METHOD OF WORKING

LONGWALL MINING

BY

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NOTES ON

Longwall mining: Methods of driving gate roads; single and multiple heading gate roads; longwall face layout – advancing and retreating faces; orientation of longwall face; support system for longwall gate roads; powered supports; face transfer, operation of shearer and plough; roof management and hard roof management; periodic and main fall; design of high productive longwall panel; mini/short-wall mining.

Longwall mining

Longwall mining is a form of underground coal mining where a long wall of coal is mined in a single slice (typically 0.6 – 1.0 m thick). The longwall panel (the block of coal that is being mined) is typically 3 – 4 km long and 250 – 400 m wide.

The basic idea of longwall mining was developed in England in the late 17th century. Miners would undercut the coal along the width of the coal face, removing coal as it fell, and using wooden props to control the fall of the roof behind the face. This was known as the *Shropshire method* of mining. While the technology has changed considerably, the basic idea remains the same, to remove essentially all of the coal from a broad coal face and allow the roof and overlying rock to collapse into the void behind, while maintaining a safe working space along the face for the miners.

Starting around 1900, mechanization was applied to this method. By 1940, some referred to longwall mining as "the conveyor method" of mining, after the most prominent piece of machinery involved. Unlike earlier longwall mining, the use of a conveyor belt parallel to the coal face forced the face to be developed along a straight line. .{You are reading it on mineportal.in}.{You are reading it on mineportal.in} The only other machinery used was an electric cutter to undercut the coal face and electric drills for blasting to drop the face. Once dropped, manual labor was used to load coal onto the conveyor parallel to the face and to place wooden roof props to control the fall of the roof.

Such low-technology longwall mines continued in operation into the 1970s. The best known example was the New Gladstone Mine near Centerville, Iowa "one of the last advancing longwall mines in the United States". This longwall mine did not even use a conveyor belt, but relied on ponies to haul coal tubs from the face to the slope where a hoist hauled the tubs to the surface.

Longwall mining has been extensively used as the final stage in mining old room and pillar mines. In this context, longwall mining can be classified as a form of retreat mining.

Applicability conditions

1. Ore strength - Any but it should crush under roof pressure rather than yield; preferably material that is weak and can be cut by continuous miner.
2. Rock strength- Weak to moderate, must break and cave, intermediate roof should be thin bedded, floor should be firm
3. Deposit shape - tabular
4. Deposit dip- preferably flat and uniform (dip should be less than 12 degree)
5. Deposit size- large areal extent, thin bedded, uniform thickness
6. Ore grade- Moderate
7. Depth- Moderate to very deep

Advantages of Longwall mining

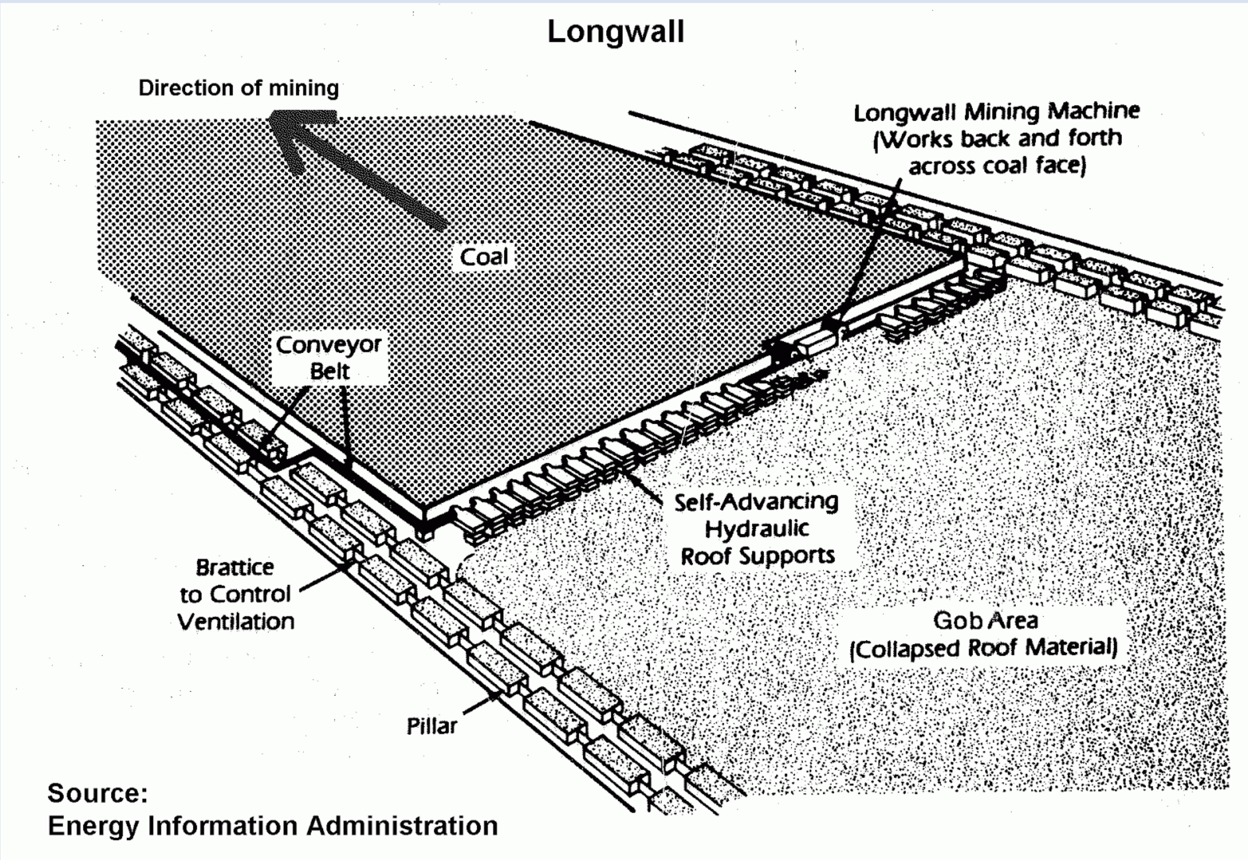
1. Highest underground productivity, outstanding continuity of operations and low labour intensity resulting in high output.
2. Fairly low mining cost, one of the most economical underground mining method
3. High production rate
4. Suitable for total mechanization, remote control and automation
5. Low labor requirement
6. Fairly high recovery
7. Concentrated operations, facilitating transport, supply and ventilation
8. Most suited for deep seam under bad roof condition
9. Good for Miner's health and safety

Disadvantages of Longwall mining

1. Damage to the surface features as caving and subsidence occurs over wide area.
2. Method very inflexible and rigid in layout and execution
3. Rate of advancement of face should be uniform to avoid roof-support and subsidence problems
4. Very high capital cost.
5. Reliance on single production unit means costly delays, interruptions in production
6. High moving costs
7. Chances of fire due to self-heating in gob

Layout

Gate roads are driven to the back of each panel before longwall mining begins. The gate road along one side of the block is called the maingate or headgate; the road on the other side is called the tailgate. Where the thickness of the coal allows, these gate roads have been previously developed by continuous miner units, as the longwall itself is not capable of the initial development. The layout of Longwall could be either 'advancing' type or of 'retreat' type. In the advancing type, the gate roads are formed as the coal face advances. In thinner seams the advancing longwall mining method may be used. In the retreat type, the panel is formed by driving maingate, tail gate and a face connecting the both. Only the maingate road is formed in advance of the face. The tailgate road is formed behind the coal face by removing the stone above coal height to form a roadway that is high enough to travel in. The end of the block that includes the longwall equipment is called the face. The other end of the block is usually one of the main travel roads of the mine. The cavity behind the longwall is called the goaf, goff or gob.



Ventilation

Typically, intake (fresh) air travels up the main gate, across the face, and then down the tail gate, known as 'U' type ventilation. Once past the face the air is no longer fresh air, but return air carrying away coal dust and mine gases such as methane, carbon dioxide, depending on the geology of the coal. Return air is extracted by ventilation fans mounted on the surface. Other ventilation methods can be used where intake air also passes the main gate and into a bleeder or back return road reducing gas emissions from the goaf onto the face, or intake air travels up the tail gate and across the face in the same direction as the face chain in a homotropal system.

To avoid spontaneous combustion of coal in the goaf area, gases may be allowed to build up behind seals so as to exclude oxygen from the sealed goaf area. Where a goaf may contain an explosive mixture of methane and oxygen, nitrogen injection/inertisation may be used to exclude oxygen or push the explosive mixture deep into the goaf where there are no probable ignition sources. Seals are required to be monitored each shift by a certified mine supervisor for damage and leaks of harmful gases.

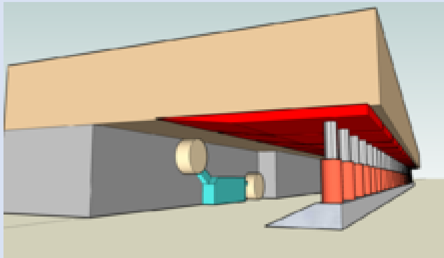
Equipment

Hydraulic chocks

A number of hydraulic jacks, called *powered roof supports*, *chocks* or *shields*, which are typically 1.75 m wide and placed in a long line, side by side for up to 400 m in length in order to support the roof of the coalface. An individual chock can weigh 30–40 tonnes, extend to a maximum cutting height of up to 6 m and have yield rating of 1000–1250 tonnes each, and hydraulically advance itself 1 m at a time.

Hydraulic chocks, conveyor and shearer

The coal is cut from the coalface by a machine called the *shearer* (*power loader*). This machine can weigh 75–120 tonnes typically and comprises a main body, housing the electrical functions, the tractive motive units to move the shearer along the coalface and pumping units (to power both hydraulic and water functions). .{You are reading it on mineportal.in}At either end of the main body are fitted the ranging arms which can be ranged vertically up down by means of hydraulic rams, and onto which are mounted the shearer cutting drums which are fitted 40–60 cutting picks. Within the ranging arms are housed very powerful electric motors (typically up to 850 kW) which transfer their power through a series of lay gears within the body the arms to the drum mounting locations at the extreme ends of the ranging arms where the cutting drums are. The cutting drums are rotated at a speed of 20–50 revs/min to cut the mineral from coal seam.



Chocks providing support to allow shearer to work

The shearer is carried along the length of the face on the *armoured face conveyor* (AFC); using a chain-less haulage system, which resembles a ruggedized rack and pinion system specially developed for mining. Prior to the chainless haulage systems, haulage systems with Chain were popular, where a heavy duty chain was run along the length of the coal face for the shearer to pull itself along. The shearer moves at a speed of 10–30 m/min depending on cutting conditions.

The AFC is placed in front of the powered roof supports, and the shearing action of the rotating drums cutting into the coal seam disintegrates the coal, this being loaded onto the AFC. The coal is removed from the coal face by a scraper chain conveyor to the main gate. Here it is loaded onto a network of conveyor belts for transport to the surface. At the main gate the coal is usually reduced in size in a crusher, and loaded onto the first conveyor belt by the *beam stage loader* (BSL).

As the shearer removes the coal, the AFC is snaked over behind the shearer and the powered roof supports move forward into the newly created cavity. As mining progresses and the entire longwall progresses through the seam, the goaf increases. This goaf collapses under the weight of the overlying strata. The strata approximately 2.5 times the thickness of the coal seam removed collapses and the beds above settle onto the collapsed goaf. This collapsing can lower surface height, causing problems such as changing the course of rivers and severely damaging building foundations.

Comparison with room and pillar method

Longwall and room and pillar methods of mining can both be used for mining suitable underground coal seams. Longwall has better resource recovery (about 80% compared with about 60% for room and pillar method,[8] fewer roof support consumables are needed, higher volume coal clearance systems, minimal manual handling and safety of the miners is enhanced by the fact that they are always under the hydraulic roof supports when they are extracting coal.

Longwall Mining Automation

Longwall mining has traditionally been a manual process in which alignment of the face equipment was done with string lines. Technologies have been developed which automates several aspects of the longwall mining operation, including a system that aligns the face of the retreating longwall panel perpendicularly to the gate-roads.

Briefly, inertial navigation system outputs are used in a dead-reckoning calculation to estimate the shearer positions. (Dead reckoning is the process of estimating the current position based on previous estimates of position and direction of travel.) Optimal Kalman filters and smoothers can be applied to improve the dead reckoning estimates prior to repositioning the longwall equipment at the completion of each shear. Expectation-maximization algorithms can be used to estimate the unknown filter and smoother parameters for tracking the longwall shearer positions.

Compared to manual control of the mine equipment, the automated system yields improved production rates. In addition to productivity gains, automating longwall equipment leads to safety benefits. The coalface is a hazardous area because methane and carbon monoxide are present, while the area is hot and humid since water is sprayed over the face to minimize the likelihood of sparks occurring when the shearer picks strike rock. By automating manual processes, face workers can be removed from these hazardous areas.

Subsidence

There have been cases of surface subsidence altering the landscape above the mines. At Newstan Colliery in New South Wales, Australia "the surface has dropped by as much as five meters in places" above a multi- level mine. In some cases the subsidence causes damage to natural features such as drainage to water courses or man-made structures such as roads and buildings. "Douglas Park Drive was closed for four weeks because longwall panels . destabilized the road. In 2000, the State Government stopped mining when it came within 600 meters from the twin bridges. A year later there were reports of 40-centimetre gaps appearing in the road, and the bridge had to be jacked sideways to realign it."

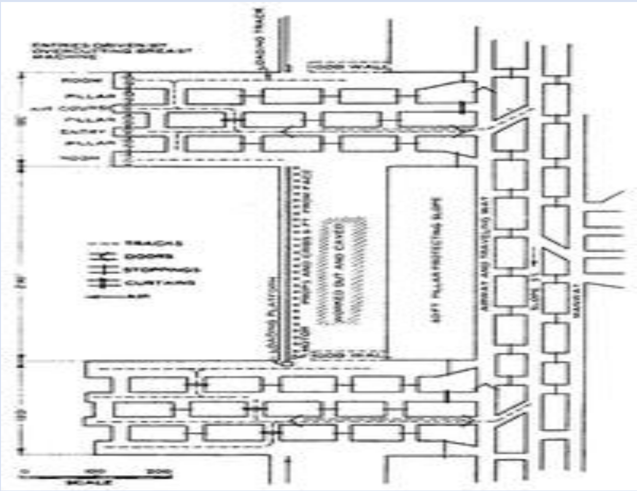
A 2005 geotechnical report commissioned by the NSW RTA warns that "subsidence could happen suddenly and occur over many years".

However, there are several mines, which were successfully mined with little to no measurable surface subsidence including mines under lakes, oceans, important water catchments and environmentally sensitive areas. Subsidence is minimized by increasing the block's adjacent chain pillar widths, decreasing extracted block widths and heights, and by giving consideration to the depth of cover as well as competency and thickness of overlying strata.

ADVANCING AND RETREATING LONGWALLS

Advancing Longwalls

In this method, the face start point is close to the main headings, usually leaving a barrier pillar to protect them. Once the face equipment is installed, extraction commences working away from the main headings towards the block limit. Obviously the main and tailgates do not exist prior to the start of extraction and have to be formed at each end of the face as mining progresses. The gate roads are effectively in the goaf and a false rib has to be installed on one side, usually by constructing a small pillar, sometimes using stone cut from the roof in thin seams or using some type of cementitious material brought into the mine. Such gate roads tend to require a very heavy support system (yielding steel arches have often been used).



Advancing longwalls were once common in Europe in relatively thin seams where packs were constructed using stone, which had to be cut in order to produce sufficient height for the gate roads, and sometimes using coal fines which were not very marketable at one time.

Usually a pillar of coal referred to as a "chain pillar" would be left between adjacent longwall blocks, wide enough to remain intact when carrying the load between two goaves and protect the gate road. Occasionally two longwalls would be operated simultaneously, one each side of a shared maingate (in this case referred to as a "mother gate").

Retreating Longwalls

In this method, the gate roads are first driven from the main headings to the block limits and then connected with a roadway to install the face equipment. The gate roads may be connected to another set of roadways at that point for ventilation/gas control purposes. Once the face equipment is installed, production commences with the face retreating from the limit back towards the main headings, usually to finish at a position so that a barrier pillar is left to protect the latter headings.

Because the gate roads are long, it is normally necessary to drive at least two (sometimes more) at each side of the block. That set of roadways which will be used for access onto the longwall and for coal clearance off the longwall are typically called the "maingate roadways" whilst the other roadway or set of roadways is typically referred to as the "tailgate roadways". The latter are used for primary access on occasions, but this is not generally the case.

As the face retreats, the roadways forming the face ends are destroyed and become part of the goaf. The other roadways will remain open if adequately supported and it is common practice for one of the remaining roads at the maingate end (usually only one in any case) to become the tailgate of the next block. The gate road first working pillars then become the chain pillars between the blocks.

Retreating longwall mine .

Comparison of Advancing and Retreating Longwalls

The advantages of retreating longwalls compared to advancing are:

a. Gate road formation is remote from face operations (less congestion at face ends, less supplies into longwall face area, face not held up waiting for gate road preparation or vice versa, no problems of dust production from gate road workings affecting longwall personnel).

b. No gate or roadway side packs required, so less supplies overall

c. Longwall block is surrounded by roadways before the longwall starts so knowledge of strata conditions is much better

d. Gas drainage of adjacent blocks can be carried out starting during development; with longwall advancing the drilling can only be done behind the face after longwall extraction, allowing less drainage time before the next block commences production

e. With retreat longwall mining, additional gateroads or bleeder roadways behind the goaf area can be developed for ventilation by the development unit if required. Such additional roadways are much more difficult to mine with an advancing longwall

f. There are more options for ventilation/gas control using additional roadways at the limit of the block

g. Advancing longwall gateroads typically require extensive maintenance to maintain the roadway cross-section (roof and floor brushing) during the life of the longwall block, whereas retreat longwall gateroads are allowed to collapse behind the retreating face

h. The only real advantages of advancing longwalls are:
i. Production can begin earlier as the mine does not have to wait for the gate roads to be developed before longwall production can commence (provided development rates are adequate this should only apply for the first longwall in a mine)

j. It provides an opportunity for disposal of stone which has to be excavated into gate side packs (this benefit is probably more than offset by the costs involved in pack construction)

LONGWALL BLOCK SIZE

As a general principle, within reason, the bigger the longwall block sizes the better. Factors which favour large blocks are:

- Relocating a longwall face from one block to another is a major logistical exercise involving high costs and during which there is no production (and hence no income) – the fewer moves required the better and the longer the blocks the fewer moves are required;
- There may be more initial development required to open a new longwall mine, or area of a mine, to create long blocks, but once this is done the main road development required is usually the same for each new block (unless additional airways are required to obtain satisfactory ventilation) regardless of block length. The longer the block, the lower the total development metres required per longwall tonne made available;

- When cutting, the slowest part of each cutting cycle (shear) is at the face ends, so the wider the face, the greater the amount of coal available to cut at the high rate;
- A large proportion of strata control problems, or work required to avoid such problems, occurs at the face ends in the gate roads. The wider the face the fewer the number of gate roads within the longwall area and the less the cost per tonne for gate road maintenance;
- Much of the gate road equipment is the same regardless of the face width, or at least the extra cost for gate road equipment for the extra width is relatively small. The wider the block the lower the capital cost per longwall tonne; and
- Beyond a certain block width, the chain pillar size required for stability is not affected. The wider the face, the less coal is sterilised in these pillars.

Factors which constrain longwall block sizes are:

- The size and shape of the coal lease, the location of existing workings (including those in under or overlying seams) and the desirability of avoiding geological structures can all constrain the size of longwall blocks;
- The wider a face the slower the face advance rate for a given cutting rate, which can have bad repercussions on strata control;
- Generally the wider the face the more difficult it becomes to control alignment and to steer;
- Long blocks require large conveyor installations, sometimes making an additional conveyor drive (usually a tripper drive) necessary;
- The larger the blocks (length and width), the more arduous the ventilation duty, whichever style of ventilation is used, particularly for lower seam heights. The long gate roads affect both longwall and development ventilation; driving more than 2 headings per panel will ease the ventilation duty, but this is seldom an economic option even if there is time available for this additional drivage;
- The higher the production rate, the greater the gas make which the ventilation and gas drainage system will have to accommodate;
- The larger the block size, the more arduous are the water supply, compressed air supply and possibly pumping duties because of the increased pipe friction losses which arise from increased pipe length (larger diameter or additional lines may be required to overcome this problem);
- The larger the block, the more likely it becomes that major equipment will have to be removed for repair/overhaul during the life of the block, instead of the more common practice of overhauls being carried out during relocations. With good planning, much of the equipment can be changed out reasonably easily at suitable locations during the block life, but in most cases a spare item needs to be available as it would not be good practice to stand the wall for long periods while overhauls are carried out. It is also necessary to be confident that strata control can be maintained during the inevitable standing time involved, even where spare equipment is available;

- The wider the block, the higher the initial capital cost involved. It is an option to increase block widths incrementally over several blocks to spread this cost, but certain items would have to be specified for the later higher duty when first purchased; and
- Power requirements for a wide face coupled with space available (with regard to seam height) can restrict possible face widths. AFC chain size is also a consideration here and in some mines heat produced can be restrictive.

Once a block size is chosen and the operating conditions are known as far as possible, then the specifications for the equipment can be drawn-up and design can commence. It may, of course be necessary to revisit the plan several times as the size, cost and production capacity of equipment becomes known.

EQUIPMENT SELECTION

a) Armoured Face Conveyors (AFCs)

Although frequently considered after supports and cutting machines, AFCs can be the ultimate limitation on face width. Given that geological, ventilation, financial and such issues would not limit a longwall width, it will almost certainly be the AFC which governs the final size. In most cases, adding extra supports to widen a face would have minimal effect other than the additional unit cost. There is not even a large effect on pump capacity as the same number of supports will be operated at any time as for a shorter face (though there will be extra pressure losses in longer hoses and increased leakage). The cutting machine would be the same size regardless of face length and a longer face would only entail additional incremental costs for longer cables, hoses, etc.

A surface partial or "mini" build of the
equipment for a thick seam face.
Photo courtesy of Joy Manufacturing Company Pty Ltd

A typical modern 2 leg support shown
with its AFC pan section in place.
Photo courtesy of Joy Manufacturing Company Pty Ltd

With the AFC however, any additional length requires additional power in the drives, additional strength in the transmission equipment and additional strength (which usually involves additional size) in the AFC chain.

The additional power will normally entail an increase in physical size of drive units. In thinner seams, simply fitting the larger equipment in the seam height may be difficult, will minimise any clearance left to allow for strata movement (roof or floor) and will increase the resistance to ventilation, all factors which merit close consideration at the planning stage. In thicker seams, the effect of these factors will of course be less, though increased ventilation resistance may still be important, but in all seams the increased power will give rise to increased heat production, heat which is bound to be released into the ventilating air at some point. This will be a major consideration in mines where heat is a problem.

Additional power will also dictate an increased chain size (48mm chain is quite common on large longwalls and 50mm chain is now in use) together with increased sprocket size which in turn impacts on the size of face end equipment. Large chain brings with it material handling difficulties when sections have to be moved on or off the face.

Calculation of power requirements is by no means a simple procedure and large factors have to be included to allow for worst case situations. The starting point is simply the power required to drag the empty chain around the pan line and overcome the friction involved. Consideration of the "payload" to be carried must allow for the possibility of a completely full load, at least to the top of the AFC back plates, and the possibility that a proportion of this may be stone (cut intentionally or because of a loss of roof control). There is then the probability (certainty in some respects) that the AFC will seldom be completely straight horizontally and vertically, and the face grade can be in favour of or against the load, frequently some of each. Added to all this is the possibility that all this resistance may have to be overcome from standstill. While there are drive arrangements that can ensure a "soft start" and avoid the very high initial demand of a direct start, the energy still has to be transmitted through the system in some way to get things moving.

There are computer programs available to calculate power requirements for AFC's, given that the properties of the material to be transported and seam conditions are known, but a decision based on experience (especially if there are other mines in the vicinity) or on a risk assessment has to be made to decide the economic optimum level of "overdesign" to cater for the worst cases. Inadequate power or strength of components may require the excess load to be shovelled off the AFC by hand in order to restart after a stoppage, a lengthy and therefore expensive exercise. Faces with 2 or even 3 motors rated at 1MW each (including main and tail gate drives) are in use successfully on some faces.

Large AFC capacity entails more robust and wear resistant line pans, unless a shorter life is acceptable, which is seldom the case. Improved design and construction methods and use of more wear resistant materials have all led to better AFC performance. Separation of structural and wear components enables the use of abrasion resistant steel on the wear sections. Thicker, and in some cases replaceable deck plates help to extend the working life. The improved design and construction methods have led to closer tolerances being possible which in turn assists in reducing wear and increasing life and in allowing greater flexibility. Modern pan lines are capable of around 7° of vertical flexure and around 1° horizontal.

All this extra power and load capacity leads to a requirement for stronger attachments between supports and pans, particularly at face ends where all the larger equipment has to be advanced by the few main gate and tail gate supports, usually the same number of supports being involved regardless of face width.

While dog bones have had improvements in design and manufacture, some being rated at up to 450t, these are still intentionally designed as the weak link to fail before more expensive damage occurs (i.e. equivalent to shear pins).

Shearer drive racks have also increased in size along with larger and/or faster shearers. While the racks are carried on the AFC pans, they are essentially an add-on item and can be changed out if an increased size is required, possibly with an upgraded mounting arrangement. However, extra weight would still be involved.

Increased AFC capacity entails increased BSL, crusher and boot end capacity all of which results in increased power requirements and increased component strength in their turn.

All the above power and strength increases may be desirable from the point of view of longwall operators, but all come at an additional financial cost and with additional weight. The latter is not usually a problem during operation, unless a soft floor is involved which fails under the extra load, but may become an issue during face relocation when the heavier items have to be transported. As usual, the optimum size will probably be a compromise between several factors.

b) Face Supports

While it is frequently argued (correctly) that all items on a longwall face have to be considered as parts of an integrated unit and be fully compatible, it cannot be denied that poorly designed and inadequate roof supports are likely to have a greater effect on results than other items, and to that extent are more important. Inadequate shearer or AFC designs will usually only slow the face and reduce production (which may be financially disastrous) whereas inadequate support design may lead to loss of roof control with significant safety risks and damage to equipment as well as loss of production. Overdesign will avoid the risk of loss of control but at a cost and possibly will introduce other problems associated with size and weight.

A thinner seam 2 leg support shown almost fully closed.
Photo courtesy of Joy Manufacturing Company Pty Ltd

A support for a very thick seam shown fully extended.
Photo courtesy of Joy Manufacturing Company Pty Ltd

Support design/selection considerations should include:

- Ability to control the rate of closure of the mined opening i.e. strata control (mainly roof control but floor movement cannot be ignored). This will include setting and yield loads, support stiffness, hydraulic system pressures and support geometry;
- Load distribution on roof and floor strata, closely allied with the above but with different connotations;
- Hydraulic systems with regard to ease of operation, minimisation of potential leakage points, speed of operation (supports must be capable of being advanced at the same rate as the shearer cuts) and pressure requirements;
- Support operating height range to handle seam thickness variations and to enable transport around the mine when fully closed;
- Overall size and weight for transport purposes; and
- Ergonomic aspects both for operation (largely ease of travel along the face) and for maintenance/parts replacement purposes.

Cost has intentionally been excluded from the list of considerations. It may be a factor in deciding between two otherwise satisfactory offers or may be the deciding factor in continuing or abandoning a project. It should not be a consideration in determining the adequacy of support systems which should be a purely engineering exercise.
Most of the following comments are applied to roof movement which is usually the major concern, certainly from safety aspects, but it should not be forgotten that they can apply equally to floor movement. The roof may be controlled satisfactorily, but excessive closure of the face as a result of floor heave can still occur leading to major production losses.

The starting point for design/selection is to decide the support load requirement to maintain control of the roof, usually expressed as tonnes/m2, sometimes as tonnes/m run of face which ignores the area of the support. This will be an exercise in geomechanics and its success will depend greatly on the extent of knowledge of the resource to be worked. Some knowledge is required of coal and surrounding strata strengths, extent of discontinuities (joints, bedding planes, etc), depth of cover and stress magnitudes. Ideally information can be obtained from experience already gained at the mine concerned or from adjacent workings, but at a "greenfield" location some estimation will be necessary. With the large number of longwalls which have been worked it is likely that similar conditions will have been encountered elsewhere and useful information may be available.

Factors and methods involved in designing/evaluating support performance are well described by Mitchell(1) who in turn refers to work by C Wilkinson for Broadmeadow mine.

The degree of roof movement possible before failure occurs, usually considered to be in the range 15 to 35mm of sag(1), varies with roof material properties and must be determined or estimated for the given conditions.

The roof movement is made up of two parts, an initial largely uncontrollable portion which occurs before supports can have full effect and a portion which is allowed by the support during operation. Additional movement may occur if the support is inadequate and can lead to roof failure and loss of control.

There may be some degree of even earlier movement if gas drainage is used and removal of the gas pressure allows relaxation of the coal reducing the degree of restraint on roof and/or floor strata.

The initial movement starts as soon as the coal which previously supported and constrained the roof strata is removed and no support is in place to resist such movement, other than the bridging effect of strata supported by uncut coal ahead of the face and the supports set between the face and the goaf. This largely unrestrained freedom to move continues until supports are moved forward under this freshly cut roof and even then can continue while any clearances or looseness in the support structure connections are taken up and any broken material or any unevenness of the roof or floor crushes and allows full application of the support load to the strata. While this has been termed "uncontrollable" as there is no effective support immediately below the portion of roof in question, the supports and coal do provide an overall control system by providing the sides for the bridging effect, which does affect the stability of this otherwise unsupported roof.

The main control of the roof is provided by the supports once they are set below freshly exposed roof, so for best control it is obvious that the support should be brought forward and reset as soon as practical behind the shearer. If roof conditions are poor, the support should be brought forward almost immediately, but some methods of working require some delay before supports are advanced. There is no problem in this delay, provided the overall support system is designed to cater for it.

The attitude of the support is an important factor in providing this early application of load to the roof. With a rigid canopy, any unevenness or cavities in the roof can prevent the front of the canopy from touching the roof thus delaying any effective resistance to movement of that section of freshly exposed roof until either the support is advanced again or until roof movement closes the gap, this delay possibly leading to roof failure. Some supports have been provided with an articulated tip actuated by a small cylinder attached to the canopy to provide early tip load. Such loading is relatively low but can be effective in maintaining roof control.

However effective the support setting, this does not normally cause roof movement to cease (nor is it intended to as no support can provide enough load to do this) unless the stage is reached where the roof strata is bridging across the supports onto reconsolidated material in the goaf. After setting, further closure of the roof will occur (the second part of the two parts of roof movement referred to above) as the fluid in the support legs is compressed and dilation of the hydraulic system occurs. This will cause an increase in the hydraulic pressure above the setting pressure until this is relieved through valves at the designed support yield pressure. This continuous increase in pressure is the mechanism providing roof control, but it must be allowed to yield before damage to the support occurs.

To minimise roof movement, the setting pressure should be as close as practical to the yield pressure. It is not practical for them to be too close, and setting pressure (and reset pressure after yield occurs) is normally of the order of 95% of the yield pressure. The degree of movement involved in this process is referred to as the "stiffness" of the support, and for good roof control the stiffer the better. (A solid support rather than yielding would obviously provide the greatest stiffness but would not be able to be moved once loaded). The closer the reset pressure to the yield the more frequent is the yield/reset cycle which has an effect on the life of a support; supports are usually designed to last for a particular number of cycles taking into account the fatigue effect of continuous cycling.

Regarding the support geometry, to minimise the total support load to be provided, the shorter the support (in the face to goaf direction) the better, as less strata has to be supported. There must of course be adequate room to provide access for personnel to carry out production and maintenance operations, and space to contain the hydraulics and control equipment. Other factors to be considered however are the chock "aspect ratio" (the ratio of the support canopy length in front of the legs to the length behind the legs) and the bearing pressure on the strata. The aspect ratio affects the canopy tip load which can be applied and the attitude of the canopy, especially if strata crushing occurs at the goaf edge and it has been found that an aspect ratio of 2.6:1 or less is preferred (1). If the bearing pressure on the strata is too great, this can cause roof failure in itself, especially at the goaf edge where horizontal constraint is lost. A longer canopy may be required to reduce this bearing pressure, though it is rare for this to be a controlling factor in support design.

The load/m^2 provided by a support can be varied at the design stage by changing the area of the support canopy, though this is largely fixed for practical purposes. Load is most frequently varied at the design stage by changing the hydraulic pressure and/or the support cylinder diameter. In order to obtain the high pressures required on many faces it is common to have a two-pressure set, a lower set pressure (commonly at a pressure of the order of 320 bar) providing relatively high quantity, with a lower quantity high pressure set (of the order of 400-420 bar) to complete the operation. A higher load is more easily obtained by increasing the leg cylinder diameter, but this of course comes with increased cost and more difficult material handling if a leg replacement is required. As conditions become more onerous, both high pressure set and large leg diameters are required.

From the fact that support stiffness depends on the yield/reset cycle, it is readily apparent that leakage from the section of the hydraulics which is closed after resetting (the leg cylinders and controls) must be minimised. This becomes a vital component of support maintenance. Leakage from other parts of the hydraulic circuit will lead to wastage, increased pumping and fluid costs and to slower setting times. The latter can have some effect on roof control, but leakage from the closed section will lead to the worst consequences. There are advantages in being able to isolate individual legs on each support, allowing other legs to maintain their design capacity despite a leak on one leg. The ability to isolate and control legs individually can also assist in maintaining canopy alignment in the event of uneven roof conditions being encountered. This ability requires manual operation and means the automatic operation, if used, would have to be turned off, at least for that support or group of supports.

Because any looseness in support linkages has to be taken up before any pressure is applied to the roof, minimising this looseness by good maintenance is important to enable more rapid attainment of full set pressure during initial setting, but also to minimise misalignment between the support canopy and base, particularly in the thicker seams where such misalignment can become a serious problem. The misalignment can lead to an inability to obtain the full setting load and can lead to damage to legs, leg attachments and support structure.

Misalignment between canopy and base can also be caused by poor design and/or misuse of the support side shields. The side shields are intended to ensure a full cover of the roof when all supports are set in order to prevent loose material from falling between supports causing injury to personnel or damage to equipment. They are not intended to be used as an aid to steering of supports which may be tending to move down dip or towards the leading end of the face, and such use will lead to the misalignment problems noted above. This effect can occur unintentionally if the supports are tending to move in one direction and the side shield cylinders are allowed to provide their maximum push. .To try and avoid the misuse of side shields, some supports are fitted with a spring to push the side shields into place, thus restricting the force which can be applied, and only use hydraulics to retract them. Others limit the extent of side movement available, but this can lead to larger than desirable gaps between supports, particularly in high seams.

c) Shearers

Although this section should strictly be titled "coal cutting machines", in most of the world, and certainly in Australia, this involves shearers. The only other cutting machines of note with modern longwalls are ploughs. There are no ploughs in use in Australia at present.

While there have been many developments in shearers over the years, these have been in the main related to increases in size, power, control, monitoring and removal of cut coal. Such developments have certainly been major, but the essentials of the shearer are little changed since they were originally introduced into Australia.

A modern large shearer (thick seam)
viewed from the goaf side.
Photo courtesy of Joy Manufacturing Company Pty Ltd

Some of the biggest developments have been in size and power, largely associated with high production mines in thick seams. Shearers weighing up to 100 tonnes supplying 1 MW of power to the ranging arms are now in use with increases in the size of shearer haulage equipment to match speed to the power available.

The basic shearer support frames are similar to older versions only more robust, though where space allows a hydraulically controlled roll frame is included so the shearer is less dependent on the attitude of the AFC for controlling the profile of the next shear.

A view of the shearer in the previous illustration
from the face side.
Photo courtesy of Joy Manufacturing Company Pty Ltd

While the strength of all components has been increased over the years (usually involving an increase in size), for thinner seams as well as thick seams, and in particular the strength of ranging arms, close attention has been paid to maximising clearance around the shearer for coal flow. A badly designed shearer can incur a lot of wasted power pushing or grinding coal which has been cut but is trapped within the shearer structure.

As part of this attention to coal clearance, shearer drum design is of major importance, involving the number, depth and space between vanes, pick lacing and size and drum rotation speed, all related to shearer speed and coal strength and structure (and roof or floor strata if that is to be cut on a regular basis). Ventilation of the face side of cutter drums, assisted by the use of hollow shaft drums allowing air flow to the face side, is important in gassy seams.

Dust minimisation and control is another area which has received considerable attention by correct design of cutter picks and pick lacing, by good coal clearance and by the use of water sprays. The latter can be considered in two parts, application of spray water to the pick points and the use of spray curtains to keep dust between the shearer and the face so that it is prevented from mixing with the airstream in which personnel are operating. Usually the spray water is also used for machine cooling and is applied via a pump to increase the pressure above that of the mine reticulation system.

Summarising, the main points for consideration in shearer design are:

- Power requirements must be able to handle the likely worst conditions as well as the required production levels;
- Component strength, in particular for ranging arms;
- Ability to cut far enough into the gate roads to allow excavation when required to enable advance of the AFC drives;
- Ability to cut below the AFC pan line to correct cutting profiles;
- Machine size with regard to transport around the mine with minimum dismantling (cutting drums and ranging arms can be removed reasonably easily);
- Coal clearance ability around the shearer, including beneath the shearer body and when cutting in either direction;
- Machine size in relation to face ventilation (a shearer is effectively a mobile regulator on the face);
- Available control systems (radio, umbilical, local manual – it is usually beneficial for all 3 to be available);
- Availability of automated systems; and
- Availability of monitoring and fault diagnostics.

LONGWALL TOP COAL CAVING

A partial or "mini" build on the surface of the equipment for a Longwall Top Coal Caving face. Photo courtesy of Joy Manufacturing Company Pty Ltd

Despite the fact that there are numerous coal seams in Australia which are thick enough for LTCC to be used to increase recoveries, potentially at a reduced cost/tonne, there has been little application of the process to date. It is possible that several sites with potentially suitable seams have examined the process closely and concluded it is not suitable for their conditions. However the process is widely used in China, apparently successfully, and it seems unlikely that all Australian seams are different from seams in China, which in turn implies that there is a possibility of a wastage of resources occurring as no attempt is being made to recover the full seam thickness of most very thick seams in Australia. Certainly there is a financial risk involved in purchasing new equipment, possibly replacing equipment which is still operating satisfactorily, in order to trial a process which may not be successful. The potential benefits may be large, but so is the risk of spoiling a successful operation already providing a satisfactory return on investment, and avoiding this risk is probably the correct financial decision in most cases. LTCC certainly involves many unknowns and untried equipment and it may be the case that maximising the use of resources will require financial support of some kind by government (possibly an effort of the type being applied to longwall automation described in the following part).

Actual and potential problems of LTCC include:

- Caving characteristics of the top section of the seam are not known with certainty. At other than a greenfield site, observation of the caving of current goaves can be monitored, although the current cutting height may be different from any proposed LTCC cutting height (the current height could be reduced for a period for test purposes). It is essential for good recovery that the top coal caves readily immediately behind the support canopies, fractures into small enough pieces to allow good flow through the rear canopy doors or openings and small enough not to cause problems along the rear AFC. The Chinese industry has developed a caving index based on observable or measurable properties which may be applicable to Australian conditions, in principle if not in actual values. Failure to cave satisfactorily would result in an expensive failure of the system;

- It is not only coal caving properties which have to be considered, but the caving properties of the overlying strata. If this strata does not cave with the coal there is the possibility of large voids forming with detrimental effects on support loading, ventilation, possible accumulations of gas close to the face and possible wind blast effects;
- Considerable design work is required for the face ends to include the drives for the rear AFC, particularly at the main gate face end where the BSL would have to extend through the main gate supports to the rear AFC and be protected from being buried by the goaf. In China the main gate is frequently supported by manually set hydraulic props rather than powered supports, but this system may not be acceptable in Australian mines;
- The overall length of the roof supports would be greater than normal supports to cover the rear AFC and allow space for access. The main canopy length may be little different but the rear section would have to be considerably longer. This may affect the design load capacity required as well as the ability to transport supports during recovery and relocation operations;
- Control of the floor strata beyond the support base may be difficult, leading to floor heave beneath the rear AFC which may be very difficult to correct;
- Face ventilation requirements will be more onerous, especially along the rear AFC and particularly if there is an appreciable gas make from the floor. Such floor gas is likely to issue from close behind the support base, below the rear AFC;
- Because the face will have two production locations (the shearer and the current cave draw point), there will be two potentially major dust production zones and, unless the two processes are separated (i.e. shear first, then stop shearing and draw the caved coal for the full length of the face), some personnel will be forced to work in dusty conditions at some location for considerable periods. Separating the two processes may still lead to good production, but will slow the rate of face advance and make it intermittent, factors which are generally detrimental to good longwall mining practice;
- In some cases, the additional surface subsidence resulting from an increased thickness of coal removed may be a problem;
- Poor face alignment may not only cause face roof control problems but also affect the caving process leading to lost production;
- If the possible high production rates are achieved with both mining processes in operation concurrently, the capacity of the outbye coal clearance system must be able to cope; and
- Even if the system is able to work perfectly, an extended learning period is likely to be required for personnel to gain sufficient experience to achieve the required results. Experienced trainers can possibly be sourced from China in

the early stages but periods of training and gaining experience locally are likely to be at lower than optimum production.

So many risks, many with potentially serious consequences, present a major disincentive for any company to attempt LTCC without some form of financial assistance or guarantee. If successful the process could be extremely valuable in increasing recoverable resources and in reducing production costs. If not successful, it could result in a very expensive failure, and it is believed few companies are large enough to accept such a risk.

It is not known if the possibility of forced caving has been considered or attempted in cases where the top coal does not cave successfully. Holes could be drilled between supports by retracting side shields (or providing suitable size holes in these shields for drilling if retraction is considered unsafe). Shotfiring such holes probably involves too great a risk, especially in gassy seams. Pulsed infusion firing with the charge close to the face may be less risky, but in any case there are types of high expansion materials available which could be pumped into the holes and be successful in forcing the coal to cave as desired. The economics of such an operation would be a major consideration.

AUTOMATION

Progress has been made in Australia in many aspects of automation, largely as a result of research initiatives under the auspices of ACARP's Longwall Automation Steering Committee, carried out by CSIRO and CRC Mining in co-operation with Original Equipment Manufacturers (OEMs) and operating mines. The research has been/is being carried out in ten sections(1):

- Face alignment, focusing on lateral direction control and face geometry.
- Horizon control looking at maintaining cutting horizons as required by the mining process.
- Open communications by developing open architecture between systems hardware and software components.
- OEM involvement/commitment by creating mechanisms for exchange of information between the project team and equipment manufacturers.
- An information system consisting of a monitoring station, automatic sequence design and operator displays.
- Production consistency and reliability focusing on condition monitoring and reliability, and then on optimising coal flow and finally on collision avoidance between components.
- Determining the redefined functions of face operators and subsequent training requirements.
- An implementation plan for introducing automation system components at strategic selected sites.

- A commercialisation plan for transferring the proven automation technology to the industry.
- A progressive automation implementation plan for all longwall mines in Australia by identifying the status of their existing technical specifications and the potential to implement appropriate levels of automation.

It is obvious from the above that this is a very ambitious program and complete success in all aspects by he planned completion date suggested in some ACARP publications is unlikely. Further research is likely to be required in some areas while good progress has already been made in others. Putting all aspects together to create a fully functioning automatic longwall operation is also likely to be challenging and piecemeal progress is more likely (and is in fact occurring now, as most existing longwalls have extensive monitoring arrangements and some aspects of automation are already in use, or at least available).

One result of increased automation will be changes to the qualifications of the operating crews. While the traditional electrical and mechanical skills will still be required there will be an increased requirement for electronic and IT expertise on a regular basis. Alternatively, changes to equipment make-up and equipment fault self-diagnosis may result in fault correction merely involving changing out plug-in modules as instructed by the machine diagnostic system and fewer personnel with electronic qualifications will actually be required on site. The replaced item may be discarded or repaired elsewhere.

While automation should, in theory, reduce the need for skilled mining operators while operating normally, it is considered the need for such personnel will remain unchanged, at least in the near future. An automated face probably will not be able to recognise all changing conditions and the skilled mining operators will still be required to recognise when changes to the automated program are required to avoid loss of control or to prescribe the actions required if control is lost.

CUTTING METHODS

The basic cutting methods are Bi-di and Uni-di. "Half Web" methods have been introduced to provide several benefits, mainly in productivity, but at a cost of being more complex (not a cost once crews are fully trained). Some Half Web systems can provide similar benefits to Pre-cutting.

Depending on the equipment used and the mine's coal haulage capacity, Bi-di mining is often more productive than Uni-di, especially with longer faces, and can handle poor strata conditions better. Uni-di however is a simpler system to operate, enables operators to keep away from high dust concentrations, can allow the loading onto the coal haulage system to be made more even and can at times equal or even exceed Bi-di production. The Half Web systems can be used to gain the benefits of both Bi and Uni-di if operated correctly and in the right conditions.

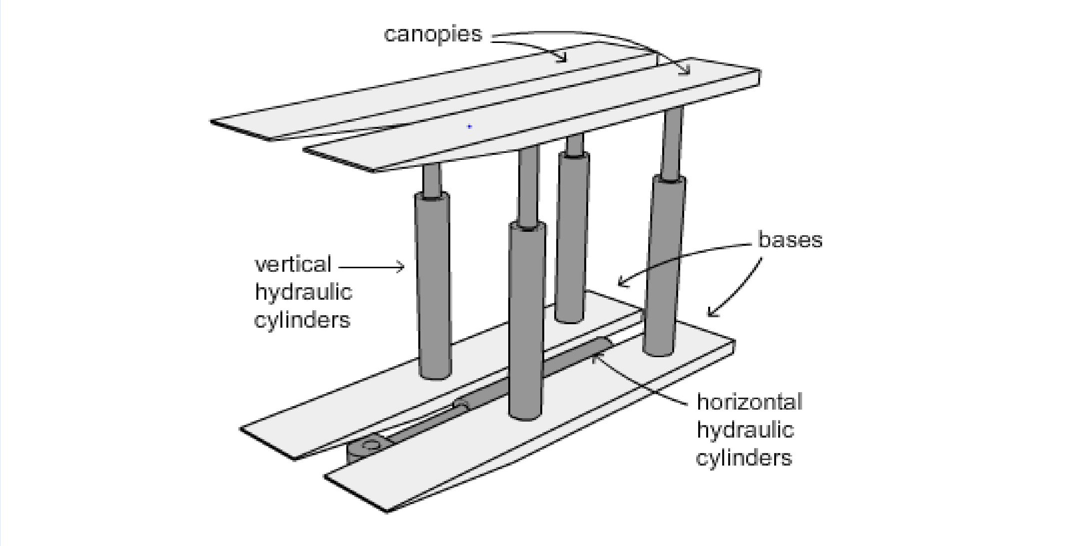
EQUIPMENT

Chocks (also known as "Powered Supports", "Supports" or "Shields")

Roof support in early longwalls (in the days of hand mining) was by timber props and bars, withdrawn from the goaf side as the faces advanced and re-used if still intact. Eventually these were replaced by steel bars supported by yielding props (eg friction props where resistance to yield was provided by a wedge system or hydraulic props which were individual props filled with fluid which could be pumped with an internal hand pump and released using a valve).

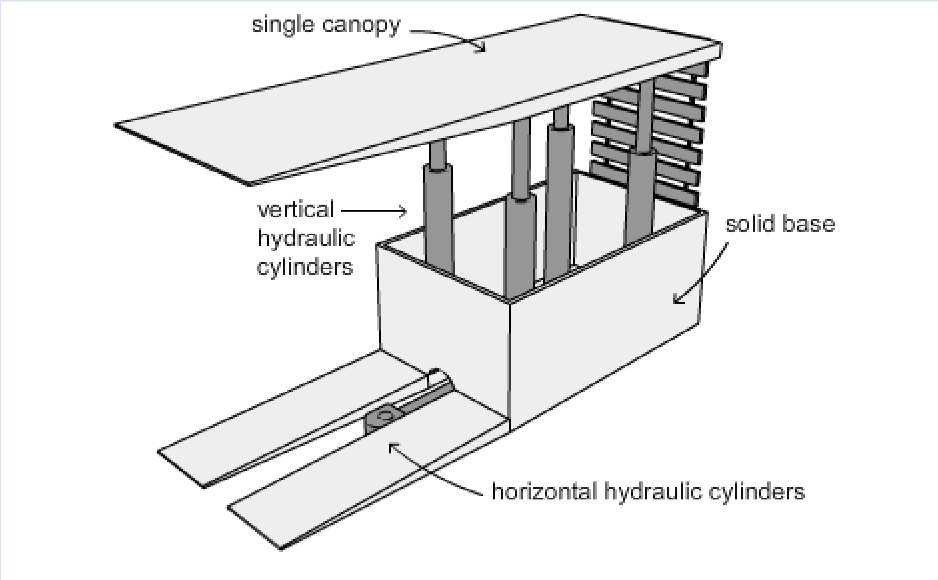
In time the hydraulic props (now referred to as legs) were combined in pairs, mounted on a base and joined with a roof canopy, with adjacent pairs being connected by a frame containing a horizontal hydraulic cylinder. This enabled each "chock", as the 4 leg sets were called, to advance itself with one pair of legs, released from the roof, pushing against the 2^{nd} pair which remained set. Such chocks were set along the length of the face forming a continuous line of "self-advancing supports", sometimes also referred to as"goal post supports or chocks".

Simplified sketch of "goalpost chock"



Further development saw the legs being mounted closer together on a single solid base with a solid, cantilevered roof canopy allowing the front line of legs to be a little further from the face while still providing adequate support close to the freshly exposed roof. The horizontal cylinder in these chocks attached the chock base to the face coal haulage system. The cylinders were used to push the AFC forward and then drag the chocks forward one at a time as the face advanced. The chocks were interconnected with hydraulic hoses and connected back to a pumping arrangement in the gate road by a hydraulic fluid reticulation system. The hydraulic fluid used was (and still is) mostly water with a low concentration of soluble oil, partly to assist in lubrication but mostly to inhibit corrosion. At times 6 leg chocks were used with 4 close together at the rear and 2 close to the AFC, leaving a travelling way between them.

Simplified sketch of early longwall chock



Over time the rear of the chocks was partially closed-in with flexible arrangements of steel plates, chains and timber to try and prevent broken material from flushing through from the goaf into the face area.

Similar support systems were developed where the 4 vertical legs were replaced by 2 larger legs set at the rear of the base and angled towards the face. These had a somewhat larger canopy with a rear section connected to the base with a "lemniscate" linkage which enabled the base and canopy connection to be fully covered, while the main canopy remained essentially parallel to the base at whatever set height was used. These supports were called "shields" instead of chocks.

Again, over a period, further developments combined the best aspects of chocks and shields into what were referred to as "chock-shields" but are now often referred to using either term by itself, but the terms "supports" is probably the most common term now used.

Most modern supports are two leg types, though four leg shields and chock-shields are also in use, all four legs mounted towards the rear of the base with the front pair angled towards the face and the rear pair towards the goaf.The rear section of the chocks which contains most of the operating valve systems and the legs, is partially covered by side plates as well as being enclosed from above and behind. These side plates have a top cover and can slide sideways, pushed by small hydraulic cylinders, so that the chocks can stand skin-to-skin and provide continuous cover over the full length of face.

In a further development the front tip of the roof canopy is articulated and connected to another small cylinder allowing a greater load to be applied to the roof at this point to improve roof control.

For thick seams, where coal falling from the face can be a hazard supports can be fitted with an articulated plate attached to the support tip which can swing down and provide a horizontal support to the exposed face. Once again this is controlled hydraulically and obviously has to be lifted clear again before the next web of coal is cut.

Most modern supports are fitted with "base lifters" another hydraulic arrangement which allows the base to be lifted up, a very useful function in soft floor conditions where support bases may sink into the floor and limit the ready advance of the face.

Modern supports are very complex pieces of equipment, made even more complex by the primary method of operation being via remote control or automated, requiring electronic control and monitoring, with manual control also being fitted. Longwall faces are also now extensively illuminated and these functions require hydraulic and electrical connections from support to support and back to the maingate area, a typical face carrying many hoses and cables. The hoses and cables between supports need to be flexible and have sufficient length to allow for the distance any support will be advanced ahead of adjacent supports.

Because of the high setting pressures of modern supports a two stage setting system may be used, an initial "low pressure" set (of the order of 320 bar) being boosted to the final set by a high pressure supply (of the order of 420 bar) - yet another set of hoses to be included.

Supports are designed to operate through a range of heights to accommodate variations in working heights and possibly some degree of unplanned loss of roof or floor. It is also necessary that they can be closed down low enough to allow transport around the mine in whatever height is available. Support legs are often multi-stage legs to allow additional travel.

Support widths vary, mostly between 1.5 and 2m. Note that as a chock is made wider the load/metre run along the face which can be applied to the roof reduces for a given leg capacity. Also the wider a support, the heavier it becomes and more difficult to handle. As a support is made narrower it becomes less stable if subjected to uneven ground. Also the narrower a support, the more supports are required for a given face length and with each support requiring a set of control valves and interchock hoses, so the greater the cost. The ideal support width will be the best compromise between the conflicting aims.

An aspect of support design beginning to receive more attention is ergonomics. When a face is "closed-up", especially in lower height seams, travelling along the face can be very arduous. To increase the width of walkways involves extending the length of the supports which has ramifications on roof loading and strata control (as well as costs), so some degree of compromise is required.

There is a tendency to design for "average" size personnel which, by definition, means that half the workforce are likely to experience some difficulty or discomfort. It may be better, within reason, to design for the tallest and widest person likely to travel the face. Allowance has to be made for equipment being carried on a regular basis (cap lamps, self rescuers, etc).

The purpose of the roof supports on a longwall face is not to prevent roof movement but to control it so that the immediate roof remains essentially intact where the coal is cut and within the area of the face where personnel have to work. Once the work area has moved forward it is acceptable, indeed desirable, that the roof collapses or "caves" (a term frequently used). The ideal situation is that the roof caves immediately behind the supports as they are moved forward; if the collapse is delayed the roof strata will hang out into the goaf in a cantilever putting extra load on the supports.

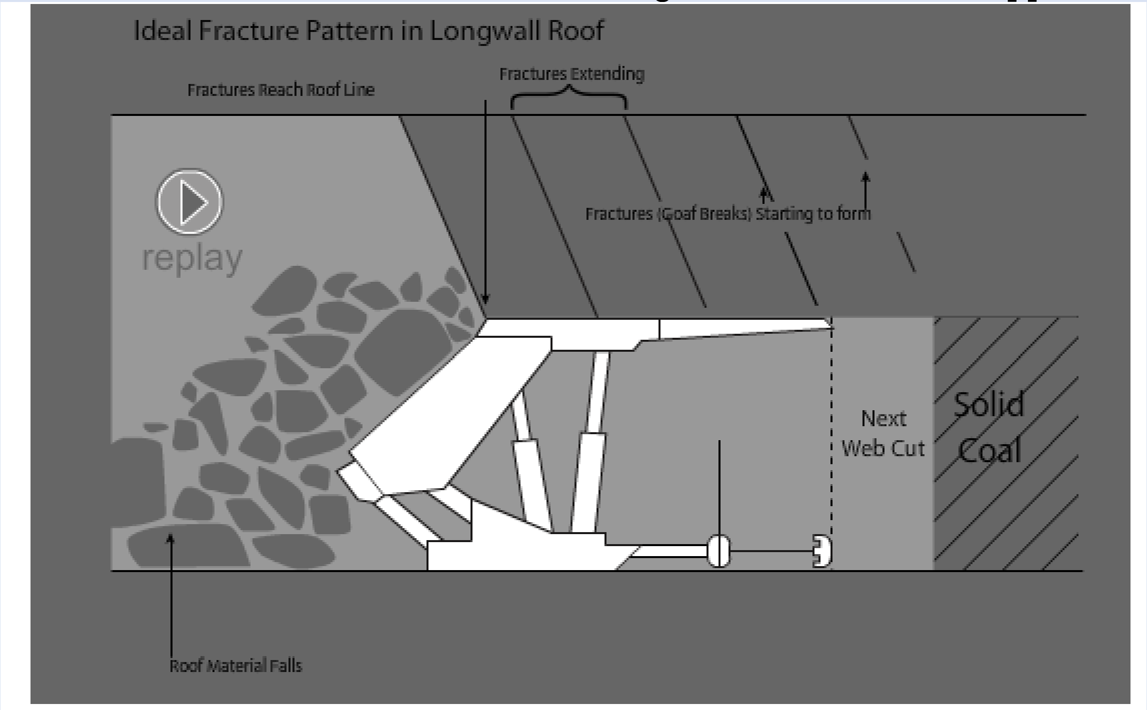
This cantilever effect was largely responsible for early failures of longwall mining in Australia, circa 1970. Supports at that time had been developed in Europe where roof strata was generally weaker and laminated and caved readily. Support loading capacities of 100 tonnes or less were adequate to control the roof. In Australia more massive strata is common which breaks and falls less readily and the cantilever effect leads to very high chock capacities being required, sometimes over 1000 tonnes.

In order to prevent damage to the hydraulic legs, chocks are designed to yield (ie release the hydraulic pressure) at a set value, so the roof is allowed to lower in a controlled fashion.

For best roof control a high chock set pressure is required, as close as possible to the yield pressure – it is not practical to set at the yield pressure.

As the roof lowers, the strata above will begin to bend and then beds will fracture under tension from the higher levels extending down towards the immediate roof (these fractures being known as "goaf breaks"). Ideally these fractures should reach the roof as the rear of the supports passes that point, allowing immediate caving.

Ideal formation of roof fractures, reaching seam roof at rear of support

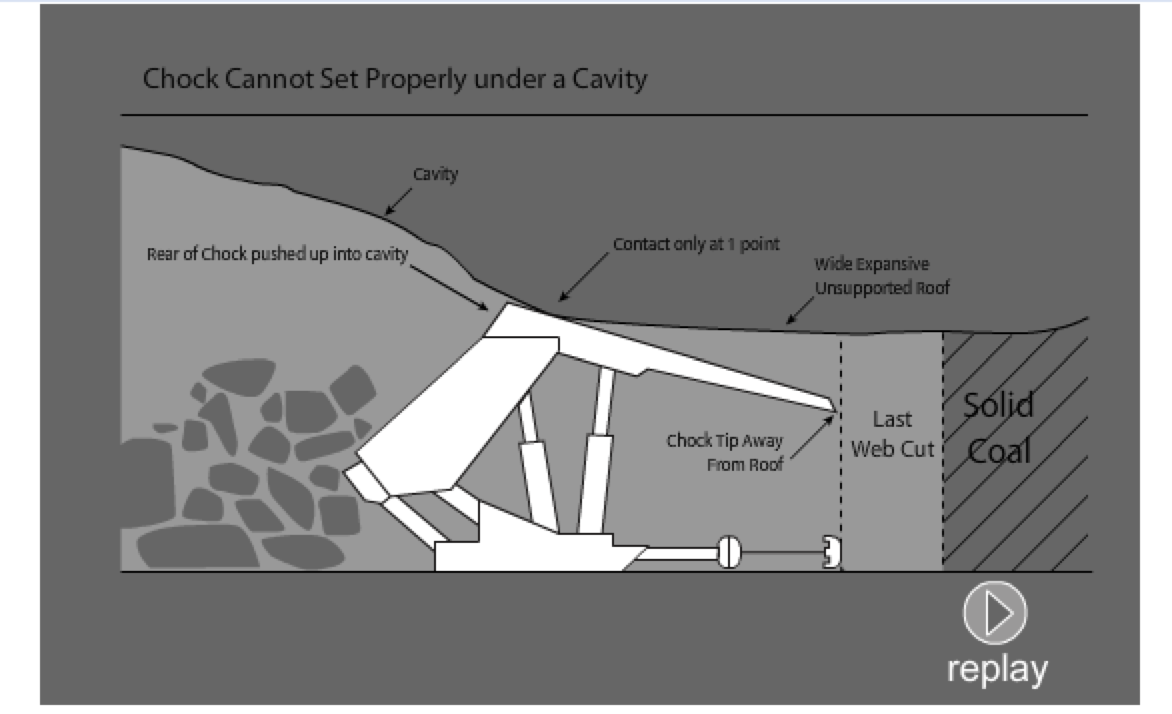


If it occurs ahead of this point then the roof above the chocks will break-up and this may give rise to problems on the face itself. Broken roof above the canopies is not usually a big problem as the broken strata will remain more or less in place as long as the chocks are not lowered away from the roof when they are moved. Broken roof ahead of the canopies can lead to total loss of control and major falls on the face itself.

Cavities or broken roof above the canopies can result in:

- Not being able to set the supports to the roof at the designed set pressure because the canopy has to remain reasonably level; if there are cavities one end of the canopy may push up into the cavity causing the other end to come away from the roof
- Very high point loads at contact points between strata and canopy which have to take the share of load which cannot be transmitted through the cavity.

Sketch to show effect on support setting (if full pressure set used) of cavity in roof over chock



Armoured Face Conveyor (Face Coal Haulage)

Once the coal is cut, it has to be removed from the face to the maingate, so some form of coal haulage system is required. In early hand-working days, belt conveyors were used, but these are not amenable to high production rates as the whole conveyor had to be moved sideways to advance the face.

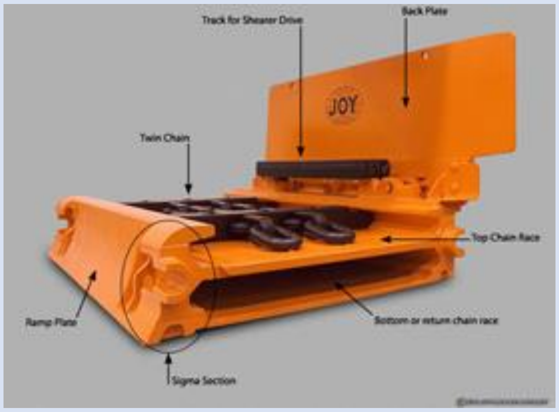
The major development, apart from self-advancing supports, allowing high production longwalls to evolve was the "Armoured Face Conveyor or AFC" which was originally developed in Germany and was frequently (less so nowadays) referred to as a "Panzer" conveyor.

Essentially an AFC is a one-sided trough scraper conveyor, the second side of the trough being formed by the coal face. Cut coal falls into this trough which has an endless chain with scraper flights attached running along the base plate and returning below the base plate in an enclosed lower section or "race". The coal is dragged along the base plate by the flights.

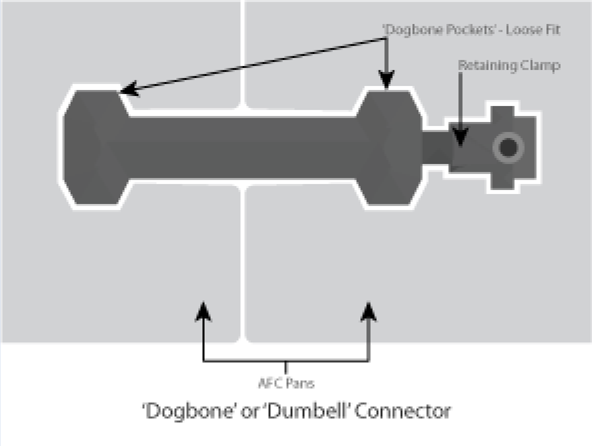
Early AFC's had two outboard chains, one running each side of the base plate with flights connecting them. Later developments saw the two outboard chains replaced by a single centre chain with flights cantilevered each side. With increasing longwall size, the single centre chain has now become a double or triple centre chain on almost all longwalls.

The outer ends of the AFC flights (or outer chains on old AFC's) are kept in place on the base by a channel section lip at each side of the base. A similar arrangement for the return chain below the base leads to this part of the AFC being known as the "Sigma Section" because of its shape.

The chain is driven, via sprockets, by electric motors at both ends of the face (maingate and tailgate, although earlier AFC's were driven by maingate drives only). These drives must sit in line with the face at each end and the size of drive(s) required is often the limiting factor on the practical face length (along with AFC chain strength and size). To accommodate the sprockets and shafts, specially shaped AFC sections have to be used at each end of the AFC to raise the chain path.



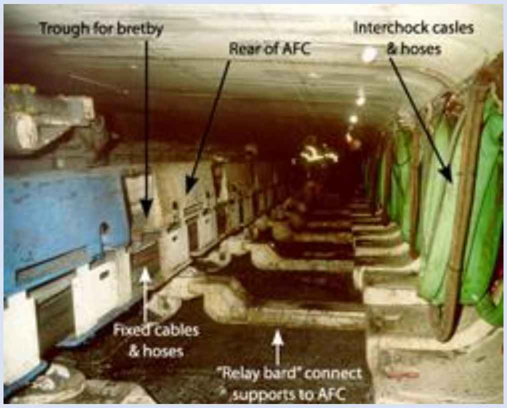
To provide a degree of horizontal and vertical articulation in the AFC along the face, the AFC consists of many individual sections joined together with flexible couplings, these sections being known as"pans". The length of each pan is normally the same as the chock width, so the face has the same number of chocks as pans over most of its length (some special arrangements are required at the face ends). The flexibility allows the AFC pans to be moved in differing amounts along the face so that short lengths can be pushed towards the face forming a bend or"snake".



The connection between pans is by means of dumbbell shaped bars (known as "dog bones") slotted quite loosely into brackets mounted on the back plates and the front of each pan.

Pans with a removable section in the base plate, known as "inspection pans", are spaced along the face, to allow access to the bottom race for cleaning or repair purposes as well as inspection.

In order to move the AFC forward after the shearer advances along the face, hydraulic cylinders mounted horizontally on the support bases are attached to the steel (or goaf) side of the AFC trough, referred to as the "back plate" or"spill plate". These cylinders are double acting, and besides being able to push sections of the AFC forward, are utilized to pull forward or advance the supports, once they are lowered from the roof.



The main and tailgate drives are mounted on base plates which have to be rigid (the drive trains can only have minimal flexibility), so support attachments at the face ends are generally via rigid structures and the drives are pushed forward by several chocks at the same time.

Coal is actually cut in front of the AFC, so a lot of the coal initially sits on the base of the cut section and does not fall directly onto the AFC. This coal is lifted onto the AFC as it is pushed forward. Because it is necessary to cut the maximum amount of coal on each web, it is important that this coal is able to be cleaned up as the AFC advances and not left on the floor, thereby preventing the AFC being pushed right to the solid coal. For this reason the front edge or toe of the pan is normally built in a wedge shape, to ramp up the coal at the face and assist in the clean up (hence the toe of the pan is sometimes called the "ramp plate"). In the past small ploughs were sometimes fitted to the front of pans and dragged along the length of the face to improve the loading (commonly referred to as "activated ramp plates"). Improved toe plate design on modern longwalls has obviated the need for theses activated ramp plates.

Apart from the AFC's primary function of removing the cut coal from the face, its structure is used for other purposes:

- The edges of the base are used as tracks for the coal cutting machines (shearers) to run on
- Tracks or chains utilized for haulage of the shearer are mounted on the AFC in most cases, and
- Troughs and attachments are mounted on the back plates to carry fixed and trailing cables and hoses

Cutting Machines - Introduction

There have been several different types of cutting machine developed over the years, but "shearers" and"ploughs" have proved to be the most successful, especially the former and will be the only types dealt with in any detail here. Some types, such as "trepanners" were designed to produce a larger size of coal required for boilers, particularly for railways, and have dropped out of use as this need disappeared.

Coal Ploughes

Coal Ploughs have had little application in Australia, and their main use has been in Europe, particularly in Germany where they were first developed. Essentially a plough is a large mass of steel, usually of a more or less triangular shape when viewed from the coal face or goaf sides, fitted with large "picks" (more like small agricultural plough blade tips) angled from the steel body towards the coal face. The plough height is the working height in the seam being mined (possibly a bit lower if the coal tops can be guaranteed to fall once the coal below is cut. These "picks" act in a fashion similar to chisels and break a narrow web of coal off the face (of the order of 300-400mm thick). In most cases there are no moving parts on a coal plough.

The plough itself is mounted on the front of the AFC and is pushed into the face by push cylinders mounted in the supports. The plough has an endless chain haulage attached to the rear, and is driven through sprockets on electric drive(s) at the face end(s).

The main advantages of ploughs compared to shearers are:

- Cheap
- Simple (no moving parts on the cutting machine itself)
- Relatively low dust make
- Able to keep exposed roof area very small (but a large number of chock movements would be required to maintain this.

Though only a small web is taken, in the right conditions production rates can be comparable to a shearer as the plough is operated at a relatively fast speed along the face.

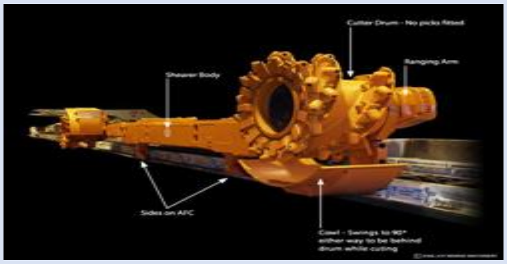
Some disadvantages are:

- Cutting height is fixed
- Ability to cut stone is limited
- With increasing cutting height, machine stability becomes more problematic

- Grading can only be done using the AFC angle
- There are safety implications with an exposed chain haulage.

Shearers

Shearers are by far the most commonly used cutting machines, both in Australia and overseas. A shearer consists of a machine body containing electric motors, hydraulic equipment and controls which is mounted over the AFC. Horizontal cutting drums are mounted on the face side of the machine, laced with cutting picks and rotating in a plane parallel to the face. If the AFC is pushed towards the face as the cutting drums are rotated and the shearer travels along the face, it is able to cut into the face for the full web width, moving along a snake in the AFC. This is known as "sumping in". Once fully into the web, the shearer can advance the full length of the face cutting out the web. The snake can also be reversed to cut the wedge shaped portion of coal left while sumping in.



The first shearers had a single drum rigidly attached to the body of the machine and so had a fixed cutting height and had to be able to get the body of the machine past the edge of the face at one end to cut the full length. Further developments saw the cutting drum being mounted on a hydraulically operated arm to vary the possible cutting height and a second cutting drum being fitted to provide a drum at both ends of the machine. Such machines were originally known as "Double Ended Ranging Drum Shearers or DERDS" but being almost the only type of shearer now used, the term "shearer" is generally understood to refer to this configuration.

The double ended ranging drum arrangement also allows smaller drums to be fitted while still cutting the required height, either cutting part of the seam with each drum in one pass or as two separate passes. The ranging arms allow much easier grading or changes in cutting height where required.

Shearers are mounted onto AFC's using either steel rollers or sliders running on the two edges of the AFC structure, either trapped or sufficiently guided by flanges to ensure derailing will not occur. The clearance between the bottom of the shearer body and the AFC base is an important design consideration as all coal cut on the tailgate side of the shearer has to pass under the shearer body on its way to the maingate. Any severe restriction at this point can cause problems, either causing coal to build up and spill over the AFC back plate or causing the AFC to jam. This is particularly the case where coal frequently comes off the face as large blocks or where roof stone has fallen on the AFC. In some instances a small coal breaker is fitted under the shearer at the low point to prevent blockages occurring.

Most shearers use radio control for normal operations although they are also fitted with manual and/or pendant controls for use in emergency conditions.

Shearer Haulage

Chain haulages have been used to drive shearers along the face, but modern units use a rack and pinion arrangement for this purpose. Rack sections are attached on the face side of the AFC back plates and drive pinions, usually hydraulically driven, are attached to the goaf side of the shearer body. Stops are built into or placed in the rack sections at each end of the face to limit shearer travel and so prevent damage to other equipment in those locations.

Shearer Cutting Drums and ancillaries

Shearer drums are not continuous cylinders but have a small diameter central section with scrolls or vanes from there to the full diameter, the picks being attached to the outer edges of the scroll. The pick lacing is set to optimize cutting for the seam being worked, while the vanes are used to move the coal efficiently from the cutting area onto the AFC. Inefficient loading will lead to a lot of the power applied to the drums being wasted in churning and crushing coal already cut in the drum area. The pick design is also important to maximize efficiency in cutting and avoid wasting power in this process.

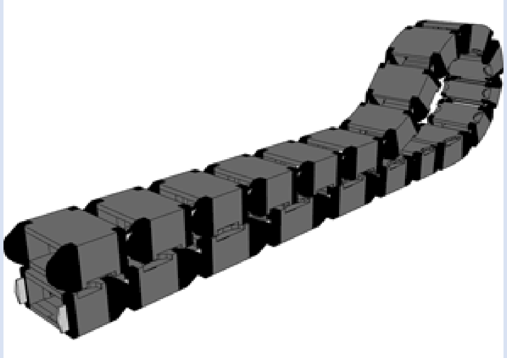
Another device used to improve coal loading is a "cowl". This is a curved steel plate mounted on supporting arms so that it sits behind a cutting drum to prevent coal passing from the back of the drum into the area where the web has been cut, the coal being forced to move sideways towards the AFC instead. The cowl is fitted with a motor and drive pinion so that the cowl can be rotated from one side of the drum to the other when the cutting direction is reversed.

Some shearer drums now have a hollow shaft venturi ventilation system design to ventilate the area on the face side of the drum while cutting, thereby preventing a possible ignition of gas given off in the cutting process.

High pressure water is directed through the drums to the picks for dust suppression and pick cooling purposes, including water which has been used for cooling the shearer motors. Additional sprays are also fitted on the shearer for dust control, often acting as curtain sprays to keep dust on the face side of the shearer away from personnel rather than actually trying to settle the dust.

Cable and Hose Handling (Bretby)

A shearer has to be supplied with electric power and water and as it is continuously moving, trailing cables and hoses are required. Unlike a continuous miner which is mostly only moving forward with intermittent reversals, a shearer is continuously moving backwards and forwards along the entire face length and in a very congested area. A means of managing and controlling the trailing cables is therefore required, The "Bretby" (named after the location of the research and development section of the British National Coal Board, where the device originated) is a flexible carrier consisting of a series of flat "plates" usually of a plastic material connected in the form of a chain but which only flexes along its length, similar in appearance to tracks on a "caterpillar" driven machine but with a double row of plates. With the aid of pins on the outer edges, this forms a container which is flexible in one direction only (in a vertical plane when installed on the face).



Fixed cables and hoses are run in a services tray attached to the back plate of the AFC from the services supply end of the face (normally the maingate) to mid-face. At this point the shearer trailing cables and hoses, enclosed within the Bretby, are attached to the fixed cables and hoses, with the Bretby then being run in a cable trough attached to the AFC back plate (above the services tray) to both ends of the face as the shearer moves from end to end. The other end of the Bretby is attached to an arm fixed to the goaf side of the shearer of a length to hold the end of the Bretby directly above the cable trough and at a height such that the cable is bent at an acceptable radius when dragged back on itself.

When the shearer is at the tailgate, the trailing cable is fully stretched from mid-face. As the shearer travels to the maingate, the Bretby doubles back on itself to the mid-face position and is then dragged out to full extension again when the shearer is at the maingate. The process is reversed when going back to the tailgate.

Care is needed if the shearer is taken backwards and forwards in one area as the Bretby will be trying to go back on itself several times. There is a height limit to the number of times it is possible to do this, as well as a limit on the radius the cable can be bent to.

Beam Stage Loader and Crusher

When the coal has been hauled to the maingate, it then has to be transferred through a 90° turn, and loaded onto the maingate conveyor. This function is carried out by the "Beam Stage Loader or BSL", which is another scraper conveyor, in this case with steel plates on both sides and runs from the maingate drive to the maingate conveyor (belt). The BSL has a change of elevation (a vertical curve) along its length in order to discharge coal onto the maingate conveyor and. in almost all cases, a crusher or breaker is mounted on the BSL to improve loading onto and to prevent damage to the outbye conveyors. The BSL ends above the boot and is normally attached to it with an attachment able to rotate horizontally (and with a limited amount of vertical flexibility).

In most cases, the BSL runs past the maingate drive to the goaf side and the AFC is slightly elevated where the two scrapers cross. There being no base plate to the AFC at this point the coal drops from the AFC onto the BSL. This design is known as a 'side discharge'and a curved guide plate is often mounted across the BSL at the loading point to assist in guiding the coal around the right angle turn.

An alternative arrangement is where the AFC stops at the face side edge of the BSL scraper so that the two scrapers do not overlap. This is known as "end discharge" and could be preferred where slabby stone may have to be handled.

The crusher or breaker is usually located along the low section of the BSL and is usually a rotary breaker with the drum acting against the floor of the BSL scraper conveyor.

The BSL section with the vertical curve or "Goose Neck" normally comes as one piece, but the horizontal section between the maingate drive and the crusher is made up of a number of "pan " sections which have some flexibility, similar to AFC pans.

The BSL is usually attached to the maingate drive and is moved forward by the maingate supports at the same time as the drive is advanced.

Maingate Conveyor & Boot End

Because the face moves continuously along the maingate, at a steady rate if operating correctly, the required maingate belt length is continuously changing. To allow for this and to avoid too frequent stoppages to shorten the belt, it is normal for special arrangements to be made at both ends of the belt.

At the face end, a special boot end is installed. Reference has already been made to the attachment of the outbye end of the BSL. The boot is either pushed outbye through this connection along with the maingate drive and BSL when it has to be moved or it may be a self propelled, caterpillar track mounted unit. The boot is anchored against the belt tension by its connection through the BSL or may be wedged to the roof with jacks – it is not pinned or fixed in any other way as is a conventional belt.

The type pushed outbye by the maingate supports will seldom move in a straight line and in order to keep it in line with the belt there are arrangements included on the boot to enable it to be steered and/or to move it sideways to line it up as required. These arrangements normally involve hydraulic cylinders. Levelling jacks are also required.

At the drivehead end of the conveyor it is normal to install a special "Loop Take-up" ("LTU") with a structure much longer than a conventional LTU and also including rollers to allow the bottom belt to fold back over itself a number of times thus greatly increasing the length of belt which can be stored in the LTU. With this arrangement it is only necessary to stop the longwall to remove belting infrequently and this operation can usually be tied in with other planned stoppages (e.g. maintenance).

With the above arrangements it is possible to change the belt length in frequent small increments for a considerable total distance. The conveyor structure is removed in short sections immediately outbye the boot and for this purpose a pinned type of structure (as opposed to bolted) is preferred (for easy removal).

Face Services

Face services include high and low voltage electric power, water, compressed air, hydraulic fluids and possibly pumping. These are required for:

- High voltage power, increased over time from 415v to the present preference for 3.3kv; required via trailing cables to power the shearer and via fixed cables for the main and tailgate drives
- Most faces now have lights attached to the chocks on a 240v supply via interchock cables
- Low voltage power is required for monitoring (typically operating parameters and location of the shearer, chock operating parameters and gas concentrations at certain locations) and for control of chock operations.
- Water is required for cooling of motors (main and tailgate drives and shearer), for dust control (on the shearer drums and the crusher) and for fire fighting precautions. It may also be needed for secondary roof bolting or drilling for various purposes on the face, particularly during face recovery operations.
- Compressed air is seldom used on the main face equipment but is frequently required for pumping, roof bolting or drilling as above and possibly for ventilation purposes such as operation of venturi's. It may also be useful for machine maintenance and repair purposes.
- Hydraulic fluid, usually a soluble oil/water emulsion, is required throughout the face from the boot end to the last support at the tailgate. It is normally supplied in a closed circuit with a return line running the length of the face back to the pumps in the maingate.
- The pumping referred to here is of waste water which may collect at low points on the face or in the gate roads and needs to be removed, possibly from one end of the face to the other.

It is normal for water and compressed air to be supplied from the mine reticulation systems, which would be in steel pipes in the gate roads up to a point close to the face and would then be carried onto and across the face in hoses with outlets at suitable intervals. The water supply to the shearer would usually be run onto the face separate to the supply for other purposes and may require a high pressure line for dust suppression purposes.

Electric power and hydraulic fluid is supplied from equipment usually in the maingate or adjacent roadway, consisting of transformers, switchgear and electrical distribution boxes, hydraulic pumps and tanks. There is also usually an enclosure containing face monitoring and communication equipment.

On early mechanized longwalls, all this equipment was mounted on skid or rail mounted bases in a long line alongside the maingate conveyor. The bases were all connected together and were dragged forward as the face advanced, in some cases with a winch and in others with a large hydraulic cylinder mounted on a base with a large staker prop to the roof, also hydraulically operated. The latter usually used a chain to pull the equipment forward.

All this train of equipment is known as a "Pantechnicon", a name generally shortened to "Pantec".

Later developments have included mounting the pumps remotely, sometimes in the main headings in specially constructed pump stations, with the fluid being carried in steel pipes for most of the distance to the face. This has the advantage of the pumps being set in good, clean conditions and not having to be moved frequently, sometimes being used for more than one longwall block. The disadvantage is having to have pipes in the maingate carrying high pressure fluids with associated safety issues and pressure losses in the pipes.

More recent longwalls have a set-up between the two set-ups described above, where the pumps are located in a roadway adjacent to the face some distance outbye the face position (of the order of 500m) and relocated when the face is somewhat closer (usually before their location becomes part of the "hazardous zone" where additional protection such as flame-proofing is required). In this way the pumps, etc can be located in relatively good conditions away from the immediate face area, are moved less frequently than if they were in the face area, but have a smaller reticulation system than the outbye stations. The transformers supplying power can also be located a similar distance from the face.

A large number of cables and hoses have to be run between the Pantec or remote pump stations and transformers and this requires some type of cable handling arrangement. To protect them in the area closer to the face, they are normally run in a covered steel trough run alongside the BSL, known as a "Rigid Bretby". The enclosed trough runs around the back of the maingate drive onto the face itself.

With a Pantec there is only a very short distance from the Rigid Bretby to the end of the Pantec and the cable/hose lengths required are fixed.

With a remote pump station/transformer set-up the cable/hose lengths are very variable Rather than try to manhandle and add/remove lengths, a monorail system is typically used. Loops of cables and hoses are attached to monorail trolleys such that the loops can hang almost roof to floor or can be stretched out almost horizontally, the difference in overall length being the distance required for the pump/transformer relocation. The monorail consists of short sections of rail which are hung from roof bolts. As the face approaches, rail sections are removed and re-hung outbye ready for the next relocation of the pumps/transformer.

Other items (eg tool boxes, fire fighting equipment) can also be hung from the monorail if desired.

On the face itself, some cables and hoses require connection to every chock. These are made in short lengths, known as interchock hoses or cables, and often hung together in a cover to make handling easier. Sockets/connectors are mounted on the chocks for attachment of these services with through connections installed to connect to the next chock in line.

Cables and hoses which require no or infrequent outlets/connections are usually run along brackets on the AFC back plates.

Lighting

To improve safety and working conditions in general, longwalls now include lighting on the face, often to the extent of a light unit set in every chock. Area lighting is also frequently included on the shearer, on maingate equipment and at remote pump stations and transformer installations.

For a longwall to work at its best it should operate as near to continuously as possible, so high maintenance standards are desirable to minimize unplanned stoppages. Good maintenance of hydraulics and electrical/electronic equipment requires close attention to cleanliness and good lighting is of assistance in this aspect.

Pre-cutting

In thick seams where coal is prone to fall from the face in large slabs, these can cause problems in that:

- they can bridge across the AFC and need to be broken before they are carried away
- they can create a hazard if they protrude out of the AFC into the chock area where personnel are located
- in part they can fall into the chock area and damage personnel, hosing, cabling and apparatus
- they can jam under the shearer or at the maingate/BSL transfer and cause a blockage and spillage.

To prevent this some thick seam shearer drums are fitted with an extension of the centre of the drum which cuts a small section out of the next web as a web is being cut. Any potential slab is thereby pre-cut and will fall in smaller sections than would otherwise be the case.

STRESS/STRATA CONTROL

Stress and strata control around a longwall can be considered from two aspects:

- Along the longwall face
- In the gate roads and associated roadways

Along the Longwall Face

Along the face, the roof control function is a matter of providing sufficient resistance to control the dead load of the mass of strata which is breaking, or has broken, away from the bulk of the overlying strata. The chocks are not resisting the whole vertical stress field, most of which is redistributed to the solid coal ahead of the face (the "front abutment load") and to re-compacted material in the goaf behind the face (the"rear abutment load"). Provided the chock load capacity is sufficient to control the movement of this dead load so that the roof in front of (and preferably above) the chocks remains intact, face conditions will remain good.

Horizontal stress is not really a major factor in roof control along the face. There will be zero stress in the direction perpendicular to the face once a goaf has formed and stress on the long axis of the face will not create problems. Horizontal stress in the floor however can give rise to floor heave under the pans which can become a major issue. The angle of the pans controls the attitude of the shearer and if the rear of the pans is lifted it often becomes necessary to straighten them before continuing, a difficult and time consuming operation.

Because reliance is being placed on controlling roof lowering which is expected to occur, major problems may arise if a face is stopped for any length of time. It is possible that a stable condition may be attained where movement ceases. It is equally possible that movement will continue to the point where the roof starts to fail ahead of the face and even to the point where the chock legs are fully closed and cannot yield further, a state known as being "iron bound".

Ideally a longwall face should operate as near to continuously as possible and at a steady rate in order to maintain good face conditions. It has often been the case that a relatively minor problem which has caused the face to stand for a period leads to a deterioration in conditions which then compounds into a major event.

Some degree of movement not only ahead of the chocks but slightly ahead of the face can be useful. This movement and resulting load on the uncut coal can cause crushing and fracturing leading to a much reduced power demand on the shearer, sometimes to the extent that it becomes a loading machine rather than a cutting machine. While this effect can be beneficial it should not be a design aim as it occurs near the point where roof control is lost and this would not be an ideal operating state. Even if roof control is maintained initially, too much load on the face coal will lead to slabbing, creating a wider expanse of unsupported roof with roof control problems possibly ensuing, which is exacerbated in high seams.

In the Gateroads and Associated Roadways

The gate roads are frequently major causes of concern with regard to strata control around a longwall panel. .{You are reading it on mineportal.in} As well as having to have sufficient support to remain stable during development and for the period before the longwall passes, they have to withstand the redistributed stresses and abutment loads which arise ahead of, behind and to the sides of the advancing face

Around the face ends there is also a redistribution of horizontal stresses. It is often the case that one end of the face is protected by an adjacent goaf (in a "sress shadow" area) and horizontal stresses are carried in higher strata horizons away from the immediate roof. If the current longwall is extended beyond the old goaf, the stress shadow effect is lost and a stress concentration is likely instead. There will be a stress concentration at the other end of the face and such stress concentrations can be up to twice the normal stress levels and can cause compressive failure of the roof strata. It is preferable for the face to be oriented so that such concentration is applied to the minor principal stress rather than the major, but this is not always practical, especially where stress fields change orientation.

During development, roof strata support design is based on high residual horizontal stress levels. These dissipate after the longwall has retreated, often resulting in roof failures due to the lack of confining stress.

Additional support to that required during development is often needed and is usually provided by means of extra long bolts of some kind (or even trusses in weak strata), sometimes in the floor as well as the roof.

In the tailgate it is often practical to install passive support before it is affected by the face, usually in the form of "cans" (large diameter, thin walled steel tubes filled with lightweight concrete) or timber chocks. It is preferable for these to be set to one side of the roadway and still allow vehicle access to the tailgate drive area. If this is not possible, then it is best to leave the installation until just before the effects of the face become apparent.

Because of the conveyor and equipment in the maingate and associated roadway(s) passive support is not usually an option and fairly heavy secondary bolting is often carried out. It is best if this can be done well in advance of the approach of the longwall, ideally between the completion of development and before the longwall start, if there is sufficient time available.

It is ideal if an assessment of strata conditions can be made during development and the extent of secondary support required in different sections planned in detail rather than a blanket approach to the full length of the roadways. More dense support, which may include full mesh coverage of ribs is often applied adjacent to longwall start and finish positions, particularly the latter which has to remain stable under the abutment load for an extended period during longwall recovery.

As the longwall retreats, passive support is often installed in cut-throughs as the face passes when access is no longer required. This is to prevent the goaf running down the cut through and affecting the intersection in the remaining roadway, if it is required to remain open.

Another important issue affecting the stability of the gate roads is the design of chain pillars. These have to be such that they remain intact during the life of two longwalls (assuming longwall retreat and re-use of the second roadway at the maingate end as a tailgate for the second longwall).

SHORTWALL MINING

This method of mining was developed in the late 1960's to take advantage of the then recent development of suitable hydraulic longwall supports, coupled with the productivity and low capital cost of continuous miners and shuttle cars. In effect it gained some of the advantages of longwall mining without the cost of installing a complete set of longwall equipment

An installation roadway was driven as for a normal longwall, but only supports were installed. A continuous miner was then utilized to cut 3.5m wide open ended lifts off the face, with shuttle cars being used to transport coal along and off the face to the maingate belt in lieu of an AFC.

The face length was therefore limited by the length of shuttle car cables then available, but in practice most shortwall faces were considerably shorter than this (<90m).

Supports were connected to a reference rail which was then utilized to pull the 2 or 3 leg supports forward, in a similar manner to the use of an AFC to advance longwall supports.

Shortwall faces could be installed between two gate roads as for a longwall face, but in some cases were mined to a blind end and ventilated by auxiliary fan (not very suitable for gassy seams as the fans could draw from the goaf).

Shortwalls were used in an endeavour to increase the productivity of continuous miners at relatively low capital cost, sometimes as a transition stage while changing a mine to full longwall. In some cases, because they were somewhat more flexible, shortwalls were used to obtain the benefits of longwalls in mines, or parts of mines, where seam discontinuities or mine geometry made the use of full longwalls impractical

The main disadvantages of shortwalls compared to longwalls are:

- The width of the unsupported roof ahead of the chocks is governed by the width of a continuous miner as opposed to a shearer drum.
- Personnel have to work adjacent to the face which presents safety issues unless rib support is installed which would greatly slow production.
- The use of shuttle cars is by its nature not continuous and brings in all the disadvantages of trailing cables in the face area.
