1.1 DEFINITION OF GROUND CONTROL

Consider the stratigraphic cross-section of a coal mine. Before any opening is made, rock mass everywhere in the cross-section is in equilibrium in its virgin state. Once an opening such as an entry is made in the coal seam, the coal seam and rock mass in the vicinity of the opening are no longer in equilibrium. Because the rock mass in the roof has lost support from below, the floor rock no longer has an applied load from above, and the coal seam is no longer constrained along the sides (ribs) of the opening. Therefore, the surrounding rock mass and coal tend to deform into the opening. If no artificial supports are erected, the time interval between exposure of the roof, floor, and ribs and their collapse will depend on rock properties and local conditions.

Ground control is the science that studies the behavior of rock mass in transition from one state of equilibrium to another. It provides a basis for the design of support systems to prevent or control the collapse or failure of the roof, floor, and ribs both safely and economically.

The easiest way to maintain the original equilibrium of the structure under consideration is to put in rigid supports immediately after excavation so that no deformation of surrounding rock mass is allowed. On the other hand, yield supports allow yield continuously as the excavated opening deforms and exert no resistance. Between these two extremes, there exist an infinite number of types of support. Each provides a new state of equilibrium for the support–opening system. The problem of ground control is, then, to determine the optimum support method in terms of safety, economy, and integration with other mining activities.

In designing the best suitable support system for ground control, the basic principles of rock mechanics are frequently used. Each candidate system is structurally analyzed in terms of stress (force) and strain (displacement) distributions, which allows the stability of the system to be determined by using appropriate fracture criteria. However, the structural analysis of underground coal mines differs from those engineering applications that deal with manufactured materials in that designers have no choice in either structural elements (i.e., rocks) or the location of the structures. Mine support structures have to be made where coal is mined; therefore the properties of the structural elements are uncontrollable and have to be determined as they exist. Since coal mines generally extend over a large area, rock properties and structural arrangements within them can vary considerably. The determination and selection of representative rock properties for structural design are very difficult tasks.

After structural analysis of each candidate system has been performed, the most suitable system can be selected by considering factors such as stability of the combined structures, cost effectiveness, and suitability for integration with other mining subsystems.

1.2 CONSTRAINTS ON GROUND CONTROL DESIGNS

To assure the stability of an underground structure its designer must consider the principles of rock mechanics to determine:

1. Overall mine layouts—the relative locations and interaction of entries and pillars, sections, or panels.
2. Shape, size, and number of entries.
3. Shape, size, and number of pillars.
4. Optimum support systems for structural stability or controlled failures (e.g., longwall gob caving and surface subsidence).

However, since the sole purpose of the structural layout of an underground coal mine is to provide access for the extraction of coal and its transportation to the surface in the safest and most economical manner, the application of rock mechanics to ground control has to be considered in the context of mining operations as a whole. Three additional subsystems that always exist in normal mining operations are coal extraction, coal haulage, and ventilation. To integrate ground control with these subsystems, there are several constraints, some of which are so predominant that rock mechanics principles are completely ignored.

1.2.1 Room-and-Pillar Mining

Coal Extraction

1. Due to the length of continuous miners (from 30 to more than 40 ft), mine faces are usually advanced to the maximum allowable distance (approximately 20 ft) beyond the last row of permanent roof supports before any temporary support is erected. Consequently, a period of time

elapses between exposure of the roof and establishment of supports. For some weak roofs, immediate support is necessary to prevent the initiation of delamination or roof falls.

2. The number of temporary roof-to-floor supports that can be established in the face is limited due to the need to allow free passage of continuous miners.

3. Entries driven by ripper-type continuous miners are rectangular, whereas those made by the borer types are oval. Furthermore, most continuous miners are not designed to cut rocks in the roof and floor. Therefore, the maximum height of the entries is in most cases the seam height.

Coal Haulage

1. A sufficient prop-free area in the face is needed for passage of shuttle cars or belt conveyors.

2. All entries traveled by shuttle cars or belt conveyors must be under permanent supports for safety reasons.

Ventilation

1. The minimum number of entries (which defines the spacing of pillars between entries) and/or the minimum width of entry are determined by the amount of fresh air needed to reach the last open crosscut. The minimum amount of fresh air at the last open crosscut and at the working face is set by the 1969 Coal Mine Health and Safety Act. Furthermore, the intake airway must be separated from the return passage.

2. During the development of entries or rooms between pillars, the intake airway to the working face is frequently formed by hanging a brattice along and near the rib. The amount of fresh air that reaches the face diminishes as the distance from the last open crosscut increases. Therefore, to maintain a supply of fresh air to the face the maximum distance that can be advanced without crosscutting is determined by the ventilation requirements. The distance between crosscuts is one of the three dimensions of the pillars.

3. Floor-to-roof temporary supports tend to obstruct ventilation flow and should be kept to a minimum.

1.2.2 Longwall and Shortwall Mining

Coal Extraction

1. The length of the support unit must be such that it provides a sufficient prop-free front for free movement of the cutting machines,

conveying units, and operating crew. For longwall faces, the prop-free front runs between 6.3 and 8 ft, whereas it requires between 8 and 16 ft for a shortwall face.

2. Self-advancing powered supports should be able to move easily over steps and cavities left in the roof and/or floor by the cutting machines.

Coal Haulage

1. The prop-free area between the support units and the face must be sufficient to allow free movement of conveyors or shuttle cars.

2. Support units must be able to advance the conveyors that generally follow the cutting machines.

Ventilation

1. A sufficient open area for air flow must be maintained.

2. Gob shields are needed to prevent fresh air leakage and dilution of fresh air with contaminated air from the gob area.

Fig. 2.2.2 Continuous mining method. Courtesy Joy Manufacturing Co.

2.2.2 Roof Support Practices

Room or Entry Development

Figure 2.2.3 is a typical roof control plan for the development of entries, rooms, and crosscuts using drum-type continuous miners. During the cutting operation, miner runs of varying lengths are made on alternate sides of the centerline. When the face has been advanced to a maximum distance of 18 to 24 ft, which usually permits the operator to remain under the last row of the bolted roof, coal cutting stops and the miner retreats. Temporary supports shown in rows *A*, *B*, and *C* are installed for the subsequent bolting cycle and/or methane inspection. The temporary supports in each row may be removed as soon as roof bolts are installed. An additional row of temporary supports is needed if the distance from row *C* to the face is greater than 5 ft. The last row of roof bolts (considered the permanent support) must be installed within 5 ft of the face before any new cut is made to advance the entries, rooms, or crosscuts.

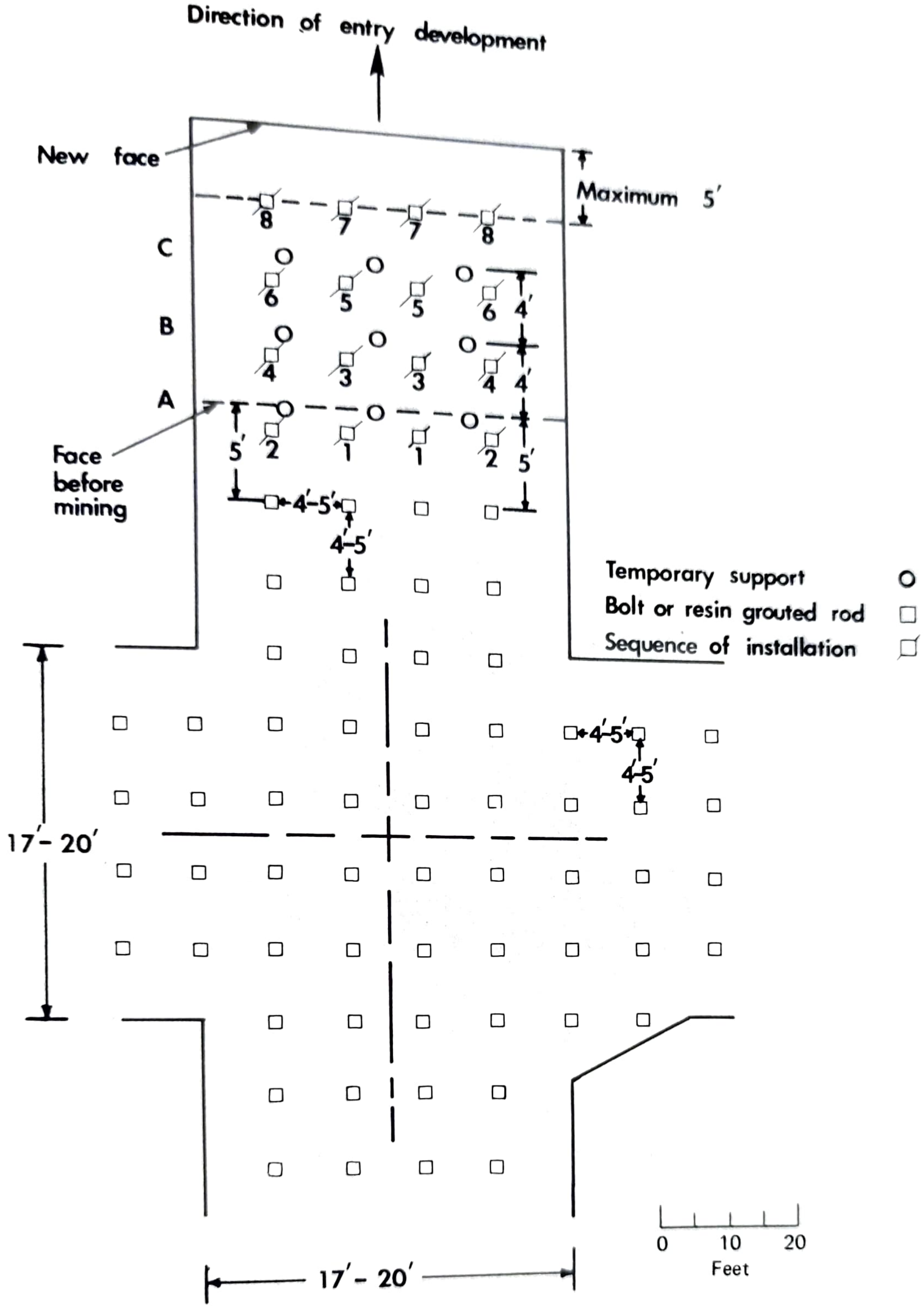


Fig. 2.2.3 Typical roof control plan for entry, room, and crosscut development using drum-type continuous miner.

The normal sequence of roof bolt installation is shown by the numbers in the upper portion of the figure. However, when roof conditions dictate, the sequence may be altered for safety reasons. It may also vary with different types of roof bolting machines. Roof bolts are generally installed at 4-, 5-, or 6-ft intervals. When the final row of bolts is installed within 2 ft of the face, the hydraulic jacks mounted on the roof bolting machines are

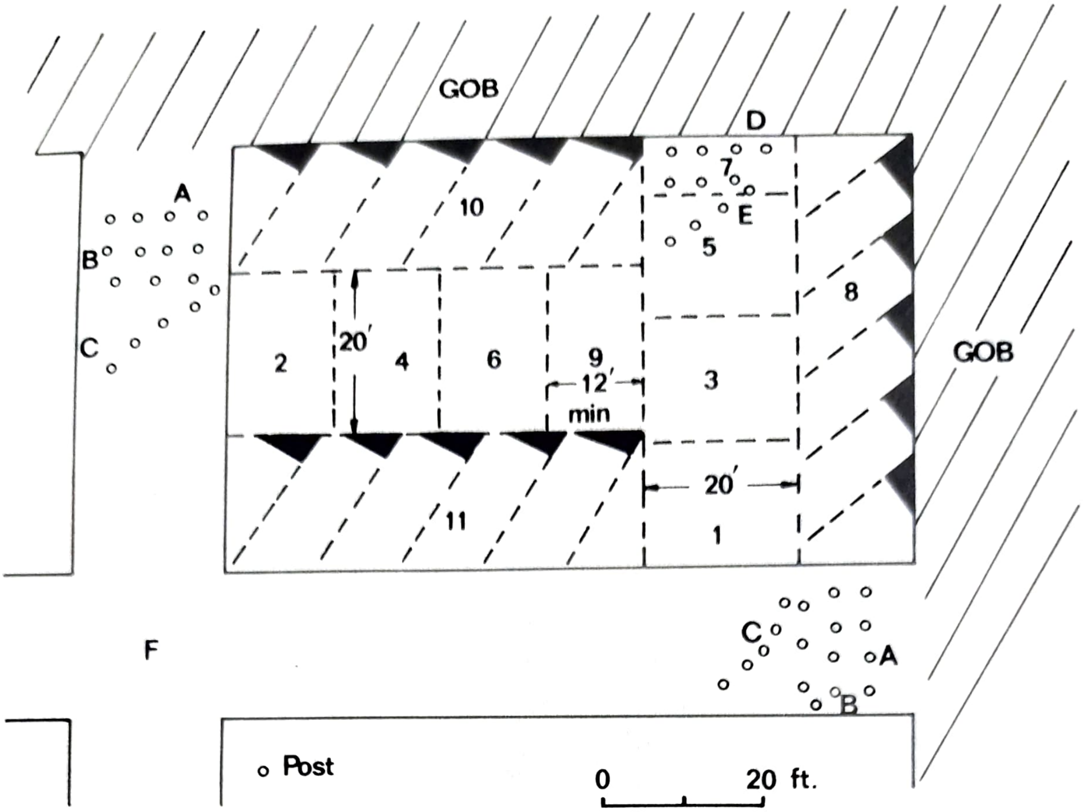


Fig. 2.2.4 Pocket-and-fender method of pillar recovery. Courtesy U.S. Bureau of Mines.

the only temporary support. Other temporary supports may be wooden posts, screw jacks, or hydraulic jacks.

Pillar Recovery

There are two methods of extracting pillars. In the "pocket-and-fender" or "split-and-fender" method, the pillar is first split into several blocks and then slices or lifts are taken from the interior of the pillar. Figure 2.2.4 shows the sequence of pillar recovery steps using the pocket-and-fender method. Numbers in the figure indicate sequences of miner's cuts or lifts. Breaker posts at location *A* are installed promptly after mining is completed inby to prevent roof cave-ins. Two line posts at location *B* are set before mining the second cut in each split. Five turn posts at location *C* are then set immediately after the second cut. Breaker posts at location *D* are set before starting the wing extraction (lift 8). Similar breaker posts must be set along the gobline before wing 10 is mined. If a second cut is required in any open-end lift, a triset must be installed and a double row of posts extended to within 10 ft of the face. Posts as shown in location *E* are set for each wing lift. All posts shown are set on 4- to 5-ft centers. Intersection *F* must be supplemented with spot roof boltings prior to pillar mining.

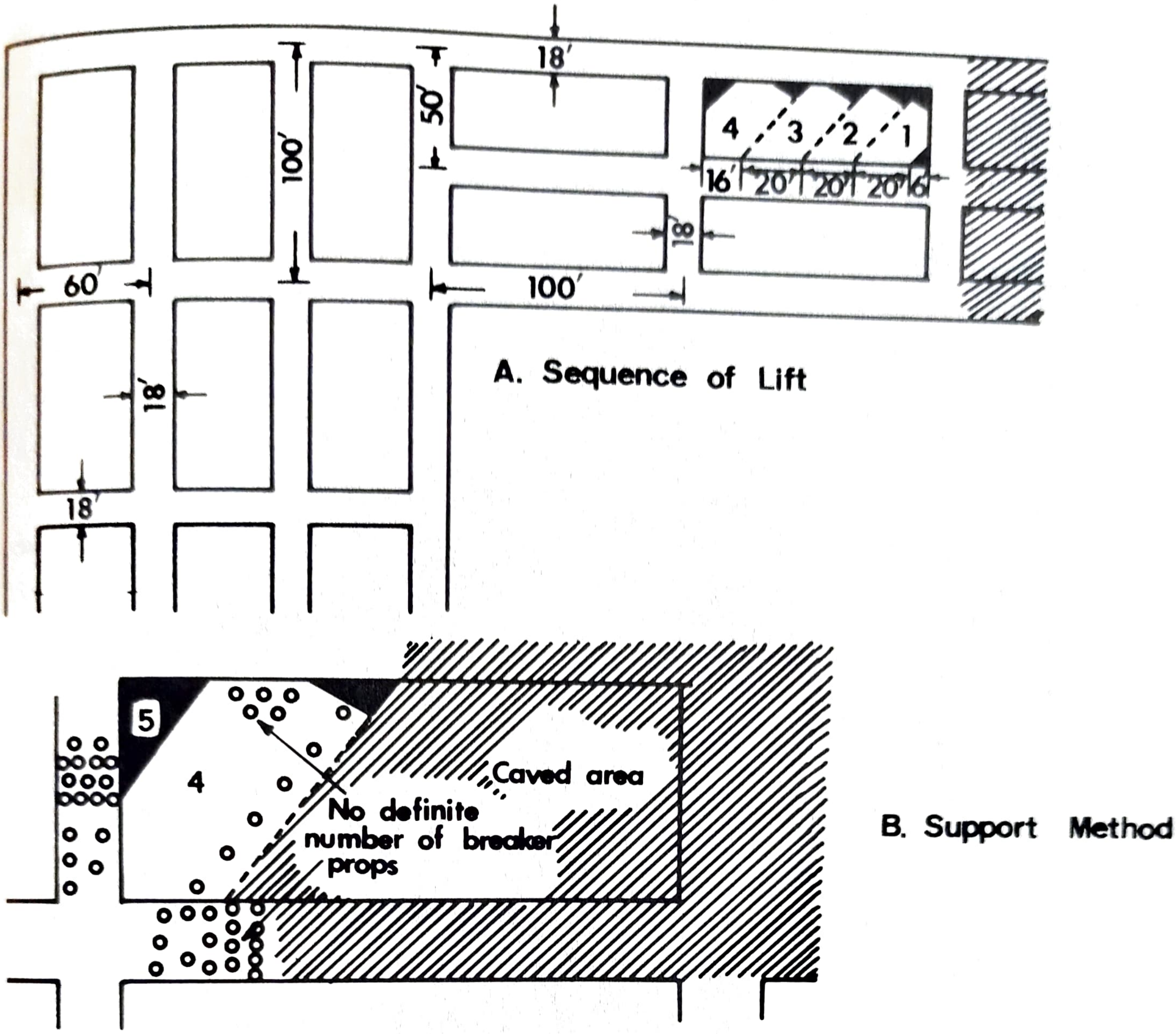


Fig. 2.2.5 Open-end method of pillar recovery. Courtesy U.S. Bureau of Mines.

Another method of pillar recovery is the open-end method, whereby the cuts or lifts of coal are started from the gob side of the pillar and proceed outby the mined-out gob. In some mines no pillars (Fig. 2.2.5) are left between the continuous miner and the gob, but in others thin pillars (Fig. 2.2.6) known as fenders are left to support the roof. The fender also serves to protect machines and crew. The roof control plan is quite similar to that of the pocket-and-fender method in that breaker posts and line posts are set before each lift. The only difference is that in the open-end method a row of posts is set along the gob side to prevent premature roof caving (Fig. 2.2.5). In extracting the final lift or stump of the pillar, additional supports are required as shown in Fig. 2.2.6C, where the entries on both sides of the lifts are reinforced with rows of four posts at 8-ft centers and with headboards and occasional crib sets.

A very important point in pillar mining is that the boundary (or pillar line) between the unmined pillars and the gob must always be kept to a

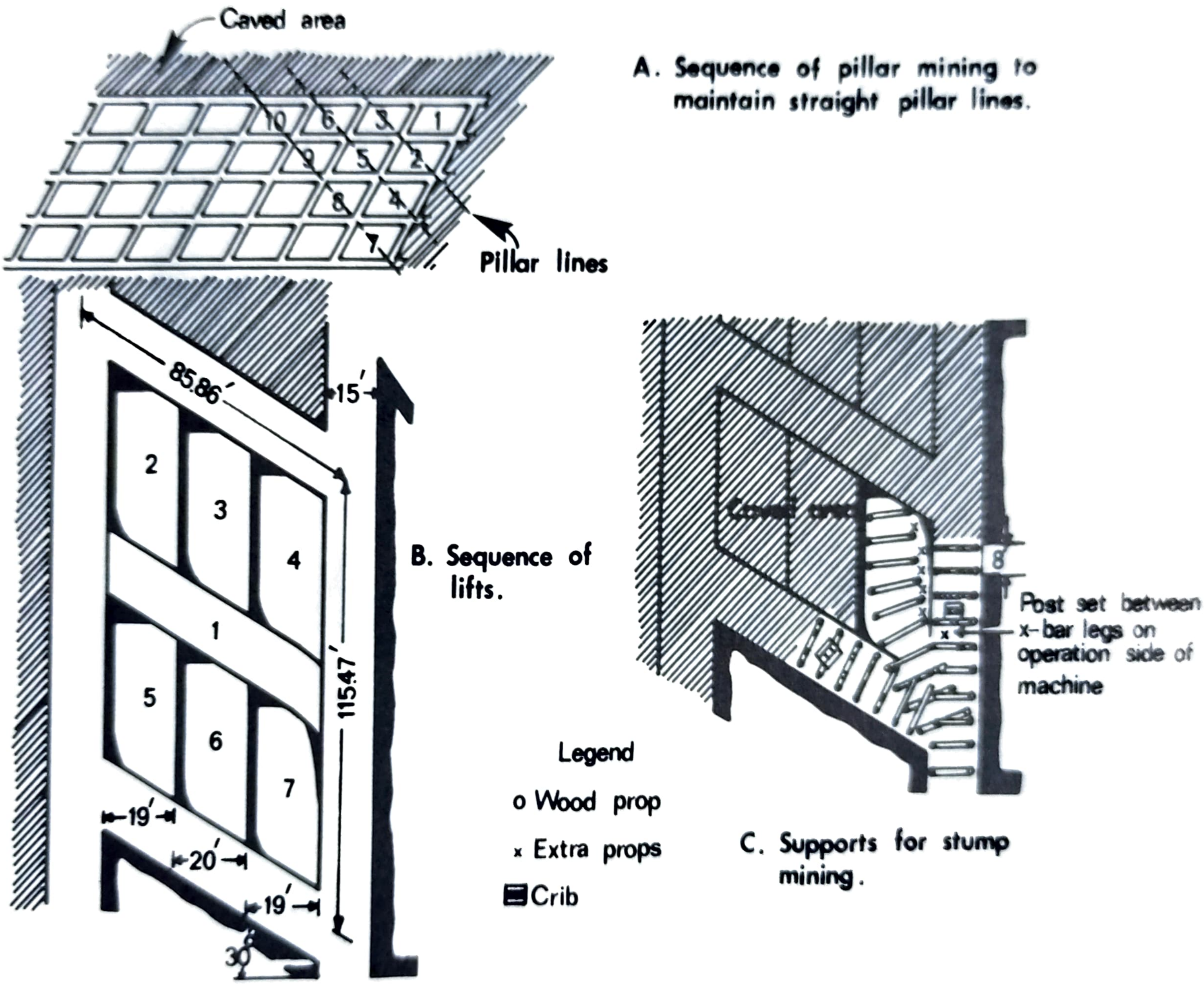


Fig. 2.2.6 Modified open-end method of pillar recovery. Courtesy U.S. Bureau of Mines.

straight line at some angle to the sides of the sections. If any of the unmined pillars along the pillar line protrudes into the gob, it will receive a higher-than-normal pressure from the roof, which may cause it to collapse violently.

chain enclosed in a special guide channel on either the face side or the gob side of the armored face conveyor. The endless chain is powered by two drive units located at both ends of the face. The cutting force is provided by the endless chain that pulls the plow. The widths of cut for the plow range from 2 to 9 in.

Coal cut by the shearer or plow is loaded onto the armored face conveyor and transported to the headentry T-junction. A T-junction is the intersection of the longwall face and the head- or tailentry. At the headentry T-junction, coal is dumped onto a stage loader, which in turn empties to the entry belt conveyor some distance outby the T-junction. A stage loader is a chain conveyor like an armored face conveyor, but it is mobile and capable of moving along with the face. Therefore, the unloading end of a stage loader where it dumps coal into the entry belt conveyor is raised and can be pushed to overlap the entry belt conveyor. Stage loaders are 30 to 150 ft long.

Powered roof supports advance in several steps (Fig. 2.3.2). When the cutting machine cuts and passes several support units beyond the support in question, the hydraulic ram of the support is extended for a distance equal to the width of cut and pushes the conveyor forward (step 1). The support legs are lowered (step 2) and pulled forward by retracting the hydraulic ram (step 3). The hydraulic ram acts against the conveyor panline, whose position is held unchanged by the fully extended hydraulic rams of the supports set on both sides of the support in question. The support is then reset against the roof and ready for the next cut (step 4).

2.3.2 Roof Support Practices

Panel Development

The roof control plans in the panel entries employ the same techniques as in room-and-pillar mining.

Retreat Mining

Face Area. Longwall faces are generally supported by self-advancing roof support units (chock, frame, 2-leg shield, and 4-leg shield or chock shield supports) with capacities ranging from 300 to 1200 tons. These support units are lined side by side along the face with center-to-center distances between 4 and 5 ft.

There is an unsupported roof gap of approximately 1 ft between the tip of the canopy and the faceline. This gap is required because the shearer may cut into the canopy if no gap is left. The prop-free front before the

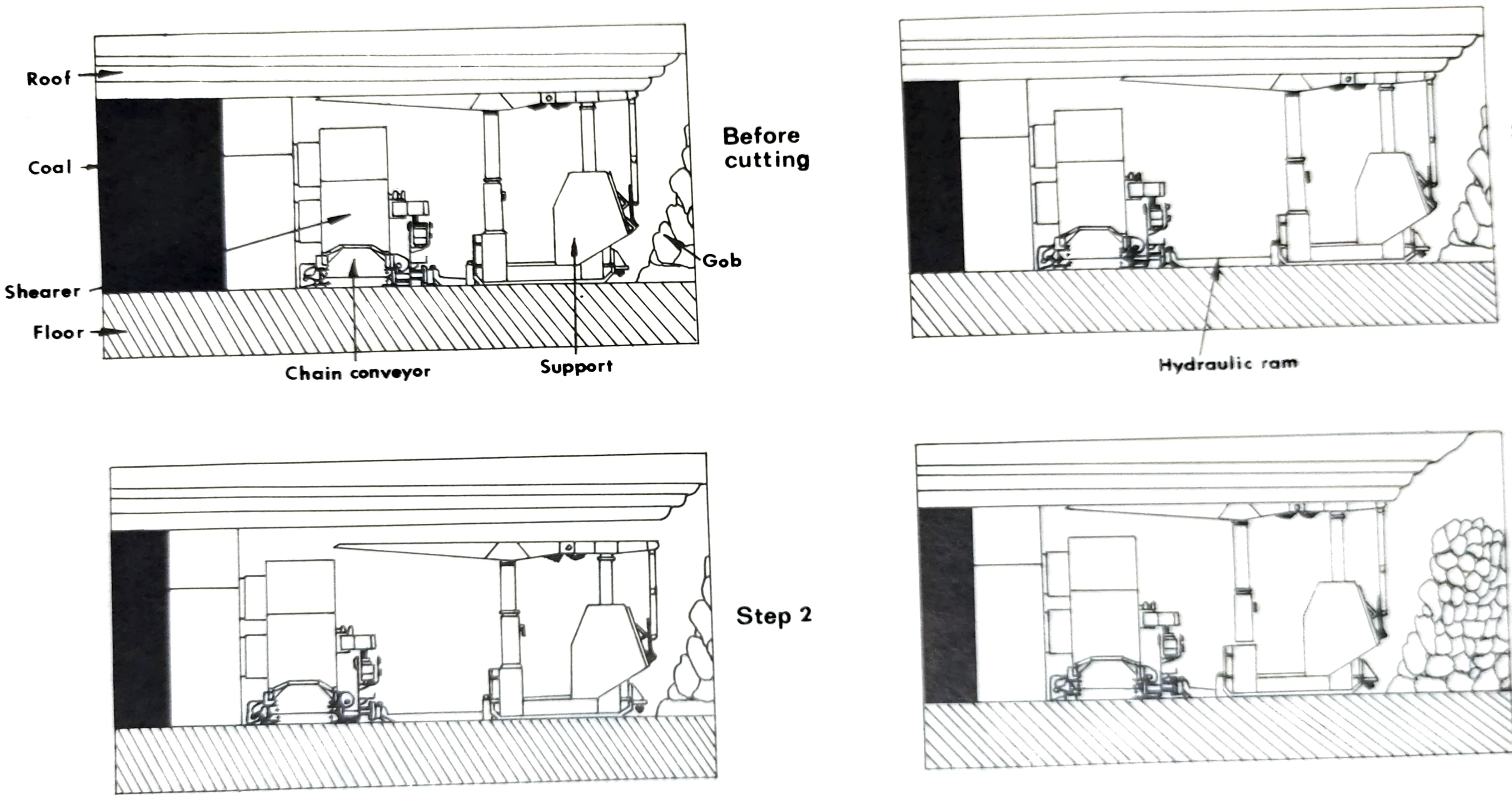


Fig. 2.3.2 Advance sequence for longwall supports.

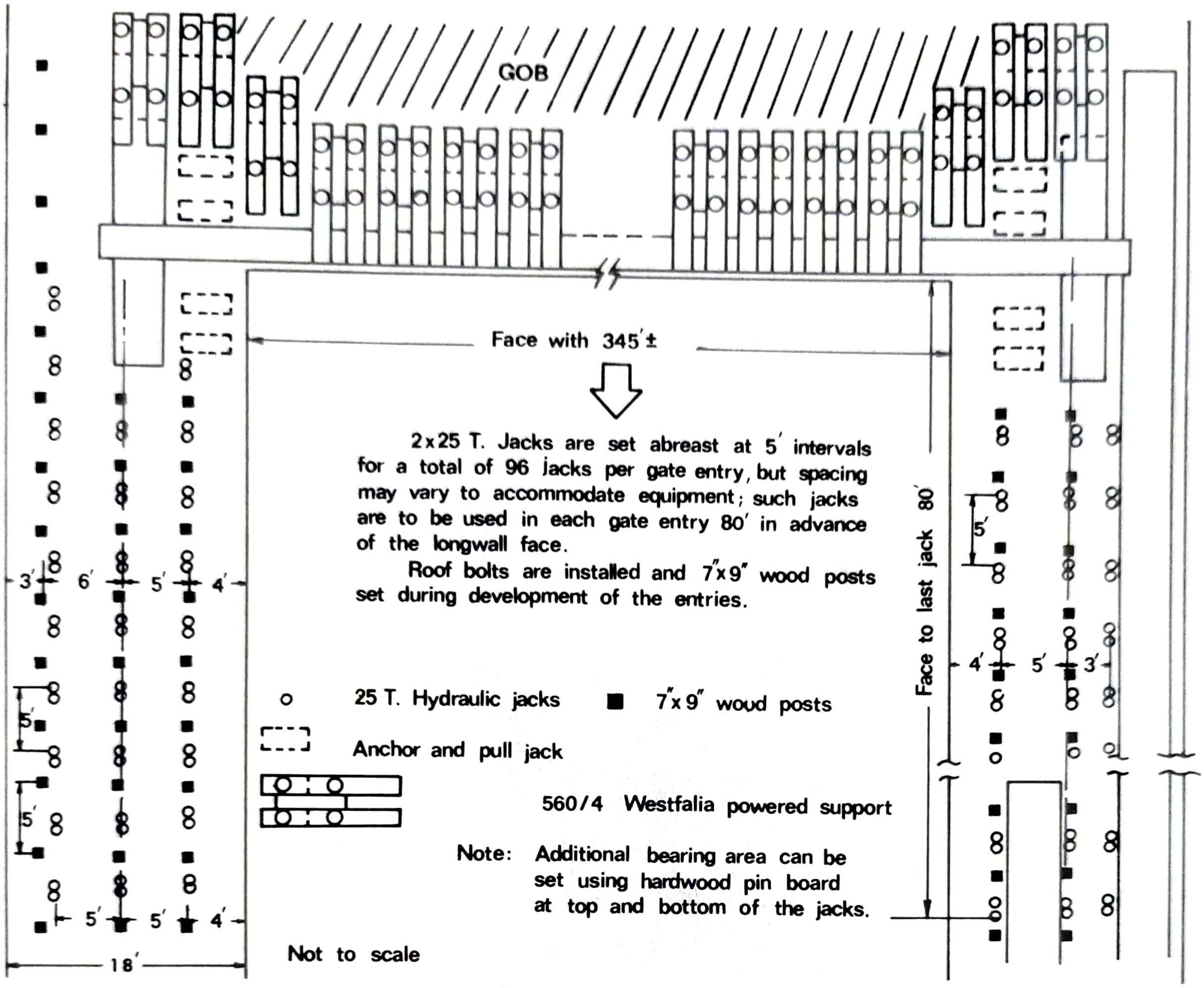


Fig. 2.3.3 A T-junction reinforcement plan employed for mining the Pocahontas No. 4 seam.

cutting machine makes the cut is approximately 6 ft 3 in. It increases by an amount equal to the width of cut immediately after the cutting machine passes but before the support is advanced.

Headentry. The headentry at the T-junction and some distance ahead of the longwall face (from 0 to 500 ft outby) is generally reinforced with supports of some type to increase support density to cope with the moving front and side abutment pressures (Fig. 2.3.3). Supports may be wooden posts, hydraulic jacks, spot roof bolting with headboards, or three-set pieces using steel channel beams. When the face approaches a crosscut intersecting the head- or tailentry, the inby end of the crosscut is generally reinforced by two or three rows of wooden posts spaced 4 to 6 ft apart center-to-center.

Tailentry. In the retreating mining method the tailentry generally experiences a heavier pressure than the headentry because it is usually

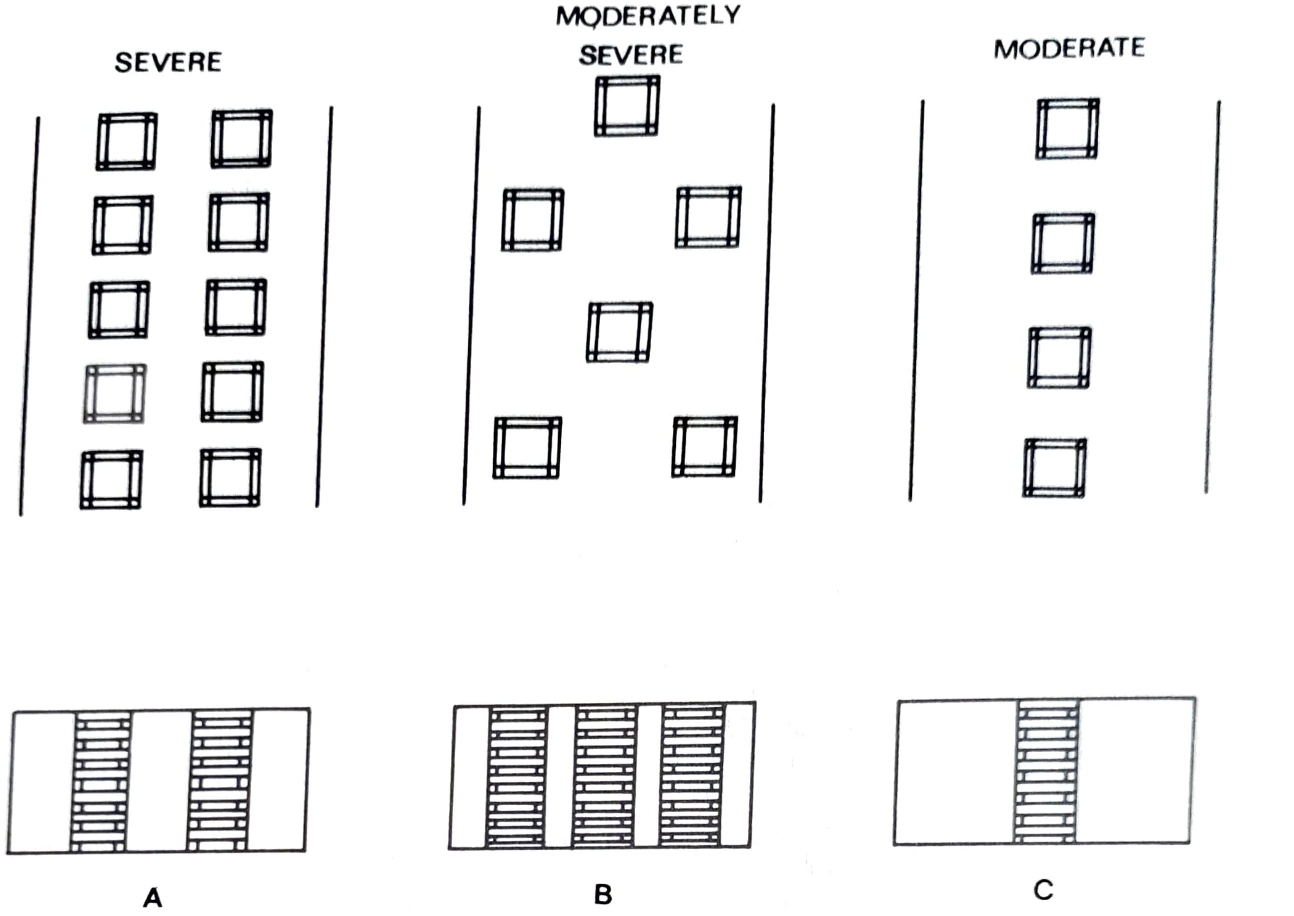


Fig. 2.3.4 Crib patterns used for tailentry reinforcement.

located on the side near the mined-out panels. Since the tailentry is mainly for return air passage, a complete clearance is unnecessary. For this reason, cribbing is almost exclusively used for tailentry reinforcement support. Several forms of cribbing are employed for different types of ground pressures as shown in Fig. 2.3.4. The distance between cribs varies, ranging from 6 to 10 ft center-to-center.

2.4 SHORTWALL MINING

2.4.1 Typical Panel Layout

The layouts of a shortwall panel resemble that of the longwall except the face length is shorter, from 100 to 200 ft. The trend is toward a longer face (2,3). The cut is much wider, ranging from 6 to 11 ft. The length of the panel varies from 400 to 3000 ft, the most popular being 2400 to 3000 ft. The retreating method is exclusively employed (Fig. 2.4.1).

The shortwall mining method is a hybrid between continuous and

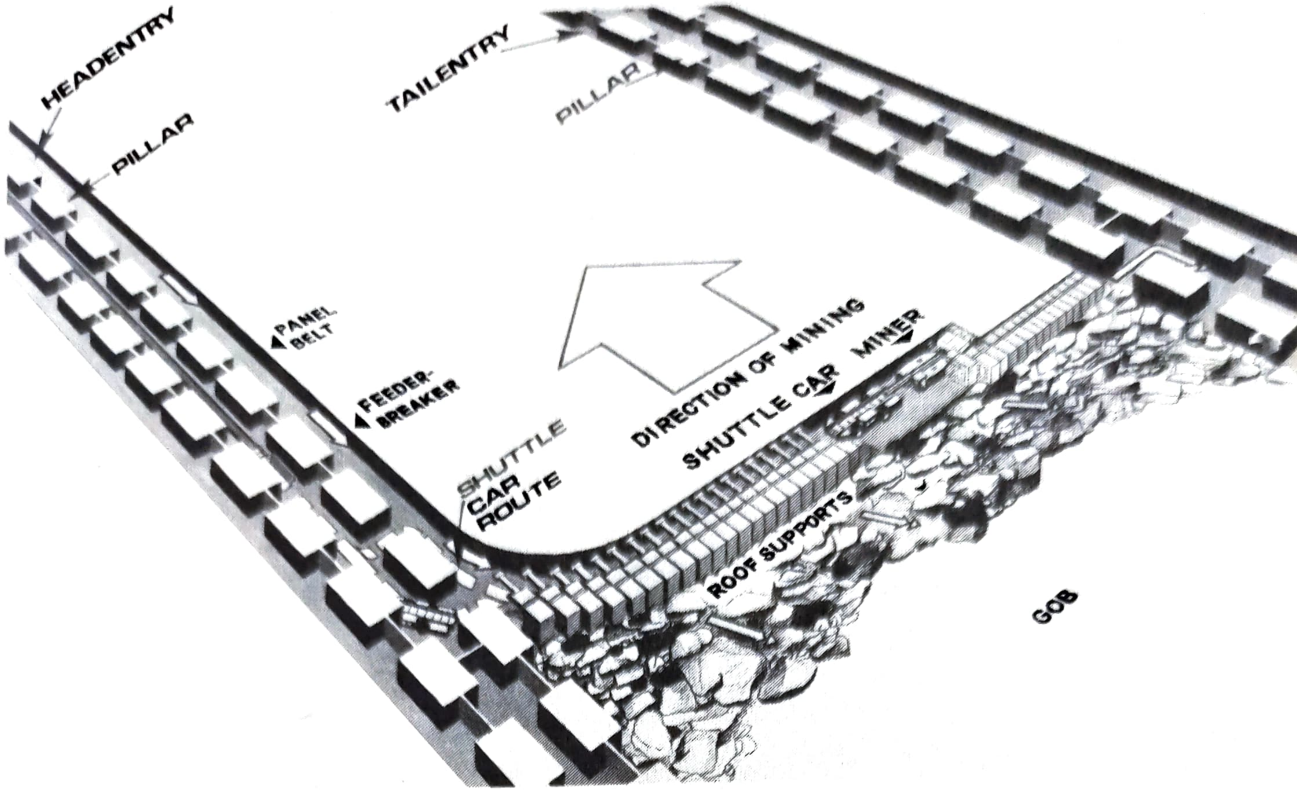


Fig. 2.4.1 Shortwall mining method. Courtesy Joy Manufacturing Co.

longwall mining; that is, coal is extracted by continuous miners and transported by shuttle cars, but the face is supported by the self-advancing powered supports traditionally used in longwall faces. Physically, it resembles the open-end method of pillar recovery in that a short face is supported by self-advancing powered supports. The shortwall powered supports are similar to those of the longwall except that an extensible canopy is added to the front end of the roof canopy, which, when it is fully extended, provides a prop-free front area wide enough for a continuous miner to operate.

A continuous miner cannot make a 90° turn at the beginning of each cut because it is too long to make such a turn at the headentry. Instead it negotiates a turn of approximately 18 to 20°, leaving a roof span considerably longer than the width of the entry. To prevent roof falls at this area, the powered supports are set and guided by an inclined dozer beam (Fig. 2.4.2) so that the roof span will not become excessive.

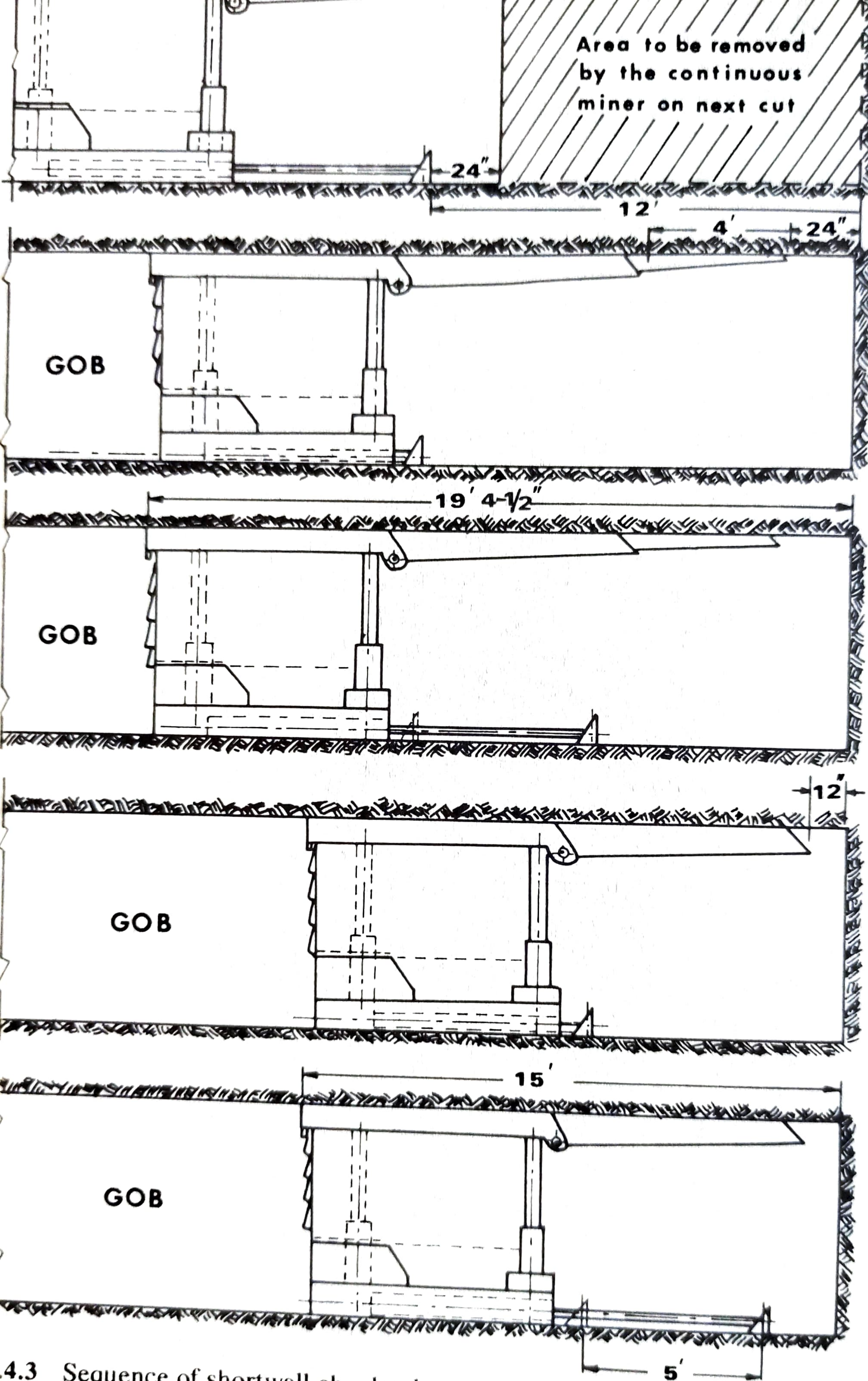
Coal is loaded to a shuttle car waiting behind the miner. The shuttle car runs back and forth between the continuous miner in the face and the panel belt conveyor in the headentry, where the loading point is generally from 200 to 400 ft ahead of the T-junction. Frequently two shuttle cars with different traveling routes (Fig. 2.4.1) are employed to reduce the waiting time of the continuous miner. Under this arrangement the panel belt conveyor is usually installed in the second entry from the panel so that both shuttle cars have smoother and separate travel routes for changeout. A coal loader is also frequently used between the miner and the shuttle car to clean out any spillage of coal on the floor. A clean floor is essential for machine movement and support advance. The armored face conveyor has been introduced in 1976 and set up along the face in front of the powered supports in much the same way as in longwall mining. Coal is loaded directly from the miner to the conveyor, which then loads to the monorail at the headentry T-junction. A monorail is a belt conveyor traveling on a monorail hung from the roof. At the end of the monorail, which may be up to 275 ft long, coal is dumped to the panel belt conveyor. Therefore, a monorail acts like a stage loader in longwall faces.

The sequence in which supports are advanced is different from that in longwall mining (Fig. 2.4.3). It consists of five distinctive steps although three are transitional. Step A is one of the two key steps. Before a new web cut is removed, there is a 12-in. unsupported roof gap between the tip of the canopy and the faceline. On the floor the hydraulic ram is fully extended, leaving a gap of 24 in. between the dozer beam (or spill plate) and the faceline. The prop-free front between the front legs of the chock and the faceline is approximately 8 to 9 ft. When the continuous miner cuts a 10-ft web and passes the chock, the chock is lowered and advanced 5 ft, while simultaneously the extensible canopy (or forward bar) is extended to its full length of 4 ft. The hydraulic legs are then raised and set the chock (step B). The unsupported roof gap is thus increased to 24 in., but the prop-free front increases to 12 to 14 ft. When the miner completes the cut and retreats back to the headentry, the dozer beam is hydraulically pushed forward 5 ft (step C). The chock is lowered and advanced forward (step D). And finally in step E the chock is set against the roof and the dozer beam is again pushed forward. It is now ready for a new cut. Therefore steps C to E are transitional steps. Step A is the initial chock position before a new cut is made. Step B is the chock position from the time immediately after a continuous miner cuts a new web and passes it to the time when the continuous miner finishes the cut along the width of the panel and retreats back to the headentry.

2.4.2 Roof Support Practices

Panel Development

The roof support plans in all panel entries in shortwall mining are similar to those described for room-and-pillar or longwall mining.



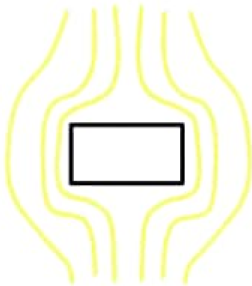
.4.3 Sequence of shortwall chock advance. (A) Area to the right shows the continu-
ner removing a 10-ft wide section. (B) Chock is lowered and advances forward 5 ft.
anopy is then set to roof and forward bar extended 4 ft to cover exposed roof. (C) On
etion of cutting pass, the miner fully retreats and the spill plate is hydraulically pushed
d 5 ft. Courtesy Joy Manufacturing Co.

Retreat Mining

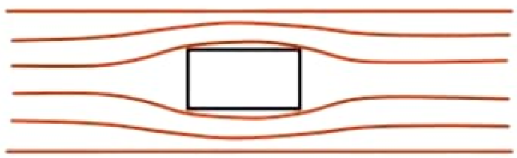
Face Area. Self-advancing shortwall chocks or frames are used solely to support the face. Shortwall supports are similar to longwall supports except that an extensible canopy of approximately 4 to 5 ft is added to the front canopy of the longwall supports. This canopy can be extended (or retracted) to cover one half of the newly exposed roof created by the cutting width of a continuous miner. The other half of the new roof is covered by the immediate forward advancement of the supports. Therefore, each miner's cut requires at least two laps of support advances (Fig. 2.4.3).

Headentry. Because of its considerable length, when a continuous miner starts off a new cut at the headentry, it cannot turn off at a 90° angle. Instead it has to cut along a curvature (Fig. 2.4.2) that results in a T-junction with a very large roof span. Therefore, the dozer beam is set at 30° angles to the faceline so that supports in this area maintain a 30° angle and reduce the unsupported roof area. Frequently the roof in this area is further reinforced by placing wooden headboards spanning between the tip of the extensible canopy of the chocks and hydraulic jacks set along the coal rib. In addition to roof bolting established during entry development, additional supports such as hydraulic jacks or spot bolting are usually erected at the T-junction, often extending for some distance outby.

Tailentry. Generally, cribs are added at the T-junction to keep the tailentry open. Occasionally, hydraulic jacks are set for some distance ahead of the T-junction. In other cases, a smaller pillar is left at the end of each alternate cut to protect the tailentry.



VERTICAL STRESS CONCENTRATED IN RIBS



HORIZONTAL STRESS CONCENTRATED IN ROOF & FLOOR

Characteristics of Coal Measure Roof Strata

Cover - The overburden of any deposit.

Overburden – Layers of soil and rock covering a coal seam. Overburden is removed prior to surface mining and replaced after the coal is taken from the seam.

Lithology - The character of a rock described in terms of its structure, color, mineral composition, grain size, and arrangement of its component parts; all those visible features that in the aggregate impart individuality of the rock. Lithology is the basis of correlation in coal mines and commonly is reliable over a distance of a few miles.

Bed - A stratum of coal or other sedimentary deposit.

Roof
The stratum of rock or other material above a coal seam; the overhead surface of a coal working place. Same as "back" or "top."

Characteristics of Coal Measure Roof Strata

Immediate Roof

The roof strata that is immediately above the coal seam. This is the strata requires support for the mine openings to remain competent.

Primary roof - The main roof above the immediate top. Its thickness may vary from a few to several thousand feet.

Secondary roof - The roof strata immediately above the coalbed, requiring support during the excavating of coal.

Competent rock - Rock which, because of its physical and geological characteristics, is capable of sustaining openings without any structural support except pillars and walls left during mining (stalls, light props, and roof bolts are not considered structural support).

Characteristics of Coal Measure Roof Strata

Fissure - **An extensive crack, break, or fracture in the rocks.**

Fracture - **A general term to include any kind of discontinuity in a body of rock if produced by mechanical failure, whether by shear stress or tensile stress. Fractures include faults, shears, joints, and planes of fracture cleavage.**

Joint

A discontinuity in the rock strata where there is no sign of relative movement.

A divisional plane or surface that divides a rock and along which there has been no visible movement parallel to the plane or surface.

Cleat

The vertical and Parallel cleavage planes or partings crossing the bedding. The main set of joints along which the coal breaks more easily than in any other direction.

Face cleat - **The principal cleavage plane or joint at right angles to the stratification of the coal seam.**

Characteristics of Coal Measure Roof Strata

Butt cleat - A short, poorly defined vertical cleavage plane in a coal seam, usually at right angles to the long face cleat.

Slickenside - A smooth, striated, polished surface produced on rock by friction.

Slip - A fault. A smooth joint or crack where the strata have moved oneach other.

Fault - A slip-surface between two portions of the earth's surface that have moved relative to each other. A fault is a failure surface and is evidence of severe earth stresses.

Fault zone - A fault, instead of being a single clean fracture, may be a zone hundreds or thousands of feet wide. The fault zone consists of numerous interlacing small faults or a confused zone of gouge, breccia, or mylonite.

Fig: Joints exposed in the sandstone roof

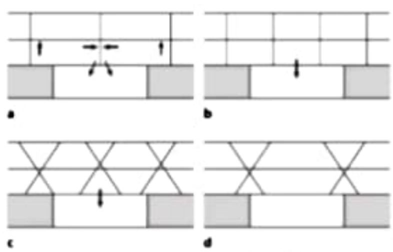


Fig.: Influence of Joints

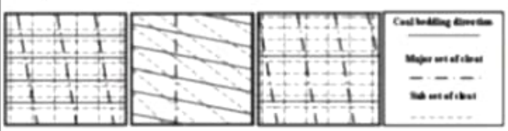


Fig: orientation of Cleats and coal seams

Fig: Face and Butt Cleats in the Coal Pillar