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Report of Activities of the Advanced Coal Extraction Systems Definition Project for the Period 1979-1980

Milton L. Lavin
Lionel Isenberg

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ABSTRACT

The activities of the Advanced Coal Extraction System Definition Project are described for the period January 1979 through December 1980. During this period Project effort was devoted to; (1) formulation of system performance goals in the areas of production cost, miner safety, miner health, environmental impact, and coal conservation, (2) survey and in depth assessment of promising technology, and (3) characterization of potential resource targets. Primary system performance goals are to achieve a return on incremental investment of 150% of the value required for a low risk capital improvement project and to reduce deaths and disability injuries per million man-hour by 50%. Although these performance goals were developed to be immediately applicable to the Central Appalachian coal resources, they were also designed to be readily adaptable to other coals by appending a geological description of the new resource. The bulk of the work done on technology assessment was concerned with the performance of the slurry haulage system, an attractive new scheme for transport of coal away from the face. Finally, internal results are presented from a characterization of domestic coals, with the intent of estimating the tonnage associated with various combinations of mining conditions, thus, identifying resources of commercial importance beyond the year 2000.

FOREWORD

This document presents the annual report of activities for the Advanced Coal Extraction Systems Definition Project for the period January 1979 through December 1980.

The project is a part of a multi-year program by the Office of Coal Mining, U. S. Department of Energy, to define, develop, and demonstrate advanced systems for underground coal mining. The primary focus of the effort for this contract period was the formulation of overall systems requirements as the first step in initiating conceptual design activity. A second activity was the assembly of background information and ideas relevant to design.

This work is performed by the Jet Propulsion Laboratory, the California Institute of Technology, via interagency agreement No. DEA101-76ET12548, between the National Aeronautics and Space Administration (NASA) and the Office of Coal Mining, The United States Department of Energy (DOE). Mr. William B. Schmidt is the Technical Project Officer for DOE.

CONTENTS

Figures

Tables

SECTION I

BACKGROUND AND OVERVIEW

During the past five years, the Jet Propulsion Laboratory (JPL) has worked, first, with the U.S. Department of the Interior, now, with the Department of Energy (DOE) to define, develop, and demonstrate advanced systems for underground coal mining. Advanced systems are designated as those which: (1) are suitable for extraction of the significant resources remaining in the year 2000, and (2) promise a significant improvement in production cost and miner safety, with no degradation in miner health, environmental quality, or difficulty in recovering the coal left unmined.

During the first four years, the project concentrated on developing systems requirements and methodologies for evaluating advanced mining concepts. In 1977, the methodology developed to that point was used to assess the performance of a scheme to mine coal from the surface using a hydraulic jet cutter and a downhole pump. As a result of this evaluation and related efforts, tools were developed to: (1) project the system life cycle cost, (2) estimate the percent recovery of impacted coal, (3) identify health/safety hazards, and (4) determine cost strategies for protecting the physical environment. The effort during 1978 and 1979 documented the evaluation methodology and prepared systems level requirements for the designated initial target region, Central Appalachia. The bulk of 1980 was spent documenting the systems requirements and publishing supporting studies in each of the five performance areas. The latter portion of 1980 was devoted to three products of central importance to the long-term project goals: (1) conceptual design requirements specific to the Central Appalachian coal resources, (2) design requirements for one or more additional coal resources of national importance, and (3) preliminary work on a Program Opportunity Notice (PON) soliciting industry participation in the design effort. The development of requirements for non-Appalachian resources began with a comprehensive description of domestic coal resources, which is reviewed in Section V of this report. Discussion of the conceptual design requirements and the PON will, however, be deferred to the 1981 Report of Activities.

During the period covered by this report the project can point to six major accomplishments:

(1) Prioritization of the five system performance areas:

   (a) Production Cost
   (b) Safety
   (c) Health
   (d) Environmental Impact
   (e) Conservation (protection of unmined coal)

(2) Formulation of performance measures for each area, together with appropriate objectives.

(3) Identification of broad opportunities to meet these goals of improved performance.

(4) Completion of in-depth studies in each of the five performance areas, documenting the rationale for the overall requirements.

(5) Initiation of a nationwide inventory and classification of coal resources (without restriction on thickness, depth, dip, etc.).

(6) Completion of two studies oriented toward the next phase (design):

   (a) A survey of recent R&D (both industry and government funded) of probable relevance to advanced system design.

   (b) An in-depth study of slurry haulage, a very promising technique for underground transport.

Thus, the primary focus of project effort during the 1979-80 period was Systems Definition (Figure 1-1).

The first four accomplishments are discussed in Section II and III. Accomplishment five is discussed in Section IV - Identification of Significant Resources. The sixth accomplishment, together with some preliminary ideas on conceptual design, is addressed in Section V: Concept Development and Technology Assessment.

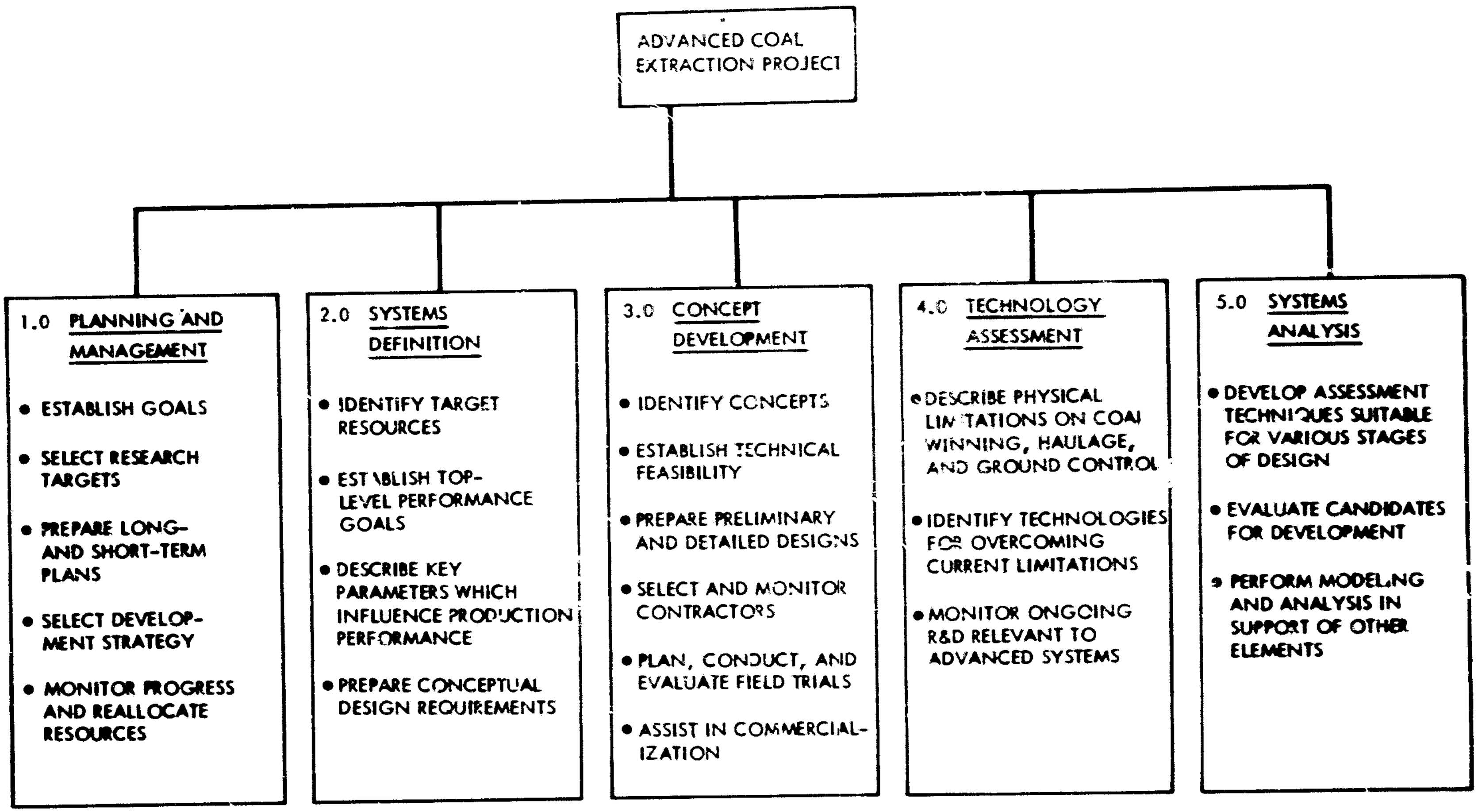


Figure 1-1. Overall Scope of Activities in the Advanced Coal Extraction Project

SECTION II

SYSTEM DEFINITION AND REQUIREMENTS

For several years system definition has been the major focus of the Advanced Coal Extraction Project. The emphasis being the formulation of systems level requirements, together with the requisite tools for evaluating the performance of advanced concepts against these requirements. Logically, the approach has been to first identify those characteristics of mining systems which have a major impact on performance, and then to present requirements in a realistic fashion, but as independent of specific technology as possible. To do this job, the project members decided it was most useful to select a resource that would permit the formulation of requirements within a concrete context. For several reasons, the coals of Central Appalachia seemed appropriate to this purpose:

(1) A substantial amount of coal resources are expected to remain in the ground in the time frame projected for these systems applications.

(2) These resources exhibit a wide variety of mining conditions, including ranges of seam thickness, overburden, dip, mode of seam access, roof and floor conditions, and the anomalies normally associated with coal deposits.

(3) This region is currently a very important source of underground coal production and is expected to be a substantial supply area for coal through the end of this century and beyond, because of its proximity to the industrial heartland and the urban centers of the east coast. Central Appalachia is currently a high cost coal production area. This cost of production is expected to escalate in real terms between now and the end of the century. Thus, productivity increases would be especially attractive.

(4) The results of a systems requirements effort for this region can be generalized fairly easily to all of Appalachia, and with some effort to the flat lying coals of modest thickness in other coal provinces.

This section summarizes the results of the effort to develop systems requirements for the Central Appalachian resource, with generalizability to the coals of other provinces. The reader will note that the organization of this discussion follows the outline of the overall systems requirements document published previously by Goldsmith and Lavin (1980). Section III presents the status of the work to identify coals of potential significance to underground mining other than those in Central Appalachia. This work is viewed from a perspective as of the end of 1980 when some preliminary tonnage estimates, together with a reasonably good estimate of those resources likely to be of substantial geologic importance were available.

The purpose of the overall systems requirement document is to delineate the top-level requirements for an advanced coal mining system. The progression of development for any advanced engineering system must begin with the fundamental objectives, or system requirements for that genera of systems. The creation of requirements enables the setting of program goals, provides understanding of the constraints, and generally establishes a focus on important issues.

In this advanced coal mining system requirement document, the effort was made to develop and present in a comprehendable fashion, a yardstick against which future mining systems could be measured. To be considered as an advanced system, a concept would have to exceed in performance, when compared by this measure, with what existing systems or their logical derivatives might offer. The utility of such a standard is that those who are trying to conceive new mining concepts can clearly understand their goals, and because of having a common basis for comparison will share a common understanding with those who might wish to sponsor, buy, or utilize such a development.

The standards by which a coal mining system might be judged have been grouped into five attribute areas, which can be separately considered. These areas are conservation of resource, environmental effects, miner health effects, miner safety, and production cost. In later sections, each of these areas will be reviewed and quantitative relationships for measuring the worth of a potential advanced mining system will be developed. It will be noted that the mining system is treated as an entity (a so-called "black box") in these considerations. The requirements of the mining system as a whole are stated in terms of overall performance; specific technical operating requirements are at a more detailed level, and are not included in this document.

Before a set of mining equipment can be judged, however, its operating environment must be defined. The environment includes the physical factors of geology and geography, and also market and economic situations, applicable laws, and even the business and social customs of the region. Both mining regions and their mines can vary widely within the United States, and it is unlikely that a universal system, applicable to all mines everywhere, can be developed. Therefore, the effort was initiated by identifying a specific defined geographic region for examination, and reviewing the characteristics of the resource there and the nature of the existent mines. Further, an advanced system can only be identified when a standard for comparison exists. Therefore, the present state-of-art, and its logical evolution over the chronological period for application of an advanced system is outlined as a "moving baseline." To be desirable, an "advanced" system must improve on the "moving baseline" in some significant way.

Having established a basis for comparison, the five attribute areas are considered to determine their relative importance to advanced mining system performance. Each attribute area is examined, and both quantitative goals and appropriate evaluation methods are outlined. In most cases, details of methodology are described in other more specialized reports. Advanced system goals and constraints are put in the context of the performance of the moving baseline.

A. CHARACTERIZATION OF TARGET RESOURCES

This section provides information on the operating environment of an advanced underground mining system. Since mining conditions vary greatly from one coal field to another, Central Appalachia, a significant source of underground production, was selected as the target region to illustrate the kinds of factors which define the operating environment. This section begins with an overview of the domestic coal resources, notes those aspects of coal geology of general interest to the mining engineer, and then focusses on the specific needs of Central Appalachia. The geology of this region is described in some detail, with emphasis on considerations which impact seam access, ease of inseam operation, and protection of unrecovered coal.

1. Overview of Domestic Coal Resources

As indicated by Figure 2-1 and Table 2-1, the United States has an abundance of coal resources. Averitt (1974) estimates an aggregate tonnage of 4 trillion tons, of which 1.7 trillion tons are classified as "identified", i.e., substantiated with a fair amount of borehole data. Subsequent work by JPL, discussed in Section III, indicates that the in-ground resources may be as much as double Averitt's estimate.

It is known that the various mining regions of the United States differ in their geological characteristics. It is as yet unclear how those differences would affect the specification of requirements for an advanced underground coal mining system. However, it is possible to identify general characteristics of coal deposits which are very likely to be of interest to the designer of advanced equipment. Appendix A summarizes what is known about the depositional environment. The next section of the text presents those geological features deemed important to the design of advanced mining systems for Central Appalachia.

2. The Coal Geology of Central Appalachia

Two wide plateaus comprise the bituminous coal mining province of Eastern Kentucky. These plateaus, which together occupy about 10,000 mi^2, are carved into steep hills and sharp ridges by drainage systems that in the north and east flow to the Ohio River, and in the south and west to the Kentucky River system. This landscape has significantly affected mining development by constraining transportation of the coal to market. The hilly terrain exposes most of the coal measures in outcrops. Coal resources buried below drainage are mostly unexplored. The Breathitt Formation, a 1,000 ft thickness of Middle Pennsylvanian sediments hosting most of the region's coal, has only slight structural deformation except along the southern frontier, and so lies nearly flat across the region.

All of Eastern Kentucky's coals are bituminous. As indicated by Table 2-2, production from Eastern Kentucky contributes about seven percent of the aggregate domestic coal production. The total resource in the region, including hypothetical resources, is estimated to be 55 billion tons, excluding the resources in the western half of the State (see Table 2-3).

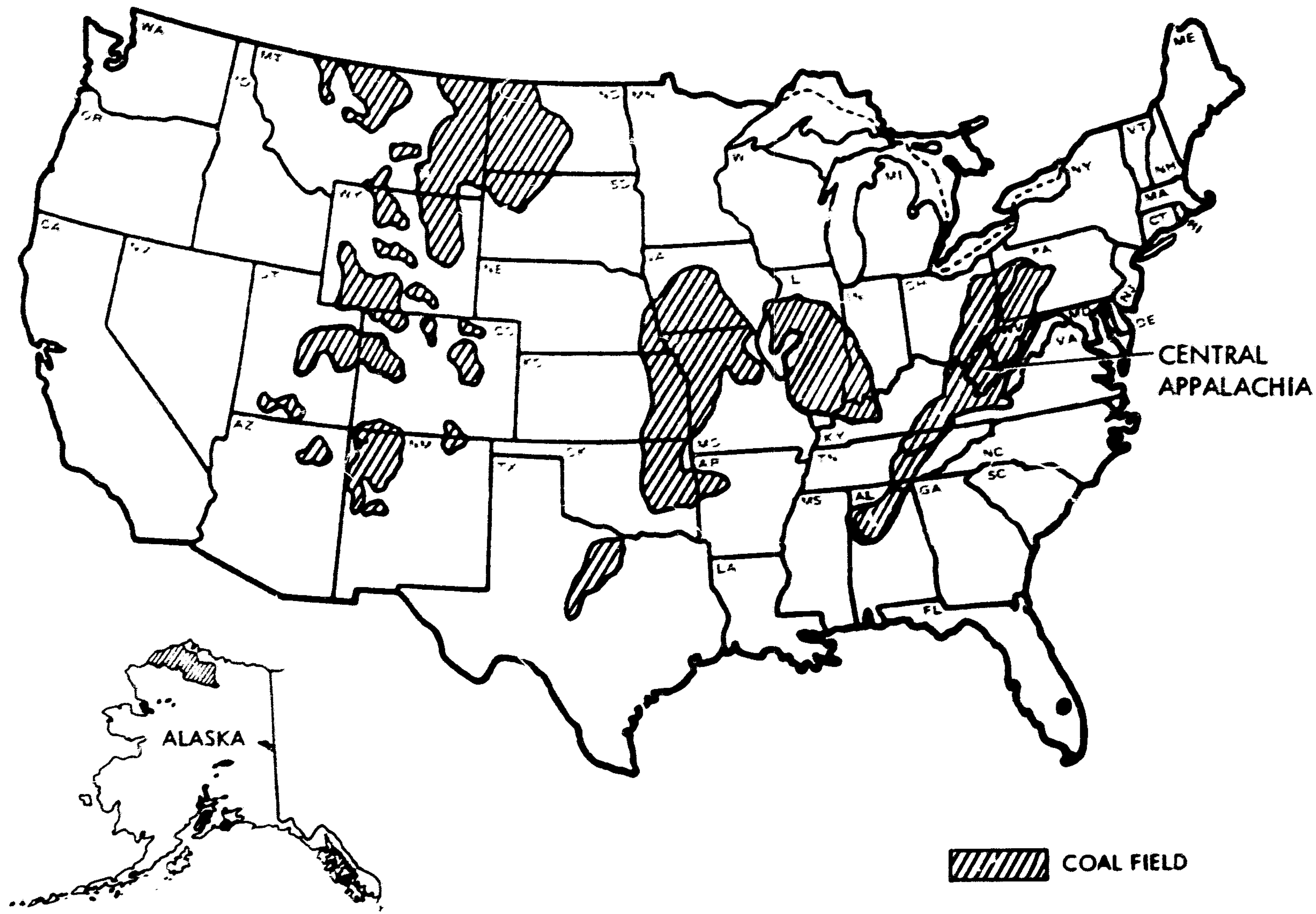


Figure 2-1. Major United States Coal Fields

Table 2-1. Total Estimated Remaining Coal Resources of the United States, January 1, 1974, in Millions of Short Tons

[In millions (10^6) of short tons. Estimates include beds of bituminous coal and anthracite generally 14 in. or more thick and beds of subbituminous coal and lignite generally 2½ ft or more thick, to overburden depths of 3,000 and 6,000 ft. Figures are for resources in the ground]

State	Overburden 0-3,000 feet: Remaining identified resources, Jan. 1, 1974 (from table 2): Bituminous coal	Subbituminous coal	Lignite	Anthracite and semianthracite	Total	Estimated hypothetical resources in unmapped and unexplored areas[1]	Estimated total identified and hypothetical resources remaining in the ground	Overburden 3,000-6,000 feet: Estimated additional hypothetical resources in deeper structural basins[1]	Overburden 0-6,000 feet: Estimated total identified and hypothetical resources remaining in the ground
Alabama	13,262	0	2,000	0	15,262	20,000	35,262	6,000	41,262
Alaska	19,413	110,666	([2])	([3])	130,079	130,000	260,079	5,000	265,079
Arizona	[4]21,234	([5])	0	0	21,234	0	21,234	0	21,234
Arkansas	1,638	0	350	428	2,416	[5]4,000	6,416	0	6,416
Colorado	109,117	19,733	20	78	128,948	161,272	290,220	143,991	434,211
Georgia	24	0	0	0	24	60	84	0	84
Illinois	146,001	0	0	0	146,001	100,000	246,001	0	246,001
Indiana	32,868	0	0	0	32,868	22,000	54,868	0	54,868
Iowa	6,505	0	0	0	6,505	14,000	20,505	0	20,505
Kansas	18,668	0	([6])	0	18,668	4,000	22,668	0	22,668
Kentucky									
Eastern	28,226	0	0	0	28,226	24,000	52,226	0	52,226
Western	36,120	0	0	0	36,120	28,000	64,120	0	64,120
Maryland	1,152	0	0	0	1,152	400	1,552	0	1,552
Michigan	205	0	0	0	205	500	705	0	705
Missouri	31,184	0	0	0	31,184	17,489	48,673	0	48,673
Montana	2,299	176,819	112,521	0	291,639	180,000	471,639	0	471,639
New Mexico	10,748	50,639	0	4	61,391	[7]65,556	126,947	74,000	200,947
North Carolina	110	0	0	0	110	20	130	5	135
North Dakota	0	0	350,602	0	350,602	180,000	530,602	0	530,602
Ohio	41,166	0	0	0	41,166	6,152	47,318	0	47,318
Oklahoma	7,117	0	([6])	0	7,117	15,000	22,117	[8]5,000	27,117
Oregon	50	284	0	0	334	100	434	0	434
Pennsylvania	63,940	0	0	18,812	82,752	[9]4,000	86,752	[10]3,600	90,352
South Dakota	0	0	2,185	0	2,185	1,000	3,185	0	3,185
Tennessee	2,530	0	0	0	2,530	2,000	4,530	0	4,530
Texas	6,048	0	10,293	0	16,341	[11]112,100	128,441	([11])	128,441
Utah	[12]23,186	173	0	0	23,359	[13]22,000	45,359	35,000	80,359
Virginia	9,216	0	0	335	9,551	5,000	14,551	100	14,651
Washington	1,867	4,180	117	5	6,169	30,000	36,169	15,000	51,169
West Virginia	100,150	0	0	0	100,150	0	100,150	0	100,150
Wyoming	12,703	123,240	([2])	0	135,943	700,000	835,943	100,000	935,943
Other States[14]	610	[15]32	[16]46	0	688	1,000	1,688	0	1,688
Total	747,357	485,766	478,134	19,662	1,730,919	1,849,649	3,580,568	387,696	3,968,264

Table 2-2. Coal Production in 1977
(Millions of Tons)

TOTAL U.S. PRODUCTION	689
U.S. UNDERGROUND TOTAL	272
CENTRAL APPALACHIA UNDERGROUND	146
EASTERN KENTUCKY UNDERGROUND	41

Source: U.S. Bureau of Mines (1977)

Table 2-3. Eastern Kentucky Coal Resources
(Millions of Tons)

PROVEN RESOURCES	
IN-PLACE ABANDONED	3,100*
THIN SEAMS (28 in.)	9,100
THICK SEAMS	9,100
SHALLOW COAL	4,400
INFERRED RESOURCE	5,200
HYPOTHETICAL	24,000
TOTAL RESOURCES	54,900

*JPL estimate

Sources: Averitt (1975) and Huddle, et al (1963)

This resource is inventoried in a family of about sixty coal beds, none of which appears consistently across the sample territory. The very discontinuous beds have been mapped and correlated mainly by their respective positions in the Carboniferous sediments. Only a part of the resource is known in detail, and only some smaller part is economical to mine.

Several characteristics of the coal resource lend themselves to quantification which may be useful to the operations planner and systems designer. Most of the analysis which follows draws heavily on Huddle et al. (1963).

3. Nature of Topographic Slopes

The typical slope of the landform surfaces in Eastern Kentucky was analyzed in order to describe resource accessibility. Analysis of a large number of 7-1/2 minute quadrangle sheets led to the following generalizatons:

(1) In areas dominated by the typical non-resistant facies of the Breathitt formation (shales, siltstones and some sandstones), the hill and ridge slopes tend to fall at or about 12 degrees.

(2) Where the more resistant Breathitt members outcrop, the slopes steepen, and tend to fall at or near 26 degrees.

Generally, the softer members, lower relief and more mature landscapes are situated in the northern part of the province where the Ohio River begins to dominate the topography.

4. Dip and Thickness of Coal Seams

The dip (angle of slope) of the resources of Eastern Kentucky was examined in some detail. Of the 220 quadrangle maps that cover the province, 52 were analyzed, giving emphasis to areas where significant deformation is known. The results are shown in Table 2-4. It is clear that the overwhelming bulk of the resource is essentially flat-lying, and that therefore, the capability of mining steeply dipping seams need not be a requirement for the advanced system.

5. Relation of Resource to Outcrop

In order to provide the designer with additional information on seam access, an analysis was made of the relationship of topography and outcrop of the multiple coal seams in the sample province of Eastern Kentucky. Here, the relatively flat coal bodies of the Breathitt Formation outcrop extensively. Mapped seams were measured by planimeter, and a typical seam thickness hypothesized from the literature. Contours were plotted inwards 200, 500, and 1000 ft from the outcrop. Thus, for each seam, the resources were categorized to establish amounts available to surface mining (within 200 ft of the outcrop); and that interior coal, deeper than 1,000 ft from outcrop, which probably must be extracted by some underground method.

Table 2-4. Relationship Between Seam Thickness and Dip (Millions of Tons)

Seam Thickness	Dip $0^\circ - 3^\circ$	$3^\circ - 10^\circ$	$+11^\circ$	Total
14" - 28"	9,100	nil	nil	9,100
28" - 42"	5,257	33	nil	5,290
42" - 120"	3,871	5	nil	3,876
TOTAL	18,228	38	nil	18,266

The present effort measured only two quadrangles: the Grayson in District 1, and the Broadbottom in District 4. The results are displayed in Table 2-5. Note that about 60 % of the coals in the Grayson Quadrangle lie within 200 ft of the outcrop, whereas only 20 percent of the resource in the Broadbottom Quadrangle are similarly situated. Examination of the landforms and their relationship to where each quadrangle lies in the drainage network, reveals the principal reason for the observed differences in seam access: The Grayson Quadrangle, characterized by low, narrow ridges, is located near the floodplain of the Ohio River in relatively mature topography. In contrast, the Broadbottom Quadrangle, exhibiting higher relief and broader ridges, is situated in the headwaters of its drainage network, near the Allegheny uplift. Thus, these two quadrangles probably portray the extremes of the range of topographic conditions which determine what fraction of the coals lies within a certain distance of the outcrop.

6. Buried Resources

Buried coal bodies in Eastern Kentucky have not been fully described in the public record. This is partly because the Breathitt Formation, which contains practically all of Eastern Kentucky's exploited coal, is almost entirely exposed to its basement by the drainage system; in other words, there has been no impetus to map buried resources.

7. Relationship of Coal Resources and Interburden

For advanced extraction system conceptualization and design, it was necessary to identify, analyze and characterize the sedimentary rock interburdens between coal members. Two avenues of inquiry were followed: one, to quantify the interburden dimensions; the other, to characterize the nature of the interburden materials. The first was completed and its conclusions are indicated in Figure 2-2. No meaningful generalization was possible about the composition of the interburden material.

Table 2-5. Resource Tonnage as a Function of Distance From the Outcrop for Selected Quadrangles

(Millions of Tons)

Seam Thickness In.	Feet From Outcrop				
	0-200'	200'-500'	500-1000'	+1000	Total
	GRAYSON QUADRANGLE, Carter County, Kentucky				
14 - 28	27.5	15.1	4.7	3.5	50.8
28 - 42	2.2	0.8	0.1	0	3.1
42 - 120	0	0	0	0	0
Total	29.7	15.9	4.8	3.5	53.9
	BROADBOTTOM QUADRANGLE, Perry County, Kentucky				
14 - 28	27.0	19.6	11.2	12.4	70.4
28 - 42	126.1	57.4	69.8	367.4	620.5
42 - 120	0	0	0	0	0
Total:	153.2	77.0	81.0	379.8	691.0

Note: Resources less than 14 in. thick not included.

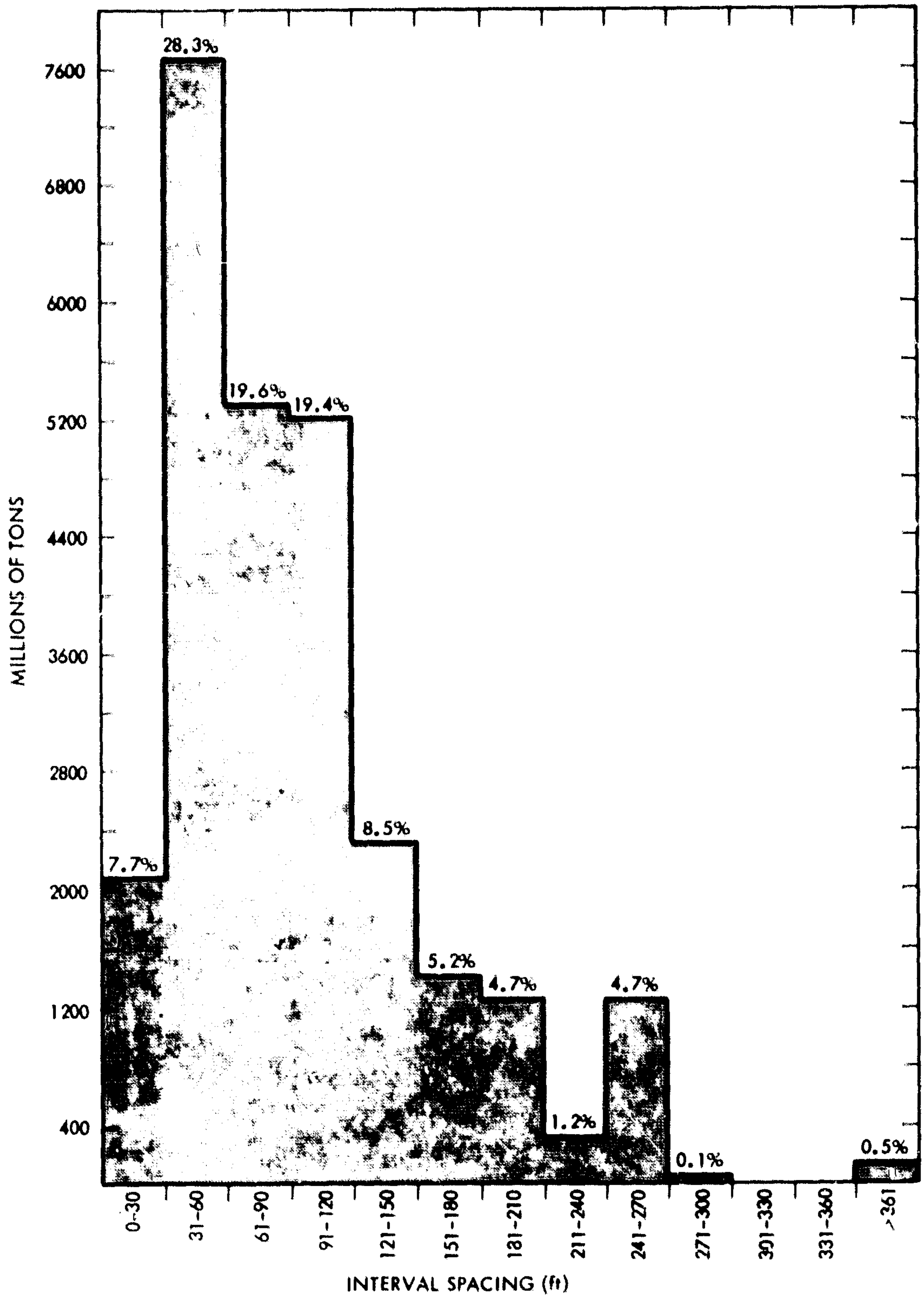


Figure 2-2. Percentage of Eastern Kentucky Coal Resources which are Separated from Adjacent Coal Seams by a Specified Interburden Distance

The results presented in Figure 2-2 indicate that 65% of Eastern Kentucky's coals lie in bodies separated from the next superimposed body by a sedimentary rock thickness of over 60 ft. The balance is more narrowly separated, and the mining of one seam may impede or prevent access to neighboring seams. This has consequences for the conservation of the resource.

8. Overburden Above the Uppermost Coal Member

The mature, rolling topography of the sample province, superimposed upon an irregularly spaced series of almost flat coal horizons, establishes a haphazard overburden pattern. An analysis, paralleling the interburden study, indicated the variation in overburden thickness across the province presented in Table 2-6.

Table 2-6. Overburden Above the Uppermost Coal Seam in Eastern Kentucky

	Vertical Feet			
District*	Min	Seam**	Max	Seam**
1	85	# 77	453	# 67
2	10	104	320	96
3	20	111	263	100
4	101	91	433	96
5	190	111	585	212
6	398	135	443	104

*Districts after Huddle (1963).
**Seam numbers after Bureau of Mines Information Circular No. #8655

B. CHARACTERIZATION OF MINES

A description of both existing and planned Central Appalachian mines was assembled as a guide to the scale of the mining operation for which a new system would be developed. Data on existing mines emphasizes operations in Eastern Kentucky and West Virginia; data on planned mines was obtained from the 1980 Keystone Coal Industry Manual.

1. Existing Mines

Data concerning mine size and ownership have been analyzed for Central Appalachia, with emphasis on Eastern Kentucky and West Virginia. According to data published by the National Coal Association (1979), West Virginia, Virginia, Tennessee, and Eastern Kentucky together have 75% of the underground coal mines in the United States. These mines employ 61% of the nation's underground coal miners and produce 53% of the deep mined coal. In general, the mines are small and scattered.

West Virginia produces 55% of the area's underground coal. In 1978, only one West Virginia operator, Consolidation Coal Co., had mines with individual production over 1,000,000 tons/year. This operator is atypical. Recent data reported by the West Virginia Department of Mines (1978) indicates that West Virginia underground coal production can be divided as follows:

(1) Consolidation Coal Co.'s 10 largest mines produce 19.7%.

(2) The 430 mines belonging to operators with a total annual production of less than 100,000 tons produce 14.4%.

(3) The remaining 65.9% comes from 346 mines.

Analysis of the data in this last category produces the following characteristics of the "typical" West Virginia mine:

(1) Mine size - 125,000 tons/year.

(2) Employment - 85 workers.

(3) Two sections working two shifts.

(4) 160 average annual working days in 1978.*

Although mining operations are scattered, ownership and control are not. A great many operators have more than one mine. All of the largest operators have several mines. Over 75% of West Virginia production is ultimately owned or controlled by large oil or steel companies, electric utilities, or industrial conglomerates. Thus, capital availability for most mines will reflect national rather than regional conditions. The "typical" mine operator, among the larger West Virginia producers:

(1) Works 4.3 mines simultaneously.

(2) has total annual production of 540,000 tons/year.

---

* 1978 was a year marked by significant work stoppages.

(3) Employs 365 people.

(4) Is owned or controlled by a larger entity.

These results are quite consistent with the data on mine size reported by the Kentucky Department of Mines and Minerals (1978). Figure 2-3 presents the cumulative production for all Kentucky mines, ordered by size. Excluding the smallest 965 mines which together aggregate 25% of Eastern Kentucky's production, the remaining 263 mines produce 75% of the region's deep mined coal and have an average output of 123,000 tons/year. The median-sized Eastern Kentucky mine produces between 100,000 and 125,000 tons/year. Aggregation of the West Virginia and Eastern Kentucky data indicates that the median sized mine produces 175,000 tons/year.

2. Planned Mines

Coal Age and The Keystone Manual, each year, publish a list of new mines (or expansions of existing mines) planned over a ten year span. This list represents actual plans and not arbitrary projections into the future. Therefore it should be representative of the development trend for the next ten years.

This data permits inferences about trends in the capacity of new mines scheduled for development in Kentucky and West Virginia during the period 1979-1988. The Keystone data tabulates the mining company, mine name, type of mine, the use, capacity at full production, present capacity, if any, and planned additions to the capacity by calendar year. Only those mines labeled deep are used in the analysis reported here.

A summary picture of these published plans is presented in Figure 2-4. Planned additions reported by Keystone total 53 million annual tons, which should be compared with the 156 million tons produced by deep mines in Kentucky and West Virginia during 1979. Note that 71% of the planned additions have an annual capacity between 600,000 and 2,000,000 raw tons, with the average new mine having a planned capability of 890,000 tons/yr. Only 15% of the planned mines are smaller than 600,000 tons/yr, and 14% are larger than 2,000,000 tons/yr.

It must be recognized that these data on planned mines have two biases. First, it is quite likely that the smaller operator would be under-represented in any survey of planned additions to capacity simply because of the difficulty of obtaining a representative sample from this segment of the industry. Second, it is a well-known fact that near-term plans tend to be much firmer than long-term plans, leading to the suspicion that large mines may be under represented in absolute terms in the planning data for the period after 1983. Whatever the import of these two biases, it is clear that large mines (with a capacity of about 1,000,000 tons/yr.) will have a sharply increasing role in Central Appalachian underground production in the future, with substantial impacts likely within the next ten years.

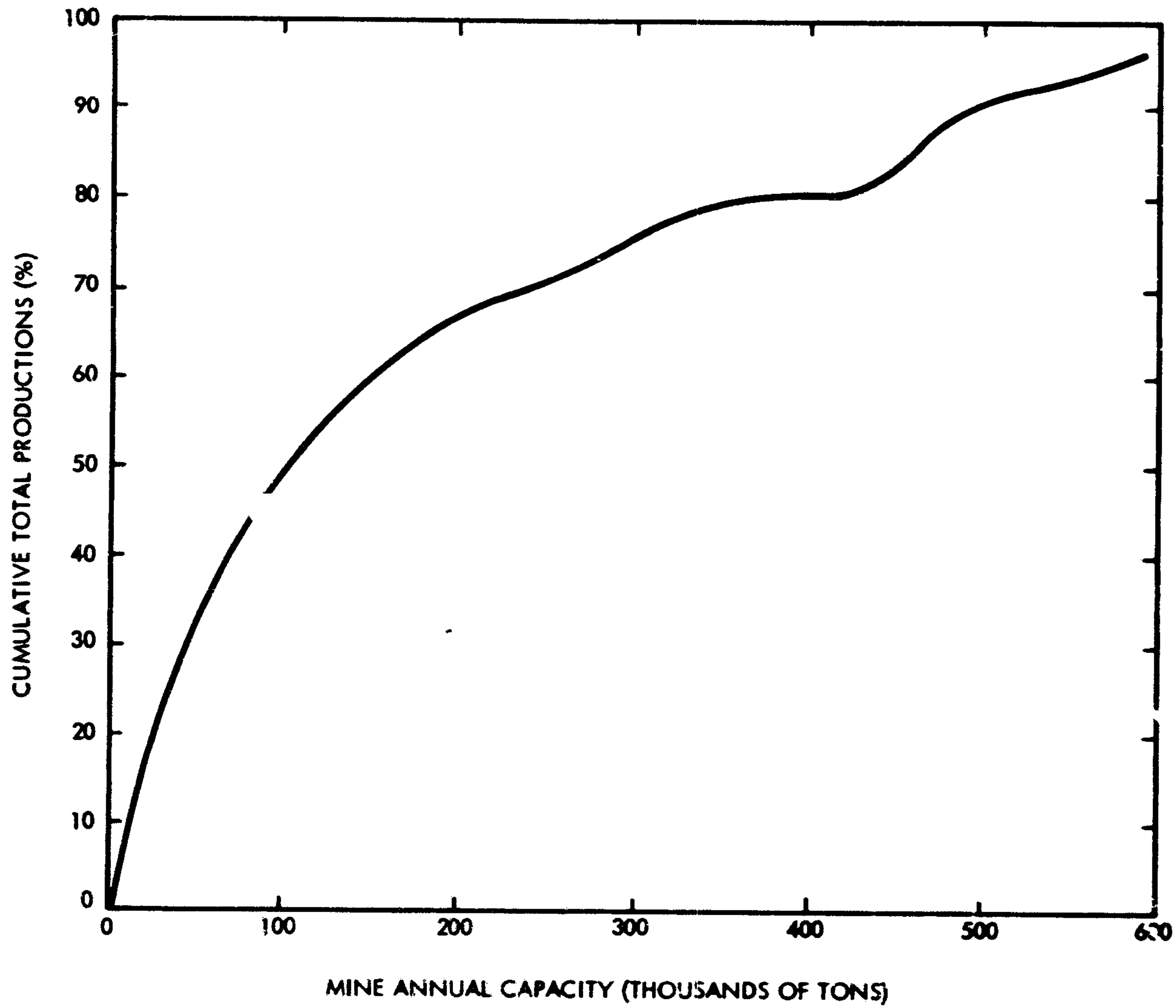


Figure 2-3. Cumulative Underground Coal Production from Eastern Kentucky as a Function of Mine Size

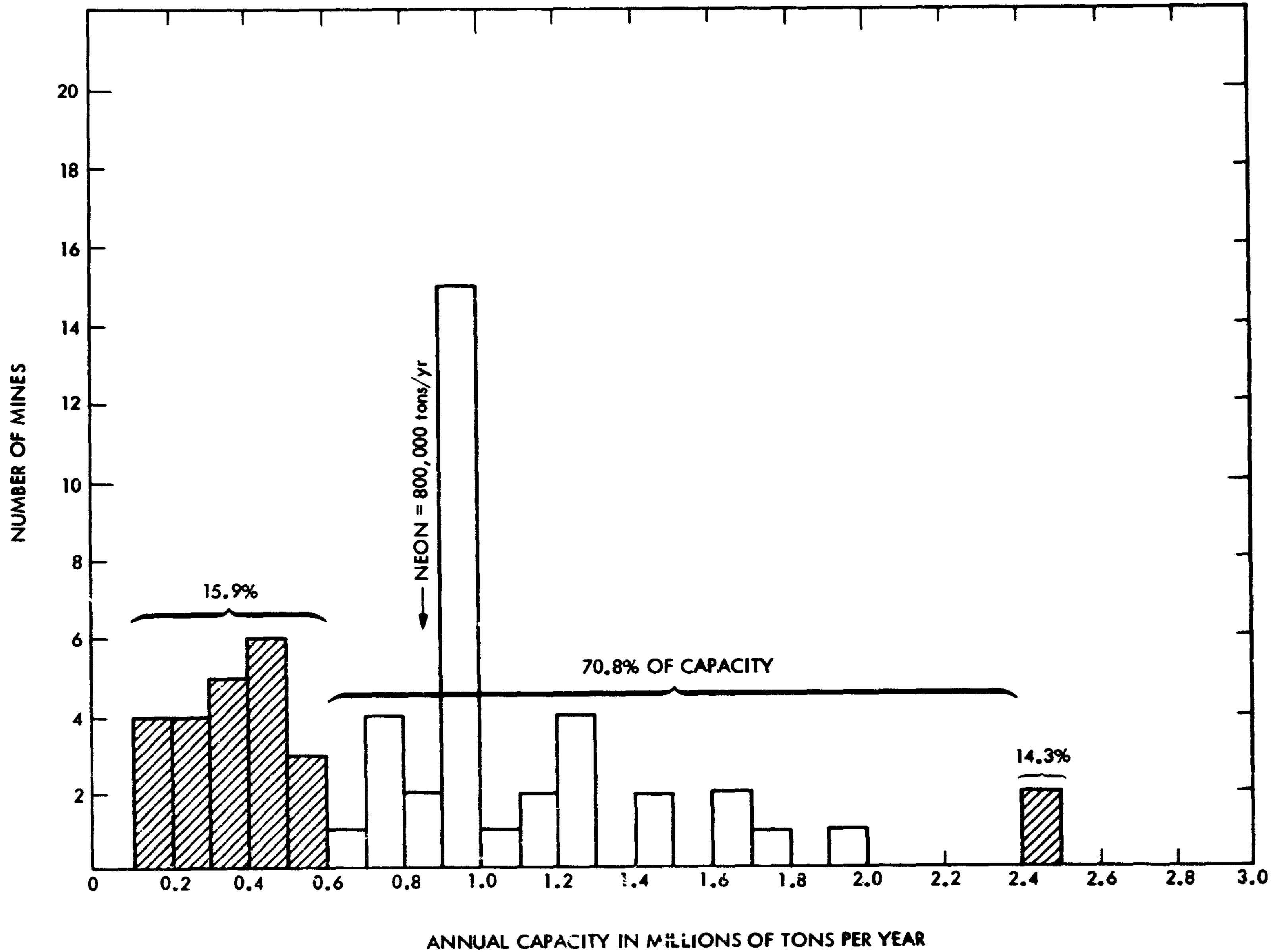


Figure 2-4. New Underground Coal Mine Capacity Planned in Kentucky and West Virginia during 1979 - 1978 Number of Mines vs Capacity

3. Implications for the Capability of an Advanced Mining System

The data on existing and planned mines indicates that for the Central Appalachian Region, advanced mining systems are appropriate for two very different scales of operation:

(1) A one-to-two section mine with an annual capability of less than half a million tons per year.

(2) A larger mine (probably multi-section) with an annual capacity of the order of one to two million tons/year.

Recall that the data on existing mines revealed an ownership pattern wherein it is very common for one firm to operate several small mines. This has two implications for the designer addressing the needs of the small mine. First, the financial resources of the typical small mine operator would not appear to place an onerous cost constraint on the acquisition of new mining equipment. Second, the smaller mine will continue to have a requirement for equipment which can be readily maintained by the mining personnel.

Finally, the multiplicity of small mines, together with the modular structure of existing mines, points to a ready market for an advanced system which could replace the small operator's equipment as it wears out and at the same time be integrated into an existing multi-section mine.

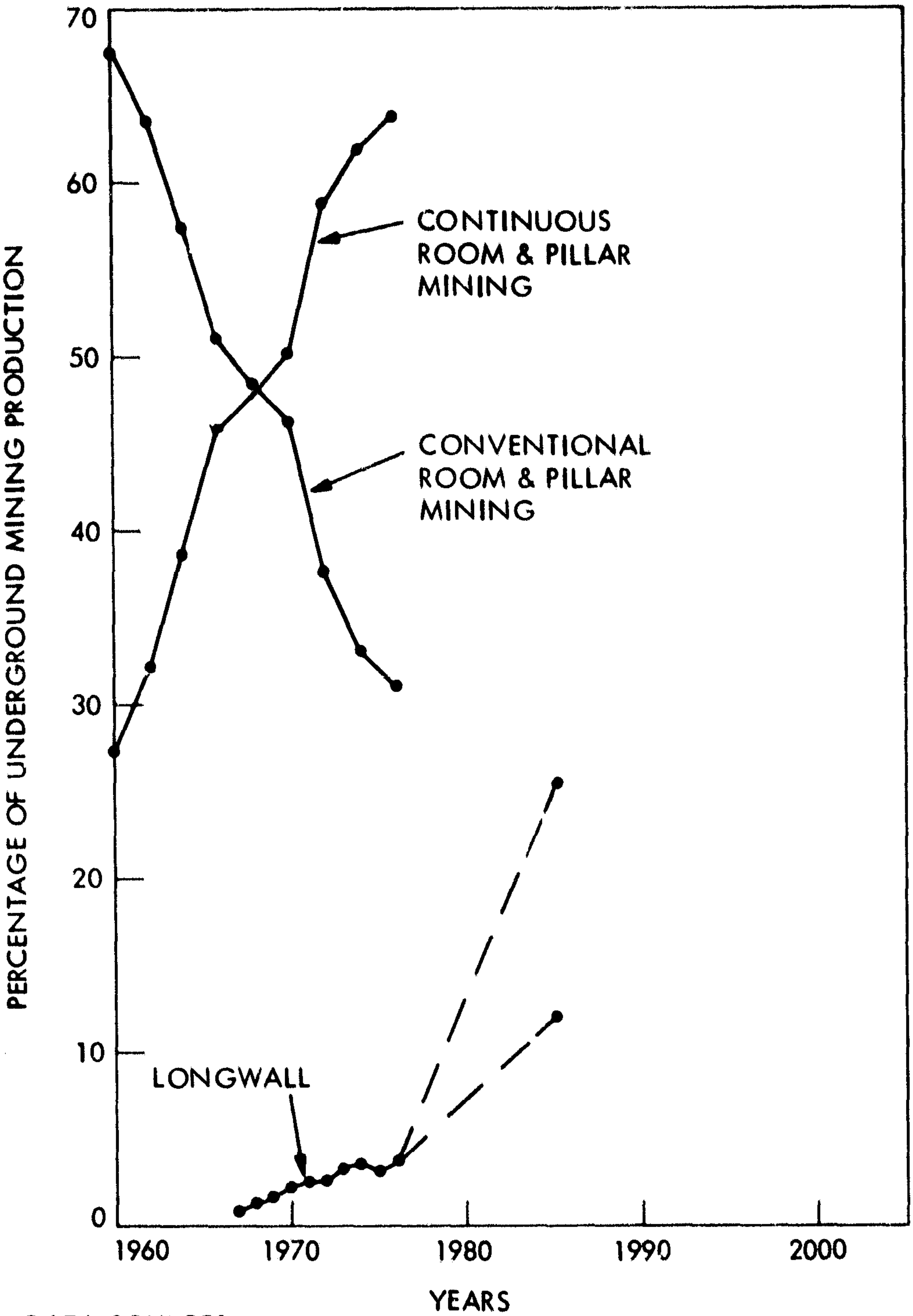
C. DEFINITION OF THE BASELINE TECHNOLOGIES

In order to determine whether a concept is actually an advanced system, its performance must be compared with the other underground mining systems which will exist at the time the new concept has matured to a commercially acceptable form; contemporary mining systems that have evolved over time. Since the year 2000 has been selected as a nominal target year for the advanced coal extraction system to be in operation, present systems will be projected to their conjectural states in the year 2000. This is the moving baseline.

First, the systems to be studied will be selected and extrapolated to the year 2000. Then, the performance parameters needed to determine system productivity will be identified and quantified. After productivity is established, the mining systems can be partnered with appropriate mine plans so that discounted cash flow analyses can be performed in order to arrive at measures of economic performance.

1. Selection of Technologies

Three of the technologies currently used by the underground coal mining industry were considered appropriate for this study. They are room and pillar with a continuous miner, longwall, and shortwall. Continuous room and pillar was selected because this mining method accounts for over 60% of U.S. underground production today, and as Figure 2-5 shows, has made a rapid entry into the industry over the past years. Although there are many possible system configurations, examination of equipment available for



DATA SOURCES:

1960 - 1976: NATIONAL COAL ASSOCIATION, COAL FACTS, 1978 - 1979

HIGH LONGWALL PROJECTION: BUSINESS WEEK (1978)
LOW LONG WALL PROJECTION: KUTI (1979)

Figure 2-5. Underground Coal Mining Methods in the United States continuous room and pillar by Frantz et al (1977) indicated that the most common scheme uses a rotary-drum continuous miner partnered with shuttle car haulage and supported by a dual-boom roof bolting machine. Therefore, this system configuration will be used as representative of the contemporary, 1980, case.

Longwall, which is applicable to mines at the upper end of the size range, was selected for several reasons. First, as shown in Figure 2-5, longwall has a consistent rate of entry into the market. Additionally, according to Kuti (1979) and Business Week (1978), longwall may account for a considerable portion (12 to 25%) of underground production by 1985. If these projections are extended at the same rate to the year 2000, longwall could contribute from 26 to 61% of underground production. Moreover, longwall mining systems account for the majority of underground coal production in many European countries. All of these factors suggest that longwall systems hold great promise for the U.S. coal industry. Kuti (1979) reports that the most commonly used longwall configuration in the U.S. incorporates a double-drum shearer with an armored face conveyor and chock-type hydraulic roof supports. This system is chosen to represent the contemporary, 1980, case.

For longwall panel development, the system selected contains a rotary-drum continuous miner and a mobile bridge carrier (MBC) haulage unit. The system is basically room and pillar technology applied to panel development. The MBC unit was selected because it provides better haulage service to the continuous miner than shuttle cars, thus affording a higher potential productivity. At this time, the MBC unit is second only to the shuttle car in utilization and is increasing in popularity. Thus, it was thought appropriate to team this system with the longwall.

Shortwall technology, which presently accounts for only a small fraction of the U.S. total production, is relatively new. Shortwall has several advantages over longwall and continuous room and pillar in proper circumstances. Shortwall is a caving system and functions well at shallow depths under massive roof strata; Stefanko (1977) notes that it is more flexible than longwall for skirting undesirable geological and man-made situations. Shortwall also is felt to have better health and safety characteristics than room and pillar, and can offer production cost advantages as well. Pollard (1975) reports that the European and Australian mining establishments have considerable interest in the future application of shortwall because of its flexibility. The shortwall systems that have been tried in the U.S. (there have been about 11 of them) normally used the panel development equipment in conjunction with chock-type roof supports for production. The same development system for shortwall and longwall was used as a basis for projections. Thus, both development and production systems for shortwall will contain a rotary-drum continuous miner and a MBC haulage unit.

Conventional room and pillar technology was not selected for study because it is believed that the technology has reached its maturity, and will not experience significant changes in the future. Evidence of the drastic decline in conventional production is shown in Figure 2-5. This prolonged trend suggests that it will not be an important alternative in the future.

2. Year 2000 Extrapolations

The projection of the contemporary systems to the year 2000 will emphasize three mining functions: cutting, haulage, and roof support. While other mining functions have impact on system productivity, the above-mentioned functions were most important. For all three technologies, it is anticipated that improvements will be made in dust control, gas control, equipment safety, and equipment reliability. Projections of the progress to be expected in each of the major technologies was based in part on a survey of current research and development activities reported by Gordon (1980), a member of the Jet Propulsion Laboratory project staff. Continuing review of published reports and journal articles, and contact with the responsible government agencies provided supporting information.

In addition to determining the focus of research and development activities in the industry, significant production constraints were identified for each of the major mining methods. For room and pillar, two major constraints were identified, the frequency of continuous miner place-changes and the intermittency of shuttle-car haulage service. The key constraints on the longwall's materials handling capability are the capacity of the armored face conveyor and its utilization, and the advance rate of the supports. An analysis of the contemporary, 1980 shortwall system identified the cutting width of the continuous miner and the mode of support advance as the major production constraints. The following paragraphs provide more detail about the constraints and the system modifications that might be expected to improve the situation in the future.

3. Extrapolated Room and Pillar

A statutory provision of the Federal Coal Mine Health and Safety Act of 1969 prohibits movement of personnel beyond the last permanent support unless adequate temporary support is provided. To comply with this provision, many mine operators elected to shorten continuous miner advance distances so that the locally positioned operator remains under permanent support. The advance distance is generally 18 to 22 ft, depending on the machine design. This leads to the first production constraint mentioned for room and pillar, but also effects panel development in all systems. Several equipment manufacturers have developed devices for remote control operation, and also offer locally controlled miner-bolter machines that permit permanent support placement in conjunction with entry advancement. Both developments allow continuous miner advancement to approach the pre-1969 situation of breakthrough-to-breakthrough length lifts (60 to 100 ft). However, each development has its drawbacks. Remote-control operations are limited by operator vision, the position of haulage operators with respect to the last permanent support, the stability of the roof, and other factors. Most miner-bolter machines experience roof-bolting delays that erode the potential time savings.

Stefanko (1975), Frantz (1975 and 1977), and National Mine Service (1979) have described various aspects of the Department of Energy's program to develop an automated remote-controlled continuous mining system. The aim of the program is to develop a miner-bolter machine that can function without the aid of on-board operators. To date, the program has not demonstrated a fully automated system.

Harold (1979) describes another approach which has increased production by 27% in initial tests. This involves hydraulically activated roof support beams that are advanced with another set of hydraulic cylinders. This roof support system allows a locally-controlled continuous miner to advance further by providing adequate temporary support. The support unit is only in the initial development phase, but has great promise because it provides a very simple, straightforward solution to the roof support problem, and it can continue to be used with present equipment as it evolves.

The second room and pillar constraint, the intermittent service provided by shuttle cars, may be eliminated by use of a continuous haulage system, such as the mobile bridge carrier (MBC). While there are several reasons why industry uses shuttle cars more extensively than MBC units, Frantz, et al (1977) note that the major reasons concern surge capacity, maintainability, and ease of operation. Positive steps are being taken to make improvements in these three areas. Some chain-conveyor MBC units have a surge bin option. Arthur D. Little (1977) conducted a conceptual study of an automated remote-controlled continuous room and pillar mining system with a surge feeder unit between the continuous miner and MBC unit. This conceptual system is part of a long-term development program in the Department of Energy. Mayercheck (1979) reported that the Department of Energy is also developing an "autotrack" MBC unit in order to improve the tracking and guidance of an MBC deployed behind a continuous miner. With a feedback control system, the MBC unit will straddle and follow an induction cable that is laid on the mine bottom by the lead segment of the unit which is under local, manual control. This addition will ease guidance and control problems.

As the previous discussion indicates, there are several future system options that directly address current room and pillar constraints. For this moving baseline, we have selected a standard rotary-drum continuous miner, with a ten-foot wide cutter head, partnered with an auto-track MBC unit, hydraulic temporary roof support units, and dual-boom roof bolters. This system is seen as an obvious evolution of existing equipment that does not require a great deal of sophisticated hardware and at the same time minimizes functional interactions. This system will also be used for the year 2000 longwall and shortwall development cases.

4. Extrapolated Longwall

The major constraints of longwall systems are the capacity of armored face conveyors, the under-utilization of face conveyor capacity, and the advance rate of roof support units. It is not clear that any major improvements in face conveyor capacity will take place in the near future. Conveyor capacity is governed by the cross-sectional area of the conveyor pan and the speed of the conveyor chain. The cross-sectional area is presently constrained by the design of the roof supports and the shearer, and conveyor flexibility requirements. Therefore, an extensive system redesign would be required to provide an increased cross-sectional area of the conveyor. How this redesign might be accomplished is not clear.

Present chain speeds are limited in order to minimize the wear rate of chain links. While several attempts have been made to develop lubrication systems, Dumbrack (1979) indicates that none seem acceptable. The ultimate solution may be more abrasive-resistant materials for the links, or friction-reducing liners for the conveyor pans. While it is certain that manufacturers and researchers are investigating this avenue, no positive results have been published. Thus, it is evident that present conveyor capacities may be the major limiting factor for longwall production. We have adopted that viewpoint in constructing the moving baseline. Schroeder, et al (1978) and Rybak (1979) reached this same conclusion in their studies of future systems.

On the other hand, conveyor capacity is presently under utilized because operational cycles for shearers have a considerable amount of nonproductive time. Bickerton (1980), illustrated this point in his analysis of the half-face method currently used most commonly in the U.S. The non-nonproductive segments of the shearer cycle for the two cases examined ranged from 30 to 47% of the total cycle time. To improve conveyor utilization, two shearers (or more) could be placed on the face. The National Coal Board (1976) reports that this practice is quite common in the United Kingdom. Each shearer would be assigned to a particular segment of the face. With the use of an interactive control system, each shearer would cut its segment of the face in such a manner that the conveyor is not overloaded and collision of the shearers is avoided. Analysis of this configuration, by Bickerton (1980), showed a 33% decrease in the cycle time while obtaining the same production per cycle as the one-shearer scheme.

While the dual-shearer face approach increases shift production by better utilizing conveyor capacity, other approaches found in the literature address the instantaneous production rate of shearers, and health and safety impacts on longwall workers. Gross (1979) and Schroeder, et al (1978) describe efforts to increase the production rate by cutting a wider web. In one of these studies maximum shearer production rate was constrained by conveyor capacity.

The present approach to automated longwall promises to improve miner health and safety by removing personnel from critical areas, and should, in addition, lay the groundwork for a future dual-shearer configuration. The development program underway at the George C. Marshall Space Flight Center (1977) has identified three basic systems required for automated, remote-controlled longwall mining: a vertical control system for the shearer, a face advancement system, and a master control system. Efforts are underway to develop these systems for application to existing longwall configurations. Such developments will clearly support the application of automation and remote control to the dual-shearer configuration. Summers (1979) has observed that British attempts at automated longwall were partially successful, but encountered labor/management problems. Their experiences, nevertheless, will benefit American developments. Therefore, the prediction of automated longwall options by the year 2000 does not seem unreasonable.

A third longwall production constraint is the slow advance rate of the roof support units. Cominec (1976) reports that the "state-of-the-art" cycle time for a support is about 10 seconds. This value transforms into a support advance rate along the face of 30 ft/min because supports are normally placed on 5 ft centers. Therefore, the shearer travel rate along the face should be limited to 30 ft/min.

According to Olsen (1977), under ideal circumstances, most roof support systems can be advanced along the face at a rate of 50 ft/min. However, several factors limit support advance rates: (1) movement of the face conveyor; (2) the loss of fluid pressure and fluid flow; (3) lowering the support from the roof in preparation for advancement; and (4) raising the support to the roof after advancement. The first factor led to the development of the "one web-back" method of face advance. This method, which is being adopted by many American operators, eliminates face advance delays caused by conveyor movement needs, improves roof control, and increases the available travel space between the conveyor and supports. The second factor is related to the inadequacy of hydraulic power pack capacity and the buildup of back pressure in the return hydraulic line. Olsen (1977) notes that these problems can be alleviated by increasing the capacity of the hydraulic system.

The last two factors result from the design of longwall powered supports. In order to improve upon the situation, the basic support design must be modified. Casanova (1979) describes an attempt by the French Collieries Research Institute to develop a crawler-mounted hydraulic roof support. This design permits advancement under load, thereby eliminating the vertical roof-beam movement required with conventional longwall support designs. It is not yet known whether the crawler sliding support is superior to the conventional support, but Casanova reports that there are several prototypes in the field.

The following system components are proposed for the extrapolated year 2000 longwall system:

(1) Two double-ended ranging drum shearers having vertical control systems.

(2) Powered roof support units.

(3) An armored face conveyor (AFC) with a peak capacity of 1,500 ton/hr.

(4) A stageloader that can adequately handle peak loads from the AFC.

(5) A face advancement control system for the supports and AFC.

(6) A master control system that effectively coordinates all face activities.

5. Extrapolated Shortwall

As previously mentioned, the cutting width of continuous miners presently used in shortwall systems is normally 10 ft. Because of this width, the roof supports must be operated in a manner which constricts optimum production performance. These situations involve the rate of face advance and strata control.

During a face advance cycle, each support unit is moved forward twice, about 5 ft each time. The first advance occurs as the continuous miner cuts along the face, and results in little, if any, production interference. The second advance does not start until the continuous miner finishes cutting and starts tramming out of the face area. The resulting production interference is quite significant, accounting for 21 to 28% of the cycle time, according to Bickerton (1980).

Katen (1979) and Mayercheck (1979) note that several shortwall operations have failed or have experienced serious production delays because of poor roof conditions. While these basic conditions are a result of natural processes, the unsupported roof area and quality of roof support that exists at the face accentuates the problem. While the continuous miner is cutting, the unsupported area is typically in the range of 400 ft^2 (40 ft x 10 ft). After the initial support advance, a span about 5 ft wide along the entire face length (180 to 200 ft) is poorly stabilized by the forepole devices of the supports. This situation, along with the unsupported roof span, promotes rock falls along the face. These not only delay production during their clean-up, but the resulting cavities also reduce support effectiveness and accentuate the problem.

Because these problems exist at present cut widths, Pollard (1975) and Stefanko (1977) have suggested narrower cuts. The extrapolated year 2000 shortwall system design incorporates this suggestion. A review of current continuous miner specifications identified 7.75 ft as the narrowest miner chassis width with a cutter head minimum of 8.5 ft. Discussions with equipment designers, including Freed (1979) suggested the possibility of a narrower body and cutter head. Therefore, a 7-ft wide cutter head was elected for the extrapolated case. To complement this narrow continuous miner, Kiskis (1979) designed a support to achieve a 7 ft advance. It is also assumed that a continuous haulage system can be designed to accommodate the space limitations. An analysis of this extrapolated system revealed shift production increases ranging from 17 to 35%, depending upon the mining conditions.

6. Summary of the Moving Baseline

The moving baseline is summarized in Table 2-7, where the features of room and pillar, longwall, and shortwall development and production equipment are listed. Table 2-8 presents estimates of the ranges of production rates expected from the moving baseline systems. The basis for these estimates is reported in Bickerton (1980). The corresponding production cost, per ton of clean coal is summarized in Table 2-9. These costs are stated in 1980 dollars and allow for a 15% return on invested capital.

7. Discussion of Results

As mentioned earlier, the study approach was tailored to fulfill two requirements. The 1980 cases were to provide a check to insure that the approach produced reliable results. The average conditions cases for the 1980 systems should provide results, both system productivity and production costs, that correspond well to current experiences. Secondly, the results of the year

Table 2-7. Description of Moving Baseline for Room-and-Pillar, Longwall, and Shortwall Technology

System	1980	2000
Room-and-Pillar	Continuous miner	Continuous miner
	20-ft lift length	Breakthrough length lifts
	Shuttle car haulage	Mobile bridge carrier haulage with automatic tracking
	Roof bolter	Roof bolter
		Mobile, powered temporary roof support system (MTRS)
Longwall and Shortwall Development	Continuous miner	Continuous miner
	20- to 40-ft lift lengths	Breakthrough length lifts
	Mobile bridge carrier haulage	Mobile bridge carrier haulage with automatic tracking
	Roof bolter	Roof bolter
		MTRS
Longwall	One double-ended ranging shearer (DERS)	Two DERS's
	Armored face conveyor (AFC)	AFC
	Chock-type supports	Chock-type supports
		Automatic control of DERS's, AFC, and supports
Shortwall	Continuous miner (10-ft head)	Continuous miner (7-ft head)
	Mobile bridge carrier haulage	Continuous haulage
	Powered supports	Powered supports permitting one-step advance

Table 2-8. Estimated Shift Production for the Moving Baseline Technologies*

	Raw Tons Per Machine - Shift	
	1980 Systems	2000 Systems
Continuous Room-and-Pillar**	290 - 680	560 - 1590
Longwall and Shortwall Panel Development*	450 - 1330	530 - 1390
Longwall Panel Production	830 - 1770	1210 - 2530
Shortwall Panel Production	520 - 1110	660 - 1260

* Those numbers are estimates of the performance of "best available" technology operating in from average to ideal conditions by an experienced workforce.

** Room and Pillar differs from Longwall and Shortwall Panel Development in 1980 because of the influences of tons per panel and panel move time. The year 2000 cases differ because of mine plan differences.

Table 2-9. Estimated Production Costs

	Production Cost per Clean Ton	
Technology	1980	2000
Room-and-Pillar	$22.59-39.84	$15.71-26.66
Longwall	$17.50-29.05	$16.48-25.71
Shortwall	$18.53-31.36	$18.30-29.41

2000 cases were developed to establish a measure of economic performance against which advanced concepts will be compared. It is assumed, and quite possible, that these year 2000 systems will provide the competition to any advanced coal extraction system developed by the Jet Propulsion Laboratory through contract to the U.S. Department of Energy. In order to be competitive, the advanced system must at least match the economic performance of the industry workhorses at the time of its commercialization.

Because difficulty will be encountered in trying to estimate the effects of mining conditions on the productivity of new concepts, comparison with future competitors should be based on ideal mining conditions; hence, the year 2000 ideal conditions cases. However, should an attempt be made to establish performance levels of conceptual systems as conditions deteriorate, any comparison should consider both the average and ideal conditions cases of the extrapolated year 2000 systems.

Before further discussion is presented concerning the results, two major limitations of the study should be noted. First, the data required for the production analysis approach were not easily available. Secondly, the data that were available did not always appear in consistent usable form. Some of the data, therefore, had to be modified. Finally, the seam height assumption proved to be quite significant. Because the "bottom-up" approach required selection of a seam height, the results of the entire study are only applicable to a 6-ft seam. Before a conceptual comparison is made, this limitation must be acknowledged or eliminated. It is suggested that similar studies be initiated for other representative seam heights.

A review of the literature showed a close correlation between the productivity values of other studies and the 1980 average conditions cases. One study that analyzed 326 continuous room-and-pillar systems, established an average productivity of 281 tons per machine-shift (TPMS) with an average seam height of 63 in.; whereas the average conditions case in this study resulted in 290 TPMS for a 6-ft seam (Suboleski, 1978). Other room-and-pillar studies presented similar results: 300 to 310 TPMS for a 6-ft seam (Katell et al., 1975; Duda, 1978). It is quite evident from these comparisons that the productivity estimate for the 1980 average conditions room-and-pillar case is extremely realistic.

While the moving baseline study estimated 830 TPMS for the 1980 average conditions longwall system, five double-ended ranging shearer faces working 6-ft seams, obtained a combined average productivity of 790 TPMS (COMINEC, 1976). Another longwall study calculated an average productivity of 900 TPMS for a 7.5-ft seam (Schroeder et al. 1978). Although the productivity result of the baseline study compares well with these other study results, the sample size of the comparators is too small to judge the accuracy of the baseline estimate. It is hoped that future information-gathering will permit a sound judgment on the moving baselines.

Although available shortwall studies did not report productivity with respect to seam height, the 520 TPMS estimate of this study was close to the midpoint of the ranges reported - 200 to 980 TPMS (Pollard, 1975; Peng and Park, 1977). Additionally, it was suggested that current shortwall productivity should not vary, appreciably, from panel development productivity

(Green and Palowitch, 1977). The same holds for the baseline study - 450 TPMS for the shortwall panel development and 520 TPMS for the shortwall production unit. Again, the small sample size of the comparators does not warrant a firm judgment as to the accuracy of the baseline estimate. However, as with the 1980 longwall case, the initial comparison is quite encouraging.

A comparison of study cost results, that includes a 15% return on investment, with current spot market prices and long-term contract prices, established an acceptable correlation. Current spot market prices range from 20 to 43 $/ton, producing an average value of $31.50 per ton (Wall Street Journal, 1980). Also, long-term contract prices are presently in the mid-to-high twenty dollar range (Suboleski, 1980). The 1980 average conditions case costs for longwall ($29.05) and for shortwall ($31.36) reflect favorably. However, the 1980 R&P cost ($39.84) does not. This discrepancy can be easily justified, however. The value represents the selling price requirement for a new room-and-pillar mine including plant site, development openings, and preparation plant. The result is a high initial capital investment per annual ton at a rather low labor productivity (9.5 tons/worker-day). But, the labor productivity figure is close to that experienced today. A 1976 study of a hypothetical room-and-pillar mine established a selling price of $31.50 per ton in order to achieve a 15% return on investment (Frantz, King, and Bartsch, 1977). The investigators recognized then that underground mines were not achieving such a high realization for their coal. Several reasons for their discrepancy were given, and in all probability, apply here since their 1976 selling price escalated to 1980 dollars ($44.61), exceeds the value presented in this study ($39.84). Operating mines were either developed before inflation escalated capital investment items to their current levels, have lower mining costs, or may not be achieveing a 15% return on investment.

Scrutiny of the break-even production cost per clean ton for the year 2000 cases indicates that longwall technology will produce coal most cheaply in average conditions, but will lose its supremacy to continuous room-and-pillar as ideal mining conditions are approached.

This trend can be easily seen in Figure 2-6, where production cost is plotted against mining conditions. The assumption underlying this plot is that a linear relationship exists between production cost and degree of geological difficulty. The implication of Figure 2-6, considering that ideal mining conditions rarely exist in nature, is that longwall technology should be the comparator for advanced systems technology. However, if longwall technology does not match well with the general characteristics of a selected target resource, then other technologies should be given consideration. Such is the case for the coal fields of Eastern Kentucky where mine size is generally small (less than 200,000 tons/yr) and the lateral extent of coal blocks may not be appropriate for longwall technology. Therefore, attempts to develop new systems for Eastern Kentucky should recognize room-and-pillar technology as the probable competitor.

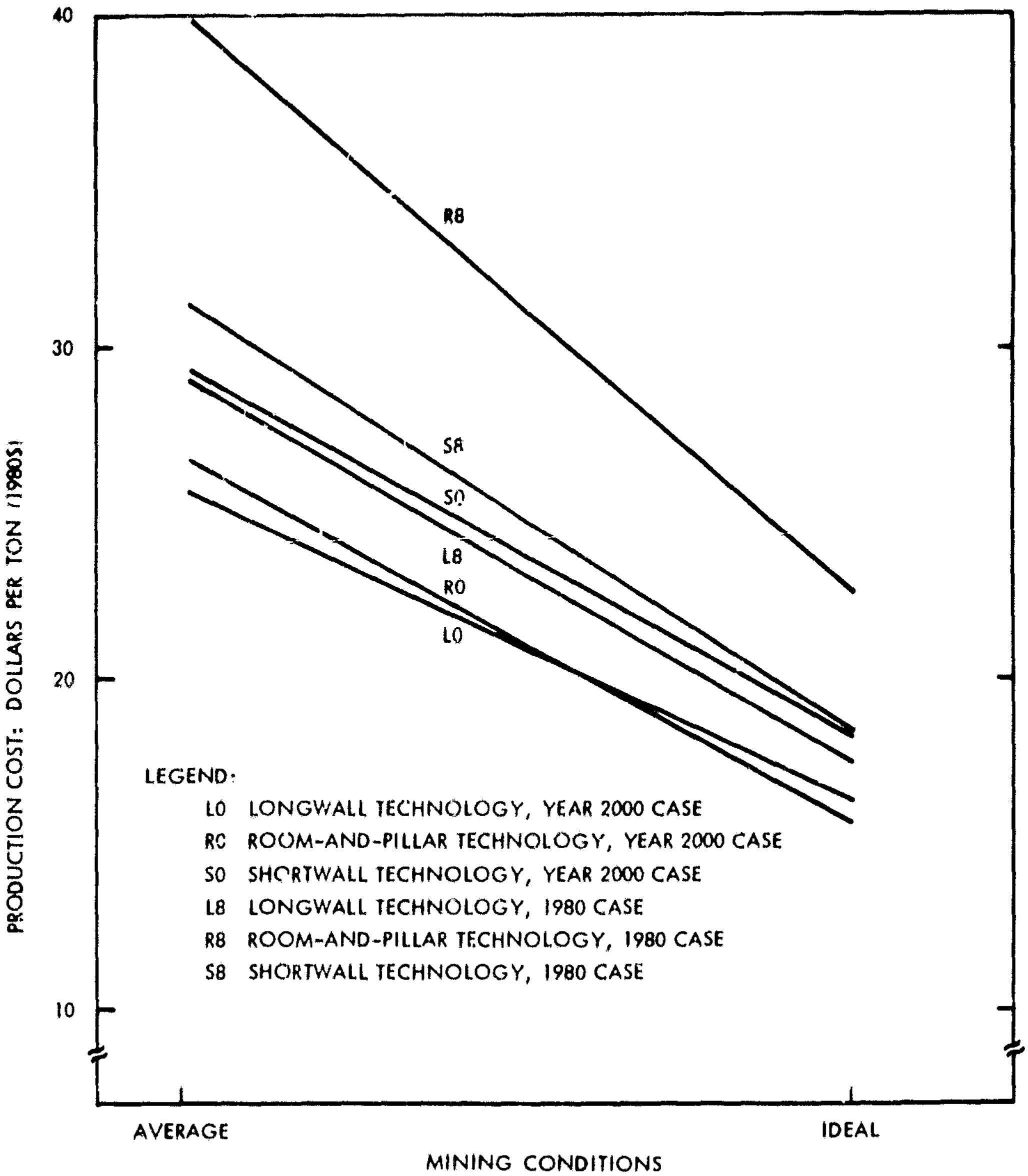


Figure 2-6. Production Cost per Clean Ton vs Mining Conditions for Room-and-Pillar, Longwall, and Shortwall

SECTION III

SYSTEM REQUIREMENTS

Consideration of the concerns that have been expressed by miners, operators, government personnel, and the public suggests that five areas should be separately identified in the system requirements. These are, in order of priority:

(1) Resource Conservation.

(2) Environmental Protection.

(3) Miner Health.

(4) Miner Safety.

(5) Production Cost.

Each subject is covered in this section of the report, with the objective of developing specific goals or targets for an advanced underground coal mining system. These requirements fall into two categories: the first type are stated in terms of a constraint; the second are stated as numerical goals. It is proper to point out that there is seldom a rigorous basis for the setting of system requirements; in most cases there are many debatable issues. The authors of these requirements have exercised judgment, and have subjected these judgments to the scrutiny of others in order to test their reasonableness. It is recognized that system requirements are subject to revision as development progresses and issues are clarified.

A. RESOURCE CONSERVATION

Underground coal mining necessarily disrupts the ground, and affects the conservation of the resource in two ways: (1) the amount of resource (sometimes rubbilized) that might be left in place because of inefficiency, technological limitation, or economic penalty; and (2) the disturbance or damage caused to any nearby coal body. No underground coal mining method or system has demonstrated, or promised, the ability to remove all of the mineral, either because of safety or environmental constraints. Even the most successful and productive methods have limits imposed by their basic design and by their need to assure safety and productivity. Extraction efficiency (resource recovery) is greatly affected by the technique used to create and maintain the mining work space.

1. Mining

Some mining methods seek an optimum extraction ratio while maintaining full support of overhead rock and earth to delay caving or collapse (e.g. room and pillar with no pillar robbing); other methods intentionally cave the cover rock in a controlled manner to relieve the otherwise unmanageable accumulation of rock stress (e.g. longwall). Still other methods combine the two techniques of ground control (e.g. room and pillar with pillar robbing). The choice of a particular technique depends upon many conditions and the impact upon production cost.

Limits are imposed upon the extraction efficiency of any system by the necessity for support of openings used for access, ventilation and transportation, all of which occupy significant area. Other support limits may be imposed where the protection of superimposed strata or the ground surface is necessary. In some states, the utilization of efficient caving systems which cause surface subsidence is often precluded by law or regulation.

Other factors can cause coal to be left in the ground unrecovered. For example, contemporary mining machinery is bulky so that it cannot be deployed in thin coal beds; no support method can control unconsolidated or shattered roof rock; and non-coal material (partings) within the seam may reduce or eliminate the value of the product. The motivation for seeking an increased recovery fraction is apparent; to waste a valuable energy source by rendering it inaccessible seems inappropriate. However, there are sharply differing viewpoints on this issue. In a separate paper which examines the issue of setting minimum coal recovery standards, O'Toole and Walton (1980) conclude that preserving fossil fuels beyond the economically efficient level is not necessarily beneficial to future generations even in terms of their own preferences. Setting fossil fuel conservation targets for intermediate products (i.e., energy) may increase the quantities of fossil fuels available to future generations and hence lower the costs, but there may be serious disadvantages to future generations as well. For example, the use of relatively inexpensive fossil fuels in this generation may result in more infrastructure development and more knowledge production available to future generations. The value of fossil fuels versus these other endowments in the future depends on many factors which we cannot possibly evaluate today. Although we are not on the verge of resolving these questions with any precision, the next twenty years should give us information which can be used to place bounds upon the cost of renewable energy sources. If the real cost (excluding inflation) of utilizing a renewable energy technology is twice that of fossil fuels, then society's view of conserving fossil fuels might be quite different than if the real cost were fifty times as high. The capability exists of waiting this long for more information without using a major portion of the physical coal reserve. Acknowledging that both over-conservation and under-conservation involve cost to society, the danger of acting on limited information is very high.

These arguments are flavored by the fact that the U.S. has several hundred years supply of economically recoverable coal at present consumption rates. There is also the implicit expectation that in such a time frame, coal will be replaced by alternative energy sources. Perelman (1980) and other proponents of the opposing viewpoint (see Appendix E.) argue that should the use of coal increase at even a modest exponential rate, our coal supplies would look more valuable and conservation would appear much more attractive. Clearly, no resolution of this fundamental issue is on the horizon. Fortunately, the time periods are large, and even an incorrect decision made today could be rectified in 20 (or even 50) years without significant or permanent harm.

The economic motivations for increased resource recovery were also examined. Assuming acquisition of mineral rights via option-lease, the principal impact of greater recovery is realized via a change in the capital recovery factor, which declines as the mine life lengthens. A rough feeling for the magnitude of the impact may be obtained by analyzing the 2.0 million ton/year longwall mine described by Bickerton (1981). If the required return is 15% and the effective Federal tax rate is 50%, recovering the $100 million investment over 40 years instead of 20 years, reduces the capital recovery factor by about 0.01, and the minimum acceptable selling price by approximately $1/ton. Clearly, the economic incentive for an operator to conserve the resource is rather weak.

In light of the above discussion of intergenerational equity, and the economic motivation to conserve coal, the conclusion was made that there is no defensible basis for requiring an advanced mining system to achieve a recovery factor substantially in excess of the capability of current technology.

2. Statement of the Conservation Requirement

<u>The Conservation Systems Requirement is Stated as a Constraint.</u> THE ADVANCED EXTRACTION SYSTEM WILL HAVE A CONSERVATION PERFORMANCE AT LEAST AS GOOD AS EXISTING EQUIPMENT OPERATING IN COMPARABLE CONDITIONS. The purpose of this section is to identify the preferred set of conditions for each of the existing technologies and then estimate a recovery ratio for this combination of technology and conditions.

The first step in developing a table of target recovery ratios is to assess the relative attractiveness of each technology for the range of conditions found in the primary resource--flat lying seams of moderate thickness. An examination of the relative attractiveness of the four underground technologies now in use in the United States suggests that the conditions which discriminate most sharply among the various technologies are the following:*

(1) Depth of overburden.

(2) Seam thickness.

(3) Regularity of the seam (i.e., absence of partings, roof and floor rolls, faults, etc.).

(4) Relative cavability of the roof.

(5) Relative stability of the immediate roof.

We have examined all possible combinations of either "favorable" or "unfavorable" sets of these five conditions, and identified the preferred technology, if one exists. Figure 3-1 presents the results of this analysis, which may be summarized as follows:

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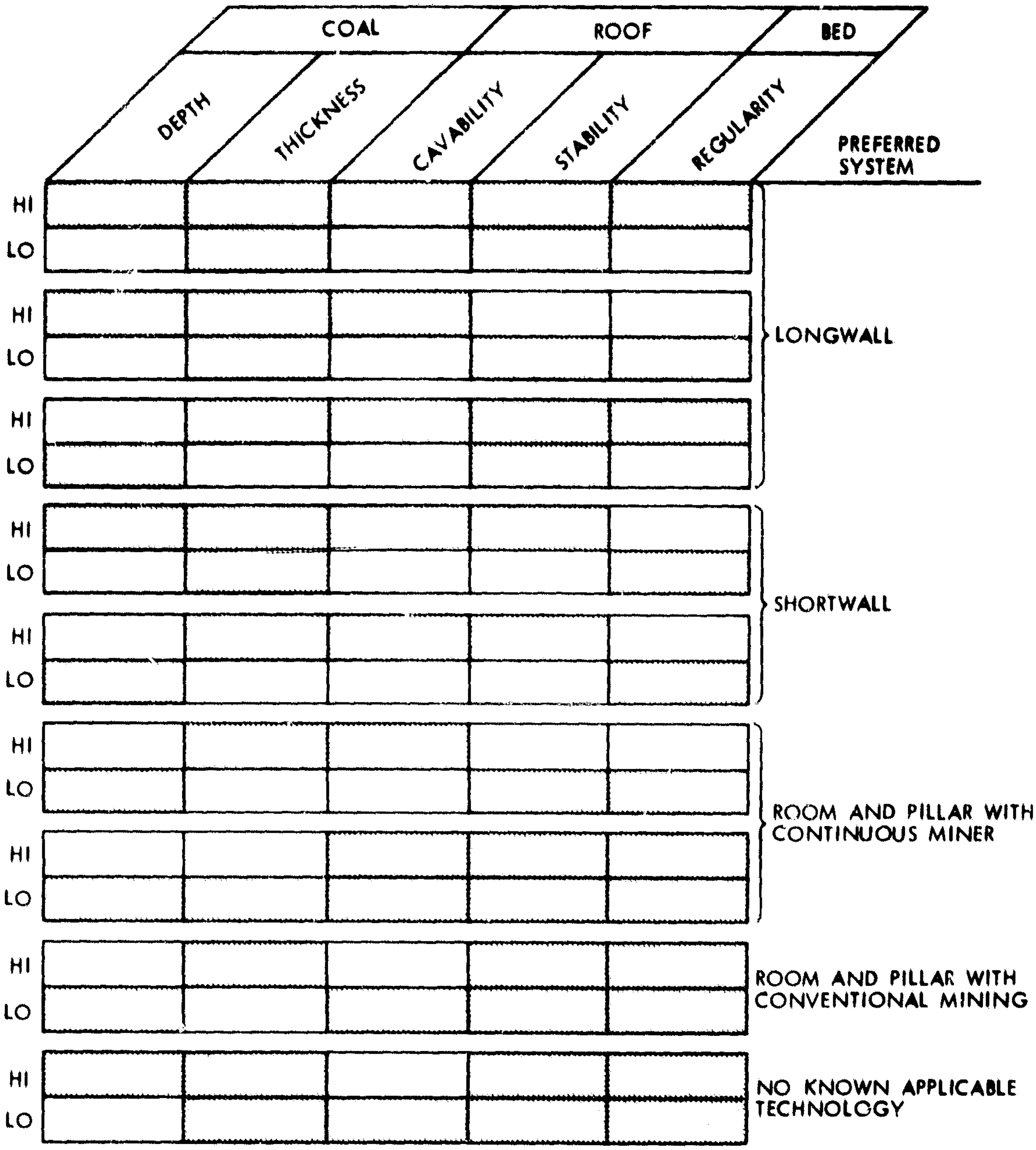
* Table B-1 in Appendix B presents a judgmental evaluation of the relative attractiveness of four technologies now in use in the United States for a variety of expected mining conditions. These judgments are based in part upon previous assessments by Stefanko (1977), Cominec (1975), and Kuti (1975).



NOTE: CONDITIONS SELECTED TO HIGHLIGHT THE RELATIVE ADVANTAGES OF EACH SYSTEM
"HI" INDICATES OVER 1500 ft, OVER 40in., VERY CAVABLE, VERY STABLE, OR VERY REGULAR

Figure 3-1. Preferred Technology for Various Combinations of Mining Conditions

(1) Caving systems are designated as baseline technology for depths over 1500 ft, with the proviso that the roof must be cavable; No known baseline technology is available if the roof is not easily cavable.

(2) For depths exceeding 1500 ft, with cavable roof, longwall is preferred if the seam is very regular; if the seam is irregular shortwall is preferred for stability; otherwise, longwall is the baseline technology.

(3) For shallow depths (less than 1500 ft), room and pillar with a continuous miner is the baseline technology for low coal (less than 40 in.).

(4) Conventional mining ("cut-and-shoot") is the applicable technology for high coal at shallow depths under roof that is not very cavable.

(5) Caving systems form the baseline for thick coal at shallow depths under easily cavable roof, unless the roof is very unstable and the seam is very irregular, in which case, room and pillar with the continuous miner is preferred; longwall is preferred for highly regular seams, and shortwall is applicable to irregular seams so long as the roof is stable.

Finally, recovery ratios for each technology-conditions combination were prepared from currently available data which included:

(1) Empirical analysis of recovery achieved from production panels only, as reported by Reese, et al (1978).

(2) Determination of the fraction of a model mine devoted to mains and submains as opposed to production panels, as described by Harris (1980).

(3) Estimates of recovery from the non-production panel portion of a property (analyses of model mines, plus consultations with mine operators).

Because of the need to set recovery targets appropriate for Central Appalachia, two different scales of operation were chosen: (1) A 250,000 ton/yr room and pillar mine (the same mine used in the environmental impact and conservation impact analyses reported in companion documents), and (2) A 1,000,000 ton/year longwall or shortwall mine (scaled down from the mine used in the projection of a moving technological baseline).

Recovery results are presented, by technology, in Table 3-1. These values represent minimum targets for an advanced mining system operating in the conditions specified in Figure 3-1.

3. Method of Evaluation

Altho[illegible] ie primary thrust of the conservation requirement is high recovery from [illegible] m being mined, protection of neighboring seams is

Table 3-1. Target Recovery Ratios for Each Major Mining Technology

Technology	Production Panels		Mains and Submains		
	Fraction of Mine Area	Recovery Factor	Fraction of Mine Area	Recovery Factor	Aggregate Recovery Factor
Room and Pillar, Continuous Miner: Full Pillar Extraction*	0.7	0.7	0.3	0.4	0.6
Room and Pillar, Conventional Mining: Partial Pillar Extraction*	0.7	0.5	0.3	0.4	0.5
Longwall**	0.8	0.7	0.2	0.4	0.7***
Shortwall**	0.8	0.7	0.2	0.4	0.6

*250,000 ton/year mine

**1,000,000 ton/year mine

***Differences are due to rounding off to one significant digit.

also a matter of concern. Thus, it is recommended that any assessment of conservation performance address impacted coals throughout a property and not merely limit itself to a computation of the conventional recovery ratio, for which quantitative requirements have been set.

The recovery ratio from the seam being mined may be easily obtained by analyzing a mine plan for a representative site. For consistency, this plan should be the same one used in assessing conformance with the other systems requirements. It is a simple matter to compute the fraction of coal recovered once the entry widths and pillar sizes have been determined, and a decision has been made on the proportions of coal to be recovered from pillars, barriers, and fenders. Reese, et al. (1978) provides excellent guidance on the details of making such an analysis. Once this analysis is complete, a parameter study should be made to determine the sensitivity of in-seam recovery both to mine size and to variations in geology which span the range of conditions for which the candidate system is suited.

Estimation of impacts on neighboring seams is considerably more difficult. Four types of impact can be defined:

(1) Rubbilization of nearby seams due to cavity collapse.

(2) Increased stress concentration and reduced roof competence due to mining out an overlying seam.

(3) Seams badly jointed but top and bottom surfaces remain fairly continuous; this phenomenon occurs in the region between the rubbilized zone and the pressure arch, and in subsidence zones.

(4) Seams not discernibly impacted, i.e., seams lying above the pressure arch.

The appropriate categorization of coals at the representative site can be made once the extent of the rubbilized zone, pressure arches, and subsidence troughs are calculated. Empirical formulae suited for these purposes, as well as other information pertinent to these calculations, may be found in Harris (1980), Peng and Chandra (1980) and Stingelin, et al. (1976). Given the coal tonnages in each category, it is a straightforward matter to compute the fraction of coals adversely impacted (categories 1, 2, and 3, plus coal left in the seam being mined).

B. ENVIRONMENTAL IMPACT PROTECTION

The environmental impact of coal mining has long been considered important. Sullivan (1980) has evaluated the modes of impact commonly experienced. Several factors make impractical the formulation of specific quantitative environmental requirements. First, the environmental impacts associated with mining are determined by the interaction between the mining system and the specific site being mined. Second, many environmental impacts are the result of a mine existing and are independent of the particular system being used; for example, any underground mining system will result in generation of refuse and alteration of ground water flow. To propose specific environmental requirements not associated with the site would be impractical.

There is no fundamental way of determining how much environmental degradation is acceptable. In practice, the level of damage is weighed against economic benefits by a political process. This is the origin of most environmental and other regulation, and represents a judgment. Present environmental regulation pertaining to coal mining represents society's judgment (which can change). Thus, any mining system must mitigate effects, as required by law.

At this point, rather than put forth <u>specific</u> system requirements, two <u>general</u> requirements will be outlined which will serve as criteria against which systems will be judged. The methods by which proposed systems will be compared with existing technology will also be described. In addition, an estimate of costs commonly associated with current environmental impact mitigation practices will be presented. Finally, potential environmental problems to be considered in the development of advanced equipment will be described. This discussion will serve as a guide to system design by describing (1) how proposed systems will be evaluated, (2) the economic significance of environmental impact mitigation associated with mining, and (3) general guidelines to achieve environmental advance over existing technology.

The Statement of the Environmental Impact includes the following:

a. Requirements. An advanced system should minimize adverse environmental impact during mining operations and maintain land suitability for future use. Two system design requirements are proposed which reflect these objectives. The first requirement addresses the costs of mitigating those environmental impacts which have a potential for degradation of off-site environmental quality. Required mitigation of potential off-site impacts is not a productive part of the mining enterprise. Consequently, innovation in system design which proportionally reduces these "non-productive" costs will result in a cost advantage over current systems.

The second requirement addresses the range of potential land uses of the mine site and adjacent lands following mine closure. The effects of mining upon subsequent land use potential are considered on-site impacts which are dealt with during reclamation. Successful reclamation should maintain the surface value of the land at the pre-mining land value. Proportionate reductions in reclamation costs, while maintaining land value, are viewed as an advancement over current technologies. Thus, it is possible to state two environmental requirements:

I. AN ADVANCED UNDERGROUND MINING SYSTEM SHOULD NOT RESULT IN HIGHER COSTS OF OFF-SITE ENVIRONMENTAL IMPACT MITIGATION THAN THOSE ASSOCIATED WITH CURRENT MINING TECHNOLOGY.

A desirable level of performance is a significant cost reduction over current technology. In any case, the cost of environmental mitigation is added to the mining cost, so that any trade-off between productivity and environmental impact is made automatically.

II. AN ADVANCED UNDERGROUND MINING SYSTEM WILL MAINTAIN THE VALUE OF MINED AND ADJACENT LANDS AT THE PRE-MINING LAND VALUE, FOLLOWING MINE CLOSURE AND COMPLETION OF RECLAMATION.

Again, site reclamation costs are added to the levelized cost of production.

b. Method of Evaluation. The mode of system evaluation used in assessing compliance with the environmental requirements employs contemporary mining technology as a standard of performance. Both contemporary and advanced underground mining systems can be evaluated in light of existing mitigation and reclamation technologies. Several assumptions underlie this approach. The first is that all potentially adverse environmental impacts can be mitigated to an acceptable level. If impacts cannot be mitigated to levels prescribed by law and regulation, it is assumed that the mining activity would be prohibited and the system will not be evaluated further. A second assumption is that the total cost of mitigating adverse environmental impacts to acceptable levels is a reasonable surrogate for the significance of the aggregated impacts. In adopting this approach, the need for assessing the relative importance of individual impacts is avoided.

In determining the impacts of a mining system upon the site and adjacent lands, it is assumed that either a land use plan exists or that the range of possible uses can be projected for the mining region. The assessment of the suitability of the mine site is performed within a specific use category, rather than for all possible uses. In using this approach, the designated or projected land use category is assumed to reflect public opinion concerning the most appropriate potential use for the land in question. The cost of reclamation can then be made on a consistent basis.

Evaluations will be comparative. To achieve consistency, actual sites representative of conditions in Central Appalachia will be selected; conventional mining systems and proposed advanced systems will be conceptually implemented at each of the selected sites. For Requirement I, environmental impacts associated with each mining system will be identified using the approach described by Sullivan (1980). After potential impacts have been identified, cost figures for their mitigation must be determined. For Requirement II, actual pre-mining land use and potential land uses as described by regional land-use plans will be identified for each mine site. Reclamation costs associated with returning the land to its original or planned use will be determined via methods similar to those employed to cost the mitigation of off-site impacts. This method of evaluation has been demonstrated by Dutzi, et al (1980). An abridged version of a conceptual level assessment may be found in Appendix F.

c. Cost of Impacts. Cost estimation for mitigation of off-site environmental impacts associated with coal mining systems will be accomplished by methods and data developed by Doyle, et al (1974). This report describes a comprehensive analysis of pollution control costs associated with current coal mines in the Monongahela River Basin of West Virginia, Pennsylvania, and Maryland. Samples included a variety of mining methods, mine sizes, and pollution control measures. Although the data are specific to the Monongahela River Basin, the cost estimates are representative of Central Appalachia and other areas with similar topography, mine drainage pollution problems, and mining history.

Costs of controlling mine drainage pollution, erosion, and sedimentation, which are the major causes of off-site environmental problems, are highly site-specific and dependent upon variables such as local geology, soil characteristics, hydrology, ground water flow, and amount of water allowed to enter the mine. These local variables result in a wide range of costs associated with environmental impact mitigation, even within a small and apparently homogeneous region. Three impact mitigation techniques (mine drainage treatment, mine sealing, and refuse bank sealing) are discussed here in order to illustrate the range in mitigation costs.

Mine drainage treatment costs vary considerably according to method of treatment chosen and amount and quality of the water to be treated. According to Doyle, et al (1974), the installed capital costs for a sample of 10 mine drainage treatment plants range from $9,850 to $1,094,000 in 1971 dollars. EPA (1975) indicates that mine drainage treatment costs per ton of coal mined range from $0.03 to $0.10 in 1975 dollars (EPA-240-1-75-0586).

Costs of mine sealing are affected by condition of the opening, condition of the rock, overburden thickness, hydrology, accessibility, haul distances, type of seal to be constructed, and number of seals. Doyle, et al (1974) report that the cost of a single mine seal can range from $2,100 to $21,000. Refuse bank reclamation, including clearing and grubbing, contouring, grading, soil cover, and revegetation, costs an average of $4,200/acre.

The most significant impacts of mining activity on land value are the disposal of refuse and land subsidence. Generally speaking, refuse piles or fills are judged to be liabilities due to potential instability and effect on water quality. Non-uniform or unplanned subsidence can result in land being made unsuitable for urban and agricultural uses.

Dutzi, et al (1980) performed a site-specific analysis of a contemporary room and pillar mine at a site in Clay County, which appears to be representative of Eastern Kentucky. Environmental impact mitigation, including sediment control, water treatment, mine sealing, and revegetation of disturbed areas, was estimated to cost from $0.04 to $0.05/ton of coal mined. Before mining, the site was covered with natural forest and had no urban or agricultural uses. According to Reynolds (1979), values of such lands without mineral rights in this part of Kentucky range from $150 to $250/acre. No land-use plan exists for this region, so it was assumed that the land would be returned to its original use (i.e., forest land). General reclamation for the site, including removal of access roads, backfilling of all disturbed areas to original contour, soil cover, refuse bank grading and soil cover, and revegetation, constituted a minor portion of the total cost of environmental impact mitigation (less than $0.01/ton of coal mined).

d. Design Guidelines. Water quality degradation is a significant problem associated with underground mining. Acid mine drainage (AMD) occurs in many parts of Central Appalachia. AMD is caused by oxidation of ferric materials in the coal itself or in the surrounding rock. Water cannot be prevented from entering a mine, and hydraulic sealing of a mine is often ineffective. Liquids contained in a mine will ultimately reappear. Thus, Laird (1979) cautions that control of the movement of water during and after mining is not a realistic requirement. However, it is sometimes possible to restrict the movement of air into the mine and thus retard or arrest oxidation through complete collapse of the mine roof. Systems which can achieve uniform and complete subsidence are likely to have fewer AMD problems in areas where AMD is a consideration.

Systems which rely on the use of potentially toxic working fluids in the mine are likely to be penalized in an environmental assessment. For example, it is possible that solvents might be used in cutting. As noted, liquids cannot always be effectively contained, and will reappear elsewhere. In areas where AMD is a problem, even the use of water as a cutting agent could present significant problems.

Control of sediment is another major water quality concern. Production of sediment by a mine is a function of the area disturbed by the total mining activity, the magnitude of surface flow disruption, amount of runoff, and the volume of water pumped from the mine. Generally, the largest portion of the sediment is derived from access or haulage roads. Parker (1979) indicates that reduction of the area covered by roads through the use of alternative haulage methods would present a definite advantage over existing techniques.

Refuse piles are an important source of sediment and potentially toxic materials. Moreover, they are also judged to have a negative impact on land value. According to Parker (1979), a desirable feature of an environmentally advanced coal extraction system would be underground disposal of refuse. In addition to potential processing economies, underground disposal would reduce the possibility of negative land value impact, while possibly contributing to control of both subsidence and AMD. However, the potential for aquifier degradation must be considered for subsurface disposal schemes.

Finally, subsidence itself is of major importance in determining subsequent land value. Obviously, regular, uniform, and controlled patterns of subsidence are desirable in order that potential land uses not be restricted.

C. MINER HEALTH

1. Introduction

The coal mining industry has been considered inherently unhealthful because of the difficulty with controlling worker exposure to a wide variety of hostile working conditions. Furthermore, the nature of these conditions is such that they affect all of the major physiological subsystems; the respiratory system, cardiovascular system, hormonal system, and sensory system (see, for example, Rockette (1977)). The key factors contributing to physiological degradation are dust (e.g., coal, silica), methane gas, diesel emissions, poor lighting, noise, and vibration. In addition to these physiological factors, Lorenz (1966) points to the psychological problems resulting from working in small, closed, unlighted spaces. Therefore, it appears reasonable that any advanced coal extraction system should provide a substantial improvement in working conditions, either by making the mine environment more hospitable, or by isolating miners from the environment.

2. Figure of Merit

The basic philosophy behind the development of the system health requirements was to establish, if possible, figures of merit (criteria) against which to compare the projected performance of new systems. During the study it became apparent that no such figure of merit was available. Whereas in the case of safety one might select an average yearly injury rate based on that of other similar industries as a figure of merit, health cannot be viewed in the same manner because the effects of unhealthful working conditions must be measured over a period much greater than one year. For example, pneumoconiosis, a respiratory disease typically related to the coal mining industry, usually takes twenty years before its physical effects become apparent. Therefore, the figure of merit for health must measure the differences between coal miners and workers in other industries, recognizing that these differences might not materialize for the greater part of a lifetime. The first measure that seemed to fit this criterion was "mortality". However, when the mortality rates for coal workers and the average male population were examined, the mortality rates were not significantly different. Figure 3-2 illustrates that the life expectancy of coal miners is only 3% less than the general male population. This is not a significant enough difference to make any firm comparative statements about coal miner mortality rates and those of the general male population. In addition to this, the data presented in Figure 3-2 do not provide the best comparison of mortality because all classes of laborers are included in the general population. A more accurate comparison would have been two similar populations (e.g., coal miner mortalities from West Virginia compared to construction laborer mortalities from the same geographical area). However, epidemiological studies relating to groups as specific as these generally do not exist. Because miner mortalities are not significantly different from those of the general population, it was decided not to use mortality as a figure of merit. One useful result the mortality study did provide, however, was that coal miners deviated significantly from the general male population in the incidence of respiratory disease. Table 3-2 indicates that a significantly larger number of coal miners will die from respiratory diseases, compared to the general male population. Pneumoconiosis is caused by coal dust which results from the cutting process. Very small particles inhaled into the lungs become trapped in the lung alveolar tissues, and, if inhaled in a large enough quantity, eventually cause a reduction in air transfer into the blood. Other diseases such as emphysema, bronchospasm, or pneumonia are aggravated by both small and large dust particles generated during the cutting process, as well as the cold and damp environments typical of many underground mines. Thus, the mine environment is the prime reason why miners suffer respiratory ailments to a larger degree than the rest of the male population.

One interesting aspect of most of the respiratory diseases shown in Table 3-2 is that they are chronic, or long-term, in nature. Although a miner may not necessarily die sooner than the average person, he may be disabled for a longer period of time. This phenomena led to the consideration of "morbidity" as a figure of merit. However, the use of a morbidity comparison presented a problem because it was difficult to obtain detailed information on respiratory disability in the coal mining industry compared to other similar industries such as metal and non-metal mining. Though some disability information was obtained for a small number of industries, it was difficult to evaluate a new design and quantitatively determine to what degree respiratory disability might be reduced. Variation in an individual's susceptibility to disease, and varying lengths of exposure to harmful elements in the environment contribute to the uncertainty with which one can project reductions in disability. Therefore, morbidity also appeared unusable as a figure of merit. Though morbidity measured in terms of "active years lost", for example, was not practical for evaluating new systems, the idea of reducing "disability" still appeared to be a valid goal. It was therefore decided to identify the major causes of disability, and evaluate new systems based on their ability to reduce or remove these causes, rather than evaluate them against a figure of merit.

3. Statement of the Health Requirements

Epidemiological studies comparing the incidence of various diseases in coal miners to the general population, indicate that respiratory disease is the primary factor contributing to mortality and early disability. Rockette (1977) indicates that where other diseases do exist (such as malignancies, diabetes, and cardiovascular disorders), their relative frequencies are not significantly different from the general population. This is evident in Table 3-2. Therefore, it is apparent that the major thrust of

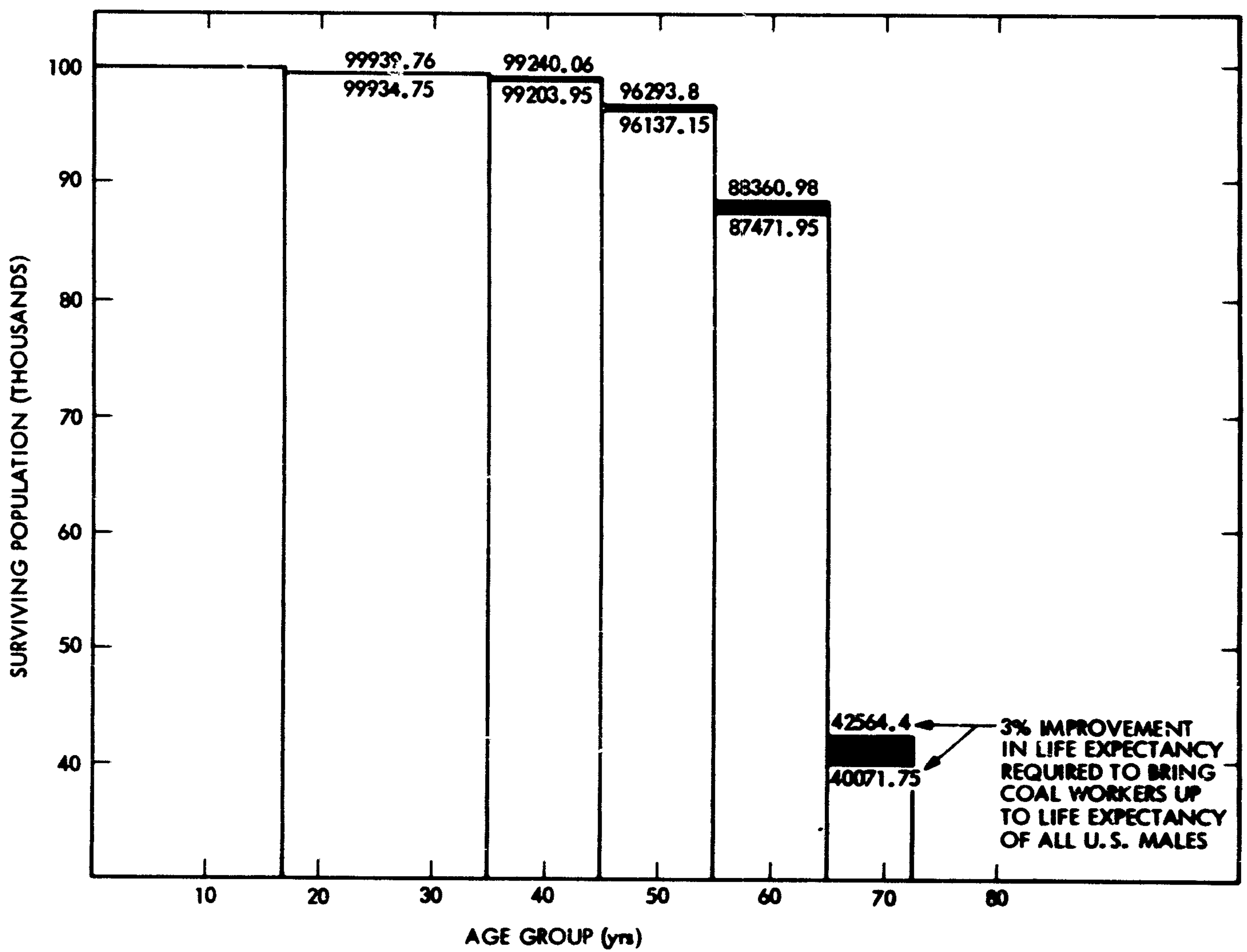


Figure 3-2. The Life Expectancy of Coal Miners Compared with all U.S. Males as a Function of Age

Table 3-2. Observed Coal Miner Deaths, Compared With Expected Deaths for All U.S. Males, for Selected Diseases (Data from a Sample of 22,998 Coal Miners)

Cause of Deaths	Observed Deaths	Expected Deaths	SMR*
Major Cardiovascular Diseases	4285	4525.9	94.7
All Malignant Neoplasms	1223	1248.2	98.0
Diabetes	64	110.2	58.1
Non-malignant Respiratory Disease	741	471.6	157.1
Influenza	28	8	340.6
Primary Atypical Pneumonia	23	12.8	179.7
Chronic Interstitial Pneumonia	58	16.4	353.7
Bronchiectasis	11	9	122.1
Emphysema	170	134.6	126.3
Pneumoconiosis	187	20.2 (est.)	925.7

* Note: The SMR is determined by dividing the observed deaths by the expected deaths, and multiplying by 100. An SMR of 100 implies no distinguishable difference between coal miners and the general population.

Source: Rockette (1977). The 1965 U.S. male population was used to compute the expected number of deaths.

the health requirement should be reduced exposure to elements in the underground mining environment that contribute to respiratory disorders.

Since coal worker mortality rates are not significantly in excess of the rest of the general male population (i.e., approximately 3%), it appears that a better measure of the potential impact of a new extraction system on health, would be some indication of the ability of the system to reduce the incidence of respiratory related disability among coal miners. Research by Morgan (1975), Nacye (1971), and others indicates that these respiratory problems are caused by dust and aggravated by high temperature and humidity. Therefore, the primary health requirement is to remove or protect the workers from these major contributors to pulmonary disease. Secondary requirements address additional elements in the environment that affect other bodily functions, (e.g., lighting, work space, vibration, and noise).

4. Primary Health Requirements

Research reported by King (1960), Morgan (1975), Nacye (1971), Penman (1970) and others indicates that the onset and development of coal worker's pneumoconiosis (CWP) and progressive massive fibrosis (PMF) is dependent on the presence of coal dust of particle size less than 5 microns in the mine atmosphere. Recent discussions with Drs. Stuermer and Hatch (1980), of Lawrence Livermore Laboratories, indicate mutagenic compounds (such as aromatic hydrocarbons) can potentiate the occurrence of CWP, PMF, and other kinds of lesions.

Rockette (1977) notes that cigarette smoking is a habit common with miners, and known to be a contributing factor in contracting CWP. Although coal workers with CWP frequently do not show a significantly altered ventilatory capacity, they do exhibit a marked decrease in oxygen transfer. A failure of the pulmonary system to transfer oxygen at at least 1250 cc/min prevents an individual from being gainfully employed in an occupation which requires continuous moderate physical activity. Rasmussen (1970) established a relationship between the concentration of coal dust of greater than 5 micron particle size and chronic bronchitis which is known to be excessive in coal miners. Summarizing, there is ample evidence that a reduction in coal dust concentration will diminish the incidence of CWP, PMF, and bronchitis and thereby lessen excessive coal miner morbidity in comparison to the rest of the working population. A more complete physiological discussion of these diseases and the contributing factors may be found in Zimmerman (1980b).

Davies (1974) and Lyons, et al (1972) have shown that the actual susceptibility of miners to CWP or PMF varies widely in both the exposure time and the allowable threshold. However, there is good agreement between health experts in both the Department of Labor and the United Mine Workers that worker exposure to dust levels of less than two milligrams per cubic meter of dust at all locations in the mine would greatly reduce the incidence of respiratory disability. Maintenance of this maximum level of exposure is now required by regulation, and therefore, forms the basis for the primary health requirement on an advanced system (see Title 30 of the Federal Code of

Regulations). Thus, the primary health requirement can be stated as follows:

ADVANCED COAL MINING SYSTEMS MUST NOT EXPOSE MINERS TO DUST LEVELS HIGHER THAN TWO MILLIGRAMS PER CUBIC METER. THE COST OF MITIGATION MUST BE ADDED TO THE PRODUCTION COST OF THE COAL PRODUCED.

The establishment of requirements on exposure to known carcinogenic, mutagenic, and toxigenic compounds is difficult due to the wide variation in susceptibility of workers to related disorders. As a consequence, no standard can be set at this time. Moreover, advanced systems may introduce compounds into the mining environment that are equally dangerous yet different from the list of compounds known to be unhealthful. Current research by Stuermer and Hatch (1980), indicates that there are four generic groups of mutagenic compounds which can be present in the mining environment. These are released through the mechanisms of heat and pressure, or in combination with solvents. Because we cannot assign defensible threshold levels, the safest course is to require that workers not be exposed to any of these compounds in the course of performing their tasks.

5. Secondary Health Requirements

The underground mining environment frequently has a high humidity. It is well documented that a high humidity atmosphere increases the likelihood of developing bronchospasm in susceptible individuals (see, for example, Fraser (1977) and Heitzman (1973). The suspected unhealthful effects of prolonged exposure to high humidity in a mine is corroborated by the statistically high incidence of asthma among underground coal miners, as reported by Rasmussen (1970). This health problem is discussed in more detail by Zimmerman (1980b).

In addition, although no quantitative relationship can be determined from the data available, Rasmussen (1970) argues from theoretical considerations that temperature extremes, as well as humidity extremes, will seriously increase the development of respiratory disease among underground coal miners. Thus, any new system should attempt to reduce exposure to high humidity and temperature extremes in order to promote the general pulmonary health of miners.

A secondary health requirement relating to the working environment is:

ANY NEW TECHNOLOGY SHOULD ATTEMPT TO CREATE AN ATMOSPHERE WHERE THE RELATIVE HUMIDITY IS BETWEEN 50 PERCENT TO 75 PERCENT AND THE TEMPERATURE IS BETWEEN 65 F AND 78 F WITH NO PROLONGED EXCURSION OUTSIDE OF THESE LIMITS.

Other environmental factors which affect health are: (1) lighting, (2) noise, (3) working space, and (4) vibration. Requirements for lighting and noise are well documented. Experience in other industries, as well as current research in mining, suggests that any advanced system should comply with the following design requirements for working space and vibration:

(a) Working Space. Human engineering studies pertaining to operator performance under varying space and vehicle control constraints, have indicated a direct relationship between fatigue, cramped working space, and poor positioning of controls. Studies done on the psychological effects of operating in cramped space (see Lorenz (1966)) also imply a relationship between irritability, fatigue, and limited working space. The mining environment cannot practically allow for ideal working space conditions. Nonetheless, it is recommended that at a minimum, advanced systems be designed with consideration given to established anthropometric standards. The basic standards are summarized by Zimmerman (1980b).

(b) Vibration. Prolonged exposure to vibration from equipment can result in "vibration disease."* This condition is characterized by: a reduction in pain sensation, decrease in vibration sensation, pains in the joints (particularly the hands), hyperactivity, and decrease in libido. As the threshold for these effects varies widely by individual, it is difficult to set a design standard for acceptable machinery vibration levels. Nonetheless, advanced systems should be designed to include vibration supression equipment. For ease of reference, all the primary and secondary health requirements developed above are summarized in Table 3-3.

6. Method of Evaluation

The major problems with evaluating health hazards on new designs are that: (1) levels of exposure are difficult to predict in a rigorous way, and (2) effects of exposure are poorly understood and do not show up for a long time. Therefore, it is necessary to use a subjective basis in the evaluation of new technology, compliance with requirements.

As described by Zimmerman (1980a), the evaluation is divided into two steps. The first step starts with a complete operational analysis of the system. Here, the system is examined to understand how the coal is cut, how the face is ventilated, how the coal is hauled, how the roof is supported, and what the salient aspects of the working environment are. In addition to the operational analysis, a task time analysis is conducted to establish the amount of interaction of the workers with the various operational elements. Once this information is assembled, all of the system components are compared with similar contemporary systems (not necessarily mining systems) to determine whether the new system may generate the same health hazards as contemporary systems, whether new health hazards have been introduced and whether exposure to these health hazards will be reduced. In this analysis, reduced exposure to health hazards (as in a dust containment system) is as important as elimination of a hazard (reduction of dust via jet cutting).

The second step requires a subjective assessment of the kinds of design changes and alterations in worker protection that show promise of meeting the system requirements on exposure. Five levels of compliance with the requirements may be distinguished:

(1) Beneficial effect.

(2) Uncertain, but likely to be beneficial.

(3) No effect.

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* See Paranko, et al (undated).

Table 3-3. Summary of Advanced Coal Extraction System Health Requirements

Health Characteristics	Goal	Requirement
Primary Requirements		
Dust	Reduce miner mortality and morbidity resulting from lung disease to that of U.S male population	No greater than 2 mg/m^3
Carcinogens and Mutagens	Same as above	Concentrations no greater than that in air of large urban areas
Secondary Requirements		
Temperature	Permit miners to work in environment satisfying OSHA and MSHA standards for other industries	Between 65°F and and 78°F with no extreme swings
Humidity	Same as above	Between 50% and 75% with no extreme swings
Noise	Same as above	Meet MSHA standards
Lighting	Same as above	Meet MSHA standards
Working Space	Same as above	Accommodate most body configurations
Vibration	Same as above	Provide vibration damping for machinery operators

(4) Uncertain, but likely to be a detrimental effect.

(5) Detrimental effect.

These judgments are obtained via consultation with experts in the field of occupational health who are provided with the data on system operation, identified health hazards, and projected exposure levels. As there is no way to quantify precisely how much dust will be generated, or the volume of toxic fumes emitted, it is necessary to identify the presence of the various health hazards and place a subjective weight on the ability of the design to minimize the hazards. Although non-quantitative, this approach offers an organized method of comparing new designs against existing technology.

D. Miner Safety

Coal mining is regarded as one of the more hazardous occupations in the United States. This can be substantiated by the known rates of temporary and permanently disabling injuries and fatalities in mining, as compared to other occupational categories. Table 3-4 presents a summary of injury statistics compiled by the U.S. Department of Labor for the period 1972 through 1978. The data of Table 3-4 suggest that mining is not enormously more dangerous than other industries in terms of total injuries, but that the rate of serious injury and death is a factor of two to three higher.

Analysis of injury data by MSHA indicates that the major hazards contributing to the rate of injuries and fatalities in coal mining have been (1) roof or face falls, (2) slips and falls, (3) electrical burns or shocks, (4) fires and explosions, (5) unsafe handling of material, (6) impact by machinery, and (7) being pinched or squeezed by equipment. The major factors mediating the impact of these hazards are the time an individual worker is exposed, the amount of body protection, the number of people exposed in a confined area, and the unpredictable nature of the mining environment. A detailed discussion of the hazards associated with current day equipment may be found in Appendix H.

1. Statement of the Primary Safety Requirement

In light of the above analysis it appears that the systems requirement for safety should focus primarily on the reduction of deaths and disabling injuries, and secondarily on the reduction of total injuries. In setting quantitative goals, we pursued two different approaches: (1) seek a statistically significant reduction in underground coal mining injuries or, (2) require that underground coal mining match the safety performance of some set of comparable industries. In practice, the two approaches yielded very similar numerical results. The second approach (comparable industries) was selected because the first approach requires an arbitrary choice of the percent reduction needed to achieve a statistically significant difference. The four industries listed in Table 3-4 were chosen as comparable because of their similarity to mining in terms of both the types of hazards encountered and the severity of accidents which occur.

Table 3-4. Average Injury Rates in Selected Industries for The Period 1971 Through 1978

Industry	Aggregate Inj. Per Million Man-Hours	Fatalities Per Million Man-Hours	Disabling Inj. Per Million Man-Hours
Underground Coal Mining	105.3 (for the target region)	0.43	58.2
Construction	84.6	0.18	29.2
Primary Metals	90.5	--	32
Non-Metal/ Metal Mining	30.7-37.7	0.3	18-23.3
Petroleum	63.4	0.28	28.5

Sources: MSHA injury statistics (1971-1979)
OSHA injury statistics (1972-1978)

However, examining the seven year average for these industries does not give the complete picture. It would not be reasonable to set an overall safety requirement without considering the possibility that in the year 2000 coal mining, as well as other industries, may have significantly different injury rates. In fact, if one examines the trends of the above industries (Appendix C), it is clear that the aggregate injury rate for coal mining is already within the range of the four industries chosen as comparable. In particular, the coal mining trend appears to be approaching a range of 65-75 injuries/million man-hours in comparison to 25-85 injuries/million man-hours for similar industries. However, the yearly fatality and disabling injury rates for coal mining have consistently been approximately two times higher than the industries selected as comparable.

In sum, the safety requirement should stimulate a reduction in deaths and disabling injuries and must be stated in a way to allow for long-term trends in both coal mining and the designated set of comparable industries. These ideas lead to the following statement of the requirement:

> AT THE ANTICIPATED TIME OF FIRST COMMERCIAL USE, ANY ADVANCED UNDERGROUND COAL MINING SYSTEM MUST HAVE RATES FOR FATALITIES, DISABLING INJURIES, AND TOTAL INJURIES WHICH FALL WITHIN THE RANGE OF RATES EXPERIENCED BY INDUSTRIES WHICH ARE JUDGED TO HAVE COMPARABLE HAZARDS.

The requirement, as stated, requires a projection for all three categories of injuries. Examination of the fatality and disabling injury ratios reveals no trend for coal mining or for any one of the comparison industries. Therefore, we will project the fatality and disabling injury requirement by extrapolating the experience of the seven year period 1972 through 1978. Total injuries will be projected by using the trends presented in Figures C.1-1 and C.1-2 (Appendix C), and by assuming that the ratio of severe and disabling injuries to total injuries will remain constant at the values shown in Table 3-5. This analysis yields the following target rates for Central Appalachia in the year 2000:

(1) Total injuries: 40-45/million man-hours.

(2) Disabling injuries: 30/million man-hours.

(3) Fatalities: 0.2/million man-hours.

These targets are based upon projections which may prove to be pessimistic as the baseline technology evolves. Indeed, considering the length of time required to develop, test and introduce a new system, (i.e., 10-15 years), it is quite possible that the baseline technology will satisfy all aspects of the safety requirement. In that case, advanced mining systems would not need to possess advantages in safety performance.

A caveat should be expressed concerning this method of characterizing system safety performance. Incidents per million man-hours is useful for comparing mining to other industries, however, it masks the overall impact on society. Consider a system which does not reduce the man-hour injury rate, but results in a lower injury rate per ton due to greatly increased labor

Table 3-5. Distribution of Fatalities, Disabling Injuries, and Non-Disabling Injuries in Underground Coal Mining for the Years 1971 Through 1978

Injury Category		Percent*
Fatalities		0.5
Disabling, Non-Fatal Injuries		61.3
Non-Disabling Injuries		38.2
	Total	100.0

*Note: Even though the overall injury rate for coal mining is decreasing, the above percentages remain relatively constant.

Source: Annual statistics on injuries compiled by the Mine Safety and Health Administration, U.S. Department of Labor productivity. Socially, coal now costs fewer injuries, but our index shows no improvement.* No simple resolution of this difficulty can be made.

2. Secondary Safety Requirements

In selecting opportunities for improving safety performance, available data suffice to identify which hazards are presently most severe. Examination of fatality and disabling injury rates for Central Appalachia for all the various hazard categories indicates a consistently high yearly contribution from roof and face falls, haulage accidents (mostly in the form of pinch and squeeze injuries), other machinery-related accidents, and injuries sustained while handling material. According to MSHA statistics, these four hazards have continually accounted for at least 75% of all disabling injuries during the period 1971 through 1978. Therefore, it appears that the designer of an advanced system should consider reducing the incidence of these existing hazards in any attempt to meet the injury rates which form the primary safety requirement.

3. Method of Evaluation

It will be necessary to estimate the expected performance of a proposed system long before actual operating experience can be evaluated. Otherwise the system requirements will be of no value in determining research and development priorities. Such a safety evaluation method has been created by JPL, and is separately reported by Zimmerman (1981). A summary is given here.

The analytical approach is divided into two phases. The first starts with a complete system failure analysis. This initial step is important because an advanced system may have a different architecture than existing equipment, and therefore, different failure modes. This information is the basis for the next step, the system hazard analysis. Both isolated and large-scale accident modes will be included.

The matching of system failures with potential human interfaces is done considering all factors related to system operation. These factors include possible adverse weather, hostile geology, machine failures, and human error. In this manner, the attempt is made to identify and describe all potential hazards to which workers will be exposed in the performance of their tasks. In addition to the system hazard analysis, data are assembled on task times and descriptions, production rates, crew sizes, protective devices, and machinery redesign possibilities.

At this stage, a suitable comparison is chosen from existing technology. This similarity can be functional or non-functional in nature. Functionally similar systems are those which operate in a similar environment and operate in the same fashion (e.g., both extract coal using a boring process). Non-functionally similar systems are those which have only one

* There is evidence that the converse situation has evolved following the passage of the Coal Mine Health and Safety Act of 1969: Injuries per million man-hours are lower, but, because of lower labor productivity, <u>injuries per ton</u> have increased somewhat over the pre-1969 level.

thing in common, (e.g., they are both material handling machines with the existing hardware being used to load ore boats). This type of comparison is used when it is not possible to find a mining analogue for some portion of the advanced system. The same data pertaining to task times, production rates, crew size, etc., are collected for the contemporary equipment selected for comparison. In addition, historical injuries related to the major hazards associated with the various conventional tasks are also tabulated.

Next, the two systems are compared from the standpoint of hazards, and the fractional reduction (or increase) in exposure times, people exposed, and body protection afforded. For each task, man-hours at risk are multiplied by the injury rates observed for analogous equipment, and then total system safety performance is estimated by aggregating the rates for the various tasks.

Phase two of the evaluation involves an interview process, during which experts conversant with coal mining safety are provided all of the above comparative data and asked for an estimate of injuries based on their experience. This approach is deemed reasonable considering that the degree of exposure to a hazard and the resultant number of injuries, are not necessarily directly proportional to each other, and that some hazards interact with each other to increase exposure. Therefore, combining expert judgment with the projected injury estimates provides a more accurate depiction of system performance.

The experts are asked to make an initial judgment on the injury rates expected for the new system by considering: (1) the system design, (2) a comparison of hazards between the two systems, and (3) existing injury levels. If so indicated, the equipment or operating procedures are redesigned, and new hazard projections are made. The experts are provided the new data on projected injuries and asked to modify their original estimates until a final "range" of expected system injuries is reached. The final consensus on expected system performance is then compared against the requirement to measure the degree of compliance.

4. Economic Trade-Off Considerations

Present expenditures for fatalities and disabling injuries indicate that less than one dollar out of the price per ton of coal is actually spent for total compensation. Even if compensation were increased with further internalization of the social cost of death and injury, it appears that the cost of safety would still be a relatively small part of operating costs. Appendix C.2 provides a more detailed discussion of the cost of safety in comparison with operating costs.

This does not mean that safety is not an important factor in the design of advanced extraction systems. It does mean that the cost of safety (given that the new system is close to meeting the prescribed requirements) need not be traded off against performance. That is, the requirement for safety exists apart from economic issues, and is socially derived. The cost of providing an appropriate level of safety is simply added to the cost of the coal produced. The validity of this approach can be easily demonstrated. Suppose we estimate the cost of accidents today to be as much as say, \$3/ton. Two advanced systems are being compared. One reduces the cost of coal by \$3/ton while not reducing accidents, the other achieves no reduction in cost, but eliminates accidents. While the comparison is presumably a wash in terms of dollar costs, there is little doubt which mining system would be chosen.

E. PRODUCTION COST

In previous sections the goals for advanced performance in miner health and safety, environmental impact, and conservation have been identified. A second and more important thrust for the advanced system is to design new hardware incorporating performance advances that will be commercially attractive to the industry.* In short, commercialization is viewed as the ultimate goal of the program, and production cost or profitability comparisons with contemporary technology will be an important consideration in judging the commercial attractiveness of a new system. Accordingly, the production cost requirement must be set with commercialization in mind.

Production cost of coal is understood to include all of the out-of-pocket expense and a normal profit, as reflected in a minimum acceptable selling price. To be acceptable to the operator, this price must cover reasonable payments to the debt and equity holders. In addition, this minimum price must cover all of the internalized costs of assuring a safe, healthful workplace, and mitigating adverse environmental impact. Conservation performance, narrowly defined, impacts cost as well, principally through the capital recovery factor in those cases where mineral rights are leased, rather than purchased outright. With other factors held constant, the higher the recovery, the lower the capital cost per ton over the life of a mine. In sum, production cost is impacted materially by either the need or the desire to meet certain levels of performance in the areas of environmental impact, miner health and safety, and coal recovery.

Accordingly, production cost is a good overall measure of system performance, to the extent that it does reflect responsible management practice and compliance with regulations. More formally, production cost may be used (1) to aggregate the internalized cost of meeting the constraint levels set for the other requirements, and (2) to assess the cost impacts of achieving higher goals for safety or other attributes. Inevitably, however, this operator-oriented view of cost excludes certain factors such as:

(1) Society's need to cope with an abandoned leaky mine seal which was constructed according to best available technology.

(2) The external social costs of fatalities, disabling injuries, and impaired health, which resulted in spite of both the designer's and operator's best efforts to provide a safe, healthful workplace, according to regulation.

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* White (1978) presents a recent industry perspective on the characteristics of an innovation, which *a priori* favor commercial success.

(3) Possibly higher energy costs for future generations as a result of permitting current economics alone to dictate level of recovery, or the mineability of unrecovered coal.

These issues, although important, are not accounted for in production cost as defined for the purposes of judging commercial merit. Recognizing their importance, we have discussed these and other issues of similar nature elsewhere in the document.

1. Factors Involving Time and Risk

There are three general groups of factors to consider in setting a cost requirement: (1) the long lead time from conceptualization to commercial use, (2) trends in the industry and society as a whole, and (3) the economic risks in developing and commercializing a new technology.

It should be anticipated that it will take ten to fifteen years to develop, test and market a new mining system, plus up to another five to ten years before it enjoys substantial use by the industry. Both the market for coal, and the equipment available to the industry can change considerably over this period. Thus, the competition for new technology is not today's technology. Moreover, the price of coal to be used as a research and development target must consider the evolution of the demand for coal, and changes in the cost of transport from mine to market; interfuel substitution, etc. These long lead time effects can be handled via the concept of a moving baseline, which examines both improvements in mining technology and their cost implications, together with forecasts of future demand by end users to project target prices for the resource in question. Because these price targets are based on a detailed scenario of how the hardware will evolve, they are called "bottom-up" targets. As a check on these bottom-up projections, one can separately forecast prices from a "top-down" analysis which incorporates aggregate forecasts of growth in regional demand; trends in pollution control regulations; changes in freight rates; the productivity changes due to resource depletion, technological progress, evolution of labor force experience, etc.

The trends in the industry and society as a whole point to increased mechanization and automation as a solution to many problems.* The industry, concerned about inflation, sees mechanization and the corresponding productivity increases as a way to meet the impact of continually rising wages and benefits. It is reasonable to expect coal miners' wages to rise more rapidly than the wages received by the average manufacturing worker, in view of the significantly higher health and safety risks and generally unappealing working conditions, coupled with increased mobility and rising expectations within the traditional sources of mine labor. Recent legislation and regulatory authority, plus a generally more assertive and independent posture of those entering the workforce in the 70's and 80's reinforce the factors which favor substantial increases in real labor cost.

Risk is the final issue to be discussed as a background to setting a production cost requirement. Because new, possibly revolutionary technology

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* The reader is referred to an ensemble of articles treating this topic in Coal Age (July 1975).

is being dealt with, risk is <u>the</u> central cost issue. The question is, can risk be quantified, and if not, how should it be handled within the context of a system requirement?

Before trying to answer this question, it is important to explore the nature of risk as a project progresses through the research and development cycle. At the beginning of a project, when it exists only as a conceptual design, technical risks predominate. A central concern is, will it work at all; which later gives way to will it work as designed? Once feasibility is established and the design is firm, manufacturing risk becomes the issue: can the machines be made on a production line basis for the cost projected by the designers? Will the equipment match the performance of the prototype in terms of both output and reliability? When the manufacturing problems are overcome, the focus shifts to the coal operator who will use the new equipment. The operator confronts a substantial applications risk: will the new machines live up to the manufacturer's promises of tons per machine shift and operating cost? What is the risk of an equipment failure jeopardizing the ability to meet contract deliveries and subsequently, financial obligations to debt holders? What is the possibility that this new equipment will, within its lifetime, be made obsolete by evolutionary development of predecessor technology, or by even more advanced machines now under development?

In theory, the attempt can be made to describe, quantify, and aggregate all of these various risk factors into one or a few carefully constructed figures of merit. Indeed, the practitioners of decision analysis do just that, with the degree of success being highly situation dependent.* This effort views decision analysis as most useful in those cases where there is one well defined locus of decision. In this case, the decision involves a wide spectrum of participants from both the public and private sectors, and thus, the practicality of a formal decision analysis is questionable, whatever its merits might be.

Another, traditional, approach to risk evaluation in such a case is break-even and sensitivity analyses, with searching examination of the possible negative factors, coupled with step-by-step, limited scope design, modeling, and experimentation, leading eventually to a laboratory mock-up, limited field trials, and finally demonstration test. As significant new information on feasibility and equipment performance is obtained, the production cost projections are refined, but no great amount of confidence is attached to these calculations until the demonstration test is complete and its results scrutinized. It appears that this second mode of handling risk is more appropriate to the development of a radically new mining system.**

Before moving to a formal statement of the production cost requirement, it is useful to summarize the points made in the introductory discussion:

(1) Commercialization must be the primary consideration, with profitability set high enough to attract a substantial group of users.

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* Hertz (1979) describes a formal procedure for treating risk in an explicit fashion.

** Frantz (1979) and Suboleski (1979) indicate a strong industry preference for this process-oriented way of dealing with risk.

(2) Profitability must be assessed against a moving target, with production cost goals predicated on reasonable projections of equipment capability 10 to 20 years into the future.

(3) Although quite important, near-term profitability is not the only consideration: insulation from continuing labor cost inflation, plus the ability to cope with the risks inherent in new technology are major concerns of the potential user.

(4) Finally, the assessment of economic risk as seen by an operator 10 to 20 years hence must figure explicitly in the production cost requirement -- in effect, the evaluation of risk is at the heart of this requirement. However, any quantification of risk must be appropriate to the incomplete data and many unresolved issues characteristic of the early stages of systems definition and development.

2. Statement of the Production Cost Requirement

Development of a commercially acceptable system is the primary goal of the advanced mining system program, and production cost advantage is presumed to be the major determinant of commercial attractiveness. The statement of the requirement itself will be immediately followed by necessary definitions and clarifications which, taken together, will provide the rationale for the requirement. This is followed by a brief description of how to evaluate system performance against the requirement.

ANY ADVANCED MINING SYSTEM WHICH IS A SERIOUS CANDIDATE FOR DEVELOPMENT AS A COMMERCIALLY ATTRACTIVE MEANS OF EXTRACTING A SPECIFIED RESOURCE, MUST SHOW PROMISE OF YIELDING A RETURN ON INVESTED CAPITAL (ROI) OF AT LEAST 1.5 to 2.5 TIMES THE MINIMUM TARGET ROI REQUIRED BY THE INDUSTRY FOR ITS AVERAGE CAPACITY EXPANSION OR REPLACEMENT PROJECT AT THE PROJECTED TIME OF FIRST COMMERCIAL USE.

3. Definitions and Clarifications

a. Mining System. The mining system includes mine design, site and seam access, initial development, the period of nominal capacity production, and mine close. In addition, the system includes all of the activities required to break the coal away from the seam, transport it to the surface, and prepare it for shipment to the customer.

b. Commercially Attractive. Some development is undertaken primarily to (1) demonstrate that a technically feasible solution exists to a previously posed research or development problem, or (2) explore possible solutions to such problems. This production cost requirement is not meant to apply to such exploratory research or proof-of-concept experiments, but rather to projects whose technology is developed to the point where it is meaningful to contemplate design, fabrication, and test of a system expected to be ready for commercial use, more or less in the form originally conceived. As indicated in the statement of the systems requirement, the degree of commercial attractiveness will be judged against an ROI criterion.

c. Specified Resource. Any evaluation of a candidate system must consider the range of mining conditions characteristic of the resource for which the system is designed. These conditions typically include such factors as seam geometry (thickness, dip, and access); roof and floor quality (includding any factors such as water, joints, residual stress which could affect bearing properties of the ground); methane liberation rate (which may vary with the cutting technology); and various anomalies (faults, folds, partings, pinch outs, undulations in roof and floor, etc.). To permit a determination of commercial attractiveness one must define the range of conditions in which the system may be used and specify a set of nominal conditions judged to be representative of average conditions likely to be encountered. To the extent possible, the system designer is urged to describe how production, equipment, manning, and consumables are expected to vary over the range of conditions for which the system is designed.

d. Show Promise. As indicated in the introductory discussion, risk is the central issue in any calculation of commercial attractiveness. For purposes of demonstrating commercial promise, it should be assumed that all problems of technical feasibility, and manufacturability will be solved for an R&D expenditure typical of a development project of the type and magnitude proposed. Thus, amortization of R&D expense per system sold should be in line with current and past industry experience, and should be considered in all projections of equipment cost. Moreover, the assumptions used to determine overall production cost, or cost savings should reflect conservative assumptions about initial construction and development expenditures, equipment cost, manning, consumables, and time actually available for production (in view of specified mining conditions, and cycle times). "Conservative," means average day-in/day-out performance, when operating in average conditions, using a work force of average ability and experience, working under rules typical of the region in which the resource is located, etc. Performance projections corresponding to ideal operating rates, equipment availabilities, work rules, etc., are not appropriate for the conservative meaning accorded to the phrase "show promise."

e. Return on Invested Capital. Return is defined as the discounted cash flow return on the incremental investment in new equipment over the life of a representative mine. This so-called "internal rate of return" is that value of the discount rate which yields a net present value of zero, when applied to all of the incremental cash flows. In order to determine the incremental cash flows, one must identify (1) the incremental investment, and (2) the incremental change in the contribution to costs and income as a result of using the new equipment. These incremental flows are used to compare the life cycle cost of a representative mine equipped with the advanced system, to the life cycle cost of a mine of the same size equipped with technology specified in the moving baseline for the year 2000. The baseline technology selected for this comparison is the one deemed most suitable for the mining conditions of interest, and the mine size is chosen to be appropriate to the advanced system.

Although it is possible to design a system that is totally new from mine opening and initial development to mine close, it is also possible that the investment in new hardware will be concentrated in certain areas, such as face equipment or main haulage. To fix ideas, consider a system whose only new element is face equipment. In this case, the incremental investment is merely the difference between the expenditures on the new and the old equipment including both initial outlays and subsequent rebuilds or replacement of components important enough to qualify as capital items.

The incremental change in contribution to fixed cost and profit is determined in a similar fashion. Let us continue with the example of changed face equipment. Given a selling price f.o.b. mine* and a section production rate, one may compute the gross revenue generated by both the new and the old section equipment. To compute a contribution to fixed costs and profit, for each complement of machines, subtract from the gross revenue, the operating and maintenance costs, and account for the tax impacts of depreciation, depletion, investment tax credit, etc. The resulting incremental cash flow generated by the new equipment is discounted until its present value just equals the present value of the incremental investment. This is the unique internal rate of return associated with the new equipment.

f. Target Return on Investment (ROI). The target return on investment is based on the after-tax return required of a relatively risk-free capacity expansion or replacement project at the time of first commercial use (presumed to be approximately the year 2000). The addition of a new section to an existing mine, or the replacement of old section equipment are good examples of relatively risk-free capacity expansion and replacement projects. Because of a presumed effective federal tax rate of 50%, the before-tax return is generally held to be twice the after-tax return. Economists break the after-tax return into two components: (1) the long-term real rate of return, which is highly correlated with the growth in productivity, and (2) the long-term rate of inflation. If a long-term growth in productivity of 2.5 to 3% and a sustained inflation rate of 8 to 9% is assumed, then a reasonable after-tax rate of return would be 10 to 12%.

The above discussion is relevant to investments with relatively little risk, in contrast to the purchase of advanced equipment, which is generally perceived to embody considerable risk in the early stages of commercial use, no matter what level of performance may have been indicated in field tests and demonstrations. Mansfield (1978), conducted extensive research to determine what "risk premium" innovators (early adopters) demanded for the purchase of new equipment. Table 3-6 presents Mansfield's results for twelve innovations which eventually saw wide use in railroading, iron and steel production, coal mining, and brewing. Mansfield's measure of relative profitability was the ratio of the payback required on the firm's typical capital project to the payback projected for the innovation. Note that for the innovations studied, this ratio varied from 1.2 to 5.0, with 1.6 to 2.0 being the range for the three coal mining innovations. Payback has now given way to more sophisticated measures of profitability, such as internal rate of return and net present value (corresponding to a fixed rate of return). In consequence, the effort was made to transform Mansfield's profitability ratio into a ratio of rates of return. The method for making this transformation is presented in Appendix D. It is shown there that, using reasonable estimates for bounding parameters, the payback ratios for mining innovations found by Mansfield of 1.6 to 2.0 can be translated into ROI ratios of approximately 1.5 to 2.5.

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* This calculation is directly dependent on the market price, perhaps twenty years in the future, which is difficult to predict with any accuracy.

Table 3-6. Empirical Basis for the ROI Target

	Innovation	Sample Size	Profitability* Ratio
	Diesel Locomotive	25	1.59
	Centralized Traffic Control	24	1.48
	Car Retarders	25	1.25
	Continuous Wide-Strip Mill	12	1.87
	By-Product Coke Oven	12	1.47
	Continuous Annealing	9	1.25
o	Shuttle Car	15	1.74
o	Trackless Mobile Loader	15	1.65
o	Continuous Mining Machine	17	2.00
	Tin Container	22	5.07
	High-Speed Bottle Filler	16	1.20
	Pallet-Loading Machine	19	1.67

* The ratio reported is anticipated payback for the firm's average capital project, divided by the payback projected for the innovation.

o Coal mining innovations.

Adapted from Table 7.1, Mansfield (1968)

Of course, this profitability ratio or risk premium concept is strictly applicable only at the point in time when an innovation is ready for use by the industry. The systems requirement on production cost must be stated in a way that provides for the evaluation of a conceptual design, whose performance is considerably less certain than a commercially available piece of hardware. Two ways around this difficulty are suggested. First, as indicated above, preliminary estimates of profitability should be based on conservative projections of production, manning, equipment cost, and consumables. Second, the preliminary screening of a concept should identify areas of performance where uncertainty implies a substantial variance in the profitability projections. It is presumed that further work on a concept would focus on resolving those uncertainties, and subsequently recomputing the return on investment.

Finally, the minimum ROI ratio was set in terms of a range of 1.5 to 2.5. This is regarded as a lower bound on the ratio. For riskier developments, it may be desirable to require a correspondingly higher ratio.* How to quantify risk and relate a particular judgment about risk to an ROI ratio is not at all clear, or may not be the best way to deal with the issue. As indicated above, it may be more illuminating to perform some simple experiments bearing on feasibility.

g. Target Price. The target price is that price presumed to exist in the marketplace at the time the new equipment is introduced. This is the price that will be employed to determine the incremental revenue generated by the new equipment. Because coal is a commodity often sold under long-term contract, it is recommended that the target price be the contract price forecast for the time period of first use. Under a contract from JPL, Energy and Environmental Analysis, Inc., of Arlington, Virginia has projected long-term contract prices for the years 1985 and 2000 (see Table 3-7). These prices are quoted in 1980 dollars and reflect the after-tax rate of return currently realized by the industry.

h. Method of Evaluation. The procedure for evaluating the production cost performance of a candidate advanced mining system is a straightforward application of existing tools. The analysis begins with the description of the advanced system, as it would be employed in a mine of a size large enough to realize the inherent economies of scale. Estimated capital investment and operating costs should be assembled in a format suitable for subsequent discounted cash flow analysis. A good example of the recommended format may be found in Duda (1978). Section production, and ultimately the annual capability of a mine may be determined from the sort of equipment cycle analysis employed by Floyd (1977) and Bickerton (1980).

Next, a baseline system for the resource in question should be selected, and both cost and production performance should be projected to the year when the advanced system is expected to be commercially available. Bickerton (1980) has projected year 2000 performance for room and pillar, longwall, and shortwall technology. (To ensure a close comparison, the baseline technology should be analyzed while operating at the same site as the

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* Hill (1979) and others indicate that this is the sort of heuristic procedure industry typically uses to handle varying degrees of risk.

Table 3-7. Target Prices for Central Appalachia for the Years 1985 and 2000

Coal Type	1985 Price ($/Ton)	1985 Annual Production Million Tons	2000 Price ($/Ton)	2000 Annual Production Million Tons
Compliance Coal (1.2% Sulfur)	$29.50-31.50	128	$33.00	174
Low Sulfur Coal (1.2-2.0% Sulfur)	$28.00-28.50	93	$33.00	180
High Sulfur Coal (2.0% Sulfur)	$28.00-29.00	26	$31.00	46
Total Production		247		400

Source: Energy and Environmental Analysis (1980) one selected for the advanced system.) Clearly, the information describing the baseline technology should be put in the same format as the one used for the candidate advanced system.

Revenues from each mine must be projected by using the market prices presumed to hold at the time the candidate system is introduced. An overview of prices expected in Central Appalachia for period 1985-2000 has been given. A more extensive discussion of price projections for all major regional resources may be found in Terasawa (1980). A summary of this document is provided in Appendix I.

Once the cash flow profiles have been assembled for the candidate and baseline systems, it is a simple matter to obtain a schedule of the cash flow difference, on a year-by-year basis. This schedule of incremental cash flow may then be analyzed using standard techniques (see Duda (1978) for example) to determine the internal rate of return.

Next, this rate of return must be compared with that rate which industry is expected to require at the time the candidate advanced system will be commercially available. As indicated above, the comparison rate is related to productivity gains and the expected rate of inflation.

Finally, a sensitivity analysis should be conducted on those parameters which are relatively uncertain, for example, market price, mine size, and the projected cost and cycle times for the new equipment.

SECTION IV

IDENTIFICATION OF RESOURCES

The objective of this task is the identification of those domestic coal resources having commercial significance in the year 2000 and beyond. In Phase II of the 1980-81 work, system requirements will be developed for one or more of these resources judged to be suitable for advanced mining technology. This effort is divided into the following three sub-tasks, each of which is discussed in detail below:

(1) Basin identification and description.

(2) Characterization of geologically important resources.

(3) Identification of commercially important resources.

A. BASIN IDENTIFICATION AND DESCRIPTION

The objectives of this task are to identify the resource targets and develop the classification scheme to be used in the remainder of the analysis.The resource targets, nominally called basins or coalfields, are delineated by applying recently developed theory on depositional environments (as described by Horne et al., 1978) to the information contained in geologic maps and written reports. The major units of analysis are the following five major coal provinces:

(1) Alaska, focusing primarily on the Brooks Range coals.

(2) Appalachia, broken into the Dunkard, the Pocohontas, and Warrior Basins.

(3) The Gulf Coast lignite deposits.

(4) The Interior Province, partitioned into Eastern and Western coal groups.

(5) The Western coals, which encompass some 25 distinct basins or areas.

Typically, each of these basins is itself, broken into sub areas for an indepth analysis of the coal resources. Drilling logs and borehole data are used selectively, wherever available. A first cut analysis indicates that there are about thirty major basins within the continental United States.

The classification scheme, basically an elaboration of the list of standard mining conditions, was constructed to encompass the diversity of conditions in the 15 original type locales, designated at the beginning of this project (Table 4-1). Results obtained to date indicate that it is feasible to include the following factors in the classification scheme:

Table 4-1. Salient Mining Conditions Associated with the Project's Fifteen Initial Type Locales

Type Locale	Salient Mining Conditions
Marylee Group, Alabama	Multi-seam, deep, met.
Multiple Seams, S.E. Uinta, Colo.	Deep (6000), thick, some steep
Herrin #6 (Macoupin), Illinois	Shallow, thick, limestone cover
Punch Mines, Kentucky	Inefficient practice in zone below strip
Thin Seams (Met), Kentucky	Flat, abundant resource in seam [illegible]"
Abandoned Mines (Pitts), Penna.	Approx. 50% of original resource
Anthracite, Southern Field, Penna.	Contorted geology, weak market
Pittsburgh Seam, Penna.	Typical contemporary industry
Sewickley Seam, Greene Co., Penna.	Dirty, modest thickness
Kaiparowits Plateau, Utah	Spectrum of conditions, undeveloped
Pitching Seams, Tabby Mtn., N.W. Uintah	Steep dips in steep terrain
Deep Seams, Virginia (S.W.)	Depth ranges below contemporary capabilities
Vertical Seams (Roslyn), Washington	Thin, steep, dirty
Deep Seams, Wyoming Co., W. Va.	Mountainous terrain issues deep cover
Thick seams, Wyoming	Thick to 2100', deep to 10,000, some steep

(1) Overburden.

(2) Dip (regional only).

(3) Seam thickness and interburden between seams.

(4) Presence of locally important faulting.

(5) Coal quality; especially Btu value, percent sulfur, and ash content.

An analysis of this scope cannot consider roof and floor conditions, methane content, and the potential for water flooding. Note that the above classification scheme is adequate to describe all of the fifteen type locales listed in Table 4-1.

This task was completed with a brief estimate of the aggregate tonnage in each of all basins identified, with no attempt at classification according to conditions. These estimates were obtained by focusing on closely associated groups of seams or formations, with the precision of the estimates being one billion tons. Results of this preliminary analysis of tonnage are summarized in Table 4-2, where the September 1980 estimates prepared by this task are compared with figures published in the 1980 Keystone Coal Industry Manual.

The preliminary estimates of Table 4-2 permit an obvious narrowing of focus of the next phase of resource identification--detailed basin description. Table 4-3 lists those basins excluded from further analysis because the tonnages are relatively insignificant. Note that the aggregate tonnage of the excluded basins is only about 0.7% of the total preliminary estimates for the domestic resource.

B. CHARACTERIZATION OF GEOLOGICALLY IMPORTANT RESOURCES

The objective of this task is to identify, in terms of the classification scheme developed previously, those resources in place which account for substantial fractions of the national total. This is done by classifying the total tonnage in place in each coal-bearing formation of each basin, and then aggregating tonnages corresponding to selected combinations of conditions (i.e., particular cells in the classification scheme). No adjustments are made for reasons of potential environmental impact, presumed difficulty of mining, lease restrictions, etc. This analysis is proceeding on a basin-by-basin basis, and is building upon the preliminary work done in the Basin Identification Task. The work of detailed basin description begins by breaking each basin into subareas which exhibit more or less uniform geology. The number of subareas used ranges from two for the Interior Province to twenty-five for the Western Coals. Typically, the depositional setting and coal thickness within coal bearing formations is inferred from a set of 15 to 20 boreholes selected to give a good overview of each subarea, in addition to boreholes and well logs. Tonnage estimates will make extensive use of published reports and maps. Tonnages are obtained via planimeter analysis of mapped regions for which average coal thickness has been estimated. The uncertainity for all basin tonnage components will be characterized in terms of 95% confidence intervals, obtained via standard statistical techniques

Table 4-2. Preliminary Estimates of the Domestic Coals Resources by Basin (in Billions of Short Tons) as of 8/31/1980

Basin	Preliminary Estimate	Keystone*	Difference
Eastern Anthracite Basin	89.4	30.3	+ 59.1
**Eastern Triassic Fault Basins	2.8	0.3	+ 2.5
Pittsburgh (Dunkard) Basin	306.6	146.6	+ 160.
Pocohontas Basin	180.0	137.1	+ 42.9
**Cumberland Plateau	13.2	0.8	+ 12.4
**Michigan Basin	6.3	0.7	+ 5.6
Warrior Basin	182.3	48.7	+ 133.6
**Cahaba & Coosa Fields	16.3	25.3	- 9.0
Arkoma Basin	17.9	37.0	- 19.1
Illinois Basin	202.7	354.2	- 151.5
Western Interior Basin	223.3	93.5	+ 129.8
Gulf Coastal Plan	931.7	28.8	+ 902.9
High Plains Tertiary Basins	1,188.4	966.7	+ 221.7
Rocky Mountain Intermontaine Basins	1,611.7	1,073.0	+ 538.7
Alaska Intermontaine Basins	7,181.4	238.5	6,942.7
**Cordilleram Fault Bounded Basins	17.0	52.0	- 35.
Alaskan Cord. Fault Bounded Basins	834.9	26.4	+ 808.5
	13,005.9	3,259.9	9,746

* Estimates computed from the 1980 Keystone Coal Industry Manual

** Excluded from an in-depth analysis

Table 4-3. Resources Excluded from an In-Depth Analysis

Basin		JPL Resource Estimate (Billions of Tons)
Eastern Triassic Fault Basin		2.8
Cumberland Plateau		13.2
Michigan Basin		6.3
Cahaba & Coosa Fieldss		16.3
Arkoma Basin		17.9
Cordilleron Fault Bounded Basins		17.0
	Total	73.5

(Table 4-4). No attempt will be made to reconcile this treatment of uncertainty with the hierarchical breakdown of "measured, inferred, and hypothetical" resources. As in the preparation of preliminary basin tonnages, the level of precision is one billion tons, which in turn dictates a scale of about 1:500,000 for the maps used in the planimeter work. The geological work described above is being done under a subcontract with the University of Kentucky, with Professor John Ferm as the principal investigator.

C. IDENTIFICATION OF COMMERCIALLY IMPORTANT RESOURCES

The objectives of this task are to: (1) identify resources of commercial significance, and (2) recommend which of these resources should be targeted for the development of advanced underground mining systems. This task is, in essence, a limited-scope policy analysis with the primary focus being resource exploitation and its impacts on national energy policy. More specifically, the overall objective is the identification of those resources which will permit the production of coal of a cost low enough and quantities high enough to reduce the aggregate cost of energy as much as possible and diminish the dependence on foreign sources of hydrocarbons.

The work began with a consideration of energy demand scenarios for the year 2000+. Four scenarios were selected: a baseline scenario, as described by Terasawa and Whipple (1980), one "high", and two "low", demand scenarios, based upon previous work by DOE, DOI, EPRI, and others with interests in long-range demand projections.

Table 4-4. Illustrative Treatment of the Uncertainty in Aggregate Tonnage Estimates

Example: Mesaverde Formation, located in the Wasote Basin, Utah	
Measured planform area of the formation	5200 sq mi
Mean thickness of coal within the Mesaverde Formation	15.5 ft
Standard deviation of coal thickness	4.0 ft
Best Estimate of aggregate tonnage within the Mesaverde Formation	96 billion tons
95% Confidence Interval for aggregate tonnage estimate	73-119 billion tons

These scenarios will be assembled in light of the results of the geological work described above. A major effort will be the assessment of the impacts of the various scenarios on the variables determining the economic value of the coal expected for the era 2000+. Analysis of these demand scenarios, together with the previously assembled geological information, will consider factors which bear upon the commercial significance of a resource, for example:

(1) Location of the likely markets served, including synfuel and export markets.

(2) Projected prices and market shares of fuels competing for these markets.

(3) Projected transportation cost from resource to market, including a consideration of new railroad lines, barge canals, slurry pipelines, etc.

(4) Projected minimum selling prices for competing resources which are presumed to be in production in the era 2000+.

(5) Likely ease with which a new resource could be developed once the technology is ready.

Commercially important resources will be those which might reasonably be expected to compete with currently forecast sources of supply in the era 2000+. For example, it is expected that strippable coal will always be cheaper than deep coal, that thick coal will always be more attractive than thin coal, and deposits containing much rock will always be more costly to work than seams with little waste. Finally, recommendations about the commercially important resources attractive for research and development effort will take into account the readiness of the technologies which are likely to be incorporated into new extraction systems.

The findings of the Resource Identification Task including all of the geological work and subsequent formulation of recommendations about resource targets will be documented in the form of a final report to the DOE sponsor. Delivery of the draft report has been promised for December 1981. Thus, the results described above must be regarded as a report of work in progress.

SECTION V

TECHNOLOGY ASSESSMENT AND CONCEPT DEVELOPMENT

This section summarizes activities during 1979-80 in two additional areas of direct relevance to the long-term project goal of designing building, and field testing new underground coal extraction systems. The first part of the section discusses two recently completed assessments of new technology identified as potential building blocks of an advanced mining system. The second portion describes the results of a brief exercise in conceptual design undertaken in early 1979 to gain a better appreciation for feasible ways to improve overall system performance, with a view to reflecting the results in design requirements.

During the period January 1979 through December 1980, the Advanced Coal Extraction Systems Definition Project completed two technology assessments:

(1) A broad survey of selected government sponsored research, supplemented by a few privately funded research efforts which were accessible.

(2) An in-depth look at slurry haulage adapted for in-mine transport of coal.

The first survey, performed by Gordon (1980), was a necessary initial step in determining two broad state of the art in mining R&D. Slurry haulage was chosen as the subject of the second assessment by Maynard (1981) because of its potential to alter dramatically the low utilization of face equipment (currently averaging about 30% of the time the equipment is available to work). Key portions of each assessment are excerpted below.

A. A REVIEW OF UNDERGROUND MINING EQUIPMENT RESEARCH AND DEVELOPMENT

1. Scope of Assessment

This report summarizes the findings of the technology assessment task, of the Advanced Coal Extraction Systems Definition Project. Time limitations and economic constraints dictated that the study primarily address mining technology in the United States, with emphasis on the active research and development projects having near-term application to underground coal mining systems. As a result, the assessment was concerned with hardware-oriented projects. Institutional problems were not addressed.

2. Objective

The objective was to describe and evaluate the physical capabilities and costs of technologies which might be incorporated into future mining systems. These technologies included the evaluation of present-day coal mining system components, existing technologies not presently used in coal mining but having potential for incorporation into advanced systems, and totally new concepts developed to advance coal mining techniques.

3. Assessment Method

The resources available for this task were limited and it was recognized that only a limited number of the many research and development projects could be investigated. Although a great deal of information was available through published reports and journal articles, it was evident that personal visits to the organizations involved in the research and development projects would provide a valuable insight into the details of the project. Therefore, visit selection criteria were established. It was agreed that the visit should: (1) Provide representative R&D projects by government agencies, industrial firms and universities; (2) Sample R&D in the areas of coal cutting, roof control, haulage, mine development and ventilation; (3) Examine projects that might contribute to advanced mining systems; (4) Look at organizations that were likely to cooperate in providing needed information.

4. Data Sources

Twelve organizations were visited and twenty-two projects discussed:

Organizations Visited

(1) Food Machinery Corporation (FMC) - San Jose, CA
(2) Bureau of Mines (BOM) - Spokane Center, Spokane, WA
(3) Virginia Polytechnic Institute and State University College of Engineering - Blacksburg, VA
(4) West Virginia University - Morgantown, VA
(5) Bureau of Mines - Bruceton Center, Pittsburgh, PA
(6) Pennsylvania State University - State College, PA
(7) Joy Manufacturing Co. - Franklin, PA
(8) Foster Miller - Waltham, MA
(9) Continental Oil Company - Ponca City, OK
(10) John T. Boyd Associates - Pittsburgh, PA
(11) D'Appolonia - Pittsburgh, PA
(12) Department of Energy (DOE) Headquarters - Washington, DC

Projects Reviewed

(1) Coal Cutting

(a) Hydraulic
(b) Kloswall Longwall Mining System
(c) Underground Auger Panel Extraction System
(d) Double Conveyor Longwall
(e) Miner/Bolter System
(f) Automatic Extraction System
(g) Automated Longwall System
(h) Automated Longwall Shearer

(2) Roof Control

(a) Portable, Remote Operated Crib
(b) Automatic Bolters
(c) Greater than Seam Height Roof Drills and Bolters
(d) Resin Bolt Bonding Verification
(e) Narrow Roof Bolter

(3) Haulage

(a) Coarse Coal Slurry Transport System
(b) Coal Injector for Coarse Slurry Transport
(c) Auto-Track Bridge Conveyor Train

(4) Mine Development

(a) Blind Shaft Borer
(b) Tunnel Boring Machine
(c) Raise Boring Shotcrete Support System
(d) Slip-Form Tunnel Walls

(5) Others

(a) Mine Sealing System
(b) Automated monitoring of mine ventilation

5. Survey Conclusions

Approximately two months were spent in project visits and review. The information obtained was analyzed and in-house reports were prepared. The results of this effort are presented in this document. While the general physical capabilities of the subsystems were obtained, detailed performance information and cost estimates were not available. Nonetheless, some general observations regarding current R&D are possible:

(1) Most of the projects are modifications or extensions of present technology.

(2) Continuous miners produce coal for only a small fraction of available time, being mainly limited by haulage and roof control and secondarily by machine reliability and mining operations.

(3) Longwall applications will continue to grow but at a slow rate unless steps are taken to match the equipment with United States mining conditions, reduce equipment moving time, and reduce the capital and operational costs of the system.

(4) There is a trend toward developing multiple function subsystems (combined miner-bolter, for example) which are extremely complex and may prove to be less reliable since a failure of one function results in complete shutdown of production from that machine.

(5) Total automation of mining systems might eventually be developed but for the foreseeable future (year 2000) the reliability, maintainability, and high cost of such complex systems will prevent commercialization. Semi-automated, remote controlled systems, having a considerable degree of human intervention, will be the more likely development leading to removal of mining personnel from the hazardous environment.

(6) Because of the detrimental environment of an underground mine and the severe operating demands, machinery has poor reliability (one hour maintenance for every hour of operation of a continuous miner, for example). Efforts should be made to improve machine design and component reliability as well as improving ease of maintenance and repair.

(7) There appears to be a problem in commercialization of many of the developments from the government-funded R&D and demonstration projects. The reasons for this are not clear but one major cause may be the reluctance of operators to invest in equipment which has not had years of proven service.

Additional detail about this survey may be found in Gordon (1980).

B. CRITICAL PARAMETERS FOR COARSE COAL UNDERGROUND SLURRY HAULAGE SYSTEMS

1. Background

Current projections of areas deserving emphasis for future mining systems technological innovations include haulage. Two of the primary reasons haulage is considered to be one of the most critical areas of coal mining activity to be improved are: (1) it is important to the continuity of the mining activity, and (2) it has a significant effect upon the safety of the mine workers. Continuous mining machines cannot be operated without interruption as a result of the inability of other operations, including haulage, to handle its capacity of coal output. Accidents involving contemporary haulage systems are second only to rockfalls in regard to their potential hazard to the miner and they may occur at any location throughout the mine. The objective of this technology assessment task was to provide an understanding of the basic parameters which directly influence the behavior of a slurry haulage system and determine its performance limitiations.

A coarse particle slurry haulage system has been considered by many investigators to be a viable alternative to the conventional method of in-mine coal transportation via shuttle cars and belt conveyors. Such a slurry haulage system would consist of hydraulic subsystems for conveying run-of-mine coal via pipeline from the continuous mining machines located at the face of the underground coal seam to a surface loading destination or preparation plant. Alleged advantages of this means of conveyance include improved health and safety for the miners and increased mine productivity. According to Poundstone (1977), these benefits are anticipated because the slurry haulage system will:

(1) Reduce the number of transporation transfer points and lessen the quantity of moving equipment located underground, leading to fewer operational delays and safety hazards.

(2) Entrain the coal dust and methane gas within the water, thus reducing the possibility of dust explosions and eliminating the generation of airborne dust at locations away from the seam face.

(3) Minimize the occurrence of spillage in transit which creates hazardous situations for miners and results in lost time for clean-up along the tracks and belts.

2. System Concept

References made in this report to the slurry haulage system are, in actuality, describing only a portion of the overall slurry transport system. Figure 5-1 illustrates this concept, and the boundaries of the slurry haulage system which were considered in the report are shown by dashed lines. It should be noted that neither the winning nor breaking functions, which are necessary operations that must be performed at the front end of the transport system, are within the system boundaries. The usual dewatering, cleaning, and sizing operations, which are typically performed at the coal preparation facility located on the surface, also lie outside the scope of this study. Consequently, the intent of this report is to provide useful information for (1) characterizing the slurry flow, (2) describing the influences and operational limitations which result from varying significant parameters, and (3) furnishing equipment design and utilization considerations for the conveyance of mined and broken coal via a water slurry from the seam face in an underground mine to a surface destination immediately outside of the mine entrance.

3. Equipment and Operation

The design of a slurry haulage system for the purpose of transporting mined coal from the seam face out of the mine requires, at a minimum, equipment to perform the following functions. A feederbreaker is necessary to crush the coal from the mining machine to a size which is compatible with the capabilities of the hydraulic components of the haulage system. Pumps are used both to overcome the friction losses which are present in any hydraulic system and to satisfy the lift requirements which are mandated by the elevation difference between the coal seam face and the desired delivery point for the coal slurry. The slurry is conveyed in pipes with their attendant valves and miscellaneous hardware.

This section is limited to a brief discussion of the primary hydraulic components of the system, namely the pumps and piping. It is assumed that the coal is supplied in a satisfactory particle size consist and provisions have been made to introduce it into the hydraulic system.

In addition to satisfying the slurry haulage system requirements, there are other equally important considerations regarding the mechanical equipment. Safety is of prime importance, and the operation of the equipment must be such that it functions in a nonhazardous manner and does not create a

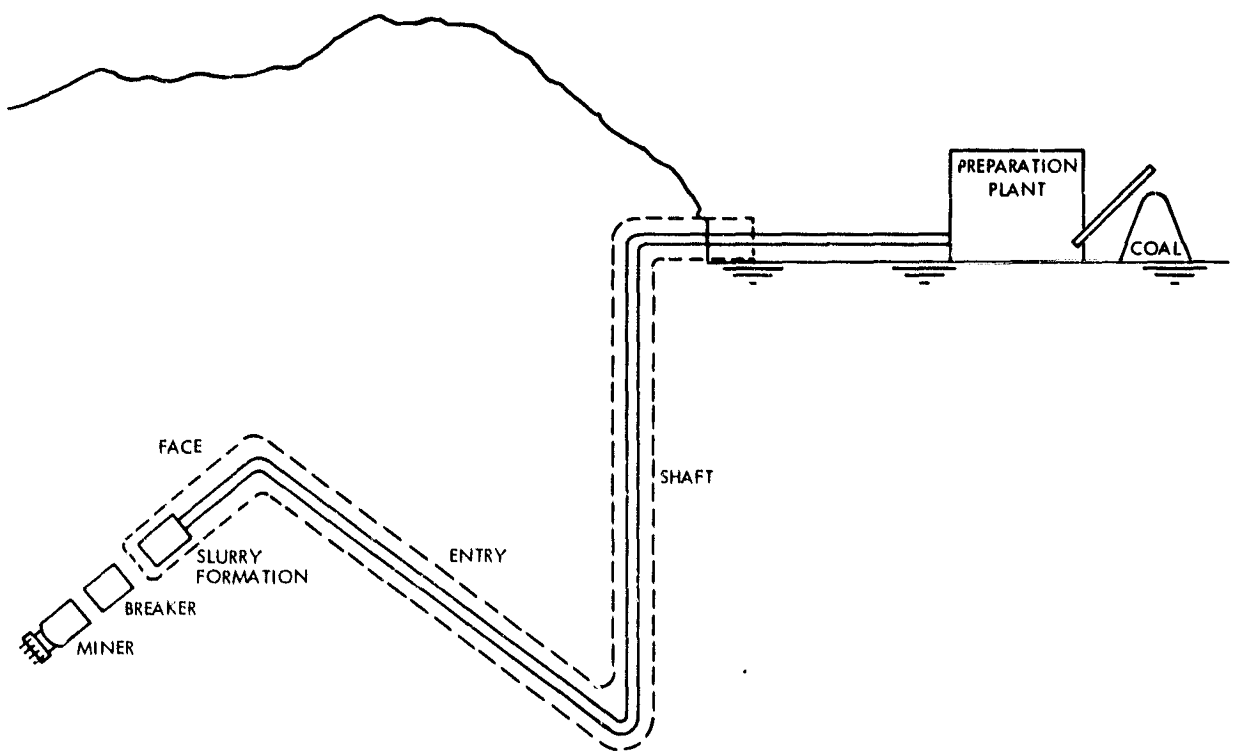

Figure 5-1. Abbreviated Schematic for an Underground Slurry Haulage System

hazardous situation in the event of failure. The reliability of such a slurry haulage system is a significant factor, as it is with any pipeline transportation system, and its advantages over more conventional transportation schemes are rapidly diminished if the system is not reliable. The third factor, which is of paramount importance when designing a slurry haulage system for the underground transport of coal, is consideration of the equipment size constraints which are imposed upon the system components as a result of the mine environment and its limitations.

4. Pumps

The pumps which are utilized for slurry transportation may generally be categorized by two generic types. These classifications are centrifugal, and positive displacement pumps. Each type has certain advantages which, depending upon the specific application, usually determines which is the more suitable type. Generally speaking, centrifugal pumps are capable of handling quite high rates of flow but at a discharge pressure less than the positive displacement units. The centrifugals also have the ability of pumping a slurry with a maximum allowable particle size that is considerably greater than the positive displacement pumps can transmit. This attribute is considered significant for mine haulage systems since the reduction or elimination of crushing operations performed on the coal in the mine prior to its introduction in the slurry is an important factor.

The maximum size of solid particles which can be handled by centrifugal pumps is usually determined by dropping spheres of graduated sizes through the passages of the pump impeller and volute. This procedure will ascertain the maximum size sphere which can be passed through the pump without becoming lodged, but in no way implies an optimum size for hydraulic operation. Due to the reduced passages which exist in positive displacement pumps as a result of their inherent tight valve seating configurations and close tolerances, more severe limitations exist regarding particle size transmission. However, the positive displacement pumps have the capability of generating very high discharge pressures and usually operate at a higher efficiency than the centrifugals at the hydraulic conditions typical of slurry haulage applications. Consequently, positive displacement pumps are excellent for high pressure applications such as providing the driving force for pipe feeders or lockhoppers to obtain large vertical hoists. Tabulation of the performance limitations which apply to both centrifugal and positive displacement pumps is shown in Table 5-1.

Centrifugal pumps are usually chosen for underground haulage systems and for moderate vertical hoist requirements due to their ability to handle large particles and their smaller equipment size. The centrifugal pump is a very adaptable piece of mechanical equipment. Provided sufficient power has been included in the driver, it is possible to vary the head-capacity performance of the pump as a function of its rotational speed. The relationships which govern its operation are as follows:

$$\frac{Q_1}{Q_2} = \frac{RPM_1}{RPM_2}$$

$$\frac{H_1}{H_2} = \frac{(RPM_1)^2}{(RPM_2)^2}$$

$$\frac{HP_1}{HP_2} = \frac{(RPM_1)^3}{(RPM_2)^3}$$

where, Q = capacity or flow rate,
H = discharge pressure or head,
HP = horsepower, and
RPM = rotational speed of pump.

Table 5-1. Slurry Pump Performance Limitations

Pump Type	Maximum Discharge Pressure, (psi.)	Maximum Flow, (U.S. gpm)	Mechanical Efficiency, (%)	Maximum Particle Size
Plunger	3,500 - 4,000	1,000	85 - 90	0.094 in.
Piston	2,500 - 3,000	3,000	85 - 90	0.094 in.
Centrifugal	600 - 700	50,000	40 - 75	6.0 in.

Source: Aude (1976)

In the previous equations, subscript 1 designates the values of the flow parameters which exist at the original pump rotational speed, RPM_1. Subscript 2 signifies the modified values of pump capacity, head, and horsepower that result from operating the pump at some new rotational speed, RPM_2. These operational characteristics of a centrifugal pump can be quite advantageous when the driver is a variable speed unit. With such equipment, the flow rate of a system can be maintained that provides adequate slurry velocity if the system head curve varies. A centrifugal pump always operates at the intersection of its head-capacity or performance curve with the system head curve. The system head curve represents the head or pressure which is necessary to cause the flow through a system of piping, valves, etc. at various flow rates. A typical system head curve consists of three components:

(1) Static head, defined as the head which is necessary to overcome elevation differences.

(2) Pressure head, representing the desired discharge pressure or head required at the outlet of the system.

(3) All losses; i.e., friction, entrance, and exit losses which result from the presence of piping, valves, and fittings and are a function of the flow rate.

To illustrate, reference is made to Figure 5-2 where the normal operation is shown as point 1 for the system as it was designed. If, however, the system head curve is modified by wear or the addition of extra pipe, then the modified system head curve would force the unit running at RPM_1 to now operate at point 2. If this corresponding capacity, Q_2, is too low, then deposition of solids could occur as the velocity of flow is decreased by the reduction in capacity. By using a variable speed drive, it is possible to increase the centrifugal pump rotation and raise the head-capacity curve to some new level as shown by the RPM_2 curve. In this situation, the pump would now operate at point 3 and the flow velocity would be increased to an acceptable value.

For the in-mine applications of a coal slurry haulage system, centrifugal pumps are better suited to satisfy the equipment size limitations, which are imposed by the mining constraints, than the positive displacement pumps, which are, as a rule, substantially larger pieces of equipment for an equivalent volumetric capacity. That, coupled with their capability of handling slurries composed of large particle sizes, and the lower installed capital cost which they possess as compared to positive displacement pumps, make centrifugal pumps the natural choice for an underground mine coal slurry haulage system.

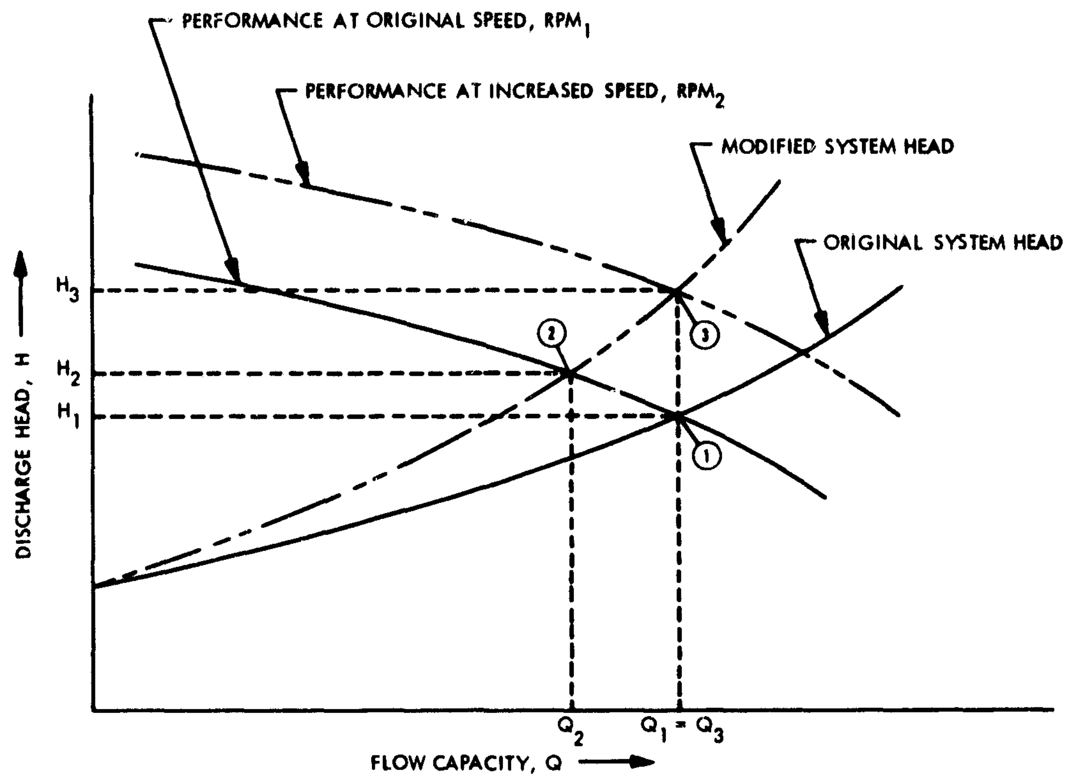

Figure 5-2. Variable Speed Centrifugal Pump Performance

The suspension property variables which have been found to have the greatest effect upon the hydraulic performance of centrifugal slurry pumps are particle size, specific gravity, and concentration of the solids con tained within the slurry mixture. The presence of solids in a slurry suspension unfavorably affects both head or discharge pressure developed and power consumed by the pump. These adverse effects can primarily be attributed to:

(1) The presence of slip between the carrier liquid and the solid particles being conveyed which occurs, particularly during acceleration and deceleration of the slurry, as a result of the pump operation.

(2) The elevated viscosity of the slurry mixture which has a tendency to lower the head developed by the pump and reduce its hydraulic efficiency.

Abrasion which is experienced in solids-handling pump applications can be categorized by three different types:

(1) Gouging - coarse particles impinge with sufficient force to cause high impact stresses.

(2) Grinding - particles are crushed between two moving surfaces.

(3) Erosion - free-moving particles impinge on the wearing surface.

Experience has shown that for pumps handling slurry mixtures consisting of abrasive solids, the pump velocity should be kept as low as possible in order to minimize the resultant wear. It has been found that the pump wear is approximately proportional to the cube of the velocity (wear α velocity^3). Therefore, since the pump velocity is directly related to the pump developed pressure, it follows that pumps utilized for high head applications will generally wear more rapidly than similar units operating at a lower head service. Also, the wear is usually found to be inversely proportional to the hardness (BHN) of the wetted pump components (wear $\alpha \frac{1}{BHN}$) and wear normally varies directly with particle concentration (wear α C_v).

In centrifugal pumps, unless the solid particles are quite fine, closed impellers are considered to be preferable to open construction impellers. Open impellers are those without a suction side shroud, whereas, closed impellers possess shrouds on both sides of the impeller vanes. Normally, closed impellers are chosen because more uniform wear of the impeller flow passages results, closed impeller geometry provides a structurally stronger design, and closed impellers are less sensitive to increased clearance between the impeller and the casing wall of the pump.

Natural and synthetic rubber is sometimes used to protect the pump's internal surfaces against wear if the solid particles of the slurry are small and relatively round. However, rubber liners or coatings are unsuitable if the solid particles are sharp and hard since they have a tendency to cut or tear the rubber. Rubber is generally not appropriate for applications with pumps having discharge heads in excess of 150 ft, peripheral impeller tip speeds in excess of approximately 5000 ft/min, or if the solid particles in the slurry are greater than 1/4 in. in diameter. Consequently, rubber lined slurry pumps are not considered a viable choice for a coal slurry mine haulage system due to these limiations. For this type of service, pumps constructed of wear resistant materials such as Ni-hard are found to be compatible with the application and useful for extending the life of the pumping equipment.

The previously discussed, conventional approach for satisfying the pumping requirements of an in-mine coal slurry haulage system consists of (1) mixing the coal and water in an open tank at the coal seam face, (2) pumping the mixture to the mine shaft, and (3) subsequently lifting it by pumping to the mine entrance. However, there are also several other concepts, some of which will be mentioned, for both forming the slurry and providing the pipeline pressure necessary to transport the mixture.

One of the most difficult tasks in the design of a face haulage subsystem due to the limitations resulting from the low headroom is the introduction of the coal into the pipeline. A concept currently under development is commonly referred to as a high pressure injection system. Malsbury et al. (1979) note that in such a design, a combination crusher-injector would be used to inject the coarse coal particles into a high pressure stream of water. A diagram illustrating this approach is shown in Figure 5-3. Two significant advantages to such a concept are:

(1) It could be feasible to utilize the substantial gravitational forces generated on the return water line of a closed system to provide the high pressure source for the jet injector.

(2) If pumps are used to generate the pressure for the jet injector, then the wear on the pumps would be substantially less than the conventional system since they would be handling only water and not a slurry mixture.

Such a design scheme provides an attractive alternative for the horizontal transporation of the coal from the seam face but would probably not be capable of generating sufficient head for vertical lifts to the mine entrance at the surface.

One approach for vertically hoisting the coal slurry from a deep mine is the lockhopper or pipe feeder system (Wasp et al., 1977: Hartman and Reed, 1973). This method would also separate the pumps that generate the necesssary high pressure from the abrasive slurry. An illustration of a lockhopper system is shown in Figure 5-4. Such a scheme would consist of two or more pressure vessels that are alternately charged with the slurry mixture and, in turn, injected with a high pressure flow of water. It is possible to utilize a lockhopper system to satisfy the slurry hoisting requirements from an underground mine where the pump is located on the surface and the pressure vessels and valves are in the mine. Once again, this arrangement would also gain the advantage of gravity to assist in generating the necessary pressure. The constraint upon the allowable particle size of coal which the pump could pass without plugging and the equipment size limitations imposed upon an in-mine pump application would not be a factor for consideration in such a system and it is possible that a positive displacement pump may be a more advantageous choice. Hartman and Reed (1973) reported that the lockhopper

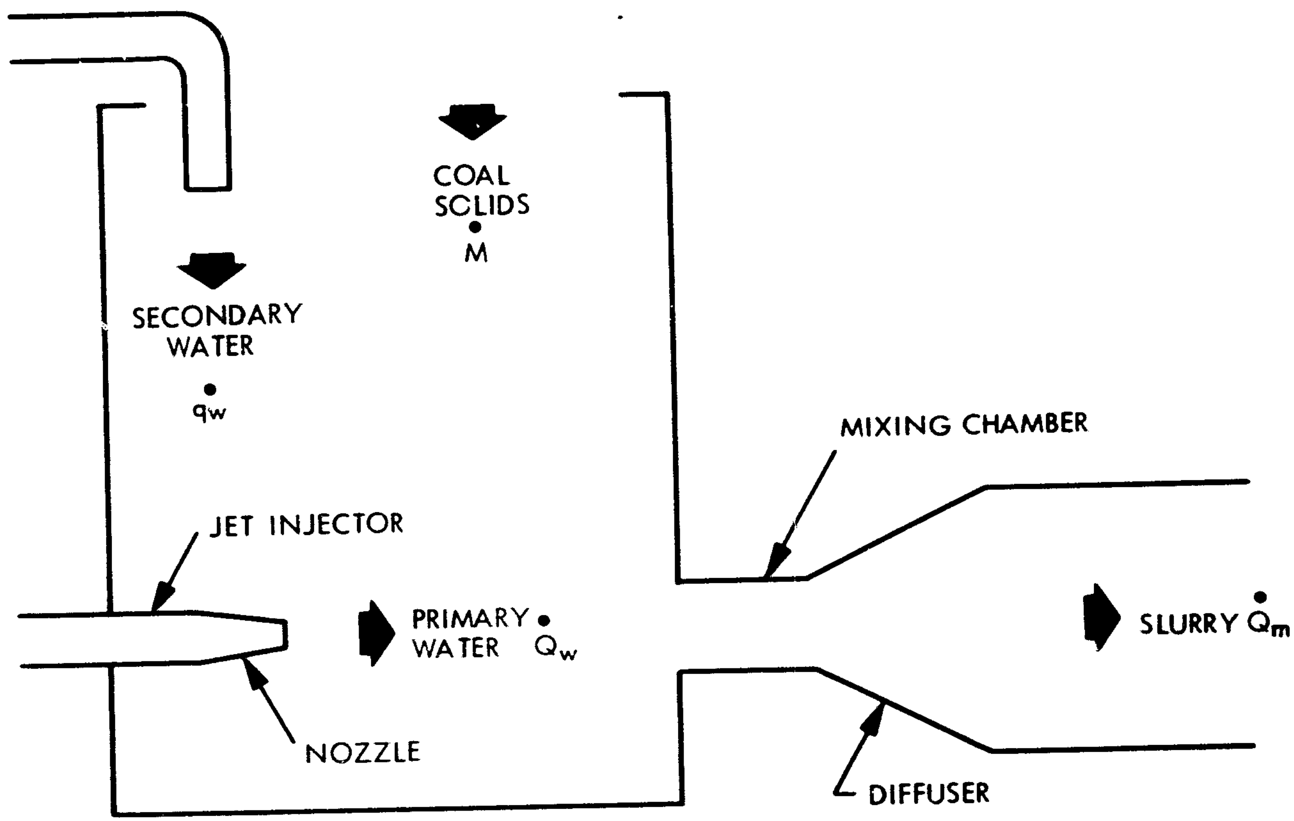

Figure 5-3. High Pressure Injection Scheme

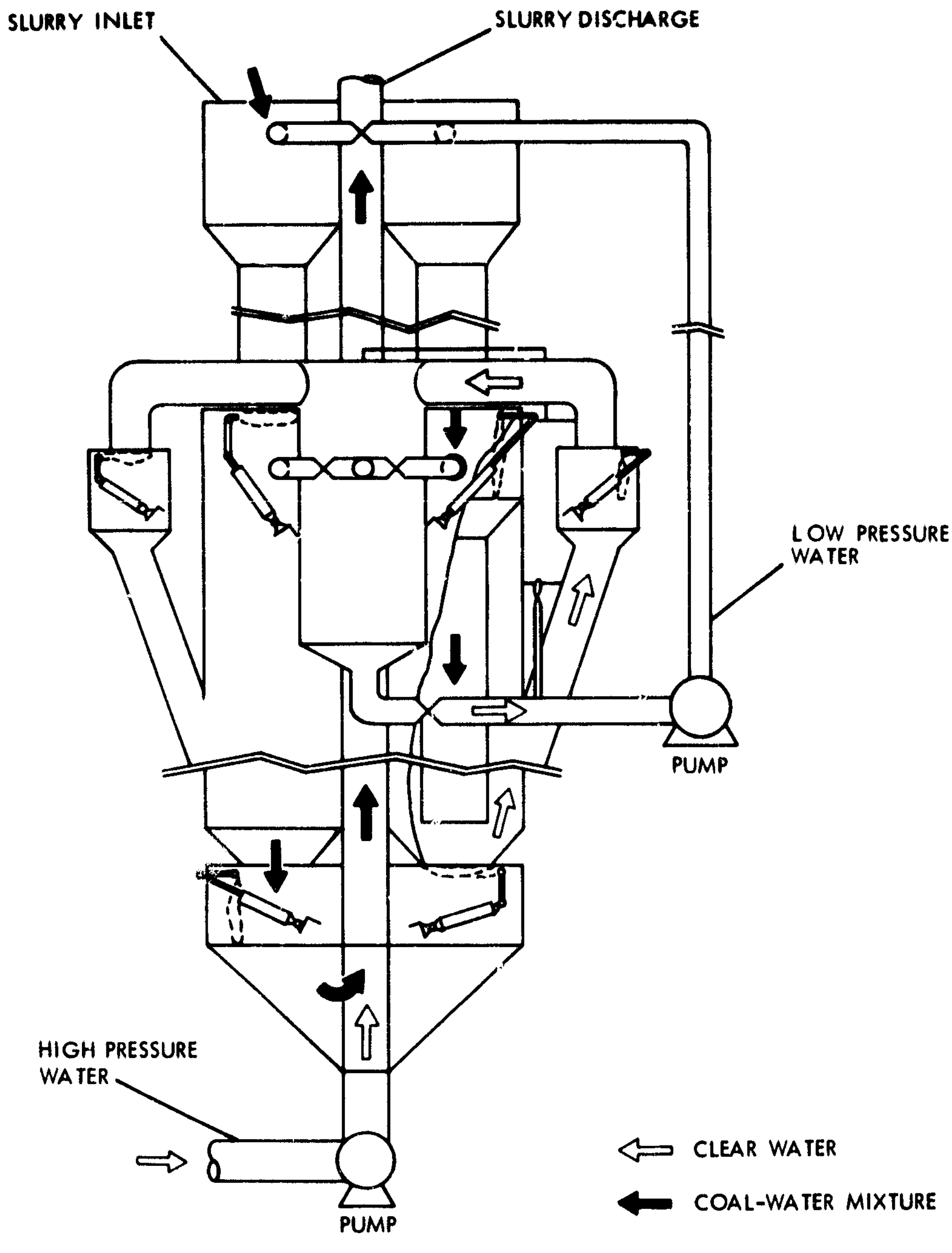

Figure 5-4. Lockhopper System Diagram

system can hydraulically lift coal with a particle size as great as 75% of the inside pipe diameter, but it is recommended that the maximum particle size should not exceed 33% of the pipe diameter.

The costs for utilizing centrifugal pumps for vertical hoisting are substantially less than the lockhopper costs. However, when the vertical distance which the coal slurry must be lifted exceeds about 500 ft, consideration should be given to lockhopper and other alternative hoisting subsystems. This is because the large number of centrifugal pumps that would be required to operate in series would probably adversely affect the larger coal particles causing size degradation and producing excessive fines.

5. Pipe and Valves

The problem of wear which was discussed in the preceeding section on pumps handling slurry mixtures is also experienced in the piping system, but to a lesser degree than for pumps. For a given quantity of slurry throughput, the selection of the pipe diameter determines the transport velocity. Consequently, as the flow velocity of the mixture increases so does the erosion of the components and the resultant friction losses.

The abrasion which is experienced in the pipe can be categorized as either deformation wear caused by impact of the solid particles, or cutting wear resulting from the sliding action of the solids. Abrasive wear is primarily governed by the following factors:

(1) Characteristics of solids - size and size distribution, hardness, density, shape, and composition.

(2) Characteristics of carrier liquid - corrosiveness, density, and viscosity.

(3) Condition of flow regime - laminar or turbulent, heterogeneous or homogeneous, and velocity.

(4) Pipe material - strength and ductility.

Homogeneous slurries normally can operate at velocities up to approximately 10 ft/s before pipe wear becomes an item of concern. However, for heterogeneous slurries, such as would be experienced in a coal slurry mine haulage system, the abrasive wear on the piping system can be severe at slurry flow velocities greater than 5 ft/s, and the wear increases exponentially as approximately the cube of velocity.

One distinct advantage held by a mine slurry haulage system is that the accessibility and length of the pipe subjected to extreme wear conditions lends itself to periodic rotation of the pipe in order to extend its service life. This is a useful attribute of the system since the majority of the abrasive wear will be experienced on the bottom of the pipe as a result of the pronounced solids concentration gradient in the horizontal pipe runs. The pipe which is oriented in a vertical position to achieve the lifting of the slurry mixture can have a lesser wall thickness for wear allowance than the horizontal pipe due to the formation of the previously discussed water annulus which will lead to a negligible abrasion loss.

Commercially available pipe of standard wall thicknesses is capable of containing the anticipated slurry pipeline operating pressures. However, the determination of the particular pipe grade, necessary wall thickness, and desired corrosion/erosion allowance must be evaluated for each specific application. Good design practice suggests that in order to avoid operational problems in conventional systems caused by pipe blockage, the minimum inside diameter of the pipe used for slurry transport should be at least three times as great as the diameter of the largest particle size being conveyed. Additionally, in order to minimize abrasive wear, it is recommended that the minimum radius used for fabricated pipe bends should be at least five times the pipe diameter and three times the pipe diameter for cast bends. While it is not always possible to adhere to these criteria, particularly at the seam face of the mine where rubber hose may be used to obtain the system flexibility necessary to follow the mining machine, it is always advisable to minimize abrupt changes in the direction of flow.

The abrasive service under which the valves in the system must operate should also be given consideration. It is preferable that the valves possess a full line-size opening at the non-throttled position since restrictions in the valve will cause abrasion downstream. If possible, the valves should not contain recesses or voids that could collect solids and impair operation, nor should they be dependent upon finish machined-metal surfaces for sealing since these can rapidly deteriorate.

6. International Experience

There has been a significant amount of activity by numerous countries in the area of hydraulic transport of solids in a mining environment. Canada, China, Czechoslovakia, France, West Germany, Japan, New Zealand, Poland, United Kingdom, the United States, and Russia have all had operating experience with hydraulic coal transport of either an experimental or production nature. All of these aforementioned instances relate directly to mine installations and not simply laboratory test facilities. There are over thirty applications of the use of hydraulic transport to satisfy one or more aspects of the coal haulage requirements associated with mining operations around the world. For instance, the Baydayevskaya-Severnaya-1 mine in the U.S.S.R. utilizes a slurry haulage system on the surface to convey the mined coal 32,800 ft. The Hansa mine in West Germany has two underground pipelines of 6,900 ft and 10,500 ft in length and a pipe feeder or lockhopper system to hoist the coal slurry 2,800 ft vertically. Both of these applications are for production coal mine installations.

Many of the in-mine hydrotransport endeavors to date have been associated with hydraulic mining activities where a high pressure jet of water is used for the coal cutting function. This is commonly referred to as hydromining. Those countries involved in hydromining activities have also investigated the possibility of hydraulic transport of coal via gravity operated flumes where the water used for the solids extractions can also be used for the loading and transport of the coal (Gregory, 1977). However the majority of the international work in hydrotransport appears to have been involved with the development of hydraulic hoisting systems for the vertical conveyance of the mined coal (Miscoe, 1977). While most of these efforts are concerned with the use of lockhopper or pipe feeder mechanisms, there have been significant achievements by China, West Germany, the United States and Russia in the use of pumps to directly satisfy the vertical lift requirements of conveying the coal/water slurry from the mine to the surface.

7. Summary

For the in-mine applications of a coal slurry haulage system, centrifugal pumps are better suited to satisfy the equipment size limitations, which are imposed by the mining constraints, than the positive displacement pumps which are, as a rule, substantially larger pieces of equipment for an equivalent volumetric capacity. That, coupled with their capability of handling slurries composed of large particle sizes and the lower installed capital cost which they possess as compared to positive displacement pumps, makes centrifugal pumps the natural choice for an underground mine coal slurry haulage system. The most advantageous use of positive displacement pumps in a slurry haulage system may be for accomplishing the slurry hoisting requirements by supplying the high pressure fluid to a lockhopper or pipe feeder mechanism where the slurry is segregated from the pump and the positive displacement pump(s) may be located on the surface.

The abrasive wear experienced in slurry pumps, pipe, and valves is proportional to the cube of the flow velocity, inversely proportional to the material hardness of the components, and directly proportional to the volumetric concentration of solid particles conveyed. The strong influence that the slurry flow velocity has on the rate of wear tends to set the upper limit on the acceptable flow velocity relatively close to the mininum operating velocity. The minimum operating velocity was previously defined as at least 30% greater than the deposition velocity.

There continues to exist a definite need both to accumulate engineering design and operating data and to develop and test equipment. Gregory (1977) notes that of the countries which are members of the International Energy Agency, there are currently extensive research and development activities being pursued in Canada, West Germany, United Kingdom, and the United States that are relevant to the underground hydraulic transport of coal.

8. Overall System Performance

The analysis of slurry rheology (as discussed by Maynard, 1981) together with the above assessment of equipment performance indicates that the parameters which have the greatest effect upon limiting the performnce of the slurry haulage system are:

(1) Flow velocity.
(2) Pressure.
(3) Coal particle size.
(4) Solids concentration.

The velocity of the slurry must be high enough to prevent excessive settling of solids and the subsequent plugging of the pipeline but, at the same time, be no greater than necessary in order to prevent excessive friction head losses and pipeline wear. For such a system as considered in this report, the optimum velocity of the slurry is expected to be from 10 to 15 ft/sec. Figures 5-5, 5-6, and 5-7 illustrate the performance envelopes of hydraulic transport systems operating within this velocity range utilizing nominal pipe sizes from 4 to 20 in. in diameter and conveying 30, 40, and 50% volumetric concentrations of coal respectively and 0% rock. Pipe sizes less than 4 in. in diameter would not be capable of transporting a great enough quantity of slurry to be practical, and pipe sizes much larger than 20 in. in diameter are difficult to manage in an underground mine environment. The curve representing a slurry containing a 30% volumetric concentration of coal, Figure 5-5, is well within the boundaries of existing design and operating capabilities. Figure 5-6, where C_V = 40%, illustrates the system performance that approximates the limits of current technology. Figure 5-7 depicts the anticipated output of a slurry haulage system if, as a result of technological innovations, the volumetric concentration of coal could be increased to 50%.

The operating pressure of the slurry system must be sufficient to overcome frictional losses and static lift requirements. The conveyance of coal from the seam face can normally be accomplished by 100 psi. of pressure, which is well within the capabilities of existing centrifugal pumps that also satisfy the capacity or flow rate requirements and size constraints imposed by an underground mining application. Centrifugal pumping equipment currently exists that can produce a discharge pressure of approximately 600 psi (1386 ft of water) to accommodate the vertical lift requirements for bringing the slurry to the surface. However, vertical hoists of greater than about 1000 ft are usually not considered when using centrifugal pumps. This is due to the necessity of staging or operating a number of pumps in series which will result in excessive particle size degradation, the reduction of generated pump discharge pressure that occurs when pumping a slurry, and the presence of pipe friction losses that must be overcome.

It is most desirable to operate the haulage system with run-of-the-mine coal as it is produced by the continuous mining machine. However, it is necessary to process the coal through a breaker or crusher before introducing it into the slurry since the maximum allowable particle size currently appears to be approximately 4 in. in diameter. This limitation is primarily a function of the size of solids particles that can physically be passed through the pump, but a general guideline is that the maximum particle size should be no larger than one-third of the interior pipe diameter.

From practical considerations, it is obviously preferable to transport as great a concentration of solids in the slurry as possible. Previous experimental work by others has shown this limit to be approximately 40% coal by volume or about 48% coal by weight in a slurry with water.

While investigation has been performed on the subject of coarse particle slurry transport (Poundstone et al., 1977; Dahl and Petry, 1977), unfortunately the majority of the data is held as proprietary information by the private companies performing this research. The areas exhibiting the greatest need for further study are large particle slurry flow characterization and pipeline plugging phenomena. The work currently being performed under the U.S. Department of Energy sponsorship in their hydrotransport continuous face haulage subprogram (Miscoe, 1979) is intended to satisfy the equipment development need for an improved method of solids injection so that pump wear is minimized.

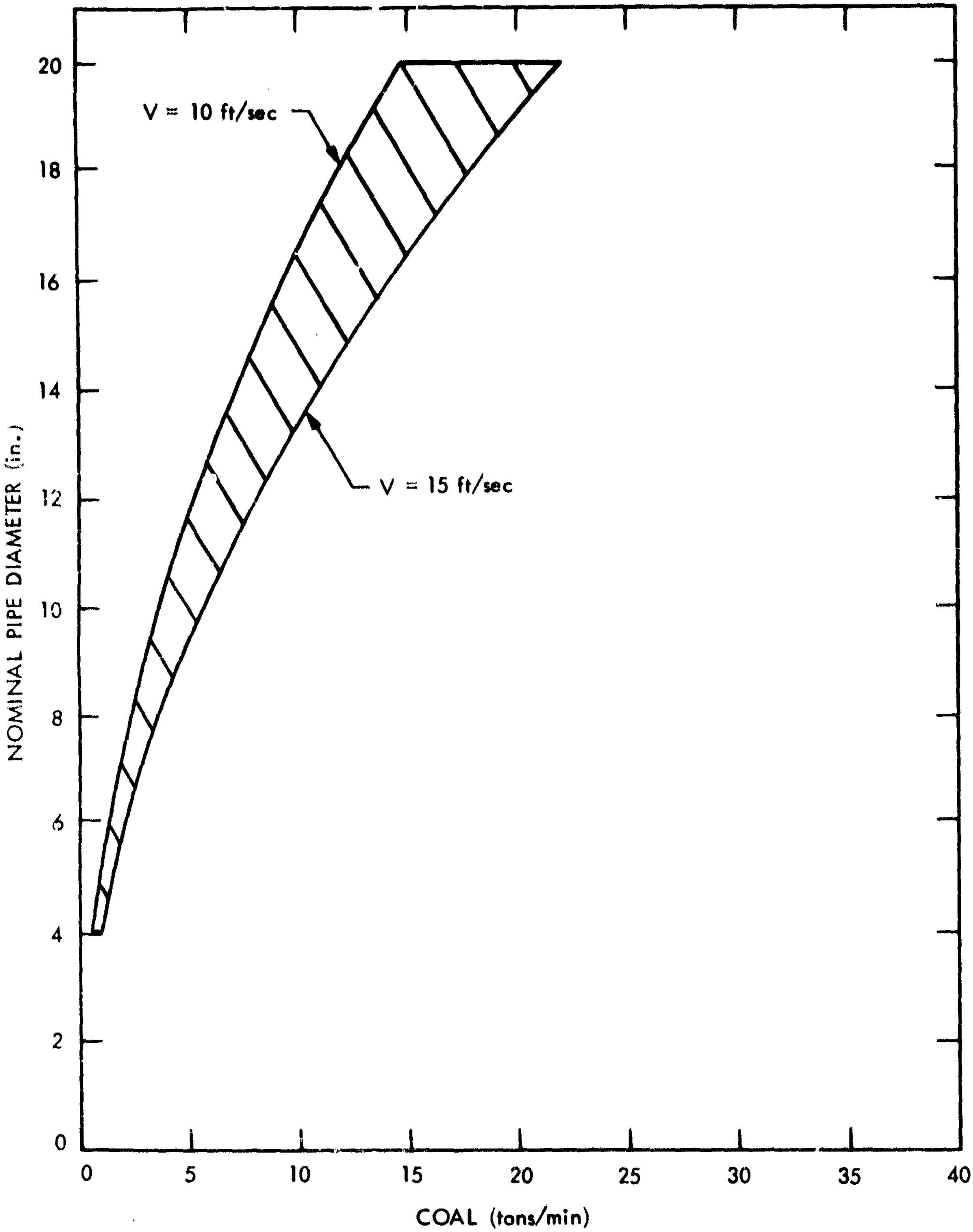

Figure 5-5. System Performance Envelope, C_V = 30%

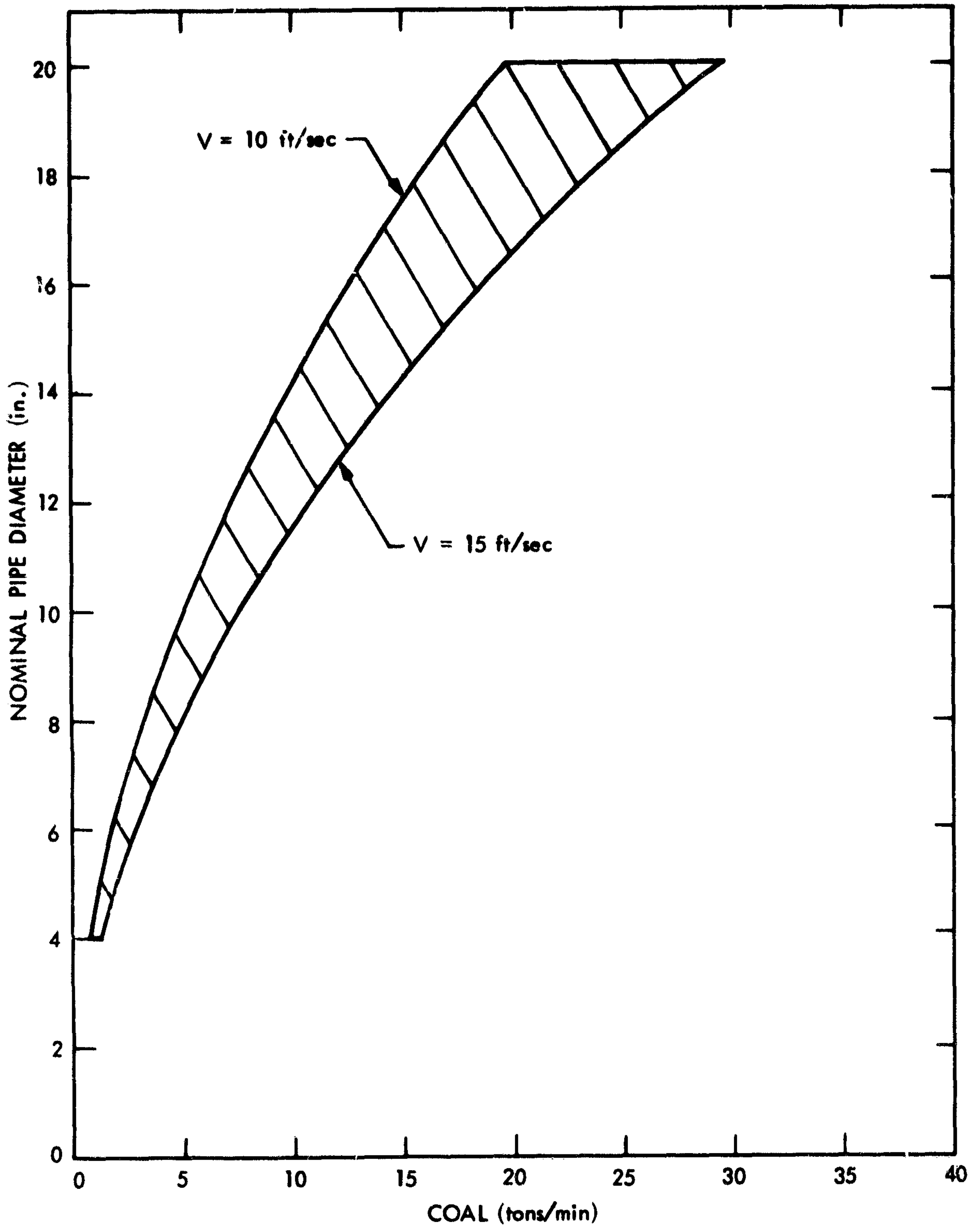

Figure 5-6. System Performance Envelope, C_V = 40%

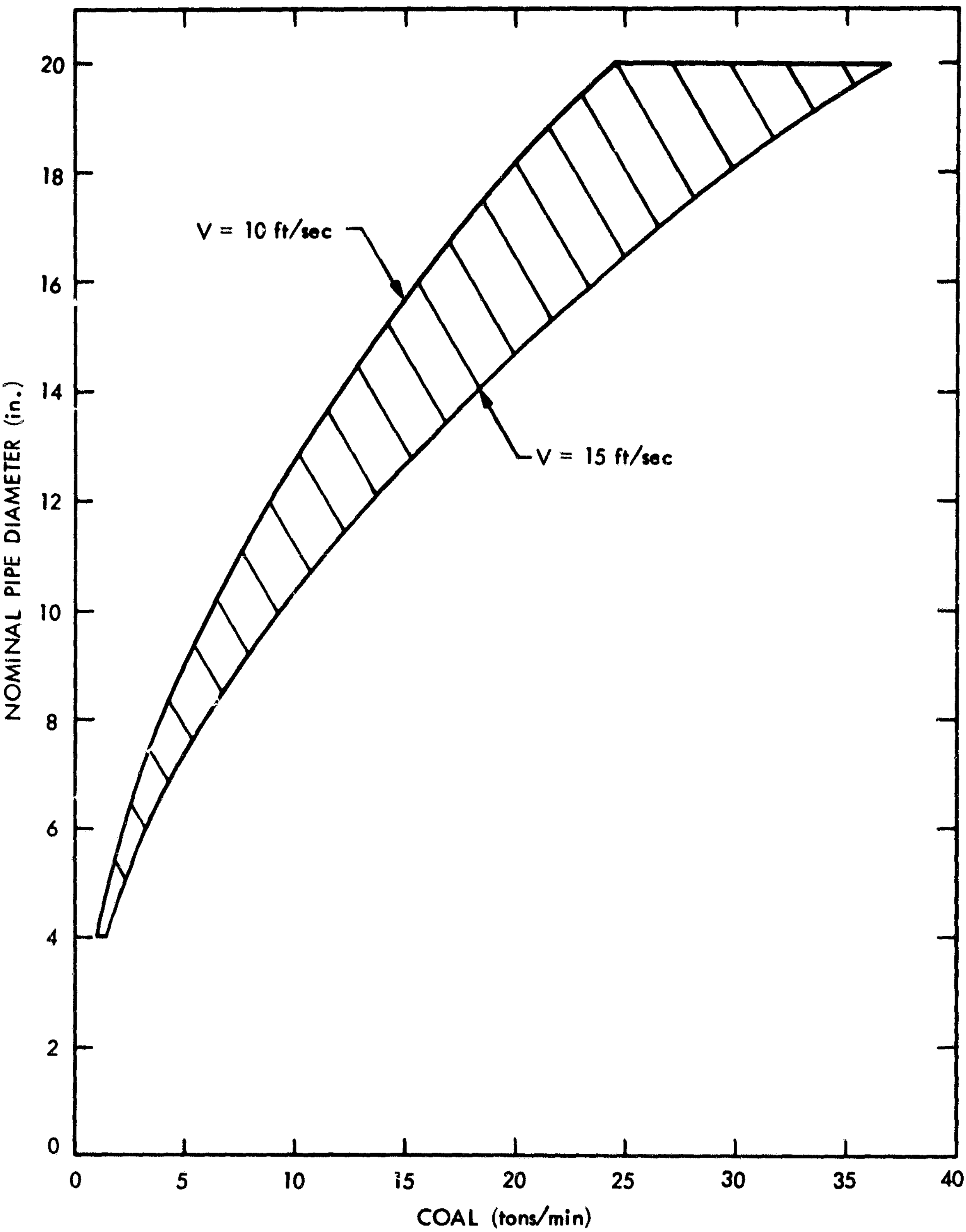

Figure 5-7. System Performance Envelope, C_V = 50%

A study by Link et al. (1975) concluded that, while the necessary capital investment for an in-mine hydraulic transport system is greater than for the conventional haulage system using shuttle cars and conveyor belts, the potential increase in coal production should result in a mine mouth cost of coal which is less. This report concluded that a mine utilizing a coal slurry haulage system, instead of the ordinary means of conveyance, will experience an approximately 40% increase in productivity.

Additional detail about the slurry haulage assessment may be found in Maynard (1981).

C. EARLY DESIGN ACTIVITY

In early 1979 it was decided to launch a brief conceptual design exercise in order to (1) gain a deeper appreciation for the most attractive areas for improving several systems performance, and (2) explore the feasibility of actually achieving this level of performance. Although system requirements and conceptual design requirements had not been formalized at that time, work by Duda (1978), J. J. Davis (1977), Marrus, et al. (1976), Douglas and Herhal (1976), Mabe (1979) and others pointed clearly to the following considerations as a likely focus of design effort:

(1) Operations during the production era of mining; operations associated with mine opening, initial development, and mine closing have a much lower relative impact on the fully amortized cost of coal.

(2) Within the production era, attention is drawn to the face operations, simply because the underground production support activities are so diverse as to defy the discipline requisite for greatly increased efficiency.

(3) In face operations, the major variables are uptime, uprate, and of course, the direct costs of equipping and operating a section.

Because cost is not a prime consideration in the early stages of conceptual design (so long as the likely cost appears reasonable), it was natural that this activity focus on:

(1) Increasing the uptime, by reducing the built-in dead time caused by equipment moves and set-ups.

(2) Increasing the block of coal removed by a major equipment set-up, thereby amortizing this non-productive time over a greater tonnage of coal.

(3) Increasing the uprate while the equipment is producing coal; the uprate may, in turn, be improved by boosting the flux of coal per unit area of cut face and/or expanding the instantaneous ones being cut.

Accordingly, the 1979 design effort examined several innovative ways of accessing panels of coal, and subsequently extracting the panels accessed by the development effort.

Because Central Appalachia had been designated as the initial target resource, this design activity was primarily concerned with the flat lying coals accessible from the outcrop. The concepts examined fall into three broad categories:

(1) Systems which could extract coal remotely from a bench (much like current day augers) while attaining a high recovery.

(2) A system for rapid in-mine development, which resembles but extends the current mine-boiler concept.

(3) Advanced caving systems for panel extraction which exhibited (1) little or no dead time between panel moves (no set-up at the end of a cut), (2) elevated uprates, and (3) in some cases, a longer increase in the ones in contact with the mining machine.

Altogether, about a dozen different concepts were explored to varying depths of analysis. Written descriptions of each design, together with drawings and preliminary performance calculations are maintained in the project archives.

SECTION VI

SUMMARY AND CONCLUSIONS

As indicated by the detailed discussion in Chapters II through V, the primary focus of the Advanced Coal Extraction Systems Definition Project during 1979-1980 was formulation of system level performance goals and the translation of these goals into conceptual design requirements. When examined as an ensemble, the overall performance goals, although presented as specific to the Central Appalachian resource, are really quite general in all areas except mine size and regional geology. In particular, whatever the resource, it is reasonable to expect the priorities established by Goldsmith and Lavin (1980) to apply, and the same figures of merit to be used:

(1) Production cost: return on incremental investment at time of first use.

(2) Miner safety: reduction in deaths and permanently disabling injuries per million man-hours to a level experienced by comparable industries.

(3) Miner health: protection of long-term functional capability by adherence to well established standards for a healthful workplace, with special emphasis on threats to pulmonary health.

(4) Environmental impact: preservation of land use appropriate for the mine site prior to commencement of mining operations; protection of adjacent lands from environmental degradation, as required by law and regulation.

(5) Coal conservation: attainment of a coal recovery performance at least as good as the best of contemporary technology operating in comparable conditions.

During the latter portion of 1980, project attention turned to transformation into conceptual design requirements the previously identified opportunities to meet the systems requirements. The analysis of safety hazards presented in Appendix H is a good example of the thrust of this effort. As in the case of the system level requirements, it is expected that many of the conceptual design requirements will be applicable to a broad spectrum of resources. Of course, the geological component of design requirements will be specific to the initial target resurce--the flat lying deep coals of Central Appalachia. Completion of these design requirements is targeted for mid-1981.

The Central Appalachian coals were chosen as the focus of the early system definition work on the basis of a brief analysis. Thus, an important portion of the system definition effort during 1980 was devoted to a comprehensive examination of all significant domestic coal resources with the objectives of assigning priorities to potential candidates for systems R&D efforts. This analysis began by constructing a classification scheme embracing the traditional determinants of mining conditions, and from obtaining a preliminary set of tonnage estimates for all fields containing at least one billion tons of coal. These preliminary estimates indicated some very substantial deposits of coal in the Gulf Coast and the Brooks Range region of Alaska. Estimates of the very modest tonnage present in fault and fold basins sufficed to eliminate these coals from further study.

At the close of 1980, this resource study was in the midst of an in-depth analysis of substantial coal deposits within the five major coal provinces--Appalachia, the Interior, The Rocky Mountains, the Gulf Coast, and Alaska. An interesting feature of this analysis is its attempt to use statistical sampling theory to put numerical bounds on the amount of resouce in the ground. The results of this study, complete with an analysis of the likely commercial importance of the various resources, will be available in late 1981.

Finally, in order to gain a better appreciation of the possibilities for improving system performance, the project launched a brief conceptual design activity in early 1979, and performed a broad survey of current R&D in underground mining technology. Subsequent work in the area of technology assessment focused on underground slurry transport, as a very promising means of reducing non-productive mining time. The thrust of this effort was a description of the engineering parameters which significantly impact the performance of a slurry system.

In sum, at the end of 1980, the project was well along the road to establishing design requirements for flat lying coals, and looking forward both to (1) adapting these results to other resources, and (2) taking the next step in system development by initiating an in-house conceptual design activity.

APPENDIX A

COAL GEOLOGY OF GENERAL RELEVANCE TO UNDERGROUND MINING

This appendix describes those aspects of coal geology of particular relevance to underground mining, whatever the resource might be. This description is unusual in that it attempts to relate observable mining conditions to features and phenomena of the depositional setting.

Most of the discussion below is adapted from Horne et al (1978), as elaborated in Camilli et al (1981).

A. DEPOSITIONAL ENVIRONMENT

Every aspect of mining is intimately related to the geologic environment associated with the coal body. Most mining systems have dimensional limitations either as maximum horizontal reaches or minimum operating heights. If a mining system is dependent on a particular coal geometry or a mining sequence for successful operation, the continuity and consistency of the coal seam and its attendant components (roof, floor) become critical. Engineering parameters such as material strength and deformability are also generally a direct result of the lithology and geologic history of a given coal bed. Though many of the factors appear to be random, the appropriate data plus an understanding of the formative processes, permit the geologist to identify a number of trends. Trend identification requires considerable detective work, which is often concerned with "academic" type evidence. Such esoteric details as whether the sandstone was graded or mottled, or whether the shale contained burrows or roots are of little use to the layman but are of vital importance in reconstructing the original depositional environment.

Coal is a product of thick vegetal accumulations most commonly associated with the peats found in swamp and marsh-like environments. Looking at today's wetland ecologies, it can be seen that very distinct classes of wetland environments exist. For example, the marsh of the Mississippi Delta differs widely in process, sediment and vegetal content, and physical distribution from the Florida Everglade swamps or the barrier island marshes of New Jersey. Yet each is capable of producing peat in sufficient quantity to form coal. Likewise, individual seams and their associated strata will reflect their original depositional setting. Intuitively, it can be seen that an Everglade type environment will form broad, evenly distributed coal beds as found in the Illinois coal basin. On the other hand, deltaic and near-short coal of Central Appalachia will be more fragmented in distribution and quantity, and have greater variability in terms of associated strata. The depositional settings can be classified into subenvironments, each with recognizable features. Horne et al (1978) has categorized each of the subenvironments common to Appalachia, with their distinguishing features.

Figure A-1 illustrates and Table A-1 summarizes the various depositional environments. Although the analysis requires a great deal of insight and base data to supplement the insight, a knowledge of the depositional environment can provide a number of clues to possible ground control problems for a given coal field. Local soil structure, topography and drainage dictate the growth densities and, therefore, the thickness of the peat. Relative rates of vegetal growth, erosion, and sedimentation effects the spatial distribution of partings, splits, benches, roof and floor types, etc. For example, in Eastern Kentucky, the southern half of the field experienced pronounced subsidence as evidenced by the thickening of the time-rock sequence. As a result, the peat forming environments could not develop and the coals south of the hinge line (Paint Creek-Irvine fault zone) are thinner and more limited in lateral extent than the coal seams to the north (e.g., seams corresponding to the transitional upper delta plain subenvironment are typically 10-15 miles in breadth in the south as compared to 60 mile widths in the north). As a counterweight to this dimensional disadvantage, coals forming in the rapidly subsiding areas generally contain significantly lower amounts of sulfur and trace elements.

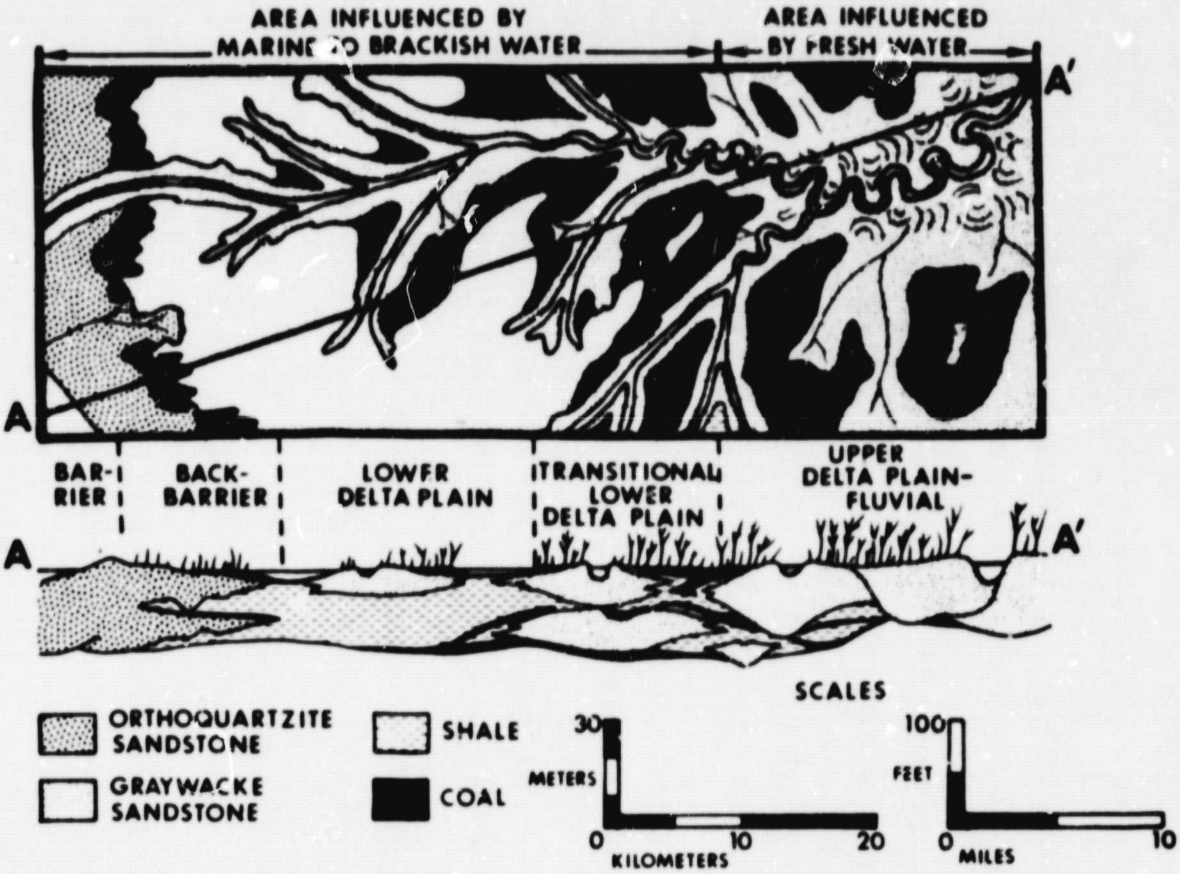

Figure A-1. Depositional Model for Peat-forming (Coal) Environments in Coastal Regions (Adapted from Horne and Ferm, 1978)

Table A-1. Depositional Environments

Environment	Coal Geometry
Fluvial and Upper Delta Plain	- Less continuous than lower delta plain - Pod shaped - Occurs in flood plains between wandering streams - Rapid variations in thickness - Elongate parallel to the depositional dip
Lower Delta Plain	- Continuous along the depositional dip direction - Discontinuous along the depositional strike - Splits common
Transitional-Lower Delta Plain	- Most important resource - Extensive in large interdistributary bays - Splits common near levees - Roughly elongate parallel to the depositional strike
Barrier	- Pod shaped - Elongate parallel to depositional strike - Channeling common

adapted from Camilli et al (1981)

B. MINING CONDITIONS OF UNIVERSAL RELEVANCE TO SYSTEM DESIGN

Once the stratigraphic and tectonic styple of a region has been characterized, the mining geologist can then translate these results into terms of engineering significance. Vital aspects such as partings, splits, sandstone channels, areas of good and bad roof, to name a few, can, to some degree, be predicted. Finally, the geotechnical engineer will transform the above information into design parameters. Whatever the extraction system, it must be able to cope with both the commonly encountered values, together with likely variations in the following:

(1) Roof strength.

(2) Floor strength.

(3) Partings.

(4) Sandstone channels.

(5) Sedimentary structures impacting coal and adjacent rock.

(6) Mechanical discontinuities.

Each of these topics is addressed briefly below.

1. Roof Structure and Strength

All of the strata immediately associated with coal in Central Appalachia are sedimentary in origin (not necessarily so in other coal basins). The roof and floor composition and distribution are, therefore, subject to the same depositional constraints as coal. The dominant roof types are shale, siltstone, sandstone, channel coal, limestone, and clay. All grades or combinations of these lithologies are possible and are likely to be encountered somewhere within a coal basin. Floor lithologies are most commonly fireclay and shale, though siltstones, carbonaecous shales and sandstones do occur. Table A-2 summarizes the roof lithology types, some of the inherent problems associated with each type, and their depositional environment (modified from Horne et al, 1978).

Inspecting Table A-2, it can be seen that the more competent strata are associated with the coarser grained lithologies. This in turn implies that deposition occurred within a relatively high energy environment such as the fluvial, onshore, or barrier island. Unfortunately, high energy environments imply complimentary erosional forces. In fact, returning to Table 2-3, it can be seen that the coals associated with high energy environments are spotty or relatively discontinuous.

The peat bogs were naturally within the active biosphere. Plant and animal alternations to the rock structure frequently affect structural behavior. For example, worm burrowings homogenize the rock mass destroying interlocking mineral fabric. Plant roots decay and fill with clays of little shear strength. Roof rock dissected by such dense root structures or dessication cracks (cube/rock) is very weak and often must be avoided. Petrified tree stumps (called kettles) also exhibit very low shear strengths along their sides as a result of differential compaction along the clay interface. Kettles frequently drop out of the roof without warning. Differential compaction of horizontally adjacent lithologies often create slickensides along the shear face. Sands compact 10-15% of their volume while shales compact 15-50% under a 2,000-4,000-foot overburden load. Slickensides have little or no bond strength and are also found along sandstone channels, clay veins, and concretions. Mining systems which hope to sustain unsupported openings in these kind of rocks have little chance of success, whereas greywacke, sandstone, and some shale roofs can remain stable over wide widths and long periods of time.

Aside from the previously mentioned kettles, concretions may abound in the roof. Concretions are semispherical nodules of claystone (shale), pyrite, siderite, or calcite that range in size from icroscopic to several feet in diameter. Concretions are the result of chemical precipitation which usually forms around some nucleus, commonly a fossil. Differential compaction around the nodules formed slick surfaces that have little adhesion to the surrounding rock, thereby allowing concretions to fall out of the roof. In some shale roof, concretion populations are quite dense and cause severe roof problems.

2. Floor Structure and Strength

The strata underlying a coal seam acts as the foundation material which supports all overlying strata, artificial supports, and the mining machinery. Although floor material of a particular coal seam within the area of a mine property is usually fairly consistent, the material properties may vary greatly over the entire region of a seam. The floor materials of Central Appalachia range from soft, plastic clays (also called underclays, fireclays, seatearth, etc.), shales and siltstones, to an occasional hard sandstone or shale. The majority of floors contain large percentages of clay minerals (kaolin and illite) which cause them to deteriorate when exposed to water.

The structure of the floor is essentially the mirror image of the roof. The floor materials and geometry are a function of the depositional environment prior to formation of the peat bogs. In Central Appalachia, this environment was essentially deltaic near-shore and barrier island. Floor lithologies can vary from several inches in thickness to tens of feet. The most troublesome underclays may form the immediate floor or they may be overlain by shales and siltstones.

The wide range of floor materials is reflected in their complicated behavioral response to in-mine stresses. The soft clays are ruled by soil mechanics principles, while the stiff shales and sandstones are described by the very different rock mechanics formulae. The presence of the large percentages of clays in siltstones and shales provides a grey zone of both soil and rock-like behavior.

Although the roof, floor, and coal generally control the mining system, many secondary geologic features can have a drastic effect on roof stability or cutting efficiency. Therefore, the occurrence and character of such features, partings, sandstone channels, clay veins, sedimentary structures, and mechanical discontinuities, will now be discussed.

Table A-2. Roof Lithologies

Roof Stability	Lithology	Conditions & Remarks	Depositional Environment
Excellent	greywacke	- lag deposits; shale, coal pebbles can weaken base of strata	- result of active, laterally migrating streams - Upper delta plain-fluvial - trans. delta plain
Good-excellent	orthoquartzitic	- often closely jointed-can cause severe roof falls	- back barrier - barrier
Poor-good	flat bedded sandstone interbedded sandstone	- stability depends on thickness - thicknesses less than 2' can exhibit separation along bedding planes - thick beds develop slickenside due to differential compaction (greater than 10') - 2'-10' thick beds best structurally	- found along flanks of distributary mouth bars - lower delta plain - trans. lower delta plain
Poor-good	coarsening upward shale, shale with sandstone, sandstone with shale sandstone	- separates at sandstone/shale bedding	- bayfill - lower delta
Very poor	seat earths, draw slates, (silty clays, roof penetrated)	- little strength, poor anchorage, must remove or leave coal - roots, slickenside - fine grained	- upper delta plain-fluvial - transitional lower delta plain

(Compiled from South Carolina Preprint
"Depositional Models in Coal Exploration and Mine Planning,"
J.C. Horne et al. 1978)

Table A-2. Roof Lithologies (Continuation 1)

Roof Stability	Lithology	Conditions & Remarks	Depositional Environment
Poor-very poor	cube rock (carbonaceous block shale, highly jointed)	- always requires bolting - brittle, nonbedded, highly jointed	- occurs during drowning in low energy-reworking environment, common to all settings - most frequently in: lower trans., lower delta lower delta plain
Poor-very poor	Burrowed siltstone, shale, shale with sandstone	- must bolt - if burrowing extensive, may require removing or leaving coal	- areas of slow sedimentation - marine, brackish, back barrier lower delta plain, trans. lower delta plain
Very poor	kettles (petrified tree)	- requires removal or bolting - tree core filled with sediment, outer skin scalifiers- poor strength, geometrically unstable - several hundred pounds	- common to all areas - frequent in upper delta plain-fluvia trans. lower delta plain
Poor	slickensides along sandstone-shale interfaces	- must bolt - slickenside developed due to differential compaction	- lower delta plain - back barrier
Very poor	Channel Bank Slump Blocks	- should be avoided, bolting, bracing ineffective - ancient landslide (slump) surfaces	- along cutbank side, laterally migrating, meandering stream channels - upper delta-fluvial - trans. lower delta plain - back barrier

Table A-2. Roof Lithologies (Continuation 2)

Roof Stability	Lithology	Conditions & Remarks	Depositional Environment
Poor-very poor	Rider coals	- if rider seam is within 20' of main seam - will fail up to upper seam - avoid or mine on retreat - roof usually heavily rooted - frequent partings	- crevassed levees, splay deposits - delta plain - all

3. Partings

Partings are layers of clay, shale, siltstone, or sandstone which horizontally intersect a coal seam. The relative position may vary from the roof to the floor, and may be paper thin to several feet in thickness. Partings were formed by floods or high tides that carried heavy sediment loads over the peat bogs, by changes in the overall depositional setting, or by flocculation of clays as caused by a chemical change in the waters. Flood-originated partings are thickest near the river banks, berms, or levees, and thin outward. The coarsest grained sediments also lie near the sediment source. These flood (crevasse splay) deposits can range from 5 to 500 m wide and 200 to 1000 m long. Parting lithology reflects the energy balance from the turbulent stream or tidal channel sandstones to lower energy sandy siltstones to siltstones to shales.

4. Sandstone Channels

The intersection of sandstone channels on a mine property can have severe consequences on the dollar value of the property and the effectiveness of a mining system. Sandstone channels are the infilled remnants of old stream or river beds which dissected the peat zones contemporaneously or eroded them after deposition. Sandstone channels (commonly filled with greywackes, siltstones, and shales) may range from several miles in width and several hundred miles in length (Walsville Channel in the Illinois basin, on the Herrin #6 seam) to a size equivalent of small stream beds. The channel may completely obliterate the coal seam or may only partially intersect it. Ancient river beds had the same random appearance as some of today's equivalents, containing meanders, tributaries, islands, oxbows, etc. In such cases, exploration is difficult and is further confused by coal pockets bracketed by sandstone channels. Aside from the obvious loss of coal, sandstone channels are associated with numerous roof problems. The Mine Safety and Health Administration (MSHA) has compiled a list of features affiliated with channels, some of which may act as omens of a possible nearby encounter with a sandstone channel (McCabe, 1978).

Figure A-2 illustrates the general shape of a typical sandstone channel. Channel fill is most commonly greywacke, although siltstones, shales or sandstones may also provide fill. The lag deposits or coarse grained material at the trough (apex) varies widely in strength but is generally difficult to cut. As the channel is approached, partings either appear or thicken. Coal may actually split into benches. Coal elevations may change as the peat climbs the river levees. Bed thickness may increase (contemporaneous) or thin (post-contemporaneous). The coal chemistry also tends to exhibit some variation. Secondary pigmentation from the more porous sandstone increases the sulfur content. Ash, phosphorous, trace elements, and moisture contents can also change due to the changed micro-ecological environment on a river bank (holds for contemporaneous channels only). Water seeps through roof joints and boreholes may occur several hundred feet out by actual intersection. Clay swell in roof bolt holes could weaken anchor points. Bedding planes in claystones and shales could disapear as a result of flow associated with differential compaction. Slickensides, goatbeards, and joints also increase as the channel is approached, again due to differential compaction. Technically, the change in thickness and elevation of the coal bed immediately adjacent to the eroded or nondepositional "want" area is termed a roll. This is not simililar to tectonic folds, pinches, or areas where the predepositional topography was rolling.

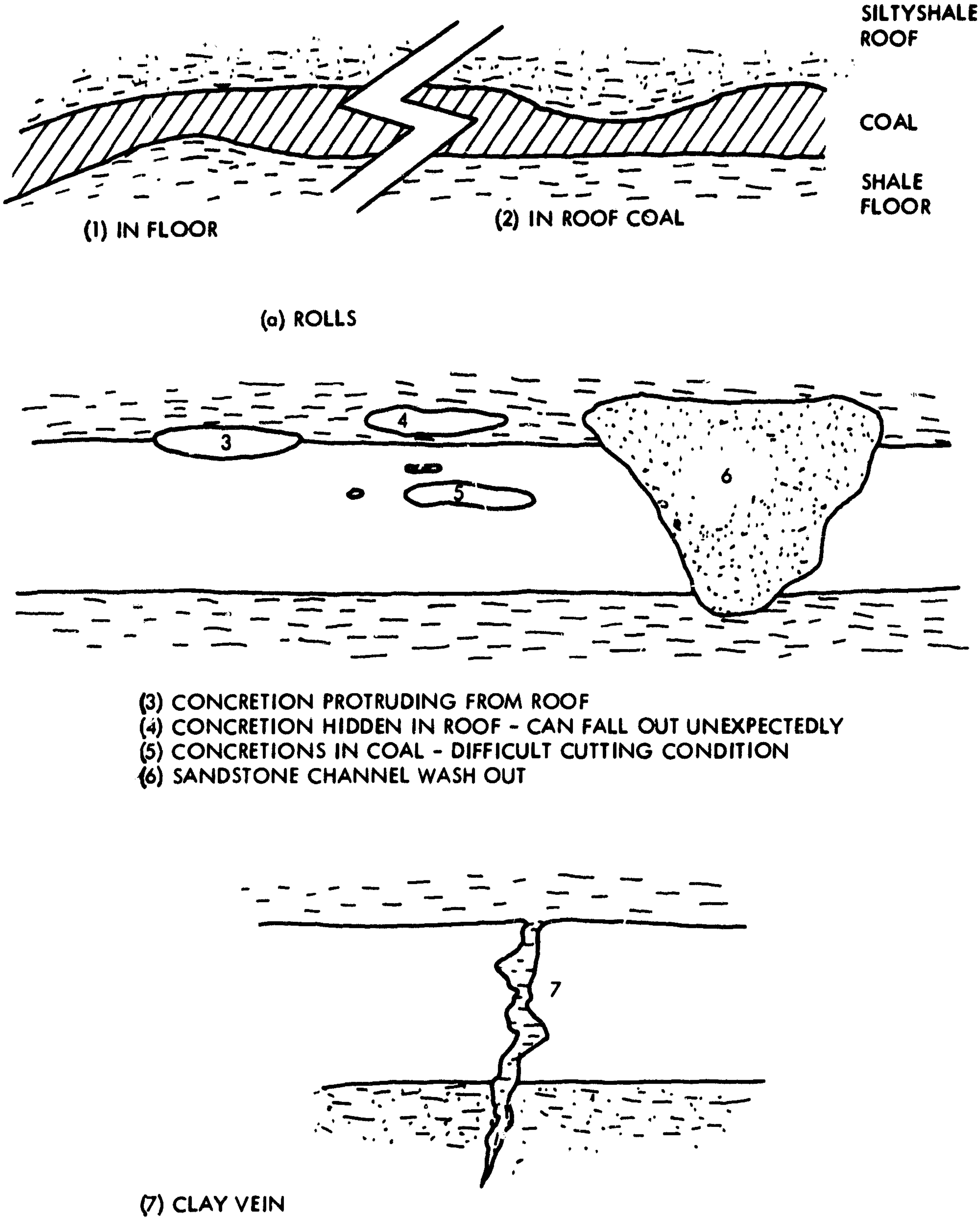

Figure A-2. Depositional Features Associated with Coal Deposits (Adopted from Camilli et al, 1981)

5. Clay Veins

Clay veins or dikes are masses of clay or shale which have been injected from the roof (occasionally the floor) into the coal bed. Clay veins may be inches to several feet in thickness, and may extend laterally several hundred feet. Clay veins are usually associated with fold axis' and tend to exhibit a parallelism with the predominant cleat directions. The invested, conical structure (also called a horseback) exhibits little cohesion and easily falls out, creating a domal cavity. The impermeable nature of clay veins can cause water and methane buildups.

6. Sedimentary Structures

Primary sedimentary structures are features that are contemporaneous with initial depositional, erosional, and diagenetic phases. They are primarily a result of parametric variations in current velocities, sedimentation rates and composition. Differences in cohesion due to fabric contrasts or grain type and size variations may create mechanically weak zones along the interface of the sedimentary structure. Examples are bedding planes, dessication cracks, ripple marks, crossbedding, lithologic contacts, and scour and fill interfaces. The tendency of sedimentary rocks to exhibit a pronounced fabric introduces a problem of anisotropy to the material behavior characterization. On the scale of a mine entry this anisotropy causes the roof to behave as a series of beams or plates. The plate boundaries are defined by bedding planes or stratigraphic contacts.

Secondary sedimentary structures are caused by penecontemporaneous physical or chemical accretionary processes. Examples are concretions or sulfur balls. Concretions are semispherical nodules of claystone (shale), pyrite, siderite, or calcite that range from microscopic size to several feet in diameter. Concretions are the result of chemical precipitation which usually forms around some nucleus, commonly a fossil. Concretions include the host rock, but tend to be denser. Differential compaction around the nodules formed slick surfaces that have little adhesion to the surrounding rock, thereby allowing concretions to fall out of the roof. In some shale roof, concretion populations are quite dense and cause severe roof problems. Concretions within the coal seam play havoc with cutting machines and must be blasted out occasionally. Pyritic concretions or sulfur balls and coal balls (iron carbonates), in addition to being a pollutant source, are extremely hard and may pose a safety hazard due to sparking.

7. Mechanical Discontinuities

Coal strata rarely act as homogeneous isotropic masses, due partly to the material variation mentioned above and also to mechanical fractures. In fact, it is likely that the majority of roof failures occur along pre-mining discontinuities as opposed to soils or weak rock where the failure surface cuts through the intact material. Structural discontinuities are the result of syngenetic or post-diagenetic stress applications. Stresses may have originated from the diagenetic process or from tectonic sources. The distinction between classes is morphologically difficult but the causitive processes are important to understand for predictive purposes. Mechanical discontinuities include slickensides, cleats, joints, faults and fractures zones.

a. Slickensides. Slickensides are surfaces of little cohesion distinguished by a shiny face and caused by differential movement across that surface. Slickensides are associated with a number of phenomena and are not usually categorized in a class by themselves. In fact, slickensides only imply a surface with little cohesion and have little genetic meaning. For practical mining purposes, however, the redundancy and inaccurate definition may be overlooked.

Coal measure sediments exhibit a wide range of compactional properties. Shales may reduce in volume from 15-50%, whereas sandstones will compact only 10-15% of its original volume. The difference in compaction causes relative movement across a shale-sandstone interface and creates slickensides. Slickensides are also associated with kettles (petrified tree stumps), concretions, clay veins, and can be seen across any other feature which has different consolidation rates. Slickensides are found along the current failure surfaces of slump features. Minor "slicks" may also be found in shale clasts or argillaceous salt or sandstones. More significantly, slickensides generally can be associated with poor roof. Recognition of the cause will determine the extent of bad roof, and provide hints to optimize corrective support systems.

b. Cleating. Cleats are high angle, orthogonal joint systems that pervade most coal beds. The better developed set is called the face cleat, while the less prominent set is referred to as the butt cleat. Cleat formation has been variously attributed to dehydration, devolitization, and/or tectonic stress application. Though usually limited in extent to the coal or (occasionally) the immediate roof and floor, cleats tend to parallel regional tectronic features. Local structures (faults, folds) can cause deviations in the orientation of the cleat surface. Anomalous changes in cleat attitude may be seen as a precursor to the nearby presence of tectonic structures. Though the lateral extent of any given cleat plane is limited, the homogeneous interfacing and connection with regional joint systems provides ideal pathways for methane and groundwater. Pronounced differences in methane emissions can be easily effected by face orientation with respect to face cleat attitude.

Cleat spacings vary from millimeters to meters and due to their nature, exhibit no tensile and little shear strength across the cleat plane. Proper orientation of the mine plane to the cleat has reduced cutting energies by as much as 50% resulting also in decreased bit wear, increased cut coal block size, and decreased respirable dust generation. When entries are parallel to face cleats, rib sloughing can be a cause of concern.

c. Joints. Joints are planar fractures caused by natural forces, across which negligible movement has occurred. Joint planes may be restricted to single beds or transect entire rock sequences. Their occurrence, frequency and orientation reflects tectonic influence. A knowledge of the local structural geology can, therefore, provide information on joint occurrence. Joints rarely occur alone. Spacing frequencies range from inches to hundreds of feet. Joints provide little strength in tension and shear. Joint strength depends on cohesion, normal stresses across the plane, and the rock's coefficient of friction.

The translithologic extent of some joints makes them excellent conduits for groundwater and gas migration, particularly across impermeable strata. "Open" joints can prove to be troublesome when mining under bodies of surface water or mine pools. A higher density of joint occurrence in the roof (near fault zones) can also spell problems.

d. Faulting. Faulting is a deformational manifestation of tectonic stresses. Faulting in Central Appalachia is of two basic types: post-depositional and postcontemporaneous.

Post-depositional faulting can extend hundreds of miles and disrupt strata with thousands of feet of offset. The shear zone (gauge zone) is often filled with crushed rock (breccia) which can act as an excellent hydraulic conduit, or if weathered to clay, can act as an impermeable barrier. High water and methane pressures have been occasionally associated with fault zones causing in-rushes or gas bursts. The density of joints also tends to increase in the vicinity of a fault, thereby causing severe roof problems. The obvious hazard of an unsuspected fault is in the disruption in the mining cycle. Minor faults with displacements of several feet will require the construction of ramps up or down to the continuing seam. This requires exposing fractured roof or floor of dubious structural integrity. Central Appalachia has been a relatively stable tectonic region with severe faulting only occurring in the Pine Mountain area, Ressell Fork, and Irvine Paint Creek area.

Post-contemporaneous faulting does not cause the failure problems that post-depositional faulting can, although both can adversely affect the coal geometry. Tectonic activity occurring simultaneously with peat formation will cause severe changes in the depositional-erosional scheme. For example, the Paint Creek-Irvine fault zone which transects Eastern Kentucky was active during and immediately after peat deposition. The resulting fault scarps acted as dams to lateral stream migration on the down faulted side, thereby causing severe erosion immediately adjacent to the downthrown side of the fault. This area in Eastern Kentucky probably has a high incidence of sandstone channels and should be approached with caution. The upthrown side created structural highs and allowed the development of soil structures (roots, burrows, desication cracks, etc.). These areas exhibit poor roof characteristics.

Fracturing of the roof, coal, and floor strata can also occur as a result of the high stress modifications caused by the mining activity. Recently, certain geomorphic features identifiable by aerial photo and remote sensing techniques have been correlated to poor roof conditions (Rinkenberger, 1979). Such features as lineaments, stream valleys, and confluences may be either structurally controlled (joints and faults) or may effect the roof by overburden stress relief.

APPENDIX B

THE RELATIONSHIP OF RECOVERY RATIO TO MINING CONDITIONS

Mining Conditions will inevitably have an important impact on coal recovery, whatever the technological solutions to ground control, coal winning, haulage, etc. Accordingly, the conservation requirement defines recovery targets as a function of conditions. The data presented in Table B-1 represent a judgmental assessment of the relative attractiveness of the four comtemporary mining methods when faced with a variation in one geological factor (e.g. dip, floor quality, seam thickness, etc.) with all other factors held constant. These judgments were based, in part, upon assessments of the various methods made by Stefanko (1977), Kuti (1975), and Cominec (1975). The information thus obtained on individual conditions was examined to identify these factors which discriminated best among the technologies. As indicated in Section II of the text, they are:

(1) Depth.

(2) Seam thickness.

(3) Roof cavability.

(4) Roof stability.

(5) Seam regularity.

A dichotomization of each factor into high and low values produced unique sets of conditions, for which a preferred technology was nominated. Figure 3-1, on page 3-4 of the text, presents the results of this analysis, which when combined with the recovery percentages of Table 10, establishes recovery minimums for each set of conditions to be expected in the target resource.

Table B-1. Relative Attractiveness of Various Mining Methods as a Function of Geological Conditions

(Rated as 1 for most attractive to 6 for least attractive)

	Room and Pillar Methods				Caving Methods		
	Continuous		Conventional		Longwall	Shortwall	
	Rob Pillars	No Rob	Rob Pillars	No Rob		Shuttle	AFC
Thick Seam	5	3	4	2	1	3	3
Depth 100'-500'	2	1	2	1	4	3	3
500'-1,000'	2	1	3	2	2	2	2
1,000'-1,500'	3	2	4	3	1	2	2
1,500'-2,000'	4	3	4	3	1	2	2
Weak Immed. Roof	5	4	5	4	1	6	5
Strong Immed. Roof	2	1	2	1	6	5	5
Weak Prin. Roof	5	4	5	4	1	2	2
Strong Prin. Roof	1	2	1	2	6	5	5
Weak Floor	4	3	4	2	6	6	5
So-So Floor	2	1	2	1	2	2	1
Strong Floor	1	1	1	1	1	1	1
Surplus Water	5	4	5	4	3	5	3
Faults-Frequent	3	2	2	1	6	4	5
Wants, Washouts	[illegible]	2	2	1	6	4	5
Partings	2	1	2	1	2	2	2
Rolls	3	2	3	1	5	4	4
Pitch 5°	1	1	1	1	2	1	1
5°	2	2	2	1	4	3	3
Methane-Strong	6	4	5	3	2	3	3
Inclusions-Major	3	3	1	1	2	2	2
Hard Cutting	3	3	1	1	2	3	3

APPENDIX C

DETAILED INFORMATION RELEVANT TO THE SAFETY REQUIREMENT

C.1 Injury Rates for Coal Mining and Comparable Industries

C.2 The Cost of Safety in Relation to the Overall Cost of Coal

APPENDIX C.1

INJURY RATES FOR COAL MINING AND COMPARABLE INDUSTRIES

Figures C.1-1 and C.1-2 portray, respectively, the injury experience of underground coal mining and historical injury rates for industries judged to have comparable hazards. Note that the trend for coal mining is sharply downward, having decreased to a total injury rate of less than 70 per million man-hours in 1978, from over 120 per million man-hours in 1972. It is not surprising that the West Virginia statistics closely parallel the industry aggregate since West Virginia accounts for a large portion of total U.S. underground production. In contrast to underground coal mining, injury rates in metal mining, and petroleum production have been rising somewhat, while the rates for primary metal manufacturing, construction, and non-metal mining (except coal) exhibit a modest decline.

These data have two implications for setting a safety requirement for underground coal mining. First, in terms of total injuries, coal mining at a rate of about 70 per million man-hours in 1978 falls near the upper portion of the range of 85 to 25 per million man-hours for comparable industries. Thus, some improvement in the total injury rate is indicated for coal mining, but it need not be regarded as a current pressing problem. However, as pointed out in the discussion of Section III-D of the text, rates for fatalities and disabling injuries remain very high for coal mining relative to the above set of comparable industries.

Second, it is reasonable to expect long term changes in injury rates, both for coal mining and the industries with which it is compared. Thus, the safety requirement must incorporate the notion of a moving baseline whose role in setting safety goals is very similar to the role of the moving technological baseline in setting production cost goals.

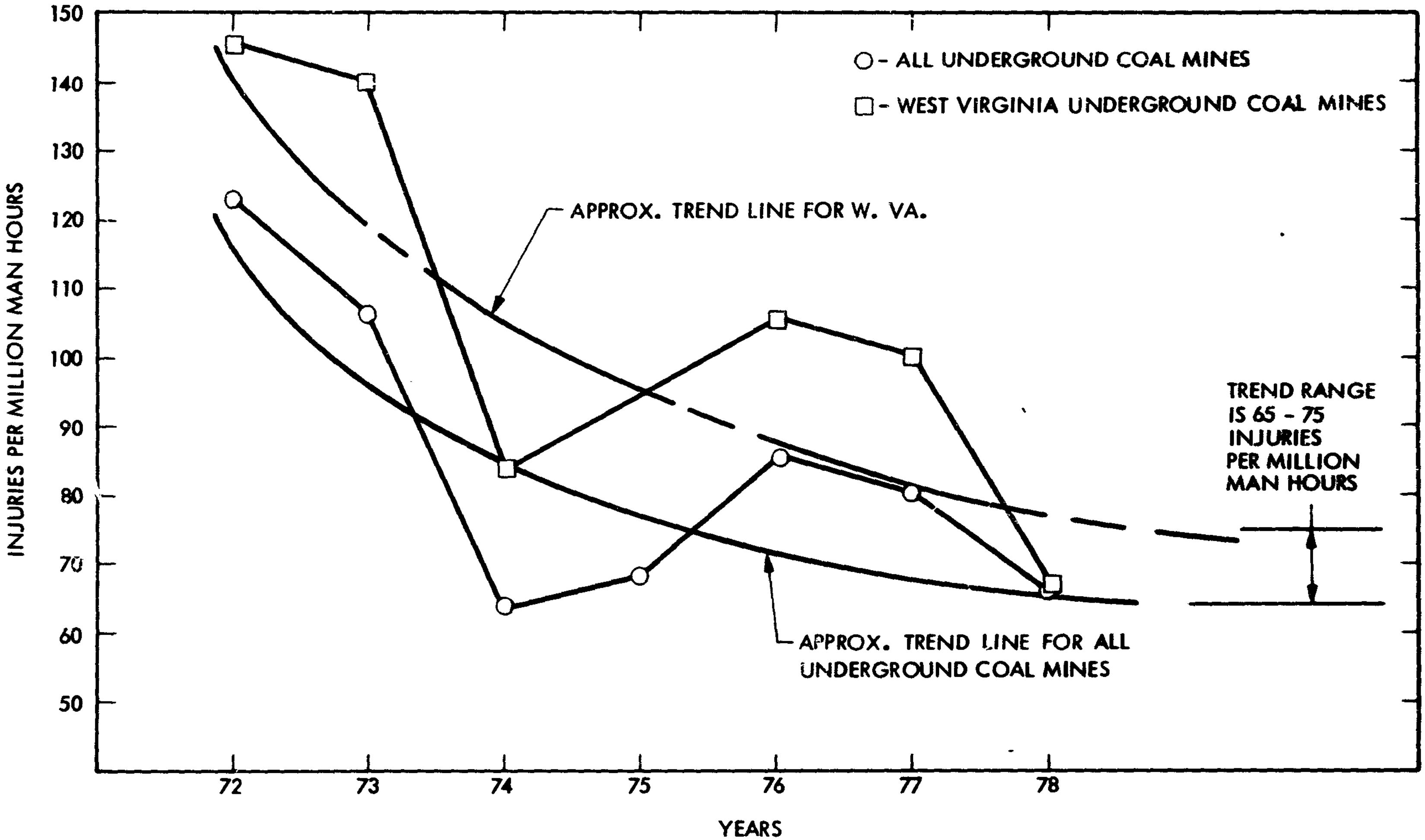

Figure C.1-1. History of Total Injuries per Million Man Hours for Underground Coal Mining with West Virginia Reported Separately

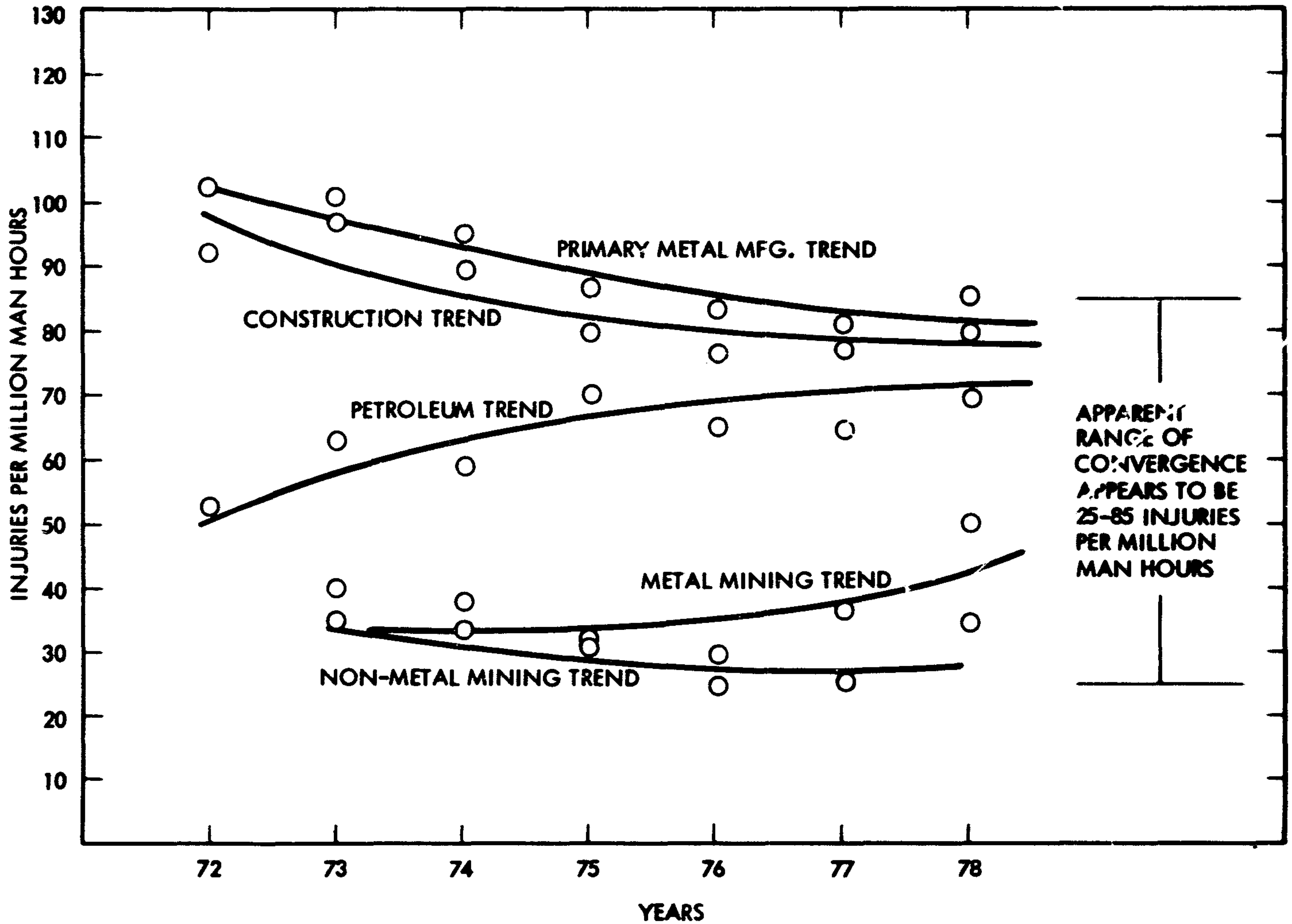

Figure C.1-2. History of Total Injuries per Million Man Hours for Industries Comparable to Underground Coal Mining

APPENDIX C.2

THE COST OF SAFETY IN RELATION

TO THE OVERALL COST OF COAL

In estimating the extent to which safety impacts the cost of coal, the following steps were taken: first, understand the variables affecting the value of a fatality or disabling injury; second, talk with labor, management, and government personnel to establish what changes have taken place over the years in compensating for loss of life and limb; and third, determine whether or not it is practical to place a value on fatalities and disabling injuries based on all the information previously assembled.

Research on the "value of a human life" has employed two major approaches, (1) the human capital notion, and (2) the willingness to pay concept. The human capital approach states that the value of a life is basically what a person's net worth would be, based on his earnings. This is the approach preferred by industry because it is easily quantifiable, and it is the cheapest. The willingness to pay concept essentially looks at what a person would be willing to pay to improve his chances of surviving, or not being disabled. The main difference between these two philosophies is that the human capital approach looks at a person's worth in relation to the GNP (separate from the person), and the willingness to pay approach tries to consider the person's measure of own worth (separate from the GNP). If these two philosophies could somehow be reconciled, then perhaps a practical, socially acceptable value could be reached.

However, each approach has its shortcomings. The human capital approach does not account for the value of home production, the value of leisure, upward mobility, the ripple effects within the family structure (e.g., offspring altering long-range educational goals, and subsequent higher wages, to deal with the immediacy of compensation for lost wages and family unity), and changes in mortality and injury rates. Schelling (1967) states "there is no reason to suppose that a man's future earnings, discounted in any pertinent fashion, bear any particular relation to what he would pay to reduce the likelihood of his own death". In view of the above discussion, McGuire (1979) observes that the legitimacy of the use of the human capital approach revolves around it being used "<u>only</u> as a quantification of costs that are directly or indirectly quantifiable - not as a tool for evaluating programs that potentially prevent injury or save lives". Accordingly, many think that the human capital approach yields a lower bound for the value of human life.

A. AN EXAMPLE OF THE HUMAN CAPITAL APPROACH

This conclusion is indirectly supported by Dicanio, et al.'s (1976) study of costs to industry and society from work-related injuries and deaths in underground coal mines. The model used by Dicanio, et al. is based on the human capital approach. The cost factors considered by this model include:

(1) Compensation payments by companies.

(2) Lost coal production by companies.

(3) Investigative costs of companies.

(4) Wage losses to miner and family.

(5) Compensation payments by public agencies.

(6) Investigative costs of public agencies.

Dicanio, et al. recognize other cost variables as well, and these will be discussed later. However, this model ignores a number of other factors because they are difficult to quantify:

(1) Lawsuits for deaths and disabling injuries.

(2) The ripple effect of losses in production in all sections of a mine resulting from a fatality or major disabling injury.

(3) The cost of retraining and rehabilitating disabled workers.

(4) Long-term medical treatment expenses (i.e., usually only the immediate short-term expense required to repair the disability is considered).

In commenting on long-term medical expenses, Dicanio et al. state that "long-term medical costs can be several times that of the short-term costs". One may argue that public and corporate compensation cover most of the above variables. However, Kerr (1979) notes that compensation benefits have not changed much within the last decade (excluding black lung, where legislation has improved benefits). The Department of Labor has been actively seeking an adjustment in public agency compensation to keep up with inflation, with only modest success. Corporate compensation programs are perhaps even less likely to keep abreast of inflation. Dicanio et al. report recent industry-wide averages for the costs of fatalities and disabling injuries as follows:

(1) Fatalities - $125,000/injury.

(2) Disabling injuries - $4,000/injury.

B. WILLINGNESS TO PAY APPROACH

The "willingness to pay" concept supposedly reflects a more personal measure of worth. The problem comes in assessing the value of increasing safety for an individual. Schelling (1967) recommends questionnaires. However, a questionnaire seems to be an inadequate tool to measure this complex, emotionally charged issue. Thaler and Rosen (1973) attempted to infer what people require as compensation for risk by analyzing acceptable wage rates for various jobs. This technique yielded a value for life of approximately $200,000. Critics of this approach point out that people who are generally insensitive to risk enter hazardous jobs. Therefore, it is felt that Thaler and Rosen's figures underestimate the true value, or willingness to pay, of the population at large. However, given that "willingness to pay" more accurately addresses the true value of life, Thaler and Rosen's result of $200,000 may be taken as a low estimate for the value of life. This value of life will be used in an initial attempt to quantify the trade-off between production cost and safety.

C. Analysis of Cost-Safety Trade-off

Consider a general cost expression which describes the total cost as a function of fatalities, permanently disabling injuries, and non-permanently disabling injuries.

$$C_{TOT} = (I_{TOT}/P)\ (f\ C_f + d\ C_d + n\ C_n)$$

Where:

C_{TOT} = Total cost of injuries (\$/ton)

P = Production (tons/man-hour)

I_{TOT} = Total injury rate/10^6 man-hours

f = Proportion of fatalities in relation to the total injuries

C_f = Assumed cost of a fatality (\$)

d = Proportion of permanently disabling injuries in relation to the total injuries

C_d = Assumed cost of a permanently disabling injury (\$)

n = Proportion of non-permanently disabling injuries in relation to the total injuries

C_n = Assumed cost of a non-permanently disabling injury (\$)

Using the data provided in the requirements, the following estimates may be made:

f = .005

d = .03

n = .58

The value for a disabling injury may be estimated by assuming that it bears the same relationship to the cost of a fatality as the ratio implied by the data reported by Dicanio, et al. above, i.e.:

$$C_d = 200{,}000\ (4{,}000/125{,}000) = 6{,}400$$

Moreover, to be conservative, assume that:

$$C_d = C_n$$

Now for Central Appalachia,

$$I = 105 \text{ injuries}/10^6 \text{man-hours}$$

and $P = 1$ ton/man-hour

Upon substituting the above numerical values into Equation (1), a lower estimate for the cost of injuries is obtained:

$$C_{TOT} = \frac{105}{(1)(10^6)} \left(.005(.2\text{x}10^6) + .61(.0064\text{x}10^6)\right)$$

$$= \$0.52/\text{ton}$$

This value is less than 2% of the current long-term contract price for steam coal.

Now, repeat the above calculations using more liberal figures for the cost of the three types of injuries:

$$C_f = \$10^6$$

$$C_d = \$10^5$$

$$C_n = \$10^4$$

Then,

$$C_{TOT} = \frac{105}{(1)\ (10^6)} \left(.005(10^6) + .03(.1\text{x}10^6) + .58(.01\text{x}10^6)\right)$$

$$= \$1.47/\text{ton}$$

It appears that even with more liberal assumptions about the value of life and limb, the cost of injuries is a small percentage of the cost of coal. This is not to be construed as saying that safety is not important. Rather, it says that the value placed on safety may outweigh *pro forma* cost calculations. In sum, there is no justification for any serious attempt to make trade-offs between safety and production cost.

APPENDIX D

DETAILED INFORMATION RELEVANT TO THE PRODUCTION COST REQUIREMENTS

APPENDIX D.1

THE RELATIONSHIP BETWEEN RETURN ON INVESTMENT AND PAYBACK

In the work by Mansfield (1968), ROI was not directly measured, nor did the researchers attempt to translate the reported payback ratios into ROI terms. The relationship between the two profitability measures is the subject of this appendix.

Consider a piece of equipment which costs P, lasts n years, and generates r dollars of net cash flow each year. Assume that the equipment is renewed continually, resulting in an infinite sequence of investments and cash flows. Let i be the internal rate of return (ROI) generated by the investment. It is easy to show that the Present Value (PV) of the infinite sequence of investments is

$$\text{PV (investment)} = P/(1-e^{-in}) \tag{1}$$

under the assumption of continuous compounding. Similarly, the expresion for the Present Value of the cash flow generated is

$$\text{PV (cash flow)} = \int_0^{\infty} re^{-it}\,dt = r/i \tag{2}$$

which is a well known result from engineering economy. The internal rate of return is determined by equating the two present value expressions:

$$\text{PV(investment)} = \text{PV (cash flow)}$$

whence

$$P/(1-e^{-in}) = r/i$$

$$\text{or} \qquad (P/r) = m = (1-e^{-in})/i \tag{3}$$

which is the explicit relationship between the payback period m and the internal rate of return i.

In the cost requirements definition, data was used on a ratio of payback periods. To interpret the result of Equ. 3 in terms of this ratio, denote the average capital project with a zero subscript (o) and the innovation with a unity subscript (1). Then, formally, the ratio of payback periods may be expressed as:

$$\frac{m_o}{m_1} \quad \frac{i_1}{i_o} \; \frac{1-e^{-i_o n_o}}{1-e^{-i_1 n_1}} \tag{4}$$

The problem is to find the return on investment which is required to justify the innovation, given a constraint on the minimum payback ratio. Note that m_o/m_1, i_o, and n_1 are all known quantities. Using this information, we define a new variable K which permits Equ. (4) to be put into a form suitable for numerical solution:

$$i_1 + Ke^{-i_1 n_1} - K = 0 \tag{5}$$

where:

$$K = \frac{m_o}{m_1} \quad \frac{i_o}{(1-e^{-i_o n_o})}$$

Equ. (5) is readily solved by Newton's method. The initial estimate for i_1, given below insures rapid convergence to a solution:

$$i_1^{(o)} = \frac{(e^{Kn_1} - Kn_1 - 1)}{(e^{Kn_1}/K) - n_1} \tag{6}$$

Table D.1-1 presents the results of a parameter study of Equ.(4), which spans a broad range of values for n_o, n_1, i_o, and i_1. Analysis of the tabulated values indicates that bracketing cases contain the following sets of values:

low minimum ROI: $n_o = 10$, $n_1 = 5$, $m_o/m_1 = 1.6$

high minimum ROI: $n_o = 5$, $n_1 = 10$, $m_o/m_1 = 2.0$

The relationship between minimum ROI and payback ratio is plotted in Figure D.1-1 for these two cases, with the cross hatched area representing the region in which innovations are expected to fall. Examination of the cross hatched region reveals that an ROI range of 1.5 to 2.5 corresponds well to a payback range of 1.6 to 2.0, in view of the probable variability in Mansfield's (1968) data, and the need to require a minimal risk premium.

Table D.1-1. Minimum Required ROI for an Innovation (i_1) as a Function of Payback Ratio (m_0/m_1), Project Lifetimes (n_0, n_1) and ROI for an Average Capital Project

m_0/m_1		1.6				2.0			
n_0		5	10	10	5	5	10	10	5
n_1		5	10	5	10	5	10	5	10
i_0 =	.08	.311	.203	.118	.380	.431	.272	.188	.481
	.10	.334	.228	.142	.399	.458	.301	.220	.505
	.12	.358	.253	.168	.419	.486	331	.254	.529
	.15	.394	.293	.211	.450	.529	.337	.308	.567
	.18	.431	.333	.256	.481	.572	.425	.365	.605
	.20	.456	.360	.288	.503	.602	.458	.403	.632
	.25	.520	.430	.370	.559	.677	.542	.501	.700
	.30	.585	.502	.455	.617	.755	.630	.600	.772
	.50	.860	.805	.790	.871	1.085	1.007	1.000	1.089

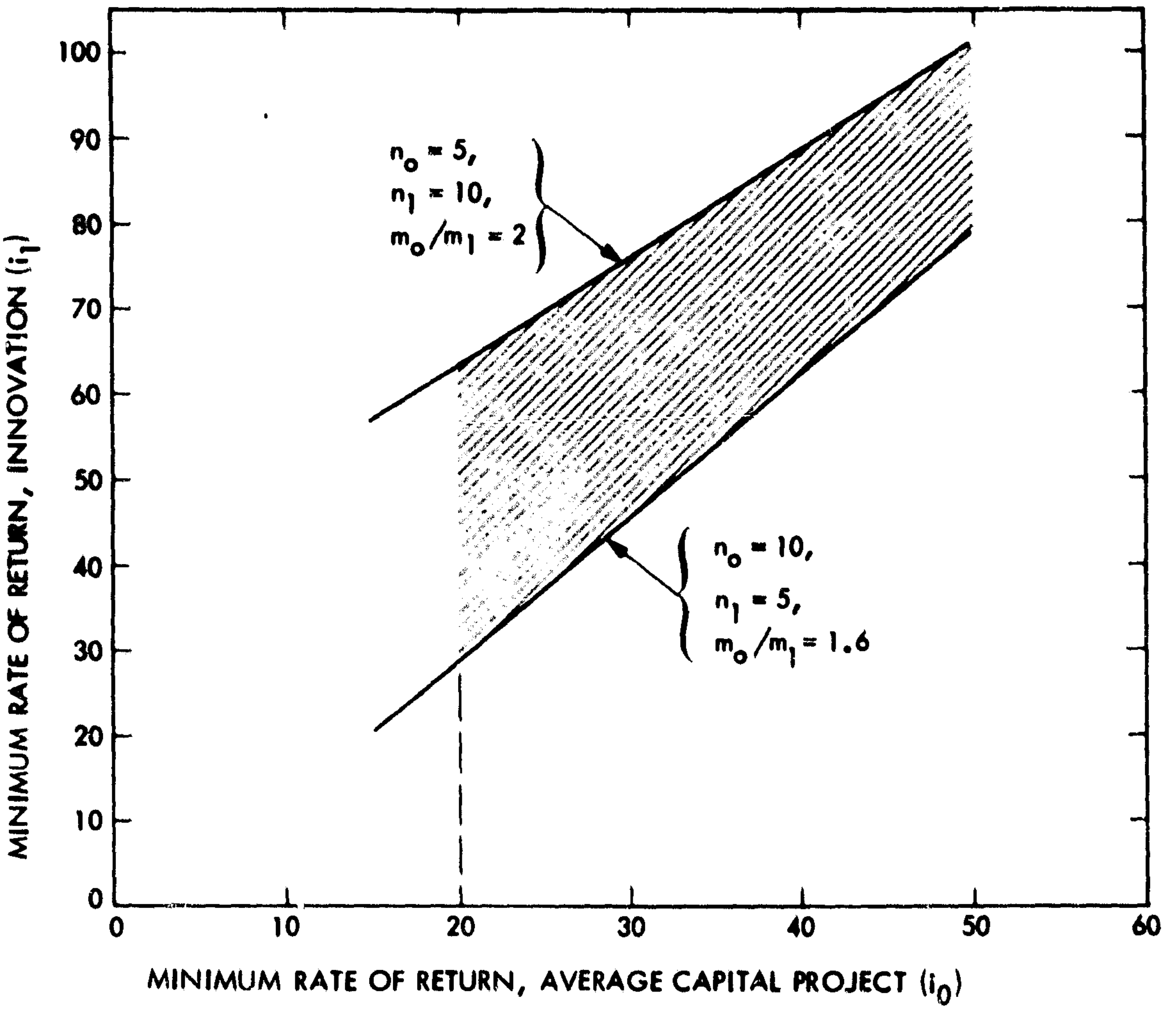

Figure D-1. Interpretation of Payback Ratio as a Ratio of Minimum Rates of Return

APPENDIX D.2

OPPORTUNITIES TO MEET THE PRODUCTION COST REQUIREMENT

There is no formula for identifying opportunities for meeting the requirement posed for production cost. There are, however, a number of different approaches, all of which boil down to sensitivity analysis of some quantitative description of underground coal mining.

Lavin, et al (1978) developed an algebraic description of deep coal mining, and subsequently used this model to compute price sensitivity coefficients. Formally, a price sensitivity coefficient is defined as the percent change in price as a result of a one percent change in the variable of interest. It is rare to find a variable whose impact on price is not moderated by the effects of other factors, thus, price sensitivities tend to be substantially less than one.

The algebraic description mentioned above has two drawbacks which make it less than ideal for the present purpose. First, it is a relatively onerous task to update all of the model coefficients to reflect first quarter 1980 costs. Second, the model is structured around labor and capital productivities which are derived variables. We now feel it is more meaningful to talk about tons per section-shift (or section-hour). Consequently, we elected to use an existing computer-based model for the sensitivity work needed to identify opportunities for meeting the cost requirements.

The model used was developed by the NUS Corporation for EPRI, and subsequently made available to JPL by DOE. The price sensitivities reported in Table D.2-1 were developed by Mabe (1979) for a 1.37 million ton/yr room and pillar mine, operating in a 5 ft seam in Central Appalachia in the year 2000. These sensitivities were derived by making successive one variable changes, with all other variables held constant. In each case the variable of interest was increased or decreased by 20%. Thus, the tabulated values approximate the results of rather substantial changes in either response inputs or coal output.

Note that the sensitivities are listed in decreasing order of importance, with section output being far and away the most important, at a value of 0.77. In order of decreasing importance there are hourly labor costs at 0.31, operating supplies and other consumables at 0.25, and the cost of production section equipment at 0.07.

These figures, which are in general agreement with the algebraic results of Lavin, et al (1978), have fairly clear implications for the overall architecture of a system with substantially improved cost performance. A certain amount of improvement can be realized by reducing manning costs (and possibly the cost of operating supplies) if the corresponding increase in the cost of underground equipment is kept within bounds. However, a much more attractive strategy is to develop face equipment which is much more productive together with any required upgrading of fixed haulage. To oversimplify it a bit, it is possible to either (1) reduce the resource inputs, or (2) expand the coal output. Expansion of output is more appealing even when one recognized that this expansion will be achieved at some cost. Note that the sensitivity to the cost of section equipment is an order of magnitude less than the sensitivity to increased shift output.

Table D.1-2

APPENDIX E

AN ALTERNATIVE PERSPECTIVE ON RESOURCE CONSERVATION

The basic question explored here is whether a case can be made for conservation of coal resources (extraction efficiency) as a criterion for evaluating "advanced" coal extraction systems. A corollary question is, if so, can an objective standard of conservation be defined.

First, it is necessary to state the case in favor of conservation. The case can be made on the basis of (at least) the following five points:

(1) Foreseeable Scarcity Under Conditions of Exponential Growth. Domestic coal reserves currently are estimated to last roughly 500 yr at the existing rate of consumption. But as Figure E-1 indicates, sustained exponential growth in production/consumption would reduce the lifetime of this nonrenewable resource considerably.

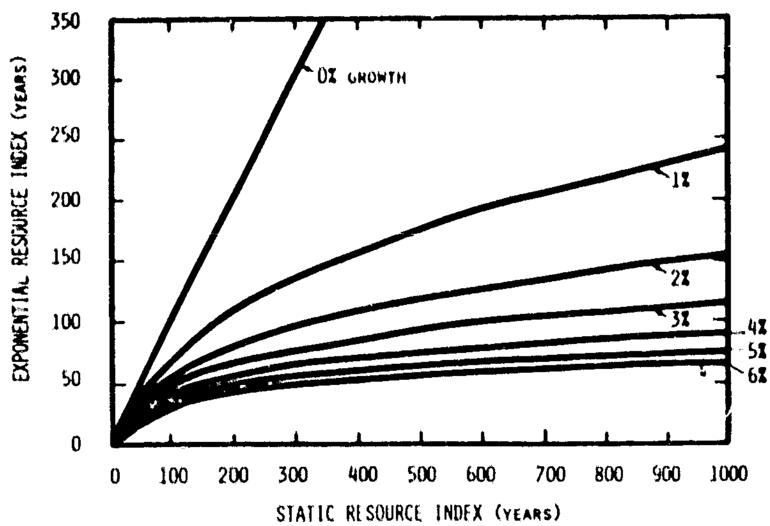

Figure E-1. Exponential Versus Static Resource Indices as a Function of Annual Growth Rates

Thus, a 2% annual rate of growth in production would reduce a 500-yr reserve to only a 100-yr reserve. In reality, it seems unlikely that coal production would grow at such a rate over the period of the next century. But there are circumstances under which the rate of coal production would be called upon to increase rapidly. For example, a prolonged interruption in the flow of oil from the Middle East, perhaps as the result of warfare, would leave Europe and Japan with a desperate need for alternative sources of fuel. Under such circumstance, the U.S. might well decide to expand its coal production dramatically, to export coal to its allies, and/or to displace its own use of petroleum to make petroleum more available to its allies, and/or to rapidly develop synfuels. The result of any of these would be to make coal resources appear more scarce.

(2) Risk and Regret. Georgescu-Roegen (1979) suggests that the purpose of conservation (of any kind) is to minimize the amount of future regrets. Note that this implies a bias which values commitment costs

more highly than opportunity costs. This is related to the fact that real socio-ecological systems (as opposed to idealized econometric models) are characterized by high-order feedback, instability, novelty, and what Forrester labels "counterintuitive" behavior. As suggested in the previous point, contingencies may arise which may make people in the future need coal more desperately than we now anticipate. One argument for conservation is simply to avoid painting our society into a corner.

(3) Need for Resource Types. The above point suggests the need to conserve coal as a generic resource. However, there is a wide variety of specific types of coal resources with different geological, physical, and chemical properties. Qualitative characteristics which affect existing coal markets are surface vs. deep seams, energy density, coking vs. steam application, and sulfur content. Future technologies/applications may make the value of specific coal types different from what current markets indicate. For example, synfuels technologies seem highly dependent on qualitative characteristics of coal feedstocks. This indicates a possible need for "strategic reserves" of particular types of coal resources, as distinguished from generic coal reserves.

(4) Stakeholder Benefits. Just as the last point disaggregates the coal conservation issue into specific coal types, it is also possible to disaggregate the general public interest in conservation into specific stakeholder interests. Some stakeholders may see benefits from the increased lifetime of particular mining units that conservation implies. For example, local communities whose economic base depends heavily on coal mining might consider it beneficial if the productive lifetime of the local mine could be extended significantly. The local mine workers might feel similarly. Under some conditions, the coal mining company also might view extended mine lifetime as desirable.

(5) Possible Cross-Benefits. Particular methods for conservation of coal resources conceivably could have positive impacts in other areas which would make the conservation procedures more attractive. Plausibly, conservation practices might reduce environmental degradation, increase health and safety, or even reduce overall production costs. Even where conservation entails marginal increases in production cost, these might be offset by consideration of the total benefits of the practice beyond the value of conservation itself.

In further assessing the issue of coal resource conservation, the following points also should be considered:

(1) Nonzero Value of Conservation. Though in reality coal conservation may not be an urgent concern at present, there probably is some positive additional cost society (i.e., political decision makers) is willing to pay to insure a higher efficiency of resource recovery than what market prices alone would induce. For example, it seems plausible that most people would be willing to pay 2% more for coal if recovery efficiency could thereby be increased from 50% to 90%. (But this does not mean that the market alone would offer the opportunity to make such a choice.)

(2) Lack of Analytical Solutions. There is no analytical derivation possible of the "correct" standard or marginal cost for coal resource conservation. The positive value of conservation suggested above can bedetermined only through political processes. Different decision makers will attach different values (including zero in some cases) to resource conservation; the distribution of political power at any time will determine which specific value would be effective.

(3) Alternatives for Conservation. Assuming conservation is a desirable goal, there are ways to achieve resource conservation other than through innovations in coal extraction technology, and these may be more cost-effective (according to some appropriate measures). One set of options to consider is regulatory policies. Another set to be taken into account is what might be called default constraints--that is, environmental, labor, societal or other factors which reduce or prevent coal production, thereby achieving conservation by default. The latter may be more influential than any regulatory policy or technical process intended to conserve coal resources.

(4) Correct Price. Finally, there are reasons to believe that the market price of coal does not reflect the real value to society of coal production. A variety of external factors would affect the price of coal both positively (e.g., petroleum displacement) and negatively (e.g., the CO_2 hazard). Determining the "correct" price of coal, again, is a political question, and one on which noneconomic criteria can and should be brought to bear. For example, Georgescu-Roegen suggests that the marginal price of any nonrenewable resource should be made not less than that of the most available renewable substitute. This may not be the most tractable rule, but the definition of some rule for determining the shadow price of coal is a problem which deserves thoughtful analysis.

APPENDIX F

THE ENVIRONMENTAL IMPACTS OF A CONTEMPORARY MINE: AN ILLUSTRATION OF THE ENVIRONMENTAL ASSESSMENT METHODOLOGY

A contemporary underground mine was asssessed to determine its environmental impacts. The evaluation was completed using <u>A Methodology for the Environmental Assessment of Advanced Coal Extraction Systems</u> by Sullivan, et al., (1980). This assessment occurred in four steps: (1) the characterization of the mining system, (2) the characterization of the physical environment where the mining system was implemented, (3) a conceptual evaluation, which identified generic impacts associated with the mining system, and (4) a preliminary evaluation, which quantified the impacts. In addition, a discussion of land use and land value following mine closure and reclamation was included.

The results of the evaluation identified the potential major environmental impacts that could result from implementing the system in the given environmental setting, along with quantitative data necessary to estimate the magnitude of each impact. These data were utilized to calculate the costs associated with (1) mitigation of the identified impacts and (2) reclamation of the land to its original or planned use.

This appendix excerpts only the highlights of the conceptual level evaluation, which is intended to flag potential environmental impacts associated with mining systems at the conceptual design stage. The conceptual environmental assessment consists of (1) a general description of the mining system, (2) an environmental identification checklist and checklist summary, and (3) impact identification sheets which describe the impacts in detail.

A. PHYSICAL SITE CHARACTERIZATION

The Fantasy Mine #1 site is located in the southeastern portion of Clay County in eastern Kentucky. This location is in the mountain physiographic region on the western border of the Appalachian plateau. It is an area of narrow flood plains, flanked by long, steep mountainsides extending from long, narrow ridgetops composed of Pennsylvanian shale, siltstone, and sandstone. For details of the mine site location see Figure F-1.

1. Land Use

In this region, over 80% of the land surface is covered by natural vegetation. The mine site is bounded to the west and southwest by broad valley flood plains which are covered with grass, herbaceous plants, and cultivated crops. On the gentle slopes above the flood plain and within the narrow upland stream valleys, the land cover and land use are a mixture of residential (no cities or urbanized areas are within the mine boundary), pasture land, and cropland (approximately 10 to 20% of the mine area). The rest of the mine area is natural woodland. Yellow poplar, white oak, black walnut, and other hardwoods dominate the north and east slopes. Black oak, scarlet oak, and hickory dominate the south and west slopes. Chestnut oaks, together with a few shortleaf and pitch pines, occupy most of the upper slopes and the narrow ridges (ESRI, 1980; McDonald and Blevins, 1965).

2. Uniqueness of Area

There are no known archaeological, paleontological, historical, or ecological critical areas located in or near the mine site (ESRI, 1980).

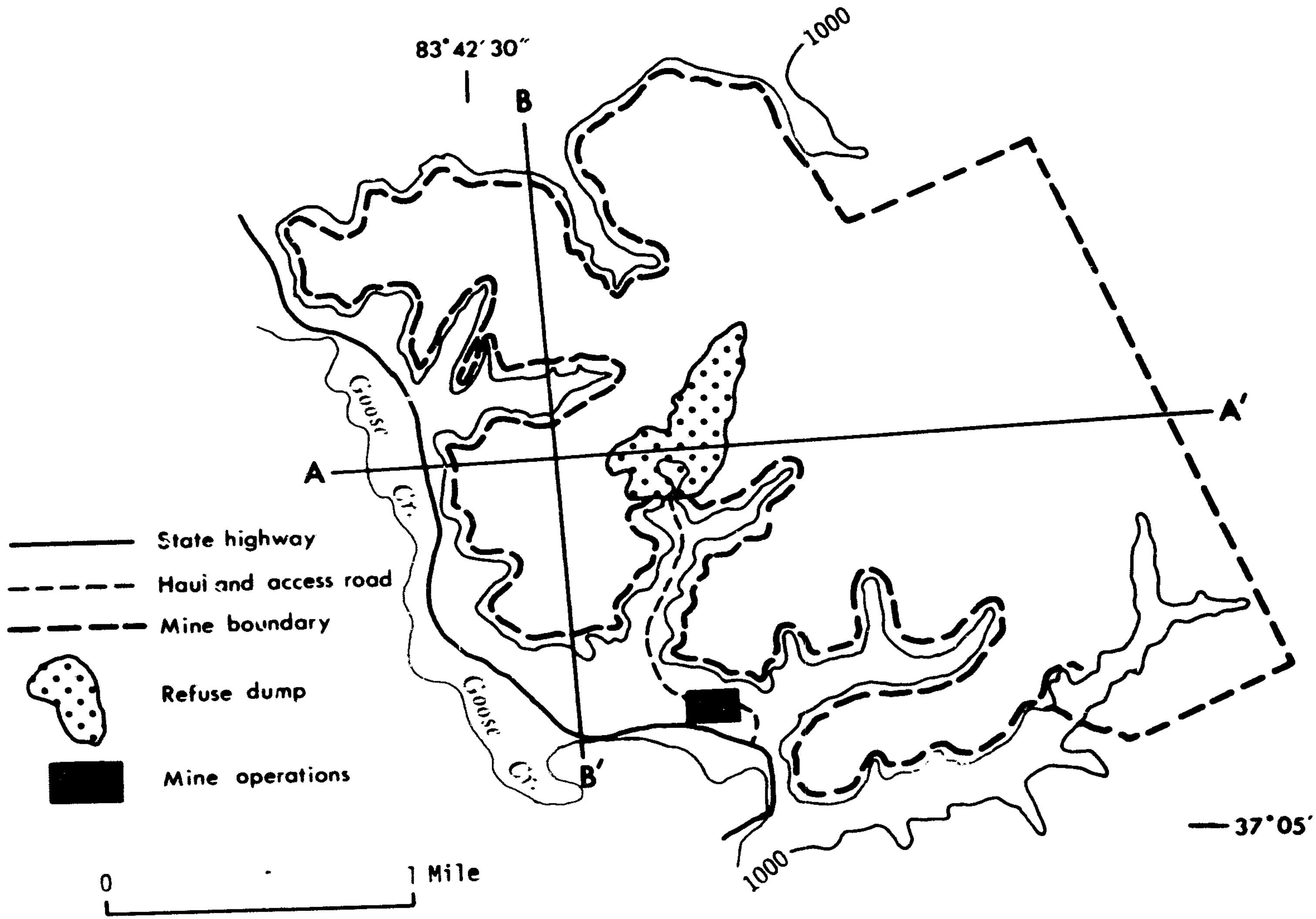

Figure F-1. Mine Site

3. Topography

The physiographic region as well as the mine site are composed of numerous steep ridges and narrow valley floors. The landforms are a combination of ridgetops (20%), sideslopes (60%), and toeslopes (20%) that blend into a complex configuration of concave and convex slopes. Over 70% of the region has a slope gradient between 35% and 50%. Near the ridgetops the slope gradient decreases to a range of 12% to 20%. At the toeslope (where most of the mining activity will occur) the slope gradient ranges from 2% to 6% with local increases to 35% (ESRI, 1980).

The maximum elevation (1686 ft above sea level) occurs in the eastern portion of the mine site and decreases to an elevation of 1185 ft in the west. The broad alluvial valleys that occur at the western and southwestern boundary of the mine site range in elevation from 800-to-900 ft. In general, the local relief averages between 300 and 600 ft.

4. Geology

The major part of the coal in the eastern Kentucky fields and in Clay County occurs in the Breathitt formation (Pennsylvanian period, 280-320 million years ago). The Breathitt formation within the mine site is composed mainly of shales, siltstones, arkosic sandstones, some carbonates (Magoffin member; Pa in Figure F-2), and minor amounts of ironstone concretions. Within the lower Pennsylvanian of the Breathitt formation (Map symbol Pc in Figure F-2) is the Jellico coal zone, containing the seams worked by the Fantasy Mine #1.

The Jellico coal zone is up to 25 ft thick and contains as many as three coal beds. Partings between the coal beds contain thin lenses of siderite, shale, and sand. Roof materials are predominantly shale, while the floor is mainly sandstone with some shale. The overburden thickness ranges from 300 ft in the west to over 600 ft in the east; overburden thickness from north to south ranges from 200 to 400 ft (Ping and Sergeant, 1978). A very high probability exists that the overburden and coal materials contain sulfide materials, and thus have the potential for producing acid mine drainage (Sullivan, et al., 1980; McDonald and Blevins, 1965).

5. Climate

The long-term averages of temperature and precipitation for eastern Kentucky are presented in Dutzi et al. (1980). In the summer the temperature may reach 100°F., but rarely for more than a few days. Temperatures below 0°F occur with moderate frequency in December, January, and February, but long cold spells are always broken by intervals of moderate temperatures. The average growing season is 175 to 180 days. Snowfall varies considerably from year to year but annually averages about 20 in. The ground seldom remains covered with snow for more than a few days after a storm (McDonald and Blevins, 1965).

6. Water Resources

a. Groundwater. The Breathitt Formation supplies very little water from drilled wells in the sideslopes and ridgetops of the mining region, but ground-water is available in adequate amounts for most domestic uses.

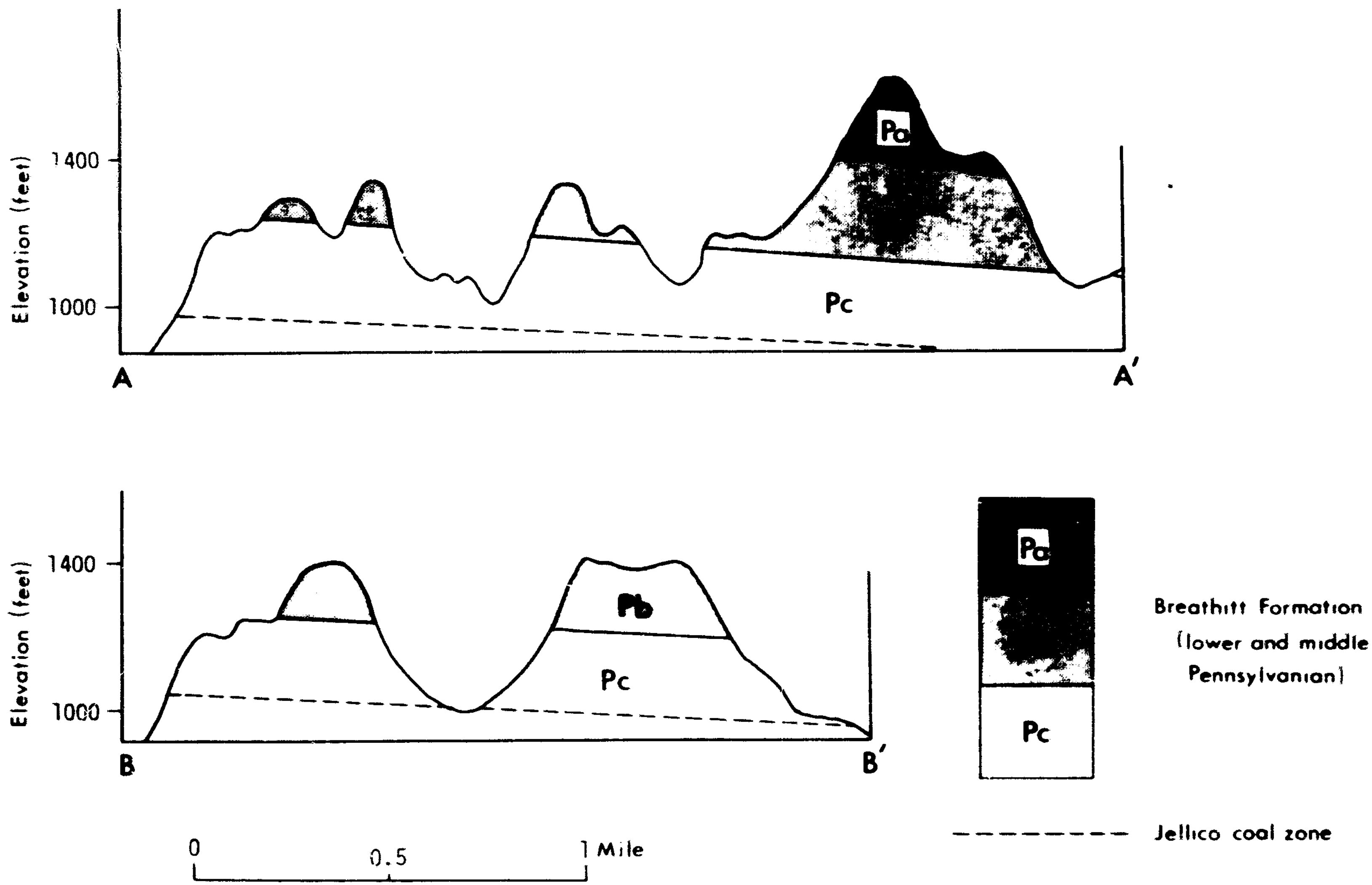

Figure F-2. Geology of the Mine Site

According to Kilburn, et al., (1962), very few wells have been drilled within the Fantasy Mine #1 region, and no data on yields are available for Clay County. However, some wells drilled in the valley bottoms have been recorded to produce at least 500 gpm.

The water in the Breathitt Formation does contain iron and is moderately hard. Most of the groundwater is fresh, but salty water may be found less than 100 ft below drainage. Nevertheless, Kilburn, et al., (1962), indicate that salty water should not be a concern within or near the mine site.

b. Surface Water. Goose Creek is the only major stream that occurs adjacent to the mine site. From the data presented by Kirkpatrick, et al., (1963), the discharge rates can be assumed to vary from 89 to 720 gpm for 98% of the year. The rest of the mine site is dissected by numerous first order streams and several second order streams. Surface water from these channels would provide a significant amount of water for the mining operation.

Although the available water resources are not abundant, there are no other competing industrial users for the existing resources. Water quality information for this region was not available.

7. Soils

The majority of the mine site is composed of the Dekalb-Muskingum-Berks soil association (McDonald and Blevins, 1965). This association makes up 96% of the soils that occupy the ridge tops and very steep side slopes. All of these soils are very stony, are shallow to moderately deep, and are derived from acid sandstone and siltstone. Soils that occur between steep uplands and broad stream bottoms (the region of active mine operations) belong to the Jefferson-Muskingum-Holston-Dekalb soil association. The Jefferson soils make up about 32% of the association and occur on the foot slopes below steeper Muskingum and Dekalb soils. The Jefferson soils are generally deep and have a gravelly loam surface layer over a clay loam or loam subsoil. The capability classes of the soils are predominately II to III on the foot slopes and VI to VII on the steeper slopes.

B. DESCRIPTION OF THE MINING SYSTEM FOR THE FANTASY MINE #1:

(1) System: Contemporary room and pillar technology using continuous miners.

(2) Coal resource: Assumed 6-ft coal bed, mostly below drainage.

(3) Mining method: 5 main entries will be utilized for access, coal clearance, and ventilation. The seam will be accessed by a drift driven from a bench. The coal will be mined by the room and pillar method, with partial extraction of the pillars. Continuous miners are electrically powered and extract the coal by mechanical cutting. Coal is removed from the working face to a processing plant outside the mine by a belt conveyor.

(4) Coal haulage: Coal from the preparation plant will be moved by conveyor to a stockpile. From the stockpile, the coal will be transported a short distance to a rail spur for loading. All outside conveyors are assumed to be covered. The rail haulage will not be considered in the environmental analysis.

(5) Access and support facilities: One two-lane gravel road will be constructed to the site of mining operations. The road will continue beyond the operations site to the refuse dump. All large refuse will be transported by truck to the dump site and stored by the valley fill method. Drainage from the dump site, operation site, and stockpile will be controlled by drainage ditches and sediment ponds. All water generated by the mine will be pumped directly to water treatment facilities.

C. IDENTIFICATION OF IMPACTS

The conceptual environmental assessment methodology focuses on potential environmental impacts which are generic to coal mining processes. Sullivan, et al., (1980), grouped mining activities under 6 general mining processes:

(1) Construction of access and haul roads.

(2) Removal of overburden.

(3) Development of systems access.

(4) Coal cutting.

(5) Coal hauling.

(6) Coal processing.

Application of Sullivan's environmental identification checklist results in the impacts summarized in Figure F-3.

D. IMPACT SUMMARY

The environmental assessment of the Fantasy Mine #1 identified the following potential environmental impacts.

(1) The refuse removed from the mined coal will be stored above ground in a valley fill. The refuse has the potential for acid water drainage, thus creating the potential for long-term pollution of the region's water resources. Even though the refuse site will be reclaimed, the possibility for erosion and structural damage in the future will present a continuing environmental hazard.

(2) In addition to the potential for acid water drainage from refuse storage, there will also be acid water production from the mine workings. Although the mine will be sealed following mine closure, water may still leave the mine from fractures in weak rock. In addition, there is also the possibility of mine seal failure. A failure of just 1 of the 4 seals would release millions of gallons of acid water directly into Goose Creek. This would not only damage aquatic life, but would also cause flood damage in the immediate vicinity of the mine and possibly downstream.

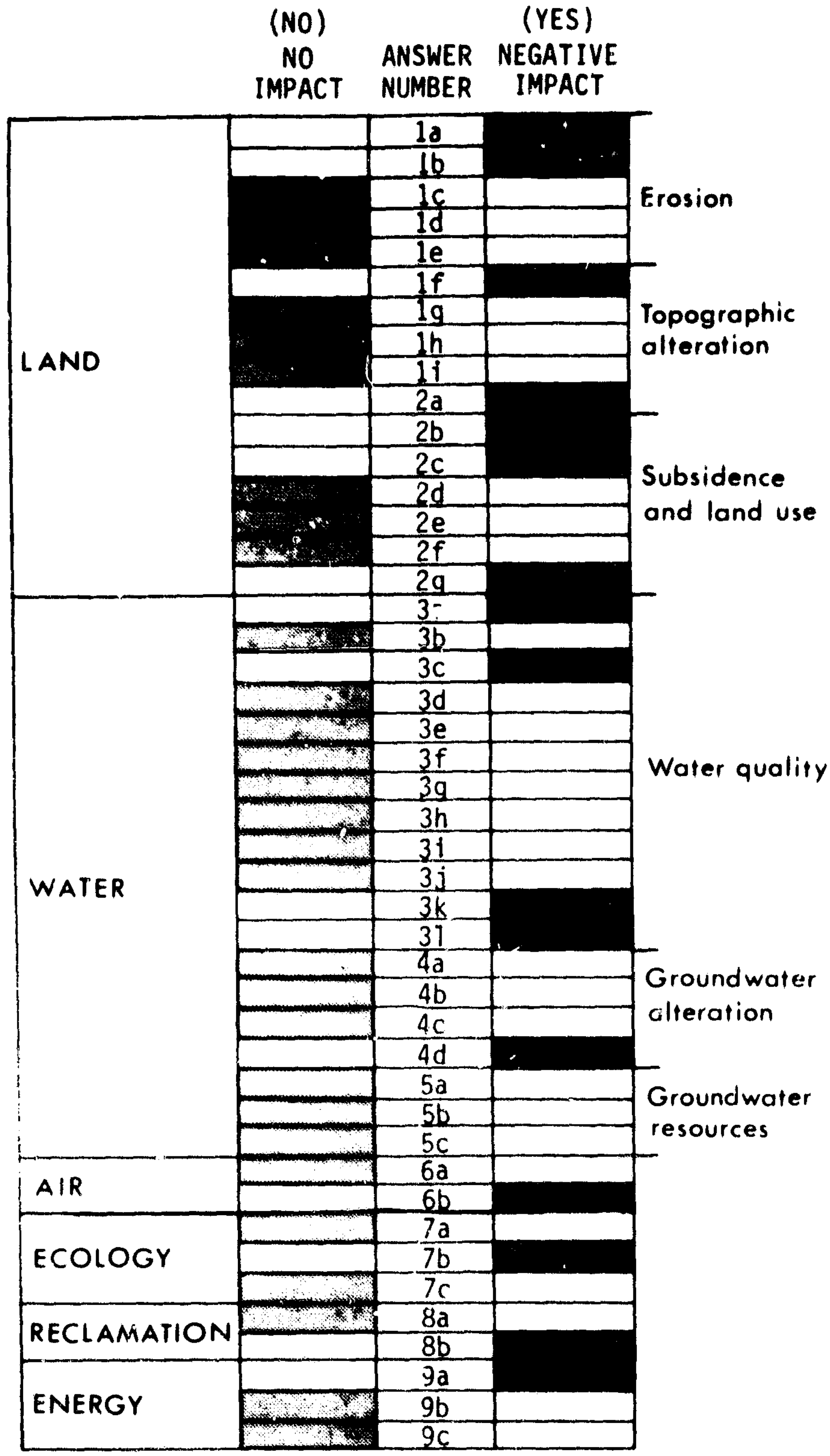

Figure F-3. Impact Summary

(3) The future land use of the mine site will be affected. Some areas will be deforested as a result of the mining operations, resulting in a loss of mature timber land; however, the cleared land will be revegetated following mine closure. Uneven subsidence will probably occur as a result of mining activity, although impacts on land use will be minimal since no urban or agricultural uses are planned f the land. Aquifers in the subsidence zone will be altered; this could result in a loss of needed ground water to local residents.

(4) The cost of environmental impact mitigation to levels prescribed by law and regulation came to less than $1/ton of coal mined. However, there will probably be long-term environmental impacts due to subsidence of the surface, acid mine drainage leaking from the mine, and possible erosion of the stored refuse.

The characterization of impacts for the Fantasy Mine #1 is concluded with a detailed description of how each impact occurs, together with appropriate mitigation measures.

E. DETAILED IMPACT DESCRIPTION AND APPROPRIATE MITIGATION MEASURES

In this section, each adverse impact identified by the checklist is discussed in the format of Sullivan's impact identification sheet. These are impacts which could occur if the defined system is implemented at the identified mine site.

1.(a) Road construction.

Nature of activity. One two-lane haul road will be constructed from the main state highway to the site of mining operations for personnel access. The road will continue from the office and bath-house to the preparation plant and then to the refuse dump. The road segment from the prep plant to the refuse dump will be paved with gravel and used by trucks to haul refuse, and to provide personnel access to the dump area.

Probable impacts. The major impacts will result from construction, maintenance, use and operation of the road. Construction will remove vegetation and change the natural contour of the land. As a consequence, there will be increased erosion from the zone of construction. Haulage operations will also create a dust problem from moving vehicles. Moderate short-term impacts will result during active mine operation.

Impact mitigation. These impacts could be substantially mitigated by following proposed construction criteria for haul roads. In addition, during active mining, haulage roads could be sprayed with water or suitable stabilizing chemicals; however, consideration should be given to possible water pollution problems that could result from these dust control techniques.

1.(b) Road construction under unsuitable conditions.

Nature of activity. The haul road may have a very long slope length up to the refuse dump.

Probable impacts. With such long slopes there is a high probability that erosion on the road surface could be severe. This would add to sediment yields and cause unsafe road conditions.

Impact mitigation. Proper engineering of the road would help to mitigate these impacts.

1.(f) Spoil production and storage.

Nature of activity. Even though the coal seam is assumed to be 6-ft thick, there is a good indication that numerous partings will be encountered. Approximately 5% of run-of-mine coal will be refuse, which will be stored above ground.

Probable impacts. If this spoil is stored above ground there could be a major long-term impact from erosion of the spoil.

Impact mitigation. If the spoil is stored using proper engineering methods and vegetation is established, erosion can be minimized. However, any artificial structure has the potential for structural failure and, hence, major long term impacts from erosion and sedimentation.

1.(i) Highwall and benches.

Nature of activity. A highwall and bench must be cut in order to provide access to the coal seam.

Probable impacts. With creation of a highwall, erosion potential in the area is increased. However, only a small area should be affected as the highwall is cut only for mine access.

Impact mitigation. Backfill area and revegetate following mine closure.

2.(a) Subsidence.

Nature of activity. The extraction of coal by underground methods ultimately leads to the collapse of the overlying strata.

Probable impacts. Subsidence of the overburden will result in slump structures at the earth's surface. As a consequence, land use above the mine will be severely limited. Additionally, the disturbance and breaking of the overlying geologic strata will irreversibly change any aquifers that might be intersected by the collapsed zone.

Impact mitigation. The degree of slumping at the surface can be reduced somewhat by artificial support. Land use, however, will still be restricted.

2.(b) Incomplete removal.

Nature of activity. No more than one-half to two-thirds of the resource will be removed. Moreover, the remaining coal resource will be in coherent blocks, leading to the potential for future removal.

Probable Impact. The fact that a large proportion of coal will remain underground means that there is a possibility that the region could be mined again in the future. This will result in further disturbance of the mine site.

Impact mitigation. None.

2.(c) Backfilling.

Nature of activity. The mined-out areas left by the removal of coal will not be stabilized by backfilling or other mechanical supports.

Probable impacts. Because the mined-out areas will be allowed to cave, differential subsidence will occur at the surface. In addition, no precautions are taken against disruption of aquifers. All of these impacts will be major and long-term.

Impact mitigation. None.

2.(g) Planned subsidence.

Nature of activity. The room and pillar method of mining does not extract the entire coal seam. As a result only portions of the earth's surface undergo subsidence.

Probable impacts. Subsidence will not be uniform, but may take many years to express itself. Thus, utilization of the land on the mine site will be constrained. It is important to note that the land will be limited to only those activities that do not involve urban or agricultural land use. This is not a major problem, nor is it likely to be in the future, since this region will probably remain forested.

Impact mitigation. None.

3.(a) Storage of coal.

Nature of activity. Coal will be stored in a large pile outside of the mine mouth as a ready supply for rail shipment. Coal stored in this fashion is subject to leaching by rain. Discharge of leachate away from the mine site may occur.

Probable impacts. Because this coal has a very high potential for containing acid-producing materials, the water that infiltrates the storage pile and runs off of the coal will probably be acidic. This water may contain high concentrations of iron as well as sulfate. The introduction of these materials into aquatic and terrestrial environments can result in major long-term damage to wildlife and vegetation.

Impact mitigation. A leachate and runoff collection system must be constructed to channel polluted waters to an appropriate water treatment plant. Once the water has been treated to comply with standards it may then be released to the environment.

3.(c) Mine sealing.

Nature of activity. After mine operations cease, 4 mine seals will be constructed.

Probable impacts. Mine seals are notoriously unreliable. With 4 mine seals there will be the possibility of a mine seal failure. In this event, the release of acid materials will pollute water supplies and cause widespread ecological destruction to aquatic and terrestrial life.

Impact mitigation. The mitigation of these potential impacts is based upon proper engineering and construction of mine seals and monitoring of mine seal pressures. There will be sufficient mine water to limit oxidative conditions if the seals hold; however, if a seal should fail, a considerable amount of water will be released. As a consequence, there is a potential for major long-term impacts.

3.(k) Processing.

Nature of activity. The extracted coal will be crushed and refuse will be removed outside of the mine.

Probable impacts. On-site processing of coal increases the potential for acid water drainage away from the mine site (see above).

Impact mitigation. The use of drainage diversions so that acid water may be collected and sent to a water treatment plant will effectively mitigate potential impacts.

3.(l) Extraction below drainage.

Nature of activity. The coal bed to be mined is below drainage for almost 95% of the mine site.

Probable impacts. The fact that mining operations will occur below drainage means that the underground openings will certainly produce acid water. In spite of the operator's best efforts to pump this water out of the mine and neutralize the acid, there is a good possibility that a substantial fraction of this water will infiltrate the surrounding geologic strata, and pollute the groundwater which flows through the mine site.

Impact mitigation. Mine sealing may help to alleviate this problem; however, infiltration of polluted water into the surrounding strata may be unavoidable.

4.(d) Pumping.

Nature of activity. Groundwater must be removed by pumping to allow the operation of equipment.

Probable impacts. The pumping of groundwater can increase the yield of groundwater and reduce the base flow of nearby streams. The resulting loss of surface water could have an adverse effect on wildlife using the disturbed water resources. These effects can have a major impact but should lessen somewhat when the pumping ceases.

Impact mitigation. None.

6.(b) Unpaved roads.

Nature of activity. The one road used for transportation to the mine site and the refuse dump will not be paved, but graveled.

Probable impacts. Haulage occurring on unpaved surfaces may result in excessive amounts of dust. Because the mine site is in an attainment region there will be no violation of existing air quality regulations. However, there is a potential health hazard to employees and persons located near the active haul road and service road.

Impact mitigation. These impacts can be mitigated to a very large extent by applications of water or other appropriate chemicals to the road surface. If the road is not maintained properly, however, wind erosion during and after active mining could be severe. Such impacts would be moderate but long-term.

7.(b) Overburden dumping.

Nature of activity. Refuse removed from the mined coal will be put into a valley fill near the mine site.

Probable impacts. In order to install a valley fill, all vegetation must be removed from the fill site, resulting in the destruction of wildlife habitat. However, as most of the region is heavily forested, reestablishment of wildlife habitat should not be a problem. As a result, there should not be a significant impact in this region.

Impact mitigation. At the end of active mining, reestablish vegetation to produce a usable wildlife habitat.

8.(b) Secondary extraction.

Nature of activity. With low extraction efficiency, there will be a high potential for secondary extraction. As a result, once-reclaimed land may be disturbed again.

Probable impacts. The disturbance of previously reclaimed land could have a serious effect on establishing secondary reclamation. As a consequence, there would be a greater potential for long-term erosion and sediment yields.

Impact mitigation. None.

9.(a) Efficiency.

Nature of activity. All the coal resource will not be removed.

Probable impacts. The amount and use of energy required to remove the resource may not be as efficient as when more coal is removed. Additionally, the resource that is left may be unavailable in the future.

Impact mitigation. None.

In summary, the conceptual environmental assessment indicates that the following major impacts could occur if the mining system were implemented at the site: (1) spoil storage above ground, resulting in sediment loss and acid drainage; (2) subsidence, resulting in groundwater alteration; (3) generation of acid water from the mine; and (4) low extraction efficiency.

F. LAND USE IMPACTS

The Fantasy Mine is located in a part of eastern Kentucky where over 80% of the land is in commercial forest and wood-based industries are important (Karan and Mather, 1977). The Fantasy Mine site itself is covered with natural hardwood deciduous forest; most of the site is privately owned land, but includes a small portion of the Daniel Boone National Forest (ESRI, 1980). The region of the mine site is rural, sparsely populated, and poor (more than one-half of the population of Clay County was below the poverty level in 1977) (Karan and Mather, 1977). Rural land values are extremely low in the eastern mountain region of Kentucky. Natural timber land in this part of Kentucky is worth from $150 to $250/acre without mineral rights, depending on the quality of the timber. Land value including mineral rights can range from $250-$1000/acre, depending on the amount of mineable coal present (Sizemore, 1979, and Reynolds, 1980). The mine site is not valuable for agriculture, not located near any large urban centers, and not noted for scenic or aesthetic values.

The impacts of underground coal mining upon subsequent potential land uses are minimal for this site. Further, no general land-use plan exists for the area; there is no projected use for the land other than its pre-mining use as timber land. Environmental regulations require that after mine closure, the land must be restored to a condition capable of supporting the uses which existed prior to mining. All surface areas disturbed by mining operations must be reclaimed. Reclamation actions include removal of access and haul roads, regrading of disturbed areas to approximate original contour, soil replacement, and revegetation. As pointed out in the previous section, the cost of such general reclamation for the Fantasy Mine site came to $73,500 in 1974 dollars. The reclaimed mine site would be less valuable than the undisturbed site, because the newly planted vegetation would be worth less than mature timber; however, the value of the land should increase with time as the trees mature.

APPENDIX G

EFFECTS OF COAL DUST INHALATION

The Effects of Coal Dust Particles of Less Than 5 Microns on CWP and PMF

The cause of coal worker's pneumoconiosis (CWP) and progressive massive fibrosis (PMF) is inhaled particles of coal dust smaller than 5 microns in diameter (King, 1960; Morgan, 1975; Naoye, 1971; Penman, 1970). Other kinds of dust particles, within the same size range, can also be inhaled. These are mineral particles that are often found within, or adjacent to, the coal seam (Park, 1975). Minerals such as beryllium and zinc can assist in the development of CWP, PMF, other non-malignant respiratory diseases, and lung cancer (Dept. of Labor, 1979). Particles larger than 5 microns are cleared at a level above the alveoli and, though they can irritate the upper bronchial tract, do not contribute to CWP or PMF (Fraser and Pare, 1977; Heitzman, 1973). Although, without coal dust, coal worker's pneumoconiosis would not exist, other factors are also of importance in the development of this disease. The job of the individual is an important consideration. A job requiring a high energy expenditure will require a greater air volume, thus presenting to the alveoli a greater load of coal dust per unit time. There also seems to be an individual variation in the ability of the lung to clear these smaller dust particles from the alveoli, thus yielding a varying susceptibility of the individual to the development of pneumoconiosis (Davies, 1974; Lyons et al., 1972).

A threshold exists for each individual when the alveoli can no longer clear the coal dust by the normal mechanism. (The particles are engulfed through the normal process of digesting cells and the mucous transport system clears these particles from the lung via expectoration or swallowing.) When this threshold level is overcome by an excessive load of coal dust, the engulfed coal dust particles remain in the alveoli. This yields a primary lesion consisting of a mixture of coal dust and distorted cells enmeshed in a fine network. These coal macules, when present in sufficient quantities in a sufficient number of alveoli, yield the well known radiographic abnormalities of coal worker's pneumoconiosis (CWP). When these coal macules become incorporated into the interstitial spaces of the alveoli, oxygen transfer is reduced. This mechanism accounts for the fact that disabled coal miners with CWP frequently have little or no alteration in their ventilatory capacity but demonstrate a marked decrease in oxygen transfer (Guyton, 1976). Cessation of exposure to coal dust inhalation in these individuals usually prevents any further progression of these pathologic changes (Morgan, 1975). However, in approximately 10% of those individuals initially developing CWP, some factors, as yet unknown, come into play causing a progressive destruction of lung tissue. The individuals developing progressive massive fibrosis (PMF) seem to have an autoimmune factor aiding the progression of this disease. Autoimmune factors usually result from the destruction of some of the body tissues which can initiate a reverse immunity reaction. The resulting immune products attack the body's own tissues (Guyton, 1976). Tests for factors such as anti-lung antibodies (Fraser and Pare, 1977; Heitzman; 1973; King, 1960) are positive in a disproportionate number of these coal workers. The lesion of PMF is usually restricted to the posterior segments of the upper lobe and the superior segments of the lower lobe. These lesions are ill-defined bundles of coarse connective tissue frequently obliterating the normal lung architecture. Stuermer and Hatch (1980), suggest that nitrogen aromatics, aromatic hydrocarbons, and oxygenated hydrocarbons found in trace amounts within the coal can also contribute to the mutation of the cellular structure of lung tissue. This could materialize as lung cancer, cellular mutations similar to PMF, or other pulmonary problems.

The Effect of Coal Dust Particles of Greater Than 5 Microns on Bronchitis

The diagnosis of chronic bronchitis is a purely clinical one. The definition is arbitrary in order to have some means of separating the diagnosis of bronchitis from similar diseases such as the common cold. It was set forth by the American Thoracic Society (1962) and the report to the Medical Research Council by the Committee on the Aetiology of Chronic Bronchitis in 1962 and 1965. Expectoration, i.e., productive cough, must occur several days out of the month for at least 3 consecutive months, during 2 successive years. The diagnosis excludes other causes of productive cough such as asthma and pulmonary edema as well as non-productive cough. In many cases these exclusions are difficult to define and some overlap does, on occasion, occur.

Coal dust greater than 5 microns in size has been documented to be an etiologic factor in the development of bronchitis (Rasmussen and Nelson, 1970). Becklabe and coworkers (1959) have demonstrated a direct relationship between impairment of exercise tolerance in miners with normal chest x-rays and the concentration of coal dust in the mines. A study of 8,555 bituminous coal miners (by Kibelstis) demonstrated a statistically significant decrease in the incidence of bronchitis among non-smokers working on the surface compared to non-smoking workers at the coal face. This again confirms the etiologic factor played by coal dust in the development of this disease.

APPENDIX H

HAZARDS ASSOCIATED WITH CONTEMPORARY MINING SYSTEMS

Given a new design, one might argue that the regulations address all the hazards associated with equipment. This is not always true for the following reasons: (1) new designs may introduce new sources of familiar hazards as well as totally new hazards, and (2) even though regulations exist and are enforced, injury data show that workers continue to be seriously hurt at a greater frequency than other relatively similar industries; this suggests that we are not fully aware or in control of the more serious hazards. For these reasons a systematic approach to isolating key hazards and designing new systems accordingly is necessary.

The results of the hazard identification analysis are of immediate use in finding where major problems in the design exist. Identifying the major problem areas tells the evaluator if a new design offers major improvements over existing systems. As many new designs may have hazards similar to existing systems, a good first step is to describe these major hazards as a function of their percentage contribution to serious injuries. The following tables display the relative ranking of the nine most prominent accident classes of the eleven accident types identified.

Table H-1. Breakdown of Major Accident Causes by Fatalities (F) and Nonfatal Disabling (NFD) Injuries (References - MSHA Injury Statistics, 1972-1978)

Accident Causal Category (in order of severity)	Avg % Contribution to Serious Injuries: Fatalities (F)	Avg % Contribution to Serious Injuries: Nonfatal Disabling (NFD)
Roof/Face/Rib falls	47	15
Haulage	23	16
Machinery	14	15
Handling Material	0	32
Subtotal	84	78
Explosion/fire	9	1
Electricity	7	3
Slips/falls	0	8
Handtools	0	7
Suffocation	0	3
Total	100	100

The above list of major accident causes clearly indicates that roof, rib and face falls, haulage, machinery, and handling materials contribute the largest portion to fatalities and disabling injuries. The remaining accidents contribute less than one fourth to the total serious injuries. Each of the nine accident causes are broken down into their respective contributing hazards in the remainder of this section. The percent contributions of detail hazards to the total injuries associated with each accident category are provided where data were available. The discussion after each table further illuminates the causes with emphasis given to the worst hazards.

Table H-2. Major Contributing Hazards to Roof, Face and Rib Falls

Roof/Face/Rib Fall Hazards as Related to Ground Control	% Contribution to Fatalities
Support adequate but is not placed close enough to the face, is improperly placed, or is not placed in sufficient time to prevent strata overstress (i.e., roof fails at time support is being placed or moved).	54 - 60
Support inadequate because workers are careless, inexperienced, or improperly trained on how and where to set support (i.e., failure to recognize bad roof and use adequate support).	23
Unusual geologic conditions met (kettle bottom, fault, rock burst, etc.).	8
Total mine system improperly designed (openings and entries improperly designed or executed).	3
Inadequate roof support plan (insufficient or inaccurate data available on roof/rib conditions).	2
Other hazards which contribute to accidents in lesser degrees:	
- Supervision is inadequate.	
- Support material has quality flaw and fails.	3 - 1.1
- Support is adequate but fails as a result of being struck by a vehicle (mechanical failure or operator failure).	
- Support is adequate but fails as a result of being struck by a tool or material being handled by workers.	
Total	100

Aggravating factors - largest majority of serious injuries occur within 25' of face; low coal does not provide room to escape hazards; most serious injuries involve workers with less than 5 years experience in their task area.

Roof/Face/Rib Fall Hazards as Related to Equipment Protection.	Contribution to fatalities.
Protection built into equipment is adequate but workers are still required to move outside of protection in order to perform certain tasks or to have better visibility of operations.	MSHA experience indicates this is largest source of serious injuries related to equipment (discussion with MSHA Safety and Injury Statistics Branch).

Aggravating factors - low coal does not require canopy protection.

References

"Tables for Falls of Roof, Face, and Rib Fatalities in Underground Bituminous Coal Mines", MSHA, Gadash, C., 1977 to 1978.

"Nonfatal Injuries from Falls of Roof, Face, and Rib (Includes Pressure Bumps or Bursts) in Underground Coal Mines", MSHA, Gadash, C., 1977 and 1978.

"Analysis of Fall of Rib, Roof and Face Accidents in Underground Coal Mines", MSHA, Heim, M., 1978.

"Comparison of Injury Hazards in Different Coal Seam Heights", MSHA, Hudson, S., 1976.

Hazards Related to Roof, Face, and Rib Falls

The hazards which far outweigh all other hazards deal with roof support being adequate (but unable to provide support in the proper place at the right time), or inadequate due to worker inexperience, or carelessness, during the installation process. The term "adequacy" implies that if the support had been placed in the right location and at the proper time, it would have prevented the rock fall. For example, machinery geometry and volume often prevent temporary and permanent support from being placed as close to the face as preferred. Similarly, machinery geometry and volume, and floor conditions, often prevent temporary support being placed in a solid position to hold the roof up. The variable nature of stress release in strata is also a factor since this is not predictable and can occur at the time support is placed. In all of these examples the support is adequate to protect workers from rock falls but fails because it cannot be installed under the conditions for which it was designed. The problem of "inadequate" support is basically due to an insufficient amount of temporary or permanent support being installed. The source of this problem becomes apparent when it is recognized that most of the serious injuries involve workers with less than five years of experience in their task area (see aggravating factors). The complicated nature of strata mechanics demands considerable experience in knowing where, and how many supports should be placed.

Venturing under unsupported roof to install temporary support, or moving away from the protection of a cab to inspect a possible equipment failure under unsupported roof, seem to be inherent in most conventional mining systems. Though it appears that workers accept this as a job-related risk, it seems that considerable effort should be expended to reduce this inherent problem in these task areas.

Table H-3. Major Contributing Hazards to Haulage Injuries

Pinch or Squeeze as Typically Related to Haulage Type Activities	% Contribution to Fatalities and Disabling Injuries
Worker struck by machine or is in between objects which can move as part of normal operations.	50
Worker struck by machine out of control as a result of mechanical malfunction, obstructed vision, machine striking an object, or carelessness.	12
Worker is in between vehicle and roof or rib while operating machine.	6
Collision of vehicles pins operator in cab.	4
Workers are exposed to pinch points on moving or rotating machinery. - Poor design of maintenance and operating areas around machinery. - Bad lighting. - Carelessness.	3 - largest source of injuries is due to performing maintenance on moving machinery.
Workers fall while jumping in or from cars (mostly a surface related hazard).	5
Workers pinched while rerailing haulage vehicles (rail car related).	3
Undefined causes (miscellaneous)	17
Total	100

Aggravating factors - low coal constrains space workers have to escape hazards and also decreases space available to perform maintenance.

References

"Nonfatal Injuries Caused by Haulage Related Accidents in Underground Coal Mines", MSHA, Gadash, C., 1977.

"Analysis of Injuries Involving Conveyors in Metal to Nonmetal Mines", MSHA, Stahl, R., 1976.

"Comparison of Injury Hazards in Difrerent Coal Seam Heights", MSHA, Hudson, S., 1976.

The major contributor to haulage type injuries (pinches and squeezes) is the necessity for miners to work in proximity to the haulage equipment. For example, workers perform clean-up tasks during the loading process in the immediate vicinity of shuttle cars and bridge conveyors. These machines can pinch workers between the machine and rib. Workers couple and uncouple rail cars as part of normal haulage operations and are required to be in-between the cars during the performance of this task. Understanding that many haulage type tasks are performed in a poorly lighted environment where machine operators may not see other workers, further clarifies why these kinds of tasks are extremely hazardous.

Table H-4. Major Contributing Hazards to Machinery Injuries

Machinery	% Contribution to Fatalities and Disabling Injuries
Workers are struck by machinery in the process of tramming and moving machinery in close quarters at the face. (Non-stationary equipment such as roof support and loading type machinery are major contributors to this hazard). Workers are struck or caught by machinery during maintenance clean-up, or support operations. - Insufficient guards. - Bad lighting and obscured vision prevents operators seeing other workers in vicinity. - Non-stationary equipment is difficult to control as a result of forces imparted on machinery during cutting or roof support operations.	59
Other contributing hazards are: - Inexperience/carelessness. - Machinery fails as a result of being overstressed.	41
TOTAL	100

Aggravating factors - low coal restricts the movement of workers close to machinery and prevents workers from being able to escape hazards.

References

"Industrial Engineering Study of Hazards Associated with Underground Coal Mine Production", Vol. I and II, Theodore Barry & Associates, 10 Dec. 1971.

"Study of Fatal Accidents Involving Underground Coal Loading Machines", MSHA, 1978.

"Comparison of Injury Hazards in Different Coal Seam Heights", MSHA, Hudson, S., 1976.

"MSHA Detailed Injury Summary", Report #CM341L2, 1976-1979.

The major problem often encountered with tramming or moving equipment in underground mines is negotiating the narrow entries. Large, slow moving equipment such as longwall systems allow workers time to move out of the way. Other types of lighter, less stationary equipment such as face drills, cutters, loaders or roof bolters move more quickly and are subject to rapid, unstable movement when traveling over an uneven floor. This same unstable movement also occurs when these machines are operating. For example, a face drill or cutter which encounters a very hard parting in the coal may bind and impart a torque large enough to displace the machine sideways. Workers in the vicinity may not be expecting this kind of movement, or may not see the machinery if lighting is poor or their vision obstructed. Inexperience is also a major contributor to these hazards because workers may not have the awareness necessary to always position themselves safely while equipment is operating.

Table H-5. Major Contributing Hazards to Handling Material Injuries

Handling Material Hazards	% Contribution to Disabling Injuries
Worker's physical capabilities are exceeded because objects are too heavy or cumbersome to handle. Tasks which act as the major sources of this hazard are:	
- Handling supplies (such as timber, tools, etc.) and equipment (fans, pumps, conveyors, etc.)	39
- Performing machine maintenance and handling machine components.	15
- Handling power cables or cable reeling.	6
- Handling coal, rock, and other waste.	8
- Coupling, blocking or chocking mine cars.	3
Workers are struck by material as a result of material being stored in an unstable position (mostly roof support materials).	6
Other miscellaneous or not otherwise classified causes.	23
TOTAL	100

Aggravating factors - low coal displays a statistically significant larger number of injuries because workers are constrained by restricted space in which to handle materials, and are usually in awkward positions when handling material.

References

"Comparison of Injury Hazards in Different Coal Seam Heights", MSHA, Hudson, S., 1976.

"MSHA Detailed Injury Summary", Report #CM341L2, 1976-1979.

The problem of workers' physical capabilities being exceeded composes almost three fourths of the disabling injuries in the category of handling material. Since most of these injuries occur during lifting or pulling various materials, it appears that weight, size, and the physical mechanics a worker employs during lifting or pulling, work together to cause injuries. For example, an extremely strong individual can be injured if he attempts to lift a cumbersome component without using the proper technique (i.e., not using the leg muscles in conjunction with the back muscles). It is also understandable that numerous injuries are caused by dropping material since many supplies and machine components are not easily grasped. It is important to note that existing data indicate low coal operations considerably increase the chance of handling material injuries because the restricted space requires awkward physical positions.

Table H-6. Major Contributing Hazards to Explosion/Burn Injuries

Explosion/Burn Hazards	% Contribution to Fatalities and Disabling Injuries
Workers in area of methane release, and - Poor ventilation causes gas and dust buildup in presence of an ignition source. - Inadequate monitoring of gas allows gas buildup and ignition.	38
Workers are in-line of explosive blast.	28
Explosive materials are detonated prematurely due to age, or improper design of detonating device (stray signals).	1
Workers exposed to high pressure releases from hoses on machinery while performing maintenance.	8
Miscellaneous or not otherwise classified causes.	25
TOTAL	100

References

"Analysis of Injuries Associated with Explosives in Coal Mines", MSHA, Tierney, M. P., 1976.

"MSHA Detailed Injury Summary", Report #CM341L2, 1976-1979.

Gas and dust explosions are a considerable problem because of the many ignition sources present in the mining environment. Cutting machines generate sparks when striking rock. Sparks are also generated when pounding spikes into brattice cloth, by the static discharge between closely operating machines, or by shorts on equipment or power cables. Difficulty with controlling spark generation, coupled with monitoring gas at the right time and in all the right locations, contribute to making the explosion hazard unpredictable and difficult to control.

Injuries caused by explosives, suffer the same degree of variability in controlling the causes. Variability in coal and rock strata effect the direction and degree to which fracturing occurs from explosive forces. For example, the explosive shot could be directed out of the charge hole if the surrounding strata is extremely hard. Workers supposedly out of the line of the blast could experience a higher exposure to deflected, flying debris as a result of the denser outburst. Similarly, strata conditions could direct the blast through a weak rib into a working area in another entry. Worker inexperience or carelessness cannot be overlooked as another contributing hazard in both personal exposure and in exposing other workers because of poor communication at the time of detonation.

Table H-7. Major Contributing Hazards to Electrically Related Injuries

Electrical Hazards	% Conbribution to Fatalities and Disabling Injuries
Worker must handle energized electrical components and contacts conductor with tool. (major component is handling /splicing cable, 25.0)	57
Workers handling rail car related electrical components (trolley wire or pole).	20
Miscellaneous or not otherwise classified causes.	23
Total	100

Reference:

"Analysis of Electrical Injuries in the Coal Mining Industry", MSHA, Mason, W. and Seale, E., 1980.

The underground mining environment contributes significantly to the electrical hazard. For example, power cables experience substantial wear from being run-over by machinery, abraded by rough or sharp corners or rock, and corroded by acidic standing ground water. The failure rate of cables varies depending on the degree to which they are affected by these variables. This results in workers not knowing if the cable is shorted when they handle it and subsequently exposing themselves to potential electrical shock. The electrical hazard is further aggravated by poor lighting (such that workers cannot readily see if power switches are activated on machinery, or if cables are abraded), and mine conditions such as standing water which may hide a shorted cable.

Trolley wires are usually exposed to allow good electrical contact with the trolley pole. These bare high voltage lines are an ever present hazard to those walking or crossing the roadways which contain the lines, as well as those who must pass near a line while boarding, riding, or leaving a railcar. Low coal aggravates this hazard considerably.

Table H-8. Major Contributing Hazards to Slip and Fall Injuries

Slips/Fall Hazards	% Contribution to Disabling Injuries
Worker in a position to be caught off balance due to:	
- Loose or slippery footing	31
- Loss of footing due to carelessness	
Worker in a position to be caught off balance or struck by machinery due to:	
- Improper placement/lack of adequate guards on machines or elevated structures to prevent falls.	
- Being careless getting on or off elevated structures.	19
- Guards or equipment failing when stressed in normal working conditions (i.e., defective scaffolding, railings, ladders, etc.).	
- Being careless getting on or off machines.	
- Not observing machinery operating in proximity.	
Worker in a position to be caught off balance or struck as a result of:	
- Tool or materials handled by another worker.	15
- Handling cumbersome material.	
Worker in a position to be caught off balance due to:	
- Tool or object being worked on breaking.	3
- Worker not properly trained or careless, choosing to use tool not suited for task resulting in it breaking or slipping.	
- Sliding material such as loose rock or mud.	1
Worker receives an electrical shock.	1

Table H-8. Major Contributing Hazards to Slip and Fall Injuries (Continuation 1)

Worker in a position to be caught off balance due to:	
- Escaping another hazard such as an explosion or out of control vehicle.	1
Miscellaneous or not otherwise classified causes	29
TOTAL	100

Reference:

"MSHA Detailed Injury Summary", Report #CM341L2, 1976-1979.

The greatest contributors to slip and fall injuries are 1) loss of footing, 2) being caught off balance or struck when operating or working around equipment, and 3) handling material. Though carelessness is sometimes a factor in these injuries, it is equally important to recognize that poor lighting, obstables (such as fallen rock, stored materials, etc.), machinery operating in close quarters, and a wet floor are aggravating factors. For example, a worker handling heavy timber in a wet environment may more easily slip and fall or strike another worker causing him to fall. Sometimes the chain of events leading to an injury is complex. For example, in cases of machine related incidents a worker not injured when bumped by a machine may be knocked off balance and suffer a severe injury as a result of the fall.

Table H-9. Major Contributing Hazards to Handtool Injuries

Handtool Hazards	% Contribution to Non-Disabling Injuries
Tools are handled carelessly causing them to slip or break.	68
Workers are struck by chips of objects being worked on, or broken tools.	18
Workers are struck by tools in the hands of coworkers.	2
Tool is defective and slips or breaks.	2
Tool is dropped on self or dropped from above.	1
Other Miscellaneous or not otherwise classified	9
Total	100

Reference:

"Handtool Injuries in Coal Mines", MSHA, Seale, R., 1979.

The source of the major handtool hazards is often the type of tool chosen for a job and the manner in which the tool is used. For example, an incorrectly sized wrench used to loosen or tighten bolts could very easily slip. Similarly, a crowbar (which is usually applied to bar down loose rock) used as a jack to install a machine component could very easily slip or be overstressed and break. Another problem experienced is the application of too much force on a tool which results in the tool breaking. A secondary consequence of the incorrect use of tools is workers often being struck by the broken tool or chips from the object worked on. It should be noted that this hazard category considers other sources besides carelessness. For example, tools used for breaking rock (such as sledge hammers) often expose workers to injury from fragments even though the tools are being handled properly.

Table H-10. Major Contributing Hazards to Suffocation Injuries

Suffocation Hazards	% Contribution to Fatalities
Workers exposed to refuse slides in normal operations.	26
Inrush of Water.	26
Gas seepage into working area as a result of insufficient or inoperative sensing systems.	14
Surface related fatalities	34
Total	100

Reference:

"Suffocation, Drowning and Asphyxia Fatalities in Coal and Metal/Nonmetal Mining", MSHA, Mason, W., 1979.

In the underground environment it is important to note that mine refuse such as dirt, rock or mud must be stored and periodically transported out of the mine. Refuse stored in overhead bins often becomes clogged in the process of filling rail cars. Workers trying to unclog the bin from underneath or the top are entrapped in the rapid slide of material when the bin is freed of the obstruction.

The presence of water and gas in underground mines is a natural occurrence. Occasionally large pockets of water (such as artesian wells or underground springs) are intersected resulting in a rapid inrush of water which engulfs workers. The gas seepage hazard usually occurs in sections that have already been worked. Workers unaware of the seepage suffocate due to lack of oxygen.

The consistent theme throughout the above discussion is that workers are exposed to hazards because of (1) inexperience or lack of training, (2) working in an extremely dangerous environment, and (3) the nature of the tasks they perform. It appears that equipment design is also strongly related to hazard exposure as a function of task. This was especially clear in injuries associated with rock falls, machinery, and haulage. Examination of equipment and how it is used reveals existing designs require workers to install temporary support under unsupported roof, perform support tasks close to operating machinery, or work in between moving machinery. Also coupled with task exposure is the requirement for workers to handle heavy, cumbersome materials and machine components. Although some of these hazards are somewhat mitigated by regulation, it appears accidents could be substantially reduced via designs that are more sensitive to built-in hazards, with particular attention to the unforgiving mine environment, and worker error.

After identifying familiar hazards and determining which will be major problems in a new design, new hazards are isolated. Because operating data are not available it is necessary to use the system failure and human interaction method to identify new hazards. A brief example to provide the reader with the sort of analysis intended is the rapid variation of temperature to fracture coal. The ways in which this process could go out of control through sudden release of heat or coolant, would represent unique failures compared to existing technology. Identifying the tasks that would bring workers into contact with these failures completes the description of the unique hazards associated with this system.

The system may be envisioned as having a reservoir to contain the gas or liquid used as the injection medium, a temperature inverter to add or remove heat, a pressurizer to build up injection pressure, and an injector which vents the medium into drill holes in the face. Workers monitoring the temperature inverter and pressurizer components could be exposed to severe burns due to high or low temperature release if valves or piping failed. Similarly, injector operators could be exposed to the same hazard if injector nozzles or valves failed. Inhaling or touching the injection medium could also be hazardous. This would not only be a hazard to system operators and helpers in the event of pressure release, but also to workers performing routine maintenance and handling refill containers.

Though there would probably be no information available to determine the degree of exposure of workers to new hazards, the hazard analysis at least indicates that workers could be injured by these hazards. Whether or not workers will be injured is a function of how well the design reduces their exposure. This will be addressed when the remainder of the methodology is developed.

APPENDIX I

SUMMARY OF THE ANALYSIS OF REGIONAL PRICE TARGETS

This report presents the results of an effort to develop an appropriate set of regional coal price targets for the years 1985 and 2000 to guide the development of an advanced coal extraction system. This major research and development project has as its overall objective the eventual development of hardware associated with a new underground coal extraction system which must be commercially attractive to the coal mining industry when developed, and demonstrate a measurable improvement in the safety of the miners using the system hardware. Further, there must be no degradation in miner health, conservation,* or the environment as a result of the adoption of the new technology. Specifically, this effort is designed to assist in the determination of how much more firms would be willing to pay to obtain the new technology in various coal supply regions and reserve blocks, and thus provide an estimate of the potential marketability in various target markets. Also, this report is intended to serve as a guide to the geologic characteristics to which advanced coal extraction technology would be applicable.

Section I** identifies the major generic difficulties in doing long-term forecasting, drawing especially on the results of the 1979 JPL Conference on Coal Models and Their Use in Government Planning (Quirk et al., 1979), which is summarized in Appendix J. The present research effort reflects an attempt to mitigate the impact of such conference-identified forecasting difficulties on the derivation of the target prices and market for an advanced coal extraction system. JPL reviewed existing coal models to determine whether they provided the information necessary to construct such estimates. It quickly became apparent that none of the existing coal forecasting models generated sufficiently precise and comprehensive estimates of the resource base, mining and transportation costs, and coal demand on a regional basis. Since it was determined that such estimates were an absolute necessity as input in the present project, JPL contracted with Energy and Environmental Analysis, Inc. (EEA) to develop a set of the basic data/estimates that could then be used to derive the requisite regional price targets. The results of the EEA effort and JPL's modification and use of the data are the major subjects of this report.

Section II outlines the methodology used to estimate the location and magnitude of coal reserves in the year 2000, and the most salient geologic characteristics of this reserve base. To this end "inferred" reserves were estimated and added to more traditional estimates of "measured" and "indicated" quantities, distributed among 15 supply regions (Table I-1). The results of this procedure are contained in Table I-2. Of the 852.8 billion tons of total reserves estimated, over 78%, 666.7 billion tons are estimated to be underground reserves. Of these underground reserves, over 30% (204.2 billion tons) are estimated to be in the San Juan Region with an equal amount in the regions which collectively comprise Appalachia.

Next, in order to describe these reserves at a level of detail which would facilitate the linkage to a specific mining method, each region's total reserves were subdivided into "reserve blocks" of specific tonnage, sulfur

*Used here, conservation refers to an attempt not to damage coal reserves proximal to mine areas, they may be cost-effective for mining at some future date beyond the target year 2000.

**Section numbers refer to the report from which this summary was extracted: Terasawa, K. L., and Whipple, D. R., "Regional Price Targets Appropriate for Advanced Coal Extraction Systems", Jet Propulsion Laboratory Report No. 80-91 (1980).

Table I-1. Supply Regions and Coal Types

	SUPPLY REGION	RANK	BTU/LB.	% of Total Coal by Sulfur Content		
				Compliance	Low	High
1	OHIO	BITUMINOUS	12,500	---	.03	0.97
2	PENNSYLVANIA MARYLAND NORTHERN W.VA.	BITUMINOUS	13,500	---	.10	0.90
3	SOUTHERN W.VA. EASTERN KENTUCKY VIRGINIA NORTHERN TENNESSE	BITUMINOUS	13,500	.45	.43	0.12
4	SOUTHERN TENNESSE ALABAMA	BITUMINOUS	13,500	.12	.63	0.25
5	WESTERN KENTUCKY INDIANA ILLINOIS	BITUMINOUS	11,000	---	.05	0.95
6	KANSAS MISSOURI NEBRASKA IOWA	BITUMINOUS	11,000	---	---	1.00
7	OKLAHOMA ARKANSAS	BITUMINOUS	13,000	---	.65	0.35
8	TEXAS LOUISIANA ARKANSAS	LIGNITE	7,000	---	---	1.00
9	MONTANA NORTH DAKOTA	LIGNITE	6,000	---	.80	0.20
10	MONTANA	SUBBITUMINOUS	8,500	.30	.70	---
11	WYOMING (PRB)	SUBBITUMINOUS	8,000	.30	.70	---
12	SOUTHERN WYOMING NORTH CENTRAL COLORADO	SUBMITUMINOUS	9,000	.40	.60	---
13	NORTHWEST COLORADO NORTHERN UTAH	BITUMINOUS	12,500	.40	.60	---
14	SOUTHERN UTAH SOUTHERN COLORADO	BITUMINOUS	11,000	.20	.80	---
15	NEW MEXICO ARIZONA	SUBBITUMINOUS	12,000	.40	.60	---

Table I-2. Total* Estimated Reserve Stocks by Region and Mining Method (millions of tons)

Supply Region	Surface	Underground	Total
1	6,396	22,844	29,242
2	6,932	50,819	57,751
3	13,250	44,136	57,386
4	383	2,727	3,110
5	29,148	86,000	115,148
6	6,398	4,150	10,548
7	752	1,902	2,654
8	10,829	-	10,829
9	39,059	-	39,059
10	33,213	69,200	102,413
11	20,664	74,057	94,721
12	5,324	8,622	13,946
13	2,327	64,508	66,835
14	1,596	33,563	35,159
15	9,848	204,151	213,999
TOTAL	186,121	666,689	852,810

*This is the total estimated <u>stock</u> of Measured, Indicated, and Inferred reserves.

content, and major geologic parameter values. The result is an initial base reserve estimate broken into a total of 1164 "reserve blocks" and characterized as one of 180 "mine types". This result is in turn restructured by keying on underground mine reserve blocks and the three geologic parameters chosen as having the largest potential impact on the new technology (seam thickness, block size, and overburden depth). This allows the number of mine types to be aggregated from the original 180 to a more manageable 16. Table I-3 contains a summary of these key mine-type codes which are used in the final regional price target forecasts, while Table I-4 presents the division of the estimated regional underground reserves (in terms of a maximum yearly recovery rate) among these constructed mine types.

Table I-4 shows that the "yyy" mine type contains almost 60% of the estimated underground reserves. A "yyy" reserve/mine has seams greater than 42 in. thick, a block size of greater than 20 million tons, and lies under more than 500 ft of overburden. No other mine-type is estimated to contain more than 18% of estimated underground reserves. Further, if the "yyl" type reserves (reserves with thick seam, large block size and less than 500 ft of overburden) are added to the "yyy" type, a full 75% of the reserves are estimated to lie in these moderately thick, large block size mines.

Section III of this report addresses the set of methodologies used to estimate the costs of traditionally mining these 1164 reserve blocks in 1985 and 2000 in the form of a Minimum Acceptable Supply Price (MASP). The MASP concept (of an average supply price per time period) is detailed and the major assumptions involved indentified and evaluated. Again, underground mines are the focus. Emphasis is placed on identification and explication of the necessary assumptions involved in constructing the required mine cost models. Ideally, the JPL moving baseline model and data would have been available for inclusion by EEA in the work described in this section (EEA, Final Report, March 7, 1980). However, given the fact that the moving baseline was still being developed at that time, EEA's assumptions of fixed productivity increases over the period from 1980 to 2000 may be viewed as proxys for the more detailed output of the moving baseline.

Table I-3. Definition of Mine Types by Characteristics

Characteristics	Parameter values	Mine Type lll	lyl	lly	lyy	yll	yyl	yly	yyy
Seam, in.	42"	X	X	X	X				
(a_2)									
Thickness, in.	42"					X	X	X	X
Block million	6 MM	X		X		X		X	
(a_4)									
Size, million metric tons	20 MM tons	X		X		X		X	
Overburden, ft	0-500'	X	X			X	X		
(a_5)									
Depth, ft	500'			X	X			X	X

Table I-4. Estimated Yearly Rate of Underground Reserves by Regions and Mine Type (MM tons per year)

Region	lll	lyl	lly	lyy	yll	yyl	yly	yyy	Total
1	24.2	40.7	0.3	127.8	33.6	46.5	0.6	184.8	458.5
2	74.4	291.6	0.9	542.4	72.9	169.8	0.3	739.7	1892.0
3	36.7	78.6	0.3	428.1	41.1	117.0	1.0	679.5	1392.3
4	3.4	7.8	0.3	18.2	2.4	5.4	0.6	18.0	56.1
5	3.7	162.9	27.8	251.7	6.6	597.2	26.3	2240.8	3317.0
6	0	73.9	0	654.7	0	21.6	0	180.3	930.5
7	3.2	5.6	9.5	30.1	6.5	0	1.9	18.7	75.5
8	0	0	0	0	0	0	0	0	0
9	0	0	0	0	0	0	0	0	0
10	107.8	254.3	355.8	62.8	0	1309.5	0	2899.0	4989.4
11	0	201.3	0	328.6	0	994.6	0	1622.9	3147.4
12	0	18.0	1.3	39.1	0	115.0	8.7	170.7	352.8
13	0	49.5	0	635.8	0	148.5	0	1873.6	2707.1
14	21.4	20.0	54.9	189.1	50.7	115.4	67.2	816.1	1334.8
15	0	80.8	0	1545.4	0	243.9	0	4636.0	6506.1
TOTALS	274.8	1285.0	451.1	4853.8	213.8	3884.4	106.6	16098.3	27139.4

Section IV describes the derivation of the demand estimates, by region (Table I-5) and coal type, together with the forecast transportation costs between supply and demand regions. The results of these estimates are a set of forecast regional production and market price (MASP) levels by coal and mine type for the years 1985 and 2000. The latter half of Section IV contains the breakdown and discussion of this forecast data according to the 16 underground mine types, and suggests caveats regarding the appropriate use of these data in the JPL project. The major results of this section fall into the following two categories and are located in the Tables referenced below:

(a) Regional production, surface and underground (Table I-6)

(b) Regional and sulfur category MASPs (Table I-7)

(c) Regional production (2000) by mine type (Table I-8)

(d) Remaining regional reserves (2000) by mine type or sulfur category (Table I-9)

The demand estimates, when compared to the aggregate forecasts of other major models, appear reasonable in the sense that there are alternative estimates which lie above and below those given here for 1985 (see Table I-10). Likewise, when the estimates utilized in the present study are broken down into their sectoral components and compared to those generated by Data Resources, Inc. (DRI), the same conclusion can be drawn (see Table I-11). As we note, however, significant increases in the demand for coal over the next 20 years is a possibility with potentially far reaching (positive) ramifications for the commercial attractiveness and appropriate development characteristics for an advanced coal extraction system.

The forecast production levels contained in Table I-6 are of significant importance. First, while total underground production is forecast to be essentially the same in 1985 as it was in 1976, it is forecast to increase dramatically (by almost 160%) by 2000. This foreshadows the potential for a large new market for an advanced underground mining technology. It is important to note that the largest projected increases in underground production are in Central Appalachia and the Uinta Basin.

The forecast marginal MASP's for these production levels and regions are presented in Table I-7 and indicate that those of the underground mines in Appalachia and the Uinta Basin are expected to be $25-30 per ton in 1985 and $26-32 per ton in 2000, and still be competitive in some markets with $7-8 per ton (mine-mouth) surface coal from Montana/North Dakota and Powder River Basin regions. These prices are consistent with the National Coal Model estimates ($23-30 in 1985 depending on supply demand/scenario), are lower than the ICF/CEUM estimates for Central Appalachia for 1990 ($29-38) and 1995 ($31-42) and are in the same general range as those predicted by Bechtel's RESPONS model.

Tables I-8 and I-9 present the breakdown of forecast regional production and remaining reserves by mine type. The primary interest is in moderately thick seam, large block size mines. It should be noted that 214 million tons per year are forecast to be mined from Uinta region with overburden of more than 500 ft (this will be 70% of production from all such mines). On the other hand only 0.2 million tons are forecast to be extracted from Central

Table I-5. Component States of Regions

Region	Demand	Supply
1	New England, New York	Ohio
2	New Jersey, Delaware Maryland	West Virginia (North), Pennsylvania, Maryland
3	Pennsylvania	West Virginia (South), E. Kentucky, Virginia, Tennessee (North)
4	Ohio	Tennessee (South), Alabama
5	Virginia, North Carolina	W. Kentucky, Indiana, Illinois
6	South Carolina, Georgia, Florida	Kansas, Missouri, Nebraska, Iowa
7	Alabama, Mississippi	Oklahoma, Iowa (Bit.)
8	Texas, Louisiana	Texas, Louisiana
9	Tennessee, Kentucky	North Dakota, Montana
10	Kansas, Nebraska, Iowa, Minnesota, Missouri	Montana*
11	Oklahoma, Arkansas	Wyoming*
12	Wisconsin, Indiana	S. Wyoming
13	Montana, Wyoming, North Dakota, South Dakota	Colorado (N.W.), Utah (North)
14	Arizona, Colorado, Utah, New Mexico	Colorado (South), Utah (South)
15	California, Oregon, Washington, Idaho, Nevada	Arizona, New Mexico

*Power River Basin portion only.

Table I-6. Coal Production Forecast by Region
(Millions of Tons per Year)

Region	Year	Mine Type: Deep	Mine Type: Surface	Total	Coal Sulfur Category: Compl.	Coal Sulfur Category: Low	Coal Sulfur Category: High
1. (Ohio)	1976	17	30	47	-*	-	-
	1985	15	26	41	0	6	35
	2000	31	21	52	-	12	39
2. (N. Appalachia)	1976	88	55	143	-	-	-
	1985	58	27	85	-	36	49
	2000	141	20	162	-	105	57
3. (C. Appalachia)	1976	113	77	190	-	-	-
	1985	128	119	247	128	93	26
	2000	256	144	400	174	180	46
4. (S. Appalachia)	1976	10	16	26	-	-	-
	1985	20	43	64	11	38	15
	2000	42	53	95	13	60	22
5. (Illinois Basin)	1976	55	81	136	-	-	-
	1985	4	103	107	-	20	87
	2000	59	108	167	-	79	88
6. (Central Midwest)	1976	0	18	18	-	-	-
	1985	0	91	91	-	-	91
	2000	0	113	113	-	-	113
7. (Oklahoma)	1976	0	4	4	-	-	-
	1985	0	27	27	0	18	9
	2000	0	29	29	-	18	11

*A dash signifies no production of this type

Table I-6. Coal Production Forecast by Region (Millions of Tons per Year) (Continuation 1)

Region	Year	Mine Type: Deep	Mine Type: Surface	Total	Coal Sulfur Category: Compl.	Coal Sulfur Category: Low	Coal Sulfur Category: High
8. (Texas Lignite)	1976	0	14	14	-	0	0
	1985	0	62	62	-	-	62
	2000	0	229	229	-	-	229
9. (MT/ND Lignite)	1976	0	21	21	-	-	-
	1985	0	47	47	-	33	15
	2000	0	103	103	-	62	41
10. (Powder River Basin--Montana)	1976	0	19	19	-	-	-
	1985	0	50	50	50	-	-
	2000	0	180	180	100	80	-
11. (Powder River Basin--Wyoming)	1985	0	138	138	120	18	0
	2000	0	178	178	169	9	0
12. (S. Wyoming)	1976	1	12	13	-	-	-
	1985	0	0	0	0	0	0
	2000	0	0	0	0	0	0
13. (Uinta)	1976	10	14	24	-	-	-
	1985	63	2	66	55	11	-
	2000	215	29	244	110	134	-
14. (4 Corners)	1976	0	5	5	-	-	-
	1985	0	35	35	34	1	-
	2000	0	94	94	66	28	-
15. (San Juan)	1976	1	5	6	-	-	-
	1985	0	35	35	34	1	-
	2000	0	94	94	66	28	-
Total USA	1976	295	385	680	-	-	-
	1985	288	804	1,092	405	301	389
	2000	744	1,335	2,079	639	794	646

Note: Numbers may not add due to rounding.

Table I-7. Forecast Market Mine-Mouth Prices by Year, Region, and Coal Type (1979 Dollars)

Region	1985 Compliance	1985 Low	1985 High	2000 Compliance	2000 Low	2000 High
1	--*	$28.95	$23.20	--	$32.93	$22.90
2	--	31.10	27.22	--	34.67	27.05
3	$29.59	27.81	27.81	$32.24	32.24	31.29
4	34.75	28.46	28.46	39.52	32.10	31.24
5	--	24.68	21.08	--	25.92	21.60
6	--	--	16.21	--	--	16.61
7	--	18.90	18.56	--	19.47	19.47
8	--	--	11.07	--	--	11.98
9	--	5.41	5.41	--	5.62	5.62
10	8.38	--	--	8.81	8.81	--
11	7.39	7.36	--	7.73	7.70	--
12	--	--	--	--	--	--
13	24.23	24.15	--	25.85	25.85	--
14	12.10	11.84	--	12.54	12.30	--
15	15.14	15.14	--	16.22	15.74	--

* A dash signifies no production of this coal type.

Table I-8. Estimated Underground Production (Year 2000)
by Region and Mine Type (MM Tons)

Mine/ Region	lll	lyl	lly	lyy	yll	yyl	yly	yyy	Total
1	1.1	.2	0	0	22.9	.8	.6	4.7	30.3
2	5.0	24.6	0	14.7	51.1	11.2	.3	29.3	136.2
3	36.7	78.6	.3	69.8	41.1	36.8	1.0	.2	264.5
4	2.3	7.2	.3	2.3	2.4	5.4	.6	17.8	38.3
5	0	0	0	0	1.9	18.8	.2	37.6	58.5
6	0	0	0	0	0	0	0	0	--
7	0	0	0	0	0	0	0	0	--
8	0	0	0	0	0	0	0	0	--
9	0	0	0	0	0	0	0	0	--
10	0	0	0	0	0	0	0	0	--
11	0	0	0	0	0	0	0	0	--
12	0	0	0	0	0	0	0	0	--
13	0	0	0	0	0	0	0	214	214
14	0	0	0	0	0	0	0	0	--
15	0	0	0	0	0	0	0	0	--
Totals	45.1	110.6	.6	86.8	119.4	73.0	2.7	303.6	741.8
%	(6%)	(15%)	-	(12%)	(16%)	(10%)	-	(41%)	--

Table I-9. Remaining Reserves (Year 2000) oy Region, Coal, and Mine Type (MM Tons)

Region	lll	lyl	lly	lyy	yll	yyl	yly	yyy	Total
1 (H)	22.9	40.3	0.0	127.8	10.7	45.7	0.0	179.7	427.1
1 (L)	0.2	0.2	0.3	0.0	0.0	0.0	0.0	0.4	1.1
1	23.1	40.5	0.3	127.8	10.7	45.7	0.0	180.1	428.2
2 (H)	66.9	267.0	0.9	465.0	21.8	153.0	0.0	665.9	1640.5
2 (L)	2.5	0.0	0.0	62.7	0.0	5.6	0.0	44.5	115.3
2	69.4	267.0	0.9	527.7	21.8	158.6	0.0	710.4	1755.8
3 (H)	0.0	0.0	0.0	42.9	0.0	9.6	0.0	81.7	134.2
3 (L)	0.0	0.0	0.0	153.6	0.0	34.6	0.0	291.9	480.1
3 (C)	0.0	0.0	0.0	161.8	0.0	36.0	0.0	305.7	503.5
3	0.0	0.0	0.0	358.3	0.0	80.2	0.0	679.3	1117.8
4 (H)	0.3	0.6	0.0	4.2	0.0	0.0	0.0	0.2	5.3
4 (L)	0.8	0.0	0.0	9.9	0.0	0.0	0.0	0.0	10.7
4 (C)	0.0	0.0	0.0	1.8	0.0	0.0	0.0	0.0	1.8
4	1.1	0.6	0.0	15.9	0.0	0.0	0.0	0.2	17.8
5 (H)	3.0	156.3	26.6	221.7	4.2	567.9	25.2	2129.6	3134.5
5 (L)	0.7	6.6	1.2	30.0	0.5	10.5	0.9	73.6	124.0
5	3.7	162.9	27.8	251.7	4.7	578.4	26.1	2203.2	3258.5
6 (H)	0.0	73.9	0.0	654.7	0.0	21.6	0.0	180.3	930.5
6	0.0	73.9	0.0	654.7	0.0	21.6	0.0	180.3	930.5
7 (H)	1.5	2.2	4.1	11.3	2.2	0.0	0.7	7.2	29.2
7 (L)	1.7	3.4	5.4	18.8	4.3	0.0	1.2	11.5	46.3
7	3.2	5.6	9.5	30.1	6.5	0.0	1.9	18.7	75.5
8	--	--	--	--	--	--	--	--	--

C-3

Table I-9. Remaining Reserves (Year 2000) by Region, Coal, and Mine Type (Continuation 1)

Region	lll	lyl	lly	lyy	yll	yyl	yly	yyy	Total
9	--	--	--	--	--	--	--	--	--
10 (L)	75.5	222.0	146.5	0.0	0.0	1106.8	0.0	2148.5	3699.3
10 (C)	32.3	32.3	62.8	62.8	0.0	202.7	0.0	750.7	1290.1
10	107.8	254.3	209.3	62.8	0.0	1309.5	0.0	2899.2	4989.4
11 (L)	0.0	140.9	0.0	230.0	0.0	696.3	0.0	1136.0	2203.2
11 (C)	0.0	60.4	0.0	98.6	0.0	298.3	0.0	496.9	944.2
11	0.0	201.3	0.0	328.6	0.0	994.6	0.0	1622.9	3147.4
12 (L)	0.0	10.8	0.9	23.5	0.0	69.0	0.8	102.9	207.9
12 (C)	0.0	7.2	0.4	15.6	0.0	46.0	7.9	67.8	144.9
12	0.0	18.0	1.3	39.1	0.0	115.0	8.7	170.7	352.8
13 (L)	0.0	29.7	0.0	381.5	0.0	89.1	0.0	993.9	1494.2
13 (C)	0.0	19.8	0.0	254.3	0.0	59.4	0.0	664.7	998.2
13	0.0	49.5	0.0	635.8	0.0	148.5	0.0	1658.6	2492.4
14 (L)	17.3	16.0	44.1	151.0	39.9	92.9	54.1	634.7	1050.0
14 (C)	4.1	4.0	10.8	38.1	10.8	22.5	13.1	181.4	284.8
14	21.4	20.0	54.9	189.1	50.7	115.4	67.2	816.1	1334.8
15 (L)	0.0	48.3	0.0	927.3	0.0	146.3	0.0	2781.8	3903.7
15 (C)	0.0	32.5	0.0	618.1	0.0	97.6	0.0	1854.2	2602.4
15	0.0	80.8	0.0	1545.4	0.0	243.9	0.0	4636.0	6505.1
Totals									
High	94.6	540.3	31.6	1527.6	38.9	697.8	25.9	3244.6	6201.3
Low	98.7	477.9	198.4	1988.3	44.7	2251.1	21.0	8219.7	13299.8
Comp.	36.4	156.2	74.0	1251.1	10.8	762.5	57.0	4311.4	6659.4
Total	229.7	1174.4	304.0	4767.0	94.4	3811.4	103.9	15775.7	26260.3
% Total	1%	4%	2%	18%	--	14%	--	60%	100%

Table I-10. Predicted Coal Production (MM Tons)

Group	1985	2000
EEA	1092	2145
DRI	1081	1910
NCM	1027-1034	--
Bechtel	1127	--

Table I-11. Predicted Sectoral Coal Demand Growth (Quads)

Sector	1985 EEA	1985 DRI	2000 EEA	2000 DRI	Annual Growth EEA	Annual Growth DRI
Electric	17.3	15.7	29.6	26.3	3.4	
Industrial	2.7		9.5		8.7	
		5.8		9.2		
Met.	2.2		2.6		1.1	
Exports	1.3	1.7	1.8	2.3	2.2	
Syn Fuels	-	-	2.5	3.2	-	
TOTAL	24.0	23.6	46.0	41.3	4.4	

Appalachian mines with the same characteristics (note that 80 million tons are forecast for all thick seam, large block size mines regardless of the overburden in Central Appalachia). The characteristics of remaining reserves in 2000 presented in Table I-9 provide additional data relevant to the choice of target markets and technical features desirable in an advanced coal extraction system. The final section of Part IV contains suggestions of the most appropriate ways this data may be used.

Finally, Section V summarizes the qualifications associated with the data and recommendations concerning future modifications and refinements.

APPENDIX J: EXCERPTS FROM "COAL MODELS AND THEIR USE IN GOVERNMENT PLANNING"

Proceedings of a Conference Held in Carmel, California
July 16 and 17, 1979

CONFERENCE SUMMARY, CONCLUSIONS AND RECOMMENDATIONS

INTRODUCTION

The importance of the energy sector in the future of the United States is unquestionable. The debate over the most appropriate source of the nation's future supply of electricity has narrowed to four candidates: coal; nuclear power; solar energy; and thermonuclear fusion. A recently issued report of a four-year, $4.1 million study sponsored by the National Academy of Sciences emphasized the crucial role of the coal sector.* The main panel of 61 energy experts assembled by the NAS for this study has concluded that only coal and nuclear energy are realistic alternatives to oil for the generation of electricity over the next 30 years. The study discounts the likely role of solar energy because of cost considerations and that of fusion on the basis of its unproven technical feasibility. However, the debate over whether to rely on "coal" or "nuclear" rages on.

We are pleased to participate in this debate by presenting the ideas from a group assembled in the summer of 1979 to analyze and critique the contributions of long-term forecasting models to the government's energy planning and policy functions with emphasis on coal supply. The major topics considered in the invited papers and subsequent discussion examined both descriptive validity and normative relevance to policymaking. We were motivated by the concern that the models developed and used to predict and understand the likely supply and demand for coal through the year 2000 be "valid" representations of reality. To this end, the group attempted to identify the weaknesses extant, weigh their probable impact on the derivation of accurate policy prescriptions, and propose ways in which the deficiencies could be corrected.

The aforementioned NAS report crystallizes the motivation for attempting to consider such a diverse range of questions at a single conference. A preliminary analysis of this report's conclusions** indicates that, although the experts assembled agreed that the risks of adopting a coal <u>vs</u> nuclear strategy could be assessed with reasonable technical precision, a clear choice between the two could not be made on that basis alone. Rather, because the experts themselves were "deeply divided over coal and nuclear hazards along philosophical rather than technical grounds", it is left to "the public" to choose between energy sources based on individual values and beliefs about social ethics and <u>not</u> on the advice of technical experts.

We believe the reader will find that the papers reprinted in these proceedings, while certainly not definitive, are thoughtful and suggestive efforts to discuss and outline paths which may lead eventually to satisfactory resolution of these thorny problems. The necessarily abridged versions of the participants' comments and discussions of the papers are integral parts of the

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* <u>Energy in Transition: 1985-2010</u>, National Academy of Sciences, December 1979.

** Sandra Blakeslee, "Energy Panel Finds It Impossible to Advise on Coal <u>vs</u> Nuclear Power," Los Angeles Times News Service, November 13, 1979.

picture portrayed here and are recommended as the linkage between the sometimes seemingly disparate ideas presented in the papers themselves.

The next section presents a succinct summary of each session's papers and discussion and is followed by a brief distillation of the conclusions and recommendations drawn from the efforts.

SUMMARY

PART ONE PAPERS

In order to provide a foundation for a productive discussion by the conference participants over the succeeding two days, the papers of the first session (Part One below) were commissioned as analyses of long-range forecasting models extant in the energy field in general, and in the coal sector in particular.

Milton Holloway describes the recently begun Texas Energy Advisory Council/University of Texas Bureau of Economic Geology (TEAC/BEG) project which attempts to estimate regional coal resources as explicit functions of the major causes of depositional uncertainty. This is contrasted with the present tendency to use only "price", "capacity", and "cost of production" in the derivation of the supply curves for the Coal Supply Model (CSM) of the Mid-term Energy Forecasting System (MEFS). The goal is to provide a dynamic, regional supply function methodology valid for use in all representative coal basins in the United States. The significant potential utility of the TEAC/BEG approach is illustrated with a hypothetical experiment.

Next, Craig Roach examines the subject of the growing dependence of energy modeling on technological process descriptions. His discussion of the advantages and disadvantages of the use of such constructs is concise and to the point: Can we be detailed enough in our modeling efforts for precise estimates at a reasonable cost? Using boiler models to illustrate his point, Roach recommends that models be structured explicitly considering more of the important but usually ignored details affecting the ability to make decisions. He leaves us with only a glimpse of the "disaggregation problem", one of the most crucial and controversial issues encountered in many technology models and a topic of much discussion over the following two days.

The Coal and Electric Utilities Model (ICF/CEUM), widely used for trade-off analyses of production vs. environmental concerns, and the ongoing MIT evaluation study of it is the subject of David Wood's paper. His concern is for the environmental and social consequences of deploying energy technologies. Given the predominance of this dimension, his paper, in conjunction with Saul Gass' in the following session, provides a valuable sketch of the requirements for accurate model validation, verification, and assessment.

A major question in the formulation of our national energy policy is to what extent, and in what form, the various sectors should be regulated. David Montgomery's paper dealing with the requirements and impact of the Powerplant and Industrial Fuel Use Act (FUA) is, therefore, an appropriate capstone to the material of the initial conference session. Montgomery highlights the areas in which the structure, data, and policy relevance of models caused difficulties for the required analysis. This paper demonstrates that careful analysis and effective communication between the analyst and the policy-maker can lead to a clearly superior decision-making process.

PART ONE DISCUSSION

This initial set of papers succeeded in stimulating a wide-ranging discussion of modeling difficulties peculiar to the coal and energy industry, emphasizing the features and elements necessary for effective policy analysis. The two major underlying concerns were the need to provide analytical tools useful to those who have the "policy responsibility" and the meaning and method of assessing a model's "validity".

Milton Holloway asserts that efforts to reduce the uncertainty surrounding the measurement and definition of coal deposits and their geographical distribution will provide better input data for the coal policy models. While intuitively persuasive, the discussion raised a number of questions and caveats. First, it is imperative to agree upon what it means to "reduce uncertainty" as well as the objective of so doing. At present there are so many unresolved subjective factors that this has not been done, which makes it very difficult to conduct the cost-benefit analyses required to ensure that research dollars are well-spent. There was some feeling that, although the exact benefits of reduced uncertainty are not yet clearly definable, the potential is so large that more attempts must be made. At the same time, analysis of existing data may yield important insights without costly new data collection efforts.

Another major issue raised during the initial discussion period was the increasing tendency to rely upon process or technology (as opposed to econometric) models as forecasting methodologies. The absence of any but the most primitive economics in the "process versions" makes it exceedingly difficult to accurately analyze and characterize a future which is significantly different from the present. Process models such as the National Coal Model also tend to ignore the behavioral elements which influence, and are influenced by, the <u>expectations</u> of the actors. The price we pay for being able to specify more detail in the process model may be an increasing lack of confidence that we are capturing the way people actually make decisions.

It was pointed out that all of the coal and energy models used for policy discussions are in reality only partial equilibrium in nature and rarely consider the overall feedback mechanisms. For example MEFS does not consider the changes one would naturally expect in price and output of final commodities resulting from the changes in energy prices. Attempts to make broad policy on the basis of forecasts generated by such models therefore entail significant risk. However, having said this, it is not clear in what form such macro impacts should be "captured" by energy forecasting models. One possibility is the use of sensitivity analyses which identify the implications of various parameter values on the appropriate policy choice. The recent attempt to integrate the Hudson-Jorgensen and Brookhaven Energy System optimization models to provide the linkages endogenously illustrates another approach.

A generic question intimately related to those just discussed is the determination of the appropriate level of model disaggregation or specificity. The problem is to have a model which balances an adequate level of detail with the ability to trace the effects and influences exogenous to the energy sector. Although such a decision should be based on an explicit consideration of the costs and benefits of further levels of disaggregation and expansion of scope, in reality this is seldom the case.

The quantity and quality of data available have proven to be significant constraints in the long-term energy forecasting area. For example, in many instances the level of disaggregation has been forced by the availability of regional or industry specific data, whose quality was questionable at best. Obviously this lack of sufficient quantities of quality data render the possibility of rigorous model validation difficult.

PART TWO PAPERS

During Part Two the emphasis continues to be on determining the "validity" of a model and its utility to policymakers at the federal level.

Robert Major's basic thrust is that, although various degrees of abstraction are required in any modeling effort, more care must be taken to tailor these abstractions to the purpose for which the model is intended. In other words, the use which decision makers have in mind for the model's output must largely determine the structure and content of the model. This is, of course, intimately related to the need to determine the appropriate level of disaggregation, discussed at length in Part One.

The succeeding paper by Charles Mann focuses on the use of the National Coal Model (NCM) in the Department of Interior's renewed coal leasing program. His message is much the same as Major's, that the disaggregation problem is real and significant. The NCM is extremely "unrealistic" in many of its details, which increase in importance as the geographical area of interest diminishes. Thus, the use of the NCM projections to support coal leasing decisions is probably inappropriate.

Phil Childress is less bothered by the disaggregation-type problems of the NCM than with his perception that its use fails to "allow for" sufficient nuclear penetration of the energy market beyond 1995. His paper recognizes and discusses the difficult conceptual questions which must be resolved in order to correctly specify and construct a model that will ameliorate this problem.

Saul Gass provides a fitting capstone to the first half of the conference by reviewing and integrating many of the assessment techniques for large-scale models. His discussion of the "verification" and "validation" components of large-scale model assessment, with emphasis on the federal government's interest and involvement in it, is especially useful.

PART TWO DISCUSSION

Comments on the papers presented in Part Two centered on the question: "Does the federal government belong in the long-term energy forecasting business, and if so, to what extent?" Although the discussion of this question revealed a number of reservations about the governmental role, the collective response was heavily in favor of continuing public sponsorship of methodology development.

The idea that the use of more detailed forecasting models may permit better decisions was also discussed at length. However, more disaggregated (detailed) models are not likely to improve the treatment of institutional rigidities and constraints, a major source of forecasting error. On the other hand, more detailed models may be quite effective in portraying direct and obvious connections between incentives one might introduce and changes in coal production. These partial models, concentrating on what one does know may prove more useful than more comprehensive models for which verification is difficult at best.

The crucial question of clearly identifying the benefits of long-term energy forecasting remains. Is it worth all the resources devoted to do it? Do the results of long-term forecasts lead to and support "correct" policy decisions, or do they convince policymakers to accept an advocate's preconceived position, which may have been derived from an extremely naive (and inexpensive) model? There is no simple answer to these questions. A question more amenable to analysis is whether the model is a "good" one? The answer depends on an "assessment" of the model, and hence on the skill of the "assessor". Although there was no agreement on the specific skills required, there was apparent unanimity in concluding that the assessor should not be an advocate. Expense generally precludes extensive pilot testing of models, and time constraints usually eliminate prospective prediction as a basis for judging a model's validity and utility. However, validity can and should be judged both by the internal consistency of the model and by its ability to capture the essence of past or present situations. Such assessment will be facilitated by the modeler's documentation of assumptions, known "holes" in the model, behavioral bounds, etc. In this way we may be able to judge the extent to which the model is reliable since the role of assessment is not to determine whether the model is the best, but rather whether it is any good at all.

Another point of view holds that we may be expecting "too much" from these forecasting models and from econometric models in general. For many reasons it is unrealistic to expect energy models to be able to replicate reality exactly. It was posited that at least some of the models are being misused if extremely short-run, or extremely disaggregated, results are desired. For example, we should not be too surprised to get less than perfect results from general models which have been manipulated to provide some type of sensitivity analyses, or broken into more disaggregated segments. Since the expressed needs and problem characterization of the client naturally influence the assumptions, descriptive parameters, and structure of a model, doing a good job for that client may result in a model of limited utility to someone else.

This does not rule out a decision to adapt an existing model if this option is more cost effective than "ground-up construction." One would expect to see a series of modifications to such "borrowed" models as their inadequacies become more apparent.

Although a good modeler may tailor the product to the expressed needs of the client (and hence increase the probability that the assumptions and structure are appropriate), there are also clients who can't (or won't) define what they want. This leaves a great deal of latitude to the modeler. The result will often be a model of minimal utility because of the resulting ambiguities or erroneous assumptions for the specific problem to be solved. This is particularly true in the way institutional considerations are handled.

A final point should be made regarding an issue that was indirectly raised at various times during the discussion. There seemed to be an inordinately strong preference for models whose validity can be established. Because of this bias "process" models were generally said to be "scientifically based" while those incorporating behavioral components were suspect from a "scientific" point of view. In addition, there seemed to be inadequate recognition of the indispensable role which "art" plays in the construction and use of forecasting models. The sooner these perceptions are corrected, the sooner will the quality of the models and their usefulness increase.

PART THREE PAPERS

There are difficulties associated with portraying the interaction of supply and demand in the energy sector and the influence of the larger environment within which these forces are embedded. These difficulties are compounded by additional problems inherent in attempting to model processes and predict events twenty years in the future. The five papers presented in this section attempt to clarify the sources of error in the use of long-term forecasting models to support government policymaking.

Toby Page begins by considering whether the market rate of interest is an appropriate choice for discounting the future social costs and benefits of contemporary government policy. By examining conditions under which a Pareto Optimal intergenerational resource allocation might be achieved and utilizing an axiomatic social choice theory, Page develops a decision rule which satisfies the notion of "Kantian fairness" and has direct implications for choosing among present energy options.

This concern for the well-being of future generations is further pursued in the paper by William Schulze and David Brookshire. The authors point out the inadequacy of standard cost-benefit techniques in dealing with the potentially catastrophic social and environmental costs of various energy options. As an alternative approach they suggest an analysis of the logical implications of adopting specific ethical principles.

Mohamed El Hodiri proposes an innovative approach to model evaluation based upon recent research involving the maximal root properties of positive matrices, and outlines the significant potential gains such a method may bring to the long term forecasting arena. Analysis of the general forecasting problem under a reasonable set of conditions underlines the need for periodic revision of forecasts, a result consistent with intuitive notions of good planning practice.

In an exploratory paper, Louis Wilde focuses on the potential uses of the theory of optimal planning in the development of improved long term forecasting procedures. Recognizing that important developments in optimal control theory, dynamic programming, and other optimization techniques have occurred over the past two decades, Wilde identifies the major obstacles to their more extensive use in actual forecasting practice: difficulties in determining the appropriate objective function to be used; meeting the substantial computational requirements associated with such models; and collecting the necessary data within the allowable budget. Wilde suggests ways to mitigate the conceptual problems associated with specification of the objective functions and to reduce both the computational time and the data requirements for the application of such techniques to forecasting problems.

The final paper in this section addresses the problem of identifying a method for incorporating the expectations of decision makers in long-term energy forecasting models. Edward Cazalet and his research group consider both the theoretical and empirical issues associated with the formulation of expectations in intertemporal models used in the production, interpretation and implementation of long-term energy forecasts.

The group discussion of these papers naturally reflected the shift in focus from energy model-specific questions to more general long-term forecasting problems.

PART THREE DISCUSSION

The discussion naturally fell into two distinct but related segments: first, Page and Schulze-Brookshire's attempts to incorporate the well-being of future members of society into an analysis of current policy options; and second, the accuracy and efficacy of long-run forecasts, as examined by El Hodiri, Wilde, and Cazalet.

The question of whether the social discount rate should be subject to adjustment for reasons independent of either social choice or ethics, or alternatively as a function of its effect on the efficiency of irreversible choices under uncertainty prompted a lengthy and spirited discussion. It was conjectured that a firmer grounding of the analysis in a "full fledged general equilibrium contingent claims market" would help. Although this is clearly an area which could use an increasing amount of scrutiny and research, the wholesale difficulties in such attempts are revealed by the present state-of-the-art in static social choice theory. A more heuristic posture was suggested as an alternative. Specifically, one might look at the ebb and flow of forces presently (and increasingly) at work in a quasi-market with non-traditional prices. A dictatorship of the present does not necessarily mean that current actions are void of concern for the future. For example, contemporary environmental concerns have generated competing causes which agitate and lobby for changes in policy and regulations which, in turn, constrain present actions and in a real sense help to shape the initial conditions confronting tomorrow's society. The role of the government may well be best limited to ensuring that the barriers to entry in this market are few (e.g., providing tax exemptions), to monitoring its "outcomes", and to incorporating faithfully these outcomes in the set of effective constraints on our resouce use and development.

The discussion of the three remaining papers centered on the ubiquitous uncertainty issue and attempts to handle it more effectively in long-term energy models. El Hodiri chose to tackle the uncertainty problem in a novel way by attempting to apply the relatively recent Theory of Evidence to the long-term forecasting problem. Because of the embryonic nature of this theory and its complexity, there were many who questioned whether it is really appropriate to attempt such an application. However, it is easy to lose sight of the potential utility of such a method by focusing on the internal intricacies. El Hodiri chose to emphasize that, if after additional research, it proved to be an appropriate tool, it would be one additional method which could be employed by policymakers to choose among the usually disparate set of energy forecasts generated by the various models in use.

It was asserted that although Louis Wilde's paper considers the role of uncertainty in optimization models while Cazalet and his colleagues emphasize the impact of uncertainty through the formation of changed expectations, both are related to the need to switch from the use of static to dynamic models.

Although the incorporation of expectations formation is a step in the right direction, there is no well-developed technique for representing this process in an analytical fashion. One possibility is to identify general heuristics or rules-of-thumb which are both theoretically sound and empirically valid.

PART FOUR PAPERS

The final section of these proceedings considers the problems encountered in constructing models which are affected by or which provide inputs to comprehensive long-term forecasting models.

Ronald Cummings reports on the research in which he and his colleagues analyze additions to local municipal infrastructure as a result of sudden, massive development of new energy industry nearby. The possibility of a significant decrease in the quality of life may effectively forestall the pursuit of one or more energy development options in specified regions or subregions. Alternatively, the unforeseen negative impacts of municipal requirements development may greatly alter the accuracy of model-based forecasts. To obtain an initial estimate of the importance of this problem, one must develop measures of the social benefits attributable to municipal infrastructure. Subsequently, one may search for the "optimal" investment in infrastructure to counteract the various problems likely to accompany the "boomtown" phenomenon.

Stuart Burness examined the relationship between Loss-of-Load-Probability (LOLP), and additions to generating capacity, when the objective is generating cost minimization. Changes in LOLP levels are one policy measure that can be implemented with relative ease and yet can have profound effects on the level of generating capacity through their effect on the mix of plants of various size (and hence fuel efficiency).

The final paper by James Quirk and Katsuaki Terasawa contrasts the levels of research and development (R&D) activities which are likely to be undertaken in two cases: First, where there is a consensus of belief about the future prospects of an industry; and second, where the beliefs are divergent. The relevance of this topic to energy forecasting is immediate. As the role of government in the energy industry expands, government activities must be coordinated and rationalized. A major role of long-term energy forecasting models is to facilitate performance of this task.

PART FOUR DISCUSSION

The discussion of the Cummings paper was dichotomous, with the difference in emphasis seemingly one of scope. Those participants who saw the problems of "boomtowns" in terms of the logistics of providing for the work force looked for ways in which the direct impact could be minimized. Others were concerned with a prior and much broader question--how can one assess the aggregate direct and indirect costs of an energy development? This group pointed out that modeling efforts by Cummings and those of similar interests do not deal with the impact on the environment, part of which may well be irreversible. Moreover, one must recognize that the "impact" on a community implies more than an investment requirement for municipal infrastructure. Community needs of a more service-intensive nature have historically received inadequate attention--counseling for alcoholism and drug abuse, family relations, etc. In sum, the second group argued for models of regional extent, addressing a variety of problems in addition to the demand for local services.

The discussion of Stuart Burness' paper emphasized the need to model the policy process whereby the target LOLP is determined. Some felt that the LOLP can only represent a small portion of the uncertainty associated with loss-of-load relevant to actual investment decisions. Others commented on Burness' tentative result that existing overcapacity is due in part to the explicit consideration of LOLP. One discussant suggested that observed overcapacity was rather a function of forecast, but unrealized, demand growth, and that the quality of existing capacity may be increased by this "questionable" investment. Finally, it was pointed out that the potential benefits of the power pools may go unrealized because of regulatory constraints on the industry. Two examples give a feeling for the kinds of deterrents to pooling: transfer prices for energy transfers are often prohibitive, and agreements on interstate power sales typically involve two regulatory authorities with possibly different philosophies on rate fixing.

Most of the comments on Quirk's and Terasawa's paper focused on issues which were not covered in the paper. In particular, as the authors readily agreed, the major question is how much R&D is optimal for the coal industry, and, indeed, for the energy industry in general? Although very little can be said about the optimality of competitive markets under uncertainty, there is a substantial amount of casual empirical evidence indicating that industries which have experienced the greatest rate of cost reduction through innovation are dominated by "new" firms and are characterized by large changes in market shares. In the energy sphere, one would like to know whether a relatively free market will automatically stimulate an optimal amount of R&D. If "diversity of opinion" fosters increased R&D expenditures as the Quirk-Terasawa exploratory model suggests, and if increased R&D leads to cost reducing innovations, then long-term forecasting may be counterproductive. In this case, the government would not want to engage in any activity which would reduce this atmosphere of "productive uncertainty". However, if such R&D expenditures are likely to result in needless duplication of effort and/or to dissipate resources required to reach a critical scale of research, the present "forecasting strategy" may be the best.

CONCLUSIONS AND RECOMMENDATIONS

Although the conclusions and recommendations to be drawn from the papers themselves and the accompanying discussions cover a diverse spectrum of issues, they have been aggregated into two major categories: (1) the role of forecasting models in energy policy making, and (2) the use of models to forecast future coal prices. We hope that these ideas will stimulate a more in-depth examination of the issues.

A. Forecasting Models as a Policy Tool

1. Formal models serve a very useful function in forcing the modeler to explicitly state his assumptions. In addition, once the model is chosen there is a consistent framework for evaluating alternative policy options. This formality facilitates communication among groups with opposing views by helping them to focus on their specific areas of disagreement.

2. Although the initial investment can be substantial, some models provide an economic way to explore the implications of many different sets of assumptions quickly.

3. Especially with long-term forecasts, there is a need to consider the implications of a broad spectrum of scenarios. Using a single baseline set of assumptions for policymaking is both dangerous and delusory.

4. Because of the explosive growth of forecast variance with time, with its concommitant impact on the ability of policymakers to discriminate between options, it is appropriate to think in terms of constructing a sequential strategy. One must identify the time at which current decisions should be evaluated, which in turn determines the planning horizon for the first stage of the sequential decision process. Thus, it may be totally inappropriate and unnecessary to prepare a forecast for 2000 or 2020 if the question were properly structured.

5. Uncertainty pervades most areas of our lives, including energy supply and demand. It is likely that efforts to "reduce" uncertainty through detailed description of physical processes and resources may largely serve to make more salient the remaining uncertainty attributable to factors over which we have little control--future human behavior. The consensus of the group seemed to be that behavioral factors will tend to dominate the energy sector and that resources are better spent trying to understand behavioral sources of uncertainty than uncertainty due to incomplete description of energy production and utilization technology.

6. On the appropriate level of disaggregation, it is apparent that we can never hope to construct one large-scale energy system model which will provide all things to all people. However, although there always seem to be very tight deadlines for policymaking, perhaps if we were more realistic and recognized that most policy processes continue for a couple of years, we would go ahead and do the modeling the right way at the outset.

7. The concensus of the group seemed to be that neither process nor econometric models alone could provide accurate answers to policy questions. Rather, there was significant support for more hybrid models. The problem is that we seem to be headed toward the replacement of econometric demand models

with process models. The real question, then, is once the full-scale model becomes process structured, how will we derive insights from the model on consumer and producer behavior? The answer seems to lie in the objective function and the form of the constraints, into which we need more systematic research. This is in contrast to further research into the process detail, which has dominated research interest recently. We need more effort in such areas as economic analysis of energy investment behavior, for example, and not into finer and finer disaggregation of our technical information on reserves. The existence of a hybrid model of manageable size would facilitate consideration of a much more complete set of policy options without the inherent drawbacks of previously tried comprehensive models. These drawbacks have tended to force policy analysts to rely on the output of much more specific, partial equilibrium models. The need to predict the effects of changes in regulatory strategies is an example. In view of the substantial impact which regulatory policy can have, it seems clear that additional research and resources should be devoted to Hudson-Jorgenson/MEFS-type projects.

8. There are significant limitations on the utility of existing models, and on the construction of more appropriate ones, imposed by the lack of data of sufficient quality. More conscious and consistent effort should be applied to the generation, collection, and management of a data base, whose precision is appropriate to the statement of the model requirements rather than vice versa, as in the traditional mode of practice.

9. Although the problem of intergenerational responsibilities is an extremely difficult one, a consensus of sorts did emerge on this thorny ethical question. First, it is clear that neither the market rate of interest nor a social discount rate in any usual sense can carry the weight of incorporating the utility of future generations into present decision processes (models). We must develop better ways to incorporate in the models both the present generation's concern for their future colleagues and the rights of those future members of society to be considered in present-day decisions which significantly impact their choices and environment. As was pointed out repeatedly, discounting (direct or indirect) costs or (dis)utilities which will accrue to members of generations even 30 years hence, permits the domination of short-term interests in resource program choice. Almost unanimous agreement existed that there are few ethical systems under which such domination could be considered equitable.

B. Use of Models to Forecast Coal Prices

The application of the principles enumerated in the previous section to the issue of coal price forecasting yields the following points:

1. Given that it will take from fifteen to twenty years for new coal technology that is under development today to be introduced into the marketplace, forecasts of the 1995-2000 time frame are pertinent to issues concerning new coal technology.

2. Over a period as long as 20 years there will be numerous shocks to the energy marketplace. These shocks may arise from foreign sources (e.g., instability in the Middle East) or from domestic changes in environmental, health and safety, or utility regulation. Thus, scenarios which capture a spectrum of possible assumptions are necessary for any meaningful analysis.

3. Models can be useful in coal price forecasting by establishing a consistent framework and explicitly describing the scenarios for transportation, environmental, and health and safety regulation as well as marginal extraction costs. Such models should combine features of both process and econometric approaches in order to make the models as robust as possible.

4. The spectrum of scenarios which drive coal price forecasts imply a range of coal prices which are bounded on the low side by a highly elastic coal supply which permits forecasts far into the future. The upper bound is the world price of oil which might occur if non-competitive elements dominate the production and transportation of coal. This wide range of possible coal prices implies that it is desirable to maintain flexibility in R&D programs via a sequential decision making process. When some of the current uncertainties are resolved the set of R&D options can be narrowed for further development.

Finally, as an editorial addendum, we should note that there exists a fundamental, and logically prior, question which is not sufficiently addressed by the papers and discussion presented in this volume; namely the usefulness of attempting very long-term forecasts. The "rational expectations" or "efficient markets" theory would imply that the current market price of energy already reflects all of the information that is currently possessed by those participating in the market with respect to future possibilities. Moreover, there is evidence that private firms did not engage in long-term forecasts of demand, supply, and prices of energy until government began to engage in this activity. This suggests that private firms did not find long-term forecasts worthwhile in terms of expected benefits.

In reply, it should be noted that there are contradictions in the efficient market hypothesis: if the market price is an efficient statistic, it pays no one to gather information; but, the market price can reflect no information about the future, because no information has been collected. Clearly, at best only a modified version of the efficient market hypothesis can hold. In addition, long-term forecasts might well be designed to elicit information that market prices, even in efficient markets, cannot provide. For example: How much oil should be imported, and from which countries; where will the costs and benefits of price rises be concentrated; and a host of other socio-political considerations.

However, even if one agrees that a strong case can be made for the government to support and fund such long-term forecasting efforts, a worrisome question remains: How much weight should be put on forecasts so far in the future and so subject to changes in the behavior of nation-states? Although these are questions clearly beyond the scope of the present work, it is important that they be noted and that future research efforts are undertaken with them in mind.

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