAM (TOTAL)
	HIGH SPEED MINE DEVELOPMENT	AUTOMATED LONGWALL MINING	ADVANCED ROOM & PILLAR MINING	THICK & MULTIPLE SEAM MINING	SURFACE MINING	
DISCOUNTED R&D FUNDING ($MILLIONS)	36.55	73.78	92.36	20.29	62.80	285.78
BENEFIT/COST RATIOS						
TOTAL COST SAVINGS ($R&D$)	16.38	24.48	12.93	5.91	85.20	31.74
CAPITAL COST SAVINGS ($/R&D$)	1.70	12.57	1.27	0.60	31.70	10.88
MANDAYS SAVED (MANDAYS/R&D$)						
- DUE TO ACCIDENT REDUCTION	--	0.04	--	--	0.29	0.07
- DUE TO INCREASED PRODUCTIVITY	--	1.90	2.95	--	--	1.44
ADDITIONS TO RECOVERABLE RESERVES (TONS/R&D$)	--	1.39	2.07	1079.79	--	77.69
BENEFITS						
REDUCTION IN PRODUCTION COSTS ($/TON)	--	4.07	1.87	5.69	1.51	2.35
INCREASE IN RESOURCE RECOVERY	--	LARGE	LARGE	LARGE	SMALL	LARGE
REDUCTION IN ENVIRONMENTAL DAMAGE	--	MODERATE	MODERATE	SMALL	MODERATE	MODERATE

also indicate reductions in production costs, increases in resource recovery and reductions in environmental damage.

The data of Table 1-2 indicate that the surface mining program is anticipated to be the most productive in total cost savings (benefit/cost ratio = 50.30). By comparison the next most productive program, automated longwall mining, has a total cost savings benefit/cost ratio of 16.88, only one third that of strip mining.

The largest reductions in production costs occur for thick and multiple seam mining ($5.69/ton) and automated longwall mining ($4.07/ton). The reduction in production costs per ton for surface mining is small even though for the total cost the benefit/cost ratio is large. This indicates that the proportion of R&D funding spent on the surface mining technology is relatively small compared to total capital expenditures and production from surface mines.

The R&D program produces large increases in resource recovery for all technologies except surface mining. In addition, it produces large additions to recoverable reserves for thick and multiple seam mining (1079.79 tons/R&D$). Reduction in environmental damage is moderate for all programs except the thick and multiple seam program.

The data for Scenario 2 (Table 1-3) is quite similar to the data for Scenario 1 (Table 1-2). The total R&D program funding remains the same in both scenarios. The primary differences occur in total cost savings and capital cost savings which increase for all technologies in Scenario 2. Scenario 2 assumes a greater production of coal over

Scenario 1, requiring the opening of larger numbers of mines and the greater utilization of the new technologies producing greater cost savings. The largest increases occur for strip mining where total cost savings ($/R&D$) increase from 50.30 to 85.20 and capital cost savings ($/R&D$) increase from 19.10 to 31.70. Other benefits such as man-days saved and reduction in production costs differ very little between Scenarios 1 and 2.

The benefit/cost data presented in Table 1-2 and Table 1-3 depend on a variety of assumptions and limitations of data sources which are discussed in Section 2. Section 3 develops the methodology, subject to the assumptions and limitations discussed in Section 2, used to generate the benefit/cost data. Section 4 presents general sensitivity studies designed to show how various subprogram goals affect the benefits associated with the various hypothetical new systems. The sensitivity studies also show how benefits are affected by many of the assumptions and limitations discussed in Section 2.

2.0 ASSUMPTIONS AND LIMITATIONS OF DATA SOURCES

The data presented in the summary and later sections of the report entailed a large number of assumptions and limitations of data sources in order to allow the development of meaningful projections concerning the effect of R&D activity. These assumptions and limitations consist of simplifications of complex relationships to the point that they can be analytically treated. In addition, many estimates were made to overcome the lack of data, especially that associated with future projections.

2.1 Limitations

2.1.1 Data

During the course of this study, numerous problems with identifying data sources and obtaining data from these sources were encountered. These data problems are reflected in the benefit/cost analysis in that additional assumptions had to be generated in order to perform the analysis. The detailed assumptions were made with respect to specific subprograms and are described in Sections 5.0 through 9.0 in which the detailed analysis is described.

2.1.2 Industrial R&D

There is a severe lack of information about industrial R&D plans in the development of mining equipment and mining systems. For the purpose of this study it was assumed that the R&D conducted by industry will not significantly affect the characteristics of the new mining systems or their availability dates as specified in the later sections of this document. This assumption is supported by a recent survey

conducted by Battelle which indicated that only "evolutionary" changes are being considered which involve only design modifications to current equipment and systems.

2.1.3 Mine Models

During the course of this study MITRE reviewed available mine models. Several computerized models were identified that were designed for site-specific applications and, therefore, required large amounts of data to operate. In view of the data limitations described above, it was considered infeasible to use these models in the analysis. The models used were those developed by the Bureau of Mines by Sidney Katell and are described in Information Circulars 8535, 8632, and 8641. Although these mine models were sufficient for the analysis presented in this document, they are very general and any future benefit/cost analysis should be supported through the development of more detailed models.

2.2 Assumptions

As with any study which attempts to predict the future, a number of assumptions had to be made to facilitate the analysis presented in this document. These assumptions can be separated into two categories: (1) general assumptions which apply to the complete study and (2) those which pertain only to a particular subprogram. The general assumptions are presented in this section, and the assumptions specific to a subprogram are presented in the sections associated with the particular subprogram.

2.2.1 Subprogram Independence

Throughout the course of the study it was assumed that the benefits derived from one subprogram did not have any impact upon other subprogram benefits. Of course, this is not completely true. Some of the subprograms are to result in competing systems such as the Automated Continuous Miner vs. the Automated Longwall Miner. In these cases, success of one research effort will impact upon the benefits to be derived from the other research efforts. Throughout the study, attempts were made to estimate market potential and therefore the benefits that would occur if all research efforts were successful and met their objectives. If, however, in reality, one such program research effort is less than successful, then its benefits will be lower than projected and the benefits derived from competing subprogram research efforts will be higher. However, the impact of this assumption on benefits can be expected to be small.

2.2.2 Hypothetical New Systems

The improved equipment and techniques that are to be developed by the R&D program are examined in terms of five hypothetical new mining systems made up of a specified combination of equipment and techniques. It is assumed that these hypothetical new systems represent a reasonable configuration of these packages that coal mine operators will put together for coal mine operation. These new systems do not completely define or limit the expected results or goals of the individual subprograms--they are merely used as a tool of analysis.

It is assumed that future government policy with respect to use of coal will not significantly increase the risk associated with

investments in coal mining. If either of these assumptions are violated, then the required rate of return will be higher than that used in this study.

2.2.3 Constant Dollars

All analysis in this study was conducted using constant 1975 dollars. Although this approach does provide a convenient framework for the analysis, its simplicity assumes that inflation rates for the resources required to mine coal are the same; i.e., the inflation rate for labor is or will be the same as the inflation rate for capital equipment.

2.3 References

References in each section are indicated by a superscript number in parentheses which refers to a numbered item on the reference page at the end of the section. The reference lists at the end of each section refer to the alphabetical list of complete reference citations at the end of the document.

3.0 BENEFIT/COST METHODOLOGY

This section describes the methodology by which the benefits and the benefit/cost ratios were developed for the Coal Extraction R&D Program. These benefits and benefit/cost ratios were calculated based on:

1. the description of the Coal Extraction R&D Program,
2. the description of the hypothetical new systems that are a result of the R&D program,
3. the description of the "current systems" which are those systems that will be used by the coal mine operators if there were no R&D program, and
4. the description of the coal production scenarios which specifies the number of new mines opening each year during the period 1980 through 2020.

The basic procedure used in developing benefits was to compare the hypothetical new systems to the appropriate current systems for the typical mine and determine the savings that would result. These savings were then applied to all new mines in the production scenario that would use the specific system, and the benefits were then accumulated over the evaluation period to get the total benefits. All dollar benefits and all costs were discounted back to the base year of 1975. The details of this procedure are described in more detail in the following subsections.

3.1 Benefit Types

The benefits examined in this analysis are classified as follows:

1. Dollar Benefits
   a. Total Cost Savings
   b. Capital Cost Savings

2. Quantitative Benefits that are a Function of Production

   a. Manpower saved due to increases in productivity
   b. Manpower saved due to accident reduction

3. Quantitative Benefits Independent of Production: Number of tons of coal added to recoverable reserves

4. Qualitative Benefits

   a. Reduction of Environmental Damage
   b. Increase in Resources Recovery

3.1.1 Dollar Benefits

The various dollar benefits are initially calculated for a typical mine for each year of mine life (20 years) and then discounted at 16 percent before taxes to the first year of mine life. Then, these benefits are multiplied by the number of new mines using the particular new system for a given year, giving the total benefit amount for each benefit type for the particular year in the production scenario. These benefits are then discounted back to the base year 1975 at a 12 percent discount rate for each year of the production scenario starting from the first year of system availability and then summed over the benefit evaluation period 1980 through 2020.

3.1.2 Quantitative Benefits that are a Function of Production

Quantitative non-dollar benefits that are a function of production are initially expressed as man-days saved per millions of tons of coal mined. Thus, the total benefit is equal to the product of total production and man-days saved per million tons. The factors used for increased productivity and for accident reduction are given in each of the sections on the individual new system descriptions.

3.1.3 Qualitative Benefits

Qualitative benefits are expressed in terms of judgmental values such as "small", "moderate", and "large". There are two benefits in this category and they are "impact on the environment" and "increase in resource recovery."

3.2 The Coal Extraction R&D Program

The Bureau of Mines' Coal Extraction Program is an integrated set of R&D projects designed to produce improved equipment and techniques to be used by coal mine operators in various combinations to meet their specific needs. This program is an on-going effort by the Bureau of Mines to continually initiate new concepts and develop these concepts in the form of commercially useful technology packages. The R&D program is divided into technological and management areas called subprograms which are described in terms of project and subprogram goals that can be used as a criteria for measuring the success of subprogram projects. These subprogram goals are based on engineering estimates of projected results if the projects are successful.

The subprograms and their funding schedule for the time period 1975 through 1980 are given in Table 3-1. Table 3-2 presents the discounted funding schedule for the subprograms in which the funds are discounted at 12 percent to the base fiscal year 1975. Table 3-3 ranks the subprograms by percent of total R&D program funding over the cost evaluation period 1975-1980.

TABLE 3-1

COAL EXTRACTION R&D PROGRAM FUNDING BY FISCAL YEAR
FOR PERIOD 1975 THROUGH 1980

	SUBPROGRAM	R&D FUNDING BY FISCAL YEAR ($MILLION)						TOTAL	% OF TOTAL R&D FUNDING
		75	76	77	78	79	80		
1.	HIGH SPEED MINE DEVELOPMENT	4.0	7.0	6.0	6.0	5.5	8.0	36.5	9.5
2.	AUTOMATED LONGWALL MINING	8.0	6.0	6.0	11.0	15.0	10.0	56.0	14.5
3.	AUTOMATED CONTINUOUS MINER	2.0	3.0	4.0	5.0	3.0	1.0	18.0	4.7
4.	CONTINUOUS HAULAGE	5.0	9.0	8.0	9.0	8.0	8.0	47.0	12.2
5.	AUTOMATED ROOF SUPPORT	3.0	4.0	5.0	5.0	5.0	4.0	26.0	6.7
6.	WESTERN COAL	1.0	3.0	4.0	5.0	5.0	6.0	24.0	6.2
7.	ENVIRONMENTAL PROTECTION	5.0	6.0	5.0	5.0	5.0	6.0	32.0	8.3
8.	METHANE REMOVAL & RECOVERY	2.0	3.0	2.0	2.0	1.5	1.0	11.5	3.0
9.	SURFACE MINING-EXTRACTION	4.6	5.0	4.5	5.0	5.0	8.0	32.1	8.3
10.	SURFACE MINING-EQUIPMENT	4.0	7.5	7.0	8.0	8.0	10.0	44.5	11.5
11.	ADVANCED MINING SYSTEMS	3.0	6.0	9.0	13.0	13.0	14.0	58.0	15.0
	TOTAL	41.6	59.5	60.5	74.0	74.0	76.0	385.6	100.0

TABLE 3-2

COAL EXTRACTION R&D PROGRAM DISCOUNTED FUNDING BY FISCAL YEAR
(12% DISCOUNT TO BEGINNING OF 1975)

		DISCOUNTED R&D FUNDING BY FISCAL YEAR @12% ($MILLION)						
	SUBPROGRAM	75	76	77	78	79	80	TOTAL
1.	HIGH SPEED MINE DEVELOPMENT	4.00	6.25	4.78	4.27	3.50	4.54	27.34
2.	AUTOMATED LONGWALL MINING	8.00	5.36	4.78	7.83	9.53	5.67	41.17
3.	AUTOMATED CONTINUOUS MINER	2.00	2.68	3.19	3.56	1.91	0.57	13.91
4.	CONTINUOUS HAULAGE	5.00	8.04	6.38	6.41	5.08	4.54	35.45
5.	AUTOMATED ROOF SUPPORT	3.00	3.57	3.99	3.56	3.18	2.27	19.57
6.	WESTERN COAL	1.00	2.68	3.19	3.56	3.18	3.40	17.01
7.	ENVIRONMENTAL PROTECTION	5.00	5.36	3.99	3.56	3.18	3.40	24.49
8.	METHANE REMOVAL	2.00	2.68	1.59	1.42	0.95	0.57	9.21
9.	SURFACE MINING-EXTRACTION	4.60	4.46	3.59	3.56	3.18	4.54	23.93
10.	SURFACE MINING-EQUIPMENT	3.00	5.36	7.17	9.25	8.26	7.94	32.72
11.	ADVANCED MINING SYSTEMS	3.00	5.36	7.17	9.25	8.26	7.94	40.98
	TOTAL	41.60	53.14	48.23	52.67	47.03	43.11	285.78

TABLE 3-3

SUBPROGRAM RANKING BY PERCENTAGE OF TOTAL FUNDING

RANK	SUBPROGRAM	% OF TOTAL R&D FUNDING
1	ADVANCED MINING SYSTEMS	15.0
2	AUTOMATED LONGWALL MINING	14.5
3	CONTINUOUS HAULAGE	12.2
4	SURFACE MINING-EQUIPMENT	11.5
5	HIGH-SPEED MINE DEVELOPMENT	9.5
6	SURFACE MINING-EXTRACTION	8.3
7	ENVIRONMENTAL PROTECTION	8.3
8	AUTOMATED ROOF-SUPPORT	6.7
9	THICK & MULTIPLE SEAM MINING	6.2
10	AUTOMATED CONTINUOUS MINER	4.7
11	METHANE REMOVAL & RECOVERY	3.0
	TOTAL	100.0

It is the subprogram descriptions that serves as a basis for developing hypothetical new systems used in this benefit/cost analysis.

3.3 The Hypothetical New Systems

In order to determine the benefits that will flow from the R&D program it is necessary to postulate the new systems that will result from a successful R&D program. These hypothetical new systems are described in terms of characteristics specified as subprogram goals. From these descriptions a comparison can be made between these new systems and those current systems that the coal mine operators would use if these new systems were not available (i.e., no R&D program). The new systems postulated for this analysis are:

1. High-speed Mine Development Systems.
2. Advanced Room and Pillar Mining Systems.
3. Automated Longwall Mining Systems.
4. Thick and Multiple Seam Mining Systems.
5. Surface Mining Systems.

Table 3-4 ranks these new systems by funding level. These new systems are based on the eleven subprograms identified in Section 3.2 and their R&D costs are distributed among these new systems as specified in Table 3-5. This cost distribution is based on the percent of subprogram support to the development of the new system, and is critical in determining the benefit/cost ratios of the individual new systems.

Each new system is described in more detail in individual sections later in this document.

TABLE 3-4

HYPOTHETICAL NEW SYSTEMS RANKED BY PERCENTAGE OF R&D FUNDING

RANK	NEW SYSTEM	FUNDING $ MILLION	% OF TOTAL
1	ADVANCED ROOM AND PILLAR MINING	123.25	32.0
2	AUTOMATED LONGWALL MINING	100.55	26.1
3	SURFACE MINING	85.30	22.1
4	HIGH-SPEED MINE DEVELOPMENT	48.00	12.4
5	THICK & MULTIPLE SEAM MINING	28.50	7.4
	TOTAL	385.60	100.0

(See Table 3-5 for detailed funding)

TABLE 3-5

HYPOTHETICAL NEW SYSTEM FUNDING

HYPOTHETICAL NEW SYSTEMS	SUPPORTING SUBPROGRAMS	% OF SUBPROGRAM FUNDING	NEW SYSTEM FUNDING	% SYS. FUND
HIGH SPEED MINE DEVELOPMENT (HSMD)	HIGH SPEED MINE DEVELOPMENT	100%	36.50	76.0
	METHANE REMOVAL & RECOVERY	100%	11.50	24.0
		TOTAL	48.00	100.0
ADVANCED ROOM & PILLAR MINING (ARPM)	AUTOMATED CONTINUOUS MINER	100%	18.00	14.6
	CONTINUOUS HAULAGE	75%	35.25	28.6
	AUTOMATED ROOF SUPPORT	100%	26.00	21.1
	ENVIRONMENTAL PROTECTION	65%	20.80	16.9
	ADVANCED MINING SYSTEMS	40%	23.20	18.8
		TOTAL	123.25	100.0
AUTOMATED LONGWALL MINING (ALWM)	AUTOMATED LONGWALL MINING	100%	56.00	55.7
	CONTINUOUS HAULAGE	25%	11.75	11.6
	ENVIRONMENTAL PROTECTION	30%	9.60	9.6
	ADVANCED MINING SYSTEMS	40%	23.20	23.1
		TOTAL	100.55	100.0
THICK & MULTIPLE SEAM MINING (TMSM)	MINING SYSTEMS FOR WESTERN COAL	100%	24.00	84.2
	ENVIRONMENTAL PROTECTION	5%	1.60	5.6
	ADVANCED MINING SYSTEMS	5%	2.90	10.2
		TOTAL	28.50	100.0
SURFACE MINING (SM)	SURFACE MINING-EXTRACTION	100%	32.10	37.6
	SURFACE MINING-EQUIPMENT	100%	44.50	52.1
	ADVANCED MINING SYSTEMS	15%	8.70	10.3
		TOTAL	85.30	100.0
	R&D PROGRAM TOTAL		385.60	--

3.4 The "Current Systems"

The "current systems" are those systems that will be used by coal mine operators in the absence of any Coal Extraction R&D Program, and serve as a basis for comparison with the new system from which benefits (positive or negative) can be derived. The assumption has been made that the R&D conducted by the mining equipment industry will not significantly effect the characteristics of mining systems, and thus, the current system in use will not change significantly in the near future. This assumption was discussed in Section 2.1.2 of this document.

The current systems are described in terms of economic mine models and in terms of performance characteristics. Economic models developed by Sidney Katell of the Bureau of Mines were used to determine cost comparisons between new systems and associated current systems. These economic models are very general but were sufficient for the purpose of this analysis. Certain modifications were made to these models to take into account various engineering developments in equipment related to individual new systems so that certain cost comparisons could be made.

The comparison of the current system with the new system assumed that if the new system replaced the current system in a typical mine then the cost savings could be estimated over the life of the mine.

Descriptions of current systems are presented in later sections of this document.

3.5 Coal Production Scenarios

Since the magnitude of the benefits is a function of the use of the hypothetical new systems, coal production scenarios were developed for the benefit evaluation period 1975 through 2020, over which the benefits were to be evaluated. These coal production scenarios were expressed in terms of annual production in millions of tons of coal mined over the evaluation period and in terms of new mine openings each year over the evaluation period.

There are two scenarios used in this analysis. Scenario 1 is based on the business-as-usual scenarios developed by the Project Independence Blueprint (PIB) Coal Task Force and represents a feasible low coal production situation. This scenario is called the low production scenario. Scenario 2 is based on MITRE's analysis of potential coal demand and represents a high coal production situation. This scenario is called the high production scenario.

In Appendix A, Tables A-1 and A-2 present the annual coal production for scenario 1 and 2 for underground and surface mines. The values in these two tables are given in millions of tons.

Table A-3 and A-4 present the two scenarios in terms of new mine openings assuming that the average mine life is twenty years. New mine openings are expressed in terms of million tons per year. Thus, an entry in the table under "mine size" of 5.0 means that for

that particular year new mines of a size that produce 5 million tons of coal per year started production and they will generate 3.4 million tons of coal per year. This implies that 0.68 ($=\frac{3.4}{5}$) new mines of the specified size will start production that year. In the tables that specify new mines these new mines are classified by:

a) Province

b) Mine Type

c) Mine Size

In order to generate meaningful benefits and benefit/cost ratios, it was necessary to allocate the total annual coal production for each scenario among the various new systems so that there is no overlap of production between new systems that are production systems. Only the High Speed Mine Development system will overlap all underground mining systems in terms of production as it is not a production system but a development system.

3.6 Market Penetration

Once the subprograms for a particular new system have been completed successfully it takes time to convert these results into a commercially acceptable system that will be manufactured by private industry and bought by coal mine operators. Each new system has a system availability date associated with it that indicates the year

it becomes commercially available. Included with the system availability data is a market penetration schedule which is an estimate, by year, of what percent of those new mines that are potential users will actually use the system.

It is assumed that the new system, when it becomes commercially available will start replacing the current system. This replacement will occur only in newly opened mines since retrofitting a new system to existing mines is not considered to be practical. If retrofitting were to occur, a separate analysis would be required.

The coal production scenario that is selected must be categorized in ways that permit the new system market potential to be clearly defined. If this is not done, the specification of market penetration factors will be exceedingly difficult. As a minimum, it will be necessary to define production on the basis of region (Eastern, Interior, Western, Rocky Mountain) and type of mine (surface and underground mines). Additional refinements, which may be essential for the analysis of some systems, include mine depth, seam thickness, coal type, methane content, level of acidity, mining method, and geological conditions. The required level of detail will be determined by specific characteristics and limitations of the new systems developed by the R&D program.

The result of this scenario development activity is the establishment of a realistic coal production schedule that delineates the potential coal production for the new system. The potential production thus defined must be adjusted to reflect the requirement that

new systems be implemented only on new mines as they are opened. This adjustment is accomplished by the following steps:

- Determine the initial year of new system implementation
- Equate production in prior years to zero
- Reduce production in future years by the amount of coal produced from mines that opened prior to the initial year

The adjusted potential production is divided by the assumed yearly production of the model mine to yield the potential number of new mines that could employ the new system. The new system market potential thus defined forms one of the basic data inputs to the benefit/cost calculation procedure.

Market penetration factors indicate the fraction of market potential that will be utilized by the new system. The rate at which market penetration increases from year to year depends upon many factors--the primary ones being the ability to manufacture and transport new mining equipment, the cost of the new equipment, and the reluctance of mine developers to accept new or unproven techniques.

The number of new mines (market potential) is multiplied by the penetration factor to yield the actual number of new mines that will employ the new system. Total production from the new system is calculated by assuming that once a mine opens, it produces according to the model mine. That is, each mine produces the assumed average yearly production for a number of future years equal to the assumed mine lifetime.

4.0 GENERAL SENSITIVITY ANALYSIS

A general sensitivity analysis was performed to determine how the benefits of a new system are affected by variation in their major characteristics as represented by major system parameters. It has been assumed that the effects of parameter changes are additive and several test cases show that this is the case within one or two percent.

In performing this general sensitivity analysis for new mining systems it was assumed that these new systems could be adequately characterized by the following set of parameters:

a) P--Percent increase in mine productivity at the working face.

b) M--Percent change in direct labor per production unit.

c) S--Percent change in operating supply costs.

d) E--Percent change in environmental protection costs.

e) I--Percent change in capital costs.

f) MPL--The maximum level of market penetration.

g) MPT--The length of time to maximum market penetration.

h) AT--System availability date.

This analysis was performed for two areas, underground mining, and surface mining, each of which were evaluated in terms of the two production scenarios.

The benefits measured in this sensitivity study are:

a) Total cost of savings.

b) Capital cost of savings.

c) Man-days saved due to accident reduction.

d) Reduction in production costs per ton.

The system parameters, the system models, and the results of this sensitivity study are described in the following subsections.

4.1 System Parameters

This subsection defines the system parameters used in the sensitivity analysis.

4.1.1 P--Percentage Increase in Mine Productivity

The increase in mine productivity (P) is a measure of the increased rate of coal production by a production unit (a collection of underground or surface mining equipment that can operate as a unit). Since the annual production for the model mine is fixed in this analysis, any increase in the production rate means that the equipment operates for a shorter period of time, or fewer production units are needed to maintain a fixed annual production rate.

Thus, if R_0 is the original rate of production for a production unit for the current system, and R_1 is the new rate of production for the production unit associated with the new system, then:

$$P = [(R_1 - R_0)/R_0] \times 100$$

As an example, if P = 100 percent then the new system will produce twice as much coal per production unit as the current system, and as a result, only half as many production units are needed to maintain the annual production rate for the mine.

4.1.2 M--Percentage Change in Direct Labor

The percentage change in direct labor per production unit is given by the parameter M. If M_0 is the number of men per production unit in the current system and M_1 is the number of men per production unit in the new system then:

$$M = [(M_1 - M_0)/M_0] \times 100$$

4.1.3 S--Percentage Change in Supply Costs

The percentage change in the supply costs associated with the new system is given by the parameter S. If S_0 is the supply cost per unit time for the current system and S_1 is the supply cost per unit time for the new system, then:

$$S = [(S_1 - S_0)/S_0] \times 100$$

4.1.4 E--Percentage Change in Environmental Protection Cost

The percentage change in the environmental protection cost per ton of coal mined is given by the parameter E. If E_0 is the environmental protection cost per ton of coal mined in the current system and E_1 is the environmental protection cost per ton of coal mined in the new system, then:

$$E = [(E_1 - E_0)/E_0] \times 100$$

4.1.5 I--Percentage Change in Capital Costs

The percentage change in capital costs is given by the parameter I. If I_0 is the annualized capital cost for the current system and I_1 is the annualized capital cost for the new system, then:

$$I = [(I_1 - I_0)/I_0 \times 100$$

4.1.6 MPL--Maximum Market Penetration Level

The maximum market penetration level is that percentage of all mines (underground or surface, but not both) that will eventually use the new system and is given by the parameter MPL. Thus, if MPL = 50 percent for a new underground mining system, then, eventually this system will be used in 50 percent of all underground mines.

4.1.7 MPT--Maximum Market Penetration Time

The time it takes for a new system to reach maximum market penetration from its initial availability date is given by the parameter MPT. It is assumed that market penetration is a ramp function as illustrated in Figure 4-1.

4.1.8 AT--System Availability Time

AT is the parameter that specifies the year the new system will be available.

4.2 New System Benefits

There are four basic new system benefits and these are measured over the 40-year period from 1980 to 2020. These benefits are:

a) Total Cost Savings: Total cost savings are the discounted cost savings that are achieved through the use of the new system in those coal mines which have the potential for use of the new system.

b) Capital Cost Savings: The capital cost savings are the discounted capital cost savings over the 40-year period as a result of using the new system. These savings represent the fact that the mining equipment used by the new system may be either more expensive or less expensive than the equipment currently being used. In most cases, there are capital cost savings due to the use of more efficient equipment.

c) Man-days Saved Due to Accident Reduction: In this study, the number of man-days saved over the 40-year period due to accident reduction is a result of reduced exposure of the men to the mine environment per ton of coal mined as a result of productivity increases.

d) Reduction in Production Costs: The reduction of production costs per ton as a result of the new system is independent of the production scenarios.

Table 4-1 contains conversion factors that enable the benefits derived from one scenario to be converted to benefits associated with the other scenario.

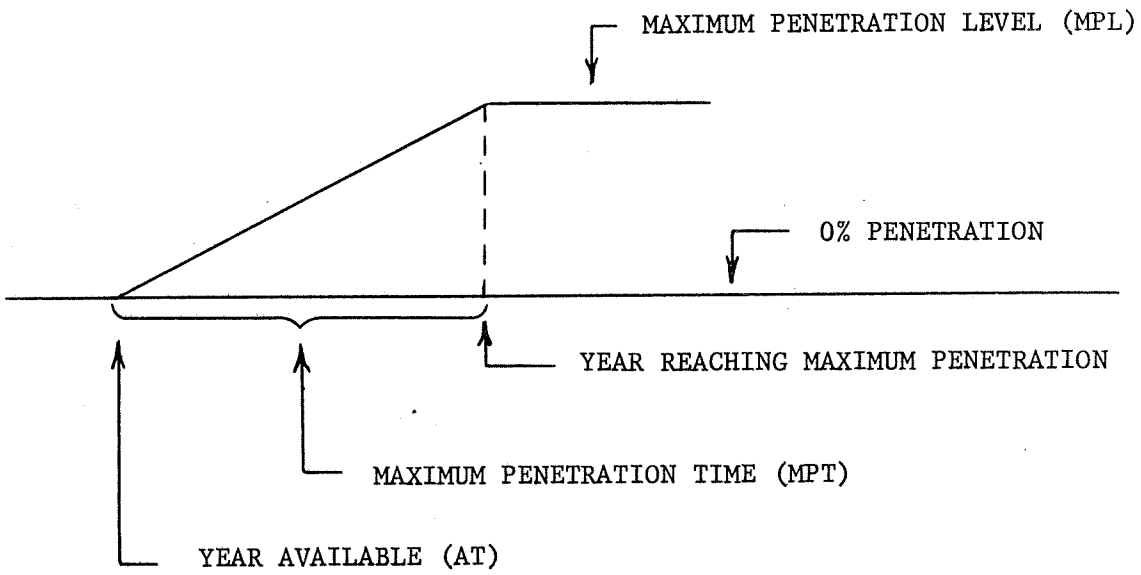

FIGURE 4-1

MARKET PENETRATION SCHEDULE

TABLE 4-1

RATIOS OF ANALYSIS RESULTS BETWEEN

SCENARIO 2 AND SCENARIO 1

$$R_i = \frac{V_i \text{ (Scenario 2)}}{V_i \text{ (Scenario 1)}}$$

R = ratio
V = value

Underground Mine Systems

i	R_i	V_i
1	1.297	Total Savings
2	1.288	Capital Savings
3	1.184	Man-Days Saved

Surface Mining Systems

i	R_i	V_i
1	1.674	Total Savings
2	1.632	Capital Savings
3	1.799	Man-Days Saved

4.3 Sensitivity Models

The sensitivity study models are based on the mine models described in Section 3.0. These models incorporate the system parameters which enable the costs to be calculated for various values of each parameter. Table 4-2 and Table 4-3 present the underground mine and surface mine sensitivity models.

All sensitivity calculations were made using the two production scenarios and the resulting dollar savings are the discounted savings. In these sensitivity models, it is assumed that any increase in productivity will reduce fixed costs per ton but will leave variable costs per ton of coal unchanged. Thus, the factor $\frac{1}{1+P}$ reduces the fixed cost per ton of coal to reflect the increase in productivity.

4.4 Sensitivity Analysis Results--Underground Mine Systems

The basic procedure in generating these sensitivty studies is first to calculate the benefits associated with the new system vs. changes in productivity with:

M = 0, S = 0, E = 0, I = 0, MPL = 100%, MPT = 0, AT = 1981

The benefits from the basic system configuration defined above are expressed in the form of curves. Then, each of the parameters is varied to determine its impact on the benefits. There is a curve or a set of curves for each parameter, which will enable the reader to combine various parameter value combinations representing the specific new system and to determine the associated benefits.

4.4.1 The Effect of an Increase in Productivity (P)--Underground Mines

Figure 4-2 shows how total cost savings and capital savings over

TABLE 4-2

THE UNDERGROUND MINE MODEL

ONE MILLION TON/YEAR MINE

COST COMPONENT	COST FORMULA ($/Ton)
Direct Labor	$2.58\left(\frac{1+M}{1+P}\right)$
Maintenance Labor	$0.38\left(\frac{1}{1+P}\right)$
Supplies	$1.84\ (1+S)$
Power	0.30
Payroll Overhead	1.20
Union Welfare	1.05
Environmental Protection	$0.4\ (1+E)$
Indirect Cost	$0.73\left(\frac{1}{1+P}\right)$
Taxes and Insurance	$0.43\left(\frac{1}{1+P}\right)$
Capital Costs	$1.30\left(\frac{1+I}{1+P}\right)$

TABLE 4-3

THE SURFACE MINE MODEL

COST COMPONENT	COST FORMULA ($/Ton)
Direct Labor	$0.275\left(\frac{1+M}{1+P}\right)$
Maintenance Labor	$0.145\left(\frac{1}{1+P}\right)$
Reclamation Road Labor	$0.047\left(\frac{1}{1+P}\right)$
Utility Labor	$0.024\ \frac{1}{1+P}\Big)$
Operating Supplies	$0.742\left(\frac{1+S}{1+P}\right)$
Power	0.464
Reclamation (Contract)	0.039
Payroll Overhead	$0.173\left(\frac{1}{1+P}\right)$
Union Welfare	1.550
Royalty, Licenses, & Bonds	0.470
Indirect Costs	$0.185\left(\frac{1}{1+P}\right)$
Taxes & Insurance	$0.221\left(\frac{1}{1+P}\right)$
Capital Cost	$0.805\left(\frac{1+I}{1+P}\right)$

the 40-year period from 1980 to 2020 vary as a function of P. These curves represent a new system that becomes available in 1981 with MPL = 100% and MPT = 0.

Figure 4-3 shows how total man-days saved as a result of accident reduction due to the new system vary as a function of P. Both Scenario 1 and 2 are represented in Figures 4-2 and 4-3.

As an example, if the new system for an underground mine had the following parameters:

P = 200%, M = 0, S = 0, E = 0, I = 0, MPL = 100%, MPT = 0, AT = 1981,

for Scenario 2

then

Total cost savings = $\$6.96 \times 10^9$

Capital cost savings = $\$2.24 \times 10^9$

Man-days saved due to accident reduction (non-fatal) = 12.59×10^5

Man-days saved due to accident reduction (fatal) = 22.22×10^5

Figure 4-4 represents the reduction in production costs per ton of coal mined as a function of P. This curve is independent of the production scenarios.

4.4.2 The Effect of Changes in Direct Labor (M)--Underground Mines

Figure 4-5 presents the change in total cost savings vs. M when P = 0, the system is immediately available in 1981, and with full penetration the first year after it becomes available. To determine the total cost savings for values of P other than zero, the P = 0 value is mulitplied by the fraction associated with the specified P from Figure 4-6.

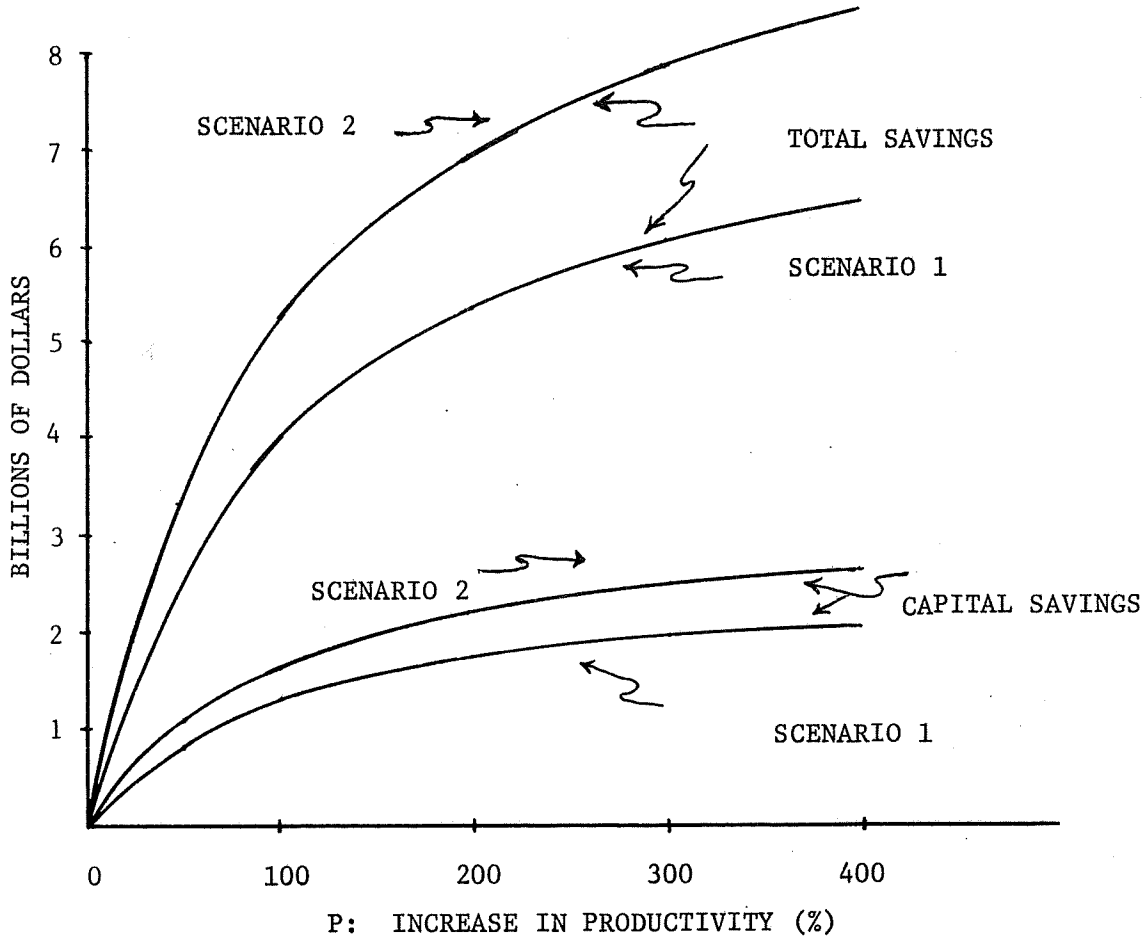

FIGURE 4-2

UNDERGROUND MINING SYSTEMS

TOTAL COST SAVINGS AND CAPITAL COST SAVINGS VS. P
(M=0, S=0, E=0, I=0, MPL=100%, MPT=1YR., AT=1981)

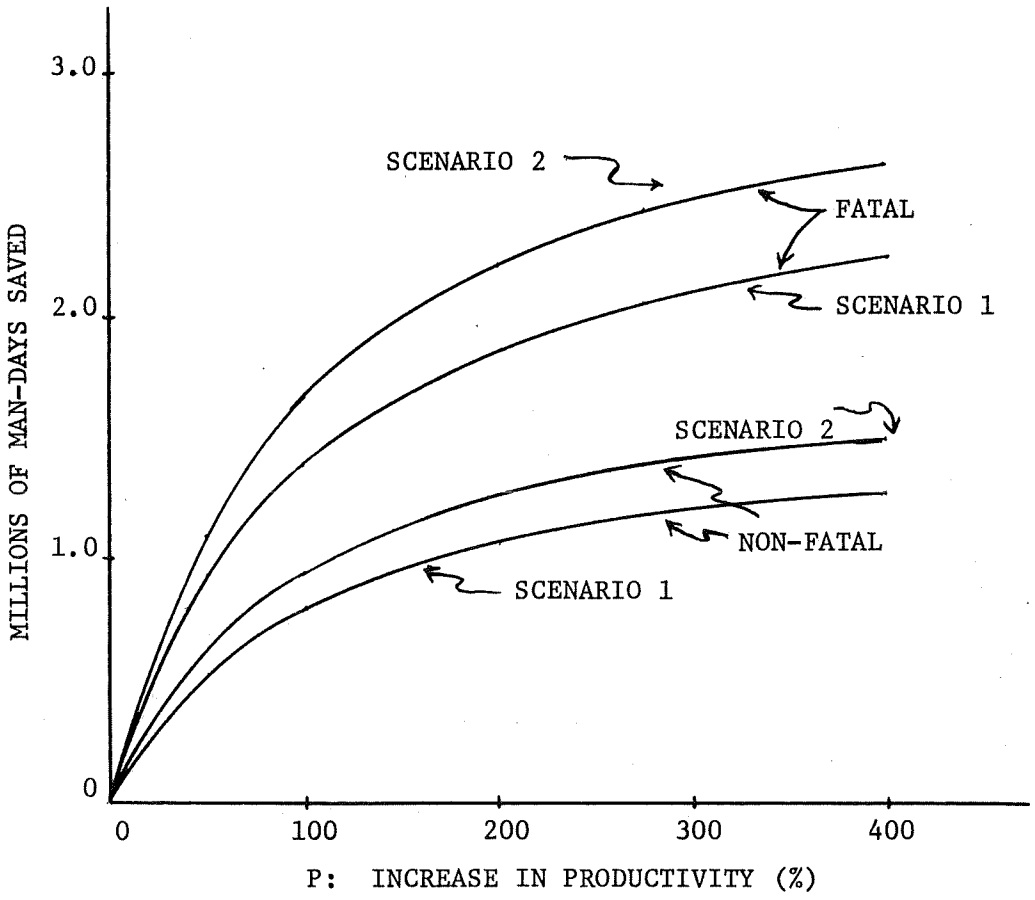

FIGURE 4-3

UNDERGROUND MINING SYSTEMS

CHANGE IN MAN-DAYS SAVED DUE TO ACCIDENT REDUCTION VS. P
(M=0, S=0, E=0, I=0, MPL=100%, MPT=1YR, AT=1981)

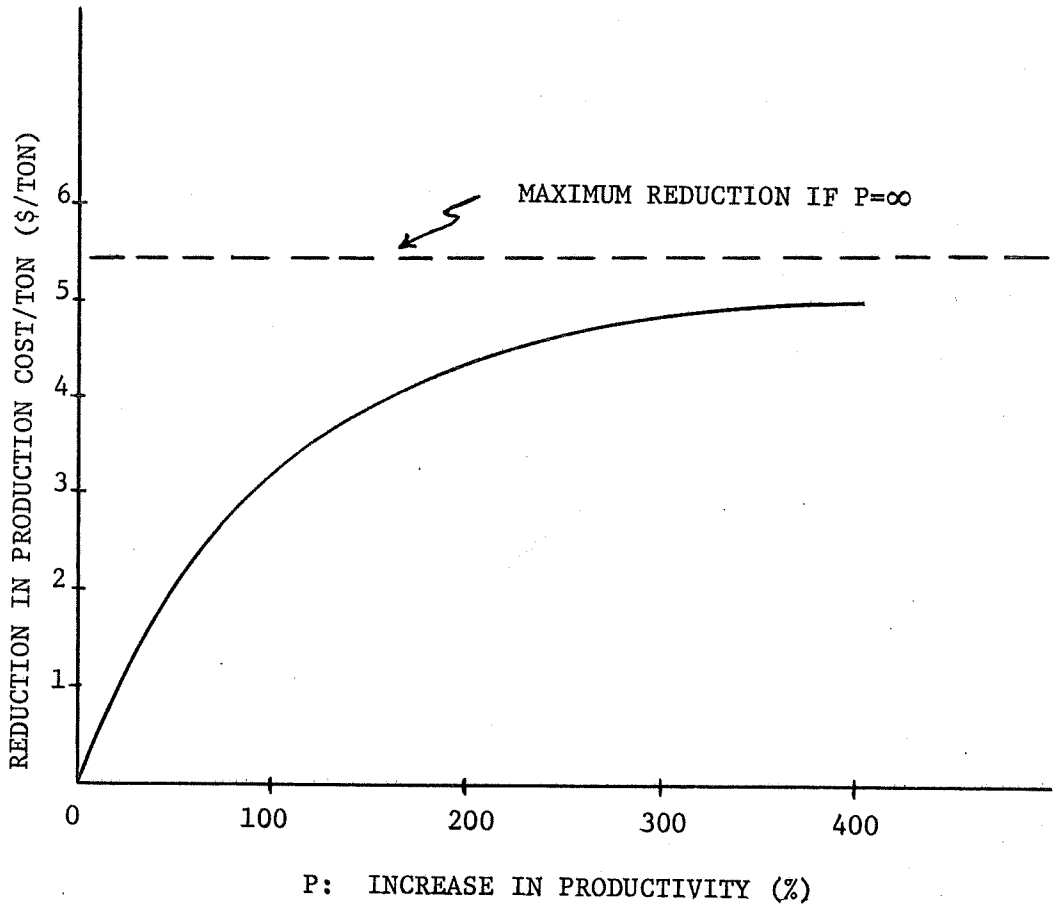

FIGURE 4-4

UNDERGROUND MINING SYSTEMS

REDUCTION IN PRODUCTION COST/TON VS. INCREASES IN PRODUCTIVITY
(M = 0, S = 0, E = 0, I = 0)

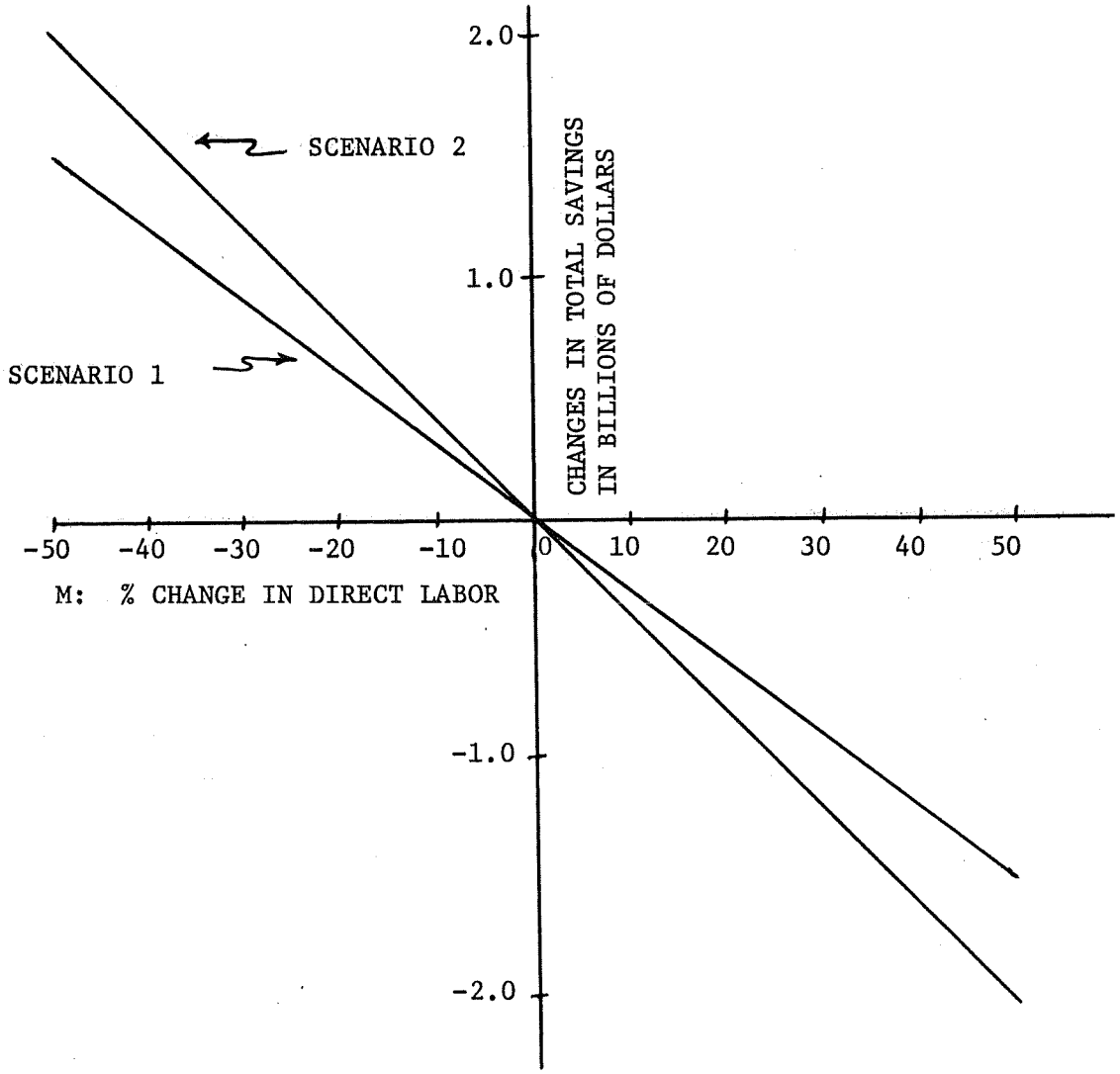

FIGURE 4-5

UNDERGROUND MINING SYSTEMS

CHANGE IN TOTAL COST SAVINGS VS. M FOR SCENARIO 2
(P=0, S=0, E=0, I=0, MPL=100%, MPT=1YR, AT=1981)

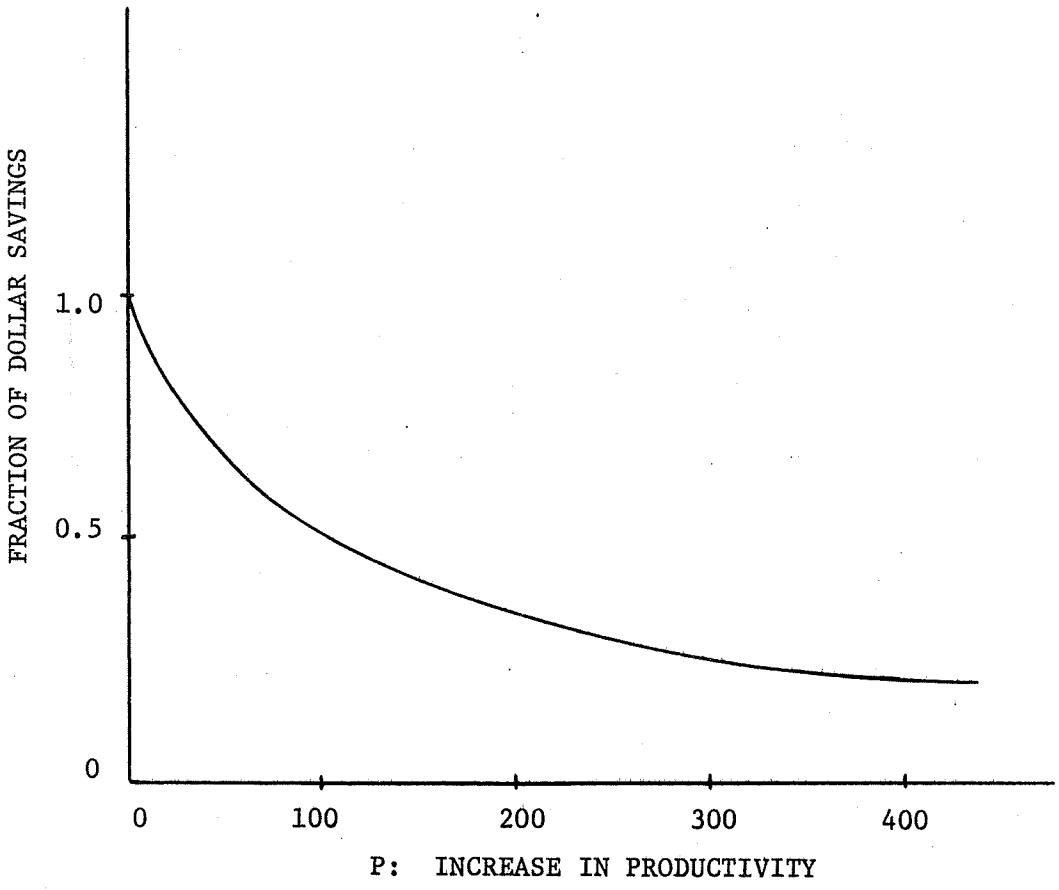

FIGURE 4-6

UNDERGROUND MINING SYSTEMS

FRACTION OF DOLLAR SAVINGS DUE TO I FOR P = 0 VS. P

As an example, if P = 75% and M = -30%, then the first step is to extract the savings value from Figure 4-3, which is $1.2 x 10^9 (for P = 0). Since P = 75%, the next step is to multiply $1.2 x 10^9 by the value of 0.57 from Figure 4-6, which gives $0.684 x 10^9 for P = 75% and M = -30%. These are the total cost savings due solely to manpower reduction over the 40-year period of 1980 to 2020.

Changes in direct labor do not affect directly the capital savings or the man-days saved as a result of accident reduction. These effects are achieved through changes in productivity.

4.4.3 The Effect of Changes in Supply Costs (S)--Underground Mines

Figure 4-7 presents the change in total savings as a function of a change in supply costs. The curve in Figure 4-6 represents the case where M = 0, S = 0, I = 0, MPL = 100%, and MPT = 0, and AT = 1981, where the curve is independent of P.

As an example, if S = -30% then the total savings will be $0.4 x 10^9 or $400,000,000.

4.4.4 The Effect of Changes in Environmental Protection Costs (E)--Underground Mines

Figure 4-8 plots the total savings as a function of E with M = 0, S = 0, I = 0, MPL = 100%, MPT= 0, and AT = 1981, where the curve is independent of P. These costs are yearly operational costs for environmental protection.

As an example, if E = -15%, then the total cost savings will be $70,000,000 over the period of 1981 to 2020.

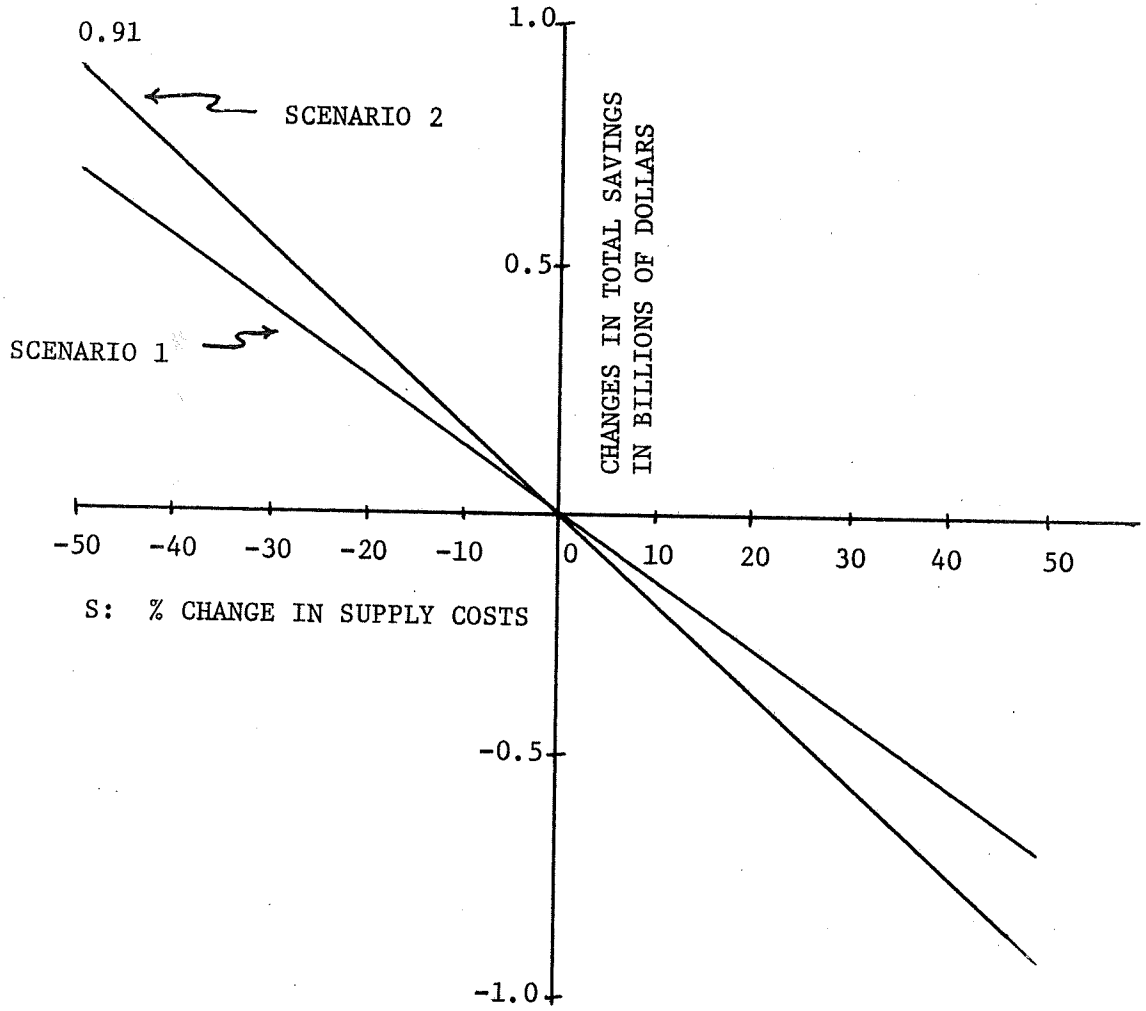

FIGURE 4-7

UNDERGROUND MINING SYSTEMS

CHANGE IN TOTAL COST SAVINGS VS. S
(M=0, E=0, I=0, MPL=100%, MPT=1YR, AT=1981)

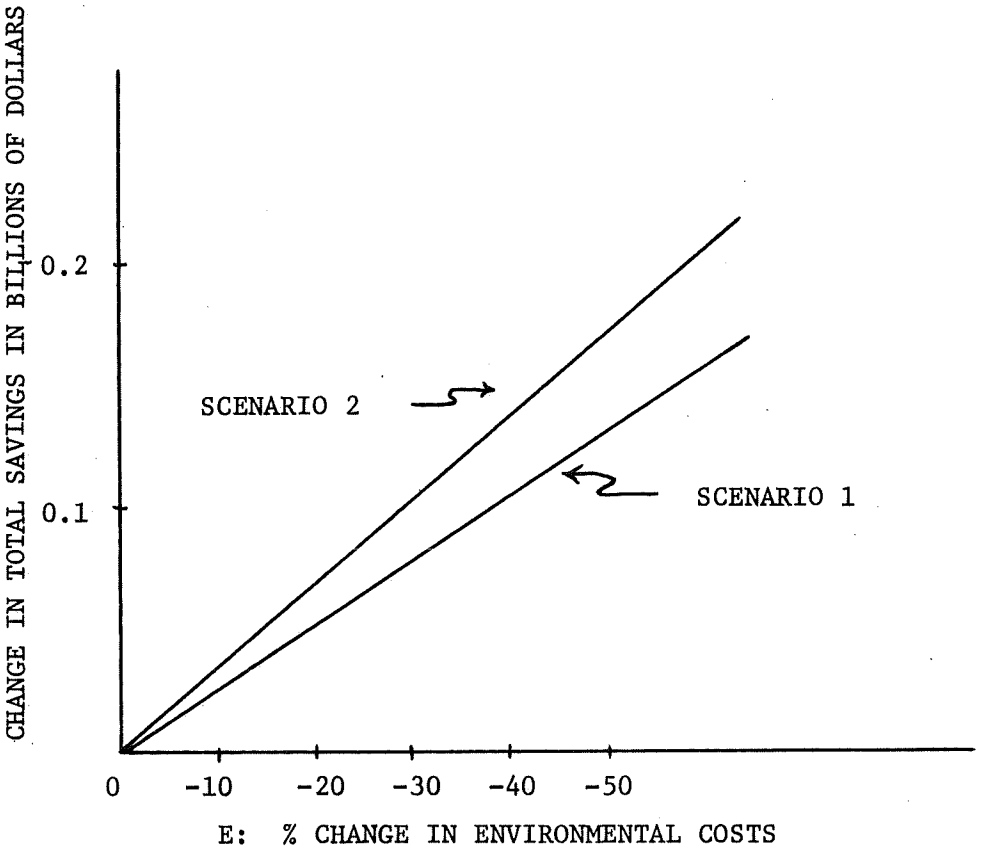

FIGURE 4-8

UNDERGROUND MINING SYSTEMS

CHANGES IN TOTAL COST SAVINGS VS. E
(M=0, S=0, I=0, MPL=100%, MPT=1YR., AT=1981)

4.4.5 The Effect of Changes in Capital Costs (I)--Underground Mines

Figure 4-9 plots the changes in capital cost savings for P = 0 for various values of I, with M = 0, S = 0, E = 0, MPL = 100%, MPT = 0 and AT = 1981. To adjust the savings for other than P = 0, the savings are multiplied by the appropriate fraction taken from Figure 4-10 for the specific value of P.

4.4.6 The Effect of Market Penetration Level (MPL)--Underground Mines

The market penetration level (MPL) represents the maximum percentage of 1-MTY mines that will use the new system. Total cost and capital savings and man-days saved are multiplied by MPL x .01 to arrive at the appropriate level of benefits.

4.4.7 The Effect of the Number of Years to Full Market Penetration (MPT)--Underground Mines

Figure 4-11 plots the fractional multiplier that reduces the total and capital savings as a function of the length of time from system availability to maximum market penetration (MPT). It is assumed that market penetration can be represented as a ramp function as depicted in Figure 4-11. Figure 4-12 plots the fractional multiplier that reduces the man-days saved as a result of accident reduction as a function of MPT. In both cases the curve is independent of P.

4.4.8 The Effect of System Availability (AT)--Underground Mines

Figure 4-13 plots the fractional mulitplier of the dollar savings to reflect the fact that the system availability (AT) has a

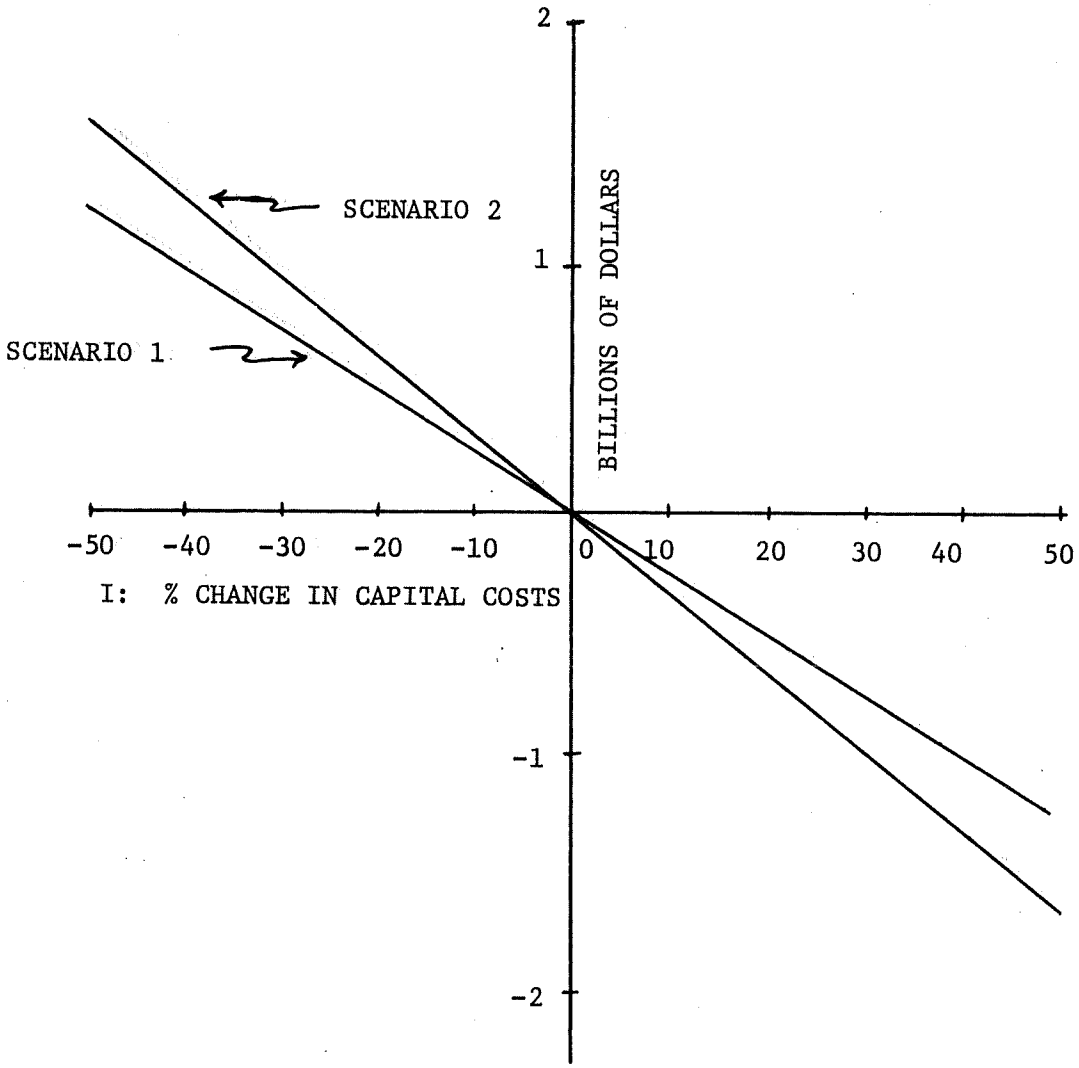

FIGURE 4-9

UNDERGROUND MINING SYSTEMS

CAPITAL COST SAVINGS VS. I
(P=0, M=0, S=0, E=0, MPL=100%, MPT=1YR, AT=1981)

SCENARIO 2

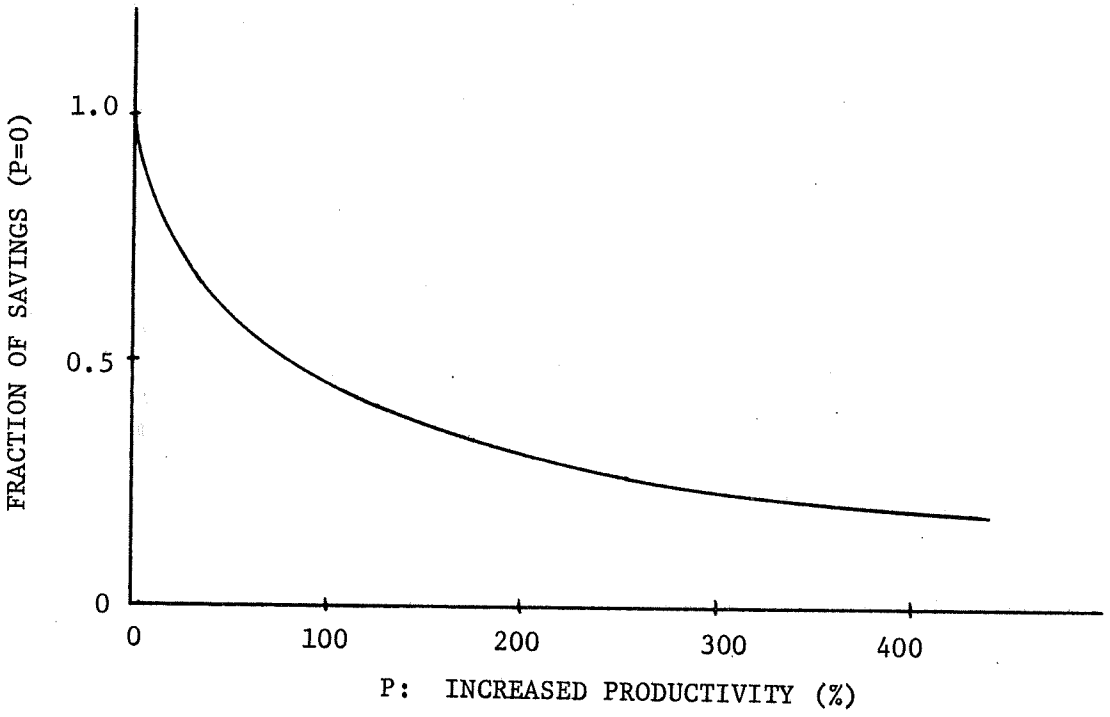

FIGURE 4-10

UNDERGROUND MINING SYSTEMS
FRACTION OF CAPITAL COST SAVINGS VS. P

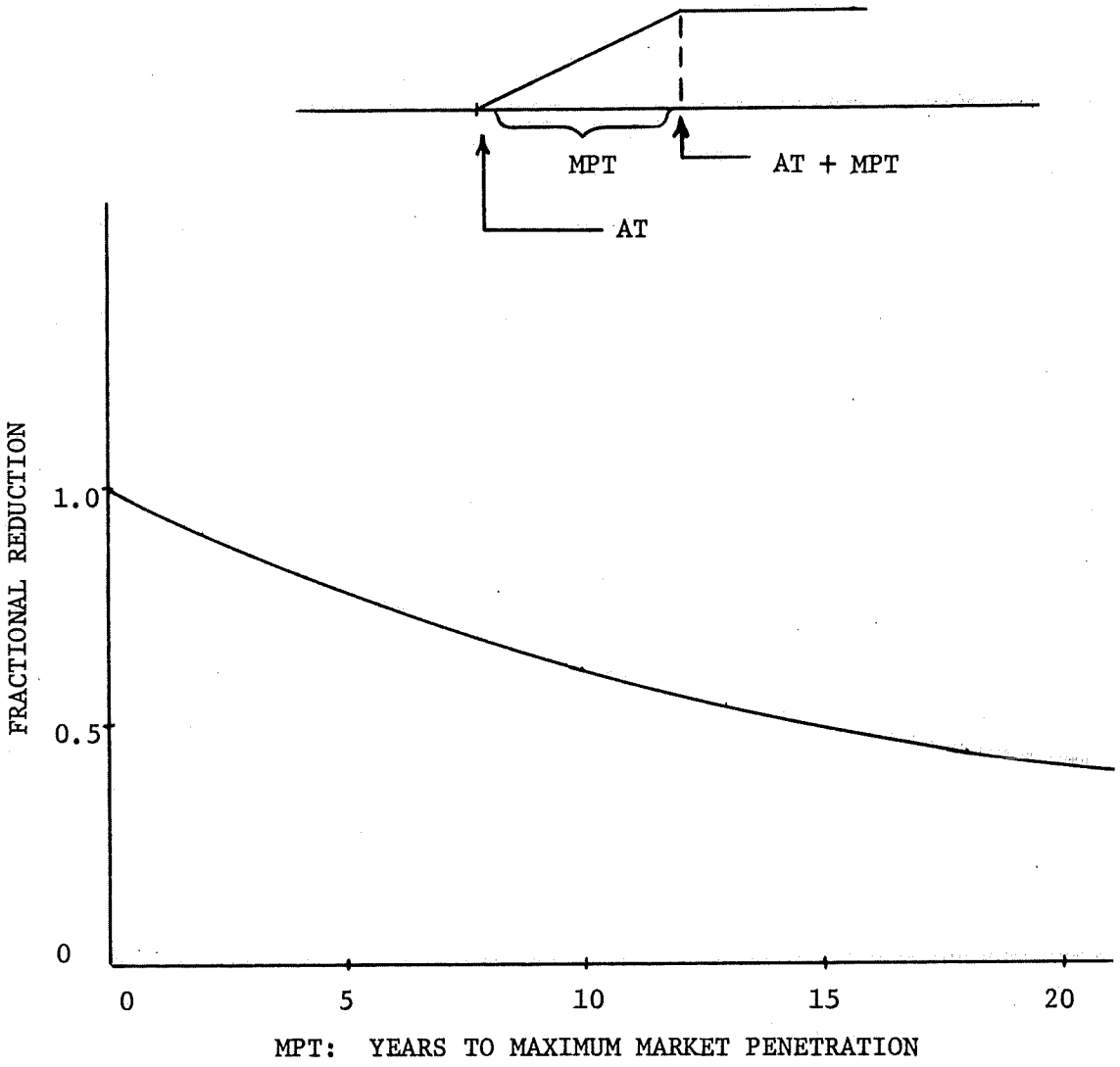

FIGURE 4-11

UNDERGROUND MINING SYSTEMS
FRACTION OF SAVINGS VS. MPT

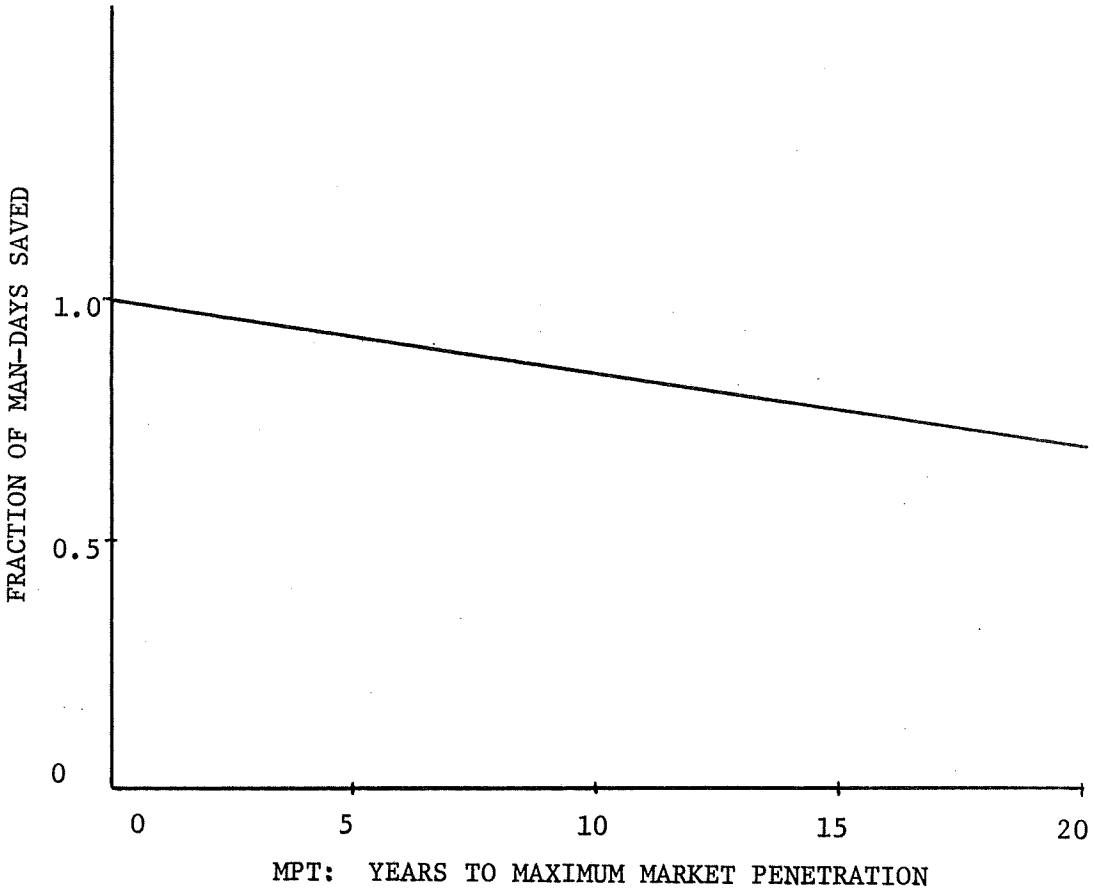

FIGURE 4-12

UNDERGROUND MINING SYSTEMS

FRACTION OF MAN-DAYS SAVED VS. MPT

FIGURE 4-13

UNDERGROUND MINING SYSTEMS

FRACTION OF DOLLAR SAVINGS VS. AT

particular date. Thus, if a new system is to be available in 1985, then the benefits calculated on the basis of AT = 1981 must be multiplied by 0.625.

Figure 4-14 plots the fractional multiplier of the man-days saved as a function of AT.

4.5 Sensitivity Analysis Results--Surface Mining Systems

The same procedure was used in generating the sensitivity curves for new surface mining systems as were used for new underground mining systems.

4.5.1 The Effect of Productivity Increases (P)--Surface Mines

Figure 4-15 plots the total cost savings and the capital cost savings as a function of P for both Scenarios 1 and 2. The other parameters are M = 0, S = 0, I = 0, MPL = 100%, MPT = 0, and AT = 1981.

Figure 4-16 plots man-days saved due to accident reduction as a function of P for both Scenarios 1 and 2, with the other parameters the same as in Figure 4-15.

Figure 4-17 plots the reduction in production costs as a function of P. This curve is independent of the scenarios.

4.5.2 The Effect of Changes in Direct Labor (M)--Surface Mines

Figure 4-18 presents the change in dollar savings vs. M when P = 0, S = 0, I = 0, MPL = 100%, MPT = 0, and AT = 1981. To determine the total cost savings for a specified value of P other than zero, P = 0 value of the savings is mulitplied by the fraction associated with the P of interest in Figure 4-6. The curve in Figure 4-6 is the same for both surface and underground mines.

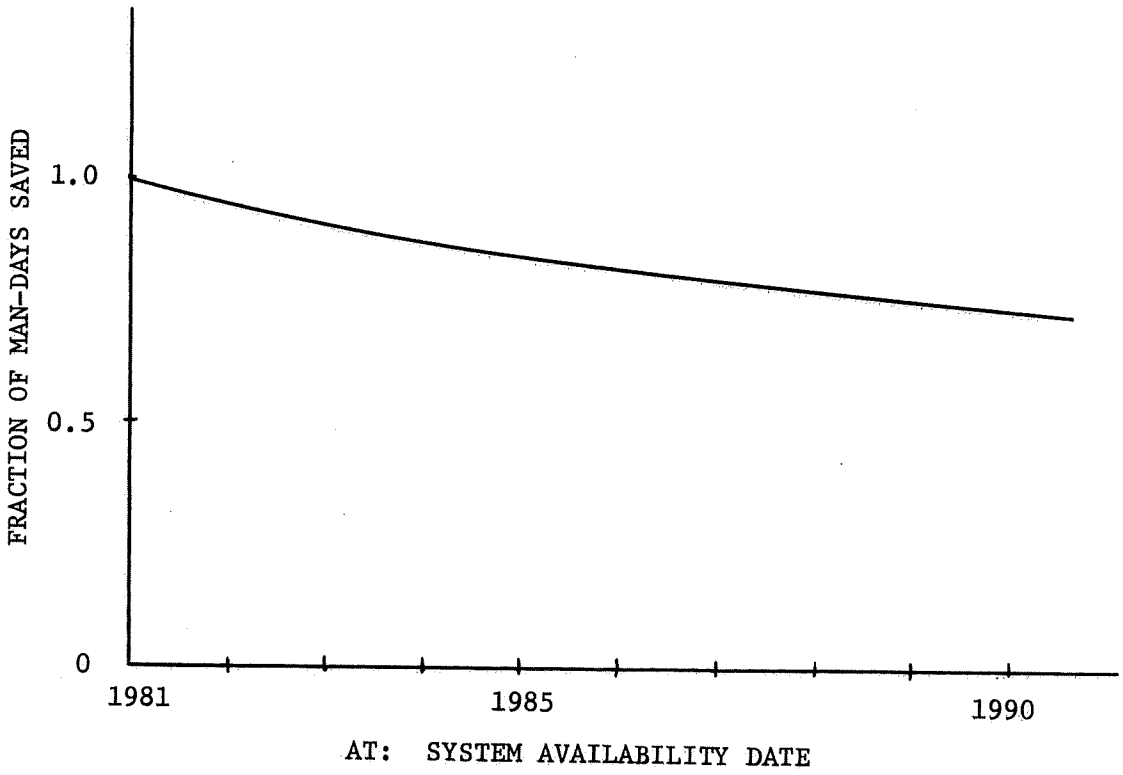

FIGURE 4-14

UNDERGROUND MINING SYSTEMS

FRACTION OF MAN-DAYS SAVED VS. AT

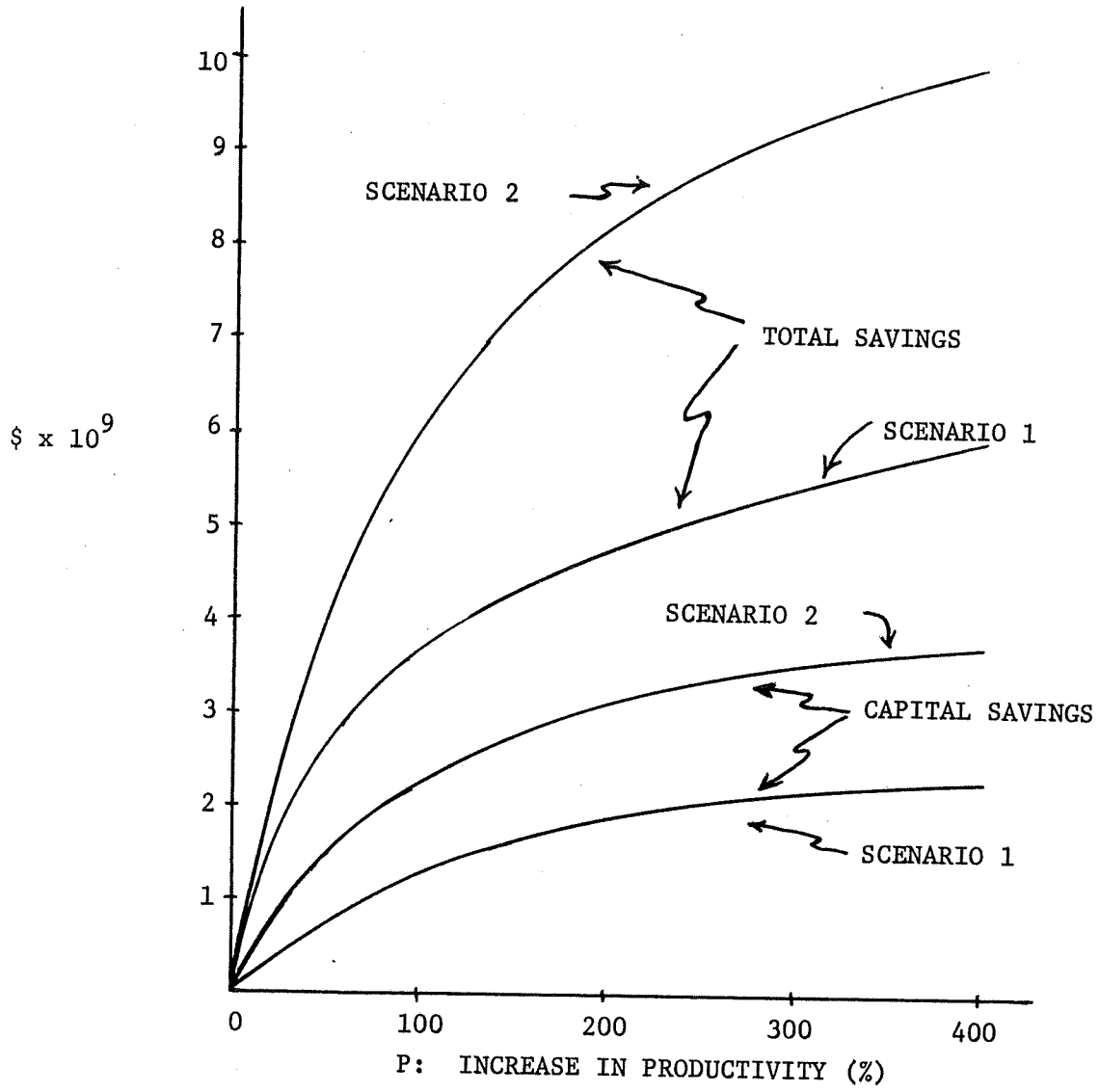

FIGURE 4-15

SURFACE MINING SYSTEMS

TOTAL COST SAVINGS AND CAPITAL COST SAVINGS VS. P
(M=0, S=0, I=0, MPL=100%, MPT=1YR, AT=1981)

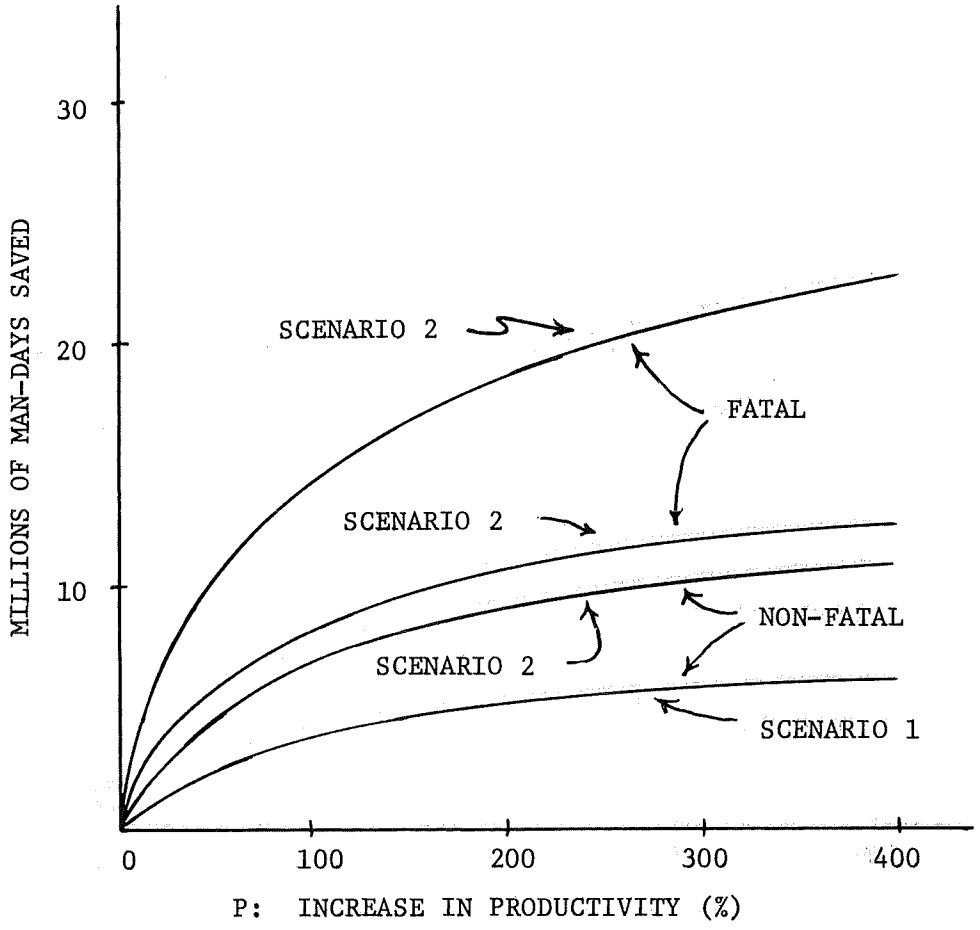

FIGURE 4-16

SURFACE MINING SYSTEMS

CHANGE IN MAN-DAYS SAVED DUE TO ACCIDENT REDUCTION VS. P
(M=0, S=0, I=0, MPL=100%, MPT=1YR, AT=1981)

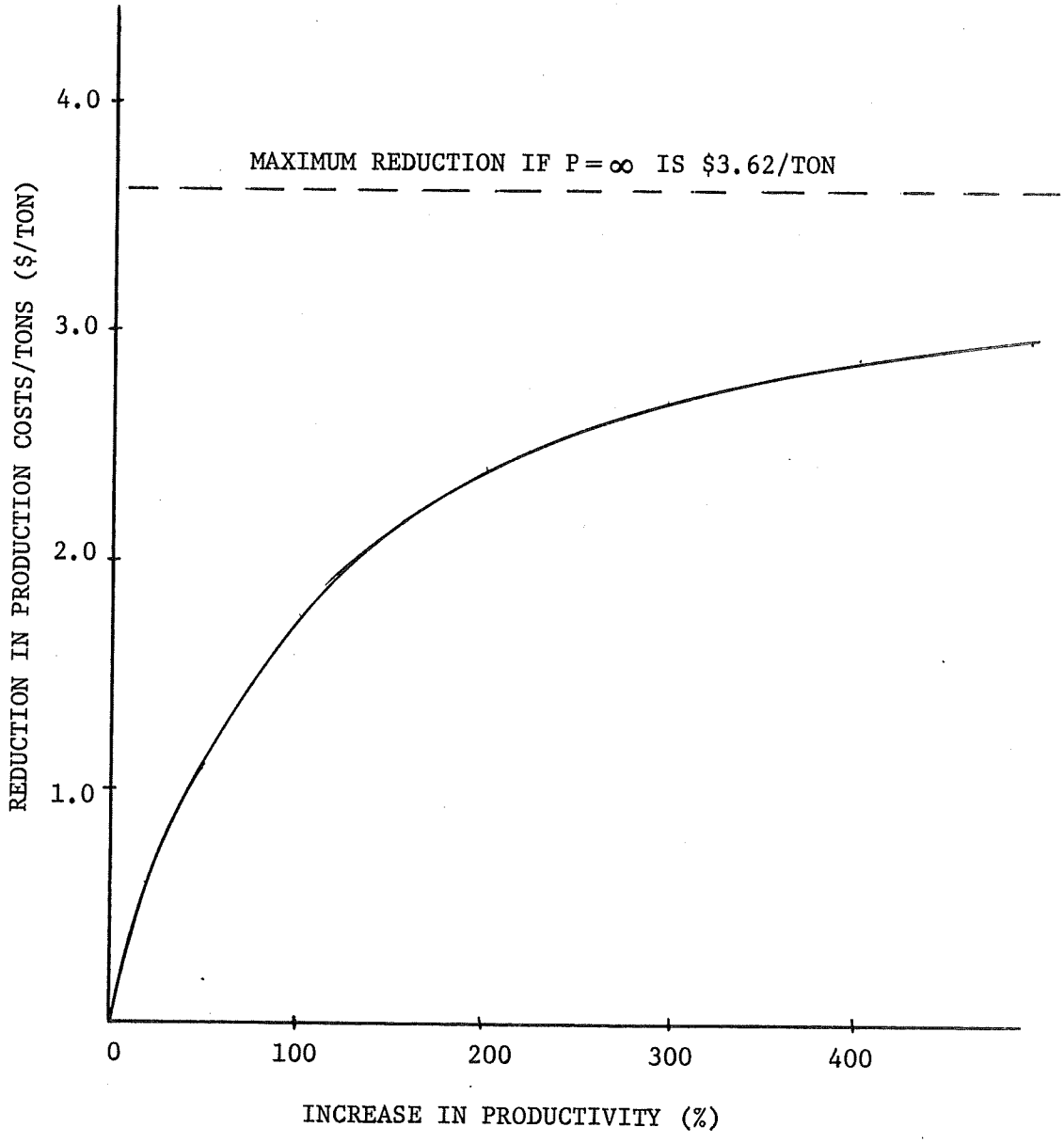

FIGURE 4-17

SURFACE MINING SYSTEMS

REDUCTION IN PRODUCTION COST/TON VS. INCREASES IN PRODUCTIVITY
(M = 0, S = 0, E = 0, I = 0)

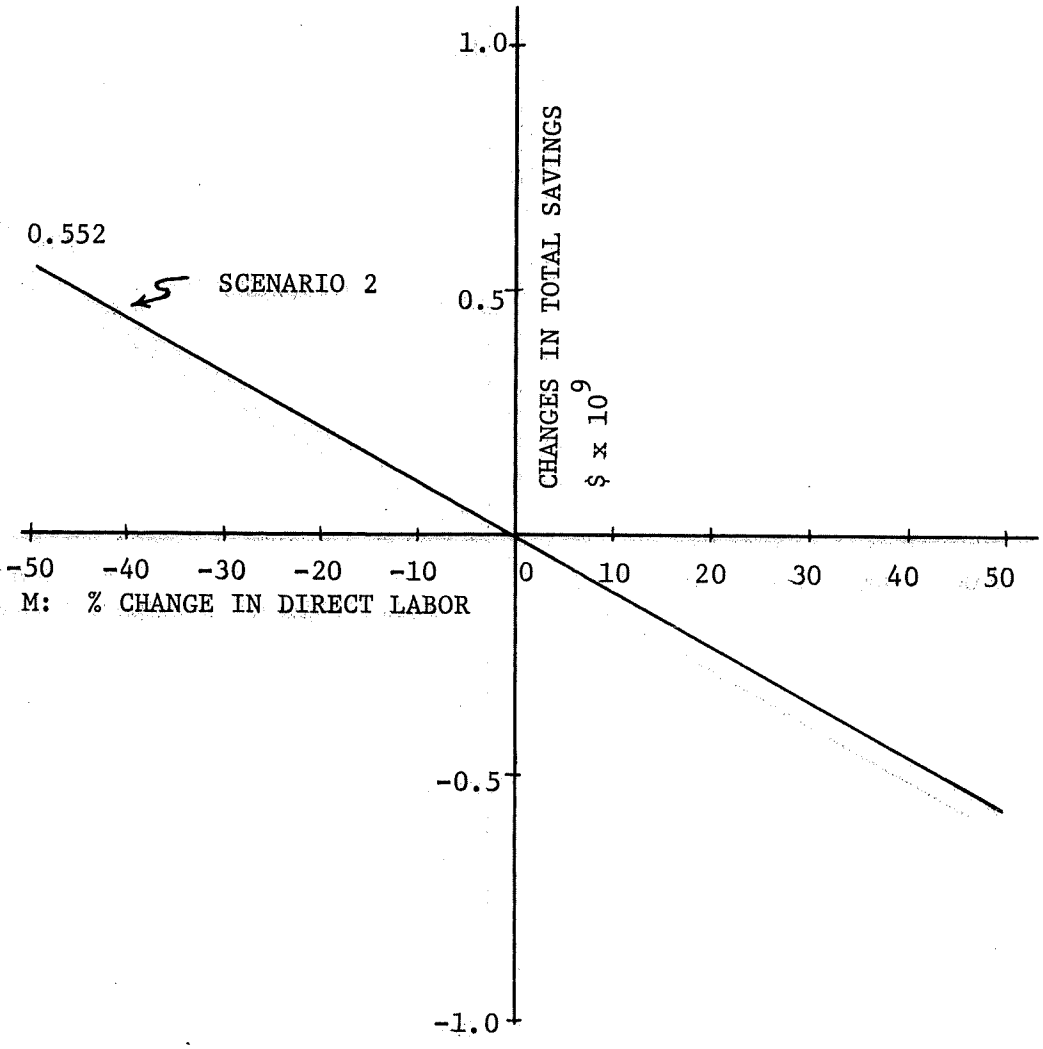

FIGURE 4-18

SURFACE MINING SYSTEMS

CHANGE IN TOTAL COST SAVINGS VS. M FOR SCENARIO 2
(P=0, S=0, I=0, MPL=100%, MPT=1YR, AT=1981)

4.5.3 The Effect of Changes in Supply Costs (S)--Surface Mines

Figure 4-19 presents the change in total savings vs. S for P = 0, M = 0, I = 0, MPL = 100 percent, MPT = 0, and AT = 1981.

To determine the total savings for values of P other than zero, the P = 0 value is multiplied by the fraction associated with the P of interest from Figure 4-6.

4.5.4 The Effect of Changes in Capital Costs (I)--Surface Mines

Figure 4-20 plots the changes in capital cost savings for P = 0 for various values of I, with M = 0, S = 0, MPL = 100 percent, MPT = 0, and AT = 1981. To adjust savings for other than P = 0, the savings is multiplied by the appropriate fraction taken from Figure 4-10 for the specific value of P.

4.5.5 The Effect of Changes in Number of Years to Maximum Level of Market Penetration (MPT)--Surface Mines

Figure 4-21 plots the fraction of dollar savings as a function of the number of years to maximum penetration (MPT). As an example, if the new system has a saving of $2 billion for an MPT = 0, then the actual value of the savings at MPT = 10 would be $2 billion multiplied by the fraction 0.67, which is the value of the curve at MPT = 10 years.

4.5.6 The Effect of Changes in the Delay of System Availability (AT)--Surface Mines

Figure 4-22 plots the fraction of dollar savings vs. system availability (AT with 1981 as the base year. Thus, if the total cost savings for a new system were $2 billion for AT = 1981, then for an AT = 1985, the total cost savings would be $2 billion x 0.68 = $1.36 billion, where 0.68 is the value of the curve at AT = 1985.

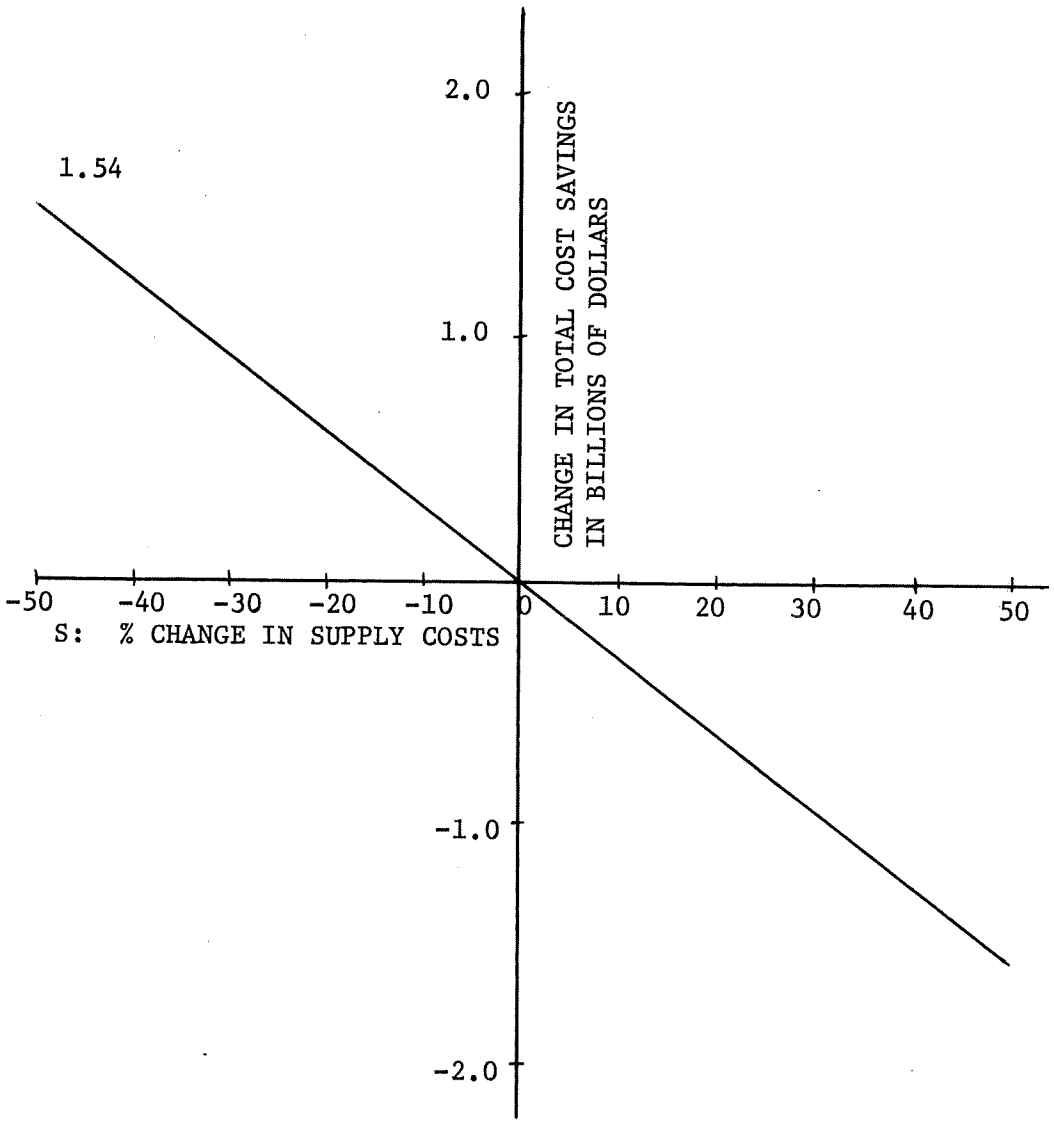

FIGURE 4-19

SURFACE MINING SYSTEMS

CHANGE IN TOTAL COST SAVINGS VS. S
(P=0, M=0, I=0, MPL=100%, MPT=1YR., AT=1981)

SCENARIO 2

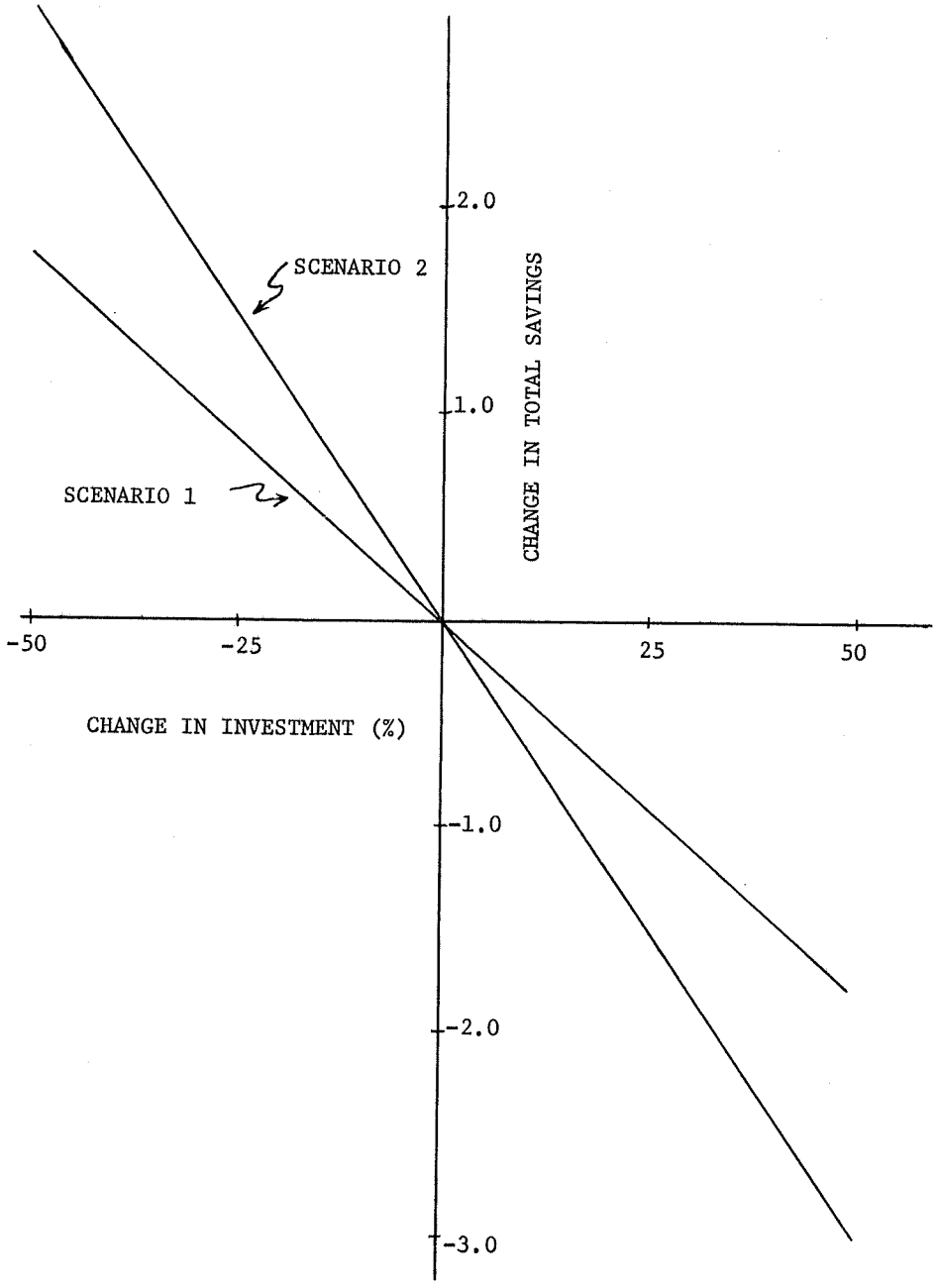

FIGURE 4-20

SURFACE MINING SYSTEMS

CHANGE IN TOTAL SAVINGS (10^9 DOLLARS)
VS.
CHANGE IN INVESTMENT (%)

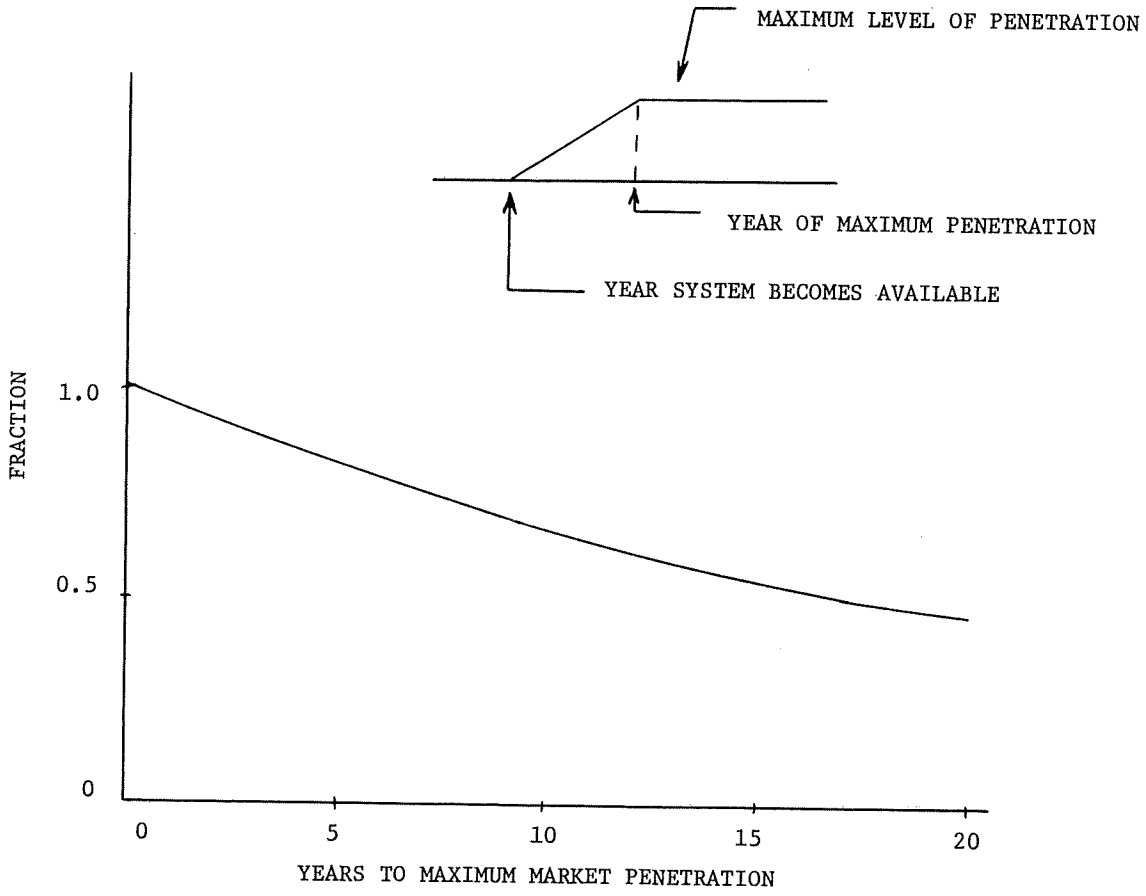

FIGURE 4-21

SURFACE MINING SYSTEMS

FRACTION OF TOTAL SAVINGS
VS.
NUMBER OF YEARS TO MAXIMUM MARKET PENETRATION

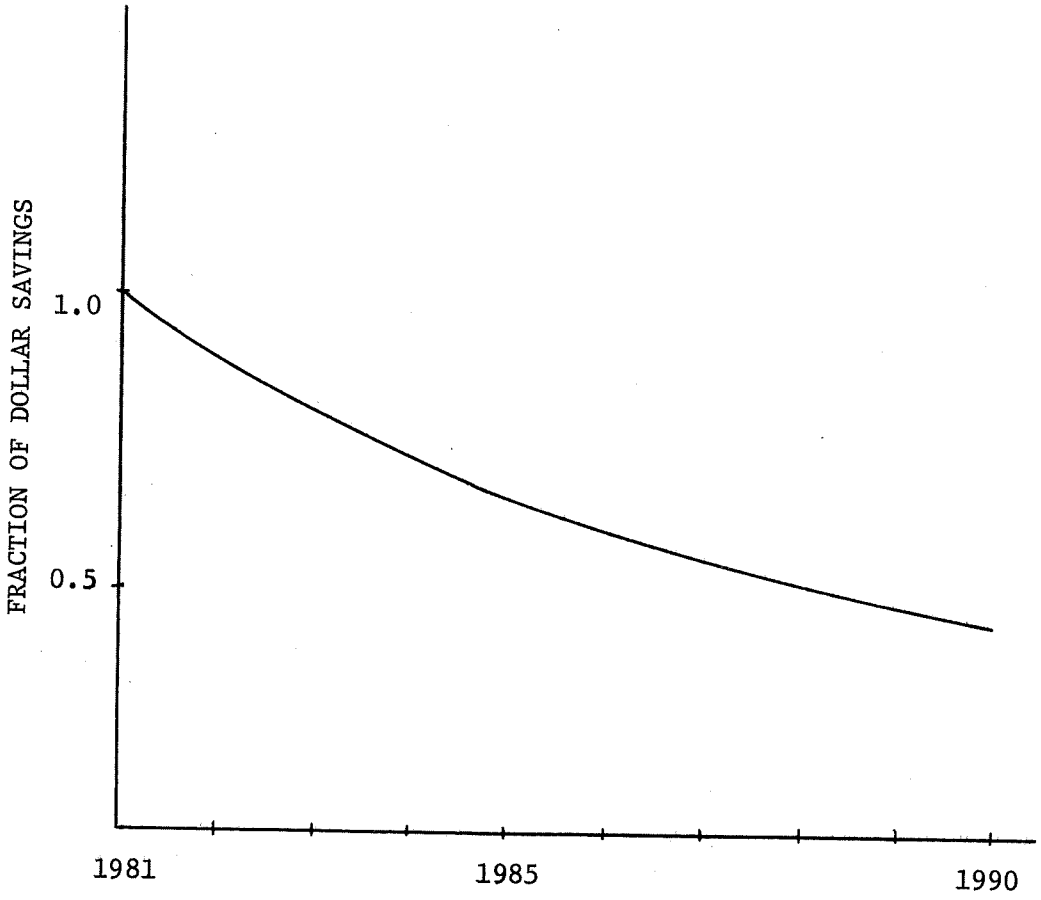

FIGURE 4-22

SURFACE MINING SYSTEMS

FRACTION OF DOLLAR SAVINGS VS. AT

5.0 HIGH-SPEED MINE DEVELOPMENT (HSMD)

This and the following sections (Section 5.0 through 9.0) present the detailed analysis of the five hypothetical new systems. The format of these sections generally follow the outline given below:

- .0 Introduction
- .1 Production Scenarios
- .2 State-of-the-Technology
- .3 Related Subprograms
- .4 Methodology
- .5 Current System Description
- .6 Hypothetical New System Description
- .7 Analysis

This section (Section 5.0) describes the High-Speed Mine Development System.

The development of an underground coal mine is a process that must start years before the mine becomes fully productive and must continue throughout the life of the mine. The major problem limiting the construction of new underground mines and expanding established mines are the high initial investment and the long development times needed before production can start. As an example, a modern mining operation capable of producing 2 million tons of coal per year for a mine life of 20 to 30 years requires a $30 million investment and takes 5 to 6 years to reach full production. Furthermore, expansion of an established mine can take 3 to 5 years.

Mine development requires large numbers of entries to be driven and sunk to meet ventilation requirements and provide access for men and materials. Conventional methods of shaft and entry development are slow, inefficient, dangerous and, as a consequence, expensive. For example, in sinking large diameter shafts (15 to 20 feet in diameter), advance rates of 60 feet per month at a cost of $1,000 per foot are not unusual. Although essential to production, this phase of development contributes relatively little coal to the mine operator's production.

Compounding the problem of entry and shaft development are several factors. The 1969 Coal Mine Health and Safety Act set stringent ventilation requirements dictating that for most mine operations an increase in the number or size of airshafts and entries. Production rate potential has increased, but to realize this potential also requires more or larger shafts and entries. Much of the remaining coal reserves are at greater depth than mined today. As coal resources are mined at these greater depths, development costs and time will increase proportionately. Most importantly, there is a critical shortage of skilled crews needed for shaft sinking and entry development.

In order to meet the projected demand for coal, numerous new mines must be developed and the capacity of existing mines must be expanded. To accomplish these goals, new technology has to be introduced to accelerate underground coal mine development and reduce development costs.

5.1 Production Scenarios for HSMD

The two standard production scenarios, as defined in Section 3.0, are used as the basis for evaluation of the HSMD system. For this particular application the new mines specified in these two scenarios are grouped as follows:

Province: New mines associated with the Rocky Mountain province and those associated with the Eastern province are combined, and those of the Interior province and the Great Plains province are combined. This reflects the assumption that the mines in the Rocky Mountain and the Eastern provinces are similar in those characteristics of interest, and the mines in the Interior and Great Plains provinces are also similar.

Capacity: All underground mines in the HSMD production scenarios are grouped by capacity in terms of:

a) less than one million tons/year
b) greater than or equal to one million tons/year

The rationale for this grouping of new mines is given in Section 5.4.

5.2 State-of-the-Technology

The feasibility of developing the equipment and procedures necessary for a high-speed mine development system rests on several factors, one of which is the state-of-the-technology in the areas of:

a) Horizontal Excavation
b) Vertical Excavation
c) Degasification of coal seams

These areas are discussed in the following subsections.

5.2.1 Horizontal Excavation: Entry Development

Present entry development uses a drill-and-blast cycle or continuous miners. In the drill-and-blast operation, a cutting machine undercuts the coal at floor level after which holes are drilled in the coal face, then loaded, and shot. Because the operation is cyclic and requires explosives, it is inherently inefficient and dangerous.

The use of continuous miners is not a continuous operation as the name would imply. The machine can only advance 20 to 25 feet in most cases before being trammed out of the new excavation to permit installation of permanent roof supports. The continuous miner is generally not capable of cutting roof or floor rock. Consequently, entry depth is determined by the coal seam thickness, which in many cases, is restrictive in terms of ventilation and materials-handling requirements. In addition, only flat roofs are obtainable, whereas arched or circular entries are more desirable from the standpoint of ground support and air flow characteristics.

Tunneling-boring machines have found wide acceptance in civil works projects. These machines produce stable, aerodynamically smooth openings at maximum rates of more than 200 feet per day. Current tunneling machine excavation records exceed conventional drill-blast-muck records for complete footage per week by a factor of 4. The average rate of progress for these projects ranges from 17 to 90 feet per calendar day, which varies with the ground condition and construction schedule.

Most of the mechanical excavators that have been fabricated for the underground construction industry have been designed to cut circular openings. Such machines consist of a rotating head on which free-rolling cutters are mounted. These cutters contact the tunnel face in a series of regularly spaced concentric paths. Thrust is exerted through the head to the cutters by several hydraulic cylinders acting against gripper shoes parallel to the centerline of the tunnel. The gripper shoes are, in turn, forced against the tunnel walls by transverse hydraulic cylinders. Steel buckets around the circumference of the cutter head pick up the cuttings from the bottom of the bore as the head rotates. Cuttings are elevated to the top of the machine and dropped through a hopper onto a conveyor for transport to the rear of the boring machine. At this point, the cuttings are discharged into a material-handling system for removal from the tunnel.

Because of the speed of excavation, the ability to excavate a wide variety of materials, the inherent stability of the resultant circular opening, the labor-saving potential and a safe working environment, tunnel-boring machines should have application in the coal mining industry.

But there are some drawbacks in existing tunnel-boring machine designs. They do not cut level floors, therefore, special provision must be made prior to utilization by some mining equipment. Neither are present machines capable of cutting short radius curves nor capable of rapid mobilization.

In the room-and-pillar mining method used in the majority of domestic underground coal mines today, a 10-entry system is commonly used. The system was adopted because ventilation and haulage requirements could be met and it was compatible with the generating characteristics of the continuous miners and the drill-and-blast system. In addition to the ventilation and haulage problems already discussed multiple entries compound roof support problems because of stresses created by the existence of neighboring openings. An effort will be made under the subprogram to develop a computer program that will aid in obtaining an optimum entry development system given the mining method and the equipment available whether existing or proposed. It is anticipated that new equipment and the computer program will result in new entry concepts that will reduce the time and cost of development significantly.

5.2.2 Vertical Excavation--Shaft Development

The common vertical development is by shaft-sinking. It has traditionally relied upon a drill, blast, and muck system for rock excavation. The mucking procedure involves leading shaft cars with an overshot loader or an air-operated clambshell bucket and revolves around alternate excavation and concrete-placement cycles. This conventional system is labor intensive, expensive and time-consuming, especially in the development of large-diameter and deep shafts.

As outlined above, drill-and-blast techniques are presently employed to sink the large-diameter transportation and ventilation shafts because these techniques offer rapid mechanized excavation. Raise-boring is sometimes used, but it requires an existing chamber at the bottom of the raise to remove the cutting. Shaft development is thereby delayed for the period of time required to drive the entry and install a haulage system. In addition, shaft diameters attainable with the raise-boring system are limited and the muck generated from the raise-boring burdens the mine haulage system.

In FY 75, a contract was awarded to a major manufacturer of boring equipment to determine the technical and economic feasibility of developing a blind-shaft borer capable of sinking 20- to 30-foot-diameter shafts at speeds of 5 to 10 feet per hour. If feasible, the design of such a system and the conduct of controlled laboratory tests of critical subsystems such as the materials-handling system will be included under the contract.

Development of a blind-shalf borer will be a long-term project. To fill the needs of the coal mining industry, to some extent, on a more immediate basis; an effort will be made to modify conventional oil-drill rig systems to permit excavation of large-diameter shafts. The objective will be to obtain, with a minimum of modification to standard components, a drill rig capable of excavating blind shafts of 12 to 15 feet in diameter to a minimum depth of 100 feet at rates comparable to raise-boring systems for the same diameters.

A new concept in boring systems that has shown great potential is the conical borer developed under Bureau of Mines Contract Number H0232057. It uses a wedging, screw type action of the cutters to generate the thrust and reaction forces required to break rock. Consequently, only one-tenth of the normally required thrust load is needed. This permits the usage of smaller, lighter, more mobile support equipment. The thrust-saving feature and the adequacy of the pneumatic muck-handling systems used with the borer have been verified by controlled experiments conducted on a 40-inch-diameter prototype unit. Preliminary field tests to debug the system (including support equipment followed by a full-scale field demonstration) are scheduled for FY 76. Also to be conducted in FY 76 is a study to determine whether a large-diameter conical borer (one suitable for development entries or shafts) is technically and economically feasible, and if so, in what size range. If the results of the demonstration and study indicate that development of a large-diameter conical borer is warranted, the development work will begin in the transition quarter of FY 76.

There are some cases where a raise-boring system would be the most efficient method of shaft excavation--e.g., in sinking shafts where entries have been developed. But, as mentioned previously, a limitation of the raise-boring system is the diameter attainable, due to torque limitations of the drill string. A project will be initiated in FY 76 to develop a powered raise head that will enable excavation of a 20-foot-diameter shaft in a single pass. By powering the cutterhead, the need to transmit torque through the drill string can be reduced or eliminated. This will permit usage of currently available raise-boring machines without modification.

The lining of shafts excavated by raise-boring systems is both time-consuming and dangerous. Present methods require crews in the shaft to manually set forms, pour concrete, or install steel supports. A solution being investigated involves mixing of shotcrete on the surface, pumping it down the unlined shaft, and spraying it on the walls with revolving nozzles. Application rates up to 30 feet per hour in 8- to 12-foot-diameter shafts are expected.

5.2.3 Degasification of Coal Seams and Methane Recovery

All coal beds contain methane. This methane was formed from plant materials by biochemical and bacterial transformations during the peat stage of deposition and, subsequently, by metamorphic processes as the peat was buried and increased in rank to become coal. The fine pore structure of coal and degraded peat, with their attendant absorptive capacity, accounts for much of the retention of methane produced during the coalification phase.

Because it is porous, coal tends to retain methane both as free gas in cracks and fractures and by absorption on the surface of micropores. The amount of gas within a coal bed is related to pressure, temperature, moisture, depth, degree of fracturing, and permeability of adjacent strata. When the coal is mined, gas emission depends upon the crack and pore structure, size of coal particles, natural fracture spacing, moisture content, and differential pressure.

The methane content of coal beds is known to be highly variable. Coal at depths of less than 1500 feet has been found to contain from 0.001 cubic feet of methane per ton to 390 cubic feet of methane per ton. Assuming that deeper coal beds contain more methane because of higher pressures, a conservative estimate can be made that the average methane content of all coal reserves with less than 3,000 feet of cover is 200 cubic feet of methane per ton. Thus, based on the estimate that there is a potential reserve of 1.3×10^{12} tons of coal with less than 3,000 feet of cover, there is a potential reserve of 260 trillion cubic feet of pipeline--quality gas (950 to 1,000 Btu per cubic foot).

In current mining practice, methane emission from the coal face and gob areas is controlled by the volume and direction of ventilation air. Maximum emission rate occurs at the working face during the actual mining process at a time when exposure is greatest to a potential ignition source from mining machinery. It follows that ventilation of the working face is very critical and methane concentration must, by law, be maintained below 1 percent. Any time this

concentration exceeds 1 percent, coal production must cease and not be resumed until the methane concentration is below 1 percent by ventilation. Time delays in coal production due to excessive methane emission and concentration can be significant in a mine operating in a gassy seam and can amount to as much as 40 minutes of lost time per shift.[1] Blocks of coal left in place during mine development continue to emit methane at a declining rate, but even this lesser contribution to the ventilating load has a significant effect on reducing ventilation efficiency and capability to dilute concentrations at the working face. As a consequence, a greater volume of air is required resulting in increased ventilation costs. Any effective procedure to drain and utilize methane from coal seams in advance of mining will result in the availability of a natural energy resource (heretofore wasted), increased coal production, increased safety, and lower ventilation costs.

Approximately one in eight deaths in U.S. coal mines during the last 70 years has been attributed to the ignition of methane and coal dust. Unfortunately, this ratio has increased in recent years, in spite of the fact that the frequency of occurrence of major mine explosions and the fatality rate associated with coal mining have both decreased. This may, in part, be due to the increased liberation rate of methane in today's mines. Studies have shown that the methane emission rate depends on both the mine depth and on the coal production rate. Studies have also shown that related explosions can be expected

as mining rates and depths increase, unless steps are taken to remove this gas before it enters the ventilation air.

A systematic study of methane control techniques was initiated 12 years ago by the Bureau of Mines.[2] One of the most promising of these techniques for methane control appears to be the degasification of coal seams in advance of mining operations and the degasification of gob areas.

Since the passage of the Federal Coal Mine Health and Safety Act and Section 75.316 of the mandatory safety standards in Title 30, U.S. Code of Federal Regulations, the use of ventilation to control methane in gassy mines is becoming very difficult and expensive. Huge quantities of air would have to be circulated in large mines (as much as 10 to 15 tons of air per ton of coal removed in some cases). Thus, it would be desirable to remove the methane before mining starts. Projects to explore various degasification techniques are being conducted by BOM. There are two major activities which are:

- Use of large vertical shafts with horizontal degasification holes at the bottom of these shafts,
- Use of directional drilling techniques in drilling wells directly from the surface.

These two activities are described briefly in the following paragraphs.

(a) Degasification Using Large Vertical Shafts. BOM initiated a project for degasifying a virgin coal bed in northern West Virginia in 1972.[1] This project involved drilling a large multipurpose bore

hole down 839 feet to a coal bed that, at that time, was not expected to be mined for 2 years. The bottom of the bore hole was enlarged to 14 feet in diameter and 7 horizontal holes were drilled radially into the 8-foot-thick coal seam. These 7 horizontal shafts were then connected to a common collector at the bottom of the vertical shaft. These horizontal holes averaged about 500 feet in length. After 180 days, 91 million cubic feet of gas had been removed from the affected coal bed. This represents 60 to 70 percent of the gas calculated to be contained in the area affected by the holes. Based on the average length of the horizontal holes, at least 9 million cubic feet, or 384,000 tons of coal were involved. The total gas emitted per estimated ton of coal after 180 days was 237 cubic feet. This volume is approximately 98 percent of the total estimated volume of methane (250 cubic feet per ton) contained in a ton of virgin coal in that particular area. A percentage of this volume could be methane migrating from outlaying coal.

At the present time, it appears that mining operations will not reach the well site for another five years. Methane emissions will be monitored at the mining face during the development of the mains as they arrporach the well to determine the actual area of effectiveness of this degasification technique.

As of this writing, a total of 544 million cubic feet of methane has been removed; of this, 139 million cubic feet have been sold to a gas company. The degasification well is still producing and BOM expects ot to continue to produce methane at the current level for at least the next five years.

A second degasification well was drilled in 1973 by BOM in the same general area as the first well. It consisted of 5 horizontal holes drilled into a coal bed at the bottom of an 18-foot-diameter shaft sunk by the coal company. These horizontal holes range in length from 1,000 to 2,000 feet. To date, this second well has produced 434 million cubic feet of gas at the rate of 1 million cubic feet per day. The results of this particular project have not yet been published.

(b) Degasification Wells Using Directional Drilling. In June 1973, using directional drilling equipment and techniques, a 3-inch-diameter hole was drilled from the surface into a coal bed 8 feet thick lying 800 feet beneath the surface. The shaft was started at 18° from the vertical and curved into the coal bed in a horizontal direction. The length of the hole was 1,300 feet from the surface to the edge of the coal bed, and 413 feet horizontally into the coal bed. A 2-inch pipe was used as a temporary casing and a pump was installed to remove the water in the hole. By late 1973, the hole was enlarged to 4 inches in diameter and two additional horizontal holes were drilled branching off from the main hole.

The activity has not been documented to date due to the fact that this project was interrupted by plans of the mining company to flood the area. The project area has not yet been flooded, but operations have been suspended and only a minimal amount of gas has been produced.

5.3 Related Subprograms

The hypothetical new system HSMD is based on the High-Speed Mine Development Subprogram and the Methane Removal and Recovery Subprogram. The Methane Removal and Recovery Subprogram is associated with degasification of coal seams ahead of mining operations so that the coal in these degasified seams can be mined more easily. Thus, coal seam degasification is an activity that should be part of a coal mine development system.

These two subprograms are described in general terms in the following subsections.

5.3.1 The High-Speed Mine Development Subprogram

The High-Speed Mine Development Subprogram has as one of its major objectives the development of equipment and procedures that will reduce the time, cost, and safety hazards involved in sinking shafts and driving entries required for coal mine development. The projects of the subprogram are designed to encourage industry participation so that they will be willing to use the new technology once it is developed. This will be done through cost sharing contracts and cooperative demonstrations.

This subprogram consists of three main project areas:

a) Mine Planning
b) Horizontal Development
c) Vertical Development

These areas are described briefly in the following subsections. Table 5-1 presents the funding schedule for this subprogram.

TABLE 5-1

HIGH-SPEED MINE DEVELOPMENT SUBPROGRAM FUNDING

FISCAL YEAR	FUNDING BY FISCAL YEAR ($MILLION)	DISCOUNTED FUNDING @12%
1975	4.00	4.00
1976	7.00	6.25
1977	6.00	4.78
1978	6.00	4.27
1979	5.50	3.50
1980	8.00	4.54
TOTAL	36.50	27.34

5.3.1.1 Mine Planning. This activity area will look at the various procedures used in mine planning which cover such things as:

a) Mine configurations
b) Development schedules
c) Mining techniques
d) Production Schedules
e) Capital requirements over mine life

The object of this activity area is to develop an integrated set of procedures which optomize mine development in that the development time and cost are reduced to a minimum consistant with maximum profitability. This can include the use of computer systems and various computer mine models to be used as a tool by the mine operators.

Planning procedures for both longwall and room-and-pillar mining will be addressed so that optimum use can be made of these two mining systems.

5.3.1.2 Horizontal Development. The purpose of this activity is to apply the latest in domestic and foreign tunneling and boring technology to the development of equipment for driving mains, submains, and production headings in U.S. coal mines.

In some coal seams, sufficiently large entries for ventilation and for access by men and materials can be obtained within the seam; in others, it is not possible. Therefore, equipment capable of excavating the coal, and non-coal materials associated with the roof and floor, will be developed, as well as equipment strictly designed for in-seam excavation.

For example, a tunnel-boring machine modified for compatibility with the coal-mine environment will be demonstrated in an operating mine. The machine will cut an 18-foot-diameter entry removing 8 feet of roof, 7 feet of coal, and 3 feet of floor. Using the data from the demonstration, an advanced boring machine or machines will be developed.

5.3.1.3 Vertical Development. The purpose of this activity area is to replace the conventional drill-and-blast method of blind-shaft excavation with rapid mechanized means of blind-shaft excavation and to improve and expand the capabilities of raise-boring systems.

Blind-shaft development is extremely important as it does not require in-mine development to the site as does raise-boring. This then permits concurrent shaft and entry development. In this area, a blind-shaft borer incorporating tunnel-boring technology will be developed by The Robbins Company. To fill the needs of the coal-mine operators on a more immediate basis, conventional oil drill rigs will be modified to permit excavation of large-diameter ventilation and material-handling shafts.

Raise-boring is an accepted means of rapid mechanized shaft excavation where entry development has already taken place. A limitation of the raise-boring system is the diameter attainable, which is due to torque limitations of the drill string. Therefore, a powered raise bore cutterhead will be developed to expand these capabilities. The lining of raise-bored shafts is time-consuming and dangerous as present methods require that crews in the shaft manually set forms,

pour concrete, and install steel supports. To alleviate this problem, a remotely controlled shotcrete support system will be developed. Raise-boring requires pilot-hole drilling, whose accuracy must increase as the depth of the seam increases. Consequently, a conical stabilizer is under development to improve pilot-hole accurracy.

5.3.2 The Methane Removal and Recovery Subprogram

The object of the Methane Removal and Recovery Subprogram is to demonstrate the feasibility of recovering commercial quantities of pipeline - quality methane from virgin coal beds and from underground gob areas to increase the production potential and productivity from United States coal deposits.

This subprogram consists of five major projects:

a) Development of methods for placing degasification wells in advance of mining operations.

b) Development of directional drilling equipment and techniques.

c) Development of methods for degasification of gob areas.

d) Development of a coal resources data base.

e) Testing of areas for gas production in little-mined coal seams in Texas, Michigan, Iowa, Wyoming, Utah and Colorado.

These five projects will lead to the development of a complete methane removal and recovery system that should be available in FY81.

The funding for this subprogram is given in Table 5-2.

TABLE 5-2

METHANE REMOVAL AND RECOVERY SUBPROGRAM FUNDING

FISCAL YEAR	FUNDING BY FISCAL YEAR ($MILLION)	DISCOUNTED FUNDING @12%
1975	2.00	2.00
1976	3.00	2.68
1977	2.00	1.59
1978	2.00	1.42
1979	1.50	0.95
1980	1.00	0.57
TOTAL	11.50	9.21

5.3.2.1 Development of Methods for Placing Degasification Wells

The goals of this project are to:

a) Determine methods of locating degasification wells with respect to coal seams in advance of mining.

b) Determine methods of scheduling the drilling of degasification wells based upon the gas content of the coal seam and upon the mining schedule.

c) Determine methods for specifying the physical characteristics of the degasification well based on the coal seam characteristics.

This project will entail the drilling of experimental degasification wells into gassy coal seams ahead of mining operations. The production of methane by these wells will be monitored and analyzed in an effort to determine the best methods for locating degasification wells with respect to the coal seam and to determine optimum well densities. After various lengths of time, the coal will be mined from the degasification areas of the seam to determine the degree to which the methane has been removed. Thus, the optimum scheduling of degasification wells can be determined along with the detailed specification of their configuration.

5.3.2.2 Development of Directional Drilling Equipment and Techniques

This project seeks to develop a technique for drilling horizonal holes in a coal bed without the use of an expensive vertical bore hole. Thus, through directional drilling techniques; a hole can be drilled from the surface on a slant down to the coal bed and the

direction of the hole can be changed so that it enters the coal seam horizontally. One goal of this activity is to be able to drill a 3,500 foot horizontal hole into a coal bed from the surface.

5.3.2.3 Development of Methods for Degasification of Gob Areas. Degasification wells will be drilled into gob areas of both abandoned and operating mines to evaluate the methane production potential of these areas. Methods will be developed for assessing of areas for their useful methane production potential and in terms of how much methane these gob areas contribute to the mine atmosphere. Studies will be conducted to determine the effect of gob degasification on ventulation costs, production, and safety.

5.3.2.4 Development of a Coal Resources Data Base. In conjunction with the state geological surveys, the Bureau of Mines will develop a data base containing information about coal reserves throughout the United States that might have a potential for gas production. This data base will serve as a means for planning coal mine gas production and will enable the coal mining companies and gas production companies to pinpoint potential degasification wells. This project will result in a stable data base and methods by which this data base can be used to help optimize the location of degasification wells.

5.3.2.5. Testing of Gas Production in Selected Areas. This project will involve drilling degasifications wells in virgin or little-mined coal seams in areas of Texas, Michigan, Iowa,

Wyoming, Utah and Colorado. Data from this activity will be used to evaluate methods of degasification as applied to various geological regions.

5.4 Methodology

The evaluation of the hypothetical HSMD System in terms of benefits as a result of it replacing the current technique of mine development must be done in terms of the projected production as expressed in the two coal production scenarios defined earlier. The new mines specified in these scenarios must be grouped by those parameters that have a major impact on development costs. These parameters are:

a) Province in which the mine is located.

b) Production capacity.

c) Entry type.

d) Mine depth.

These parameters and the assumptions associated with them are described in the following:

5.4.1 Classification by Province

It is assumed that the new coal mines opening in the Rocky Mountain Province will have the same characteristics in terms of mine development as those new mines in the Eastern Province and those in the Great Plains Province will be similar to those in the Interior Province. This assumption is based on a survey of active mines contained in the Keystone Coal Industry Manual (1972) in terms of

the characteristic of interest.

5.4.2 Productive Capacity

Development costs are a function of mine productive capacity and as a result all mines were divided into two capacity categories: those mines that will produce less than 1 million tons per year, and those that will produce 1 million, or more, per year.

5.4.3 Entry Type and Mine Depth

Mines are classified into three depth categories:

a) 300 to 599 feet

b) 600 to 899 feet

c) 900 to 1200 feet

The assumption made in this classification is that there is an unsignificant number of underground mines with depth of less than 300 feet or greater than 1200 feet. Each depth class will have its own characteristic development costs.

Entry type will be a function of depth as follows:

a) All slope and drift development will take place in mines with depth in the 300-599 feet range.

b) All shaft development will take place in mines with depth in the 600-1200 feet range.

For each combination of mine parameters defined in Table 5-3 a cost and saving schedule was developed for each year of mine life. In the evaluation of the HSMD system, it is assumed that the mine has a productive life of 20 years but that mine development starts three years prior to the start of production. Thus, coat and saving

schedules consist of 23 sets of yearly data. In these cost and saving schedules are costs that are just associated with development cost and savings - both operational and capital. All other costs and savings are associated with production by a particular mining system. Tables 5-3 and 5-4 present summaries of new mine distribution for the HSMD analysis.

5.5 The Current System

Currently the initial development time for a new coal mine is approximately five years or longer. The first two years of this period are associated with mine planning, and financing and the final three years are associated with shaft sinking and main entry development. For simplicity of analysis, mine development will be considered to start with the start of construction as this assumes that the cost associated with pre-construction development planning are the same in both the current system and the hypothetical new HSMD System. Therefore, there will be 23 years of cash flow for a new mine with an average productive life of 20 years.

No attempt has been made to estimate the exact number of shafts, slopes, or drifts required for approaching the coal seam of a new mine, or the number of entries to be developed. The number and type of entries to be developed will be dependent upon mine depth, mine productivity, ventilation requirements, length of entries, etc. For the purpose of comparison with HSMD in terms of cost, a schedule of costs is developed for selected combinations of shafts, slopes, drifts and entries. Tables 5-5 and 5-6 present cost comparisons for various types of both vertical and horizontal mine development.

TABLE 5-3

CUMMULATIVE NEW MINE OPENINGS AND COAL PRODUCTION BY MINE PARAMETERS FOR THE PERIOD 1981-2020

PROVINCE	DEPTH	CAPACITY MTY	FOR PERIOD 1981 TO 2020			
			NO. OF NEW MINES		TOTAL PRODUCTION (Millions of Tons)	
			SCENARIO 1	SCENARIO 2	SCENARIO 1	SCENARIO 2
EASTERN AND ROCKY MOUNTAIN	300 - 600	<1	108	101	1,525	1,445
EASTERN AND ROCKY MOUNTAIN	300 - 600	≥1	554	571	7,224	7,968
EASTERN AND ROCKY MOUNTAIN	600 - 900	<1	6	6	90	85
EASTERN AND ROCKY MOUNTAIN	600 - 900	≥1	81	84	1,037	1,167
EASTERN AND ROCKY MOUNTAIN	900 - 1200	<1	3	2.5	39	35
EASTERN AND ROCKY MOUNTAIN	900 - 1200	≥1	35	35	452	499
INTERIOR AND GREAT PLAINS	300 - 600	<1	22	31	316	432
INTERIOR AND GREAT PLAINS	300 - 600	≥1	34	54	449	728
INTERIOR AND GREAT PLAINS	600 - 900	<1	0.2	0.4	3	5
INTERIOR AND GREAT PLAINS	600 - 900	≥1	64	101	840	1,361
INTERIOR AND GREAT PLAINS	900 - 1200	<1	0.1	0.2	2	2.6
INTERIOR AND GREAT PLAINS	900 - 1200	≥1	27	43	480	582
TOTAL			934.3	1,029.1	12,457	14,309.6

TABLE 5-4

DISTRIBUTION OF NEW MINES BY DEPTH, TYPE OF DEVELOPMENT, PRODUCTION, AND PROVINCE

PROVINCE	TYPE	DEPTH (FT)	SCENARIO 1		SCEANRIO 2		PRODUCTION-% OF TOTAL (AVERAGES FOR BOTH SCENARIOS)	
			<1 MTY	≥1MMTY	<1 MTY	≥1 MTY	≤1 MTY	≥1 MTY
EASTERN & ROCKY MOUNTAIN	SLOPE & DRIFT	300 - 599	1,525	7,224	1,445	7,968	92.26	82.80
	SHAFT	600 - 899	90	1,037	85	1,167	5.44	12.02
		900 - 1200	39	452	35	499	2.30	5.18
TOTALS			1,654	8,713	1,565	9,634	100.00	100.00
INTERIOR & GREAT PLAINS	SLOPE & DRIFT	300 - 599	316	449	432	728	98.35	26.52
		600 - 899	3	840	5	1,361	1.05	49.57
		900 - 1200	2	480	2.6	582	0.60	23.91
TOTALS			321	1,769	439.6	2,671	100.00	100.00

TABLE 5-5

COMPARISON OF ESTIMATED COSTS FOR VERTICAL MINE DEVELOPMENT

SYSTEM PROFILE	HSMD SYSTEM		EXISTING SYSTEM
	BLIND-DRILLED SHAFT	RAISED-BORED SHAFT	CONVENTIONAL SHAFT SINKING
DEPTH	1000 FT	1000 FT	1000 FT
DIAMETER	20 FT	20 FT	20 FT
CREW SIZE	26 MEN	10 MEN	36 MEN
DAYS TO COMPLETE	45 DAYS	54 DAYS	167 DAYS
SYSTEM COSTS			
EQUIPMENT COSTS	$1,950,000	$2,000,000	$514,000
OPERATING COST PER DAY	$ 4,400	$ 3,300	$ 3,000

TABLE 5-6

COMPARISON OF ESTIMATED COSTS FOR HORIZONTAL MINE DEVELOPMENT

	HSMD SYSTEM	EXISTING SYSTEM
DEVELOPMENT ACTIVITY	TUNNEL-BORING MACHINE DEVELOPMENT	CONTINUOUS-MINERS DEVELOPMENT
MAIN ENTRIES	4	10
SUBMAIN ENTRIES	4	8
BUTT ENTRIES	1	3
EQUIPMENTS	4	8
INITIAL LONGWALL MINING	MOST REMOTE LONGWALL PANEL	CLOSEST LONGWALL PANEL
CAPITAL COST		
PER MACHINE UNIT	$ 800,000	$300,000
PER SYSTEM UNIT	$1,252,000	$858,000
TOTAL FOR DEVELOPMENT	$5,008,400	$6,961,000

SOURCE: USBM Unpublished Report 1974, "Study of the Economic Feasibility of Tunnel Boring Machines for Coal Mine Development."

The basic mine model for an underground mine developed by Katell which has a production of 1 MTY serves as a basis for comparison of the current system with the HSMD System.

The following costs were assumed for the current system.

a) The cost of certical development for large-diameter shafts was assumed at $1,000/ft to $1,100/ft depending on mine location and depth.

b) The cost vertical development for small-diameter raise was assumed at $450 to $600/ft depending on size of raises and mining depth.

5.6 The Hypothetical HSMD System Description

The hypothetical HSMD System consists of an integrated set of procedures and equipment that will do the following:

a) Mine Planning.

b) Degasification of the coal seams.

c) Horizontal and vertical boring.

The funding schedule for this new system is given in Table 5-7. This hypothetical system is described in terms of various savings over current mine development systems. This comparison between the HSMD System and the current development system is expressed in terms of a mine model developed by Sidney Kattel and modified for use in this benefit/cost study. The costs in this model are split into capital costs and total cost. Revenue from the recovered methane is added to the model as a negative cost.

It is assumed that in the area of degasification average values would be used in terms of methane concentration and drilling costs.

TABLE 5- 7

FUNDING SCHEDULE FOR THE HSMD SYSTEM

FISCAL YEAR	R&D COSTS ($MILLIONS)		PRESENT VALUE @12% ($MILLIONS)		TOTAL PRESENT VALUE ($MILLIONS)
	HSMD	METHANE REMOVAL	HSMD	METHANE REMOVAL	
1975	4.0	2.0	4.00	2.00	6.00
1976	7.0	3.0	6.25	2.68	8.93
1977	6.0	2.0	4.78	1.59	6.37
1978	6.0	2.0	4.27	1.42	5.69
1979	5.5	1.5	3.50	0.95	4.45
1980	8.0	1.0	4.54	0.57	5.11
TOTAL	36.5	11.5	27.34	9.21	36.55

The average coal seam is assumed to have 200 cubic feet of methane/ton, and that directional drilling cost $10/ft with each well being 3,500 feet long.

The HSMD System will have the following advantages over the current system:

a) Shorter lead time required for underground mine development.

b) Rapid development of coal reserve.

c) Greater production flexibility.

d) Improved safety and reduced accident cost.

e) Reduced ventilation, maintenance, and timing/support costs.

f) Reduced number of entries for mains and submains.

The development of high-speed shaft construction techniques by vertical boring will offer the following advantages:

a) Rapid excavation.

b) Methane removal and recovery.

c) Ability to cope with water inflow, bad ground; and gas in-flows

d) Delivery of muck to the surface without interference with production.

5.6.1 Degasification of Coal Seams

One of the initial activities associated with the HSMD System is a coal seam degasification. This will consist of drilling clusters of degasification wells into the coal seam at least two years ahead of mining operations. These degasification wells will be drilled using directional drilling techniques which will enable a hole to

be curved from the near vertical into the horizontal as it enters the coal seam and will penetrate the coal seam for at least 3,5000 feet. These degasification wells will produce pipe-line quality gas at a production rate governed by the methane concentrations of the coal seam--the deeper the seam, the greater the methane concentration in general. The gas thus produced will be sold to a gas company and will thus provide revenue for the coal mine operator even during the mine development stage.

The horizontal boring into coal seams and the development of main and submain entries will be speeded up due to the fact that the coal seam in the area of activity will have been degasified thus cutting time and costs.

As the mine is expanded over mine life degasification wells will be drilled into the coal seam at least two years ahead of mining operation.

5.6.2 Cost Reduction Objectives

Three basic cost reduction objectives were defined by the sub-program and they were:

a) reduce entry development time by 50 percent with a corresponding cost reduction of 25 percent,

b) reduce blind-shaft construction time by a factor of 4 with a corresponding cost reduction of 15 percent, and

c) reduce raise-bore construction time by 50 percent with a corresponding cost reduction of 35 percent.

These cost objectives are assumed in cost comparisons with the current system and are shown as benefits associated with the HSMD system.

5.6.3 Market Penetration

The market penetration of the HSMD system is estimated at an annual rate of 5 percent until it reaches maturity. It is a conservative estimate even though the technology is partially innovative since the coal mine operators will be aided by the construction contractors and equipment manufacturers in graudally replacing the existing system. Thus, it will take approximately 20 years for full market penetration.

5.7 Analysis

The first step in the analysis is to identify the various benefit schedules that are to be developed. These schedules will assume a three year development period before the start of production. Thus, in this analysis, the model mine life will last at least as long as 23 years. Thus, the comparative benefits will consist of a 23 year schedule. The results of calculating the benefits over the benefit evaluation period 1980-2020 are summarized in Table 5-8 and given in detail in Tables 5-9 and 5-10. These benefits are the capital cost savings and the total cost savings.

It can be seen that most of the benefits accrue to mines that are between 300-600 feet, which is to be expected as the majority of production during the benefit evaluation period comes from mines with depths in that range (See Figure 5-1).

Manpower savings due to increases in productivity and accident reduction were not calculated for the HSMD system since it is not a production system.

TABLE 5-8

SUMMARY OF BENEFITS AND BENEFIT/COST RATIOS
FOR THE NEW HIGH-SPEED MINE DEVELOPMENT SYSTEM

BENEFIT-BENEFIT/COST RATIO	SCENARIO 1	SCENARIO 2
TOTAL COST SAVINGS	13.34	16.38
CAPITAL COST SAVINGS	1.34	1.70
MANPOWER SAVED DUE TO ACCIDENT REDUCTION	--	--
MANPOWER SAVED DUE TO INCREASED PRODUCTIVITY	--	--
ADDITION TO RECOVERABLE RESERVES	--	--
IMPACT ON ENVIRONMENT	--	--

TABLE 5-9

DETAILED BENEFIT/COST RESULTS

PROVINCE MINE SIZE MINE DEPTH	BENEFIT/COST RATIO			
	TOTAL COST SAVINGS ($/R&D$)		CAPITAL COST SAVINGS ($/R&D$)	
	SCENARIO 1	SCENARIO 2	SCENARIO 1	SCENARIO 2
EASTERN & ROCKY MOUNTAIN PROVINCES				
<1 MILLION TONS/YEAR				
300 - 599 FEET	1.88	1.83	0.17	0.18
600 - 899 FEET	0.11	0.11	0.01	0.01
900 - 1200 FEET	0.03	0.03	0	0
TOTAL	2.02	1.97	0.18	0.19
≥1 MILLION TONS/YEAR				
300 - 599 FEET	7.71	9.31	0.77	0.95
600 - 899 FEET	1.14	1.38	0.12	0.15
900 - 1200 FEET	0.27	0.34	0.05	0.06
TOTAL	9.12	11.03	0.94	1.16
INTERIOR & GREAT PLAINS PROVINCES				
<1 MILLION TONS/YEAR				
300 - 599 FEET	0.39	0.50	0.04	0.05
600 - 899 FEET	0.01	0.01	0	0
900 - 1200 FEET	0	0	0	0
TOTAL	0.40	0.51	0.04	0.05
≥1 MILLION TONS/YEAR				
300 - 599 FEET	0.48	0.77	0.05	0.08
600 - 899 FEET	0.92	1.46	0.09	0.15
900 - 1200 FEET	0.40	0.64	0.04	0.07
TOTAL	1.80	2.87	0.18	0.30
TOTAL FOR ALL MINES	13.34	16.38	1.34	1.70

TABLE 5-10

BENEFIT/COST RATIOS SUMMED OVER MINE SIZE

PROVINCE MINE DEPTH	BENEFIT/COST RATIOS			
	TOTAL COST SAVINGS ($/R&D $)		CAPITAL COST SAVINGS ($/R&D $)	
	SCENARIO 1	SCENARIO 2	SCENARIO 1	SCENARIO 2
EASTERN & ROCKY MOUNTAIN				
300 - 599 FEET	9.59	11.14	0.94	1.13
600 - 899 FEET	1.25	1.49	0.13	0.16
900 - 1200 FEET	0.30	0.37	0.05	0.06
TOTAL	11.14	13.00	1.12	1.35
INTERIOR & GREAT PLAINS				
300 - 599 FEET	0.87	1.27	0.09	0.13
600 - 899 FEET	0.93	1.47	0.09	0.15
900 - 1200 FEET	0.40	0.64	0.04	0.07
TOTAL	2.20	3.38	0.22	0.13
TOTAL FOR ALL MINES	13.34	16.38	1.34	1.70

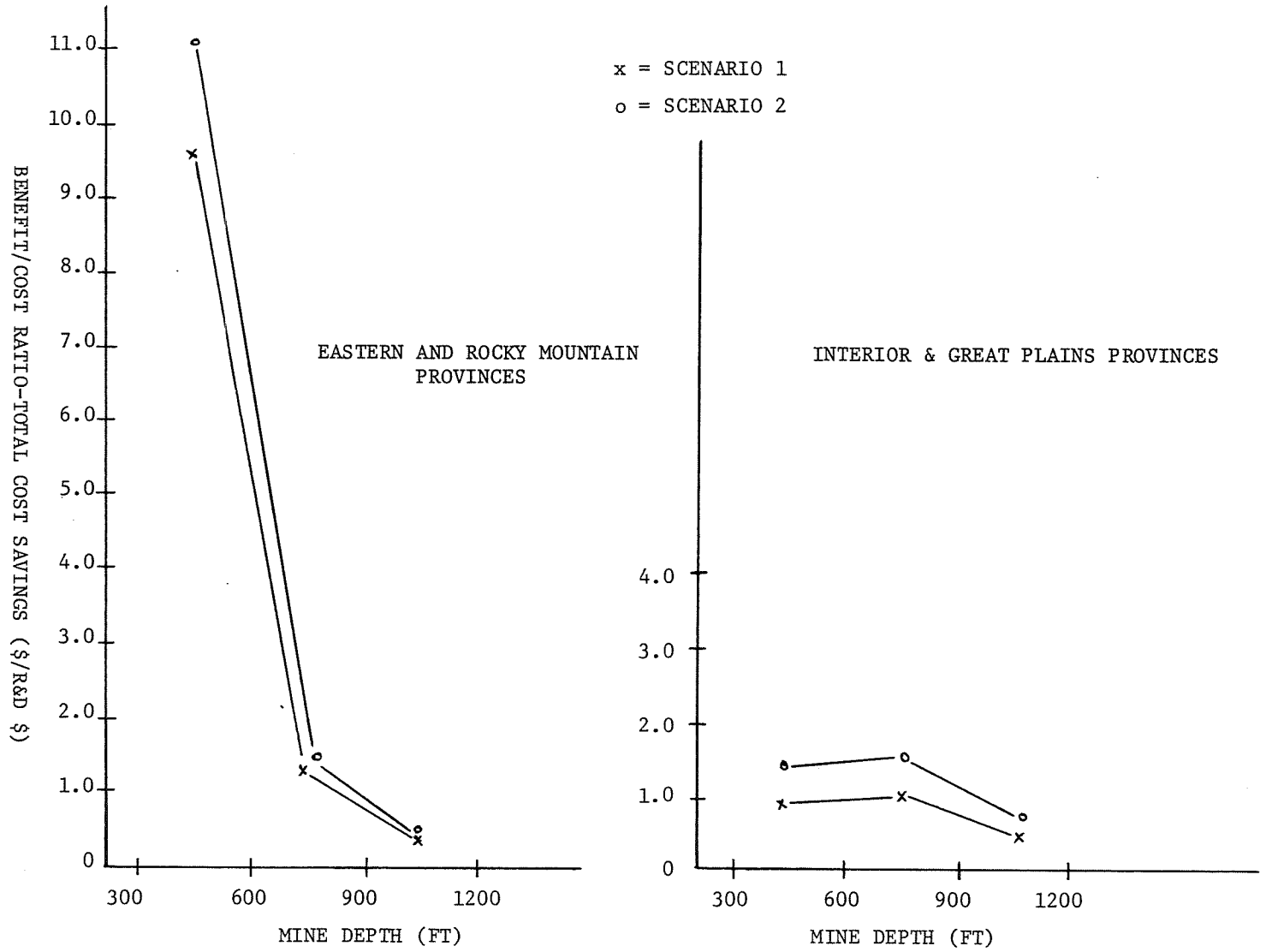
x = SCENARIO 1
o = SCENARIO 2
BENEFIT/COST RATIO-TOTAL COST SAVINGS ($/R&D $)
11.0
10.0
9.0
8.0
7.0
6.0
5.0
4.0
3.0
2.0
1.0
0
EASTERN AND ROCKY MOUNTAIN PROVINCES
300
600
900
1200
MINE DEPTH (FT)
INTERIOR & GREAT PLAINS PROVINCES
4.0
3.0
2.0
1.0
0
300
600
900
1200
MINE DEPTH (FT)

FIGURE 5-1

Table 5-11 presents the results of assuming that the subprogram goals are not fully achieved and that full market penetration is delayed five years.

Coal production cost reductions reflect the reduction in capital costs associated with the HSMD.

TABLE 5-11

SENSITIVITY ANALYSIS

CASE*	BENEFIT COST RATIOS ($/$ OF R&D COST) TOTAL COST SAVINGS	CAPITAL SAVINGS	PRODUCTION COST REDUCTION $/TON	TOTAL DISCOUNTED BENEFITS @12%**
A	13.33	1.42	$1.19	$487.21 MILLION
B	9.85	1.42	$0.85	$360.02 MILLION
C	9.38	1.01	$1.20	$342.80 MILLION

* CASE A: Basic Case - aggressive approach (100% of cost reduction objectives achieved)

CASE B: Modification of Basic Case - 50% of cost reduction objectives in horizontal development achieved.

CASE C: Modification of Basic Case - full market penetration delayed 5 years.

** Benefits computed over 1981-2020 period.

REFERENCES

1. Fields, 1973

2. BOM IC 8621, 1973

6.0 AUTOMATED LONGWALL MINING

This section analyzes the Bureau of Mines' efforts to develop and promote the use of improved, remotely controlled and automated longwall mining systems. These mining systems are related to four subprograms of the Bureau of Mines R&D program:

a) Automated Longwall

b) Continuous Face-to-Preparation Plant Coal Haulage Systems

c) Environmental Protection of Surface Areas

d) Advanced Mining Systems

Two representative mining systems will be analyzed. They are:

System 1: Improved longwall mining system

System 2: Improved longwall mining system with high capacity armored conveyor

These systems incorporate improved hardware, subsystems and procedures developed in the related subprograms. They will be described in more detail in Section 6.5.

Benefits will be generated principally from improved coal extraction and system productivity. Benefits will accrue directly from:

- increased production per man-shift,
- increased production per machine-shift,
- reduced accidents per man-shift.

Benefits also result from:

- reduced manpower requirements per million tons of coal, resulting in reduced exposure to hazards, thereby reducing accidents.
- improved mining techniques resulting in--
  - increased recovery factors,
  - recovery of additional resources.

Since most of the benefits are related directly or indirectly to improved coal extraction, the analyses concentratedon operations directly relating to coal extraction at the face and removal of coal from the face to the next stage of haulage (panel belt).

6.1 Production Scenarios

Two scenarios are used for this analysis, identified as Scenario 1 and Scenario 2. They are defined in Table 6-1, in terms of new mine openings by year from 1975 through 2020 in three sizes of mines--0.5 MTY, 1.0 MTY, and 3.0 MTY. The total mine opening scenarios for the United States are used rather than the individual province scenarios, since no cost differences have been identified by province.

Production scenarios are in clean tons while equipment productivity is usually reported in tons mined at the face. Clean tons are about 80 percent of the face tons. The distinction will be made clear wherever necessary.

6.2 State-of-the-Technology

Most of the major coal-producing nations of the world mine nearly all of their underground coal by longwall methods.

Longwall is the only mining system in production today that has coal cutting, roof support, and haulage components designed, matched, and integrated for high, continuous production rates and for high recovery factors. However, despite the excellence of the longwall concept and the tremendous capability to mine coal, longwalls have not achieved anything like their potential in the United States.

TABLE 6-1

UNDERGROUND MINING

DISTRIBUTION OF NEW MINE REQUIREMENTS BY TONNAGE
(Millions of Tons)

	SCENARIO 1			SCENARIO 2				SCENARIO 1			SCENARIO 2		
Year	.5MTY	1MTY	3MTY	.5MTY	1MTY	3MTY	Year	.5MTY	1MTY	3MTY	.5MTY	1MTY	3MTY
1975	6.3	11.3	14.5	6.3	14.4	18.4	1998	6.3	14.8	18.9	6.3	19.6	25.2
1976		9.5	12.3		8.7	11.1	1999		14.8	18.9		21.8	25.0
1977		9.5	12.3		11.7	15.1	2000		15.7	20.1		17.0	21.8
1978		9.5	12.3		11.7	15.1	2001		16.1	20.7		21.8	28.0
1979		9.5	12.3		14.3	18.4	2002		15.7	20.1		20.1	25.7
1980		9.5	12.3		12.2	15.6	2003		15.7	20.1		20.5	26.3
1981		9.9	12.8		16.1	20.7	2004		15.7	20.1		18.7	24.1
1982		9.5	12.3		16.1	20.7	2005		16.5	21.2		20.9	26.9
1983		9.5	12.3		16.1	20.7	2006		14.8	18.9		18.3	23.5
1984		9.5	12.3		15.2	19.6	2007		14.8	18.9		17.4	22.4
1985		9.5	12.3		17.0	21.8	2008		14.8	18.9		18.3	23.5
1986		9.5	12.3		15.7	20.1	2009		14.8	18.9		17.8	22.9
1987		9.5	12.3		15.2	19.6	2010		15.2	19.6		17.0	21.8
1988		9.5	12.3		15.7	20.1	2011		13.5	17.3		13.1	16.7
1989		9.5	12.3		15.2	19.6	2012		13.5	17.3		10.9	13.9
1990		9.9	12.8		14.8	19.0	2013		13.5	17.3		12.2	15.6
1991		8.7	11.1		10.9	13.9	2014		13.5	17.3		10.0	12.8
1992		8.7	11.1		8.7	11.1	2015		21.4	27.4		20.5	26.3
1993		8.7	11.1		10.0	12.8	2016		20.1	25.7		16.6	21.2
1994		8.7	11.1		7.8	10.0	2017		20.1	25.7		22.3	28.5
1995		14.8	18.9		18.3	23.5	2018		20.1	25.7		22.7	29.1
1996		14.8	18.9		13.5	17.3	2019		20.1	25.7		24.9	31.9
1997		14.8	18.9		19.2	24.6	2020		20.5	26.3		19.2	24.6

The longwall system consists of three principal parts:

a) a coal cutter

b) an armored chain conveyor

c) a set of self-advancing hydraulic chocks or shields to support the roof.

6.2.1 Longwall Mining

The longwall miner is either a planer type (plow) or a shearer. The planer is the simpler system--a saw-toothed blade which rides on the side of the conveyor. It is dragged across the face by a heavy chain and takes a 3 to 6 inch continuous web of coal off the face. The shearer has either a single or double rotating drums--double drums for thicker seams--with rows of cutting bits set around the surface of the drums. It rides on the armored conveyor, is powered by electric motors, and is driven by means of sprockets which engage the links of a heavy stationary chain which runs the length of the face.

Efforts to automate longwall systems are not new. In some respects longwalls are well suited for automation. Most of the operations are mechanical in nature. One operation that requires considerable skill on the part of the operators, however, is the height control of the shearer drums or the plows.

Some of the best-known attempts at automation of longwalls were carried out by Britain's National Coal Board (NCB) during the 1960's. Remote-Operated Longwall Faces (ROLF 1 and 2) were developed, installed, and demonstrated. They incorporated remote-controlled shearer operation, roof supports and conveyor pushover, with full remote monitoring. While the demonstrations were interesting and

impressive, the systems were never commercially implemented. The reasons for this are not clear. It can be inferred that the costs and operational problems were not considered worth the return. NCB research is now concentrating on machine guidance and steering controls, automatic sequencing of banks of 4-10 roof chocks, and automatic conveyor pushover.

Longwall systems exhibit certain advantages over continuous or conventional mining systems. They can be operated successfully in almost any type of geologic conditions. They are particularly suitable for thin pitching seams under deep cover and under more friable roof strata than can be mined by room-and-pillar methods. Friable roof conditions are in fact an advantage for longwall systems as the roof will subside uniformly behind the roof supports. However, recent installations in the thick Pittsburgh coal bed have been successful under highly competent roof. Surface environmental effects are minimized by uniform subsidence of the roof. Unrecoverable roof support costs are minimized as the roof chocks move with the miner.

Health and safety are important aspects of longwall mining. Under the continuous steel canopy, face workers are less exposed to the dangers of roof and rib falls. The linear configuration facilitates methane and dust control and reduces hazards from haulage and material handling operations. Rock dusting is not required across the face.

Coal recovery with longwall systems is high--near the maximum attainable with conventional or continuous mining systems. However,

the average productivity of 194,000 tons in 1972 represents only about 850 tons per day. On the assumption that most longwalls operate two shifts per day, the average productivity is about 425 tons per shift. Longwalls are actually capable of much more. Bethlehem's mine in Western Pennsylvania in a 42-inch seam set a record of over 3,250 tons in one shift and 9,000 tons in a day.

There are a number of factors that limit productivity achieved by longwall systems to far below their potential.

- System availability: Reliability has been a major factor in limiting the availability of the machine for cutting coal. Even at a cutting rate of 5 tons per minute, it takes only 85 minutes to cut 425 tons of coal. Clearly, the average system availability of longwalls is fairly low. Low system availability can be due to malfunctions, such as in the hydraulic systems, or parts failures such as breakage of a chain. Inadequate routine maintenance and improper operations can cause failures. Delay in diagnosing the malfunction may contribute to system unavailability. Replacement of parts will depend on the availability of that part. The part may be close at hand, or it may be available in store. In some cases, it may only be available from the manufacturer, which usually means in Europe.
- Problems on the face: These usually arise from the occurrence of clay veins, rock inclusion, thickening and

thinning of the seam, or changes in the type of roof such as from shale to sandstone. In some cases, it may be impossible for the machine to cut through. It will then be necessary to go in with cribs to shore up the roof and physically move the longwall past the fault. The stoppage causes immediate roof control problems. Pressure on the roof supports begins to build up, and, if the pressure reaches the load limit, the chocks will yield. The roof may fracture and control can be lost. Recovery can be a hazardous, costly, and time-consuming operation.

- Face-to-face moves: Moving the longwall system from one face to another, after a panel has been mined out, is a time-consuming and difficult operation. It usually takes from two weeks to two months. The time is dependent on mine layout and panel design, affecting the distance that the longwall has to be moved and the difficulty of the route.
- Institutional constraints: Differences between U.S. and foreign practice with respect to health and safety matters affect the productivity of the system and the complication of mine development. Most state and federal laws have been written for room-and-pillar mining methods. Some restrict longwall mining practices and result in lower productivity without increasing safety. Better, no-less-safe techniques

may be developed. For example, the requirement for isolated haulage and multiple escapeways precludes the use of single head and tail entries which is the custom in all other major coal-producing countries.

- Manual functions: In a double-drum ranging shearer, one operator must keep the upper drum at the top of the seam while the other operator keeps the lower drum at the lower interface. These operations must be performed on a machine that rides the conveyor which may itself be severely contorted. The operators must assess the integrity of the roof and be sure that ventilation and water sprays are adequate to control coal dust. Patricularly in thin seams, the operators may be working close to the limits of human capability in controlling and operating these huge machines.

6.2.2 Roof Support

The roof support system consists of powered hydraulic chocks or shields. A chock unit consists of a minimum of four hydraulic jacks mounted on a frame. A pair of jacks support a beam, which supports the mine roof. The chocks are self-advancing when manually actuated.

Shields consist of a base, a gob-shield hinged to the base, and a canopy jointed to the gob-shield. Angled hydraulic jacks support the gob-shield and thereby the canopy.

Chocks have greater load carrying capacity than shields as they are equipped with larger jacks and work at less of a mechanical disadvantage than do shields. However, chocks have little ability to

compensate horizontal motion in the roof. Consequently, they tend to become unstable when the roof is high and are unsuitable for seams over 8 feet thick. For the same reason, they cannot be advanced when in contact with the roof as they will tend to rack backwards. On the other hand, the kinematics of the shield support are such that the loading action of the roof results in forward motion of the canopy. This motion produces forces on the roof directed towards the coal face, opposing forces exerted on the canopy from the roof which tend to be away from the face, and counteracting the tendency for cracks to open in the immediate roof. As a consequence, shields are more stable in high roof situations and are used successfully in seams up to 11 feet thick. Another consequence of these kinematic properties of the shield is that they can be advanced while in contact with the roof. Shields also have the advantage over chocks in that the gob-shield prevents broken rock from falling into the support.

A recently developed variation of the chock is the chock-shield, which combines the load carrying capacity of the chock with many of the properties of the shield. It is kinematically designed so that the canopy tends to oppose horizontal forces directed away from the face. Thus, the chock-shield has similar stable properties to the shield and it can also be advanced under contact with the roof. It may, therefore, be quite suitable for automation.

Roof control factors that operate to limit the productivity of longwall systems include:

- Manual Operations: The chockmen have to actuate each set of powered roof supports. This is a very routine,

highly mechanical function, requiring four manual operations for every four feet of progress of the shearer across the face. The operators must actuate the valves to push the armored conveyor ahead, lower the legs of the chock, advance the chock, and then reset the legs. This must be done while scrambling through the maze of steel machinery in a vertical clearance of 36 inches or less; while maintaining the pace of the shearer which may be as high as 42 feet per minute.

- Roof control problems: Roof control can cause problems over the longwall face and in the head and tail entries. Problems can be particularly serious over the face if mining is interrupted for any length of time and pressure starts to increase as stresses build up in the roof. Non-uniform caving of the overburden behind the face can also cause problems. Ideally, the roof should collapse within a short period and this relieves some of the pressure on the roof supports.

6.2.3 Haulage

The armored chain conveyor, also called a "panzer" conveyor, carries the coal parallel to the face, in front of the roof support chocks, to the head entry where it is transferred to a panel belt running down the entry. Hydraulic rams on the chocks push the conveyor forward, when activated by the chockmen, after passage of the miner. The conveyor continues to operate while it is being pushed forward.

Most longwalls operating in the U.S. must restrict the speed of the shearer to production rates compatible with the capacity of the armored conveyor. Conveyor capacities vary, and are dependent on the speed of operation of the conveyor. The highest capacity conveyors in operation in the U.S. today can haul about 7 tons per minute. Conveyor capacity is a factor that tends to limit productivity of U.S. longwall systems.

6.3 Related Subprograms

This section describes each of the subprograms that contribute to the development of improved and automated longwall mining systems.

6.3.1 Automated Longwall Mining Subprogram

6.3.1.1 Subprogram Goals. Because the United States is not presently a manufacturer of longwall equipment, the goals of the Automated Longwall Subprogram do not envisage any substantial redesign of existing longwall equipment. The Bureau of Mines' program for automation of longwall systems will be selective in the degree of automation and remote control and the direction in which it will proceed. Goals have been set that will achieve most of the benefits to be derived from automation but will limit the additional complexity of the equipment so as not to compromise reliability.

The goals of the Automated Longwall Mining Subprogram are:

a) To improve existing subsystems.

b) To develop technology to minimize subsystem interface problems.

c) To develop improved systems.

d) To develop criteria for determining the suitability of coal seams for longwall mining.

e) To accelerate the application of developed technology through cost-sharing demonstrations.

6.3.1.2 Subprogram Description. To achieve its goals, the longwall R&D effort addresses five major areas:

a) increasing longwall production and productivity,

b) promoting use of longwall mining,

c) development and demonstration of advanced longwall systems,

d) development of automated longwall equipment and systems,

e) development of surface test facilities.

Automation is expected to proceed along the following lines:

a) Auomtatic control to keep the lower shearer at or within some predetermined range of the interface of the bottom of the seam and the floor, and the upper shearer at or within some predetermined range of the interface of the top of the seam and the roof.

b) Automated advance of the roof supports. This is likely to take the form of remote-controlled actuation of a battery of up to 10 - 12 chocks (or shields or chock-shields). Then the battery would sequentially operate automatically.

c) Automated advance of the conveyor. This is likely to be automatically sequenced with the automatic advance of the chocks.

It is possible to increase the longwall production of coal by the following methods, the feasibility of which is to be investigated:

- Automation. Although automation will not directly increase productivity, more accurate guidance of the shearer drums will result in more coal mined and less rock. Furthermore, the longwall miner will be operated more uniformly, which should improve reliability. Also, by automating very

routine operations, the miners may be motivated towards improved operation and in circumstances, such as working in thin seams, where the operators would have great difficulty in keeping up with the miner and in accurately guiding the shearers, higher productivity may be possible than with the manually controlled system.

- Improved reliability and maintainability. Mechanical reliability is a very important factor in limiting average productivity to only 425 tons per shift (see Section 8.6) Improved mechanical reliability will promote more continuous operation. This in turn will reduce stresses, which build up in the roof when stoppages occur, and will thus contribute to reductions in the frequency of mechanical breakdowns, further improving reliability.

- Improved roof control. Inability to control the roof across the face and in the head and tail entries is a frequent cause of lost time and a hazard to personnel. Newly developed roof control techniques that have not been fully utilized by coal operators will be tested and evaluated.

- Improved and faster face-to-face moves. Currently, this takes up to two months. By improved procedures and special equipment and improved panel design, it is hoped to reduce the move to an average of no more than two weeks. A Z-system panel layout can be very effective in reducing the distance to be traversed to a minimum and leaving a simple straight route for the move.

- Operation of longwalls 24 hours per day, or four 6-hour shifts, instead of the present average of about two shifts per day.

- Operation of longwalls seven days per week instead of the current five days per week. This would be achieved by operating more shifts so that any one man is not expected to work more than his 220 days per year.

- Increasing the number of longwalls. Studies in this area will determine criteria for successful longwall implementation.

In support of the above efforts, aimed at increasing the production and productivity of longwalls, there are a number of efforts to develop and demonstrate advanced longwall systems and components, including:

- Development and testing of coal/rock interface detectors. The nucleonic sensor developed by the NCB is the only one currently available. It is of the deep penetration type which measures coal thickness up to 35 cm. However, dirt bands in the roof and air gaps greater than about 1'4 inch seriously affect their accuracy. Recent studies have shown that the limitations are fundamental and that little improvement can be expected. Recent developments in pick-type sensors, which measure rock hardness directly, show considerable promise.

- In-mine demonstrations will test roof supports, advanced longwall equipment components, and shortwall mining systems. Surface test facilities will be developed with a mock longwall face and a simulated mine roof. Component testing will be conducted in these facilities.

6.3.2 Continuous Haulage Subprogram

6.3.2.1 Subprogram Goals. In many U.S. mines, longwall shearers are forced to run at reduced cutting rates because the conveyor limits the rate at which coal can be carried away from the face. Typically, the rate of cutting coal may have to be limited to 5 tons/minute whereas shearers can be capable of 18 tons/minute. Reliability of the conveyor is also a serious problem.

The goal of the Continuous Haulage Subprogram in relation to longwall mining is to develop a high-capacity armored conveyor that will be capable of transporting all the coal that can be cut by existing longwall machines.

6.3.2.2 Subprogram Description. To achieve the goal of developing a high-capacity longwall conveyor the Continuous Haulage R&D effort will be addressed to:

- Improve the design and structural strength of longwall haulage equipment to increase reliability.

- Modify or redesign conveyor dimensions to provide the capacity to transport all the coal that can be cut by existing longwall machines.

Specific objectives are to design a system having 90 percent availability while operating at surge rates as high as 18 tons/minute, an hourly rate of 1,000 tons, and a daily rate of 9,000 tons.

An articulated chain conveyor will be developed for a shortwall face. The conveyor will be similar to a longwall face conveyor.

6.3.3 Environmental Protection of Surface Areas

6.3.3.1 Subprogram Goals. The goals of this subprogram are:

- to evaluate the magnitude, complexity, and nature of environmental problems associated with underground mining, i.e., mine drainage, subsidence and disruption of hydrology, and coal mine refuse;
- to identify and evaluate current practices and recommendations for dealing with these environmental problems;
- to develop methods for predicting the occurrence and magnitude of environmental problems, and methods for eliminating or minimizing resultant damage to the environment.

Here are not related to beyond

6.3.3.2 Subprogram Descriptions. The subprogram is divided into three elements, based upon the environmental problems to be investigated: mine drainage, subsidence and hydrology, and coal mine refuse disposal.

Mine Drainage. This element is designed to evaluate the magnitude of the mine drainage problem and to develop techniques for minimizing its formation and/or discharge.

In a broad-based approach to the acid drainage problem, a wide range of possible methods will be analyzed, compared, and developed to provide various treatment alternatives for differing situations. Disciplines to be considered in this approach include geology, hydrology, chemistry, engineering, and biology.

Prevention or inhibition of acid formation by establishing and maintaining alkaline pH conditions in mine water will be investigated. Recent advances in understanding the chemisty of acid formation indicate that this may be the most efficient, economical, and environmentally safe approach to the overall problem.

A survey of the industry will be made to estimate the amount of water discharged during mining, in order to determine the magnitude of the mine drainage problem. This will be useful in determining differentials in production costs as a function of varying hydrologic conditions.

Subsidence and Hyrdology. This subprogram element is designed to evaluate the impact of underground mining on surface environments, surface structures, and surface and subsurface hydrology; and evaluate currently employed or recommended backfill methods.

The effects of underground mining on surface stability and hydrology will be determined, thereby facilitating prediction of damage potential and development of preventive techniques.

Currently proposed or recommended methods for backfilling which may be utilized for reducing subsidence and/or underground disposal of mine waste will be evaluated. Particular emphasis will be placed upon developing backfill methods which will not disrupt mining production or safety.

The relationship between subsidence and damage to man-made surface structures will be analyzed. This will promote development of improved prediction and prevention techniques.

An analysis of subsidence over the mined Pittsburgh coal bed will be performed. This will improve understanding of the relationships between mining geometry and resulting subsidence, and will provide valuable input for developing methods to minimize subsidence.

Coal Mine Refuse Disposal. This subprogram element is designed to evaluate current industry practices for disposing of mine refuse and develop underground disposal methods.

Current methods for disposing of coal mine refuse with an emphasis upon surface disposal areas will be surveyed in the Pennsylvania coal mine industry.

Various methods for disposing of waste in active underground mines will be evaluated. Development of economical, efficient methods of backfilling which will not disrupt coal production or safety will be emphasized.

Although each subprogram element emphasizes one of the three major environmental problems, they are interrelated to yield an overall environmental protection system. Mine drainage and hydrology are intimately related in that hydrology projects will minimize the quantity of mine drainage, and mine drainage projects will minimize or prevent the formation of acidity in water present in the mine.

Subsidence and mine refuse disposal projects will be integrated to produce a backfill system that will minimize subsidence and resulting damage while eliminating environmentally damaging coal refuse piles.

6.3.4 Advanced Mining Systems

6.3.4.1 Subprogram Goals. The goals of this subprogram are:

- to develop and put into practice new and innovative mining systems and equipment that will increase productivity substantially,
- to develop and put into practice new and innovative mining systems that will make possible the mining of deposits that are presently unsafe or uneconomic to mine.

6.3.4.2 Subprogram Description. The Advanced Mining Systems (AMS) Subprogram is made up of a number of program elements that involve systems and/or equipment development and culminate in field demonstrations. The subprogram will make available to the mining industry proven new mining systems that will increase coal mining productivity significantly. The AMS Subprogram is divided into two major areas: (1) Existing Concepts--concepts and systems that are currently undergoing evaluation and development; and (2) Future Concepts--research, that does not have specific advanced mining systems or equipment in mind at this time, but is specifically related to (a) anthracite, (b) bituminous and (c) lignite coal mining.

Existing Concepts. A promising concept for steeply pitching very thin, very thick or multiple seams, too deep for surface mining, is Borehole Mining. This system would remotely extract coal through a borehole by combining a high-pressure water-jet cutting system with a slurry pumping system. In this system, a borehole is drilled from the surface to the bottom of the coal seam. An equipment module

consisting of a high-pressure waterline, nozzles, and a slurry pumping system is lowered into the hole. The nozzles are rotated and slowly raised thereby eroding the coal and forming a slurry. The slurry drains to the base of the hole where it is collected by the slurry pumping system and pumped to the surface.

Low-volume and high-volume hydraulic systems will be investigated. Candidates for evaluation include cavitation jets, abrasive jets, percussive jets and continuous jets.

Methods for underground augering will be evaluated. An auger mining system capable of drilling in excess of 500 feet into the coal seam will be constructed and demonstrated. The Variable Wall Miner, based on augering principles, will be evaluated and tested if found promising.

Underground haulage for strip mining pits will be evaluated. If the evaluation is favorable a field demonstration on a cost sharing basis will be initiated.

Future Concepts. Studies will be carried out to evaluate the feasibility of underground mining of lignite and of revitalizing the underground anthracite industry. Foreign bituminous coal mining technology will be reviewed and the technology needed to meet the coal mining production requirements of the year 2000 will be forecast.

New concepts for bituminous coal mining, and for anthracite and lignite mining, if studies indicate that the mining of these types of coal is feasible, will be developed and evaluated in cooperative field demonstrations.

6.3.5 R&D Costs

For the purposes of benefit analyses, R&D funds of the related subprograms are assigned to the Automated Longwall Mining Systems as follows:

Automated Longwall Mining 100%
Continuous Haulage 25%
Environmental Protection of Surface Areas 30%
Advanced Mining Systems 40%

The R&D costs are shown in Table 6-2.

TABLE 6-2

AUTOMATED LONGWALL MINING SYSTEMS
RELATED SUBPROGRAM R&D FUNDS

	$MILLIONS BY FISCAL YEAR					
	1975	1976	1977	1978	1979	1980
AUTOMATED LONGWALL MINING	8.0	6.0	6.0	11.0	15.0	10.0
CONTINUOUS HAULAGE	1.25	2.25	2.0	2.25	2.0	2.0
ENVIRONMENTAL PROTECTION	1.5	1.8	1.5	1.5	1.5	1.8
ADVANCED MINING SYSTEMS	1.2	2.4	3.6	5.2	5.2	5.6
TOTAL	11.95	12.45	13.1	19.95	23.7	19.4

Total R&D cost, discounted at 12% to 1975 is $73.78 million.

6.4 Methodology

6.4.1 Assumptions and Limitations

In order to define the problem and maintain consistency with analyses of other subprograms, essentially the same basic assumptions and limitations that were made for the Advanced Room and Pillar Mining Systems have been applied to the Automated Longwall System:

- Developments and improvements in haulage systems, other than the face conveyor, do not affect the cost per ton of coal.
- The cost of operating the total coal mine, other than coal face operations, is dependent only on the size of the mine and is not changed significantly by improvements to mining operations at the face.
- O&M costs will be the same in the new systems as they are in the existing system. Operations costs may be slightly increased due to some added complexity of the equipment. Maintenance should be reduced as a result of the reliability improvement program and as a consequence of more continuous operation.
- New improved longwall systems will be implemented only in newly opened mines.
- It is assumed that, in the existing system, longwalls are operated two shifts per day, 5 days per week, 220 days per year (44 weeks). Part of the R&D program is to investigate the feasibility of 24 hours per day, 7 days per week operation. On the assumption that this is found to be feasible, and assuming that there are 6 hours work at the face per shift, the Automated Longwall System will be operating 4 shifts per day, 7 days per week, 44 weeks or 308 days per year. Total annual production from the Automated Longwall System is therefore from 1,232 shifts.

6.4.2 Cost Escalation and Equipment Spares Factors

Costs for longwall equipment are based on recent figures quoted by manufacturers.[1] They are therefore considered to be current 1975

prices. Operating costs are based on the new UMW pay agreement and are therefore also current for 1975.

Longwall equipment cannot, in general, be taken out of service for maintenance. Repairs must be made on the spot with spare parts brought in. Accordingly, only spare parts will be considered. A factor of 4 percent for spares has been used.

6.4.3 System Parameters - Seam Thickness

Seam thickness can impact the cost of longwall mining in two ways. Firstly, in thin seams, the productivity could be limited by the speed of progression of the shearer across the face which in turn requires that the chocks be advanced sequentially after the shearer has passed. The rate at which the chocks can be advanced is limited either by the time required for each chock to go through the advance sequence or by the time required for the chock operators to scramble through the chocks actuating the advance sequence. In practice, neither of these is a limiting factor as the capacity of the conveyor effectively limits productivity in the existing system. In the improved system, chock operation will be largely automatic, so that operators will not experience the same difficulty in keeping up with the shearer. The analysis does not assume a shearer rate which is expected to exceed the capability of the chocks to advance.

Secondly, in seams over 8 feet thick and up to about 11 feet thick, chocks are not stable enough and shields, or chock-shields are required. This adds about $1 million to the cost of the longwall

system. However, only about 8.5 percent of underground coal is mined from seams over 8 feet thick. Probably even less of the longwall production is mined from seams over 8 feet thick as longwall has advantages over room-and-pillar mining in thin seams more than in thick seams. Up to 8 feet, seam thickness does not significantly affect system cost.

In view of the above considerations, seam thickness is not used as a parameter in the longwall analysis.

6.4.4 Computation of Benefits

6.4.4.1 Cost Benefits. To compute benefits, it is necessary to estimate the difference between the cost of mining by the new systems and the cost of mining by existing systems. The technique adopted is to estimate the capital cost of equipment and the operating cost to mine the same tonnage of coal by the existing system and by the new improved system. Then, the capital cost per ton and the operating cost per ton are derived.

The estimation of costs is therefore limited to the coal face operations, including haulage on the armored conveyor to the panel belt in the head entry. O&M costs are eliminated from the analysis. The following categories of costs are developed:

a) Annual operating costs: section crew costs

b) Equipment costs:

Double drum ranging shearer
Stage loader
Power center
Electrics
Chocks or chock-shields
Armored conveyor

Model mines are analyzed to compute capital investment and operating costs and cost savings over the life of the mine, and to discount at 16.3 percent to present value at the opening of the mine.

Capital costs occur during the first and eleventh year of operation of the mine, assuming a 10-year life for equipment. The capital costs are amortized over the 10-year life at 16.3 percent discount to obtain the annualized capital cost and thence the annualized capital savings. Adding the annual operating cost savings gives the total annualized cost savings. Dividing the capital savings and the total annualized cost savings by the production of the mine gives the capital savings per ton of coal and the reduction in coal production costs per ton of coal.

The reduction in coal production cost and the annualized capital savings are applied to the Scenario 1 and Scenario 2 schedules of new mine openings (Section 6.1) from the availability dates defined in Section 6.6 at the penetration rate of Table 6-5 to the market penetration of 57 percent of 3 MTY mines defined in Section 6.6.

The reduction in coal production cost per ton of coal is accumulated through the year 2020, discounted at 12 percent to present value at 1975, to develop the benefits and present value of benefits. The capital savings per ton of coal are accumulated, discounted at 12 percent to present value at 1975, to develop the capital reduction and present value of capital reduction.

The present value of benefits and the present value of capital reduction are divided by the present value of the R&D cost to give

the benefit/cost ratio and the capital reduction per dollar of R&D cost.

6.4.4.2 Labor Savings. Man-days saved per ton of coal is the difference between the automated system and the existing system. Then total labor savings through the year 2020 per dollar of R&D costs (present value) are derived.

6.4.4.3 Analysis of Injuries. Injury data for 1970[2] have been reviewed and evaluated. Data related to longwall mining have been extracted under the following categories:

a) fall of roof;

b) fall of rib or face;

c) operating, setting up, or moving longwall machine.

The source of data is not the same as was used for the Advanced Room-and-Pillar mining injury analysis. The data which was used for that analysis does not deal with longwall mining injuries.

Data used in the analysis did present some problems. Longwall mining produced only 7,132,000 tons of coal in 1970[3]--a little over 2 percent of the total coal mined underground. Consequently, the injury data for longwall mining are not as reliable as the data for continuous mining and for roof bolting. It appears, however, that the injury rate for longwall mining is much higher than it is for continuous mining and roof bolting. In particular the man-days lost due to "permanent" injuries is approximately the same in longwall mining as it is in either continuous mining or in roof bolting (for only 2 percent of the coal mined). It is assumed that this is a freak

occurence in a small data sample. Days lost due to permanent injuries in continuous mining and roof bolting are about 20 percent of days lost due to temporary injuries. The days lost due to permanent injuries in longwall mining are therefore reduced to 20 percent of the days lost due to temporary injuries.

The "man-days lost per year" are converted to "man-days lost per million man-days." As a result of automation of the longwall shearer and of the chock-shield advance, safety will be improved. It is assumed that the rate of man-days lost per million man-days is reduced 20% as a result of improved safety.

This factor is applied to each of the automated longwall systems analyzed, at the rate of man-days per ton applicable to that system for the total coal mined by that system through the year 2020, to estimate man-days lost due to injuries. This estimate is compared with the man-days that would have been lost to mine the same amount of coal by the existing system, to dervice the man-days saved.

6.5 Current System Description

6.5.1 Current System Productivity

In 1972, the average production of longwall installations was about 194,000 tons or 850 tons per day. Assuming that the average longwall was operated two shifts per day, this is equivalent to 425 tons per shift. This is an average shift productivity over a year and includes time lost due to face-to-face transfers.

6.5.2 Current System Costs

Annual operating costs are computed in terms of costs for section crew pay, overhead, and indirect costs. The section crew

composition is based on discussion with BOM engineers. Operating costs total $1.897 per ton of coal. One-time capital costs to produce 1 million tons annually for 10 years total $15.123 million at 425 tons per shift and 440 shifts per year. Equipment is assumed to have a life of 10 years and the costs are therefore incurred in the first year and again in the eleventh year of the mine life. The mines are assumed to have a life of 20 years.

6.6 Hypothetical New System Descriptions

Development and demonstration of automated longwall mining systems will be a continuing process. Currently planned projects will develop a set of improved hardware items, subsystems and procedures and will investigate the feasibility of a variety of methods for increasing longwall production. Within the next 5-7 years this first phase of development will culminate in the assembly of a package of automated subsystems and system improvements for longwall mining.

These advances will result in evolutionary development of longwall mining systems. In general these do not represent system options for the mine operator. Advances that are proven out in demonstrations and in practice should result in substantial benefits at relatively modest cost and as such are expected to be adopted without question. However, there are alternatives that are a result of uncertainties as to the course of system development--for example the choice between chocks, shields and chock-shields depends on which of these alternatives proves feasible to automate.

Two alternative systems will be analyzed. They represent the possible extremes of system configurations in terms of cost and productivity.

6.6.1 System 1--Improved Longwall System

This is a moderate-cost system with a moderate improvement in productivity. It incorporates roof chocks with automated advance, automatically controlled shearers, improved subsystems and an armored conveyor of the currently available type with automated advance.

6.6.2 System 2--Improved Longwall System with High-Capacity Armored Conveyor

This is a high-cost system with high productivity. Automated roof shields (or chock-shields) are incorporated for roof support with automatically controlled shearers, improved subsystems and a high-capacity armored conveyor with automated advance.

6.6.3 New System Productivity

Two levels of productivity have been used. System 1 productivity is based on a 5 tons per minute cutting rate, which is approximately the rate at which longwalls are operated today. It is a rate which is limited by the capacity of the conveyor. System 2 productivity is based on an increased cutting rate of 7-1/2 tons per minute with an improved conveyor which is capable of up to 18 tons per minute. It is assumed that the longwall miner, as a consequence of R&D programs to improve reliability and improve roof control, is able to operate for much more of the shift. Minimum operating time per shift is taken to be 200 minutes, most likely is 250 minutes, and maximum is 300 minutes.

The minimum/most likely/maximum productivities for the two systems are:

System 1 (5 tons/minute): 1,000/1,250/1,500 face tons/shift

System 2 (7 1/2 tons/minute): 1,500/1,875/2,250 face tons/shift

From these productivity figures, the average productivities are calculated allowing two weeks (28 shifts--4 shifts per day, 7 days per week) for face-to-face moves and converting to clean tons:

System 1: 740/908/1,069 clean tons/shift

System 2: 1,070/1,300/1,520 clean tons/shift

6.6.4 New System Costs

Annual operating costs for the Automated Longwall System are shown in Table 6-3. The section crew composition is based on discussion with Bureau engineers. Operating costs per ton of coal are shown for Level 1 and Level 2 productivities.

Equipment costs are shown in Table 6-4 including equipment cost per annual ton of coal for System 1 and System 2. System 1 equipment includes a standard armored conveyor and roof chocks. System 2 is an upgraded system for higher productivity and includes the high-capacity armored conveyor which will be developed in the Continuous Haulage Subprogram. This system has been configured with chock-shields instead of chocks.

Chock-shields are considerably more expensive than chocks. It may be that the chock-shields will be necessary for automated sequencing--

TABLE 6-3

LONGWALL MINER - AUTOMATED SYSTEM
OPERATING COSTS

	ANNUAL COST
Shearer Operators	$ 12,254
Operator helper/chockman	11,594
2 Headmen	23,188
1 Tailman	11,594
Mechanic	12,254
Foreman	13,860
SUBTOTAL	84,744
Overhead + indirect cost 2 50%	42,372
TOTAL	$127,116
Operating cost/million tons, 308 days/year	
System 1:	
@ 740 tons/shift	$781,000
@ 908 tons/shift	636,000
@ 1069 tons/shift	541,000
System 2:	
@ 1070 tons/shift	540,000
@ 1520 tons/shift	380,000

TABLE 6-4

LONGWALL MINER-AUTOMATED SYSTEM
EQUIPMENT COSTS

	COST (1975 $)	COST WITH SPARES
SYSTEM 1		
Longwall miner	$ 470,000	$ 489,000
Chocks	2,200,000	2,288,000
Conveyor	250,000	260,000
TOTAL	$2,920,000	$3,037,000
Capital cost/million tons[1]:		
@ 740 tons/shift		3,330,000
@ 908 tons/shift		2,714,000
@ 1069 tons/shift		2,305,000
SYSTEM 2		
Longwall miner	470,000	489,000
Chock-shields	3,220,000	3,349,000
High-capacity conveyor	375,000	390,000
TOTAL	4,065,000	4,228,000
Capital cost/million tons[1]:		
@ 1070 tons/shift		3,207,000
@ 1300 tons/shift		2,639,000
@ 1520 tons/shift		2,257,000

[1]One-time capital cost to produce 1 million tons annually for 10 years (life of equipment).

this is not clear at present. The two alternatives are presented, therefore, in System 1 and System 2. Chocks and chock-shields are not necessarily options in the Automated Longwall System.

6.6.5 New Systems--Markets

a. Market Share. Longwall mining is expected to develop only in the largest mines (3 MTY) due to the very high productivity projected for a longwall installation--approximately 1 million tons per year. It is anticipated that longwall mining systems will take somewhat more than 50 percent of the market in these largest mines. This is because of its very high potential productivity and high recovery factors. Productivity per man is particularly high. Accordingly, longwall mining has been assigned 57 percent of the market in the 3.0-MTY mines.* The remainder of the market will be produced by room-and-pillar mining.

b. Market Potential. Market potential for automated longwall systems is 100% of the market share for longwall systems.

c. Market Penetration. The improved longwall systems are evolutionary developments of existing longwalls. There are no radical system differences. The principal differences are the automatically sequenced chocks and the automatically guided shearers. The differences in cost are not great--10 percent more for automatically sequenced

* This percentage is all from Eastern and Interior Provinces. Northern Great Plains and Rocky Mountain production are considered in the Western Coal Systems Subprogram.

chocks and a negligible difference on the automatically guided shearers. It is anticipated that the hardware improvements will be proven in trials and will be accepted by industry. It can not be envisaged that there will be any situation where the old systems will be preferred, nor where the old system would be applicable and the new system inapplicable. It must therefore be anticipated that the improved systems will penetrate 100 percent of the longwall market share.

d. Market Penetration Rate. As an "evolutionary" system, there is some potential for the improved longwall system to be implemented in existing longwall installations by retrofit. The market penetration of improved longwall systems into new mine openings is likely to be faster than the penetration of automated room-and-pillar mining systems as the automated longwall is less of a radical change from existing systems than are the automated room-and-pillar mining systems. It has been assumed that the market penetration reaches 100 percent of new longwall installations. To be conservative, retrofitted longwall installations will not be estimated for the benefit/cost analyses.

Penetration of new longwall installations may be expected to be slow at first and then to build up rapidly. A build-up period of five years has been adopted. The annual penetration factors during these first five years are shown in Table 6-5.

No separate analysis is made of shortwall mining. To the extent that shortwall mining develops, it is likely to be as a lower cost alternative to longwall mining, to which it is most closely related. Its

market penetration would probably be part of the longwall market penetration. Benefits would not, therefore, be additive but would be substituted for part of the longwall benefits. As such, it may be considered that potential shortwall benefits are covered in the longwall analysis.

TABLE 6-5

AUTOMATED LONGWALL MINING--PENETRATION RATE FACTORS

YEAR	FACTOR
1	.1
2	.35
3	.65
4	.9
5	1.0

6.6.6 System Availability

Based on discussions with Bureau engineers, the earliest, most likely and latest availability dates for the automated longwall mining systems are taken to be 1980, 1981, and 1982, respectively.

6.7 Analysis and Results

6.7.1 Automated Longwall Systems

Measures of benefit and value are presented in Table 6-6. The benefit/cost ratio is excellent for both scenarios and both systems, ranging from $16.02 to $16.88 for Scenario 1 and from $23.24 to $24.88 for Scenario 2 per $ R&D. Particularly interesting is the small difference in the benefit/cost ratio between System 2 and System 1. The reason is that the cost of face operations per ton of coal in System 1

TABLE 6-6

MEASURES OF BENEFIT AND VALUE
AUTOMATED LONGWALL MINING
(per $ R&D)

	SYSTEM 1	SYSTEM 2
SCENARIO 1		
Benefit/Cost Ratio	16.02	16.88
Capital Reduction	8.67	8.72
Labor Reduction (man-days)	1.29	1.47
Safety--man-days saved	.028	.031
Coal Cost Reduction/Ton	3.86	4.07
SCENARIO 2		
Benefit/Cost Ratio	23.24	24.48
Capital Reduction	12.50	12.57
Labor Reduction (man-days)	1.66	1.90
Safety--man-days saved	.034	.038
Coal Cost Reduction/Ton	3.86	4.07

is less than 25 percent of the existing system costs. Consequently, even substantial percentage improvements to System 1, such as those incorporated into System 2, result in small additional percentage improvements. This question will be treated further in sensitivity analysis. Capital reduction is about $8.70 for Scenario 1 and about $12.50 for Scenario 2 per $R&D. There is little difference between System 1 and System 2; largely because the inclusion of the more expensive chock-shields in System 2 offsets the higher productivity. Other measures of benefit and value are also very high and show similar but not large differences between System 1 and System 2.

The difference between Scenario 1 and Scenario 2 is substantial--Scenario 2 is about 45 percent higher on the benefit/cost ratio and on the capital reduction. The improvement is partly due to the higher production of large mines in Scenario 2 and partly due to the earlier implementation of large mine openings in Scenario 2.

6.7.2 Sensitivity Analyses

Parameters which cannot be accurately predicted and which will affect the measures of benefit and value include:

Market share

Market penetration and penetration rate

Productivity

Capital costs

System configuration

Market share will not be treated here. The reduction in coal cost per ton is independent of market share and all other measures of benefit

and value are proportional to market share. It should be observed however, that market share of longwall systems is not independent of room-and-pillar mining. An increase in longwall mining will be at the expense of room-and-pillar mining production.

Market penetration and market penetration rate have not been tested in the sensitivity analyses. Since automated longwall is an evolutionary development of existing systems, there would seem to be little doubt that the improved hardware and systems will rapidly achieve full market penetration.

Capital costs may change due to changes in productivity, changes in the costs of equipment, or changes in the lifetime of equipment which has been assumed to be 10 years.

Automated longwall miners, as a consequence of higher productivity and continuous operation, will cut a great deal more coal in the assumed 10-year life than existing systems cut. This does not necessarily mean that the lifetime will be less than 10 years. Automated guidance and control should result in cutting a higher percentage of coal than existing longwalls and excessive wear due to cutting rock should be reduced. Continuous operation, 24 hours per day, 7 days per week, should result in less strain on the equipment, particularly on the roof supports. It may well happen that the lifetime of automated longwall miner in the future will be longer than existing system lifetimes rather than shorter.

Systems 1 and 2 differ in two primary respects--productivity and capital cost. The difference in capital cost is almost entirely due to the inclusion of chock-shields in System 2 which has no effect on

productivity for this analysis. It will therefore be instructive to examine an alternative System 3 which has the chock-type supports of System 1 and the productivity of System 3 (with the high-capacity conveyor). Then capital costs of System 3 will be almost the same as in System 1 whereas productivity will be the same as in System 2. A comparison of System 3 with System 1 will reveal the effect of productivity on the measures of benefit and value. A comparison of System 3 with System 2 will show the effect of variation in capital costs on the measures of benefit and value. Measures of benefit and value for System 3 are compared with corresponding values for Systems 1 and 2 in Table 6-7.

TABLE 6-7

AUTOMATED LONGWALL MINING
MEASURES OF BENEFIT AND VALUE--SYSTEM 3 VS SYSTEMS 1 & 2

	SYSTEM 1	SYSTEM 3	SYSTEM 2
Coal Cost Reduction	$ 3.65	$ 4.00	$ 3.86
Scenario 1			
Benefit/Cost Ratio	16.02	17.43	16.88
Capital Reduction	8.67	9.16	8.72
Scenario 2			
Benefit/Cost Ratio	23.24	25.29	24.88
Capital Reduction	12.50	13.22	12.57

Measures of labor reduction and safety improvement are dependent on system productivity and are unchanged from System 2.

It is seen that the measures of benefit and value are little changed from System 1 or from System 2. It is concluded that the automated longwall system will be so much improved in performance over the existing system that the measures of benefit and value are relatively insensitive to changes in system costs or system performance.

SECTION 6.0 REFERENCES

1. Palowitch, 1975.

2. Moyer, 1973

3. National Coal Association, 1973a.

7.0 ADVANCED ROOM AND PILLAR MINING SYSTEMS

This section analyzes improved and automated room and pillar mining systems. These mining systems are related to five subprograms of the Bureau of Mines R&D program:

a. Automated Continuous Miner with Remote Override

b. Automated Continuous Roof Support

c. Continuous Face-to-Preparation Plant Coal Haulage Systems

d. Environmental Protection of Surface Areas

e. Advanced Mining Systems

Four representative mining systems will be analyzed. They are:

System 1: Improved continuous mining system

System 2: Automated continuous mining system

System 3: Automated continuous mining system for weak roof conditions

System 4: Improved conventional mining system.

These systems incorporate improved hardware, subsystems and procedures developed in the related subprograms. They will be described in more detail in Section 7.4.

Benefits will be generated principally from improved coal extraction and system productivity. Benefits will accrue directly from:

- increased production per man-shift,
- increased production per machine-shift,
- reduced accidents per man-shift.

Benefits also result from:

- reduced manpower requirements per million tons of coal,

resulting in reduced exposure to hazards, thereby reducing accidents.

- improved mining techniques resulting in--
  - increased recovery factors,
  - recovery of additional resources.

Since most of the benefits are related directly or indirectly to improved coal extraction, the analyses concentrate on operations directly relating to coal extraction at the face and removal of coal from the face to the next stage of haulage (panel belt).

7.1 Production Scenarios

Two scenarios are used for this analysis, identified as Scenario 1 and Scenario 2. They are defined in Table 6-1, in terms of new mine openings by year from 1975 through 2020 in three sizes of mines--0.5 MTY, 1.0 MTY, and 3.0 MTY. The total mine opening scenarios for the United States are used rather than the individual province scenarios, since no cost differences have been identified by province.

There is a distinction between the tonnages of total coal production figures, such as those in Minerals Yearbook and the scenarios of this study, and equipment productivity figures, such as those used by Katell.[1] Total coal production is reported in clean tons--that is, the output of the coal cleaning plant. Equipment productivity is usually reported in tons of coal mined at the face. Clean tons are usually about 80 percent of face tons. The distinction will be made clear wherever necessary.

7.2 State-of-the-Technology

7.2.1 Mining

Since the introduction of continuous mining machines around 1950, their share of underground mining production increased steadily from less than 2% in 1950 to 60% in 1972. Average annual productivity of continuous mining machines increased rapidly from about 54,000 tons in 1950 to 112,000 tons in 1964. At that point machine productivity reached a plateau, rising only to 117,000 tons by 1967 then declining gradually to 108,000 tons by 1970. In 1971 productivity of continuous mining machines declined about 20% to 86,000 tons, possibly due in some measure to the impact of the new health and safety standards of the 1969 Coal Mine Health and Safety Act. Then in 1972 a partial recovery was made to 96,000 tons.

Reliable time and motion data on continuous mining operations are not available. Certainly no data that could be regarded in any sense as representative of or approximating national averages for continuous mining can be obtained. However, some approximate conclusions may be drawn from the fact that the national average productivity of continuous mining machines, as determined in Section 7.6, is 300 tons per shift. Assuming that the continuous miner cuts coal at a rate of 3 to 4 tons per minute during its cutting cycle, 75 to 100 minutes are required to cut 300 tons in a shift. Approximately 400 minutes are available per shift for mining, indicating an average duty cycle of 20 to 25 percent. Since 300 to 325 minutes elapse during which the miner is not cutting coal, continuous miners are falling far short of their potential productivity.

Many problems and non-producing activities may contribute to the loss of these 300 to 325 minutes per shift, including:

a. Place change (to allow roof bolting)

b. Equipment breakdown

c. "Gasouts" due to excessive methane

d. Shuttle car delays

f. Belt delays

g. Roof bolting, roof support delays

h. Maneuvering

i. Cleanup

j. Loss of power

k. Brattice work

Although each of these problem areas may contribute to losses in productivity, the most significant factors in lost mining time are frequent changes of place to allow for roof bolting and noncontinuous shuttle car haulage. Although technical advances have been made in both areas, problems remain which reduce continuous miner productivity significantly. Basically, this is a system problem. The continuous mining system is composed of three independent operations--mining, face haulage, and roof support. Two of these operations--face haulage and roof support--are non-continuous and as a consequence the third operation--mining--is forced to be noncontinuous. Although the three operations are combined as well as equipment limitations will permit, the system is not well integrated.

7.2.2 Roof Support

The state-of-the-art in roof support has come a long way, particularly with the development of roof-bolting. The development and improvement of roof support techniques is currently an active field. Recently there have been advances in developing resin grouted steel bolts, fully pumpable polyester resin and fiberglass bolts, and in the stabilization of roof strata with epoxy resins. These techniques show great promise but are not yet operational.

Despite these advances, roof bolting remains an independent mining function that is not currently well integrated with the continuous mining system. Bolters have not yet been designed to operate close behind and in close coordination with continuous miners. In fact, continuous mining is anything but "continuous." The present state-of-the-art in roof-bolting techniques requires that the miner pull back after penetrating no more than 20 feet and then change place to mine another section while the roof bolters go in to support the roof. For an efficient operation, at least five entries must be worked simultaneously, in order to establish a balance between mining and roof bolting and avoid unnecessary delays in the operation.

The roof-bolting operation itself is a hazardous occupation. The roof bolters are manually operated and controlled and the operators are required to handle both the drill steel and the roof bolts. Operators must work in the vicinity of the bolting site and must install temporary roof jacks before any work commences. Roof bolting is currently a bottleneck operation in many mines.

7.2.3 Haulage

Coal haulage is another of the most serious bottleneck operations in underground mining. In both conventional and continuous mining, most face haulage is provided by shuttle cars, although a few face conveyor systems are in use. In good ground conditions and thick seams, shuttle cars can provide very efficient and highly productive operation. Two shuttle cars serving a continuous miner can haul as much as 1200 tons of coal in a shift. However, in practice this is rarely achieved and the average productivity of continuous miners is only about 300 tons per shift. The noncontinuous nature of shuttle car haulage is a significant factor in the limitation of continuous miner productivity, although reliable supporting data are difficult to obtain.

Face conveyors suffer from some serious limitations at present. Reliability is a very critical limitation. If one shuttle car of a two-car operation breaks down, the remaining shuttle car can serve the mining operation at about 70 percent of the two-car capacity. If any part of a conveyor system breaks down, however, the coal stops flowing. As yet, conveyor systems have not been produced that can match shuttle car productivity in thick seams with good ground conditions.

Face conveyors are also limited in length. Bridge systems are the only continuous systems currently available, but there is no practical and cost-effective bridge conveyor system available that is long enough for operation in more than three entries. As stated previously, at least five entries must be worked simultaneously to establish an efficient balance between mining and roof bolting.

7.2.4 Environmental Protection of Surface Areas

The State of the Technology in environmental protection of surface areas near underground mining operations has been dealt with in Section 6.2.4.

7.3 Related Subprograms

This section describes each of the subprograms that contribute to the development of improved and automated underground room and pillar mining.

7.3.1 Automated Continuous Miner Subprogram

7.3.1.1 Subprogram Goals. The goals of the Automated Continuous Miner Subprogram are:

a. To develop, design, fabricate, test and demonstrate an automated, full-face, continuous mining machine and remote-operator control module.

b. To integrate the automated continuous miner with an automated continuous roof support subsystem and a continuous haulage subsystem into a system capable of continuous operation and high productivity.

c. To develop improved conventional mining subsystems.

7.3.1.2 Subprogram Description. In order to achieve subprogram objectives a variety of projects are planned in the following areas:

- Interface sensor development
- Guidance and control
- Machine reliability
- Hardware development

- Systems analysis
- Management systems
- Conventional mining

Projects are planned to test and develop sensors for locating the limits of the coal seam. Other projects will develop and test sensors and subsystems for guidance and control of the automated miner itself.

An automated full-face continuous mining machine will be developed incorporating automated guidance and control, automated control of the mining head and remote operator control features. A special continuous miner will be developed for use in seams with weak roof conditions where the roof is not self-supporting more than 12 feet from the face. This machine, called the Mini-Miner, will be equipped with a hydraulic roof chock on either side of the miner that will support the roof to within 8 feet of the face.

Reliability will be addressed and analysis will also be performed on human factors, mining plans, continuous mining systems, and management techniques. Several hardware improvement projects are planned including development of a new cutting head for continuous miners.

7.3.2 Automated Continuous Roof Support Subprogram

7.3.2.1 Subprogram Goals. The goals of the Automated Continuous Roof Support Subprogram are:

a. To develop and demonstrate a remote-controlled roof bolter that can be used in conjunction with and immediately behind a remote-operator controlled continuous miner.

b. To develop and demonstrate an automated miner/support system that combines the functions of continuous mining and roof support.

c. To develop remote-controlled roof bolting techniques that will reduce exposure of the operator to the hazards of roof bolting.

d. To develop a retreat mining support system to improve resource recovery in retreat mining.

7.3.2.2 Subprogram Description. The Automated Continuous Roof Support subprogram will have efforts in the following areas:

- Development of manual/programmable, remote-controlled roof bolters.
- Development of automated continuous miner systems with built in roof-bolting capabilities.
- Studies of roof bolting practices and problems.

The remote-controlled roof bolter will install bolts up to 10 feet long behind an automated continuous miner. The automated continuous miner with remote-control override is being developed in the Continuous Miner Subprogram. The bolter operator will ride directly on his machine, which can be operated manually, or in an automatic programmed mode. The bolting operation takes place in front of the bolter operator and is remote-controlled in the sense that the operator does not handle drill steel or roof bolts directly; drilling and bolting operations are remote controlled. The bolter will operate

beside a belt conveyor in a 16-foot entry or will have pass-through capability for the coal. Thus, the miner will be able to penetrate continuously 100 feet or more. The roof bolter will follow immediately behind the miner to support the roof. The miner operator will control the miner from behind the roof bolter.

A "miner/bolter" system is being developed. This will be a continuous mining machine, which will cut a full 16-foot entry, with built-in roof bolterswhich will install, under remote operator control, a 4-bolt pattern in front of the miner operator. The bolters will install bolts up to 1 foot shorter than seam thickness. The miner/bolter will be able to penetrate a full breakthrough of about 100 feet before changing place and the operator will always remain under supported roof.

A further development of the miner/bolter program will be a "miner/supporter" system. In this system the bolt length will not be limited by seam thickness and bolts up to 8 feet long will be installed in a 4-foot seam.

The miner/bolter system has most of the functional capabilities of a continuous miner together with a remote-controlled roof bolter. The principal difference is in the capability to install long bolts. The miner/bolter is limited by seam thickness whereas the remote-controlled roof bolter is not. The miner/supporter is functionally equivalent to and can replace a continuous miner with a remote-controlled roof bolter. Neither the miner/supporter nor the remote-controlled

roof bolter are limited by seam thickness although the remote-controlled roof bolter will be able to install a longer bolt than the miner/supporter.

Studies will consider roof-bolting bottlenecks, pillaring practices and retreat mining systems. Improving retreat mining will lead to greater safety, increasing the practice of pillar recovery which in turn will lead to higher recovery factors and more uniform subsidence of the overburden which will impact surface environmental considerations.

7.3.3 Continuous Face-to-Preparation Plant Coal Haulage Systems Subprogram

7.3.3.1 Subprogram Goals. The goals of the Continuous Haulage subprogram are to develop and demonstrate haulage systems and subsystems to increase productivity significantly, including:

a. Face haulage conveyor systems.

b. Improved belt conveyor subsystems for secondary haulage.

c. Improved rubber tire and rail systems.

d. Hydraulic underground transportation systems.

e. Improved deep hoisting apparatus.

7.3.3.2 Subprogram Description. The objective of this subprogram is to develop continuous haulage systems in the sense of being "continuously available," i.e., available when required. The subprogram, therefore, covers systems such as shuttle cars and mine cars which are not considered "continuous" but can be continuously available. Haulage includes systems for haulage of men and supplies as well as coal. To meet these objectives, this subprogram considers all aspects of coal

haulage, for all mining systems, between the coal face and the preparation plant. The subprogram covers many aspects of different types of underground haulage systems including conveyor systems, shuttle cars, hydraulic systems, rail haulage, cascading systems, and deep hoisting. The haulage systems serve conventional mining, continuous mining, longwall mining and shortwall mining; however, only those projects that will contribute to continuous miner operations are considered in this section. Those haulage systems that will enhance longwall and/or shortwall mining are analyzed in Section 6.0 of this report.

Considerable emphasis is given to face haulage systems as this is where current coal haulage bottlenecks principally occur. A "continuous" haulage system here means essentially a conveyor system. As stated earlier, reliability is critical as a breakdown in a conveyor system stops the coal flowing, whereas a breakdown in one shuttle car of a two-car operation may reduce the haulage capacity no more than 30%. A design objective of the Continuous Haulage subprogram is to develop face conveyor systems that provide haulage as reliably as two shuttle cars, i.e., conveyor systems that will be non-operational no more than both shuttle cars of a two-car face haulage system would be non-operational simultaneously.

Although there can be a large payoff from continuous haulage systems that can be integrated into highly productive continous mining systems, the payoff is likely to be dissipated if bottlenecks are

simply transferred from the face mining operations elsewhere in the mine. Therefore, it is highly important to develop all mine haulage systems in parallel to provide compatible and highly productive, continuously available haulage of coal.

7.3.4 Environmental Protection of Surface Areas

A description of the Environmental Protection of Surface Areas Near Underround Mining Operations subprogram has been given in Section 6.3.3.

7.3.5 Advanced Mining Systems

A description of the Advanced Mining Systems subprogram has been given in Section 6.3.4.

7.3.6 R&D Costs

For the purposes of benefit analysis, R&D funds from the related subprograms are assigned to the Advanced Room-and-Pillar Mining Systems as follows:

Automated Continuous Miner	100%
Continuous Roof Support	100%
Continuous Haulage	75%
Environmental Protection of Surface Areas	65%
Advanced Mining Systems	40%

The R&D costs are shown in Table 7-1.

TABLE 7-1

ADVANCED ROOM-AND-PILLAR MINING SYSTEMS
RELATED SUBPROGRAM R&D FUNDS

	$ Millions by Fiscal Year					
	1975	1976	1977	1978	1979	1980
Continuous Mining	2.0	3.0	4.0	5.0	3.0	1.0
Continuous Roof Support	3.0	4.0	5.0	5.0	5.0	4.0
Continuous Haulage	3.75	6.75	6.0	6.75	6.0	6.0
Environmental Protection	3.25	3.9	3.25	3.25	3.25	3.9
Advanced Mining Systems	1.2	2.4	3.6	5.2	5.2	5.6
TOTAL	13.2	20.05	21.85	25.2	22.45	20.5

7.4 Methodology

The principal goals of the BOM R&D program for underground mining are to increase production and productivity of mining systems and to reduce the cost of coal. The principal benefits of the R&D program are computed in terms of the cost of coal. The most significant factors in the cost of coal that are affected by the R&D program are the cost of mining at the coal face and the productivity of the mining operations.

7.4.1 Assumptions and Limitations

In order to bound the problem and to formulate it in a manner such that benefits can readily be calculated, certain limitations are adopted and assumptions are made:

a. The impact of developments and improvements in haulage systems, other than face conveyors, does not affect the cost per ton of coal: such improvements are needed to permit full utilization of improvements at the coal face.

b. The costs of operating the total coal mine, other than the costs of coal face operations, is dependent only on the size of the mine and is not changed significantly by improvements to mining operations at the face.

c. O&M costs will be the same in the new systems as they are in the existing system.

d. New mining systems are implemented only in newly opened mines. This is not completely true as some of the larger existing mines will experiment with the new systems; however, this will tend to compensate for lower productivities during an initial learning period.

e. It is assumed, for the purpose of computing equipment cost per annual ton of production, that continuous mining operates 3 shifts per day, 220 days per year, or 660 shifts per year.

7.4.2 Cost Escalation and Equipment Spares Factors

The M&S equipment cost indices[2] show a rise from 1973 through the third quarter of 1974 from 344 to 414--a 20 percent increase. It is assumed that the total increase from 1973 through 1975 will be 25 percent. Costs based on 1973 prices[1] will be incremented 25 percent to 1975. Estimated costs for conveyors have been obtained more recently[3] and are escalated only 12.5 percent. Operating costs are based on the new UMW pay agreement and are therefore current for 1975.

Spare units of equipment based on Katell[1] are estimated at 20 percent to allow for units out of operation for maintenance. However, the factor for continuous miners is higher; that is, only two-thirds of the total units in use are actually in operation at any time.[4] A 50 percent spares factor is used for continuous miners.

7.4.3 System Parameters

Two aspects of mining are considered critical in the analysis: seam thickness and productivity.

7.4.3.1 Seam Thickness. Katell[5] has obtained productivity information from a limited sample of continuous mining operations. Productivities derived from this information are:

48-inch seam--312 tons/shift

72-inch seam--344 tons/shift

The difference between the Katell productivities at 48 inches and 72 inches is about 10 percent. A linear productivity relationship between these points can be assumed since the data do not justify a

more complex relation. Extrapolation below 48 inches and above 72 inches should be done with extreme caution. It may be valid over a limited range, e.g., between 3 feet and 7 feet, which, as stated above, includes 75 percent of underground production.

Assuming that productivity varies over a narrow range (20 percent) and is a linear function of seam thickness between 36-inch and 84-inch seams, the cost of mining as a function of seam thickness will also vary over a narrow range and will be nearly linear. If cost of mining coal is a linear function of seam thickness, it follows that the cost of mining coal averaged over all seam thicknesses is equal to the cost of mining coal at the average seam thickness (approximately 60 inches for the nation in 1970).[6]

The benefit/cost analyses are performed for coal mined from 5-foot seams. The result is the same as it would be if the coal were distributed and benefits calculated, as functions of seam thickness, and summed.

7.4.3.2 Existing System Productivity. Existing system productivity may be one of the most significant parameters in determining benefits. The potential benefits of automated continuous mining to a mining operation that is achieving low productivity may be much greater than the potential benefits to a highly productive mining operation.

There are no published data on coal production as a function of productivity. In fact, productivity data are usually guarded closely by the mine operators and released to very few. It is knownthat some continuous mining operations achieve as much as 1200 tons per shift

although it is not known what average productivity they maintain. At the other end of the scale, some mining operations may be operating at below 200 tons per shift although it is not known how low a productivity can turn a profit.

It has not been determined how productivity of an automated continuous mining system depends on the factors that cause the productivity of existing systems to vary as much as 6 to 1 or more. More data on these factors would be necessary and these data are not readily available at present.

Consequently, the new continuous mining systems are analyzed relative to an existing system with the current average productivity.

7.4.4 Computation of Benefits

7.4.4.1 Cost Benefits. To compute benefits, it is necessary to estimate the difference between the cost of mining by the new systems and the cost of mining by existing systems. The technique adopted is to estimate the capital cost of equipment and the operating cost to mine the same tonnage of coal by the existing system and by the new improved system. The capital cost per ton and the operating cost per ton are then derived.

The estimation of costs is limited to the coal face operations, including face haulage to the section haulage or panel belt. O&M costs are not considered in the analysis as they are assumed to be equal for both systems. The following categories of costs are developed:

a. annual operating costs: section crew costs

b. equipment costs: mining machines, roof bolters and haulage.

Each of the four systems described in Section 7.4 is compared with the existing system. Cost savings over the life of the mine, discounted at 16.3 percent to present value at the opening of the mine, are developed.

Capital costs occur during the first and eleventh year of mine operation, assuming a 10-year life for equipment. The capital costs are amortized over the 10-year life at 16.3 percent discount to obtain the annualized capital cost and thence the annualized capital savings. Adding the annual operating cost savings gives the total annualized cost saving. Dividing the capital savings and the total annualized cost savings by the production of the mine gives the capital savings per ton of coal and the reduction in coal production costs per ton of coal.

The reduction in coal production cost and the annualized capital savings are applied to the Scenario 1 and Scenario 2 schedules of new mine openings (see Section 7-1) from the availability dates of Table 7-7, at the penetration rates of Table 7-6, to the market penetration of Table 7-5.* Each system is calculated separately. The reduction in coal production costs per ton of coal is accululated for the market penetration of each system through the year 2020, and discounted at 12 percent to present value at 1975, to develop the capital reduction and the present value of capital reduction.

The present value of benefits and the present value of capital reduction are divided by the present value of the R&D cost to give the benefit/cost ratio and the capital reduction per dollar of R&D cost.

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* Tables 7-5, 7-6, and 7-7 are presented in subsection 7.6.

7.4.4.2 Labor Savings. Man-days saved per ton of coal is the difference between the new system and the existing system. Total labor savings per dollar of R&D costs (present value) are derived by applying the labor saving per ton to the coal production for each system through the year 2020.

7.4.4.3 Analysis of Injuries. FMC Corporation[7] has developed estimated national accident statistics based on an analysis of accident and injury data from 43 mines. The data cover the period January 1970 to June 1971 and are related to mine operations at the coal face and between the face and the point of first transfer, for continuous mining and conventional mining operations. Estimated man-days lost per year due to accidents in roof bolting, in extracting coal with a continuous miner, moving and loading coal with shuttle cars, maintenance and "other" functions have been derived from the report.

The "man-days lost per year" are converted to "man-days" lost per million "man-days." This factor is applied to each of the improved mining systems, at the rate of man-days per ton applicable to that system for the total coal mined by the system through the year 2020, to estimate man-days lost due to injuries at the present (1970) injury rate. This estimate is compared with the man-days that would have been lost to mine the same amount of coal with the existing system, to derive the man-days saved. The savings are 49 percent of the man-days that would have been lost and represent the biggest part of the savings in lost time due to injuries--a consequence of the increased productivity of the new systems and therefore the reduced labor per ton of coal.

Further reduction in work time lost due to injuries will result due to improved safety of advanced and automated roof bolting, continuous mining and haulage systems. Accidents from these systems account for 60% of lost time due to accidents in coal-face operations. It is estimated that, as a result of remote control of roof bolting and of continuous miners, lost time due to accidents will be reduced 50%, and where shuttle cars are replaced by conveyors, lost time will be reduced 25%. The net saving of lost time due to injuries is increased to 56%.

7.5 Current System Description

7.5.1 Current System Productivity

The estimated average productivity of a continuous miner is based on 1972 published data of coal mined, average number of men working daily, and average number of days worked.[8] Using operating parameters for underground coal mines, the average number of face crews daily is estimated and thereby the average productivity is determined to be 240 clean tons per shift or 300 face tons per shift.[1] It is also estimated that each continuous miner in active use was operated an average of 2.63 shifts per day. On the average, one continuous miner out of three is out of active use (undergoing repair, maintenance, or rebuilding) at any time.

7.5.2 Current System Costs

Annual operating costs and equipment costs are presented in Table 7-2. Both operating and equipment cost data are limited to those cost items which will be affected by the BOM R&D subprograms and the analyses in this section.

Annual operating costs are computed in terms of costs for section crew pay, overhead, and indirect costs. The section crew composition is based on information from Katell.[1] Total operating costs are $3.351 million per million tons of coal at 240 tons per shift and 220 days per year. Equipment is assumed to have a life of ten years and the costs are therefore incurred in the first year and again in the eleventh year of the mine life. One-time capital costs to produce 1 million tons annually for 10 years totals $3.333 million at 240 tons per shift and 660 shifts per year. The mine is assumed to have a life span of 20 years.

In calculating operating cost per million tons and capital cost per million tons, the clean tons per shift productivities are used. The costs are inputs to the analysis procedure in which they are applied to the production scenarios, which are also defined in clean tons of coal.

7.6 New System Descriptions

Development and demonstration of advanced room-and-pillar mining systems will be a continuing process. Currently planned projects will develop a variety of specialty mining systems, subsystems and hardware. Within the next 5-7 years this first phase of development will culminate in the assembly of a range of possible system options for room-and-pillar mining. Four systems have been selected as representative of possible system options for the early 1980's. These four systems will be referred to as System 1, System 2, System 3 and System 4 and will be described in this section.

System 1 is an evolutionary option which can be implemented at an earlier date than System 2. It relies, as far as possible, on current hardware and is basically a remote-operator-controlled system. It appears to have considerable merit in its own right as a lower cost alternative to System 2 and it is carried on as an option throughout the period of analysis (1975-2020).

System 2 is a more automated system that will have higher productivity than System 1. It is supported by the sensor development projects and by the projects to develop guidance and control systems. The Mini-Miner has all the automated features of the System 2 miner, and will be analyzed in System 3.

System 4 is basically room-and-pillar mining with continuous haulage. Whereas Systems 1, 2 and 3 are not expected to penetrate the continuous mining market 100%, the remainder of the continuous mining market is considered to be open to penetration by System 4. Conventional mining will also be penetrated by System 4, but the system will be analyzed as continuous mining.

7.6.1 System 1--Improved Continuous Mining

This is the least automated and least expensive option that is capable of penetrating one breakthrough without changing place and it takes advantage of continuous face haulage. The only equipment required that is not currently available is the continuous roof bolter. This is the same bolter that is required for the System 2 implementation and is the pacing item for System 1. The other required items of equipment are all currently available and include:

a. Remote-controlled continuous miner (Remote-control modules are available for existing continuous miners.)

b. Mobile extendable conveyor

c. Two shuttle cars

System 1 operates in the following mode. The remote-controlled continuous miner is followed immediately by the continuous roof bolter with pass-through capability. The miner operator controls the miner with his remote-control module from behind the continuous roof bolter. The coal passes through the roof bolter onto the conveyor, which carries the coal down the entry to dump into shuttle cars waiting at a crosscut. The shuttle cars carry the coal to the panel belt. The panel belt moves up when the miner reaches the limit of its trailing cable (about 500 feet).

Any existing continuous miner can be modified for remote control at a cost of about $10,000. Upon request, new continuous miners can be delivered with the remote-control module at no extra cost.

7.6.2 System 2--Automated Continuous Mining

System 2 is the most highly automated and most highly productive option to be developed in the Automated Continuous Miner Subprogram. The components of the system are:

a. Automated continuous miner with remote override. The miner is programmed to go through a complete cutting cycle atuomatically.

b. Remote-controlled roof bolter with pass-through capability.

c. Manually controlled mobile face conveyor. It is assumed that the conveyor system will be a cascading unit, a flexible conveyor train or a bendable belt, and will require one operator at each end of the conveyor.

The mode of operation of System 2 is as follows. The continuous miner is set to operate in the programmed automatic mode. Sensors control the vertical cut of the miner to keep it in the coal seam. A guidance and control system keeps the miner cutting forward in a straight line. The miner is followed immediately by the remote-controlled roof bolter and the miner operator is located behind the bolter. The miner operator has remote control of his mining machine and can override the automated operation. The continuous face conveyor follows the bolter. The coal is passed through the roof bolter onto the face conveyor, which carries it all the way back to the panel belt.

When the miner has penetrated as far as it can and must change place, the conveyor is driven out ahead and is parked temporarily alongside the panel belt. The bolter follows the conveyor and lastly the miner changes its place. The roof bolter and the conveyor now follow it into the new entry.

7.6.3 System 3--Automated Continuous Mining--Mini-Miner

In the Mini-Miner development program a continuous miner is being modified for operating under roof conditions that are not considered self-supporting for more than about twelve feet behind the face. Modifications include the attachment of a large hydraulic roof chock to each side of the miner. The roof support pad of the chock will support the roof to within 8 feet of the face. The pad is supported by two hydraulic jacks and it can be elevated to the roof. It is released and

lowered in order to advance the chock as the miner penetrates the coal face.

The Mini-Miner is equipped for remote control and for automatic operation. It will be integrated with a remote-controlled roof bolter and a continuous haulage conveyor unit. System 3 will have the total range of capabilities of System 2 and will be slightly less productive.

7.6.4 System 4--Conventional Mining

In this option, continuous face haulage is implemented without the automated continuous miner or remote controlled roof bolter. The market for the option is:

a. Continuous mining operations that do not implement System 1, System 2, or System 3.

b. Conventional mining operations, which are not amenable to automation, remote-control, or truly continuous operation.

The type of continuous haulage that would be implemented would be the face conveyor system that emerges from the Continuous Haulage Subprogram as the most promising system. It would be the same type of face conveyor system that would be implemented as part of System 2 or System 3 and would require the same capabilities of extension, capacity, and maneuverability.

7.6.5 New Systems--Productivity

Estimated productivities for Systems 1, 2, and 3 have been derived.[9] The starting point is the average productivity of the existing system--300 tons per shift at the face or 240 clean tons per shift. It is assumed that the seam thickness is 5 feet and the entry

width is 16 feet. Six-ton shuttle cars are used in a two-car operation. Average penetration before changing place is 20 feet in the existing system and 100 feet in the improved systems.

The derivation uses empirical values for the following parameters:

a. average cutting and sumping time;

b. average "busy time" at the face--maneuvering to make two passes, per 20 feet of penetration;

c. average lost time to change place;

d. average measured idle time per shuttle car load--waiting for shuttle car change-out.

The average rate of mining coal is determined to be 3.8 tons per minute from the cutting and sumping time to mine 300 tons. Then the number of place changes to mine 300 tons of coal per shift is derived and thereby the lost time due to place changes and due to busy time is determined. The number of shuttle car loads per shift is calculated and thereby the lost/idle time waiting for shuttle car change-out is determined.

The near-term goal for System 1 is to double productivity by:

a. mining 100 feet before changing place,

b. eliminating busy time by using a full face miner, and

c. eliminating idle time by using a continuous belt.

The mining rate for System 1 is not increased over the 3.8 tons per minute of the existing system. It is found that sufficient extra mining time can be obtained to achieve the goal of doubling productivity. Allowance is made for additional time needed to move the belt during place changes. Minimum productivity is assumed to be a 50

percent increase over the existing system. Maximum productivity is achieved by utilizing the full additional mining time at the same mining rate.

The goal for System 2 is to increase the mining rate to 6 tons per minute with the automated continuous miner. This is well below the mining rate of 12 to 18 tons per minute of which current continuous miners are capable. An allowance is made for moving the conveyor but, since the conveyor is designed for mobility, the allowance is less than for System 1. Minimum productivity is assumed to be that which would be achieved at 3.8 tons per minute (i.e., the same as the maximum productivity for System 1). Maximum productivity is obtained by assuming a faster place change.

System 3 (Mini-Miner) productivity is obtained by the same approach as for System 2. Minimum productivity is assumed to be the same as for System 2. Most likely productivity is obtained at the mining rate of 6 tons per minute with a time allowance made for moving the conveyor and a time allowance made for operating the roof support chocks. Maximum productivity is obtained by assuming more efficient operation of the roof chocks and a faster place change.

It is estimated that System 4 implementation of continuous face haulage would increase the productivity of a continuous miner or of a conventional mining section by 40 percent. The confidence limits on this estimate are from 20 percent to 60 percent, i.e., ±50 percent of the increase.

7.6.6 New Systems-Costs

Annual operating costs and equipment costs are presented in Table 7-3 for the new Systems 1, 2, 3, and 4. Equipment is assumed to have a life of 10 years and the costs are therefore incurred in the first year and again in the eleventh year of the mine life. The mines are assumed to have life spans of 20 years.

7.6.7 New Systems-Markets

a. Market Shares: The market shares of new mine openings that have been adopted are shown in Table 7-2. The rationale for this allocation is as follows. It is assumed that there will always be a need for conventional mining since this need depends on the nature of the coal seam; if there is rock in the seam, continuous miners are not effective. Since conventional mining requires a higher degree of skill than continuous mining, it is unlikely that conventional mining will participate in the growth projected in the scenarios. It is assumed that conventional mining will maintain approximately its present tonnage levels, or about 110 to 120 million tons per year; and that conventional mining will tend to be used more in smaller mines where advanced systems are not implemented to the extent that they are in larger mines.

TABLE 7-2

COAL MINING SYSTEMS - MARKET SHARES

Mine Size	.5 MTY	1.0 MTY	3.0 MTY
Longwall Mining			57%
Continuous Mining	70%	85%	37%
Conventional Mining	30%	15%	6%

TABLE 7-3

ROOM-AND-PILLAR MINING--CAPITAL AND OPERATING COSTS (SECTION)

	EXISTING SYSTEM		SYSTEM 1		SYSTEM 2		SYSTEM 3		SYSTEM 4	
CAPITAL COSTS	Number of Units	Cost	Number of Units	Cost	Number of Units	Cost	Number of Units	Cost	Number of Units	Cost
Continuous Miner	1	$200,000	1	$212,500	1	$265,000	1	$310,000	1	$200,000
Shuttle Cars	2	135,000	2	135,000						
Conveyors			1	225,000	1	450,000	1	450,000	1	450,000
Roof Bolter	1	55,000	1	125,000	1	125,000	1	125,000	1	55,000
Spares		138,000		205,000		250,000		270,000		201,000
TOTAL		$528,000		$902,000		$1,090,000		$1,155,000		$906,000
	Tons/Shift		Tons/Shift		Tons/Shift		Tons/Shift		Tons/Shift	
Capital Cost/Annual Ton	240	$ 3,333	360	$ 3,796	515	$ 3,207	515	$ 3,398	290	$ 4,734
			480	2,887	750	2,202	620	2,823	335	4,098
			515	2,654	810	2,039	675	2,593	380	3,612
OPERATING COSTS	Number of Men	Annual Costs	Number of Men	Annual Costs	Number of Men	Annual Costs	Number of Men	Annual Costs	Number of Men	Annual Costs
Miner/Miner Bolter Operator	2	$ 23,848	2	$ 24,508	2	$ 24,508	2	$ 24,508	2	$ 23,848
Shuttle Car Operators	2	21,978	2	21,978						
Conveyor Operators			2	21,010	2	21,010	2	21,010	2	21,010
Roof Bolter Operators	2	24,508							2	24,508
Braticemen, Utility Men Mechanics	3	33,748	3	33,748	3	33,748	3	33,748	3	33,748
Foreman	1	13,860	1	13,860	1	13,860	1	13,860	1	13,860
Overhead, Indirect @50%		58,971		57,552		46,563		46,563		58,487
TOTAL		$176,913		$172,656		$139,689		$139,689		$175,461
	Tons/Shift		Tons/Shift		Tons/Shift		Tons/Shift		Tons/Shift	
Operating Cost/Ton	240	$ 3,351	360	$ 2,180	515	$ 1,233	515	$ 1,233	290	$ 2,751
			480	1,635	750	0,846	620	1,025	335	2,380
	515	1,524	515	1,524	810	0,785	675	0,940	380	2,099
Annualized Cost/Ton	240		480		750		620		335	
Operating		$ 3,351		$ 1,635		$ 0,846		$ 1,025		$ 2,380
Capital		,697		,596		0,461		0,591		0,857
TOTAL		$ 4,048		$ 2,231		1,307		$ 1,616		$. 3,237

Longwall mining is expected to develop only in the largest mines (3 MTY) due to the very high productivity projected for a longwall installation--approximately 1 million tons per year. It is anticipated that longwall mining systems will take somewhat more than 50 percent of the market in these largest mines. This is because of the very high potential productivity and high recovery factors. Productivity per man is particularly high. Accordingly, longwall mining has been assigned 57 percent of the market in the 3.0 MTY mines.* The remainder of the market will be produced by continuous mining.

b. Market Potential: Approximately 75 percent of continuous mining is performed in seams with competent roof that is suitable for Systems 1 and 2. The other 25 percent of continuous mining is performed under weak roof conditions for which the Mini-Miner is designed. These weak seams can be worked by System 3 or by System 4. All of conventional mining is a potential market for System 4.

The market potentials for each system are shown in Table 7-4. Potentials overlap so that they do not add up to 100 percent.

c. Market Penetration: Automated continuous mining systems (Systems 1 and 2) are expected to penetrate their market potential to about 50 percent. Thus it is assumed that Systems 1 and 2 will share 26.25 percent of the .5 MTY mines, 31.8 percent of the 1.0 MTY mines and 13.85 percent of the 3.0 MTY mines.

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* This percentage is all from Eastern and Interior Provinces. Northern Great Plains and Rocky Mountain production are considered in the Western Coal Systems Subprogram.

TABLE 7-4

NEW MINING SYSTEMS--MARKET POTENTIAL

CONTINUOUS MINING AND CONVENTIONAL MINING
(% OF MARKET BY MINE SIZE)

MINE SIZE	.5 MTY	1.0 MTY	3.0 MTY
System 1	52.5	63.75	27.75
System 2	52.5	63.75	27.75
System 3	17.5	21.25	9.25
System 4	100	100	43

The high productivity of System 2--about 500,000 tons of coal annually--is likely to restrict its implementation to mines of 1 MTY or larger. System 1 is assumed to mine the full 26.5 percent of the market in the .5-MTY class of mines. Systems 1 and 2 are assumed to share equally the penetration of the 1-MTY mines, or 15.9 percent of the 1-MTY market each. The greater productivity of System 2 is likely to be more appropriate to the largest class of mines where 4.7 percent is assigned to System 1 and 9.25 percent to System 2.

System 3 (Mini-Miner) will have high productivity--about 400,000 tons per year--and is unlikely to be operated in small mines of the .5 MTY class. In the 1-MTY and the 3-MTY mines, however, the Mini-Miner system is expected to achieve a high penetration of about 75 percent of the 25 percent of mines with weak roof conditions--that is, 15.9 percent of the 1-MTY mines and 6.9 percent of the 3 MTY mines.

Of the remainder of the continuous miner market, not penetrated by System 1, System 2, or System 3, System 4 may be expected to penetrate

about 50 percent. System 4 is also expected to penetrate about 50 percent of the conventional mining market which is not suited to any components of the advanced mining systems other than continuous haulage. Thus, System 4 is expected to penetrate 36.9 percent of the .5-MTY mines, 26.1 percent of the 1-MTY mines, and 11.1 percent of the 3-MTY mines. The penetration of continuous mining systems, as a percentage of the total new mine openings by mine size, is shown in Table 7-5.

TABLE 7-5

NEW MINING SYSTEMS - MARKET PENETRATION

CONTINUOUS MINING AND CONVENTIONAL MINING
(& OF TOTAL MARKET BY MINE SIZE)

MINE SIZE	.5 MTY	1.0 MTY	3.0 MTY
System 1	26.25	15.9	4.6
System 2		15.9	9.25
System 3		15.9	6.9
System 4			
--Continuous Mining	21.9	18.6	8.1
--Conventional Mining	15.0	7.5	3.0
System 4 Total	36.9	26.1	11.1

The application of System 4 to continuous mining and to conventional mining will be handled as if it were all continuous mining.

d. Market Penetration Rate: As "innovative" systems, the improved and automated continuous mining systems will penetrate the market primarily by implementation in new mines. Full penetration of new mine openings will not be achieved immediately, but will phase-in over a

period of time. Penetration may be expected to be slow at first, subsequently to build up rapidly until full penetration is nearly achieved, and then to approach full penetration asymptotically. This may be represented by the characteristic S-curve, typical of such market penetrations. A build-up over a period of 10 years has been adopted. The annual factor multipliers of the final penetration percentage, are shown in Table 7-6.

TABLE 7-6

PENETRATION RATE FACTORS

YEAR	1	2	3	4	5	6	7	8	9	10
FACTOR	.03	.1	.2	.35	.5	.65	.8	.9	.97	1.0

7.6.8 System Availability

Based on discussions with BOM engineers, earliest, most likely, and latest availability dates have been developed and are shown in Table 7-7.

7.7 Analysis and Results

7.7.1 Room and Pillar Mining Systems

This section presents results of cost analyses of the individual systems and develops measures of benefit and value for the Advanced Room-and-Pillar mining systems.

Measures of benefit and value per dollar present value of R&D costs are shown in Table 7-8.

TABLE 7-7

IMPROVED ROOM & PILLAR MINING SYSTEMS
(AVAILABILITY DATES)

	EARLIEST	MOST LIKELY	LATEST
EQUIPMENTS			
Continuous Bolter	1978	1979	1980
Continuous Face Conveyor	1979	1980	1981
Automated Continuous Miner	1979	1980	1981
Mini-Miner	1978	1979	1980
SYSTEMS			
System 1	1979	1980	1981
System 2	1978	1979	1980
System 3	1979	1980	1981
System 4	1980	1981	1982

The benefit/cost ratio is 9.69 for Scenario 1 and 12.93 for Scenario 2. Systems 1, 2, 3 and 4 each show good measures of benefit and value.

System 4 compares favorably with Systems 1, 2, and 3 on measures of labor reduction and safety. However, coal cost reduction is only $1.07 per ton and capital requirements are reduced substantially less.

TABLE 7-8

MEASURES OF BENEFIT AND VALUE
ADVANCED ROOM AND PILLAR MINING SYSTEMS

(per $ R&D)

	System 1	System 2	System 3	System 4	Total
SCENARIO 1					
Benefit/Cost Ratio	2.71	2.45	2.19	2.34	9.69
Capital Reduction	.29	.29	.21	.10	.89
Labor Reduction (man-days)	.80	.81	.70	.89	3.20
Safety--man-days saved	.00088	.00087	.00076	.00069	.00320
Coal Cost Reduction/Ton	$1.93	$2.86	$2.61	$1.01	$1.87
SCENARIO 2					
Benefit/Cost Ratio	3.42	3.39	3.09	3.03	12.93
Capital Reduction	.39	.41	.30	.17	1.27
Labor Reduction (man-days)	.71	.83	.72	.69	2.95
Safety--man-days saved	.00102	.00108	.00095	.00081	.00386
Coal Cost Reduction/Ton	$1.93	$2.86	$2.61	$1.07	$1.87

7.7.2 Sensitivity Analyses

Parameters which cannot be accurately predicted and which will affect the measures of benefit and value include:

Market penetration

Market penetration rate

Productivity

Capital costs

Market penetration will not be treated here as its effect on the measures of benefits and value is straightforward. The reduction in coal cost per ton for each system is independent of market penetration. All other measures are proportional to market penetration for each system. The benefit/cost ratio, capital reduction, labor reduction and improvement in safety (reduction in man-days lost due to injuries) are additive for the total system--the combination of Systems 1,2, 3, and 4. The reduction in coal cost per ton is, however, a weighted average for the total system. Also an increase in market penetration for one system may result in a reduction in market penetration for another system.

Market penetration rate has not been tested in the sensitivity analysis. It may have a significant effect on measures of benefit and value.

Changes in capital costs can occur as a result of changes in productivity, changes in the costs of equipment, or changes in the lifetime of equipment, which has been assumed to be 10 years.

Automated continuous miners, as a consequence of greatly improved productivity over the existing system, will cut a great deal more coal in the assumed 10 year life. This does not necessarily indicate that the lifetime will be less than 10 years as continuous miners regularly undergo complete overhauls.

The four mining systems have given good indications of the effect of productivity on the measures of benefit and value. However, each system has different costs associated with it and useful indications of the effect of variation in capital costs can be obtained by varying the capital cost in each system, keeping system productivity and operating costs fixed.

Capital costs are increased 25% in a "Variation Case" for Systems 1, 2, 3, and 4 to test the effect of changes in capital cost on each system. The estimated measures of benefit and value for each system in the variation case are shown in Table 7-9. Base case values are shown in parentheses underneath each value to facilitate comparison. Measures of labor reduction and safety improvement are dependent on system productivity only and are unchanged from the base case.

The results show clearly how the systems with high productivity are least vulnerable to changes in capital costs. Benefit/Cost ratio and the reduction in cost of coal per ton (which is proportional to Benefit/Cost ratio) are relatively little affected (less than 10% reduction) for System 1, 2 and 3, whereas System 4 is reduced about 20%. Capital requirements are more seriously affected. Capital

Reduction for System 2 (with the highest productivity) declines about 20%. However, Capital Reduction for System 4 (with the lowest productivity) changes from a modest reduction to a substantial increase.

TABLE 7-9

MEASURES OF BENEFIT AND VALUE (Variation Case)
ADVANCED ROOM AND PILLAR MINING
(per $R&D)

	System 1	System 2	System 3	System 4	Total
SCENARIO 1					
Benefit/Cost Ratio	2.52 (2.71)*	2.37 (2.45)	2.08 (2.19)	1.91 (2.34)	8.88 (9.69)
Capital Reduction	.17 (.29)	.23 (.29)	.13 (.21)	-.23 (.10)	.30 (.89)
Coal Cost Reduction/Ton	1.79 (1.93)	2.75 (2.86)	2.47 (2.61)	.80 (1.01)	1.69 (1.87)
SCENARIO 2					
Benefit/Cost Ratio	3.20 (3.42)	3.27 (3.39)	2.94 (3.09)	2.51 (3.03)	11.92 (12.93)
Capital Reduction	.25 (.39)	.33 (.41)	.18 (.30)	-.25 (.17)	.51 (1.27)
Coal Cost Reduction/Ton	1.79 (1.93)	2.75 (2.86)	2.47 (2.61)	.80 (1.01)	1.69 (1.87)

*Base case values in parentheses.

SECTION 7.0 REFERENCES

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8.0 THICK AND MULTIPLE SEAM MINING

Current coal mining methods do not lend themselves to the safe or high recovery of western thick-seam deposits (20 feet or higher) or multiple seam deposits. These methods yield relatively high mining costs when an attempt is made to mine the thicker seams at the upper practical height limits for these methods. More often, much of the coal bed is sacrificed by the mining at lower, but more profitable, level. In addition, these mining methods become more hazardous as seam thickness increases above 7 feet because the ground control problem inherent in these methods increases rapidly with increases in seam thickness.

Advanced mining techniques, tailored to western coal seams, must be developed to take advantage of the great seam thickness and/or close proximity of multiple seam beds to minimize costs, maximize recovery (65 percent or higher), and preclude environmental damage and sterilization of coal reserves. The hypothetical new Thick and Multiple Seam Mining System that is to result from the Coal Extraction R&D Program is designed to meet these goals upon the assumption of successful completion of the supporting subprograms.

8.1 Production Scenarios

The scenarios used for this analysis assume that the Thick and Mulitple Seam Mining System will be used in all new underground mines in the Great Plains and Rocky Mountain provinces. As a result, the two scenarios used and that are presented in Tables 8-1 and 8-2 are subsets of the two basic scenarios described in Section 3.0. Tables 8-3 and 8-4 present the total production by the new system by year and

TABLE 8-1

PRODUCTION SCENARIO 1
FOR
THICK AND MULTIPLE SEAM MINING SYSTEMS

(NEW UNDERGROUND MINES IN THE GREAT PLAINS AND ROCKY MOUNTAIN PROVINCES USING THE NEW SYSTEM)

YEAR	NEW MINES (MTY)	YEAR	NEW MINES (MTY)
1980	0	2000	2.2
1981	0	2001	2.2
1982	0	2002	2.2
1983	0	2003	2.2
1984	0.12	2004	2.2
1985	0.24	2005	2.3
1986	0.36	2006	2.1
1987	0.48	2007	2.1
1988	0.60	2008	2.1
1989	0.72	2009	2.1
1990	0.84	2010	2.2
1991	0.96	2011	1.9
1992	1.08	2012	1.9
1993	1.4	2013	1.9
1994	1.4	2014	1.9
1995	2.1	2015	2.9
1996	2.1	2016	2.7
1997	2.1	2017	2.7
1998	2.1	2018	2.7
1999	2.1	2019	2.7
		2020	2.8

TABLE 8-2

PRODUCTION SCENARIO 2
FOR
THICK AND MULTIPLE SEAM MINING SYSTEM

YEAR	NEW MINES (MTY)	YEAR	NEW MINES (MTY)
1980	0	2000	1.9
1981	0	2001	2.3
1982	0	2002	2.2
1983	0	2003	2.1
1984	0.18	2004	2.1
1985	0.38	2005	2.0
1986	0.54	2006	1.9
1987	0.72	2007	1.9
1988	0.90	2008	1.9
1989	1.08	2009	1.9
1990	1.12	2010	1.9
1991	0.96	2011	1.5
1992	0.99	2012	1.2
1993	1.2	2013	1.5
1994	1.0	2014	1.2
1995	2.0	2015	2.2
1996	1.5	2016	1.8
1997	2.0	2017	2.4
1998	2.2	2018	2.4
1999	2.2	2019	2.5
		2020	2.0

TABLE 8-3

TOTAL PRODUCTION
FOR THICK AND MULTIPLE SEAM MINING SYSTEM

PRODUCTION SCENARIO 1

YEAR	PRODUCTION (MTY)	YEAR	PRODUCTION (MTY)
1980	0	2000	20.90
1981	0	2001	23.10
1982	0	2002	25.30
1983	0	2003	27.50
1984	0.12	2004	29.58
1985	0.36	2005	31.64
1986	0.72	2006	33.38
1987	1.20	2007	35.00
1988	1.80	2008	36.50
1989	2.52	2009	37.88
1990	3.36	2010	39.24
1991	4.32	2011	40.18
1992	5.40	2012	41.00
1993	6.80	2013	41.50
1994	8.20	2014	42.00
1995	10.30	2015	42.80
1996	12.40	2016	43.40
1997	14.50	2017	44.00
1998	16.60	2018	44.60
1999	18.70	2019	45.20
		2020	45.80
		TOTAL	877.80

TABLE 8-4

TOTAL PRODUCTION
FOR THICK AND MULTIPLE SEAM MINING SYSTEM

PRODUCTION SCENARIO 2

YEAR	PRODUCTION (MTY)	YEAR	PRODUCTION (MTY)
1980	0	2000	20.87
1981	0	2001	23.17
1982	0	2002	25.37
1983	0	2003	27.47
1984	0.18	2004	29.39
1985	0.56	2005	31.01
1986	1.10	2006	32.37
1987	1.82	2007	33.55
1988	2.72	2008	34.55
1989	3.80	2009	35.37
1990	4.92	2010	36.15
1991	5.88	2011	36.39
1992	6.87	2012	36.90
1993	8.07	2013	37.20
1994	9.07	2014	37.40
1995	11.07	2015	37.60
1996	12.57	2016	37.90
1997	14.57	2017	38.30
1998	16.77	2018	38.50
1999	18.97	2019	38.80
		2020	38.90
		TOTAL	826.10

the total production over the 40 year benefit evaluation period for scenarios 1 and 2.

8.2 State-of-the-Technology

In 1969, experiments were performed using two-level advancing longwall caving methods in thick seams at the Velenje Lignite Mine in Yugoslavia.[1] This involved mining a coal seam 50 feet thick with two longwall miners at two levels. The first longwall miner worked on a face whose bottom was 22 feet from the bottom of the coal seam. It blasted 8 feet into the ceiling and 6 feet into the floor behind it as it advanced, leaving behind stored loose coal. The second longwall miner, working 100 feet behind the first longwall miner on the floor of the seam, mined the coal at the face and blasted the coal in the ceiling behind it. This procedure caused all of the loose coal generated by the first longwall miner to flow down to the rear shields of the second longwall miner, where it was allowed to flow in through openings in the rear shields and loaded onto the face conveyor.

Several problems were encountered:

a) The manpower needed to maintain the galleries, air intake roads, and entrance ways, due to the high pressures in the gob areas, was almost equal to the manpower working at the two faces.

b) Haulage was not adequate to handle the volume of coal that could be brought in through the rear shields.

c) For safety reasons, permissive explosives were used and, as a result, it was necessary to blast two or three times in order to cave the ceiling separating the two working faces.

The major problems associated with the proposed methods for mining thick seams and multiple seams include the following items:

(a) Behavior of Coal When It is Caved: Techniques must be studied for caving part or all of a coal seam in order to produce coal that can be collected easily. The size and consistency of the broken coal will depend on several factors not only associated with the coal seam itself, but also associated with the techniques by which the caving is induced. In the caving methods, it would be desirable for the broken coal to be of such size and consistency that it would pour onto the conveyor.

(b) Stabilization of Entryways: Techniques have to be investigated for stabilizing those entryways and airways that will be subjected to great pressures due to caving. This problem is much more serious with advancing longwall mining than with retreating longwall mining.

(c) Spontaneous Combustion: Studies will be needed to determine the susceptibility of the broken coal to spontaneous combustion, especially if it is stored in place for any length of time.

(d) Methane Emission: The emission of methane from broken and stored coal will be high unless degasification of the coal seam has already taken place. The use of explosives in blasting the coal will be governed by the degree of methane released by the gob and broken coal.

(e) Cleaning Coal: The mining of coal by means of the caving method, esepcially in multiple seams, will result in a mixture of coal and rock. A low-cost method must be developed that will separate coal from rock and other material when the percentage of coal in the mixture is as low as 40 to 50 percent.

(f) Recovery of Coal from Sterilized Thick Seams: Examination of techniques for recovering coal from sterilized thick-seam coal beds should be initiated. Thick-seam mining with conventional methods results in a low recovery rate and, as a result, much of the valuable low-sulfur underground coal is left in the ground in a currently unminable state.

8.3 Related Subprograms

The Thick and Multiple Seam Mining System is based upon the results of the following subprograms:

a) Mining Systems for Western Coal

b) Environmental Protection for Underground Mines

c) Advanced Mining Systems

The Mining Systems for Western Coal Subprogram is described in Subsection 8.3.2. The Environmental Protection Subprogram and the Advanced Mining Systems Subprogram are described in Section 6.3.

8.3.1 System Funding

Funding for the Thick and Multiple Seam Mining System is given in Table 8-5. All of the Western Coal subprogram will be used to support this new system along with 5 percent of both the Environmental Protection and the Advanced Mining System subprograms.

8.3.2 Mining Systems for Western Coal

The primary objective of the Western Coal Subprogram is to develop special mining systems for mining thick-seam and multiple-seam coal beds that satisfy the following criteria:

a) Recovery rate must be greater than 65 percent.

TABLE 8-5

THICK AND MULTIPLE SEAM MINING SYSTEM FUNDING

FISCAL YEAR	R&D COSTS ($MILLION)			R&D COSTS ($MILLION) DISCOUNTED @12% TO 1975			
	WESTERN COAL	ENVIRON. PROTECTION	AMS	WESTERN COAL	ENVIRON. PROTECTION	AMS	TOTAL
1975	1.00	0.25	0.15	1.00	0.25	0.15	1.40
1976	3.00	0.30	0.30	2.68	0.27	0.27	3.22
1977	4.00	0.25	0.45	3.19	0.20	0.36	3.75
1978	5.00	0.25	0.65	3.56	0.18	0.46	4.20
1979	5.00	0.25	0.65	3.18	0.16	0.41	3.75
1980	6.00	0.30	0.70	3.40	0.17	0.40	3.97
TOTAL	24.00	1.60	2.90	17.01	1.23	2.05	20.29

b) Production costs must be equal to or less than production costs associated with current conventional systems.

c) Safety must be equal to or better than that associated with conventional systems.

These special mining systems consist of two major activities:

a) Evaluation of thick-seam and multiple-seam mining methods through a one-step procedure. This procedure involves solicitation of proposals from mining companies for a mining concept and demonstration at a site.

b) Evaluation of thick-seam and multiple-seam mining methods through a two-step procedure. The first step involves soliciting and evaluating new mining concepts. The second involves a demonstration of those concepts that appear to be the best.

This subprogram has been carefully designed to consider and evaluate a maximum number of possible techniques, thereby increasing the probability of determining and developing optimum methods. Thus, this subprogram is designed around the two main procurement efforts listed above, each of which may lead to one or more demonstrations. Taken together, these efforts provide for consideration of mining concepts from a variety of sources.

In addition to the two major procurement efforts that are described in the following subsections, the subprogram includes general support projects; such as studies of spontaneous combustion, subsidence, and coal availability.

BOM is currently considering seven different methods for mining thick- and multiple-coal seams:

a) Longwall Slicing of Thick Seams

b) Longwall Caving of Thick Seams

c) Undercut Caving of Multiple Seams

d) Hydraulic Mining of Thick Seams

e) Shallow Seam Mini-Mine Using Boreshafts

f) Bottom Slice Caving with Pillar Extraction

g) Caving into Retreating Yielding-Arch Entries

These methods are described in the following subsections.

8.3.2.1 Longwall Slicing of Thick Seams. This method involves dividing the seam into multiple faces, one on top of the other, separated by a floor of a given thickness. Each face would be worked by a longwall miner, either in ascending or descending order. The advantage of the descending method is that the highest face is advanced first so that each longwall miner will have a solid base needed by the large hydraulic face support units. When the descending method is used, it is important to fully extract the coal; any remaining pillars or stumps may cause high stress concentrations on the lower panels.

The ascending method generally requires stowing, and the upper layers of coal often become considerably fissured. The descending method is the most common in that it can be applied to a wider range of geologic and mining conditions.

A possible problem may be in the design of the development entries. As several units may be working simultaneously, it becomes necessary that the openings remain stable yet do not cause high stress concentrations on the adjacent workings.

The slicing method has the potential for mining thick coal seams with a higher recovery and productivity. Another advantage is that the

coal will be more rapidly extracted, thus reducing the possibility of spontaneous combustion of the coal.

There are many possible variations of the slicing method; some employ a conventional longwall unit, while others combine a series of longwall units in conjunction with blasting and loading coal from gob-side conveyors. An example of the combination longwall and blasting method is the so-called Velenje mining method used in Yugoslavia (see Section 8.2). In other methods, some seams are worked by more than two units at a time; some seams are worked leaving a coal layer between the units, while other techniques employ hydraulic stowing and/or the use of mats.

8.3.2.2 Longwall Caving of Thick Seams. This method involves the use of a longwall miner working on a 7- to 8-foot-high face at the bottom of the thick seam. As it moves forward, the ceiling behind it is caved using mechanical means or explosives, filling the area behind the rear shields with broken coal. As a result, production not only comes from the longwall face, but broken coal is brought in through the rear shields onto a gob-side conveyor.

A similar system is used in Europe; however, this system requires a high amount of labor. The system required in this country should be automated for higher productivity and reduced labor costs.

In the automated system, a screw conveyor is located beneath hydraulic fingers (bananas). These fingers serve two purposes: (1) they act as a grizzly to protect the screw conveyor and break large pieces, and (2) they can be "pumped" to accelerate caving of

the coal. However, a problem occurs due to the fact that the screw conveyor must be flexible enough so that the support units can be advanced; smaller steps will probably have to be used (i.e., 6 inches). Similar systems used in Europe have claimed a high recovery (80 to 85 percent), with the gob-side conveyor accounting for much of the production. In order for the automated system to work, the coal seam should be thick (>20 feet), the coal should not be too hard, and the pitch of the seam should not be greater than 15° to 20°.

8.3.2.3 Undercut Caving of Multiple Seams. This method entails driving main entries in the bottom rock (shale) from a canyon wall or from inclined or vertical shafts. Parallel subentries, which become the basic openings for obtaining ore in the undercut caving system, are driven from a main entry. The end of each subentry becomes a drawhole for the caved rock and coal. Crosscuts for ventilation can be driven between the parallel subentries at 100-foot intervals connecting the ends of subentries. These crosscuts will be destroyed as mining progresses.

If the overlying coal seam or the lower part of it is mined out first by conventional coal mining, such as by longwall, a vertical raise is bored or driven up to within 5 feet of the floor of the seam. The raise is then funneled by conventional drilling and blasting to form a circular open connection to the caved material in the floor above.

If the coal seam immediately above the bottom rock development has not been mined out and is not to be mined previous to intersecting it with drawholes, then it may be necessary to use some other method of undercutting. One such approach would be to drive fan drifts parallel

to the subentries on center-lines in the same manner as for the circular raises. Holes in the fans will be 3 to 5 feet apart. Undercutting proceeds as the rows of fanned holes are blasted.

Undercut caving of multiple coal seams is worthy of consideration because it eliminates development of each seam and also recovers coal from seams that are under 4 feet thick. It should approach 100 percent recovery of all coal in vertical columns several hundred feet thick.

8.3.2.4 Shallow Seam Mini-Mine Using Boreshafts. This system would be utilized under the following conditions: (1) a thick coal seam (20 feet), and (2) shallow overburden (200-300 feet).

Each mining section would be self-contained, having its own man shaft, ventilation shafts, and production shafts. This could result in less travel time, simplified ventilation, and decreased wait time, thereby increasing production. The proximity of the man shaft could result in increased safety, in that the miners would be able to escape quickly in the event of an emergency.

An initial slope entry would be driven for supply and ventilation purposes. The coal could be mined using familiar techniques and equipment (i.e., continuous miners, longwall, or any combination).

This method has potential in several areas if proven technically and economically feasible. One advantage may be the reduction in travel time and downtime, as there will be man and production shafts for each production section. Another advantage may be increased safety. In the event of an emergency, the escapeway to the surface is near the production crew. Communication with the surface will be improved

and ventilation may be simplified. Supplies can be brought down the shafts, thereby reducing downtime for repairs.

As each unit is self-contained, there is little reason to maintain a large area of permanent openings. This should result in a reduction of supplies and the requirement to maintain a large network of development entries. However, it may prove necessary to keep some entries open to bring in large-sized supplies, provide an emergency escapeway for the miners, and provide additional ventilation.

8.3.2.5 Bottom Slice Caving with Pillar Extraction. The concept behind this method is to employ familiar underground coal mining methods and equipment in thick coal seams.

A typical roon-and-pillar sequence could be developed using possible continuous miners or heading machines and augers for developing the breakthroughs. The new concept to be employed in this method is the use of underground augers to "pull" the pillars and also to mine the caved roof coal. This system has the potential of mining coal pillars with a high recovery and increased safety.

The optimum situation would be if the auger has the capability of extracting not only the small pillars, but also the barrier pillars. An example of a pillaring sequence could be the following:

a) extract coal from barrier pillar, working toward the panel main entries; and

b) move auger to the next entry and use the same method for the room pillars, leaving the required bleeders.

8.3.2.6 Cavings into Retreating Yielding-Arch Entries. The purpose of this system is to mine a thick coal seam using a modified

room-and-pillar method for development with yielding-arch supports for roof control.

One possible method would be to use a heading machine to drive the room entries with an underground auger for developing the crosscuts. The crosscuts, which are used only for ventilation, would be driven at the maximum spacing permitted. The yielding steel arches are placed as the development progresses.

As in room-and-pillar operations, the majority of the tonnage comes from pillar recovery. During the retreat, sections of the steel arches would be removed; these arches could then be fan-drilled and shot. A loading machine would then load the blasted coal into shuttle cars, an extensible belt, or another means of transport.

There are many variations of pillar recovery that may be possible. If the coal is naturally caving or can be pre-conditioned, the need for drilling and shooting may be eliminated. If the coal will cave, it may be possible to have several loading machines working on the retreat.

The advantage of this system is that familiar techniques and equipment can be utilized, and the miners are always under steel supports, so that development and retreat can take place quickly. This may reduce ground control problems and reduce the possibility of spontaneous ignition of the coal.

One disadvantage is that there is a high use of steel arches; however, these are recoverable. Another disadvantage may be that the

recovery percentage during the retreat sequence may not be high.

8.4 Methodology

The development of special techniques for mining thick-seam and multiple-seam coal beds will be judged on whether the cost per ton is competitive with conventional techniques. In addition, these new techniques must take advantage of thick seams and multiple seams by having a high recovery rate (65 percent or higher) while maintaining or increasing safety over conventional methods.

A hypothetical thick and multiple seam mining method, described in Section 8.6, will be compared with a current system, described in Section 8.5, and this comparison will serve as a basis for the benefit/cost study in that area of the R&D program associated with thick and mulitple seam mining.

In the analysis of this new system, the major parameter is seam thickness, which will be used to scale costs and savings.

All costs and savings will be based on the existing system described in Section 8.5. These costs and savings will be scaled as a function of increased production and changes in equipment and manpower costs with new advanced mining equipment.

8.5 Current System Description

The current system being considered uses continuous miners and is based on the Katell underground model mine which produces 1 million tons a year. In this model it is assumed that the costs of production are not a function of seam thickness, and that only 4 feet of a 6-foot seam are mined with a 57 percent recovery rate. As the thickness of

the seam increases, the recovery rate decreases (Figure 8-1). Thus, mining a 50-foot seam with the current system would leave more than 90 percent of the coal in the ground in an unminable (sterilized) condition.

The production schdeule for the model of the current system is 1 million tons per year with 220 working days a year, which results in 4.545 tons of coal per working day. Productive labor costs are $3.5 million per year, and capital invested in underground mining equipment is $5.5 million every 10 years. O&M costs are $2 million (excluding productive labor) per year.

8.6 The Hypothetical Thick and Multiple Seam Mining System Description

The new system is a hypothetical system specifically designed for mining thick-seam and multiple seam coal beds and represents, for the purpose of this benefit/cost analysis, the end product of the sub-programs specified in Section 8.3. This system is used as a basis for comparison with the current system to determine the benefit/costs.

The new system selected for the analysis is the longwall-caving system described in Section 8.3.1.2. It consists of a longwall miner with reinforced shields specifically designed to enable broken coal to flow through the rear shield onto a conveyor located on the gob side of the longwall miner. This method was selected because of its relative simplicity. It is assumed that the longwall miner used by this system is highly automated and contains two conveyors: a face conveyor, and a gob-side, or rear, conveyor. Because it has shields

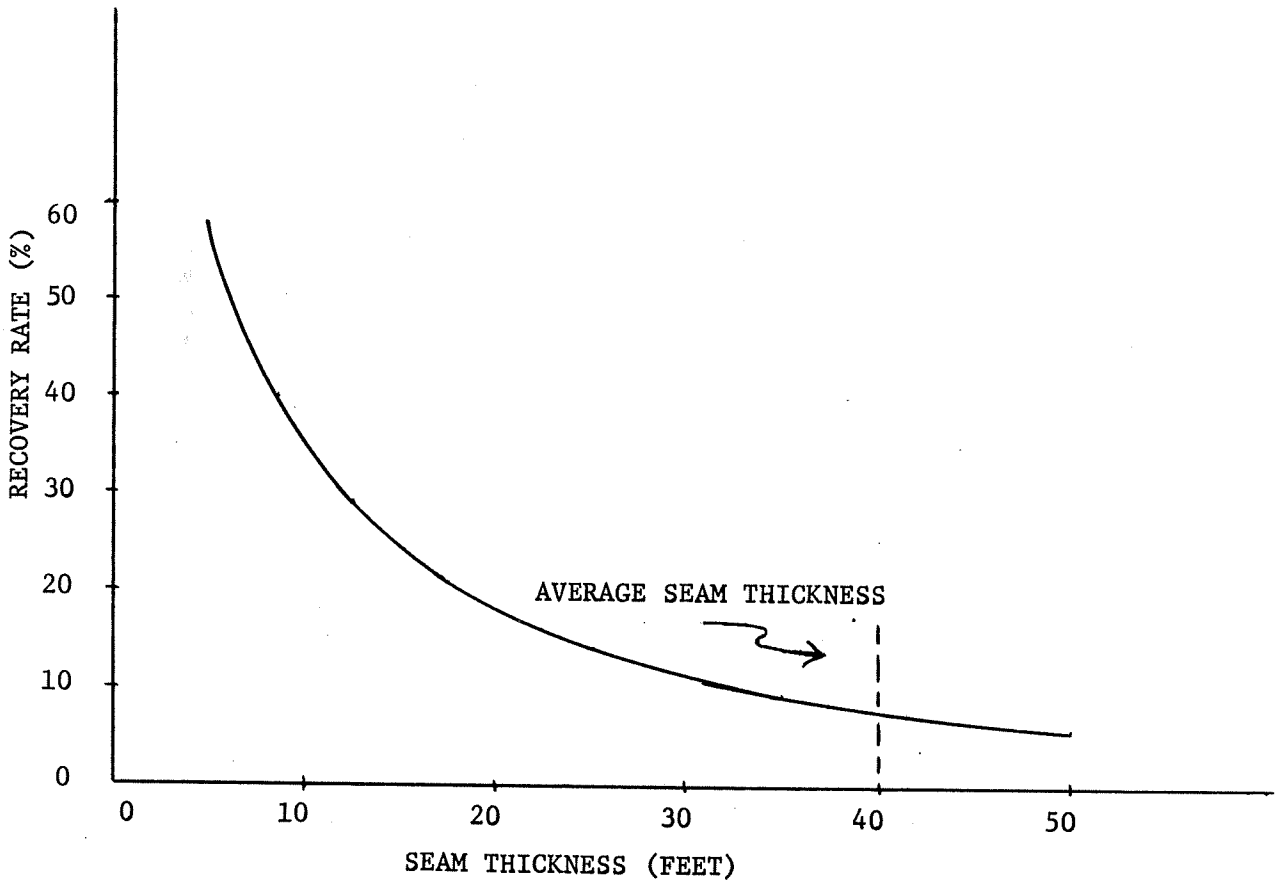

FIGURE 8-1

CURRENT SYSTEM FOR MINING THICK COAL SEAMS
RECOVERY RATE (%) VS. SEAM THICKNESS

to increase the safety of the men by preventing broken coal from flooding the work area, this longwall miner is preferable to one without shields.

In a thick seam, the longwall miner cuts coal from the working face and collapses the coal from the ceiling behind it by blasting or by mechanical means. This broken coal is then allowed to pour in through the rear shield at a controlled rate onto the gob-side conveyor. The longwall miner will not start cutting the working face so that it can move forward until the coal being taken in from the rear of the miner is 50 percent rock. The cleaning cost for coal is approximately $.65/ton at 80 percent coal; for coal that contains 50 percent rock, the cleaning cost is $.75/ton.* Thus, a higher recovery rate can be achieved with very little added expense.

Costs and savings associated with the new system are given in the following subsections.

8.6.1 Production as a Function of Seam Thickness

The major characteristic of the thick-seam mining system will be its increased productivity, which is a function of seam thickness. It is assumed that the maximum amount of coal that can be mined on the working face side of the new system will be 5,000 tons a day, and the maximum amount of coal that can be handled on the gob side will be 9,000 tons a day. However, the rate at which the coal is mined at the working face will be inversely proportional to the coal mined from the gob side. As long as coal is being brought in through the rear shield,

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* Conversation with BOM Field Engineer.

the longwall miner cannot be moved forward--which means that the forward progress of the longwall miner will be slower as the volume of coal mined from the rear increases. The rate of coal mined by the longwall miner is plotted in Figure 8-2. In this figure, the production rate in tons per day is given as a function of seam thickness. From these data, production ratios can be developed that express the difference between the current system and the new system. In Figure 8-3, the number of days per million tons per year associated with the new system is plotted. Thus, the ratio of days needed to produce 1 million tons by the current system (220 days) to the number of days needed by the new system gives the fractional improvement in production. This fraction is then used to reduce the capital costs, the O&M costs, and the labor costs.

8.6.2 New System Costs and Savings

The cost of an automated longwall miner modified to mine broken coal from the rear is estimated at $4 million. This includes a gob-side conveyor, reinforced shields, and machinery to bring in the broken coal through the rear shield. O&M costs, exclusive of labor, are assumed to be $500,000 per year; labor costs will amount to another $500,000 per year. These costs are associated with a 10-foot-thick seam. Each of these costs are reduced, per million tons of coal mined, as the seam thickness increases. These reduced costs are due to the fact that the number of days of production required to obtain 1 million tons decreases with seam thickness. Thus, the cost savings as a result of using the new system is expressed in terms of dollars per million tons of coal mined. The cost savings are calculated as follows:

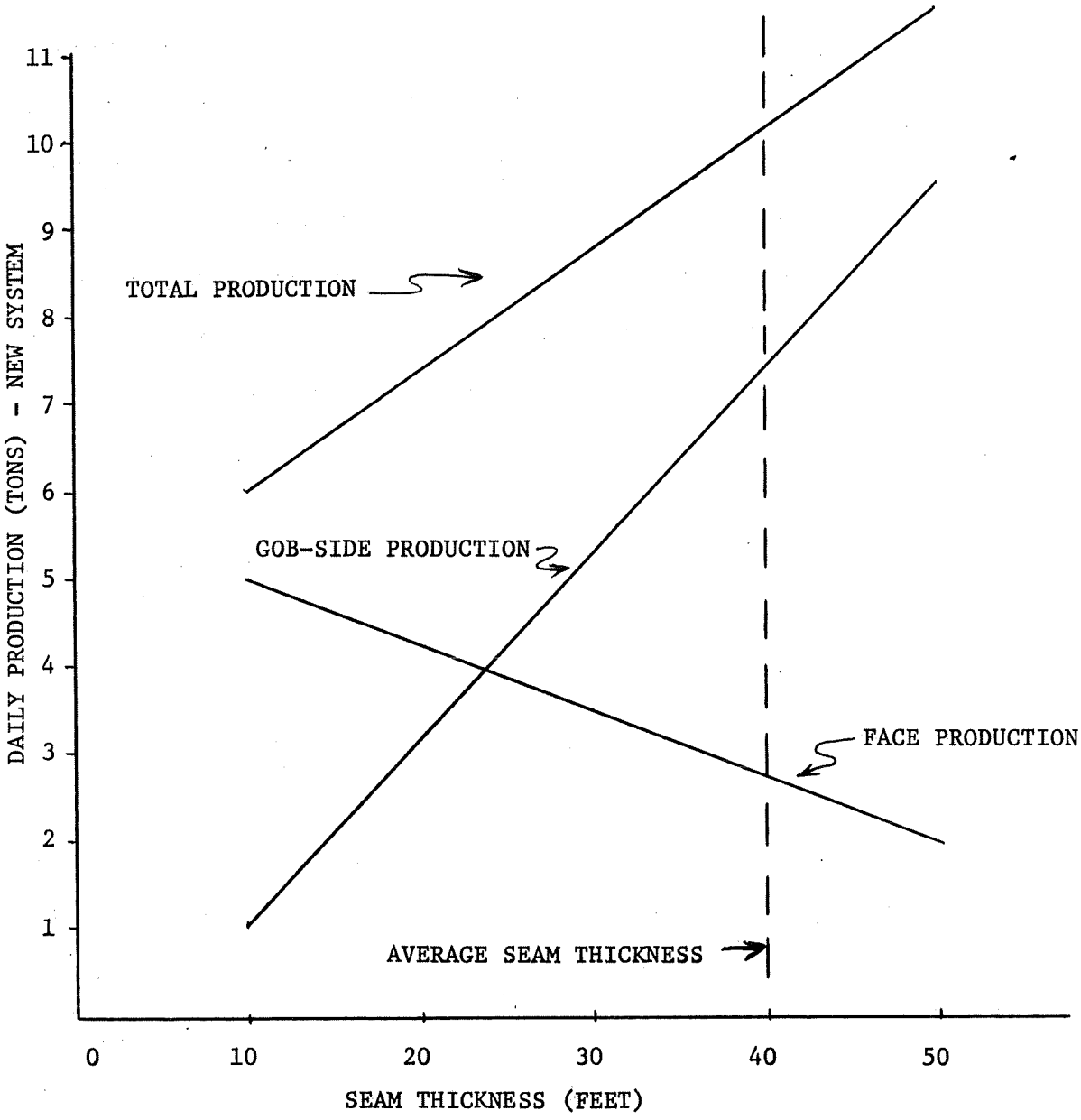

FIGURE 8-2

THICK AND MULTIPLE SEAM MINING SYSTEM
TOTAL DAILY PRODUCTION (TONS)
VS.
SEAM THICKNESS (FEET)

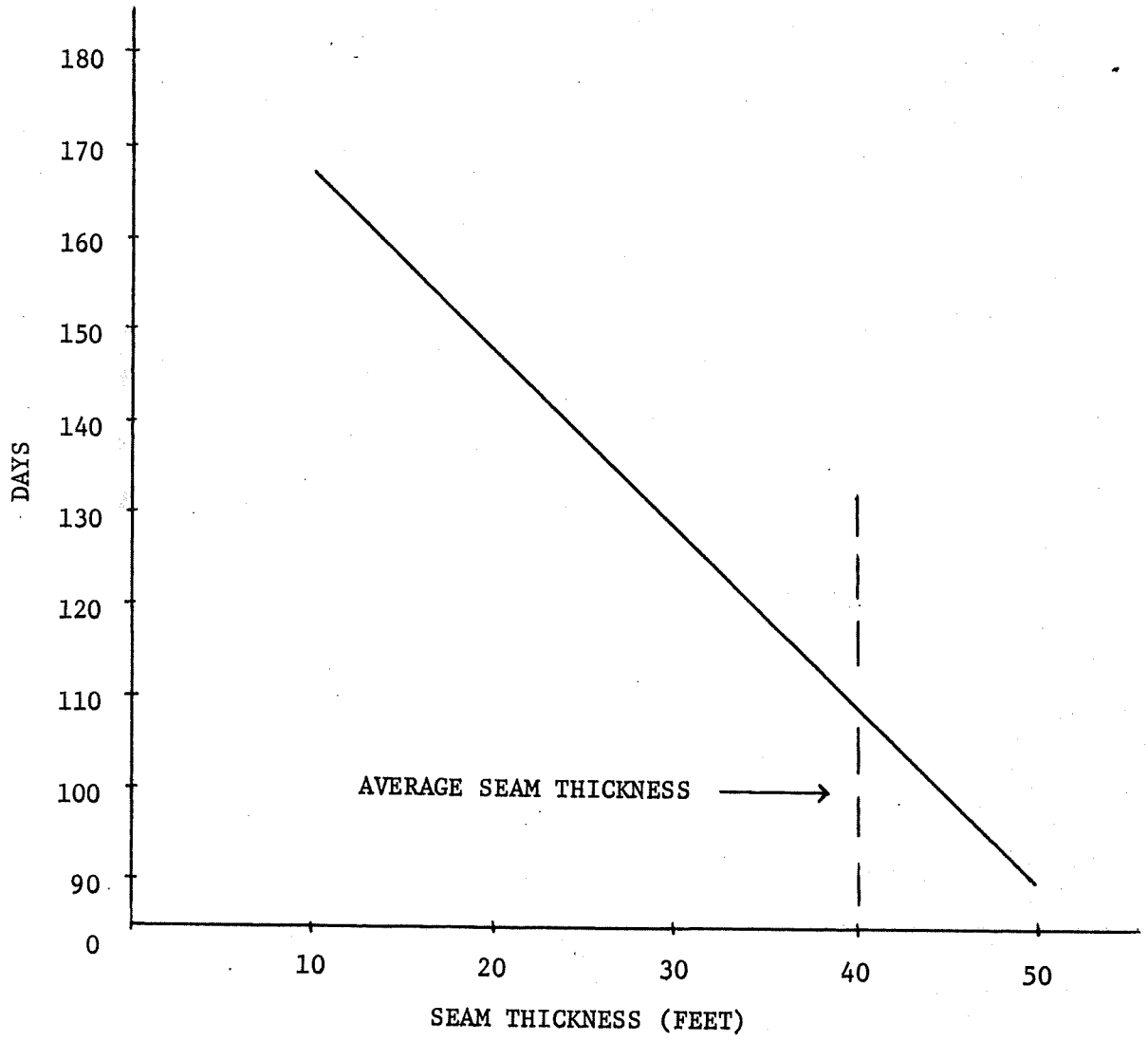

FIGURE 8-3

THICK AND MULTIPLE SEAM MINING SYSTEM

NUMBER OF DAYS PRODUCTION PER MILLION TONS OF COAL
VS.
SEAM THICKNESS (FEET)

Cost Savings for a given seam thickness =

Old System -[(New System Cost) x (Thickness Factor)]

where thickness factor = 0.75 for seam thickness = 10 feet

= 0.41 for seam thickness = 50 feet

In Figures 8-4, 8-5, and 8-6, respectively, the capital costs, O&M costs, and labor costs are presented as functions of seam thickness. The differential cost savings are computed by comparing the costs of the current system and the new system.

8.6.3 Costs and Saving Schedule vs. Seam Thickness

For the purpose of evaluating beenfit/cost ratios and determining the sensitivity of these ratios to seam thickness, cost and saving schedules were developed covering the 20-year life of the model mine. A separate schedule was developed for a seam thickness of 10 feet (see Table 8-6) and for a seam thickness of 50 feet (see Table 8-7). These schedules indicate that there are capital savings per million tons of production during the first year of mine life, at which time the longwall mining system is purchased; and in the eleventh year of mine life, when it is replaced with a new longwall miner--it is assumed that longwall mining systems have an economic life of 10 years. In these two schedules, all figures represent savings due to the large increase in production per unit time resulting from the new system.

8.6.4 Market Penetration

On the basis of past experience with new innovative mining systems, it is assumed that it will take a period of at least 10 years to achieve close to 100 percent market penetration. Thus, the market

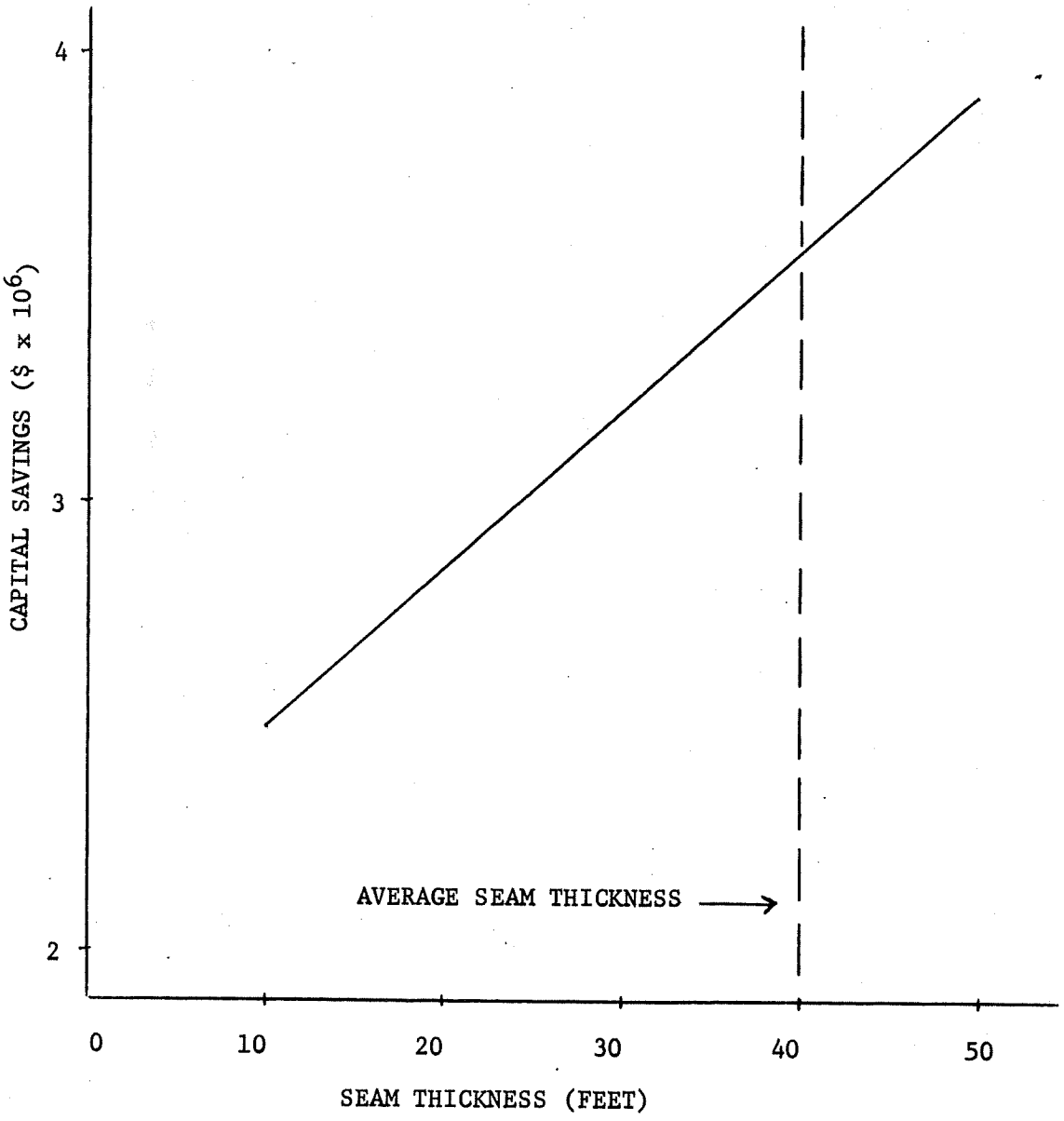

FIGURE 8-4

THICK AND MULTIPLE SEAM MINING SYSTEM

CAPITAL SAVINGS FOR 10 YEAR PERIOD
VS.
SEAM THICKNESS (FEET)

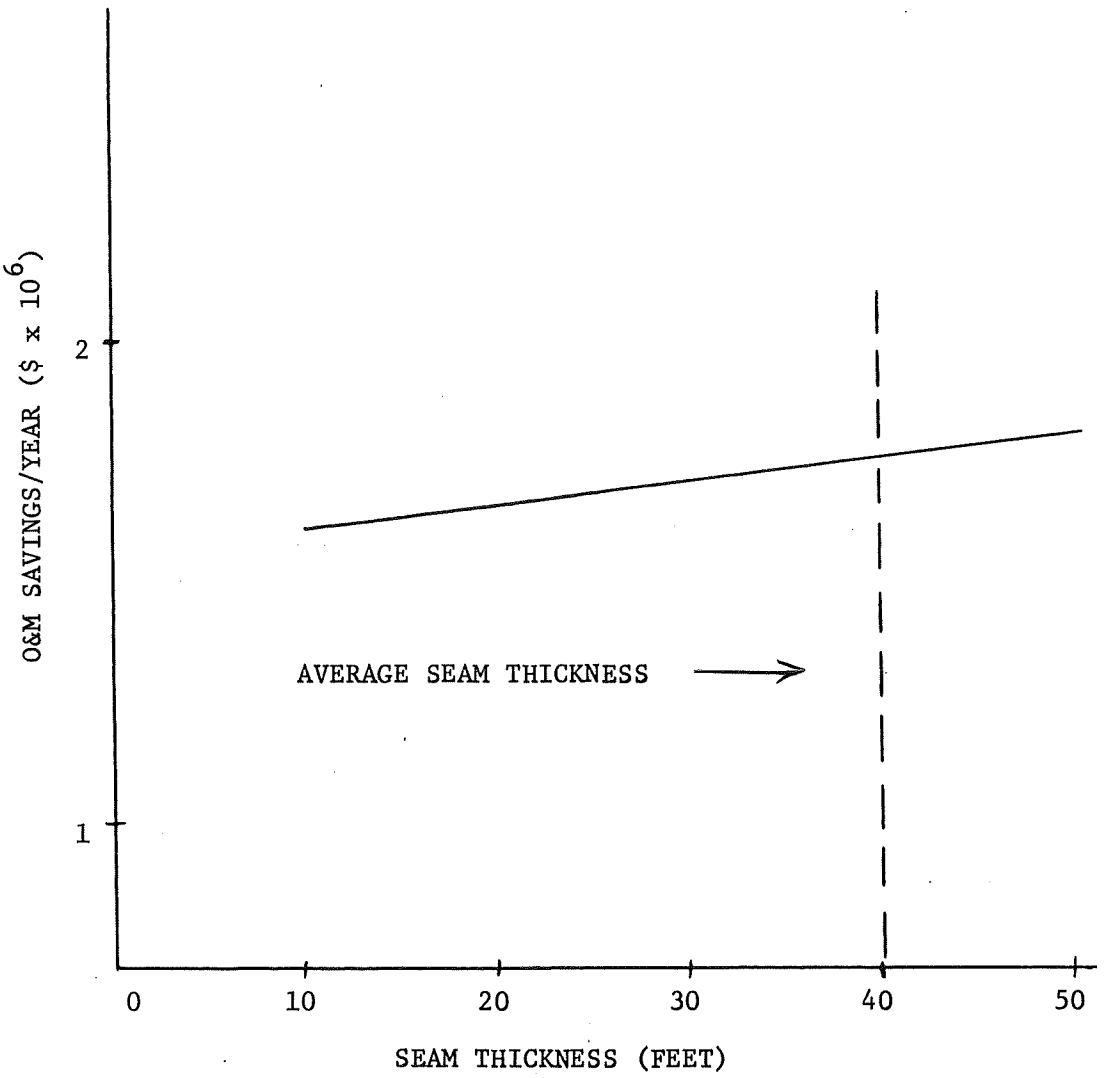

FIGURE 8-5

THICK AND MULTIPLE SEAM MINING SYSTEM

O&M SAVINGS PER YEAR
VS.
SEAM THICKNESS (FEET)

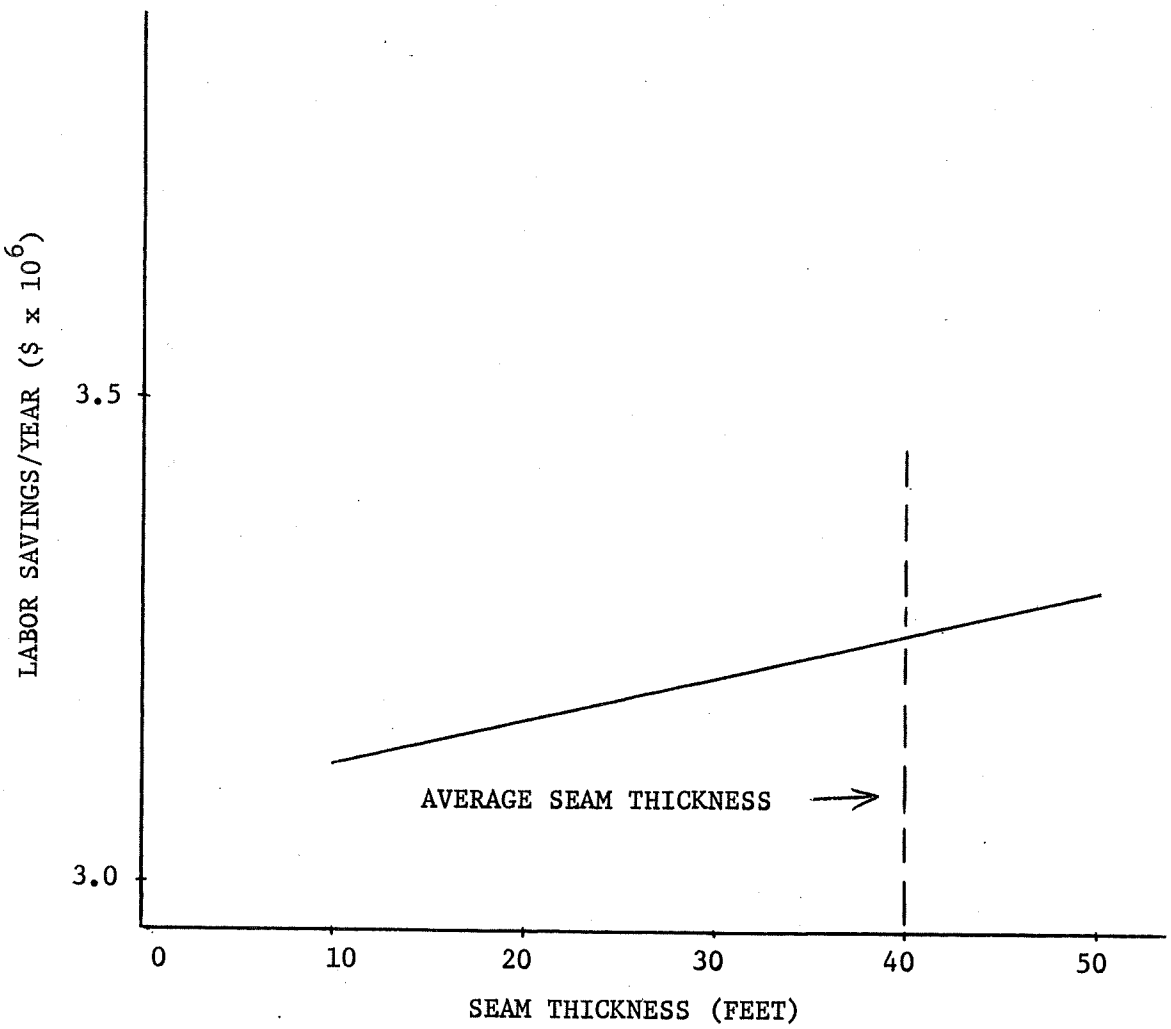

FIGURE 8-6

THICK AND MULTIPLE SEAM MINING SYSTEM

LABOR SAVINGS PER YEAR
VS.
SEAM THICKNESS (FEET)

TABLE 8-6

SAVINGS SCHEDULE FOR NEW SYSTEM FOR A SINGLE MINE

WITH SEAM THICKNESS = 10 FEET

Year	Capital Savings ($ x 10^6)	Labor Savings ($ x 10^6)	O&M Savings ($ x 10^6)
1	2.5	3.125	1.625
2			
3			
4			
5			
6			
7			
8			
9			
10			
11	2.5		
12			
13			
14			
15			
16			
17			
18			
19			
20			

TABLE 8-7

SAVINGS SCHEDULE FOR NEW SYSTEM FOR A SINGLE MINE

WITH SEAM THICKNESS = 50 FEET

Year	Capital	Labor	O&M
1	3.86	3.295	1.795
2			
3			
4			
5			
6			
7			
8			
9			
10			
11	3.86		
12			
13			
14			
15			
16			
17			
18			
19			
20			

penetration schedule reflecting this assumption is given in Table 8-8. System availability will be 1984.

8.7 Analysis and Results

The benefit/cost analysis was performed for the two production scenarios and for a seam thickness of 10 feet and 50 feet.

Figure 8-7 plots the reduction in production cost as a function of seam thickness. This reduction is independent of the production scenarios. Figure 8-8 plots the net benefits per R&D dollar as a function of production scenario and seam thickness. Figure 8-9 plots the net capital savings per R&D dollar for each scenario vs. seam thickness.

Since the production costs of the coal using continuous miners in the model miner were $10.00 per ton, then, using the thick-seam mining system will reduce production costs from this value as a function of seam thickness. Figure 8-10 plots the coal production costs using this new system vs. seam thickness. Table 8-9 gives the expected benefits when an average seam thickness of 40 feet for underground Western coal is assumed.

TABLE 8-8

MARKET PENETRATION SCHEDULE

YEAR	PENETRATION FACTOR
1	0.1
2	0.2
3	0.3
4	0.4
5	0.5
6	0.6
7	0.7
8	0.8
9	0.9
10	1.0

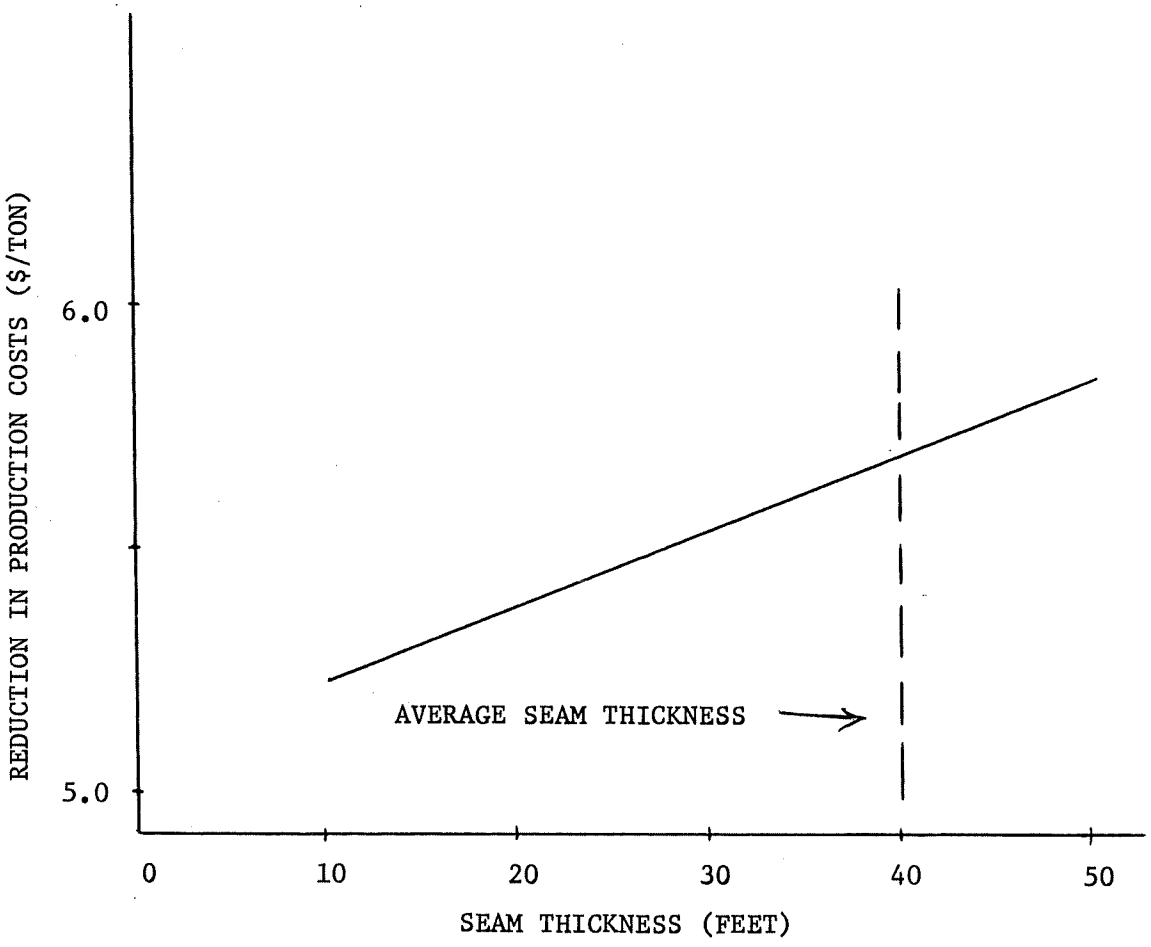

FIGURE 8-7

THICK AND MULTIPLE SEAM MINING SYSTEM
REDUCTION IN COAL PRODUCTION COSTS PER TON
VS.
SEAM THICKNESS (FEET)

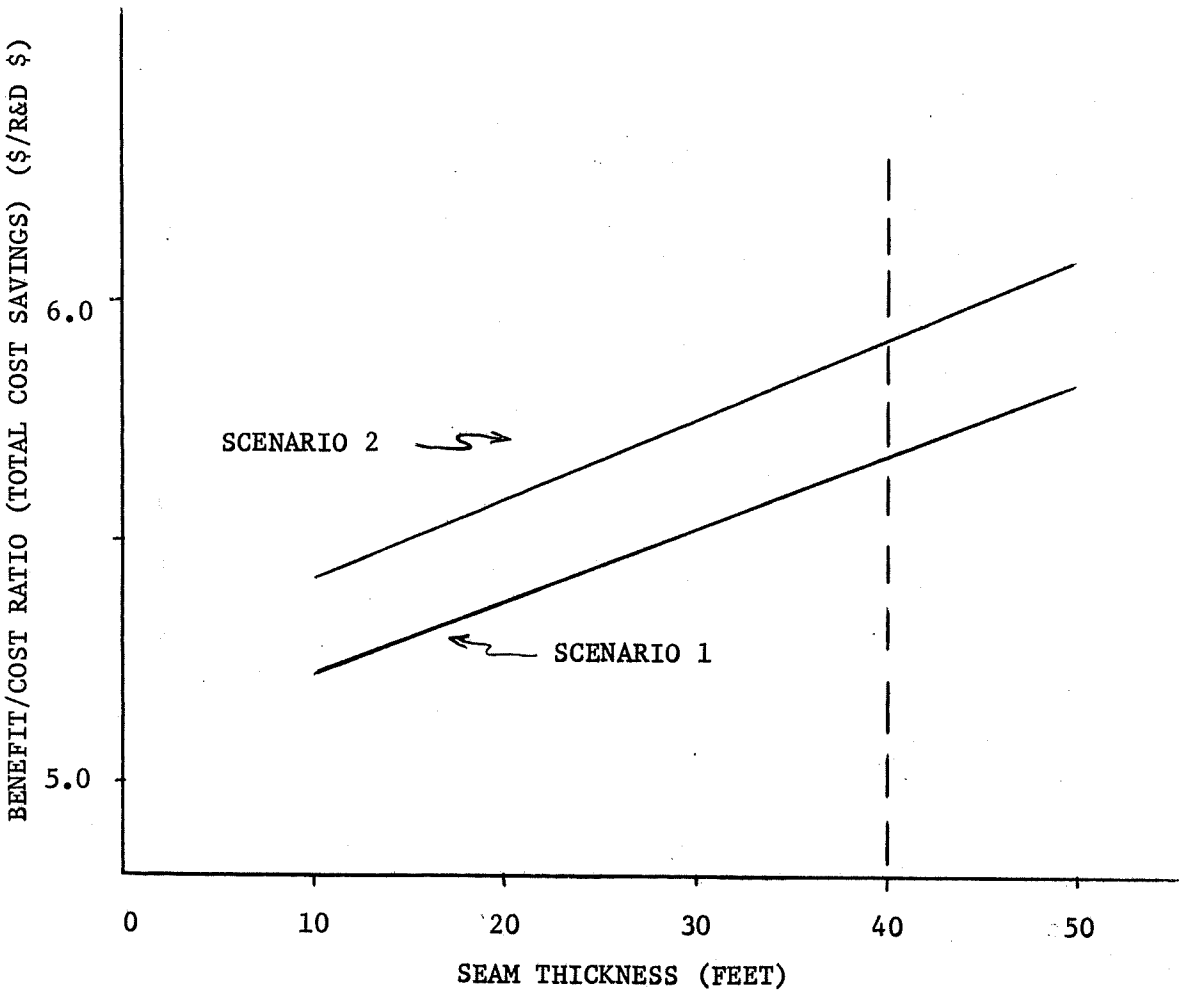

FIGURE 8-8

THICK AND MULTIPLE SEAM MINING SYSTEM

BENEFIT/COST RATIO (TOTAL COST SAVINGS)
VS.
SEAM THICKNESS (FEET)

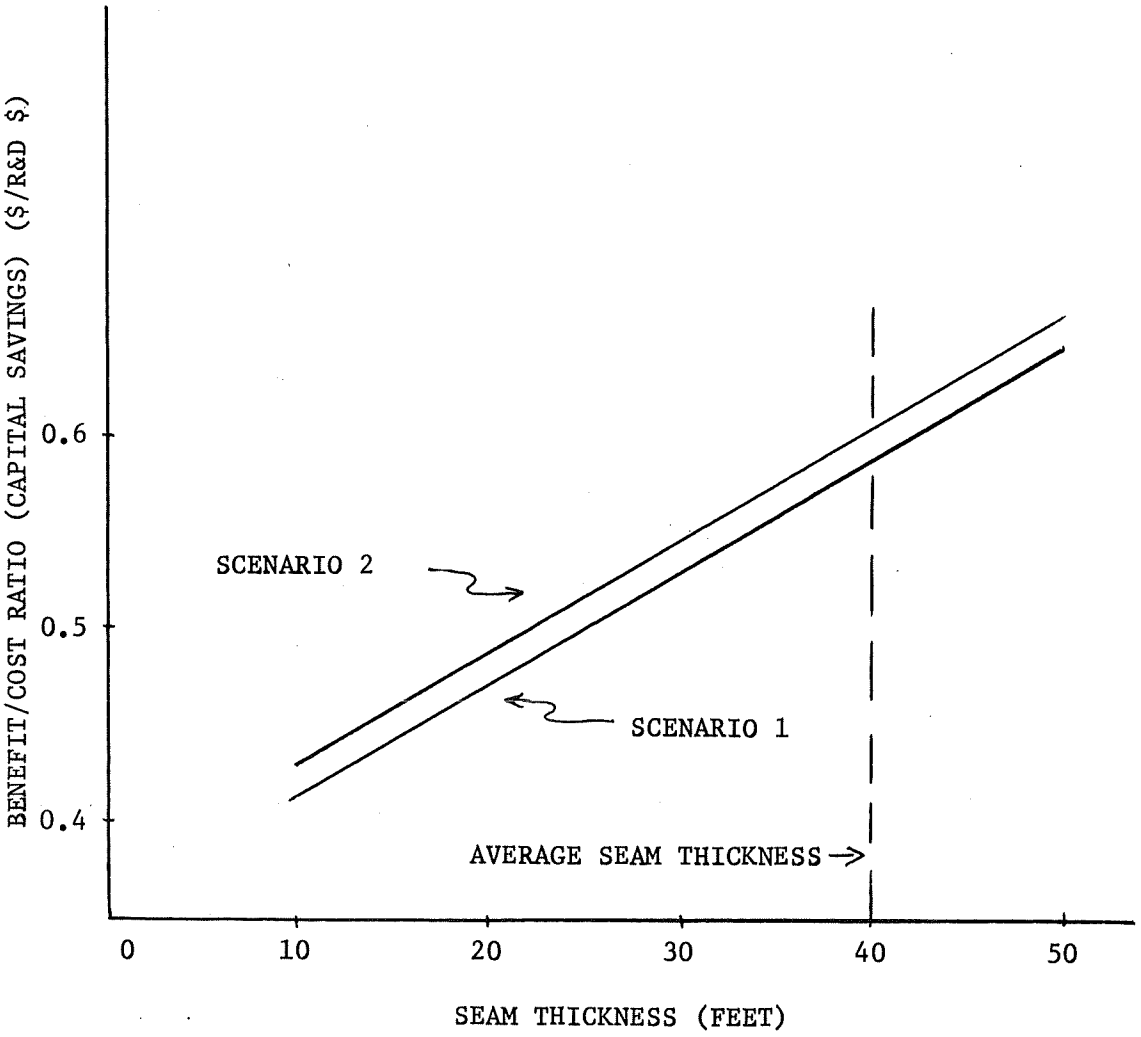

FIGURE 8-9

THICK AND MULTIPLE SEAM MINING SYSTEM

BENEFIT/COST RATIO (CAPITAL SAVINGS)
VS.
SEAM THICKNESS (FEET)

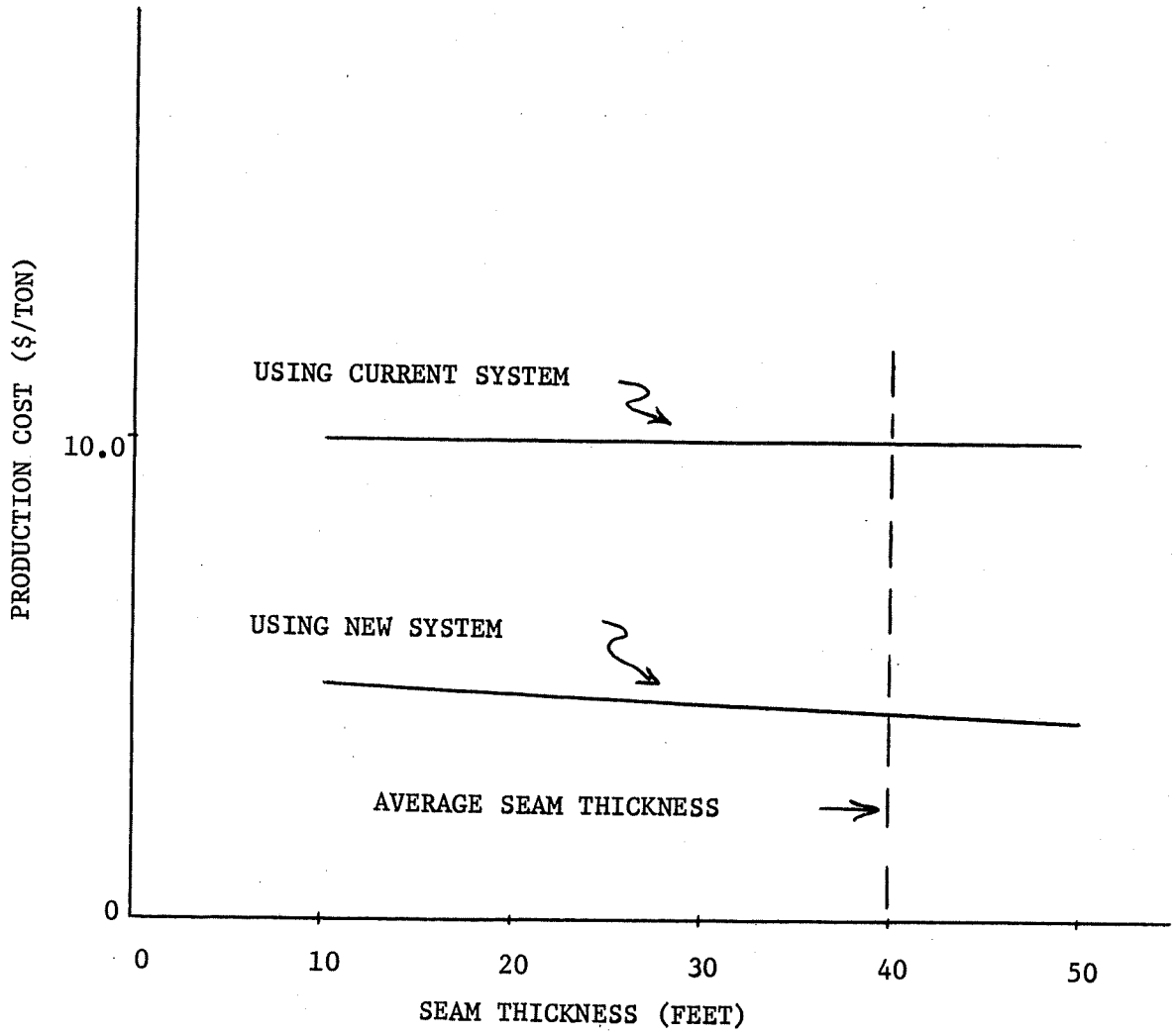

FIGURE 8-10

THICK AND MULTIPLE SEAM MINING SYSTEM

PRODUCTION COST PER TON USING NEW SYSTEM
VS.
SEAM THICKNESS

TABLE 8-9

EXPECTED BENEFITS FOR SEAM THICKNESS = 40 FEET

	Scenario 1	Scenario 2
Reduction of Production Cost ($/Ton)	$5.69	$5.69
Benefits/R&D Dollar	5.68	5.91
Capital Savings/R&D Dollar	0.58	0.60

REFERENCES

1. Ahcan, R., 1969.

9.0 SURFACE MINING

Surface-mined coal is an increasingly important source of national energy. Because surface methods offer significant advantages over underground methods, there has been a decided shift toward surface operations; surface mining accounted for only 26 percent of U.S. coal production in 1954, increasing to 50 percent in 1974. The trend toward increased surface operations is expected to continue in the near future in order to meet predicted energy requirements.

Surface methods presently offer the greatest potential for rapidly expanding production to enhance national self-sufficiency in energy because surface mines may be developed more rapidly and on a larger production scale than underground mines. When compared with underground mining, surface mining is 30 percent less labor intensive, considerably safer, and has significantly higher resource recovery efficiencies and lower production costs. Furthermore, of the 45 billion tons of potentially recoverable strippable reserves, 71 percent contains less than 1 percent sulfur, which facilitates satisfying both energy demands and national air quality standards.

Although surface methods offer great potential for expanding production, considerable problems must be overcome if surface mining is to make a significant contribution to national energy self-sufficiency. Current surface mining methods are adequate for supplying coal in today's market; however, productivity has remained relatively constant since 1969, while rising costs for labor, equipment and supplies, coupled with stricter reclamation requirements, have pushed production costs up sharply. Historically, the mining industry's response to

rising production costs has been to develop larger and larger stripping equipment. This response will no longer be adequate. The size of mining equipment, particularly large draglines and stripping shovels, has reached a plateau in the last five years as a result of the economics of utilizing very large machines. The main limitations appear to be a lack of suitable applications for very large equipment, lack of flexibility, and serious maintenance and transport problems. In addition to increasing production problems and costs, it appears that the surface mining industry will be faced with increasingly stringent environmental legislation.

Increasing surface coal production in spite of current and expected problems will require a multiple-pronged approach. Improvement and modification of current systems and equipment offer the greatest potential for rapidly increasing production in the near future. Significant advances in long-term production will require development of innovative techniques and equipment. The Bureau of Mines R&D effort in surface mining is designed to investigate and evaluate each of the problems and possible solutions. The purpose of this analysis is to determine the benefits associated with the surface mining R&D effort.

9.1 Production Scenarios

The production scenarios utilized in this analysis are based upon standard Scenarios 1 and 2 as presented in Section 3.0. Several modifications and assumptions have been made to facilitate analysis of the data:

a) Production figures have been limited to surface production.

b) Production has been limited to two mine sizes for the analysis: small and large. Small mines are assumed to have a capacity of 1 million tons per year; production figures for each year and province being determined by summing the production from all surface mines with annual production less than or equal to 1 million tons per year. Large mines in the analysis are assumed to have a production capacity of 3- to 5-million tons per year; production figures for each year and province being determined by summing the production from all surface mines with annual production greater than 1 million tons per year.

9.2 State-of-the-Technology

There are three basic categories of surface mining methods in the United States: area, contour, and auger. There are many different versions of area mining and contour mining methods in use, depending upon specific situations. Key considerations in choosing the mining method and equipment to be employed, as well as specific modifications of the basic method, include:

a) Coal seam characteristics: thickness, quality;

b) Overburden: thickness, chemical and physical characteristics;

c) Stripping ratio;

d) Surface topography: slopes, drainage;

e) Environment: land, water, wildlife, vegetation;

f) Equipment and materials: availability, cost, flexibility;

g) Desired production;

h) Haulage: distances, cost.

9.2.1 Area Mining

Area mining, the principal method of surface mining in the United States accounted for 73 percent of surface coal production in 1972. It is generally limited to those geographic regions having gently rolling to relatively flat terrain and coal beds lying under less than 200 feet of overburden. Area mining is conducted throughout the Illinios Coal Basin, Midwestern coal fields, and the far West; some operations are also conducted in the Eastern Province.

Cuts are made through the overburden in parallel rows (usually referred to as a trench or boxcut) to expose the coal seam. The overburden from the first cut is placed on unmined land adjacent to the cut and the exposed coal is then removed. Once the first cut is completed, a second cut is made parallel to the first, and the overburden from the succeeding cuts is deposited in the cut previously excavated. Large equipment, such as shovels, draglines, bucket wheel excavators, pan scrapers, bulldozers, and front-end loaders, are used to remove the overburden and cast or convey the material. Shovels and draglines are the most commonly used pieces of equipment, with draglines increasingly favored over shovels. The largest shovels and draglines in operation have bucket capacities of 180 and 220 cubic yards, respectively. Size limits have now apparently been reached, as both the largest draglines and shovels in the industry were built more than five years ago. The current trend has been against selecting the largest equipment available because of performance problems. The mine operators are continuing to depend on equipment in the intermediate

capacity ranges (e.g., bucket capacities of 20 to 100 cubic yards), which have proven performance. Smaller shovels or front-end loaders are used to load coal onto trucks, and bulldozers and scrapers are used to grade and smooth the rugged, washboard configurations formed by spoil bank ridges.

The performance of wheel excavators hold considerable promise where conditions are favorable. This machine has the capability for continuous overburden removal and selective placement of the topsoil.

In the West, where coal seams are unusually thick, open-pit extraction techniques are often used. At many operations, tractor scrapers and bulldozers are used for primary stripping; large conventional road excavation and grading equipment also are used for primary stripping.

9.2.2 Contour Mining

This mining method produces approximately 22 percent of the surface coal production in the U.S. and is employed on rolling to very steep or mountainous terrain, typified by the Appalachian bituminous coal fields. The conventional method of contour mining is conducted along horizontal benches rimming the mountainside. The overburden--topsoil and other strata--is removed from the coal bed starting at the outcrop and proceeding around the hillside. It is then deposited down the hillside and stacked along the outer edge of the bed. After the coal is removed, successive cuts are made until the depth of the overburden becomes too great for economical recovery of the coal or exceeds the physical limitations or capability of the earth moving equipment.

Conventional contour surface mining presents many more environmental problems than area mining methods. Conventional contour mining on a steep slope will inevitably result in erosion and ground slides, because the overburden from the first cut is stacked on the outslope of the mined bench and the slope of the resulting pile is equal to or exceeds the material's angle of repose. Other problems of conventional mining include:

a) Aesthetic blight (Flattened mountain tops and exposed highwalls)

b) Landslides

c) Sedimentation

d) Mud flows and rock slides

e) Acid water

f) Destruction of wildlife habitat

Because of these environmental problems, many states have imposed strict environmental regulations that have necessitated changes in the mining methods used by coal operators. These changes were designed to allow surface coal operations to maintain a profit while complying with the law.

Among the new methods developed to meet environmental requirements are: (1) the head-of-hollow fill method, (2) the mountain-top removal method, and (3) the block-cut method.

The head-of-hollow fill method utilizes narrow V-shaped, steep-sided valleys near the ridge top that are free of underground mine openings, seeps, or wet weather springs to dispose of spoils from the removal of entire mountain tops and overburden from contour

benches. If properly designed and constructed, horizontal and vertical pressures will provide adequate friction to prevent a failure in the fill and will stabilize the resulting surface. Instead of miles of unstable outslope, with its potential for slides and erosion, or islands of isolated land with no access, a large, stable, fairly level area can be constructed with this method. Utilization of this method enhances aesthetics, reduces landslides, allows for full recovery of one or more coal seams, and produces potentially valuable flat to rolling mountain-top land suitable for uses other than forestry.

The second new system developed to meet environmental restrictions is the mountain-top removal method, an adaptation of area mining to contour mining for rolling to steep terrain. Where coal seams lie near tops of mountains, ridges, knobs, or knolls, they can usually be economically strip-mined. The entire tops are removed down to the coal seam in a series of parallel cuts. Excess overburden that cannot be retained on the mined area is transported to head-of-hollow fills, stored on ridges, or placed in natural depressions. This mining method produces large plateaus of level, rolling land that may have great value in mountainous regions.

Many of the coal seams that lie high on a mountain often cannot be recovered by underground mining. Extreme surface subsidence, unsafe roof conditions, and narrowness of the coal seam make these coal reserves recoverable only by mountain-top removal.

The third special method of surface mining is the block-cut method (haul back, pit storage, put-and-take, etc.), a simple variation of the conventional contour strip mining method for steep terrain.

Instead of casting the overburden from above the coal seam down the hillside, it is hauled back and placed in the pit of the previous cut. The method is not new and is known by various names, depending on the locality. Basically, the operational procedures are similar in that no spoil is deposited on the downslope below the coal seam, topsoil is saved, overburden is removed in blocks and deposited in prior cuts, the outcrop barrier is left intact, and reclamation is integrated with mining.

When the mine is started, a block of overburden is removed down to the coal seam and disposed of. This first-cut spoil can be placed above the highwall or spread along the downslope as in conventional contour mining, or moved laterally and deposited in a head-of-hollow fill or ridge fill. The original cut is made into the hillside to the maximum depth that is to be mined. The width is generally three times that of the following cuts. After the coal is removed, the overburden from the second cut is placed in the first pit and the coal from the second cut is removed. This process is repeated as mining progresses around the mountain. Once the original cut has been made, mining can be continuous, working in both directions around the hill or in only one direction.

The cuts are mined as units, thereby making it easier to retain the original slope and shape of the mountain after mining. In all cuts, an unmined outcrop barrier is left to serve as a notch to support the toe of the backfilled overburden. Block-cut mining makes it possible to mine on slopes steeper than those being mined at present

without the danger of slides and with minimal disturbance. Approximately 60 percent less total acreage is disturbed than by other mining techniques now in use.

There are various environmental advantages of the block-cut method that make this an acceptable steep-slope mining method. Significant visual evidence indicates that the block-cut method is less damaging than the old practice of shoving overburden down the side of the mountain, a practice which resulted in permanent scars on the landscape. With the block-cut method, the tree line below the mined area and above the highwall is preserved, and the results of the mining operation generally are hidden so that they cannot be seen from the valley below.

Existing or pending state and Federal legislation makes it illegal to push overburden beyond the outcrop and over the mountain side and thus prohibits the conventional type of contour strip mining. However, the block-cut and similar methods meet the criteria of this new legislation and allow for recovery of coal reserves in mountainous regions that otherwise would be unminable.

9.2.3 Overview of Problems in Surface Mining

The surface mining methods in current use are adequate for supplying coal in today's market. However, productivity has remained relatively constant since 1969, while rising labor, equipment, and supply costs, coupled with stricter reclamation requirements, have pushed mining costs up sharply.

Historically, the mining industry has used larger stripping machines as the solution to increased costs and deeper coal depths. The size of mining equipment, particularly large draglines and stripping shovels, has reached a plateau in the last five years because of the economics of utilizing very large machines. The main limitations appear to be a lack of suitable applications for very large equipment, coupled with a lack of flexibility and serious maintenance and transport problems. Rather than increase the size of draglines and shovels, as has been the practice in recent years, the trend is now toward refining and improving the existing intermediate-size machines.

The selection and use of mining equipment, mining methods, design and layout of mine pits, operating policies, and maintenance policies are done with little preplanning and consistency throughout the industry.* The preplanning and systematic layout and operation of a surface-mining operation vary significantly from operation to operation.

Currently, mining equipment is generally used in a batch process operating mode, with the spoil radius of the dragline or strip shovel being the basic parameter governing the mining plans and sequence of mining. Because the mining method is governed by the spoil radius parameter, other dependent operation problems are generated:

a) In many cases, the working area between the highwall and the spoil piles is 100 feet or less. This reduces coal and waste loading productivity and increases maintenance costs substantially.

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*Discussions with BOM personnel and with Theodore Barry Associates.

b) Sequence of cuts and castings of spoil becomes extremely critical and invariably results in rehandling of soil material.

c) Casting results in spoil piles having large peaks and valleys requiring high-cost tractor grading operations before reclaiming the land.

d) The coal wedge must be recleaned of spoil slump material before loading.

e) Coal reserves, as determined by economic stripping ratios, are limited by geometry rather than unit productivity costs.

f) Large stripping machinery is "misused" on thin (10- to 15-foot) parting seam waste removal because the casting distance is needed for spoiling far enough from the highwall.

g) Deep coal seams or thin beds are not mined because of equipment matching problems.

h) Unproductive "dragline walking" becomes a common operating delay.

Existing state reclamation requirements and potential Federal surface mining legislation have necessitated that land reclamation become an integral part of the mining operation. In the past, reclamation has been done after mining was completed. This generally resulted in double handling of the overburden and associated increases in costs. Recent trends in state legislation have been reflected in a transition from past mining reclamation requirements to more extensive requirements calling for concurrent reclamation. Technology is adequate over much of the country to meet these requirements; however, in some areas, an increase in production costs as a result of new environmental laws may preclude surface mining unless new systems or more productive specialized equipment are developed.

The key to effective reclamation and its integration with production is preplanning. Existing and proposed state and Federal

surface mining and reclamation statutes contain permit programs that require mine operators to develop and prepare comprehensive pre-mining and reclamation plans before mining commences. Methods and procedures must be developed to consider:

a) Chemical analysis of overburden and soil materials before mining begins;

b) Water damage control;

c) Sediment control;

d) Erosion control through proper grading and initial revegetation;

e) Controlled placement of spoil;

f) Most profitable use of reclaimed land both during and following mining operations;

g) Climate of region, particularly amount of precipitation;

h) Land use before mining;

i) Special characteristics of the region, including wildlife habitation and lifestyle of local populations.

Reclamation requirements for contour mining have resulted in several new and innovative mining systems (e.g., the block-cut method) in steep terrain. However, these new systems still use existing equipment, such as loaders and trucks to remove and replace the overburden. Use of this equipment in the new system has the following limitations:

a) Investment costs for spoil haulage are increased.*

b) Complicated and time-consuming methods of drilling and blasting are required to maintain control of the

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* A haulback mining operation observed by Skelly and Loy uses 2 front-end loaders, 1 dozer and 4 trucks to remove and haul overburden; a conventional contour mining operation would use 1 or 2 front-end loaders.

overburden and to get proper fragmentation for the particular types of equipment being used in spoil removal.

c) Special precautions must be taken in scheduling the various phases of mining and reclamation so as to realize the maximum recovery of coal, while eliminating dead-time for equipment.

d) Long-term environmental consequences are not known and will require a monitoring program of pilot block-cut operations to determine if stream siltation and mineralization can be eliminated.

These new systems have been estimated by some mine operators to be from 30 to 100 percent more expensive than previous operations.* Thus, these systems must be evaluated to determine if they meet environmental requirements, and if they can result in increased productivity by better operational procedures and the implementation of different types of material handling equipment.

9.3 Related Subprograms

The major objective of the surface mining R&D effort is to develop and demonstrate surface mining systems and equipment that will increase production and productivity, reduce costs, maximize mine safety and enhance reclamation of disturbed lands.

The general approach taken by BOM for achieving the objectives is to integrate mining and reclamation activities into a "total system" that will make a significant improvement in operational efficiency.

The main emphasis of the program is the development of:

a) A primary planning capability that will satisfy existing and anticipated local, state, and Federal surface mining reclamation regulations, and will not restrict or decrease surface coal production;

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*Discussions with BOM personnel and a telephone conversation with Professor W. F. Foreman of VPI.

b) A system's approach to surface mining that will effectively integrate coal production and reclamation;

c) Equipment items that are complementary to each other, flexible, and capable of being used to achieve a continuous material handling systems operation as opposed to current batch operations.

The projects can be classified as having either "near-term" benefits or "long-term" benefits. Projects associated with near-term benefits involve the testing and demonstration of existing and proposed system concepts and equipment that may be available for implementation in the field by FY 79.

The long-term projects are geared for the development of a mining system two or three times more productive than today's methods. The Bureau of Mines' approach in this area is to develop truly continuous systems which will increase productivity significantly and reduce costs. As a result of these efforts, it is anticipated that new systems and equipment will become available starting in FY 81 to replace or augment the near-term projects.

The R&D approach is an effort designed:

a) To make a quick impact using available technology that will increase productivity and satisfy reclamation requirements in the 1977-1979 time period when significant increases in coal production are expected; and

b) To develop more advanced system concepts and equipment to improve, augment, or expand the expected benefits from the near-term effort.

The surface mining R&D effort consists of two subprograms:

Surface Mining, Extraction and Reclamation (SMER),

Surface Mining Equipment (SME).

The projects within the two subprograms provide for a comprehensive coverage of the entire surface mining system operation. They address the three primary surface mining operations--planning, production, and reclamation--and critical functions within each of these operations.

Most of the projects within the two subprograms are applicable to all regions of the country and any method of mining.

9.3.1 Surface Mining, Extraction and Reclamantion Subprogram

9.3.1.1 Subprogram Goals. The goals of this subprogram are:

a) To improve production and productivity of surface mining and reclamation operations.

b) To minimize the impact of surface mining on the environment.

9.3.1.2 Subprogram Description. The research effort is designed to integrate the planning, production and reclamation functions of surface coal mining into an efficient, continuous system. The emphasis is to improve equipment selection and scheduling, operating procedures, maintenance and manpower training using existing technology. The projects within this subprogram may be grouped into four major elements:

a) Engineering and environmental analysis,

b) Mine systems development,

c) Mining and reclamation planning,

d) Spoil stabilization and planning.

Analysis will include evaluations of current overburden handling techniques and land reclamation practices and costs.

Retreat surface area mining methods will be investigated for

moderately dipping coal beds to determine the feasibility of altering these systems in a manner that would minimize or eliminate highwalls and haul ramps through the spoil area. This would facilitate simultaneous mining and reclamation and expediate achievement of reclamation objectives. It would also reduce manpower and costly equipment currently used for highwall reduction, and maintenance and reclamation of haul ramps.

Existing surface area mining systems and underground longwall mining systems will be evaluated to determine the practicability of integrating these mining techniques to provide an efficient, economic, and safe system for recovering shallow coal deposits in a manner that minimizes direct disturbance of the overburden.

New, regionally targeted, surface mining systems will be developed that minimize material handling by integrating overburden handling and reclamation activities into a continuous operation.

New material handling techniques will be developed to selectively remove, transport, and control the deposition of overburden material from along the highwall and/or outbound edge of the contour mine pit. The desired result is to enhance the productivity and simultaneous reclamantion aspects of the block-cut mining method by eliminating the pit handling of spoil material.

Current highwall reduction practices will be analyzed and improved explosive/mechanical methods of rapid highwall elimination under varying topographic and geologic conditions will be developed and demonstrated. These methods would achieve safe, cost-effective

compliance with existing or anticipated state highwall reduction requirements, complement backgrading of spoil material to fill final cuts, cover exposed coal and toxic materials, control drainage, provide access to upland areas, eliminate hazardous conditions, and enhance revegetation.

The feasibility of employing the block-cut method of contour mining, or a suitable modification thereof, will be evaluated and demonstrated to recover coal resources in steep slopes in central and southern Appalachia in a manner that will achieve rapid reclamation while increasing economic and safe coal production.

A successful demonstration should result from the use of the "short pit" block-cut method, which would shorten the length and time the loading area is open, simplify water handling problems, increase backfilled spoil stability, and reduce cost of reclamation by diminishing the area of affected land to be reclaimed.

Logical rationale, procedures, and guidelines for pre-mining and reclamation plans will be developed to assist mine operators in their pre-mining site evaluation. The plans will analyze soil and rock properties, surface and ground resources, and other physical and cultural features that must be considered to satisfy local, state, and Federal regulatory requirements.

The utility of false color infrared aerial photographic imagery will be investigated and demonstrated as an effective tool for coordination of mining and reclamation planning and monitoring. The effort will develop and/or compile interpretative criteria to aid in effective utilization of such imagery by the mine operator.

The extent of adverse environmental effects of airblast and ground vibrations from blasting in large-scale surface coal mines will be determined. Blasting techniques to minimize these effects will be developed.

The feasibility of using urban and industrial waste as inexpensive soil amendments and fill material to enhance revegetation and reclamation will be examined. This is intended to provide the mine operator with an inexpensive alternative to fertilizer and fill. The probelms to be investigated are collection, preparation, and transportation of the waste to the mine, blending of wastes with spoil materials and, where desirable, with topsoil, and determination of proper application rates.

9.3.2 Surface Mining Equipment Subprogram

9.3.2.1 Subprogram Goals. This subprogram has the following goals:

a) Development of equipment having less capital cost per unit.

b) Development of equipment capable of being used in more flexible systems.

c) Development of systems and equipment capable of selectively depositing the overburden.

d) Development of systems capable of mining all coal seams economically.

e) Ensuring that the equipment and systems developed to meet the above objectives would be applicable to the majority of surface coal mines operating today.

9.3.2.2 Subprogram Description. A Holland Loading Company Vertical Bladed Excavator will be tested to determine its capability

to dig and selectively deposit coal and overburden in a surface coal mining operation. The capability to segregate and selectively deposit coal is essential for developing a continuous system and is likely to be important when coal is being used for conversion to synthetic fuels and blending may be required.

An Aerial Conveyor System will be tested for moving and placing overburden in open-pit mine operations.

This new materials handling system, which is to be utilized in conjunction with existing mining equipment is expected to increase the productivity of the dragline since the dragline boom can be shortened to permit installation of a larger bucket. It is expected to lower the cost of transferring spoil in place, and reduce the costs of backfilling and leveling the spoil since the spoil profile will be lower.

A wheel-bladed excavator will be tested to demonstrate the capability to extract coal less expensively than is currently being done. This equipment is lighter in weight and more efficient than existing coal removal equipment, and requires only one man for its operation. It is expected to dig coal without the need for blasting and may also be used for overburden removal under certain conditions.

An explosive rock ripper capable of more efficient fragmentation of overburden and coal will be designed, built and tested.

Information concerning the current status of surface mining operations in the United States will be analyzed to select critical operations that require new equipment development and technology and changes in equipment system configuration. The objective will be to

increase production and productivity by improving overburden handling, coal loading, and reclamation operations. Studies will be emphasized in three areas:

a) Economic-Engineering Analysis of Surface Coal Mines and Effective Land Reclamation: The objectives are to document the current technology of surface coal mining in the United States, define current problems and possible solutions, and project future developments necessary to assure adequate surface-mined coal.

b) Operations Study of United States Surface Coal Mining Systems: The objective is to gather operational data from selected surface coal mines in the United States and assess the potential production and economic capacity of these specific systems.

c) Equipment Applications and Practices/Surface Coal Mines: The objective is to determine the current practices and technology of surface mining equipment and their application in U.S. coal mines, and identify the industry's equipment needs for effective land reclamation.

Feasibility studies on innovative equipment concepts proposed by industry will be conducted. These proposals are expected to include new techniques for overburden and coal removal, better material handling schemes, and improved methods for grading and reclaiming spoil piles. Problems concerning miner health and safety, highwall and spoil stability, and environmental constraints will also be considered. Depending on the outcome of proposal analyses, prototype development and tests would then be prepared for implementation in FY 77.

Studies will be conducted to determine if there is a need in the coal surface mining industry for large tractor-dozers in excess of 500 horsepower and if construction of these dozers is technically and economically feasible with current technology. The advantages to be

derived from a large tractor-dozer are in leveling spoil banks and ripping overburden, resulting in increased productivity, reduced cost, and longer machine life. The results of these studies together with the information obtained from the specialized reclamation equipment project tests, will be integrated and used to design, develop, and test more technologically advanced reclamation equipment.

A winch-pulled dozer blade, capable of moving in excess of 3,000 cubic yards per hour at an approximate cost of $0.02 per cubic yard, will be designed and fabricated.

Dozer blades will be field tested to determine their effectiveness and use in spoil-leveling operations, and their ability to increase material handling and reclaim strip-mined land to useful productivity. At present, the dozer is the primary piece of equipment used for leveling spoil, but individually it is not capable of moving high volumes of material. The use of 40- and 60-foot angle blades in conjunction with two dozers to increase leveling capacity and reduce safety hazards and leveling costs will be investigated. Field tests will be made in an effort to achieve handling capacities of 7,000 to 9,000 cubic yards per hour at a cost of less than $0.02 per cubic yard.*

An effort to determine the strength, durability, and life of wire rope sizes greater than 2 inches will develop replacement schedules based upon usage and estimated life cycles of the rope. The objective

*The cost savings from the winch-pulled dozer and the big dude dozer are considered to be temporary, and the developed equipment is expected to be phased out by the year 2000. Replacement equipment will more likely be developed from long-term equipment projects.

is to reduce production losses due to unnecessary replacement of wire rope on draglines.

Development of a dragline simulator will be developed that can be used to train dragline operators. This will minimize production losses which result during the six-month on-the-job training period due to the lower operational efficiency of new operators with respect to experienced operators.

9.3.3 R&D Costs

Research and development costs are presented in Table 9-1. Total R&D cost, discounted at 12% to 1975, is $62.8 million.

9.4 Methodology

9.4.1 Assumptions and Limitations

Certain assumptions have been made to simplify the analysis:

- All mines have a 20-year life.
- Stripping shovels and draglines are assumed to have a 20-year life; the entire capital outlay for these items being made in the first year. All other equipment is assumed to have a 10-year life; capital expenditures being made in the first and eleventh years of mine life.

9.4.2 Cost Escalation Factors

- Cost for wages and union welfare are as specified in the current United Mine Workers of America contract, ratified December 5, 1974.
- Costs for operating supplies and power have been increased 29 percent and 100 percent, respectively; the increases based upon Katell's investigation of unit price increases for underground mines.(1)
- Royalties, strip licenses and bonds were assumed to remain constant.

TABLE 9-1

SURFACE MINING SYSTEM FUNDING

	$ MILLIONS BY FISCAL YEAR						
	1975	1976	1977	1978	1979	1980	TOTAL
SURFACE MINING EXTRACTION	4.6	5.0	4.5	5.0	5.0	8.0	32.1
SURFACE MINING EQUIPMENT	4.0	7.5	7.0	8.0	8.0	10.0	44.50
ADVANCED MINING SYSTEMS	0.45	0.9	1.35	1.95	1.95	2.1	8.70
TOTAL	9.05	13.4	12.85	14.95	14.95	20.1	85.30

- Equipment cost estimates were increased 40 percent; based upon M & S Equipment Cost Index for the mining industry.(2)

- Reclamation operations which are accomplished by contract (mulching, liming, fertilizing, and seeding) are assumed to increase 20 percent.

- Cost items, which Katell expresses as a percentage of other costs, are determined by using the same percentage of updated costs (i.e., payroll overhead, indirect labor, and taxes and insurance).

9.4.3 System Parameters

The two surface mining subprograms in the surface mining R&D effort deal with all major surface mining functions. Benefits of systems and equipments to be developed are related to specific mining functions as shown in Table 9-2. The benefit/cost analysis estimates contributions, from mining systems and equipments, to increases in productivity of each mining function, overall productivity increases for the total system, and savings that result from these productivity increases.

The method of mining is the primary basis used in the analysis for considering the benefits that the R&D program will have on different unit operations and, ultimately, on the total system. Three different mining operations were included in the analysis: (1) contour mining-block method in the Eastern Province, (2) area mining in the Interior Provinee, and (3) area mining in the Western Province. A representative mine for each of the three mining methods was used, with different combinations of equipment and manpower used for the unit operations.

TABLE 9-2

APPLICABILITY OF BOM R&D TO SURFACE MINING FUNCTION

X: Primary Function S: Secondary Function	MINING FUNCTION					
	Planning	Overburden Preparation	Overburden Removal	Coal Loading & Hauling	Backfilling & Regrading	Revegetation
Surface Mining Equipment						
1. Vertical Bladed Excavator						
2. Mine & System Development			X			
3. Future Mining Equipment			X	X	X	
4. Big Dude Dozer					X	
5. Improved Rope Efficiency			X			
6. Highwall Mining Study (Auger)				X		
7. Winch-Pulled Dozer Blade					X	
8. Forward Rotating Wheel Excavator				X		
9. Explosive Rock Ripper		X		X		
10. Cross-Pit Material Handling			X		X	
11. Large Tractor Dozer		S	X		X	
12. Dragline Simulator			X			
Surface Mining Extraction & Reclamation System						
1. Area-Longwall Mining of Shallow Coal Deposits			X		X	
2. Reverse Surface Area Mining Method			X		X	
3. Improved Overburden Handling Techniques		X	X		X	
4. Highwall and Outbound Haulage			X		X	
5. Highwall Reduction			X		X	
6. Demo-Block--Cut Method--Appalachia	X	X	X	X	X	
7. Preliminary Planning Rational	X					
8. IR Photography for Mining, Reclamation, and Monitoring	X				X	X
9. Vibration and Blast Noise		X				
10. Reclamantion with Solid Wastes						X

9.4.4 Computation of Benefits

Determination of the benefits of the BOM surface mining R&D effort is accomplished in several steps. The first step is the development of mine models and cost data to represent current systems. These cost data, which are developed in detail in Section 9.6, are then utilized in conjunction with basic production relationships to determine how production costs and savings vary as a function of changes in five critical parameters: productivity, manpower, investment, supplies, and power. The results of these calculations, based upon the production scenarios described in Section 9.1 are plotted in Section 4.5.

These curves are particularly useful in determining the sensitivity of the magnitudes of the benefits to changes in the five key parameters chosen for the analysis: the greater the absolute slope of the curves, the greater the change in the magnitudes of benefits per unit change in the system parameters.

The curves in Section 4.5 represent general relationships of savings and parameters over a wide range of values for those parameters. Estimation of actual benefits to be derived from the R&D program requires the determination of a specific point or range of points on the curves. Determination of these points is accomplished in Section 9.6 by calculating the expected changes in system parameters.

Of the five system parameters, productivity is the most critical for two main reasons. First, changes in productivity affect all

cost components except those which cannot be affected by improvements in mining technology, (e.g., union wlefare payments, royalties, bonds and licenses); changes in each of the other parameters affect only a single cost component. Second, changes in productivity* are the only changes which are sufficiently known at the present time to allow calculation of benefits which have a reasonable confidence level. Changes in input requirements for the other four system parameters are much more poorly defined at present; R&D projects may increase input requirements in one area while reducing requirements in other areas. In contrast, all of the R&D projects are designed to increase productivity (output per unit time).

Because productivity provides significant advantages over the use of other system parameters, it is the basis of the analysis. The other parameters are included qualitatively rather than quantitatively to allow assignment of specific values as they become known.

The increase in productivity that is expected from the R&D surface mining effort is calculated in Section 9.6. First, the impact of individual R&D projects is estimated, on a province basis, for each of the six major mining functions: planning, overburden preparation, overburden removal, coal loading and haulage, backfilling and regrading, and revegetation. The productivity increase is weighted by its estimated market penetration and the individual increases are summed for each mining function. The values for the three provinces are averaged assuming equal weights for each of the three provinces.

*Expressed as R&D goals which will probably be achieved.

These values are weighted by the distribution* of capital and operating costs: the sum of these weighted costs represents the overall increase in productivity expected from the R&D surface mining effort.[1]

9.5 Current System Description

This section contains a brief description of the mining methods chosen to represent current mining technology. For the benefit/cost analysis the country has been divided into three mining provinces: Eastern, Interior, and Western. The Eastern and Interior Provinces are as described in Sections 3.4. The Western Province is developed by combining the Great Plains Province and the Rocky Mountain Province described in Section 3.4.

Descriptions of mine operations are taken from the model mines developed by the Bureau of Mines.[3]

9.5.1 Eastern Province

The Eastern Mining operation is assumed to be located at a hilly site with overburden ranging from zero feet at the outcrop to 100 feet maximum, and a coal-seam thickness of 6 feet. After drilling and blasting, the overburden is removed by 100-cubic-yard shovels. Both the highwall drills and overburden stripping shovels will be supported by bulldozers. The coal is loaded, without shooting, into 120-ton trucks by 15-cubic-yard shovels.

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*Determined by dividing the sum of capital and operating costs for each individual mining function by the total costs for all functions.

9.5.2 Gulf and Interior Province

The Interior mining operation is based on a 6-foot coal bed with an overburden averaging 70 feet, which, after blasting, is removed by 20-cubic-yard electric shovels. The overburden drill and strip shovels are backed up with bulldozers and wheel tractor scrapers.

The 20-cubic-yard electric shovels, supported by front-end loaders for cleanup and auxiliary loading, load the coal without shooting into 120-ton trucks.

9.5.3 Western Province

The Western Province study is based upon a 25-foot seam of coal with an average of 70 feet of overburden. The overburden is blasted and removed by 45-cubic-yard draglines. The coal is then loaded into 120-ton trucks by 15-cubic-yard electric shovels which work with front-end loaders. Drills and draglines are backed up by bulldozers and wheel tractor scrapers.

9.5.4 All Provinces

Characteristics common to all three provinces include:

- All shooting will be done with ammonium nitrate-fuel oil (AN-FO) mixture.
- The mining high voltage will be reduced to 7,200 volts before entering the mine power system. Overburden drills, stripping equipment, coal loading shovels, pumps and lighting equipment will be operated by electrical power.
- Backup equipment for reclamation, road building and road maintenance, service, maintenance, and transportation is essentially the same for each province.

9.5.5 Current System - Costs

- Cost data are developed for two mine sizes in each Province; small mines, whose production is less than or equal to 1

million tons per year (MTY), and large mines, those with a production greater than 1 million tons per year. Cost data for the large mines are based directly upon BOM's estimates. Cost estimates for the small mines are developed by multiplying large-mine costs by 1.43;* based upon BOM's estimates that production costs per ton of coal average 30 percent less for large mines (3-MTY) than for small mines. Cost data have been modified to reflect increases since the BOM publication.

- Cost estimates per million tons of production are determined by dividing total cost estimates by the production capacity in millions of tons.

Resulting cost data for each province, by mine size, are presented in Table 9-3. All data are presented on the basis of 1 million tons per year.

9.6 New System Description

The major objective of the surface mining R&D effort is to improve efficiency and productivity of surface mining operations. The new system to be developed through the R&D effort cannot be fully described at present due to the many uncertainties regarding the outcome of feasibility studies and system development. However, the benefit/cost analysis can be accomplished by analyzing general characteristics of the new system and objectives of the R&D projects.

9.6.1 New Systems - Productivity

Analysis of the projects indicates that the new system will have significantly higher productivity than the current system due to improvements in two major areas: mining systems and equipment. Current systems will be improved through reorganization and integration

*Excluding costs which are independent of mine size, i.e., union welfare, bonds, royalties, licenses and contracted reclamation items.

TABLE 9-3

CURRENT SYSTEM COSTS BY MINE SIZE AND PROVINCE
(In Millions of Dollars Per Million Tons)

COST ITEM	EASTERN PROVINCE		INTERIOR PROVINCE		WESTERN PROVINCE	
	Small	Large	Small	Large	Small	Large
Production payroll	0.428	0.299	0.296	0.207	0.247	0.173
Maintenance payroll	0.230	0.161	0.149	0.104	0.132	0.092
Recl. & Roads payroll	0.050	0.035	0.066	0.046	0.050	0.035
Utility payroll	0.033	0.023	0.033	0.023	0.017	0.012
Operating supplies	1.088	0.761	0.885	0.619	0.646	0.452
Power	0.372	0.260	0.257	0.180	0.172	0.120
Reclamation (contract)	0.034	0.024	0.052	0.036	0.051	0.036
Payroll overhead	0.259	0.181	0.190	0.133	0.156	0.109
Union welfare	1.550	1.550	1.550	1.550	1.550	1.550
Royalty, license, bond	0.570	0.570	0.590	0.590	0.240	0.240
Indirect costs	0.275	0.192	0.215	0.150	0.164	0.115
Taxes and Insurance	0.360	0.252	0.300	0.210	0.120	0.084
SUBTOTAL	5.249	4.308	4.583	3.848	3.545	3.018
Depreciation	1.204	0.843	1.080	.755	.560	.391
TOTAL	6.453	5.151	5.663	4.603	4.105	3.409

of mining functions whenever feasible with special emphasis being placed upon planning procedures, operating procedures, equipment scheduling, and mine layouts. Future mining systems will develop from the analysis and evaluation of innovative mining concepts.

BOM Engineers' estimates of percentage increases in productivity due to improved systems and equipment, for each mining function, are tabulated in Tables 9-4, 9-5, and 9-6. BOM Engineers' estimates of most probable market penetration for each project are presented in Table 9-7. Multiplying the estimated productivity increases by the market penetration determines the productivity increase for the overall market. The productivity increases for the overall market are presented in parentheses. Total productivity increases for the overall market for each mining function are calculated by summing the weighted increases for each project. These totals represent the total increases in productivity which are expected to result from the BOM surface mining R&D effort.

The total productivity increase for the overall new system to be developed through the R&D effort is determined by weighting productivity increases for each mining function by the fraction of overall cost each represents. The overall increase in productivity for the new system is presented in Table 9-8. The Bureau of Mines surface mining R&D program will result in a productivity increase of 126 percent.

TABLE 9-4

ESTIMATED PRODUCTIVITY INCREASES*
EASTERN PROVINCE
(By Percent)

	Planning	Overburden Preparation	Overburden Removal	Coal Loading	Backfilling & Regrading	Revegetation
Surface Mining and Reclamation Systems						
Reserve Surface Area Mining						
Area Longwall Mining						
Improved Overburden Handling			25 (25)		25 (25)	
Highwall or Outbound Haulage			100 (25)		100 (25)	
Highwall Reduction			100 (50)		100 (50)	
Demonstration of Block-Cut					100 (30)	
Reclamation Planning Rationale	100 (100)					
False Color IR Photography	25 (25)					300 (300)
Vibration and Blast Noise		2 (2)				
Reclamation Using Various Solid Wastes						
Surface Mining and Reclamation Equipment						
Vertical Bladed Excavator						
Cross-Pit Material Handling						
Forward Rotating Wheel Excavator				67 (35)		
Explosive Rock Ripper		55 (10)		55 (10)		
Highwall Mining Study[2]						
Mine and System Development			10 (2)	10 (2)	10 (2)	
Future Surface Mining Technology			10 (10)	10 (10)	10 (10)	
Large Tractor Dozer					40 (8)	
Winch-Pulled Dozer Blade					100 (20)	
Specialized Reclamation Equipment						
Big Dude Dozer						
Improved Rope Efficiency						
Dragline Simulator						
TOTALS	125 (125)	57 (12)	245 (112)	142 (57)	485 (170)	300 (300)

*Figures which are not in parentheses represent estimated productivity increases for a portion of the overall market. Figures in parentheses represent the estimated productivity increases for the total market.

TABLE 9-5

ESTIMATED PRODUCTIVITY INCREASES
INTERIOR PROVINCE
(By Percentage)

	Planning	Overburden Preparation	Overburden Removal	Loading	Backfilling & Regrading	Revegetation
Surface Mining and Reclamation Systems						
Reserve Surface Area Mining		10 (9)	10 (9)		67 (57)	
Area Longwall Mining					400 (120)	
Improved Overburden Handling			25 (25)		25 (25)	
Highwall or Outbound Haulage						
Highwall Reduction						
Demonstration of Block-Cut						
Reclamation Planning Rationale	100 (100)					
False Color IR Photography	25 (25)					300 (300)
Vibration and Blast Noise		2 (2)				
Reclamation Using Various Solid Wastes						
Surface Mining and Reclamation Equipment						
Vertical Bladed Excavator		65 (10)	65 (10)	55 (9)		
Cross-Pit Material Handling			65 (65)		20 (20)	
Forward Rotating Wheel Excator				65 (33)		
Explosive Rock Ripper		55 (11)		55 (11)		
Highwall Mining Study[2]						
Mine and System Development			10 (10)	10 (10)	10 (10)	
Large Tractor Dozer					40 (20)	
Winch-Pulled Dozer Blade					100 (50)	
Specialized Reclamation Equipment						
Big Dude Dozer					100 (50)	
Improved Rope Efficiency			5 (4)			
Dragline Simulator			5 (2)			
TOTALS	125 (125)	132 (32)	195 (135)	195 (73)	722 (362)	300 (300)

TABLE 9-6

ESTIMATED PRODUCTIVITY INCREASES
WESTERN PROVINCE
(By Percentage)

	Planning	Overburden Preparation	Overburden Removal	Coal Loading	Backfilling & Regrading	Revegetation
Surface Mining and Reclamation Systems						
Reverse Surface Area Mining		10 (5)	10 (5)		65 (33)	
Area Longwall Mining						
Improved Overburden Handling			25 (25)		25 (25)	
Highwall or Outbound Haulage						
Highwall Reduction						
Demonstration of Block-Cut						
Reclamation Planning Rationale	100 (100)					
False Color IR Photography	25 (25)				5 (5)	300 (300)
Vibration and Blast Noise		2 (2)				
Reclamation Using Various Solid Wastes					20 (20)	
Surface Mining and Reclamation Equipment						
Vertical Bladed Excavator		65 (26)	65 (26)	55 (22)		
Cross-Pit Material Handling			65 (65)		20 (20)	
Forward Rotating Wheel Excavator				65 (33)		
Explosive Rock Ripper		55 (11)		55 (11)		
Highwall Mining Study[2]						
Mine and System Development			10 (10)	10 (10)	10 (10)	
Future Surface Mining Technology			10 (10)	10 (10)	10 (10)	
Large Tractor Dozer					40 (16)	
Winch-Pulled Dozer Blade					100 (40)	
Specialized Reclamation Equipment						
Big Dude Dozer					50 (15)	
Improved Rope Efficiency			5 (2)			
Dragline Simulator			5 (1)			
TOTALS	125 (125)	132 (44)	195 (144)	195 (86)	395 (194)	300 (300)

TABLE 9-7

ESTIMATED MARKET PENETRATION OF SURFACE MINING SYSTEMS AND EQUIPMENT
(By Percentage)

	GEOGRAPHICAL PROVINCE		
	Eastern	Interior	Western
SURFACE MINING EQUIPMENT			
1. Vertical Bladed Excavator		15	50
2. Mine & System Development	20	100	100
3. Future Mining Equipment	100	100	100
4. Big Dude Dozer		50	15
5. Improved Rope Efficiency		80	30
6. Highwall Mining Study (Auger)	15		15
7. Winch-Pulled Dozer Blade	20	50	40
8. Forward Rotating Wheel Excavator	50	50	50
9. Explosive Rock Ripper	20	20	20
10. Cross-Pit Material Handling		100	100
11. Large Tractor Dozer	20	50	40
12. Dragline Simulator		40	15
SURFACE MINING-EXTRACTION AND RECLAMATION SYSTEM			
1. Area-Longwall Mining of Shallow Coal Deposits		30	
2. Reverse Surface Area Mining Method		85	50
3. Improved Overburden Handling Technique	100	100	100
4. Highwall and Outbound Haulage	25		
5. Highwall Reduction	50		
6. Demo-Block Cut Method--Appalachia	100		
7. Pre-Mining Planning Rational	100	100	100
8. IR Photography for Mining, Reclamation, and Monitoring	100	100	100
9. Vibration and Blast Noise		20	
10. Reclamation with Solid Wastes			100

TABLE 9-8

ESTIMATION OF PRODUCTIVITY INCREASES

MINING FUNCTION	ESTIMATED INCREASES IN PRODUCTIVITY				COMPOSITE COST DISTRIBUTION *	WEIGHTED INCREASE IN PRODUCTIVITY**
	EASTERN	INTERIOR	WESTERN	AVERAGE		
Planning	125	125	125	125	0.03	3.8
Overburden Preparation	12	32	44	29	0.09	2.6
Overburden Removal	112	135	144	130	0.61	79.3
Coal Loading	57	73	86	72	0.15	10.8
Backfilling and Regrading	170	362	194	242	0.11	26.6
Revegetation	300	300	300	300	0.01	3.0
OVERALL ESTIMATED INCREASE IN PRODUCTIVITY						126.1

* Based upon Table 9-9.

** Calculated by multiplying the average estimated increase in productivity by the composite cost distribution.

9.6.2 New System - Costs

The distribution of capital and operating costs that is used in this analysis is based upon general models and is presented in Table 9-9. Unit cost items that will not be affected by changes in productivity (i.e., union wlefare, royalty, taxes, and bonds) have been excluded.

9.6.3 New System - Markets

a) Market Potential. Market Potential of new surface mining systems is assumed to be 100% of the surface mining market in each region.

b) Market Penetration and Penetration Rate. BOM estimates of market penetration of new surface mining systems and equipments have been given in Table 9-7. It is assumed that new systems and equipments will penetrate their markets at a constant incremental rate to achieve full market penetration over a period of ten years.

9.6.4 System Availability

BOM engineers estimate that all projects but two* will be available by 1981. Therefore, the new system to be developed through the R&D program is assumed to have an availability date of 1981.

9.7 Analysis and Results

The benefit/cost ratios for the surface mining R&D effort have been developed by utilizing the surface mine curve in Section 4.0 and the parameter values developed in Section 9.6. The benefit/cost ratios are presented in Table 9-10. The present value of the R&D cost is $62.8 million. The sensitivity of these ratios to variations

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*Mine and Systems Development and Future Surface Mining Technology.

TABLE 9-9

ESTIMATED MINING COSTS BY MINING FUNCTION

(Percentage of Total Costs)

MINING FUNCTION	CAPITAL COST	OPERATING COST	COMPOSITE*
Planning	0	4	3
Overburden Preparation	5	10	9
Overburden Removal	65	60	61
Coal Loading & Haulage	15	15	15
Backfilling & Regrading	14	10	11
Revegetation	1	1	1
TOTALS	100	100	100

* Based upon a capital to operating cost ratio of 1:3.

TABLE 9-10

BENEFIT/COST RATIOS FOR SURFACE MINING R&D

WITH P = +126%; M = 0%; S = 0%; I= 0%;
MPL = 100%; MPT = 8 years; AT = 1981*

BENEFIT/COST RATIOS	Scenario 1	Scenario 2
Total Cost Savings ($/R&D $)	50.3	85.2
Capital Cost Savings ($/R&D $)	19.1	31.7
Man-days Saved Due to Reduction in Non-Fatal Accidents/R&D $	0.05	0.09
Man-days Saved Due to Reduction in Fatal Accidents/R&D $	0.11	0.20
Reduction in Production Costs ($/ton)	1.78	1.51

* Meanings of abbreviations explained in page 1 of Section 4.0.

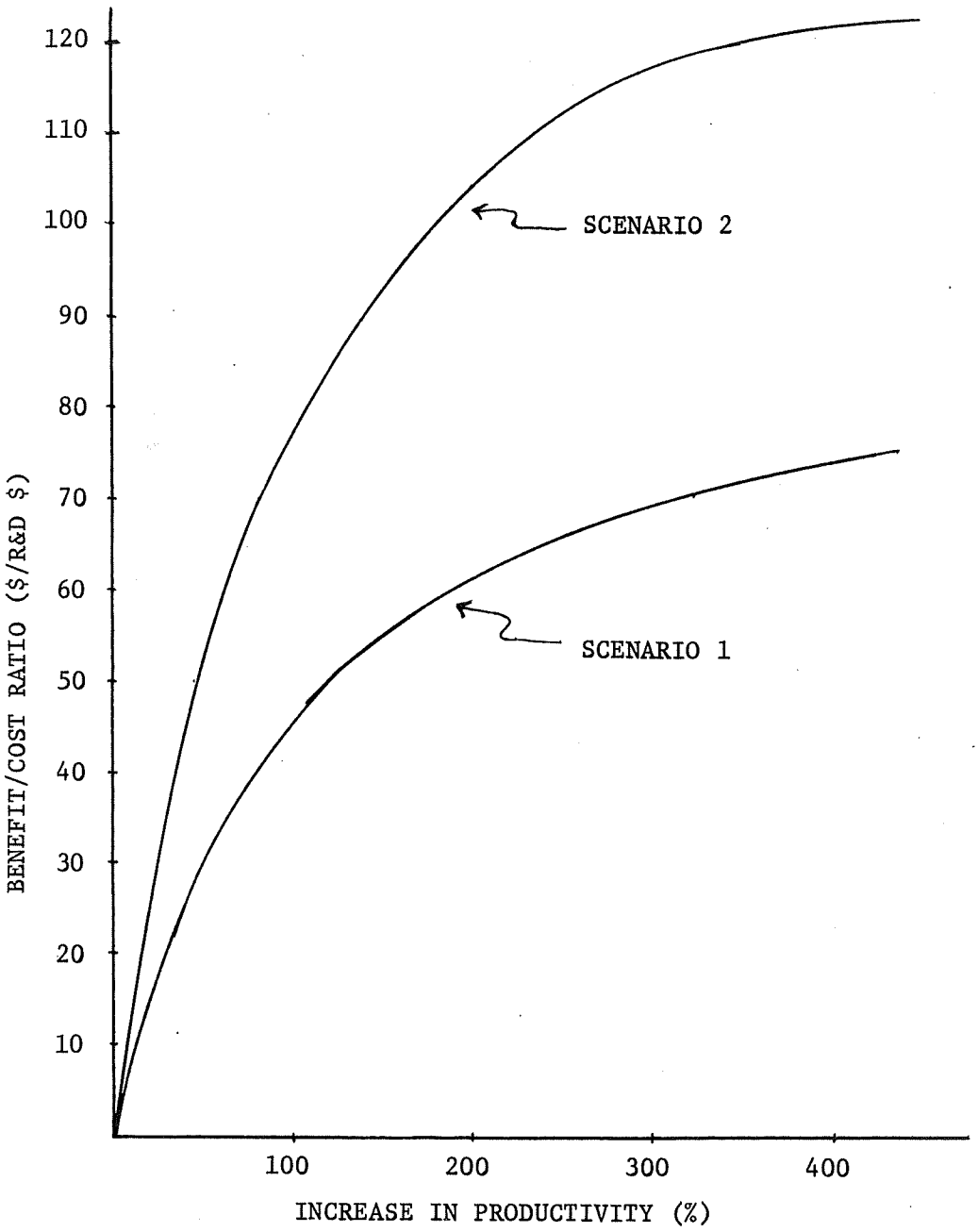

FIGURE 9-1

BENEFIT/COST RATIO VS. PRODUCTIVITY, WITH ALL OTHER SYSTEM PARAMETERS UNCHANGED

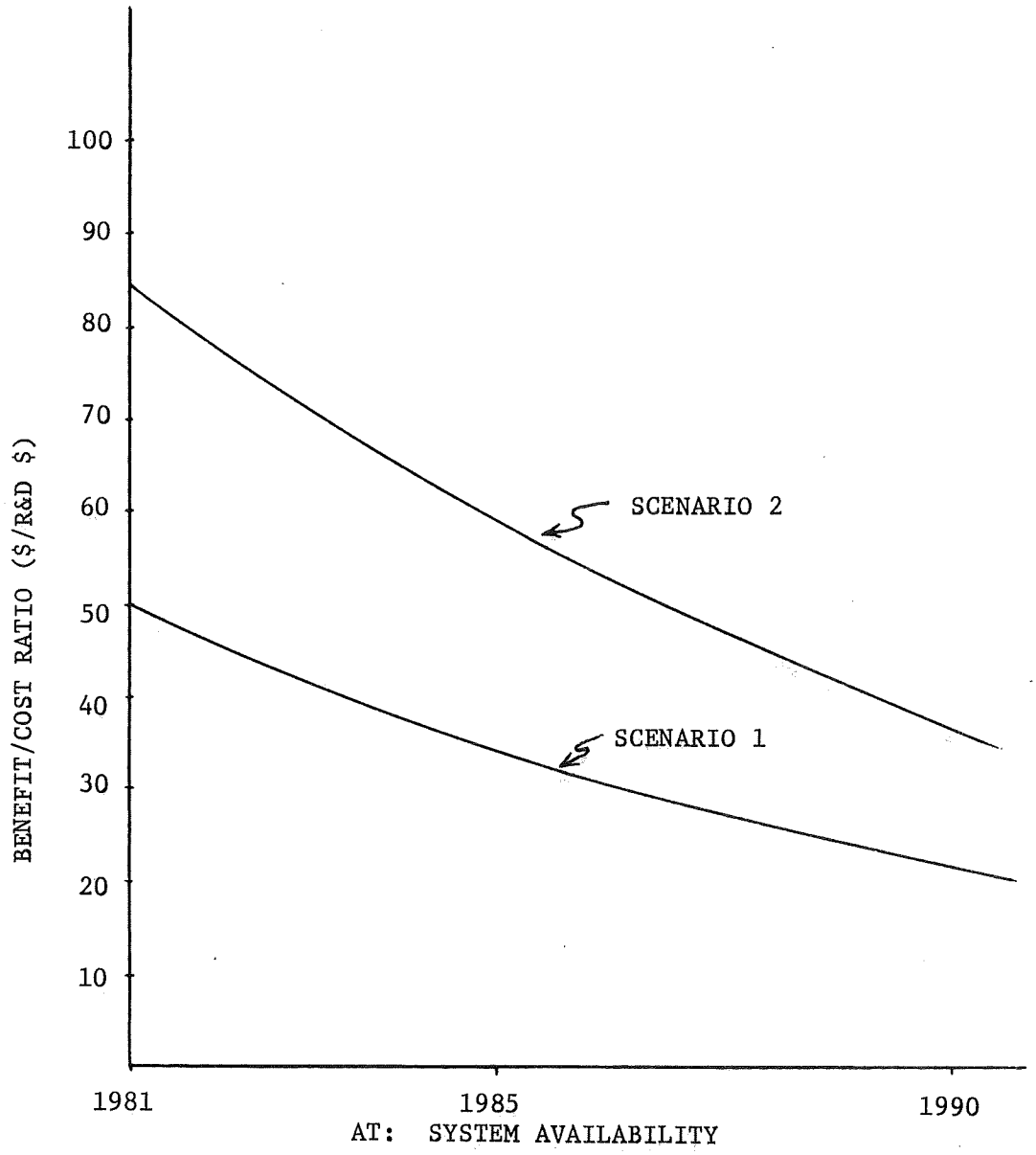

FIGURE 9-2

EFFECT OF SYSTEM AVAILABILITY
ON BENEFIT/COST RATIOS

in productivity and system availability date is shown in Figures 9-1 and 9-2, respectively.

This analysis has not determined the probable effects of the R&D program on manpower, supplies, and capital investment for the reasons described in Section 9.5. However, the general effect of varying these parameters is illustrated in Figures 9-3 and 9-4, which present the increases in savings by system parameter. The figures indicate several significant conclusions:

a) The marginal increase in savings per unit productivity is high for productivity increases of up to 200 percent, but decreases rapidly thereafter.

b) Productivity is the most critical system parameter. Increases in productivity produce greater savings than increases in any of the other system parameters.

c) Savings due to decreases in requirements for manpower, supplies, and capital decrease as productivity increases: a result of the fact that an increase in productivity, P, reduces savings due to reduced manpower, capital, and/or supplies by a factor of 1 + P.

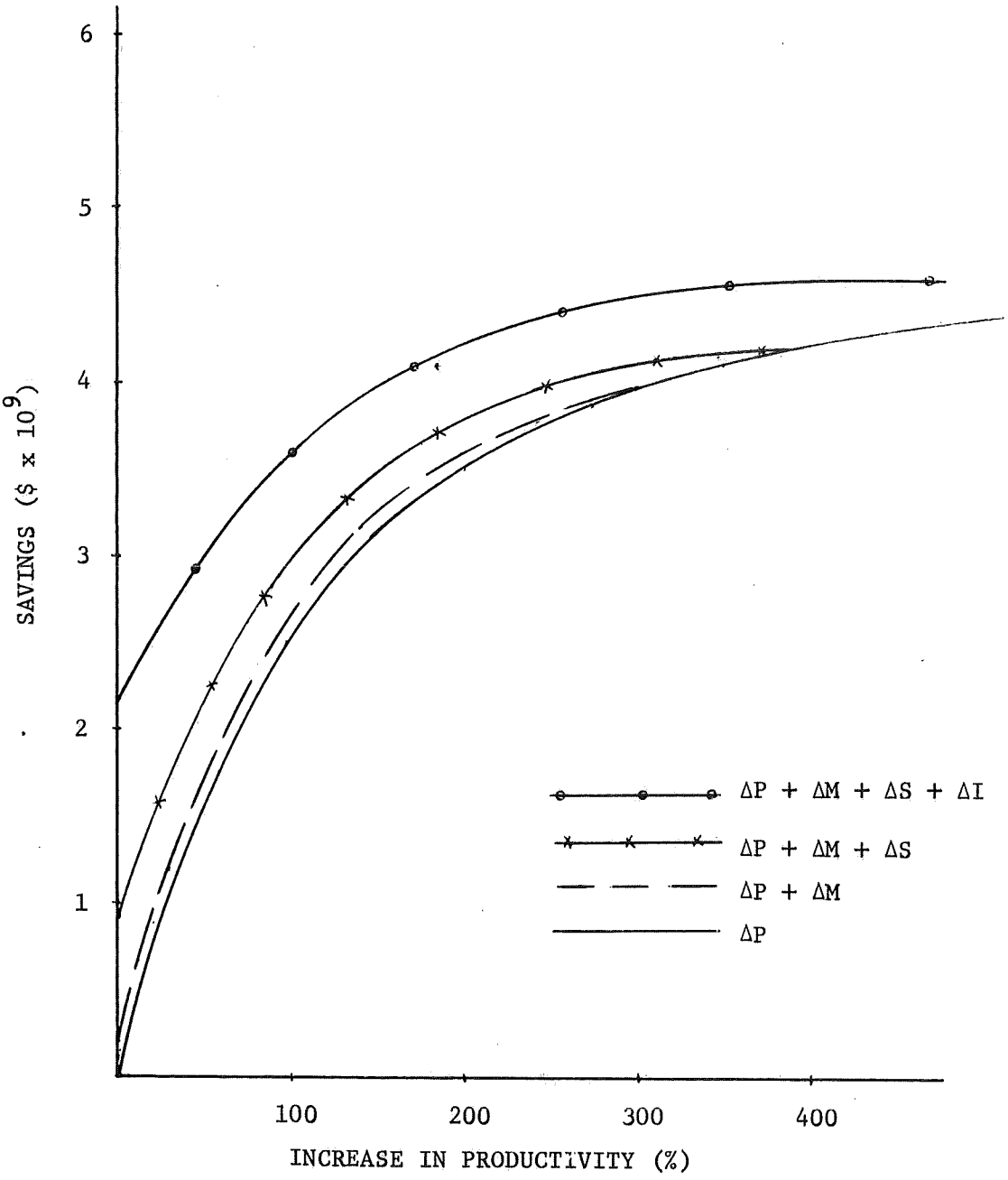

FIGURE 9-3
SAVINGS VS. PRODUCTIVITY
BY SYSTEM PARAMETER
SCENARIO 1

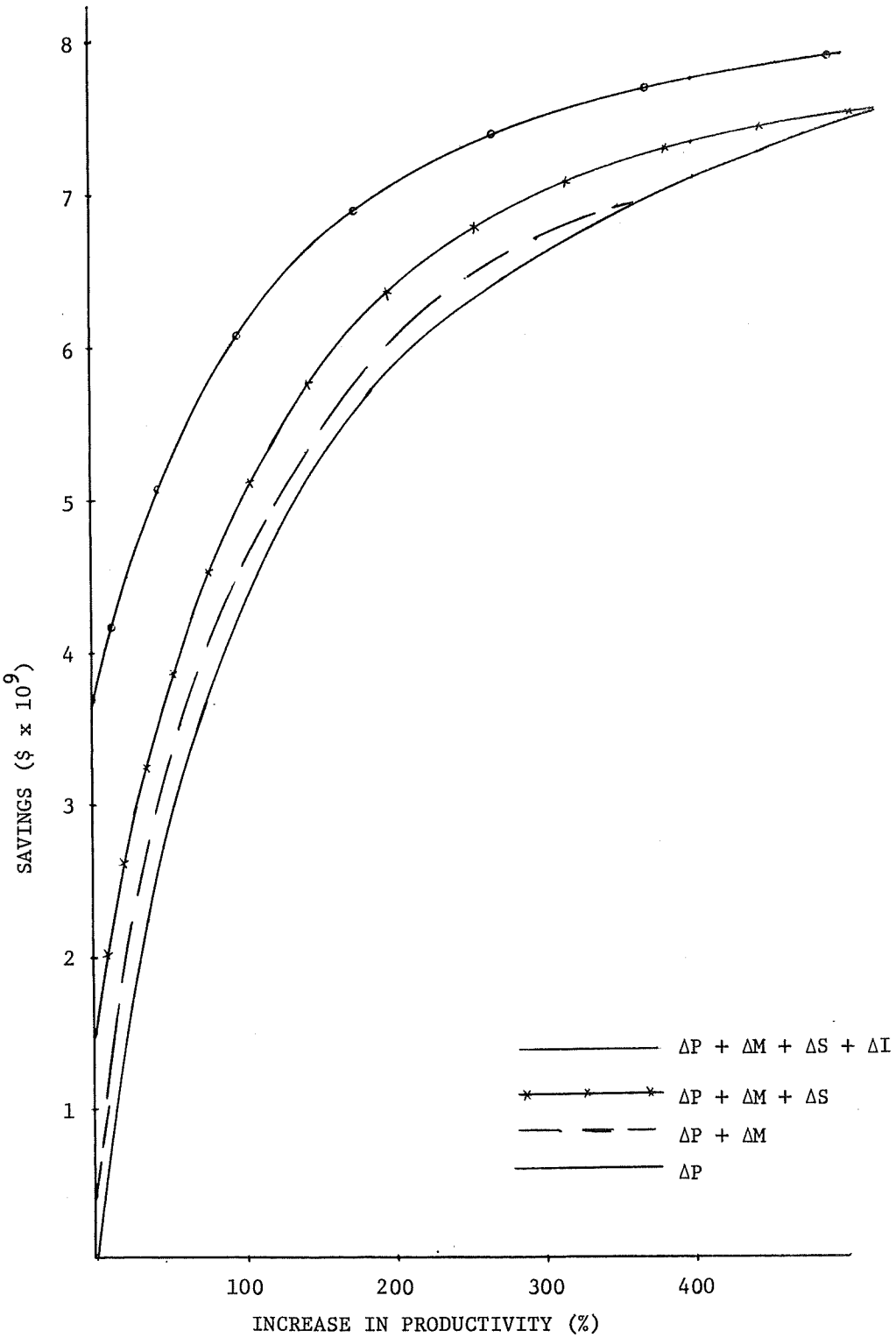

FIGURE 9-4

SAVINGS VS. PRODUCTIVITY
BY SYSTEM PARAMETER
SCENARIO 2

SECTION 9.0 REFERENCES

1. Katell, unpublished, 1975 data.

2. Chemical Engineering, March 3, 1975.

3. Katell, 1974 (a).

APPENDIX A

TABLE A-1

SCENARIO 1 - LOW PRODUCTION

ANNUAL COAL PRODUCTION IN MILLIONS OF TONS BY MINE TYPE
FOR ALL PROVINCES AND MINE SIZES

YEAR	MINE TYPE		TOTAL
	UNDERGROUND	SURFACE	
1975	336	349	685
1976	348	379	727
1977	360	408	768
1978	372	438	810
1979	384	467	851
1980	396	496	892
1981	409	525	934
1982	421	554	976
1983	433	583	1017
1984	445	612	1059
1985	459	641	1100
1986	471	669	1140
1987	483	697	1180
1988	495	725	1220
1989	507	752	1260
1990	520	780	1300
1991	530	814	1344
1992	540	849	1388
1993	550	883	1432
1994	560	917	1476
1995	568	952	1520
1996	580	987	1571
1997	592	1022	1618
1998	604	1057	1666
1999	616	1092	1714

TABLE A-1 (Continued)

SCENARIO 1 - LOW PRODUCTION

ANNUAL COAL PRODUCTION IN MILLIONS OF TONS BY MINE TYPE
FOR ALL PROVINCES AND MINE SIZES

YEAR	MINE TYPE		TOTAL
	UNDERGROUND	SURFACE	
2000	630	1127	1757
2001	644	1159	1804
2002	658	1191	1850
2003	672	1224	1897
2004	686	1256	1943
2005	702	1288	1990
2006	714	1320	2034
2007	726	1352	2078
2008	738	1384	2122
2009	750	1416	2166
2010	762	1448	2211
2011	773	1483	2257
2012	784	1518	2304
2013	795	1553	2350
2014	806	1588	2397
2015	821	1623	2444
2016	833	1653	2486
2017	845	1683	2527
2018	857	1713	2569
2019	869	1743	2611
2020	880	1772	2652

TABLE A-2

SCENARIO 2 - HIGH PRODUCTION

ANNUAL COAL PRODUCTION IN MILLIONS OF TONS BY MINE TYPE
FOR ALL PROVINCES AND MINE SIZES

YEAR	MINE TYPE		TOTAL
	UNDERGROUND	SURFACE	
1975	343	356	700
1976	353	397	750
1977	370	430	800
1978	387	458	845
1979	410	490	900
1980	428	536	964
1981	455	585	1040
1982	482	643	1125
1983	509	721	1230
1984	534	811	1345
1985	563	901	1464
1986	589	961	1550
1987	614	1036	1650
1988	640	1125	1765
1989	665	1195	1860
1990	687	1309	1996
1991	704	1366	2070
1992	714	1451	2165
1993	727	1528	2255
1994	735	1610	2345
1995	743	1654	2397
1996	755	1745	2500
1997	772	1823	2595
1998	790	1885	2675
1999	807	1953	2760

TABLE A-2 (Continued)

SCENARIO 2 - HIGH PRODUCTION

ANNUAL COAL PRODUCTION IN MILLIONS OF TONS BY MINE TYPE
FOR ALL PROVINCES AND MINE SIZES

YEAR	MINE TYPE		TOTAL
	UNDERGROUND	SURFACE	
2000	818	2006	2824
2001	831	2069	2900
2002	840	2140	2980
2003	850	2190	3040
2004	858	2257	3115
2005	867	2310	3177
2006	873	2372	3245
2007	878	2427	3305
2008	884	2491	3375
2009	890	2535	3425
2010	894	2569	3463
2011	900	2635	3535
2012	905	2695	3600
2013	910	2740	3650
2014	915	2790	3705
2015	919	2832	3752
2016	926	2884	3810
2017	933	2937	3870
2018	940	3000	3940
2019	947	3048	3995
2020	952	3090	4042

TABLE A-3

SCENARIO 1 - LOW PRODUCTION

NEW MINES BY PROVINCE, TYPE, CAPACITY, AND YEAR
(NEW MINES EXPRESSED IN TERMS OF MILLION TONS/YEAR)
-AVERAGE PRODUCTIVE MINE LIFE = 20 YEARS-

EASTERN PROVINCE

	MINE TYPE						
	UNDERGROUND			SURFACE			
	MINE SIZE (MTY)						
YEAR	<0.5	1.0	3.0	<0.5	1.0	3.0	5.0
1975	5.0	9.0	11.6	3.1	1.2	1.8	8.1
1976	5.0	7.6	9.8	3.0	2.4	2.9	13.1
1977	5.0	7.6	9.8	2.9	2.3	2.8	12.5
1978	5.0	7.6	9.8	2.9	2.3	2.8	12.5
1979	5.0	7.6	9.8	2.7	2.1	2.6	11.7
1980	5.0	7.6	9.8	2.7	2.1	2.6	11.4
1981	5.0	7.6	10.2	2.6	2.0	2.5	11.1
1982	5.0	7.6	9.8	2.6	2.0	2.5	11.1
1983	5.0	7.6	9.8	2.6	2.0	2.5	11.1
1984	5.0	7.6	9.8	2.6	2.0	2.5	11.1
1985	5.0	7.6	9.8	2.5	2.0	2.5	10.8
1986	5.0	7.6	9.8	2.5	1.9	2.4	10.6
1987	5.0	7.6	9.8	2.5	1.9	2.4	10.6
1988	5.0	7.6	9.8	2.5	1.9	2.4	10.6
1989	5.0	7.6	9.8	2.5	1.9	2.3	10.3
1990	5.0	7.9	10.2	2.5	1.9	2.4	10.6
1991	5.0	7.0	8.9	2.5	1.9	2.8	12.2
1992	5.0	7.0	8.9	2.5	2.2	2.8	12.5
1993	5.0	7.0	8.9	2.5	2.3	2.8	12.2
1994	5.0	7.0	8.9	2.5	2.2	2.8	12.2
1995	5.0	11.8	15.1	2.5	3.0	3.7	16.4
1996	5.0	11.8	15.1	2.5	3.8	4.7	20.8
1997	5.0	11.8	15.1	2.5	3.8	4.7	20.8

TABLE A-3 (Continued)

SCENARIO 1 - LOW PRODUCTION

NEW MINES BY PROVINCE, TYPE, CAPACITY, AND YEAR
(NEW MINES EXPRESSED IN TERMS OF MILLION TONS/YEAR)
-AVERAGE PRODUCTIVE MINE LIFE = 20 YEARS-

EASTERN PROVINCE

	MINE TYPE						
	UNDERGROUND			SURFACE			
	MINE SIZE (MTY)						
YEAR	< 0.5	1.0	3.0	<0.5	1.0	3.0	5.0
1998	5.0	11.8	15.1	2.5	3.8	4.7	20.8
1999	5.0	11.8	15.1	2.5	3.7	4.6	20.6
2000	5.0	12.3	16.1	2.5	3.7	4.6	20.6
2001	5.0	12.9	16.6	2.5	3.6	4.4	19.7
2002	5.0	12.3	16.1	2.5	3.6	4.4	19.7
2003	5.1	12.7	16.3	2.5	3.7	4.5	20.0
2004	5.1	12.7	16.3	2.5	3.6	4.4	19.7
2005	5.1	13.4	17.2	2.5	3.6	4.4	19.7
2006	5.1	12.0	15.3	2.5	3.5	4.4	19.4
2007	5.1	12.0	15.3	2.5	3.5	4.3	18.9
2008	5.1	12.0	15.3	2.5	3.5	4.3	18.9
2009	5.1	12.0	15.3	2.5	3.4	4.2	18.7
2010	5.1	12.3	15.9	2.5	3.5	4.3	18.9
2011	5.1	10.9	14.0	2.5	3.9	4.8	21.4
2012	5.1	10.9	14.0	2.5	4.0	4.9	21.6
2013	5.1	10.9	14.0	2.5	3.9	4.8	21.4
2014	5.1	10.9	14.0	2.5	3.9	4.8	21.4
2015	5.1	17.3	22.2	2.5	4.6	5.7	25.4
2016	5.1	16.3	20.8	2.5	5.2	6.4	28.4
2017	5.1	16.3	20.8	2.5	5.2	6.4	28.4
2018	5.1	16.3	20.8	2.5	5.2	6.4	28.4
2019	5.1	16.3	20.8	2.5	5.1	6.3	28.1
2020	5.1	16.6	21.3	2.5	5.1	6.3	28.1

TABLE A-3 (Continued)

SCENARIO 1 - LOW PRODUCTION

NEW MINES BY PROVINCE, TYPE, CAPACITY, AND YEAR
(NEW MINES EXPRESSED IN TERMS OF MILLION TONS/YEAR)
-AVERAGE PRODUCTIVE MINE LIFE = 20 YEARS-

INTERIOR PROVINCE

	MINE TYPE						
	UNDERGROUND			SURFACE			
	MINE SIZE (MTY)						
YEAR	<0.5	1.0	3.0	<0.5	1.0	3.0	5.0
1975	1.0	1.8	2.3	2.2	1.1	1.3	5.8
1976	1.0	1.5	2.0	2.2	1.8	2.2	9.7
1977	1.0	1.5	2.0	2.2	1.8	2.2	9.7
1978	1.0	1.5	2.0	2.1	1.7	2.1	9.4
1979	1.0	1.5	2.0	2.1	1.7	2.1	9.2
1980	1.0	1.5	2.0	2.1	1.7	2.1	9.2
1981	1.0	1.6	2.0	2.1	1.7	2.1	9.2
1982	1.0	1.5	2.0	2.1	1.6	2.0	8.9
1983	1.0	1.5	2.0	2.1	1.6	2.0	8.9
1984	1.0	1.5	2.0	2.0	1.6	2.0	8.6
1985	1.0	1.5	2.0	2.0	1.6	2.0	8.6
1986	1.0	1.5	2.0	2.0	1.5	1.9	8.4
1987	1.0	1.5	2.0	2.0	1.5	1.9	8.4
1988	1.0	1.5	2.0	2.0	1.5	1.9	8.4
1989	1.0	1.5	2.0	2.0	1.5	1.8	8.2
1990	1.0	1.6	2.0	2.0	1.5	1.9	8.4
1991	1.0	1.4	1.8	2.0	1.8	2.2	9.7
1992	1.0	1.4	1.8	2.0	1.8	2.2	10.0
1993	.9	1.3	1.7	2.0	1.7	2.1	9.4
1994	.9	1.3	1.7	2.0	1.7	2.1	9.4
1995	.9	2.2	2.8	2.0	2.3	2.9	12.6
1996	.9	2.2	2.8	2.0	2.9	3.6	16.0
1997	.9	2.2	2.8	2.0	2.9	3.6	16.0

TABLE A-3 (Continued)

SCENARIO 1 - LOW PRODUCTION

NEW MINES BY PROVINCE, TYPE, CAPACITY, AND YEAR
(NEW MINES EXPRESSED IN TERMS OF MILLION TONS/YEAR)
-AVERAGE PRODUCTIVE MINE LIFE = 20 YEARS-

INTERIOR PROVINCE

	MINE TYPE						
	UNDERGROUND			SURFACE			
	MINE SIZE (MTY)						
YEAR	<0.5	1.0	3.0	<0.5	1.0	3.0	5.0
1998	.9	2.2	2.8	2.0	2.9	3.6	16.0
1999	.9	2.2	2.8	2.0	2.9	3.6	15.8
2000	.9	2.4	3.0	2.0	2.9	3.6	15.8
2001	.9	2.4	3.1	2.0	2.8	3.4	15.2
2002	.9	2.4	3.0	2.0	2.8	3.4	15.2
2003	.9	2.2	2.8	2.0	2.8	3.5	15.4
2004	.9	2.2	2.8	2.0	2.8	3.4	15.1
2005	.9	2.3	3.0	2.0	2.8	3.4	15.1
2006	.9	2.1	2.6	2.0	2.7	3.4	14.9
2007	.9	2.1	2.6	2.0	2.7	3.4	14.9
2008	.9	2.1	2.6	2.0	2.7	3.4	14.9
2009	.9	2.1	2.6	2.0	2.7	3.3	15.7
2010	.9	2.1	2.7	2.0	2.7	3.4	14.9
2011	.9	1.9	2.4	2.0	3.1	3.8	16.9
2012	.9	1.9	2.4	2.0	3.1	3.8	17.1
2013	.9	1.9	2.4	2.0	3.1	3.8	16.9
2014	.9	1.9	2.4	2.0	3.1	3.8	16.9
2015	.9	3.0	3.8	1.9	3.5	4.4	19.4
2016	.9	2.8	3.6	1.9	4.0	4.9	21.7
2017	.9	2.8	3.6	1.9	4.0	4.9	21.7
2018	.9	2.8	3.6	1.9	4.0	4.9	21.7
2019	.9	2.8	3.6	1.9	3.9	4.8	21.5
2020	.9	2.9	3.7	1.9	3.9	4.8	21.5

TABLE A-3 (Continued)

SCENARIO 1 – LOW PRODUCTION

NEW MINES BY PROVINCE, TYPE, CAPACITY, AND YEAR
(NEW MINES EXPRESSED IN TERMS OF MILLION TONS/YEAR)
-AVERAGE PRODUCTIVE MINE LIFE = 20 YEARS-

GREAT PLAINS PROVINCE

	MINE TYPE						
	UNDERGROUND			SURFACE			
	MINE SIZE (MTY)						
YEAR	<0.5	1.0	3.0	<0.5	1.0	3.0	5.0
1975	.1	.1	.1	.8	.4	.5	2.2
1976	.1	.1	.1	.9	.7	.9	4.0
1977	.1	.1	.1	1.0	.8	1.0	4.6
1978	.1	.1	.1	1.1	.9	1.1	4.8
1979	.1	.1	.1	1.2	1.0	1.2	5.3
1980	.1	.1	.1	1.3	1.0	1.3	5.6
1981	.1	.1	.1	1.4	1.1	1.3	5.8
1982	.1	.1	.1	1.4	1.1	1.4	6.1
1983	.1	.1	.1	1.4	1.1	1.4	6.1
1984	.1	.1	.1	1.5	1.2	1.4	6.4
1985	.1	.1	.1	1.6	1.2	1.5	6.7
1986	.1	.1	.1	1.6	1.2	1.5	6.5
1987	.1	.1	.1	1.6	1.2	1.5	6.5
1988	.1	.1	.1	1.6	1.2	1.5	6.5
1989	.1	.1	.1	1.6	1.2	1.4	6.3
1990	.1	.1	.1	1.6	1.2	1.5	6.5
1991	.1	.1	.1	1.6	1.4	1.7	7.5
1992	.1	.1	.1	1.6	1.4	1.7	7.7
1993	.1	.1	.1	1.6	1.4	1.8	7.9
1994	.1	.1	.1	1.6	1.4	1.8	7.9
1995	.1	.1	.2	1.6	1.9	2.4	10.5
1996	.1	.1	.2	1.6	2.5	3.0	13.4
1997	.1	.1	.2	1.6	2.5	3.0	13.4

TABLE A-3 (Continued)

SCENARIO 1 - LOW PRODUCTION

NEW MINES BY PROVINCE, TYPE, CAPACITY, AND YEAR
(NEW MINES EXPRESSED IN TERMS OF MILLION TONS/YEAR)
-AVERAGE PRODUCTIVE MINE LIFE = 20 YEARS-

GREAT PLAINS PROVINCE

	MINE TYPE						
	UNDERGROUND			SURFACE			
	MINE SIZE (MTY)						
YEAR	< 0.5	1.0	3.0	< 0.5	1.0	3.0	5.0
1998	.1	.1	.2	1.6	2.5	3.0	13.4
1999	.1	.1	.2	1.6	2.4	3.0	13.2
2000	.1	.2	.2	1.6	2.4	3.0	13.2
2001	.1	.2	.2	1.6	2.3	2.9	12.7
2002	.1	.2	.2	1.6	2.3	2.9	12.7
2003	.1	.2	.2	1.6	2.4	2.9	12.8
2004	.1	.2	.2	1.6	2.3	2.9	12.7
2005	.1	.2	.2	1.6	2.3	2.9	12.7
2006	.1	.1	.2	1.6	2.3	2.8	12.5
2007	.1	.1	.2	1.6	2.3	2.8	12.5
2008	.1	.1	.2	1.6	2.3	2.8	12.5
2009	.1	.1	.2	1.6	2.2	2.8	12.3
2010	.1	.2	.2	1.6	2.3	2.8	12.5
2011	.1	.1	.2	1.6	2.6	3.2	14.1
2012	.1	.1	.2	1.6	2.6	3.2	14.2
2013	.1	.1	.2	1.6	2.6	3.2	14.1
2014	.1	.1	.2	1.7	2.7	3.3	14.6
2015	.1	.2	.3	1.7	3.2	3.9	17.4
2016	.1	.2	.3	1.7	3.6	4.4	19.4
2017	.1	.2	.3	1.7	3.6	4.4	19.4
2018	.1	.2	.3	1.7	3.6	4.4	19.4
2019	.1	.2	.3	1.7	3.5	4.3	19.2
2020	.1	.2	.3	1.7	3.5	4.3	19.2

TABLE A-3 (Continued)

SCENARIO 1 - LOW PRODUCTION

NEW MINES BY PROVINCE, TYPE, CAPACITY, AND YEAR
(NEW MINES EXPRESSED IN TERMS OF MILLION TONS/YEAR)
-AVERAGE PRODUCTIVE MINE LIFE = 20 YEARS-

ROCKY MOUNTAIN PROVINCE

	MINE TYPE						
	UNDERGROUND			SURFACE			
	MINE SIZE (MTY)						
YEAR	<0.5	1.0	3.0	<0.5	1.0	3.0	5.0
1975	.2	.3	.4	.4	.2	.2	1.0
1976	.2	.3	.4	.4	.3	.4	1.7
1977	.2	.3	.4	.4	.3	.3	1.7
1978	.2	.3	.4	.4	.3	.4	1.7
1979	.2	.3	.4	.4	.3	.4	1.7
1980	.2	.3	.4	.4	.3	.4	1.7
1981	.2	.3	.4	.4	.3	.4	1.7
1982	.2	.3	.4	.4	.3	.4	1.7
1983	.2	.3	.4	.4	.3	.4	1.7
1984	.2	.3	.4	.4	.3	.4	1.7
1985	.2	.3	.4	.4	.3	.4	1.7
1986	.2	.3	.4	.4	.3	.4	1.6
1987	.2	.3	.4	.4	.3	.4	1.6
1988	.2	.3	.4	.4	.3	.4	1.6
1989	.2	.3	.4	.4	.3	.4	1.6
1990	.2	.3	.4	.4	.3	.4	1.6
1991	.2	.3	.4	.4	.3	.4	1.9
1992	.2	.3	.4	.4	.3	.4	1.9
1993	.3	.4	.4	.4	.3	.4	1.9
1994	.3	.4	.4	.4	.3	.4	1.9
1995	.3	.6	.8	.4	.5	.4	2.5
1996	.3	.6	.8	.4	.6	.6	3.2
1997	.3	.6	.8	.4	.6	.7	3.2

TABLE A-3 (Concluded)

SCENARIO 1 - LOW PRODUCTION
NEW MINES BY PROVINCE, TYPE, CAPACITY, AND YEAR
(NEW MINES EXPRESSED IN TERMS OF MILLION TONS/YEAR)
-AVERAGE PRODUCTIVE MINE LIFE = 20 YEARS-

ROCKY MOUNTAIN PROVINCE

	MINE TYPE						
	UNDERGROUND			SURFACE			
	MINE SIZE (MTY)						
YEAR	<0.5	1.0	3.0	<0.5	1.0	3.0	5.0
1998	.3	.6	.8	.4	.6	.7	3.2
1999	.3	.6	.8	.4	.6	.7	3.2
2000	.3	.6	.8	.4	.6	.7	3.2
2001	.3	.6	.8	.4	.6	.7	3.0
2002	.3	.6	.8	.4	.6	.7	3.0
2003	.3	.6	.8	.4	.6	.7	3.1
2004	.3	.6	.8	.4	.6	.7	3.0
2005	.3	.7	.8	.4	.6	.7	3.0
2006	.3	.6	.8	.4	.5	.7	3.0
2007	.3	.6	.8	.4	.5	.7	3.0
2008	.3	.6	.8	.4	.5	.7	3.0
2009	.3	.6	.8	.4	.5	.7	2.9
2010	.3	.6	.8	.4	.5	.7	3.0
2011	.3	.5	.7	.4	.6	.8	3.4
2012	.3	.5	.7	.4	.6	.8	3.4
2013	.3	.5	.7	.4	.6	.8	3.4
2014	.3	.5	.7	.4	.6	.8	3.4
2015	.3	.9	1.1	.4	.7	.9	4.0
2016	.3	.8	1.0	.4	.8	1.0	4.5
2017	.3	.8	1.0	.4	.8	1.0	4.5
2018	.3	.8	1.0	.4	.8	1.0	4.5
2019	.3	.8	1.0	.4	.8	1.0	4.4
2020	.3	.8	1.1	.4	.8	1.0	4.4

TABLE A-4

SCENARIO 2 - HIGH PRODUCTION

NEW MINES BY PROVINCE, TYPE, CAPACITY, AND YEAR
(NEW MINES EXPRESSED IN TERMS OF MILLION TONS/YEAR)
-AVERAGE PRODUCTIVE MINE LIFE = 20 YEARS-

EASTERN PROVINCE

	MINE TYPE						
	UNDERGROUND			SURFACE			
	MINE SIZE (MTY)						
YEAR	<0.5	1.0	3.0	<0.5	1.0	3.0	5.0
1975	5.0	11.5	14.7	3.1	2.0	2.4	10.8
1976	5.0	7.0	8.9	2.9	2.9	3.6	16.0
1977	5.0	9.4	12.1	2.8	2.4	3.0	13.2
1978	5.0	9.4	12.1	2.7	2.1	2.6	11.4
1979	5.0	11.4	14.7	2.7	2.3	2.9	12.6
1980	5.0	9.8	12.5	2.7	3.0	3.8	16.6
1981	5.0	12.9	16.6	2.5	3.0	3.7	16.4
1982	5.0	12.9	16.6	2.4	3.3	4.0	17.9
1983	5.0	12.9	16.6	2.3	4.0	4.9	21.9
1984	5.0	12.2	15.7	2.1	4.3	5.3	23.5
1985	5.0	13.6	17.4	2.1	4.2	5.2	23.0
1986	5.0	12.6	16.1	2.0	2.8	3.4	15.2
1987	5.0	12.0	15.5	2.0	3.3	4.1	18.2
1988	5.0	12.4	15.9	1.9	3.7	4.6	20.4
1989	4.9	11.9	15.3	1.9	3.0	3.7	16.5
1990	4.9	11.5	14.8	1.8	4.5	5.6	24.7
1991	4.9	8.5	10.8	1.8	2.5	3.1	13.5
1992	4.9	6.8	8.7	1.8	3.4	4.1	18.3
1993	4.9	7.7	9.9	1.8	3.1	3.8	16.7
1994	4.9	6.0	7.7	1.7	3.1	3.8	17.0
1995	4.9	14.1	18.1	1.7	2.6	3.2	14.1
1996	4.9	10.4	13.3	1.7	4.8	5.9	26.1
1997	4.9	14.8	18.9	1.7	4.1	5.0	22.4

TABLE A-4 (Continued)

SCENARIO 2 - HIGH PRODUCTION

NEW MINES BY PROVINCE, TYPE, CAPACITY, AND YEAR
(NEW MINES EXPRESSED IN TERMS OF MILLION TONS/YEAR)
-AVERAGE PRODUCTIVE MINE LIFE = 20 YEARS-

EASTERN PROVINCE

	MINE TYPE						
	UNDERGROUND			SURFACE			
	MINE SIZE (MTY)						
YEAR	<0.5	1.0	3.0	<0.5	1.0	3.0	5.0
1998	4.8	14.9	19.2	1.6	3.3	4.0	17.8
1999	4.8	16.6	21.3	1.6	3.6	4.4	19.6
2000	4.8	12.9	16.6	1.6	3.6	4.4	19.6
2001	4.8	16.6	21.3	1.6	3.8	4.7	20.8
2002	4.8	15.3	19.5	1.6	4.3	5.4	23.7
2003	4.7	15.4	19.9	1.5	4.1	5.1	22.6
2004	4.7	14.0	18.1	1.5	5.0	6.2	27.3
2005	4.7	15.7	20.2	1.4	4.5	5.6	24.7
2006	4.7	13.7	17.6	1.4	3.8	4.6	20.5
2007	4.7	13.1	16.8	1.4	4.0	4.9	21.9
2008	4.7	13.7	17.6	1.4	4.5	5.5	24.3
2009	4.7	13.4	17.2	1.4	3.4	4.2	18.5
2010	4.7	12.8	16.4	1.4	4.2	5.3	23.6
2011	4.7	9.8	12.5	1.4	3.7	4.5	20.0
2012	4.7	8.2	10.4	1.3	4.0	5.0	22.0
2013	4.7	9.0	11.5	1.3	3.4	4.2	18.8
2014	4.7	7.4	9.5	1.2	3.5	4.3	19.2
2015	4.7	15.1	19.5	1.2	2.9	3.6	15.9
2016	4.7	12.3	15.7	1.2	4.8	5.9	26.1
2017	4.7	16.5	21.2	1.2	4.3	5.3	23.5
2018	4.6	16.6	21.2	1.2	3.8	4.7	20.9
2019	4.6	18.2	23.3	1.2	3.7	4.6	20.2
2020	4.6	14.0	18.0	1.2	3.6	4.4	19.5

TABLE A-4 (Continued)

SCENARIO 2 - HIGH PRODUCTION

NEW MINES BY PROVINCE, TYPE, CAPACITY, AND YEAR
(NEW MINES EXPRESSED IN TERMS OF MILLION TONS/YEAR)
-AVERAGE PRODUCTIVE MINE LIFE = 20 YEARS-

INTERIOR PROVINCE

	MINE TYPE						
	UNDERGROUND			SURFACE			
	MINE SIZE (MTY)						
YEAR	<0.5	1.0	3.0	<0.5	1.0	3.0	5.0
1975	1.0	2.3	2.9	2.2	1.4	1.8	7.8
1976	1.0	1.4	1.8	2.2	2.2	2.7	12.1
1977	1.0	1.9	2.4	2.2	1.9	2.3	10.4
1978	1.0	1.9	2.4	2.1	1.7	2.0	8.9
1979	1.0	2.3	2.9	2.1	1.8	2.2	9.9
1980	1.0	2.0	2.5	2.1	2.4	3.0	13.4
1981	1.0	2.6	3.3	2.1	2.5	3.0	13.4
1982	1.0	2.6	3.3	2.0	2.8	3.4	15.0
1983	1.0	2.6	3.3	2.0	3.5	4.2	18.8
1984	1.0	2.4	3.1	1.8	3.6	4.5	19.9
1985	1.0	2.7	3.5	1.8	3.6	4.4	19.4
1986	1.0	2.5	3.2	1.7	2.3	2.9	12.8
1987	1.0	2.4	3.1	1.6	2.8	3.4	15.1
1988	1.1	2.6	3.3	1.6	3.2	4.0	17.6
1989	1.1	2.7	3.4	1.6	2.5	3.1	13.7
1990	1.1	2.7	3.4	1.5	3.7	4.6	20.3
1991	1.1	2.0	2.5	1.4	2.0	2.4	10.6
1992	1.1	1.6	2.0	1.4	2.7	3.3	14.9
1993	1.2	1.9	2.4	1.4	2.4	2.9	13.0
1994	1.2	1.4	1.9	1.3	2.4	2.9	13.1
1995	1.2	3.5	4.5	1.3	2.0	2.4	10.8
1996	1.2	2.6	3.3	1.3	3.7	4.5	20.1
1997	1.2	3.6	4.7	1.3	3.2	3.9	17.2

TABLE A-4 (Continued)

SCENARIO 2 - HIGH PRODUCTION

NEW MINES BY PROVINCE, TYPE, CAPACITY, AND YEAR
(NEW MINES EXPRESSED IN TERMS OF MILLION TONS/YEAR)
-AVERAGE PRODUCTIVE MINE LIFE = 20 YEARS-

INTERIOR PROVINCE

	MINE TYPE						
	UNDERGROUND			SURFACE			
	MINE SIZE (MTY)						
YEAR	<0.5	1.0	3.0	<0.5	1.0	3.0	5.0
1998	1.3	3.9	5.0	1.2	2.5	3.0	13.5
1999	1.3	4.4	5.6	1.2	2.7	3.3	14.9
2000	1.3	3.4	4.4	1.2	2.7	3.3	14.9
2001	1.3	4.4	5.6	1.2	3.0	3.7	16.5
2002	1.3	4.0	5.1	1.2	3.3	4.0	17.8
2003	1.3	4.3	5.5	1.2	3.2	4.0	17.7
2004	1.3	3.9	5.1	1.1	3.7	4.6	20.2
2005	1.3	4.4	5.6	1.1	3.5	4.3	19.1
2006	1.3	3.8	4.9	1.1	2.9	3.6	15.8
2007	1.3	3.7	4.7	1.1	3.1	3.8	16.9
2008	1.3	3.8	4.9	1.0	3.4	4.2	18.5
2009	1.3	3.7	4.8	1.0	2.6	3.2	14.1
2010	1.3	3.6	4.6	1.0	3.2	4.0	18.0
2011	1.3	2.8	3.5	1.0	2.8	3.4	15.2
2012	1.3	2.3	2.9	1.0	3.2	4.0	17.6
2013	1.4	2.7	3.4	1.0	2.6	3.2	14.1
2014	1.4	2.2	2.8	1.0	2.8	3.4	15.2
2015	1.4	4.5	5.8	1.0	2.3	2.8	12.6
2016	1.4	3.7	4.7	1.0	3.8	4.6	20.6
2017	1.4	4.9	6.3	1.0	3.4	4.2	18.6
2018	1.4	5.2	6.7	1.0	3.0	3.7	16.2
2019	1.4	5.7	7.3	1.0	2.9	3.5	15.7
2020	1.4	4.4	5.7	1.0	2.8	3.4	15.1

TABLE A-4 (Continued)

SCENARIO 2 - HIGH PRODUCTION
NEW MINES BY PROVINCE, TYPE, CAPACITY, AND YEAR
(NEW MINES EXPRESSED IN TERMS OF MILLION TONS/YEAR)
-AVERAGE PRODUCTIVE MINE LIFE = 20 YEARS-

GREAT PLAINS PROVINCE

	MINE TYPE						
	UNDERGROUND			SURFACE			
	MINE SIZE (MTY)						
YEAR	<0.5	1.0	3.0	<0.5	1.0	3.0	5.0
1975	.1	.1	.2	.8	.5	.7	3.0
1976	.1	.1	.1	.9	.9	1.1	5.0
1977	.1	.1	.2	1.0	.8	1.0	4.6
1978	.1	.1	.2	1.0	.8	1.0	4.3
1979	.1	.1	.2	1.1	.9	1.1	4.9
1980	.1	.1	.2	1.3	1.1	1.4	6.0
1981	.1	.2	.2	1.4	1.7	2.1	9.2
1982	.1	.2	.2	1.6	2.1	2.6	11.6
1983	.1	.2	.2	1.8	3.1	3.8	16.9
1984	.1	.2	.2	2.0	3.9	4.8	21.3
1985	.1	.2	.2	2.1	4.4	5.3	23.7
1986	.1	.2	.2	2.3	3.2	3.9	17.2
1987	.1	.2	.2	2.4	4.1	5.0	22.4
1988	.1	.2	.2	2.5	4.9	6.0	26.8
1989	.1	.2	.2	2.5	4.1	5.0	22.2
1990	.1	.1	.2	2.6	6.4	8.0	35.3
1991	.1	.1	.1	2.7	3.6	4.5	19.8
1992	.1	.1	.1	2.7	5.2	6.4	28.4
1993	.1	.1	.1	2.8	4.9	6.0	26.6
1994	.1	.1	.1	2.9	5.2	6.3	28.2
1995	.1	.2	.2	2.9	4.3	5.2	23.3
1996	.1	.1	.2	2 9	7.9	9.7	43.1
1997	.1	.2	.2	2.9	7.1	8.7	38.7

TABLE A-4 (Continued)

SCENARIO 2 - HIGH PRODUCTION

NEW MINES BY PROVINCE, TYPE, CAPACITY, AND YEAR
(NEW MINES EXPRESSED IN TERMS OF MILLION TONS/YEAR)
-AVERAGE PRODUCTIVE MINE LIFE = 20 YEARS-

GREAT PLAINS PROVINCE

	MINE TYPE						
	UNDERGROUND			SURFACE			
	MINE SIZE (MTY)						
YEAR	<0.5	1.0	3.0	<0.5	1.0	3.0	5.0
1998	.1	.2	.3	3.0	6.0	7.4	32.7
1999	.1	.2	.2	3.0	6.6	8.1	36.0
2000	.1	.2	.2	3.0	6.6	8.1	36.0
2001	.1	.2	.3	3.1	7.5	9.2	40.8
2002	.1	.2	.3	3.1	8.7	10.7	47.4
2003	.1	.2	.3	3.1	8.6	10.6	47.1
2004	.1	.2	.2	3.2	10.6	13.1	58.2
2005	.1	.2	.3	3.3	10.3	12.7	56.2
2006	.1	.2	.2	3.3	8.6	10.5	46.6
2007	.1	.2	.2	3.3	9.1	11.2	49.8
2008	.1	.2	.2	3.3	10.8	13.3	59.1
2009	.1	.2	.2	3.3	8.2	10.1	45.0
2010	.1	.2	.2	3.4	10.5	13.2	58.4
2011	.1	.1	.2	3.4	9.0	11.2	49.6
2012	.1	.1	.1	3.4	10.7	13.1	58.4
2013	.1	.1	.2	3.5	9.3	11.4	50.7
2014	.1	.1	.1	3.6	10.2	12.5	55.6
2015	.1	.2	.3	3.6	8.5	10.4	46.1
2016	.1	.2	.2	3.6	13.8	17.0	75.5
2017	.1	.2	.3	3.6	12.4	15.3	68.0
2018	.1	.2	.3	3.6	11.9	14.6	64.9
2019	.1	.2	.3	3.7	11.7	14.4	64.0
2020	.1	.2	.2	3.7	11.3	13.9	61.6

TABLE A-4 (Continued)

SCENARIO 2 - HIGH PRODUCTION

NEW MINES BY PROVINCE, TYPE, CAPACITY, AND YEAR
(NEW MINES EXPRESSED IN TERMS OF MILLION TONS/YEAR)
-AVERAGE PRODUCTIVE MINE LIFE = 20 YEARS-

ROCKY MOUNTAIN PROVINCE

	MINE TYPE						
	UNDERGROUND			SURFACE			
	MINE SIZE (MTY)						
YEAR	<0.5	1.0	3.0	<0.5	1.0	3.0	5.0
1975	.2	.4	.6	.4	.3	.3	1.4
1976	.2	.3	.3	.4	.4	.5	2.1
1977	.2	.4	.5	.4	.3	.4	1.8
1978	.2	.4	.5	.4	.3	.4	1.6
1979	.2	.4	.6	.4	.3	.4	1.8
1980	.2	.4	.5	.4	.4	.6	2.4
1981	.2	.5	.6	.4	.5	.6	2.5
1982	.2	.5	.6	.5	.6	.8	3.4
1983	.2	.5	.6	.5	.8	1.0	4.4
1984	.2	.5	.6	.5	1.0	1.3	5.7
1985	.2	.5	.7	.5	1.1	1.3	5.8
1986	.2	.5	.6	.5	.7	.9	3.9
1987	.2	.5	.6	.5	.9	1.1	4.8
1988	.2	.5	.6	.6	1.2	1.4	6.3
1989	.2	.5	.6	.6	.9	1.2	5.1
1990	.2	.4	.6	.6	1.4	1.8	7.9
1991	.2	.3	.4	.6	.8	1.0	4.3
1992	.2	.3	.3	.6	1.1	1.4	6.1
1993	.2	.3	.4	.7	1.1	1.4	6.2
1994	.2	.2	.3	.7	1.2	1.5	6.6
1995	.2	.6	.7	.7	1.0	1.2	5.4
1996	.2	.4	.5	.7	1.8	2.3	10.0
1997	.2	.6	.7	.7	1.6	1.9	8.6

TABLE A-4 (Concluded)

SCENARIO 2 - HIGH PRODUCTION

NEW MINES BY PROVINCE, TYPE, CAPACITY, AND YEAR
(NEW MINES EXPRESSED IN TERMS OF MILLION TONS/YEAR)
-AVERAGE PRODUCTIVE MINE LIFE = 20 YEARS-

ROCKY MOUNTAIN PROVINCE

	MINE TYPE						
	UNDERGROUND			SURFACE			
	MINE SIZE (MTY)						
YEAR	<0.5	1.0	3.0	<0.5	1.0	3.0	5.0
1998	.2	.6	.8	.7	1.3	1.6	7.1
1999	.2	.7	.8	.7	1.4	1.8	7.8
2000	.2	.5	.7	.7	1.4	1.8	7.8
2001	.2	.7	.8	.7	1.6	2.0	8.7
2002	.2	.6	.8	.7	1.8	2.2	10.0
2003	.2	.6	.8	.7	2.0	2.4	10.8
2004	.2	.6	.7	.7	2.4	2.9	13.1
2005	.2	.6	.8	.7	2.3	2.8	12.4
2006	.2	.5	.7	.7	1.9	2.3	10.2
2007	.2	.5	.7	.7	2.0	2.5	10.9
2008	.2	.5	.7	.7	2.3	2.9	12.7
2009	.2	.5	.7	.7	1.8	2.2	9.7
2010	.2	.5	.7	.7	2.2	2.8	12.4
2011	.2	.4	.5	.7	1.9	2.4	10.5
2012	.2	.3	.4	.7	2.2	2.7	12.1
2013	.2	.4	.5	.7	1.9	2.3	10.3
2014	.2	.3	.4	.7	2.0	2.5	11.1
2015	.2	.6	.8	.7	1.7	2.1	9.2
2016	.2	.5	.6	.7	2.8	3.4	15.1
2017	.2	.7	.9	.7	2.5	3.1	13.6
2018	.2	.7	.9	.7	2.3	2.9	12.7
2019	.2	.7	1.0	.7	2.3	2.9	12.4
2020	.2	.6	.7	.7	2.2	2.1	11.9

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