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Study of the Effect of Web Variation on the Performance of a Longwall Face

Final Technical Report
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FOREWORD

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This report is a summary of the work completed during the period October 1977 to May 1979.

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ABSTRACT

This report presents the benefits that can be derived from mining deeper webs (cuts) on a retreat longwall as practiced in the United States. The study looks at the increases in productivity (tons per shift and per annum) and the cost benefits (reduced cost per ton off the face) for a double ended ranging drum (DERD) shearer operating in the half face or modified half face mode. The gains in productivity and the cost benefits are established for web depths varying from 27 to 57 in. in a 6.5 ft seam height.

The impact of wide webs on face equipment is evaluated including all currently used types of roof supports, and sizes of shearing machines. Additionally, the expected effect of wider webs on health and safety is discussed, as well as the experience that the British Mining Industry has had mining one meter (39 in.) webs.

Finally, the most cost effective system of equipment is identified and specified along with the overall impact of its use on conditions of mining.

1. INTRODUCTION

The depth of web or depth of cut is one of the principal parameters which determine productivity on a longwall face.

In November 1974 the British proceeded to mine a one meter (39 in.) web on selected faces in their Western Mining Area. This represented a 70 percent increase in the depth of web previously mined on these faces. After mining the one meter web for some 102 shifts, the British reported that the advantages were:

a. An average increase in productivity of 35 percent

b. Much improved roof conditions

c. Improvements in working conditions, including a reduction in the levels of dust, a 75 percent reduction in accidents on the face and reduced worker absenteeism.

However, these results were achieved under decidedly difficult conditions of mining and with equipment significantly different than what is presently used in the United States.

The British mine at depths of 2500 to 3000 ft and under immediate strata consisting of friable shale. In the United States mining occurs at depths of 500 to 700 ft and often under the influence of heavy competent sandstone strata relatively close to the seam. The different roof conditions result in different types and capacities of longwall supports being used in the two countries. A majority of faces in the United Kingdom use chocks of less than 200-ton capacity, whereas in the United States, in recent years, the shield type of support of 500-ton capacity has been specified for new installations.

For this reason the United States Department of Energy, in October 1977, funded a study to investigate the feasibility of mining wider webs in the United States and to ascertain its impact on productivity, safety, and the cost of mining. This study was limited to what could be achieved by modifying existing longwall equipment to accommodate the deeper webs. This limitation restricted this study to web depths of 60 in. or less.

The scope of this report provides definitive answers to the following questions:

- Under United States conditions, are wider webs feasible?
- What determines the depth of web presently mined and what would limit the maximum web depth?
- Is mining wider webs cost effective?
- What is the expected impact on health and safety?
- What will be the impact on contemporary equipment and what modifications will be required to accommodate the deeper webs?
- What is the expected impact on geological conditions?

1.1 Feasibility

The principal factor determining the feasibility of mining wider webs is control of the roof. Roof conditions in the United States are generally characterized by the presence of limestone or sandstone within the zone of influence. This results in the need for supports of substantial capacity. This capacity would be additionally increased by the wider web. In addition to the increased capacity, it was determined that the increase in web depth would also result in:

a. A shift in this resultant roof load towards the face

b. A corresponding shift in the load distribution on the mine floor

c. The longer span of exposed roof over the wider web would increase the potential for roof falls in front of the supports.

It was determined in this study that total support capacity required is primarily a function of the nature of the roof (the presence of and the location of the main competent roof strata) and is not greatly influenced by the increase in web depth. It was, therefore, concluded that mining wider webs would not require supports of substantially greater capacity than are presently available.

Various designs of supports were investigated as to their ability to accommodate the shift in resultant roof loading. A stability analysis was done for each support type. It was determined that supports set one web back would generally be unsuitable for substantial increases in web depth. This would entirely exclude the use of the two-leg shield. The five and six-leg chock, with hinged forward canopy, can be used without modifications, but the four-leg shield would have to be lengthened to provide a manway between the front and rear sets of hydraulic legs. This would allow the support to be set forward under the resultant roof load. The four-leg shield, so modified, was judged to be the most stable support for controlling the roof over significantly wider webs.

Various special features were then evaluated for controlling the increased expanse of roof in front of the supports. These special features included sliding extensions to the roof canopy and supports set one web back. Additionally, a novel concept was presented and evaluated. This concept consists of a support whose roof canopy slides forward independent of the base.

The best overall support was the lengthened four-leg shield fitted with the novel full sliding canopy. Acceptable alternatives were the same four-leg shield with sliding extension and the six-leg chock with hinged forward canopy.

Wider web longwalling was, therefore, considered to be feasible in the United States with the determination that roof supports of required capacity were available and could be fitted with special features to accommodate the deeper web. It was further recommended that a special design of support be investigated - a design specifically applicable to very wide webs under difficult roof conditions.

1.2 Practical Limitations on Depth of Web

Most shearers in the United States are supplied by German and British manufacturers and mount a 30-in. wide drum to extract a web depth that varies from 24 to 27 in. This width of drum appears to be directly related to practices established in Europe and particularly in the United Kingdom, where the prop free front (PFF) distance is regulated. The PFF distance, in fact, determines the maximum drum width.

Today's contemporary 300 kW shearer is capable of mounting a 48 in. wide drum to extract a depth of web up to 45 in. Furthermore, a soon to be available 310 kW shearer is reported by the manufacturer to be capable of mounting a drum 60 in. wide which could extract a 57 in. web. The capability of this machine was established as representing the maximum web depth, for the purposes of this study, since no other upper limit could be quantitatively established.

For any particular mine site other factors such as roof conditions or mine haulage and preparation might establish a lesser upper limit on web depth. For this report no such limit could be established as being generally applicable.

Cost Effectiveness

To determine the expected productivity and cost effectiveness of mining deeper webs in the United States, it was necessary to establish the capability of today's best longwalls. Actual performance figures were obtained from five high production longwalls in the Eastern United States. These figures were then compiled and averaged as shown in Table 15 of the report.

The performance of this hypothetical face, mining a 27-in. web, was then used throughout the report to reflect the expected productivity and cost effectiveness of mining deeper webs. This baseline operation is referred to throughout the report as "the contemporary high production face."

The report clearly established the principal benefit of wide web longwalling as being a substantial increase in productivity with a corresponding reduction in the cost of mining a ton of coal. There was a constant increase in shift production directly proportional to the depth of web for all equipment systems analyzed. The machine systems spanned the complete range of shearers (170 to 310 kW) and face conveyor capacities (700 to 1500 tons/hr) for web depths of 27 to 57 in.

This increase in productivity where each shearer-conveyor combination is mining its maximum web depth is shown below. The percent increase in shift production is relative to the 1316 tons/shift established in Table 15 for the conventional web of 27 in.

Shearer (kW)	Conveyor (tons/hr)	Web Depth (in.)	Tons Per Shift	Percent Increase
170	700	39	1490	13.2
300	700	45	1862	41.5
300	1000	45	2212	68.1
310	1500	57	2338	77.7

The projections were arrived at by analysis that accounted for all known factors influencing shift production, particularly as effected by an increase in web depth, including:

- An expected increase in lost time on the face due to equipment malfunctions and geologically related problems
- The limitations on the capability of each machine system to mine deeper webs.

The results arrived at coincide with the published British results as to the relationship between web depth, tons per cycle, and cycles per shift. The results consistently show that for all systems and conditions of mining the increase in the tons won per cycle (with the deeper web) was substantially greater than the decrease in the cycles completed per shift. Furthermore, in all cases the impact of the nonproductive modes of the operating cycle (turnaround time at the entries and cleanup time) was reduced. Real shearer utilization (the amount of total shift time in which the shearer is *actually* mining coal) consistently increased with increasing web depth. These factors combine to account for the substantial increase in tons mined per shift.

The additional mining and operating costs associated with mining the wider webs were then applied to the productivity analysis. For all systems analyzed, the increased productivity was substantially greater than was the increase in cost. The reduction in cost per ton was directly related to the depth of web and in all cases the deepest web was the most cost effective as shown.

System	Shearer Power (kW)	Conv. Capacity (tons/hr)	Max. Web Depth (in.)	Cost Per Ton ($)	Percent Decrease in Cost/ Ton*
Low Capacity	170	700	39	3.20	14
The Contemporary High Production Face	300	700	45	3.05	18
High Capacity	300	1000	45	2.88	23
Very High Capacity	310	1500	57	2.97	20

*Based on $3.72/ton stereotypical system mining 27 in. web.

The costs reflected are the owning and operating costs on the face and the cost of the outby coal haulage system. The equipment for all systems is identified in Tables 16, 20, 21, and 22.

The contemporary high production face mined 1316 tons of coal per shift from a 27-in. web at a baseline cost of $3.72 per ton off the face. The cost per ton, for this same system, mining a 45-in. web was reduced 18 percent to $3.05. At the deeper web productivity of this system was restricted by the capacity of the face conveyor. Increasing the conveyor capacity to 1000 tons/hr (the high capacity system) further reduced the cost to $2.88 per ton.

Roof control was determined to be the major consideration in evaluating the impact of mining deeper webs. The principal capital outlay on a longwall face is the cost of the roof supports. The relative cost of mining at various web depths with the four equipment systems reflect this fact. The low capacity system, mining a maximum web depth of 39 in. employs the least expensive support - a six-leg, 450-ton chock. The very high capacity system, mining up to 57 in. of web, uses the highest capacity, most expensive 700-ton, four-leg shield. The cost of all roof support systems included the special features required to control the roof over the deeper webs. These special features included longer advancing rams, sliding canopy extensions, and longer supports depending on the depth of web and type of support employed.

1.3 Health and Safety

The impact of wider webs on productivity, equipment, and the costs of mining were calculated mathematically. The impact on health (dust) and safety (accidents and the presence of methane) cannot be derived so precisely. In the case of dust, there is a lack of a sufficient data base whereas accidents and the presence of methane are random occurrences, requiring large quantities of accurate data from which to project results arrived at statistically.

The Mining Safety and Health Administration (MSHA) did provide a data base identifying the nature and severity of all accidents occurring during 1977 on United States longwall faces. The factors, related to wider webs, that could be expected to have an influence on face accidents were then identified and qualified as to their projected impact on this data base.

It was concluded that at least the number of accidents would decrease with deeper webs which was substantiated by the results published by the British. Accidents were reported as being reduced on their one meter web faces, from 303 per 100,000 shifts to 71 per 100,000 shifts. The British concluded that accidents were reduced due to:

a. A reduction in the number of machine cycles per shift (reduced number of operations per shift)

b. Better roof conditions - less roof fallout due to stress cycling of the immediate strata

c. A reduction in levels of dust

d. An improvement in operator morale.

Dust is not expected to be a problem since it is likely that improved methods of both reducing dust production (deeper cutting slow speed drums) and suppressing dust (more efficient sprays, pick face flushing, and possibly scrubbers) will have reduced the dust problem on United States longwalls. The application of some of these techniques will be particularly effective with deeper webs, for example, deeper cutting slower speed drums.

The report does identify methane as being potentially the principal problem associated with the deeper web. This is due to the increased rate of mining (in tons per shift) and the potential for gas pockets to collect in the deeper cut. The report suggests that additional study be undertaken to investigate the use of new devices being developed - in particular the ventilating cowl.

1.4 The Impact on Equipment

The impact on the roof support system was discussed in subsection 1.1. The effect in the face conveyor is minimal, mostly affecting the capacity required to accommodate the increase in output from the shearer. In the productivity analysis, additional down time was assigned to the conveyor. This was to account for the fact that the additional pushover distance (web depth) might result in additional damage to pan connectors and flites.

The shearer will require modifications to accommodate the wider drums. These changes, identified by the manufacturers, included strengthening the ranging arms and their mountings, including the mounting arrangements for the drums and strengthening the mounting of the shearer to the conveyor.

1.5 Effect on Local Geological Conditions

The basic method of panel extraction is unaffected by mining deeper webs. The principal impact on the local geological conditions is likely to be the increase in the rate at which the face advances. This usually improves the conditions of mining. For example, the influence of the forward abutment pressure on conditions in the entries is likely to be reduced. The magnitude should not be affected, but it will advance more rapidly thus reducing its deteriorating effect on roof, floor and pillars.

Conditions on the face can also be expected to improve since the time related effect of roof convergence is minimized with the faster advancing face.

1.6 The Potential for Wide Web Mining

The increased productivity of mining the deeper webs, with the corresponding reduced cost per ton, plus the fact that these results can be achieved with present proven equipment and methods should result in longwalling becoming economically attractive to more operators.

There are, however, other alternatives to increasing longwall production with present longwalling equipment and methods. They are:

a. Mine with faster shearers

b. Mine longer faces

c. Increase overall system reliability (reduce lost time).

This report evaluated and compared the potential of these alternatives, as they would impact the productivity of the contemporary high production face.

Each of the four parameters (face length, web depth, shearer speed and system reliability) were changed while all other parameters were kept constant. In each case, the change in the parameter of interest was within the limitations of the machine system. Referring to Table 15, the 300 kW shearer mining the 27 in. web traverses the face (cutting speed) at 13.1 ft/min. At this speed it is mining coal at a rate of 517 tons/hr yielding a shift production of 1316 tons.

This shearer-conveyor combination is underutilized since the face conveyor has a maximum rated capacity of 800 tons/hr, which is a rate a 300 kW shearer is capable of mining. At 800 tons/hr the shearer speed could be increased to 20 ft/min mining a 27-in. web. At this increased speed, cycle time would be reduced and shift production would increase from 1316 tons to 1646 tons, all other parameters being constant.

Alternatively, the web depth could be increased to 42 in. to load the conveyor at 800 tons/hr at a shearer speed of 13 ft/min. Under these conditions, with all other parameters constant, shift production would increase to 2043 tons.

The impact of increasing face length and decreasing machine downtime (increasing reliability) were also evaluated in the same manner. The percent increase in shift production, from the baseline 1316 tons, for an equivalent increase in each of the four operating parameters, is shown below. These results were arrived at in subsection 2.3.3 and summarized in Table 4 of the report.

Operating Parameter	Increase in Operating Parameter	Resulting Increase in Production
Increased web depth	55.6	55.2
Increased reliability	55.0	53.6
Longer face	55.0	37.2
Faster shearer	54.2	25.1

An increase in productivity alone does not justify mining wider webs as the preferred alternative to mining longer faces or with faster shearers. The choice must also take into consideration the cost effectiveness of these alternatives and their effect on health and safety.

An increase in face length is not likely to be cost effective as it involves an increase in the number of supports - the most expensive equipment system on the face.

Faster shearers are very likely to increase the dust problem and result in an increase in accidents. These effects, of course, can be neutralized by removing the operators from the face through automation.

This report concludes that production increases, with corresponding reductions in cost of mining, can best be realized by mining deeper webs and by increasing system reliability (reducing lost time). The NCB in the United Kingdom have arrived at the same conclusions as, in addition to their wide web program, they are concentrating their efforts at improving machinery reliability.

The immediate gains in productivity that can be achieved on United States longwalls is best illustrated by the recommendation of Section 7 where an underground demonstration of wide web mining is recommended. This recommendation includes a specification for equipment, panel dimensions, sequence of extraction, and mine site specifications.

A 300 kW shearer with a state-of-the-art, 1000 tons/hr face conveyor and four-leg shields are recommended to mine a 45 in. deep web on a 500 ft long face in a 6.5 ft high seam. Shift production is projected at 2200 tons/shift with the shearer advancing along the face at 14.5 ft/min. Under these conditions the conveyor would be loaded at its rated 1000 tons/hr.

To achieve the same shift production under the same conditions, a faster shearer mining the conventional 27 in. web would have to traverse the face at 50 ft/min mining coal at the rate of 1980 tons/hr. This is well beyond the capability of today's shearers and conveyors.

Similarly, a shearer taking a 27 in. web, at 1000 tons/hr, would be required to mine a 1360 ft long face to achieve a shift production of 2200 tons.

1.7 Recommendations

There are two recommendations made in Section 7. The first is to demonstrate underground the benefits of mining a 45 in. web with available equipment. This face would use the 300 kW shearer and a 1000 tons/hr capacity face conveyor in a 6.5 ft seam height. This demonstration could be undertaken immediately.

The second recommendation is to develop a roof support specifically for controlling the roof over a very wide web. It is felt that this development is required to project wide web longwalling into the 57 in. depth range, which should further reduce the cost of longwalling.

The report identifies a special design feature that would greatly enhance a support's stability at these web depths. This special feature involves mounting the main roof canopy on slides which is pushed forward by hydraulic rams. The report suggests that alternatives to this particular support also be investigated.

2. LONGWALL MINING - A BRIEF INTRODUCTION

The longwall equipment supplied to the American market in the early years was basically of a size and capacity that had proved itself in British and European mines. The fully mechanized longwall face with the shearer type of cutter-loader is essentially a British development. The standard face developed by the NCB used the chock type of roof support exclusively (1).

Some of the earliest equipment introduced in the United States was of British manufacture and featured a chock support of limited capacity. As a result of some early failures, it quickly became apparent that heavier supports were required. This was primarily due to the shallower depths of mining in the United States and the presence of massive roof strata immediately over the seam. By comparison, British experience was in controlling friable immediate roof strata that more easily broke over the gob end of the support, which minimized the actual roof load carried by the props (1).

Longwall mining with the shearer is primarily a British development, whereas the plow has been perfected and applied primarily in Germany. Figure 1 shows the distribution of shearers and plows in the United Kingdom, West Germany, and the United States in 1976. As can be seen, the shearer is the predominant machine in both the United Kingdom (1) and the United States. In contrast, 78 percent of German faces are equipped with plows (2). The plow has had success in West German mines because of the geological conditions, the coal being relatively soft and/or friable with good parting between the seam and roof.

The development of longwall equipment in the United Kingdom and Germany reflects itself in the equipment supplied to the United States as shown in Figure 2 (1,2). This bar graph is a breakdown of longwall equipment, as to type and country of origin on shearer-equipped United States longwall faces as of 1976. Forty-two of the fifty shearer faces used chocks and six used shields. Of the forty-two chock supported faces, thirty were supplied by British manufacturers. However, all six of the shield systems were supplied by German manufacturers with the first shield face going into operation in April of 1975 at Consolidation Coal's Shoemaker Mine near Wheeling, WV (3).

Three companies have supplied shearers to the United States market. Sale of the majority of earlier single drum shearers, shared equally by the German and British manufacturers, were

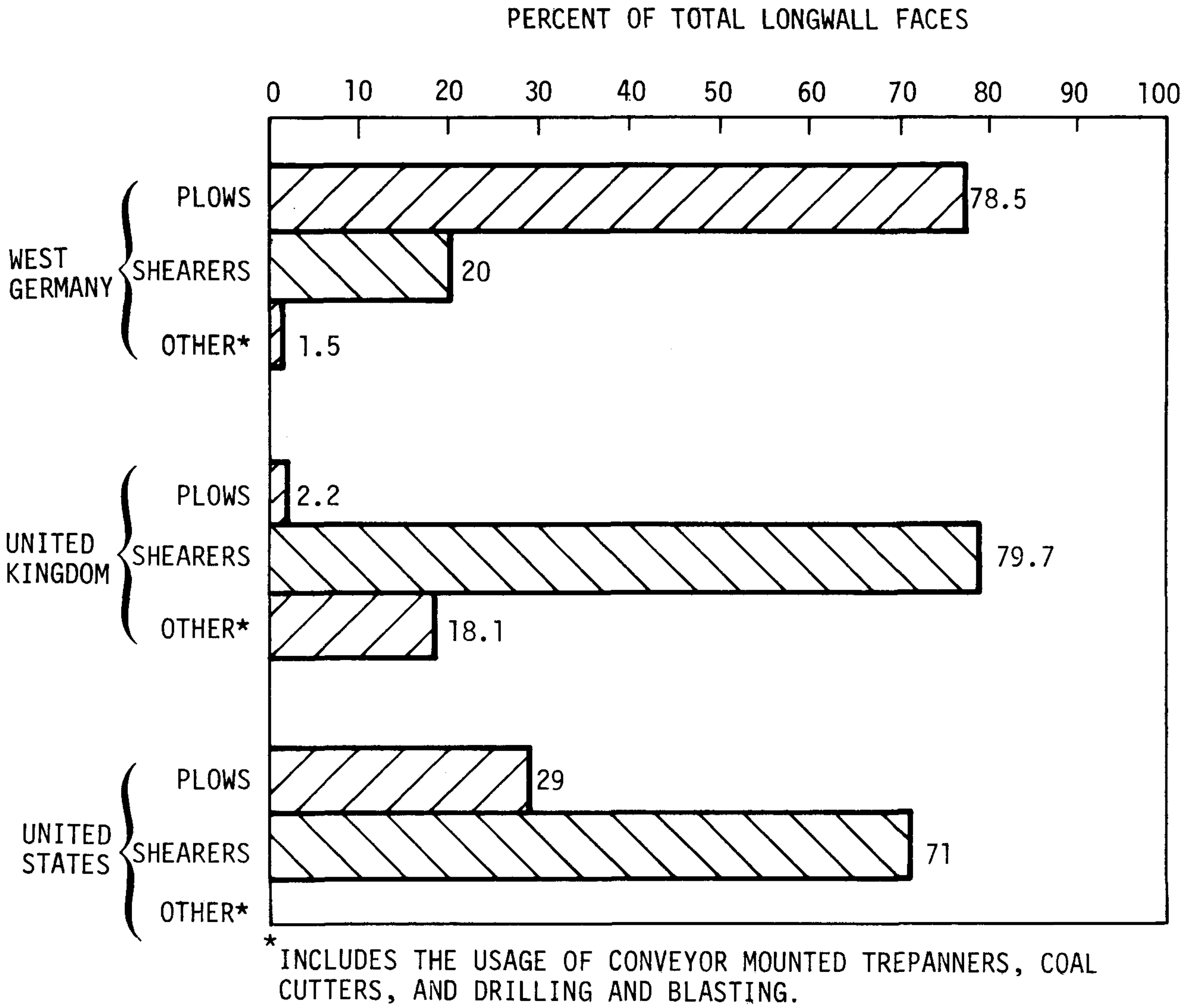


FIGURE 1. - Distribution of longwall cutter-loader applications in West Germany, United Kingdom and United States - 1976.

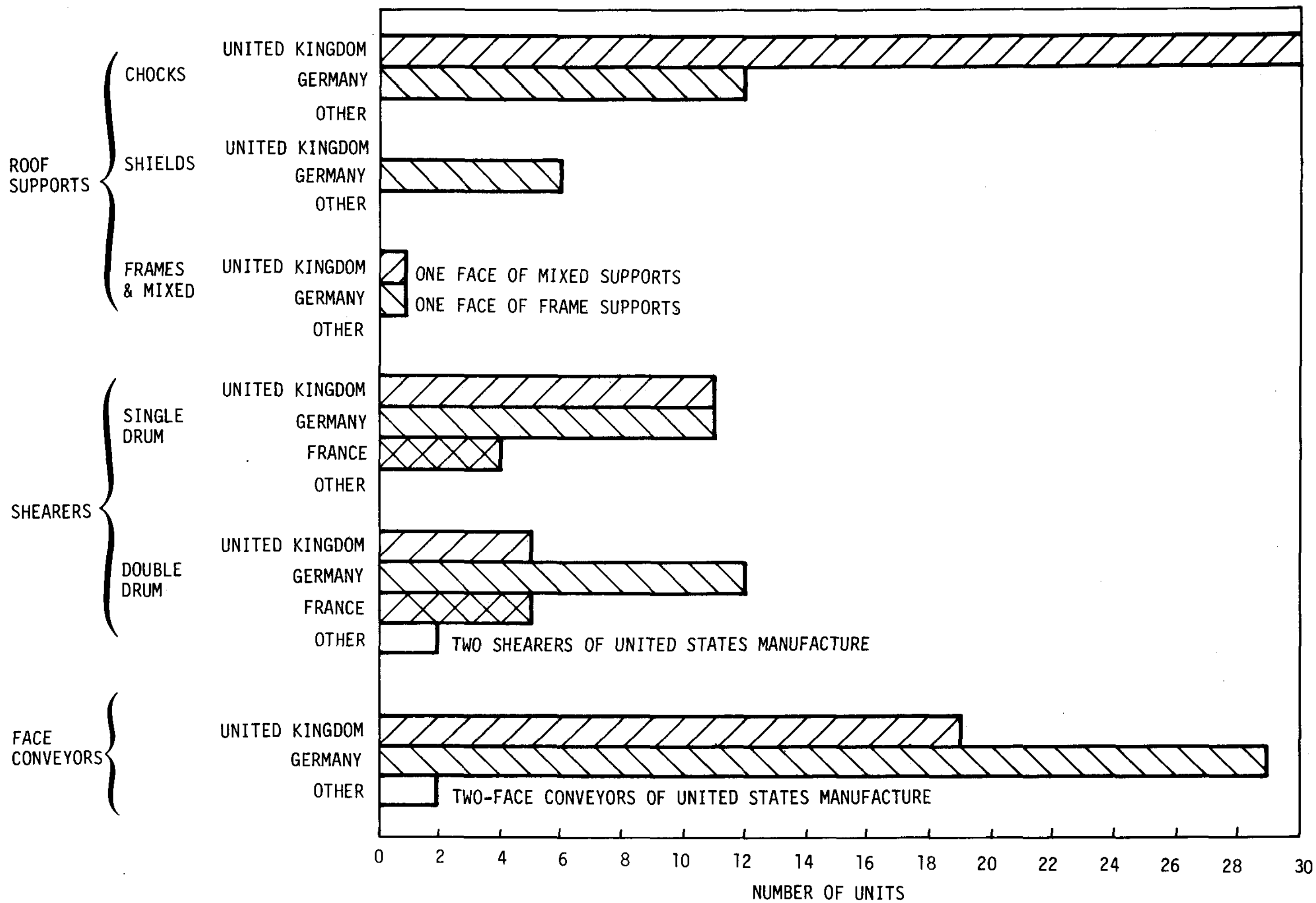


FIGURE 2. - Distribution of longwall equipment on United States shearer faces by country of origin - 1976 (4).

smaller machines of some 200 to 270 hp. The German manufacturer was quick to respond to the market need for increased production and increased power, resulting in a clear lead in the sale of the higher production, higher horsepower double-ended shearers. These machines range in size from 300 to 460 hp.

It is important to note that historically all early pieces of longwall equipment (shearers, supports and face conveyors) were supplied by foreign manufacturers and that most of this equipment came from the United Kingdom or West Germany.

It is of further importance to note that the equipment used in the United States has been influenced by mining conditions prevalent in the country of origin.

It will be pointed out later in this report that although the European manufacturers have reacted to the difference in needs, some practices, particularly where determined by legal restrictions in the country of origin, still effect equipment supplied to the United States market. The classic example is the relationship between the depth of web mined in the United States and the British legal restriction of allowable prop face front distance. This will be discussed in some detail in subsection 2.5.

In applying longwall mining, the United States mining community has developed some of its own techniques best suited for prevailing conditions and foreign equipment suppliers have provided the hardware in response to the need - as this need has been identified. It is expected that the American market will continue to be characterized by the demand for increased production and/or the need to maintain state-of-the-art productivity at reduced cost per ton of coal mined.

Most of the applications of longwall equipment in the United States is in the intermediate seam heights of 48 to 84 in. (4). These seam heights have been mined by a combination of shearer and chock-type support. Low seams under 42 in. are essentially mined by the plow where coal cutting conditions will allow. There is a substantial amount of work being done in the United Kingdom to perfect an in-web or "buttock" shearer for low seam longwalling. Development of this type of shearer will extend the use of longwall mining into low seam heights where the plow might not be suitable.

High seam longwalling was initially attempted at Kaiser Steel's York Canyon mine in 120 in. of seam height with an Eickhoff shearer and a set of Hemscheidt 320-ton, two-leg caliper shields shown in Figure 3. This support is fitted with a face "sprague" which prevents sloughing of coal off the face, a common occurrence in high seams.

Contemporary longwall faces can generally be associated with equipment systems composed of higher horsepower double-drum shearers and the shield-type of roof support. Very recently, the preference among purchasers has been for the four-leg version of the shield, with rated capacity of greater than 500 tons, and the inclusion of the chainless type of haulage system on the shearer.

2.1 Contemporary Equipment

The following comments cover the salient features of contemporary equipment here defined as hardware purchased in the last 2 to 3 years and for the most part representing high capacity equipment.

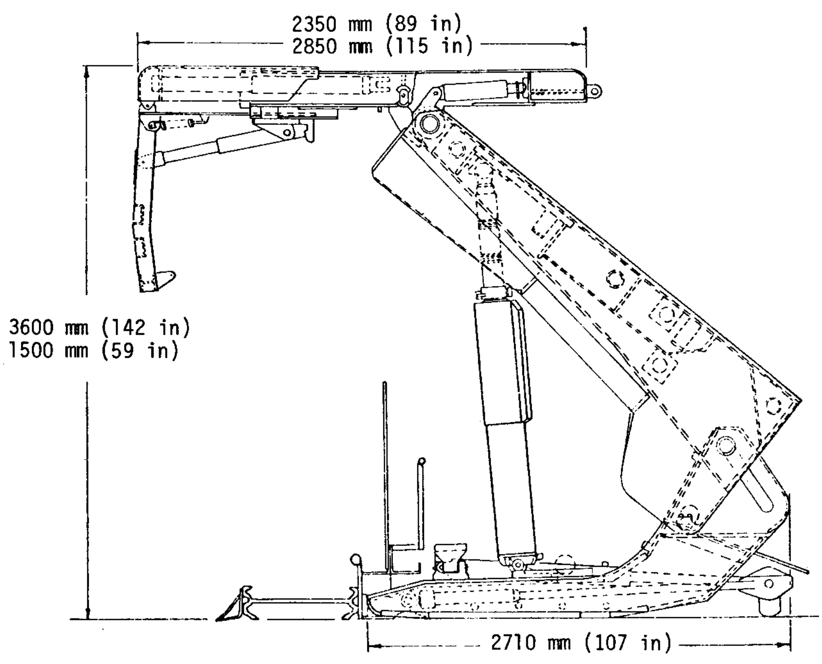


FIGURE 3. - Hemscheidt 320-ton, two-leg caliper shield Kaiser Steel's York Canyon Mine.

2.1.1 Shearers

A majority of shearers purchased are of the double-ended (double drum) type with ranging arms. These machines range in size from 300 to 460 hp. The trend to higher powered shearers in the United States appears to be based on increasing production as a result of:

a. A heavier, more rugged, more reliable machine

b. Reserve power to cope with tough cutting conditions which may appear periodically.

This philosophy suggests that shift production is at least as heavily dependent on shearer reliability as it is on shearer capacity (cutting and loading capability in tons per hour). This would certainly be the case where the capacity of the haulage system (face conveyor, section conveyor, mine cars, etc.) restricts production.

These double-ended shearers are equipped with ranging arms for extracting full seam height with each pass of the machine on the face. A machine of this type is shown in Figure 4. The latest shearers sold are almost all equipped with the new chainless haulage systems. The standard drum width is 30 in. (5) which mines a 24 to 27 in. web, depending on face conditions. The largest machines in use today have 460 hp available from a single motor and can cut and load 1000 to 1200 tons/hr of coal. These machines maximize production by cutting in both directions on the face in a half or modified half face mode of operation. The ranging arms provide for cutting into the entries at both the head and tail gate.

The latest design high capacity shearer is no longer mounted off the conveyor pan line. Its weight is now carried on the static ramp plates and gob side furniture. In addition to removing the machine weight from the pan line, the resulting wider mount also provides the heavier machines with a more stable base of support.

A single electric motor provides all the power to the cutting drums, the haulage (propelling) system and the hydraulic control cylinders. In some cases the shearers have been supplied with a second motor. In this configuration, one motor drives each cutting drum with auxiliary power (for haulage and control) additionally being taken off one of the two motors. This does not double the capacity of the machine to mine coal, since most of the coal is cut and loaded by a single (the leading) drum. This drum only sees a fractional increase in available power due to the addition of the second motor.

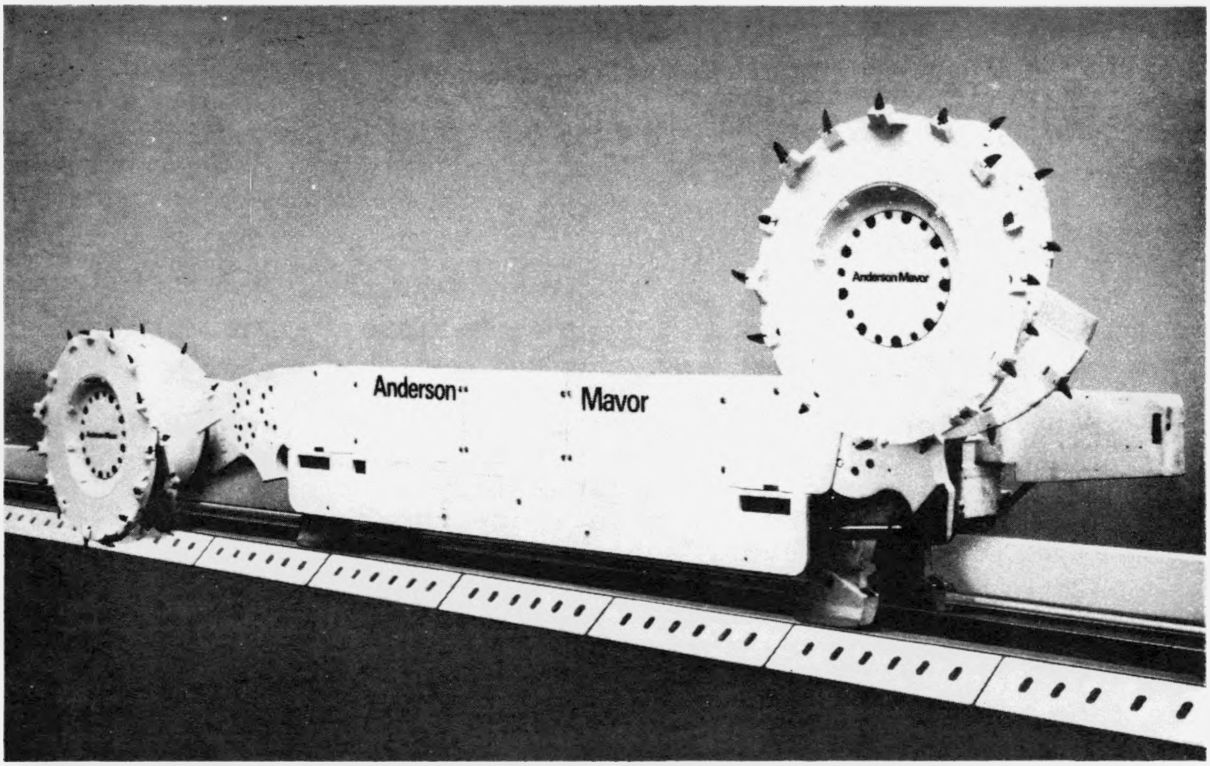


FIGURE 4. - Typical double ended ranging drum shearer Anderson Mavor 500 kW (670 hp) twin motor.

2.1.2 Roof Supports

Until recent years, all United States longwall faces equipped with shearers employed the chock support, and plows were used with the frame. In the last 3 years, there has been a decided shift to the two-leg and four-leg shield support on shearer equipped faces, and some plow faces have been equipped with chocks.

The basic advantages of the shield type of support, as compared to the chock, are:

a. More complete coverage of the roof by skin-to-skin contact of the gob shields - complete shielding of the manway from roof debris

b. By design, the shield is an inherently stable structure able to resist lateral loads from the roof or gob

c. Under the same conditions, the required length of roof canopy on the shield is less than the required length on an equivalent chock.

The length of the canopy is of significance since it is related to the required capacity of the support and to the number of times any portion of the roof is subjected to a cycle of stress. The length of roof, and therefore the weight of roof being carried by the support, is directly related to the length of canopy.

The number of stress cycles to which any part of the roof is subjected to is also canopy length related as:

$$n \text{ (number of stress cycles)} \approx \frac{\text{length of canopy}}{\text{depth of web}}$$

Stress cycles, or roof tramping, is generally detrimental as it works to break up the strata over the roof canopy.

The original shield installations in the United States were two-leg shields of the caliper type as shown in Figure 3. The canopy is connected to the gob shield by a single point pivot. Discontinuities in the roof strata, however, result in poor roof-to-canopy contact as the canopy could tilt up into a roof cavity. The second major disadvantage associated with this type of support is the single point pivot. When the canopy is raised and lowered, the face tip traverses an arc. Operating this support in a higher position (assuming a variation in seam height) would result in the canopy tip being set too far from the face, which could lead to roof fallout in front of the canopy.

To eliminate both these difficulties, the lemniscate linkage (panograph linkage) was added as shown in Figure 5. This linkage stabilizes the roof canopy and eliminates the arcing motion of the canopy when the support is raised and lowered.

Most recently a second set of props has been added between the base and the underside of the gob shield structure resulting in today's latest support innovation - the four-leg shield shown in Figure 5. The second set of legs increases the available load density (tons per square foot) for better roof control.

The basic four-legged chock has been similarly modified to add lemniscate linkage and full gob shielding as shown in Figure 6. This support design is referred to as a chock-shield. The linkage provides the lateral stability missing from the earlier chocks, but it still requires a longer canopy than the shield.

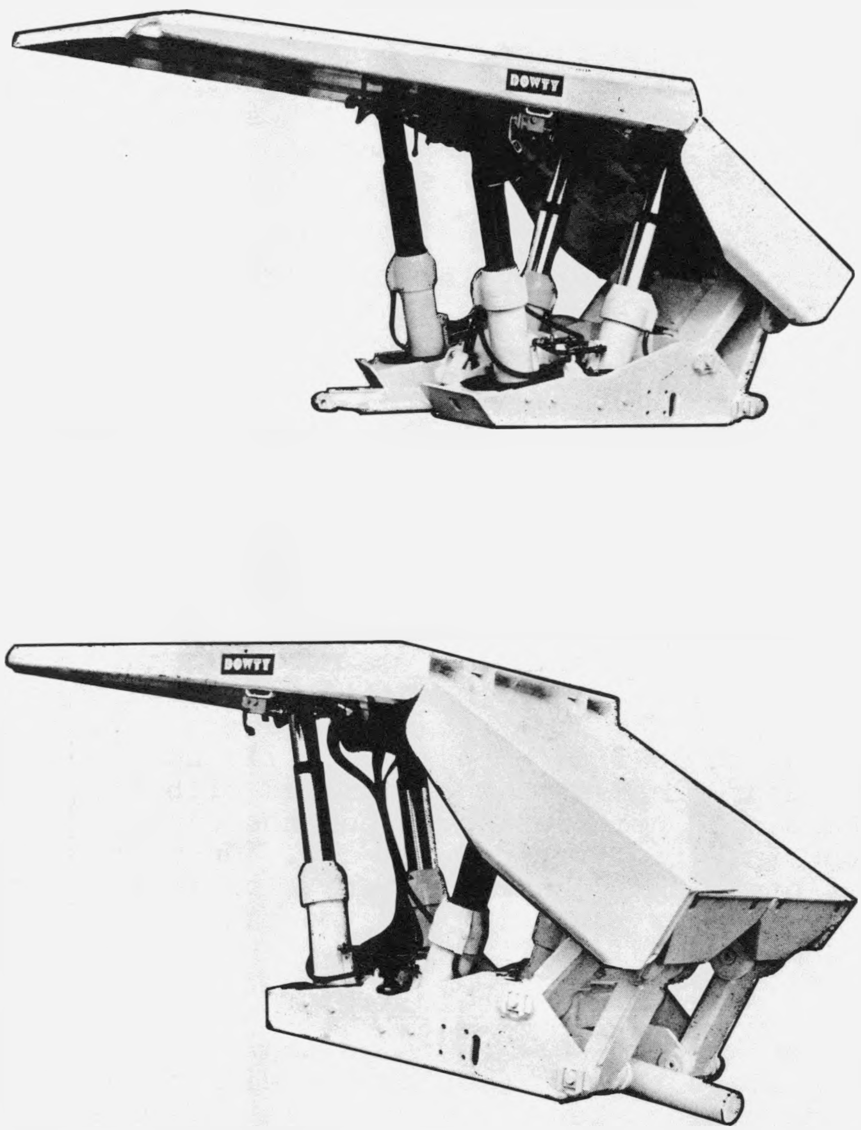


FIGURE 5. - Typical 360-ton, four-leg Dowty shield with the lemniscate linkage

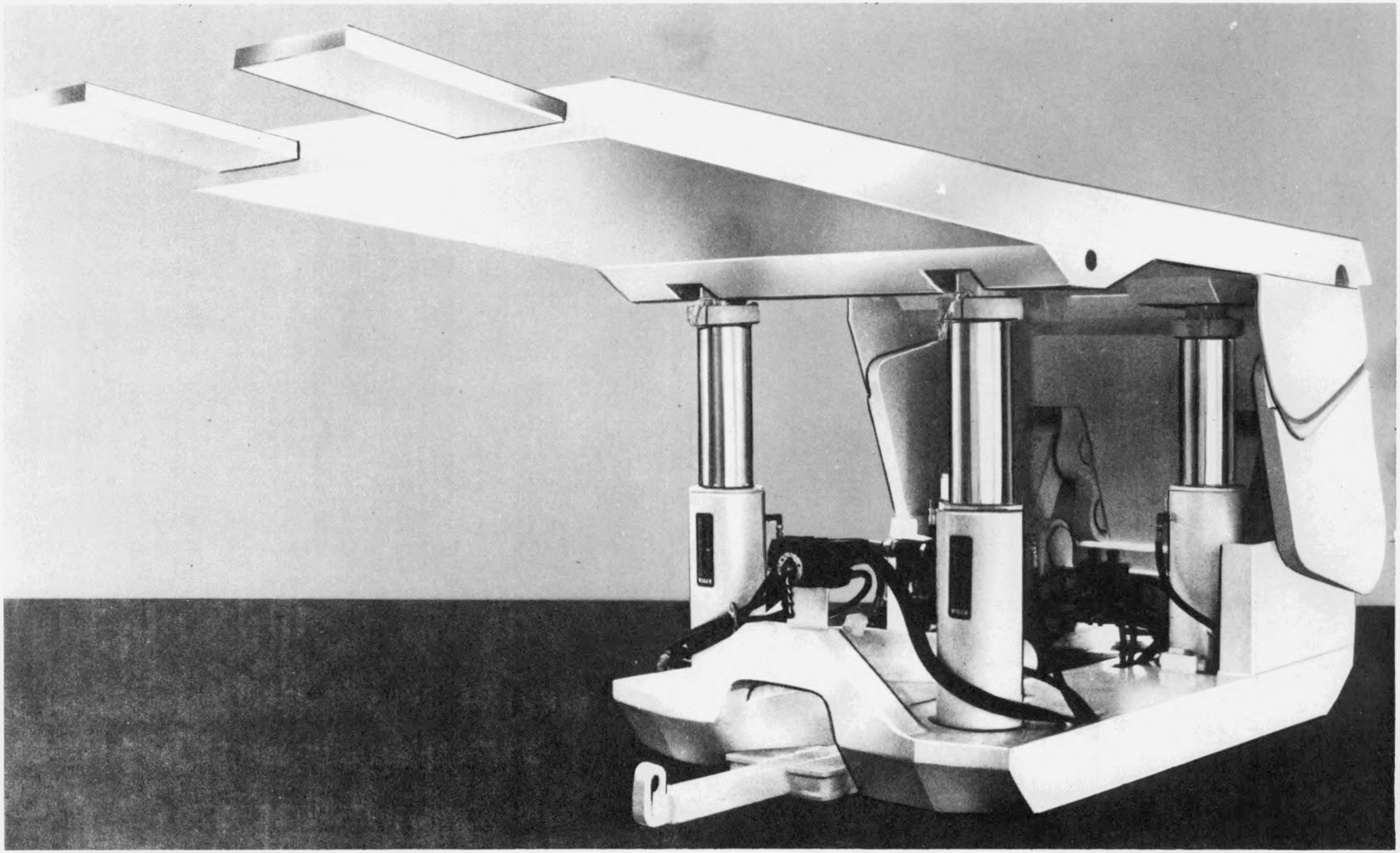


FIGURE 6. - Chock shield support with extendable canopy for IFS Gullick Dobson 500-ton capacity.

2.1.3 The Armored Face Conveyor

The face chain conveyors vary in type and capacity. The types vary based on the number and the position of the chains connecting the flights. There are four basic types: single and double center strand, twin outboard and triple strand. Chain sizes vary from 18 to 30 mm. Capacity (tons per hour) is mostly related to pan dimensions (width and height) and flight speed. The NCB has standardized the available pan dimensions, and rate conveyors in tons per hour of capacity per meter per second of flight speed for various pan cross sections.

The state-of-the-art conveyor on American longwalls is represented by Eickhoff's EKF-3 conveyor which has pan dimensions of 764 mm (30 in.) of width and 222 mm (6-3/4 in.) of height. The capacity of this conveyor is rated at between 700 to 800 tons/hr at chain speeds of 235 to 270 ft/min. This conveyor is the single center strand type with one 30 mm chain.

A similarly rated conveyor is the 26 mm twin-inboard type manufactured by Dowty Meco, Huwood Irwin and Westfalia. This conveyor has approximately the same pan dimensons as the EKF-3.

Power requirements vary with conveyor type, face length, pitch of the face and expected peak load (startup torque requirement). The 30 mm center strand conveyor is fitted with drives of 300 to 400 hp whereas the twin strand conveyor has 375 to 600 hp drives. The additional power required by the twin strand conveyor is at least partly due to the additional friction associated with running two chains outboard in the pan race.

Although the 700 to 800 tons/hr conveyor is the state-of-the-art system on American longwalls, higher capacity conveyors will soon be available. These machines will be discussed in the subsection that follows.

As part of this study, equipment manufacturers were surveyed by FMA to establish the near future trends in equipment based on size and types sold, but not delivered, and quoted but not sold. The results of this survey are summarized in the following subsections.

2.2 Recent Trends in Equipment

The major longwall manufacturers of supports and shearers were surveyed with regard to the specifications of equipment recently sold or quoted to the United States mining industry. This survey was conducted to establish what trends might exist in the application of contemporary equipment.

Most of the information presented in this section has been extracted from conversations with two shearer manufacturers and a survey of six roof support manufacturers conducted in early March 1978 (5).

2.2.1 Shearing Machines

As expected, shearer manufacturers anticipate that the demand for larger, more powerful shearers will continue. The average power on machines that will be sold in the next 5 years will be 500 hp. One manufacturer does not anticipate recommending a machine of less than 500 hp for any new purchase. The first of his new 500-hp shearers will be delivered to an American operator in June of 1978.

The second manufacturer surveyed expects to introduce a new 670-hp shearer to the American market by the fourth quarter of 1979. He further expects to be the first major supplier to offer a shearer manufactured and assembled in the United States.

Of the 19 shearers sold by both manufacturers surveyed for delivery in 1978 or early 1979, all were double drum bidirectional machines and 13 out of the 19 were sold with 400 hp or more. The remaining six machines had 225 hp motors.

Both manufacturers felt that the size of shearing machines would probably peak at a maximum of about 1000 hp and this might well occur within 5 to 7 years. The shearer market will also see an increase in the number of suppliers competing for the business. Historically, there have been three suppliers of longwall shearing machines to the United States mining industry, as previously shown in Figure 2. In the near future, the number of suppliers is likely to expand to include two additional British and one American manufacturer.

Additional changes that will impact the market will include the shift to chainless haulage. The initial systems were installed during 1977 and, if successful, both manufacturers expect that the old chain haulage system will be entirely phased out. Chainless haulage was developed in the United Kingdom at the insistance of the NCB because the chain haulage systems had proven to be hazardous.

Other recent innovations such as through the drum ventilation, remote control of shearers and electronic speed control of electric motor powered haulage have not, as yet, been applied to United States longwalls.

2.2.2 Roof Supports

The successes of the Shoemaker installations, the record setting Robinson Run longwall, the successful application of the shield under the very difficult roof at the Old Ben Mine and the application of shields to high seam extraction at Kaiser's York Canyon Mine appear to have shown the shield to be a support of superior performance.

The growing preference for the shield support is clearly shown in Figure 7. This bar graph shows the shift in favor of the shield that occurred in 1977 as well as the projected use of the shield in 1978 and 1979. The estimated projection for the shield is a result of a survey conducted in support of this report.

The results of the survey conducted among three British and three German support manufacturers as to the type of equipment sold but not delivered is summarized in Table 1. Of the 23 systems sold, 22 were two-leg or four-leg shield type supports and one was a set of chock shields. Of the two manufacturers who were willing to provide information on equipment quoted but not sold, 26 quotes were for shields and only one was for chocks.

TABLE 1. - Roof support types to be installed on United States longwall faces in 1978 and 1979

Country of Origin	Sold but not delivered	Sold and installed	Delivered but not operating	Quoted but not sold
Germany	Shield faces (4)	None	None	Not sold
Britain	Four-leg shield faces (4)	None	None	Shield faces (19) Chock face (1)
Germany	Shield faces (4)	Shield faces (2)	None	Not surveyed
Germany	Shield faces (3)	None	Shield faces (3)	Four-leg shield faces (7)
Britain	Two-leg shield faces (2)	None	None	Not surveyed
Britain	Four-leg chock shield (1)	None	None	Not surveyed

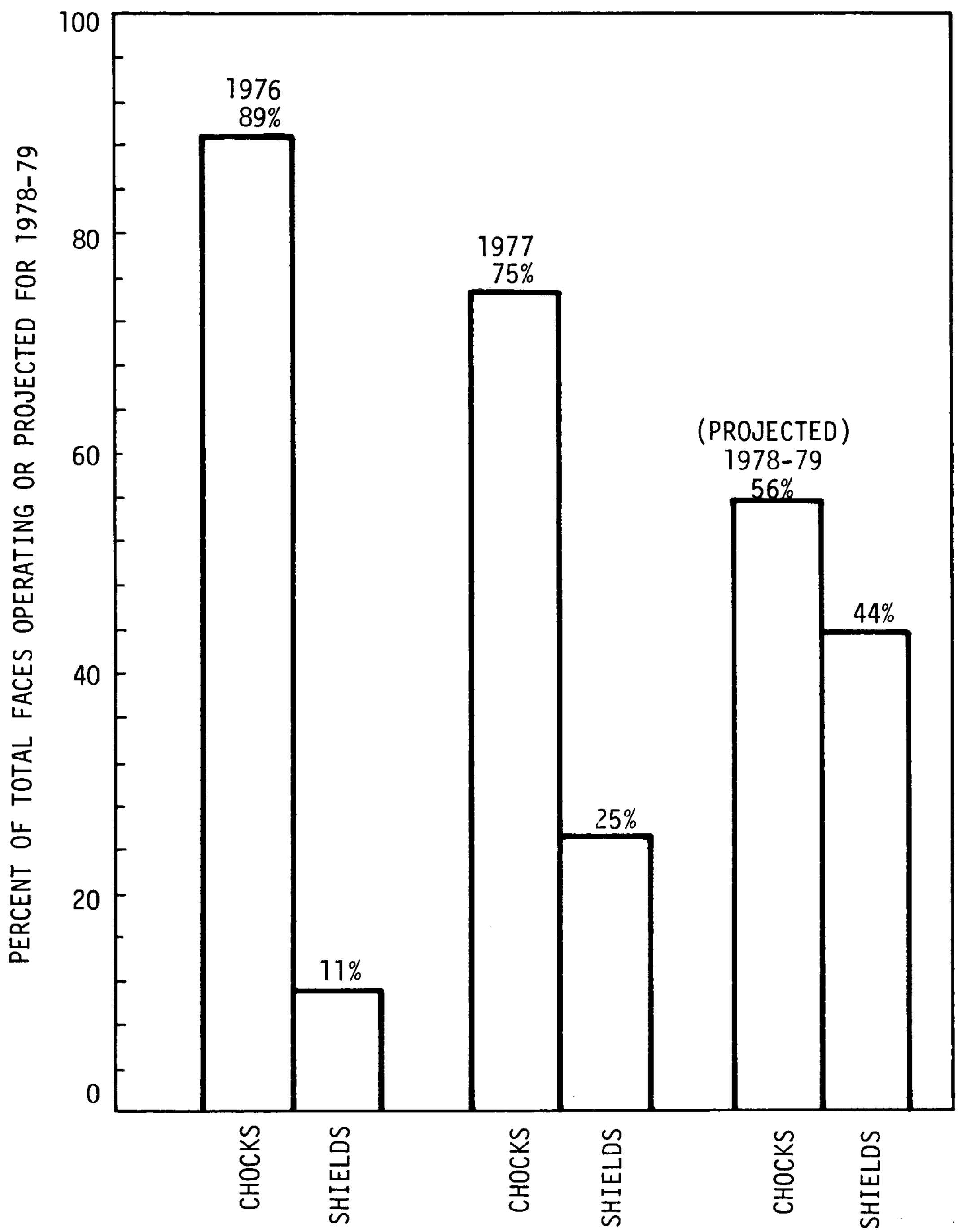


FIGURE 7. - Comparative use of chocks and shields on United States longwalls - 1976 through 1979.

As discussed in subsection 2.2 and illustrated in Figures 3 and 5, the shield type of support has either two or four hydraulic legs. The first of the shields used in the United States were of the two-leg type.

More recently, however, the preference appears to be shifting to the four-leg shield. The manufacturers surveyed felt that the four-leg shield is the support of the future. This support can be constructed with a relatively short canopy and a roof support density in excess of 6.5 tons/ft^2 (70 tons/m^2). The lemniscate linkage would maintain the forward edge of the canopy within 15 in. of the face within the full range of seam heights that the support is designed to accommodate.

Such a support is illustrated in Figure 8 where the support density is 9.6 tons/ft^2 when operated conventionally and 6.8 tons/ft^2 when operated one web back. The closed-to-open height ratio is 4.6 to 1.

2.2.3 The Armored Face Conveyor

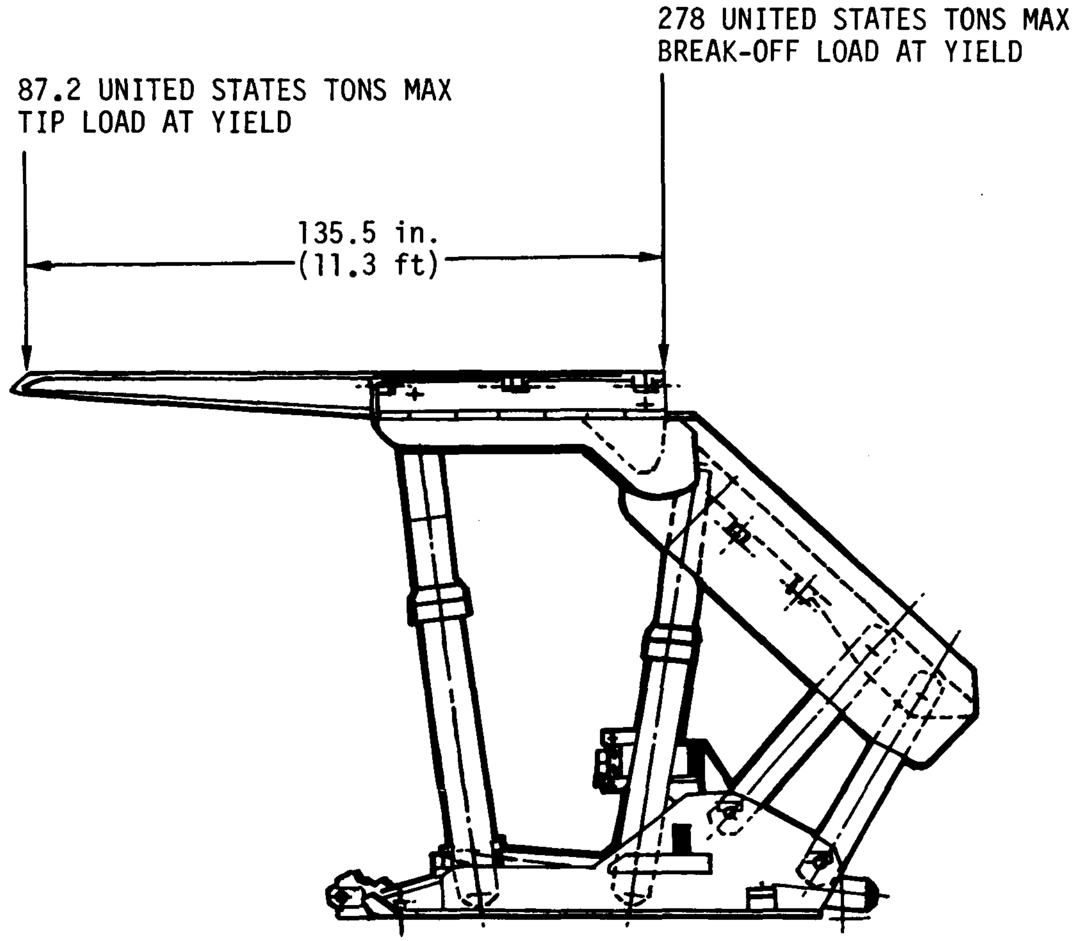
In subsection 2.2 the contemporary face conveyor was identified as having a rated carrying capacity of 700 to 800 tons/hr. In some applications, this conveyor could restrict the rate of mining; for example, when used with the higher horsepower (300 kW) shearers in seam heights of 7 ft or greater.

There is presently under development a family of conveyors with carrying capacities of 1000 to 1200 tons/hr, a capacity better suited for the 300 kW shearer in relatively high seams. The conveyors being developed are the Eickhoff DMKF-4 and the DOE-FMA high capacity conveyor (6).

The German conveyor is offered with either single 34 mm or twin 30 mm center strand chain operating at 260 ft/min. The pan dimensions are 832 mm (32.75 in.) by 250 mm (9.84 in.) high.

The DOE-FMA conveyor has twin 26 mm center strand chain operating at 270 ft/min. Pan dimensions are 838 mm (33 in.) wide by 255 mm (10 in.) high.

The performance of these conveyors will impact the industry in the very near future as both conveyors are scheduled to be operating underground in the last quarter of 1979.



HEIGHT CLOSED 910 mm (35.8 in.)*
HEIGHT OPEN 4180 mm (164.5 in.)*
ROOF CANOPY AREA 53.7 ft^2
SUPPORT DENSITY (30 in. WEB)

IFS WORKING
BEFORE CUT 58.5 $tons/m^2$ (6.0 $tons/ft^2$)
AFTER CUT 48.6 $tons/m^2$ (5.0 $tons/ft^2$)

CONVENTIONAL WORKING
BEFORE CUT 73.1 $tons/m^2$ (7.48 $tons/ft^2$)
AFTER CUT 58.5 $tons/m^2$ (6.0 $tons/ft^2$)

*REFERS TO ENTIRE RANGE OF MODELS - TYPICAL EXTENSION RATIOS ARE IN THE AREA OF 2.2:1 FOR A DOUBLE TELESCOPIC LEG

FIGURE 8. - High support density for four-leg shield Dowty 500-ton capacity for one web back operation.

2.3 Problems in Longwall Mining

Although introduced in the early 1950's, longwall mining accounted for only 4 percent of the total United States underground production in 1977. This figure is expected to reach 15 percent by 1985, and longwall production is expected to match annual continuous mining production by the year 2000 (7).

This subsection of the report will briefly discuss some of the problems that may have restricted the use of longwall in the last 30 years. Additionally, some of today's problems and possible solutions to these problems will be discussed particularly as they relate to productivity and cost per ton.

2.3.1 Longwall Problems - Historical

Fully mechanized longwall mining was a European development. When introduced in the early 1950's, it had to compete with the room and pillar method of mining, which was developed in this country. European manufacturers entered the market with equipment that was designed for the mining conditions in Europe. As a result, some of the early applications ended in failure because the roof supports did not have adequate capacity (tons of support).

Because of the different roof conditions in the United States mines, over the years, foreign manufacturers of roof supports have also responded to the need for substantially greater capacity than is generally required in Europe. This is clearly evident when a comparison is made between equipment used on British longwalls and on United States longwalls as shown in Table 2 (1). Only 22.3 percent of the faces in the United Kingdom use heavy duty (high capacity) supports whereas in the United States 85.7 percent do. The same situation is true of heavy duty (high capacity) shearers and face conveyors. In the United States, 53.2 percent of longwalls use cutter-loaders with motor capacity greater than 230 hp, whereas in England only 1.2 percent do.

Longwall mining equipment being supplied to the industry today, although still designed and constructed by foreign manufacturers, is tailored to mining conditions prevalent in the United States. As the American market continues to expand, machinery specifically tailored for the United States market will, in all probability, be manufactured in the United States as well.

The high initial investment required to develop and equip a longwall panel, however, is a major problem today, as will be discussed in the following subsection. This is evidenced by the fact that only the larger coal companies in the United States use the longwall method of mining (4).

TABLE 2. - Distribution of heavy duty longwall face equipment in the United States and the United Kingdom - 1975 (33)

Country	Equipment	Number	Percent of total faces
United Kingdom (730 faces)	Supports	163	22.3
	Conveyors	9	1.2
	Cutter/Loaders	9	1.2
United States (77 faces)	Supports	66	85.7
	Conveyors	26	33.8
	Cutter/Loaders	41	53.2

Notes:

1. Heavy duty equipment defined as:

   Supports in excess of 200-ton capacity; cutter/loaders in excess of 230 hp; conveyors heavier than the NCB standard specification (for example, H&B EKF-3, Dowty Meco 10 in. NCB Model 222).

2. Largest support in the United Kingdom - 250-ton capacity

3. Supports in the United States:

   Minimum capacity - 168 tons

   Maximum capacity - 700 tons

   Majority of faces - 400 to 500 tons

4. Cutter/loader includes shearers, plows and trepanners.

2.3.2 Longwall Problems Today

The principal problems generally associated with longwall mining today are:

a. Geologically related problems

b. Dust on the face

c. Entry (panel) development

d. The high cost of longwall equipment.

Geologically related problems are always present in any underground mining operation. On any longwall face, at any time, the principal problem can be geological abnormalities. The impact of these problems can vary from very little effect to complete loss of the face. The impact of these difficulties can be reduced but never eliminated. Panel layouts can be planned around better knowledge of the presence of, and the locations of, such abnormalities as major faulting of the seam or overlying strata.

Dust is another major problem on United States longwall faces. A majority of longwall operations are not in compliance with Federal regulations for maximum respirable dust in suspension. Operational procedures are often modified to reduce the impact of dust. For example, shearers will be limited to cutting in one direction on the face so as to situate the shearer operator on the fresh air side of the cutting drums. Under more extreme conditions, face crews will be cycled in a shift to limit time of exposure.

Improved designs of cutter drums, picks and pick lacing patterns can have a significant effect on dust. Slower drum speeds increase cutting efficiency and reduce the effect of recirculating cut coal in the drum helix.

Dust control techniques presently being developed in Europe should, in the near future, have a favorable effect on dust control in the United States. These techniques include pick face flushing and through the drum ventilation. Additionally, the USBM is sponsoring work in this country to combat the dust problem. The physical effort, at this time, is directed towards the use of dust "scrubbers" on longwall shearers.

Geologically related problems and dust are operational difficulties which generally do not restrict the increased use of the longwall mining method. However, the high cost of longwall equipment and the cost of entry (panel) development represent a capital investment that appears to be restricting its use. This is substantiated by the fact that all longwalls presently operated in the United States are owned by the larger mining companies who can afford the investment (4).

Longwall mining's contribution to total underground production can be improved by reducing the impact of the high cost of longwall equipment and by reducing the time and cost of panel development. This can be best accomplished by:

a. Developing techniques or methods that will reduce the time required and/or the economic impact of panel development

b. Increasing the productivity of the longwall face to reduce the impact of the high cost of longwall equipment.

Starting a new longwall panel generally requires a year or more to drive the entries. The retreat method of longwalling requires that the panel development be completed before the production equipment (the longwalling face) is installed. During the development stage, production is limited to the coal extracted from the development of the entries. The high rate of production, from the longwall face itself, is not available for payback until the panel development is completed and the longwall equipment is operational. Additionally, entry development for subsequent panels may lag longwall production. This oftentimes results in a longwall section being idle while development of the next panel is completed.

Because of legal requirements, most panels are developed by driving three separate entries with cross cuts which form two rows of coal pillars. By employing barriers in the cross cuts, the entries can be isolated from one another to comply with legal requirements for escapeways and ventilation. Driving an entry system of this complexity is a slow process.

Single entry development has been suggested as an alternative. One entry is driven and the requirements for separate ways is provided for by building man-made barriers (walls) as the entry is driven. This has the potential to substantially increase the rate of development.

A second approach to reducing the economic impact of entry development is to employ the advance method of longwall mining, the European method. Since with advancing longwall the panel is extracted coincidental with development, coal production from the longwall face is immediately available. Single entry development with the advance method of longwalling would, in combination, minimize the economic impact of panel development. Single entry development has the additional advantage of maximizing resource recovery as there are no coal pillars left unmined.

Finally, increasing productivity from the longwall face has the potential to reduce the economic impact of the high cost of longwall equipment, provided that the increased production is obtained at minimum additional cost. In the subsection that follows, the factors that affect or limit production off the face are identified and discussed. Alternative approaches to increasing production, including the mining of wider webs, are discussed and compared.

2.3.3 The Potential for Wide Web Longwalling

Subsection 2.3 identified recent trends in the purchase of new longwall equipment. Among those trends were the specification of higher horsepower shearers and the development of higher capacity face conveyors. The capability of the higher horsepower shearers can result in an increase in machine system capacity (in tons mined per hour) or an increase in shearer reliability. The higher rate of mining can be achieved if the shearer is not restricted by other factors such as coal haulage systems capacity. The larger shearers can be more reliable since they are of heavier construction with reserve power for mining through abnormalities, such as rock bands and sulphur balls.

For a given mineable seam height, the larger shearers will have the potential to mine the standard web depths at a faster advancing speed (rate of advance along the face). Alternatively, the increased power could be used to mine deeper webs at slower advancing speeds.

Subsection 2.3 also identified the shift to the chainless haulage systems in the purchasing of new longwall shearers. This will increase the potential to mine longer faces, as the older haulage systems required that a chain be stretched from entry to entry along the face. The length of chain that could be operated, and therefore the face length, was often limited when mining pitching or undulating coal seams.

The recent trends in the design and utilization of contemporary longwall equipment suggests that increases in shift production can be achieved by:

- An increase in machine system reliability
- Increasing the shearer speed (increased rate of mining)
- Mining deeper webs
- Mining longer faces.

There are many factors which must be considered in assessing these alternatives which include safety, cost effectiveness, percent resource recovery, and the impact on other operations such as entry development. However, the relative potential for each of these alternatives to increase shift production can be evaluated using the formula shown in Table 3 and the performance data for a contemporary high capacity longwall face. For this study, FMA surveyed five high production faces for the purpose of establishing an average high performance operation to be used as a data base for projecting the cost effectiveness and productivity of changes in operating parameters. The operating parameters of this contemporary high capacity longwall face are summarized in Tables 15 and 16 in subsection 4.1. The results of the five mine surveys are individually shown in the tables of Appendix A.

The formula of Table 3 is developed in Appendix B and represents the performance of a longwall face using a bidirectional shearer in the half face or modified half face mode of operation. The factors taken from Table 15 for use in the formula of Table 3 are:

t_t = lost time off the face = 79 min

H = height of extraction = 6.5 ft

T_t = total shift time = 8 hr (480 min)

L = face length = 500 ft

S_c = tramming speed = 30.1 ft/min

t_e = turnaround time = 7.54 min

t_ℓ = lost time in the face = 120.5 min

W = web depth = 27.2 in.

TABLE 3. - Shift production formula - half face sumping double ended shearer

Tons per shift (TPS) = tons per cycle × cycles per shift

$$TPS = \frac{LWH}{266.7} \times \frac{T_t - t_t - t_\ell}{\frac{L}{S_{c\ell}} + \frac{L}{S_c} + 2t_e}$$

where:

T_t = total shift time in minutes (480 for an 8-hr shift)

t_t = lost time off the face in minutes (travel time, etc.)

t_ℓ = lost time on the face in minutes (down time related to machine system failure, geological conditions, etc.)

L = face length in ft

W = web depth in in.

H = height of extraction (seam height) in ft

$S_{c\ell}$ = shearer speed in ft/min while mining coal

S_c = shearer speed in ft/min while tramming (flighting)

t_e = turnaround time at the entries in min

The relationship between machine system capacity (C_c) in tons/hr and shearer speed in ft/min is:

$$S_{c\ell} = \frac{4.44\ C_c}{WH}$$

The constants 4.44 and 266.7 are based on in situ specific weight of coal at 90 lb/ft^3.

NOTE: This formula applicable where cycle time is a function of shearer performance restricted by either shearer power, coal haulage system capacity or shearer control.

The use of these values in the formula for shift production will provide a projection of the relative impact of an increase in reliability (a reduction in lost time t_{ℓ}), an increase in shearer speed ($S_{c\ell}$), mining deeper webs (an increase in W), and mining longer faces (an increase in L). The comparative analysis will project the impact of selected increases (or decreases) in the parameter of interest while holding all other parameters constant.

The Potential for Increasing Shearer Speed

The contemporary high capacity face of Table 16 is equipped with a 300 kW shearer and an EKF-3 conveyor. The capacity of this combination is usually limited by the rated capacity of the conveyor, which is some 800 tons/hr. Therefore, the first alternative to increasing shift production is to speed up the shearer to cut and load out coal at the rated capacity of the conveyor. The relationship between shearer speed ($S_{c\ell}$), web depth (W), height of extraction (H) and system capacity (C_c) is:

$$S_{c\ell} = \frac{4.44\ C_c}{WH}$$

From the data of Table 15 the shearer is mining at a capacity of 517 tons/hr and at a shearer speed of 13.1 ft/min. The full capability of the mining machine system is not being utilized since the 800 tons/hr capacity conveyor is only being loaded with 517 tons/hr. Shearer speed would have to increase to load the conveyor at its rated capacity where the new shearer speed ($S_{c\ell}$) would be:

$$C_c = 800 \text{ tons/hr}$$

$$W = 27 \text{ in.}$$

$$H = 6.5 \text{ ft}$$

$$S_{c\ell} = \frac{4.44\ (800)}{27\ (6.5)}$$

$$S_{c\ell} = 20.2 \text{ ft/min.}$$

If the shearer speed is increased to 20.2 ft/min (a 54.2 percent increase), then shift production will also increase, all other parameters remaining constant. Using the formula on Table 3 with a shearer speed of 20.2 ft/min, shift production increases from 1316 tons/shift to:

$$TPS = \frac{LWH}{266.7} \quad \frac{T_t - t_t - t_\ell}{\frac{L}{S_{c\ell}} + \frac{L}{S_c} + 2t_e}$$

$$TPS = \frac{500(27)\ (6.5)}{266.7} \times \frac{480 - 79 - 120.5}{\frac{500}{20.2} + \frac{500}{30.1} + 2(7.54)}$$

$$TPS = 1645.9 \text{ tons/shift}$$

This represents a 25.1 percent increase in shift production from a 54.2 percent increase in shearer speed.

<u>The Potential for Mining Wider Webs</u>

As an alternative to increasing the speed of the shearer, the rated capacity of the conveyor can be achieved by increasing the depth of web mined. For example, if it is desired to maintain the shearer speed at 13 ft/min because of difficult conditions of mining, the depth of web that could be mined without exceeding the 800 tons/hr capacity of the conveyor would be:

$$W = \frac{4.44\ C_c}{S_{c\ell}\ H}$$

where

$$C_c = 800 \text{ tons/hr}$$

$$S_{c\ell} = 13 \text{ ft/min}$$

$$H = 6.5 \text{ ft}$$

$$W = \frac{4.44(800)}{13(6.5)}$$

$$W = 42 \text{ in.}$$

As will be shown later in this report, the 42 in. web depth (45 in. wide drums) is well within the capability of the 300 kW class of shearers. Under these conditions of mining, shift production of 1316 tons/shift will increase to:

$$TPS = \frac{500(42)(6.5)}{266.7} \times \frac{480 - 79 - 120.5}{\frac{500}{13} + \frac{500}{30.1} + 2(7.54)}$$

$$TPS = 2042.7 \text{ tons/hr}$$

This represents a 55.2 percent increase in shift production for a 55.6 percent increase in web depth.

<u>The Potential for Mining a Longer Face</u>

The increase in production achieved by mining a wider web was a result of substantially increasing the tons of coal mined per pass of the shearer. The same objective can be achieved by mining a longer face. If the face length is increased by 55 percent, from 500 to 775 ft, then there will be a shift production increase. If the face is operated at a shearer speed of 13 ft/min, loading out coal from the 27 in. web at 517 tons/hr, then the increase in face length will result in a shift production of:

$$TPS = \frac{775(27)(6.5)}{266.7} \times \frac{480 - 79 - 120.5}{\frac{775}{13} + \frac{775}{30.1} + 2(7.54)}$$

$$TPS = 1424.3 \text{ tons/shift}$$

This represents only an 8.3 percent increase in production while the face is operating below machine system capacity. If the face is upgraded to mine coal at 800 tons/hr at a shearer speed of 20.2 ft/min, then the effect of the longer face will be:

$$TPS = 510 \times \frac{480 - 79 - 120.5}{\frac{775}{20.2} + \frac{775}{30.1} + 2(7.54)}$$

$$TPS = 1806.2 \text{ tons/shift}$$

This represents a 37.2 percent increase in shift production over the 1316 tons/shift.

The Impact of Increasing System Reliability

Average time lost in a shift due to all causes occurring on the face or in the entry (including outby haulage stoppages) was 120.5 min for the contemporary high capacity face of Table 15.

As in the case of increasing the face length, the calculated impact of a reduction in this lost time will also be dependent on the rate of mining coal. For example, a reduction of 55 percent in lost time will increase shift production from 1316 tons/shift to 1626 tons with the shearer mining coal at 517 tons/hr. If the same increase in mining system reliability is applied, when mining coal at 800 tons/hr, then shift production increases to 2021 tons/shift, a 53.6 percent increase.

The combined results of this analysis are summarized in Table 4. As expected, shift production increases are realized in all cases. However, for this particular operation, an increase in web depth and an improvement in system reliability have the greatest potential to increase productivity.

It should be realized that this type of analysis requires simplifying assumptions. For example, in considering mining a deeper web, or mining with a faster shearer, lost time (t_ℓ) on the face was treated as a constant. This assumes that roof control problems will not effect performance when mining a wider web, and that operations with the faster shearer will not be effected by increases in levels of dust on the face.

TABLE 4. - Potential impact on increasing shift production - stereotypical United States longwall

Variable	Percent change in variable	Percent increase in shift production*
Shearer speed	54.2	25.1
Web depth	55.6	55.2
Face length	55.0	37.2
System reliability	55.0	53.6

*Based on 1316 tons/shift typical high production face of Table 15

These assumptions may or may not be true, but it is valid to the extent that this analysis is an attempt to show the "relative potential" that these alternative courses of action may be expected to have. To further qualify these results requires that other factors be considered.

More reliable equipment and reductions in lost time will positively increase productivity under all mining conditions. In the United Kingdom, where the NCB is operating over 700 longwall faces, measuring the performance of and identifying the requirements for increasing machine system reliability is one of the major efforts directed towards increasing productivity. The NCB does not see the need for increasing mining machinery capacity, but rather to develop more reliable equipment.

Mining with faster shearers may increase production, but it may also produce more dust. Additionally, the entire operation on the face must be speeded up, which may increase the number of accidents and may require more sophisticated methods for steering the shearer. Automating the shearer or the entire face may, in the future, result in the operation of very high speed shearers.

Although the longer face showed a potential production increase of 37 percent, it may be the least cost effective alternative. Lengthening the face requires the purchase of additional roof supports and additional length of face conveyor.

Roof supports represent the most expensive equipment on the face costing $6,000 to $8,000/ft depending on type of support and capacity. Additionally, increasing the length of the face conveyor may require larger drives and chain tensioning systems.

Mining longer faces, however, does maximize the percentage of the seam mined and does reduce the impact of entry development. The cost and time required to develop the entries is a major problem in longwalling. Lengthening the production face maximizes the tons of coal won per foot of entry developed and reduces the amount of coal left in entry pillars.

Finally, Table 4 indicates that mining deeper webs has the best potential for increasing productivity. Furthermore, this increased production can be achieved with reduced shearer speeds. This can be expected to have a positive impact on safety and to reduce the problem of effectively controlling the shearer. The major factor to be considered is the effect of wider webs on roof control. Mining wider webs will result in less tramping (stress cycling) of the immediate strata and a faster advance of the face. Both of these factors can be expected to improve roof conditions. However, careful consideration must be given to design and selection of supports since stability and floor loading patterns could be a problem. Finally, wider webs result in a greater area of roof exposure in front of the canopy. Control of the roof over this wider area of exposure may be the key factor in determining the success or failure of a wider web face.

At the beginning of this section it was pointed out that longwall face equipment is designed and manufactured primarily in West Germany and the United Kingdom. The depth of web mined in the United States has been largely determined by experience in the United Kingdom where the shearer-type cutter/loader was developed and is most extensively used.

In the subsection that follows, this influence will be discussed, particularly with regard to the prop free front (PFF) distance which has been legally imposed on the British coal mining industry. However, it will be additionally shown that the British have had recent experience with mining wider webs, and that the results published to date also substantiate that wider webs increase shift production.

2.4 Wide Web Longwalling in the United Kingdom

In the very early days of longwall mining, coal was hand loaded from the face onto belt conveyors and the roof was supported by hand set wooden props. To insure a degree of safety, the British Inspectorate regulated the minimum spacing of the wooden props and the maximum distance the first line of props could be set from the operating face.

One of the very early mechanized machines deployed to cut and load coal from a longwall face was the Meco-Moore wide web miner shown in Figure 9 (8). This machine extracted a cut 6 to 8 ft deep (web) with hand set props and beams supporting the roof. At that time, the maximum distance the first row of supports could be set from the face was some 3 ft. This distance will hereafter be referred to as the PFF distance.

The introduction of the self-contained powered support, and subsequent elimination of hand set props and bars, obviated the requirement for the minimum support spacing (density) regulation. The PFF regulation, however, could still be applied in principle and therefore remained in effect.

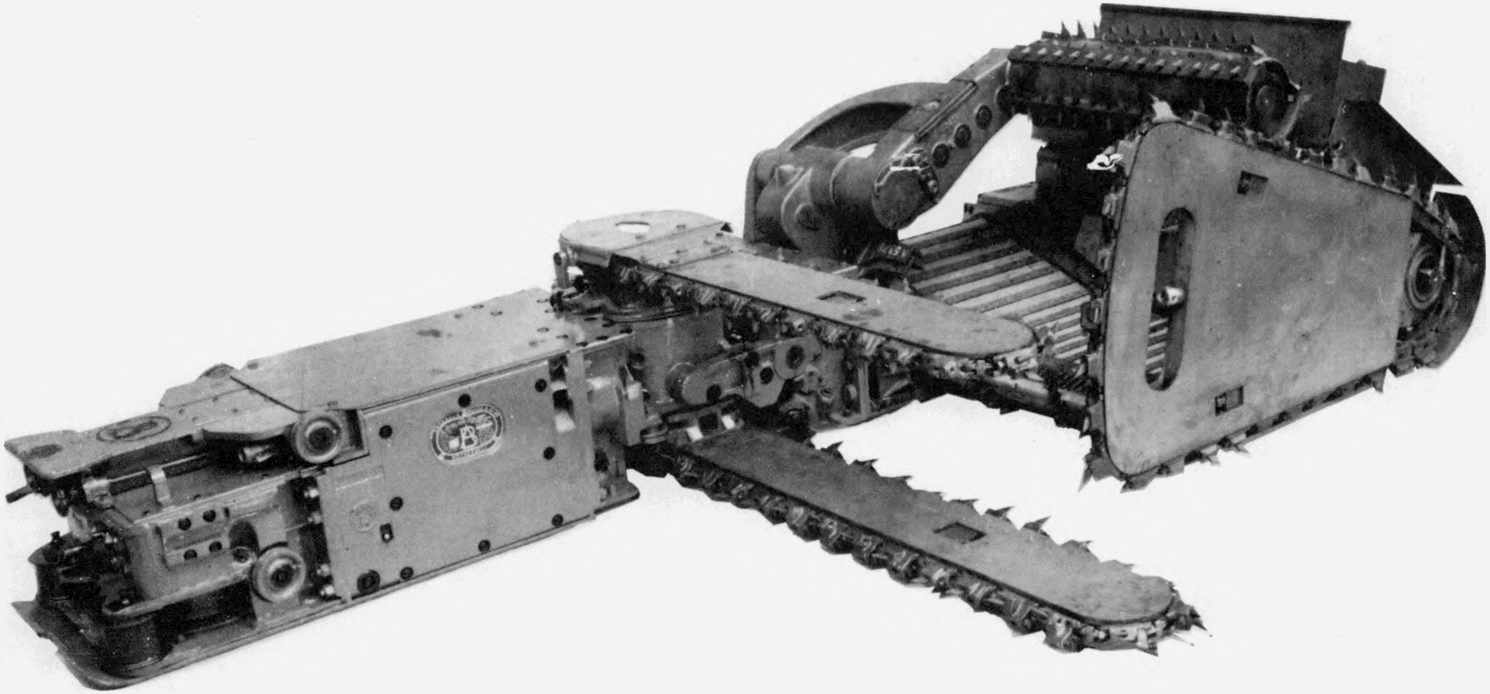


FIGURE 9. - The Meco-Moore mechanized longwall miner.

The PFF regulation has been historically relaxed over the years with the introduction of the German developed armoured (panzer) face conveyor, the development of the pan mounted modern shearer and the introduction of the self-advancing chock with solid roof canopy. These improvements, in combination with the constantly increasing demand for production and increased equipment size, has resulted in the present day regulation - which allows for a PFF up to 6 ft, 6 in.

2.4.1 Web Depth - Present Practice

The regulated PFF distance which must accommodate space for the shearer body, the conveyor and gob side furniture, shearer ranging arm width, and the drum width in fact dictates the depth of web that can be cut. This is illustrated in Figure 10.

As equipment has grown in size and complexity over the years, the depth of web that could be taken within the 6 ft, 6 in. PFF distance has been periodically reduced. This has, of course, reflected itself in decreased production off the face (9). This situation was substantially affected when the double ended shearer with ranging arms was introduced to British mines in 1966 (the Eickhoff EDW-170L) and was again in 1967 when the Anderson Boyes Mk II DERD was initially installed on a standard 30 in. wide twin 18 mm chain conveyor. This became the standard face, but could only accommodate a 21 in. wide drum, within the 6 ft, 6 in. allowance because of the additional space occupied by the ranging arms.

To provide for wider webs the 6 ft, 9 in. PFF was introduced in 1970, which allowed the mounting of 24 in. wide drums. Under these conditions production was limited by the capacity of the 18 mm chain face conveyor. To overcome this limitation, the 26 mm single strand face conveyor was introduced in 1973 on the 6 ft, 9 in. PFF face.

To take advantage of the full capability of this heavy duty conveyor, the 7 ft, 3 in. PFF face was recently designed to accommodate a 27 in. drum. It is expected that this face design will be extensively used throughout the United Kingdom in the future.

Finally, the Inspectorate granted permission to operate an 8 ft, 3 in. PFF in the NCB's western area for the purposes of accommodating a 1-m (39 in.) drum width. This arrangement has been successfully used on three faces at the Hem Heath and Holditch Collieries. Table 5 summarizes the face designs that are presently working in the United Kingdom with the drum widths that are accommodated within the prop face front distances.

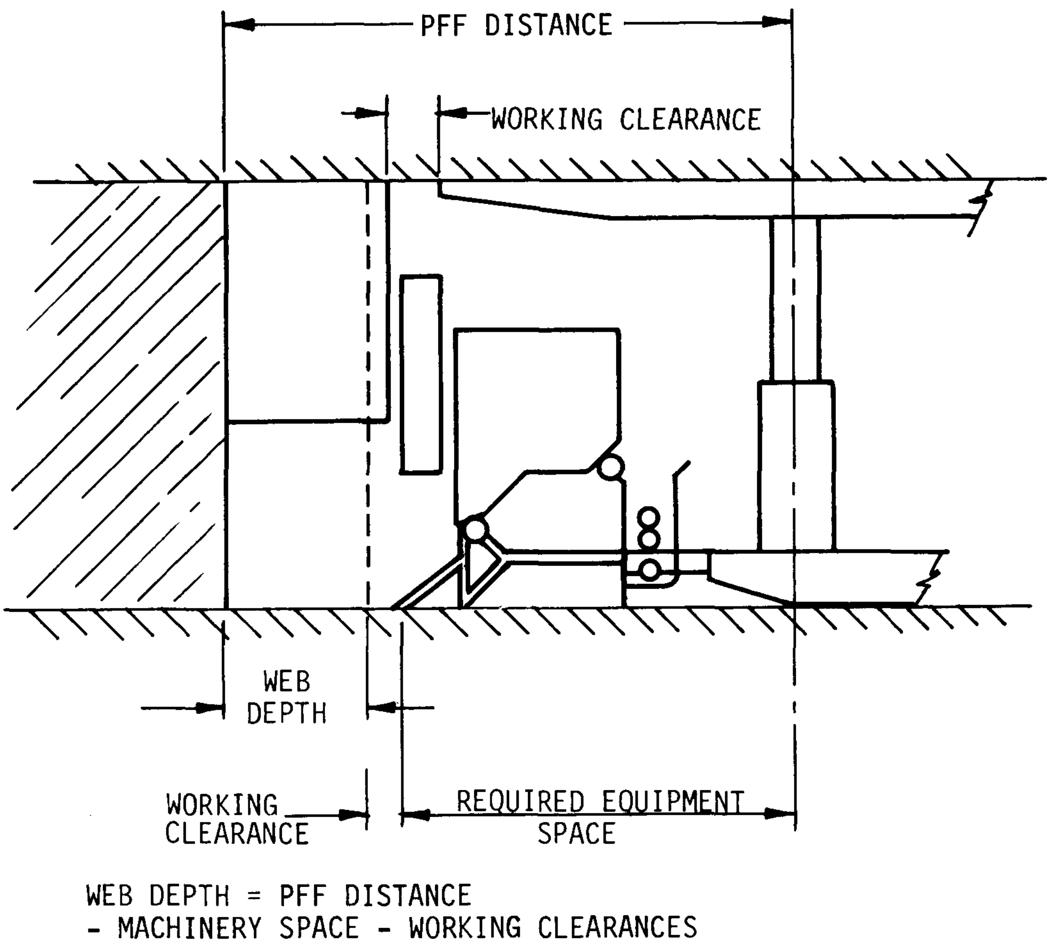


FIGURE 10. - Effect of PFF regulation on web depth.

TABLE 5. - Progression of longwall face development in the United Kingdom with DERD shearers (2)

PFF	AFC	Shearer	Drum Width	Remarks
6 ft, 6 in.	30 in. twin 18 mm	200 hp DERD	21 in.	Standard face for DERD shearer starting in 1967
6 ft, 9 in.	30 in. twin 18 mm	200 hp DERD	24 in.	Face design initiated in 1970. Some 82 faces operating
6 ft, 9 in.	30 in. single 26 mm	200 hp DERD	24 in.	High production face introduced in 1973
7 ft, 3 in.	30 in. single 26 mm	200 hp DERD	27 in.	Latest design of high production face
8 ft, 3 in.	30 in. single 26 mm	200 hp DERD	39 in.	One meter web faces at Holditch and Hem Heath

The equipment supplied to the American market historically was designed to fit within the PFF regulations. This included shearers, roof supports and conveyors. In the absence of working experience, the United States operator historically accepted the PFF distance as having a proven safety record. However, as Table 5 illustrates, the PFF distance is not a fixed parameter in the United Kingdom. It has increased with the introduction of changes in equipment designs and the need for more production from larger equipment.

There is no reason to believe that wider webs cannot also be mined in the United States and that, as in the United Kingdom, wider webs will increase production. This increase in productivity has been demonstrated on the 1-m web face at the Holditch and Hem Heath Collieries. As the following subsection will also show, roof control and dust also improved on these wide web faces.

2.4.2 One-Meter Web Mining at Holditch and Hem Heath (10)

The primary goal of the NCB during the 1950s and 1960s was to increase productivity by fully mechanizing longwall faces. This has essentially been achieved. However, productivity peaked in early 1970, and since then there has been a steady decline in both total production and productivity (11).

As a result, in 1974 the NCB and its research arm, the Mining Research and Development Establishment (MRDE), began the Advanced Technology Mining (ATM) program. One of the stated goals of this program was to increase production and safety by developing and introducing new extraction techniques. One of the new techniques of extraction that received immediate consideration was the mining of wider webs. The development of ranging arm shearers and the introduction of automatic cable handling had steadily reduced the average web on a mechanized longwall face to less than 21 in. The initial demonstration to reverse this trend by mining a 1-m web was planned for the Holditch Colliery in the western area.

The Holditch Colliery (10)

Underground mining of a planned 1-m web required a PFF distance of 8 ft, 3 in. and a roof support that would provide:

a. 240 tons of capacity in a heavy duty chock with forepoling

b. Forepoling to within 12 in. of the face

c. Setting and yielding tip loads of 5 and 10 tons, respectively with forepoling fully extended

d. Remote control advance of the supports.

Permission for the required 8 ft, 3 in. PFF was granted by the Inspectorate and underground mining commenced in November 1974. The face was initially mined with a conventional drum width taking a 21 in. web. The wider drum was then mounted and a 37.2 in. web was extracted. The equipment specifications and comparative results are shown in Table 6, and the arrangement of the equipment on the face is illustrated in Figure 11.

The increase in weekly production was directly related to the web increase as tons/shear increased 65 percent with the wider web while the number of shears/shift only decreased 13 percent. It was also reported that roof control over the wider web improved because of less tramping (stress cycling) of the immediate roof strata. Stress cycling is the number of times the supports are released, advanced and reset under each segment of roof which, of course, is reduced with a wider web.

TABLE 6. - One meter web mining - Holditch Colliery comparative results

Equipment:		
Shearer		
Type	Anderson Mavor DERD	
Horsepower	200	
Drum type	3 start spiral vane	
Diameter and width	54 in. diam x 39 in. wide	
Drum speed	45/22.5 rpm	
Cowls	Manual HD	
Supports		
Type	Gullick-Dobson 6/240	
Canopy area	39.0 ft^2	
Extension:		
Width	8.5 in.	
Length	30.0 in.	
Ram stroke	42 in.	
Face Conveyor		
Type	25 mm single strand	
Speed	230 ft/min	
Horsepower	240	
Performance:	Normal Web	Wide Web
Web depth (in.)	22.2	37.2
Working weeks	8	29
Production/week (tons)	4479	6358
Shears/week	22	19
Tons/shear	201	331
Tons/manshift	22.1	31.0

Percent increase in web = 67 percent
Percent reduction in shears = 13 percent
Percent increase in production = 42 percent
Percent increase in productivity/manshift = 36 percent

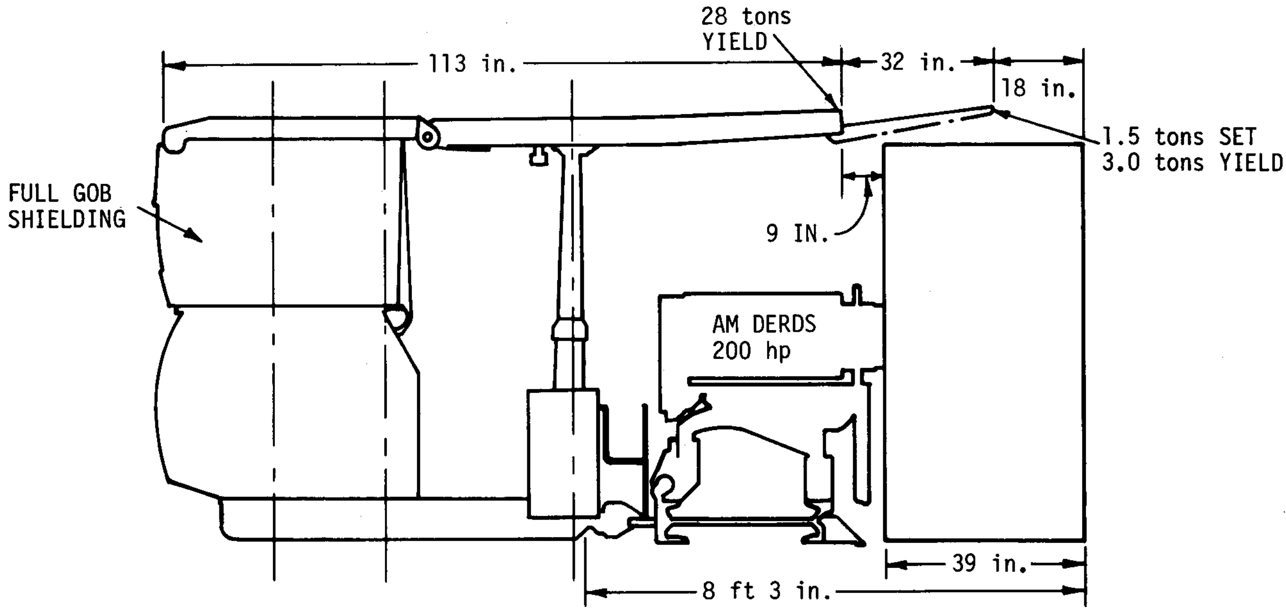


FIGURE 11. - Equipment arrangement at Holditch Colliery.

The Hem Heath Colliery

NCB-MRDE planning for a high production demonstration face at the Hem Heath Colliery, in the newly developed Yard/Ragman coal seam, began in 1974. The goals and objectives of this demonstration face were established as requiring:

a. Equipment selection that would result in one million tons of production between major overhauls

b. The mining of a wider web to increase production, provide for a more comfortable working environment and increase safety

c. The development of a chock support specifically designed for 1-m web extraction.

As in the demonstration at Holditch, an exemption was granted and the face was designed and operated with an 8 ft, 3 in. PFF for two operating faces. Underground mining commenced in November 1975 with the arrangement of equipment shown in Figure 12. Equipment specifications and comparative mining results are shown in Tables 7 and 8.

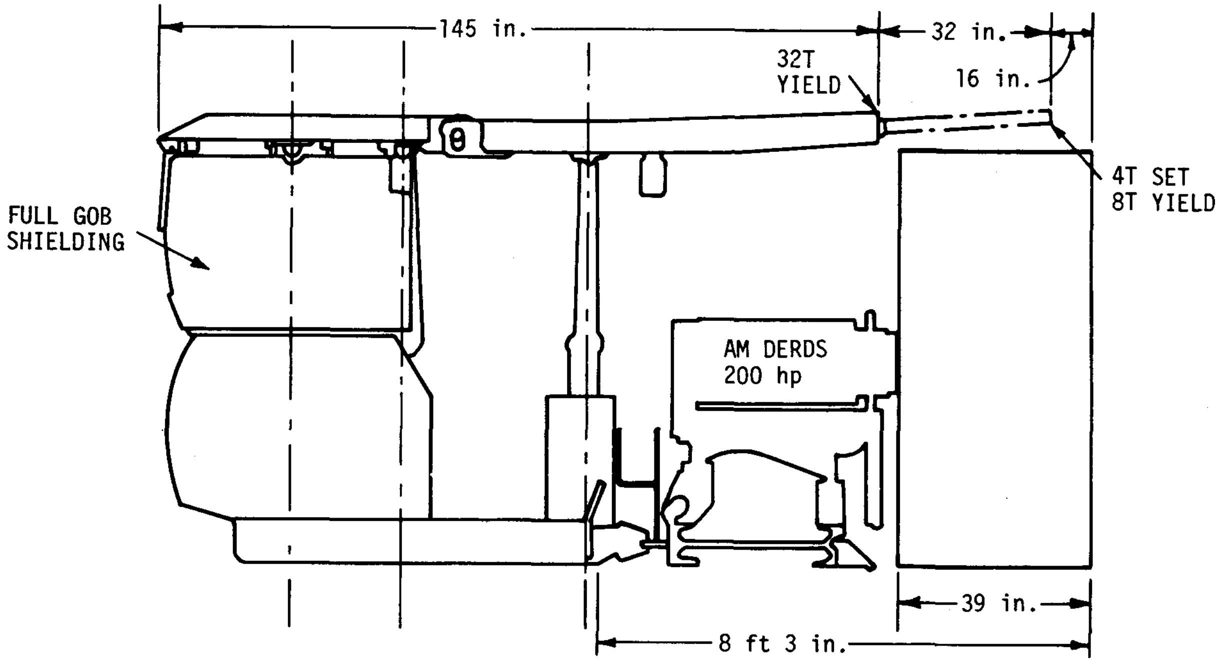


FIGURE 12. - Equipment arrangement first face Hem Heath Colliery.

The comparative results, as in the case at Holditch, showed an increase in production directly related to web depth. The meter web has, at this writing, been extracted on two faces at Hem Heath. The rate of mining on the two 1-m web faces was restricted to 400 tons/hr by outby haulage limitations. As a result, shearer advancing speed was limited to between 8 and 12 ft/min, which provided for easier working conditions on the face. The extended PFF also allowed for an increase in machine clearances which resulted in a more reliable operation.

At the Hem Heath Colliery, the mine floor was relatively soft. Some difficulty was experienced when mining the wider web because the supported roof load was heavier and the pressure pattern shifted towards the toe of the base. In addition, particular care had to be exercised when advancing the conveyor across the wider web since pan parting can result in damage to the pan section connectors.

The highest wide web production was achieved on Hem Heath's second face, which can be credited to a combination of more tons per shear and more completed shears per week. When compared to

TABLE 7. - One meter web mining - Hem Heath Colliery comparative results - first face

Equipment:		
Shearer		
Type	Anderson Mavor DERD	
Horsepower	200	
Drum type	3 start spiral vane	
Diameter and width	50 in. diam x 39 in. wide	
Drum speed	45/22.5 rpm	
Cowls	Manual HD	
Supports		
Type	Gullick-Dobson 6/240	
Canopy area	35.6 ft^2	
Extension:		
Width	8.5 in.	
Length	30 in.	
Ram stroke	42 in.	
Face Conveyor		
Type	18 mm twin strand	
Speed	169 ft/min	
Horsepower	155	
Performance:	Normal Web	Wide Web
Web depth (in.)	22.3	37.0
Working weeks	10	39
Production/week (tons)	5121	6585
Shears/week	18.5	17.3
Tons/shear	277	380
Tons/manshift	23.2	24.4

Percent increase in web = 66 percent
Percent reduction in shears = 6 percent
Percent increase in production = 28 percent
Percent increase in productivity per manshift = 5 percent

TABLE 8. - One meter web mining - Hem Heath Colliery production results - second face

Equipment:	
Shearer	
Type	Anderson Mavor
Horsepower	200
Drum type	3 start spiral vane
Diameter and width	60 in. diam x 39 in. wide
Drum speed	45/22.5 rpm
Cowls	Manual HD
Supports	
Type	Gullick-Dobson 6/240
Canopy area	35.6 ft^2
Extension:	
Width	15.75 in.
Length	32 in.
Ram stroke	42 in.
Face Conveyor	
Type	26 mm single strand
Speed	158 ft/min
Horsepower	210
Performance:	
Web depth (in.)	37.0
Working weeks	17
Production/week (tons)	7828
Shears/week	19.3
Tons/shear	406
Tons/manshift	36.8

the first face at Hem Heath, the higher production operation completed more passes per week. When compared to the Holditch wide web operations, the major increase was in tons per shear. This was due to the longer 510 ft face at Hem Heath as compared to the 405 ft face at Holditch.

It was reported (10) that face conveyor and/or outby haulage limited production on all three faces to 350 to 400 tons/hr. Preliminary surface cutting tests conducted at Holditch indicated that full power performance of the shearer was uneffected by the increase in web depth, being some 540 tons/hr for both the 22 and 37 in. web depths.

In actual operation the Holditch shearer cut and loaded 360 tons/hr using 66 percent of available power. The same shearer operating on both faces at Hem Heath mined 350 tons/hr from the wide web with 60 percent of available power. This strongly suggests that in actual performance on all three faces there was no significant increase in power required to mine the wider webs - confirming the preliminary surface tests at Holditch.

2.4.3 Wide Web Mining in Low Seams

Unlike conditions in the United States, much of the more accessible coal in the United Kingdom has already been mined. For this reason, a substantial effort has been directed to the development of equipment for extracting adequate production from very low seams of coal. This requirement has lead to the development of the in-web or buttock type of shearer. A typical bidirectional (double ended) buttock shearer with ranging arms is shown in Figure 13.

The in-web shearer is particularly suited to low seam mining because the body of the shearer is alongside of the face conveyor (in the web) rather than over the conveyor. The face conveyor is straddled only by the control canopy as shown in Figure 13. In addition, the repositioning of the machine body, ranging arms and cutting drums allows the extraction of a wider web within a restricted PFF.

The Florence Colliery

The wider web with the in-web machine was demonstrated at the Florence Colliery in Western Area where an in-web shearer was placed on a low seam face within the regulation 6 ft, 6 in. PFF. This machine commenced mining in October 1976 with 35 in. wide drums in an inclined seam (25 percent gradient). This shearer extracted a 33 in. web loading in favor of the gradient.

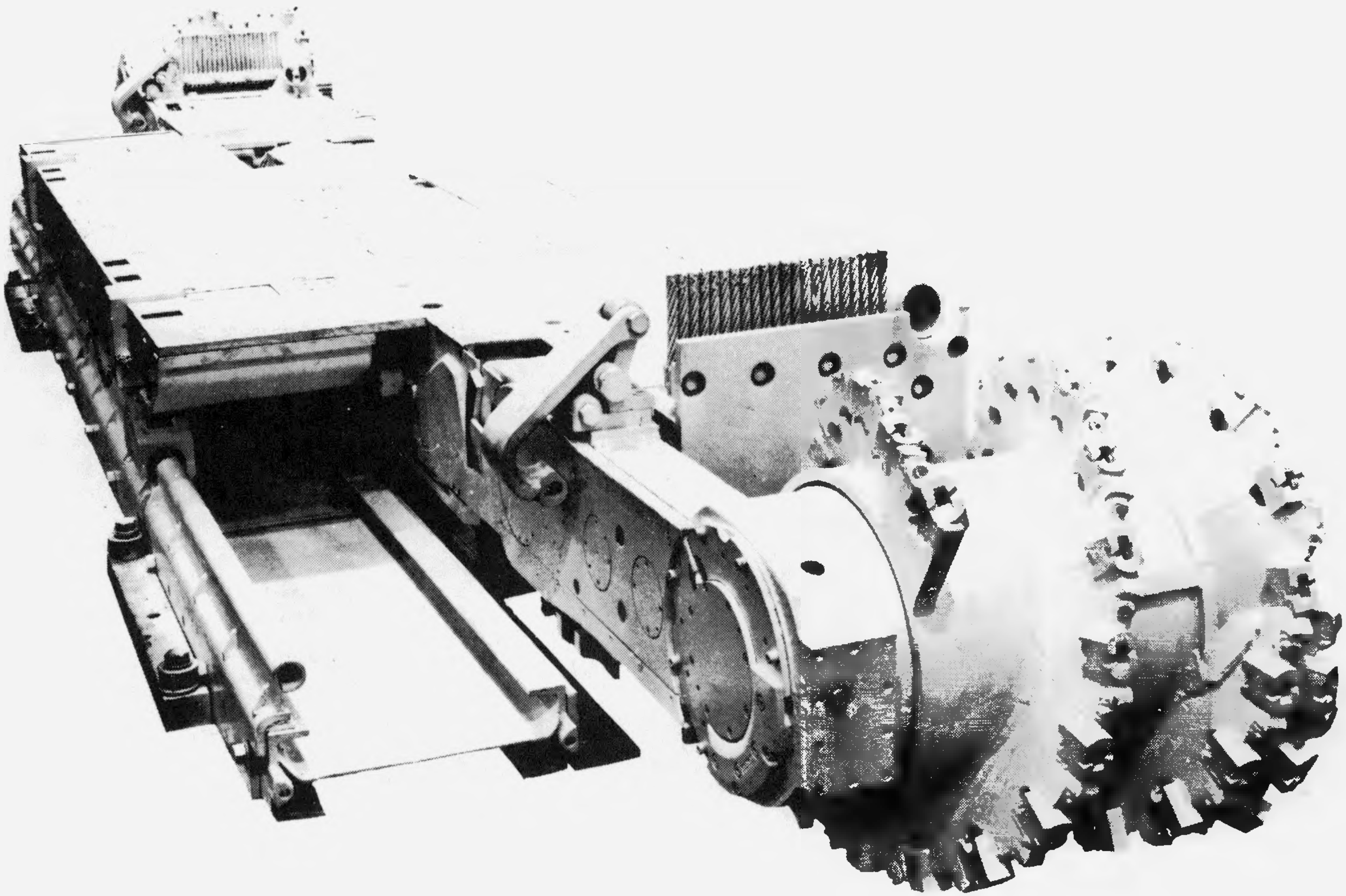


FIGURE 13. - Typical in-web (Buttock) double ended shearer.

This represents a 57 percent increase over the average 21 in. web normally taken with an on-pan shearer within the same 6 ft, 6 in. front. The wider web is possible because with the in-web shearer the ranging arms are situated so that their width does not detract from the PFF distance. This operation was helped by loading in favor of the gradient which is beneficial in low seam mining since effectively loading the conveyor is oftentimes a major problem. Equipment specifications, production results and equipment arrangement are shown in Table 9 and Figure 14.

Surface Tests - Very Wide Drum in Web Shearer

Based on the favorable results achieved at the Florence Colliery, NCB-MRDE is planning an additional demonstration face deploying an in-web shearer to extract a very wide web. The new face will use an Anderson Strathclyde shearer equipped with 48 in. wide drums. The very wide web will be extracted with a PFF exemption to 9 ft, 1 in.

Roof support, in the form of four-legged chocks, will be supplied by Gullick-Dobson with a capacity of 300 tons. The chocks will be equipped with 40 in. of forepoling providing a 4 ton set and an 8 ton yield load at the face with forepoling extended to within 14 in. of the face.

Shearer loading tests, with the 48 in. drums, were conducted at MRDE's surface test facilities in Swadlincote beginning in December of 1977. The face equipped as shown in Figure 15 is planned to be placed in operation at a colliery in the North Derbyshire area in 1979.

Increased productivity was not the only reported benefit derived from mining the 1-m web at Holditch and Hem Heath. It was claimed that all these faces experienced better roof control, a reduction in accidents due to roof falls and less dust with the wider web (10). Better roof control was credited to less stress cycling of the roof strata which, when combined with the need for immediate roof coverage when taking a wide web, resulted in a reduction in roof fall related accidents.

Airborne dust was also reduced when mining the wider web, because the increased tonnage of coal was being cut from deep in the web out of the ventilating air stream and the drum speeds of 20 to 30 rpm were relatively low.

The results at Hem Heath and Holditch were achieved with basically standard equipment modified to accommodate the 1-m

TABLE 9. - Equipment specifications and production statistics - Florence Colliery

Equipment specifications:	
Shearer	
Type	AM in-web
Horsepower	270
Drum type	3 start spiral vane
Diameter and width	48 by 48 in.
Drum speed	38 rpm
Supports	
Type	Gullick-Dobson 5/200T
Ram stroke	42 in.
AFC	
Type	18 mm twin strand
Speed	169
Horsepower	120
Production statistics retreat operation:	
Web depth (in.)	33.5
Output to date (tons)	77,745
Production per week	5000
Shears per week	38.8
Tons per shear	129

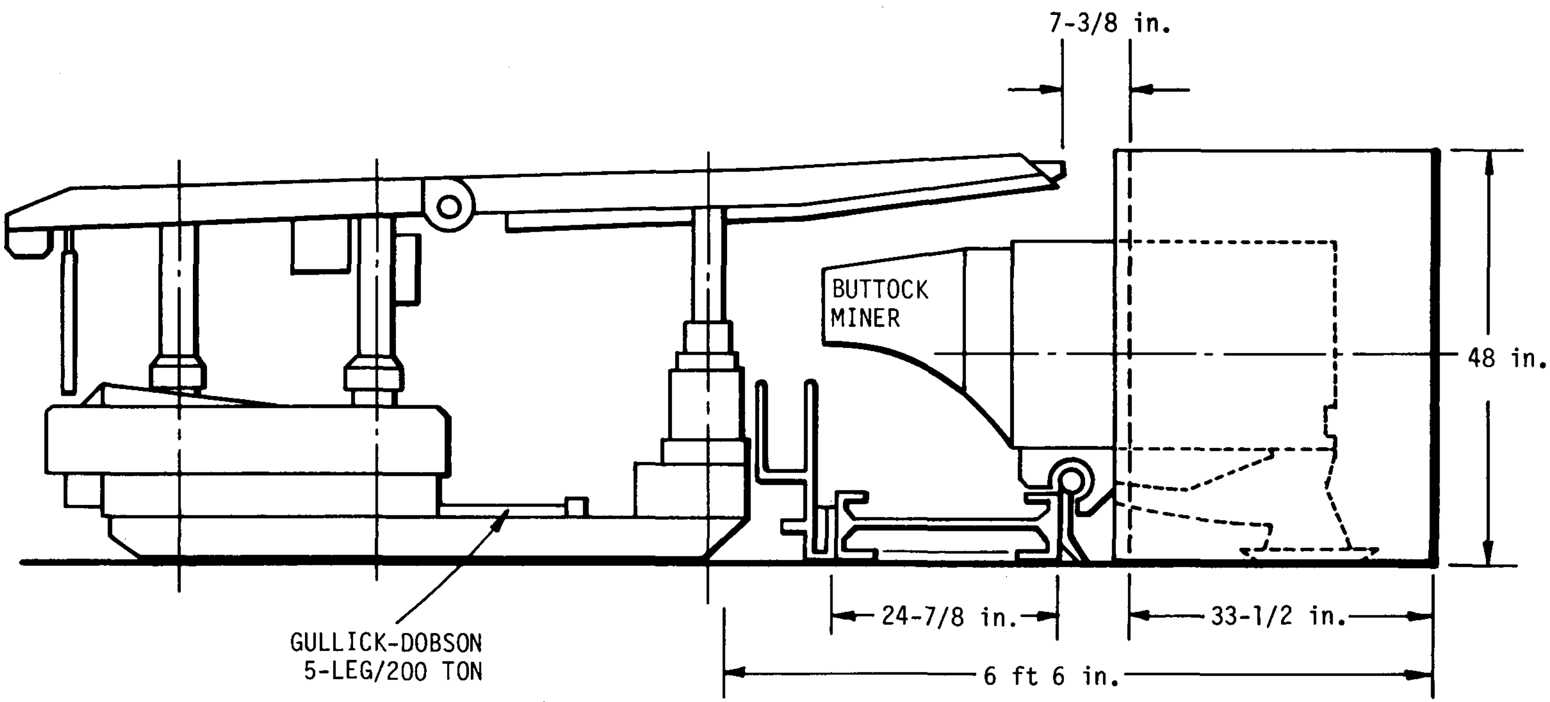


FIGURE 14. - Equipment arrangement Buttock shearer face - Florence Colliery.

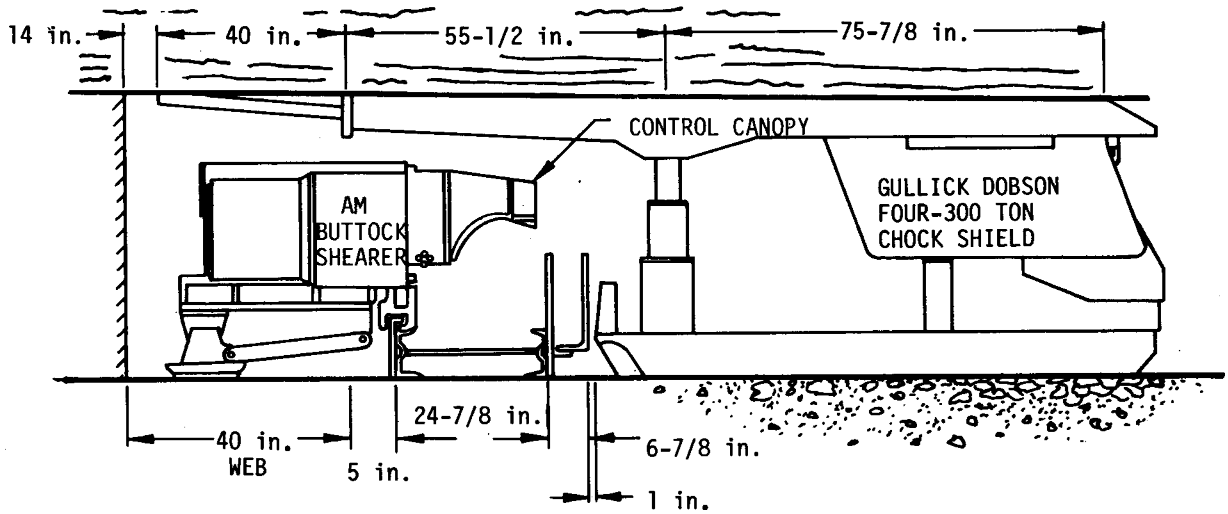


FIGURE 15. - Equipment arrangement - very wide web (48 in.) with Buttock shearer.

web. Further improvements in the productivity of wider web faces in the future are expected to be derived from:

a. Still deeper webs - up to 48 in.

b. Increased capacity outby haulage

c. Equipment designed specifically for wider web mining.

At Holditch and Hem Heath, MRDE used the six-leg chock with forepoling extensions to control the roof over the 1-m web. At the Florence Colliery, the five-leg chock with extensions was used. This represents an adaptation of existing equipment for the application as the chock-type support is the NCB standard. Until very recently, the shield-type support had not been used on a British longwall.

In the United States, however, the shield type of support has recently demonstrated superior performance under the different roof strata overlying American coal. In further assessing the benefits of wide web longwalling as applied in the United States, this report compares the relative benefits of both the chock and the shield for varying depths of web under different conditions in roof strata.

3. IMPACT ON EQUIPMENT

In this section of the report, the impact of wider webs on equipment is limited to its effect on the mechanized roof supports, the shearer cutter/loader and the armoured face conveyor. In Section 6, which covers the economics of wide web mining, the requirements for outby haulage, in terms of additional capacity and cost, are considered. The impact on entry development is taken into account in Section 7 under mine planning.

To quantify the effect that wider webs would have on the supports, shearer and conveyor, it was necessary to make the following principal determinations:

a. What was the maximum web depth that could be considered within the scope of this study?

b. How would the roof strata react? What would be the expected change in roof loading on the supports (capacity and distribution of roof loads)?

c. What support types would be best able to accommodate this roof loading pattern and what modifications or special features would be required?

d. What impact would wider webs have on the shearer - particularly with regard to increased power requirements?

The purpose of this study was to determine what benefits could be realized from mining wider webs with conventional longwall equipment and methods. The study was to determine if wide web mining was cost effective and what impact it would have on safety. Equipment that could be considered was limited to conventional hardware with modifications or special features necessary to accommodate the deeper web.

Early in the study it was determined that the width of drum that state-of-the-art shearers could accommodate would limit the depth of web to be considered. The maximum drum width that contemporary 300 kW shearers could mount was 48 in. This drum width would apply to the Eickhoff 300 and the Anderson Stratchclyde AM 500 double ended machines with modified ranging arms and shearer mountings.

So as to project the near future possibilities in shearer capability, the study includes consideration of the new but untried Mining Supplies 310 kW double ended shearer. This machine, in the opinion of the manufacturer, will be capable of mounting 60 in. wide drums because of its heavier ranging arms and wider supporting base.

To quantify the expected change in total roof load as a shift in load distribution on the supports, the roof strata was modeled (as a semifloating block of strata) according to the theory presented by Wilson (21). This model was used primarily to account for the anticipated shift in roof loading towards the face as the web depth increased.

Total support capacity, however, was calculated using Wade's formulation (22), which accounts for the influence of wider webs, depth of influencing strata and conditions of roof overhang into the gob. This formula was applied to roof conditions that the report defines as best, intermediate and worst conditions of loading depending on the type of roof strata column above the seam.

Wilson's model was used to reflect the comparative stability of different support types only. Wade's formula was used to project the required capacity of roof supports for different web depths under different roof conditions. This report does not suggest that the two formulas are compatible nor were they required to be for the purposes of this study. The absolute numerical values arrived at in the stability calculations are comparative in nature only. These values only reflect the differences in basic geometry of the various support types as these types are able to react to the shift in roof load.

Support capacities projected from Wade's formula, however, are more meaningful inasmuch as the cost of support systems in the economics of Section 6 reflect these capacities. The use of both Wilson's and Wade's model and formula are illustrated in Appendix C.

The report additionally identifies the modifications to the shearer that are required to mount and to operate with the wider drums. The cost of these modifications are included in the work of Section 6.

Finally, this section of the report analyzes the shearer power requirements for mining the deeper cut. This analysis is based on previously referenced work on tests to determine the specific energy of cutting coal with shearer drums. The productivity calculations of Section 4 take into account the change in specific energy of mining as the web depth increases.

3.1 Roof Support Systems

The key to the success of any longwall operation is roof control for which there are various types of basic supports available. The different types include the four-, five-, and six-leg chocks, the two- and four-leg shields, the chock-shield and various types of frame supports. Although the shields are presently favored in this country, no one type is best suited for all conditions. This would also be true when controlling the roof over a wider web. For this reason all the support types were evaluated, with two exceptions: the frame and the chock-shield.

The frame supports were not considered since they are used almost exclusively on very narrow web plow faces. The chock-shield was not considered "separately" since it is essentially a four-leg chock with lemniscate linkage and full gob shielding. Therefore, the stability calculations, floor loading and the capacity requirements of the four-leg chock would generally apply also to the chock-shield whereas lateral stability (to resist loading parallel to the canopy) and manway production are about equal to the four-leg shield.

The quantitative comparative analysis that reflects the relative performance of the various support types was limited to factors directly related to the increase in web depth. Other considerations normally applied in the selection of a proper support are not included in this report for simplicity. Such considerations would include closed-to-open height ratio and cross section of open area for ventilation.

The factors that were considered to be directly associated with the wider webs were: the additional weight of roof to be carried, the shift in roof load towards the face and the greater distance of unsupported roof exposed in front of the support canopy when the cut is taken.

The additional roof weight and the shift in this weight toward the face could result in support instability and will result in a shift in floor loading towards the toe of the base. Peak loads near the toe could result in the support digging into a soft floor.

The greater span of unsupported roof between the support canopy and the coal face will increase the potential for local roof falls. To prevent this from occurring requires that the roof be supported immediately after the shearer has taken the cut (immediate forward support). Immediate forward support was considered to be required for web depths in excess of 30 in.

For supports set one web back, wider webs will require longer roof canopies. This longer canopy increases the distance from the forward-most hydraulic prop to the front end of the canopy (PFF). This adversely affects the efficient transfer of supporting load from the props to the forward tip of the canopy (tip loading).

The comparative results presented in the remainder of this section rate the various support types to various depths of web for:

1. Support stability - the potential for a support to tip towards the face as the web deepens

2. Floor loading - the peak loading pressure and its location relative to the toe of the base

3. Immediate forward support - the percent of exposed roof effectively supported immediately after the shearer takes the cut

4. Tip loading - the efficient transfer of supporting load to the roof at the forward end of the canopy.

Total support capacity was not considered as a significant factor that would influence the selection of a support type and therefore does not appear in the selection process. The impact of support capacity is, however, reflected in the economics of Section 5, where increased capacity appears in the cost of the support systems.

Before the various types of supports could be evaluated, consideration had to be given to how each particular support could be applied to wider web longwalling. For example, the two-leg shields and four-leg chocks were set one web back to allow for adequate manway. The addition of special features was, for the most part, restricted to presently available hardware such as the forward canopy extension (forepoling) applied to the five and six-leg chocks or changes that represented modifications to existing designs.

Two supports in particular fall into the latter category. The four-leg shield set conventionally would be modified to increase the distance between the forward and rear sets of legs. This modification would be required to provide for adequate manway between the supports.

The second support type that represents a more extensive modification is the four-leg shield with full sliding canopy.

This support, in face, is not available today but was classified as a modification falling within the scope of this study. This support is discussed in some detail in Appendix D.

In summary, the supports comparatively evaluated in the following subsections are:

1. The four-leg chock with solid roof canopy set one web back - this analysis could apply equally to the chock-shield

2. The six-leg chock with hinged forward canopy and fore-poling operated with conventional set back - this analysis would apply equally to the five-leg chock

3. The two-leg shield operated one web back

4. The four-leg shield set one web back - this support could be modified to increase the distance between forward and rear hydraulic legs. As modified, this support could be set conventionally as discussed at the conclusion of this section.

5. A four-leg shield set conventionally with full sliding canopy - see Appendix D.

The various supports are rated in performance against the four criteria discussed previously. The weighted results are tabulated in Table 13. Of the strictly conventional supports, the four-leg shield and six (or five) leg chock scored the highest. The full sliding canopy added to either the two-leg shield or four-leg shield scored the highest as a special purpose wide web support.

3.1.1 Comparative Stability Analysis

In analyzing the various supports for their inherent stability, typical dimensions, as shown in the figures, were taken from manufacturer's catalogs. These same dimensions were used to reflect the shift in floor load distribution in the section on floor loading.

The figure for each support type considered show the shift in loading on the front, rear and - where applicable - the center sets of hydraulic legs. As the web depth increases, generally the load increases on the front legs and decreases on the rear legs. For purposes of this study, if the resultant roof load acts along a line that intersects the floor in front of the base of the toe, the support is considered to be unstable. The conventional two-leg shield set one web back became unstable, by this definition, for all web depths in excess of 35 in. The graphs also show the increase in total support capacity as the web depth increases. These capacities are not necessarily related to the capacities predicted by Wade's formula in subsection 3.1.3.

As mentioned earlier, the maximum web depth considered in the stability analysis was 60 in.

The Four-Leg Chock

The average dimensions for, and the position of, the four-leg chock as analyzed is shown in Figure 16(a). The support is set one web back to provide for adequate manway in front of the forward props. In this position, the chock also provides for immediate forward support of the exposed roof. Figure 16(b) shows the increase in support capacity with increasing web depth, and the shift in load to the front set of cylinders. Because this support has a relatively short canopy and is set one web back, the load shifts rapidly to the front supports.

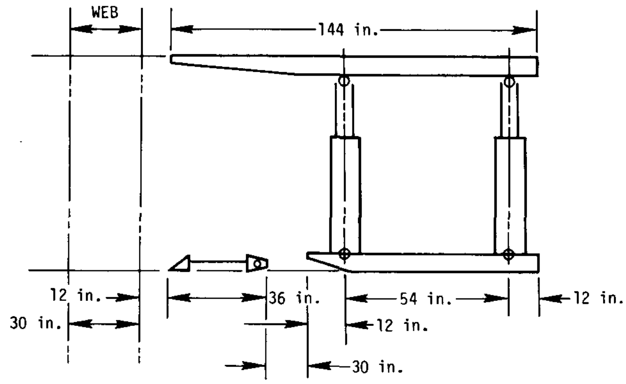
The Six-Leg Chock

Typical dimensions and the location of the six-leg chock is shown in Figure 17(a). This support provides for adequate manway behind the front row of props. The hydraulics are situated between the back rows of cylinders under the main canopy. For this reason, the support is not set one web back which considerably improves the roof load distribution as shown in Figure 17(b) as compared to the four-leg chock of Figure 16. This support is equipped with hydraulically actuated sliding extensions (not shown) for immediate forward support. This arrangement is typical of the NCB support arrangement in the British 1-m web faces.

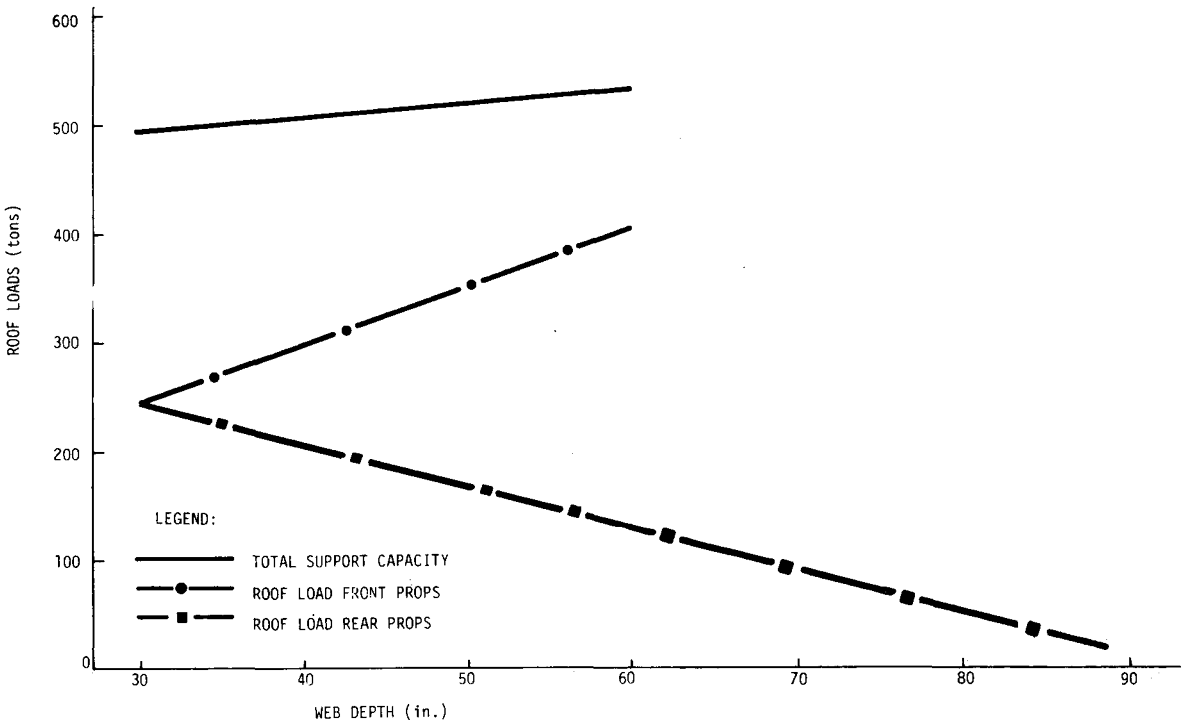
The improvement in load distribution is also due to the longer length of canopy. The percent increase in total roof load, with increasing web, is less than for the shorter canopy of the four-leg chock. Therefore, the shift in load is not as dramatic.

The Four-Leg Shield

Typical dimensions for the four-leg shield are shown in Figure 18(a). Like the four-leg chock, this shield is set one web back to provide for adequate manway and immediate forward support. This shift in roof load to the front supports, with increasing web depth (as shown in Figure 18(b)) is similar to the four-leg chock. However, the shift is not as rapid because the shield canopy is longer than that of the chock. At the

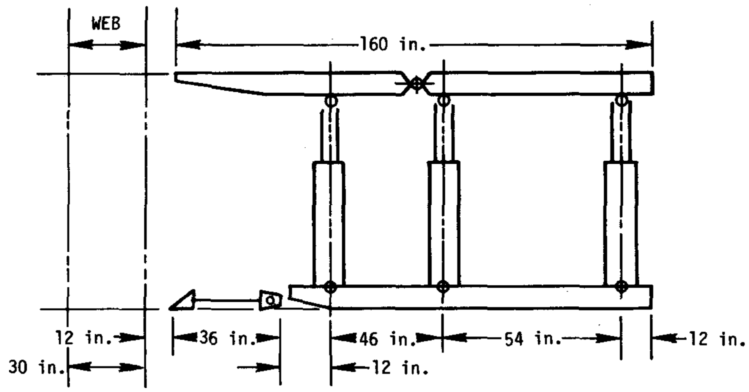


(a) Typical Dimensions Set One Web Back
30 in. Web Depth

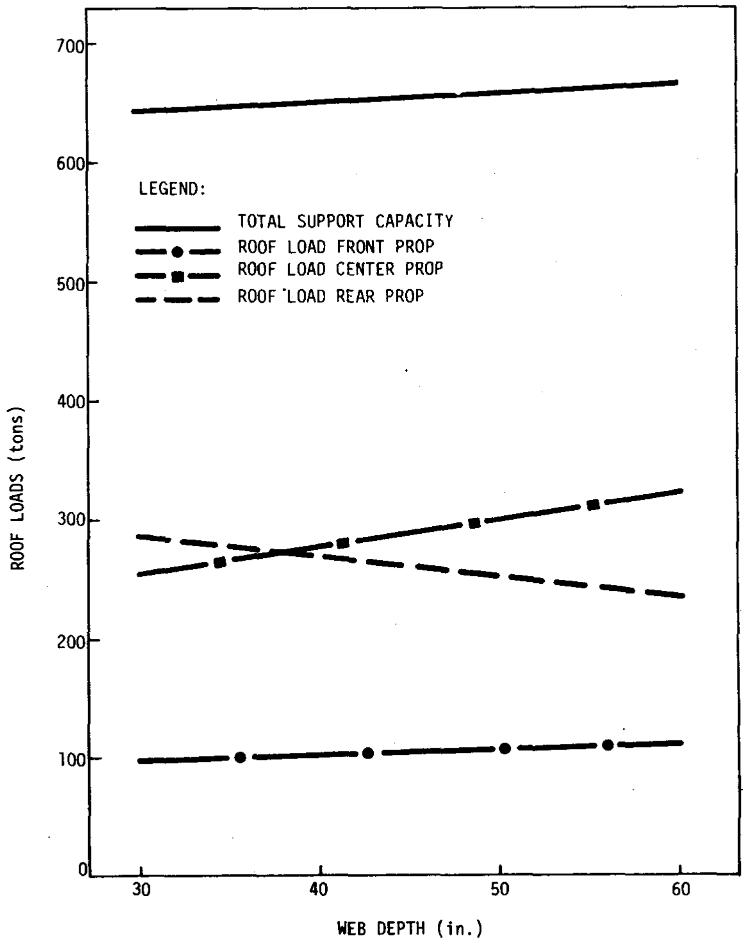


(b) Capacity and Shift in Roof Load
with Increasing Web Depth

FIGURE 16. - Application of the four-leg chock for wide web roof control.

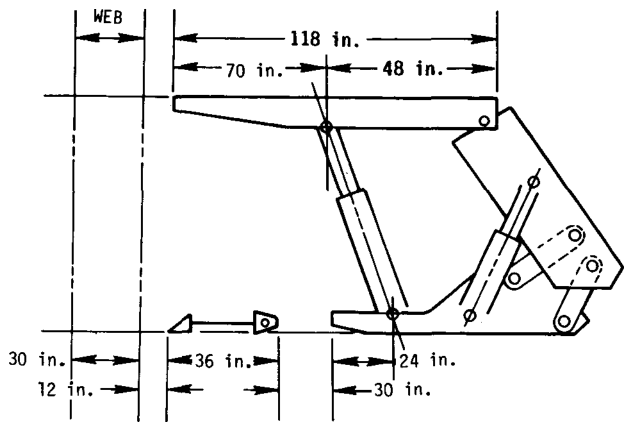


(a) Typical Dimensions Set One Web Back
30 in. Web Depth

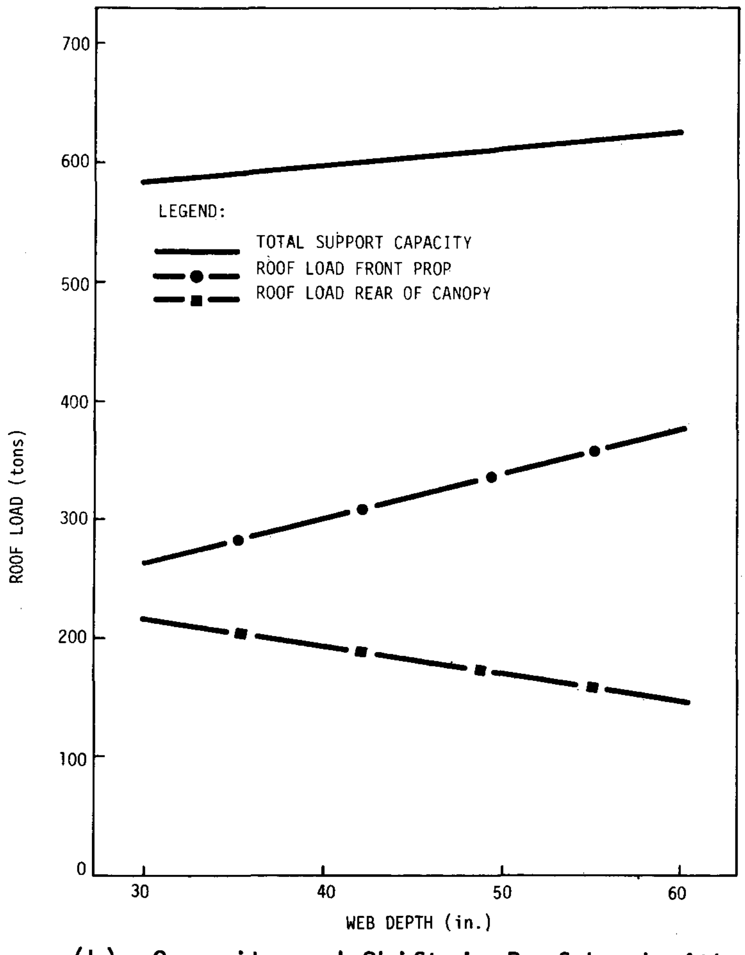


(b) Capacity and Shift in Roof Load with
Increasing Web Depth

FIGURE 17. - Application of the six-leg chock for
wide web roof control.



(a) Typical Dimensions Set One Web Back 30 in. Web Depth



(b) Capacity and Shift in Roof Load with Increasing Web Depth

FIGURE 18. - Application of the four-leg shield for wide web roof control.

60-in. web depth, the front support loading is 430 tons on the four-leg chock, but only 380 tons on the shield. There is a slight decrease in total roof load with the four-leg chock reflecting the 10 in. shorter canopy. Modifications to this support are discussed at the conclusion of this section.

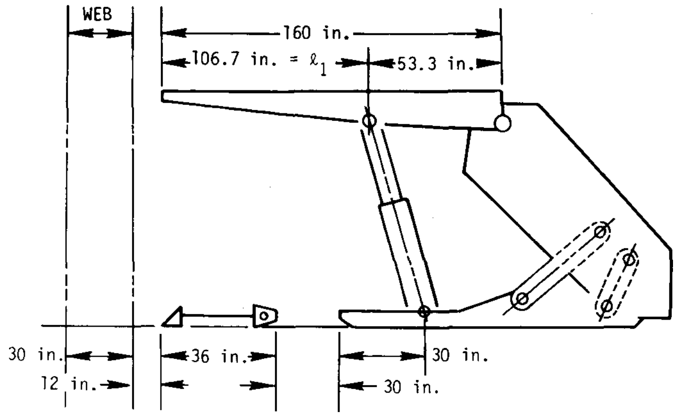
The Two-Leg Shield

A typical length of canopy on a two-leg shield is 110 in. Like the four-leg shield, this type of support would be set one web back for manway and immediate forward support. With this length of canopy, this support became unstable at a web depth of about 35 in. As typically configured, the short canopy is carried quite far forward of the toe of the base with the base set back one web. When the canopy is extended forward to cover the wider web, the resultant roof load moves forward of the toe of the base, resulting in an overturning movement. For purposes of this study, this represents a condition of instability. The results of the analysis for the two-leg shield with conventional canopy lengths was not illustrated for this reason.

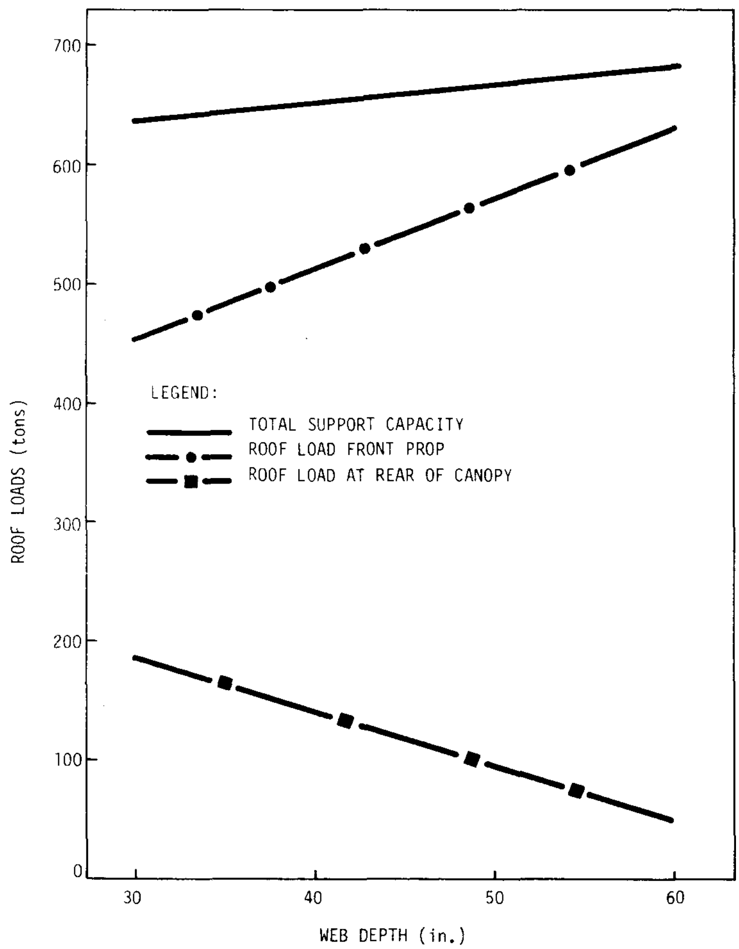
The two-leg shield that is illustrated in Figure 19(a) is modified to mount a canopy whose length was increased to 160 in. This length of canopy was chosen because at any length less than 160 in. the support becomes relatively unstable at the 60-in. web depth. The stability characteristics are shown in Figure 19(b). Although the increase in canopy length improves stability, the roof load distribution is still relatively poor. As shown, 90 percent of the roof load is carried on the front supports at the 60-in. web depth. Low support densities at the gob end of the canopy could result in some difficulty in breaking the immediate roof.

Finally, Figure 20(a) illustrates a specially modified two-leg shield. The entire roof canopy is mounted on slides and is pushed forward by a hydraulic cylinder. The canopy slides forward independent of the main support frame (consisting of base, gob shield and support cylinders). The frame is later advanced off the conveyor pans in the usual manner. As configured, this support provides for immediate forward support without the base being set one web back. This entirely changes the roof loading distributions as shown in Figure 20(b). With increasing web depth, the loading shifts to the rear supports rather than to the front supports.

This type of canopy could be applied to any of the previously considered supports. It was evaluated as applied to the two-leg shield since this support represented a worse case condition. The features of the full sliding canopy as applicable to the four-leg shield is discussed in more detail in Appendix D and appears as Support F in Table 13.

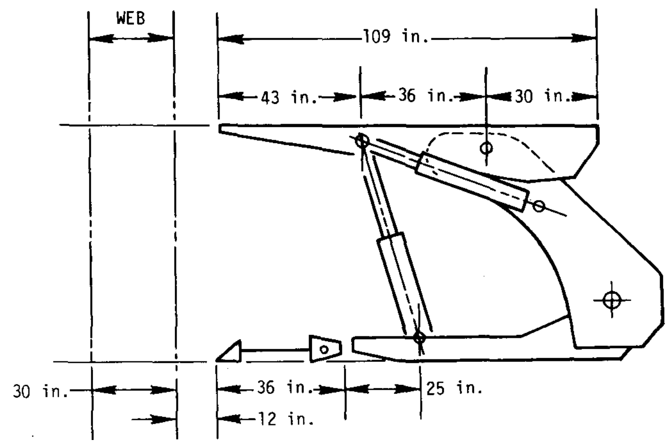


(a) Typical Dimensions Conventional Set Back 30 in. Web Depth

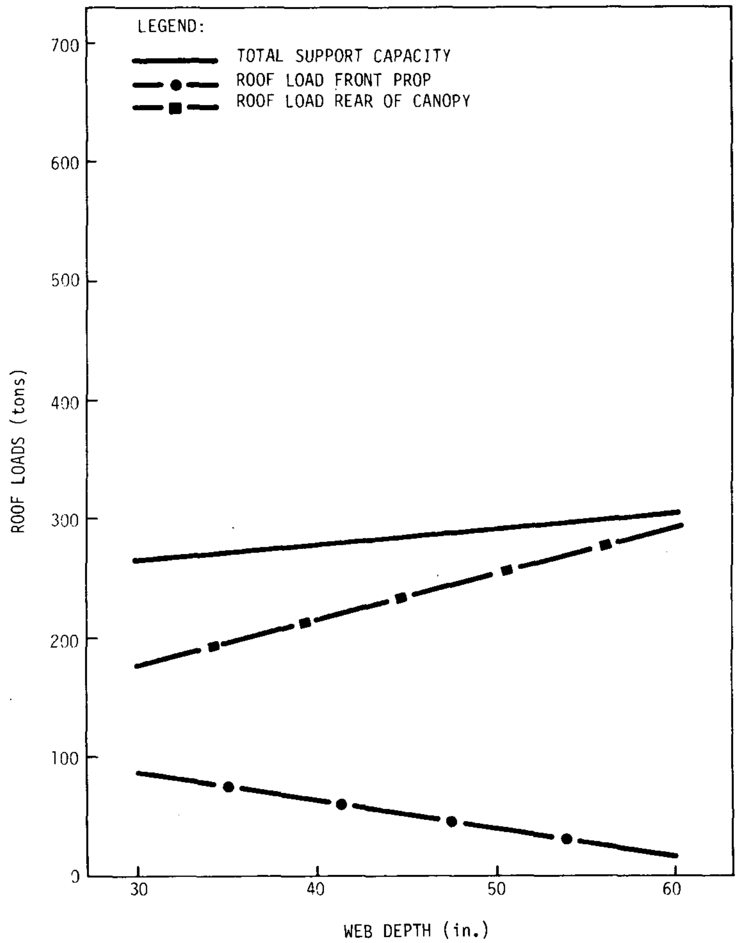


(b) Capacity and Shift in Roof Load with Increasing Web Depth

FIGURE 19. - Application of the two-leg shield for wide web roof control.



(a) Typical Dimensions Conventional Set Back 30 in. Web Depth



(b) Capacity and Shift in Roof Load with Increasing Web Depth

FIGURE 20. - Special two-leg shield with full sliding canopy.

3.1.2 Comparative Floor Loading Analysis

The increase in roof load and the shift in this load towards the base, developed in the previous section, will be reflected in the floor loading pressure distribution patterns. Although a support may not be unstable, it may transmit to the floor high peak bearing pressures particularly at the toe of the base. High pressures at the toe can result in the support digging into a soft floor, making it very difficult to advance the support. Therefore, in evaluating the various support types, it was considered necessary to compare both the peak pressures and the location of this maximum floor loading relative to the toe of the base.

To provide both criteria of comparison, the average bearing pressure was calculated for each support type at each web depth. This pressure was based on the total roof load at the particular web depth and typical base dimensions for the type of support. These average pressures were then redistributed to reflect the unequal roof loadings at the front and rear of the roof canopies. The floor pressure distributions were developed for the 30 and 60 in. web depths only, as that was sufficient to show the shift in the pressure pattern and the maximum peak pressures for each support. An example analysis is shown in Appendix E for the four-leg chock.

The peak pressures as well as the forward shift in peak pressures towards the toe of the base is maximum for supports set one web back. This includes the four-leg chock and the two and four-leg conventional shields. The floor pressure distribution patterns for these supports are shown in Figure 21(a,b,c). The worst case condition was under the modified two-leg shield of Figure 21(c) where peak pressure was 31.6 tons/ft^2 (440 lb/in.^{2}) at a 60-in. web depth. The best of the one web back supports was the conventional four-leg shield with a maximum peak pressure of 16.8 tons/ft^2 (233 lb/in.^{2}).

The six-leg conventional chock was not set one web back. The floor pressure distribution pattern for this support is shown in Figure 22(a). Peak pressures occurred well back from the toe of the base and are less than peak pressures experienced on the one web back supports.

The special two-leg shield, with full sliding canopy, exhibits the best roof load distribution (relative stability) and floor loading patterns of all of the supports analyzed. Deployment of this type of support does require the development of the sliding canopy feature.

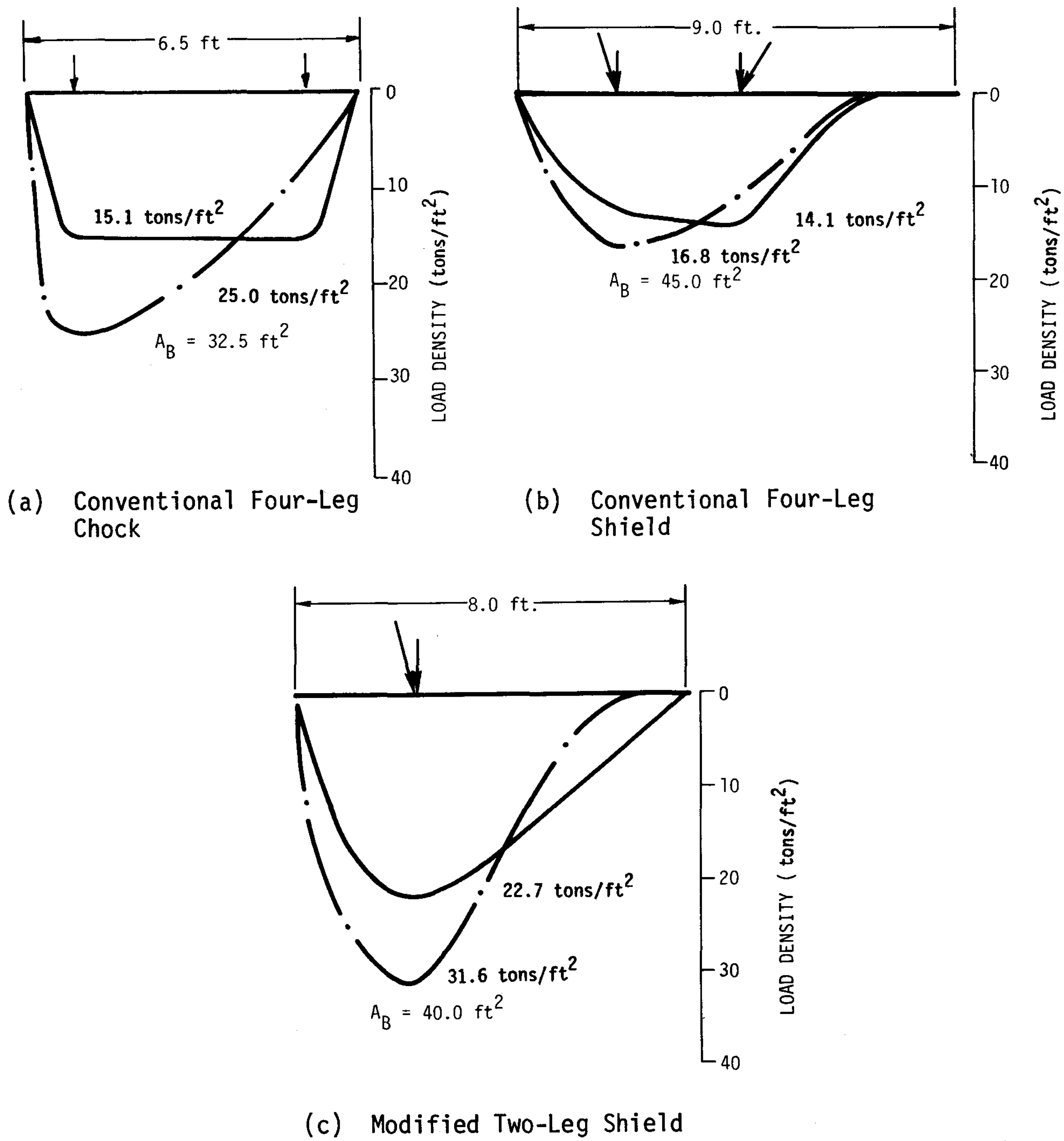
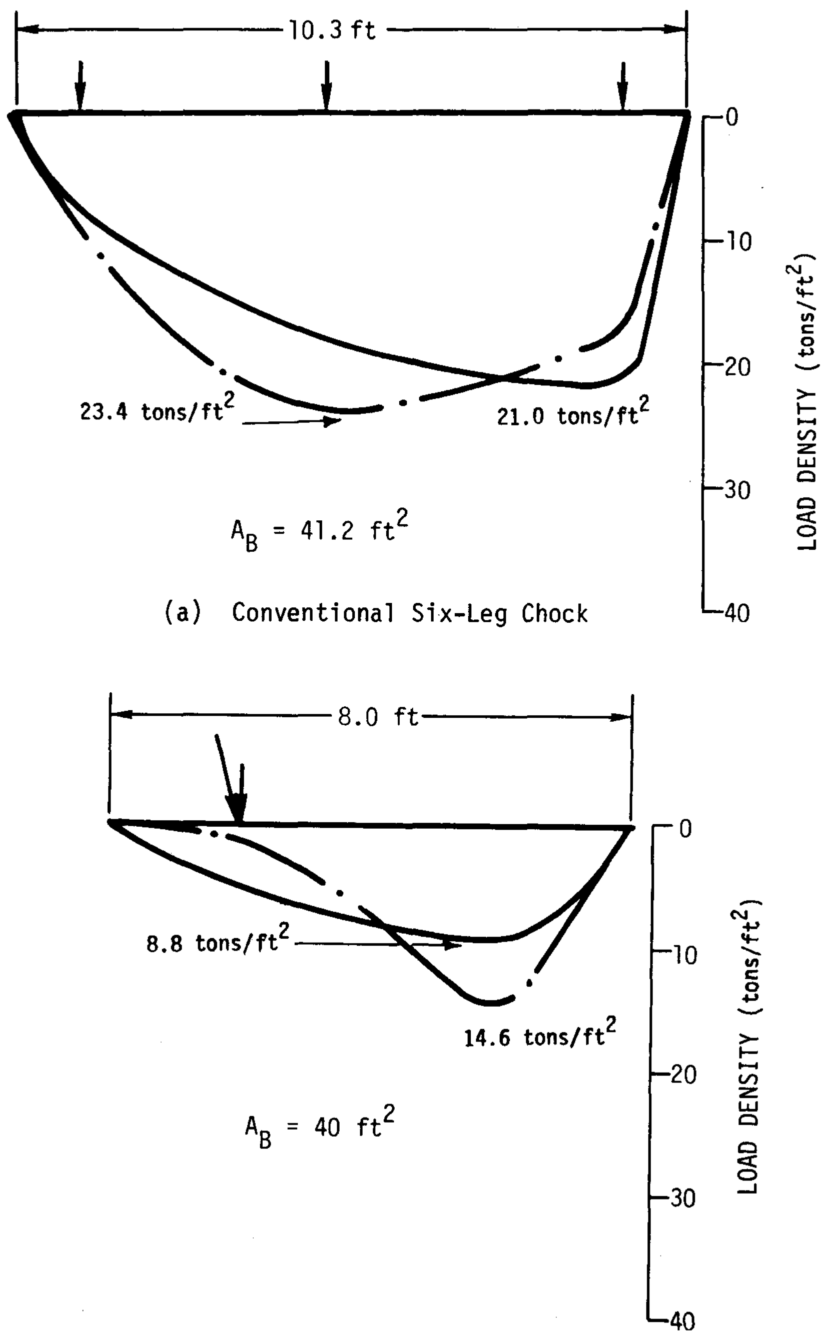


FIGURE 21. - Floor load distributions for conventional supports set one web back.



(b) Special Two-Leg Shield - Full Sliding Canopy

FIGURE 22. - Floor load distribution for support not set one web back.

Of the strictly conventional supports, the six-leg chock and four-leg shield have the best stability and floor loading characteristics. However, other performance parameters, including tip loading and control of the immediate roof, remain to be analyzed.

3.1.3 Required Capacity of Supports

In the previous subsection, the roof was modeled as an unstable block according to Wilson. Increases in web depth resulted in a rapid shift in loading towards the face. Wilson's model, therefore, represented a worst-case condition with regard to support stability and floor loading.

In this subsection the support capacity requirements will be established by Wade's formula. Required support capacities will be established for different web depths, roof conditions and support types. This will provide a measure of the full range of support requirements and a determination as to which factor most influences capacity: roof type, web depth or support type (canopy dimensions).

Wade's formula establishes the required load density (tons/ft^2) of support capacity as:

$$\text{Load density (tons/ft}^2\text{)} = \left(1 + \sum_{n=1}^{5} C_n\right) \times 4\ \delta h \quad (\underline{22})$$

The term 4 δh accounts for the weight of roof (tons per square foot) that the support is theoretically required to carry, where δ is the specific weight (tons per cubic foot) of the strata and h is the height of coal extraction in feet. In this formula the influencing height of roof is four times the seam height (h). This represents a common method of estimating support capacity requirements on United States longwall faces.

However, Wade's formula goes further in estimating the effect of depth of web, caving roof overhang and location of the main roof relative to the top of the seam. These influencing factors are accounted for in the term:

$$\sum_{n=1}^{5} C_n$$

This term is the summation of magnification coefficients (five in total number) that increase the loading on the support. For example, a deeper web, greater overhang and main roof strata close to the seam would all increase the required capacity of a support. These multiplying factors are identified in Appendix C along with a sample calculation illustrating the use of the formula.

Once the support densities (tons per square foot) have been determined for variations in roof type and web depth, then the total capacity of required support is:

$$\text{Capacity (tons)} = \text{support density} \left(\frac{\text{tons}}{\text{ft}^2}\right) \times \text{canopy area (ft}^2\text{)}$$

Canopy area in this formula is the total area of roof supported, which is the length of canopy and the unsupported span in front of the canopy times the center-to-center spacing of the support. Capacity thus determined becomes a function of roof type, web depth and the type of support since canopy area varies with support type.

The results of these calculations for roof conditions identified as best, intermediate and most difficult appear in Tables 10, 11 and 12. The factor that establishes the degree of roof difficulty is the distance from the top of the coal seam to the bottom of the competent noncaving strata. This varied from 26 ft for the best conditions to 19.5 ft for the most demanding conditions as shown in the tables.

The required capacities were calculated for four basic support types: the four and six-leg chock (also applicable to the five-leg chock) and the two and four-leg shields. The web depths varied from 30 in. (representing the standard web) to 60 in. which is the maximum web depth established for this study. The dimensions shown in the tables for each support are length of the support canopy (L), center-to-center spacing of the supports (W) and the resulting calculated area (A). It should be noted that the two-leg shield has a standard canopy length and is not, therefore, the shield of Figure 19 which has an extra long canopy.

The tables show that support capacity is influenced most significantly by roof type and support type (canopy dimensions) and least significantly by depth of web. Depth of web, therefore, should not have a measurable impact on the cost of the longwall support system in terms of increased support capacity. This, of course, does not account for the cost of special features and modifications that may be required for wider webs.

TABLE 10. - Basic support requirements with variation in web depth - best roof conditions

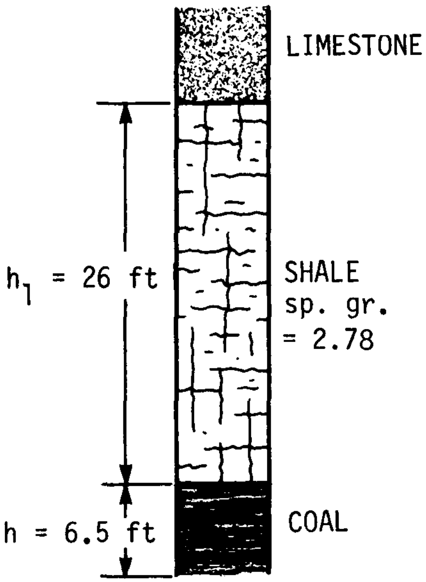
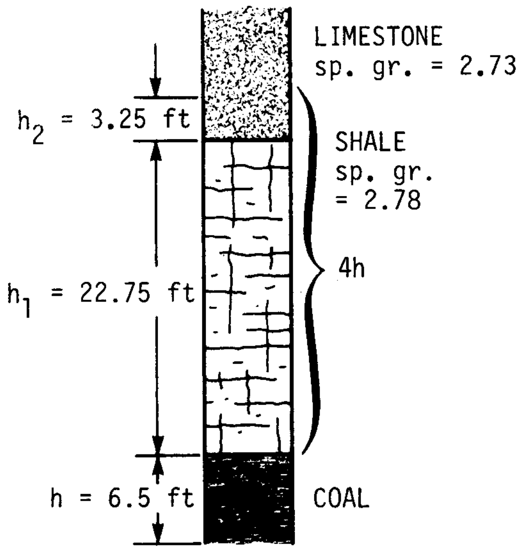
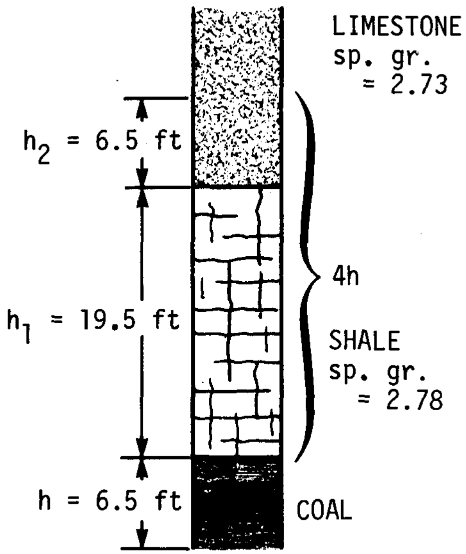
Roof type	Support type	Web depth (in.)	Calculated tons/ft^2	Calculated capacity (tons)
	Four-leg chock	30	3.86	232
	L = 144 in.	40	4.02	241
	W = 60 in.	48	4.14	248
	A = 60 ft^2	60	4.33	260
	Six-leg chock	30	3.81	203
	L = 160 in.	40	3.95	211
	W = 48 in.	48	4.06	217
	A = 53.3 ft^2	60	4.23	226
	Two-leg shield	30	4.01	184
	L = 110 in.	40	4.21	193
	W = 60 in.	48	4.38	201
	A = 45.8 ft^2	60	4.62	212
	Four-leg shield	30	3.83	246
	L = 154 in.	40	3.98	256
	W = 60 in.	48	4.09	263
	A = 64.2 ft^2	60	4.27	274

TABLE 11. - Basic support requirements with variation in web depth - intermediate roof conditions



Roof type	Support type	Web depth (in.)	Calculated tons/ft^2	Calculated capacity (tons)
	Four-leg chock	30	9.24	555
	L = 144 in.	40	9.40	564
	W = 60 in.	48	9.53	572
	A = 60 ft^2	60	9.71	583
	Six-leg chock	30	9.18	490
	L = 160 in.	40	9.32	497
	W = 48 in.	48	9.43	503
	A = 53.3 ft^2	60	9.60	512
	Two-leg shield	30	9.39	430
	L = 110 in.	40	9.59	439
	W = 60 in.	48	9.76	447
	A = 45.8 ft^2	60	10.00	458
	Four-leg shield	30	9.21	591
	L = 154 in.	40	9.36	601
	W = 60 in.	48	9.48	608
	A = 64.2 ft^2	60	9.65	620

TABLE 12. - Basic support requirements with variation in web depth - worst roof conditions

Roof type	Support type	Web depth (in.)	Calculated tons/ft^2	Calculated capacity (tons)
	Four-leg chock	30	11.42	685
	L = 144 in.	40	11.58	695
	W = 60 in.	48	11.70	702
	A = 60 ft^2	60	11.89	713
	Six-leg chock	30	11.41	608
	L = 160 in.	40	11.55	616
	W = 48 in.	48	11.66	622
	A = 53.3 ft^2	60	11.83	631
	Two-leg shield	30	11.56	530
	L = 110 in.	40	11.77	539
	W = 60 in.	48	11.93	546
	A = 45.8 ft^2	60	12.18	558
	Four-leg shield	30	11.39	731
	L = 154 in.	40	11.54	741
	W = 60 in.	48	11.65	748
	A = 64.2 ft^2	60	11.83	759

Although there is no claimed compatibility between Wilson's roof model and Wade's formula, it is interesting to note that the calculated capacities for the intermediate roof conditions of Table 11 are fairly consistent with the capacities of Figures 16 through 20 arrived at using Wilson's roof model.

3.1.4 Control of the Roof at the Face

As the shearer progresses along the face it exposes a section of roof whose span is related to the depth of web. Immediately behind the shearer the newly exposed roof is unsupported until the chocks or shields are advanced forward and reset.

The time required to bring the supports forward depends on whether the supports are set conventionally or one web back. With conventional supports, the conveyor must be pushed forward before the supports can be advanced. This results in a delay in which this area of roof is left unsupported.

One-web-back supports can be brought forward before the conveyor is advanced. Therefore, the area of unsupported roof is substantially reduced as is the period of time that the roof is unsupported. The roof problem that can be experienced are related to both the area or span of roof exposed and the time required to bring the support forward.

Localized roof falls and/or a general deterioration of the local roof can seriously reduce the effect of the main roof canopy. Cavities created by localized falls will result in poor contact between the canopy and roof. Deterioration of the immediate roof can result in a poor transfer of supporting load to the upper heavier strata. This will allow the upper strata to converge more rapidly, substantially increasing the load on the supports.

In evaluating the various methods that can be employed to provide roof support behind the shearer, it is assumed that the depth of web and the conditions of the roof require immediate forward support (IFS). Under those conditions of evaluation, the two criteria of importance are:

a. The percent of exposed roof that is covered by the IFS

b. The pattern of roof contact that the IFS feature provides.

For example, Figure 10 shows the supports used on the 1-m wide web longwall face at the Holditch Colliery in the United Kingdom. These supports were fitted with forward sliding extensions for

IFS. However, these extensions only covered a percent of the exposed roof and actual contact with the roof only occurred at the very tip of the extension.

Alternatively, shields set one web back provide for more effective IFS. Since the roof canopies are set skin to skin, 80 percent IFS is accomplished when the supports are advanced and set. Secondly, contact between canopy and roof is ideally along the entire length and width of the canopy. However, one web back IFS has some disadvantages. The supports must be retracted from the roof immediately behind the shearer. Loss of roof support at this particular time can have a detrimental effect, as observed by J.J. Bates (12).

The one-web-back type of support has an additional disadvantage that is particularly related to an increase in the depth of web. In addition to adequate control of the immediately exposed roof discussed above, the roof support system must also provide for adequate support at the face *after the main supports are advanced and set.* In this report, this requirement at the face is referred to as *tip loading* and is related to the previously discussed PFF regulation imposed on the British mining industry.

The PFF distance is the maximum distance that the first line of hydraulic supports can be from the face and therefore from the forward-most tip of the roof canopy. This maximum distance is regulated by law in the United Kingdom so as to insure adequate support up close to the face. As dicsussed in subsection 2.4.1, this allowable distance can vary with permission from the Inspectorate depending on particular roof conditions and design of equipment. This maximum distance is regulated for two fundamental reasons, assuming a one-piece canopy (not hinged as in the case of the five or six-leg chock). The one-piece canopy reacts to the force system of roof loads and reactions from the hydraulic cylinders like a continuous beam. The front section of this beam that extends out over the conveyor acts as a cantilever. The greater the length of this cantilever (PFF) the less efficient will be the transfer of supporting load to the tip and the greater will be the potential for poor roof contact of the canopy at the tip. The reasons for this are explained in some detail in Irresberger's paper on longwall roof control (13).

In the two subsections that follow, the various types of supports are evaluated for their ability to:

a. Provide for immediate support of the exposed roof, considered to be of prime importance when extracting a deep web. The criteria to be used in assessing the various supports and special features are:

- Percent of exposed roof covered by the immediate forward support feature
- The pattern of contact between the forward support canopy and the roof

b. Provide for effective tip loading at the face when the supports are advanced and reset. The criteria of evaluation is: The distance between the forward tip of the main support canopy and the first line of hydraulic supports.

In evaluating effective tip loading at the face, it is assumed that the roof canopies for all supports considered are of equal rigidity.

Finally, the conventional and special supports that are evaluated in the following two subsections and summarized at the end of this section are:

a. Conventional Supports:

- The four-leg chock set one web back. The features of this support would also apply to the chock-shield.
- The six-leg chock (or five-leg chock) set conventionally with sliding forward canopy extensions. A particular feature of this support is the hinged forward section of main canopy.
- The modified (longer canopy) two-leg shield set one web back.
- The conventional four-leg shield set one web back.

b. Nonconventional Supports

- Special four-leg shield set conventionally with forward sliding extendable canopy. This support would be lengthened to provide adequate manway between the front and rear sets of hydraulic supports.

- The specially lengthened four-leg shield with the special full sliding canopy feature of Appendix D. This shield would be set conventionally.

Immediate Forward Support (IFS)

There are presently available two techniques that can be used to provide for IFS. The support can be set one web back or the support can be equipped with a forward sliding extension. The latter method is referred to as "forepoling." Although either technique may be applied, both also have disadvantages which become increasingly more of a factor as the web depth increases. A third technique that will be considered is the full sliding roof canopy, which is an innovative technique that is not presently used on longwall supports. The development of such a feature, however, could allow the mining of very wide webs under more difficult roof conditions than present techniques allow.

In this subsection the advantages and disadvantages of the three techniques will be discussed.

One-Web-Back Feature

Longwall supports are set one web back to provide an adequate manway. Operating personnel are situated between the conveyor gob side furniture and the front row of hydraulic props. If this space is made equal to the depth of web, then the support is set one web back. Under these specific conditions the support can provide for immediate coverage of the exposed roof. As the shearer passes, the support is immediately advanced until the base of the support comes up to the back of the conveyor.

The four-leg chock and the two and four-leg shields were positioned in this manner to provide adequate manway. However, as discussed in subsections 3.1.2 and 3.1.3, when the web depth increases, stability and floor loading are adversely affected when supports are set on web back. However, the performance of the conventional four-leg chock, four-leg shield and modified two-leg shield (longer canopy) were considered acceptable as analyzed.

Percent of roof coverage is best with the shield type of support. They are set skin-to-skin for complete protection from roof falls.

The five- or six-leg chocks have typically 3.5 ft wide canopies and are set on 4 ft centers. Roof coverage, as an IFS feature, is therefore 87.5 percent. Four-leg chocks have 4.5 ft wide canopies and are set on 5 ft wide centers, providing roof coverage of 90 percent. The two- and four-leg shields are also set on 5 ft centers with 4.5 ft wide canopies. Although the percent roof coverage by the actual canopy is the same as the four-leg chock, the shields are equipped with side shielding for skin-to-skin contact. This feature prevents any material from dropping out of the roof through the space between the roof canopies - the side shields close the gap. Therefore, the shields are rated higher than the four-leg chocks in the scoring of Table 13.

The second measure of performance for the IFS feature was canopy contact. All of the supports set one web back were rated equally since support is by the main roof canopy set in the normal manner.

Forepoling

Figure 23 shows the forepoling extensions that were fitted to the six-leg chocks for covering the 1 meter web at the Hem Heath Colliery in the United Kingdom. Although the roof canopy was 3.28 ft wide, the extension is only 1.3 ft (15.75 in.) wide. If it is assumed that the supports are set with a 3 in. gap between (typical for the six-leg chock), then the center-to-center spacing would be 3.78 ft. Under these conditions the percent roof coverage is 34.7. All supports fitted with the sliding extension feature were rated poorly for percent roof coverage in the analysis of Table 13.

As also illustrated in Figure 23, the forepoling extension slides out from under the main roof canopy. The extension is then powered into contact with the roof by either leaf springs or hydraulic cylinders. However, roof contact is only at the tip as shown. Fall of roof material can occur and lodge itself in the gap in front of the main roof canopy. This has the potential to allow the immediate roof to deteriorate and to cause some difficulties when advancing the support. As with percent roof coverage, supports fitted with sliding extensions were also rated poorly for canopy-to-roof contact.

The major advantage of forepoling is that, unlike the one web back supports, the sliding extension can be set in place without disturbing the set of the main support canopy. This has the potential to minimize covergence of the lower strata just after the shearer has passed, which is considerable as previously

TABLE 13. - Overall relative performance of various supports for wide web mining

Performance parameter	Weighting factor (W)	A		B		C		D		E		F	
		n	w·n	n	w·n	n	w·n	n	w·n	n	w·n	n	w·n
Immediate forward support	3	2	6	1	3	4	12	4	12	2	6	4	12
Support stability	3	2	6	4	12	1	3	3	9	4	12	4	12
Tip loading at the face	2	1	4	4	8	2	4	2	4	3	6	2	4
Floor loading:													
Distribution	2	2	4	3	6	1	2	3	6	3	6	4	8
Peak pressure	1	2	2	3	3	1	1	2	2	3	3	4	4
Total $\left(\sum^{5} w \cdot n\right)$			22		32		22		33		33		40

Support types:

A = four-leg chock set one web back (also chock-shield)

B = six-or five-leg chock set conventionally with forepoling extension

C = two-leg shield set one web back - modified for longer canopy

D = conventional four-leg shield set one web back

E = modified four-leg shield set conventionally with forepoling extension

F = modified four-leg shield set conventionally with full sliding canopy

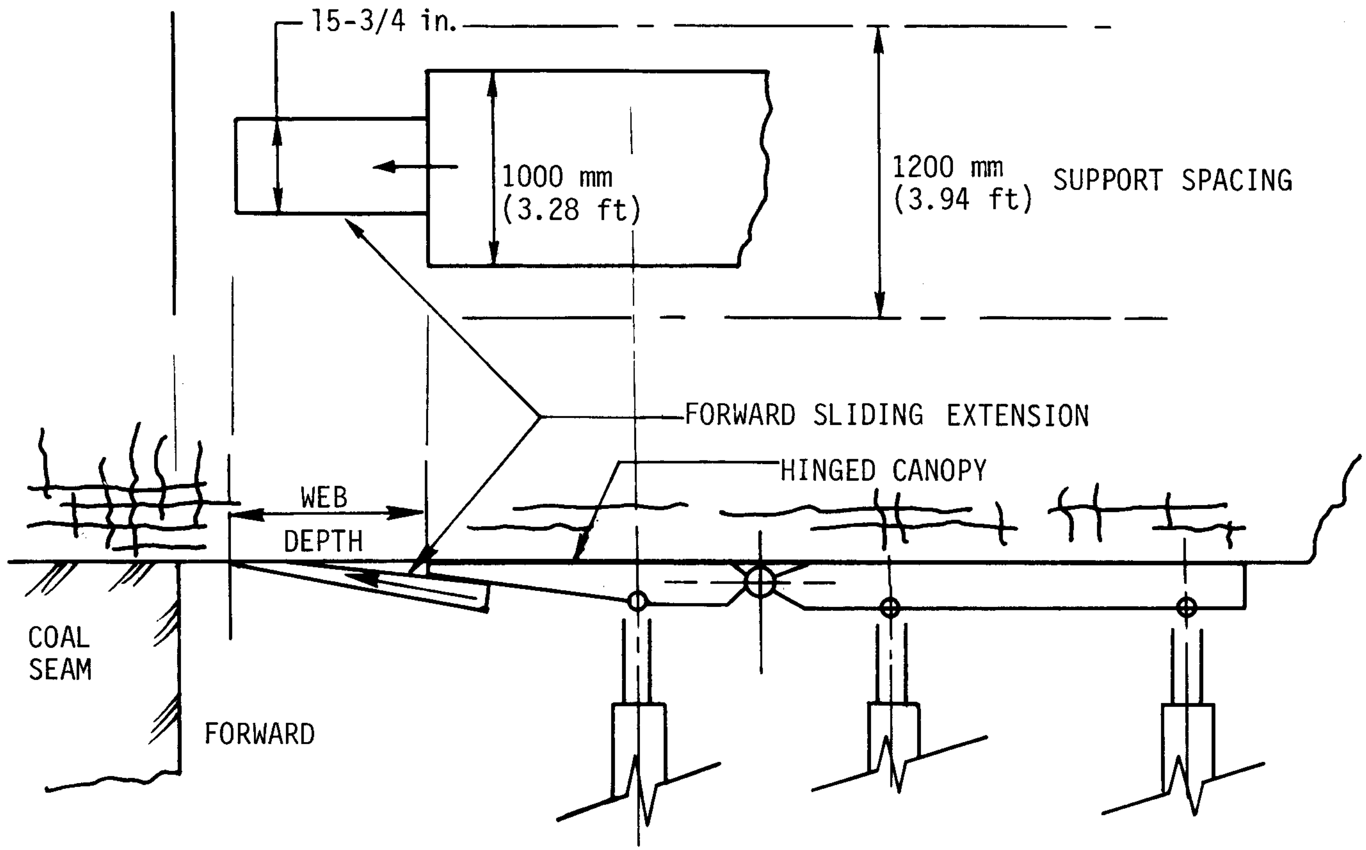


FIGURE 23. - Sliding forward canopy extension as applied to the six-leg chock.

referenced observations have noted (13). If this strata movement is not restricted and the immediate strata is a very friable material, then localized roof falls can occur.

Forepoling, as an IFS feature, can be improved by increasing the width of the extension and by redesigning the mechanism to improve canopy-to-roof contact. The latest designs of wide web chocks being developed in the United Kingdom do provide for improved roof contact. The extensions are powered up to the roof by hydraulic cylinders and a scrissor-type mechanism which places a substantial area of the extension canopy flat against the roof.

Full Sliding Canopy

The full sliding canopy detailed in Appendix D provides the good contact and percent roof coverage features of the one-web back supports. As applied to the two- and four-leg shields, this feature would provide the same full roof coverage as the shields set one web back. The canopy could be fitted with side shielding to close the gap between the roof canopies.

The major advantage of the full sliding canopy feature over the one-web back support is the improved stability and floor loading characteristics as previously illustrated in Figures 20 and 22. Although these stability and floor loading patterns were developed for the two-leg shield with full sliding canopy, these same improvements would equally apply to the four-leg shield.

The disadvantage of the full sliding canopy feature, like the one web back supports, is that the support must be lowered to advance just after the shearer.

Tip Loading

Under the best of conditions the roof of a mine is an irregular surface. The support canopy acts as a nonrigid beam as it is loaded against the roof by the hydraulic cylinders. Contact between the nonrigid beam and the rigid, but irregular roof surface results in a random, unpredictable pattern of contact.

Some portion of the flexible roof beam can be expected to deflect away from the roof under the influence of the loads. Generally, this effect will be mimimal for that portion of the canopy situated between the hydraulic cylinders, hereafter referred to as the main support canopy. However, for that part of the canopy that acts as a cantilever beam, deflection may be substantial.

Good roof control can often depend on adequate support being applied at or near the forward tip of the cantilever at the face. Failure to provide this support can result in localized roof falls, deterioration of the roof surface and excessive sloughing of coal from the face.

Supporting the roof near the face depends on the forward end of the canopy contacting the roof, which is oftentimes not the case. German observations report that the first contact point of the bar with the roof can be as much as 50 cm (1.64 ft) from the front tip of the canopy on narrow web plow faces. This condition could be expected to worsen with wider webs when a one-web-back type of support is used. For example, a four-leg chock set one web back has a length of cantilevered canopy proportional to the width of the conveyor plus the depth of web. As the web increases the cantilever becomes longer.

For the non-one web back supports, the length of cantilever is not associated with web depth, but is dependent on the particular geometry of the support type. For example, the forward inclination of the hydraulic cylinders on the four-leg shield causes the proper to contact the canopy out over the conveyor. This shortens the effective length of the cantilevered portion of the roof canopy.

Therefore, the relative measure of performance that was used to evaluate the various supports for controlling the roof at the face is the length of cantilever in front of the forward props. This length was defined as the distance from the canopy tip to the forward support cylinders where those cylinders attach to the roof canopy. This varies for the different types of supports depending on the position of the support and the inclination of the forward set of cylinders.

Further justification for this distance as a measure of performance is presented in Appendix F. The comparative rating of each support type appearing in Table 13 is qualified in the following discussion for the supports considered.

The Four-Leg Chock

This support has a rigid roof canopy and is set one web back. The length of canopy, cantilevered out beyond the toe of the base, is essentially equal to the width of conveyor assembly plus a depth of web. This distance determines the PFF when the support is advanced to the face. Unlike the shields, the legs are vertical; and therefore, this support exhibits the maximum PFF for any given web depth. For this reason, the four-leg chock was scored lowest in performance.

The Six-Leg Chock (or Five-Leg Chock)

Since this support is not set one web back, the length of canopy is reduced to that length required to cover the conveyor assembly irrespective of web depth. The web depth only influences the length of sliding canopy extension used for IFS. The forward section of the canopy is hinged and is loaded to the roof by the forward most set of cylinders. This is an advantage since tip loading can be applied regardless of the set of the main support canopy. This is not true of any of the supports which mount a one-piece roof canopy. Therefore, this support received the highest rating for control of the roof at the coal face.

It should be noted that four-leg chocks can also be equipped with a hinged forward canopy section. The hinged section is loaded to the roof by an inclined hydraulic cylinder. This is decidedly inferior to the six-leg arrangement since the inclined cylinder cannot support very much roof load. This style of four-leg chock was not considered.

The Conventional Two-Leg Shield

This support is also set one web back, but the forward inclination of the cylinders reduces the length of cantilever as compared to the four-leg chock. In addition, when the shield type of support is advanced properly it slides forward in contact with the roof. This minimizes the potential for roof debris to build up on the canopy, which increases the potential for good tip loading.

The Conventional Four-Leg Shield Set One Web Back

The comments above pertaining to the two-leg shield apply equally to the four-leg shield.

Modified Four-Leg Shield - Set Conventionally

This four-leg shield is modified to increase the distance between the front and rear sets of supports for increased manway. Therefore, this shield is the only shield type support that can be set non-one web back with a rigid canopy. Because of the forward inclined front hydraulic legs, this support has the shortest cantilevered length of canopy. This support received the highest rating followed by the same support with full sliding canopy.

Modified Four-Leg Shield - Full Sliding Canopy

This shield is set non-one web back and featured the full sliding canopy. The length of required cantilever on the canopy would be the same as the above four-leg shield. However, the sliding canopy was considered to be not as rigid and this support was derated to reflect this difference.

Based on the considerations summarized above, the various supports were rated for *support of the roof at the face* in the following order of preference:

a. Six- (or five-) leg chock with hinged forward canopy set conventionally

b. The modified four-leg shield with rigid canopy set conventionally

c. The modified four-leg shield with full sliding canopy

d. The conventional four- or two-leg shield set one web back

e. The four-leg chock set one web back.

The various support types have now been evaluated individually for those influences pertaining specifically to mining wider webs. In the subsection that follows, the relative qualities of the various supports are combined by weighting factors so as to project the best support for controlling the roof over a wide web. It should be noted that the relative rating of supports is based on the assumption that the wide web is a very wide web - greater than 45 in.

3.1.5 Support Selection and Recommendations

Selecting the right support for a particular application requires consideration of many factors, many of which are site dependent. For example, a support is being selected for an application where the seam height is expected to vary considerably. Under these particular conditions, the operating range of the support becomes a major consideration where for a different site, it may be of far less importance.

The factors identified and discussed in this report are only those which are directly affected by the depth of web mined. The relative importance of each of those factors is determined by the actual depth of web being considered and the particular mining conditions. The importance of this family of factors, in the overall selection process, would increase with increasing web depth. Within the family of factors, the relative importance of a particular parameter could vary with mining conditions. Under certain conditions, for example, floor loading could be the most important consideration, where normally roof conditions would be.

In addition, the following selection process considers only the roof conditions that are represented by the selected models. Conditions can and often do exist which do not conform to either of these models. For example, certain types of sandstone strata will overhang the supports and deflect to the floor of the mine with little or no caving. Web depth would have no effect on support requirements under these conditions. However, the models that were used, and thus, the comparative measures of support requirements for varying web depths that were developed from them, cover a wide range of United States roof conditions. This limitation should always be kept in mind when applying the results of this study.

The process selected for demonstrating the comparative qualities of each support type involved a cumulative scoring process. This process required that:

a. The relative importance of each parameter, influenced by the change in web depth, be established and based on this determination a weighting factor (w) be assigned.

b. Each support type was then performance rated for each of the operating parameters. The relative scale of performance (n) ranged from four for best performance of a support type to unity for poorest performance.

In the order established for their relative importance, the operating parameters were:

Operating parameter	Relative importance	Weighting factor (w)
IFS	Very important	3
Support stability	Very important	3
Tip loading at the face	Important	2
Floor loading:		
Pressure distribution	Important	2
Peak pressure (value)	Lesser importance	1

This report considers support stability and immediate forward support of the roof to be of prime importance. This is based on the assumption that the web depth will be 45 in. or greater and that the immediate roof strata will be friable shale - a condition that is common in the United States.

Tip loading at the face is considered important as is the distribution of pressure on the mine floor. The distribution of floor loading was considered to be of more importance than the actual peak pressure values established in subsection 3.1.2. This suggests that the average mine floor would be more sensitive to pressure at the toe of the support base than it would be to pressures more evenly distributed.

The scale of performance (n) for each support relative to the other supports considered is, of course, based on the discussions and figures of subsections 3.1.1 through 3.1.4.

The cumulative score of each support type for the five parameters of importance the total score (N) is:

$$N = \sum^{5} w \cdot n$$

The results of this comparative scoring process appear in the performance matrix of Table 13.

The conventional two-leg shield is considered unacceptable for wider webs because it proved to be unstable. The two-leg shield evaluated in Table 13 was modified to present a longer canopy. As modified, this support was still the least stable of all the supports considered.

The conventional four-leg chock is also not recommended because of poor stability and because of the excessive PFF distance required to accommodate the one web back operation. This situation would also apply to the chock-shield type of support.

Of the strictly conventional supports, the four-leg shield and the six-leg (or five-leg) chock were rated the highest. The six-leg chock exhibited good stability because of the long length of canopy. The forward hinged section of canopy on this support would provide the best support of the roof at the face (tip loading). The comparative weakness of the six-leg chock is the rather poor IFS provided by the forepoling feature.

The conventional four-leg shield set one web back, however, would provide excellent coverage of the exposed roof. Stability and floor loading with this support set one web back were rated as good. However, the stability of this shield can be considerably improved by extending its length. The extended length would be required to provide additional manway between the forward and rear sets of props. With the manway between the props this modified shield would not have to be set one-web-back. Stability and floor loading would be improved by moving the base forward to the conveyor and by the resulting increased length of canopy.

However, as in the case of the six-leg chock, support of the forward exposed roof would be compromised because of the requirement to use the extendable sliding canopy (forepoling).

Eliminating this deficiency on the modified four-leg shield led to the full sliding canopy concept as presented in Appendix D. The overall rating of this support was the highest of all supports considered. Deployment of this support would, of course, require the development of the full sliding canopy feature. This should be considered a viable concept as this feature would be particularly desirable for covering very wide webs (45 to 60 in.) under difficult roof conditions.

The potential for the modified four-leg shield, set non-one web back, would also be considerably enhanced with an improved design of forward sliding extension. These improvements would be required to provide more extensive coverage and better contact between the extendable canopy and roof.

Recommendation

The full scope of this program includes an underground demonstration of wide web longwalling. The recommendation of a roof support for this demonstration would, of course, depend on the maximum depth of web to be demonstrated and the roof conditions at the selected mine.

However, the analysis and discussion above suggests that the development of new or improved features would considerably enhance control of the roof, particularly over very wide webs. Based on the long range possibilities for wide web longwalling, the following order of preference is suggested:

a. Develop the sliding canopy concept to be deployed on the modified (lengthened) four-leg shield.

b. Develop improved sliding extensions (forepoling) to also be deployed in the modified four-leg shield.

c. Lengthen the four-leg shield and mount state-of-the-art forepoling - if roof conditions at the selected mine are suitable.

d. Mount state-of-the-art forepoling on the five- or six-leg chock if the excellent tip loading features of this support are clearly required.

Under most mine conditions considered to be suitable for longwalling, the limitations both in terms of maximum web depth and rate of mining (tons per hour) will be established by the capability of the shearer cutter-loader.

Therefore the next subsection discusses the impact of wider webs on the shearer and in particular on the power required to mine these depths.

3.2 The Shearer Cutter-Loader

The first of the two subsections that follows is concerned with the dependence of the shearer's cutting and loading performance on depth of web while the second subsection discusses the machine modifications that will be necessary to mount the wider drums.

3.2.1 Shearer Power Consumption

The efficiency with which the shearer cuts coal is an important parameter that will affect the economics of mining wider webs. This is because power available for a given machine is of necessity limited, and a significant increase in specific power consumption could reduce the rate of mining coal. In this subsection the two bodies of data that are available on the effect of web depth on shearer specific power consumption are examined and conclusions drawn that will then be applied to determine productivity and cost effectiveness in Sections 4 and 5.

German Data

The first set of data was published by Ostermann (14) and later incorporated in the Cominec report on the conceptual design of an automated longwall system (15).

Ostermann describes the conditions prevailing during a trial of an Eickhoff single drum shearer-loader of 80 kW (107 hp). The seam height of 900 mm (35 in.) was worked by an 1100 mm (43 in.)

diameter drum. The coal was hard (compressive strength up to 4300 lb/in.^{2}) with no discernible cleats and included ventricular intrusions of very hard pyrites (compressive strength up to 7650 lb/in.^{2}). The experiment consisted of varying the web depth over the range of 15 to 30 in. Drum shaft speed was varied from 80 to 102 rpm. The loading spiral angle was constant at about 9 deg and drum rotation was clockwise for all tests.

The results of the experiment showed both a decrease in production rate and an increase in specific energy consumption with increasing web depth over the range 15 to 30 in. Figure 24(a) and 24(b) demonstrate these results graphically.

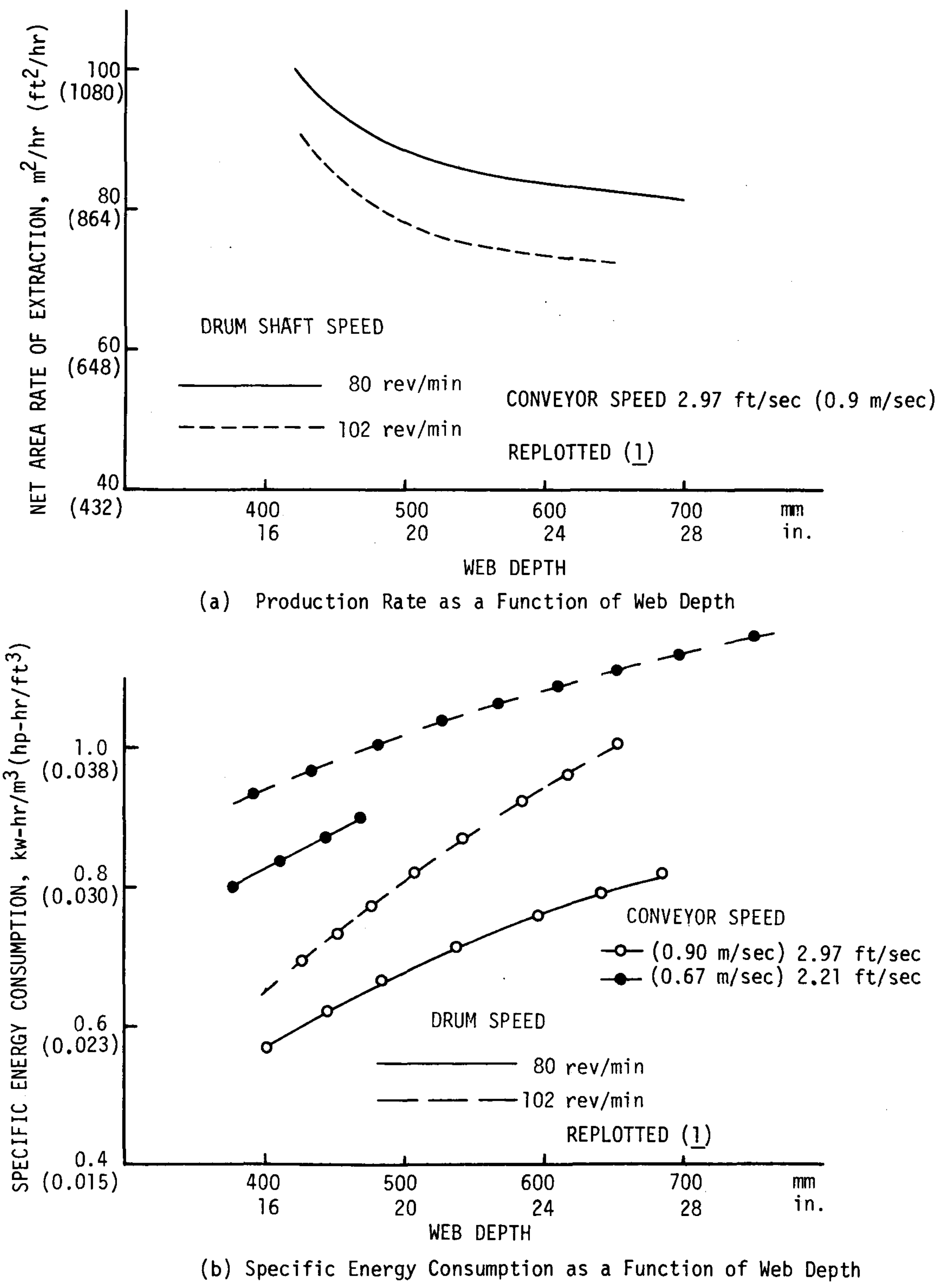
Figure 24(a) shows that the rate of extraction decreased as web depth increased at both of the drum shaft speeds employed. Figure 24(b) demonstrates the increase in specific energy consumption that occurred with web depth. It should be noted that in this case the specific energy consumption was also strongly dependent upon conveyor and drum shaft speed. The curves suggest that the conveyor may have been overloaded at 0.67 m/sec.

Ostermann concludes that the increase in specific energy consumption at the wider web is associated with the decrease in "cutting depth" (penetration) prevailing at a given drum shaft speed as haulage speed is reduced. His data shows that this effect can be mitigated by reducing drum shaft speed and thereby increasing penetration at a given haulage speed.

Mining Research and Development Establishment (MRDE) Surface Test

The surface test conducted by MRDE involved a 200 kW Anderson Mavor Buttock Shearer cutting a block of simulated coal using drums 30, 36, 42, and 48 in. wide. Mechanical haulage was employed to cut at a finite number of haulage speeds. No consistent attempt was made to investigate the effect of changes in drum shaft speed. The primary purpose of the surface trial was to confirm the ability to load simulated coal out of a very wide web.

The raw test data were analyzed to ascertain how the efficiency of the shearer (in terms of specific energy consumption) was affected by variation in web depth.



(a) Production Rate as a Function of Web Depth

(b) Specific Energy Consumption as a Function of Web Depth

FIGURE 24. - Shearer performance with increasing web depth - German test results by Ostermann.

The analysis showed that the specific energy increased significantly as web depth increased both at constant production rate and constant power consumption. Further consideration suggests that the apparent decrease in efficiency is not associated with the increase in web depth but, as with the Ostermann data, was due to the change in the haulage speed that occurred as the web depth was increased. At a constant production rate (tons per hour) as the web depth is increased the haulage (advancing) speed must be reduced. Since the drum design and drum shaft speed were not varied, the reduced haulage speed at wider webs resulted in a significant reduction in pick penetration and a consequent decrease in cutting efficiency.

Most of the MRDE data was gathered at a single drum shaft speed of 56 rpm. For the few tests where the drum shaft speed was increased to 64 rpm, there was approximately a 25 percent increase in specific energy consumption. Unfortunately, very little data was obtained at the higher drum shaft speeds, and it is not possible to determine its full effect.

The data shows no dependence of haulage power requirements on web size for a fixed production rate. The tests also offer no data on the power required to helix the coal out of deeper drums although this must be expected to increase with web depth.

Application to United States Conditions

Mutmansky (16) has attempted to apply Ostermann's results to United States conditions. As noted earlier, the coal used in the Ostermann tests was very hard (4300 lb/in.^{2} compressive strength) as compared to United States conditions (usually less than 3000 lb/in.^{2}).

This was done by plotting the trend of Ostermann's data through a point defined by a 300 kW machine which, under United States conditions, can cut and load coal from a 27 in. web at about 1000 tons/hr, (Figure 25). This figure shows that, while specific energy consumption is likely to fall with thin webs, the specific energy curve flattens out in the region of the width of webs (25 to 30 in.) that are currently mined.

The strength of the material used for the MRDE surface test is unavailable and, in any case, would not be relevant since this material lacks any of the cleats or faults which may greatly affect the cutting properties of coal. The material employed was described as being harder to cut than coal.

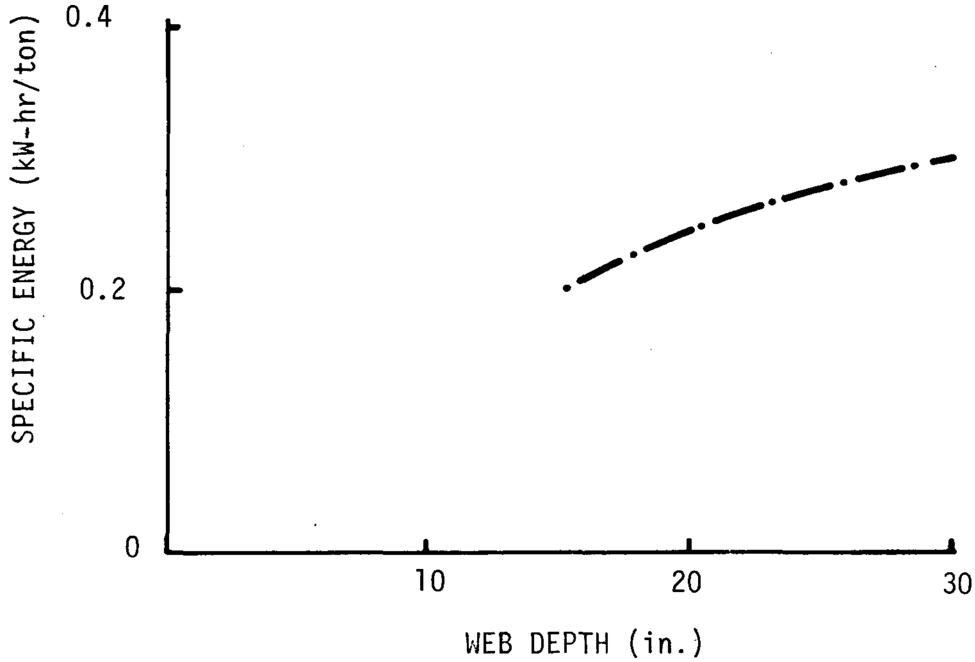


FIGURE 25. - Specific energy versus web depth - Mutmansky's curve to depths of 30 in.

Prediction of Specific Energy at Wider Webs

To determine the theoretical output of various types of shearing machines over a range of web depths, the first step is to determine the effect of web depth on specific energy consumption.

Power input to a shearer is split three ways between:

- Haulage
- Cutting of coal
- Loading coal through the drum helix.

Both the German and MRDE data show that coal cutting power requirements are likely to be independent of web depth, provided that:

a. Cutting conditions, especially pick penetration, are not changed.

b. A large proportion of the cutting does not take place in the prefractured zone within a few inches of the free face.

If these conditions are met the cutting power requirement should be proportional to the production rate alone.

The power required to load coal out of the web will increase as web depth increases. The additional energy required is not large since it does not exceed 30 hp on a current high capacity machine. As a first approximation, the power required to load a solid material through a helix is proportional to the length of the spiral (in this case the depth of the web).

The assumptions stated above were applied to a current shearer whose overall (483) and haulage (40) horsepowers were known and whose production is 1000 tons/hr in actual operations. Applying assumed efficiencies of 75 and 50 percent to the mechanical drives and hydraulic haulage, respectively, it was possible to calculate specific energy consumption as a function of web depth. This theoretical relationship is shown graphically in Figure 26.

The conclusion to be drawn therefore is that if webs are decreased to values well below the standard web (27 in.) a decrease in specific energy consumption occurs, but that if webs are increased over the range considered in this study, only a slight increase in specific energy requirement is to be expected in the range 30 to 60 in.

This view is borne out by the empirical statement of J.R. Hunter (9) that there was no discernible decrease in shearer performance when 1-m webs were introduced at Hem Heath and Holditch Collieries.

The relationship between web depth and specific energy consumption determined above is used in Section 4 of this report to determine the coal cutting capacity of shearers of known rated power.

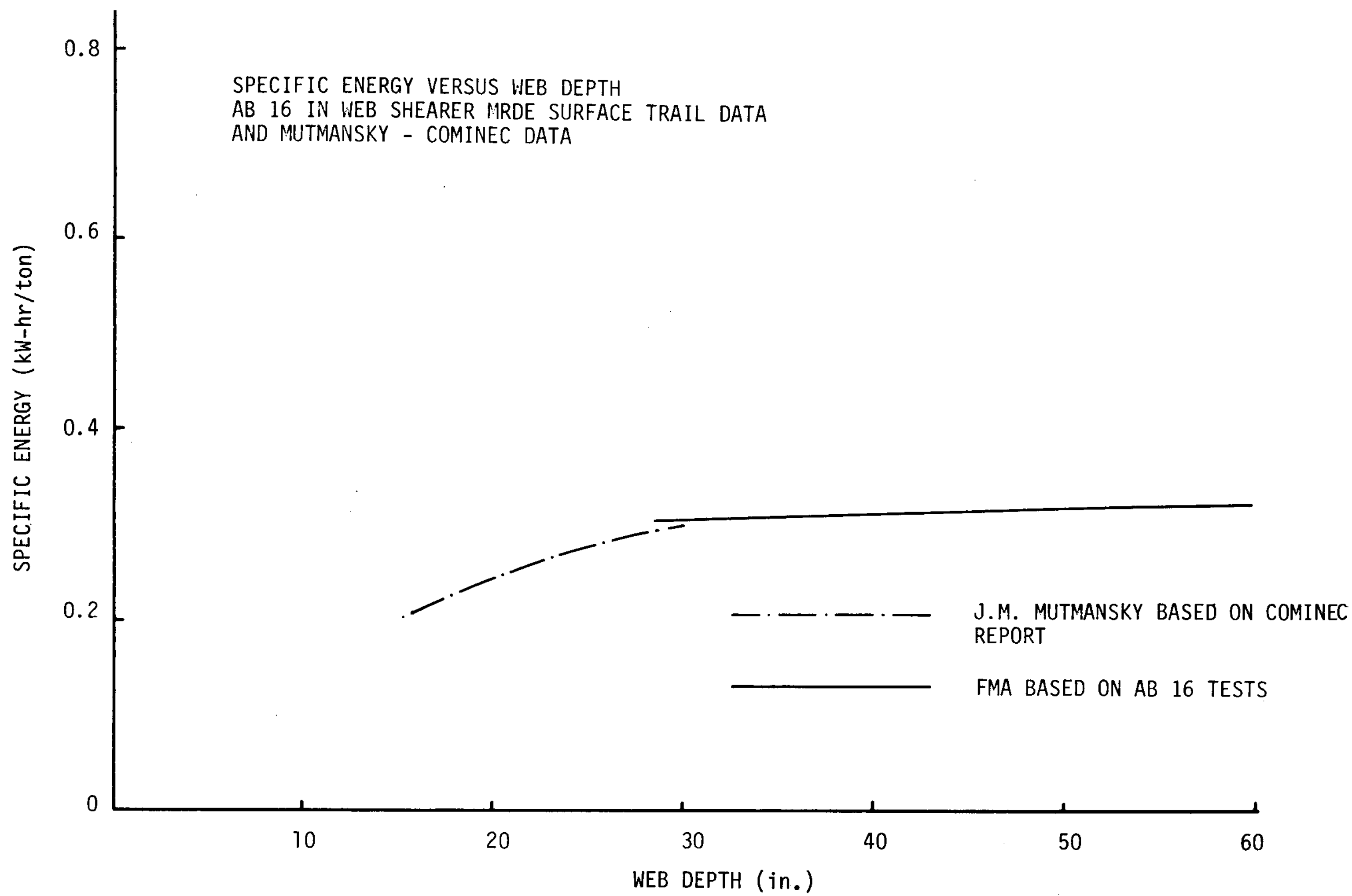


FIGURE 26. - Specific energy versus web depth - Mutmansky's curve and FMA's extrapolation.

The determination of this relationship which is fundamental to the overall analysis has been, of necessity, based upon the extrapolation of somewhat incomplete data. It is strongly recommended that underground tests be conducted to measure shearer power consumption at varying web depths and as a function of drum shaft and haulage speed.

The objective of this test would be to mine a 27-in. web and a wider (say, 40-in.) web at constant power (at or near full available shearer power). The two web depths would have to be mined at constant pick penetration. This would require reduced shearer haulage speed and reduced drum speed (or increased bit "bite") at the 45-in. web depth.

For both tests shearer total power, shearer speed, haulage systems power, and average web depth would be recorded. Power to the cutting drums would be total power minus haulage power. It would then be possible to calculate power consumption per cubic foot of coal mined at the web depths.

3.2.2 Modifications to Existing Shearers for Wider Drum Mountings

Shearer manufacturers were surveyed to determine the maximum drum widths that could be mounted on today's shearers and what kinds of modifications would be required to accommodate these drum widths. The survey was extended to include a soon-to-be-available shearer normally designed to mount 1.0m (39 in.) wide drums. This machine could also be modified to mount 1.5m drums (60 in.) and is identified as the prototype 310 kW machine.

As a result of the survey, it was determined that today's shearers can mount the maximum drum widths for mining the indicated depths of web as indicated in Table 14.

The capability of the 300 kW machine to mount a 48-in. drum and load out of a 45-in. web was established as the limit by both Eickhoff and Anderson Mavor. This was based on what they considered to be reasonable modifications to the machine.

Mining Supplies, the British conveyor manufacturer, has built a prototype 310 kW shearer designed to mount a 1.0m (39 in.) wide drum. With modifications, the manufacturer indicates it could mount a 1.5m (60 in.) drum. Mining Supplies plan to offer this machine to the market within 8 to 12 months. This machine with 60-in. drums therefore was included in this study.

Additional changes to mount still wider drums were considered, by the manufacturers, to be a redesign of the machine, not a modification to an existing machine. Therefore, these limits on web depth were used throughout the report to define the maximum depth of web within the scope of this feasibility study.

TABLE 14. - Maximum web depths for available shearers

Shearer Size (kW)	Maximum Drum Width (in.)	Maximum Web Depth (in.)
170	42	39
300	48	45
Prototype 310	60	57

The modifications that the manufacturers identified as being required to accommodate the above maximum drum widths were generally:

a. Strengthening the ranging arms and their mounts on the shearer body

b. The strengthening of the drum mountings and drum hub assemblies

c. Modifying the shearer underframes to widen the mountings for additional stability and to remove the loads from the conveyor pans.

The extent of the modifications required would vary both with the size of the machine and the particular manufacturer.

The shearer manufacturers further felt that the coal could be loaded out of the indicated web depths using the helixed type of drum. Based on experience to date these drums would undoubtedly be three start helixes with possibly tapered drum hubs. There might also be some considerations given to variable pitch helixes. Chainless haulage systems were recommended for all shearers at all depths of web.

Throughout the remainder of the report the performance and costs associated with these shearers reflect these modifications and features.

3.3 The Armored Face Conveyor

Of the equipment on the face, the armored conveyor is least affected by an increase in web depth. The conveyor must be advanced a greater distance with increased depth of web. Wider webs will also result in heavier shearers and increased loads on the mounting guides and trappings.

To accommodate the increase in conveyor advance, push rams with longer strokes will be required. Mounting and guiding the heavier shearers requires modifications to the shearer mounted guidance hardware.

Pan connectors can and oftentimes are overloaded when the conveyor is advanced across the present 27-in. wide web. An increase in depth of web increases the potential to damage these connectors. However, the nature of the problem does not change, only the severity. This is accounted for in the productivity calculations as additional lost time due to conveyor malfunction. Stronger connectors may be required even though the problem only occurs as a result of the equipment being misused.

The requirement for higher capacity conveyors, with increasing web depth, is also accounted for in this study. Under certain conditions, conveyor capacity does limit production. The impact of higher capacity state-of-the-art conveyors, as well as capacities beyond the immediate state-of-the-art, are discussed in Section 4 of this report.

4. SYSTEM PERFORMANCE (PRODUCTIVITY)

In subsection 2.3.3 the potential productivity of mining wider webs was compared against mining with faster shearers, mining longer faces or mining with more reliable equipment. In the simplified analysis of subsection 2.3.3, only the potential of the contemporary high capacity longwall was investigated. This involved the Eickhoff EDW-300L shearer and the Eickhoff EKF-3 conveyor (700 tons/hr rated capacity).

In this section the productivity analysis (tons of coal per shift) will be expanded to include various combinations of shearer-face conveyors over a range of web depths. The productivity calculations of this section will then be combined with the owning and operating costs to arrive at the cost effectiveness of mining wider webs in Section 5.

4.1 Mining Conditions

As in subsection 2.3.3 this expanded productivity analysis will reflect the potential of the mining conditions associated with the contemporary high capacity face. The performance parameters of this operation are given in Table 15 with the equipment specifications of Table 16.

The factors extracted from Table 15 to be used in this analysis are:

t_ℓ = lost time off the face = 79 min

H = height of extraction = 6.5 ft

T_t = total shift time = 8 hr (480 min)

L = face length = 500 ft

S_c = tramming speed = 30.1 ft/min

t_e = turnaround time = 7.54 min.

These factors will be treated as constants in the shift production equation of Table 3. This equation representing the shift performance of double-ended shearer operating in the half face or modified half face mode is developed in Appendix B. Therefore, the productivity calculations of this section and the economics of Section 5 are for double-ended shearers cutting both ways on the face. Unlike the analysis of subsection 2.3.3, lost

TABLE 15. - Performance parameters contemporary high capacity longwall

Panel	
Face width	500 ft
Panel length	3000 ft
Height extracted	6.5 ft
Web depth	27 in.
General	
Working days/year	220 days
Production shifts/day	2 shifts
Maintenance shifts/week	6 shifts
Total shift time	480 min
Travel time and miscellaneous	78 min
Available working time at face	401 min
Downtime Analysis Per Shift	
Total down or lost time	120.5 min
Armored face conveyor	37.1 min
Shearer	33.6 min
Stage loader	4.9 min
Roof supports	4.6 min
Outby haulage and miscellaneous	35.4 min
Geological and other	4.9 min
Panel Development	
Manshifts required	572 manshifts
Entry development rate	57.2 ft
Average tons/shift	268 tons
Shift Breakdown	
Tons/pass	329 tons
Passes/shift	4.0 passes
tons/shift	1316 tons
Cutting and loading rate (tons/hr)	517.1 tons
Shearer speed	
Cutting (ft/min)	13.1 ft/min
Cleaning (ft/min)	30.1 ft/min
Cycle time (min)	70.1 min
Shift Utilization	
Loading (min)	280.5 min
Cutting (min)	152.7 min
Cleaning (min)	66.7 min
Turnaround (min)	61.1 min
Scheduled maintenance (min)	0 min
Unscheduled maintenance and miscellaneous	120.5 min
Travel and miscellaneous	79 min
Annual Summary	
Working shifts/year	440 shifts
Maintenance shifts/year	300 shifts
Tons/panel	438,750 tons
Working shifts/panel	334 shifts
Panels/year	1.106 panels
Tons/year	485,050 tons

TABLE 16. - Contemporary high capacity longwall face equipment specifications

Item	Quantity	Description	Unit price ($)	Total cost ($)
Shearer	1	Eickhoff EDW-300-L shearer, chain type haulage with Eickomatic, water cooled	761,000	761,000
Conveyor	1	Eickhoff EKF-3 armoured face conveyor, 26 mm single strand chain, 300 hp, 510 ft		257,000
(includes)	1	Elevating pan		
	1	1-m pan		
	1	0.75m pan		
	1	0.5m pan		
	1	Drive units, approximately 30:1 reduction ratio		
	1	Set of Eickhoff face accessories for EKF-3		310,000
(to include)	1 set	Ramp plates		
		Tube guide sections		
		Splash pans with connection plates		
		Chains		
Stage Loader	1	Eickhoff EKF-3 stage loader - interchangeable with face conveyor, 2 by 50 hp		81,000
Roof Supports	102	Kloeckner shield supports, 2/352 ton	27,000	2,754,000
	1 set	High pressure Hauhinco hydraulic pumps	78,000	78,000
	2	100-hp electric motors, 950 Vac	3,100	6,200
Conveyor Belt	1	42-in. belt drive - long airdox, 250 hp		32,557
	1	Long airdox "super 300" takeup unit for 500 ft of belt storage		22,653
	1	Self-advancing tailpiece - long airdox - crawler mounted		37,495
	583	Top belt rollers - long airdox	42.95	25,040
	292	Bottom belt rollers - long airdox	34.64	10,407
	7000 ft	5/8-in. wire rope	0.49/ft	3,430
	7000 ft	42-in. conveyor belt	9.33/ft	65,310
	1	Belt power center		17,000
Electrical equipment	1	Ensign Hubgel 750 KVA power center, 7200/950V	19,000	19,000
	2	Control boxes with line starters		59,705
	1	Master control box		
	20	Pull chord switches for face conveyor		31,556
	1 set	Davis of Derby face communication equipment with master control consol		
	1600 ft	2/0 3C SHD-GC 2.00 OD permissible electrical cable		26,209
	500 ft	No. 2 3C SHD-GC permissible cable		
	700 ft	No. 4 3C SHD-GC permissible cable		
	250 ft	No. 6 3C SHD-GC permissible cable		
Miscellaneous	25	Dowty 25-ton props with handles	295	7,375
		Miscellaneous tools, connectors, hoses, etc.		10,000
		Total cost (4/28/78)		$4,614,928

TABLE 17. - Lost time* on the face (t_ℓ) with increasing web depth

Item	Extracted web depth in inches										
	27	30	33	36	39	42	45	48	51	54	57
Face conveyor	37.1	38.0	39.0	39.6	40.8	41.8	42.7	43.6	44.5	45.4	46.3
Shearer	33.6	34.4	35.3	36.1	36.9	37.8	38.6	39.4	40.2	41.0	41.8
Stage loader	4.9	4.9	4.9	4.9	4.9	4.9	4.9	4.9	4.9	4.9	4.9
Roof support	4.6	4.7	4.9	5.0	5.2	5.3	5.5	5.6	5.8	6.0	6.1
Outby haulage	35.4	35.4	35.4	35.4	35.4	35.4	35.4	35.4	35.4	35.4	35.4
Geological and other	4.9	5.1	5.3	5.4	5.6	5.7	5.9	6.0	6.2	6.3	6.5
Total	120.5	122.5	124.7	126.4	128.8	130.9	133.0	134.9	137.0	139.0	141.0

*All times are minutes lost per normal 8-hr shift.

time on the face (t_ℓ) will vary with web depth as shown in Table 17. As the table shows, this lost time is due to all causes including outby haulage delays and geologically related problems. The delay times associated with the 27 in. extractions are compiled from various downtime studies (17, 18). The actual effect of wider webs on equipment and geologically related delays is unknown. Therefore, the increase in delay times assigned to the wider webs is arbitrary. However, increasing downtime was only assigned to that equipment or those factors that would actually be influenced by the deeper web - specifically the shearer, face conveyor, roof support system and geologically related problems. The accuracy of the productivity projections will be enhanced by the inclusion of the data of Table 17 since the increased delay times have been projected from a reasonably accurate data base and have been assigned to only those factors affected.

Shearer tramming speed (S_c) was not considered to be significantly affected by an increase in web depth, particularly at the 6.5 ft height of extraction. Tramming speed, therefore, was treated as a constant in the shift production equation.

Turnaround time at the face ends (t_e) was also considered to be a constant for this analysis. However, it might be argued that the shearer may be more difficult to maneuver at the entries with the wider drums and cowls. It might also be argued that the shearer tramming speed would be reduced for the same reason.

To determine the sensitivity of the productivity calculations to these assumptions a calculation was made where the tramming time and face end cleanup time were arbitrarily increased by 20 percent. For a system extracting a 45-in. web, shift production decreased 139 tons/shift. This represents a 7.5 percent change in productivity as the result of a 20 percent change in the assumed tramming and cleanup times. It was, therefore, concluded that the accuracy of the production projections arrived at with S_c and t_e constant are not significantly compromised.

With the above assumptions, web depth (W) and shearer haulage speed ($S_{c\ell}$) remain as the only variables in the shift production equation of Table 3. Web depth will vary according to the limitations of the particular shearer being evaluated.

Shearer haulage speed (while cutting and loading coal) is related to system capacity (C_c) in tons per hour and web depth as shown in the following equation:

$$S_{c\ell} = \frac{4.44\ C_c}{WH}$$

The limitations imposed on $S_{c\ell}$ for different shearer-conveyor combinations and depths of web will be discussed in the subsection on equipment performance since they are equipment related limits. However, one additional limitation was imposed in the study which is related to the mining conditions.

Under mining conditions classifed as good, the maximum shearer haulage speed was limited to 20 ft/min. There are oftentimes conditions of mining which would limit the haulage to less than 20 ft/min. These conditions might include:

a. The operators ability to control the machine while negotiating undulations in the roof and floor which requires the ranging of the shearer drums

b. Cutting conditions, such as the presence of rock bands, sulphur balls and other difficult to mine inclusions

c. Dust generation which will increase with increasing haulage speed and the associated necessary increase in drum speed. This may impose a limitation affecting the operators vision and consequently his safety as is discussed in subsection 5.2

d. Methane gas liberation, the rate of which is a function of web depth and haulage speed, since the rate of new surface exposure determines the rate of gas leakage from the unmined seam. Also, the rate of coal mined determines the rate of release of residual gas in the cut coal. The combination of wider webs and/or faster haulage speeds and drum speeds (pick speed) can lead to the ignition of gas pockets in the deeper webs.

These conditions of mining were classified as "difficult" limiting the shearer speed to 13 ft/min. The upper and lower limits of 20 and 13 ft/min and the one tramming speed of 30 ft/min was established by the data from the five mine surveys of high production operators that is summarized in Table 18. There are obviously combinations of the above conditions that would not even allow speeds of 13 ft/min, but these conditions were considered not to be potentially high production faces and, therefore, were outside the scope of this study. Finally, as a matter of interest, production figures were also established for the situation where haulage speed was completely unrestrained by mining conditions.

TABLE 18. - Cutting and tramming speeds from the five-mine survey

Mine	Cutting Speed (ft/min)	Tramming Speed (ft/min)	Seam Height (ft)
1	11.3	30.0	4.39
2	20.0	20.0	6.50
3	15.0	20.0	6.50
4	12.0	38.0	7.00
5	12.3	42.0	7.00

4.2 Mining Equipment Systems

Various shearer-conveyor combinations were considered in this productivity analysis so as to project the potential for wider web mining over a reasonably broad spectrum. Additionally, it was desired to determine in the economic analysis of Section 5 what combination of equipment and depth of web would be the most cost effective. Therefore, in addition to projecting the productivity of the contemporary high production longwall at various web depths, shearer-conveyor combinations of both greater and lesser capacity were also investigated.

Four combinations were included in the study defined as:

a. The Contemporary High Capacity System - This is the system of Table 16 which utilizes an Eickhoff EDW-300-L (300 kW) and a conveyor of 700 tons/hr capacity. The performance of this system was investigated for a range of web depths varying from the current 27 in. to a maximum of 45 in., the upper limit established by the shearer manufacturer.

b. The Low Capacity System - This face differs from the contemporary high capacity only in that a shearer of lower capacity, 170 kW, is employed. The AFC employed is still the EKF-3 700 tons/hr conveyor. For this shearer, the maximum drum width that can be mounted is 42 in.

c. The High Capacity System - The 300 kW shearer with 48 in. drums is combined with a state-of-the-art conveyor capable of handling 1000 tons/hr. This conveyor is representative of the Eickhoff DMKF-4 or the DOE-FMA high capacity conveyor.

d. The Very High Capacity System - This system uses the Mining Supplies soon to be available 310 kW shearer with their new 1500 tons/hr conveyor. The width of conveyor and therefore shearer body mounting allows the mounting of 1.5m (60 in.) wide drums.

The maximum web depth for each system considered was established in consultation with the relevant manufacturers as:

System	Maximum drum width (in.)	Maximum web depth (in.)
Contemporary High Capacity	48	45
Lower Capacity	42	39
High Capacity	48	45
Very High Capacity	60	57

The difference between the maximum drum width and web depth is due to the imperfect nature of the conditions of operation. For example, 30 in. wide drums widely used in the United States typically extract a 27 in. web. The productivity of these four shearer-conveyor combinations at various web depths up to the maximum is basically a function of shearer haulage speed that can be achieved within the limitations discussed in the following subsections.

4.2.1 Equipment Systems Performance-Rated Capacities

The formula developed in Appendix B and summarized in Table B establishes shift production as a product of tons won per cycle and cycles completed per shift. The tons won per cycle (TPC) is:

$$TPC = \frac{LWH}{266.7}$$

With face length (L) and height of extraction (H) constants, the tons per cycle varies directly with the depth of web.

The cycles completed per shift (n) is a more complex function being established as:

$$n = \frac{T_t - t_t - t_\ell}{\frac{L}{S_{c\ell}} + \frac{L}{S_c} + 2\ t_e}$$

where the numerator is the available time in a shift and the denominator is the average cycle time.

As established in subsection 4.1, all factors in this relationship are constants with the exception of shearer haulage (cutting) speed $S_{c\ell}$ and lost time on the face t_ℓ. Lost time varies as a function of web depth as established in Table 17. Therefore, the only factor remaining to be quantified is shearer cutting speed $S_{c\ell}$. As defined, cutting speed and equipment system capacity (in tons per hour) are in fact synonymous as established by the second relationship in Table 3 which is:

$$S_{c\ell} = \frac{4.44\ C_c}{WH}$$

where C_c is system capacity in tons per hour and W is web depth in inches.

System capacity, and therefore shearer speed, for varying depths of web (the height of extraction being constant) can be limited by:

a. Shearer capability - a function of available shearer power and the specific energy of cutting coal at the various depths of web

b. The rated capacity of the face conveyor in tons per hour.

The specific power consumption was established in subsection 3.2.1 and Figure 27 for varying depths of web up to 60 in. Given the available shearer power and the depth of web, the rate of mining (C_c) and therefore the shearer speed ($S_{c\ell}$) can be established.

In conclusion, for the purposes of this analysis, the performance of the four mining systems for varying depths of web was established by shearer capability, face conveyor capacity or mining conditions, where the limitations imposed by the mining conditions were established in subsection 4.1 as 20 ft/min for good conditions and 13 ft/min for difficult conditions.

The capability of today's longwall equipment is summarized in Figure 27 where shearer cutting speed, as a measure of rate of mining, is shown for various equipment systems mining at various web depths.

The following observations can be made from these performance curves:

a. The nonlinear curves reflect the increase in specific energy of cutting coal as the web depth increases

b. The performance curves of the high capacity and very high capacity systems coincide at web depths less than 45 in. This reflects the fact that the performance of both systems are limited by shearer capability over the entire range of web depths. The capability of the 1500-ton capacity conveyor is never used - the 310 kW shearer of the very high capacity system simply extends longwall mining capability out to a web depth of 59 in. Utilization of the 1500-ton capacity conveyor would require a larger shearer with more power.

c. Conveyor limitations imposed on the capability of the contemporary 300 kW shearer are clearly shown by the performance of the contemporary high capacity system (curve A-A). Shearer speed over the entire range of web depths is less than that for the same shearer operating with a 1000 ton/hr capacity conveyor (curve C-C). This decrease in performance will reflect itself in decreased shift production. This relative difference in performance would also occur at seam heights of less than 6.5 ft, as the family of performance curves would simply shift up on the graph. Actual performance in a lower seam, of course, might well be limited by mining conditions rather than by equipment capability.

d. The effect of limited shearer power is shown by the performance curve B-B. Over the entire range of web depths, this system is limited by shearer capability.

e. The effect of conditions of mining are illustrated by the upper (20 ft/min) and lower (13 ft/min) limits on shearer speed identified as presenting *good* and *difficult* conditions. With good conditions:

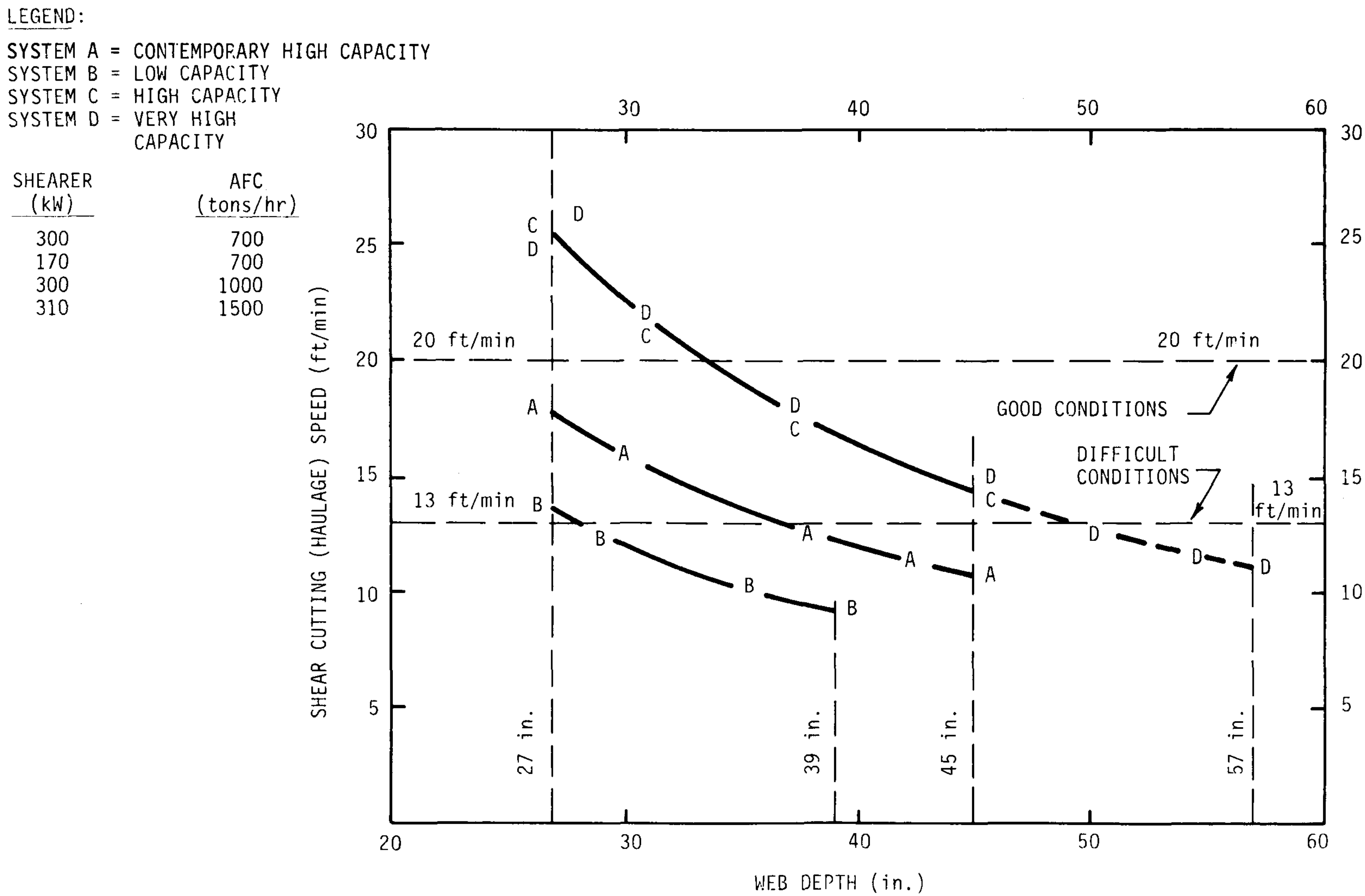


FIGURE 27. - Shearer speed versus web depth - a measure of longwall system performance.

- Maximum performance would be achieved with the high capacity and very high capacity systems. The maximum allowable web depth should be mined with this equipment and web depths less than 34 in. should not be mined at all.

- The contemporary high capacity and low capacity equipment systems restrict production, at all web depths, with good mining conditions.

With difficult conditions:

- The low capacity and contemporary high capacity systems would mine their potential at web depths greater than 28 and 37 in., respectively.

- The full potential of the 300 kW class of shearer would only be used at web depths greater than 49 in.

- The greatest potential for wide web mining to increase productivity occurs under difficult mining conditions. For example, at a web depth of 27 in., all systems are operating at less than their potential. Systems C and D are operating at half of their potential whereas System A is operating at 73 percent of its potential. Only the low capacity system is operating near its capability.

All factors, both constants and variables, appearing in the shift production equation have now been quantified for the four equipment systems at the depths of web within their capabilities. To this point, this subsection has discussed primarily the capability of the four mining systems in terms of rate of mining coal (tons of coal per hour mined and transported off the face) at various web depths.

The subsection that follows expands this work to reflect the shift production that can be achieved by these systems under various conditions.

4.2.2 Equipment Systems Performance - Shift Productivity

The detailed results of the complete analysis for the four systems at all appreciable web depths appear in Appendix G. These tables of results were compiled by assigning the appropriate constants and variables defined in the preceding sections to the equations of Table 3. Also included in these tables is the tons of coal mined per annum which is applied to the economics of Section 5. The tables, as they appear in the Appendix, are:

a. G-1 - The Low Capacity System - Operating over the range of web depths 27 to 39 in. This table looks at performance with no restrictions imposed by conditions of mining. Tons mined per shift is a function of shearer-conveyor capability. This is the only table of results compiled for this system since performance is never restricted by mining conditions (see curve A-A of Figure 27).

b. G-2 - The Contemporary High Capacity System - Operating over the range of web depths 27 to 45 in.; no restrictions imposed by conditions of mining.

c. G-3 - The Contemporary High Capacity System - This table shows performance where difficult mining conditions impose a maximum shearer speed of 13 ft/min. This limitation accrues for web depths less than 37 in.

d. G-4, G-5 and G-6 - The High Capacity System - These tables look at shift performance for three conditions of operation:

    1. Where mining conditions impose no restrictions (G-4)

    2. Where difficult conditions restrict shearer speed to 13 ft/min (G-5)

    3. Where good conditions allow the shearer to operate at 20 ft/min (G-6)

e. G-7, G-8 and G-9 - The Very High Capacity System - As in the case of the high capacity system, these tables look at performance of the very high capacity system for the three conditions of operation.

These tables of performance show that production per shift always increases with increasing web depth for all equipment systems even when mining conditions restrict shearer speed to 13 ft/min. As would be expected, the number of cycles per shift decreases with increasing web depth. This is because in cases where shearer capability or conveyor capacity determines the rate of mining, shearer speed decreases with increasing web depth (for a constant height of extraction). Where mining conditions limit shearer speed to a constant, with increasing web depth, the number of cycles per shift still decreases because of the increase in lost time (t_e). However, the percent *decrease* in cycles completed per shift is less than the percent *increase* in tons won per cycle with increasing wep depth. The net effect is therefore an increase in overall production. This same pattern was experienced by the British when they mined 1 meter webs at Holditch and Hem Heath as shown in Tables 6, 8, and 9 of subsection 2.4.2.

The reduced shearer speed, with increasing web, also appears in the tables as a constant increase in percent shearer utilization. Shearer utilization, in this study, is defined as the percent of total available shift time that the shearer is *actually* cutting and loading coal.

$$\text{Percent shearer utilization} = \frac{\text{time shearer in the cutting mode}}{\text{total available time in a shift}}$$

where total available time is total shift time minus lost time off the face.

This is a more meaningful measure of system performance than the definition that is more commonly used. Shearer utilization usually is a measure of the time in a shift in which the shearer is operating as a percentage of total shift time. Shearer operating time in this definition includes tramming time and turnaround time at the entries, basically nonproductive modes of the operating cycle.

The definition as applied to this report is a better measure of efficiency of the mining systems because:

- The system *is not* penalized for time off the face
- The system *is* penalized for the time that the shearer is operating in its nonproductive modes.

It should, however, be recognized that for the results of Tables G-1 through G-9 tramming time (shearer flitting speed) and turnaround time was constant for all web depths. As mentioned in the previous subsection, it could be argued that this may not be the case.

It was previously shown that for a 45-in. web a 20 percent error on both assumed tramming speed and turnaround time only resulted in a 7.5 percent error in projected shift production. In a 6.5 ft seam it is not likely that shearer tramming speed would be restricted. It is more likely, however, that turnaround might be increased due to the deeper web, but the impact on the results of Tables G-1 through G-9 should be less than 5 percent.

Finally, although the values of Table 17 are assumed values, they are projected from a real data base and increases in lost time were logically assigned to factors likely to be effected by an increase in web depth.

The potential for each shearer-conveyor combination is shown in Figure 28 where production per shift is plotted against depth of web. The impact of mining conditions is also shown by the curves of constant shearer speeds of 20 and 13 ft/min. For example, where mining conditions limit shearer speed to 13 ft/min, the contemporary high capacity system should not mine a web depth of less than 38 in. since lesser web depths do not use the full potential of this equipment (the conveyor capacity of 700 tons/hr). Under the same conditions, the full capability of the 300 kW shearer cannot be used at a web depth of less than 49 in. Since the maximum drum width that this shearer can mount is 48 in. due to structural limitations, the best performance this shearer can achieve is 2100 tons/shift at a 45-in. web depth. For good mining conditions, where shearer speed of 20 ft/min can be achieved, shift productions would increase to 2200 tons/shift using the full capability of the 300 kW shearer.

The larger, soon to be available, 310 kW shearer can mine a 57-in. web with 60-in. drums. However, this capability does not increase shift production significantly as the curve D-D shows. The deeper web only increases production 6.3 percent to 2340 tons/shift. This suggests that in 6.5 ft of coal, the greatest potential for increasing shift production is by increasing the capacity of the face conveyor from 700 to 1000 tons/hr. This is clearly shown by comparing the curves C-C and A-A. Both systems use the same 300 kW shearers, but system A is performance-limited by the 700 tons/hr face conveyor.

Table 19 summarizes the maximum potential capability of each system at its maximum web depth as a percent increase in shift production compared to the contemporary high capacity system mining a 27-in. web. In this case the performance of the contemporary high capacity system is upgraded to 1535 tons/shift, loading the conveyor at 700 tons/hr. This production was used rather than the 1306 tons/shift reported in Table 16, because the full potential of the 700 ton/hr conveyor was not being utilized. These reference points are also shown in Figure 28.

LEGEND:

SYSTEM A - CONTEMPORARY HIGH CAPACITY
SYSTEM B = LOW CAPACITY
SYSTEM C = HIGH CAPACITY
SYSTEM D = VERY HIGH CAPACITY

SHEARER (kW)	AFC (tons/hr)
300	700
170	700
300	1000
310	1500

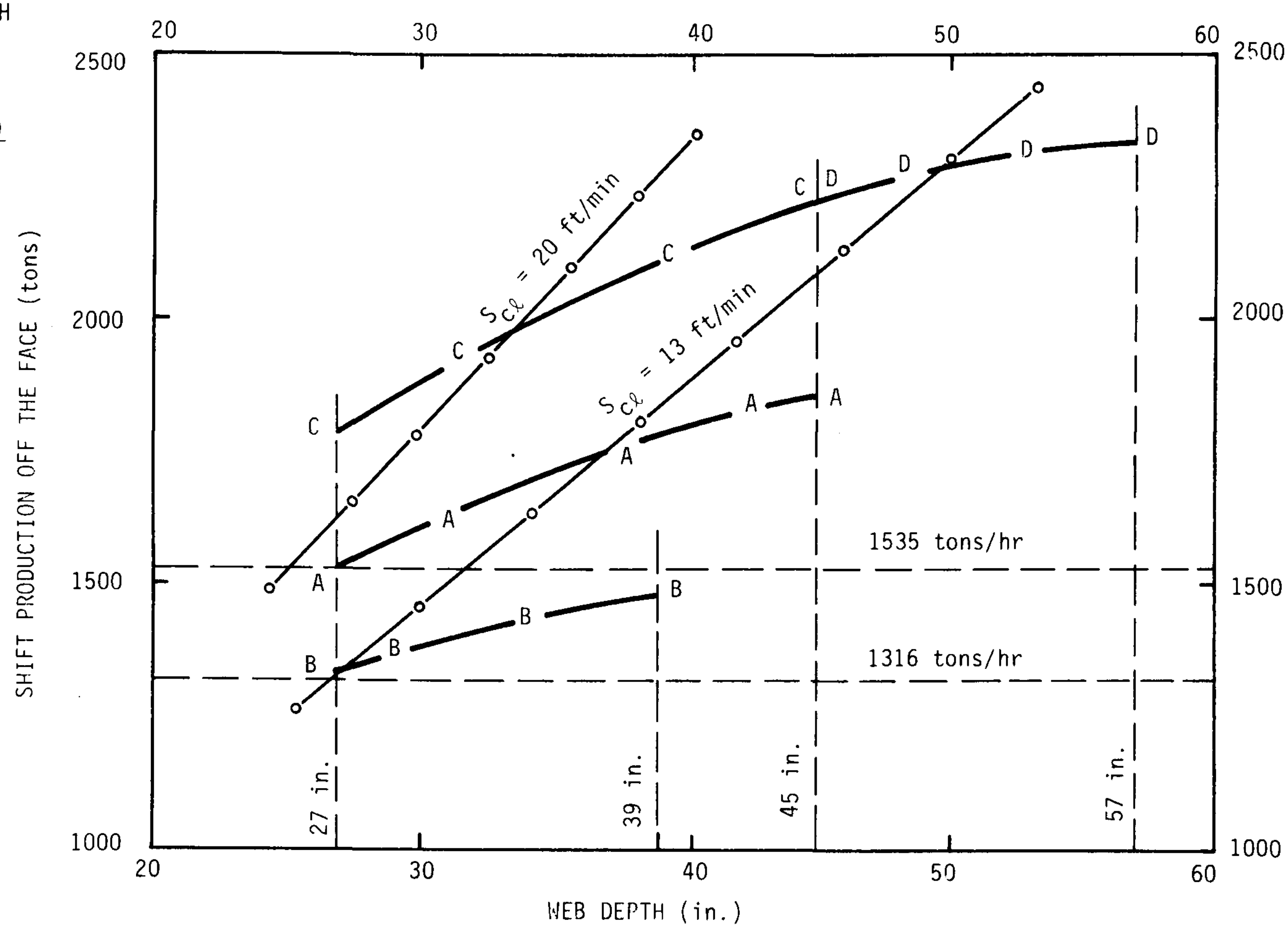


FIGURE 28. - Shift production versus web depth in a 6.5-ft seam.

TABLE 19. - Maximum potential of mining systems at maximum web depth in 6.5-ft seam

System	Maximum web depth (in.)	Percent increase in shift production
Contemporary high capacity	45	21.3
Low capacity	39	2.9
High capacity	45	44.1
Very high capacity	57	52.3

Finally, the performance of the low capacity system of 170 kW shearer and 700 tons/hr conveyor (curve B-B) is always limited by shearer power. It was included in this study so as to compare the capability of a low cost system, mining a wider web, against the contemporary high capacity system taking a conventional web. This comparison appears in the next section as part of the economical analysis.

5. ECONOMICS OF WIDE WEB MINING

The economic analysis of this section establishes the cost effectiveness of mining wider webs over a range of possibilities previously established by the work of Sections 3 and 4. Section 3 established the requirements for roof supports over the full range of web depths and for roof conditions identified as best, intermediate and worst case. Section 4 established the potential productivity of four shearer-conveyor combinations over the same range of web depths.

In this section the additional cost factors associated with the wider webs are applied to the productivity analysis of Section 4 to establish the relative cost per ton of mining various depths of web under varying conditions. The economic analysis here is limited to the cost per ton off the face which involves the operating face equipment and personnel and the additional cost of increased section coal transport (outby haulage). Estimates are also included for the additional cost of maintenance associated with the additional tonnage mined as the web depths increase.

To establish the cost of mining over a range of possibilities, four systems were investigated. The lowest cost system consisted of the smallest, least expensive shearer (170 kW capacity) with the least expensive type of support (the six-leg chock). To further minimize the cost of this system, the capacity, and thus the cost of the supports, was established as being for the *best* roof conditions. The highest cost system involved the use of the 310 kW shearer with the largest face conveyor (1500 tons/hr capacity). This face is equipped with the most expensive type of support (the four-leg shield) requiring the maximum capacity under the worst-case roof conditions.

Also included in the tables of this section are some projected costs involving annual production. This analysis includes consideration of the shifts per year required to move the equipment to a new panel, but does not include consideration of the cost of or time required to develop the panels. Entry development is considered in the work of Section 7, which establishes 12 different strategies for mining wider webs as well as FMA's detailed recommendation for an underground demonstration.

5.1 Equipment Systems

The productivity increases established in Section 4 certainly favor a resulting reduction in cost per ton. However, it remains to be shown under what conditions and at what web depths the range of systems investigated can be expected to minimize the cost per ton.

The systems identified for analysis, therefore, are:

a. The Low Capacity System - a 170 kW shearer with a conveyor of 700 ton/hr rated capacity. The supports selected for this system were 450 ton, six-leg chocks since they are the least expensive supports. This system, therefore, represents the least capital intensive longwall system operating under the best roof conditions.

b. The Contemporary High Capacity System - the 300 kW shearer with 700 ton/hr capacity conveyor and 350 ton, two-leg shield supports. This is the equipment system identified in Table 17 of subsection 4.1.

c. The High Capacity System - a 300 kW shearer with a 1000 ton/hr rated capacity face conveyor. This combination represents the best contemporary equipment available in today's market. This system uses 500-ton, four-leg shields suited for intermediate roof conditions.

d. The Very High Capacity System - a system employing a "500 hp" shearer designed to mount 1.5m wide drums (60 in.). This shearer is mounted on a high capacity (1500 ton/hr) face conveyor having a width of 1m. The supports selected for this system were 700-ton, four-leg shields recommended in subsection 3.1.3 for the most severe roof conditions. This combination, therefore, represents the most expensive mining system.

The maximum drum widths (and, therefore, web depths) were estimated by the equipment manufacturers, as was the cost of modifications to ranging arms. ranging arm mountings and, where necessary, the under-frame of the machine to accommodate these drums.

In the case of the 300 kW machine, the cost of the Eickhoff EDW-300 was used even though it is more expensive than the Anderson Mavor AM500, since the Eickhoff machine is more widely used in the United States. Both Eickhoff and Anderson Mavor agreed that these machines, as presently designed, could mount the 48-in. drums.

The 310 kW wide drum machine is a new design. At the present time, a prototype machine has been built, and is presently undergoing surface trials at MRDE Swadlincote.

The cost of all three shearers with modifications were estimated for double-ended ranging drums for bidirectional operation. Productivity figures for these machines (Section 4 and Appendix G) were for bidirectional cutting, using the half face or modified half face sumping method.

The 700 ton/hr conveyor is the most common size of conveyor presently used in the contemporary United States longwalls. The 1000 ton/hr conveyor represents state-of-the-art. This conveyor should be the conveyor for high production longwalls in the very near future. The 1500 ton/hr conveyor is being developed in the United Kingdom and is in the prototype stage.

The type of supports assigned to the four systems reflect both the maximum spread in the cost of support types and the recommendations of subsection 3.1.5 The most expensive type of support is the four-leg shield and the least expensive is the six-leg chock. Table 14 in subsection 3.1.5 rated the four-leg shield and the six-leg chock as being the best of the conventional supports for wide webs.

The estimated costs of these various supports were provided by the support manufacturers and represent costs in effect as of the second quarter of 1978. They include the costs of special wide web features, such as extensions to the roof canopies (where required), hydraulically actuated forepoling and lengthened advancing rams. Additionally, the cost of all supports reflect the additional capacity required for both the deeper web and type of roof. These capacities were established in Tables 10, 11 and 12 of subsection 3.1.3.

To the cost of the shearers, face conveyors and supports was added the cost of outby haulage, electrical equipment and stage loaders as summarized in Tables 16, 20, 21, and 22. The cost of the outby haulage systems (conveyor belts and stage loaders) reflect the additional capacity, where required, to accommodate the additional tonnage mined from the wider webs. All costs reflect those existing in 1978.

5.2 Method of Analysis

The economic analysis prepared for this report is comparative in nature. The owning and operating costs associated with the three selected systems are compared with the costs of a contemporary high production system taking a standard 27 in. web under intermediate roof conditions.

In all of the shearer-conveyor combinations analyzed in Section 4, production per shift and annual production increased with increasing web depth for the total range of webs. The cost

TABLE 20. - Equipment cost breakdown - low capacity system

Subsystem	Quantity	Description	Total cost
Shearer webs (27 to 39 in.)	1	Eickhoff EDW-170-L double-ended ranging drum shearer with chainless haulage	$ 480,000
Face conveyor	1	Eickhoff EKF-3 armoured face conveyor, single strand, 26 mm, 2 x 150 hp	257,000
	1	Set of face accessories for an EKF-3 conveyor including chainless haulage	460,000
Stageloader	1	Eickhoff EKF-3 stageloader, 2 x 50 hp	81,000
Roof supports	130	Gullick Dobson 450-ton, six-leg chock supports, 9.71 ton/ft^2	2,013,700
	2	High pressure hydraulic pumps with 200 gal storage tank, 2 x 50 hp	47,000
Conveyor belt	1	42 in. conveyor belt drive, 250 hp	32,557
	1	Long airdox "super 300" takeup, 500 ft storage	22,653
	1	Long airdox self-advancing tailpiece	37,495
	583	Top belt rollers at $42.95 each	25,040
	292	Bottom belt rollers at $35.64 each	10,407
	7000 ft	5/8 in. wire rope at $0.49/ft	3,430
	7000 ft	42 in. conveyor belt at $9.33/ft	65,310
Electrical equipment	1	Belt power center	17,000
	1	750 kVA section power center	29,933
	1	Set of electrical controls	59,705
	1	Set of conveyor controls and communication system	31,556
	1	Set service machine lighting system	61,628
	1	Set electrical cables	26,200
Miscellaneous	25	Dowty 25-ton props with handles	7,375
		Miscellaneous tools, connectors, hoses, etc.	10,000
Total equipment cost			$3,778,989

TABLE 21. - Equipment cost breakdown - high capacity system

Subsystem	Quantity	Description	Total cost
Shearer webs (27 to 45 in.)	1	Eickhoff EDW-300-L double-ended ranging drum shearer with chainless haulage	$ 761,000
Face conveyor	1	Eickhoff DMKF-4 armoured face conveyor twin inboard, 2 x 30 mm, 2 x 300 hp	300,000
	1	Set of face accessories for a DMKF-4 conveyor including chainless haulage	460,000
Stageloader	1	Eickhoff EKF-3 stageloader, 2 x 50 hp	81,000
Roof supports	102	Dowty 500-ton, four-leg shield supports	2,958,000
	1	Set high pressure Hauhinco pumps	78,000
	2	100 hp, 950V, 60 Hz electric motors	6,200
Conveyor belt	1	48-in. conveyor belt drive, 400 hp	61,111
	1	Long airdox "super 300" takeup unit	31,500
	1	Long airdox self-advancing tailpiece	46,000
	583	Top belt rollers at $51.20 each	29,850
	292	Bottom belt rollers at $40.55 each	11,841
	7000 ft	5/8-in. wire rope at $0.49/ft	3,430
	7000 ft	48-in. conveyor belt at $12.35/ft	86,450
Electrical equipment	1	Belt power center	17,000
	1	1000 kVA section power center	31,000
	1	Set of electrical controls	59,705
	1	Set of conveyor controls and communication system	31,556
	1	Set of service machine lighting system	61,628
	1	Set of electrical cables	26,200
Miscellaneous	25	Dowty 25-ton props with handles	7,375
		Miscellaneous tools, connectors, hoses, etc.	10,000
Total equipment cost			$5,158,346

TABLE 22. - Equipment cost breakdown - very high capacity system

Subsystem	Quantity	Description	Total cost
Shearer webs (27 to 57 in.)	1	Mining supplies, 500 hp double-ended ranging drum shearer with chainless haulage	572,000
Face conveyor	1	Mining supplies, meter wide, high capacity conveyor with accessories including chainless haulage, 2 x 300 hp	900,000
Stageloader	1	Mining supplies, high capacity stageloader	150,000
Roof supports	102	Dowty 700-ton, four-leg shield, 14.80 ton/ft^2	3,366,000
	1	Set of Hauhinco hydraulic pumps and emulsion container	78,000
	2	100 hp, 950V, 60 Hz electric motors	6,200
Conveyor belt	1	48-in. conveyor belt, 400 hp	61,111
	1	Long airdox "super 300" takeup unit	31,500
	1	Long airdox self-advancing tail-piece	46,000
	583	Top belt rollers at $51.20 each	29,850
	292	Bottom belt rollers at $40.55 each	11,841
	7000 ft	5/8 in. wire rope at $0.49/ft	3,430
	7000 ft	48-in. conveyor belt at $12.35/ft	86,450
Electrical equipment	1	Belt power center	17,000
	1	1250 kVA section power center	31,000
	1	Set of electrical controls	59,705
	1	Set conveyor control and communications system	31,556
	1	Set service machine lighting system	61,628
	1	Set electrical cables	26,200
Miscellaneous	25	Dowty 25-ton props with handles	7,375
		Miscellaneous tools, connectors, hoses, etc.	10,000
Total equipment cost			$5,586,846

per ton could, therefore, be expected to decrease for all systems for the full range of webs, provided that the owning and/or operating costs did not increase substantially so as to negate the gains in productivity.

It is important to note that the number and sequence of operations at the face do not change when a wider web is mined. The number of, and the skills of the operating people are, therefore, not affected and the operating labor costs on the face remain constant.

This was not, however, considered to be true for equipment maintenance hours. Both maintenance equipment and labor costs were calculated on a per ton of coal mined basis. Therefore, with increasing tons per shift mined from the wider webs, maintenance costs per shift increased.

A typical analysis appears in Appendix B for the low capacity system which uses the 170 kW shearer. The summary of results for the four systems analyzed appears in Tables 23 through 26 and the curves of Figure 29.

5.3 Comparative Results

The purpose of this analysis is to determine what cost benefits can be derived from mining wider webs. There are a great number of factors that would have to be considered in making this determination for a particular mine site. This study has, of necessity, been limited to investigating four mining systems of varying capability for varying depths of web. The study has also included as variables two conditions of mining and three conditions of roof strata. These variables have been applied to a 6.5 ft seam height where the shearer is operating in the half face sumping mode along a 500 ft long face.

The mode of operation, seam height and face length were established as a result of the mine survey of high production contemporary United States longwalls. This approach was taken because it was felt that the study should reflect the impact that mining wider webs would have on contemporary operations employing the best of today's longwall equipment. Obviously the increases in productivity and the decreases in the cost of mining would have been considerably more dramatic if the results of this study had been compared against existing operations of lesser capability using older equipment.

TABLE 23. - Cost per ton per shift and per year with varying web depths - low capacity system

	Web depth (in.)				
	27	30	33	36	39
Annual owning cost:					
Capital investment	$3,778,989	$3,995,743	$4,017,496	$4,039,250	$4,061,004
Depreciation	377,898	399,574	401,750	403,925	406,100
Interest, Taxes and Insurance	334,640	350,958	351,905	350,902	352,792
Total	712,565	750,532	753,655	754,827	758,892
Annual operating cost:					
Labor	436,930	436,930	436,930	436,930	436,930
Supervisory	136,800	136,800	136,800	136,800	136,800
Power and Water	16,743	16,789	16,875	16,868	16,939
Maintenance:					
Parts	271,597	279,841	286,500	292,272	296,202
Labor	74,072	76,320	78,136	79,710	80,782
Total	936,142	946,680	955,241	962,580	967,653
Annual production (tons)	493,813	508,802	520,907	531,403	538,549
Cost per ton per annum	3.34	3.33	3.28	3.23	3.20
Shift performance:					
Mining shifts per year	369	366	365	362	362
Shift production (tons)	1,342	1,391	1,431	1,466	1,490
Cost per ton per shift	3.33	3.33	3.37	3.24	3.20
Capital cost per ton production	1.44	1.48	1.45	1.42	1.41

TABLE 24. - Cost per ton per shift and per year with varying web depths - contemporary high capacity system

	Web depth (in.)						
	27	30	33	36	39	42	45
Annual owning cost:							
Capital investment	$4,614,928	$4,842,720	$4,881,811	$4,920,903	$4,959,995	$4,999,086	$5,038,178
Depreciation	461,493	484,272	488,181	492,090	495,999	499,909	503,818
Interest, Taxes and Insurance	407,590	423,060	419,445	414,536	416,506	415,124	418,370
Total	869,083	907,332	907,626	906,626	912,505	915,033	922,188
Annual operating cost:							
Labor	436,930	436,930	436,930	436,930	436,930	436,930	436,930
Supervisory	136,800	136,800	136,800	136,800	136,800	136,800	136,800
Power and Water	19,752	19,400	18,932	18,454	19,720	18,264	18,145
Maintenance:							
Parts	266,501	289,309	310,742	331,670	342,121	348,646	354,358
Labor	72,682	78,902	84,748	90,456	93,306	95,085	96,643
Total	932,665	961,341	988,152	1,014,310	1,028,877	1,035,725	1,042,876
Annual production	484,547	526,016	564,986	603,037	622,039	633,901	644,287
Cost per ton per annum	3.72	3.55	3.36	3.19	3.12	3.08	3.05
Shift performance:							
Mining shifts per year	368	364	358	351	350	346	346
Shift production	1,312	1,448	1,580	1,713	1,781	1,824	1,862
Cost per ton per shift	3.73	3.55	3.35	3.19	3.11	3.09	3.05
Capital cost per ton production	1.79	1.72	1.61	1.50	1.47	1.44	1.43

TABLE 25. - Cost per ton per shift and per year with varying web depths - high capacity system

	Web depth (in.)						
	27	30	33	36	39	42	45
Annual owning cost:							
Capital investment	$5,158,846	$5,386,638	$5,425,729	$5,464,821	$5,503,913	$5,543,004	$5,582,096
Depreciation	515,885	538,664	542,573	546,482	550,391	554,300	558,210
Interest, taxes and insurance	442,010	449,892	444,042	444,618	445,156	444,327	446,121
Total	957,895	988,556	986,615	991,100	995,547	998,627	1,004,331
Annual operating cost:							
Labor	436,930	436,930	436,930	436,930	436,930	436,930	436,930
Supervisory	136,800	136,800	136,800	136,800	136,800	136,800	136,800
Power and water	26,233	25,390	24,700	24,723	24,730	24,656	24,703
Maintenance:							
Parts	317,580	343,492	367,982	381,470	390,305	398,042	404,717
Labor	86,613	93,680	100,359	104,037	106,447	108,557	110,377
Total	1,004,156	1,036,292	1,066,771	1,083,960	1,095,212	1,104,985	1,113,527
Annual production	577,419	624,531	669,059	693,581	709,646	723,713	735,849
Cost per ton per annum	3.40	3.24	3.07	2.99	2.95	2.91	2.88
Shift performance:							
Mining shifts per year	357	348	341	339	337	334	333
Shift production (tons)	1,623	1,790	1,954	2,047	2,108	2,164	2,212
Cost per ton per shift	3.39	3.25	3.08	2.99	2.94	2.91	2.88
Capital cost per ton production	1.66	1.58	1.47	1.43	1.40	1.38	1.36

TABLE 26. - Cost per ton per shift and per year with varying web depth - very high capacity system

	Web depth (in.)										
	27	30	33	36	39	42	45	48	51	54	57
Annual owning costs:											
Capital investment	$5,586,846	$5,780,669	$5,817,029	$5,855,621	$5,894,804	$5,932,213	$5,971,396	$6,009,988	$6,048,579	$6,087,171	$6,125,763
Depreciation	558,685	578,067	581,703	585,562	589,421	593,280	597,140	601,000	604,858	608,717	612,576
Interest, taxes and insurance (average)	536,532	531,065	523,655	524,038	524,380	523,114	524,941	525,160	528,532	530,297	530,425
Total	1,085,217	1,109,132	1,105,358	1,109,600	1,113,800	1,116,394	1,122,080	1,126,160	1,133,390	1,139,014	1,143,000
Annual operating cost:											
Labor	436,930	436,930	436,930	436,930	436,930	436,930	436,930	436,930	436,930	436,930	436,930
Supervisory	136,800	136,800	136,800	136,800	136,800	136,800	136,800	136,800	136,800	136,800	136,800
Power and water	28,450	27,529	26,775	27,979	26,804	26,722	26,772	26,728	26,731	26,789	26,671
Maintenance:											
Parts	317,580	343,492	367,982	381,469	390,305	398,042	404,717	410,363	415,143	418,940	421,908
Labor	86,613	93,680	100,359	104,037	106,447	108,557	110,377	111,917	113,221	114,256	115,066
Total	1,006,373	1,038,431	1,068,846	1,086,033	1,097,286	1,107,051	1,115,596	1,122,738	1,128,825	1,133,715	1,137,375
Annual production	577,419	624,531	669,059	693,581	709,646	723,713	735,849	746,115	754,805	761,710	767,105
Cost per ton per annum	3.62	3.44	3.25	3.17	3.12	3.07	3.04	3.01	3.00	2.98	2.97
Shift performance:											
Mining shifts per year	357	348	341	339	337	334	333	331	330	330	328
Shift production (tons)	1,623	1,790	1,954	2,047	2,108	2,164	2,212	2,253	2,288	2,316	2,338
Cost per ton per shift	3.61	3.45	3.26	3.16	3.11	3.08	3.04	3.02	2.97	2.97	2.97
Capital cost per ton production	1.88	1.78	1.65	1.60	1.57	1.54	1.52	1.51	1.50	1.50	1.49

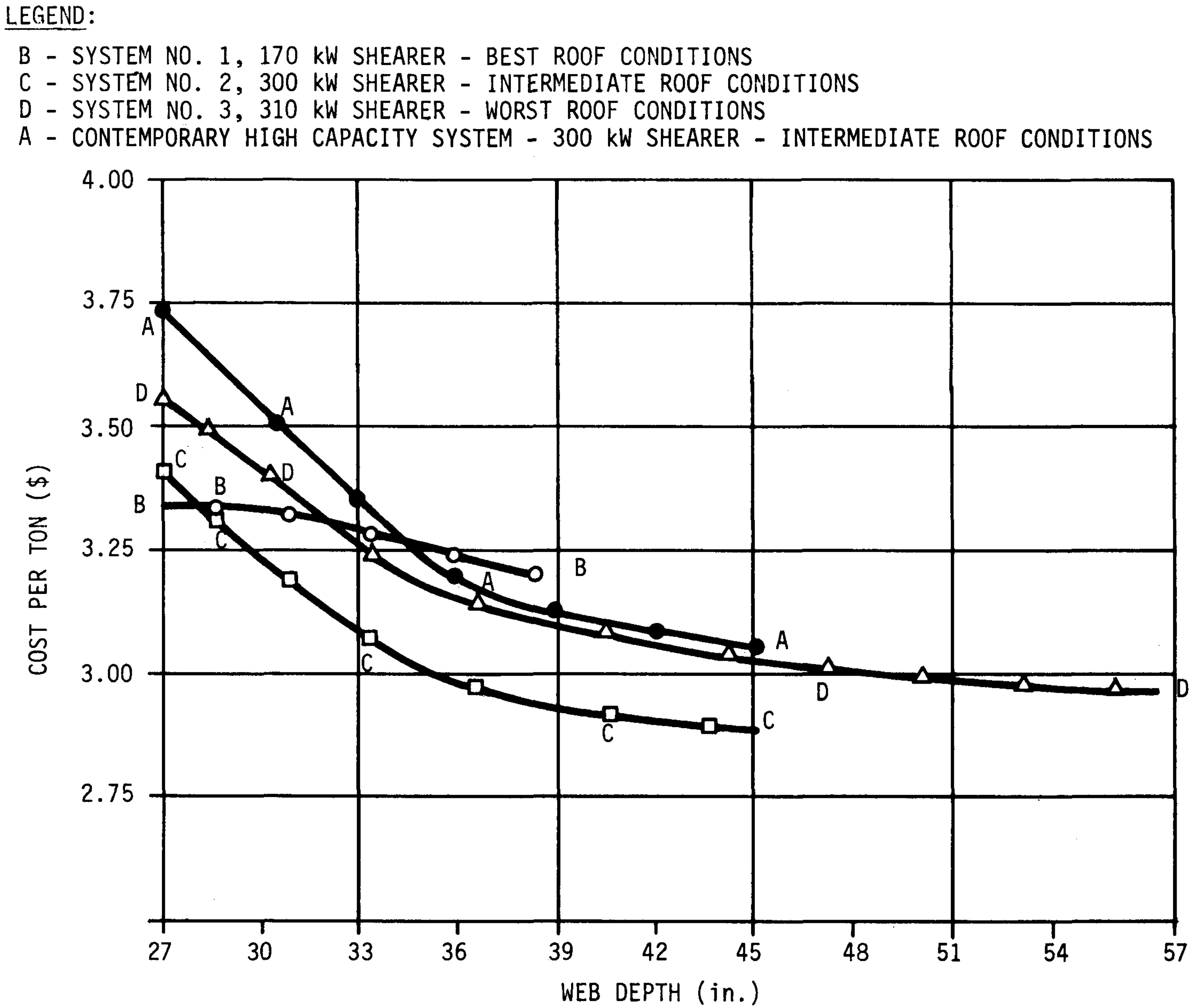


FIGURE 29. - The comparative range of costs associated with wide web mining.

In arriving at the potential cost benefits of mining wider webs, it was first required to determine under what conditions the four mining systems should be used. This determination was made from the curves of Figure 28 as follows:

a. The Low Capacity - Curve B-B, in Figures 28 and 29 - Under all conditions of mining, productivity is limited by shearer power and productivity is always less than it is for the other four systems as shown in Figure 28. It therefore remains to be determined if these are conditions under which this system mines coal at a lower cost per ton. Figure 29 shows that the cost per ton consistently decreases with increasing web depth, but at a rate that is less than the other three systems considered. This is because the curve for increasing production (Figure 28, curve B-B) is relatively flat.

   The cost per ton is lower than for the other systems for narrow webs as shown in Figure 29. However, it is unlikely that this system would mine coal more economically than the high production system, even in the region of narrow webs, for equal conditions of roof strata. This is a valid assumption since the cost of supports is the major capital expenditure on a longwall face.

b. Contemporary High Capacity System - Curve A-A, Figures 28 and 29 - Under the conditions of mining defined as difficult where shearer speed is limited to 13 ft/min, the contemporary high capacity system is more productive than the low capacity system over the entire range of web depths as also shown in Figure 28. However, the capability of this system is not utilized at web depths of less than 36 in. This reflects itself in the cost per ton curves of Figure 29, where at web depths from 27 to 36 in. the cost per ton decreases very rapidly with increasing web depth due to the rapid increase in productivity along the line of constant shearer speed in Figure 28. The cost per ton continues to decrease, from 36 to 45 in., but at a decreased rate due to the productivity curve leveling out in this region. From 36 to 45 in. productivity is limited by conveyor capacity, not by constant shearer speed.

   The contemporary high capacity system represents the combination of shearer and conveyor that should be used in a 6.5 ft seam when mining conditions limit shearer performance.

However, if conditions allow, this system should mine a minimum web depth of 45 in. for minimum cost per ton. Under the conditions as defined, the system should not mine a depth of web less than 36 in.

c. The High Capacity System - Curve C-C, Figures 28 and 29 - The productivity curves of Figure 28 show that this system is somewhat limited under difficult conditions of mining where shearer speed is limited to 13 ft/min. Under these conditions machine system capability is never used over the entire range of webs. Therefore, the cost per ton curves of Figure 29, for this system, are based on good conditions where shearer speeds of 20 ft/min can be achieved. Even under these conditions full system capability is not utilized unless the web depth exceeds 33 in.

The cost per ton curves of Figure 29 show the high capacity system of 300 kW shearer and 1000 ton/hr face conveyor to be the most cost effective under the conditions limiting this study. When compared to the contemporary high capacity system which uses the same shearer face conveyor of lesser capacity, both productivity and cost effectiveness are considerably improved. It is valid to assume that this would also be the case if the contemporary capacity system were also running under good conditions, since the productivity of the high capacity system would still be substantially greater, particularly at the deeper web depths.

As for all the other systems, maximum productivity and minimum cost per ton are achieved at the maximum web depth of 45 in.

d. The Very High Capacity System - Curve D-D, Figures 28 and 29 - This system extends the depth of web out to 57 in. which results in the maximum production per shift as shown in Figure 28. Unlike the high production system, this equipment can be deployed under difficult mining conditions at web depths greater than 49 in. At depths from 49 to 57 in. full system capability is utilized at shearer speeds equal to or less than 13 ft/min.

Although maximum shift production is achieved by the very high capacity system, the cost per ton is greater than for the high capacity system as shown in Figure 28. However, this is almost entirely due to the worse case roof conditions which increased the cost of the supports. Under equal roof conditions, the cost curve D-D would be, for all practical purposes, an extension of curve C-C into the region of 45 to 57 in. web depths. This would be particularly true if the additional cost of the 1500-ton capacity conveyor were eliminated by using the less expensive 1000 ton/hr conveyor. This is a valid substitution since, as was pointed out in subsection 4.2.1, the full capacity of the larger conveyor was never used with the 310 kW shearer. However, the conveyor would have to be modified to support the larger shearer.

The 310 kW shearer, capable of mounting 60-in. wide drums, should be used where mining conditions limit shearer haulage speed. This machine, when employed with the 1000 ton/hr class of conveyor, will maximize production at minimum cost per ton at web depths greater than 45 in.

Although this analysis is limited to a seam height of 6.5 ft, the comparative results could certainly be expected to apply to thicker seams within the efficient operating range of the shearers considered. This would be particularly true of the improved performance of the 300 kW class of shearer when deployed with a higher capacity conveyor. In the previous section the productivity of this shearer was clearly shown to be limited by the capacity of the contemporary 700 ton/hr conveyor presently in use. As shown in Figure 28, when minimg conditions are good, the difference in performance is substantial and the disparity increases with increasing web depth.

The cost curves of Figure 29 also clearly favor the use of the 300 kW shearer with the 1000 ton/hr face conveyor. This combination forms the basis of the FMA recommendations for an underground demonstration detailed in Section 7. As would be expected, based on the results of this and the previous section, it is also recommended that the shearer mount a 48 in. drum to take a 45 in. web.

The method of analysis developed for application to the economics of this section and the productivity analysis of Section 4, although limited in scope, is applicable to any study being conducted for a specific mine site.

Economics obviously favor the mining of wider webs. The possible impact on health and safety is considered in the section that follows.

6. HEALTH AND SAFETY

The introduction of any new technique into the hostile environment of an underground coal mine requires an assessment of its expected impact on miner's health and safety. This section of the report discusses safety in terms of the potential for wider webs to affect the number and severity of accidents in the longwall face, and the expected effect on methane gas concentrations. Finally, dust concentrations, as influenced by the deeper cut, are discussed as the principal factor affecting the health of the operating people on the face.

The impact of wider webs on equipment, productivity, and costs discussed in the previous sections was presented as both a qualitative discussion and a quantitative analysis. The impact on safety, dust and methane, however, does lend itself readily to a meaningful numerical analysis.

The qualitative discussion presented in this section does identify the factors associated with wider web mining which could be expected to both favorably and unfavorably impact miner health and safety. The best available data was used to reinforce the qualitative discussion if such data existed. Finally, the experience of mining one meter webs in the United Kingdom with respect to the impact on the number of accidents on the face and the affect of wider web mining on dust is referenced (19,20).

6.1 Safety

This subsection discusses the expected impact that mining wider webs is likely to have on accidents happening to shearer operators, jacksetters and mechanics working the longwall face. The data obtained and analyzed was supplied by the United States Department of Labor, Mine Safety and Health Administration (MSHA) relating to the cause, number and severity of accidents occurring during 1977 to personnel engaged in all aspects of longwall coal mining.

The information was in the form of a computer readout of coded statistics and included details which classified the accident as to:

a. Job description of employee

b. Accident type

c. Cause of accident

d. Activity of employee when accident occurred

e. Number of days of disability as a consequence of the accident.

The preceding classifications enabled those accident types to be selected that resulted from operations on the longwall face and, in particular, those accidents that would be influenced by a wider web. The relevant face-related accidents identified fell into the following two general categories:

a. Machinery-related accidents; injuries sustained while operating, moving or repairing machinery at the face

b. Accidents related to fall of roof; injuries sustained by any fall of rock or (coal) from one elevation to a lower elevation.

Accidents were omitted that occurred to face workers not actively engaged on the face (for example, to personnel engaged in entry development or operators injured by the outby haulage equipment, etc.). Mining a wider web was considered to have no impact on the number or severity of these occurrences. Accidents were also omitted where insufficient data was available to positively identify the location and cause of the accident or the personnel involved.

The base data did not allow the distinction to be made between accidents occurring on plow faces from those occurring on shearer faces. Since a plow operator does not follow the machine along the face, all accidents that were identified as occurring to mining machine operators were considered to have occurred on a shearer face.

For the remaining relevant accidents to jacksetters and mechanics, the assumption was made that the likelihood for accidents to occur on shearer faces was equal to that on plow faces. The number of these accidents was, therefore, divided by the ratio 76 to 24, the ratio of shearer to plow faces according to the 1977 USBM Census of Operating United States Longwalls (23).

The number and severity of the accidents that could be positively identified as being relevant to this study are summarized in Table 27. The severity of accidents is reflected by the number of days of disability. For example, the most serious relevant accident occurred to a shearer operator when he

TABLE 27. - Nature of accidents occurring on United States longwall faces in 1977

General category of accident	Activity	Job title	Number of accidents	Days of disability
Machinery-related accidents	Operating shearer	Shearer operator	10	113
	Operating roof supports	Jack-setter	6	25
	Servicing equipment	Mechanic	3	68
	Operating shearer	Shearer operator	1	46
Vertical fall of material-related accidents	Operating roof supports	Jack-setter	11	106
	Servicing equipment	Mechanic	1	0
Overall total			32	358

was struck by the fall of material. However, the base data did not provide for a determination as to whether this was the fall of roof material or coal sloughing from the face.

In order to determine the influence that mining wider webs might be expected to have on these statistics, it was necessary to identify the factors that could be expected to *decrease* the number of accidents (advantages) as well as those likely to *increase* the number of accidents (disadvantages).

6.1.1 Advantages and Disadvantages of Mining Wider Webs

The principal safety related advantages of mining wider webs is associated with the reduced speed of the shearing machine (haulage speed) and related operations, such as advancing the chocks and conveyor. For example, consider a 300 kW shearer that can cut and load at a rate of 1000 tons/hr. If this machine were mining a 27-in. deep web it would be traversing the face at twice the speed required to mine the same tons per hour from a 54-in. web. The chock setters would also be required to perform their duties at a quicker pace to match the faster speed of the shearer.

Additionally, as discussed in subsection 3.2.1, slower haulage speeds require slower drum speeds for equivalent depth of pick penetration. This is likely to reduce the possibility of operators being struck by particles of rock and coal thrown from the cutting picks. The slower speeds would both reduce the velocity of the particles as well as the distance they are likely to be thrown.

As shown in the tables of Appendix G and as discussed in subsection 4.2.2, the number of operating cycles per shift is reduced when mining a wider web. This reduction in cycles occurs even when the tons per shift is increasing. This reduces the number of entry turnaround operations and the total travel of the shearer both cutting and flitting. Although the conveyor and supports must be pushed over a further distance, the number of moves per shift is reduced.

Both the number and severity of machinery-related accidents can be expected to decrease with the wider webs because:

a. The entire cycle of operation can be slowed down including the setting of supports and the advance of the face conveyor.

b. The number of times per shift that each operation has to be performed is reduced.

Table 27 shows that of the 28 relevant accidents occurring to the shearer operators and chock setters, 16 accidents (57 percent) were machinery related. These machinery related accidents can be expected to decrease with increasing web depth.

Table 27 also shows that 12 of the accidents affecting the shearer operators and chock setters were due to the vertical fall of roof rock or coal. Accidents due to coal sloughing from the face is probably more related to the height of seam than to the depth of web. This, however, would not necessarily be true of roof fall related accidents.

Roof control was reported to be improved with deeper webs (9) due to less stress cycling of the immediate roof strata The advancing cycle involves the lowering of the support (destressing the strata), advancing the support and resetting it to the roof (restressing the strata). Reducing the number of times a segment of the immediate roof is subjected to this stress cycling will result in less deterioration of the roof; and therefore, less likelihood of localized roof falls.

This advantage could be offset by the additional span and area of roof exposed by the taking of the deeper cut. This should be somewhat neutralized, however, by the deployment of some form of immediate forward support as recommended and discussed in subsections 3.1.4 and 3.1.5.

Of the 32 relevant accidents reported in 1977, only four occurred to the face mechanic and only one insignificant accident was a result of vertical fall of material (no lost time). The machinery related accidents occurring to the mechanics might be expected to increase due to the heavier nature of the machinery. The significance of this fact should have a minimal overall impact, however, since this type of accident only represents 9.4 percent of the total accidents identified in Table 27, and though the equipment is heavier, the nature and complexity of the machinery remains relatively unchanged.

An understanding of both the nature of face related accidents (Table 27) and the factors associated with mining wider webs likely to influence the number and severity of those accidents is, in itself, not all conclusive. Hunter (9) reported that on the 1-m (39 in.) web faces in the United Kingdom, conditions were substantially improved due to a reduction in overall effort required per ton of coal won. He further suggested that the increased morale, the slower speed of the shearer, and reduced dust contributed to generally safer working conditions.

These observations were expanded by the Hardman (19) with regard to accidents on the face and by Smales (20) on the subject of dust generated on deep web faces as discussed in subsection 6.2.

6.1.2 One-Meter Web Mining at the Holditch Colliery

The 1-m web operation at the Holditch Colliery was previously discussed in subsection 2.4.2 with the face equipment arrangement of Figure 11. This face commenced production in November of 1974. For the first 2 months the face was operated with 24 in. wide drums and the shearer haulage speed was 14 ft/min. The wide drums of 39 in. were then mounted and the face was mined with a required reduced shearer speed of 10 ft/min.

At the time of Hardman's reporting, accident statistics had been compiled for the wide web operation for the period November 1974 to July 1975. For this period, there were six accidents on the face for a projected rate of 70.7 accidents/100,000 shifts as shown in Table 28.

TABLE 28. - Accident statistics, Holditch Colliery normal and wide web operations

Face	Dates	Pick coverage (in.)	PFF	Accident rate per 100,000 shifts	Total Accidents	System
1s 10 ft	7/70 - 8/71	21-3/4	6 ft, 9 in.	272	77	Advance
2s 10 ft	10/71 - 9/72	24	6 ft, 9 in.	303	43	Retreat
5s 10 ft	10/72 - 9/74	21-3/4	6 ft, 9 in.	301	77	Advance
3s 10 ft	11/74 - 7/75	39 (1-m)	8 ft, 3 in.	70.7	6	Retreat

Average distance with narrow web = 20 to 23 in.

Average advance with wide web = 38 in.

Comparative figures, collected at the same mine are also shown in this table. In all cases, with the narrower webs, the accident rate/100,000 shifts was substantially greater. From these observations, Hardman concludes that there is a direct relationship between the reduced accident rate and the depth of web. The reasons given for the reduction in accidents were:

a. Better roof control with wider webs

b. Reduced speed of the shearer

c. Better overall mining conditions and morale.

In addition to a reduction in stress cycling, better roof conditions were also attributed to better contact between the support canopy and roof. The pattern of contact between the canopy and roof is influenced by the depth of web. The wider the web, the less number of "steps" in the roof/unit of advance. Better control of the roof, attributed to the deeper web, was reported as being particularly important in reducing the potential for accidents since the most dangerous time exists during the chock advance cycle, particularly at the face ends.

The reduced speed of the shearer contributed to a reduction in accidents since the shearer operators and chock setters had more time and were better able to adhere to established practices related to safety.

Finally, morale and operator attitudes were improved due to the better general conditions on the face. This, perhaps, was exemplified by the substantially lower rate of voluntary absenteeism experienced during the reported life of the wide web operation (19).

6.2 Airborne Respirable Dust

In assessing the impact of wider webs on accidents, the qualitative discussion which identified the factors likely to be of influence was supported by a data base (Table 27) and the directly related statistics of Table 28.

For the discussion on dust, no data base exists which identifies where the dust is generated in quantitative terms. Furthermore, although the British-published reports strongly indicate that dust improves with wide webs, there is no published numerical data in direct support of this contention.

In the discussion that follows, the following topics are discussed:

- The principal sources of dust on a longwall face
- The parameters affecting the rate of creation of this dust
- The factors determining the quantity of dust that becomes airborne
- The effect of the introduction of wide web working on the above in terms of the exposure of underground personnel to respirable dust.

The reports of the NCB Western Area wide web trials all address the question of dust and the limited results presented in these will be analyzed to illustrate the implications of wide web.

The impact of changing from standard to wider webs will be assessed in the same way as it is measured under field conditions by estimating the total exposure of the employee during a working shift. This determination should take into account both the quantity of dust generated by each operation and the number of times that operation takes place during a working shift.

6.2.1 Sources of Dust

Dust on a longwall face is generated principally by the operation of three major items of equipment. They are:

- The longwall shearer
- The face conveyor
- The roof support system.

Other sources of dust exist, but can be regarded as constants since they both are relatively minor and not significantly affected by the introduction of wider webs.

The longwall shearer creates dust at the pick tip whenever coal is cut. Cutting of coal by shearer drums employs an action whereby the coal is first locally crushed until sufficient stress is applied for a wedge of coal to be broken out. The crushing process is the principal source of coal dust. The translational rotation of the shearer drum means that as the picks enter and leave the coal, very shallow cuts are inevitable.

It is during this shallow cutting and during the cutting of the clearance circle at the back of the web that the majority of the dust is created.

Dust is also created during the fracture of the coal and during the passage of the coal through the shearer drum vanes. This latter process may result in coal being ground between pick boxes and cowls and additional fracture and grinding of coal as it is circulating in the drum helix.

The face conveyor is responsible for the creation of very little dust though a certain amount of grinding or breakage may take place. The principal source of dust on the coal haulage system is at the transfer points between stage loader and belt.

The roof support system does not in itself directly create dust, but can act as a reservoir on which dust may settle and from where dust may be introduced into the airstream during support move over. This dust is created during the compaction of the roof and breaking of the immediate roof during the forming of the gob. The dust created by the shearer will be principally coal dust though where a rock band, parting, roof or floor is taken rock dust will be present. The dust from the roof support will contain a significant proportion of rock. This dust also tends to have a significant fraction in the nonrespirable size range which though not particularly hazardous contributes a lot to the miners' impression of conditions.

6.2.2 Factors Affecting Dust Creation

Apart from the condition and type of the coal, several factors affect the amount of dust created during coal cutting. Over the last few years it has become apparent that the most significant of these is the depth of cut (or penetration) of the individual picks. Shallow cutting with implied high cutting speeds creates a high proportion of small coal and dust while deep cutting, usually at lower cutting speed, radically reduces dust creation. These conclusions have been drawn by several authors including those reporting on the USBM Microminer/Deep cutting experiments and the MRDE Large Pick Shearer Drum. The other principal design factor controlling dust creation is the conditions prevailing in the clearance circle at the back of the shearer drum. In this area, cutting is harder and the concentration of picks per line is high and penetration low. Special care should be taken to insure adequate clearance for the drum hub in this area to avoid crushing and grinding of the coal.

As has been mentioned above, loading drum design will affect the quantity of dust created during passage of the coal through the loading helix. If loading is inefficient, grinding will take place when the coal is recirculated in the drum.

Airborne Dust

Dust is not a significant problem on the longwall face if it does not become airborne. However, the passage of the coal from the face to the face conveyor offers several opportunities for respirable dust to become airborne. This can occur:

- Close to the cutting site where the dust is formed
- Within the drum, especially if the coal is thrown radially from the vanes by excessive drum speed
- At the point where coal is discharged into the AFC.

The dust present on the roof support canopies or gob shields may be disturbed when support movement occurs. It is reported that flushing shields make little impact on dust from this source.

The amount of dust that becomes airborne will depend upon cutting conditions, the operating condition of the shearer drum and the degree to which proper steps are taken to insure that the dust that is inevitably created during coal cutting does not have the opportunity to become airborne. Principal means of assuring this are:

- The use of pick face flushing with drum-mounted water jets. It appears that the layout of the water jets is not as important as their maintenance and the provision of a reliable supply of clean water at the appropriate pressure.
- The paying of special attention to the clearance ring in terms of pick lacing, water sprays, adequate clearance and ventilation.
- The use of loading doors and cowls which confine the coal within the drum helix. Therefore, assuring efficient loading, further opportunity exists for dust to be dampened by water sprays and partial prevention of ventilation airflow through the shearer drums.

- Optimization of pick "spacing and lacing" to provide adequate penetration and separation. This factor is inevitably linked to drum shaft and haulage speed.

- Provision of water sprays on roof support canopies and gob shields.

The above are the principal means of dust control associated with equipment design. Other factors such as panel layout, ventilation plan and operating method are also important influences which are either not relevant, since they apply equally to wide web and standard web mining, or are covered below in the discussion of the impact of wide web working on dust levels.

6.2.3 The Effect of Wider Web Working on Dust Levels

In a report of this nature, it is inevitable that resort must be made to various project and modeling techniques to determine the effect of the change in a mining parameter. In the case of wide web working, however, there are several reports of actual experience which, while they do not contain a satisfactory rigorous analysis of the effect on dust levels, do contain empirical data and the observations of trained observers.

These reports relate to the wide web trials of the NCB's Western area. The first of these was at Holditch Colliery where 39 in. web drums were fitted in January of 1975.

The background of the United Kingdom wide web trials is usually considered to be dominated by the "two strips per shift" syndrome that limits production at many United Kingdom mines. However, true though this may be, the primary stimulus for the trials at the time that they were conceived *and* the basis on which the HM Inspectorate granted exemptions for 8 ft, 3 in. PFF was the need to reduce chronic levels of airborne dust and the expectation that the use of wider webs would contribute to an improvement. The reasons for this expectation were given by Smales in his paper (20) on dust control at Holditch.

He states that it was argued that improvements may be obtained due to the fact that:

a. Only one pass of the machine is required for 39 in. of coal, whereas over one and a half passes would be required of a 26 in. wide drum.

b. The clearance ring of the drum, accepted as the main area of dust production, would have less cuts per unit ton extracted than with the narrow web.

c. Since the coal is not a hard cutting proposition, it would be practical to design a drum with graded pick lacing and arrange the dust suppression water to be concentrated at the back of the cut.

d. There would be a greater portion of the drum buried in the coal and so less propensity for the ventilation along the face to pick up dust.

e. Chock movement produces a high dust concentration during the short period of operation. By moving them 1 meter as against 24 in., there was a potential reduction in dust due to operation alone.

Smales also refers to the face that for a given production there will be fewer face end operations (turnarounds). He identifies these as being prime dust producers due to the slow cutting of roof coal and proximity to the ventilation when the panel side rib is cut out.

As is often the case, circumstances did not allow direct comparison of dust levels with narrow and wide drums, but the conclusion was drawn that there was no deterioration in dust levels due to the shearer when wide webs were taken and that there was probably an improvement in levels of dust due to support movement.

Smales' overall conclusion was that for a given tonnage a narrow web machine must be active for a higher proportion of the shift and that the shift average levels of dust will be higher when narrow rather than wide webs are taken.

Other United Kingdom trials of wide web working have been reviewed by Hunter (9). Again, there is evidence that levels of respirable dust were reduced through overall output, and output per man shift (OMS) were increased. This contention cannot be assessed directly as once again the experiment was confounded by other changes in face layout; however, the changes that were brought about on the longwalls in the 10 ft seam at Hem Heath resulted in compliance.

The above does not represent a rigorous analysis of the impact of wider webs on dust levels since the data did not allow this analysis and in any case the conditions and measurement methods employed in United Kingdom mines differ from those in the United States. However, it would appear that the use of wider webs will not exacerbate dust problems and may improve conditions even if production is increased. This conclusion is supported by the fact that the taking of wider webs will necessitate the use of state-of-the-art equipment and techniques for cutting and loading coal.

6.3 Methane

This section discusses the principal sources of methane liberation on longwall faces and the effect that mining wider webs can be expected to have on the rate of methane release.

6.3.1 Sources of Methane Liberation

Methane is capable of liberation into the air of the working area by the following mechanisms:

a. Liberation from cut coal - methane previously trapped in the coal is released by desorption into the working area as the coal is cut

b. Liberation from face - methane in the coal bed is capable of seepage out of the face and released into the working area

c. Liberation from gob material - methane in the overlying strata is released into the gob and spills into the working area as the roof falls.

6.3.2 Effect of Introducing Wider Webs

The introduction of wider web cutting techniques will affect the liberation of methane due to each of the preceding three mechanisms:

a. Liberation from cut coal - an increase in the rate of cutting as a result of wider web increases the rate of methane liberation. This should increase proportional to the increase in the rate of mining.

b. Liberation from face - since the web is wider, the rate of exposure of new face area is less. However, since deeper cutting encroaches on the pressure gradient curve more rapidly than shallow cutting, the flow of methane from the coal bed into the face area may increase slightly as a result.

c. The rate of roof caving (ft^2/hr) is directly related to the rate of coal extraction for a given height of seam. If the rate of methane liberation is directly related to the roof caving, then the higher advance rates associatd with wider webs will increase the flow of methane from the gob.

Consideration of the mechanisms of methane liberation indicates that the cumulative effect will be an increase in the rate of methane liberation with web depth on a longwall face. However, the principal concern is the effect of methane concentrations in the deeper web and the resulting potential for ignition. The deeper web will require improved ventilation techniques which are available as should be applied. For this reason, the recommendations of Section 7 include the specification of through the shaft ventilation and the possible use of ventilating cowls.

7. RECOMMENDATIONS FOR THE UNDERGROUND DEMONSTRATION

A furthering of the effort to assess the economic, health and safety benefits of mining wider webs would require an underground demonstration. The purposes of this demonstration would be to validate the relationship between web depth and:

a. Cycle time

b. Machinery and geologically related lost time on the face

c. Specific energy of mining

d. Maintenance costs

e. The impact on dust, methane and safety

f. The cost benefits as related to the total economics of a real mine situation

g. Geological or operational limitations that may limit the maximum depth of web.

The remainder of this subsection deals with the consideration of such an underground demonstration. In the subsection that follows, the economic study is expanded to include consideration of panel development, since development was identified in subsection 2.3.2 as being one of the principal constraints to longwall production. The impact of development was applied to eight possible strategies designed to reflect applications of wider web mining. This work is followed by the development of a demonstration plan which includes the scope and methodology of the monitoring program required to establish the relationship between varying web depth and the overall performance of the face. Finally, the particulars of the mine site are identified which would best suit the objectives of the wide web demonstration.

7.1 Economic Strategies

A range of possible economically defined alternatives for wider web longwalling are shown in the matrix of Table 29. The matrix is arranged to reflect:

a. Strategies based on minimizing the cost per ton while maintaining production at today's levels (strategies 1 through 4) and alternatively maximizing production at today's cost per ton (strategies 5 through 8)

b. The impact of limiting the increase in capital investment to 20 percent and the impact of no limitation on equipment costs

c. The application of web depth increases up to 45 in. (48 in. drum) and web depths from 45 to 57 in. (60 in. drum).

Web depths up to 45 in. are available with modifications to shearers presently being used. Web depths over 45 in. could be obtained from soon-to-be-available machines.

Table 30 is derived from Table 29 by inserting values for "best average" production per shift (1316 tons), best average cost per ton ($3.72) and an equipment cost of $5,538,000 which represents the allowable 20 percent increase in capital investment.

These values were taken from Tables 15 and 16 which were previously established as representing the best of today's United States longwall operations. As throughout this report, maximum drum width was established as 60 in. Panel dimensions, equipment identification, shift production, figures and mining costs were thus established for each of these strategies as shown in Table 31.

The economic factors that most influenced the results of Table 31 were:

a. Changes in the cost per ton were most affected by a change in the capital cost of equipment, since the major operating cost (wages) was fixed. The number of operating personnel on the face is unaffected by web depth.

b. The major equipment cost on the face is the cost of the roof support system followed by the cost of the face conveyor and shearer, as shown in Table 32. These cost figures were taken from Table 16 for the contemporary high capacity longwall face. Therefore when the analysis required a change in the cost of mining this change could be most affected by changing the length of the face and/or the type of supports.

c. Maximum production and minimum cost per ton are obtained from mining the maximum web depths, as shown in Figures 28 and 29.

TABLE 29. - Range of economic strategies

Strategy	Maximum drum width (in.)	Maximum equipment cost (%)	Cost per ton off the face	Production per shift
1	48	120[1]	Minimum	Best average[3]
2	48	No limit	Minimum	Best average
3	60	120	Minimum	Best average
4	60	No limit	Minimum	Best average
5	48	120	Best average[2]	Maximum
6	48	No limit	Best average	Maximum
7	60	120	Best average	Maximum
8	60	No limit	Best average	Maximum

NOTES:

1. Equipment cost limited to 120 percent of the cost of contemporary equipment using a conventional web.
2. Cost per ton off the face for the best average of contemporary United States longwall operations.
3. Shift production for the best average of contemporary United States longwalls.

TABLE 30. - The eight economic strategies

Strategy	Maximum drum width (in.)	Maximum equipment cost	Cost per ton off the face	Production per shift
1	48	$5,537,914	Minimize	1316
2	48	No limit	Minimize	1316
3	60	$5,537,914	Minimize	1316
4	60	No limit	Minimize	1316
5	48	$5,537,914	$3.72	Maximize
6	48	No limit	$3.72	Maximize
7	60	$5,537,914	$3.72	Maximize
8	60	No limit	$3.72	Maximize

TABLE 31. - Summary of specifications for selected strategies

	Strategy requirements				Mining dimensions			Equipment			
Strategy	Web depth (in.)	Equipment cost ($1000)	Annual mining cost per ton	Production per shift (tons)	Face length (ft)	Seam height (ft)	Panel length (ft)	Support type	Shearer power (kW)	Conveyor capacity (ton/hr)	Outby belt (in.)
1	36	2,899	3.63	1316	310	6.5	9000	6-leg chock	170	700	42
2	36	3,943	4.00	1316	310	6.5	9000	4-leg shield	170	700	42
3	57	3,996	4.12	1316	480	3.8	9000	6-leg chock	170	700	42
4	57	5,070	4.54	1316	480	3.8	9000	4-leg shield	170	700	42
5	45	5,534	3.72	2376	725	6.5	2060	6-leg chock	300	1000	48
6	45	9,526	3.72	2499	1000	6.5	7200	4-leg shield	300	1000	48
7	57	5,511	3.72	2480	715	6.5	2100	6-leg chock	310	1500	48
8	57	10,436	3.72	2611	1090	6.5	9000	4-leg shield	310	1500	48
9	45	5,537	3.32	2214	500	6.5	6000	4-leg shield	300	1000	48

TABLE 32. - Comparative equipment costs, contemporary high capacity face

Item	Cost	Percent of total cost
Roof supports	$2,754,000	59.7
Shearer	761,000	16.5
Face conveyor	567,000	12.3
Outby belt conveyor	213,892	4.6
Electrical equipment	136,470	3.0
Stageloader	81,000	1.8
Miscellaneous	101,575	2.2
Total	$4,614,937	100

d. Face length, for the various strategies, was generally determined by the required cost per ton for reasons explained in b. above.

e. Panel development generally was the principal determinant in establishing panel length.

f. For a given seam height, web depth system capacity and face length determine shift tonnage and thus rate of mining the panel. Increased depth of web, increased system capacity and shorter faces require that panel development progress move rapidly.

g. From the USBM survey of operating longwalls (23) the minimum length of face mined in the United States is 300 ft. This will be established as the minimum that will be considered in this study. Longer faces will be applied where possible to insure a reasonable percent resource recovery.

Maximum face width is most often restricted by the capability of the face conveyor or by considerations for maintaining a reasonably straight face. The latter is most difficult to ascertain for a generalized study so conveyor limitations dictated. This limitation will be: 600 ft for the small (700 tons/hr) face conveyor, 1000 ft for the intermediate (1000 tons/hr) conveyor, and for the largest conveyor (1500 tons/hr), 1800 ft as estimated by the manufacturer.

h. The longest panel mined in the United States in 1977 (4) was 6000 ft. The length of panel is usually limited by the time required to develop the panel, the maximum operational length of section conveying belt, or the man travel time to and from the operating face. For this study, the maximum length of panel will be extended to 9000 ft taking these limiting factors into account. The shortest panels were seldom less than three times the face width, which will be adhered to as the minimum in this analysis.

i. To minimize the complexity of the analysis, the rate of panel development was fixed as a constant 50 ft/shift. This would represent the activity of one continuous miner cutting a four-entry system. Entry widths are 18 ft on 60 ft centers with crosscuts on 100 ft centers.

Assuming 30 shifts to move to a new panel, this system of entry driving can develop panels per year as shown in Table 33. Since a different number of shifts are required to move the longwall face than the development section, mining and development must be compared in a panels-per-year basis.

TABLE 33. - Entry development schedule, panels per year and tons mined

Panel Length (ft)	Move Time (Shifts)	Development Tons per Panel	Panels per Year	Development Tons per Year
3000	30	92,169	1.74	160,374
5000	30	147,557	1.12	165,264
7000	30	202,945	0.82	166,415
9000	30	258,333	0.65	167,916

The application of these principles can best be understood by considering the development of strategy 1. Similar discussions involving the development of the other seven strategies appear in Appendix I. A sample economic analysis, typical of the type of analysis used throughout this report, is shown in Appendix H. The range of equipment available for application to this work was previously applied to the productivity and economics of Sections 4 and 5.

7.1.1 Discussion of Strategy 1

Table 30 requires that the face, identified as strategy 1, produce 1316 tons/shift from a web depth of less than 45 in. The face equipment cost must not exceed $5,538,000 and the cost per ton off the face must be minimum.

Curve B-B of Figure 29 shows that the combination of small (170 kW) shearers and 700 tons/hr capacity face conveyor can mine 1300 to 1400 tons/hr at web depths of 27 to 39 in. This combination represents the least expensive equipment available which is a logical choice to minimize the cost of mining. To insure that the cost per ton will be minimized, this face will be equipped with the least expensive six-leg chock supports. This infers that good roof conditions exist. Finally, the outby section haulage requirements of 1316 tons/hr can be handled by a 42-in. wide belt conveyor. The complete equipment package is summarized with costs in Table 34.

The face width will now be established as the shortest required, so as to further minimize the cost of the supports and face conveyor. The length of the panel will be selected to match the rate of entry development once the face width has been selected. The minimum face width requirement that will yield 1316 tons/shift would be achieved by mining the maximum web depth of 39 in.

By inserting the appropriate values into the equation of Table 3, the required 1316 tons/shift can be obtained by mining the 39 in. web from a 286-ft long face. However, this is less than the minimum acceptable face width of 300 ft.

A web depth of less than 39 in. will be required to mine the 1316 tons from a face width of greater than 300 ft. The required lengths of face were determined for the range of web depths 30 to 39 in. as shown in the first two columns of Table 35. Again, these dimensions were arrived at by inserting the appropriate values in the shift production equation of Table 3.

TABLE 34. - Summary of capital investment expenditures for strategy No. 1

Quantity		Total cost
1	Eickhoff EDW-170L shearer w/42 in. drums	$ 500,000
1	Eickhoff EKF-3 face conveyor w/accessories	383,080
1	Eickhoff EKF-3 stageloader	81,000
80	Gullick six-leg, 510 ton chocks at $16,500 each	1,292,000
1 set	Set of pumps complete with storage tanks	47,000
1	Long airdox, 42 in. belt drive	32,557
1	Long airdox, "Super 300" takeup	22,653
1	Long airdox, self-advancing tailpiece	37,495
1	Belt power center	17,000
1	Section power center, 750 kVA	19,000
1 set	Control boxes	59,705
1 set	Emergency stop and communication system	31,556
1 set	Electrical cables	26,209
25	Dowty hydraulic props with handles	7,375
	Miscellaneous tools, etc.	10,000
	Subtotal capital investments	$2,566,630
	Capital investment for panel lengths greater than 6000 ft (additional)	
1	Long Airdox, 42 in. belt drive	32,557
1	Belt power center	17,000
	Capital investment for outby haulage belt at 9000 ft panel length	
		282,796
	Total capital investment	$2,898,983

TABLE 35. - Rate of production for alternate lengths of face over various panel lengths

Face Length (ft)	Web Depth (in.)	Move Time (Shifts)	Panel Length (ft)	Tons/Panel	Panels/Year	Tons/Year
286.0	39	44	3000	250,965	1.87	469,305
			5000	418,275	1.22	510,296
			7000	585,585	0.90	527,027
			9000	752,895	0.71	534,555
313.2	36	47	3000	274,833	1.72	472,713
			5000	458,055	1.11	508,441
			7000	641,277	0.82	525,847
			9000	824,499	0.65	535,924
348.6	33	50	3000	305,897	1.56	477,199
			5000	509,828	1.01	514,926
			7000	713,759	0.74	528,182
			9000	917,690	0.59	541,437
393.7	30	55	3000	345,472	1.38	476,751
			5000	575,786	0.89	512,450
			7000	806,101	0.66	532,037
			9000	1,036,415	0.52	538,936

The number of panels mined per year was then arrived at for each face width and web depth for the range of panel lengths 3000 to 9000 ft as shown in column 6. This calculation required the use of the estimated shifts required to move the equipment between panels (column 3).

For the face width of 313 ft (36 in. web) the number of panels per year that can be mined closely matches the panel development schedule of Table 33.

The cost per ton for the various lengths of panel was then calculated and plotted as the curve of Figure 30. As shown, the minimum cost per ton is achieved at the maximum panel length of 9000 ft. It was estimated that a 42 in. single belt section was capable of a maximum run of 6000 ft. The disruption in the curve at a length of 6000 ft is therefore the result of the need to add the cost of a second belt section and drive.

A summary of the specifications established for strategy 1 appears in Table 34.

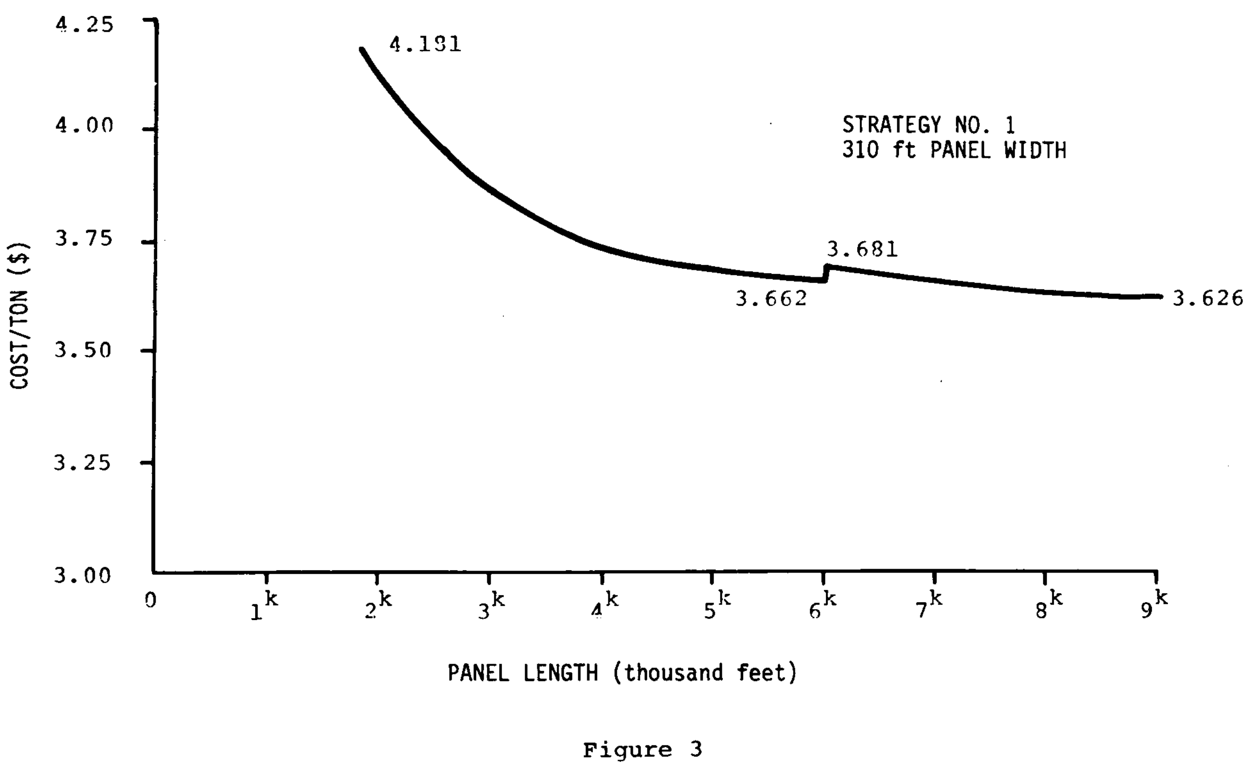


FIGURE 30. - Cost per ton versus panel length, strategy 1.

7.1.2 Discussion of Other Strategies

A similar approach was used to arrive at the specifications for the other seven strategies also summarized in this table. Detailed discussion of how these strategies were developed appears in Appendix I.

The first four strategies required that a range of wider webs be applied so as to minimize the cost per ton at today's levels of shift production. The shift production of 1316 tons is constant for these first four strategies. To minimize the economic impact of moving the equipment between panels, the maximum panel length of 9000 ft was assigned to these four strategies. Also, the minimum face widths were considered so as to reduce the cost of supports and face conveyor.

Under these conditions, only strategy 1 mines coal at a cost per ton of less than \$3.72 where the unit cost of \$3.72 was established as representing the economics of mining a 27 in. web on today's best United States longwalls. Strategies 1 through 4 do not reflect the principal advantage of wide web mining, which is increased production. The cost benefits of wider webs, previously shown in Figure 30, were due to production increases that greatly outweighed the additional cost of the equipment.

This is particularly evident in strategies 3 and 4 where production is limited on faces employing a very deep web. The practical application of these strategies would be limited to low seam work; in this case, a seam height of 3.8 ft. Strategies 3 and 4 would probably represent the application of an in-web shearer capable of mounting 60 in. wide drums. This type of machine, of course, does not presently exist.

The four production-limited strategies all display face equipment of limited capacity, principally the small 170 kW shearer and 700 tons/hr conveyor. Strategies 5 through 8, however, use the higher capacity equipment since they are not production limited. These strategies reflect the increase in production from wider webs maintaining today's unit cost of mining.

Strategies 5 and 6 represent the performance of the contemporary 300 kW shearer and 1000 tons/hr face conveyor. Strategy 5 mines 2376 tons from a 725 ft long face with a panel length of 7200 ft. Strategy 6 mines a longer face (1000 ft) from a longer panel (9000 ft) to offset the additional cost of the four-leg shields - the cost per ton being \$3.72 in both cases.

To extend the web depth beyond 45 in. requires the use of the 310 kW shearer and 1500 tons/hr conveyor. As in the case of strategies 5 and 6, strategy 8 mines a longer and wider panel than 7 to offset the cost of the more expensive shields. Strategy 8 additionally produces the maximum output of 2611 tons/shift because of its maximum web depth and face length.

As shown in the detailed analysis of Appendix H, the high production faces of strategies 5 through 8 mine the panels faster than they can be developed. This occurs even though very wide faces are specified. The time required to develop longwall panels will be more of a problem with the potentially higher production longwall faces.

With the exception of strategies 5 and 6, the shearer haulage speeds will be less than 13 ft/min. The higher speeds of 14.5 ft/min which occurred on faces 5 and 6 were a result of the very deep web being mined in a relatively low seam height.

Finally, strategy 9 represents the performance of FMA's recommended face for the underground demonstration. This recommendation is based on the potential minimum cost per ton of $3.32 and the use of immediately available hardware. Panel dimensions are also within very practical limits where face lengths of 500 ft and panel lengths of 6000 ft are presently being mined in the United States.

7.2 The Recommended Equipment System

This report has identified and included, in the system performance and economic analysis, two major pieces of equipment that are not presently available. These two items are the 310 kW shearer potentially capable of mounting 60 in. wide drums and the special four-leg shield with full sliding canopy. All other equipment that was considered is presently available and, with acceptable modifications, are capable of mining the increased depths of web identified in this report.

However, the equipment recommendation for the underground demonstration required an assessment of the 310 kW shearer and the shield with special full sliding canopy. This assessment was considered to be necessary because:

a. The 310 kW shearer was potentially capable of extending the maximum depth of web to 57 in. Presently available shearers with modifications were limited to a maximum web depth of 45 in.

b. The full sliding canopy concept has the potential to extend the applications of wide web longwalling.

Finally, immediate roof conditions and/or relatively soft floor might well limit the depth of web, at a potential mine site. A support with the special sliding canopy would, to a great degree, reduce the effect that wider webs would have on these conditions.

Consideration of this special equipment, as related to the underground demonstration is summarized as follows.

7.2.1 The 310 kW Very Wide Web Shearer

The prototype machine, designed and constructed by Mining Supplies Limited of Doncaster, England, is scheduled for initial testing at the NCB's MRDE surface test facility at Swadlincote in May of 1979. It is expected that these surface tests will require 3 to 5 months. Upon successful completion of these tests the prototype machine will be assigned to an underground operation. This machine could be considered a proven piece of hardware after completing one panel - a period of approximately 1 year.

The application of this machine to the wide web demonstration would require a minimum of 15 to 20 months of testing and underground evaluation and thus it was not recommended at this time.

7.2.2 The Special Support With Full Sliding Canopy

Section 3 identified the advantages of this support as being:

a. Superior stability - Figure 20

b. Superior floor loading characteristics - Figure 22

c. Good immediate forward support.

As a result, the full sliding canopy, as applied to the four-leg shield, had the highest rating of all the supports considered as shown in Table 13.

Its principal advantage would be to extend the potential use of wide web longwalling. No other support identified could provide full coverage of the immediately exposed roof without being set one web back. As was discussed in Section 3, one web back supports exhibit poor floor loading patterns and instability at web depths in the range of 45 in. The four-leg shield with full sliding canopy is very stable with well distributed floor loading at those web depths.

The mining conditions which would most benefit from its use could be generally defined as those sites with very friable immediate roof strata and a relatively soft floor. The use of more conventional supports would result in very modest increases in web depths (and thus production) under these mining conditions. However, the development of this support would not be essential to the underground demonstration since the site selection process could provide for conditions conducive to the use of a more conventional support. As a result, recommendations for the separate development of this support are included in the summary at the end of this section.

FMA's equipment recommendations, therefore, consist of hardware that will both minimize the risks and the costs per ton of the demonstration face. The recommended equipment consists of:

a. The shearer and coal transporting system

b. The roof support system

c. Ancilliary equipment.

7.2.3 The Recommended Shearer and Coal Transporting System

Referring to Figure 28, with the elimination of system D-D, the obvious choice is the combination represented by curve C-C. This system is comprised of the contemporary 300 kW shearer and a face conveyor with a rated capacity of 1000 tons/hr.

The shearer will be modified to mount 48 in. wide drums for a web depth of approximately 45 in. Shift production is estimated at 2200 tons/shift with a cost off the face of $2.87/ton as read from Figures 28 and 29.

It is further recommended that the double-ended shearer be equipped with chainless haulage and the latest techniques in dust suppression equipment. The dust equipment most certainly should include:

a. Through the shaft ventilation of cutting drums with water venturis for face side flushing

b. Pick face flushing of cutting bits

c. Deep cutting picks on slow speed drums for both dust reduction and increased efficiency of coal cutting.

In addition, further consideration should be given to ventilating cowls, particularly as they pertain to the extraction of methane laden air from the deeper web.

There are presently available two face conveyors of the rated capacity required. They are the Eickhoff DMKF-4 conveyor and the DOE-FMA high reliability conveyor. Both of these conveyors will have experienced comparative underground trials at the time that the demonstration face is specified. A review of this comparative performance should be reviewed prior to final selection.

There are no special features or recommendaitons being made for the stageloader or outby belt haulage system other than the capacity required to accommodate the output from shearer-face conveyor combinations.

7.2.4 The Roof Support System

The preference for the shield type of roof support on United States longwalls was referred to in subsection 2.2.2 and illustrated in Figure 7. Additionally, the more recent trend has been to the four-leg style of shield because of its superior lateral stability, complete roof coverage, gob shielding and relatively short canopy as compared to the four-leg chock or chock-shield.

All of these generally desirable features equally apply to controlling the roof over a wider web. Subsection 3.1 further identified the special considerations that were particularly related to wide web. These special considerations were:

a. The need for immediate forward support

b. Stability against tipping towards the face

c. Adequate tip loading at the face

d. Floor loading - distribution pattern of pressure.

The support types considered to be applicable to wide web longwalling were evaluated on these factors in the work of subsection 3.1. The evaluation of these various support types was summarized in Table 13, where support F, the modified four-leg shield with full sliding roof canopy, received the highest score. However, the development of this special support is not considered necessary for a successful underground demonstration.

The choice of a more conventional support for the applications, therefore, becomes the four-leg shield, of which two were evaluated in Table 13. Support D was a strictly conventional shield set one web back. Support E was modified to provide additional space inside the shield between the front and rear sets of hydraulic props. Support E can then be set immediately behind the face conveyor, since the manway will be between the props.

A comparison of the two supports shows the recommended support E to be superior in stability, tip loading at the face, and floor loading. The conventional support D, however, provides for better IFS since it is set one web back. The newly exposed roof is immediately supported by the main roof canopy.

Alternatively, support E is equipped with a forward sliding canopy extension similar to the extensions used on the British 1 meter web longwall faces - see Figures 11 and 12. These extensions are not as effective as the main roof canopy since they do not provide full roof coverage and actual contact with the roof is limited to the forward tip of the extension.

Since the deployment of this type of sliding extension over the 1 meter webs at the Holditch and Hem Heath Collieries improvements have been made. Hydraulic capsules have been added to these canopies to substantially improve roof contact. The recommended support for the wide web demonstration should include these capsules and should also specify extra wide extensions for more complete roof coverage.

With these modifications, support E will be the best available support for controlling the roof over the 45 in. web depth.

Capacity (in tons of support) for the recommended shield is site dependent. However, the present trend in four-leg shields is to provide a minimum of 6.5 tons/ft^2 as recommended in subsection 2.2.2. It is most probable that support density in the order of 7 tons/ft^2 will be recommended because of the deeper web.

A description of the equipment recommended for the wide web demonstration is summarized in Table 36. The approximate cost of this system is $5,158,846.

TABLE 36. - Equipment summary - recommended system wide web demonstration

Shearer	Type: Power: Drums: Width: Speed: Picks: Special features: Haulage:	double-ended ranging drum (DERD) 300 kW diameter - approximately 70 in. depending on seam height 48 in. 30 rpm - somewhat dependent on coal conditions deep cutting for minimum dust face side ventilation, through the shaft ventilation, pick face flushing ventilating cowls (optional) chainless rack and pinion type, drive optional
Face conveyor, DOE-FMA High Reliability Conveyor	Type: Width: Height: Capacity: Power: Special features:	twin inboard (26 mm) chain 33 in., approximately 10 in., approximately 1000 tons/hr - peak 1200 tons/hr 2 by 300 hp automatic return end tensioner
Supports	Type: Capacity: Canopy: Advanced rams: Special features:	four-leg shield with lemniscate linkage modified for additional length estimated 7.0 tons/ft^2, mine site dependent with forward sliding extension fitted with hydraulic loading capsule (45 in. extension length) capacity with 45 in. stroke batch advancing control with water sprays for dust suppression
Stageloader	Type: Width: Height: Power:	Twin inboard, 2 by 26 mm 36 in. 8-3/4 in. 100 hp drive
Outby belt	Type: Power: Conveyor speed:	48 in. belt conveyor 400 hp approximately 600 ft/min, 20 deg idlers
Approximate cost for complete system: $5,158,846		

7.3 Objectives and Requirements of the Demonstration

The objectives of the underground demonstration eventually are to:

a. Prove the feasibility of mining the recommended 45 in. web

b. Prove that mining such a web is cost effective

c. Prove that wider webs are beneficial in terms of operator comfort and safety

d. Measure and observe the impact of using the wider web on equipment, operations and mining conditions.

The demonstration plan outlined in this section will establish the general mine site requirements best suited for the demonstration. Additionally, this subsection will illustrate and discuss a panel extraction plan and the requirements for monitoring the operation to provide the quantitative data required to evaluate the above listed objectives.

7.3.1 Mine Site Evaluation

The success of the wide web underground demonstration will be greatly dependent on the suitability of the chosen mine site. Among the key factors to be considered in evaluating the prospective sites are:

a. Geological conditions

b. Mine roof safety record - the nature of accidents that have occurred historically

c. Ventilation plan - the need for face ventilation required to accommodate the increased cutting of coal from a deeper web

d. Coal handling system - the ability of the mine site to transport the additional production from the wide web face

e. Mining plan - the reserves necessary to mine the recommended number and size of adjacently located panels and the location of these relative to present operations.

7.3.1.1 Geological Conditions

The most important geological requirement is that the roof must be self-supporting over the increased span of the web. This must be particularly true of the immediate roof strata which, if excessively friable, could result in localized falls in front of the supports.

The caving characteristics of the immediate roof are also an important consideration, although the use of high capacity supports of increased stability has made this a less important consideration in recent years. The cavability of the immediate roof is reflected in the length of overhang and the caving angle. Ideally, a zero overhang and a high caving angle in the order of 60 deg is desirable. Caving angles greater than 60 deg usually indicate that the immediate roof strata is somewhat incompetent and unable to be self-supporting over longer spans.

Ideally, the chosen mine site should have an existing operating longwall so that a reasonable accurate preliminary assessment of roof and floor conditions can be made. This should then be followed by a more formal assessment based on comparative borehole samplings and a study of the general geological conditions prevalent in the area.

The coal seam itself should have several desirable characteristics, the most important of which is a low methane content. Since control of the rate of methane emission is expected to be a more severe problem with wider web mining, seams of high methane content should be avoided. Although such corrective measures as methane drainage are available, this will add unnecessary cost and complexity to the program.

It has been consistently recommended in this report that the initial attempt at wide web mining be in a medium seam height of 6.5 ft. This height of extraction will insure the maximum utlization of the high capacity mining machinery system while minimizing such problems as equipment interferences and man mobility. The wide web mining of a low seam, although highly beneficial, involves the use of a different type of shearer (the buttock or in-web shearer). This type of demonstration is presently being separately funded by the DOE at a site in eastern Kentucky.

The cutting characteristics of the coal should be evaluated so as to insure the proper design of drums and choice of cutting picks.

The capability of the floor to withstand the increased loading is a very important consideration. It has been demonstrated earlier in this report that wider web longwall mining results in an increased floor loading and a forward shift in the point of application. This forward shift in loading could result in the toe of the support base digging into the floor preventing forward movement of the support.

Area-wide geologic features should also be an important consideration. Among the features to be evaluated are the following:

a. Area hydrology

b. Depths and type of overburden

c. Presence of any adverse geologic features such as faults, fractures, linears, clay veins, washouts, etc.

7.3.1.2 Mine Safety

Current health and safety problems at the mine site would certainly have an impact on the wide web demonstration. Wider web longwall mining does provide the potential for providing less hazardous conditions, but the nature of accidents that have historically occurred would provide an indication as to the impact of wider webs on safety.

This preliminary minesite investigation should therefore include a survey of accident records as to number, severity and cause of previous longwall related accidents.

7.3.1.3 Mine Ventilation Plan

Larger quantities of air are likely to be required for wide web longwall mining to reduce the impact of increased methane emission. Although dust on the face is not expected to increase, the tons per hour transported from the face is expected to increase substantially. It is anticipated that the overall ventilation requirement for the wide web face will represent an increased load on the mine ventilation system.

The preliminary survey should include an evaluation of present ventilation requirements and conditions. The results of this study should then be used to project the quantities needed for the demonstration face and an accompanying preliminary ventilation plan.

7.3.1.4 Mine Coal Transport Requirements

The principal advantage of wide web longwalling is the reduction in the cost per ton off the face. The cost reduction is primarily due to the substantial increase in production that is achieved at a modest increase in the owning and operating costs. The economic analysis of subsection 7.1 and Section 5 included the cost of the panel belt haulage systems required to handle the increased production. However, the mine site evaluations should include an investigation into the capabilities of the downstream mine transport system. A successful demonstration will require that the overall mine coal haulage capacity not limit production on the face.

7.3.2 Recommended Mining Plan

The subsection that follows presents a recommended mining plan for the demonstration. The selected mine site will require the reserves necessary for this plan of panel extraction. The objectives of the demonstration can be best achieved if the panels mined are adjacent. This will minimize the impact of substantial differences in mining conditions.

The recommenced plan for panel extraction is illustrated in Figure 31(a and b). Both plans allow for the mining of three depths of web including the maximum of 45 in.

The three depths are recommended for reasons of safety and to provide comparative data. Behavior of the roof is the principal unknown affecting the study. The roof conditions must be constantly evaluated both as to its general behavior over the life of the panel and in particular its impact prior to and during first fall (panel startup).

Although there is no direct evidence to suggest that first fall conditions will be substantially affected by wider webs, it is only prudent to anticipate this possibility. Both panel extraction plans recommend that the initial panel be started with the narrowest web of 27 in. Behavior of the roof strata prior to and during first fall will be closely monitored. The remainder of the first panel will then be mined with increasing web depths of 39 and 45 in.

If a sufficient level of confidence is achieved during the mining of the first panel, then the plan of Figure 32(a) may be adopted. Plan 31(a) could, for example, be recommended if the selected mine site has a history of longwalling under good roof conditions. Plan 31(b) would be adopted if roof conditions experienced during the extraction of the first panel warranted a more conservative approach.

The equipment recommended in subsection 7.2 will accommodate the mining of the three web depths with three widths of shearer drums. The supports will be equipped with varying strokes in the pushover rams and the hydraulic cylinders that activate the extendable forward canopies.

The normal period of time required to extract a 6000 ft long panel is approximately 12 months. Therefore, the period of time required to execute the plan of Figure 31(a) is estimated to be 16 to 18 months. Plan 31(b) would require 28 to 30 months.

7.3.3 Monitoring the Demonstration

The monitoring program required to document the objectives of the demonstration involves both quantitative instrument measurements and qualitative observations. Projections arrived at previously in this report which affect the impact of wider webs on production, economics and safety were based on certain assumptions which are listed in the beginning of Section 7.

To validate these assumptions, the demonstration will require the taking of data to provide:

a. An identification of the mining cycle as affected by the depth of web

b. A complete downtime study (industrial engineering study)

c. Shearer power consumption as related to depth of web.

Additional specific objectives of the underground demonstration, presented at the beginning of subsection 7.3, requires that the monitoring program also include:

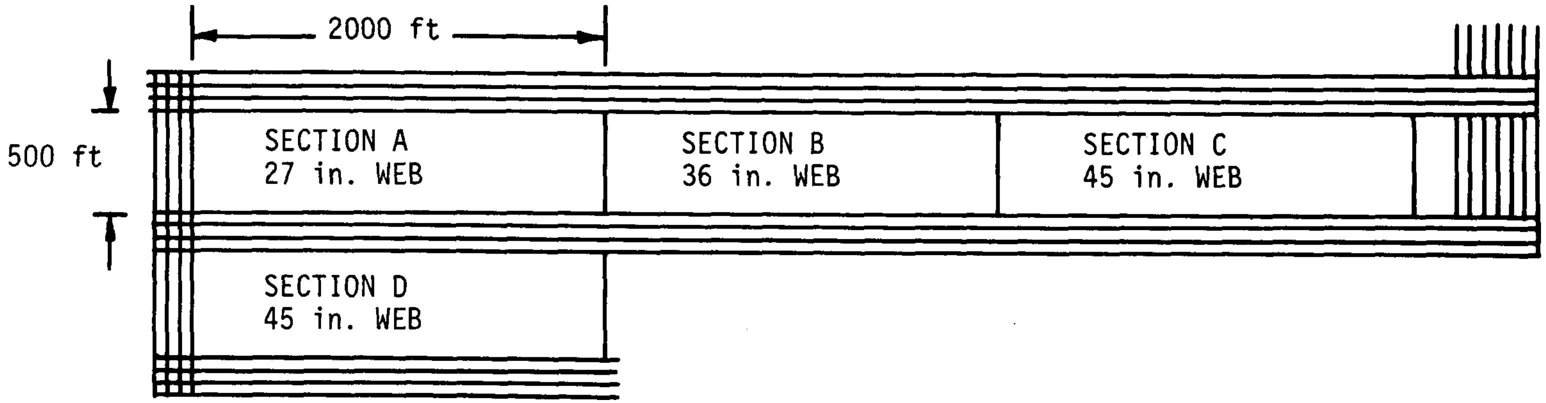
a. All cost factors directly related to the depth of web

b. Selective monitoring of both dust and methane

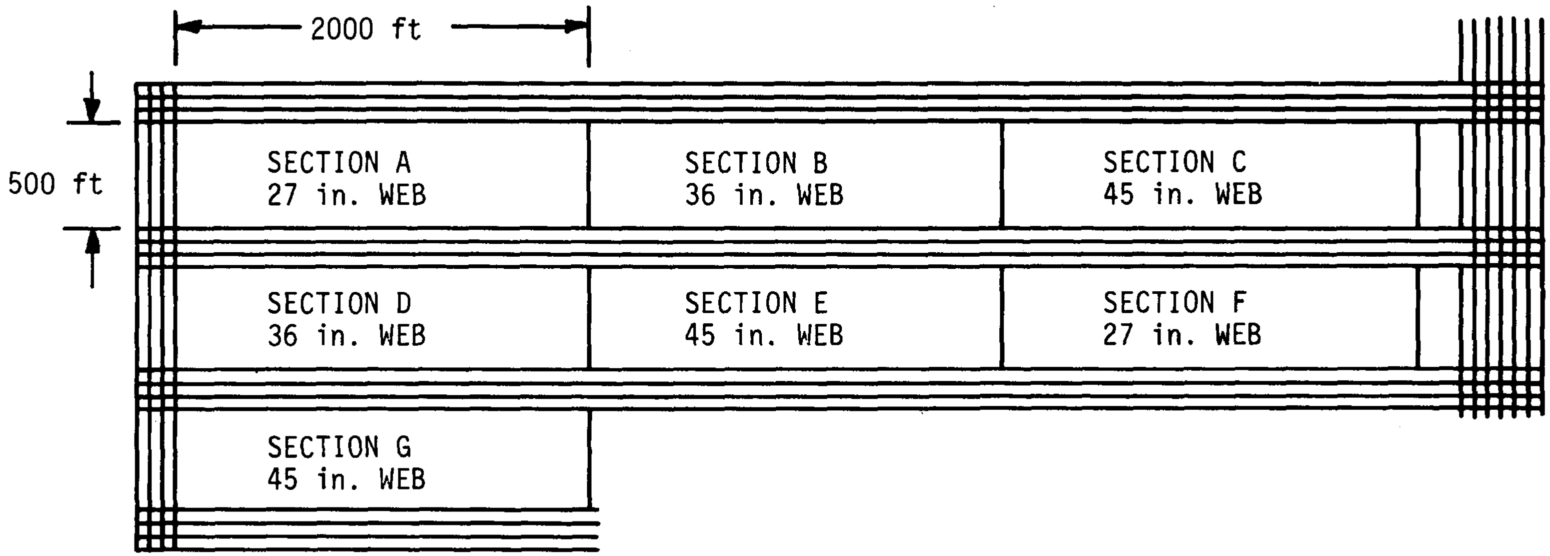
c. An accurate log of all accidents occurring on the face

d. Monitoring of geological influences - in particular, the influence of the roof for varying depths of web

e. The effect of mining wider webs on equipment performance and reliability.



(a) PREFERRED PANEL EXTRACTION PLAN



(b) ALTERNATIVE PANEL EXTRACTION PLAN

Figure 31. - Recommended panel layouts wide web demonstration.

7.3.3.1 Establishing the Mining Cycle

Throughout this report, it has been assumed that certain operations would be unaffected by the depth of web. These operations included sumping the shearer, turnaround operations at the entries, and the flitting speed of the shearer during its cleanup operation. An accurate log of these activities for the three depths of web will be essential to the demonstration results. The data taken should also include constant monitoring of shearer speed during the cutting and loading mode of the operating cycle.

The average cycle for each web depth should be established for at least five complete operating shifts. Periodically, this data should be verified and new data taken if a change in conditions affecting the cycle are identified.

7.3.3.2 The Downtime Study

The industrial engineering study will account for lost time due to occurrences on the face or conditions off the face that result in an actual shutdown of operations. Generally, lost time can be categorized as:

a. On the face:

1. Equipment related lost time

2. Lost time due to geological conditions

3. Downtime due to operational requirements (excessive methane or dust, straightening the face, cleaning excessive coal buildup from under the face conveyor, etc.)

b. Off the face:

1. Travel time to and from the face

2. Delays due to outby coal haulage

3. Loss of power, water, ventilating air or other services.

These occurrences should be logged by observers on the face equipped with hand-held tape recorders. The observer will log the total time lost and the exact reason for the delay. An additional observer should be situated in the head entry to account for lost time due to breakdowns in the outby haulage system.

7.3.3.3 Shearer Power Utilization

This report has predicted only a modest increase in the specific energy of mining with increasing web depth. To validate this assumption requires that shearer power consumption be constantly monitored during the cutting and loading mode. For the purposes of this study, the power readings have two essential purposes:

a. To insure that comparative performance data is taken for each web depth

b. The use of available shearer power is systematically accounted for.

The comparative nature of the study requires that for each depth of web mined the total shearer power used be a constant at or near full power consumption. This will insure that the resulting production figures and costs will be generated from a common basis of comparison.

Additionally, the study requires that shearer power consumption be accounted for as:

a. Power consumed by the shearer propelling system

b. Power to the cutting and loading drums for cutting coal

c. Power to the drums for loading coal out of the web.

To provide for this detailed level of data will require that in addition to total power consumption being monitored at the head gate power center, an instrument package will have to be placed on the shearer. This package will monitor and record:

a. Haulage system effort in pounds or tons of pull

b. Haulage speed

c. The axial thrust of the drums.

The first two readings will account for shearer haulage power whereas the third reading will provide a comparative measure of the effort required to load the coal out of the web.

7.3.3.4 Cost Factors for Varying Web Depths

The principal claim of this report is that mining wider webs is most cost effective. To substantiate this claim, it will be required that production off the face be accurately monitored.

Production data should include a measure of both total shift production and the rate of production (in tons per hour). The former will be measured by the placement of weighing scales at or near the stage loader. The latter can be fairly accurately estimated from the monitoring of shearer speed and by observing the average depth of web and height of extraction.

Other major factors contributing to the comparative cost of mining include:

a. Total face power consumption (to include the face conveyor, stage loader, outby belt haulage and hydraulic pumps)

b. Manpower requirements

c. Maintenance costs for replacement parts and labor.

The accurate accumulation of this data will provide the information necessary to assess the comparative costs of mining and three web depths. It is not considered essential to this study to account for other costs related to the overall operation of the mine or costs incurred on a per-ton basis particularly when these costs are not directly related to mining coal.

7.3.3.5 <u>Monitoring Safety, Dust and Methane</u>

The health and safety aspects of mining a wider web are most important. Therefore, a measure of the impact on health and safety ranks as an important objective of the underground demonstration. Of particular concern are the lost time accidents and the rate of methane and dust liberation.

The demonstrated impact on dust is expected to favor wider web mining because the increase in production will be obtained from deep in the web and because the shearer will be equipped with state-of-the-art hardware. The demonstration, therefore, is likely to prove that substantial increases in tonnage per shift can be realized without creating a dust problem.

It is recommended that the dust sampling procedure should be as recommended by the MSHA. MSHA requires that 10 shifts of data be taken using cumulative respirable dust samplers. These samplers are both worn by the operators on the face and situated in the head and tail entries. The regulated procedure is considered satisfactory for this demonstration.

The monitoring of methane, however, is considered to be of more importance. Because the demonstration will represent a high production face (potentially 1000 to 1200 tons/hr) and because the deeper web will be more difficult to ventilate, subsection 7.3.1 recommended a seam of known low methane concentration. Additionally, subsection 7.2.1 recommended that the latest in dust and methane techniques be deployed including consideration of the ventilated cowl, a recent NCB development.

Normal techniques for monitoring methane on the face and in the entries will be adequate. However, additional observations and monitoring of methane concentrations within the web may be required.

7.3.3.6 Geological Observations and Measurements

The principal unknowns associated with the consideration of wider webs of mining are geologically related. The British claimed to have experienced improved roof conditions when mining 1-m (39 in.) webs. However, detailed observations or measurements of roof behavior were not reported.

The mining plan of subsection 7.3.2 will allow more detailed comparative observations of roof behavior to be taken as the web depth varies within the same panel. Measurements and recorded observations of roof behavior for the three depths of web should include:

a. Changes in the caving characteristics, both with first fall and during normal mining of the panel. Characteristics to be observed would include length of overhang, caving angle and depth of caving.

b. Condition of the immediate strata in front of and over the support canopies. Roof falls at the face will be recorded as to area of fall, location along the face, depth of roof involved and resulting lost time.

c. Observations as to changes in the pattern of convergence along the length of the face - particularly directly behind the shearer. Actual measurement of roof convergence in this particular area is not considered to be practical and is therefore not recommended.

d. The influence of wider webs on roof conditions in the entries, in advance of the face. Changes in roof loads and convergence, as influenced by the forward abutment pressure, can be measured by strain gauges mounted on hydraulic legs.

e. Differences in the characteristics of the coal seam as influenced by changes in the behavior of the roof. If such changes should occur, it is expected to mostly influence cutting characteristics and possibly dust.

The most important geologically related data, however, is to be obtained from instrumenting the roof supports along the entire length of the face. It is recommended that pressure gauges be periodically attached to both the front and rear sets of hydraulic props on the shield supports.

These pressure gauges will transmit readings to strip-chart recorders which will be driven mechanically to continuously record over an 8-hr period. The gauge-recorder package will be equipped with quick disconnect devices for selected placement along the face. Forty of these devices are recommended for the 500-ft long face. Initial placement of these devices would be on every fifth support. The portable nature of the gauges would allow them to be quickly rearranged along the face as conditions required.

The pattern of information that is expected to be generated from these recorded measurements will include:

a. The shift in roof loading on the support as the depth of web changes. This will also reflect the change in load distribution on the floor.

b. The change in total support loading with web depth and as influenced by roof caving characteristics.

c. The general pattern of roof loading characteristics along the face as related to the cycle of mining. For example, the pattern of loading experienced by a support as the shearer passes and as the support is later advanced to the face.

d. Variations in loading experienced by the forward sliding canopy extensions.

The recorded observations and instrument readings will require careful correlation so as to provide a meaningful pattern of data that reflects the overall effect of web depth on geological conditions.

7.3.3.7 The Impact on Equipment

A quantitative measure of the effect of mining the wider webs on equipment can be provided by a careful accounting record of actual maintenance requirements. This data could be augmented by historical mine maintenance records for the longwall operation prior to the demonstration.

The comparative data that can be logged from the actual wide web demonstration will fall into two categories.

a. Normal maintenance requirements involving wearable components

b. Catastrophic equipment failures due to an extension of usage beyond practical design limits.

Category a. would involve the comparing of such maintenance records as cutting tool replacement schedules, servicing of hydraulic seals, replacement of worn hydraulic cylinder rods, etc. This category might also include major equipment overhauls which are normally scheduled for annual or semi-annual occurrence.

Catastrophic equipment failures would be those occurrences which are not normally anticipated or scheduled. These events would have to be categorized as being related to the increased web depth. Examples of these types of failures might be:

a. An increase in conveyor chain breakage due to the increased distance the face conveyor might be pushed across the deeper web

b. A main bearing failure in the shearer ranging arm due to the increased weight of the wider drums and the increased moment of loading.

Identifying equipment maintenance requirements in this manner will provide for a separation of data to reflect the normal affected increase in the cost of mining wider webs (category a.) from the requirement for additional equipment modifications to accommodate wider web mining (category b.).

The subsection has only quantified those parameters that are essential to the demonstration. They were considered to be the system of mining equipment, the depths of web and the plan for panel extraction.

The requirements of the test as related to the particulars of the mine site and the monitoring program have been treated more generally. The parameters associated with the mine site have been identified and their relative importance to the demonstration has been discussed. The specifics of the monitoring program were treated in like manner.

The contents of this report suggest that in order of importance, the underground trial is required to demonstrate and quantify:

a. The affect that wider webs have on geological conditions

b. The cost benefits of mining wider web

c. The impact on equipment performance and reliability.

The choice of demonstration site and the combined effect of the deeper web and state-of-the-art equipment should result in dust and methane being of lesser consequence.

Finally, the British reported that the attitude and morale of the operating personnel played a significant role in upgrading the overall performance of the operation. It can be expected that this rather immeasurable but significant factor will equally apply in the United States.

7.4 Additional Recommendations

The specifications for the underground trial were restricted to the use of available equipment modified to accommodate a deeper web which did restrict the web depth to 45 in.

It has been pointed out that there is a prototype shearer, presently undergoing tests, that is capable of mounting a 60 in. wide drum. Additionally, there is a prototype face conveyor undergoing tests with a design capacity of 1500 tons/hr.

Projecting the potential of wide web longwalling beyond the 45 in. demonstration face would involve the mining of 1500 tons/hr of coal from a 57 in. web. Based on the specific energy curve of Figure 13, a 500 kW (670 hp) shearer would be required to load the conveyor at this rated capacity.

From the productivity equation of Table 3 and the down time projections of Table 17 such a system could mine as much as 3200 tons/shift from a 500 ft long face in a 6.5 ft seam. This would represent an additional 45 percent increase over the projected 2200 tons/shift from the demonstration face.

The deployment of these developments, however, would require an in depth assessment of the capability and practicality of present types of roof supports. The comparative analysis, presented in Section 3 of this report, strongly indicates that present supports would be relatively unstable and that effective control of the roof, in front of the main canopy, would be marginal at these web depths.

Stability and control of the roof over the deeper web was substantially improved with the application of the full sliding canopy concept. It is, therefore, recommended that a parallel program be funded to assess new ideas and concepts in roof supports, specifically for web depth of 57 in. or greater. Phases and tasks for this program are divided into the following.

<u>Recommended Program - Roof Support Development</u>

<u>Phase I - Feasibility and Design</u>

The objectives of this phase would be:

a. The identification of specific design requirements for controlling the roof over a very deep web.

b. Based on these requirements, the assessment of the feasibility of the full sliding canopy concept - or other concepts that may be identified.

c. Projection of the impact of such a development on longwalling in the United States - where and under what conditions would such a support be required.

d. Preparation of detailed designs, specifications, and the cost of such a support.

e. Economic analysis of the cost effectiveness of very wide web longwalling with such a support.

At the conclusion of Phase I the status of wide web longwalling in the United States as well as other research programs related to increasing production could be determined. A determination could also be made as to the desirability of proceeding into Phase II.

Phase II -

The objective of this phase would be to:

a. Construct a prototype support

b. Laboratory test the support

c. Finalize the design, specifications, and costs

d. Demonstrate very wide web longwalling underground.

It is estimated that it would require about 9 months to complete Phase I and an additional 20 months to complete Phase II including 12 months for the underground demonstration.

8. REFERENCES

1. "The Application of Heavy Duty Equipment in American Mines" J.T. Pearey, The Mining Engineer (London) March 1977.

2. "American vs. German Longwall Mining Methods" Peter H. Klaar Mining Congress Journal, Vol. 58, January 1972.

3. Consols Five Record Production Runs Impetus to Use of Shield Supports for U.S. Longwalls, Nicholas P. Chironis, Coal Age Operating Handbook of Underground Mining.

4. USBM 1976 Census of Operating American Longwall Installations, Coal Age, January 1977, M.A. Gross.

5. U.S. Dept. of Energy Contract No. ET-75-C-01-9024 "High Capacity Longwall Conveyor Program" Assigned to Foster-Miller Associates, Inc.

6. "Longwall Mining: Avenue to Safety, Productivity, and Resource Recovery" P.C. Merritt and H. Davis, Coal Age, Volume 82, January 1977.

7. Methods of Working Coal & Metal Mines - Seth D. Woodruff Pergamon Press, London - Page 173 - First Edition

8. Advanced Technology Mining, W.J.W. Bourne, The Institute of Mining Engineers (London) Annual Conference, May 1977.

9. Wide Web Working, J.R. Hunter, The Institute of Mining Engineers (London) Annual Conference, May 1977.

10. Mechanization Profile of the Mining Research and Development Establishment of the NCB, May 1977.

11. Barry, A.J., O.B. Nair, In Situ Tests of Bearing Capacity of Roof and Floor in Bituminous Mines, USBM, RI 7406.

12. An Analysis of Powered Support Behavior by J.J. Bates - The Mining Engineer (London) June 1978.

13. Irresberger, H., Roof Control on Longwall Faces, Colliery Guardian International, p. 34, April 1978.

14. Ostermann, W., Cutting Performance of Drum Shearer Loader as a Function of Web Depth and Drum Speed, Gluckauf, Feb. 1966. PMSRC Translation, Dec. 1976.

15. Conceptual Design of an Automated Longwall Mining System, Cominec Inc., USBM Contract S0241051, April 1976.

16. Mutmansky, Analysis of the Effects of Changes in Cutting Width on Longwall Performance, Preliminary Report to USBM, Sept. 1976.

17. Longwall Mine Availability and Delay Analysis by Curry, Heimburgh and Metcalfe. Prepared by the Jet Propulsion Laboratory for the USBM in Document 5030-46.

18. Longwall Conveyor System Study by Herhal, Pinmentel and Douglas. Prepared by Ketron Inc. for U.S. Dept. of Energy Cont. No. ET-77-C-01-8915(2).

19. Wide Web Mining in the Ten Foot Seams at the Holditch Colliery. D. Hardman - Presented to the Institute of Mining Engineers North Staffs Branch, Nov. 1975.

20. Dust COntrol and Planned Layout in the Ten Foot Seam at Holditch. T.E. Smales - Presented at the Institute of Mining ENgineers North Staffs Branch, September 1975.

21. Wilson, A.H., Support Load Requirements on Longwall Faces, General meeting of Manchester Geological and Mining Society Branch of IMinE, Paper No. 4558, May 1974.

22. Wade, L.V., Longwall Support Load Predictions from Geological Information, Society of Mining Engineers AIME Transactions, Vol. 262, p. 209, Sept. 1977.

23. 1977 Census of Longwall Installations Operating in the United States M.A. Gross "The Longwall Newsletter" Aug. 1978.

24. Discussions with Mr. P Tragelles, Director of the NCB's Mining Research Development Establishment in April of 1975.