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LONGWALL FACE LENGTH SELECTION

by

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ABSTRACT

A review of current longwall equipment characteristics was performed to serve as a basis for an analysis of longwall face length constraints. The Armoured Flexible Conveyor (AFC) is identified as an upper constraint on longwall face length; a lower constraint on longwall face length is found to be geotechnical in nature. Subsequently, the development of a retreating longwall model for flat-lying coal seams greater than 60 inches thick is discussed. The effects of seam height, number of development entries, powered support design, panel development productivity, panel entry width, panel length, seam depth and gob overhang length on the optimum longwall face length and the optimum project Net Present Value (NPV) are examined. A test case is presented to demonstrate the NPV based face length design optimization program.

Finally, it is concluded that any change in panel geometry which causes a greater tonnage of development coal to be produced or lengthens the time to realize positive cash flows generated from a longwall operation will cause the optimum face length to increase and the optimum project NPV to decrease.

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CHAPTER I

INTRODUCTION

Longwall coal mining is becoming ever more popular in the United States as underground coal producers struggle to increase productivity, enabling them to compete with surface mined coal and other energy sources. Longwalls, perhaps the safest underground coal mining method in use, are also the most capital intensive. Investments as high as $15 million dollars may be required to equip a new production face.

Prior to installation of a new longwall, intensive engineering, planning and mine development must be performed to insure attainment of economic goals. Equipment selection must be carefully undertaken with respect to compatibility and ultimately suitability for the site specific geologic and mining environment.

To date, equipment selection and particularly face length selection have been based largely on personal preference and rule-of-thumb judgments of corporate managers. Poor judgment on the part of management with respect to optimum face length may cause a longwall project to fall several million dollars short of its maximum or anticipated return on investment.

Because the cost of mistakes in longwall face length selection is high, it is necessary to have some rational engineering and economic basis for face length selection. To jointly analyze the engineering and economic ramifications of longwall design, it is necessary to allow semi-fixed parameters such as seam thickness and geologic setting to interact with variables such as support type and panel development criteria. After numerous evaluations are made over a range of face length values, the economically and operationally optimum design may be identified. Use of computer simulation is the most efficient method by which to evaluate the criteria affecting face length and ultimately select the final optimum longwall face length. However, it should be noted that the simulator is only intended to expedite the calculation process. It is not an overall longwall design simulator. Further, its development was necessitated by the feet that no other longwall design "simulator" satisfied the objectives of this study.

In this thesis it is proposed that an economic optimum face length does exist on a site specific basis, and that it can be identified via the economic selection criteria known as Net Present Value (NPV). To simplify the optimization, the important design variables were opened for user input while reasonable values were assumed for all other possible variables. The important design variables idenfified are:

- o Overburden height
- o Thickness of gob bridging bed
- o Gob overhang length
- o Shield support style
- o Longwall face length
- o Seam extraction height
- o Longwall panel length
- o Number of development entries
- o Continuous miner productivity
- o Development entry pillar length
- o Development entry pillar width
- o Development entry width

The optimum longwall face length is defined in this thesis as that length at which the NPV for the project, over 15 million tons or 30 years, is maximized.

CHAPTER II

LITERATURE REVIEW

Within recent years several publications have addressed, either directly or indirectly, optimization and/or simulation of longwall coal mining systems. For instance, Curth (1979) developed an equation to determine optimum longwall face length based on selected costs and productivities involved in longwall mining systems. The exact equation is:

$$L_f \max = \left(\left(1+\frac{\frac{P_f}{P_g}(C_{wg}+C_{hg})-(C_{wf}+C_{hfc})}{C_{wg}\ C_{hfl}}\right)^{\frac{1}{2}}-1\right)C_{wg}$$

Where:

P_f = longwall productivity, tons/day

P_f = development productivity, tons/day/section

L_f = face length, ft

C_{wf} = longwall labor costs, \$/day

C_{wc} = development labor costs, \$/day

C_{hfc} = constant longwall hardware costs, \$/day

C_{hfl} = cost of hardware varying with face length, \$/day

C_{hg} = cost of development hardware, \$/day/section

This calculation procedure results in face lengths from 450 ft to 600 ft as the optimal range for U.S. coal mining conditions. These variations are solely dependant on seam height and therefore ignore many variables which allegedly influence longwall face lengths.

Although some of the activities and resultant consequences involved in longwall mining are considered, there appear to be many shortcomings in this technique. Some of these shortcomings are:

1) No consideration is given to the cost and timing of longwall moves.
2) No consideration is given to taxation ramifications.
3) It is assumed that longwall productivity is not a function of face length.
4) The impacts of timing and production scheduling are not considered.
5) The impact of variations in geologic and geometric parameters cannot be directly evaluated.

Each of the aforementioned shortcomings are believed to have substantial impacts on the overall economics of a longwall mining system. Therefore, the omission of these factors negates the usefulness of this technique as a method of selecting longwall face length.

Breeds, et al., (1979) developed a computer simulator to assist in the optimization of longwall face length. The optimization can be based on any one of three objectives. To quote Breeds, et al, they are:

1) Minimize average unit production cost
2) Minimize present value of average unit cost
3) Maximize internal rate of return (IRR) of the project

The results suggest that the optimum longwall face length, presumably for any condition, varies from 270 ft to 550 ft, depending only on the evaluation criteria selected. The latter face length results from minimization of average unit production cost and the former from maximization of the project IRR. This study also is believed to have numerous shortcomings, not the least of which are:

1) Equipment and support costs are assumed constant from mine-to-mine.
2) Development costs are disregarded.
3) The optimization is based solely on equipment costs.
4) No comment is included to acknowledge the effects of the cost and timing of longwall moves.
5) There is apparently no consideration of taxation, production scheduling or event timing.
6) The impact of variations in geologic and geometric

parameters cannot be directly evaluated.

As these shortcomings are believed to have significant impacts on the economics of a longwall project, this simulator is not considered useful to the current investigation. Further, it is clear that its utility in face length selection is very limited.

The other contribution in this area was a longwall simulation model by Manula (1978) at the Pennsylvania State University. This simulator was intended to evaluate the engineering selection and design of longwall mining equipment. Typically, it may operate over a span of 10 shifts. In no way does the simulator consider overall longwall panel mining or moving times. It has few if any project economic computations and could not be used to evaluate a multi-panel longwall project. As such, it is not useful to the investigation undertaken in this study.

In general, previous efforts with model development to select an optimum longwall face length have lacked sufficient detail to be of any use to this investigation or to any mine operator designing a longwall project. Therefore, it was necessary to develop a longwall mining schedule calculator detailed enough to assist in the selection of the optimum longwall face length on a site-specific basis.

CHAPTER III

STATE-OF-THE-ART IN LONGWALL COAL MINING EQUIPMENT

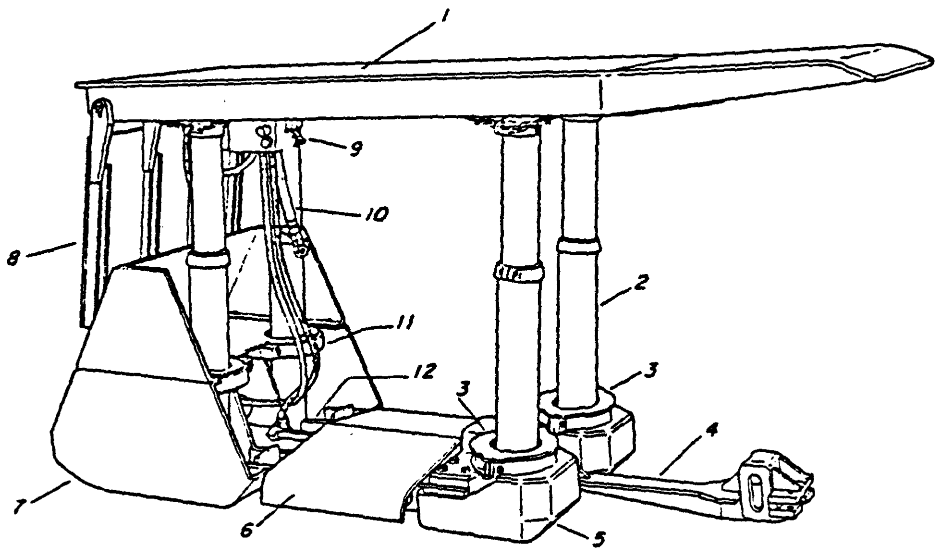
Powered Supports

The powered face supports on a longwall are perhaps the most important units on the face. Critical as they are to safety and production, their cost is the greatest of any component on the face, and considerable attention must be given to their selection. Sizing of these supports is fairly well defined. A thorough explanation of one method of support load calculation appears in Appendix A.

There are four types of face supports typically found in modern use. They are chocks, two-leg and four-leg shields, and chock-shields. Though all four of these support types are subsequently discussed, only the latter two are considered in the following state-of-the-art longwall model.

The chock support (Figure 1) has been used extensively in Great Britain and to a limited degree in the United States. Chocks may be useful in some instances, particularly because of their low cost. Chocks are only 20% - 30% as expensive as shield supports of similar capacity.
The advantages that chocks offer are:

1) Potentially larger cross-section for ventilation purposes.
2) Potentially large support capacity.



1 Full-width rigid canopy
2 Leg
3 Self-centering leg housing
4 Double-acting ram
5 Front base structures
6 Walkway floor cover
7 Rear base structure
8 Antiflushing shield
9 Hydraulic control valve
10 Hydraulic hoses
11 Stabilizer
12 Frame bars

Components of chock-type powered support

Figure 1

(After Barry, 1969)

The disadvantages of chocks are:

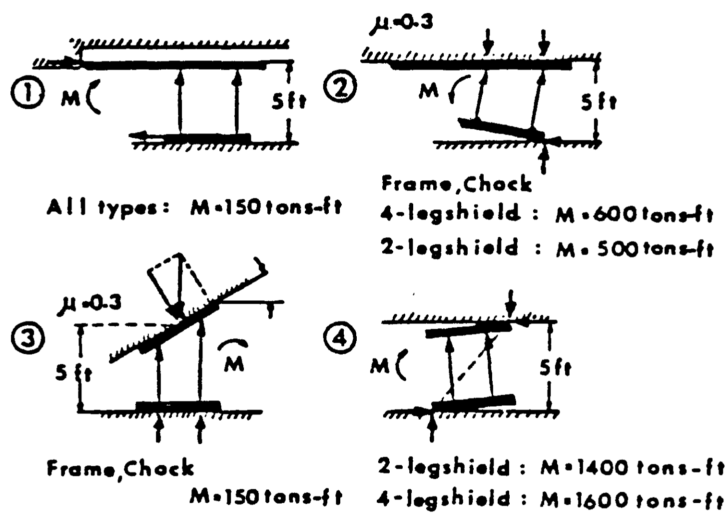
1) Gob flushing if the roof does not break in large blocks.
2) The four joint design is inherently unstable.
3) Improved leg mountings are necessary for stability.
4) Canopies are 2 - 4 ft. longer than shields.
5) Roof degradation due to repeated support setting, known as "tramping", is pronounced.
6) Chocks are subject to damage by tangential forces.

Perhaps the two most severe problems with the chock support are gob flushing and leg bending.

Gob flushing causes severe maintenance problems because of hydraulic system damage as well as increased chock clean-up. Gob flushing may also be a safety hazard to face personnel.

Leg bending in chocks arises from the chock's inability to withstand tangential forces. Tangential forces also damage leg positioning and attachment elements. These forces are produced by the support under the following conditions: (Figure 2)

1) The support encounters a roof step.
2) The support retains roof contact (tip of skid lifts).
3) The support canopy slides laterally during set.
4) The support encounters oblique loading.



Sources of Tangential Loading

Figure 2

(After Irresberger, 1977)

Chock-type supports probably find their greatest application in thin seams where the travel way is maximized.

Shield-type supports are a fairly recent advance in longwall technology. A product of the 1960's, shields were first made in W. Germany under Russian license and have since grown to become state-of-the-art technology in longwall mining. Unique to the shield support is a lemniscate gob shield. This lemniscate linkage allows the canopy tip to describe a vertical locus throughout its height range and to translate most of the load on the gob shield to the base of the support. The fact that the canopy travels a straight path precludes variation in the unsupported area between the face and the canopy tip with varying heights, and thus removes the possibility of the canopy tip interfering with the coal winning machine at varying heights.

The gob shield, along with side shields, provides complete gob sealing as well as total coverage of the supported roof area.

Shield supports are very resistant to tangential forces by virtue of their three-bar construction. This resistance to tangential forces allows the shields to be advanced while still maintaining contact with the roof. Advancing under contact enjoys the following advantages:

1) Minimizes degradation of the roof by "tramping".

2) The canopy is kept clean, prohibiting accumulation of a cushioning pile of debris.
3) Small broken strata is held in place.
4) The cycle time for setting in good floors is reduced.

Shields offer the following advantages over chocks:

1) Shields provide stable support, while creating a forward thrust during yielding which maintains roof strata integrity.
2) The shorter canopy means there is less weight to support than with other types of supports.
3) The robust design is simple and easy to operate and maintain.
4) Higher reliability, thus higher production than with other supports.
5) Provides improved safety for face personnel.
6) Provides superior ground control.
7) Reduces the face-to-gob line distance, thus being removed from the area of maximum convergence.
8) Excellent resistance to high lateral ground movement.

Shield-type supports are best applicable in medium-to-thick seams with any roof characteristic, multi-seam mining with a de-stressed seam, faces where spalling is a problem, or faces where the ventilation must be maintained at high levels along the face.

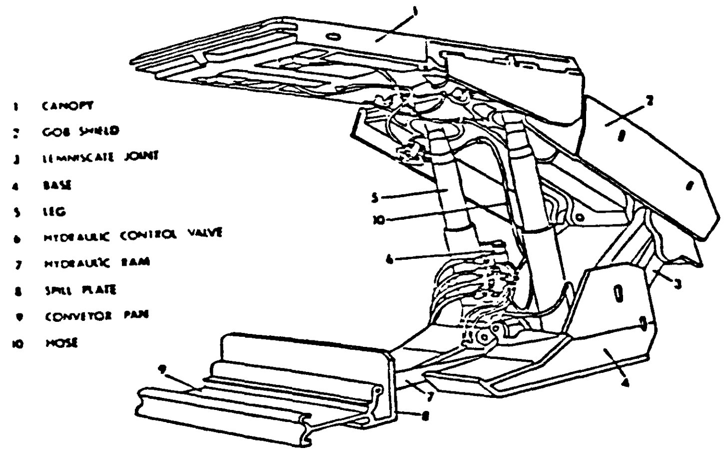
2-Leg Shields

Two-leg shields (Figure 3) have the following general characteristics.

1) Wide range of mining heights with low collapsed height.
2) High support resistance.
3) High resistance to tangential forces (good for inclined seams).
4) Travelway safe from gob flushing.
5) Smaller cross-sectional area than 4-leg shield.
6) Keeps face airflow from escaping to gob.
7) Hydraulic system protected from debris.
8) Good in friable roof conditions, minimum tramping effect.
9) Good in seams with roof steps or roof falls.
10) Good in medium-to-thick seams.
11) Generally recommended where shield capacity is less than 500 tons.
12) Can create high floor pressures near base toe.

4-Leg Shields

Four-leg shields (Figure 4) have similar characteristics to 2-leg shields with advantages as follows:



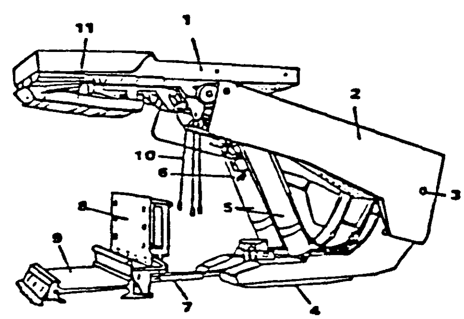
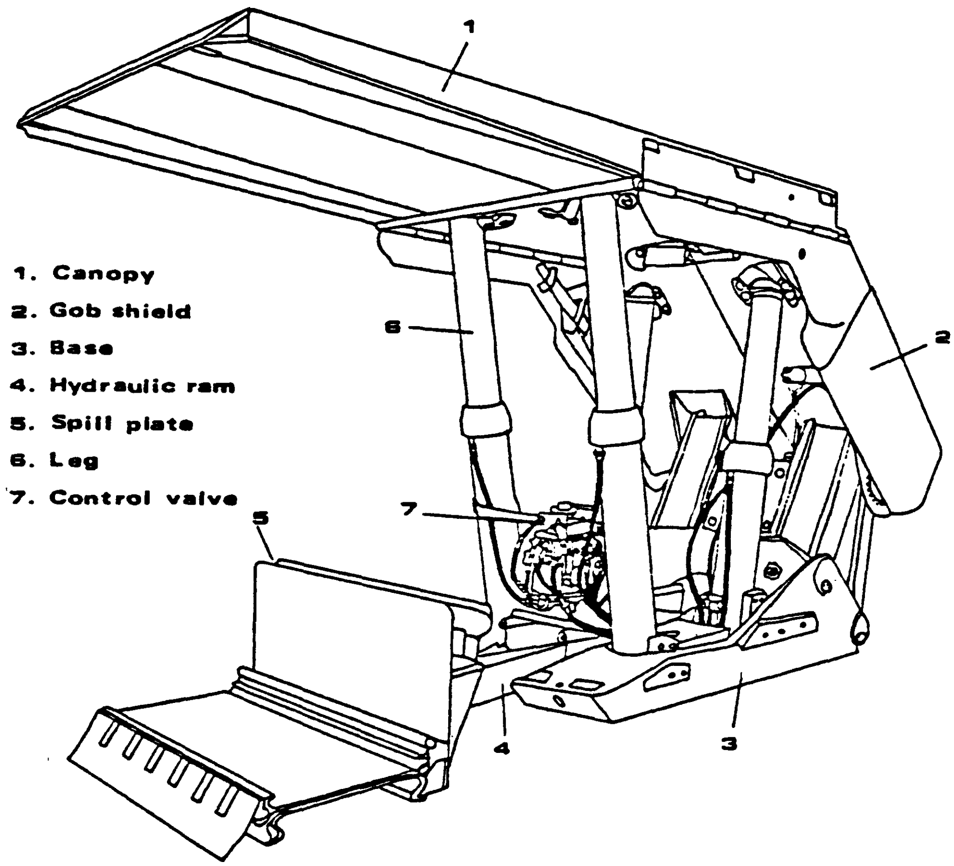


Figure 3

Components of a 2-Leg Shield

(After Peng, 1978)



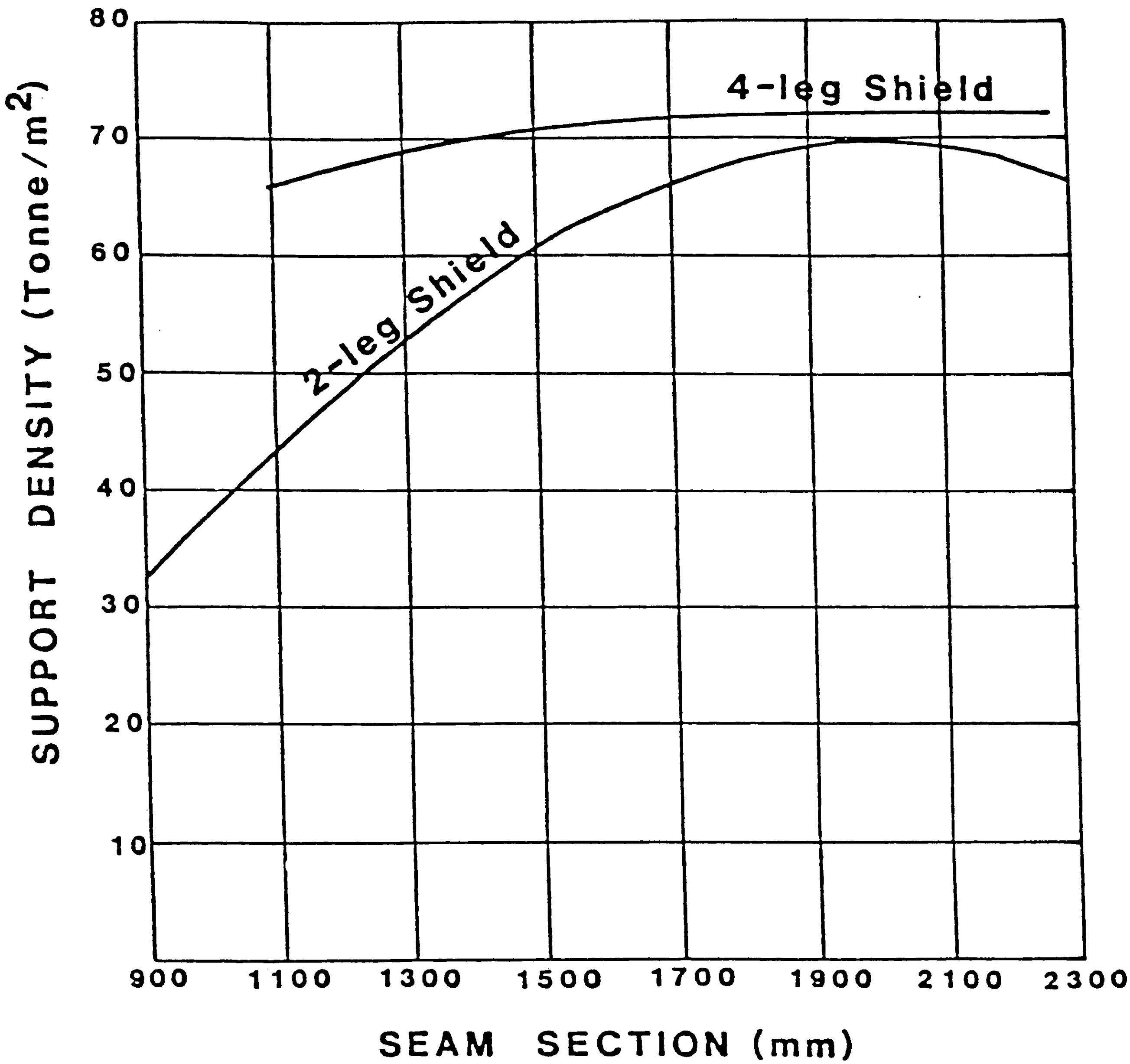
Components of a 4-Leg Shield

Figure 4

(After Peng, 1978)

1) Larger cross-sectional area than two-leg shields for methane and dust control.
2) Higher support density due to two legs acting directly on roof.
3) Little variation in support density with height because of shield geometry (Figure 5).
4) Although two extra legs are required, there is no overall increase in hydraulic gear because no hydraulic cylinders are needed between the gob shield and the top canopy to control articulation.
5) Canopy tip load is higher.
6) Stability of the support is better because the center of gravity is near the center of the base.
7) The line of action of the roof load is well inside the toe of the base which prevents toe penetration in soft floors.
8) Good at overcoming large roof cavities.
9) Allows the use of smaller, less expensive hydraulic cylinders.

Four-leg shields find particular application in areas with shallow overburden or massive sandstone roofs; under these conditions, the rear legs have the ability to create a break-line in the roof at the rear of the canopy.



Variation of Support Density with

Height for 2-Leg and 4-Leg Sheilds

Figure 5

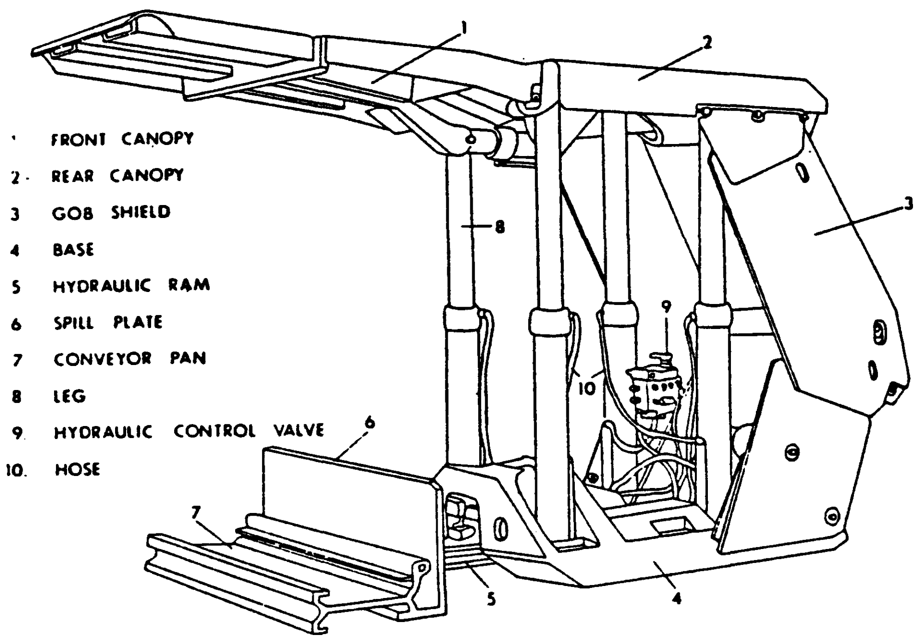
(Dowty, 1982)

Chock-Shield

The chock-shield (Figure 6) is a relatively recent arrival on the longwall scene. This new, powered roof support is a hybrid between chock and shield which incorporates most of the best aspects of both parents. Some of the chock-shield characteristics are:

1) The legs act directly on the top canopy.
2) The yield load is constant throughout the height range.
3) It has a rigid base.
4) It has a lemniscate linkage.
5) The side and gob shields eliminate gob flushing.
6) The support geometry assures a good travel way at any height.
7) The front legs afford protection from spalling to the walkway.
8) It has potentially higher support capacity than a lemniscate shield.
9) It provides a high cross-sectional area for ventilation.

The disadvantages of this unit include the rigid base and longer canopy length (thus necessitating higher support resistance). The chock-shield's ability to resist tangential forces is not well defined. Chock-shields may be most



Components of chock shield support.

Figure 6

(After Peng, 1978)

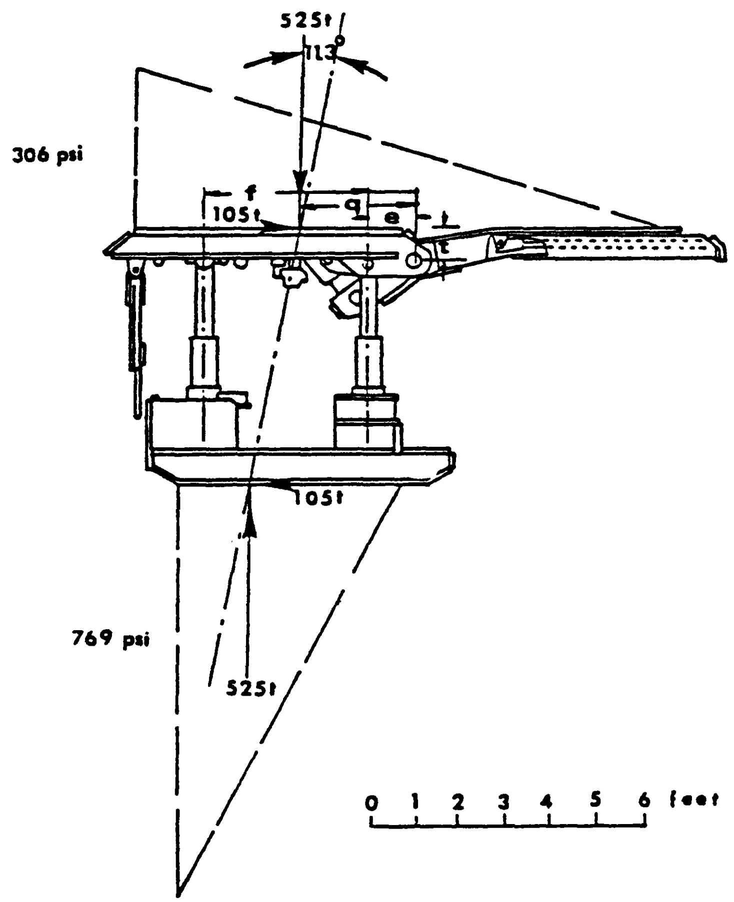
comparable overall to two-leg lemniscate shields.

The following sections describe in a general context the common components to all powered supports.

Bases

Each of the powered supports discussed has a different load distribution as depicted in Figure 7 a,b,c,d. These distributions are, however, only approximate. It is obvious that there may be some load on a support at any point along the roof canopy or base. Upon examination, it can be seen that shield-type supports, with the exception of the chock-shield, have high toe loads which tend to create toe sinking conditions. Although the "swinging base" concept of attaching the legs behind the toe of the base has minimized this condition, shield advancement can become difficult if sinking occurs. Recent shield designs which have split bases as opposed to solid bases allow independent movement of the bases in the vertical direction. Such independent movement allows the negotiation of floor steps or sinking conditions with a minimum of effort. The tunnel between the split bases also allows loose floor material to migrate through to the gob instead of "rolling" in front of the base, which requires hand removal.

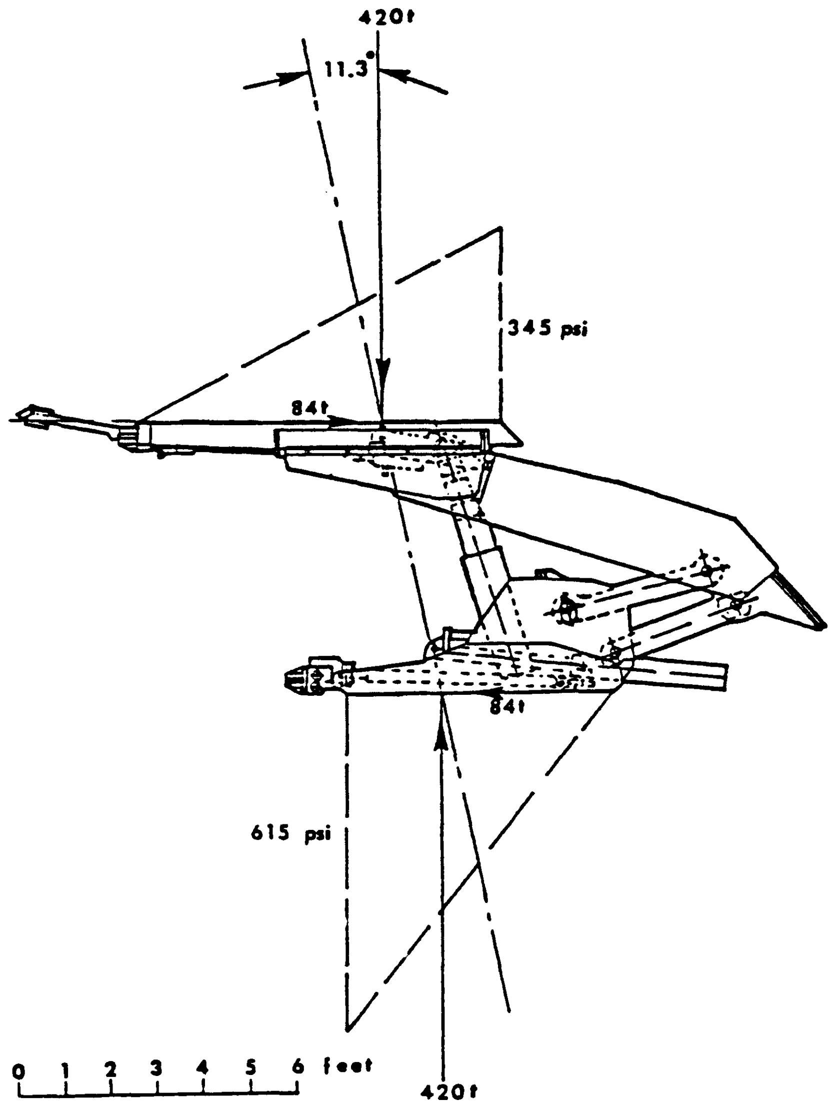
Base design should be considered with regard to plate bearing test results for seam specific conditions.



Chock Load Distribution

Figure 7a

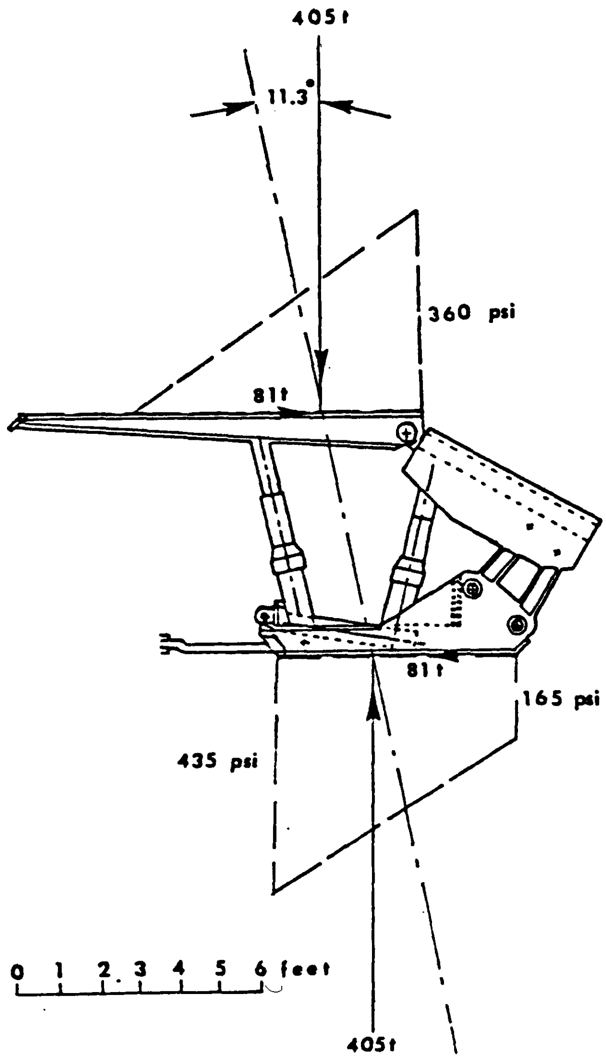
(After Peng, 1978)



2-Leg Shield Load Distribution

Figure 7b

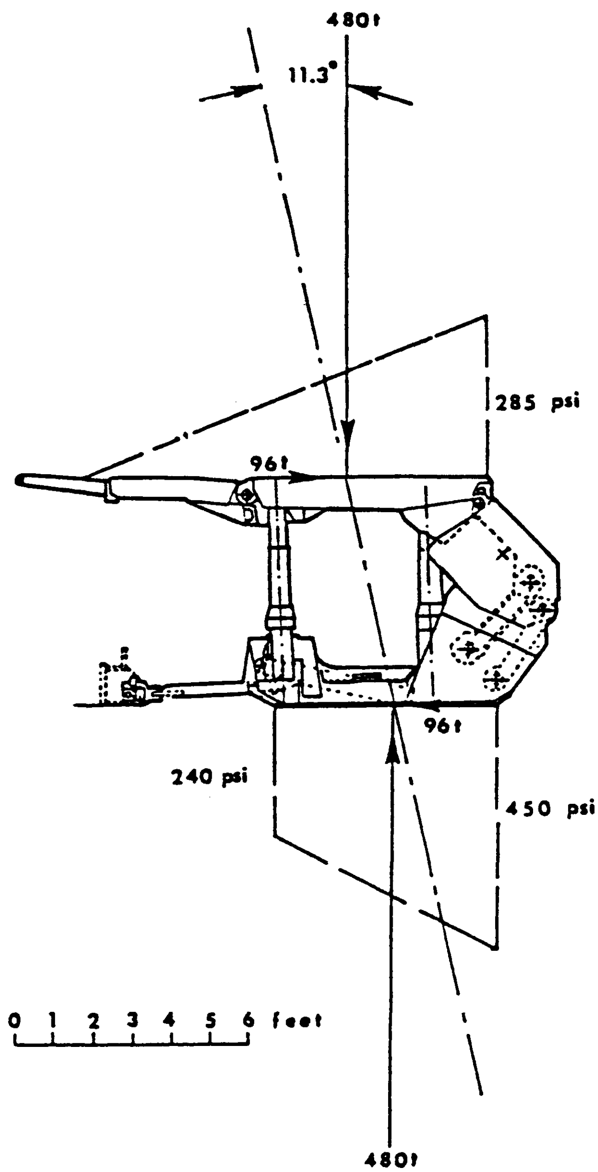
(After Peng, 1978)



4-Leg Shield Load Distribution

Figure 7c

(After Peng, 1978)



Chock-Shield Load Distribution

Figure 7d

(After Peng, 1978)

Hydraulic Cylinders

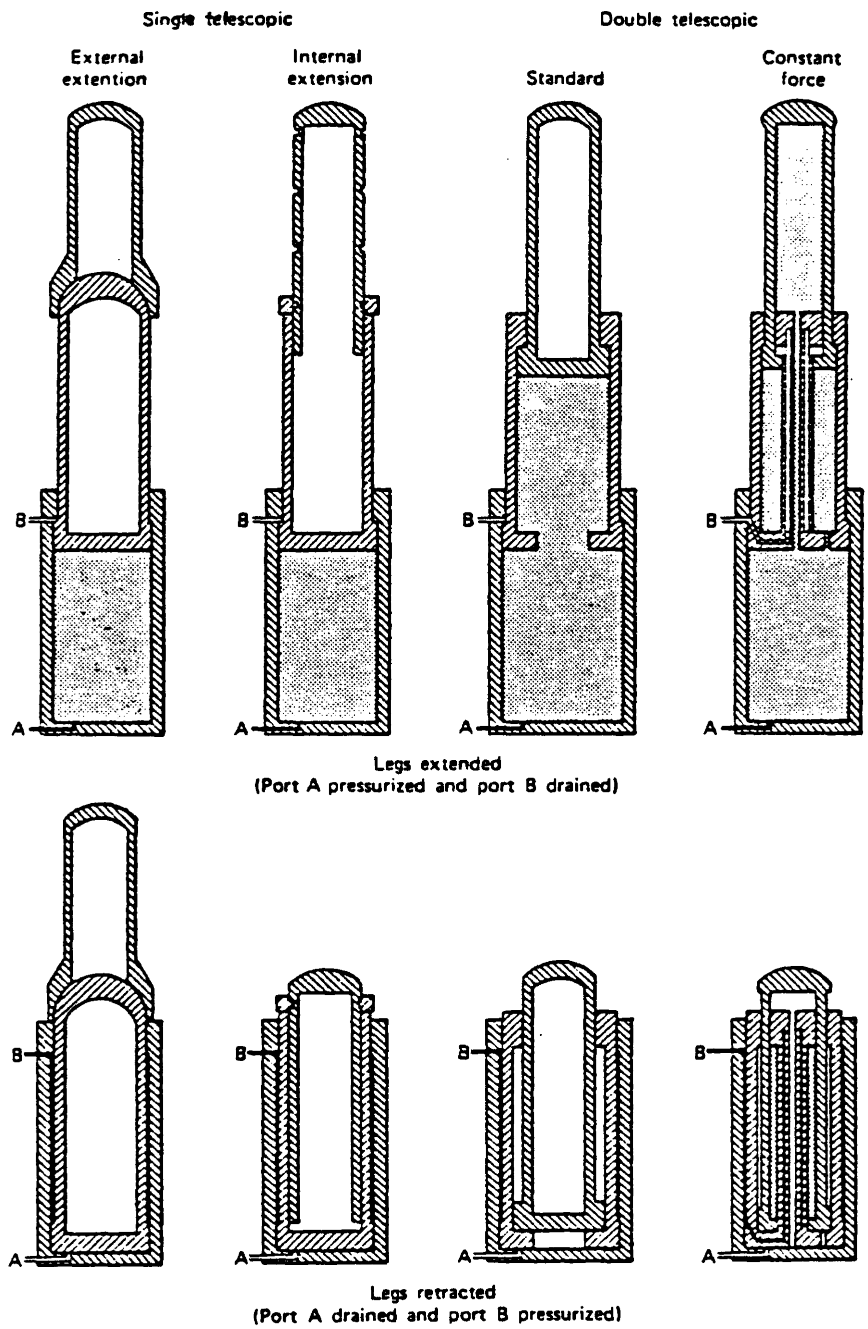
The hydraulic legs on a powered roof support device are ultimately responsible for providing the support force. The leg-cylinders are generally single, double-acting cylinders, or they may be telescopic, double-acting cylinders with as many as three telescoping units. Single telescopic units are available with internal or external extensions, while double telescopic units are available with standard or constant force configurations as depicted in Figure 8.

Standard plating thickness on the legs has been 0.0015 in. to 0.0020 in. on the hydraulic leg ram. New leg cylinders are being specified to have 0.0030 in. to 0.0040 in. of plating for wet, corrosive or abrasive environments.

Hydraulic Fluid

The fluids used in the powered supports are of four types; 5% soluble oil-in-water emulsion, 40% water-in-oil emulsion, 50% glycol-in-water solution and refined petroleum based oil. The basic requirements of the fluid are:

1) low cost
2) low viscosity
3) inflammible
4) chemically non-reactive with air
5) resistant to foaming



Telescopic Hydraulic Cylinders

Figure 8

(After Loxley et. al., 1975)

6) non-corrosive

7) provide lubrication to system

Hydraulic oil supply pressure on U.S. longwalls ranges from 3500-5000 psi while it is as low as 1500-2750 psi on British faces. This difference is attributable to differing opinions and practices on support setting pressure.

Support Control

Support controls are of three basic types; same support control, adjacent support control, and banked or remote control. Adjacent support control is preferable to same support control because of the safety afforded by this method. Banked control of four to eight supports is desirable for headgate and tailgate supports at least -- if not for all supports. Graham has shown (Graham, 1978) that manual advance of a support may require eleven seconds including support mover travel time of four seconds. Banked or remote control supports have been noted to have a comparable advancement time of six seconds with support mover time of zero to four seconds. Such a system not only saves time, but it may eliminate the need for one of the support movers.

A guaranteed set pressure mechanism may be desirable. This device guarantees that the required setting pressure is attained in every support. With manual pressure setting, this is often not the case. Set pressure in random supports may be as little as 20 percent of the desired set pressure under manual setting conditions. Uniform set pressure on the face helps maintain acceptable strata control.

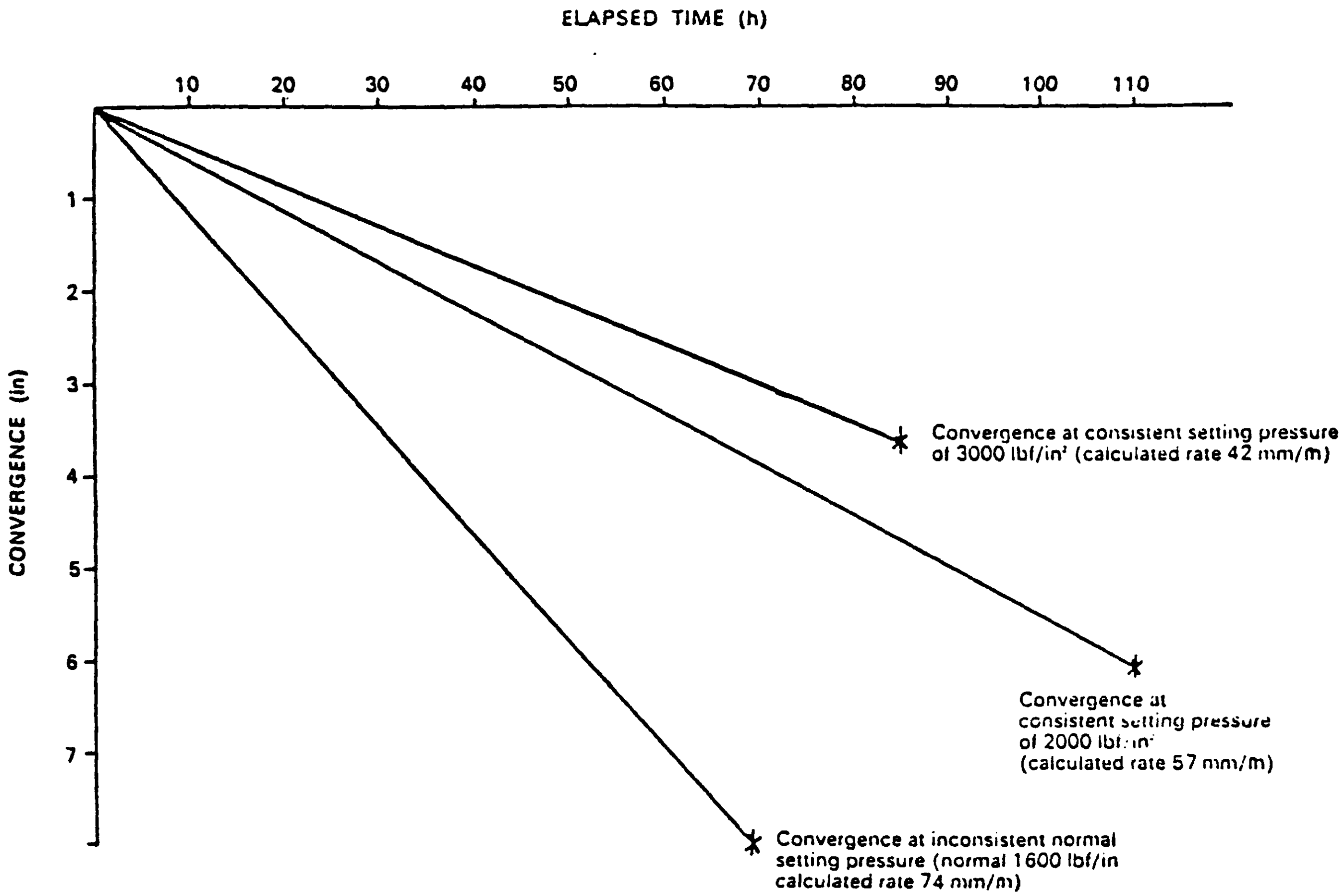
Set Pressure

The pressure at which the powered support should be set is the subject of much controversy. European and U.S. thought suggests that the set pressure should be 75% of the yield pressure, while British practice is to set supports at 50% of rated yield pressure.

There is evidence which indicates that higher setting pressures aid in limiting roof-floor convergence by limitation of bed separation or by limiting strain on jointed or cracked roofs. Figure 9 shows the results of increased setting pressures on a friable roof's convergence characteristics. Figure 10 shows the recommended support load densities and hydraulic pressure tolerances for some selected supports.

Extended Height and Vertical Hydraulic Travel

The extended height of a support must be able to vary within limits to account for convergence and varying seam thickness. Distance from the coal face is important in the determination of these limits. Figure 11 shows NCB recommended minimum vertical travel (travel below minimum working height) for various distances from the face and minimum extended heights.



Strata Behaviour at Various Setting Pressures

Figure 9

(After Bates, 1978)

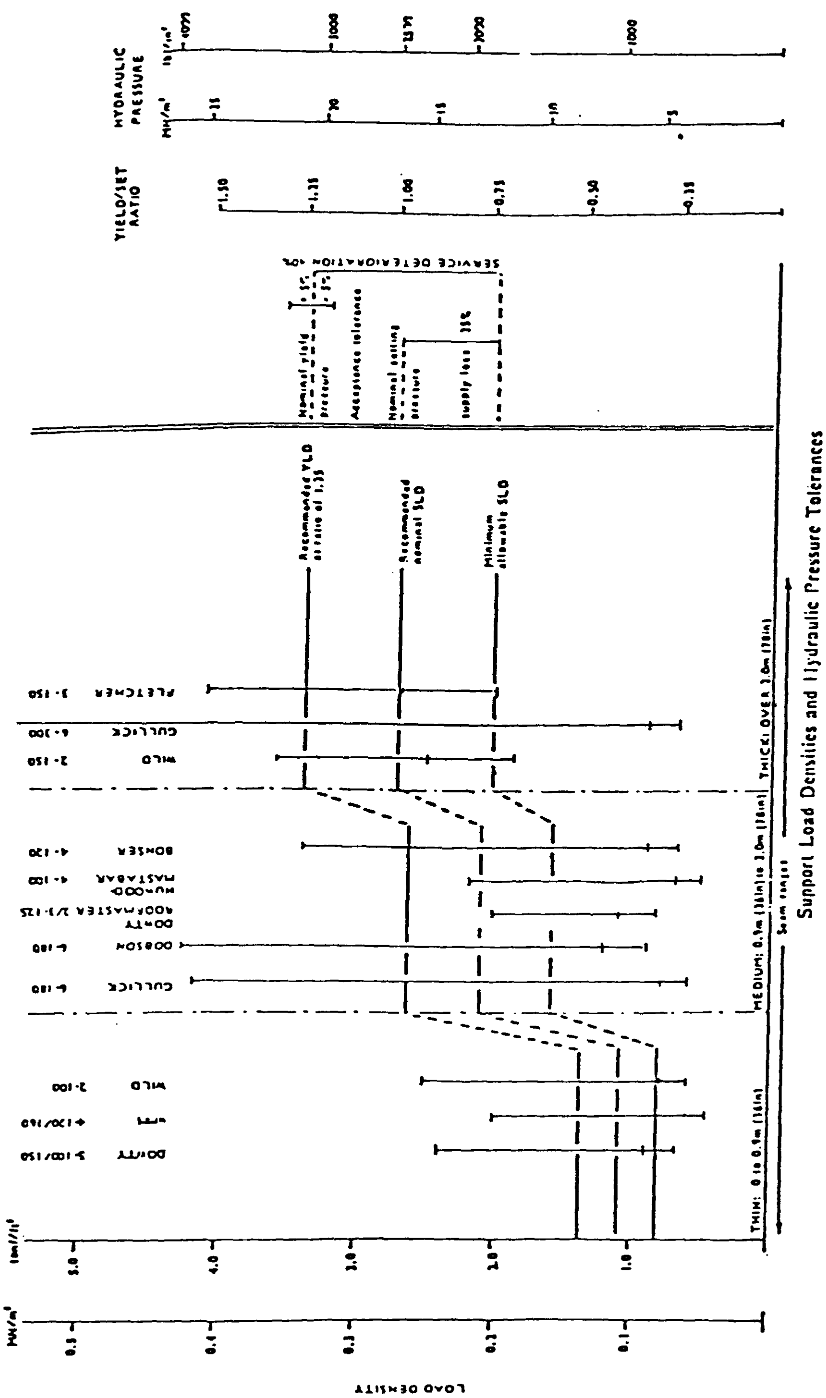


Figure 10

(Adapted from Ashwin, 1970)

Extended Height and Travel

	Conver-gence C mm/m	*Nominal Height of Extraction, H* m	(in)	*Contingency Allowance, a* m	(in)	*Minimum Extended Height, h, in metres at distance from coal face of l* l = 1.25 m (4.1 ft)	l = 1.75 m (5.8 ft)	l = 2.50 m (8.2 ft)	l = 3.00 m (9.80 ft)	*Vertical Hydraulic Travel, V* m	(in)
Thin	40	0.70	(27)	0.05	(2)	0.70	0.68	0.65	0.63		
		0.80	(31)			0.80	0.78	0.75	0.73		
		0.90	(35)			0.90	0.88	0.85	0.83	0.31	(12)
Medium	50	1.00	(39)	0.05	(2)	0.99	0.96	0.93	0.90		
		1.10	(43)			1.14	1.11	1.08	1.05		
		1.14	(45)			1.18	1.15	1.12	1.09	0.31	(12)
		1.20	(47)	0.10	(4)	1.24	1.21	1.18	1.15		
		1.30	(51)			1.34	1.31	1.28	1.25		
		1.40	(55)			1.44	1.41	1.38	1.35		
		1.50	(59)			1.54	1.51	1.48	1.45		
		1.52	(60)			1.56	1.53	1.50	1.47	1.43	(17)
		1.60	(63)	0.15	(6)	1.69	1.66	1.63	1.60		
		1.70	(67)			1.79	1.76	1.73	1.70		
		1.80	(71)			1.89	1.86	1.83	1.80		
		1.90	(75)			1.99	1.96	1.93	1.90		
		2.00	(79)			2.09	2.06	2.03	2.00	0.52	(21)
Thick	80	2.20	(87)	0.20	(8)	2.30	2.26	2.20	2.16		
		2.40	(95)			2.50	2.46	2.40	2.36		
		2.60	(102)			2.70	2.66	2.60	2.56		
		2.80	(110)			2.90	2.86	2.80	2.76		
		3.00	(118)			3.10	3.06	3.00	2.96	0.68	(27)
		3.20	(126)	0.25	(10)	3.30	3.26	3.20	3.16		
		3.40	(134)			3.50	3.46	3.40	3.36		
		3.60	(142)			3.70	3.66	3.60	3.56		
		3.80	(150)			3.90	3.86	3.80	3.76		
		4.00	(157)			4.10	4.06	4.00	3.96	0.82	(32)

Figure 11

(After Ashwin, 1970)

Fore Poles/Face Sprags

Most types of supports are available with extendible canopies (forepoles) and face sprags. Forepoles are a valuable option in jointed or friable roots subject to falls where the basic support design allows the generation of adequate tip loads. Face sprags can be valuable if face spalling is a problem, particularly during an extended period of inactivity.

Face sprags should be considered for seams thicker than 2.5 m to assist in roof control. However, they may provide no greater protection to face personnel than high spill plates.

Summary

For purposes of review, Table 1 has been prepared summarizing face support characteristics.

Table 1

CLASSIFICATION OF SUPPORT SYSTEMS ACCORDING TO SELECT CRITERIA

CRITERIA / CLASSIFICATON	RANK 1 (OPTIMAL)	RANK 2	RANK 3	RANK 4 (POOREST)
PRIMARY COST PER UNIT OF RESISTANCE	FRAME CHOCK	2-LEG SHIELD	4-LEG SHIELD	
SCREENING AGAINST CAVE DEBRIS	CHOCK-SHIELD 2-LEG SHIELD 4-LEG SHIELD	CHOCK		FRAME
CAPABILITY OF OVER-COMING ROOF CAVITIES	4-LEG SHIELD	CHOCK CHOCK-SHIELD	2-LEG SHIELD	FRAME
TANGENTIAL FORCE CAPACITY-IN DIRECTION OF ADVANCE AND TRANSVERSE	4-LEG SHEILD	2-LEG SHIELD	CHOCK CHOCK-SHIELD	FRAME
RANGE OF HEIGHT ADJUSTMENT	2-LEG SHIELD	4-LEG SHIELD	CHOCK-SHIELD CHOCK FRAME	
FLOOR PRESSURE	4-LEG SHIELD	CHOCK CHOCK-SHIELD	2-LEG SHIELD	FRAME
ROOF PRESSURE	CHOCK	4-LEG SHIELD FRAME	2-LEG SHIELD CHOCK-SHIELD	
SUPPORTED ROOF %	2-LEG SHIELD 4-LEG SHIELD CHOCK SHIELD	CHOCK		FRAME
YIELD LOAD TONS	CHOCK 4-LEG SHIELD	2-LEG SHIELD CHOCK-SHIELD		
VENTILATION CONTROL	2-LEG SHIELD 4-LEG SHIELD CHOCK-SHIELD			FRAME CHOCK

Coal Producing Machines

Types of Machines

When selecting a coal producing device, the criteria of most interest pertain to the device's ability to maintain a continuous or near continuous maximum coal load on the armoured flexible conveyor (AFC), machine flexibility, stability, and cost. When a seam thickness of under 72 inches is being considered, and particularly those below 50 in., a double-ended, conveyor-mounted trepanner (DECMT) or an in-web-shearer might be utilized. Furthermore, new coal plow technology makes plowing a viable alternative, even in hard coals.

For seam thicknesses of 72 inches or more, however, a drum-type shearing machine is generally the best alternative. Drum-type shearing machines may have single or double, fixed or ranging drums, the double-ended, ranging drum shearing (DERDS) machines being most desirable.

For purposes of the model, only seams greater than 60 in. thick were considered. Therefore, only double-ended ranging drum shearers were selected. Shearing machines are the norm on new U.S. faces and may be considered state-of-the-art in coal winning machines. Shearing machines are considered only in terms of their ability to fill the AFC to its maximum capacity and their advantages and disadvantages to the face as a unit.

Shearing-Type Machines

All of the shearing-type machines utilize a positive milling action to mine coal. The result of this positive coal breaking is that all shearing-type machines are capable of filling the AFC to its maximum capacity while moving in either direction on the face, depending on the AFC speed. Because of this fact, the various shearing techniques are not specifically discussed. It should be noted, however, that some techniques such as "half-facing" are supposedly better than "full face" techniques in regard to respirable dust production. Though this may be true, such techniques also appear to have lower machine shearing time utilization than full face shearing, and thus lower overall productivity.

In thin seams, the in-web-shearer or the trepanner-type machines are useful because they have a lower profile than AFC mounted shearers and therefore provide adequate room above the AFC pans for coal haulage.

When medium or thick seams are encountered, the machine of choice is often a double-drum, ranging-arm shearer. Fixed drum shearers are useful only where there is a fixed and constant extraction height without any geologic disturbances. Single-drum, ranging arm-shearers are of limited use because they are unable to cut out both entries in narrow entry systems.

The cutting drums of a shearing machine may be short cylinders or spiral scrolls with bits mounted on the drum or scroll facing. The scroll drum is a popular design. To complement the clean-up provided by the cutting drums, cowls are commonly used. The cowl is a large, position-variable element curved to conform to the cutting drum shape. When the scroll acts in conjunction with a variable cowl, more complete coal clean-up is made in the web which enables maximum AFC and powered support advance. This maximum advance ensures maximum web depth on the following shear. The enhanced clean-up results from the screw-conveyor effect of the drum and cowl as well as the floor grading effect of the lowered cowl. Drum speeds range from 60-72 rpm with haulage forces as high as 40 tons.

The bit lacing on the drum should, as a guideline, be such that the cutting lines are two to three times the pick penetration. Such bit lacings are generally required to minimize the number of picks required and prevent coal coring effects. Such coring effects lower machine productivity.

Dust suppression on the shearing machine, the principle dust source on a longwall, arises from water sprays mounted on the machine. Water sprays on the drums, perhaps to the extent of pick-face-flushing, along with cowl sprays and inter-drum spray bars can do a great deal to control the dust generated on and around the drums. Many U.S. longwalls, however, still are not in compliance with federal standards on respirable dust.

As a further note, shearing machines in medium or thick seams often have a rotary chunk breaker on the tail-gate end. This helps minimize face down-time due to large chunks which cannot pass through the AFC tunnel on the shearing machine. There should also be a chunk breaker on the stage loader assembly to minimize down time of a similar nature at the headgate.

Shearing machines derive their haulage force in one of two ways -- by haulage chains or by "chainless" haulage. Chainless haulage is state-of-the-art technology in longwall mining. The following chainless haulage systems are available in the U.S.

1) Joy traction rack (Joy Ride)
2) Pitcraft Rackatrack
3) Eickhoff Eicotrack
4) Perard Pera-track
5) Anderson Mavor rollrack
6) Bolton Barlock
7) Cerchar Gripping Hands

The first five methods are basically panline shearing machine rack and pinion drives. The latter two are hydraulic ram propulsion systems with some sort of chucking mechanism which engages a traction bar mounted on the AFC pans.

Chainless haulage has the following advantages over chain haulage:

1) Increased safety
2) Reduced maintenance on the haulage system
3) Reduced noise (no haulage chain on AFC)
4) No haulage anchor points
5) Positive captivation of shearer on AFC
6) Less energy consumption
7) Better shearer control on grades or undulating conditions
8) Smoother movement causing reduced wear and downtime
9) Reduced damage to face supports, hydraulics, and electrics
10) Increased haulage force
11) Increased AFC pan strength meaning less panline wear and damage

Armoured Flexible Conveyors

The capacity of a longwall is largely dependent on the capacity of the Armoured Flexible Conveyor (AFC). With this in mind, the AFC must be carefully matched to the production machine, the face support's anchoring demands, and the face length. On most modern AFC's, the most critical determination to be made by an operator pertains to the conveyor chain.

AFC chain is available in the following sizes with their respective breaking strengths.

19x64.15 mm	53 tons
22x86 mm	70 tons
26x92 mm	94 tons
30x108 mm	126 tons

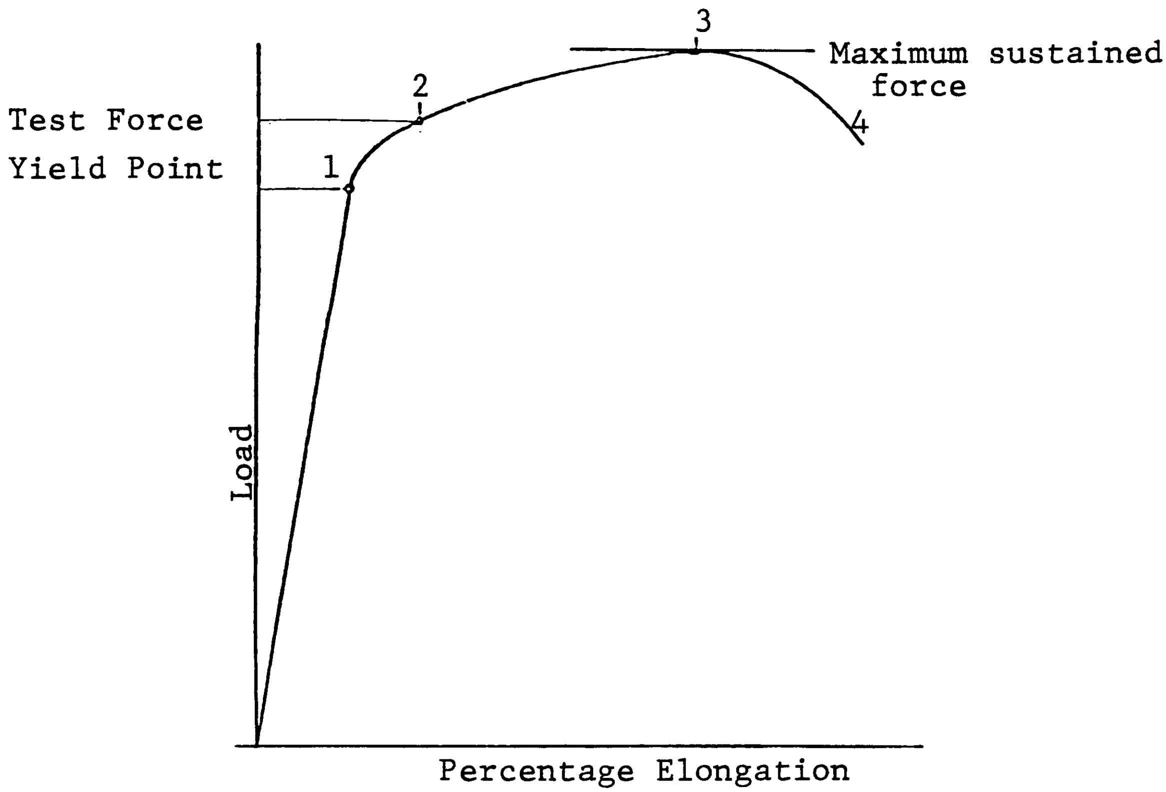
In addition to the link size choice, strands to control the scraper bars and drive the conveyor may consist of single chain, double inboard or outboard chains, or triple chains. Multi-chain strands have proportionately higher breaking strengths than single chains.

Single or twin-inboard strands are most popular in the U.S. at present. These strand configurations have advantages over outboard strand configurations in that they have a lower tendancy to freeze in the conveyor races as a result of conveyor damage or fines briquetting; they are also less sensitive to bends and kinks in the longwall face. Tests on power consumption of various strand configurations have demonstrated little difference between the types.

In order to run correctly, AFC chains must be pretensioned by stretching the chain to its elastic limit (Figure 12). Excessive pretensioning can cause high power requirements, increasing linearly with tension. Furthermore, it can create wear and maintenance problems on chain links, sprockets, and bearings. Excessive pretensioning greatly increases the amplitude of tensile force oscillations on the AFC chains, particularly when the conveyor is not loaded.

The running behavior and working life of the AFC components are improved with decreased pretensioning; however, low pretensions are not desirable either. Low pretensions can lead to excessive slack on the head drive which can result in scraper bars traveling inverted on the return race or excessive carryback of fine coal to the return race of the AFC. Excessive carryback can cause the AFC to "bog down" and stop. On single chain installations, low pretensions may allow the scraper bars to twist and come out of their races.

The following equation may be used as a guideline for determining the correct pretension to apply.



1. Elongation at yield
2. Elongation at test force
3. Elongation at maximum force
4. Total Elongation at break

Typical Stress-Strain Curve for AFC Chain

Figure 12

(After Mason, 1979)

$$\text{Pretension} = \frac{(T_1 + T_2 + T_3 + T_4)}{4}$$

T_1 = Tension induced by the tail drive at full load torque

T_2 = T_1 - frictional loss across bottom race. The loss across the bottom race being approximately 4 tons for a 600 ft face and proportional for different face lengths.

T_3 = T_2 - force induced by the head drive at full load torque.

T_4 = Tension required in the top chain at the tail drive to pull the chain off the sprocket. This value may be zero or negative if the drive tolerates slack.

It should be noted that the pretension is independent of chain size and is the total force on the chain strand.

It is important to check the pretension on new chains or installations every shift during the "bedding in" period.

In addition to poor pretensioning, the following shortcomings in chain maintenance and utilization are common:

1) Twin-inboard chains are not replaced in pairs.
2) New chain is not stored in a protected area.
3) AFC is run more than a few minutes without lubrication.

4) The longwall face is bent.
5) Poor delivery of coal to the stage loader causes excessive carryback of fines.
6) Missing or broken line pan connecting bolts allowing scraper bars to bind.
7) Damaged chain strippers are unrepaired.
8) Excessively sharp snake-over behind shearing machine causing scraper bars to bind.
9) Damaged scraper bars are not replaced.

The capacity of the face conveyor is dependent on the speed of the chains and the dimension of the conveyor. Figure 13 displays this interdependancy. Additional information on conveyor loading with shearing type machines is provided in Figure 14.

Increased conveyor capacity may be provided by spill plates. Figure 15 shows the relationship between spill plate height and AFC capacity. The figure shows that 7-in. spill plates bring the AFC to 90% of its maximum capacity.

Higher spill plates may be desirable in thick seam sections or conditions of large sized face spalling, both for protection of face personnel and confinement of large chunks to the moving AFC load.

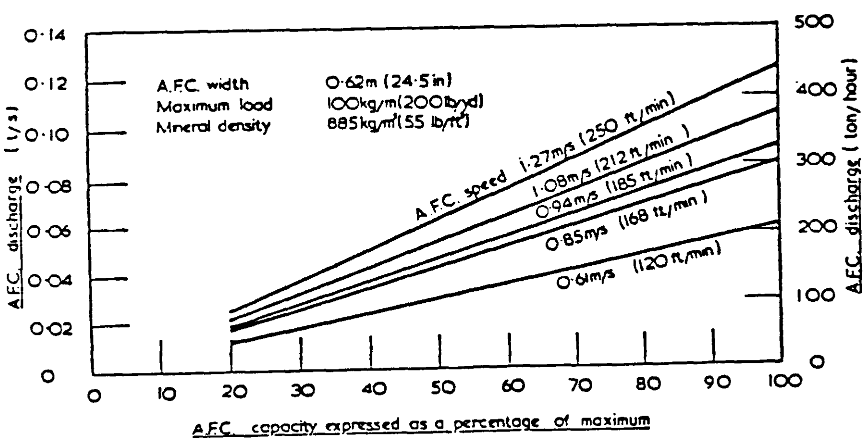
A roller-curve device at the headgate may be useful to minimize transfer problems at the stageloader. The use of a

LONGWALL FACE CONVEYOR SPECIFICATIONS

Conveyor Width x Ht (in.)	Quantity of Material (ft^3/ft)	Conveyor Speed (fpm)	Approximate Output (tph)	Lump* Size (in.)	Loading** (lbs/ft)	Discharge Rate (tpm)
		100	174			2.90
24 x 8	1.16	210	365	10	58.0	6.09
		230	400			6.67
		100	240			4.00
		210	504			8.40
30 x 10	1.60	230	552	14	80.0	9.20
		260	624			10.40
		100	360			6.00
		210	756			12.60
36 x 12	2.40	230	828			13.80
		260	936			15.60

* Not to exceed 10 percent of total volume.
** Coal density = 50 lbs/ft^3.

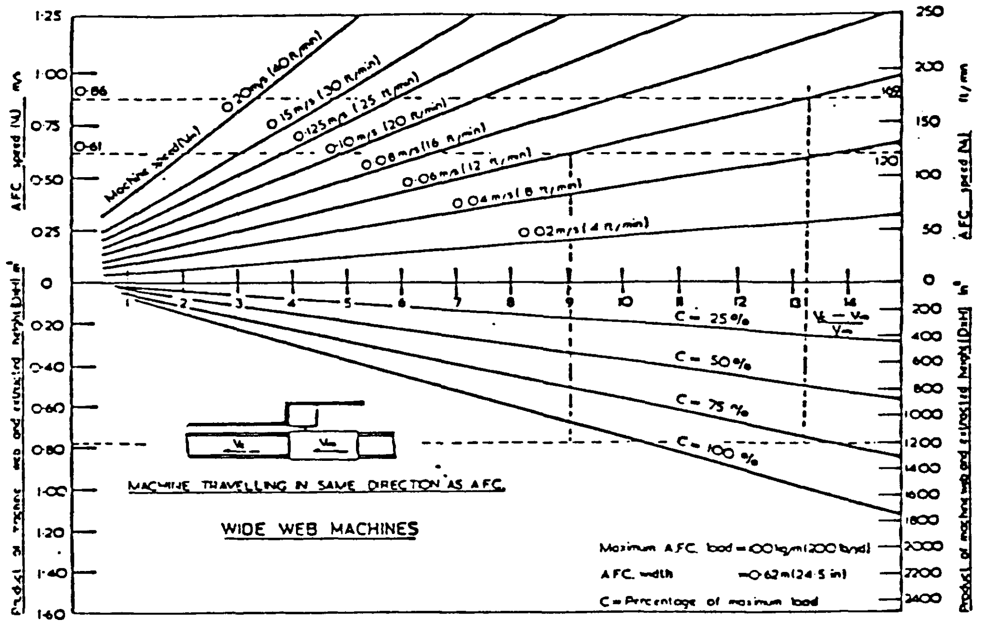
After Britton, 1980



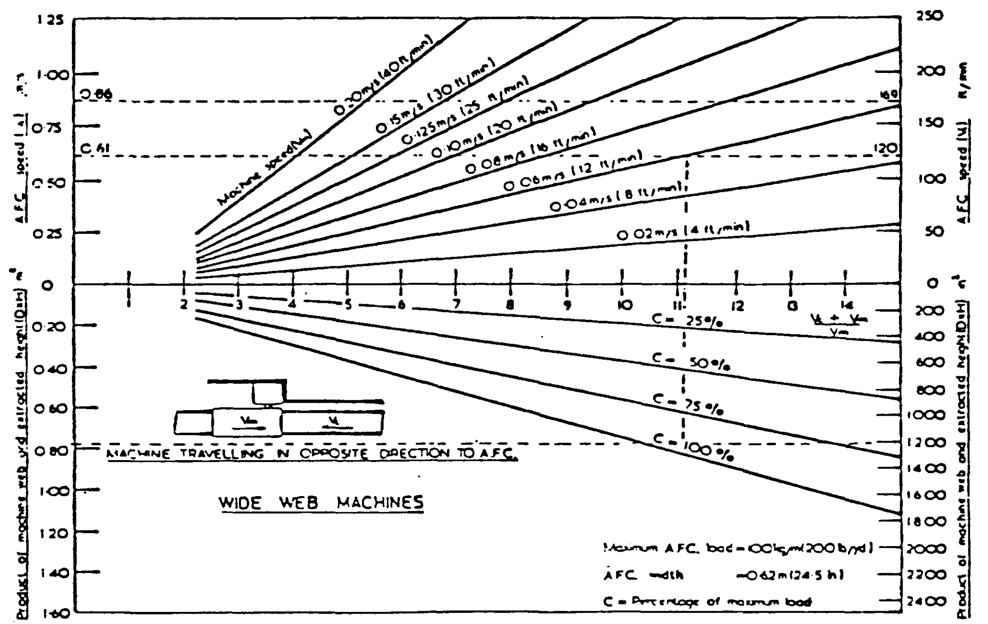
Mineral discharge rate from AFC at different speeds

Figure 13

(After Guppy, Whittaker, 1970)



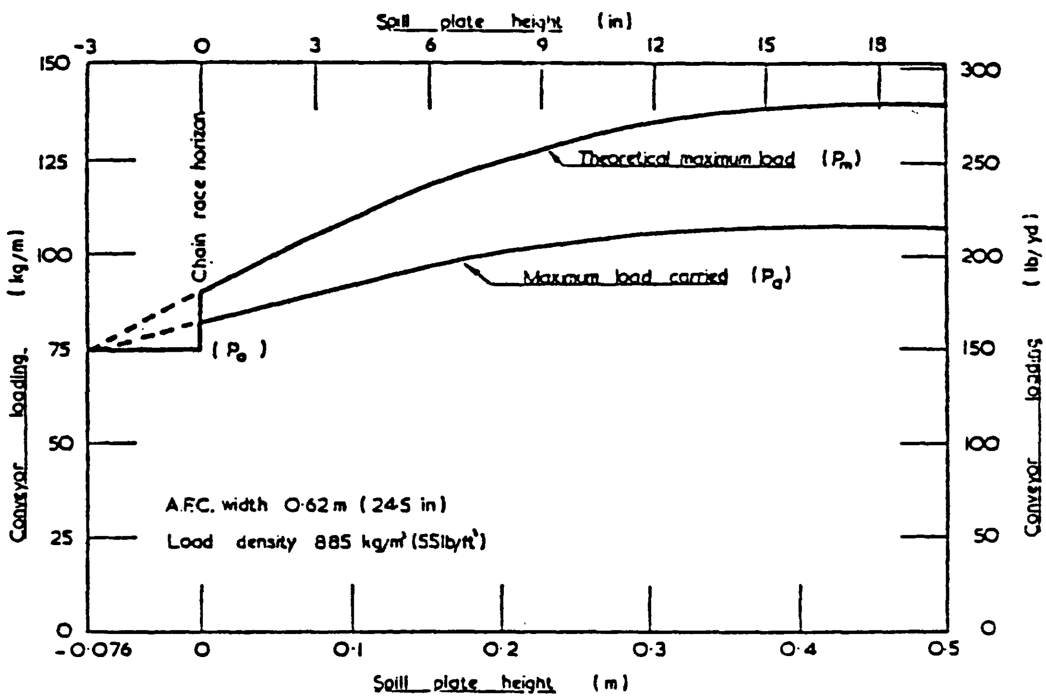
Relationship between AFC loading and machine web and extracted section for various machine and AFC speeds. (Wide web machines)



Relationship between AFC loading and machine web and extracted section for various machine and AFC speeds. (Wide web machines)

Figure 14

(After Guppy, Whittaker, 1970)



Effect of spillplate height on AFC capacity

Figure 15

(After Guppy, Whittaker, 1970)

roller curve device implies that space is available for the 5-7 ft diameter sheaves, that horsepower is limited to about 500 hp for 30x108mm chains, and that a special four link scraper bar spacing is utilized. This scraper bar spacing requires 2.5 times the normal number of scraper bars. The roller-curve system may also limit the service life of chains by 50% in some cases.

AFC Drives

AFC drives in the United States consist of an air cooled, 1800 rpm induction motor with a fluid coupling and a bevel spur gear.

German practice dictates the use of dual-speed, double-wound, water-cooled motors. Starting of the AFC is done with one-third of nominal speed which eliminates the requirement for fluid couplings. High starting torques can be maintained and current limited with this method. However, problems do exist with this technique. When starting under load, the head drive is under a higher starting load than the tail-drive; this allows the tail-drive to come to a higher torque and speed than the head drive in an equal amount of time. Because the chain connects the drives, the tail-drive becomes influenced by the slower head-drive. This mutual influencing sets up a continuous oscillation pattern for AFC operation. The drives, under oscillation, accept equal loads during equilibrium operation of the AFC. The U.S. practice of using fluid couplers eliminates this oscillating condition by eliminating mutual influencing of drives. It has been found that the load on the drives is nearly the same in both drive techniques, but start time is increased by several seconds with the fluid coupling method.

CHAPTER IV

CONSTRAINTS ON LONGWALL FACE LENGTH

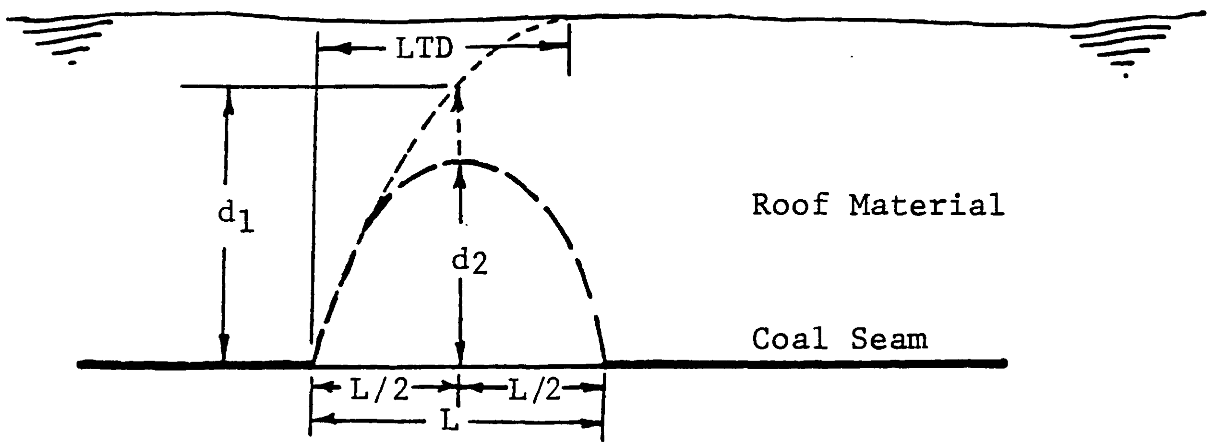
Geotechnical Constraints

There appears to be no upper limit on longwall face length imposed by geotechnical constraints. However, no matter what the face length is, the face must advance rapidly enough to avoid ground control problems related to roof convergence and face support yield with time. In general, Gorrie and Scott (1970) state that a face should advance 60 ft per week to ensure that time dependent roof closure does not give rise to ground control problems.

In contrast, there is a geotechnical lower bound to longwall face length. This bound is the minimum width required to promote good caving of the gob. In order to cave adequately, the span must collapse under its own weight.

The weight on the unsupported (mined out) span, parallel to the longwall face, is assumed to be attributable to a parabolic arch of rock. That is, the weight of all rock under the load transfer arch. The arch is depicted in Figure 16.

To arrive at the weight, W, it is necessary to use an empirical load transfer distance equation (Abel, 1981) to find the maximum arch height (d_2). The maximum arch height



L = unsupported span length

LTD = Load Transfer Distance

d_1 = height from the span to the Load-Transfer-Arch

d_2 = maximum arch height

W = weight of unsupported span (acting along d_2)

The Scaled-Down Load-Transfer-Arch

Figure 16

(After Abel, 1981)

multiplied by the unit weight of the rock yields the maximum load on the roof beam of a load transfer arch "L" ft. wide.

The load-transfer equation (1) can be scaled to arrive at the maximum arch height, d_2, for an arch less than 2 LTD 's wide (equation 2).

$$LTD = -45.0 + .373H - 0.000082\,H^2 \qquad (1)$$

$$d_1 = \frac{H}{(LTD)^2}\left(2(LTD)(L/2)-(L/2)^2\right)$$

$$LTD_2 = -45.0 + .373\,d_1 - .000082\,d_1^{\,2}$$

$$d_2 = \frac{d_1}{(LTD_2)^2}\left(2(LTD_2)(\tfrac{L}{2})-(\tfrac{L}{2})^2\right) \qquad (2)$$

After (Abel, 1981)

Where H is equal to the seam depth (ft) and d_1 is the height from the seam roof to the load transfer arch at some point other than $L/_2$.

Assuming a rock density of 144 lb/ft^3 and a beam width of one foot, the weight W which results from d_2 is $W = d_2 x 144.0$ lbs/sq. ft.

The maximum moments produced by this loading and beam configuration have been shown by Applied Engineering Resources, Inc., 1977 to be

$$M\ \text{center} = 0.0375\ WL^2 \qquad (3)$$

$$M\ \text{end} = 0.0667\ WL^2 \qquad (4)$$

It is assumed that failure of the span will occur when there is impending tensile stress in the lower fiber edge of the roof beam or when the roof beam fails at its ends in compression. Thus, when the maximum moment at the center of the beam is equal to the horizontal compressive stress (Mcenter = σ_H), tensile failure is impending. This assumes that joints have no tensile strength. For the beam in question, the stress in the outer fiber margin (exposed roof) is as follows.

$$\sigma_f = \frac{MC}{I} = \sigma_H$$

σ_c = bulk compressive strength of rock (psf)

σ_f = fiber stress (psf)

σ_H = Horizontal stress (psf)

M = Maximum moment at center

= .0375 WL^2

C = distance from neutral fiber (ft)

= h/2 ft

I = moment of inertia for beam section about neutral fiber

I = $\frac{bh^3}{12}$ (Higdon & Stilles "Mechanics of Materials")

b = Beam width = 1 ft

h = beam height

If the bulk unconfined compressive strength of the immediate roof has been exceeded at the ends of the span, it is assumed the roof will fail by compression.

In this case, a bulk unconfined strength of 5000 psi (720,000 psf) is assumed. Thus,

$$\sigma_c = \sigma_f + \sigma_h$$

$$\sigma_c = \frac{MC}{I} + 144\ H$$

The final results are that the maximum spans for failure are as follows:

Tensile failure: $L = \left(\frac{Hh^2}{(.2250\ (d^2))}\right)^{1/2}$ ft

Compressive failure: $L = \left(\frac{(5000-H)(h^2)}{(.4002\ (d^2))}\right)^{1/2}$ ft

The lesser of these two spans controls the minimum width required to propagate caving to a minimum height of four times the seam thickness. This caving height is required based on an assumed gob bulking factor of 1.25. A detailed explanation of the caving width equations appears in Appendix B.

Mechanical Constraints

There is no mechanical lower bound on the longwall face length which would practically apply. Physically, it would be impossible to operate a longwall shorter than about 75 ft because of the room required to snake the shearing machine into the coal. Such a face length is economically unjustifiable. On the other hand, there is a mechanical upper bound on longwall face length. This bound is produced by the working strength or breaking strength of the armoured flexible conveyor chains. Though the tension in the conveyor chains controls the maximum face length, such other considerations as head loss in fluid power distribution, haulage chain length, and the physical size of conveyor drives each have a face length beyond which current system technology in these areas becomes inadequate.

Head loss in the fluid power distribution system as well as reservoir capacity could become a face length constraint in the event that all pumps were located in the headgate entry. If, however, head loss or reservoir capacity became a problem, a pump station or reservoir could be installed in the middle of the face or in the tailgate entry in addition to the headgate pump and reservoir facilities.

Since shearing machine haulage has predominately changed from haulage chains to rack and pinion techniques, haulage chain tension and related safety problems have been eliminated from consideration.

The physical size of the conveyor drives could conceivably become a constraint on longwall face length. At present, the size of the drives for an 800-ft face poses no problems. As such, the face would have to be much longer than current or near future chain technology is likely to permit for the AFC drive size to be a limiting factor.

The best current estimate of the tension in an armoured face conveyor chain is incorporated in the Pennsylvania State University Longwall Simulation Model. The equation is as follows:

$$PO_{th} = \Psi\Psi'((L-X)(W_L+W_c)(CO_{th}\cos\alpha-\sin\alpha) + XW_c\ (\mu_c\cos\alpha-\sin\alpha))$$

$$P_u = LW_c\ (C_u\cos\alpha+\sin\alpha)\ \Psi$$

Where

PO_{th} = pulling force of upper chain strand with shearer travelling toward headgate (kg or lb)

Pu = Pulling force in lower chain (kg or lb)

Ψ = friction factor due to conveyor snake = 1.12

Ψ' = friction factor due to seam undulation = 1.10

L = length of conveyor (m or ft)

W = weight of conveyor load (kg/m or lb/ft)

X = length of empty conveyor after initial load has reached head drive = L Vs/Vc

V_s = shearing machine haulage speed (m/min or ft/min)

V_c = conveyor chain strand speed (m/min or ft/min)

W_c = weight of chain strand (kg/m or lb/ft)

C_o = friction factor of sliding coal on the panline $CO_{th} = CO_{tt} = .4$

α = angle of inclination of conveyor (deg)

μ_c = friction factor between chain strand and conveyor pans = 0.33

C_u = friction factor of lower chain strand =

Even though the above equation is the most complete estimate of chain tensions found, it is incomplete. To make this estimate complete, a term to account for the sliding friction of coal on the spill plates must be added. Britton (1980) advocates the use of the following term to account for spillplate contact.

$$\frac{h^2}{20} = \text{spill plate sliding friction (lb)}$$

Furthermore, information from chain conveyor manufactures (Long-Airdox) indicates that several constants for friction in the Pennsylvania State University Longwall Simulation Model are inappropriate. Long-Airdox recommends the following constants.

C_o = 0.4 for a moving coal pile

C_o = 0.60 for a stationary coal pile

μ_c = 0.33 for steel on steel

The increased friction factor for the chain strand seems reasonable because of minor conveyor damage. This increased value may also help account for such coal fines as might be carried back in the return compartment on the armoured flexible conveyor (AFC).

Prior to final discussion of the conveyor strand tension equation, some assumptions about the AFC should be noted. For purposes of this investigation, the following assumptions were made.

- The face conveyor width is 30 in.
- The AFC has minimum 12-in. high spill plates.
- The maximum operating load on the AFC is 106.67 lb/ft.
- The AFC chain strand is composed of two, 30x108mm chains at 16 lb/ft each.
- The chain strand speed is 250 ft/min.
- The breaking strength of a single AFC chain is $93d^2$ where d is the diameter of the link barrel in inches and the breaking strength is in tons.
- The endurance strength of an AFC chain is 18% of the breaking strength for cyclic loading applications.
- The longwall face is approximately level.

A twin inboard chain strand is preferred because it is probably the most satisfactory strand configuration available. Single chain strands are too weak for the longer faces currently being installed. Outboard chain-strand configurations have too much difficulty in dealing with snaking bends in the face; they also tend to hang-up in their races in the sides of the conveyor pans.

A restatement of the AFC chain tension equation follows:

$$PO_{th} = \Psi\Psi' \left((L-X)(W_L+W_c)(CO_{th}\cos\alpha-\sin\alpha) + XW_c\,(\mu_c\cos\alpha-\sin\alpha)\right)$$

$$P_u = LW_c\,(C_u\cos\alpha+\sin\alpha)\Psi$$

Though the PSU simulator does not use   in the lower chain tension, there is no reason why it should not apply. After incorporating this aspect, the new lower chain strand tension is as follows.

$$P_u = LW_c\,(C_u)\Psi\Psi'$$

The total chain strand tension is T (lb)

$$T = PO_{th} + P_u \qquad (5)$$

Though the chain is capable of withstanding 93 d^2 tons, if the chain stress exceeds two-thirds of the breaking stress, the zone of elastic deformation will be exceeded and the chain links will be permanently elongated. Such elongation would adversely affect the drive mechanism and, thus, must be avoided. Therefore, the maximum allowable load in any chain is 62 d^2 tons. Furthermore, to ensure the expected service life, of an AFC chain, the endurance strength, 18% of the breaking strength, must not be exceeded during normal continuous operation.

To calculate the maximum allowable AFC length, the type of drives utilized must be considered. A review of current literature indicates that electric motors may develop 2.5 times their rated torque when stalled. If the AFC is driven by an AC motor, the maximum torque is limited to approximately 2.5 times the rated torque. This maximum torque must not create forces which exceed the breaking strength of the chain strand.

For purposes of this analysis, it was assumed that the AFC drives are fluid coupled and that the AFC is loaded full length. Therefore,

Maximum Allowable Tension = 2.5 (T)

The worst case for breaking strength stress occurs if the AFC is fully loaded and hung-up on the carrying side of the AFC near the headgate drive.

Thus, solving equation (5) with the breaking strength parameters and stated assumptions for the maximum face length (L):

Maximum Allowable Tension = 2.5 (T)

Equating breaking strength and stated assumptions to the maximum allowable tension:

$$(2)\ 62\ d_2\ (2000) = 2.5\ (PO_{th} + Pu)$$

$$344603 = 2.5\ (PO_{th} + P_u)$$

$$137841 = (1.12)(1.10)(L)\ (106.67(.6)+(2)(.33)(32.0) + \frac{12.0^2}{20})$$

$$137841 = 1.232\ (L)\ (64.00 + 21.33 + 7.20)$$

$$137841 = 1.232\ (L)\ (92.53)$$

$$L = 1209 \text{ ft}$$

L is now the maximum face length allowed by breaking strength parameters.

However, it is possible that the endurance limit of the chains might control the maximum face length. Equating the endurance strength at the stated assumption of the strand to equation (5) reduced with the AFC operating parameters, the maximum face length by endurance criteria results.

$$93043 = (1.12)(1.10)\ L\ (106.67\ (.4) + 2\ (.33)(32.0) + \frac{12^2}{20})$$

$$93043 = 1.232(L)\ (42.668 + 21.33 + 7.20)$$

$$93043 = 1.232\ (L)(71.198)$$

$$L = 1061 \text{ ft for a fluid coupled drive}$$

Here, L is the maximum face length allowed by strand endurance strength parameters.

For the endurance strength of the AFC at maximum loading, the horsepower required is:

$$HP = \frac{(250)(93043)}{33000} = 705 \text{ horsepower}$$

AC motors composing this horsepower requirement would not be so large as to pose a physical size problem on a longwall operating in 60 in. or more thick coal.

CHAPTER V

LONGWALL MODEL DEVELOPMENT

Development Sequencing

In the longwall simulation model it is assumed that all longwall panels for each particular analysis are of identical length, width, and seam thickness over a maximum of 15 million tons. The capacity to simulate two and three entry development exists, as well as the ability to test various pillar dimensions.

The model is designed to simulate only fully retreating longwall designs. As such, continuous miner development is executed prior to face installation. The first panel is assumed to require continuous miner development on three sides, and subsequent panels are developed adjacent to the first panel with develoment being required on two boundaries. Upon completion of the development drivage for the first panel, the development section is moved to begin development of the second panel. Each time a panel is mined out after the second panel is developed, the development crew returns to develop the next adjacent panel. Providing that panel development is less time consuming than panel mining, the new panel is ready prior to abandonment of the current panel. If not, the longwall is forced to await completion of development drivage on the new panel. Delay times and costs are generated to allow for development waiting and longwall panel transfers. Maintenance and equipment rebuild costs are incurred as mining progresses as per a set maintenance schedule.

Provision is also made in the simulator to set a single row of cribs in the tail gate entry concurrent with panel mining.

Industrial Engineering Information

Information regarding crew size and event timing on longwall faces was obtained from the U.S. Department of Energy "Longwall Data Bank" (1981). Time study results pertaining to 95% of all U.S. longwall installations are contained in the data. Only data from faces operating in seams greater than 57 in. and cutting with the full-face technique was utilized in this investigation. Event times used in the simulator represent an average of several actual observations.

Longwall Moving Rates

The average longwall moving rate was found to be approximately 16 ft of linear face per three shift day. To move at this rate, a 42 person labor force over three shifts is required.

Mining Cycle Times

The cycle time for one full cut from headgate to tailgate and back may be calculated using the following relationship:

$$2\left(\frac{F-95}{V_m}\right) + 18.63 = \text{min/cycle}$$

Where

F = face length ft.

V_m = shearing machine cutting tram speed ft/minute

The 95 ft subtracted from the face length is the distance required to snake into the coal (approximately 15 shield widths plus the shearing machine length). The length of the shearing machine is approximated to be 20 ft.

The 18.63 minute term is a constant representing gate related delays. This equation may be derived logically when considering gate related delay time and the length of face available to be sheared at full shearing tram speed.

Cycle Production

The total tonnage mined in one cycle is

$$\text{Total tonnage} = \frac{2.0\text{x}F\text{x}SM\text{x}(30.0\text{ in./}12.0\text{ in./ft})\text{x }82.5\text{ lb/ft}^3}{2000\text{ lb/ton}}$$

$$\text{Total tonnage} = .20625\text{x}(F)\text{x}(SM)$$

Where

F = Face length, ft

SM = Seam height, ft

The cutting drum width on the shearing machine is arbitrarily fixed at 30 in.

Available Mining Time

Available mining time per shift is derived from the following assumptions.

Total Available Time:	480.00 min
Travel, One Way:	20.00 min
Average Delay Time:	192.10 min
Average Machine Prep:	6.24 min
Average Lunch Delay:	0.00 min
Available mining time	241.66 min

Longwall Productivity

As a result of the above information, the daily production from the longwall face for 2 shifts per day is:

$$\left(\frac{2.0 \text{x} 241.66}{2\left(\frac{F-95}{V_m}\right)+18.63}\right) \text{ x Total Tonnage = Ton/day raw coal.}$$

Finally, a management efficiency of 75% was applied to this tonnage to reflect deviations from the model such as failure to cut a full web and/or failure to cut at full shearing machine tram speed.

Manpower

Longwall production as outlined above requires a labor force of 31 people over one maintenance and two production shifts per day. The daily manpower schedule is as follows:

Production Supervisors	4
Maintenance Supervisors	1
Shearing Machine Operators	4
Support Movers	6
Headgate Operators	2
Mechanics (production)	2
Mechanics (maintenance)	4
General Labors	8
Total Personnel	31

It is assumed that longwall development and longwall production occur 240 days per calendar year.

Economic Evaluation

The economic evaluation portion of this program is based on the principle of net annual after-tax discounted cash flows. The exact format for calculating net annual cash flows is as follows:

Revenue
 - Royalty
Revenue less Royalty
 - Operating Costs
 - Severance Tax
 - OSM/Abandoned Lands
 - Property Tax
Net After Cost
 - Depreciation
Net After Depreciation
 - State Tax
Net After State Tax
 - Depletion
Taxable Income
 - Federal Tax
 - Minimum Tax
 + Tax Credits
Net Profit
 + Depreciation
 + Depletion
Operating Cash Flow
 - Capital Expenditure
 - Working Capital
Net Annual Cash Flow

The net cash flow is then discounted to the present at a fixed rate. The sum of all discounted net annual cash flows gives the net present value (NPV). The NPV is the variable selected for optimization (maximization) during longwall selection.

Each component in the cash flow analysis is discussed in the following section.

Revenue

The annual revenue generated from the longwall is assumed to be $15.00 per ton of material at the point of placement on the panel conveyor belt. It is assumed that costs out-by the panel conveyor are equal to the difference between the 15.00 price and a more realistic price, perhaps \$22.00 per ton. It is further assumed that there is no significant product dilution. The annual tonnage produced is a result of the longwall simulator.

Royalty

A production royalty of 8% of gross revenue is assumed.

Operating Costs

Annual operating costs are generated by the longwall simulator and based on the cost breakdowns detailed in Appendix C. Both continuous miner development costs, as well as longwall moving and production costs, are expensed annually.

OSM/Abandoned Lands

For bituminous coal mined underground, there is a \$0.15 per ton tax levied to assist in cleaning up and restoring previously disturbed and unreclaimed mining sites.

Property Tax

An ad valorem property tax is assumed which is assessed on both production and capital improvements. The appraised value for production is the greater of the net proceeds or 25% of the gross proceeds. In this case, gross proceeds are the revenue from production less administrative, crushing, and transportation costs. The deductible costs are assumed to total \$2.50 per ton. Net proceeds are the gross proceeds less production costs.

In the case of capital improvements, the book value of buildings and equipment, based on straight-line depreciation, is assessed at 30%.

The sum of the two assessed values is then multiplied by the mill levy, assumed to be 60 mils, to obtain the ad valorem property tax liability.

Depreciation

The depreciation schedule is based on the Accelerated Cost Recovery System (ACRS) brought into effect by the Economic Recovery Tax Act of 1981. Under this system, the mining equipment discussed in this model is subject to a five year recovery schedule. It should be noted, however, that the panel development equipment is assumed to have already been depreciated or to have its depreciation charged elsewhere. This too is true of the panel conveyor line. Furthermore, it is assumed that the equipment is placed in service between 1981 and 1984. The following depreciation schedule is then specified by the ACRS.

Years of Service	Recovered in Year
1	15%
2	22%
3	21%
4	21%
5	21%

Depletion

The depletion allowance is the greater of cost/unit depletion or statutory depletion. The statutory depletion allowance is the lesser of 50% of net after depreciation or the statutory percentage of gross income less royalty. For bituminous coal, the statatory percentage of gross income is 10%.

One intricacy is introduced where federal depletion is a deduction from state taxable income and vice versa. In the case of Colorado, the State assumed for this analysis, the depletion allowance must be determined by the following procedure.

State Tax = (0.0256) x Net after Depreciation

Depletion base on 50% Net Income = (Net after Depreciation-State Tax) x 50%

Statutory Percentage Depletion = (Gross Revenue-Royalty) x 10%

Selection of the lesser of these two depletion bases yields the claimed depletion. Cost depletion is not considered in the model because it is rarely the claimed depletion in cases such as the one under consideration.

State Income Tax

The state tax rate is assumed to be 5% of net income after depletion. In any year where there is a net loss, it is assumed that some profitable portion of the corporate entity will benefit from this tax benefit in the year it occurs. Thus, there are no loss carry forward provisions.

Severance Tax

The severance tax assumed is the tax levied by the State of Colorado. This tax is $0.30/ton on underground mined coal, and is assessed on all production beyond the first 8000 tons each quarter.

Federal Income Tax

The Federal income tax rate is 46% of the taxable income. As with the state income tax, there are no loss carry forward provisions. Further, it is assumed that the first $100,000 each year is earned by some other unit of the company and taxed at lesser rates.

Federal Minimum Tax

The Federal minimum tax is a 15% add-on tax levied on tax preference items. The minimum tax base is the total of the taxpayer's tax preferences reduced by the "greater of (1) $10,000 or (2) the full amount of the taxpayer's income tax liability" (Gentry and O'Neil, 1983). The tax liability includes any investment tax credits. Preference items considered for the purposes of longwall evaluation are excess depreciation on real property and depletion.

Excess depreciation is depreciation on real property claimed beyond such depreciation as might have been claimed under the straight-line method. However there is no real property in this analysis.

Excess depletion is that depletion claimed beyond the unadjusted basis of the property at the beginning of the year. For purposes of this discussion, the entire depletion claimed is used as a tax preference item.

Investment Tax Credits

Under the Economic Recovery Tax Act of 1981, the fraction of 5 year property qualifying for the ITC is 100%. The amount of ITC which can be claimed in one year is the lesser of the tax liability or $25,000 + 85% x (tax liability over 25,000). The ITC may be carried back 3 years or forward 15 years. In the model, provisions have been made for ITC carry forward.

Capital Expenditures

Capital expenditures are made based on manufacturers' estimates of current purchase costs of equipment. Also considered are the estimated rebuild costs to attain equipment life and productivity goals. These capital expenditures are then depreciated as described in the section on depreciation.

Working Capital

The initial working capital, estimated to be 25% of the total average annual revenue, is recaptured in equal installments during the first ten years. It is assumed that the project itself provides working capital after the initial working capital is recaptured. Because the parent organization receives the benefits of the longwall project, no interest is charged on working capital.

Present Value Calculation

The net cash flow is discounted to time zero at a nominal discount rate of 16% annually. This discount rate approximates current hurdle rates applied by coal producers in project evaluations.

It is important to note that the tax codes utilized in this document have been superceeded by the Tax Equity and Financial Responsibility Act (TEFRA) codes. The now outdated code was utilized because the simulation routine was under development prior to the TEFRA standards. Updating would have required a great deal of modification. Initial tests run to evaluate the significance of using the outdated tax codes compared to the TEFRA standards show that the TEFRA generated NPV is within 3 percent of the simulator generated project NPV. As such, efforts to revise the tax computations of the simulator to conform to TEFRA standards were deemed unnecessary. Future revisions of state or federal tax codes may require modification of the simulator to maintain acceptable accuracy in comparison to the 10% or better accuracy involved in other areas of the simulation program.

In conclusion, it is necessary to note that an implicit assumption is made that the project in question is associated with one division of a multi-division profitable corporation. This assumption validates the assumptions regarding taxation.

Test Case

A test case modeling a realistic coal mine project scenario is presented in the following sections along with the resulting output. This test case serves to demonstrate the utility of the computer longwall simulator.

The parameters utilized in the test case are as follows:

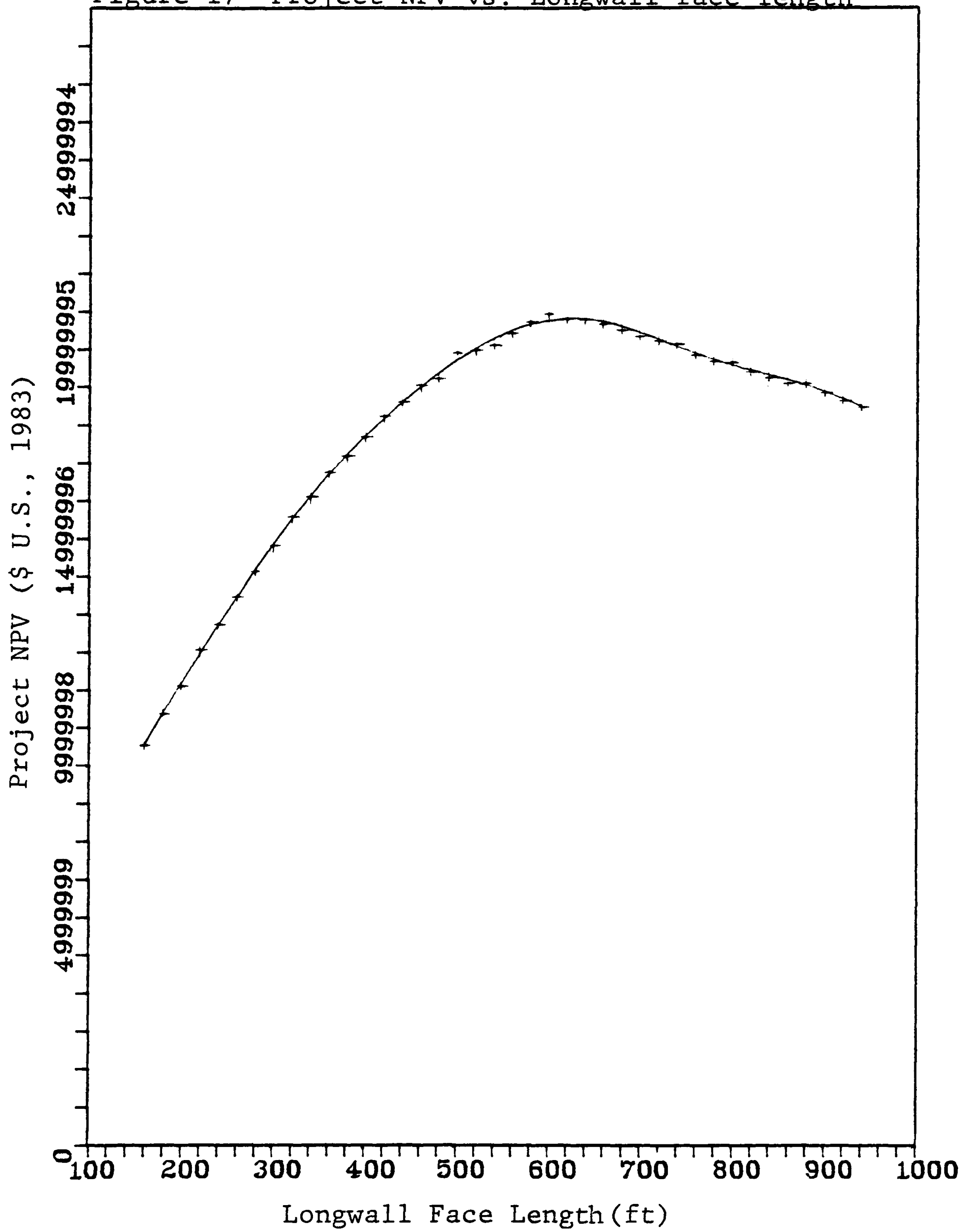
Seam Depth: 1000 ft
Thickness of the most competent overlying bed within four seam heights: 15 ft
Length of gob overhang beyond the coal face: 23.7 ft
Shield type: 2-leg lemniscate shield
Seam Extraction Height: 10.0 ft
Longwall Panel Length: 4000 ft
Panel Development System: 3-entry.
Continuous Miner Development Productivity: 350 tons/shift
Development Entry Pillar Dimensions: 80 ft x 40 ft
Development Entry Width: 20 ft

Results of the longwall simulator are presented in Figure 17. In the figure, project NPV is plotted versus longwall face length. Additional simulator output in the form of economic and design results are presented in Appendix D.

The results of Figure 17 are rather obvious. The NPV rises with increasing face length prior to reaching the economic maximum value. During this period, several factors are interacting. At short face lengths, (approximately 100-275 ft) the time required to develop a panel may exceed the time required to mine out an identical panel. This circumstance is costly because of lost productive time and waiting or delay costs. Also, shorter faces have a higher percentage of high cost development production to total production; they also have higher fixed costs per ton mined in the panel because of the increased number of inter-panel equipment moves required. These two factors tend to decrease the economic benefits of shorter faces compared to longer faces. Furthermore, longer longwall faces are more productive and thus have lower production costs than shorter faces.

At face lengths beyond the economic optimum face length, other factors work to decrease the economic benefits of longer longwall faces. Beyond the minimum amount of face equipment, increases in face length increase

Figure 17 Project NPV vs. Longwall face length



capital costs of equipment and maintenance with a near linear trend. As such, the marginal gain in productivity, and thus NPV, is exceeded by increased capital costs. Furthermore, the realization time of initial cash flows is more lengthy than for shorter faces, thus decreasing the present value of comparable cash flows.

The above stated factors all contribute to a fairly broad optimal range of face lengths. The optimal range of face length may be on the order of 100 to 200 ft, within a tolerance of $500,000 in NPV below the optimum. In general, it is preferable to have longwall faces slightly longer than the optimum face length rather than slightly shorter than the optimum face length. The reason for this statement is that the rate of change in NPV with changing face length is greater for sub-optimal face lengths than post-optimal face lengths.

For this test case, the optimum face length is 600 ft with an optimum project NPV of $21.9 million dollars (1983 U.S.). The range of optimal face lengths within $500,000 of the optimum NPV is approximately 560 ft to 690 ft.

It must be recognized that operator certainty in such variable factors as continuous miner development productivity will temper face length selection. For example, if the development productivity were over estimated, the optimum face length would increase and the optimum NPV would decrease. As such, choice of a slightly sub-optimal face length might bring the final outcome as close to optimality as possible under the prevailing conditions of uncertainty.

CHAPTER VI

GENERALIZED RESULTS

The results presented in this section are attributable to analyses of variations of a base case. Results which isolate specific variables for analysis are based on departures of the variable in question from the base case conditions.

The base case parameters are as follows:

Seam Depth: 1000 ft
Thickness of the most competent overlying bed in four seam thicknesses: 15 ft
Gob overhang beyond coal face: 23.7 ft (10 ft beyond rear of shield roof canopy)
Shield type: 2-leg
Seam Extraction Height: 10.0 ft
Panel length: 4000 ft
Number of Panel Development Entries: 3
Continuous Miner Productivity for Entry Development: 400 ton/shift
Development entry pillar dimensions: 80 ft x 40 ft
Development entry width: 16 ft
Project life: 15 million tons or 30 years, whichever comes first

The effects of seam extraction height, number of shield legs, development productivity, development entry width, and longwall panel length are presented in the following sections.

<u>Variation of Seam Height</u>

Figures 18 and 19, respectively, present the effects of varying seam height and development productivity on

Figure 18 Seam Height vs. Optimum Longwall Face Length for 3-entry Panel Development

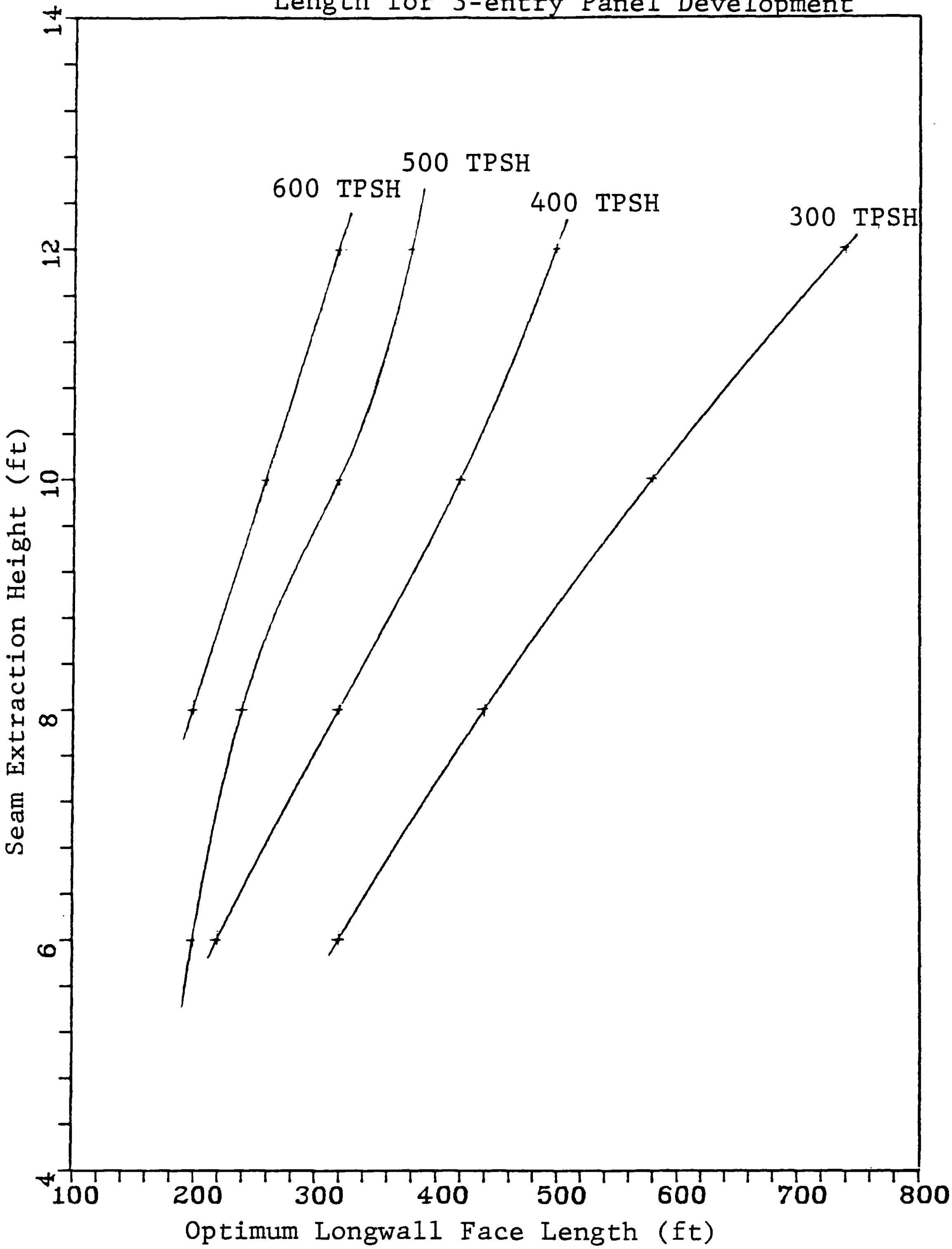
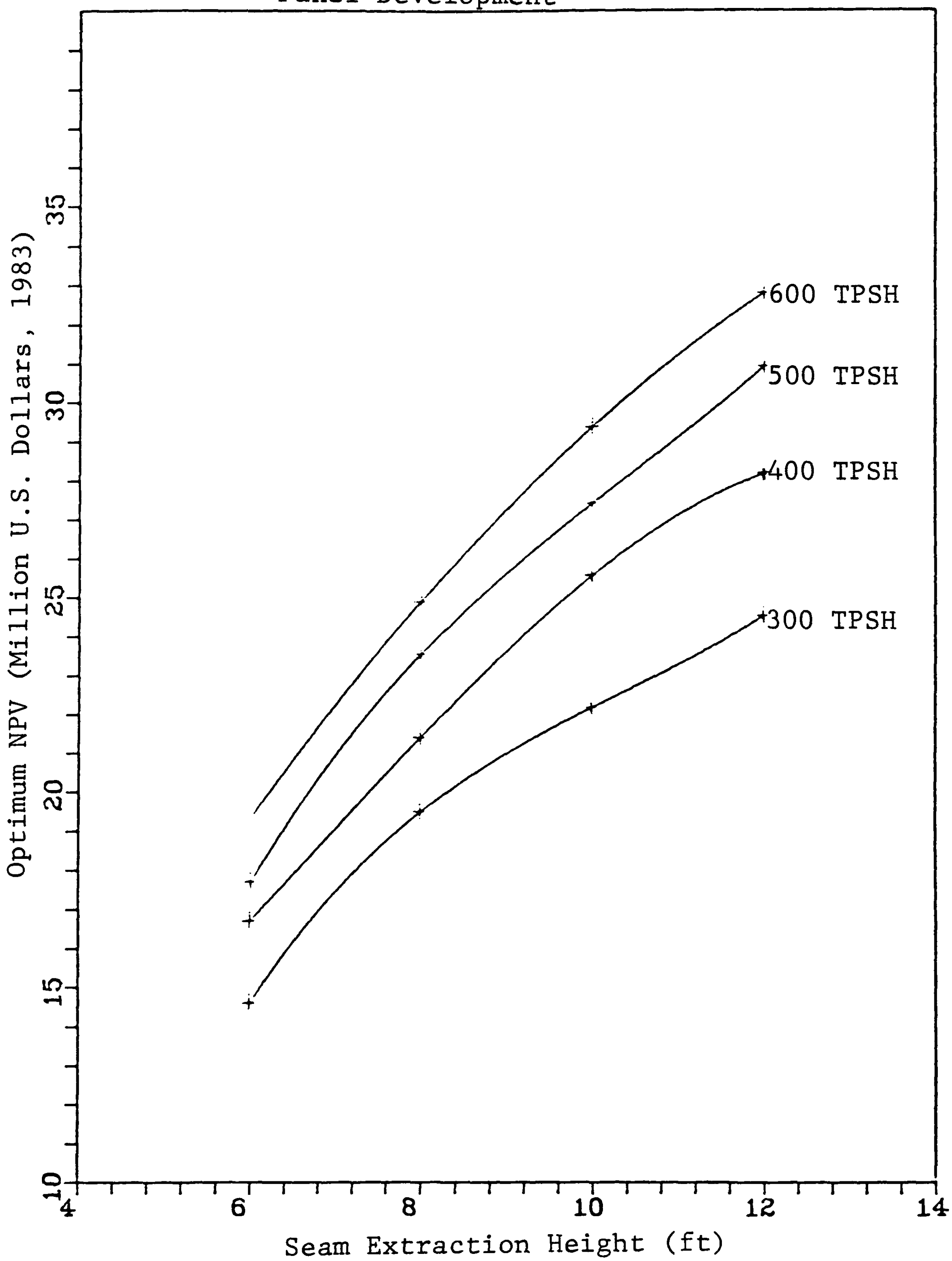


Figure 19 Optimum NPV vs. Seam Height for 3-entry Panel Development



optimum longwall face length and optimum net present value for 3-entry development systems. Similarly, figures 20 and 21 present the results of varying seam height and development productivity on optimum face length and net present value for 2-entry development systems. In both cases, increasing the development productivity decreases the optimum face length and increases the net present value of the project. The rate of decrease of optimum face length and increase in NPV decreases as the development productivity rises. Eventually, as the development productivity rises, the optimum longwall face length falls to the lower bounding condition. At that continuous miner productivity it is no longer desirable to operate a longwall face in comparison to continuous miner room-and-pillar techniques -- assuming operating conditions permit.

<u>Variation in Number of Development Entries</u>

Figures 22 and 23 display the aforementioned results for 2-entry and 3-entry development on the same plot.

It is clear that optimum face lengths for three entry development, as compared to two entry development for the same development productivity, may be several hundred feet longer in thick seams. The effects are lessened in thinner seams, however. The NPV of projects with 3-entry development, as compared to identical projects with 2-entry

Figure 20 Seam Height vs. Optimum Longwall Face Length for 2 two-entry Panel Development

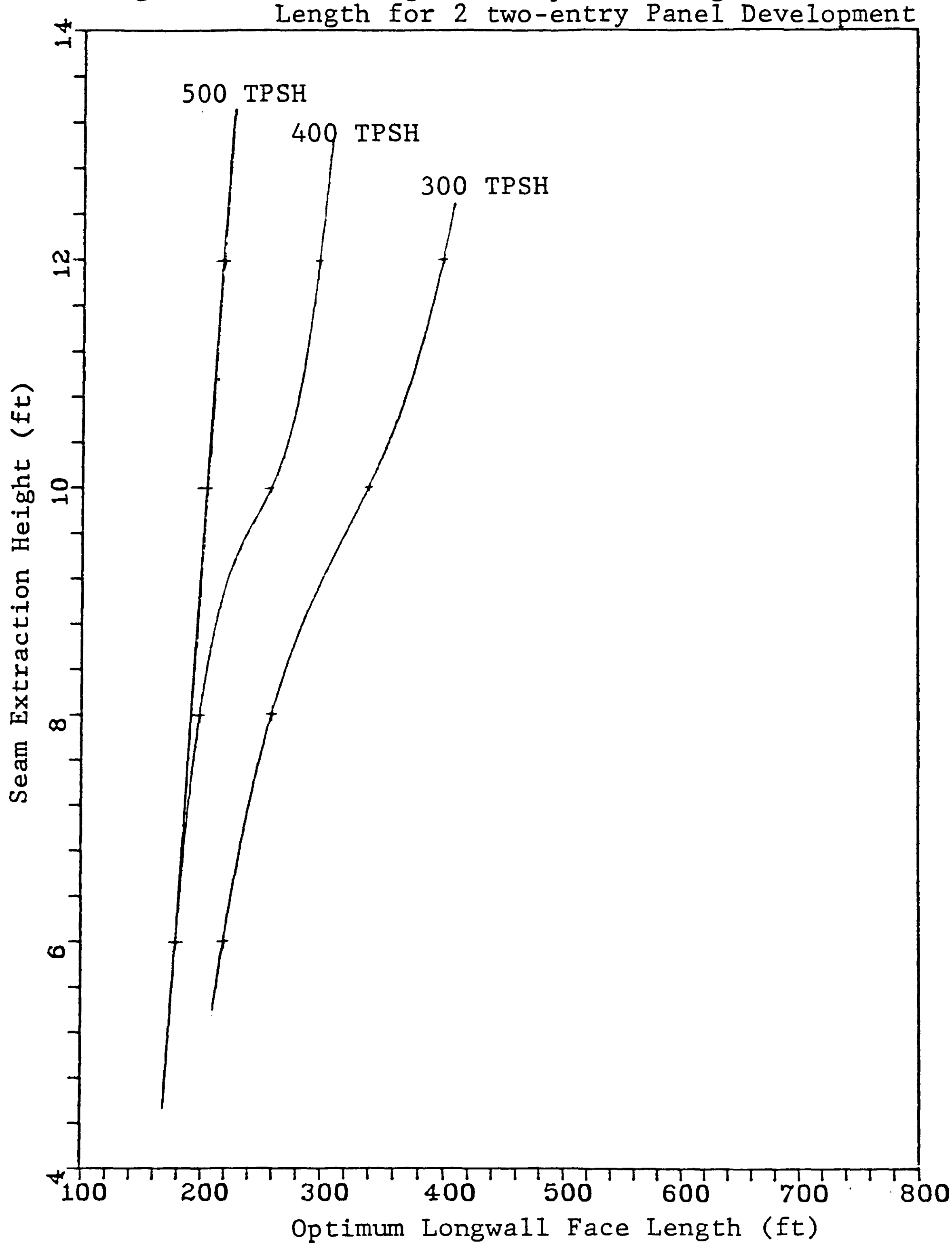


Figure 21 Optimum NPV vs. Seam Height for 2-entry Development

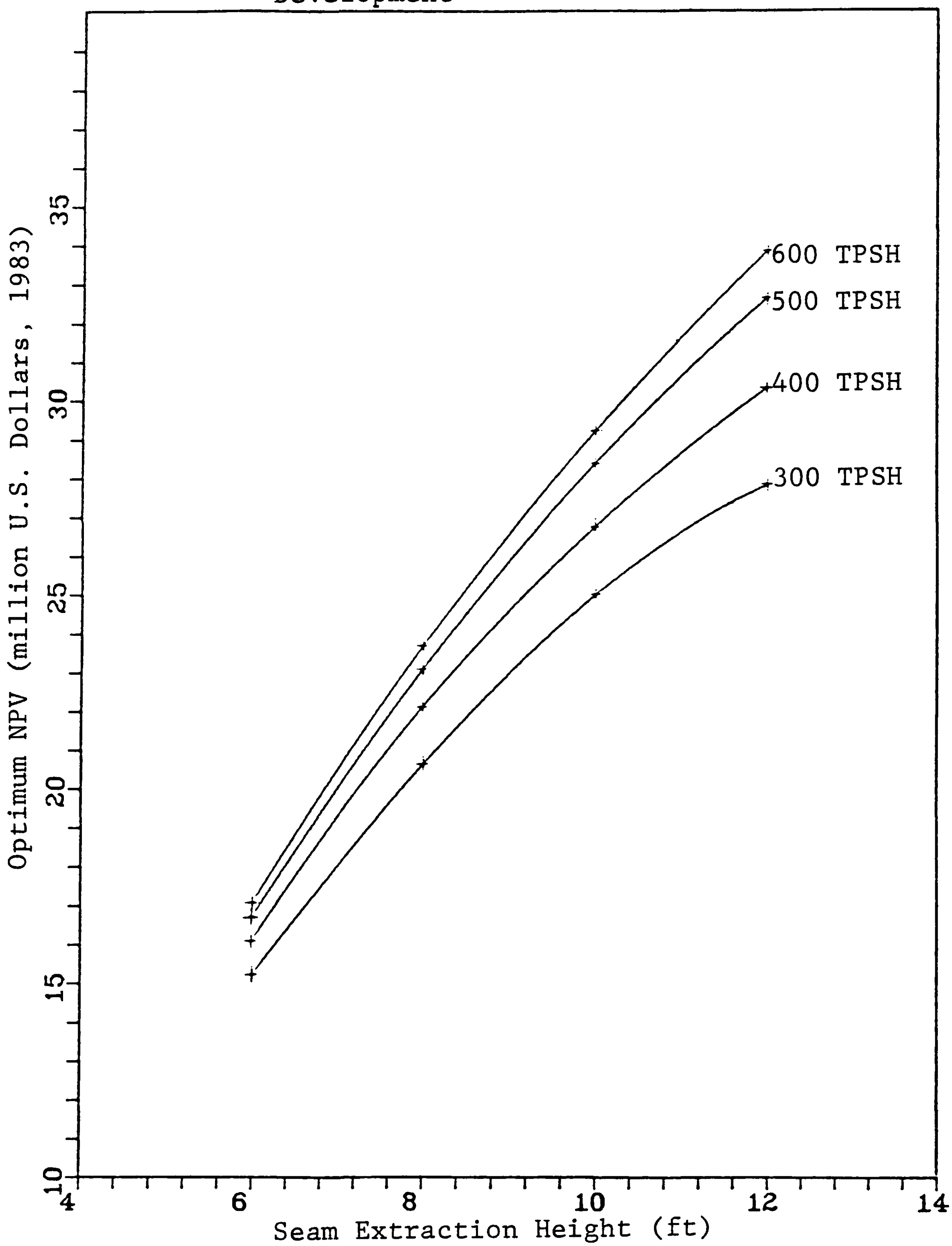


Figure 22 Seam Height vs. Optimum Longwall Face Length

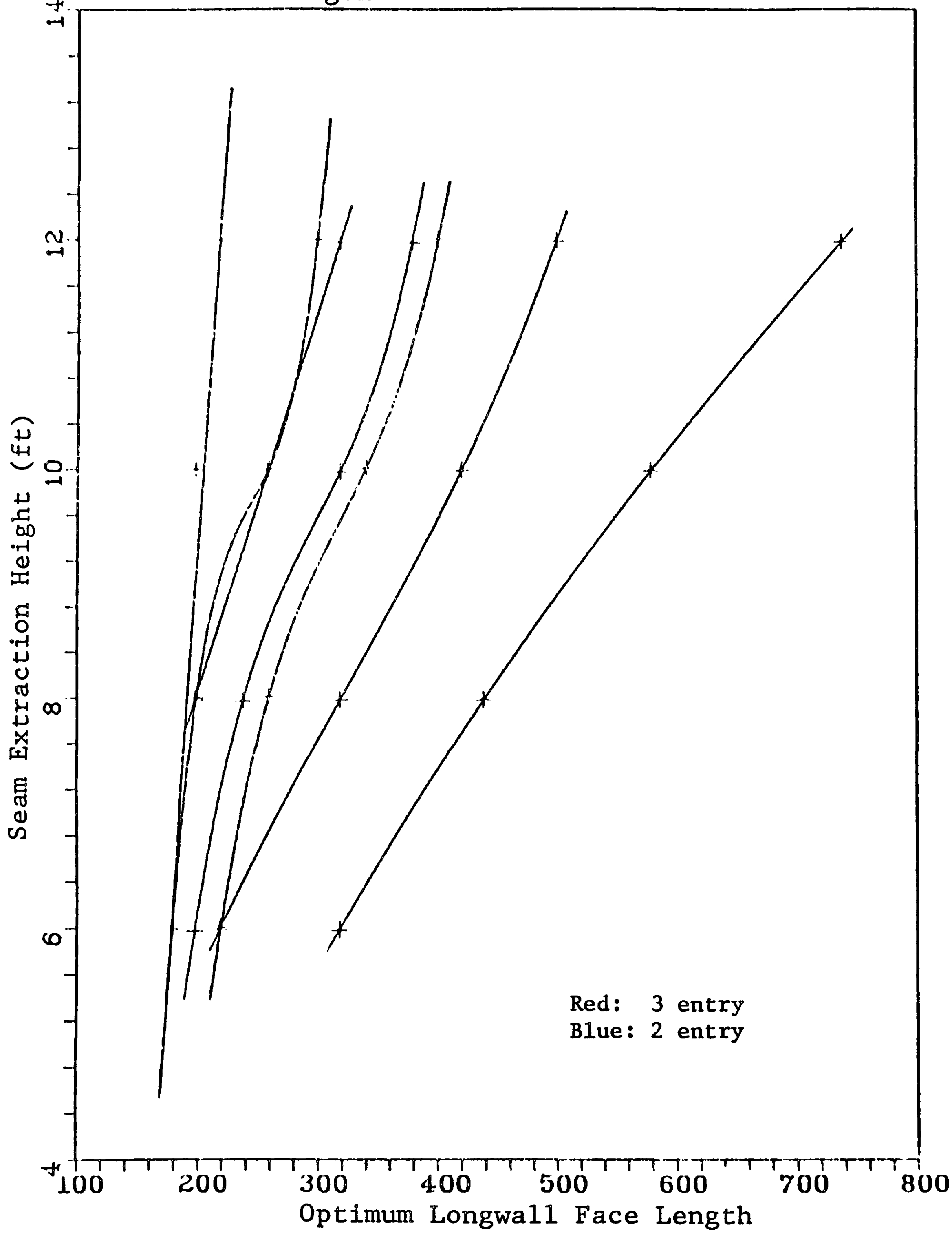
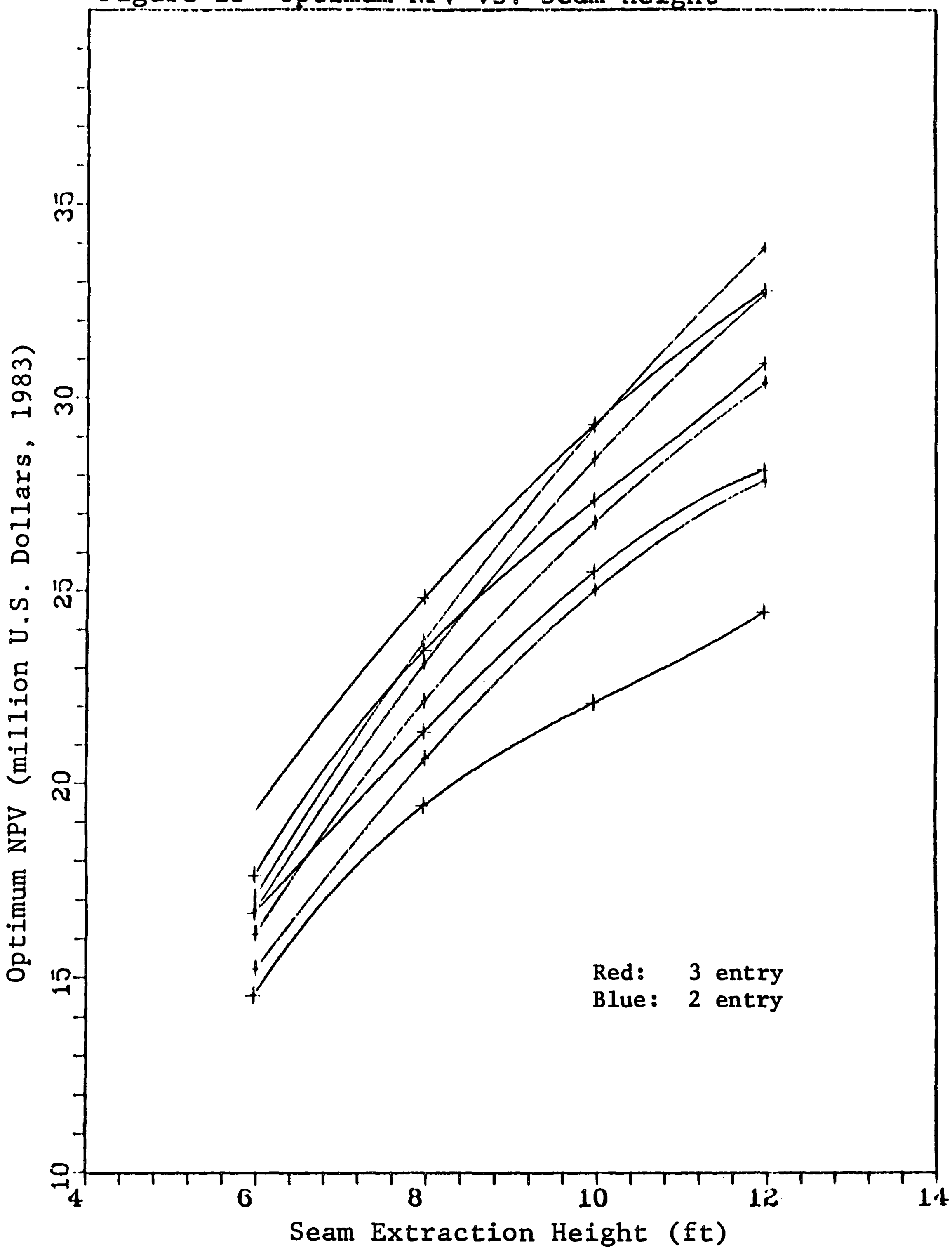


Figure 23 Optimum NPV vs. Seam Height



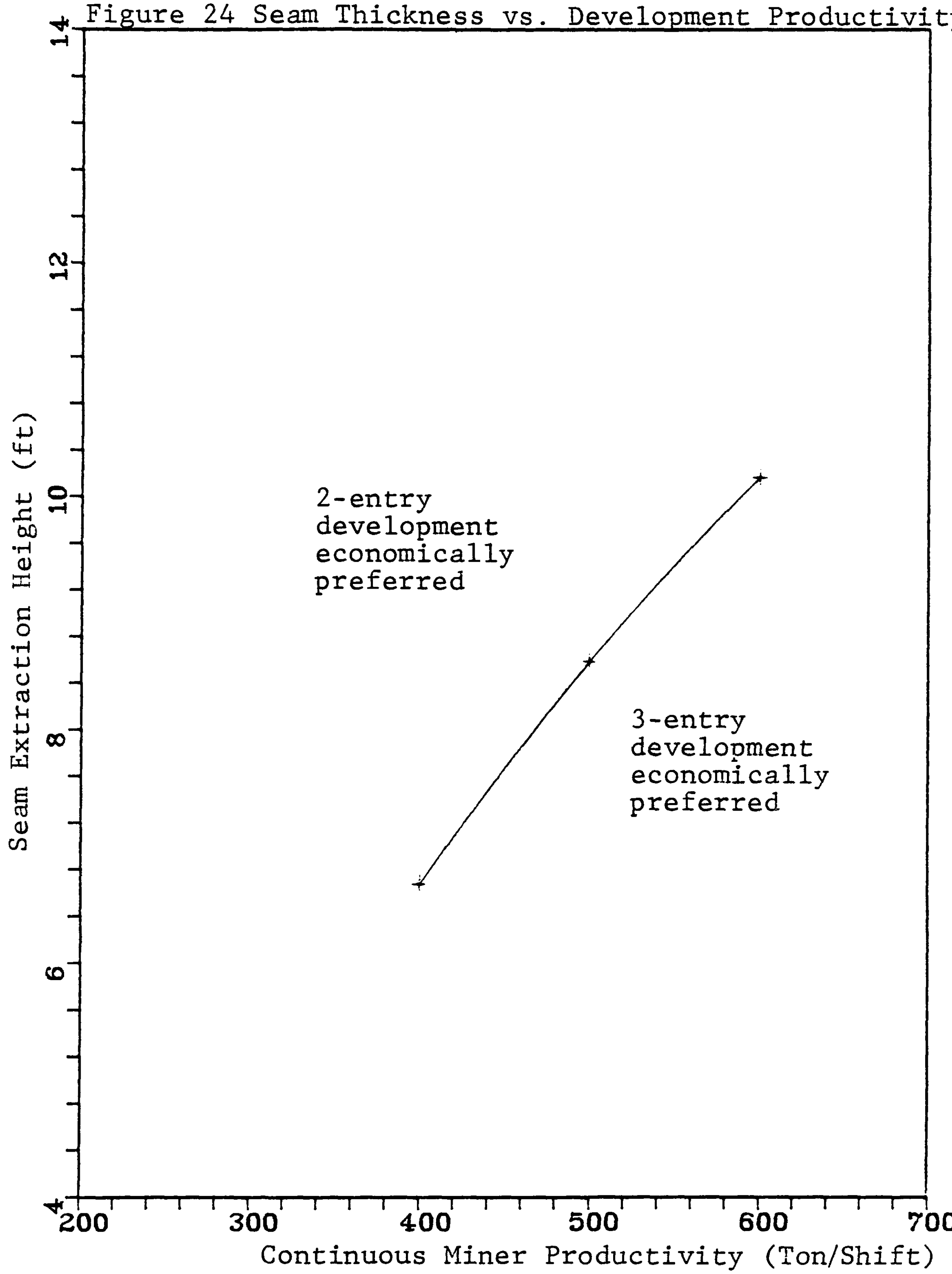
development, may vary as much as $3.5 million dollars at lower development productivities and lessens to near $1 million dollars at very high development productivities.

These results are reasonable when consideration is given to the ramifications of varying seam thickness and development productivity.

Since maximization of the project NPV is the objective in this analysis, it is clear that producing the minimum amount of development coal at the maximum possible rate is desirable. This conclusion is founded on the premise that the cost of development product is greater than the cost of longwall mined product. This is particularly true if the development product is more costly than its unit value on the panel or section conveyor belt. Minimizing the amount of development coal produced minimizes the average cost of production throughout the project and thus maximizes the NPV. For this reason, three entry development generally requires longer longwall faces with resulting lower average production costs than 2-entry developed systems.

In addition, it would appear that below a specific seam height 3-entry development is economically preferable to 2-entry development. Above that seam extraction height, the situation is reversed. Figure 24 presents these results. Economically, the line plotted on Figure 24 is

Figure 24 Seam Thickness vs. Development Productivity



the point where continuous miner produced coal has a production cost equivalent to the longwall coal production cost. However, this is not to be confused with the point where a longwall is no longer economically desirable. Because of the assumptions applied regarding continuous development costs, the tonnage where the breakover occurs is at a lower continuous miner productivity than the continuous miner production room-and-pillar/longwall production breakeven production cost. This discrepancy results from assumptions about continuous miner section depreciation, manpower and overhead costs. Thus, continuous miner productivity would have to increase to be equivalent to the longwall production costs.

Increasing seam height and productivity have the effect of lowering the fixed cost component of the total development production cost. Thus, increasing these variables has the effect of shortening the optimum face length necessary to minimize the average production cost over the project life. Therefore, at a fixed productivity, 2-entry systems are developed more rapidly at lower total production of development coal than 3-entry systems of comparable geometry and thereby require shorter longwall faces to minimize the overall coal production cost. Increasing the seam thickness also has the effect of lengthening the optimum longwall face length and raising the project NPV. This occurs because the longwall productivity is increased as the face length increases and the exposed face area per unit face length increases. Thus the overall production cost is minimized thereby maximizing the project NPV.

Variation in Shield Style

Figures 25 & 26 display the results of using 2-leg and 4-leg shields on optimum longwall face length and optimum project NPV. Within the accuracy of the analysis, there is no effect of shield design on optimum face length. Obviously, however, the slightly lower NPV of a 4-leg face (attributable to higher capital costs) requires that the optimum face length be slightly longer to raise productivity and lower the overall production cost. The lengthening involved is less than 20 ft under the conditions of the base case and varying seam heights investigated. The lower NPV associated with a 4-leg shield face is also negligible, being under $200,000 throughout the range of seam heights. investigated.

Variation of Continuous Miner Productivity

Figures 27 & 28 show the effects of varying continous miner development productivity on optimum longwall face length and optimum NPV, respectively. As the development productivity decreases, the optimum longwall face length

Figure 25 Seam Height vs. Optimum Longwall Face Length for 2-leg and 4-leg shields

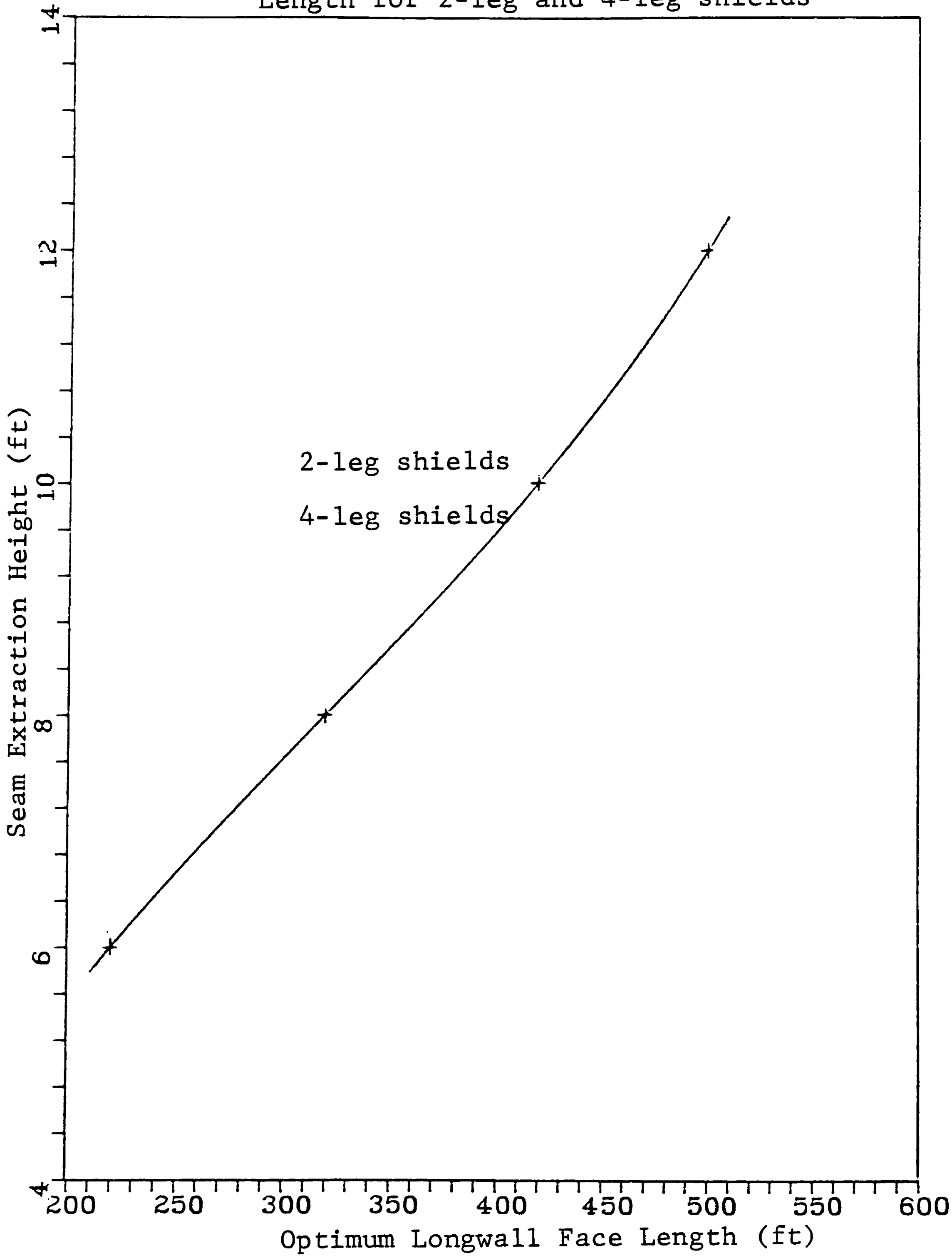


Figure 26 Optimum vs. Seam Height for 2-leg and 4-leg Shields

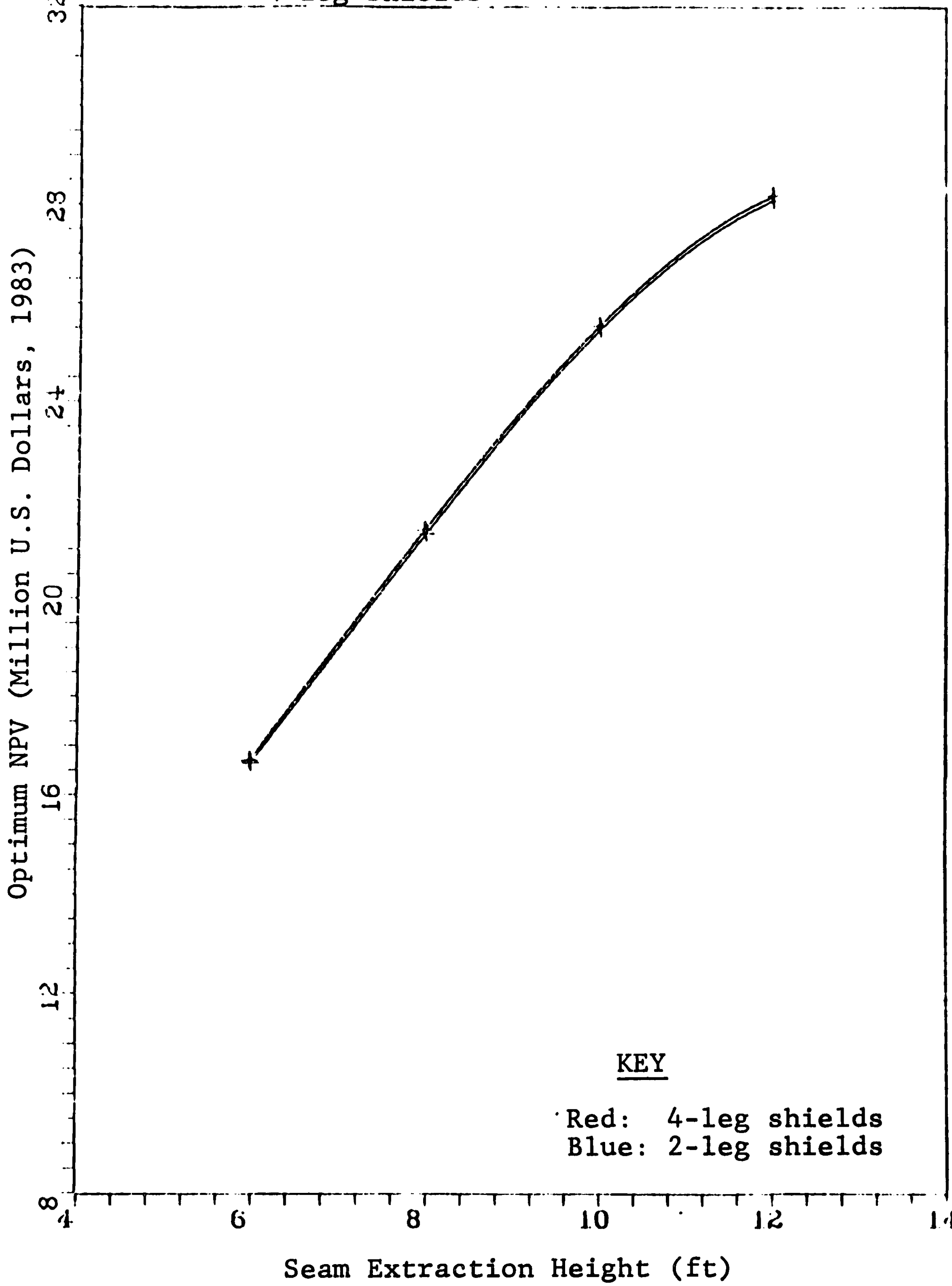


Figure 27 Continuous Miner Development Productivity vs. Optimum Longwall Face Length

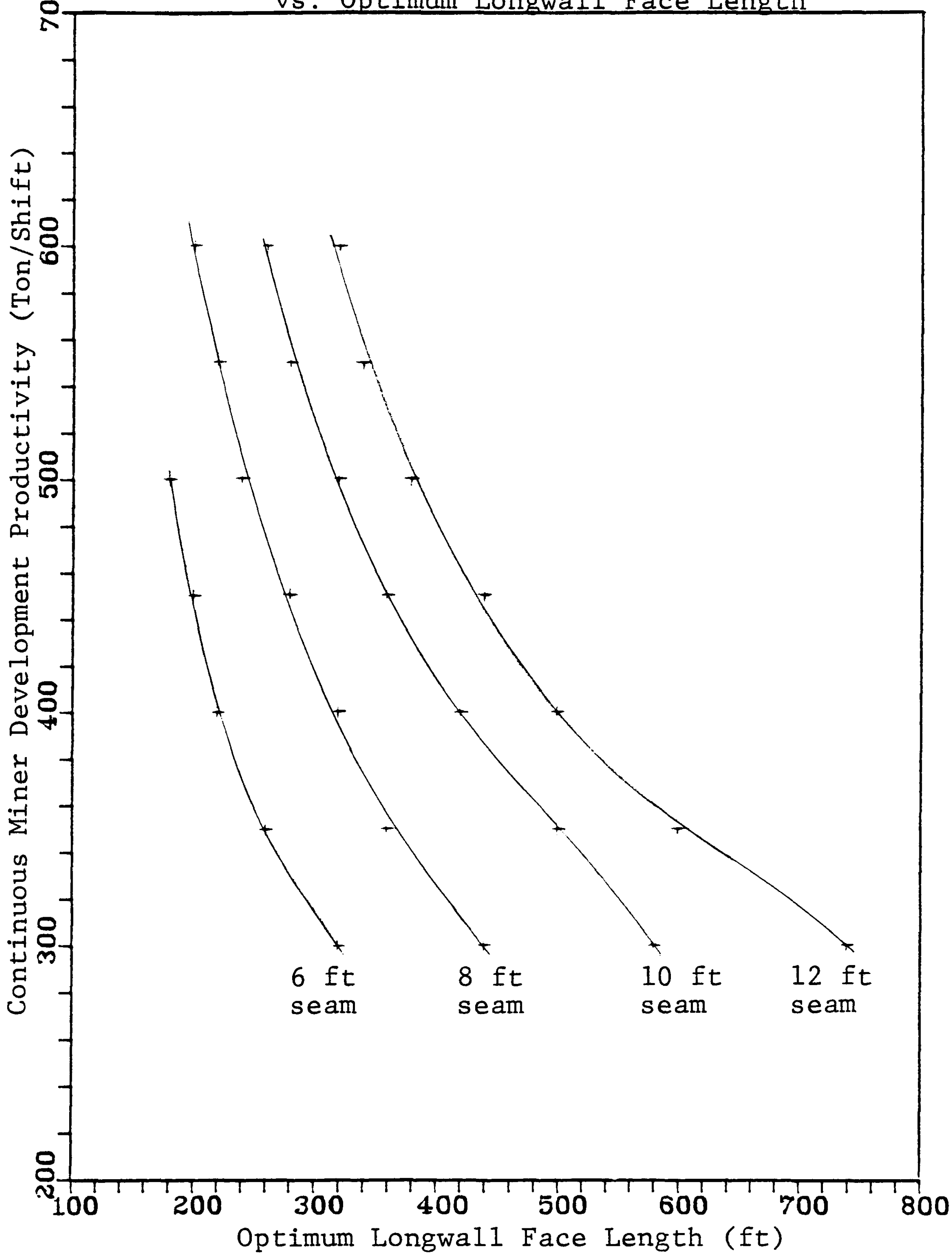
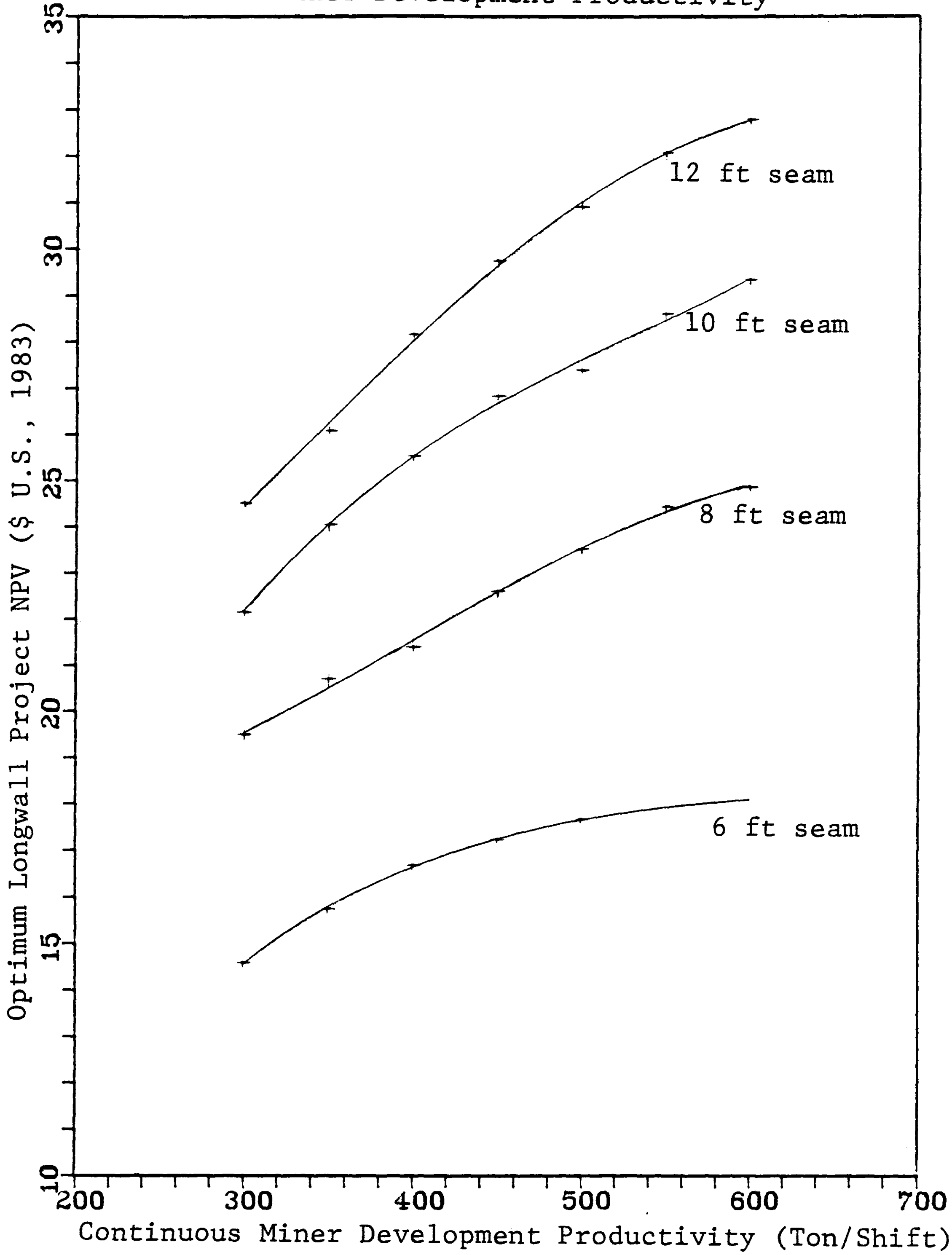


Figure 28 Optimum Project NPV vs. Continuous Miner Development Productivity



increases and the optimum NPV decreases.

The increase in optimum longwall face length results from a need for lower cost production coal to balance out the higher cost development production. In addition, the face length where panel mining time is equivalent to panel development time increases. Also, the length of time required to realize initial revenues increases, thus making the NPV of equivalent initial cash flows less.

The optimum NPV decreases with decreasing development productivity. As above, the NPV decreases because of increasing average production costs and cash flow timing. Equipment capital costs also increase with increased face length and adversely affect the NPV.

Variation of Development Entry Width

Figures 29 & 30 show the effects of variation of the panel development entry width through the range of seam extraction heights on optimum face length and optimum NPV, respectively. The optimum longwall face length increases as the entry width is increased and the optimum NPV decreases with increasing entry width. The optimum longwall face length and optimum NPV for a given entry width increase with increasing seam height.

Such responses to variation in entry width are easily explained. Increasing the development entry width requires

Figure 29 Development Entry Width vs. Optimum Longwall Face Length for 3-entry Development

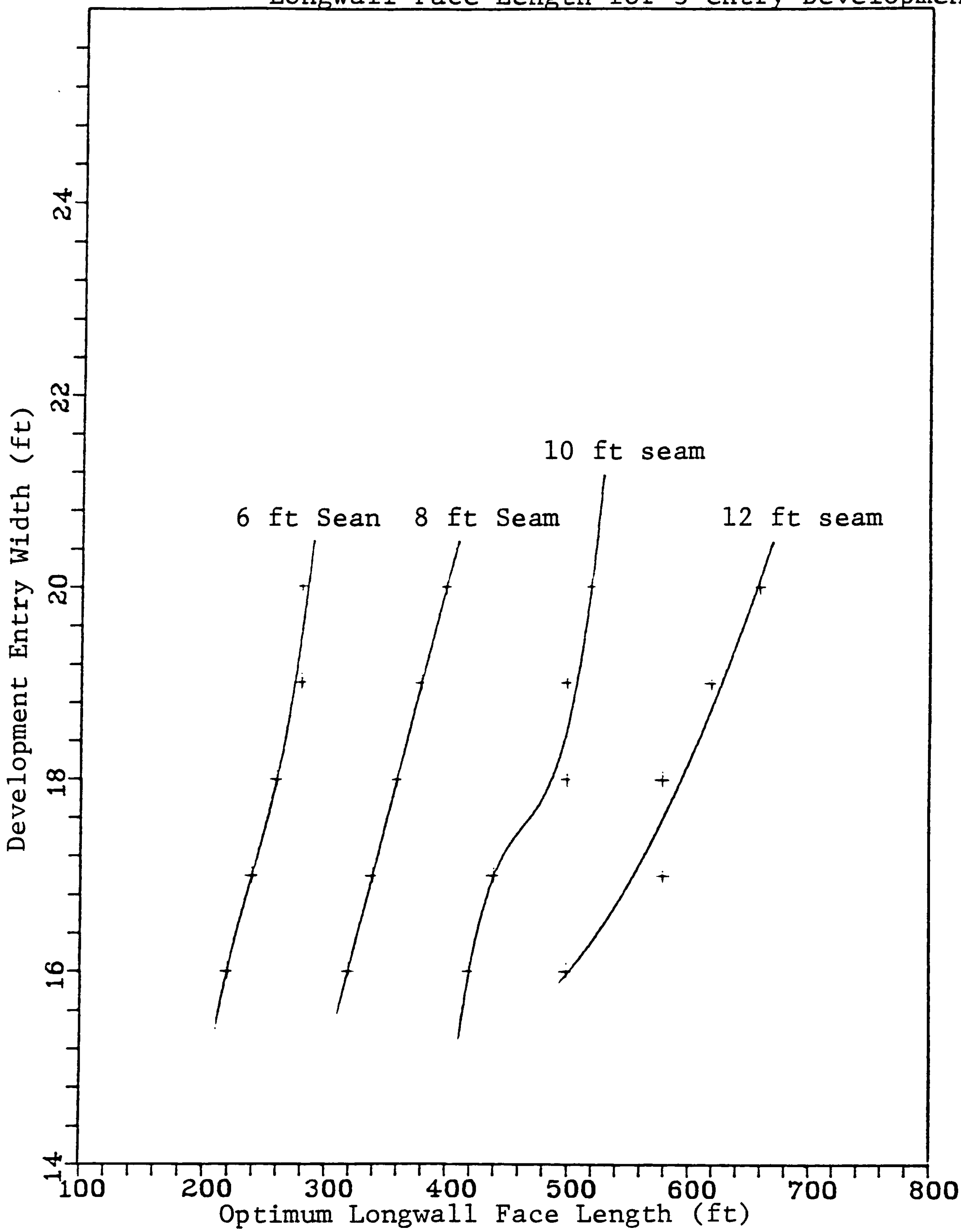
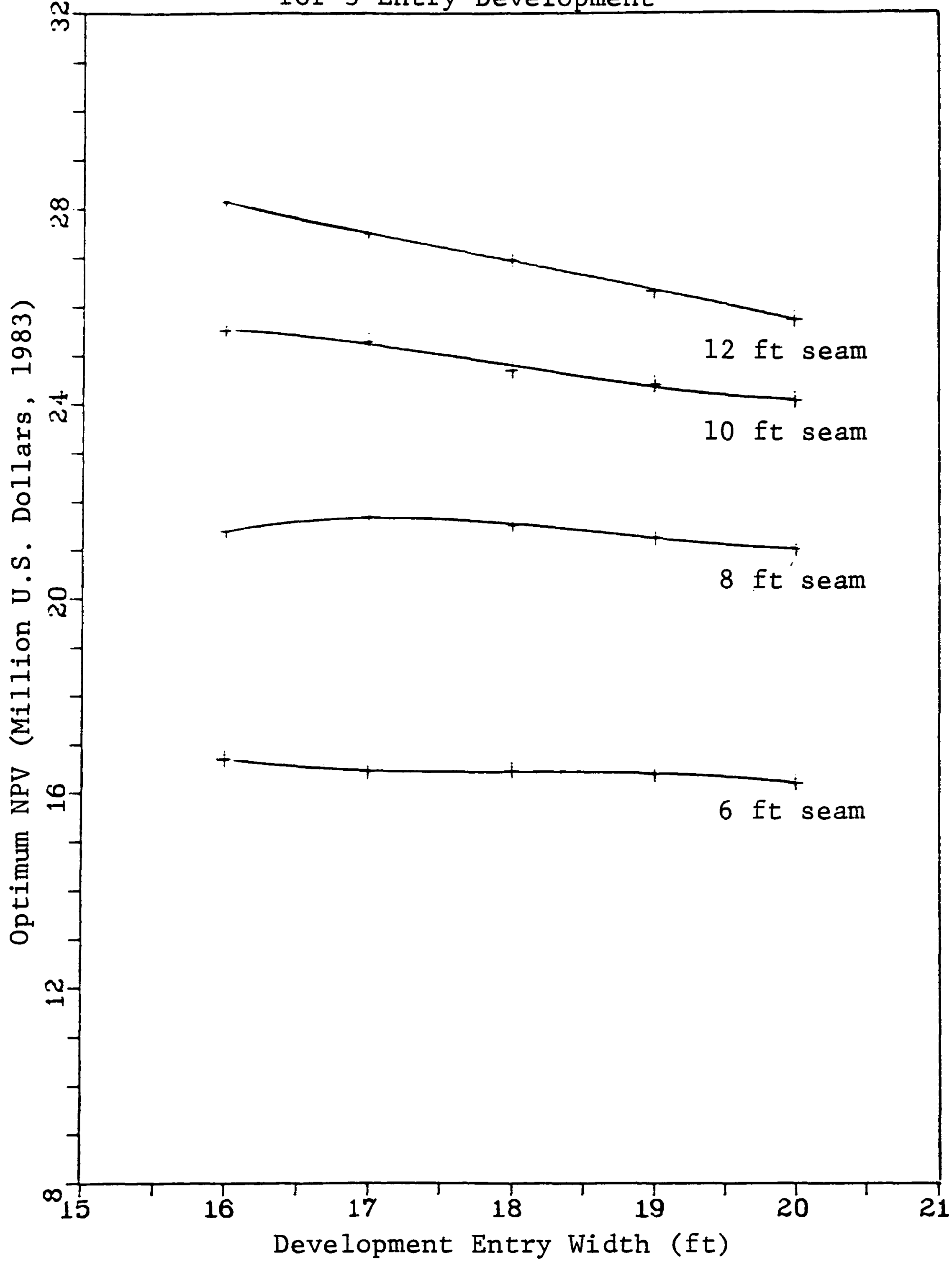


Figure 30 Optimum NPV vs. Development Entry Width for 3-Entry Development



the extraction of increasing quantitites of high cost development coal and requires longer development times at a fixed development productivity. These elements combine to create a need for longer longwall faces with higher productivities to offset the high cost development coal. The optimum NPV consequently decreases as the overall average coal production cost is increased, and the marginal differential between cost and revenue decreases.

At constant development entry width, increases in seam thickness has the effect of lengthening the optimum longwall face length and raising the optimum project NPV. This occurs because the longwall productivity is increased as the face length increases and the exposed area of coal on the face increases per unit face length. Thus, overall production cost is minimized, thereby maximizing optimum project NPV.

Variation of Panel Length

Finally, Figures 31 & 32 show the effects of variation of longwall panel length on optimum longwall face length and optimum NPV through the seam height range analyzed. The results of this section are more difficult to explain than are those in the previous sections. In general, there is little variation in optimum longwall face length or optimum project NPV with varying longwall panel length. For

Figure 31 Panel Length vs. Optimum Longwall Face Length

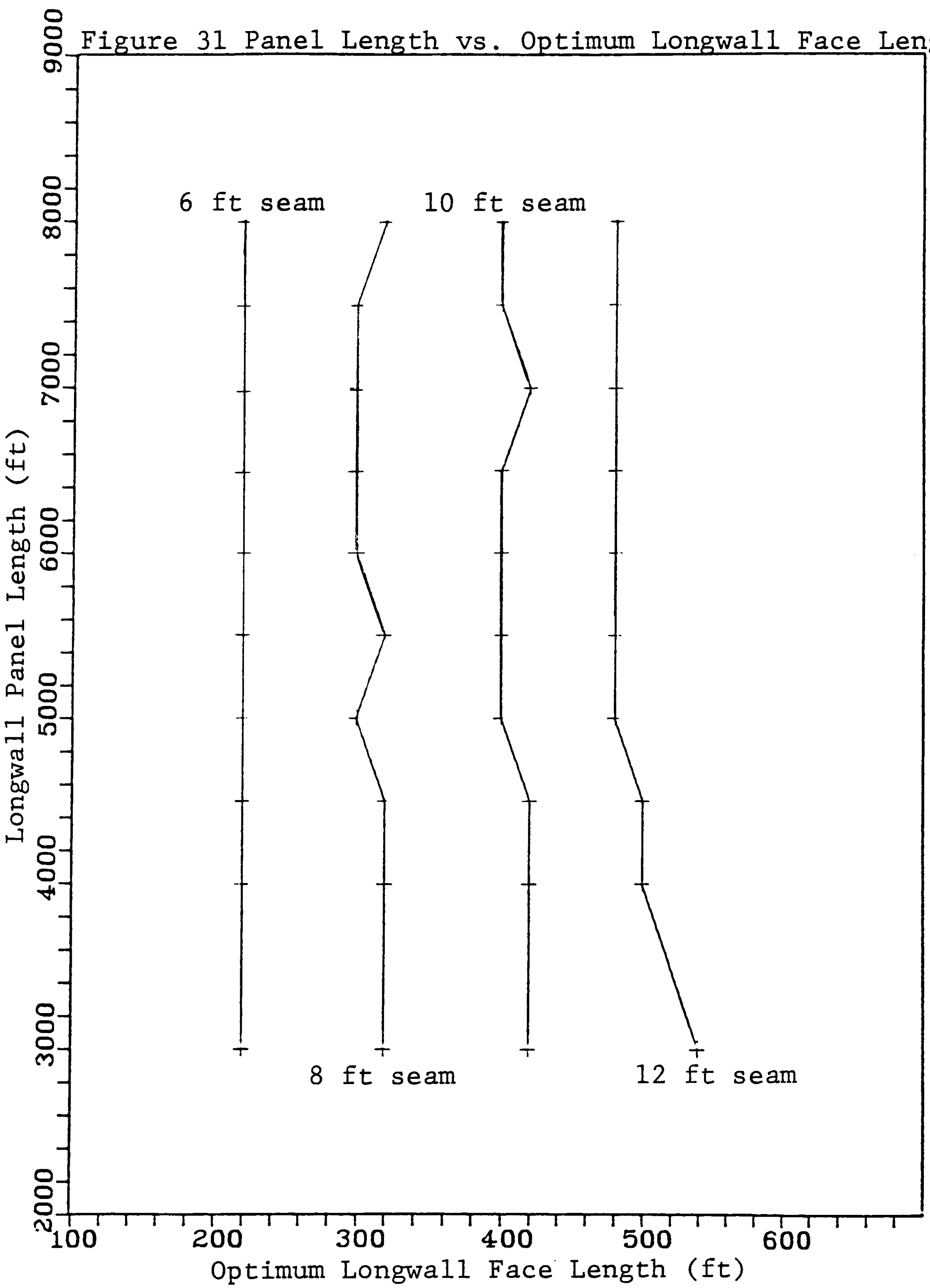
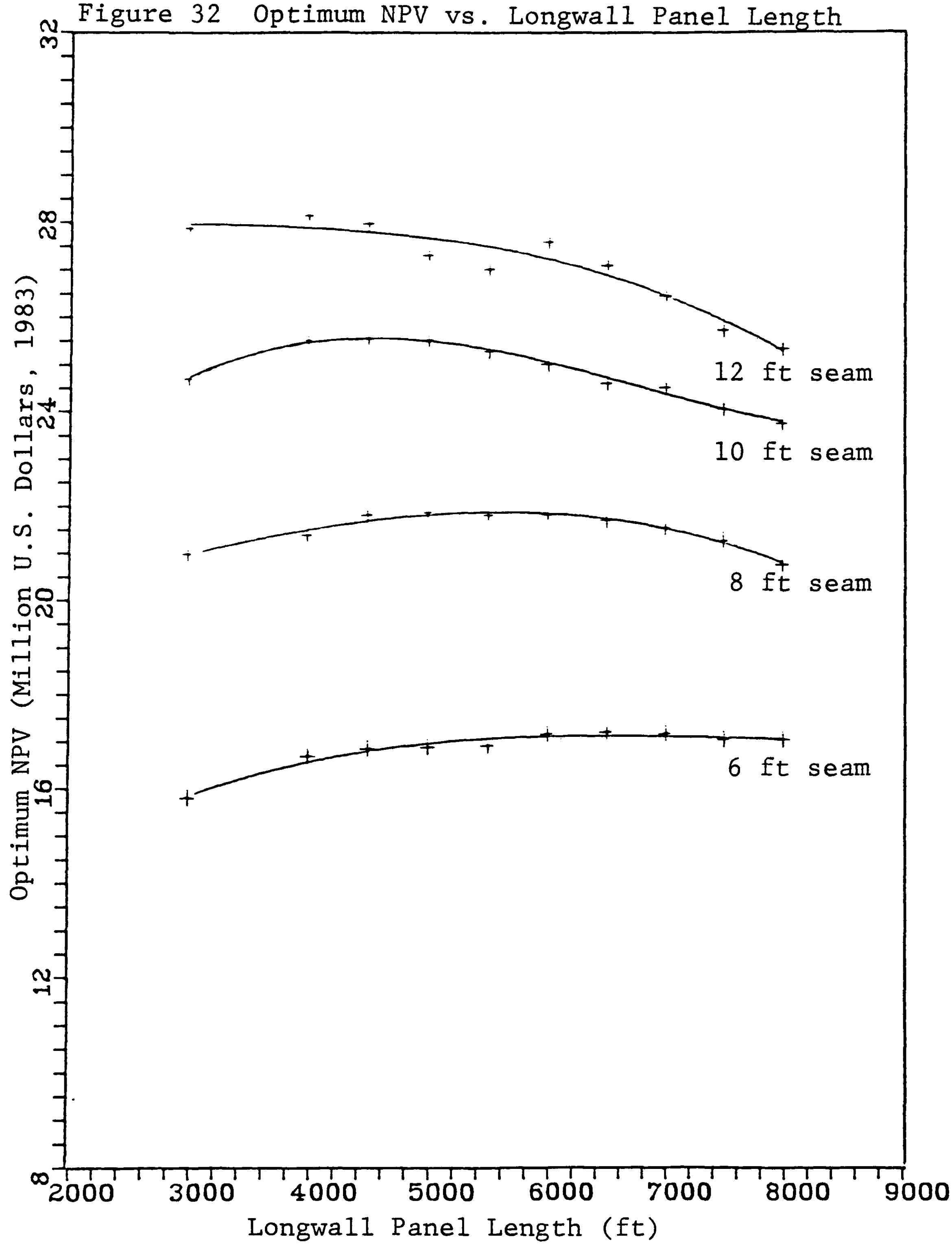


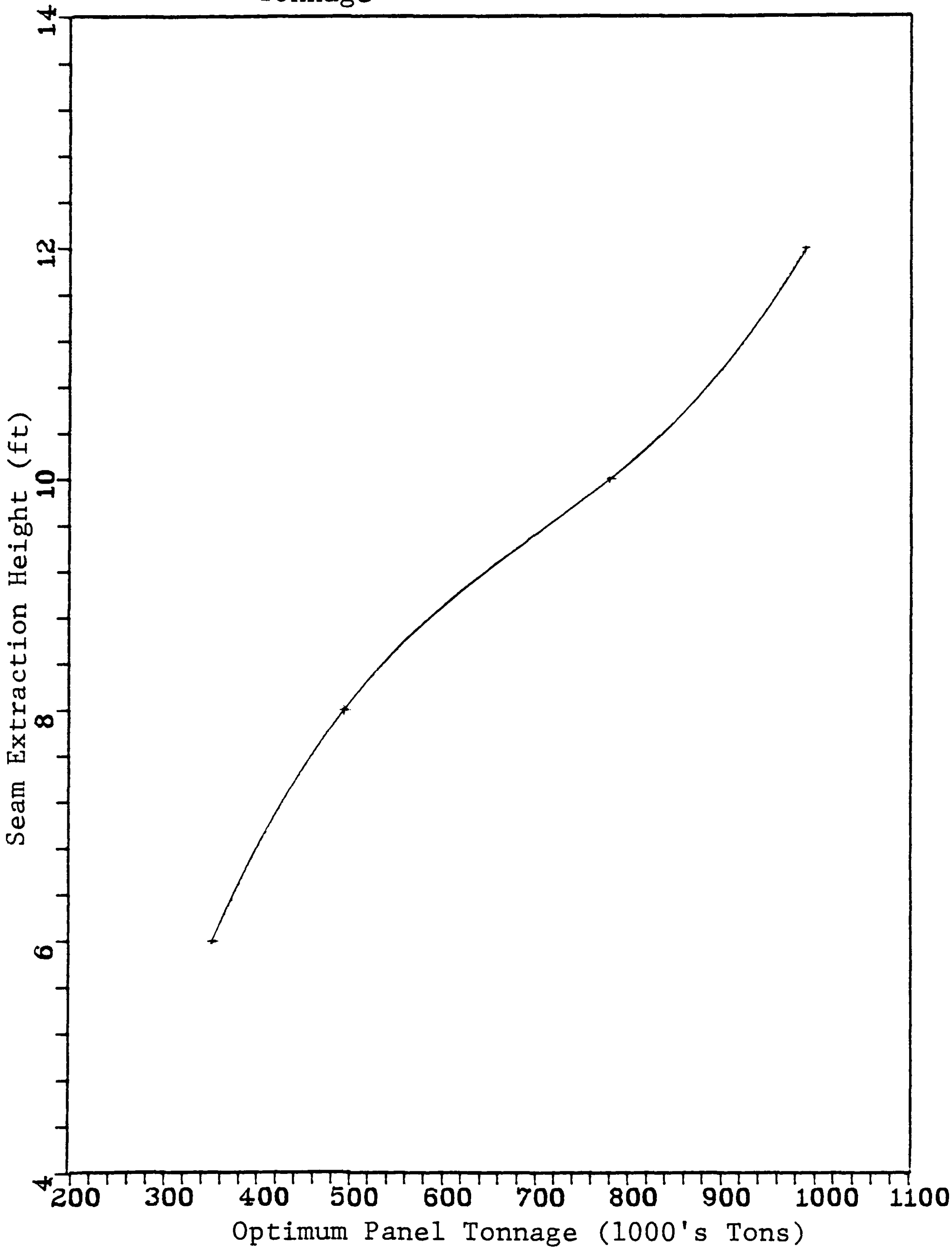
Figure 32 Optimum NPV vs. Longwall Panel Length



longwall panel lengths from 3000 ft to 8000 ft, the maximum decrease in optimum longwall face length is 60 ft, with more typical decreases being on the order of 20 ft for a given seam height. In the lower range of seam heights, the optimum face length variation is negligible. As the longwall panel length increases through its range, the optimum project NPV's appear to rise to a broad peak and then decrease with increasing panel length. As a result, there appears to be an optimum panel tonnage related to the seam height. Figure 33 shows this relationship.

The decrease in optimum longwall face length with increasing panel length may be attibutable to the reduction in fixed costs per ton associated with moving longwall equipment from worked out panels to virgin panels. The economic optimum related to panel length (Figure 32) for a given seam height is difficult to explain with certainty. Potentially, the panel length increases to an optimum as the incremental cost per ton attributable to panel equipment moves decreases. Beyond the optimum panel length, development times increase as do the quantities of high cost development coal. Possibly, the development time might exceed the time required to mine out a virgin panel, thus entailing a waiting cost. In any case, there are time value of money and scheduling problems involved with very long

Figure 33 Seam Height vs. Optimum Longwall Panel Tonnage



panels. The timing and quantity of high cost development coal production may also influence the decline of the optimum project NPV beyond the economic optimum panel length.

Other Variables

Seam depth and the length of roof overhang were examined and found to influence only the geotechnical parameters such as the minimum face length necessary to propagate gob caving and the support load density. Their effects on optimum longwall face length and optimum NPV were found to be negligible.

CHAPTER VII

CONCLUSIONS

In general, any change in panel geometry or operation which causes a greater tonnage of development coal to be produced or lengthens the time span to realize positive cash flows generated from longwall operations will cause the optimum longwall face length to increase and the optimum project NPV to decrease. The general impact on longwall face length of the variables of interest are as follows:

Variable	Variation	Effect on Optimum Longwall Face Length	Effect on Optimum Project NPV
Seam Extraction Height	increase	increase	increase
Entry Systems, 3-entry	2-entry	decrease	increase
Shield Design, 2-leg	4-leg	negligible	very minor decrease
Continuous Miner Development Productivity	increase	decrease	increase
Entry Width	increase	increase	decrease
Panel Length	increase	increase	variable
Seam Depth	increase	none	none
Gob overhang length	increase	negligible	very minor decrease

In view of the fact that an optimum panel length and face length occur at a fixed seam height, there also exists an optimum panel tonnage.

Furthermore, for a given continuous miner development productivity and seam extraction height, there is a point where 2-entry development is no longer economically preferable to 3-entry development.

Finally, it appears to be economically preferable to select a longwall face length slightly longer than the optimum face length rather than shorter than the optimum face length. This rule results from the rates of change of the NPV vs. longwall face length curve on either side of the optimum.

RECOMMENDED ADDITIONAL STUDY

During the course of this investigation, several points arose which were not completely understood or were not modeled. To make this investigation more complete and enhance its practical utility, the relationships of the following areas to the model at hand should be further investigated.

1. Variation of longwall panel length in a group of panels.
2. Variation of seam height in a panel or block of panels.
3. The effect of multi-entry panel development.
4. The effects of fully-advancing and "Z" longwall systems.
5. The effects of entry pillar dimensions.
6. The exact cause of an optimum longwall panel tonnage.

Although it might seem reasonable to extend this investigation to thin or steeply dipping seams being mined by longwalls, it would probably require major computer code revisions and extensive literature research before these cases could be modeled because of their unique operational and design requirements.

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APPENDIX A

Calculation of Support Load and Loading Geometry

The modified U.S. Bureau of Mines model used to predict face support loading is shown in Figure 34. For the purposes of this thesis, the assumed values for variables are as follows:

W = Weight on support (ton)

g = In situ density of material (lb/ft3) = 144 lb/ft3 (Wilson, 1982)

h = Height of cave (ft) = 4 times the extracted section (Wilson, 1982)

A = Angle of cave (^{o}) = 75^{o}

SM = Extraction Height (ft.)

θ = Breaking length of roof (ft)

S = Center-to-center support spacing = 5.0 ft.

W (tons)= 1.44 (SM)(1.54SM+θ)

The derivation of the support loading equation is as follows.

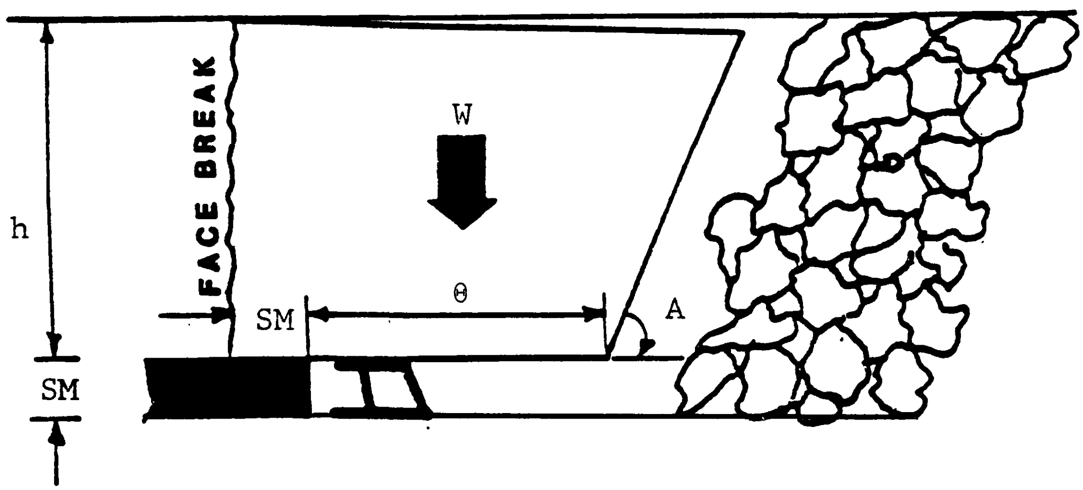
$$W = \left(\frac{(SM+\theta) + (SM+\theta) + h \text{ Tan } (90^{o}-75^{o})}{(2) \quad (2000 \text{ lb/ton})}\right) \times h \text{ (ft)} \times 5 \text{ (ft)} \times g \text{ (lb/ft}^{3}\text{)}$$

$$= \left(\frac{2x(SM+\theta) + 4 \text{ SM} \times \text{Tan } (15^{o})}{2 \times 2000}\right) \times 4SM \times 5x144$$

$$= (\ (SM+\theta) + .5359 \times SM) \times 1.44 \times SM$$

W (tons)= 1.44xSMx(1.5359SM+θ)

This method may be considered conservative because of the excess weight of the rock added to the carried block by



Face Support Loading Geometry

Figure 34

(After Gentry and Stewart, 1982)

the assumption of a vertical break ahead of the face. In reality, the break is probably stress induced and at a similar angle to the cave angle, which is also stress induced.

APPENDIX B

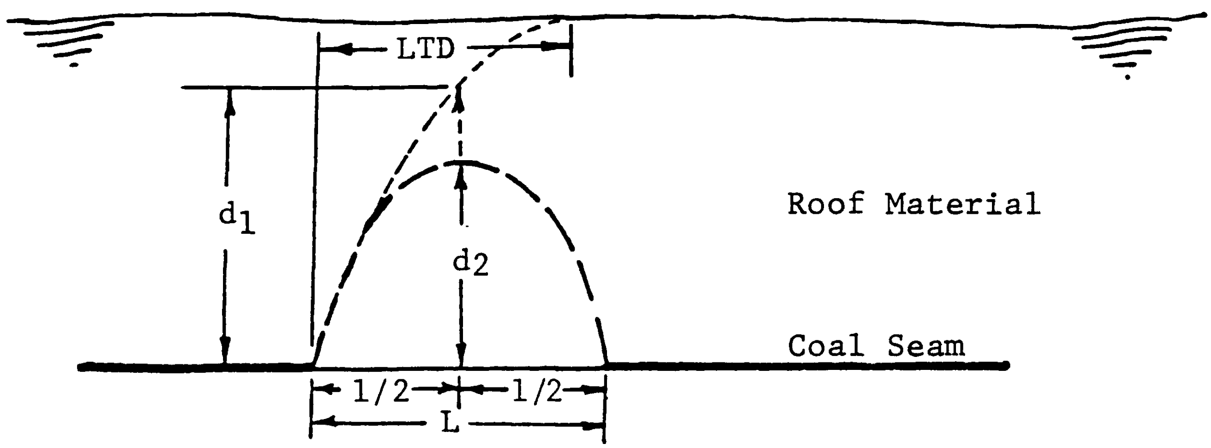
Geotechnical Bounds on Longwall Face Length

The purely geotechnical bounds on longwall face length are derived from the following assumptions and conditions.

First, it is assumed that the weight on an unsupported (mined out) span, parallel to the longwall face, is generated by a parabolic arch of rock. That is, the weight of all rock under the load-transfer-arch or a scaled-down load-transfer-arch. A load-transfer-arch is an arch, two load-transfer-distances (LTD's) wide at the seam roof. For spans shorter than two load transfer distances, the load-transfer-arch is scaled down, as shown below, to represent that portion of the load-transfer-arch affecting the opening. Figure 35 depicts the arching situation.

To arrive at the weight W, it is necessary to use an empirical load-transfer-distance equation (Abel, 1981) to find the maximum arch height (d^2), of a load-transfer-arch "L" feet wide. The maximum arch height (d_2) multiplied by the unit weight of the rock yields the maximum load on the roof beam.

The load transfer equation (1) can be scaled to arrive at the maximum arch height, d_2, for an arch less than two LTD's wide at the exposed span.



L = unsupported span length

LTD = Load Transfer Distance

d_1 = height from the span to the Load-Transfer-Arch

d_2 = maximum arch height

W = weight of unsupported span (acting along d_2)

The Scaled-Down Load-Transfer-Arch

Figure 35

(After Abel, 1981)

$$LTD = -45.0 + .373H - 0.000082H^2 \quad (1)$$

$$d_1 = \frac{H}{(LTD)^2} (2(LTD)(^L/2) - (^L/2)^2)$$

$$LTD_2 = -45.0 + .373d_1 - 000082d$$

$$d_2 = \frac{d_1}{(LTD2)^2} (2(LTD_2)(\frac{L}{2}) - (\frac{L}{2})^2) \quad (2)$$

Assuming a rock density of 144 lb/ft^3 and a beam width of one foot, the weight W which results from d_2 is:

$$W(lb) = d_2 \times 144.0$$

The maximum moments produced by this loading and beam configuration has been shown by Applied Engineering Resources, Inc., 1977 to be:

$$M_{center}\ (lb\text{-}ft) = 0.0375\ WL^2 \quad (3)$$

$$M_{end}\ (lb\text{-}ft) = 0.0667\ WL^2 \quad (4)$$

It is assumed that the span fails at the ends of the span in compression. Thus, when the compressive strength of the strongest roof member in four seam heights is exceeded, the span will collapse.

The derivation of the minimum undermined span, "L", required to promote good caving to four seam heights above the seam is as follows.

σc = bulk compressive strength of controlling roof member (psf)

σh = horizontal in-situ stress (psf)

σf = fiber stress due to bending moment (psf)

$$\sigma_f = \frac{MC}{I}$$

M = maximum moment for center or end of beam (lb-ft)

c = distance from neutral fiber to fiber edge (ft)

= h/2 ft

I = moment of inertia for beam section about neutral fiber (ft^4)

= $\frac{bh^3}{12}$ for a rectangle

b = beam depth = 1 ft

h = beam height (ft)

H = depth to seam (ft)

The total horizontal compressive stress on a rock element is equal to the generated fiber stress due to bending added to the existing horizontal stress in the ground.

$$\sigma_c = \sigma_f + \sigma_h \qquad (6)$$

Based on the hypothesis that through geologic time the compressive horizontal stess will come to equilibrium at the lithostatic stress, horizontal stress is assumed equal to the vertical stress (Wilson, 1972).

Reducing equation (6) with these assumptions, the total compressive stress on a rock element in the roof near the rib is represented by equation (7).

$$\sigma_c = \frac{MC}{I} + 144\ H$$

$$= \frac{0.0667WL^2(h/2)}{\frac{(bh^3)}{12}} + 144\ H$$

$$= \frac{(0.0667)(144d_2)(L^2)(h)(12)}{bh^3} + 144H$$

$$= \frac{(0.0667)(144d_2)(L^2)(6)}{bh^2} + 144H \qquad (7)$$

The bulk compressive strength of the controlling roof member is assumed to be 5000 psi (720,000 psf).

Setting equation (7) equal to the assumed bulk compressive strength of the rock, the minimum span required to promote adequate caving (L) can be found.

$$720{,}000 \text{ psf} = \frac{(0.0667)(144d_2)(L^2)(6)}{bh^2} + 144H$$

$$L^2 = \frac{(720\ 000 - 144H)\ bh^2}{(0.0667)(144d_2)(6)} \qquad (8)$$

Assuming b = 1 ft, then equation (8) reduces to:

$$L^2 = \frac{(720\ 000 - 144H)h^2}{(0.0667)(144d_2)(6)}$$

$$= \frac{(5000-H)h^2(144)}{(.0667)(144)(d_2)(6)}$$

$$= \frac{(5000-H)(h^2)}{(.0667)(6)(d_2)}$$

$$L(ft) = \left(\frac{(5000-H)(h^2)}{(0.400)(d_2)}\right)^{\frac{1}{2}}$$

L = maximum span length for compressive failure (ft)

A check must also be made to determine if tensile failure controls the maximum span length. It is assumed that joints are cohensionless and that failure is pending when in situ horizontal stress, assumed to be compressive, is exceeded by the tensile beam fiber stress. Therefore, at the center of the roof beam equation (9) represents the tensile fiber stress-horizontal stress equilibrium at pending tensile failure.

$$\sigma_f = \frac{MC}{I} = \sigma_H \qquad (9)$$

Reducing equation (9) with the aforementioned assumptions, the limiting span (L) may be found.

$$= \frac{(010375WL^2)(h/2)}{(bh^3/12)}$$

$$= \frac{(0.0375)(144d_2)(L^2)(h)(12)}{(2)(b)(h^3)}$$

$$= \frac{(0.0375)(144d_2)(L^2)(6)}{(b)(h^2)}$$

$$L^2 = \frac{\sigma_H\ (b)\ (h^2)}{(0.0375)(144d_2)(6)}$$

Assume σ_H = 1.0 psi/ft of depth = 144 psf/ft of depth

$$L^2 = \frac{144H\ (b)(h^2)}{(.0375)(144d_2)(6)}$$

Assume b = 1 ft

$$L^2 = \frac{(H)(h^2)}{(.0375)(d_2)(6)}$$

$$L = \left(\frac{(H)(h^2)}{(0.2250\ (d_2)}\right)^{\frac{1}{2}}$$

L = maximum span length for tensile failure (ft)

It may also be noted that the minimum span length also corresponds to the distance from ribside to the longwall face, in the direction of panel length, where initial panel caving occurs. The span to initial cave may be slightly longer if the roof bolts in the starting entry are intact. The added span length results from the increased strength of that small section. On the other hand, if the starting entry is blasted to initiate cave early, the advance achieved prior to full cave will depend on the degree of blast disturbance and the height to the cave controlling member. Panels adjacent to worked out panels may be observed to cave prior to attainment of the predicted advance because of disturbances of the roof related to the mined out adjacent panel.

APPENDIX C

Detailed Cost and Economic Assumptions used in the Longwall Simulation Model

Cost assumptions necessary to generate production cost estimates are explained in detail in the following paragraphs.

Included in the ventilation and roof control cost for longwall production is a component of between \$0.05 - \$0.16 per-ton-produced for timber cribs in the tailgate. It is assumed that a single row of 3-ft x 3-ft cribs on a 6 ft center-to-center spacing is installed. The cribs are constructed from twin timber layers of 6 in x 6 in timbers, 3 ft long.

Further, it is assumed that there is an initial longwall engineering and design fee of \$100,000 which is capitalized, as a part of depreciation, over the first 10 years of project life.

Table 2 shows the daily cost of the continuous miner personnel. These manpower figures are assumed based on common industrial practice at unionized (United Mine Workers of America) mines.

Table 3 shows the daily cost of longwall production and moving manpower. These figures are based on national statistics for longwalls in coal seams thicker than 60 in.

The production cost estimate for longwall development, mining, and moving (Table 4) is based on cost assumptions from the general literature and generally accepted operating practice.

TABLE 2

CONTINUOUS MINER PANEL DEVELOPMENT FOR LONGWALL
(as per 1981 UMW contract 12/82-3/83)

Production Personnel (2 shifts per day)

	Total	$ Wage/ Day	Total Cost/Day
Continuous Miner Operators	2	102.00	204.00
Continuous Miner Helpers	2	102.00	204.00
Shuttle Car Operators	4	95.52	382.08
Roof Bolters	4	102.00	408.00
Maintenance Personnel	2	102.00	204.00
Utility Personnel	2	95.00	191.04
Subtotal	16		1,593.12
10% Absenteeism			159.31
Total Hourly Cost			1,752.43
Supervisors	2	145.84	291.68
Total Production Labor	18		2,044.11

Maintenance Personnel (1 shift/day)

	Total	$ Wage/ Day	Total Cost/Day
Maintenance Personnel	2	95.52	191.04
10% Absenteeism			19.10
Supervisor	1	145.84	145.84
Total Maintenance Labor	3		355.98
Subtotal Development Labor	21		2,400.09
Fringes at 42%			1,008.03
Total Development Labor Cost			3,408.13

TABLE 3

LONGWALL LABOR

Production (2 production shifts, 1 maintenance shift)

	Total	$ Wage/ Day	Total Cost/Day
Production Supervisors	4	145.84	583.36
Maintenance Supervisor	1	145.84	145.84
Shearing Machine Operators	4	101.76	407.04
Support Movers	6	98.48	590.88
Headgate Operators	2	98.48	196.96
Mechanics (production)	2	101.76	203.52
Mechanics (maintenance)	4	101.76	407.04
General Laborers	8	93.20	745.60
Subtotal	31		3,280.24
10% Absentism Excluding Salaried Supervisors			255.10
Subtotal Production Labor			3,535.34
Fringes at 42%			1,484.84
Total Production Labor Cost			5,020.18

Panel Moves (3 shifts per day)

Supervisors	6	145.84	875.04
Mechanics	18	101.76	1,831.68
Scoop Operators	6	98.48	590.88
General Labor	12		1,118.40
Subtotal	42		4,416.00
10% absenteeism Excluding Supervisors			354.10
Subtotal Move Labor			4,770.10
Fringes at 42%			2,003.44
Total Labor Cost for Moving Longwall			6,773.54

TABLE 4

DEVELOPMENT PRODUCTION COST

Tonnage per shift:	input by program user	
Direct Labor:	3408.13	(From Table 2)
Maintenance:	$1.35/ton	
Ventilation & Roof Support:	0.50/ton	
Subtotal	1.85/ton + 3408.13	
10% Contingenty	.19/ton + 340.81	
Total Day Cost	2.04/ton + 3748.94	

Cost/ton = Daily Production/Total Daily Cost

LONGWALL PRODUCTION COST

	Day Cost	
Direct Labor	5020.18	(From Table 3)
Maintenance	.25/ton	
Ventilation & Roof Control	.40/ton	
Subtotal	0.65/ton + 5020.18	
10% Contingency	.07/ton + 502.01	
Total Day Cost	0.72/ton + 5522.20	

LONGWALL MOVE COST

Direct Labor	6773.54	(From Table 3)
2 Scoops @ 400/day	800.00	
Supplies	500.00	
Subtotal	8073.54	
10% Contingency	807.35	
Total longwall moving cost/day	8880. 89	

LONGWALL EQUIPMENT COSTS

Hemscheidt Shields

Ton Ratings	2 leg	4 leg
350	$30,000 ea.	$32,000 ea.
472	$35,000 ea.	$37,000 ea.
582	$39,000 ea.	$41,000 ea.

Forepole Option: $3500 ea. (Sublett, 1983)

Eickhoff EDW-300L Shearing Machine $1.2mm (Mauulakis, 1983)

Halbach-Braun Face Equipment

30-in. AFC pan line (30x108mm chain, $26.85/ft)

AFC drives

Base frames

Stage loader

Crusher

Mobile belt tail piece

Installation/Training

Total $1.3mm for a 600-ft face (Easton, 1983)

All costs given in this section are May, 1983 cost estimates supplied by the component suppliers. In most cases, the costs are approximate as exact costs (lowest cost) are considered propriatary. These equipment costs are known to be in agreement with costs quoted to recent potential clients. The specific equipment manufacturers quoted were chosen for illustrative purposes only.

APPENDIX D

Cash Flow Tables of Sub-Optimal, Optimal, and Super-Optimal Variations of Face Length for the Test Case.

Sub-Optimal: 580 ft

It should be noted that any C/F figures shown for years beyond the end of the project have no economic significance and do not impact simulator results. Such figures exist only because of the output format. The end of the project life is last year in which there is a positive gross revenue.

All figures are in 1983 U.S. dollars and net annual cash flows are discounted to present value at 16% compounded monthly (17.2% annually).

These tables are shown to enable a simulator user to examine individual annual cash flows for budgetary purposes as well as to assist the user in error recognition and rectification.

YEAR	1	2	3	4	5	6
REVENUE	2520000	6422620	16410572	16410572	16410572	16410572
ROYALTY/O&M	226800	578035	1476951	1476951	1476951	1476951
OPERATING COST	1242465	1905772	3130691	3130691	3130691	3130691
PROPERTY TAX	51452	326451	739057	726932	714677	701123
NET AFTER COST	999282	3612360	11063871	11075996	11088252	11101808
DEPRECIATION	180000	1487372	2373795	1797304	1229489	641432
NET AFTER DEPRECIATION	969282	2124988	8690076	9278691	9858762	10460376
DEPLETION	231840	590881	1509772	1509772	1509772	1509772
STATE & SEVERANCE TAXES	78672	195557	677626	707057	736060	766141
PRE-TAX PROFITS	678770	1338549	6502677	7061861	7612929	8184462
FEDERAL INCOME TAXES	312234	615732	2991231	3248456	3501947	3764852
MINIMUM TAX	0	0	0	0	0	0
TAX CREDITS	0	527122	211563	7229	7229	0
NET PROFITS	366536	1249939	3723008	3820634	4118210	4419609
CASH FLOW	608376	3328192	7606576	7127712	6857472	6570814
CAPITAL EXPENDITURE	0	7386860	0	72292	72292	0
NET CASH FLOW/INC. WORK CAP.	395791	-4271251	7393992	6842835	6572596	6358230
NET PRESENT VALUE	337628	-3108130	4589814	3623466	2968911	2450013

YEAR	7	8	9	10	11	12
REVENUE	16410572	16410572	16410572	16410572	16410572	16410572
ROYALTY/CSM	1476951	1476951	1476951	1476951	1476951	1476951
OPERATING COST	3130691	3130691	3130691	3130691	3130691	3130691
PROPERTY TAX	707464	692868	678141	703217	682622	676494
NET AFTER COST	11095464	11110061	11124788	11099712	11120307	11126434
DEPRECIATION	284602	434906	348198	754490	939124	640758
NET AFTER DEPRECIATION	10810861	10675155	10776589	10345221	10181182	10485676
DEPLETION	1509772	1509772	1509772	1509772	1509772	1509772
STATE & SEVERANCE TAXES	783665	776880	781952	760383	752181	767406
PRE-TAX PROFITS	8517423	8388501	8484864	8075064	7919227	8208497
FEDERAL INCOME TAXES	3918014	3858710	3903037	3714529	3642844	3775908
MINIMUM TAX	0	0	0	0	0	0
TAX CREDITS	122843	7229	7229	253729	0	7229
NET PROFITS	4722251	4537020	4589056	4614264	4276383	4439817
CASH FLOW	6516627	6481698	6447027	6878527	6725280	6590348
CAPITAL EXPENDITURE	1228432	72292	72292	2537292	0	72292
NET CASH FLOW/INC. WORK CAP.	5075612	6196822	6162150	4128651	6725280	6518056
NET PRESENT VALUE	1668371	1737582	1473943	842419	1170584	967793

YEAR	13	14	15	16	17	18
REVENUE	16410572	16410572	16410572	16410572	16410572	0
ROYALTY/CSM	1476951	1476951	1476951	1476951	1476951	0
OPERATING COST	3130691	3130691	3130691	3130691	3130691	0
PROPERTY TAX	670236	662807	656680	650422	645074	7169
NET AFTER COST	11132692	11140121	11146249	11152507	11157854	-7169
DEPRECIATION	449341	243466	43375	54941	46266	28916
NET AFTER DEPRECIATION	10683350	10896654	11102873	11097565	11111588	-36086
DEPLETION	1509772	1509772	1509772	1509772	1509772	0
STATE & SEVERANCE TAXES	777290	787955	798266	798001	798702	0
PRE-TAX PROFITS	8396287	8598926	8794834	8789791	8803113	-36086
FEDERAL INCOME TAXES	3862292	3955506	4045624	4043304	4049432	0
MINIMUM TAX	0	0	0	0	0	0
TAX CREDITS	7229	0	7229	7229	0	0
NET PROFITS	4541224	4643420	4756439	4753716	4753681	-36086
CASH FLOW	6500339	6396659	6309587	6318431	6309720	-7169
CAPITAL EXPENDITURE	72292	0	72292	72292	0	0
NET CASH FLOW/INC. WORK CAP.	6428047	6396659	6237295	6246139	6309720	0
NET PRESENT VALUE	814170	691133	574879	491093	423189	0

YEAR	19	20	21	22	23	24
REVENUE	0	0	0	0	0	0
ROYALTY/CSM	0	0	0	0	0	0
OPERATING COST	0	0	0	0	0	0
PROPERTY TAX	7169	7169	7169	7169	7169	7169
NET AFTER COST	-7169	-7169	-7169	-7169	-7169	-7169
DEPRECIATION	0	0	0	0	0	0
NET AFTER DEPRECIATION	-7169	-7169	-7169	-7169	-7169	-7169
DEPLETION	0	0	0	0	0	0
STATE & SEVERANCE TAXES	0	0	0	0	0	0
PRE-TAX PROFITS	-7169	-7169	-7169	-7169	-7169	-7169
FEDERAL INCOME TAXES	0	0	0	0	0	0
MINIMUM TAX	0	0	0	0	0	0
TAX CREDITS	0	0	0	0	0	0
NET PROFITS	-7169	-7169	-7169	-7169	-7169	-7169
CASH FLOW	-7169	-7169	-7169	-7169	-7169	-7169
CAPITAL EXPENDITURE	0	0	0	0	0	0
NET CASH FLOW/INC. WORK CAP.	0	0	0	0	0	0
NET PRESENT VALUE	0	0	0	0	0	0

YEAR	25	26	27	28	29	30
REVENUE	0	0	0	0	0	0
ROYALTY/OSM	0	0	0	0	0	0
OPERATING COST	0	0	0	0	0	0
PROPERTY TAX	7169	7169	7169	7169	7169	7169
NET AFTER COST	-7169	-7169	-7169	-7169	-7169	-7169
DEPRECIATION	0	0	0	0	0	0
NET AFTER DEPRECIATION	-7169	-7169	-7169	-7169	-7169	-7169
DEPLETION	0	0	0	0	0	0
STATE & SEVERANCE TAXES	0	0	0	0	0	0
PRE-TAX PROFITS	-7169	-7169	-7169	-7169	-7169	-7169
FEDERAL INCOME TAXES	0	0	0	0	0	0
MINIMUM TAX	0	0	0	0	0	0
TAX CREDITS	0	0	0	0	0	0
NET PROFITS	-7169	-7169	-7169	-7169	-7169	-7169
CASH FLOW	-7169	-7169	-7169	-7169	-7169	-7169
CAPITAL EXPENDITURE	0	0	0	0	0	0
NET CASH FLOW/INC. WORK CAP.	0	0	0	0	0	0
NET PRESENT VALUE	0	0	0	0	0	0

2.1716866E+07, 0.0000000E+00YEAR=31F=580.0000PRODUCTION=
0.0000000E+00

APPENDIX D CONTINUED

Optimal Case: 600 ft Face

YEAR	1	2	3	4	5	6
REVENUE	2520000	6187485	16639399	16639399	16639399	16639399
ROYALTY/CSM	226800	556873	1497545	1497545	1497545	1497545
OPERATING COST	1242465	1886335	3166657	3166657	3166657	3166657
PROPERTY TAX	51452	319317	751413	738938	726330	712382
NET AFTER COST	999282	3424958	11223782	11236257	11248865	11262813
DEPRECIATION	10000	1530040	2442064	1848936	1264740	659732
NET AFTER DEPRECIATION	969282	1894918	8781718	9387321	9984124	10603111
DEPLETION	231840	569248	1530824	1530824	1530824	1530824
STATE & SEVERANCE TAXES	78672	180433	685732	716012	745853	776822
PRE-TAX PROFITS	678770	1145236	6565161	7140483	7707447	8295484
FEDERAL INCOME TAXES	312234	526808	3019974	3284622	3545425	3815922
MINIMUM TAX	0	6365	0	0	0	0
TAX CREDITS	0	451537	308482	7444	7444	0
NET PROFITS	366536	1063599	3853669	3863305	4169465	4479561
CASH FLOW	608376	3162888	7826558	7243065	6965031	6670088
CAPITAL EXPENDITURE	0	7600200	0	74440	74440	0
NET CASH FLOW/INC. WORK CAP.	391716	-4653971	7609899	6951966	6673931	6453429
NET PRESENT VALUE	334152	-3386630	4723838	3681254	3014685	2486696

YEAR	7	8	9	10	11	12
REVENUE	16639399	16639399	16639399	16639399	16639399	16639399
ROYALTY/GSR	1497545	1497545	1497545	1497545	1497545	1497545
OPERATING COST	3166657	3166657	3166657	3166657	3166657	3166657
PROPERTY TAX	718685	703692	687360	713543	692487	686317
NET AFTER COST	11256510	11271503	11287835	11261652	11282708	11288879
DEPRECIATION	289796	442785	339800	752769	951739	650708
NET AFTER DEPRECIATION	10966714	10828717	10948035	10508882	10330969	10638170
DEPLETION	1530824	1530824	1530824	1530824	1530824	1530824
STATE & SEVERANCE TAXES	794982	788082	794048	772090	763195	778555
PRE-TAX PROFITS	8640907	8509810	8623162	8205967	8036949	8328790
FEDERAL INCOME TAXES	3974817	3914512	3966654	3774744	3696996	3831243
MINIMUM TAX	0	0	0	0	0	0
TAX CREDITS	125010	7444	0	262444	0	7444
NET PROFITS	4791100	4602741	4656507	4693666	4339952	4504990
CASH FLOW	6611720	6576351	6527132	6977260	6822516	6686524
CAPITAL EXPENDITURE	1252100	74440	0	2624440	0	74440
NET CASH FLOW/INC. WORK CAP.	5144961	6285252	6310473	4136161	6822516	6612084
NET PRESENT VALUE	1691166	1762378	1509420	843952	1187509	981754

YEAR	13	14	15	16	17	18
REVENUE	16639399	17062535	17062535	17062535	17062535	0
ROYALTY/OSM	1497545	1535628	1535628	1535628	1535628	0
OPERATING COST	3166657	3193539	3193539	3193539	3193539	0
PROPERTY TAX	680012	692046	685876	679572	674178	7403
NET AFTER COST	11295183	11641320	11647491	11653795	11659189	-7403
DEPRECIATION	458619	251641	44664	56574	47641	29776
NET AFTER DEPRECIATION	10836564	11389679	11602827	11597221	11611547	-37179
DEPLETION	1530824	1569753	1569753	1569753	1569753	0
STATE & SEVERANCE TAXES	788474	822647	833304	833024	833740	0
PRE-TAX PROFITS	8517264	8997279	9199769	9194443	9208054	-37179
FEDERAL INCOME TAXES	3917941	4138748	4231894	4229444	4235704	0
MINIMUM TAX	0	0	0	0	0	0
TAX CREDITS	7444	0	7444	7444	0	0
NET PROFITS	4606766	4858530	4975319	4972443	4972349	-37179
CASH FLOW	6596210	6679925	6589736	6598771	6589744	-7403
CAPITAL EXPENDITURE	74440	0	74440	74440	0	0
NET CASH FLOW/INC. WORK CAP.	6521770	6679925	6515296	6524331	6589744	0
NET PRESENT VALUE	826041	721739	600502	512966	441970	0

YEAR	19	20	21	22	23	24
REVENUE	0	0	0	0	0	0
ROYALTY/OSM	0	0	0	0	0	0
OPERATING COST	0	0	0	0	0	0
PROPERTY TAX	7403	7403	7403	7403	7403	7403
NET AFTER COST	-7403	-7403	-7403	-7403	-7403	-7403
DEPRECIATION	0	0	0	0	0	0
NET AFTER DEPRECIATION	-7403	-7403	-7403	-7403	-7403	-7403
DEPLETION	0	0	0	0	0	0
STATE & SEVERANCE TAXES	0	0	0	0	0	0
PRE-TAX PROFITS	-7403	-7403	-7403	-7403	-7403	-7403
FEDERAL INCOME TAXES	0	0	0	0	0	0
MINIMUM TAX	0	0	0	0	0	0
TAX CREDITS	0	0	0	0	0	0
NET PROFITS	-7403	-7403	-7403	-7403	-7403	-7403
CASH FLOW	-7403	-7403	-7403	-7403	-7403	-7403
CAPITAL EXPENDITURE	0	0	0	0	0	0
NET CASH FLOW/INC. WORK CAP.	0	0	0	0	0	0
NET PRESENT VALUE	0	0	0	0	0	0

YEAR	25	26	27	28	29	30
REVENUE	0	0	0	0	0	0
ROYALTY/OSM	0	0	0	0	0	0
OPERATING COST	0	0	0	0	0	0
PROPERTY TAX	7403	7403	7403	7403	7403	7403
NET AFTER COST	-7403	-7403	-7403	-7403	-7403	-7403
DEPRECIATION	0	0	0	0	0	0
NET AFTER DEPRECIATION	-7403	-7403	-7403	-7403	-7403	-7403
DEPLETION	0	0	0	0	0	0
STATE & SEVERANCE TAXES	0	0	0	0	0	0
PRE-TAX PROFITS	-7403	-7403	-7403	-7403	-7403	-7403
FEDERAL INCOME TAXES	0	0	0	0	0	0
MINIMUM TAX	0	0	0	0	0	0
TAX CREDITS	0	0	0	0	0	0
NET PROFITS	-7403	-7403	-7403	-7403	-7403	-7403
CASH FLOW	-7403	-7403	-7403	-7403	-7403	-7403
CAPITAL EXPENDITURE	0	0	0	0	0	0
NET CASH FLOW/INC. WORK CAP.	0	0	0	0	0	0
NET PRESENT VALUE	0	0	0	0	0	0

2.1933399E+07, 0.0000000E+00YEAR=31F=600.0000PRODUCTION=
0.0000000E+00

APPENDIX D CONTINUED

Super-Optimal Case: 620 ft Face

YEAR	1	2	3	4	5	6
REVENUE	2520000	6035807	16952104	16952104	16952104	16952104
ROYALTY/CSM	226800	543222	1525689	1525689	1525689	1525689
OPERATING COST	1242465	1872218	3207964	3207964	3207964	3207964
PROPERTY TAX	51452	316036	767642	754818	741857	727517
NET AFTER COST	999282	3304329	11450808	11463632	11476593	11490933
DEPRECIATION	10000	1572708	2510332	1900567	1299992	677972
NET AFTER DEPRECIATION	989282	1731621	8940475	9563064	10176601	10812961
DEPLETION	231840	555294	1559593	1559593	1559593	1559593
STATE & SEVERANCE TAXES	78672	169932	698486	729615	760292	792110
PRE-TAX PROFITS	678770	1006394	6682395	7273855	7856715	8461257
FEDERAL INCOME TAXES	312234	462941	3073902	3345973	3614089	3892178
MINIMUM TAX	0	13852	0	0	0	0
TAX CREDITS	0	397250	384103	7658	7658	0
NET PROFITS	366536	926850	3992597	3935540	4250285	4569078
CASH FLOW	608376	3054852	8062524	7395701	7109870	6806645
CAPITAL EXPENDITURE	0	7813540	0	76588	76588	0
NET CASH FLOW/INC. WORK CAP.	400646	-4966417	7854794	7111383	6825553	6598915
NET PRESENT VALUE	341769	-3613992	4875856	3765670	3080174	2542756

YEAR	7	8	9	10	11	12
REVENUE	16952104	16952104	16952104	17206386	17545428	17545428
ROYALTY/OSM	1525689	1525689	1525689	1548574	1579088	1579088
OPERATING COST	3207964	3207964	3207964	3224112	3245643	3245643
PROPERTY TAX	733779	718391	701624	740529	735782	769266
NET AFTER COST	11484671	11500059	11516827	11693169	11984913	11951429
DEPRECIATION	294989	450665	345860	774181	997021	1190124
NET AFTER DEPRECIATION	11189682	11049394	11170967	10918987	10987892	10761305
DEPLETION	1559593	1559593	1559593	1582987	1614179	1614179
STATE & SEVERANCE TAXES	817946	803932	810010	801327	809994	798664
PRE-TAX PROFITS	8819141	8685868	8801362	8534672	8563718	8348461
FEDERAL INCOME TAXES	4356805	3995499	4048626	3925949	3939310	3840292
MINIMUM TAX	0	0	0	0	0	0
TAX CREDITS	127177	7658	0	271158	7658	254354
NET PROFITS	4889513	4698028	4752735	4879081	4632066	4762523
CASH FLOW	6744096	6708286	6658189	7237051	7243267	7566826
CAPITAL EXPENDITURE	1271770	76588	0	2711588	76588	2543540
NET CASH FLOW/INC. WORK CAP.	5264596	6423969	6450459	4317733	7166679	5023286
NET PRESENT VALUE	1730491	1801274	1542904	881000	1247413	745851

YEAR	13	14	15	16	17	18
REVENUE	17545428	17545428	17545428	17121625	0	0
ROYALTY/O&M	1579088	1579088	1579088	1540946	0	0
OPERATING COST	3245643	3245643	3245643	3218729	0	0
PROPERTY TAX	758207	747148	734848	704214	31108	31108
NET AFTER COST	11962489	11973548	11985847	11657735	-31108	-31108
DEPRECIATION	1281485	879456	455982	249436	42889	0
NET AFTER DEPRECIATION	10681003	11094091	11529865	11408299	-73997	-31108
DEPLETION	1614179	1614179	1614179	1575189	0	0
STATE & SEVERANCE TAXES	794649	815304	837092	824488	0	0
PRE-TAX PROFITS	8272174	8664608	9078592	9008622	-73997	-31108
FEDERAL INCOME TAXES	3805200	3985719	4176152	4143966	0	0
MINIMUM TAX	0	0	0	0	0	0
TAX CREDITS	7658	7658	0	7658	0	0
NET PROFITS	4474632	4686547	4902440	4872314	-73997	-31108
CASH FLOW	7370297	7180183	6972602	6696940	-31108	-31108
CAPITAL EXPENDITURE	76588	76588	0	76588	0	0
NET CASH FLOW/INC. WORK CAP.	7293709	7103595	6972602	6620352	0	0
NET PRESENT VALUE	923815	767514	642651	520515	0	0

YEAR	19	20	21	22	23	24
REVENUE	0	0	0	0	0	0
ROYALTY/OSM	0	0	0	0	0	0
OPERATING COST	0	0	0	0	0	0
PROPERTY TAX	31108	31108	31108	31108	31108	31108
NET AFTER COST	-31108	-31108	-31108	-31108	-31108	-31108
DEPRECIATION	0	0	0	0	0	0
NET AFTER DEPRECIATION	-31108	-31108	-31108	-31108	-31108	-31108
DEPLETION	0	0	0	0	0	0
STATE & SEVERANCE TAXES	0	0	0	0	0	0
PRE-TAX PROFITS	-31108	-31108	-31108	-31108	-31108	-31108
FEDERAL INCOME TAXES	0	0	0	0	0	0
MINIMUM TAX	0	0	0	0	0	0
TAX CREDITS	0	0	0	0	0	0
NET PROFITS	-31108	-31108	-31108	-31108	-31108	-31108
CASH FLOW	-31108	-31108	-31108	-31108	-31108	-31108
CAPITAL EXPENDITURE	0	0	0	0	0	0
NET CASH FLOW/INC. WORK CAP.	0	0	0	0	0	0
NET PRESENT VALUE	0	0	0	0	0	0

YEAR	25	26	27	28	29	30
REVENUE	0	0	0	0	0	0
ROYALTY/OSM	0	0	0	0	0	0
OPERATING COST	0	0	0	0	0	0
PROPERTY TAX	31108	31108	31108	31108	31108	31108
NET AFTER COST	-31108	-31108	-31108	-31108	-31108	-31108
DEPRECIATION	0	0	0	0	0	0
NET AFTER DEPRECIATION	-31108	-31108	-31108	-31108	-31108	-31108
DEPLETION	0	0	0	0	0	0
STATE & SEVERANCE TAXES	0	0	0	0	0	0
PRE-TAX PROFITS	-31108	-31108	-31108	-31108	-31108	-31108
FEDERAL INCOME TAXES	0	0	0	0	0	0
MINIMUM TAX	0	0	0	0	0	0
TAX CREDITS	0	0	0	0	0	0
NET PROFITS	-31108	-31108	-31108	-31108	-31108	-31108
CASH FLOW	-31108	-31108	-31108	-31108	-31108	-31108
CAPITAL EXPENDITURE	0	0	0	0	0	0
NET CASH FLOW/INC. WORK CAP.	0	0	0	0	0	0
NET PRESENT VALUE	0	0	0	0	0	0

2.1798668E+07, 0.0000000E+00YEAR=31F=620.0000PRODUCTION=
0.0000000E+00

APPENDIX E

Longwall Test Case Design Result Tables for
Face Lengths from 160 ft to 940 ft

OVERBURDEN THICKNESS,FT	1000	1000	1000	1000	1000
THICKNESS OF STRONGEST BED,FT	15	15	15	15	15
LENGTH OF ROOF OVERHANG,FT	23	23	23	23	23
COAL EXTRACTION HEIGHT,FT	10	10	10	10	10
CONTINUOUS MINER PROD,TPSH	350	350	350	350	350
ENTRY PILLAR LENGTH,FT	80	80	80	80	80
ENTRY PILLAR WIDTH,FT	40	40	40	40	40
ENTRY WIDTH,FT	20	20	20	20	20
NUMBER OF ENTRIES	3	3	3	3	3
LONGWALL FACE LENGTH,FT	160	180	200	220	240
MAXIMUM FACE LENGTH,FT	1060	1060	1060	1060	1060
MINIMUM FACE LENGTH,FT	88	88	88	88	88
LONGWALL PANEL LENGTH,FT	4000	4000	4000	4300	4000
FACE SUPPORT DESIGN LOAD,ST	562	562	562	562	562
NUMBER OF SHIELD LEGS	2	2	2	2	2
FACE CONVEYOR HORSEPOWER	94	105	117	129	141
NPV OF LONGWALL PROJECT,$ 1983	10537161	11359334	12799866	13045772	13715938

OVERBURDEN THICKNESS,FT	1000	1000	1000	1000	1000
THICKNESS OF STRONGEST BED,FT	15	15	15	15	15
LENGTH OF ROOF OVERHANG,FT	23	23	23	23	23
COAL EXTRACTION HEIGHT,FT	10	10	10	10	10
CONTINUOUS MINER PROD,TPSH	350	350	350	350	350
ENTRY PILLAR LENGTH,FT	80	80	80	80	80
ENTRY PILLAR WIDTH,FT	40	40	40	40	40
ENTRY WIDTH,FT	20	20	20	20	20
NUMBER OF ENTRIES	3	3	3	3	3
LONGWALL FACE LENGTH,FT	260	280	300	320	340
MAXIMUM FACE LENGTH,FT	1060	1060	1060	1060	1060
MINIMUM FACE LENGTH,FT	88	88	88	88	88
LONGWALL PANEL LENGTH,FT	4000	4000	4000	4000	4000
FACE SUPPORT DESIGN LOAD,ST	562	562	562	562	562
NUMBER OF SHIELD LEGS	2	2	2	2	2
FACE CONVEYOR HORSEPOWER	152	164	176	188	199
NPV OF LONGWALL PROJECT,$ 1983	14438795	15112539	15807477	16561210	17101167

OVERBURDEN THICKNESS,FT	1000	1000	1000	1000	1000
THICKNESS OF STRONGEST BED,FT	15	15	15	15	15
LENGTH OF ROOF OVERHANG,FT	23	23	23	23	23
COAL EXTRACTION HEIGHT,FT	10	10	10	10	10
CONTINUOUS MINER PROD,TPSH	350	350	350	350	350
ENTRY PILLAR LENGTH,FT	80	80	80	80	80
ENTRY PILLAR WIDTH,FT	40	40	40	40	40
ENTRY WIDTH,FT	20	20	20	20	20
NUMBER OF ENTRIES	3	3	3	3	3
LONGWALL FACE LENGTH,FT	360	380	400	420	440
MAXIMUM FACE LENGTH,FT	1060	1060	1060	1060	1060
MINIMUM FACE LENGTH,FT	88	88	88	88	88
LONGWALL PANEL LENGTH,FT	4200	4000	4000	4000	4000
FACE SUPPORT DESIGN LOAD,ST	562	562	562	562	562
NUMBER OF SHIELD LEGS	2	2	2	2	2
FACE CONVEYOR HORSEPOWER	211	223	235	246	258
NPV OF LONGWALL PROJECT,$ 1983	17736298	18167485	18686339	19230591	19603734

OVERBURDEN THICKNESS,FT	1000	1000	1000	1000	1000
THICKNESS OF STRONGEST BED,FT	15	15	15	15	15
LENGTH OF ROOF OVERHANG,FT	23	23	23	23	23
COAL EXTRACTION HEIGHT,FT	10	10	10	10	10
CONTINUOUS MINER PROD,TPSH	350	350	350	350	350
ENTRY PILLAR LENGTH,FT	80	80	80	80	80
ENTRY PILLAR WIDTH,FT	40	40	40	40	40
ENTRY WIDTH,FT	20	20	20	20	20
NUMBER OF ENTRIES	3	3	3	3	3
LONGWALL FACE LENGTH,FT	460	480	500	520	540
MAXIMUM FACE LENGTH,FT	1060	1060	1060	1060	1060
MINIMUM FACE LENGTH,FT	88	88	88	88	88
LONGWALL PANEL LENGTH,FT	4000	4000	4000	4000	4000
FACE SUPPORT DESIGN LOAD,ST	562	562	562	562	562
NUMBER OF SHIELD LEGS	2	2	2	2	2
FACE CONVEYOR HORSEPOWER	270	282	293	305	317
NPV OF LONGWALL PROJECT,$ 1983	20051537	20226778	20906425	20981840	21106521

OVERBURDEN THICKNESS,FT	1000	1000	1000	1000	1000
THICKNESS OF STRONGEST BED,FT	15	15	15	15	15
LENGTH OF ROOF OVERHANG,FT	23	23	23	23	23
COAL EXTRACTION HEIGHT,FT	10	10	10	10	10
CONTINUOUS MINER PROD,TPSH	350	350	350	350	350
ENTRY PILLAR LENGTH,FT	80	80	80	80	80
ENTRY PILLAR WIDTH,FT	40	40	40	40	40
ENTRY WIDTH,FT	20	20	20	20	20
NUMBER OF ENTRIES	3	3	3	3	3
LONGWALL FACE LENGTH,FT	560	580	600	620	640
MAXIMUM FACE LENGTH,FT	1060	1060	1060	1060	1060
MINIMUM FACE LENGTH,FT	88	88	88	88	88
LONGWALL PANEL LENGTH,FT	4000	4000	4000	4000	4000
FACE SUPPORT DESIGN LOAD,ST	562	562	562	562	562
NUMBER OF SHIELD LEGS	2	2	2	2	2
FACE CONVEYOR HORSEPOWER	329	340	352	364	376
NPV OF LONGWALL PROJECT,$ 1983	21423551	21716866	21933399	21798667	21791132

OVERBURDEN THICKNESS,FT	1000	1000	1000	1000	1000
THICKNESS OF STRONGEST BED,FT	15	15	15	15	15
LENGTH OF ROOF OVERHANG,FT	23	23	23	23	23
COAL EXTRACTION HEIGHT,FT	10	10	10	10	10
CONTINUOUS MINER PROD,TPSH	350	350	350	350	350
ENTRY PILLAR LENGTH,FT	80	80	80	80	80
ENTRY PILLAR WIDTH,FT	40	40	40	40	40
ENTRY WIDTH,FT	20	20	20	20	20
NUMBER OF ENTRIES	3	3	3	3	3
LONGWALL FACE LENGTH,FT	660	680	700	720	740
MAXIMUM FACE LENGTH,FT	1060	1060	1060	1060	1060
MINIMUM FACE LENGTH,FT	88	88	88	88	88
LONGWALL PANEL LENGTH,FT	4000	4000	4000	4000	4000
FACE SUPPORT DESIGN LOAD,ST	562	562	562	562	562
NUMBER OF SHIELD LEGS	2	2	2	2	2
FACE CONVEYOR HORSEPOWER	387	399	411	423	434
NPV OF LONGWALL PROJECT,$ 1983	21661665	21508499	21340777	21215797	21126023

OVERBURDEN THICKNESS,FT	1000	1000	1000	1000	1000
THICKNESS OF STRONGEST BED,FT	15	15	15	15	15
LENGTH OF ROOF OVERHANG,FT	23	23	23	23	23
COAL EXTRACTION HEIGHT,FT	10	10	10	10	10
CONTINUOUS MINER PROD,TPSH	350	350	350	350	350
ENTRY PILLAR LENGTH,FT	80	80	80	80	80
ENTRY PILLAR WIDTH,FT	40	40	40	40	40
ENTRY WIDTH,FT	20	20	20	20	20
NUMBER OF ENTRIES	3	3	3	3	3
LONGWALL FACE LENGTH,FT	760	780	800	820	840
MAXIMUM FACE LENGTH,FT	1060	1060	1060	1060	1060
MINIMUM FACE LENGTH,FT	88	88	88	88	88
LONGWALL PANEL LENGTH,FT	4000	4000	4000	4000	4000
FACE SUPPORT DESIGN LOAD,ST	562	562	562	562	562
NUMBER OF SHIELD LEGS	2	2	2	2	2
FACE CONVEYOR HORSEPOWER	446	458	470	481	493
NPV OF LONGWALL PROJECT,$ 1983	20842454	20673759	20639939	20396265	20238650

OVERBURDEN THICKNESS,FT	1000	1000	1000	1000	1000
THICKNESS OF STRONGEST BED,FT	15	15	15	15	15
LENGTH OF ROOF OVERHANG,FT	23	23	23	23	23
COAL EXTRACTION HEIGHT,FT	10	10	10	10	10
CONTINUOUS MINER PROD,TPSH	350	350	350	350	350
ENTRY PILLAR LENGTH,FT	80	80	80	80	80
ENTRY PILLAR WIDTH,FT	40	40	40	40	40
ENTRY WIDTH,FT	20	20	20	20	20
NUMBER OF ENTRIES	3	3	3	3	3
LONGWALL FACE LENGTH,FT	860	880	900	920	940
MAXIMUM FACE LENGTH,FT	1060	1060	1060	1060	1060
MINIMUM FACE LENGTH,FT	88	88	88	88	88
LONGWALL PANEL LENGTH,FT	4000	4000	4000	4000	4000
FACE SUPPORT DESIGN LOAD,ST	562	562	562	562	562
NUMBER OF SHIELD LEGS	2	2	2	2	2
FACE CONVEYOR HORSEPOWER	505	517	528	540	552
NPV OF LONGWALL PROJECT,$ 1983	20392402	20083500	19847265	19623780	19458661

APPENDIX F

Retreating Longwall Simulator User's Guide

In order to utilize the retreating longwall computer simulator, the user must first identify the folowing parameters for the specific case to be analyzed.

H = depth to seam, ft.

T = thickness of most competent overlying member in 4 seam heights, ft.

O = gob overhang from coal face, ft.

SHLD = shield style, 2-leg or 4-leg lemniscate

F = longwall face length, ft.

SM = seam extraction height, ft.

PD = panel length, ft.

ENTR = number of development entries, 2 or 3.

TPSH = continuous miner development productivity, ton/shift.

X = development entry pillar length, ft.

Y = development entry pillar width, ft.

E = development entry width, ft.

OUTPT = output desired, 2.0,3.0.

H

H is equal to the depth to the seam in feet. When using values of H greater than approximately 3500 ft the load-transfer equation begins to decay because LTDs drop below the maximum. This occurs because the equation is quadratic.

SHLD

Use of 2.0 as the value of SHLD, selects 2-leg shields and 4.0 has the result of selecting 4-leg shields.

F

For purposes of face length optimization, values of F from 160 ft to 940 ft by 20 ft increments would be adequate in most cases. Only after the optimum has been bracketed might the user desire to refine optimum face length and NPV estimates.

SM

Seam heights of 5 ft to 14 ft are within the capability of the simulation routine.

Outpt

When 2.0 is input for the variable output, cashflow tables and design result tables are generated for the specific case. Use of a 3.0 will generate only design result tables for the specific case.

It is imperative that all input values have decimal points.

A data file named PAPDAT.SIT must be created. The variables listed above must be input in order from top to bottom, into the data file on one line each. The values must be separated by commas. There must be 40 lines of data.

An example of such a data file, for the test case, is shown in figure 36.

On the present C.S.M. DEC-10 computer system, a program written in Fortran '77, as this program is, must first have the command .PATH/NEW (CR) executed prior to execution of the longwall simulator.

To execute the longwall simulator, the command .EXE LWSIM.FOR (CR) is entered.

Execution will occur rapidly, often in under two minutes.

The cash flow tables and/or design result tables will be printed on the computer center line printer. These outputs may be picked-up at the dispatcher's counter.

At the same time that simulation is taking place, a file name Result.Sit (Figure 37) is created. In this file, the variable values F and TNPV exist separated by a comma. When Column 2 is plotted against column 1 a curve of net present value vs. face length. The graphical routine GRAPHM can be used to plot the data of Result.Sit and generate polynomial best-fit curves for the data. The best-fit curves are generated for a sum of least squares error reduction routine. To execute GRAPHM enter .RUN GRAPHM (12,37) (CR).

The .SED editor as well as the substitute command of the .SOS editor are extremely useful when manipulating the data file.

A listing of the simulator can be found in Appendix G.

```
1000.0,15.0,23.67,2.0,160.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,180.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,200.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,220.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,240.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,260.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,280.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,300.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,320.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,340.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,360.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,380.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,400.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,420.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,440.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,460.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,480.00,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,500.00,10.0,4300.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,520.00,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,540.00,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,560.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,580.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,2.
1000.0,15.0,23.67,2.0,600.0,10.0,4300.0,3.0,350.0,80.0,40.0,20.0,2.
1000.0,15.0,23.67,2.0,620.0,10.0,4300.0,3.0,350.0,80.0,40.0,20.0,2.
1000.0,15.0,23.67,2.0,640.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1200.0,15.0,23.67,2.0,660.0,10.0,4700.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,680.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,700.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,720.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,740.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,760.0,10.0,4700.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,780.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,800.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,820.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,840.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,860.0,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,880.00,10.0,4200.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,900.00,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,920.00,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
1000.0,15.0,23.67,2.0,940.00,10.0,4000.0,3.0,350.0,80.0,40.0,20.0,3.
```

Figure 36

Example Imput Data File

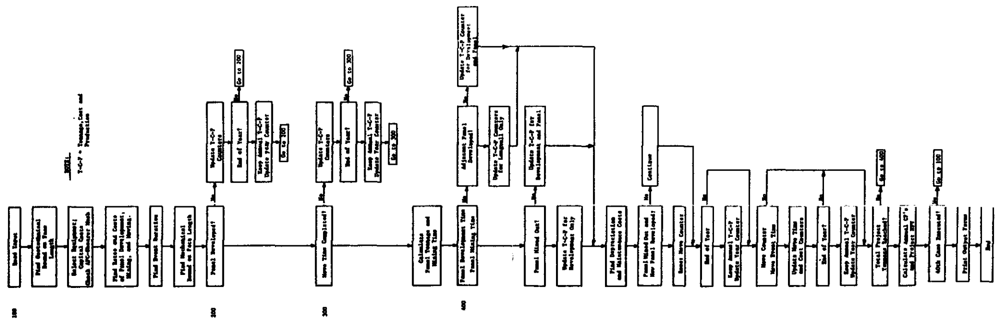
```
160.0000, 1.0537161E+07
180.0000, 1.1359335E+07
200.0000, 1.2099886E+07
220.0000, 1.3245773E+07
240.0000, 1.3715938E+07
260.0000, 1.4438795E+07
280.0000, 1.5112539E+07
300.0000, 1.5897478E+07
320.0000, 1.6561216E+07
340.0000, 1.7121168E+07
360.0000, 1.7738298E+07
380.0000, 1.8187486E+07
400.0000, 1.8686339E+07
420.0000, 1.9230591E+07
440.0000, 1.9603735E+07
460.0000, 2.0051537E+07
480.0000, 2.0226778E+07
500.0000, 2.0916426E+07
520.0000, 2.0981841E+07
540.0000, 2.1106522E+07
560.0000, 2.1423551E+07
580.0000, 2.1716866E+07
600.0000, 2.1933399E+07
620.0000, 2.1798663E+07
640.0000, 2.1791133E+07
660.0000, 2.1661686E+07
680.0000, 2.1506499E+07
700.0000, 2.1340777E+07
720.0000, 2.1215797E+07
740.0000, 2.1126224E+07
760.0000, 2.0842455E+07
780.0000, 2.0673759E+07
800.0000, 2.0639940E+07
820.0000, 2.0396265E+07
840.0000, 2.0238650E+07
860.0000, 2.0092403E+07
880.0000, 2.0080501E+07
900.0000, 1.9847285E+07
920.0000, 1.9623781E+07
940.0000, 1.9458662E+07
```

Figure 37

Example Result File

APPENDIX G

COMPUTER SIMULATION FLOW CHART



APPENDIX H

HAND SOLUTION OF SIMULATION EVENTS FOR THREE INITIAL YEARS

This appendix shows a sample simulation result over the first three years of the model. The starting conditions are those of the optimum face length of the test case. The working constraints are as follows:

- o 2 leg shields with load of 562 tons; $42,500/ea.
- o Double-ended, ranging-drum shearing machine, $1,200,000/ea. Cutting tram speed: 14.0 fpm
- o AFC cost: 600 ft x $2,167/ft = 1,300,200
- o Total equipment cost: $7,600,200
- o Continuous miner tonnage: 700 tpd
- o Continuous miner production cost: $5,176.94/day
- o Longwall tonnage: 4942 tpd
- o Longwall production cost: $9,080.29/day
- o Longwall move cost: $8,880.89/day
- o Longwall moving time: 38.04 days; therefore, 39 days
- o Three entry development; 20 ft entry width
- o Entry pillars are 80x40 ft
- o Daily development advance: 22.3 ft
- o Time to develop first panel: 385.1 days; therefore, 386 days
- o Time to develop subsequent panels: 206 days
- o Panel tonnage: 990,000 tons

The annual production results and costs are as follows:

Activity	Year	Event	Tonnage (T)	Revenue ($)	Cost ($)
Develop Initial Panel	1	240 days at 700 tpd	168,000	2,520,000	1,242,466
Complete Development	2	146 days at 700 tpd	102,200	1,533,000	755,833
Install Longwall	2	39 day longwall move	--	--	346,355
Develop Second Panel	2	55 days at 700 tpd	38,500	577,500	284,732
Start Panel	2	55 days at 4942 tpd	271,799	4,076,985	499,416
	2	Total	421,499	6,187,485	1,886,336
Complete Second Panel Development	3	151 days at 700 tpd	105,700	1,585,500	781,718
Finish Mining First Panel	3	146 days at 4942 tpd	721,503	10,822,545	1,325,722
Move Longwall	3	39 day longwall move	--	--	346,355
Start 3rd Panel Development	3	50 days at 700 tpd	35,000	525,000	258,847
Start Mining 2nd Panel	3	50 days at 4942 tpd	247,090	3,706,350	454,015
	3	Total	1,109,293	16,639,395	3,166,657

As a result of capital expenditures the depreciation and capitalized costs for the first three years are:

Year	Depreciation ($)	Capitalized Cost Cost ($)	Capital Expenditure ($)
1	0.0	10,000	0.0
2	1,520,040	10,000	7,600,200
3	2,432,064	10,000	0.0

These results are equivalent to those presented in Appendix D for the first three years of the model. The only differences result from rounding error in the hand calculations. Thus, it may be concluded from these tables that the model is correctly executing its basic function.

APPENDIX I

Retreating Longwall Simulator Code Listing

```
      DIMENSION BV(9Ø),DPC(9Ø),DPM(9Ø),P(9Ø),OC(9Ø)
      DIMENSION IAA(7Ø,3Ø),IAO(7Ø,3Ø)
      DIMENSION C(9Ø),R(9Ø),RP(9Ø),PT(9Ø),TNAC(9Ø)
      DIMENSION DP(9Ø),TNAD(9Ø),DR(9Ø),TNADD(9Ø),ST(9Ø)
      DIMENSION PTP(9Ø),FT(9Ø),TC(9Ø),TNP(9Ø),CF(9Ø)
      DIMENSION TNCF(9Ø),PV(9Ø),CTNG(9Ø),CCOST(9Ø)
      DIMENSION WCCST(9Ø),WTNG(9Ø),CP(9Ø),DPR(9Ø)
      DIMENSION ADDA(9Ø),TM(9Ø),OSM(9Ø),TPR(9Ø),FORG(9Ø)
      DIMENSION BNDS(9Ø),HS(9Ø),TS(9Ø),OS(9Ø),SMS(9Ø)
      DIMENSION XS(9Ø),YS(9Ø),ES(9Ø),ENTRS(9Ø),FS(9Ø)
      DIMENSION  ZNADD(9Ø),VPTP(9Ø),PDS(9Ø),WS(9Ø)
      DIMENSION HPS(9Ø),TNPVS(9Ø)
      DIMENSION TPSHS(9Ø),FCMXS(9Ø),SHLDS(9Ø)
      OPEN(UNIT=46,FILE='papdat.sit')
      OPEN(UNIT=17,FILE='RESULT.SIT')
      DO 2ØØØ IZ=1,4Ø
C     MINIMUM CAVING WIDTH DETERMINATION
C     WRITE(4,*) 'ENTER H,T,O,SHLD,F,SM,PD'
      READ(46,*) H,T,O,SHLD,F,SM,PD,ENTR,TPSH,X,Y,E,OUTPT
      DLTA=Ø.Ø
      AL=1ØØ.Ø
1Ø    AL=AL-DLTA/2.Ø
      ALTD=-45.Ø+Ø.373*H-Ø.ØØØØ82*H**2.Ø
      D=H/ALTD**2.Ø*(2.Ø*ALTD*AL/2.Ø-(AL/2.Ø)**2.Ø)
      ALLTD=-45.Ø+Ø.373*D-Ø.ØØØØ82*D**2.Ø
      DD=D/ALLTD**2.Ø*(2.Ø*ALLTD*AL/2.Ø-(AL/2.Ø)**2.Ø)
      ALL=(1.Ø*H*T**2.Ø/.225Ø*DD)**Ø.5
      DLTA=AL-ALL
      IF(DLTA.GT.1.Ø)THEN
      GOTO 1Ø
      ELSE
      TBND=ALL
      ENDIF
      DLTA=Ø.Ø
      AL=1ØØ.Ø
16    AL=AL-DLTA/2.Ø
      ALTD=-45.Ø+Ø.373*H-Ø.ØØØØ82*H**2.Ø
      D=H/ALTD**2.Ø*(2.Ø*ALTD*AL/2.Ø-(AL/2.Ø)**2.Ø)
      ALLTD=-45.Ø+Ø.373*D-Ø.ØØØØ82*D**2.Ø
      DD=D/ALLTD**2.Ø*(2.Ø*ALLTD*AL/2.Ø-(AL/2.Ø)**2.Ø)
      ALL=((5ØØØ.Ø-H)*(T**2.Ø)/(.4ØØ2*DD))**Ø.5
      DLTA=AL-ALL
      IF(DLTA.GT.1.Ø)THEN
      GOTO 16
      ELSE
      SBND=ALL
      ENDIF
      IF(TBND.LT.SBND)THEN
      BND=TBND
      ELSE
      BND=SBND
      ENDIF
C     END OF MINIMUM CAVING WIDTH DETERMINATION
C     WRITE(4,*)'RUNNING 1'
C
C
C     SUPPORT SELECTION CALCULATION
      W=1.44*SM*(1.5359*SM+O)
      IF(SHLD.EQ.2.Ø) THEN
      GOTO 5Ø
```

```
          ELSE
          ENDIF
          IF(W.LT.350.0)THEN
          DLR=35500.0
          GOTO 90
          ELSE
          GOTO 20
          ENDIF
20        IF(W.GT.350.0.AND.W.LT.472.0) THEN
          DLR=40500.0
          GOTO 90
          ELSE
          GOTO 30
          ENDIF
30        IF(W.GT.472.0.AND.W.LT.582.0) THEN
          DLR=44500.0
          GOTO 90
          ELSE
          GOTO 90
          ENDIF
50        IF(W.LT.350.0)THEN
          DLR=33520.0
          GOTO 90
          ELSE
          GOTO 60
          ENDIF
60        IF(W.GT.350.0.AND.W.LT.472.0)THEN
          DLR=36520.0
          GOTO 90
          ELSE
          GOTO 70
          ENDIF
70        IF(W.GT.472.0.AND.W.LT.582.0)THEN
          DLR=42500.0
          GOTO 90
          ELSE
          ENDIF
90        EN=F/5.0
          SDCT=EN*DLR
          SDMC=SDCT/2.0
          IF(SM.GE.5.0.AND.SM.LE.15.0) THEN
          DERDS=1200000.0
          V=14.0
          ELSE
          ENDIF
          DRDMC=DERDS/2.0
          AFC=F*2167.0
          AFCM=AFC/2.0
C         END EQUIPMENT SELECTION CALCULATION
C
C
C
C
          DO 100 IA=1,90
          C(IA)=0.0
          P(IA)=0.0
          OC(IA)=0.0
          CP(IA)=0.0
          FORG(IA)=0.0
          CP(IA)=0.0
```

```
      ADPR(IA)=0.0
      DPR(IA)=0.0
192   CONTINUE
      NPV=0.0
C
      CAP=0.0
      M=5000000000.0
      J=0
      K=0
      IJ=0
      JJ=0
      TWIN=0.0
      IYEAR=1
      IB=IYEAR
      CTON=0.0
      WTON=0.0
      DCMC=0.0
      DWC=0.0
      PASS=0.0
      IDAY=0
      PTCN=0.0
      TWTN=0.0
      CTPD=2.0*TPSH
C     WRITE(4,*)'RUNNING 2',IDAY
      CPC=2.04*CTPD+3743.94
      WTPD=(2.0*241.66/((2.0*(F-95.0)/V)+18.63))*.75*.20625*F*SM
      IF(WTPD.GT.6445.0)THEN
      WTPD=6445.0
      ELSE
      CONTINUE
      ENDIF
C     WRITE(1,*)'WTPD=',WTPD
      WPC=.73*WTPD+5522.20
      AMVC=6380.59
      AWVIM=F/15.769
      IF(ENTR.EQ.2.0)THEN
      DA=(X+E)*CTPD/(((X+E)*F*2.0+Y*E)*SM*(82.5/2000.))
      ELSE
      DA=(X+E)*CTPD/(((X+E)*F*3.0+2.0*Y*E)*SM*(82.5/2000.))
      ENDIF
      DAY=((2.0*PD)+F)/DA
      DDAY=(PD+F)/DA
      WC=V*SH*30.0/12.0*82.5/(250.0-V)
      IF(WC.GT.106.66)THEN
      WC=106.66
      ELSE
      CONTINUE
      ENDIF
      FMAX=93843./((.4*WC+28.53)*1.232)
      FFMAX=137541./((.5*WC+28.53)*1.232)
      IF(FMAX.GT.FFMAX) THEN
      FCMX=FFMAX
      ELSE
      FCMX=FMAX
      ENDIF
      HP=250.0*(.4*WC+34.89)*F/33000.0
210   IF(IDAY.LE.DAY)THEN
      CTON=CTON+CTPD
      DCMC=DCMC+CPC
      IDAY=IDAY+1
```

```
            ELSE
            GOTO 220
            ENDIF
            IF(IDAY.GE.240)THEN
            IB=IYEAR
            CTNG(IB)=CTON
            CCOST(IB)=DCMC
            OC(IB)=DCMC
            P(IB)=CTON
            CTON=0.0
            DCMC=0.0
            IDAY=IDAY-240
            DAY=DAY-240.0
            IYEAR=IYEAR+1
            GOTO 210
            ELSE
            GOTO210
            ENDIF
220         J=0
C           WRITE(4,*)'RUNNING 2.5',IB,IDAY
221         IF(J.LE.WMVTM)THEN
            DWC=DWC+WMVC
            IDAY=IDAY+1
            J=J+1
            ELSE

            GOTO 230
            ENDIF
            IF(IDAY.GE.240)THEN
            IB=IYEAR
            CTNG(IB)=CTON
            CCOST(IB)=DCMC
            WCOST(IB)=DWC
            OC(IB)=DCMC+DWC
            P(IB)=CTON
            CTON=0.0
            DCMC=0.0
            DWC=0.0
            IDAY=IDAY-240
            IYEAR=IYEAR+1
            GOTO 221
            ELSE
            GOTO 221
            ENDIF
230         TPAN=(F*PD*SM)*(82.5/2000.0)
            LLL=NINT(TPAN/WTPD)
            JDAY=NINT(DDAY)
C           WRITE(4,*)'RUNNING 3',IB,IDAY
1000        IF (JDAY.GT.LLL) THEN
            IF(K.LE.LLL)THEN
            PTON=PTON+WTPD
            TWTN=TWTN+WTPD
C           WRITE(4,*) PTON,TWTN
            IDAY=IDAY+1
            WTON=WTON+WTPD
            CTON=CTON+CTPD
            DWC=DWC+WPC
            DCMC=DCMC+CPC
            K=K+1
            ELSE
```

```
      ENDIF
      IF(K.GT.LLL.AND.K.LE.JDAY)THEN
      IDAY=IDAY+1
      CTON=CTON+CTPD
      DCMC=DCMC+CPC
      K=K+1
      ELSE
      ENDIF
      ELSE
      IF(K.LE.JDAY)THEN
      PTON=PTON+WTPD
      TWTN=TWTN+WTPD
      IDAY=IDAY+1
      WTON=WTON+WTPD
      CTON=CTON+CTPD
      DWC=DWC+WPC
      DCMC=DCMC+CPC
      K=K+1
      ELSE
      PTON=PTON+WTPD
      TWTN=TWTN+WTPD
      IDAY=IDAY+1
      WTON=WTON+WTPD
      DWC=DWC+WPC
      ENDIF
      ENDIF
      IB=IYEAR
      IF(TWTN.GT.0.0.AND.TWTN.LT.14000.0)THEN
      CP(IB)=SDCT+DERDS+AFC
      ELSE
      CONTINUE
      ENDIF
      IF(TWTN.GT.1494000.0.AND.TWTN.LT.1506000.0)THEN
      CP(IB)=(107.4*F)+10000.0
      ELSE
      CONTINUE
      ENDIF
      IF(TWTN.GT.2994000.0.AND.TWTN.LT.3006000.0)THEN
      CP(IB)=(107.4*F)+10000.
      ELSE
      CONTINUE
      ENDIF
      IF(TWTN.GT.4494000.0.AND.TWTN.LT.4506000.0)THEN
      CP(IB)=(107.4*F)+10000.
      ELSE
      CONTINUE
      ENDIF
      IF(TWTN.GT.5996000.0.AND.TWTN.LT.6006000.0)THEN
      CP(IB)=(107.4*F)+10000.
      ELSE
      CONTINUE
      ENDIF
      IF(TWTN.GT.7496000.0.AND.TWTN.LT.7506000.0)THEN
      CP(IB)=SDMC+(107.4*F)+12000.
      ELSE
      CONTINUE
      ENDIF
      IF(TWTN.GT.8994000.0.AND.TWTN.LT.9006000.0)THEN
      CP(IB)=(107.4*F)+10000.
      ELSE
```

```
      CONTINUE
      ENDIF
      IF(TWTN.GT.17496000.0.AND.TWTN.LT.10506000.0)THEN
      CP(IB)=(107.4*F)+10000.
      ELSE
      CONTINUE
      ENDIF
      IF(TWTN.GT.11996000.0.AND.TWTN.LT.12006000.0)THEN
      CP(IB)=(107.4*F)+10000.
      ELSE
      CONTINUE
      ENDIF
      IF(TWTN.GT.13496000.0.AND.TWTN.LT.13506000.0)THEN
      CP(IB)=(107.4*F)+10000.
      ELSE
      CONTINUE
      ENDIF
C     IF(TWTN.GT.14960000.0.AND.TWTN.LT.15006000.0)THEN
C     CP(IB)=SDCT+DADMC+AFCM
C     ELSE
C     CONTINUE
C     ENDIF
      IF(TWTN.GT.16496000.0.AND.TWTN.LT.16506000.0)THEN
      CP(IB)=(107.4*F)+10000.
      ELSE
      CONTINUE
      ENDIF
      IF(TWTN.GT.17996000.0.AND.TWTN.LT.18006000.0)THEN
      CP(IB)=(107.4*F)+10000.
      ELSE
      CONTINUE
      ENDIF
      IF(TWTN.GT.19496000.0.AND.TWTN.LT.19506000.0)THEN
      CP(IB)=(107.4*F)+10000.
      ELSE
      CONTINUE
      ENDIF
      IF(TWTN.GT.20996000.0.AND.TWTN.LT.21006000.0)THEN
      CP(IB)=(107.4*F)+10000.
      ELSE
      CONTINUE
      ENDIF
      IF(TWTN.GT.22496000.0.AND.TWTN.LT.22506000.0)THEN
      CP(IB)=(107.4*F)+10000.+SDMC
      ELSE
      CONTINUE
      ENDIF
      IF(TWTN.GT.23996000.0.AND.TWTN.LT.24006000.0)THEN
      CP(IB)=(107.4*F)+10000.
      ELSE
      CONTINUE
      ENDIF
      IF(TWTN.GT.25496000.0.AND.TWTN.LT.25506000.0)THEN
      CP(IB)=(107.4*F)+10000.
      ELSE
      CONTINUE
      ENDIF
      IF(TWTN.GT.26994000.0.AND.TWTN.LT.27006000.0)THEN
      CP(IB)=(107.4*F)+10000.
      ELSE
```

```
      CONTINUE
      ENDIF
      IF(TWTN.GT.28496000.0.AND.TWTN.LT.28506000.0)THEN
      CP(IB)=(107.4*F)+10000.
      ELSE
      CONTINUE
      ENDIF
      IF(TWTN.GT.4996000.0.AND.TWTN.LT.5006000.0)THEN
      CP(IB)=DRDMC+AFCM
      ELSE
      CONTINUE
      ENDIF
      IF(TWTN.GT.9996000.0.AND.TWTN.LT.10006000.0)THEN
      CP(IB)=DERDS+AFC
      ELSE
      CONTINUE
      ENDIF
      IF(TWTN.GT.19996000.0.AND.TWTN.LT.20006000.0)THEN
      CP(IB)=DERDS+AFC
      ELSE
      CONTINUE
      ENDIF
      IF(TWTN.GT.24996000.0.AND.TWTN.LT.25006000.0)THEN
      CP(IB)=DRDMC+AFCM
      ELSE
      CONTINUE
      ENDIF
C     WRITE(4,*)'RUNNING 3.5',IB,TWTN,IDAY,JDAY,K,PTON,TPAN
      IF(IB.EQ.1)THEN
      DPR(IB)=0.2*CP(IB)
      ADPR(IB)= 0.1*CP(IB)
      ELSE
      CONTINUE
      ENDIF
      IF(IB.EQ.2)THEN
      DPR(IB)=0.2*CP(IB)+0.32*CP(IB-1)
      ADPR(IB)=0.1*(CP(IB)+CP(IB-1))
      ELSE
      CONTINUE
      ENDIF
      IF(IB.EQ.3)THEN
      DPR(IB)=0.2*CP(IB)+0.32*CP(IB-1)+0.24*CP(IB-2)
      ADPR(IB)=0.1*(CP(IB)+CP(IB-1)+CP(IB-2))
      ELSE
      CONTINUE
      ENDIF
      IF(IB.EQ.4)THEN
      DPR(IB)=0.2*CP(IB)+0.32*CP(IB-1)+0.24*CP(IB-2)+0.16*CP(IB-3)
      ADPR(IB)=0.1*(CP(IB)+CP(IB-1)+CP(IB-2)+CP(IB-3))
      ELSE
      CONTINUE
      ENDIF
      IF(IB.EQ.5)THEN
      DPR(IB)=0.2*CP(IB)+0.32*CP(IB-1)+0.24*CP(IB-2)+0.16*CP(IB-3)
     1 +0.08*CP(IB-4)
      ADPR(IB)=0.1*(CP(IB)+CP(IB-1)+CP(IB-2)+CP(IB-3)+CP(IB-4))
      ELSE
      CONTINUE
      ENDIF
      IF(IB.GE. 6)THEN
```

```
DPR(IB)=0.2*CP(IB)+0.32*CP(IB-1)+0.24*CP(IB-2)+0.16*CP(IB-3)
1 +0.08*CP(IB-4)
ELSE
ENDIF
IF(IB.EQ.6)THEN
ADPR(IB)=0.1*(CP(IB)+CP(IB-1)+CP(IB-2)+CP(IB-3)+CP(IB-4)
1 +CP(IB-5))

ELSE
CONTINUE
ENDIF
IF(IB.EQ.7)THEN
ADPR(IB)=0.1*(CP(IB)+CP(IB-1)+CP(IB-2)+CP(IB-3)+CP(IB-4)
1 +CP(IB-5)+CP(IB-6))
ELSE
CONTINUE
ENDIF
IF(IB.EQ.8)THEN
ADPR(IB)=0.1*(CP(IB)+CP(IB-1)+CP(IB-2)+CP(IB-3)+CP(IB-4)
1 +CP(IB-5)+CP(IB-6)+CP(IB-7))
ELSE
CONTINUE
ENDIF
IF(IB.EQ.9)THEN
ADPR(IB)=0.1*(CP(IB)+CP(IB-1)+CP(IB-2)+CP(IB-3)+CP(IB-4)
1 +CP(IB-5)+CP(IB-6)+CP(IB-7)+CP(IB-8))
ELSE
CONTINUE
ENDIF
IF(IB.GE.10)THEN
ADPR(IB)=0.1*(CP(IB)+CP(IB-1)+CP(IB-2)+CP(IB-3)+CP(IB-4)
1 +CP(IB-5)+CP(IB-6)+CP(IB-7)+CP(IB-8)+CP(IB-9))
ELSE
CONTINUE
ENDIF
IF(PTON.GT.TPAN.AND.K.GE.JDAY)THEN
M=0
K=0
ELSE
CONTINUE
ENDIF
IF(IDAY.GE.240)THEN
IB=IYEAR
CTNG(IB)=CTON
CCOST(IB)=DCMC
WTNG(IB)=WTON
WCOST(IB)=DWC
CC(IB)=DCMC+DWC
P(IB)=CTON+WTON
CTON=0.0
PASS=0.0
DCMC=0.0
WTON=0.0
DWC=0.0
IDAY=IDAY-240
IYEAR=IYEAR+1
CONTINUE
ELSE
CONTINUE
ENDIF
```

```
900     IF(M.LE.NMVTM)THEN
        IDAY=IDAY+1
        PTON=0.0
        DWC=DWC+WMVC
        M=M+1
        ELSE
        GOTO 235
        ENDIF
        IF(IDAY.GE.240)THEN
        IB=IYEAR
        P(IB)=WTON+CTON
        OC(IB)=DCMC+DWC
        CTNG(IB)=CTON
        CCOST(IB)=DCMC
        WTNG(IB)=WTON
        WCOST(IB)=DWC
        CTON=0.0
        DCMC=0.0
        WTON=0.0
        DWC=0.0
        IDAY=IDAY-240
        IYEAR=IYEAR+1
        GOTO 900
        ELSE
        GOTO 900
        ENDIF

235     IF(TWTN.GT.1500000.0)THEN
        GO TO 105
        ELSE
        GOTO 1000
        ENDIF
240     JJ=0
C
C
C
C
C       CAPITAL COST GENERATION
C
105        QB=0.0
        DFFTA=0.0
        DFFTB=0.0
        RR=0.0
        DFSTA=0.0
        DFSTB=0.0
        CAP=0.0
C       START CASH FLOW CALCULATION
        TNPV=0.0
        DO 110 ID=1,30
        BV(ID)=0.0
        DPC(ID)=0.0
        DPM(ID)=0.0
        R(ID)=P(ID)*15.00
        TR=RR+R(ID)
        RR=TR
C       WRITE(4,*)'RUNNING 4'
        TNPV=0.0
110     CONTINUE
        DO 200 I=1,30
        R(I)=P(I)*15.00
```

```
IF(I.LE.10)THEN
WKCAP=0.25*(TR/30.0)
ELSE
CONTINUE
ENDIF
IF(I.GT.10)THEN
WKCAP=0.0
ELSE
ENDIF
RP(I)=0.080*R(I)
OSM(I)=0.15*R(I)
IF(I.GT.10)THEN
J=J-10
ELSE
J=I
ENDIF
IF(I.GT.15)THEN
K=I-15
ELSE
K=I
ENDIF
IF(I.GT.1)THEN
CAP=BV(I-1)
ELSE
CAP=0.0
ENDIF
BV(I)=CAP+CP(I)-ADPR(I)
IF(BV(I).LT.0.0)THEN
BV(I)=0.0
ELSE
ENDIF
GP=R(I)-P(I)*2.50
RNP=GP-OC(I)
FGP=GP*0.25
IF(FGP.LT.RNP)THEN
AVC=RNP
ELSE
AVC=FGP
ENDIF
PT(I)=((0.30*BV(I)+AVC)*60.0/1000.0)
TNAC(I)=R(I)-RP(I)-OC(I)-PT(I)-OSM(I)
IF(I.LE.10)THEN
DP(I)=DPR(I)+10000./10.0
ELSE
DP(I)=DPR(I)
ENDIF
TNAD(I)=TNAC(I)-DP(I)
STX=0.0256*TNAD(I)
TNAST=TNAD(I)-STX
FTDR=0.50*TNAST
STDR=0.10*(R(I)-RP(I))
IF(FTDR.LE.0.0)THEN
FTDR=0.0
ELSE
FTDR=0.50*TNAST
ENDIF
IF(STDR.LE.0.0)THEN
STDR=0.0
ELSE
STDR=0.10*(R(I)-RP(I))
```

```
ENDIF
IF(STDR.GT.FTDR)THEN
DR(I)=FTDR
ELSE
DR(I)=STDR
ENDIF
TNADD(I)=TNAD(I)-DR(I)
IF(TNADD(I).LT.0.0)THEN
DFSTA=DFSTB+TNADD(I)
DFSTB=DFSTA
ZNADD(I)=0.0
ELSE
BFSTA=-1.0*DFSTA
IF(BFSTA.GT.TNADD(I))THEN
DFSTB=DFSTA+TNADD(I)
ZNADD(I)=0.0
DFSTA=DFSTB
ELSE
PNADD=TNADD(I)
ZNADD(I)=PNADD+DFSTA
DFSTA=0.0
DFSTB=0.0
ENDIF
ENDIF


ST(I)=0.05*ZNADD(I)+0.30*(P(I)-32000.)
IF(ST(I).LT.0.0)THEN
ST(I)=0.0
ELSE
ENDIF
PTP(I)=TNADD(I)-ST(I)
IF(PTP(I).LT.0.0)THEN
DFFTA=DFFTB+PTP(I)
DFFTB=DFFTA
VPTP(I)=0.0
ELSE
GFSTA=-1.0*DFFTA
IF(GFSTA.GT.PTP(I))THEN
DFFTB=DFFTA+PTP(I)
VPTP(I)=0.0
DFFTA=DFFTB
ELSE
GNADD=PTP(I)
VPTP(I)=GNADD+DFFTA
DFFTA=0.0
DFFTB=0.0
ENDIF
ENDIF

FT(I)=0.46*VPTP(I)
IF(FT(I).LT.0.0)THEN
FT(I)=0.0
ELSE
ENDIF
TPR(I)=DR(I)
DIFF=TPR(I)-FT(I)
IF(DIFF.GT.0.0)THEN
TM(I)=0.15*DIFF
ELSE
```

```
      TM(I)=0.0
      ENDIF
      QC=0.10*CP(I)*1.0+QB
      TCC=25000.0+0.85*(FT(I)-25000.0)
      IF(QC.LT.TCC)THEN
      TC(I)=QC
      ELSE
      TC(I)=TCC
      ENDIF
      IF(TC(I).LT.0.0)THEN
      TC(I)=0.0
      ELSE
      ENDIF
      QB=QC-TC(I)
      TNP(I)=PTP(I)-FT(I)+TC(I)-TM(I)
      CF(I)=TNP(I)+DP(I)+DR(I)
      IF(P(I).EQ.0.0)THEN
      TNCF(I)=0.0
      ELSE
      TNCF(I)=CF(I)-CP(I)-(0.1*WKCAP)
      ENDIF
      PV(I)=TNCF(I)/((1.0+0.16/12.0)**(12.0*FLOAT(I)))
C     WRITE(6,*)R(I),RP(I),OC(I),OSM(I),BV(I),PT(I),TNAC(I)
C     WRITE(6,*)DP(I),TNAD(I),DR(I),ST(I),PTP(I),FT(I)
C     WRITE(6,*)TPF(I),TM(I),TC(I),TNP(I),CF(I),TNCF(I)
      IAA(I,1)=I
      IAA(I,2)=R(I)
      IAA(I,3)=RP(I)+OSM(I)
      IAA(I,4)=OC(I)
      IAA(I,5)=PT(I)
      IAA(I,6)=TNAC(I)
      IAA(I,7)=DP(I)
      IAA(I,8)=TNAD(I)
      IAA(I,9)=DR(I)
      IAA(I,10)=ST(I)
      IAA(I,11)=PTP(I)
      IAA(I,12)=FT(I)
      IAA(I,13)=TM(I)
      IAA(I,14)=TC(I)
      IAA(I,15)=TNP(I)
      IAA(I,16)=CF(I)
      IAA(I,17)=CP(I)
      IAA(I,18)=TNCF(I)
      IAA(I,19)=PV(I)
C     IAA(I,20)=BV(I)
C     IAA(I,21)=P(I)
      IF(R(I).GT.0.0)THEN
      TNPV=TNPV+PV(I)
      ELSE
      CONTINUE
      ENDIF
200   CONTINUE
      IF(OUTPT.EQ.2.0)THEN
      DO 101 IO=1,30,6
      KO=IO+5
101   WRITE(6,1005)((IAA(MO,JO),MO=IO,KO),JO=1,19
     + )
      WRITE(6,*)TNPV,CP(I),'YEAR=',I,'F=',F,'PRODUCTION=',P(I)
1005  FORMAT(1H1,//,
     +' YEAR                                        ',6I10,//
```

```
     +   ' REVENUE                                   ',6I10,//
     +   ' ROYALTY/O&M                               ',6I10,//
     +   ' OPERATING COST                            ',6I10,//
     +   ' PROPERTY TAX                              ',6I10,//
     +   ' NET AFTER COST                            ',6I10,//
     +   ' DEPRECIATION                              ',6I10,//
     +   ' NET AFTER DEPRECIATION                    ',6I10,//
     +   ' DEPLETION                                 ',6I10,//
     +   ' STATE & SEVERANCE TAXES                   ',6I10,//
     +   ' PRE-TAX PROFITS                           ',6I10,//
     +   ' FEDERAL INCOME TAXES                      ',6I10,//
     +   ' MINIMUM TAX                               ',6I10,//
     +   ' TAX CREDITS                               ',6I10,//
     +   ' NET PROFITS                               ',6I10,//
     +   ' CASH FLOW                                 ',6I10,//
     +   ' CAPITAL EXPENDITURE                       ',6I10,//
     +   ' NET CASH FLOW/INC. WORK CAP.              ',6I10,//
     +   ' NET PRESENT VALUE                         ',6I10,//)
         ELSE
         ENDIF
         WRITE(4,*)'NPV=',TNPV,'LOW BND =',BND,'MAX BND=',FMAX
C        WRITE(4,*) 'WORKING CAP=',WKCAP
         WRITE(17,*) F,TNPV
         BNDS(IZ)=BND
         HS(IZ)=H
         TS(IZ)=T
         OS(IZ)=O
         SMS(IZ)=SM
         TPSHS(IZ)=TPSH
         XS(IZ)=X
         YS(IZ)=Y
         ES(IZ)=E
         ENTRS(IZ)=ENTR
         FS(IZ)=F
         FCMXS(IZ)=FCMX
         PDS(IZ)=PD
         WS(IZ)=W
         SHLDS(IZ)=SHLD
         HPS(IZ)=HP
         TNPVS(IZ)=TNPV
C
         IAQ(IZ,1)=HS(IZ)
         IAQ(IZ,2)=TS(IZ)
         IAQ(IZ,3)=OS(IZ)
         IAQ(IZ,4)=SMS(IZ)
         IAQ(IZ,5)=TPSHS(IZ)
         IAQ(IZ,6)=XS(IZ)
         IAQ(IZ,7)=YS(IZ)
         IAQ(IZ,8)=ES(IZ)
         IAQ(IZ,9)=ENTRS(IZ)
         IAQ(IZ,10)=FS(IZ)
         IAQ(IZ,11)=FCMXS(IZ)
         IAQ(IZ,12)=BNDS(IZ)
         IAQ(IZ,13)=PDS(IZ)
         IAQ(IZ,14)=WS(IZ)
         IAQ(IZ,15)=SHLDS(IZ)
         IAQ(IZ,16)=HPS(IZ)
         IAQ(IZ,17)=TNPVS(IZ)
  200    CONTINUE
         DO 541 IQ=1,40,5
```

```
      KQ=IQ+4
541   WRITE(6,1036)((IAQ(MQ,JQ),MQ=IQ,KQ),JQ=1,17
     + )
1036  FORMAT(1H1,//,
     + ' OVERBURDEN THICKNESS,FT            ',5I10,//
     + ' THICKNESS OF STRONGEST BED,FT      ',5I10,//
     + ' LENGTH OF ROOF OVERHANG,FT         ',5I10,//
     + ' COAL EXTRACTION HEIGHT,FT          ',5I10,//
     + ' CONTINUOUS MINER PROD,TPSH         ',5I10,//
     + ' ENTRY PILLAR LENGTH,FT             ',5I10,//
     + ' ENTRY PILLAR WIDTH,FT              ',5I10,//
     + ' ENTRY WIDTH,FT                     ',5I10,//
     + ' NUMBER OF ENTRIES                  ',5I10,//
     + ' LONGWALL FACE LENGTH,FT            ',5I10,//
     + ' MAXIMUM FACE LENGTH,FT             ',5I10,//
     + ' MINIMUM FACE LENGTH,FT             ',5I10,//
     + ' LONGWALL PANEL LENGTH,FT           ',5I10,//
     + ' FACE SUPPORT DESIGN LOAD,ST        ',5I10,//
     + ' NUMBER OF SHIELD LEGS              ',5I10,//
     + ' FACE CONVEYOR HORSEPOWER           ',5I10,//
     + ' NPV OF LONGWALL PROJECT,$ 1983     ',5I10,//)
      END
```

APPENDIX J

Polynomial Best-Fit Equations of Curves Presented in the Text

BEST-FIT EQUATIONS OF CURVES

Figure No.	X axis	Y axis	Coefficients of X^N X^o	X^1	X^2	X^3
27	Opt. face length 12 ft.	Develop Productivity 3 entry	1681.682	-5.5837	$.81549 \times 10^{-2}$	$-.42345 \times 10^{-5}$
27	Opt. face length 10 ft.	Develop Productivity 3 entry	1676.327	-6.9368	$.12978 \times 10^{-1}$	$-.88101 \times 10^{-5}$
27	Opt. face length 8 ft.	Develop Productivity 3 entry	1491.604	-6.9485	$.14638 \times 10^{-1}$	$-.11377 \times 10^{-4}$
27	Opt. face length 6 ft.	Develop Productivity 3 entry	2045.904	-15.819	$.50273 \times 10^{-1}$	$-.55899 \times 10^{-4}$
19	Opt. face length 12 ft.	Opt. NPV 3 entry	15.844	$.1100 \times 10^{-1}$	$.88619 \times 10^{-4}$	$-.0000 \times 10^{-6}$
19	Opt. face length 10 ft.	Opt. NPV 3 entry	-6.6900	.16367	$-.27800 \times 10^{-3}$	$.17556 \times 10^{-6}$
19	Opt. face length 8 ft.	Opt. NPV 3 entry	19.782	$-.28865 \times 10^{-1}$	$.12490 \times 10^{-3}$	$-.1044 \times 10^{-6}$
19	Opt. face length 6 ft.	Opt. NPV 3 entry	-2.9014	$.95843 \times 10^{-1}$	$-.14943 \times 10^{-3}$	$.80000 \times 10^{-7}$
20	Opt. face length 2 entry	Seam Height 300 TPSH	-45.875	.47798	$-.14286 \times 10^{-2}$	$.14881 \times 10^{-5}$
20	Opt. face length 2 entry	Seam Height 400 TPSH	-120.000	1.5400	$-.61667 \times 10^{-2}$	$.83333 \times 10^{-5}$

BEST-FIT EQUATIONS OF CURVES (cont.)

Figure No.	X axis	Y axis	Coefficients of X^N X^o	X^1	X^2	X^3
20	Opt. face length 2 entry	Seam Height 500 TPSH	-21.000	.15000	0.000	0.000
18	Opt. face length 3 entry	Seam Height 300 TPSH	-1.2459	$.28065 \times 10^{-1}$	$-.19383 \times 10^{-4}$	$.7633 \times 10^{-8}$
18	Opt. face length 3 entry	Seam Height 400 TPSH	-1.3333	$.49484 \times 10^{-1}$	$-.95238 \times 10^{-4}$	$.99206 \times 10^{-7}$
18	Opt. face length 3 entry	Seam Height 500 TPSH	-36.857	.42262	$-.13393 \times 10^{-2}$	$.14881 \times 10^{-5}$
18	Opt. face length 3 entry	Seam Height 600 TPSH	-39.586	$.97792 \times 10^{-2}$	$-.67566 \times 10^{-6}$	$.16330 \times 10^{-10}$
21	Seam Height 2 entry	Opt. NPV 300 TPSH	-2.7900	2.7633	$.97500 \times 10^{-1}$	$-.95833 \times 10^{-2}$
21	Seam Height 2 entry	Opt. NPV 400 TPSH	-13.3200	6.6558	-.33125	$.66667 \times 10^{-2}$
21	Seam Height 2 entry	Opt. NPV 500 TPSH	-8.6400	5.0175	-.13125	$.11170 \times 10^{-7}$
21	Seam Height 2 entry	Opt. NPV 600 TPSH	-10.1100	5.5850	-.19000	$.25000 \times 10^{-2}$
19	Seam Height 3 entry	Opt. NPV	-33.060	13.9908	-1.2513	$.40417 \times 10^{-1}$

BEST-FIT EQUATIONS OF CURVES (cont.)

Figure No.	X axis	Y axis	Coefficients of X^N: X^o	X^1	X^2	X^3
19	Seam Height 3 entry	Opt. NPV 400 TPSH	8.9200	-.45250	.41125	$-.2000\times10^{-1}$
19	Seam Height 3 entry	Opt. NPV 500 TPSH	-27.230	12.4933	-1.0325	$.32917\times10^{-1}$
19	Seam Height 3 entry	Opt. NPV 600 TPSH	-143.4558	18.8789	-.72222	$.97531\times10^{-2}$
25	Opt. face length 2-leg shield	Seam Height	-1.3333	$.49484\times10^{-1}$	$-.95238\times10^{-4}$	$.99206\times10^{-7}$
25	Opt. face length 4-leg shield	Seam Height	-1.3333	$.49484\times10^{-1}$	$-.95238\times10^{-4}$	$.99206\times10^{-7}$
26	Seam Height 2-leg shield	Opt. NPV	9.1200	-.51416	.41750	$-.20208\times10^{-1}$
26	Seam Height 4-leg shield	Opt. NPV	9.8500	-.77750	.44625	$-.21250\times10^{-1}$
29	Opt. face length 6 ft.	Entry Width 3 entry	-138.00	1.8458	$-.75000\times10^{-2}$	$.10417\times10^{-4}$
29	Opt. face length 8 ft.	Entry Width 3 entry	0.00	$.500\times10^{-1}$	0.00	0.00
29	Opt. face length 10 ft.	Entry Width 3 entry	-929.000	6.0825	$-.13063\times10^{-1}$	$.93750\times10^{-5}$

BEST-FIT EQUATIONS OF CURVES (cont.)

Figure No.	X axis	Y axis	Coefficients of X^N: X^o	X^1	X^2	X^3
29	Opt. face length 12 ft.	Entry Width 3 entry	403.109	-2.02422	$.34766 \times 10^{2}$	$-.19531 \times 10^{-5}$
30	Entry Width 6 ft seam	Opt. NPV 3 entry	197.479	-30.1028	1.6686	$-.30833 \times 10^{-1}$
30	Entry Width 8 ft seam	Opt. NPV 3 entry	-233.780	41.5821	-2.2421	$.40000 \times 10^{-1}$
30	Entry Width 10 ft seam	Opt. NPV 3 entry	-120.168	25.2169	-1.4336	$.2667 \times 10^{1}$
30	Entry Width 12 ft. seam	Opt. NPV 3 entry	81.899	-7.9654	.40786	$-.75000 \times 10^{-2}$
31	Opt. face length 6 ft. seam	Panel Length	Equation: $X=220$			
31	Opt. face length 8 ft. seam	Panel Length	27400.0	-70.000	0.000	0.000
31	Opt. face length 10 ft. seam	Panel Length	42250.00	-89.5833	--	--
31	Opt. face length 12 ft. seam	Panel Length	Equation: $Y=.11867 \times 10^{19}+.93137 \times 10^{16}X -.27394 \times 10^{14}X^2+.35786 \times 10^{11}X^3-.17519 \times 10^{8}X^4$			

BEST-FIT EQUATIONS OF CURVES (cont.)

Figure No.	X axis	Y axis	Coefficients of X^N: X^o	X^1	X^2	X^3
32	Panel Length 6 ft. seam	Opt. NPV	10.8154	$.25822 \times 10^{-2}$	$-.34487 \times 10^{-6}$	$.14965 \times 10^{-10}$
32	Panel Length 8 ft. seam	Opt. NPV	18.7594	$.76340 \times 10^{-3}$	$.23499 \times 10^{-7}$	$-.10913 \times 10^{-10}$
32	Panel Length 10 ft. seam	Opt. NPV	15.1633	$.55579 \times 10^{-2}$	$-.92981 \times 10^{-6}$	$.46252 \times 10^{-10}$
32	Panel Length 12 ft. seam	Opt. NPV	28.0371	$-.14370 \times 10^{-3}$	$.78422 \times 10^{-7}$	$-.12923 \times 10^{-10}$
24	Develop Productivity	Seam Height	-4.1300	$.33400 \times 10^{-1}$	$-.16000 \times 10^{-4}$	--
33	Opt. Panel Tonnage (1000's)	Seam Height	-6.6425	$.57131 \times 10^{-1}$	$-.72627 \times 10^{-4}$	$.34282 \times 10^{-7}$

17 Face Length Longwall Test Case — NPV

Coefficients of X^N

X^o	$-.38182 \times 10^{8}$	X^5	$.43412 \times 10^{-3}$
X^1	1052767.0	X^6	$-.60093 \times 10^{-6}$
X^2	-10270.94	X^7	$.51111 \times 10^{-9}$
X^3	57.8685	X^8	$-.24319 \times 10^{-12}$
X^4	-.19919	X^9	$.49499 \times 10^{-16}$

APPENDIX K

Data Points Used in Generation of Polynomial Best-Fit Equations

Seam Height	Development Productivity (T/Shift)	2 Entry Development		3 Entry Development	
		Optimum Face Length	NPV ($mm)	Optimum Face Length	NPV ($mm)
12 ft.	300	400	27.85	740	24.49
	350	340	29.51	600	26.06
	400	300	30.37	500	28.15
	450	260	31.67	440	29.73
	500	220	32.67	380	30.89
	550	220	33.23	340	32.05
	600	220	33.87	320	32.77
10 ft.	300	340	25.01	580	22.14
	350	280	25.99	500	24.04
	400	260	26.78	420	25.52
	450	200	27.85	360	26.81
	500	200	28.41	320	27.37
	550	180	28.79	280	28.58
	600	180	29.24	260	29.32
8 ft.	300	260	20.65	440	19.48
	350	220	21.55	360	20.69
	400	200	22.14	320	21.38
	450	200	22.71	280	22.58
	500	200	23.10	240	23.49
	550	200	23.41	220	24.40
	600	200	23.69	200	24.83
6 ft.	300	220	15.23	320	14.57
	350	220	15.77	260	15.74
	400	180	16.13	220	16.69
	450	180	16.47	200	17.23
	500	180	16.74	180	17.67
	550	200	16.93	160	18.17
	600	220	17.10	160	18.45

	6 ft. Seam		8 ft. Seam		10 ft. Seam		12 ft. Seam	
	Face Length (ft.)	NPV ($mm)	F	NPV	F	NPV	F	NPV
80 ft x 60 ft pillars	260	16.45	360	21.45	460	24.96	580	27.16
80 ft x 80 ft pillars	280	16.38	400	21.05	520	24.19	620	25.96
17 ft entry	240	16.45	340	21.68	440	25.29	580	27.49
18 ft entry	260	16.45	360	21.49	500	24.69	580	26.96
19 ft entry	280	16.38	380	21.25	500	24.40	620	26.33
20 ft entry	280	16.19	400	21.00	520	24.06	660	25.74
4 leg shield	220	16.66	320	21.31	420	25.45	500	28.06
3000 ft panel length	220	15.82	320	20.98	420	24.70	540	27.87
4000 ft panel length	220	16.70	320	21.38	420	25.52	500	28.15
4500 ft panel length	220	16.86	320	21.80	420	25.55	500	27.99
5000 ft panel length	220	16.89	300	21.85	400	25.52	480	27.31
5500 ft panel length	220	16.92	320	21.79	400	25.27	480	27.00
6000 ft panel length	220	17.17	300	21.80	400	25.00	480	27.58
6500 ft panel length	220	17.21	300	21.69	400	24.60	480	27.09
7000 ft panel length	220	17.18	300	21.53	420	24.52	480	26.45
7500 ft panel length	220	17.06	300	21.26	400	24.07	480	25.73
8000 ft panel length	220	17.05	320	20.75	400	23.76	480	25.34