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Record Requested: **All reports on the operation of Cayuga Salt Mine from year 2000 to the present including but not limited to those submitted to DEC by Cargill and including all CDs, all maps as AutoCad or Adobe Acrobat files, all extensometer and closure readings as Excel or other digital files, all consultant reports including but not limited to those reports from ESC, RESPEC, Rock Mechanics Assist and RockTec Solutions, and any other materials submitted as part of the annual report submission process including materials only sent to John T. Boyd company and not copied to a DEC office.**

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Create Date:	6/23/2017 4:39 PM
Update Date:	7/12/2017 1:31 PM
Completed/Closed:	No
Required Completion Date:	7/24/2017
Status:	Processing
Priority:	Medium
Assigned Dept:	Regional Admin
Assigned Staff:	Region 7 FOIL Coordinator
Customer Name:	Dr. John Dennis
Email Address:	johnvdennis@gmail.com
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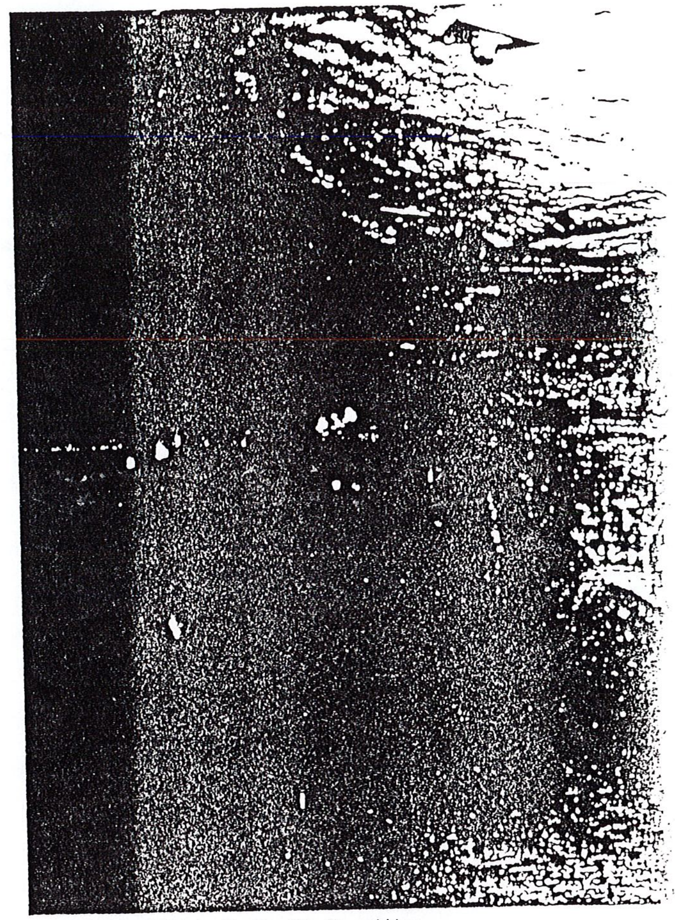
Larry,
This is the article from which I spoke of. This has been used successfully since 1979. See Fig 6. for sketch. Jerry

YIELDING PILLAR AND PRESSURE ARCHES

PLAY A KEY ROLE IN BOOSTING MINE PRODUCTIVITY AT THE CARGILL SALT DIV.



In the NE panel, this is pillar No. 26 after robbing.

Gary B. Petersen, mine manager, and **David B. Plumeau**, mine engineer, Cayuga mine, Cargill Salt Div.

Jack Parker, Jack Parker and Associates

Small pillars are being used at the Cayuga underground rock salt mine in New York state to yield and shed load onto massive abutment pillars, thus reducing roof stresses in the mining area. The new concept was developed because the standard room-and-pillar layout in the mine at depths greater than 2,500 ft resulted in unstable roof conditions. Instability occurred even though the operators left pillars measuring 8 ft high and 88 ft square, with 32-ft-wide entries—which provided an extraction ratio of only 46%.

To remedy the situation, two test areas were mined by using yielding pillars, with the second test achieving 90% extraction from a 600-ft-long x 200-ft-wide panel. Pillar stubs were as small as 12 x 28 ft, and entries were as wide as 46 ft. Nominal pillar load was 32,000 psi, giving a "safety factor" of only 0.16, but as yet there are no signs of roof or pillar failure. The test panel proved very productive, largely because of short tramming distances for LHDs and the low cost of pillar robbing. Simple instrumentation indicated that the pillars did in fact yield, that the load was shed onto abutments, and that an arch must have formed.

Further tests are under way to adapt the principle of the yielding-pillar/pressure arch to the various requirements of both development and production panels in the mine.

MINING METHODS AND EQUIPMENT

The mining horizon at Cayuga is one of several rock salt beds interspersed with beds of shale and dolomite that are part of the Late Silurian Syracuse formation (Fig. 1). Although some of the upper beds are severely distorted, the No. 6 salt bed is fairly uniform in thickness, dipping gently to the south at about 100 ft per mi. Thickness of cover varies from 2,300 ft at the shaft, near Lake Cayuga, to more than 3,000 ft under the hills east of the lake.

Conventional mining equipment is used in a room-and-pillar layout. Roof bolts 7 and 8 ft long are installed on 4- to 5-ft centers. The salt face is then undercut to a depth of 14 ft with a Joy 15 RU undercutter. A Fletcher face jumbo drills 24 1¾-in. holes in 32-ft-wide x 8-ft-high entries. The holes are loaded with either Trojamite "C" or Tovex 90 and blasted. Broken salt is hauled to a Stamler feeder-breaker by Wagner ST 5A LHD vehicles, conveyed to a preparation plant for crushing and screening, and hoisted to the surface. Daily salt production is about 3,300 st.

In the original mine design, mining of the No. 4 salt bed followed the thick rolls of that contorted bed. Thus, both pillars and stopes varied greatly in width (Fig. 2).

The layout in the No. 6 salt bed was more systematic. Entries 32 ft wide and pillars 88 ft square yielded a 46% extraction rate, which at a 2,400-ft depth produced an average pillar load of about 4,890 psi (2,400 x 1.1 ÷ 0.54). This load registered a nominal pillar safety factor of about 1.0, since the compressive strength of rock salt is generally assumed to be about 5,000 psi.

To achieve a more acceptable theoretical pillar safety factor of 2.0, it would have been necessary to aim for a pillar load of 2,500 psi. This load would have produced *zero* extraction. However, even with a safety factor as low as 1.0, no pillar failures were experienced, so the theoretical factor of 2.0 was inapplicable.

In the early years of mining, there were few ground control

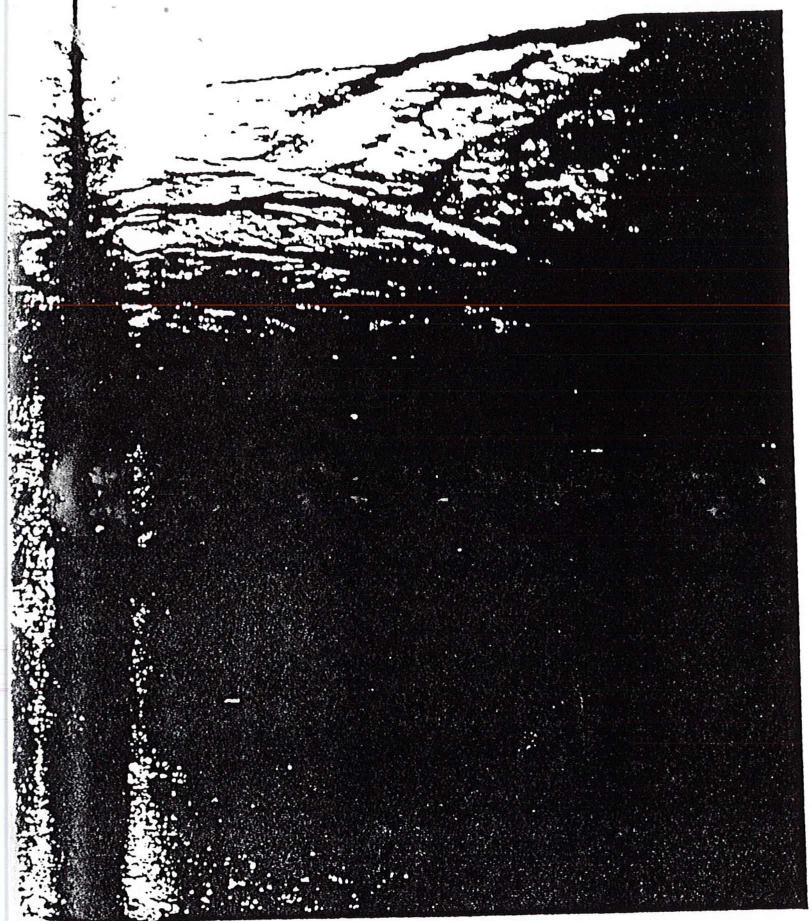


Fig. 1—Geological cross sections

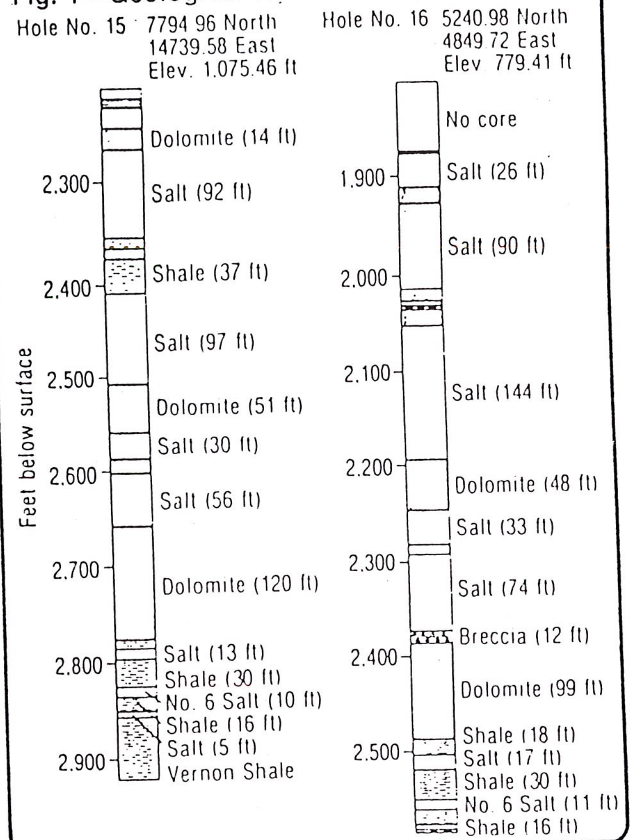


Fig. 2—Part of No. 4 level

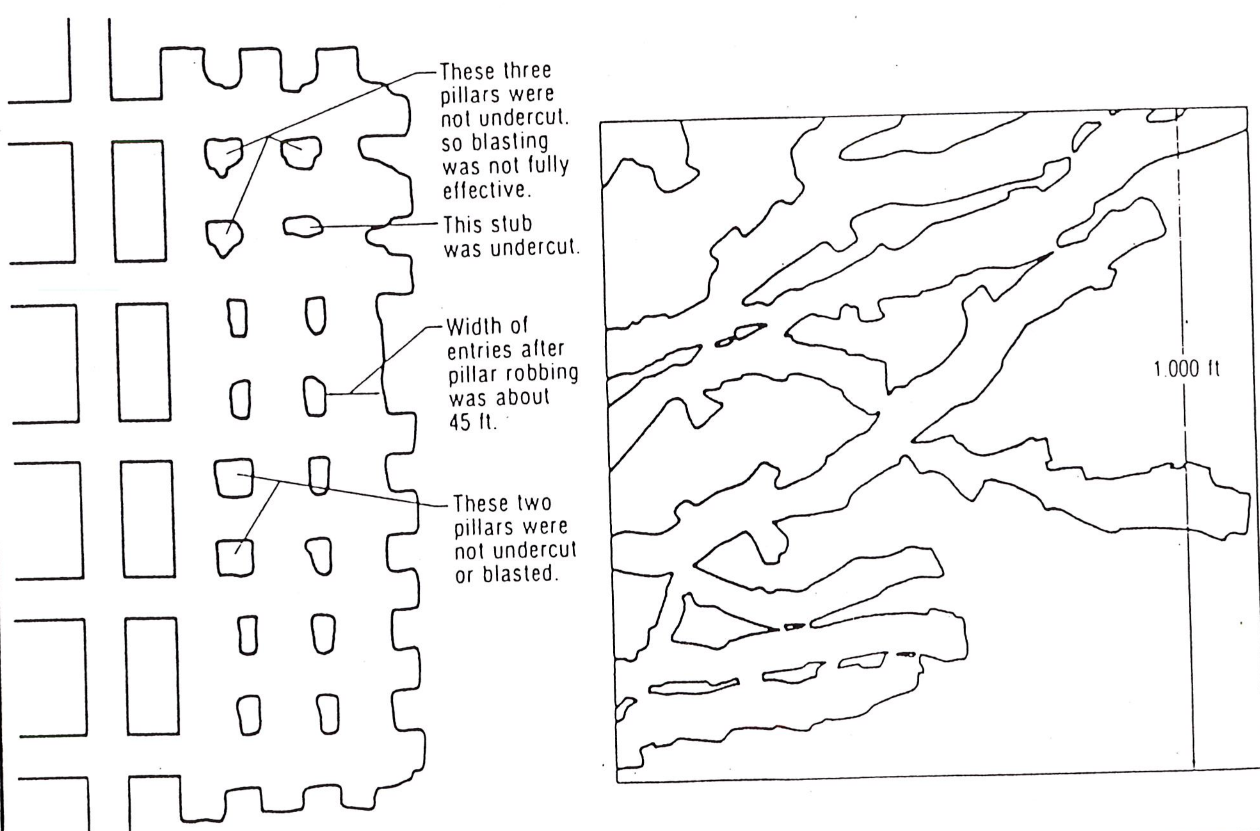
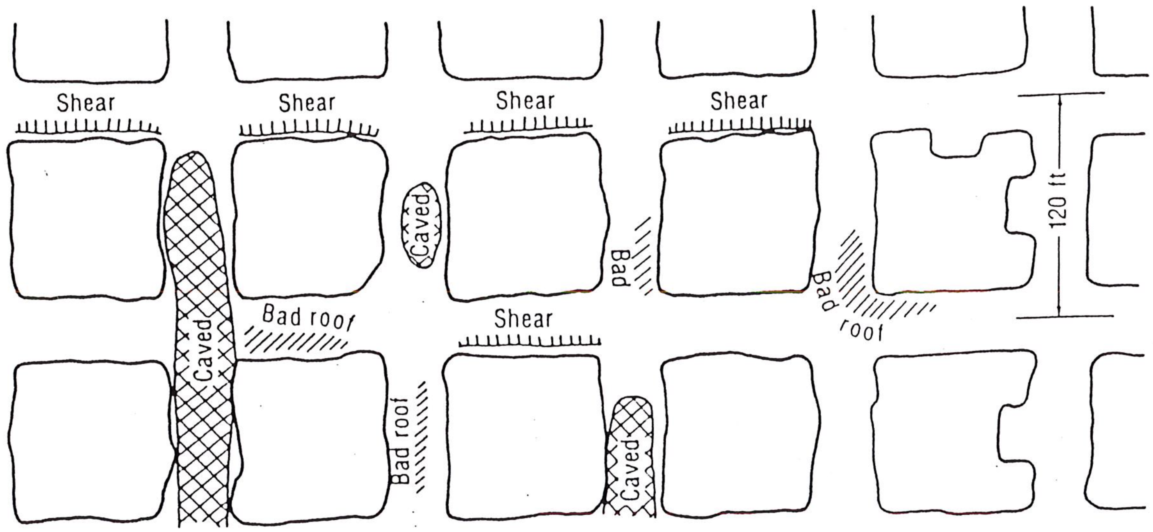


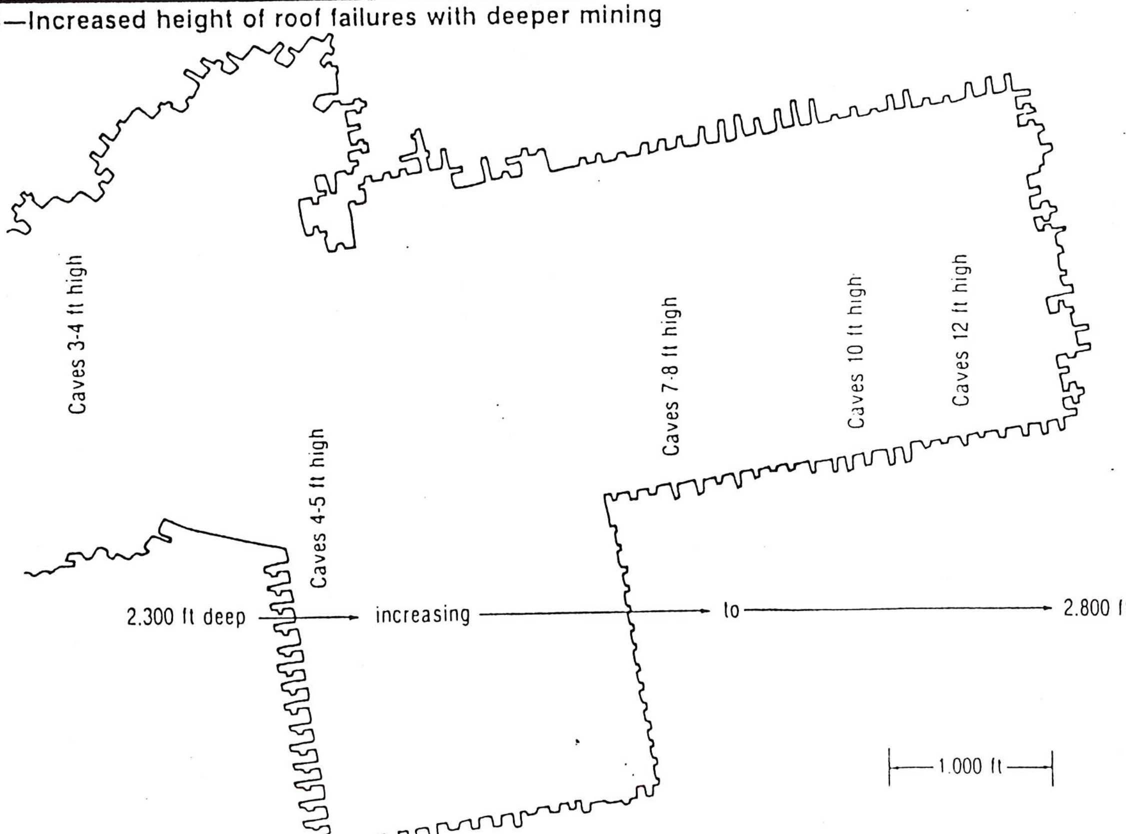
Fig. 3—Map of typical bad ground



problems, perhaps because mining was conducted beneath a protective "umbrella" of workings in the No. 4 salt bed. In recent years, ground conditions worsened, and in May 1975 a month's production was lost when the entire mining front was closed by the Mine Safety and Health Administration (MSHA). A new mining front was established perpendicular to the first, but one year later this front also was threatened by numerous roof failures. Mine design appeared to be inadequate, since only a year or two of stability was being achieved.

The mode of failure was such that, although the floor did not heave and the pillars did not fail, the load was apparently too great for the roof rocks. In most places, 1 or 2 ft of salt is left in the roof to avoid contamination of salt by broken shale. Vertical loading results in lateral motion of the salt in the roof. If there is a free face, due perhaps to overbreak, the salt moves toward it; if not, a low-angle shear develops along one rib and the roof salt is thrust sideways and downward. In an entry 32 ft wide there can be as much as 1 ft of lateral motion—sufficient to shear ¾-in. bolts, both mechanical and

Fig. 5—Increased height of roof failures with deeper mining



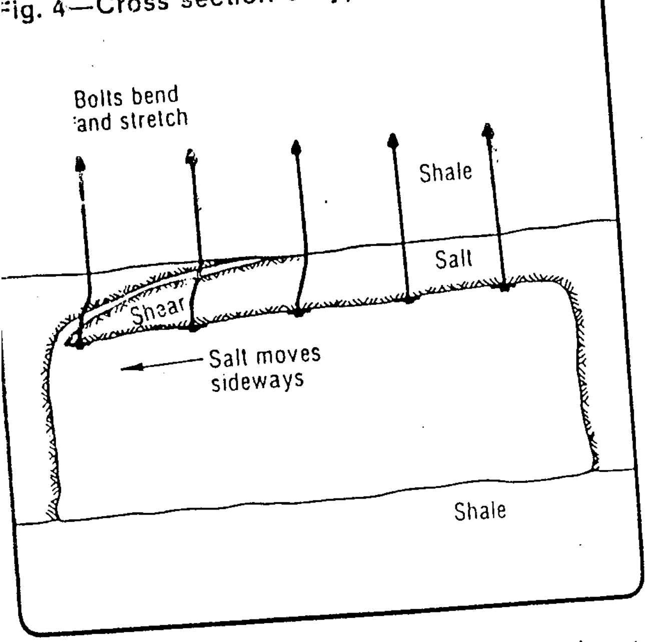


Fig. 4—Cross section of typical roof failure

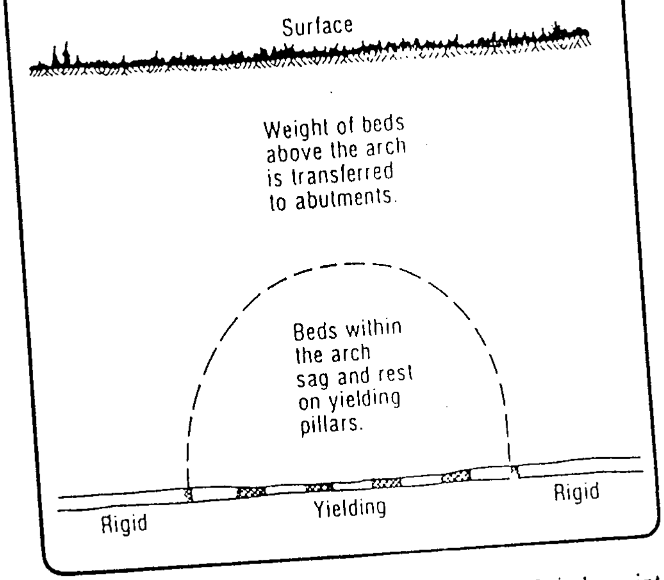


Fig. 6—Principle of yielding pillars and pressure arch

resin-grouted. (See Figs. 3, 4 and 5.) Roof failures do not favor any particular orientation, but they usually begin between pillars and not in the intersections.

SEEKING A CURE

Since salt mining at Cayuga was already being conducted at a deep level, with an extraction ratio and pillar dimensions unfavorable for efficient mining, only one solution to the roof failures appeared feasible: Pillars had to be designed small enough to yield under load, instead of being stiff to the point of overloading the roof rocks and causing roof failures. At the same time, abutment pillars would have to be made available, strong enough to support the load shed by the yielding pillars. Presumably, there would be some critical width within which the overlying strata would bridge or arch onto the abutment pillars, but beyond which the bridge or arch would collapse. (See Fig. 6.) Management suspected that a layer of dolomitic shale above the mine would help by providing a "beam" about 100 ft thick. (See Fig. 1.) A

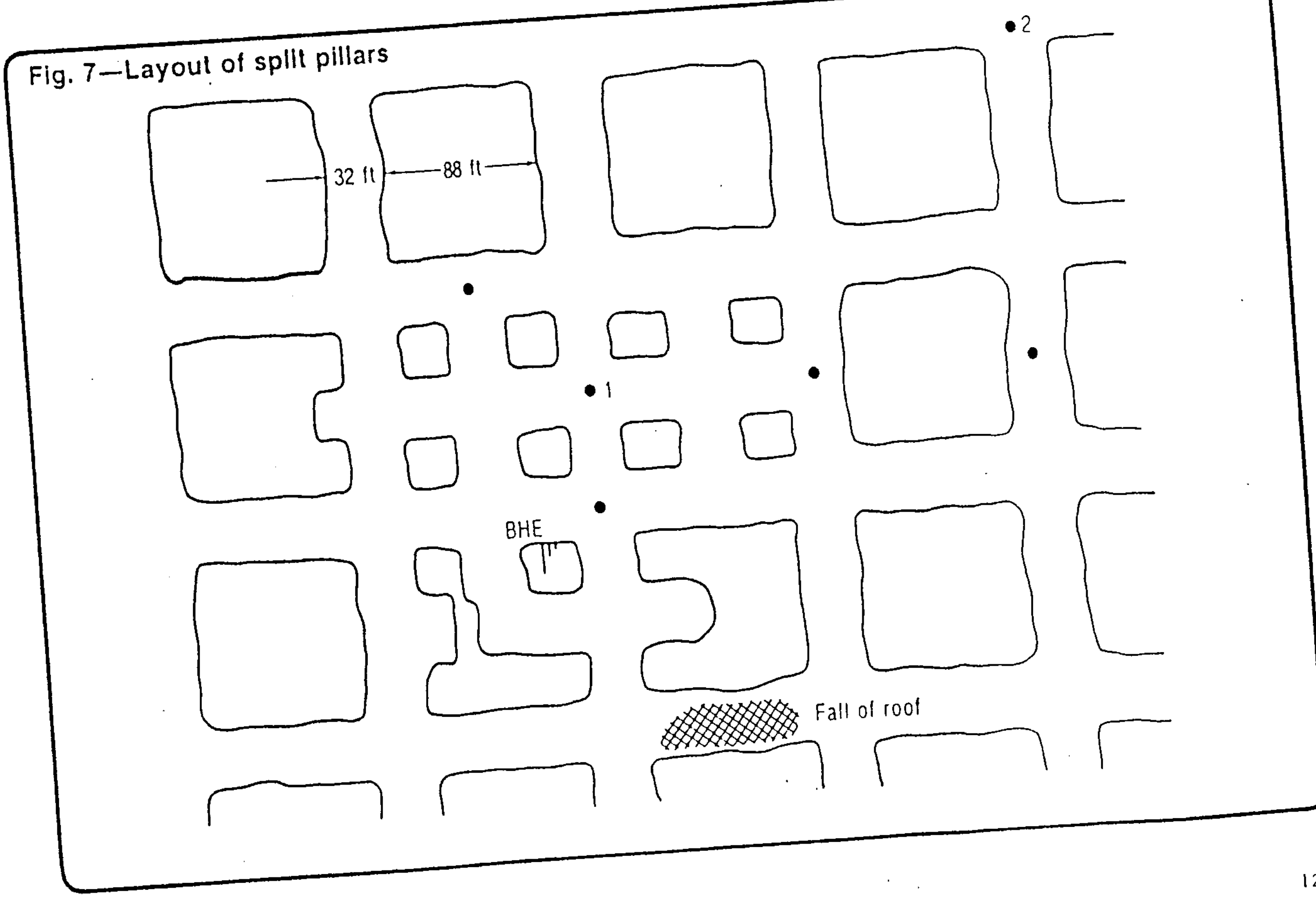


Fig. 7—Layout of split pillars

Fig. 8—Effect of mining on convergence rate at Point 1

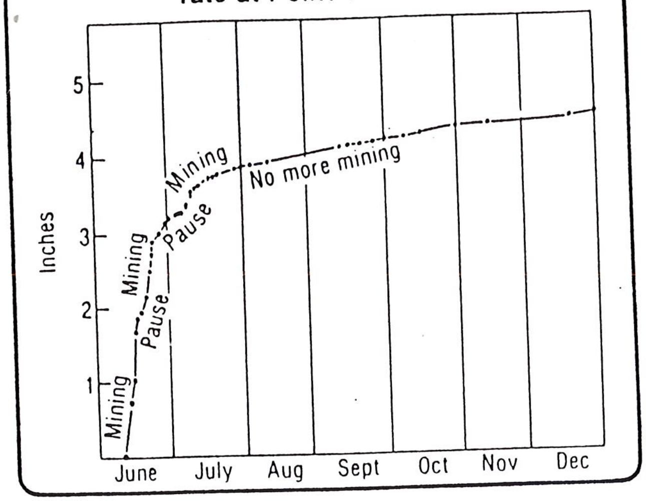
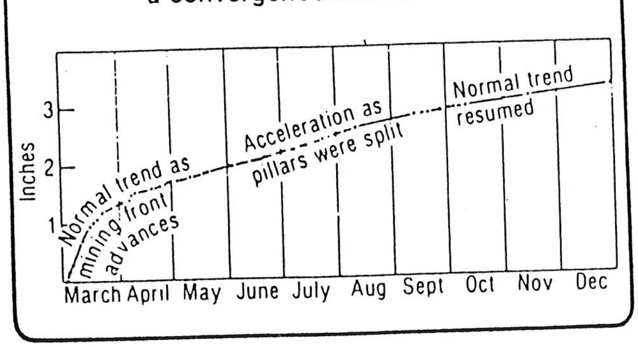


Fig. 9—Effect of splitting pillars on a convergence station



delicate balance would have to be achieved. The yielding pillars would have to support the rock within the arch, and the mining panel would have to be wide enough to allow good productivity. If the panel were too wide, however, the arch would be too high, and the yielding pillars would overload.

Management had no reliable theory to apply, so design information had to be obtained through experimentation in the mine without impeding production. It was reasonably certain that the behavior of any yielding, stiff, or very rigid pillar would depend on the width-to-height ratio. Further, it was hypothesized that the ratio for a yielding pillar should be approximately 3:1, which meant that the yielding pillars should be between 24 and 30 ft wide. By fortunate coincidence, pillars of that size could be produced by driving standard 32-ft entries through the middle of standard 88-ft-square pillars, leaving four 28-ft-square stubs.

Thus, plans were made to split four of the large pillars close to the mining front, using face equipment to perform the work—and, in the process, to obtain some easy salt. (See Fig. 7.)

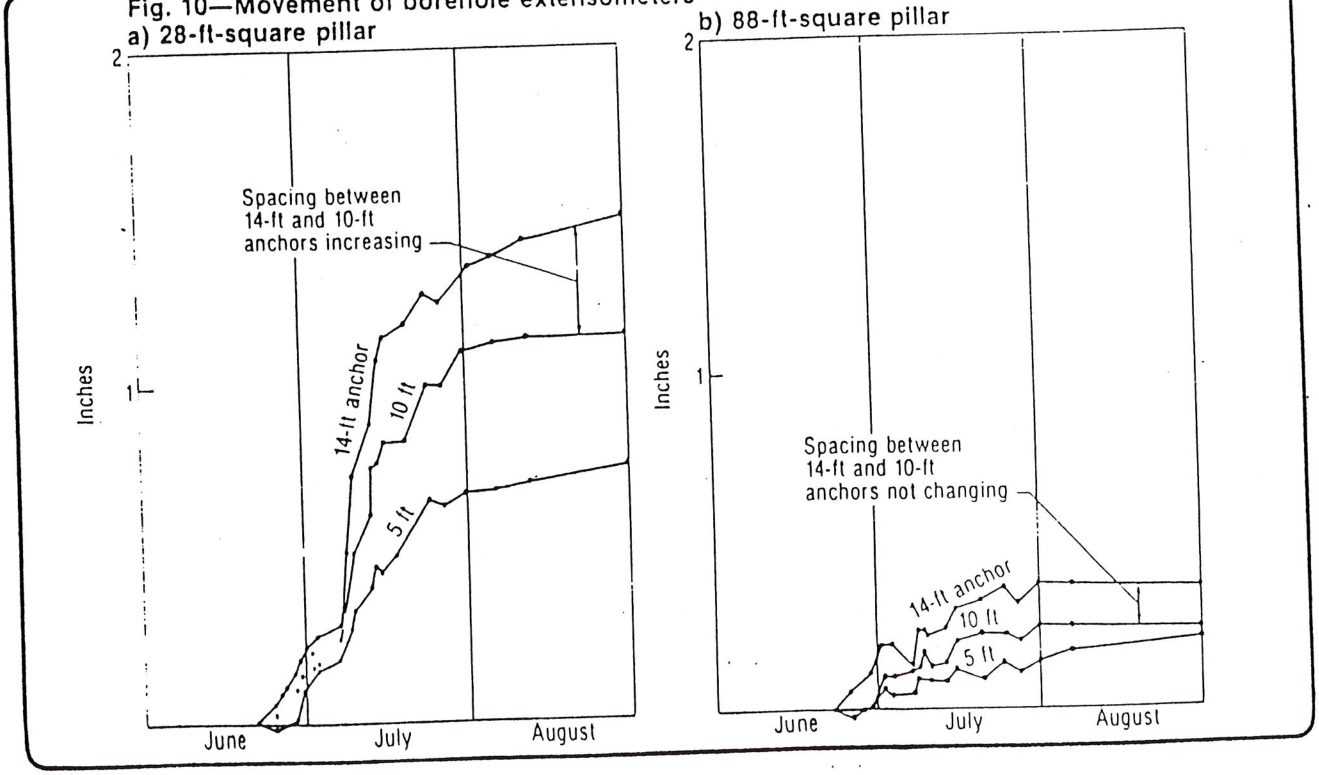
INSTRUMENTING THE TEST

The decision to reduce pillars to 28 ft square and boost the extraction ratio to 79% would increase the nominal pillar load to 13,600 psi. The plan called for the 28-ft-square pillars to shed the load onto larger adjacent pillars, thus relieving some of the stress in the roof among the small pillars. To make certain that the stress was being relieved, two forms of instrumentation were used:

- Convergence equipment. Small reference points were installed in the roof and floor, and the initial distance between them was measured to the nearest thousandth of an inch with a Reed-type extensometer. Follow-up measurements revealed any changes in distance between roof and floor, as well as the rate of change.

Types of convergence measured in this way commonly include roof sag, pillar yield, and floor heave. At the Cayuga

Fig. 10—Movement of borehole extensometers



mine, it was determined that the movement was mainly pillar yield among the small pillars and roof sag among the large pillars. Graphs of convergence measurements were drawn to show the effects of mining and the trend of convergence, toward stabilization or failure.

■ Borehole extensometers. A yielding pillar cannot have a rigid core, and the core of a rigid pillar cannot yield. To check the behavior of pillars, borehole extensometers were installed in both 28-ft and 88-ft squares. The design was simple and effective. A hole was drilled into the pillar, and wires were anchored at various depths, with the ends extending from the collar of the hole. The distance from a mark or split shot on each wire to a reference point at the collar of the hole was measured periodically. Measurement to the nearest 0.01 in. was found precise enough, since the rock salt moves a great deal. The measurements taken at the collar showed the relative movements of the anchors, indicating whether the inner parts of the pillar were rigid or yielding.

RESULTS OF PILLAR-SPLITTING TEST

The results of the experiment are shown most clearly by graphs. Fig. 8 shows how the convergence rate at Point 1, which was typical of the area of split pillars, responded to the mining of each pillar and to each delay, settling down to a moderate trend after mining ceased. The graph indicates that the small pillars *did* yield, and the moderate long-term trend shows that a small pillar could not possibly support the nominal load of 13,000 psi. Therefore, load must have been shed onto adjacent, larger pillars.

Some degree of continuing convergence is normal everywhere in the mine. The rates vary, and they seem to be related to the load on the salt.

Fig. 9 illustrates how splitting the pillars affected a convergence station 200 ft from the test. There was an immediate response when splitting began, and rates diminished when splitting ceased. These were direct indications that the load had been shifted from the split pillars onto a wide zone of the larger pillars. This also indicated that pillars 88 ft square were not big enough to serve as abutment pillars, which are meant to accept and restrict the spread of the transferred load.

Fig. 10a shows the apparent movement of borehole extensometer anchors within a 28-ft-square pillar. Since the deepest anchors moved apart, the core of the pillar must have been yielding.

Fig. 10b charts the relative movement of anchors within an 88-ft-square pillar. The outer 10 ft of the pillar moved into the entry, but the greater width of the pillar had lent rigidity to its core.

Visual observations were in agreement with the measurements: The first cuts into pillars were difficult and noisy, suggesting that the skins of the big pillars were highly stressed; roof conditions were unusually good in the new entries through the pillars, probably because of low pillar load; and roof conditions deteriorated in the entries around the test area, with a roof fall occurring just south of the test area (Fig. 7).

MINING A LARGER TEST PANEL

Based on the encouraging results of pillar splitting, plans were drawn up for a larger test panel, with nine 32-ft-wide entries to be driven eastward about 200 ft. This would allow development of a row of "transition" pillars, each 40 x 88 ft, and two rows of pillars each 28 ft square. Fig. 11 shows the layout of the larger (NE) test panel and the location of the convergence points.

Although the small pillars were under a nominal load of 13,600 psi, with a nominal safety factor of 0.37, miners expressed dissatisfaction when the panel was stopped. The short tramming distances made it easy for miners to earn a production bonus, but management was compelled to stop temporarily to assess the stability of the new panel. Convergence measurements showed that conditions stabilized quickly.

Fig. 11—NE test panel and convergence points

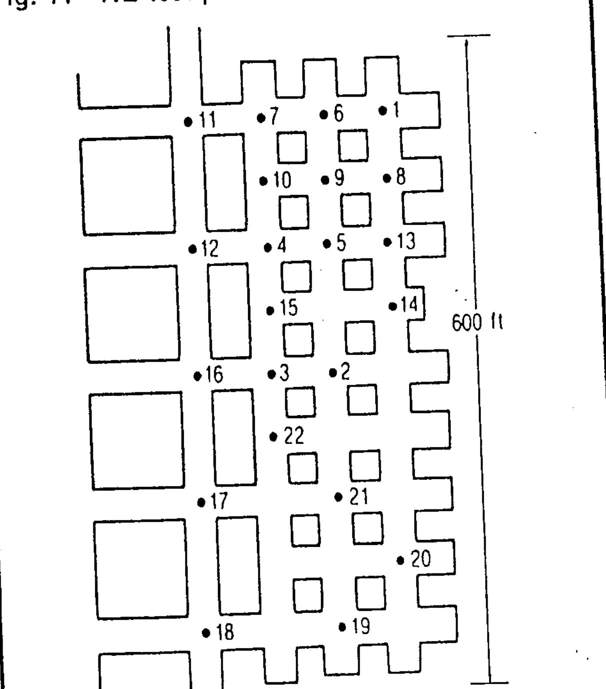


Fig. 12—Convergence rates at pillar intersections during undercutting

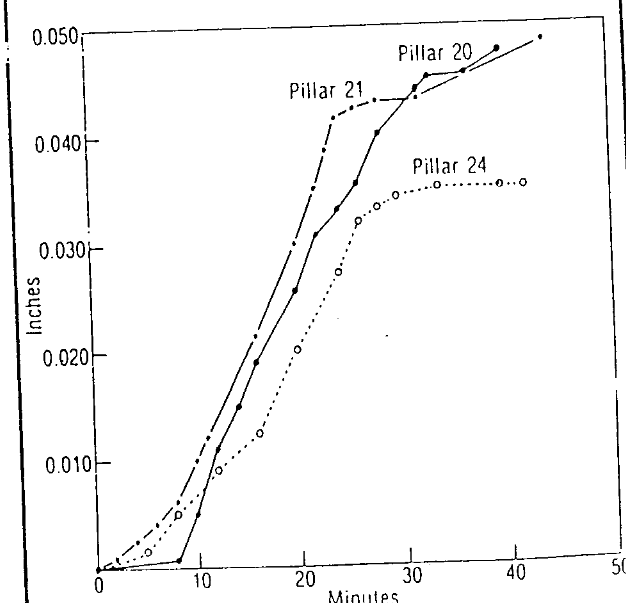
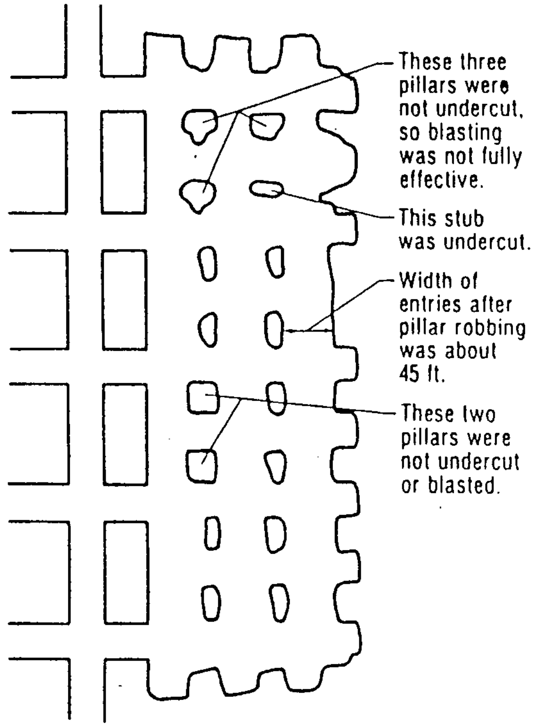


Fig. 13—NE test panel after pillar robbing



Since this panel was a good distance from the production front, it was decided to test the limits of the yielding-pillar/pressure arch concept by reducing the size of the 28-ft-square pillars.

Editor's note: This paper was first presented at the 18th US Symposium on Rock Mechanics in Keystone, Colo., June 22-24, 1977.

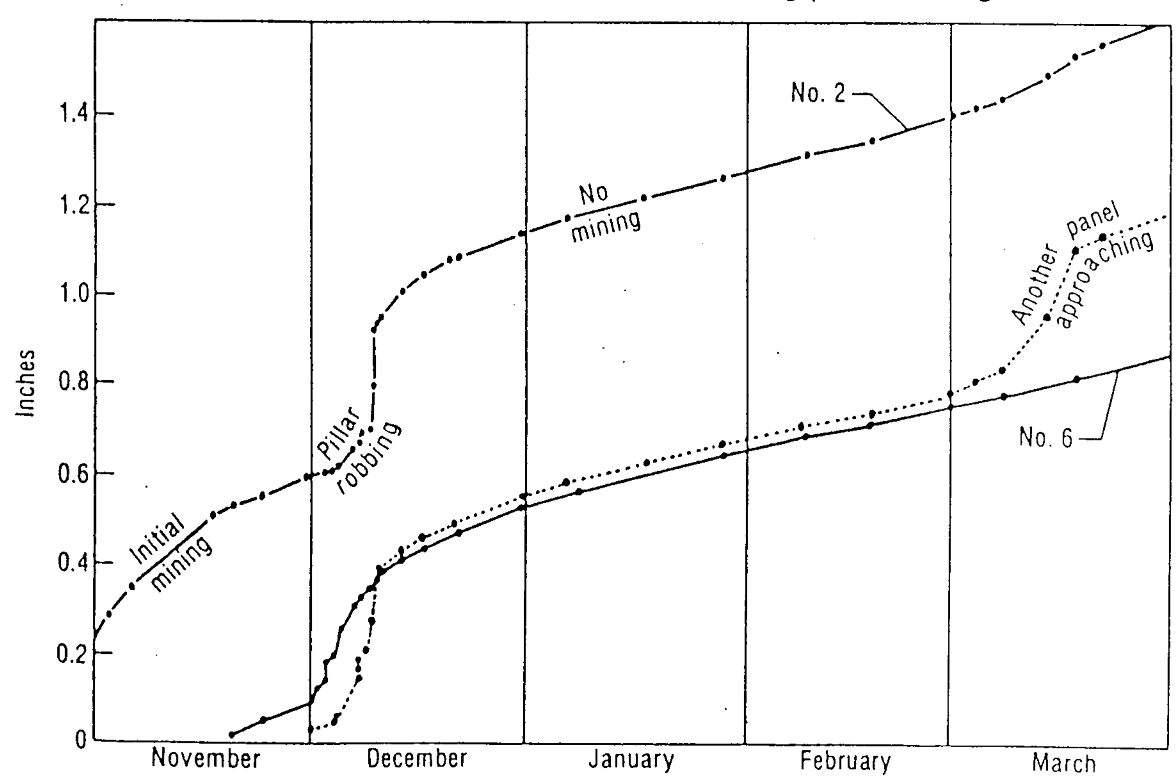
Initially, management decided not to expose the cutter operator to an unstable roof, so simple blasting of slabs from the pillars was tried. However, blasting was not very successful without the undercut. Because convergence rates settled so quickly after the blast, the cutter was used, but as an extra safety precaution, two or three extensometers were continuously monitored during cutting and mucking. At first concerned over the rates of convergence during the undercutting of each pillar, management soon was pleased with pillar behavior.

Fig. 12 graphs convergence rates measured at intersections adjacent to three pillars as the pillars were undercut. The responses to the start and finish of a cut were almost immediate; in fact, by watching the dial of the extensometer, it could be determined when a machine was cutting salt and when it was idling. Total convergence was found to be almost exactly the same for all cuts, with changes measured in thousandths of an inch over a total height exceeding 100 in. The behavior of salt appeared consistent and predictable.

Two of the pillars were not finished because a feeder-breaker was moved up in the production panel. The net result was the stubs shown in Fig. 13.

Fig. 14 shows how the rates of convergence accelerated during pillar robbing, but decelerated to stable, predictable levels soon after mining ceased. The rates are believed to depend on the height of the arch, which in turn depends on local geology and panel width.

Fig. 14—Acceleration of convergence rates during pillar robbing



Chapter 18.1

ROOM AND PILLAR MINING

IAN FARMER

18.1.1 INTRODUCTION

Bullock (1982a), quoting previous data, showed that room and pillar mining together with stope and pillar mining accounted for most of the underground mining in the United States. He estimated that 60% of noncoal minerals (about 80 million tons or 70 Mt) and 90% of coal (about 290 million tons or 260 Mt) were obtained by room and pillar methods, and it is unlikely that things are radically different today. The method is cheap, highly productive, easily mechanized, and relatively simple to design. Ultimately, and particularly with increasing depths, mechanized longwall methods will make greater inroads into both coal and noncoal mining. But longwall requires major capital investment and development costs, and even now design is difficult, and success not always certain. In particular, longwall is inflexible. The rapid advance rates required to provide an adequate return on capital mean that all except very minor geologic faults must be avoided. Thus quite large areas of reserve are not minable using longwall methods, and they often give much lower overall recovery than retreat room and pillar mining, which is highly flexible.

The *room and pillar mining* method is a type of open stoping used in near horizontal deposits in reasonably competent rock, where the roof is supported primarily by pillars. Ore—or more commonly, coal—is extracted from rectangular shaped rooms or entries in the ore body or coal seam, leaving parts of the ore or coal between the entries as pillars to support the hanging wall or roof. The pillars are arranged in a regular pattern, or grid, to simplify planning and operation. They can be any shape but are usually square or rectangular. The dimensions of the rooms and pillars depend on many design factors, which will be considered later. These include the stability of the hanging wall and the strength of the ore in the pillars, the thickness of the deposit, and the depth of mining. The objective of design is to extract the maximum amount of ore that is compatible with safe working conditions. The ore left in the pillars is usually regarded as irrecoverable or recoverable only with backfill in noncoal mines. In this case backfill costs or the potential loss of valuable resource may be a limiting factor in room and pillar mining at greater depths. In coal mining, pillars are, ideally, recovered by retreat mining, allowing the roof to cave, thus relieving stress and reducing the likelihood of bumps.

The applications of pillar mining have been discussed by Hamrin (1982) and Hittman Associates (Anon., 1976), among others. Suitable conditions include ore bodies that are horizontal or have a dip of less than 30°. A major requirement is that the hanging wall is relatively competent over a short period of time, or is capable of support by rock bolts that are used extensively in room and pillar mining. The method is particularly suited to bedded deposits of moderate thickness (6 to 20 ft, or 2 to 6 m) such as coal—the main application—salt, potash, and limestone.

18.1.2 DESIGN OF PILLARS

18.1.2.1 Pillar Stress

Much of the following discussion is based on geomechanics theory presented in Chapter 10.5 and other chapters of Section 10.

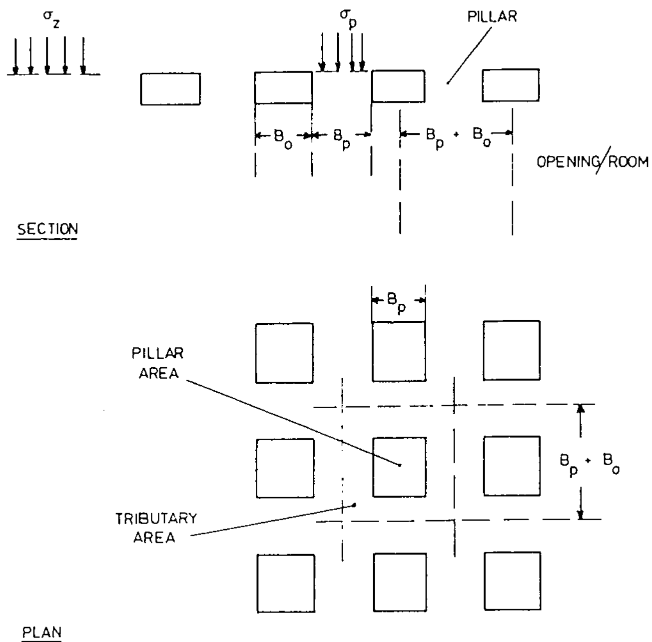


Fig. 18.1.1. Section and plans of rooms and pillars with widths and dimensions for simple analysis.

Despite the simplicity of the structure, and the detailed knowledge of rock behavior obtained over the past few years, pillar design has changed very little during the present century. It is based on the assumption that the stress in a pillar is evenly distributed and equal to the original vertical geostatic stress divided by the pillar area/original area ratio; and that pillar failure occurs when this stress exceeds the compressive strength of the pillar rock. It would be a naive assumption for any engineering structure in any material. It is particularly so in the case of pillars with high width/height ratios in a jointed, brittle material such as rock.

The major recent work on stresses acting on pillars has been carried out by Coates (1981). He started with the simplest and traditional statement of average pillar stress, known as the tributary area method. This assumes that each of the pillars left during excavation supports all the overlying strata that are "tributary" to their location. Then the average pillar stress σ_{pa} for square pillars with rooms of consistent width is

$$\sigma_{pa} = \sigma_z \frac{(B_p + B_o)}{B_p} \tag{18.1.1}$$

where B_p and B_o are width of the pillar and room, respectively (Fig. 18.1.1), and σ_z is the geostatic or premining stress acting normal to the plane of excavation. If this is horizontal, then

$$\sigma_z = \gamma z \tag{18.1.2}$$

where γ is rock average unit weight and z is depth to the mining horizon. This can be stated more simply for the common case of rectangular or irregular shaped pillars in terms of the extraction ratio R, where $R = \dfrac{B_o}{B_o + B_p}$ is the ratio of the area extracted to the total area of the orebody mined. Since $1 - R = \dfrac{B_p}{B_o + B_p}$, Eq. 18.1.1 can be more generally stated,

$$\sigma_{pa} = \sigma_z \left(\frac{1}{1 - R}\right) \qquad (18.1.3)$$

This approach assumes that the mined area is extensive and shallow, that the mined rock is horizontally stratified, and that the pillars are equidimensional. It specifically ignores the relative extent and depth of the mined area, the stress component parallel to the plane of mining, the relative deformation properties of pillar, roof, and floor rocks, and the positions of the pillars in the mining zone. Taking some of these into account, Coates (1981) obtained a more general solution, principally for deep, long, mine pillars but applicable generally, by solving the statically indeterminate net deflection of the roof and floor rocks resulting from mining. Then the solution for average pillar stress becomes

$$\sigma_{pa} = \sigma_z \left\{ \frac{\left[2R - K_o \dfrac{H}{L}\dfrac{(1 - 2\nu_w)}{(1 - \nu_w)} - \dfrac{\nu_p}{(1 - \nu_p)} K_o \dfrac{H}{L}\dfrac{E_w}{E_p}\right]}{\left[\dfrac{H}{L}\dfrac{E_w}{E_p} + 2(1 - R)\left(1 + \dfrac{1}{N}\right) + 2\dfrac{RB}{L}\dfrac{(1 - 2\nu_w)}{(1 - \nu_w)}\right]} \right\} \qquad (18.1.4)$$

where H is seam height; L is the extent of the mined area; K_o is the ratio between σ_h and σ_z or the coefficient of geostatic stress; and E_w, E_p, ν_w, and ν_p are the elastic constants of the wall (roof and floor) and pillar materials.

This is a two-dimensional elastic solution in plane strain and requires, strictly speaking, a length/width ratio of about 3 or more to be applicable. An analytical three-dimensional approach is not feasible, although finite element and boundary element methods (see for instance Tang and Peng, 1988) can be used to give a numerical solution.

Coates' (1981) approach is helpful in that it can be used to illustrate simply several of the fundamental characteristics of strata and geometry that affect pillar stresses. Some of these are illustrated in Fig. 18.1.2. For instance, as the E_w/E_p ratio rises (Fig. 18.1.2a), so the pillar stress is reduced from a magnitude close to $4\sigma_z$ (the extraction ratio has been chosen as 80%) to a level of $0.5\sigma_z$ for $H/L = B/L = 0.1$. This illustrates the bridging effect of the stiffer roof and floor layers and the tendency to transfer stress to the side abutment. Similarly, as L is decreased (Fig. 18.1.2b), the pillar stress is reduced from a maximum magnitude of $4\sigma_z$ to zero and $H/L = 0.4$ for a E_w/E_p ratio of 6. Again this can be attributed to bridging at low spans. As a further illustration (Fig. 18.1.2c), using fixed values for E_w/E_p, H/L, B/L, there is considerable variation between the tributary area calculation (Eqs. 18.1.1 and 18.1.4) for stress at increasing extraction ratios.

It should be emphasized that this is used as an illustration, and that measurements of *average* pillar stresses are very infrequent. In fact, a review of the literature shows virtually no

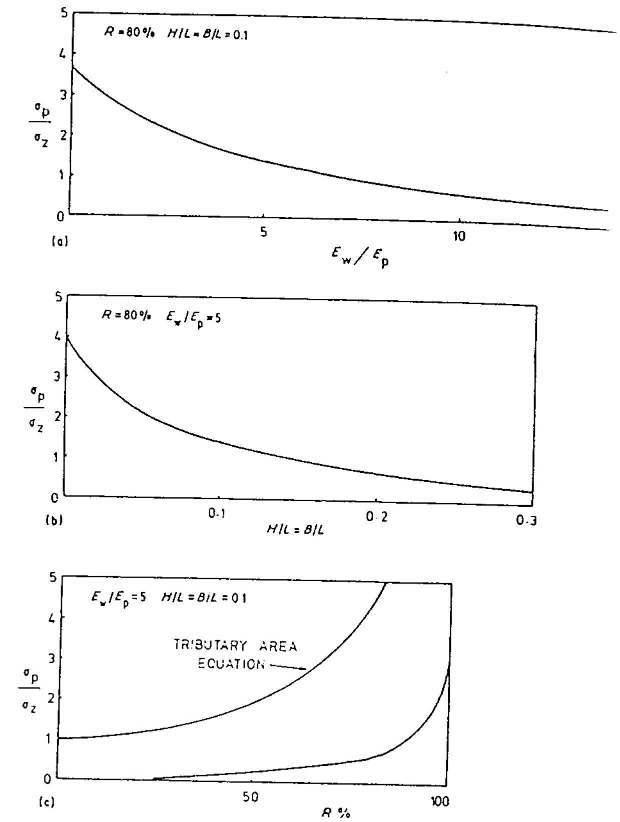


Fig. 18.1.2. Estimates of pillar stress σ_p as a proportion of ver stress σ_z based on the variables in Eq. 18.1.4, putting $K_0 = 1$, ν $\nu_w = 0.33$, and N large, so that

$$\frac{\sigma_p}{\sigma_z} = \frac{(2R - 0.5H/L) - (0.5H/L \times E_w/E_p)}{H/L \times E_w/E_p + 2(1 - R) + RB/L}$$

coal mines. He describes two case histories in which surf settlements and underground displacements were measured ing leveling and anchors in boreholes drilled from the surfac the seam level and below. The seams were at average depth 131 ft (40 m) and 223 ft (68 m). The purpose of the measureme was to test an analog model, and satisfactory simulation allo computation of pillar stresses from observed seam deformatic

The pillar geometries and data on the mining and instrum tation layouts are illustrated in Figs. 18.1.3 and 18.1.4 toget with the pillar stresses σ_{pa}, computed from seam deformati in Figs. 18.1.3c and 18.1.4c. These are quite close to the pi stresses σ_{pa} computed from the tributary area equation (18.1.1). In these cases, the E_w/E_p and H/L ratios were, resp tively, 3 and 0.01 and 2 and 0.05, and it can be seen from F 18.1.2 that such a result would be expected. It is interesting note the reduced pressure on the pillars adjacent to the ribsi and also the relatively low level of the abutment stress. T former would be expected; the latter is rather surprising a implies some weakening of the abutment.

The concept of average pillar stress is not a good one, si pillar stresses are not evenly distributed. This can be illustra simply by stress analysis. A simple two-dimensional bound element program, developed by Bray and Hocking among oth

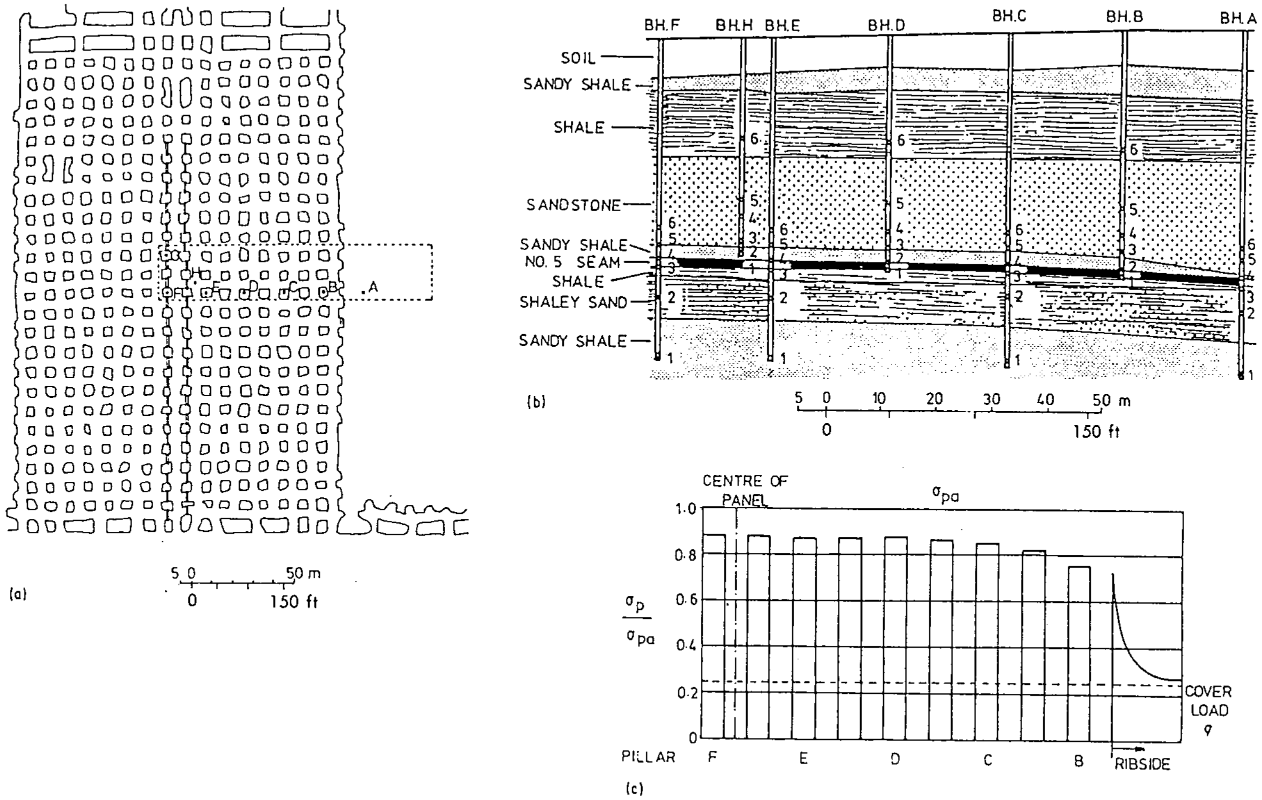


Fig. 18.1.3. Estimation of pillar stress σ_p as a proportion of pillar stress σ_{pa} computed from tributary area theory from experiments by Oravecz (1977) in No. 5 seam at Colliery A., South Africa. Data: average depth to mid-seam 40.3m; seam height 1.5m; pillar width 5.2m; room width 5.5m; percentage extraction 76.4%; panel width 176.2m (est.); deformation modulus, seam (est.) 1.54GNm^{-2}; deformation modulus strata (est.) 4.43GNm^{-2}; Poisson's ratio (est.) 0.15. Conversion factors: 1 ft = 0.3048 m, 10^6 psi = 6.894 GNm^{-2}.

ditions of plane strain in an infinite medium subjected to various combinations of uniform field stresses or external loadings. Typical solutions are given in Hoek and Brown, and the solutions for square and rectangular openings in a uniform stress field are reproduced in Fig. 18.1.5. Although the boundary conditions may be a little extreme for room and pillar mining, a simple example of how these computed stress distributions can be used in pillar design is given in Fig. 18.1.6. This takes the stress distribution in Fig. 18.1.5b and assumes initially two square rooms of dimension a at a distance 4a apart. Then the minor principal stress or confining stress in the pillar between the two can be projected on to a graph of minor principal stress against pillar width, to give the minor principal stress distribution and the average minor principal stress. This can be computed for pillars of any width (see Fig. 18.1.6c), and the resultant distribution can be used to compute the ultimate pillar strength using the strength envelope of the rock or coal in the form,

$$\sigma_{1f} = \sigma_{cf} + K_p \sigma_3 \tag{18.1.5}$$

Then σ_{1f} can be compared with the pillar stress σ_{pa} computed from the tributary area Eq. 18.1.1 to obtain an estimate of safety factor.

18.1.2.2 Pillar Strength

There is a large literature on pillar strength, much of it empirical. The most complete work is by Salamon and Monro (1967), and the best summaries by Bieniawski (1981) and Tsur-Lavie and Denekamp (1982). For detailed coverage of pill strength theory, see Chapter 10.5.

The basic problem with pillar strength is that in a britt rock, strength is dependent upon the size, and to a lesser exter the shape of a test specimen. This means that the convention method of pillar design, relating rock strength to pillar stre through a factor of safety ($FOS = \sigma_{cf}/\sigma_p$), is unacceptable brittle rocks, although it may be acceptable in more ductile rock The reason for this is evident: if failure occurs in a brittle manne the strain energy stored in a pillar will be released from a volur onto a shear or tensile failure plane, where it will be distribute as surface energy per unit area of fracture surface; a constant f a particular rock. This is the basis of the Griffith failure criteric and is explained in Farmer (1985). Since energy is proportion to the square of stress, this means that strength will be inverse proportional to the square root of the dimension of the ro specimen, an observation confirmed experimentally by Bienia ski (1981) and Singh (1981) for various rocks including coal. terms of pillar σ_{pf} and rock σ_{cf} strength, this can be express

$$\frac{\sigma_{pf}}{\sigma_{cf}} = \left(\frac{L_s}{L_p}\right)^{1/2} = \left(\frac{V_s}{V_p}\right)^{1/6} = \left(\frac{V_s}{V_p}\right)^{0.17} \qquad (18.1.$$

where L and V represent dimension and volume, respectivel and the subscripts s and p refer to the laboratory specimen f strength testing and the pillar, respectively. In the ductile cas the energy is not transferred onto fracture surfaces but even distributed in the specimen or pillar. Then the exponent a

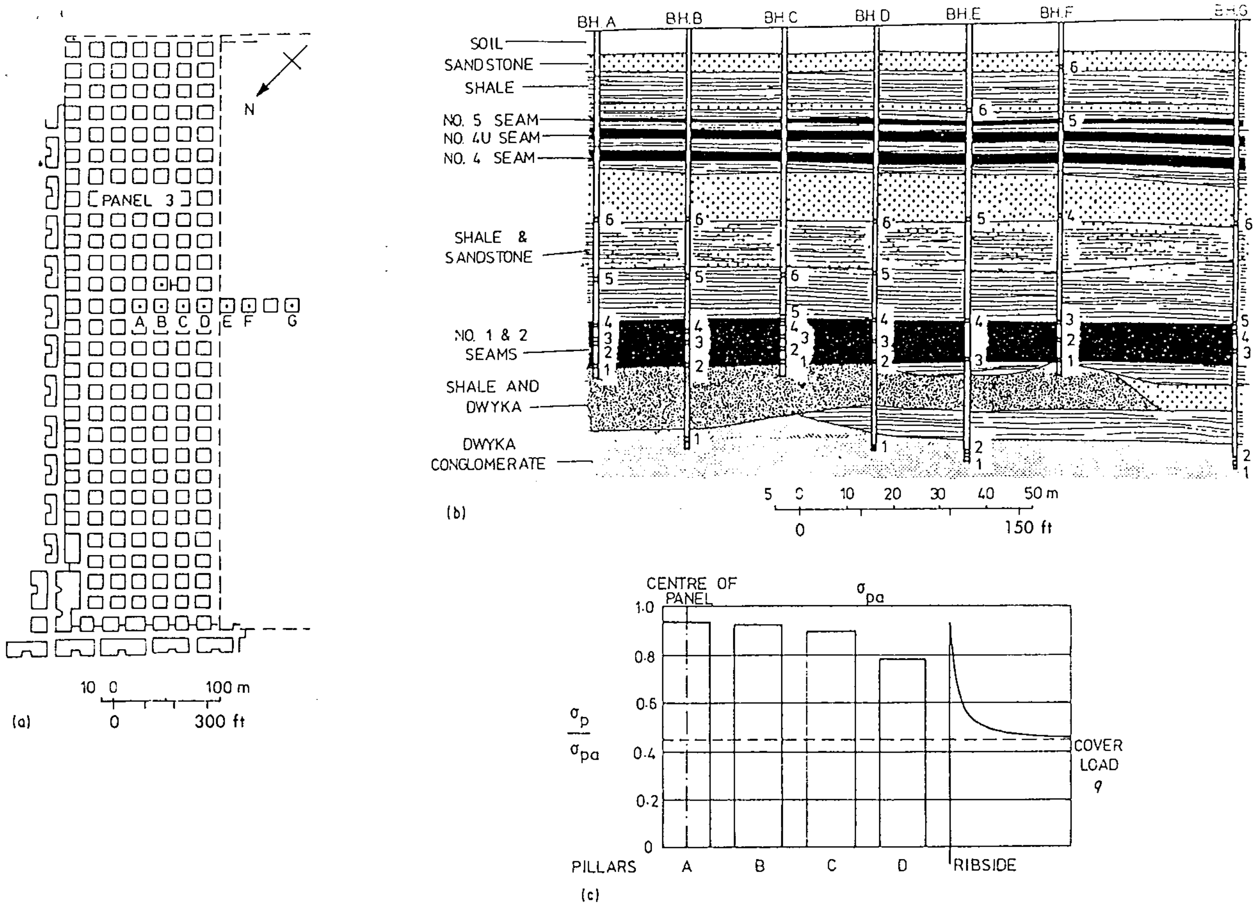


Fig. 18.1.4. Estimation of pillar stress σ_p as a proportion of pillar stress σ_{pa} computed from tributary area theory, from experiments by Oravecz (1977) in No. 2 seam at Colliery B., S. Africa. Data: average depth to mid-seam 66.7m; seam height 5.5m; pillar width 13.7m; room width 6.1m; percentage extraction 52.1%; panel width 144.8m; deformation modulus, seam (est.) 3.92 GNm^{-2}; deformation modulus, strata (est.) 6.27 GNm^{-2}; Poisson's ratio (est.) 0.15. Conversion factors: 1 ft = 0.3048 m, 10^6 psi = 6.894 GNm^{-2}.

proaches unity. Thus, in the case of wide pillars, and pillars in pseudo-ductile rocks such as rock salt, Eq. 18.1.6 can be modified.

The relevance of Eq. 18.1.6 can, however, be confirmed by the empirical work of Hardy and Agapito (1977) on oil shale pillars in western Colorado. They proposed a general pillar formula which is recommended for all *brittle* rocks—that is, where the pillars fail in tension or shear—in the form,

$$\frac{\sigma_{pf}}{\sigma_{cf}} = \left(\frac{V_s}{V_p}\right)^{0.118} \left[\left(\frac{B_p}{H_p}\right) \Big/ \left(\frac{B_s}{H_s}\right) \right]^{0.833} \tag{18.1.7}$$

where B and H are pillar and specimen width and height, respectively. There are, of course, limitations for this approach, one of which would probably be the pillar width/height ratio. If this is less than 1, and particularly if the rock is ductile, the volume exponent will increase.

For the record, although the above method is strongly recommended, it is useful also to include the conventional representations of pillar design equations, often called the Holland-Gaddy (Holland, 1964) equation in the United States, which take the form,

$$\sigma_{pf} = \sigma_{cf}\left(a + b\,\frac{B}{H}\right) \tag{18.1.8}$$

$$\sigma_{pf} = K\,\frac{B^{\alpha}}{H^{\beta}} \tag{18.1.9}$$

In this case, σ_{cf} is uniaxial compressive strength of a cube of specified dimension; a and b are dimensionless constants, usually chosen so that $a + b = 1$ and $\sigma_{pf} = \sigma_{cf}$ when $B/H = 1$; α and β are dimensionless constants; and $K = f(\sigma_{cf})$ is a constant so that $K = \sigma_{cf}$ when $\alpha = \beta$ and $B = H$. There is a reasonable agreement about constants a, b, α, and β in Eqs. 18.1.8 and 18.1.9. Some representative values from early times to more recent are quoted in Table 18.1.1, principally for coal mines. All of the constants are effectively shape factors. The basic problem is that σ_{cf} in either equation is essentially the laboratory value, and a factor of safety, usually not included in the equation, is needed to allow for size effects and ensure safe design. Quoted values of this "safety factor" are difficult to find. Wilson (1983) suggests 5 for coal, but incorrectly recommends 1 for strong massive unjointed rock and 6 to 7 for weak rock—quite the reverse of the probable actual values. Where the economic success or failure of an operation depends on correct estimation of extraction ratio, a more accurate approach is required and Eq. 18.1.7 is recommended as a starting point. This represents a safety factor of 4 to 5 for most rocks and pillar shapes.

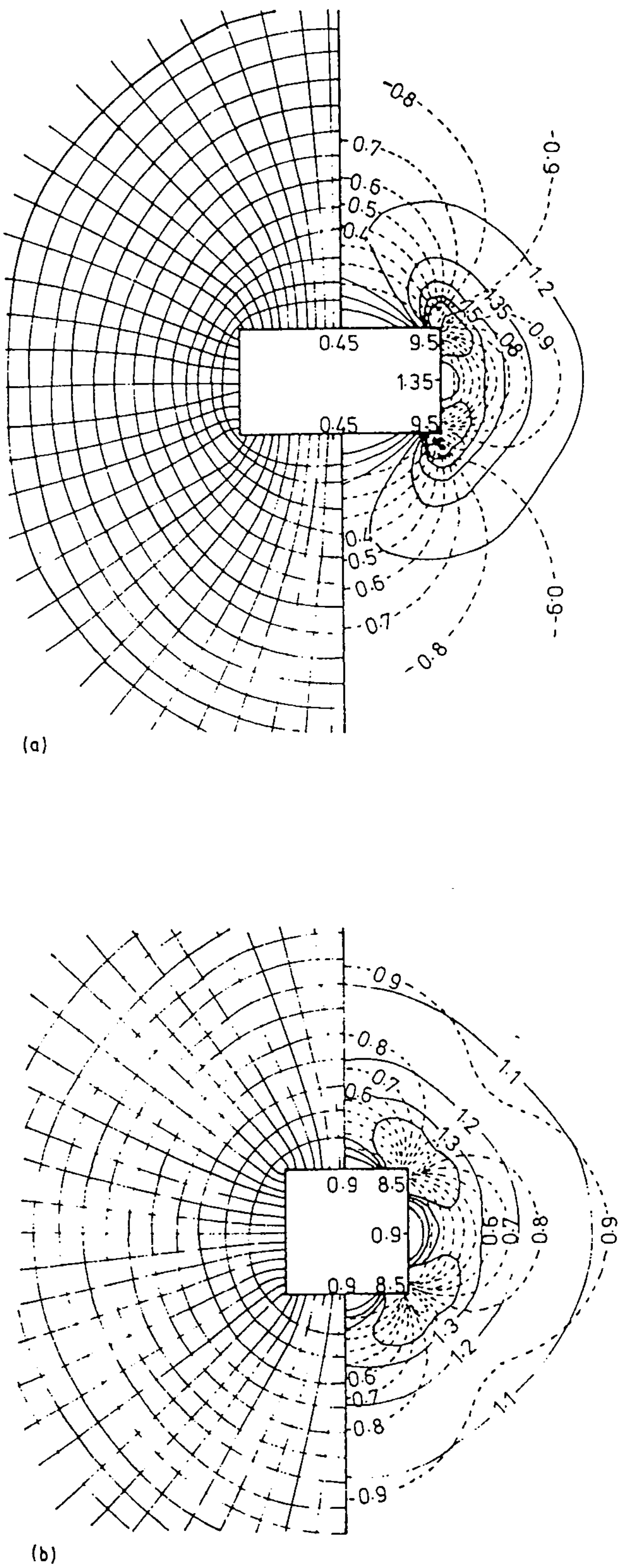


Fig. 18.1.5. Principal stress trajectories (LHS) and contours (RHS) of the ratio of major principal stress to applied stress (solid line) and minor principal stress to applied stress (dotted line) for (a) a rectangular and (b) a square opening in an infinite medium subject to a uniform stress field (using Bray and Hocking's two-dimensional boundary element analysis, in Hoek and Brown, 1981).

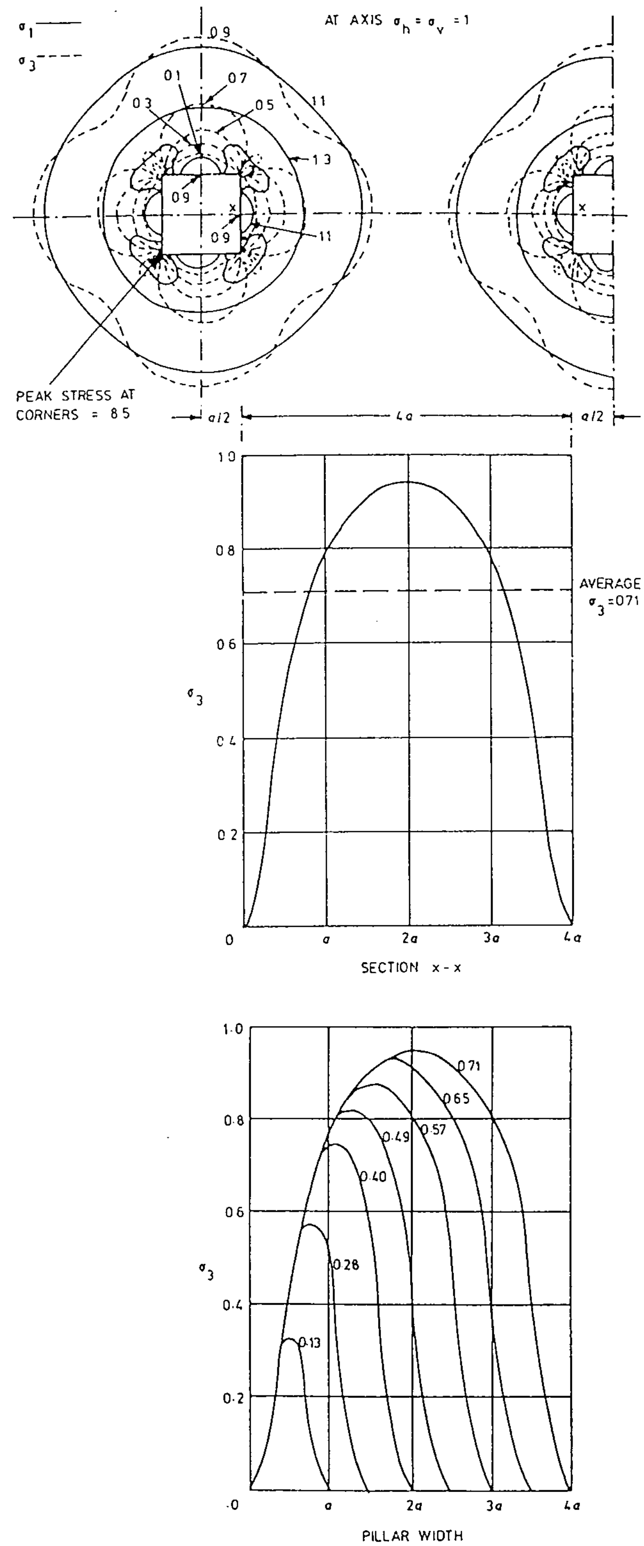


Fig. 18.1.6. (a) Contours of major (solid line) and minor (dotted line) principal stress around two rooms of dimension a separated by a pillar 4a in width, and (b) plotted to give minor principal stress (expressed as a proportion of applied stress) distribution in the pillar, and (c) relation between minor principal stress expressed as a proportion of uniform applied stress and pillar width—for pillars of varying width. Values for average σ_3 are given for each curve.

Bunting (1911)	0.7	0.3	—	—	—	Laboratory data
Obert, Windes, and Duvall (1946)	0.78	0.22	—	—	—	Laboratory data
Bieniawski (1968)	0.64	0.36	—	—	—	In situ—S. Africa
Van Heerden (1974)	0.70	0.30	—	—	—	In situ—S. Africa
Wang, Skelly and Wolgamott (1977)	0.78	0.22	—	—	—	W. Virginia mines, United States
Sorensen and Pariseau (1978)	0.69	0.31	—	—	—	Statistical—United States
Greenwald, Howarth, and Hartmann (1939)	—	—	0.5	0.83	—	In situ—Pittsburgh mines, United States
Streat (1954)	—	—	0.5	1	—	Statistical—S. Africa
Holland (1964)	—	—	0.5	1	—	Statistical—United States
Salamon and Monro (1967)	—	—	0.46	0.66	—	Statistical—S. Africa
Bieniawski (1968)	—	—	0.16	0.55	—	Statistical—S. Africa
Hazen and Artler (1972)	—	—	0.5	0.5	—	Statistical—United States
Zern (1926)	—	—	0.5	0.5	—	Empirical—United States
Morrison, Corlett and Rice (1975)	—	—	0.5	0.5	—	In situ—Canada
Greenwald, Howarth, and Hartmann (1939)	—	—	—	—	9.3	Originally in psi for B, H values in inches-Pittsburgh, United States coals
Salamon and Monro (1967)	—	—	—	—	9.1	Originally in psi for B, H values in feet-S. Africa
Bieniawski (1968)	—	—	—	—	6.9	Originally in psi for B, H values in feet-S. Africa
Jenkins and Szeki (1964)	—	—	—	—	12.4	Originally in psi for B, H values in feet-Britain
Wagner (1974)	—	—	—	—	11.0	B, H values in meters. Based on in situ tests-S. Africa

Conversion factor: 10^3 psi = 6.894 MN m^{-2}.

18.1.2.3 Barrier Pillar Design

Room and pillar mines are usually developed in a series of rectangular panels separated by barrier pillars. There is no specific design method for these pillars, but where the roof is not caved or where pillars are left in place, design of barrier pillars assumes greater importance. Fig. 18.1.2 shows that pillar stress is not necessarily evenly distributed, and where the roof and floor rocks are stiffer than the pillar rocks, stress with be transferred to an abutment. There is also the probability that deterioration—or overmining—of highly stressed pillars may lead to a reduction in load capacity of individual (or groups of) pillars, and transfer of load to other pillars that may lead to progressive failure. This is one of the most common causes of extensive pillar collapse (Mottahed and Szeki, 1982, describe a total mine collapse), and barrier pillars can control this.

Wilson (1983) analyzed this problem and suggested, for coal mines, barrier pillar widths of 1/10th of the working depth, but his approach, although applied to room and pillar workings, was designed principally to reduce entry damage in longwall entry chain pillars. A more satisfactory approach may be to consider pillar yield. Hudson, Brown, and Fairhurst (1971) in a series of tests on marble, which can be repeated on coal, showed that a pillar behaved in a yielding rather than a brittle manner if its height/width ratio was less than 1/3. The implication is that below this ratio, a pillar will deform rather than fracture, resisting rapid collapse. A yielding, barrier pillar of 3 to 4 times the excavation height can, therefore, be recommended, particularly at greater mining depths.

18.1.3 SUPPORT OF ROOMS

18.1.3.1 Rock Bolts

The key to design of rooms is support. This invariably means the use of rock bolts in room and pillar mining. At present, over 100 million bolts per year are installed in US mines. There are various types of rock bolt, and the type and method of installation can have a significant effect on performance. Classification of rock bolts into types is difficult. Conventionally, there are two methods, either as (1) grouted (usually fully grouted) or (2) mechanically anchored (usually point-anchored) bolts. A list of available bolt types from Peng and Tang (1984) is given in Table 18.1.2. A point-anchored bolt is usually tensioned; a fully grouted bolt is usually untensioned. A mechanical anchor can be installed easily, but is unreliable over a period of time; a resin bolt requires precision in installation—whether point or fully grouted—and usually has better long-term characteristics. The theory of rock bolting is developed fully in Chapter 10.5.

Conventional rock bolts are made from $\frac{5}{8}$-in. (16-mm), $\frac{3}{4}$-in. (19-mm), 1-in. (25-mm), or $1\frac{1}{4}$-in. (32-mm) steel rebar with an approximate yield force, respectively, of 7 (6), 9 (7.5), 17 (15), and 26 tons (23 tonnes). Normally, the installed bolt tension is 50% of this load. Steel bearing plates at the hole collar are usually 6 in. (150 mm) square and $\frac{1}{4}$ in. (6 mm) to $\frac{3}{8}$ in. (9.5 mm) thick and are flat or bell-shaped with a center hole. The main function is to distribute stress to the rock at the collar through a nut threaded on to the top of the bolt, and tensioned through a drill chuck. Angle or spherical washers are used to create a uniform bearing surface. To prevent falls of rock between bolts—an important factor in weaker rocks—mesh or bench bars are placed behind the anchor bearing plates. For long-term intallations, shotcreting is essential.

Bolts are usually considered temporary supports. At bolt forces close to working load, they are, like all rock stress systems, prone to deterioration with time. At differential roof deformations, greater than 1 to $1\frac{1}{2}$%, they usually cease to function, although performance can be improved with shotcreting. The reduction or change in capacity with time is not well documented and relies to a great extent on ground conditions. A particularly useful recent paper by Signer and Jones (1990) illustrates the changing reinforcement loads on fully grouted bolts during roof deformation and illustrates their very flexible response to deformation.

In the case of mechanical bolts, installation is invariably accompanied by reduction in tension with time. This was investi-

Table 18.1.2. Types of Roof Bolt

Types of bolt	Types of anchor	Suitable strata type	Comments
Point-anchored bolt (tensioned)	Slot-and-wedge	Hard rock	Used in early stages
	Expansion shell	Medium-strength to soft rock	Most commonly used in US
	Resin grout	All strata especially for weak rock	Increased usage recently. Resin length less than 24 in. (0.6 m)
	Combination shell and resin anchor	Most strata	Resin lengths greater than 24 in. (0.6 m)
Full-length-anchored bolt (nontensioned)	Cement grout	Most strata	Disadvantage: (1) shrinkage of cement (2) longer setting time
	Resin grout	All strata	Increased use recently, especially for weak strata
Special bolts: Yieldable bolt	Expansion	Medium-strength rock	An expansion-shell bolt with a yield device
Pumpable bolt	Resin	Weak strata	Complex in installation
Helical bolt	Expansion shell	Most strata	In experimental stage
Split set	Full-length friction	Weak strata	Cheap but needs special installation equipment
Roof truss	Expansion shell	Adverse roof	Recommended for use at intersections and/or heavy pressure areas
Cable sling	Cement anchor and full-length friction	Weak strata	Substitute for timber, steel, or truss support
Lateral force system	Full-length 2-piece steel	Soft strata	Applying full-length lateral force (compression) to the strata
Swellex bolt	Full-length holding	Water-bearing strata	Using high-pressure water to swell the steel tube thus holding the rock

Source: Peng and Tang, 1984.

gated by de la Cruz (1964) and Parson and Osen (1969) among others and was attributed principally to slippage of serrations on the anchor shell, rock deformation and rock breakage at the anchorage and collar, and ground movement following excavation. In addition, dynamic vibration due to blasting is a major cause of tension loss. This means that constant monitoring and retensioning of bolts is needed if long-term installation is required. Conversely excessive bed separations can lead to bolt head failure, which is not found in grouted bolts.

It has been claimed that fully or point-grouted resin or cement anchors give improved performance, both long and short term, and there is some evidence for this. Franklin and Woodfield (1971), in a series of experiments, showed that reliance on bond rather than friction means that the force take-up is much quicker, and by extrapolation, the possibility of slippage is much less. There remain dangers associated with faulty installation, excessive annulus thickness, and poor bonding in wet holes, which in practice can make resin grouting less attractive.

The action of bolts is best described through the typical theoretical stress distributions around the openings illustrated in Fig. 18.1.5. In both cases, the surface of the opening is subjected to compressive tangential stress and zero radial stress. Further away from the surface, both the radial and tangential stresses approach the primitive stress levels in the rock mass undisturbed by excavation. The tangential compression stresses are high at [illegible] in the sidewalls, but low in the roof and floor.

Rock bolts are the cheapest and most obvious way of maintaining stability in such circumstances. Provided that the rocks are suitable for an anchorage location, are not subject to swelling or slaking, and there are no high pore pressures or water flows, then bolts have two main functions acting either singly or as a pattern. These are to maintain the stability of sagging roofs, particularly in weaker stratified rocks, and to restrain blocks in well-jointed or blocky rocks where release surfaces daylight in the exposed roof. The former application is principally for roof support in room and pillar mining in stratified rocks. This is the most common use of rock bolts, and it can be improved by variations such as trusses or slings (see, for instance, Seegmiller, 1990). The latter application is principally in civil engineering works, such as tunnel and cavern construction, and occasionally in slopes, where quite-large-capacity anchors are often used.

18.1.3.2 Support Design

Where a bolt is used to restrain a single block in the roof of an entry, the volume and hence the weight of the block and where necessary its direction of sliding can be determined by stereographic analysis of the kinetics of sliding. This method is outlined in Farmer and Shelton (1980) and in Farmer (1985). Methods of support based on the common requirement that bol

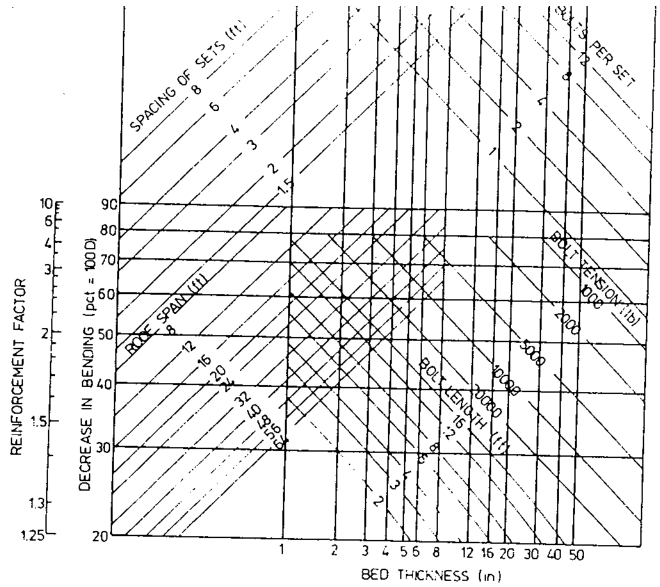


Fig. 18.1.7. Nomograph to determine the friction effect for bolting in mine roofs. $\mu = 0.7$, $\sigma = 24.6 kN/m^3$. Conversion factors: 1 in = 25.4 mm, 1 ft = 0.3048 m, 1,000 lbf = 4.448 kN. (After Panek, 1962b.)

$$P = \frac{\gamma BXL}{\left(\frac{x}{\alpha} + 1\right)\left(\frac{B}{S} + 1\right)} \qquad (18.1.10)$$

where γ is unit weight of the roof rock.

This equation, suggested by Obert and Duvall (1967), is valid if the roof above the excavation is completely suspended by bolts. For an assumed bolt load, it can also be used to estimate spacing and the number of rows. It represents the upper limit of bolt force since it ignores the important supporting effect of the abutments. It also ignores the interaction of a series of roof beds.

A more accurate approximation can be obtained by considering the effects of friction between beds and also by considering the roof span as a series of thin beams, fixed at each side of the opening. Panek (1962a,b; 1964) in a series of seminal papers considered this condition both experimentally by centrifugal testing and analytically, and developed the nomograph illustrated in Fig. 18.1.7, which has been used extensively in mine design. It is explained in detail by Panek and McCormick (1973) in the *SME Mining Engineering Handbook.* The basic variable is a reinforcement factor RF that is used to evaluate the interbed friction effect due to bolting. The roof is considered as a series of beds of equal thickness, of the same material, and without bonding between them. The bolts are assumed normal to the beds and tensioned to give normal compressive loading across the beds. Then

$$RF = \left(1 + \frac{\Delta\sigma_f}{\sigma_{fs}}\right)^{-1} \qquad (18.1.11)$$

where $\frac{\Delta\sigma_f}{\sigma_{fs}}$ is the *decrease* in bending stress from frictional resistance induced by bolting, expressed as a ratio of the maximum

cal equation:

$$\frac{\Delta\sigma_f}{\sigma_{fs}} = \frac{3}{8}\,\mu(aB)^{-0.5}\left[\frac{B}{S}\,P\left(\frac{L}{t} - 1\right)\cdot\frac{1}{\gamma}\right]^{0.33} \qquad (18.1.12)$$

where μ is the interbed coefficient of friction, a is spacing between rows, B is span, S is bolt spacing, t is average roof layer thickness, P is assumed bolt tension, and L is assumed equal to bolt length or supported thickness. For typical thin-bedded mine roof strata, RF should be greater than 2, and bolt spacing must by law be less than 5 ft (1.5 m). Spacings of 4 ft (1.2 m) are more common. Based on Eqs. 18.1.11 and 18.1.12, Panek's well-known nomogram (Fig. 18.1.7) allows rapid estimation of RF for a bolted roof, and forms a basis for rapid rock bolt pattern design.

18.1.3.3 Roof Caving

Although roof caving is not strictly speaking related to support, the mechanics are similar and it can be considered here. Caving is an important part of strata control in all mining operations. Correctly carried out, caving relieves stresses on abutments, barrier pillars, and chain pillars and improves overall mine stability. The need to cave the roof successfully determines the width of a room and pillar panel, as it does the width of a longwall face.

Cavability is a difficult concept. It is usually expressed in terms of a pressure arch, a circular, parabolic, or rectangular zone in the rock above an opening in two dimensions (see Fig. 18.1.5a) that has low radial compression stress, and where the rock sags and ultimately collapses under self weight at a critical unsupported span. This process is assisted by the presence of joints and weaknesses, which is why elastic analysis leaves a certain amount to be desired. The basics of computation of fracture onset in a roof span, analogous to the beam, plate, or "cracked arch," have been considered, with little success, by Obert and Duvall (1967) and Wright (1973). A better approach may be Terzaghi's (1946) arching theory, based on shear resistance in a frictional material above a bin hopper (the unsupported roof), and similar empirical methods that are summarized in Farmer (1985). An outline of this is given in Table 18.1.3. If a bulking factor of 1.1 is assumed for most layered rocks (Gorrie and Scott, 1970), then for caved strata to bulk sufficiently to support upper layers, the *span B* must be such that $1.1\,xB = xB + M$, where M is the excavated (or in the case of coal, seam) thickness, or

$$B = \frac{M}{0.1\,x} \qquad (18.1.13)$$

where $x = 0$ *(good) to* 2 *(poor)* depending on the rock quality in Table 18.1.3. Obviously, a hard and intact rock is not cavable. For a massive, moderately jointed rock, a span in excess of 20 times the excavated thickness (i.e., 200 ft or 60 m, for a 10 ft or 3 m, thick excavation) would be required.

18.1.4 METHODS OF ROOM AND PILLAR MINING

18.1.4.1 Hard-rock Mining

Room and pillar mining takes place in sections or panels, which are usually rectangular and regular in plan. It is important

Table 18.1.3. Relation Between Cavability and Rock Classification Systems

Terzaghi classification	Rock behavior and possible causes of instability	Approximate stand-up time	Deere classification	Rock breakage height, m
1 Hard and intact	Stable excavation unless induced stress greater than rock strength	Many years	Excellent: RQD 90–100	0
2 Hard stratified and schistose	Bed separation with time; surface spalling	1 year		0.25 *B*
3 Massive, moderately jointed	Immediately stable. Detachment of blocks, progressively releasing further blocks	1 week	Good: RQD 75–90	0.5 *B*
4 Moderately blocky and seamy	Immediately stable. Detachment of blocks, progressively releasing further blocks	1 week	Fair: RQD 50–75	0.7 *B*
5 Very blocky and seamy and shattered	Immediately fairly stable. Surface dilation of rock due to rapid block detachment	1 day	Poor: RQD 25–50	1.5 *B*
6 Completely crushed	Local roof falls during excavation. Rapid peripheral dilation	1 hour	Very poor: RQD 0–25	2 *B*
7 Sand and gravel	Immediate collapse	0		
8 Squeezing: moderate depth	Rapid yielding and deformation		Squeezing and swelling ground	

Note: *B* is excavation span. RQD is rock quality designation.
Conversion factor: 1 ft = 0.3048 m.
Source: After Terzaghi, 1946; Deere, 1963.

here to differentiate between hard-rock and coal mining. In hard-rock mining of horizontal ore bodies, the method is very similar to open stoping (see Chap. 18.2). In many cases, ore grade control may be the primary requirement in mine design, and ground control and ventilation secondary considerations. This may lead to an ad hoc room and pillar design with irregular-shaped, nonrecoverable pillars of low-grade ore. In coal mining, ventilation and ground control are major factors, and this requires carefully designed room and pillar panels isolated from the rest of the mine and with a controlled ventilation system. It may also require plans for retreat pillar mining and caving.

Hard-rock room and pillar mining is effectively a method of open stoping (stope and pillar mining) at a low angle to the horizontal, excavating rooms and leaving supporting pillars. Where mineral values vary, the method is similar to the old "gophering" method of mining where random excavations followed highly mineralized zones. Where mineral values are consistent, the mine layout can be regular. The method differs from most hard-rock mining methods in that gravity flow is limited, and ore must be loaded in the excavation where it has been blasted and transported from that point. In large operations, this involves trucks and loaders or load-haul-dumps (LHDs), although slushers may also be used.

There are various methods of room and pillar stoping. The most common are *full-face slicing* or *breast stoping* and *multiple slicing* or *bench and breast stoping,* illustrated in Figs. 18.2.5 and 18.2.7 (see Chapter 18.2). In the former, the rooms are opened to their full vertical height with no mineral or economic value left in the roof or the floor. Probably the reasonable safe limit for full-face slicing is 25 to 35 ft (8 to 10 m) depending on drilling and support equipment, and beyond this, multiple slicing is used. In the United States, most coal, trona, and potash deposits are mined in one slice. Limestone, lead, and zinc mines use multiple slicing. In multiple slicing, the face is divided into a breast or brow, which is the top slice, and a bench (or benches), which is the bottom slice (or slices). It is quite common for mining to be organized so that there is simultaneous mining on the breast and

vided a layer of broken ore is left as a working platform. Overhand stoping is, however, more dangerous since new roof is continually exposed, whereas underhand stoping can be carried out under an undisturbed, supported roof.

18.1.4.2 Coal Mining

The basic unit in room and pillar coal mining is the panel that defines the area of the mine to be worked and ventilated. In the panel, there are two main phases in which the rooms are first developed, isolating the pillars, to the extent of the panel. Then the pillars may be extracted in a reverse direction. Conveyor belts, LHD transports, and services are extended with the room advance and are taken up during retreat pillar extraction. Room advance and pillar extraction can be carried out separately, or at the same time, or the pillars can be left in place. Kauffman, Hawkins, and Thompson (1981) describe four primary methods of production room and pillar mining—principally for coal mines—although they may be adopted for any mining operation. The methods are illustrated in Fig. 18.1.8 and may be summarized as follows.

1. *Panel advanced on entry set; rooms only extracted on retreat* (Fig. 18.1.8a). Here a group of entries, or an entry set, just large enough (usually three or four) to handle the necessary ventilation, haulage, and other support services, is developed, usually in the center of the panel, to the full panel length, connecting through to the return airway gas bleeder system in the case of a coal mine. Then production rooms in sets of four or five are driven in both directions as the equipment is retreated from the panel. No pillar extraction is carried out.

2. *Full panel advanced on rooms; pillars extracted on retreat* (Fig. 18.1.8b). Here a full width panel with 10 to 12 entries is developed off the panel neck to the full panel length, connecting through to the return airway entries and chain pillars to establish a bleeder system. Pillars are then extracted in retreat until the

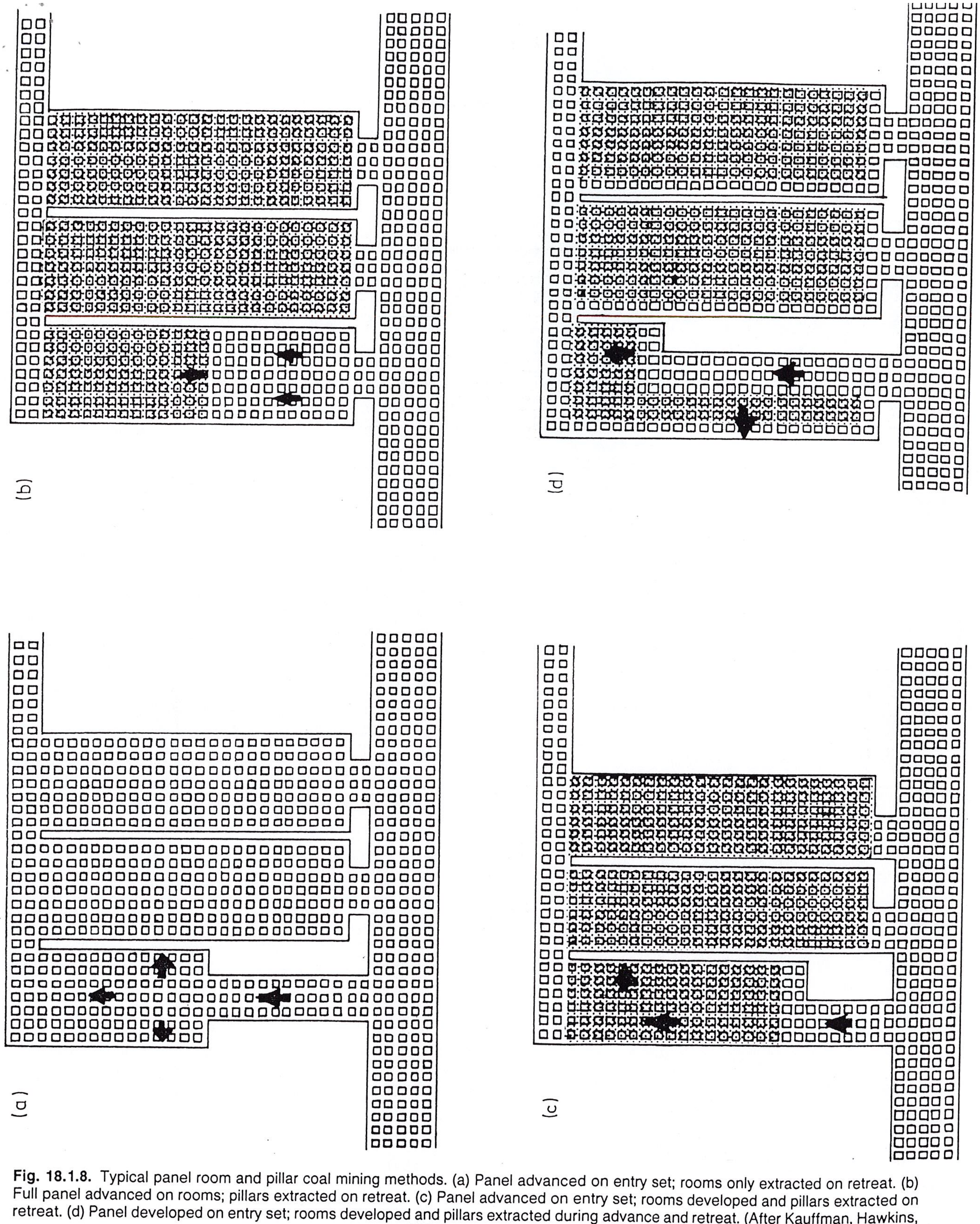


Fig. 18.1.8. Typical panel room and pillar coal mining methods. (a) Panel advanced on entry set; rooms only extracted on retreat. (b) Full panel advanced on rooms; pillars extracted on retreat. (c) Panel advanced on entry set; rooms developed and pillars extracted on retreat. (d) Panel developed on entry set; rooms developed and pillars extracted during advance and retreat. (After Kauffman, Hawkins, and Thompson, 1981.)

3. *Panel advanced on entry set; rooms developed and pillars extracted on retreat* (Fig. 18.1.8c). Here a panel entry set (three to five entries) large enough to handle ventilation, haulage, and support services is developed to the full panel length, usually on one side of the panel, although it can be in the center. After establishing a bleeder system, production rooms are developed to the side of the entry set in groups of three or four, then production and chain pillars are extracted using flat or angled pillar lines. Because of the limitation on the number of working faces, this method is only suitable for continuous mining.

4. *Panel developed on entry set; rooms developed and pillars extracted during advance and retreat* (Fig. 18.1.8d). In this method, rooms are developed and pillars extracted on one side of the panel entry set as the panel is advanced. When the entry set reaches the panel limit, and a ventilation bleeder system is established, the rooms on the other side of the entry set are developed, and the resultant pillars are extracted together with the entry set chain pillars in retreat. The pillar line can be flat or angled; the method is only suitable for continuous mining.

Kauffman, Hawkins, and Thompson (1981) consider the advantages and disadvantages of each of these methods related to the more desirable features of room and pillar mining, and these are worth repeating as they highlight the fundamental principle of this type of mining. Desirable features are listed below, and the methods not conforming are mentioned.

1. Active working places should not be near a caved area, since the increased pressures associated with caving increase the likelihood of roof falls. This is a drawback in the case of methods 3 and 4 above.

2. The length of time that openings are maintained should be a minimum. The loosening of roof bolts referred to above and exposure of roof and pillar sides to oxidation and moisture will cause deterioration. The exposure time is largest in the case of methods 1 and 2.

3. Ideally, solid coal should be retained on at least one side of the panel entry to reduce pressures on chain pillars during advance development. This is not the case in method 4.

4. Work places should be concentrated in a limited area. This reduces the area of direct supervision and improves management of the operation. This is not the case in method 2.

5. The tonnage produced between take-ups of belts and services should be maximized, and haul distances should be minimized to reduce nonproductive time. Arguably this is lowest in method 2, highest in 1, 3, and 4.

6. The ventilation system should operate with the minimum number of diversions during mining. The most difficult method to ventilate is method 4.

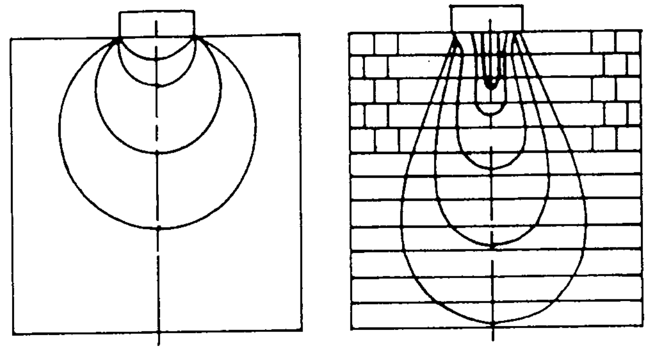
7. The bleeder system should be easy to establish and maintain in order to reduce ventilation. This is most difficult in the case of method 4.

8. The maximum amount of reserves should be recovered. Ore or coal left in the panel is lost and reduces the overall economics of mining. This is obviously a drawback with method 1.

18.1.4.3 Multiple Layer Room and Pillar Mines

A type of pillar mining that is common but not widely discussed is multiple-layer pillar mining where close vertical separation of pillars may lead to stability problems in roofs and floors. The applied mechanics approach to design is considered by Obert and Duvall (1967), and the main factors can also be identified from Figs. 18.1.5 and 18.1.6.

The main design approach must be to reduce stress concentrations in the roof. It is therefore logical to position pillars above pillars since the lower pillar will provide the better support for the upper pillar. Similarly the rock thickness between the mined layers must be sufficient to avoid excessive stress concentrations. This will depend on local conditions, but it can be seen from Fig. 18.1.5a that in the case of a rectangular excavation a roof thickness of twice the room height would be advisable.



(a) (b)

Fig. 18.1.9. Increase in major principal stress beneath a pillar in (a) homogeneous rock and (b) stratified rock. (After Gaziev and Erlikhman, 1971.)

Peng (1986) considers the particular problem in some detail, using the approach devised by Gaziev and Erlikman (1971) who demonstrated, using photoelastic models, the effect that layers of increasing or different modulus could have on the stress distribution beneath a foundation element (Fig. 18.1.9). The unavoidably high stress concentrations under pillars leads to Peng's particular recommendations for multiseam room and pillar mining:

1. *The upper seam is mined out prior to mining the lower seam.* High abutment pressure under upper seam pillars and abutments is the interaction problem most likely to be encountered in the lower seam. The design guidelines applicable to these conditions are (a) no pillars should be left unmined in the upper seam, (b) small pillars should be left in the upper seam if partial extraction is practiced, (c) pillars in the upper and lower seams should be columnized, (d) entries should not be driven under high stress zones such as abutment zones, and (e) longwalling might be the best alternative for the lower seam if pillaring is practiced in the upper seam with a few remnant pillars left.

2. *The lower seam is mined out prior to mining the upper seam.* Subsidence will be the most troublesome interaction effect. Caving induced by the lower seam mining might disrupt mining operations in the upper seam if seam separation is small. The design guidelines applicable to these conditions are (a) do not drive entries in the tensile zone of the subsidence trough, (b) reduce subsidence or arching effects by reducing opening width and extraction ratio, (c) columnize pillars, and (d) backfill the lower seam.

3. *Mining of the upper and lower seams is carried out simultaneously* with development and pillaring being kept in advance in the upper seam. Possible interaction problems are pillar stress concentrations. The design guidelines applicable to these conditions are (a) columnize pillars, and (b) keep the face of the upper seam ahead of the lower seam face by a minimum distance equal to the product of interburden thickness and the angle of draw.

18.1.4.4 Yielding Pillars

A major concept in pillar mining—although it has greater application in chain pillar design for longwall mining—is that

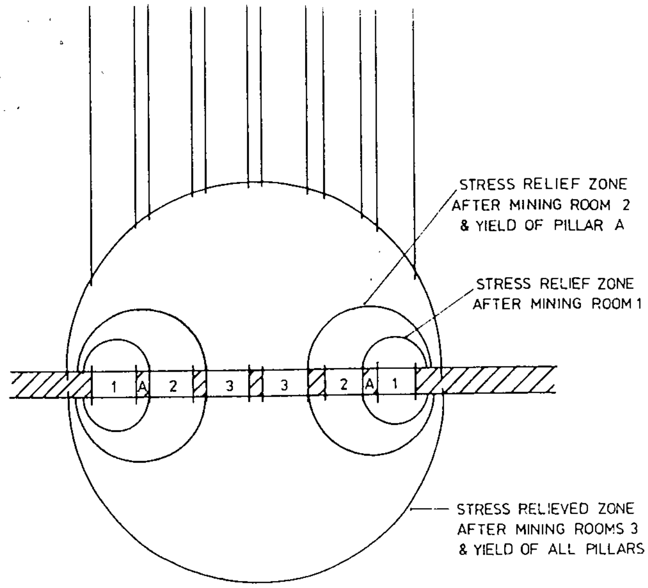


Fig. 18.1.10. A yield pillar layout for a six entry system, illustrating the development of a stress-relieved zone.

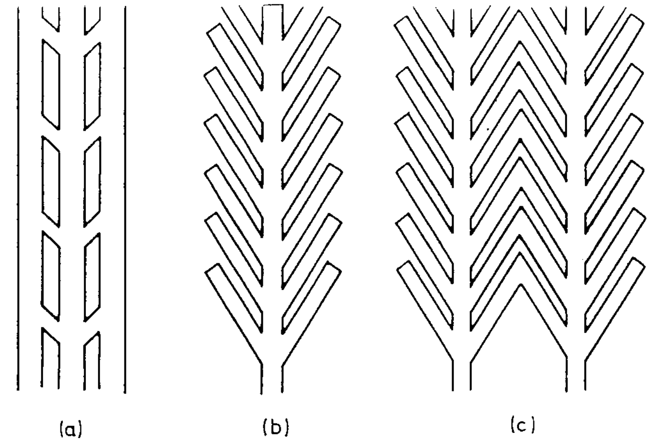


Fig. 18.1.11. Rapid development or time-control layouts used to obtain high productivity in weak deposits at depth: (a) 3-room yield pillar, (b) Christmas tree, and (c) chevron. (After Serata, 1984.)

of yield pillars. A major application has been in deep potash mines, but it is important in any application where a combination of stress and rock conditions can leads to bumps, bursts or excessive deformation.

Yield pillars are pillars that are designed to yield as soon as they are isolated, so that they transfer most of their overburden pressure to the abutment pillars of the panel. This prevents the buildup of high roof and floor pressures at the edges of the pillars at the center of panel, and should ensure improved roof conditions in most rooms at the expense of the outer rooms. The detailed mechanics of yield pillar design are explained by Serata (1983), although the method has been used—often in an ad hoc way—for many years.

Fig. 18.1.10 illustrates a typical layout for a six-entry system. The outer entries are driven first, as rapidly as possible, and the adjacent entries immediately afterwards, leaving a yield pillar. Yielding of this pillar should concentrate stresses in the abutment pillar, creating a pressure arch that will lower the vertical stresses on the remainder of the panel while damaging the outer room and abutment edge. The inner entries can then be driven in stress-relieved ground. Pillar extraction, by outside lifting (see 18.1.6.2) from the four protected rooms can then be used to complete the mining process. With suitable ground conditions, this method can be adapted to a greater or lesser number of entries. Even where pillar extraction is not considered desirable or feasible, use of the yield pillar approach allows a much higher rate of extraction than conventional tributary area design, and reduces the likelihood of bumps, bursts, and other roof falls.

An alternative approach to high extraction, used in salt, potash, and trona deposits and sometimes called the time-control technique (Serata 1983), involves rapid single-, double-, or triple-entry extraction using a "Christmas tree" or chevron approach (Fig. 18.1.11). This is designed for use in weak ground, and the objective is to excavate as much ore as possible very rapidly in a controlled way, using secondary yielding pillars to protect the central access entry, and using as little support as possible over a short time period. This method is not feasible in coal mines but has been very successful in deep potash deposits where 50% extraction ratios are possible.

18.1.5 PRODUCTION METHODS—NONCOAL

18.1.5.1 Production Cycle

It is necessary to differentiate between coal and noncoal production methods. This has been done very ably by Bullock (1982 b,c,d), utilizing a US Bureau of Mines-commissioned report by Dravo Corporation (Anon., 1974) on noncoal mining and an EPRI report by Hittman Associates (Anon., 1976) on coal mining. The difference arises from three main factors:

1. *Strength,* which means that the weaker coal can usually be cut by continuous miners.
2. *Scale,* where US coal seams are generally thinner than noncoal deposits.
3. *Gas,* where coal mines are gassy and noncoal mines are usually gas free. Thus noncoal mines are usually mined by drilling and blasting off the solid in large working excavations; coal seams are undercut and blasted or continuously mined in relatively small excavations.

There are three basic types of room and pillar mining cycles, which are illustrated as flow diagrams and element interaction bar charts in Fig. 18.1.12. For *hard-rock ore bodies,* the basic cycle (Fig. 18.1.12a) is similar to hard-rock tunneling with four main elements: (1) mark out and drill blastholes, usually in a wedge pattern; (2) charge, blast, and ventilate to remove blast fumes; (3) introduce mucker and muck and load; and (4) scale the face and walls and bolt the roof where necessary. There is considerable complexity in the interaction among these elements that make up a basic critical path. In order to estimate the cycle time, it is necessary to determine unit loading and drilling rates and task times for these elements and also to estimate how subsidiary elements and tasks such as haulage and ventilation takeup may impinge upon the critical path in a badly organized mine.

18.1.5.2 Panel Development

A panel layout for a typical room and pillar mine in a noncoal mine is illustrated in Fig. 18.2.3 (see Chapter 18.2). The

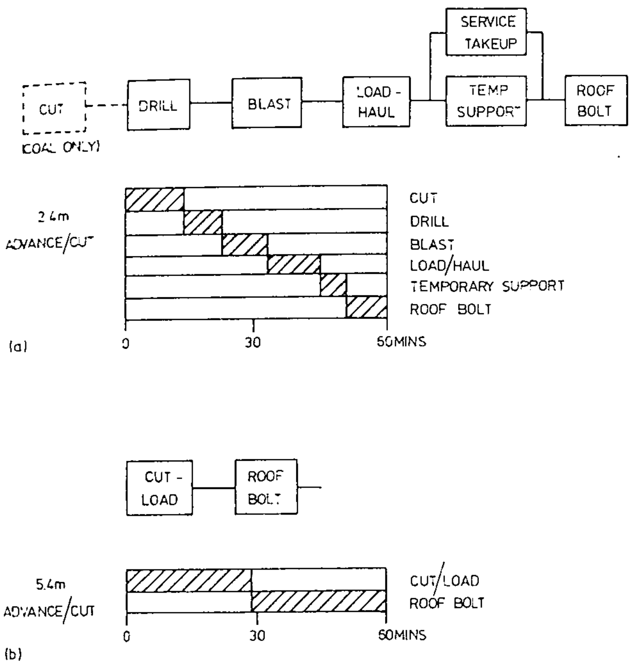


Fig. 18.1.12. Flow diagrams and element interaction bar chart for (a) conventional room and pillar and (b) continuous mining. Conversion factor: 1 ft = 0.3048 m.

excavation height is about 15 ft (4.5 m), and the normal stoping practice is to drive a single development drift about 35 ft (10.5 m) wide a distance of about four or five rooms into the ore body. This will serve as the main haulage drift. Pillars are then marked out on the drift walls and rooms driven between them.

To drill and blast the initial drive when the only exposed or free face is the drive face, some form of cut pattern is used. This is known as the "face round" or "swing" and in a 15- by 35-ft (4.5- by 10.5-m) face will comprise 60 to 70 holes (see Chapter 9.2) of about 1½ in. (38 mm) to a depth of 10 to 12 ft (3 to 3.6 m). If more than one face is exposed, a group of holes may be drilled at a low angle to the free face in what is known as a "slab round" or "slabbing" or "slashing". This requires less explosive and less drilling than a single face. The most common form of face round is a wedge or V, cut although burn cuts can also be used.

Drilling is carried out with jumbo-mounted hydraulic drills; loading is usually by gathering arm loader, although in modern mines, trackless LHD vehicles are used to carry the load to a transfer raise where it is reloaded into trucks or conveyors. Some typical productivity figures for this type of operation are given in Table 18.1.4.

18.1.5.3 Cut and Fill Pillar Mining

Where the roof can be caved, as in coal mining, high levels of extraction can be obtained by retreat mining. Where the roof is stronger—as in most non-coal mining—the pillars are normally left as semi-permanent support. In high-grade ores at depth or where roof conditions are poor, loss of from 25 to 50% of the ore body may be unacceptable and, in this case, backfill may be considered. Placing of backfill can most easily be carried out using a form of slot and pillar rather than room and pillar arrangement at a gold mine in Washington, designed to reduce subsidence in this case, is described by Tesaric, Seymour, and Vickery (1989) and Brechtel (1987). Slots or rooms (Fig. 18.1.13a) were excavated in 50-ft (15-m) vertical intervals by multiple benches 24 ft (7.3 m) wide and 24 ft (7.3 m) high. They were mined and filled in an alternating sequence from footwall drifts. The cemented fill was dumped from dumper trucks and allowed to settle at its angle of repose. At the top of the ore block, the backfill was rammed tight using a plate mounted on a LHD. It is a relatively simple system of mining that can be adapted for any room and pillar configuration. Completed stopes range in height from 30 to 130 ft (9 to 40 m), depending on their location in the ore zone. The fill comprised 55% minus 2-in. (50-m) river gravel, 40% alluvial sand, and 5% cement. A more useful mix might utilize tailings, which are often pozzolanic and require little or no cement.

A very radical approach to backfill pillar mining has been suggested by Dixon (1990). Called spiral slot and chamber mining, it is presented as a total extraction method for strata-bound deposits in a horizontal plane. The ore body is mined (Fig. 18.1.13b) in a continuous, flat, but not necessarily circular, concentric spiral pattern. There are three operations—top heading, benching, and backfilling from radial crosscuts with a chamber width of 30 ft (9 m)—the initial slot spiral being followed by a chamber spiral. The slot is backfilled with cemented fill the chamber with mine tailings or sand. Several potential benefits are claimed. The pattern should induce more even and favorable stress distributions than conventional layouts and should be more amenable to automation, leading to reductions in bursts, better strata control, and improved productivity.

One problem with backfilling, apart from the major logistical one, is the cost of cement, and in some cases its availability. Mitchell (1989) suggests using geogrid reinforcements as an alternative, and this is probably feasible. However, most silicates have some pozzolanic properties and it may be that added cementing agents, particularly in bulk fills, are unnecessary.

18.1.6 PRODUCTION METHODS—COAL

18.1.6.1 Panel Development

In coal mining, blasting off the solid is illegal, principally because of the danger associated with blown-out shots in an environment where explosive gases may be present. Where blasting is used, a horizontal cut is formed, generally in the face. This is usually a bottom cut in thin seams, a center cut in thicker seams. A vertical center cut may also be used. A similar approach may be used in rock salt and potash mining. The presence of the cut creates a free face for blasting and reduces both the amount of explosive needed and the possibility of blow-outs. A typical cutter jib is 9 to 12 ft (2.7 to 3.6 m) long, and the picks are arranged to cut a 6-in. (150-mm) slot or kerf. The cutter jib is sumped into the center of the face and moved to each side to complete the undercut. The basic cycle of operations (Fig. 18.1.12a) thus requires one more element before drilling.

Cyclic systems are usually referred to as *conventional room and pillar mining.* Much more productive and much commoner in mechanized mines is *continuous mining.* This is particularly important because, as the name implies, it reduces the number of unit operations and hence the cyclic element (Fig. 18.1.12c). There are several types of continuous miner, but they all combine the same basic elements, a cutting head above, or combined

Table 18.1.4. Typical Productivity of Noncoal Room and Pillar Mines (1970s Data)

Type of Rock	Location	No. of Mines in Sample	Range of Production 1000 tonnes/ day	Range of Productivity tonnes/ employee-shift	Average Productivity tonnes/ employee-shift
Dolomite (Lead/Zinc)	New Lead Belt, MO	8	1.6–4.5	19.2–59.0	32.6
Cherty Limestone (Zinc)	E. Tennessee	5	1.4–3.1	12.5–26.6	17.8
Shale & Sandstone (Copper)	N. Michigan	1	6.2	—	14.6
Limestone	Various	45	1.1–8.7	31.8–136.1	74.5
Phosphate	Utah	1	2.2	—	52.2
Shale & Potash	N. Mexico	6	3.6–8.3	22.0–75.0	42.8
Salt	Various	10	1.6–10.8	14.3–63.6	36.9
Sandstone	Pennsylvania	1	2.4	—	73.7
Salt (Trona)	Wyoming	3	3.2–5.6	27.9–63.1	41.1

Conversion factor: 1 ton = 0.9072 t.
Source: Bullock, 1982a.

ventilation and roof support. The reduction in the number of unit operations means that for efficient operations, a much smaller number of faces can be worked continuously. This is illustrated in Fig. 18.1.14, which shows typical plans for (a) conventional development of a six-pillar, seven-entry room and pillar panel, developed as in Fig. 18.1.8b as full panel, and (b) continuous miner development of a four-pillar, five-entry set for rapid development of a panel (see Fig. 18.1.17c or d).

In Fig. 18.1.14a, the conventional method, initial development is on 20-ft (6-m) rooms with 60- by 50-ft (18- by 15-m) pillars. Twenty feet (6 m) is the maximum room width under the Federal Coal Mine Health and Safety Act. The advance per cut is planned to be 10 ft (3 m). Unit operations have been described in detail by both Stefanko (1983) and Bullock (1982). The basic elements of the plan can be seen and can be described in terms of the working cycle in Fig. 18.2.3. Cut 1 (entry 7) is being loaded; entry 1 has just been loaded, and the loader, typically a gathering arm loader with integral armored flight conveyor on caterpillar tracks, has moved from this entry to cut 1. The blasted material is loaded into a rubber-tired shuttle car, which transports it to the feeder belt where it is dumped. There are usually two shuttle cars, and, depending on whether they are cable reel electric or diesel, they follow the same or separate paths. Cut 2 (entry 6) is being charged and prepared for blasting. Cut 4 (entry 4) is being drilled, and cut 5 (entry 3) is being undercut. Cut 6 (entry 2), after loading is being bolted, and cut 7 (entry 1) is being prepared for bolting and other service move-ups. The continuing sequence following this cycle can be seen in the numbered cuts in Fig. 18.1.14a.

The efficiency of the operation and the overall productivities and advance per shift depend on the time taken for each of the elements in the cycle, the way in which they interact, and the speed with which equipment can be moved from one entry to the next. Typical times, from the same sources as Fig. 18.1.14a, are:

Cutting cycle	11–16 min	2 employees
Drilling cycle	8–12 min	1 employee
Blasting cycle (6–8 holes)	8–12 min	1 employee
Load/haul cycle	12–20 min	3 employees
Temporary support cycle	5–10 min	1 employee
Roof bolt cycle	8–15 min	2 employees

Since these cycles are concurrent, the overall cycle time will be the longest part of the cycle, probably the load/haul cycle, which is discussed later. Ideally, it should be possible to complete a cycle in 12 to 20 minutes and to complete 24 to 40 cycles in each 8-hour shift. This may be compared with Table 18.1.5, which includes actual data. Bearing in mind the time lost in shift changes, traveling, and breakdowns, the actual achievement is remarkably good.

Continuous mining is noncyclic and utilizes a smaller crew, and the results (Table 18.1.5) are better in comparable situations. The major advantage is that the reduction in cycle time reduces the number of entries that need to be driven in order to maintain output to 3 or 4, since the only separate operations needed are cutting and roof support. Where these can be combined, and shuttle cars eliminated by extensible conveyors, then 100 ft (30 m) of continuous driving can be obtained to isolate a pillar side before moving the machine. The method then becomes very productive.

18.1.6.2 Methods of Pillar Extraction

Four basic pillaring methods are described in detail by Kauffman, Hawkins, and Thompson (1981). These are split and fender, pocket and wing, outside lift, and open ending. Each of these methods is illustrated in Fig. 18.1.15, and their basic characteristics are summarized in the following.

1. *Split and fender* (Fig. 18.1.15a) is the most commonly used pillar extraction method in the United States. The basis of

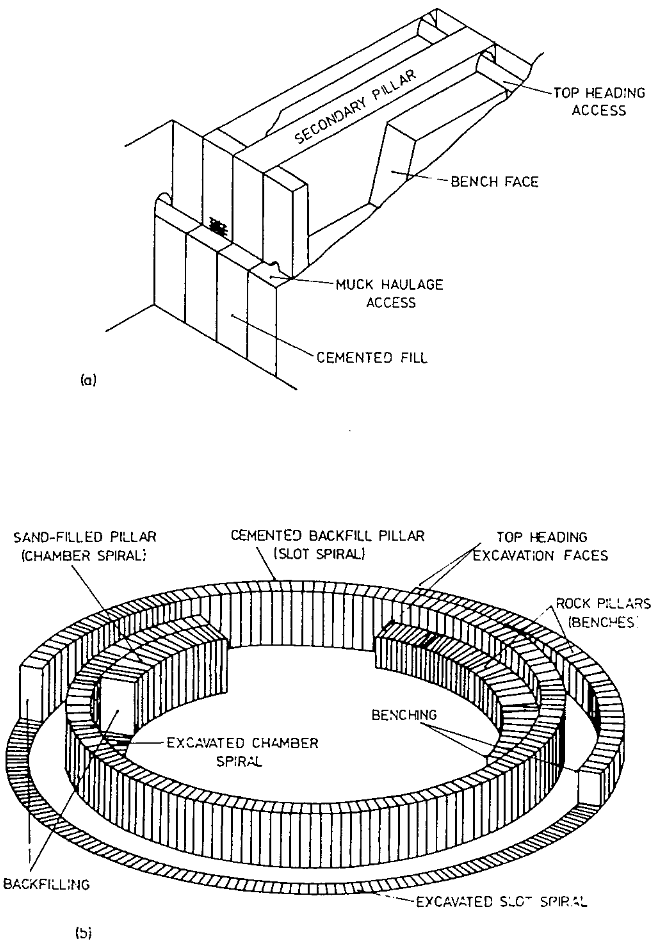


Fig. 18.1.13. (a) Slot and fill mine layout (after Brechtel, 1987). (b) Spiral slot and fill mine layout (after Dixon, 1990).

the method is to mine through the pillar center parallel to the longer side, creating a split and a fender of coal on each side of the split. Before mining, breaker posts are placed at all openings to the gob, and roadway posts are placed to reduce roadway widths to 16 ft (4.8 m). Turn posts and breaker posts are used in the split for additional support, and roof bolts are installed, as in room development, to support exposed roofs. The split is the same dimension as the rooms in the original panel, and the width of the fenders is usually fixed so they can be wholly extracted by the continuous miners without additional support. This effectively determines the maximum pillar width. Under most conditions, the minimum fender width is 8 ft (2.4 m), and the maximum about 13 ft (3.9 m). The split width can range from 10 ft (3 m) to 20 ft (6 m), giving a range of pillar widths from 26 ft (7.9 m) to 46 ft (14 m). Wider pillars can be extracted using multiple splits, but this reduces the simplicity, and the pocket and wing technique is more suitable. There is no limit on pillar length. The method usually involves mining two or more pillars simultaneously. Fig. 18.1.15a illustrates a double pillar sequence with 1 to 7 and 16 being split operations, the remainder beneath supported roof. Ventilation is difficult, involving quite complex brattice curtain erections at critical points. The process is, however, simple and can be adapted to all thicknesses from 40 in. (1 m) to 25 ft (7.5 m) and to all equipment from simple loaders to continuous miners. The method is generally not suitable for large pillars and fragile roofs.

2. *Pocket and wing* (or pocket and fender, Fig. 18.1.15b) is a single pillar extraction method used mainly in northern West Virginia. Two working places are extended in the pillar leaving wings or fenders to support the roof. It can be easily adapted to large pillars and allows concentration of working places in a pillar, and hence, rapid extraction. Ventilation and haulage are also easier. It is not as efficient as the split and fender method and is used primarily where mining at depth requires large pillars for roof control. The method is not suitable in bump conditions.

3. *Open ending* (Fig. 18.1.15c) is a method similar to pocket and wing, but the mining sequence is taken along the sides of the pillars, breaker posts being extended at the pillar edge. It has limited use; ideally, the roof should be competent enough to span the opening, but brittle enough to break off or cave beyond the breaker posts.

4. *Outside lifts* (Fig. 18.1.15d) are rarely used except to extract narrow pillars rather like the fenders of the split and fender method. The method is used in shallow mines that allow the safe use of small pillars. A variation is in deep pillar extraction, particularly in bump prone areas where residual small yielding pillars are desired. Such a plan permits rapid extraction. The method can also be used for ad hoc partial extraction of pillars where bolting is not needed.

18.1.6.3 Mobile Roof Support (MRS)

Retreat pillar mining is highly productive. Supply, power, haulage, and ventilation systems are established during panel development and knowledge of roof and water conditions obtained. It is also dangerous, particularly where the roof does not cave in a predictable manner, and where the seam is prone to bursts, floor heave, and crushed pillars. The prime factor in improving safety is successful roof control through correct design of pillars, including yield pillars and supports such as posts, cribs, and roof bolts. These supports have the disadvantage that they act in a passive way. Technology from longwall mining, where active waste edge shields give an added dimension to roof control, was urgently needed, and this has been supplied through mobile roof supports (Thompson, 1983) developed initially by the US Bureau of Mines and improved by Fletcher (1990).

These are approximately 14 ft (4.2 m) by 6 ft (1.9 m) wide. Specific design features (Fig. 18.1.16) include continuous miner-type crawler drives with variable speed motors, hydraulically operated plows, 600-ton (540-t) total load capacity (through four cylinders each with 150-ton, or 135-t, yield load capacity), lemmiscate-linked canopy with free-floating support cylinders, and a heavy-duty rear shield. The supports are self contained apart from a cable reel electric power source, and are operated through a hand-held radio remote controller.

The mobile roof supports are typically used in two-pair configurations (Fig. 18.1.17), each pair being located between a solid pillar and the pillar being extracted. After each miner cut, they are advanced one unit at a time until there is just enough space remaining for the miner at the last cut.

18.1.7 VENTILATION

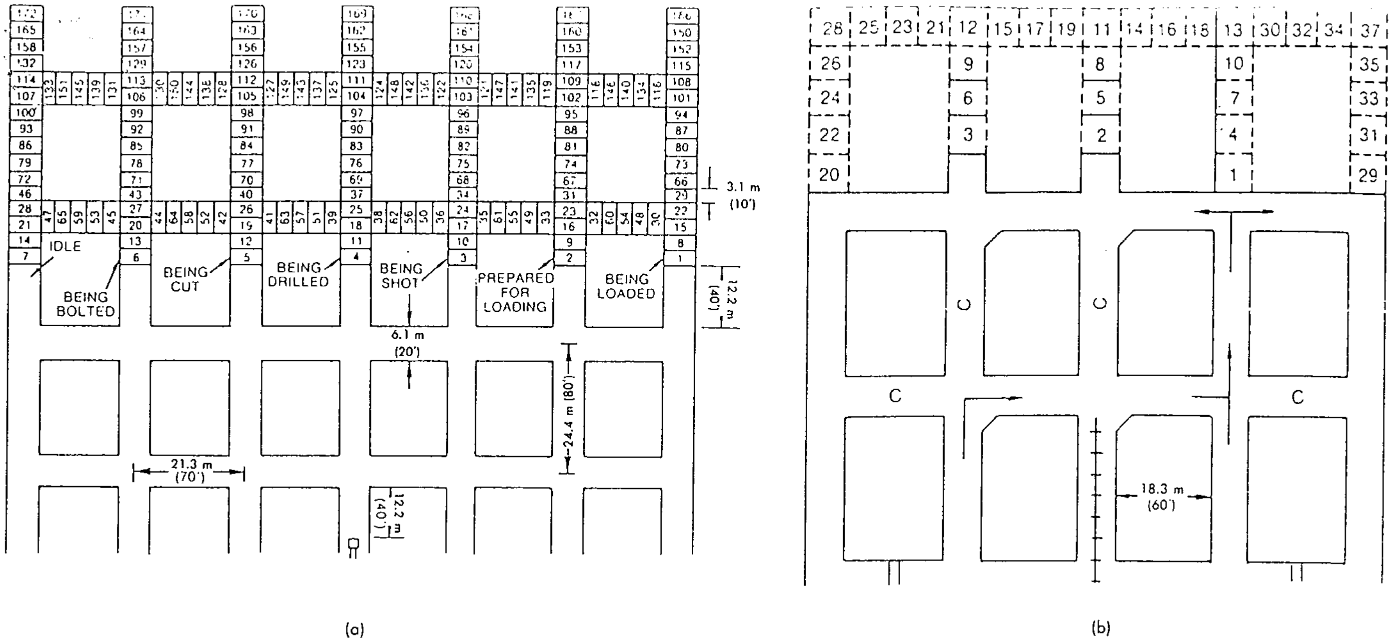


Fig. 18.1.14. (a) United operations in conventional room and pillar mining, showing the cut sequence of a seven-entry plan. (b) Mining sequence for a five-entry continuous mining operation (after Anon., 1976).

Table 18.1.5. Comparisons of Productivity Estimates for Conventional (A) and Continuous Miner (B) Room and Pillar Mining

Production Data	Conv. A	Cont. B	Conv. A	Cont. B	Cont. A	Cont. B
Seam height, ft (m)	4 (1.2)	4 (1.2)	5 (1.5)	5 (1.5)	6 (1.8)	6 (1.8)
Width, ft (m)	20 (6.1)	20 (6.1)	20 (6.1)	20 (6.1)	20 (6.1)	20 (6.1)
Depth of cut, ft (m)	8 (2.4)	18 (5.4)	8 (2.4)	18 (5.4)	8 (2.4)	18 (5.4)
Tons/cut	23	53	29	65	34	78
Cuts/shift	20	8.7	20	8.3	18	8.2
Tons/shift	464	454	581	544	617	635
Work min/shift	400	400	400	400	400	400
Face Crew Employee (mins/cut)						
Continuous miner	—	1 (31)	—	1 (33)	—	1 (33.5)
Loading machine	2 (30)	1 (31)	1 (30)	1 (33)	2 (34)	1 (33.5)
Shuttle cars	2 (30)	2 (62)	2 (30)	2 (66)	2 (34)	2 (67)
Cutting machine	2 (30)	—	2 (30)	—	2 (34)	—
Drill	1 (13)	—	1 (13)	—	1 (15)	—
Shooting	1 (17)	—	1 (17)	—	1 (20)	—
Roof bolt machine	2 (30)	2 (72)	2 (30)	2 (72)	2 (30)	2 (76)
TOTAL FACE CREW	10 (150)	6 (196)	10 (150)	6 (204)	10 (168)	6 (210)
Employee min/shift at the face	3000	1700	3000	1700	3024	1720
Employee min/ton	5.32	3.07	4.25	2.59	4.01	2.21
Tons/employee-shift	46.4	75.6	58.1	90.7	61.7	105.8

Conversion factor: 1 long ton = 0.996 t.
Source: Anon., 1977.

ventilation of room and pillar mines. A major provision is the requirement for bleeder entries and systems. *Bleeders* (Kauffman, Hawkins and Thompson, 1981) are entries surrounding an area being mined or which has been mined out. The purpose of bleeder entries is to bleed methane and other explosive gases from the gob area and into the main mine return airways, using a controlled filter of intake air. Bleeder entries should be maintained for access and examination. Only in areas liable to spontaneous combustion is sealing of caved areas permitted.

The practice of bleeding requires that a differential air pressure be maintained between the intake and return airways across a gob so that gases flow into the return airway. Where there

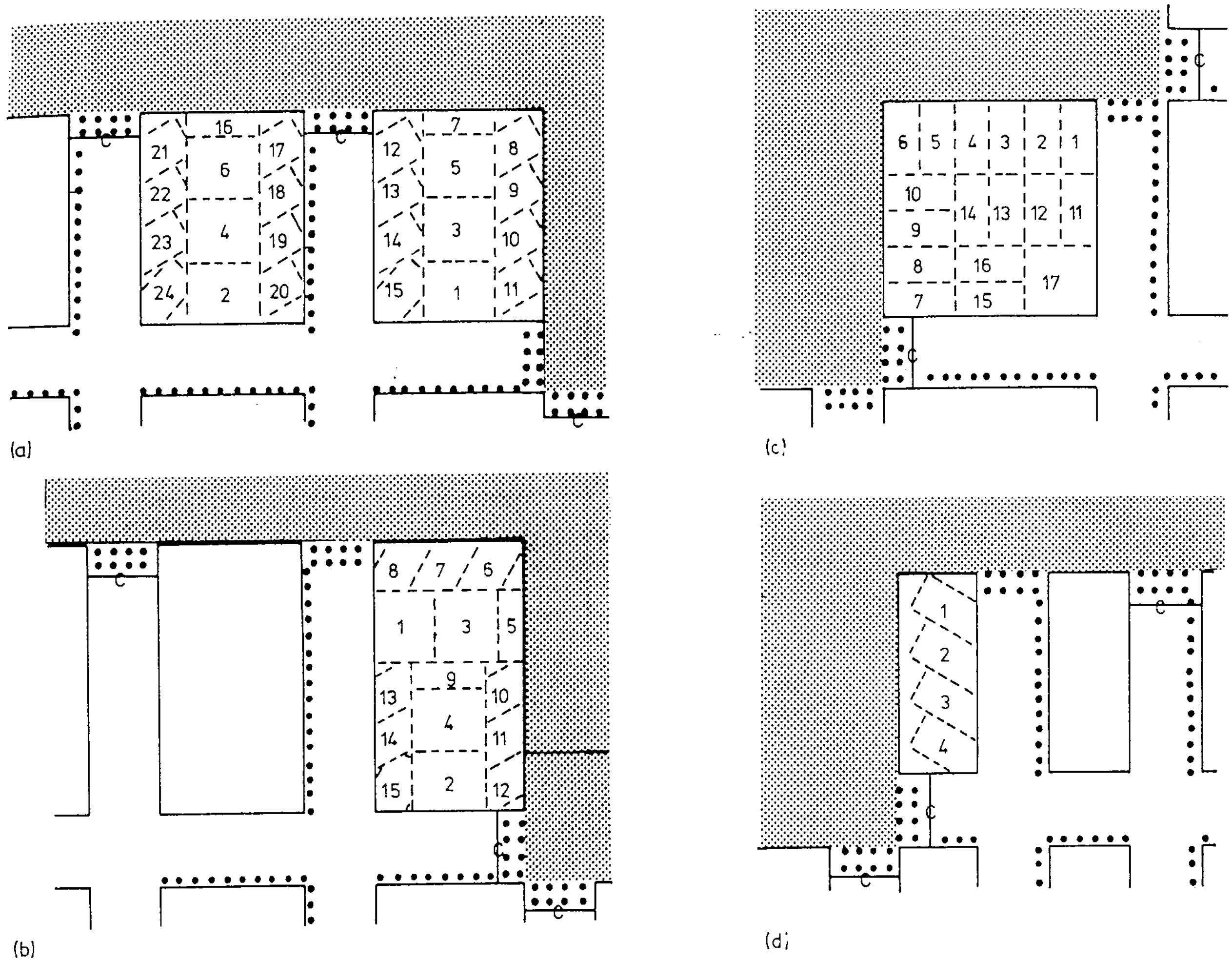


Fig. 18.1.15. Overall cut sequence for pillar extraction during retreat mining: (a) split-and-fender method, (b) pocket-and-wing or pocket-and-fender method, (c) open ending method, and (d) outside lift method (after Kauffman, Hawkins, and Thompson, 1981).

has been pillar extraction beneath a blocky roof, there will be sufficient flow through the caved area to bleed. A laminated roof may seal the caved area and an entry through the gob may be needed.

In order to simplify bleeding, it is sometimes desirable to remove barrier pillars during retreating to connect panels and to allow bleeding of an extended area of the mine. During development, it is better practice to bleed individual panels.

For details of mine ventilation theory and practice, see Chapters 11.6 and 11.7.

18.1.7.2 Section Ventilation

As discussed in Chapter 11.7.2, each working section of a coal mine should be ventilated with a minimum of 9000 cfm (4.25 m^3/s) of air to the last open crosscut. At least 3000 cfm (1.42 m^3/s) must reach each working face where coal is being mined. The air must contain more than 19.5% oxygen and less than 0.5% carbon dioxide.

During development, air going into the section is directed to the face by means of curtains across entries, line curtains, or

Blower or exhaust fans in coal mines must be capable of delivering or exhausting 3000 cfm to or from the working face. Exhaust fans have the advantage that they can remove dust, fumes, and gas from the working area more efficiently than blower fans, particularly if the tubing is close to the continuous miner head.

A typical exhaust ventilation layout for a coal room and pillar system is illustrated in Fig. 18.1.18. This is a system employing line brattice curtains and is described by Stefanko (1983). The line brattice is essentially a space divider or temporary partition made of an impervious material that is installed and maintained carefully and kept as close to the face as possible. Its purpose is to guide the airflow through the face area and last open crosscut and into the return. Brattices were formerly (and to some extent still are) made of untreated jute, but nylon-reinforced plastics and similar materials are more commonly used today.

The line brattice is installed so as to split the heading longitudinally and thus provide an inlet as well as a return from the face to the last open crosscut. Since the mining machine must

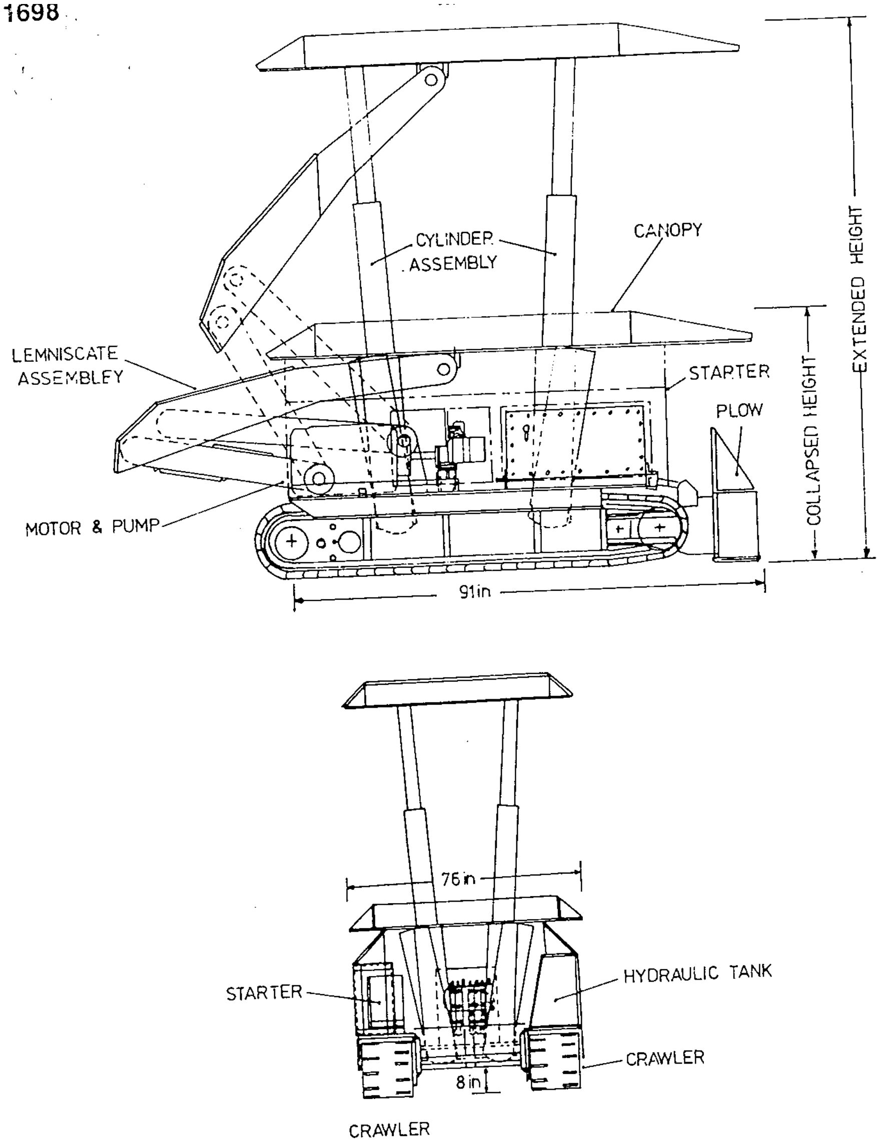


Fig. 18.1.16. Design features of the mobile roof support. Courtesy of Fletcher Mining Equipment Co. Conversion factor: 1 in. = 25.4 mm.

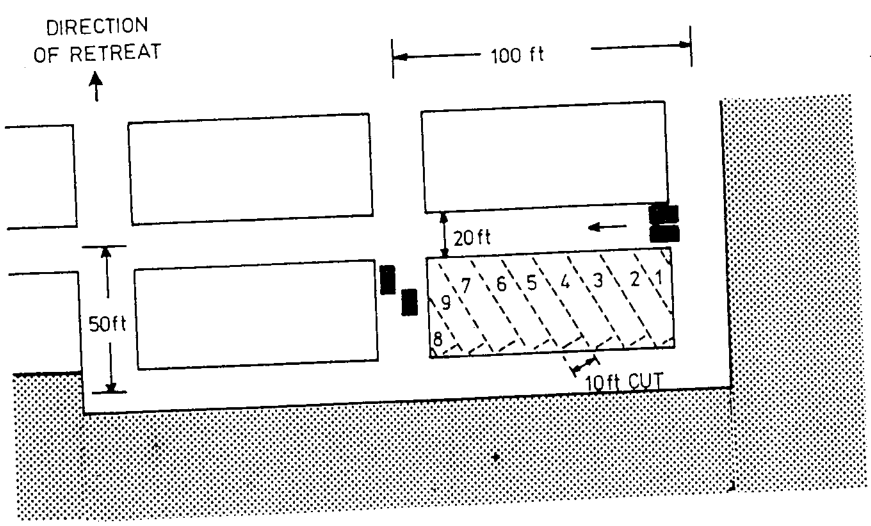


Fig. 18.1.17. A pillar extraction method using two pairs of mobile roof supports. Courtesy of Fletcher Mining Equipment Co. Conversion factor: 1 ft = 0.3048 m.

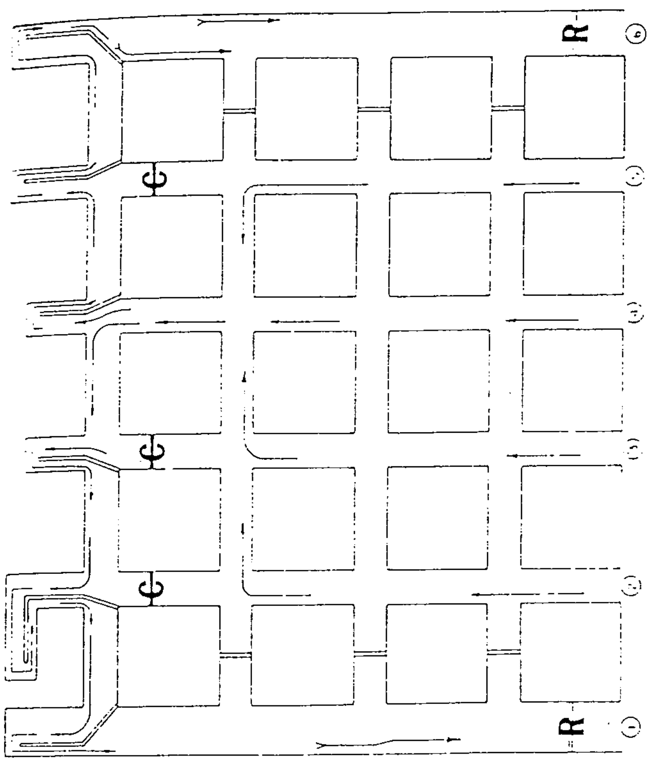


Fig. 18.1.18. An exhausting line brattice ventilation layout for a six-entry room and pillar panel (after Stefanko, 1983).

ments may be made to blow or force the air into the heading or to exhaust it. The exhaust system illustrated in Fig. 18.1.18 is more commonly used since the operators work in fresher dust-free air. Exhaust fans can be used to replace line brattices at the face, and these are effective in controlling dust.

18.1.8 SYSTEMS ANALYSIS

The room and pillar method is particularly suitable for simulation modeling and systems analysis. Particular approaches are recommended by Bullock (1982c) and Stefanko (1983), but the most sophisticated model is that of Manula and Suboleski (1982). In this *Handbook,* there is coverage of systems analysis in Chapters 8.3 and 9.4.

Simulation modeling to optimize productivity requires a detailed analysis of the mining process and the way in which the mining variables interact with and affect the selection of mining methods and equipment. Mining variables include seam height, floor quality, roof quality, methane quantities, coal hardness, depth, and presence of water. Functional relations between these and unit operations are the basis—together with observations from underground studies—of planning and simulation. For instance, in a typical cycle, *cutting* and *continuous miner* operation will be affected by the seam thickness, floor quality, water presence, and particularly the strength of the coal. *Drilling* will be affected by coal hardness, because more holes will be required to break the coal, and *blasting* will similarly take longer if more holes are used. *Roof bolting* will be affected by roof quality, and *ventilation* by methane quantities. Layout and pillar size will be determined by depth, and this will affect loading and hauling variables apart from weak or wet floors. They are also the major factor affecting conventional and continuous miner cycle times. Manula and Suboleski (1982) and Bullock (1982c) illustrate how *total cycle times* may be predicted through a simple mathematical model. Thus loader cycle time LCT for a cut is given by:

$$\mathrm{LCT} = \mathrm{LT} + \mathrm{COT} + \mathrm{WSC} + \mathrm{MISC} \qquad (18.1.14)$$

where $\mathrm{LT} = \mathrm{T}/(\mathrm{LR})$

$\mathrm{COT} = (\mathrm{N} - 1)\,(2 \times \mathrm{COD}/\mathrm{SPD})$

$\mathrm{WSC} = [(\mathrm{N} - 1)/2]\,(2 \times \mathrm{HD}/\mathrm{SPD} + \mathrm{DT}) - (\mathrm{NO} - 1) \times (\mathrm{CAP}/\mathrm{LR}) + (2 \times \mathrm{COD}/\mathrm{SPD}) - (\mathrm{NO} - 1) \times (\mathrm{CAP}/\mathrm{LR}) + (2 \times \mathrm{COD}/\mathrm{SPD}) - (\mathrm{NO} - 1) \times (\mathrm{CAP}/\mathrm{LR}) + (2 \times \mathrm{COD}/\mathrm{SPD})$

$(\mathrm{N} = \mathrm{T}/\mathrm{CAP})$

and where LT is the time loader spends loading, COT is the time the loader spends waiting for cars to travel, WCS is the time the loader spends waiting for the car to arrive at the change point after the other car has cleared the change point, MISC is the time to check for connect and disconnect water hose, hand curtain, tram, etc., T is the weight of coal in the cut, LR is the mean loading rate, N is the number of shuttle car loads in the cut, COD is the change out distance one-way, SPD is the mean shuttle car speed, CAP is the mean shuttle car payload, HD is the distance from dump to change point, DT is the mean dump time of the shuttle car, NO is the number of shuttle cars in use (normally equals two), [ ] indicates truncation of the number to the next lowest integer, and { } indicates raising of number to next highest integer.

Analysis of Eq. 18.1.14 can give several indications about production improvement methods. They are relatively insensitive to loading rate increases alone since a decrease in LT is partially offset by an increase in WSC, all other factors remaining equal. Increasing the number of cars in use at one time will decrease WSC but will have no effect on COT (in practice, COT is often increased in the case of three cars, since one of the cars may be forced to change out farther from the face than the other two); and the greatest sensitivity is experienced with a change in shuttle car payload, since this affects N, which in turn affects the value of COT and WSC. The payload also affects WSC directly.

Time studies and simulations of room and pillar mining systems indicate that *change-out time* can represent from 15 to 25% of the available time for production. (This is defined as the shift time less travel, face preparation, scheduled meetings, breakdowns, lunch, servicing, etc.; that is, the time in which the units and men otherwise are actually capable of coal production.) In general, available time for production will range from 175 to 300 min/shift with an "average" value at 225 min. Thus, 30 to 60 min could be saved if suitable continuous haulage units were available. It must be recognized, however, that not all of this time will be additional loading time. In general, this time will be distributed proportionally among the remaining loading and hauling activities.

Additional time is lost in those cuts where the car cannot go back to the change point at or prior to the time it is cleared by the previous car. The maximum distance from the dump to the change point at which an additional wait will not be encountered can be calculated by balancing the load and change-out time with the haul and dump times:

$$\mathrm{WAIT} = 0 = (\mathrm{TTD} + \mathrm{D} + \mathrm{TFD})$$

where TI $= SE \times COD$
TO $= SL \times COD$

$$TTD = SL \times HD$$
$$TFD = SE \times HD$$

Then

$$(SL + SE) \times HD + D = (N - 1)(L + (SL + SE) \times COD)$$

or

$$HD_{max} = (N - 1) L - D + (N - 1)(COD) SL + SE$$

where L is loading time, TTD is the time to travel from the change point to the dump, TFD is the time to travel from the dump to the change point, D is mean dump time, N is number of haulage units in service, SE is mean travel time unloaded, SL is mean travel time loaded, TI is the changes out time unloaded, TO is the change-out time loaded, COD is change-out distance, and HD is the distance from dump to change point.

These are examples of simple approaches to planning. A fuller listing is given in Manula and Suboleski (1982), and elsewhere in this *Handbook*.

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