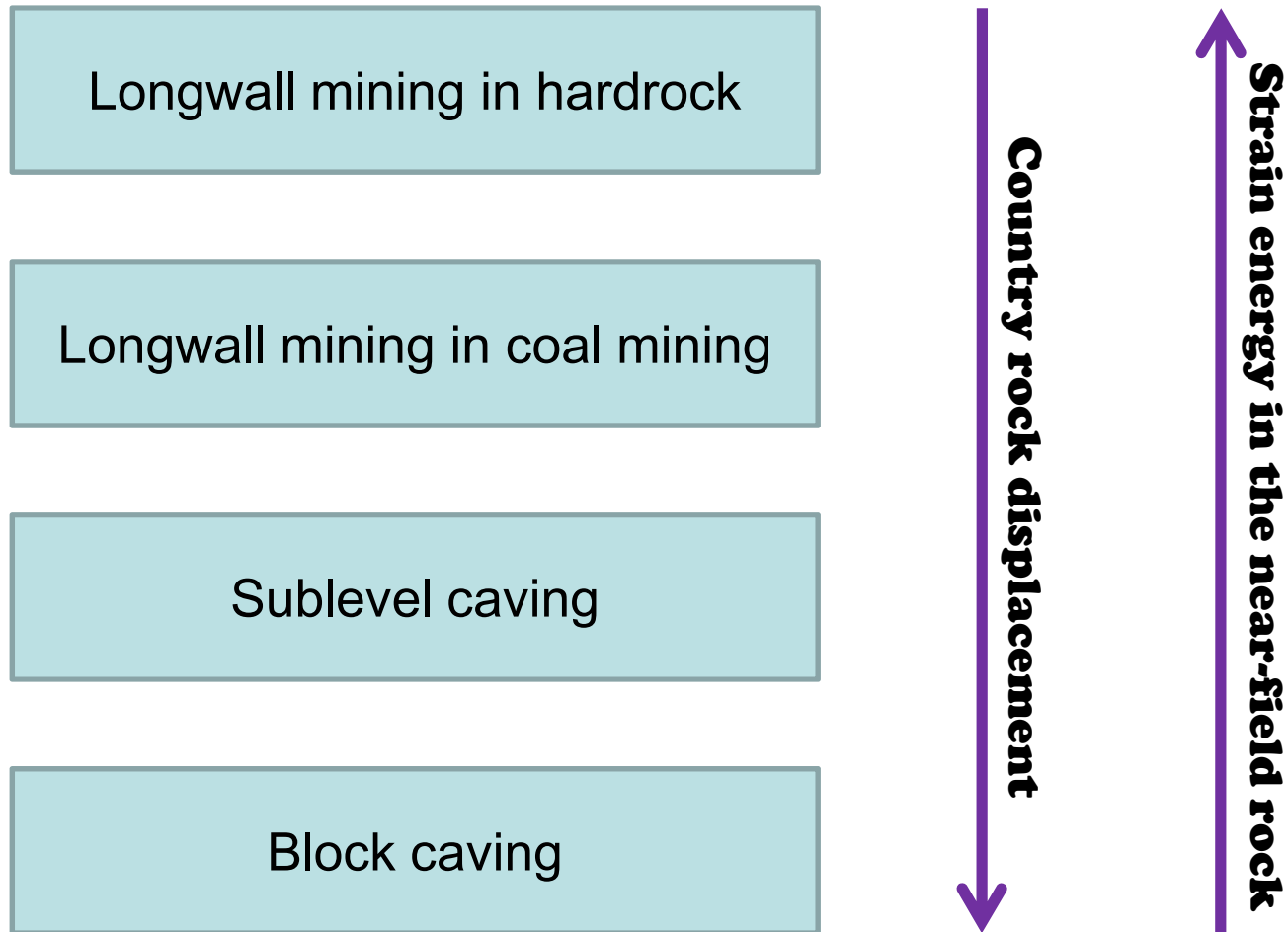


## **15. Longwall and caving mining methods**

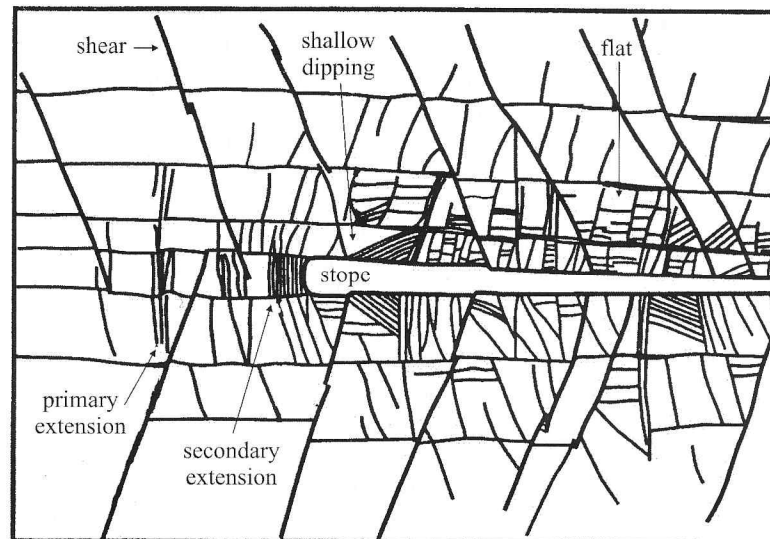
## 15.1 Classification of longwall and caving mining methods



## 15.2 Longwall mining in hard rock

### 15.2.1 Basic geomechanics considerations

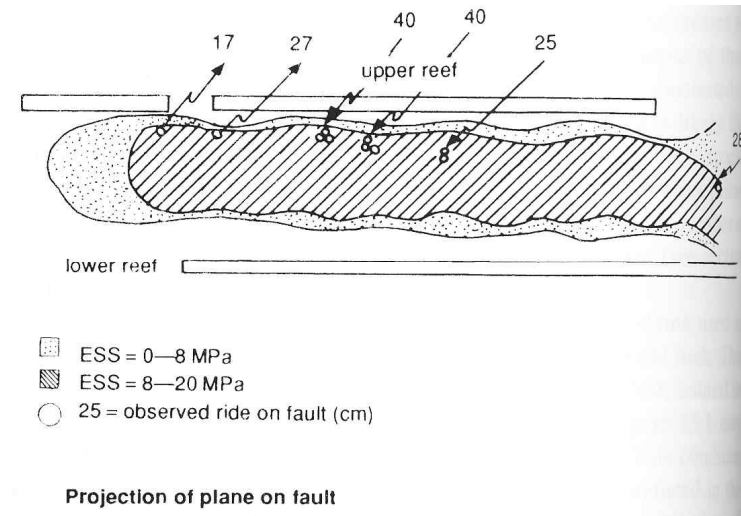
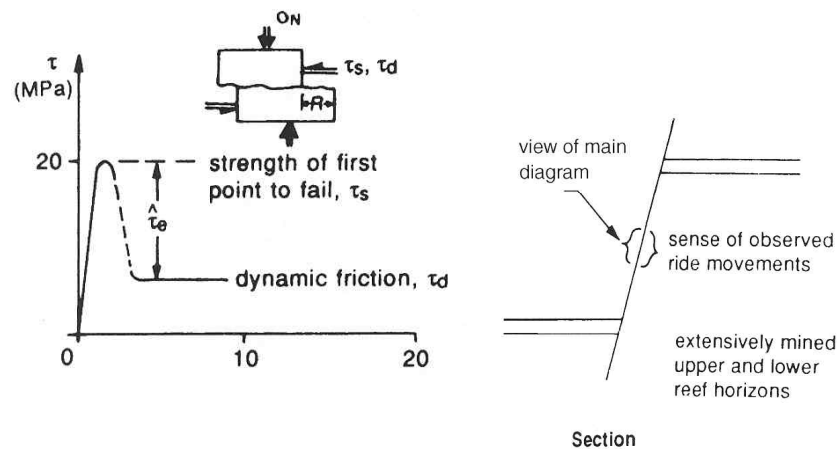
- Near-field rock is usually strong, and mining takes place at considerable depth.
- Pseudo-continuous behavior may be disturbed by two influences: natural or mining-induced discontinuities (figure below), and rockburst.
- Rockbursts are a kind of seismic events which may have damaging effects on the rock surrounding.
- Causes of rockburst: fault-slip (greater in size), and stress induced fracturing or crushing (as destructive as the fault-slip in an operating stope).



## 15.2 Longwall mining in hard rock

### 15.2.2 Rockburst controls

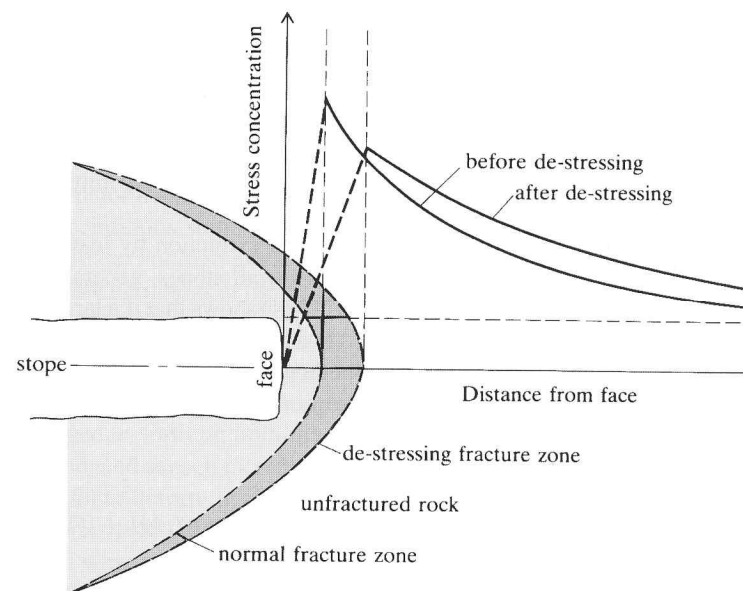
- Stress drop during fault slip is related with an excess shear stress (ESS):  
$$\tau_e = \tau_s - \tau_d \text{ (0.1 ~ 10 MPa, 5~10\% of the static shear strength)}$$
- Higher ESS causes larger shear displacements.
- General strategy of layout planning to control rockburst is to limit the energy release rate in space and time.



## 15.2 Longwall mining in hard rock

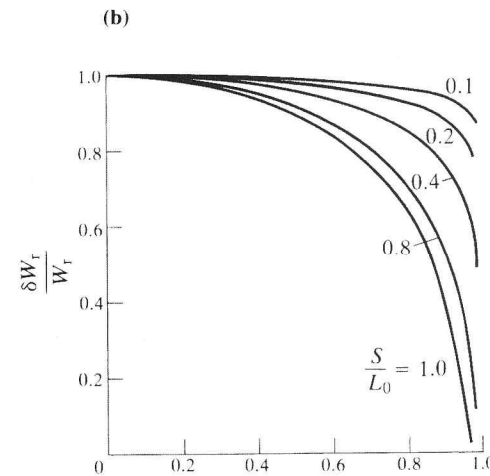
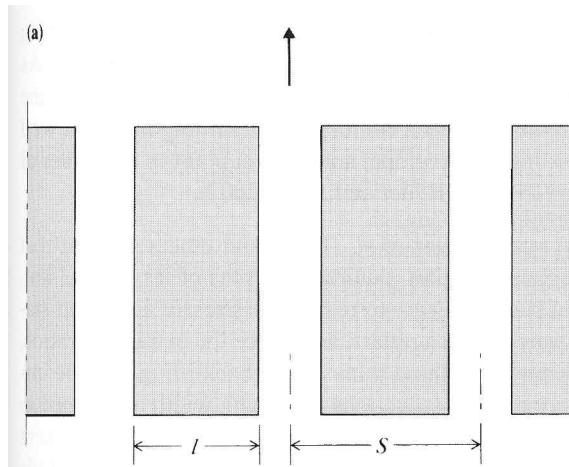
### 1) De-stressing

- An early attempt to limit the incidence of rockbursts around longwall faces.
- Fracturing the rock ahead of the face by blasting reduces the stress level in the rock.
- As mining progresses to deeper level, however, rock bursting continues.
- Goals of de-stress blasting is to reduce stiffness of rock mass and dissipate energy through fracturing /frictional sliding on internal surfaces (gouge & heat).



## 15.2 Longwall mining in hard rock

- 2) Limiting the displacement of peripheral rock: supporting
- Active support using hydraulic props with pack or stick support.
  - Leaving pillars along the entire length of longwall stopes: bigger pillars are required in deeper mines, which is the major disadvantage of this method.
  - Backfilling the mined void with tailings, sand or waste rock: high cost and operational problems have been overcome by R & D.



$W_r$ : Energy released per unit length of a stope by total extraction

$\delta W_r$ : Difference in energy released by total and partial extractions

$L_0$ : Critical span

## 15.2 Longwall mining in hard rock

### 15.2.3 Support and reinforcement systems

#### 1) Stope support

- Must be able to accommodate stope closure, support fractured hangingwall, and absorb energy rapidly in the event of a rockburst.
- Consists of hydraulic/timber props, timber/concrete packs, tendons, and backfill.
- Max. potential tributary area supported by a single support unit for static (rock fall) conditions:

$$A_T = F / \rho g b$$

$F$  = max. support unit load (N)

$\rho$  = rock density (ex.  $2700 \text{kgm}^{-3}$ )

$g$  = gravitational acceleration

$b$  = height of instability (m)

## 15.2 Longwall mining in hard rock

- Energy to be absorbed by the support in dynamic/rockburst conditions:

$$E_a = \frac{1}{2}mv^2 + mgh \rightarrow A_T = E_a / \rho b \left( \frac{1}{2}v^2 + gh \right)$$

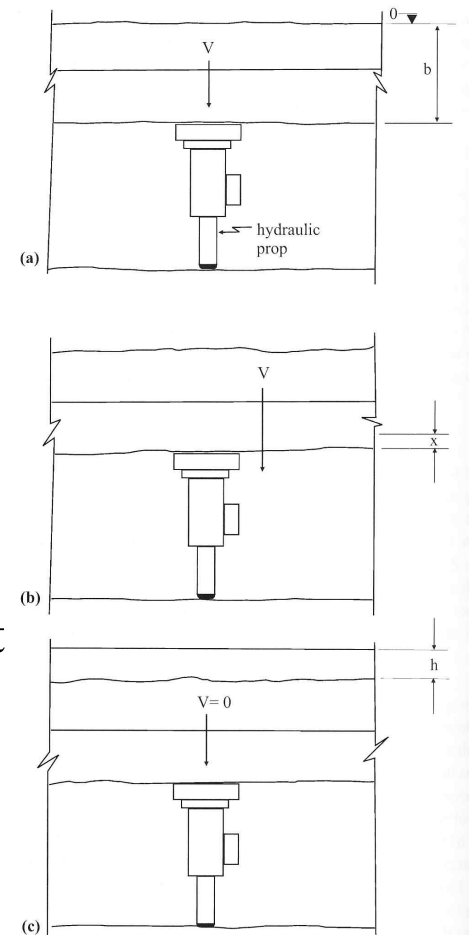
$m = \rho b A_T$  = mass of rock to be supported by the support unit

$v$  = rock ejection velocity (often taken as  $3 \text{ ms}^{-1}$ )

$h$  = hanging wall displacement during the dynamic event

$E_a$  = energy absorption capacity of the support unit (J)

- Zone of support influence: distance between the support edge and the point at which stress envelope induced by the support intersects the bedding plane that defines the upper limit of rock mass to be supported.





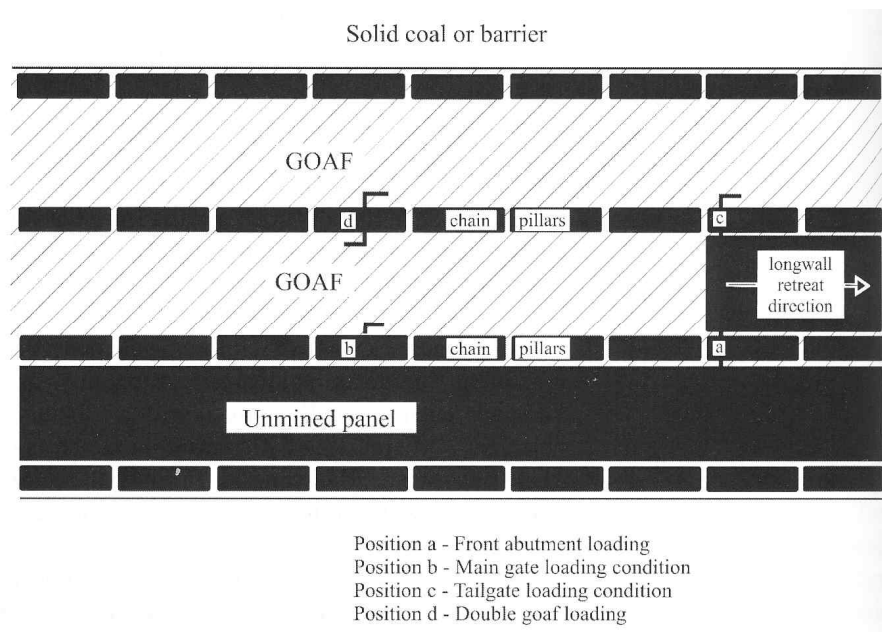
## 15.2 Longwall mining in hard rock

- 2) Tunnel (access/transportation excavation) support and reinforcement
  - Support or reinforce rock mass under both static and dynamic loading conditions.
  - Consists of rock bolt, cable bolt, studs, tendons, shotcrete (sprayed liner), mesh, straps, energy absorbing face plates, etc.
  - As in the case of stope support, the design methodology is based on tributary area and energy absorption concepts.

## 15.3 Longwall coal mining

### 15.3.1 Basic geomechanics considerations

- Primary object of mining design is to achieve pseudo-continuous deformation of the main upper strata overlying the seam.
- Lower strength of coal measures rocks: 20~40 MPa of USC and hydrostatic ( $p = \gamma h$ ) in UK.
- Retreat method is prevailing: roadways (main gate & tail gate) are developed first to the end of the panel which is then extracted on retreat.



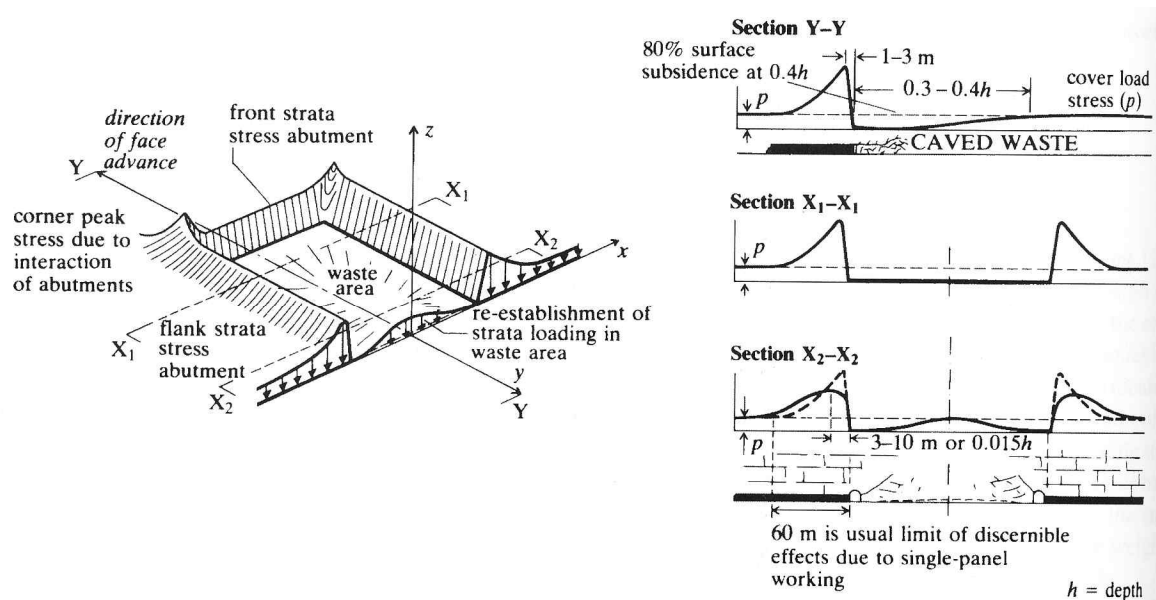
- Pillars serve to protect road ways, to isolate a panel with unfavorable discontinuities, fire, water, gas, or to control surface subsidence.
- Roadways provide access to the face for personnel, materials, air, and mined coal transportation.

## 15.3 Longwall coal mining

- Support or strata control measures at deep mines ( $> 100 \sim 200$  m):  
Face support, roadways reinforcement/support, chain or rib pillars, and support of the mined out void.

### 15.3.2 Distribution of stress around a longwall face

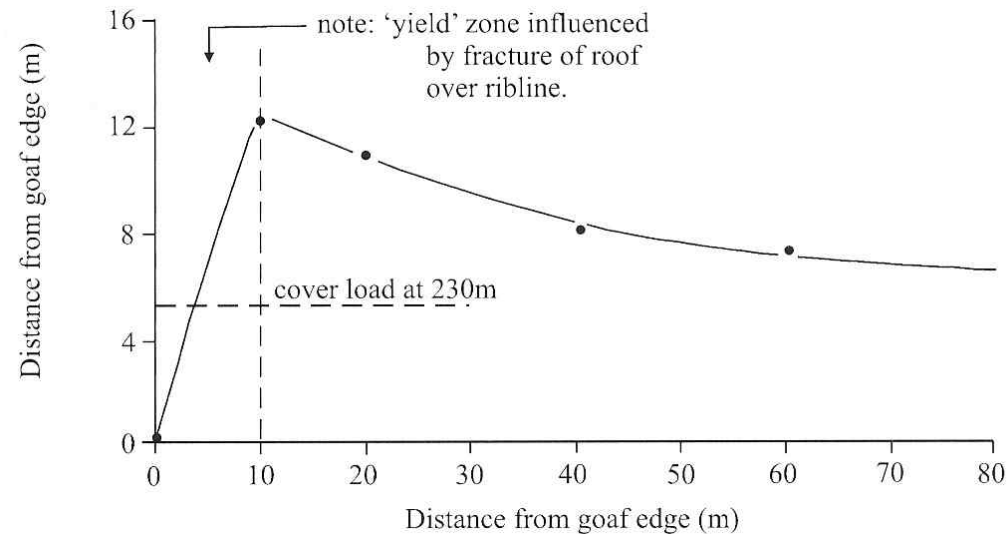
#### 1) Vertical stress distribution around a single longwall face in UK



- Max. abutment vertical stress of 4~6 times the in-situ vertical stress occurs close to the longwall face.

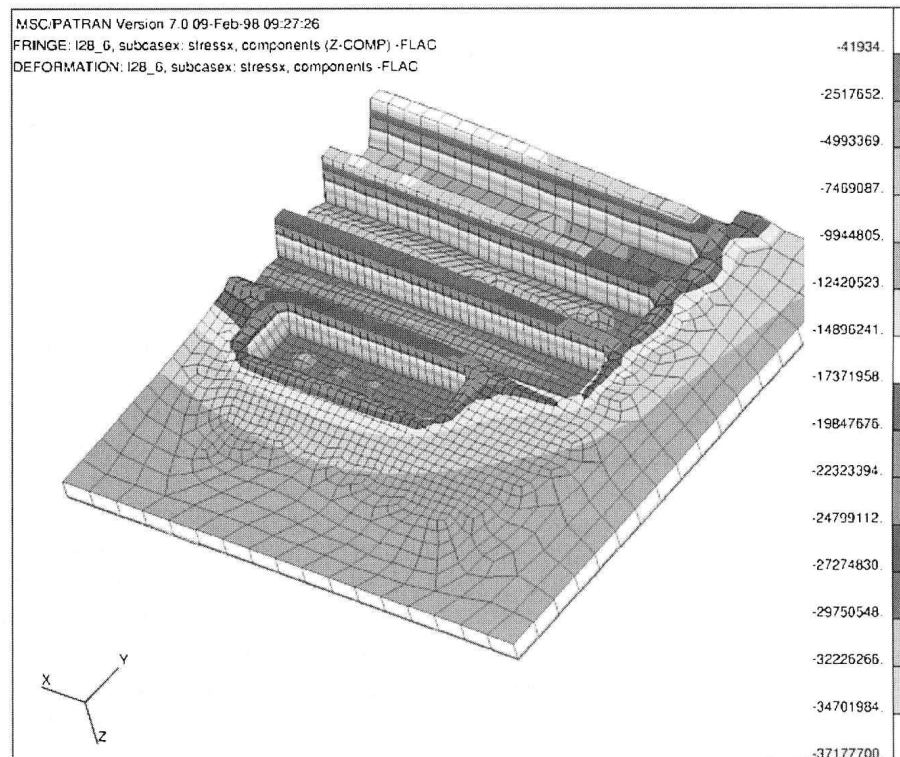
## 15.3 Longwall coal mining

- 2) Vertical abutment stress relative to the goaf edge, Gordonstone Mine, Australia
- Max. abutment stress is only twice the overburden stress and occurs about 10 m from the rib side and ahead of the longwall face.
  - Major principal in-situ stress is in horizontal, about 2.4 times the vertical stress and approximately parallel to the roadways.
  - Shearing through intact rock and along bedding planes reduces the load carrying capacity of the rock adjacent to the longwall zone.



## 15.3 Longwall coal mining

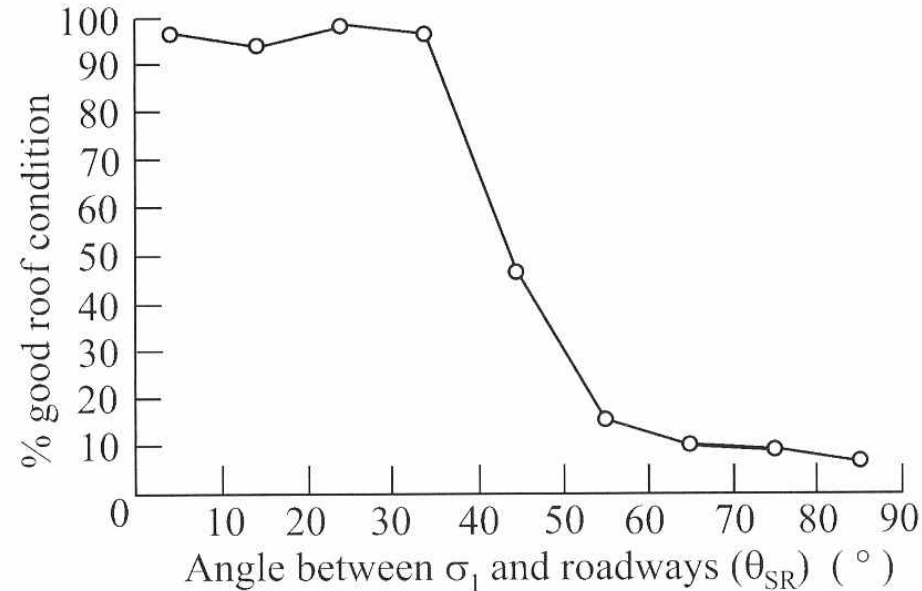
- 3) Vertical stress around four adjacent longwall panels at 500 m of depth, Australia
- Max. abutment stress is about three times the overburden stress over pillars between mined-out panels.
  - Peak abutment stress at the ends of the panels is about twice the overburden stress.



## 15.3 Longwall coal mining

### 4) Influence of the horizontal major and minor principal stresses in the Southern Coalfield, Australia

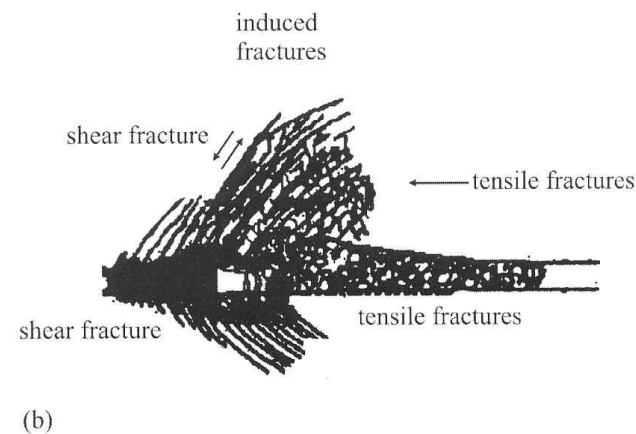
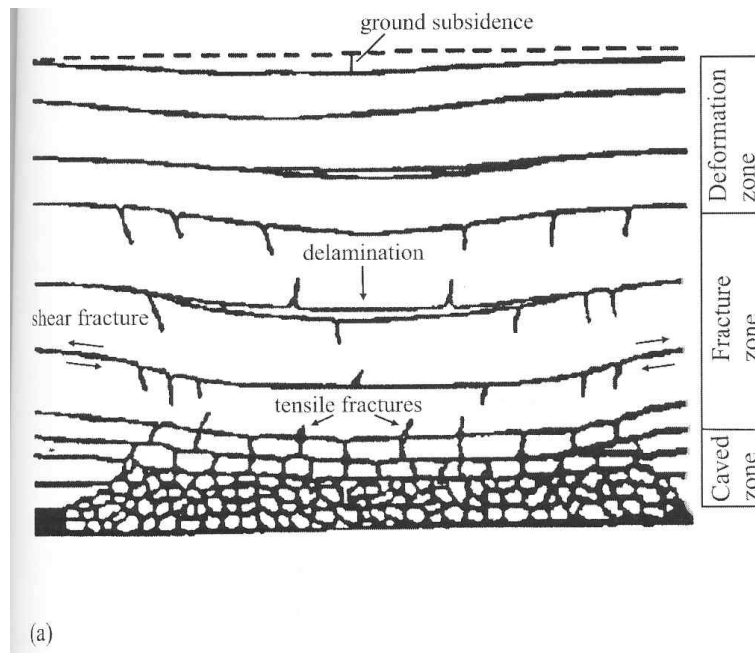
- Rectangular roadways are driven at varying angles to the major principal in situ stress
- In situ principal stress are 25, 16, and 15 MPa.
- ‘Good roof’ conditions are defined by the absence of visible shear fracturing.



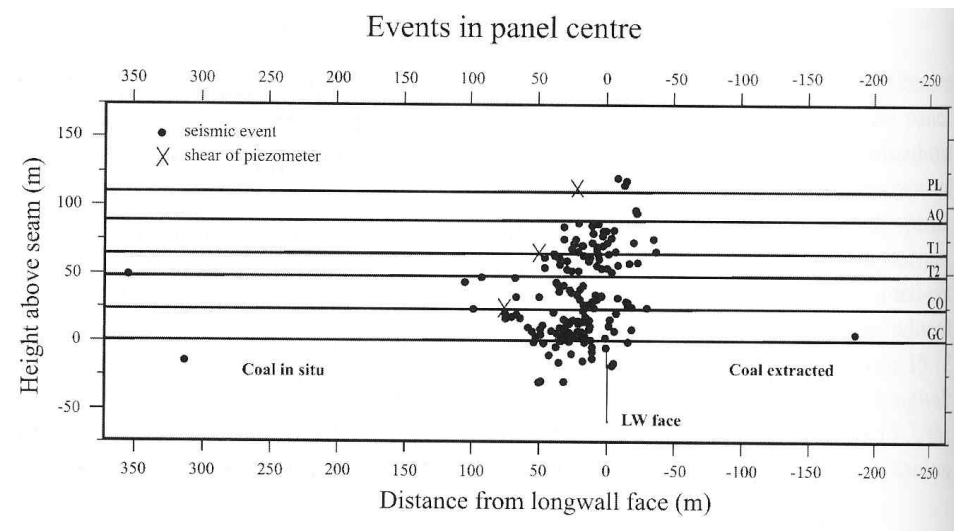
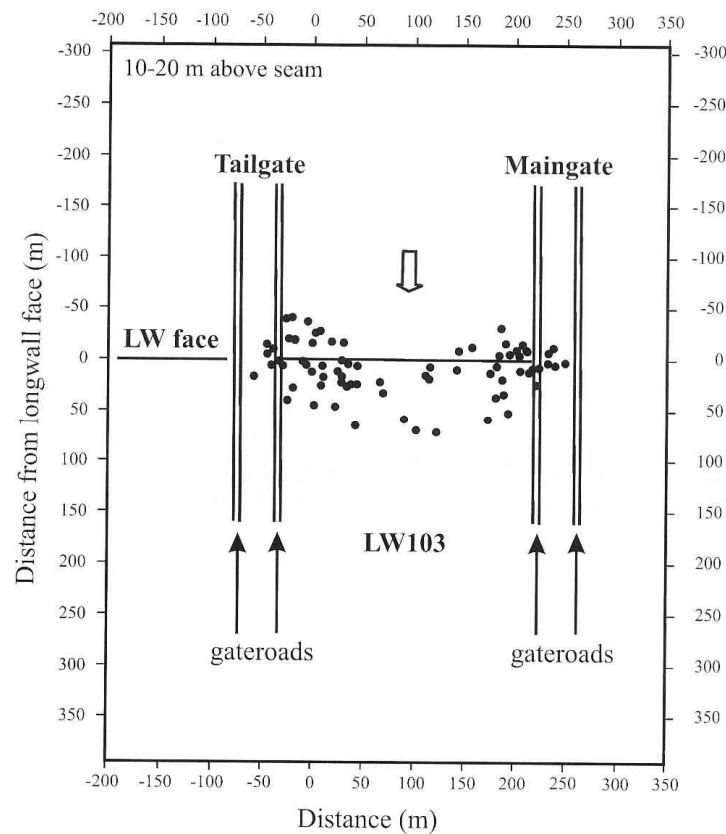
## 15.3 Longwall coal mining

### 15.3.3 Longwall caving mechanics

- Stress distributions during/following mining induce tensile fracturing, delamination and opening of pre-existing fractures which produce caving in the mined-out area, shear fracturing and slip on bedding planes and other discontinuities.



- Micro-seismic events show that rock fracture or shear occurs generally 30 ~ 50 m, but as much as 80 m, ahead of the face, and up to 100 m above or below the face.

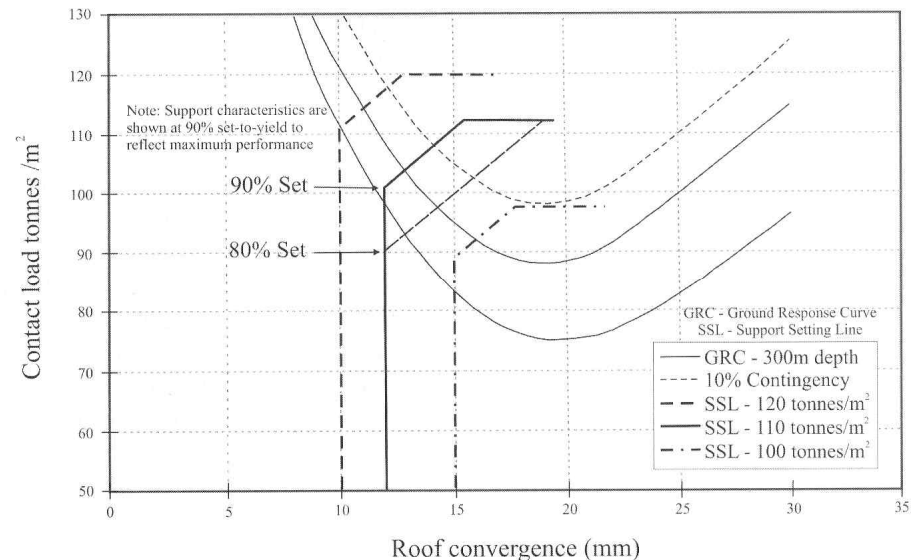
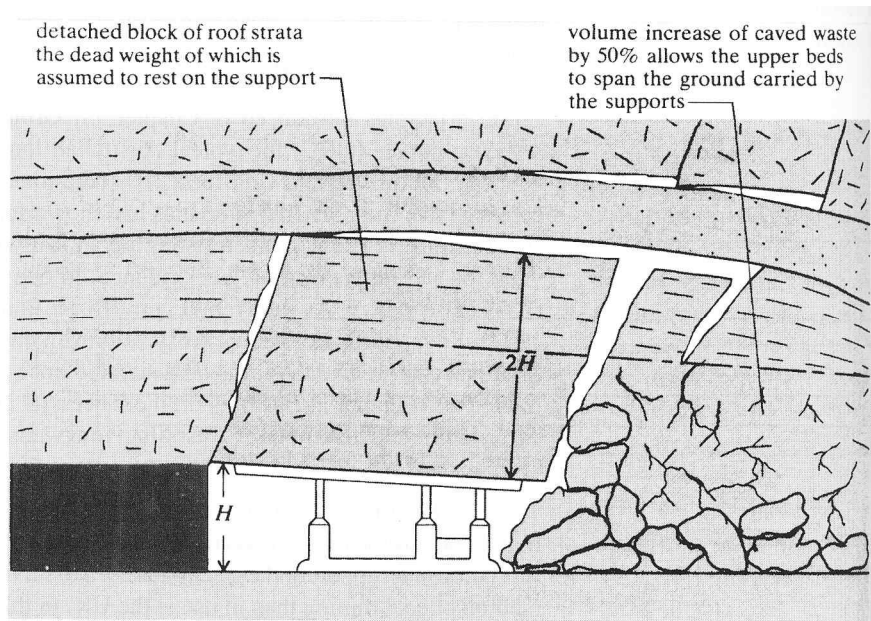




## 15.3 Longwall coal mining

### 15.3.4 Face support

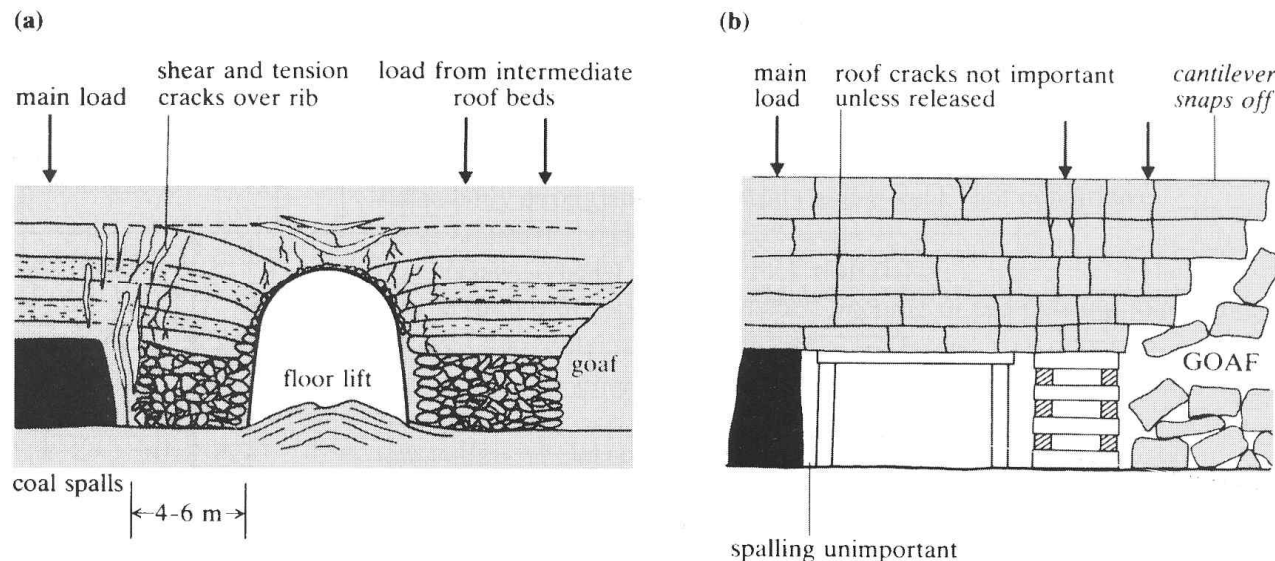
- Face support consists of a series of chocks each containing 4~6 hydraulic legs.
- The upper part of each chock is formed by roof beams or canopy.
- Rear shield supports are used where waste is friable or high horizontal waste displacement occur.
- Support system needs to support a detached block and to transmit the re-distributed stresses (at high depth).



## 15.3 Longwall coal mining

### 15.3.5 Roadway formation and support

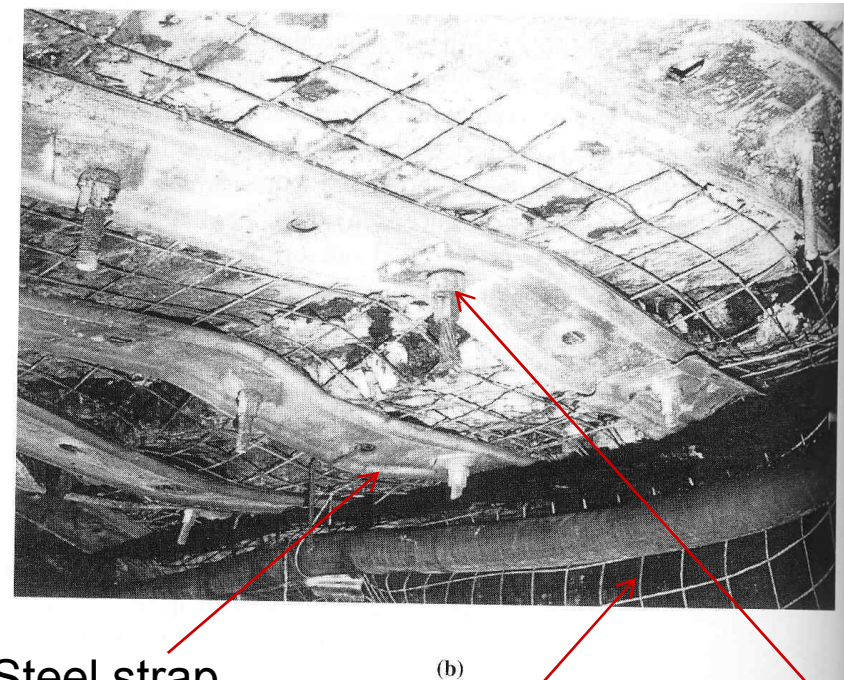
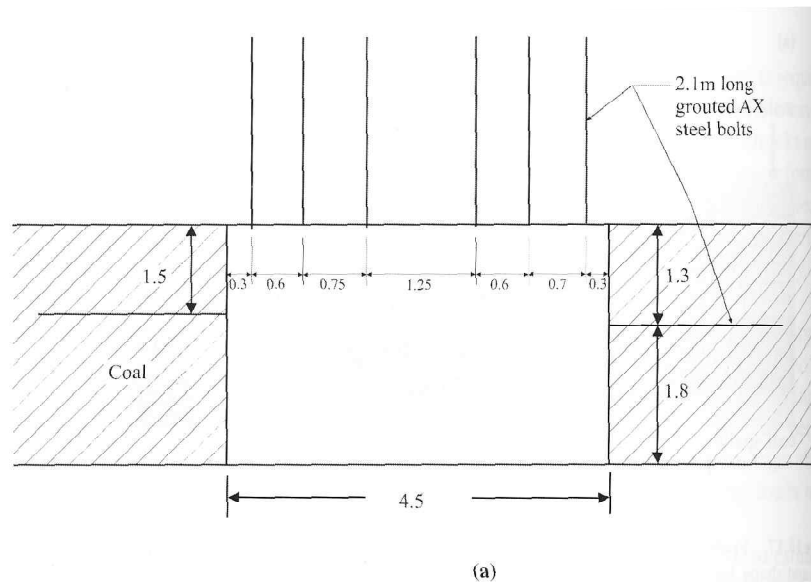
- Single roadways were often used for an advancing longwall method with packs to carry loads on the waste side, or between rib side and roadway.



- Retreat method of longwall coal mining has become almost universal where multiple entries are used widely.
- The main gate of the previous panel becomes the tail gate of the new panel to service egress and ventilation circuit.

## 15.3 Longwall coal mining

- Roadways are now usually mined to a rectangular profile.



Steel strap

Wire mesh

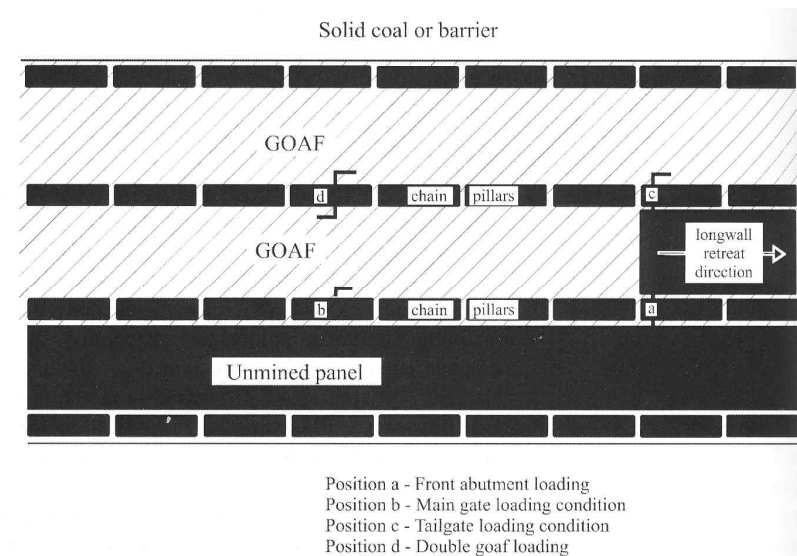
Rock bolt

Roadway support and reinforcement, Angus Place Colliery, Australia

## 15.3 Longwall coal mining

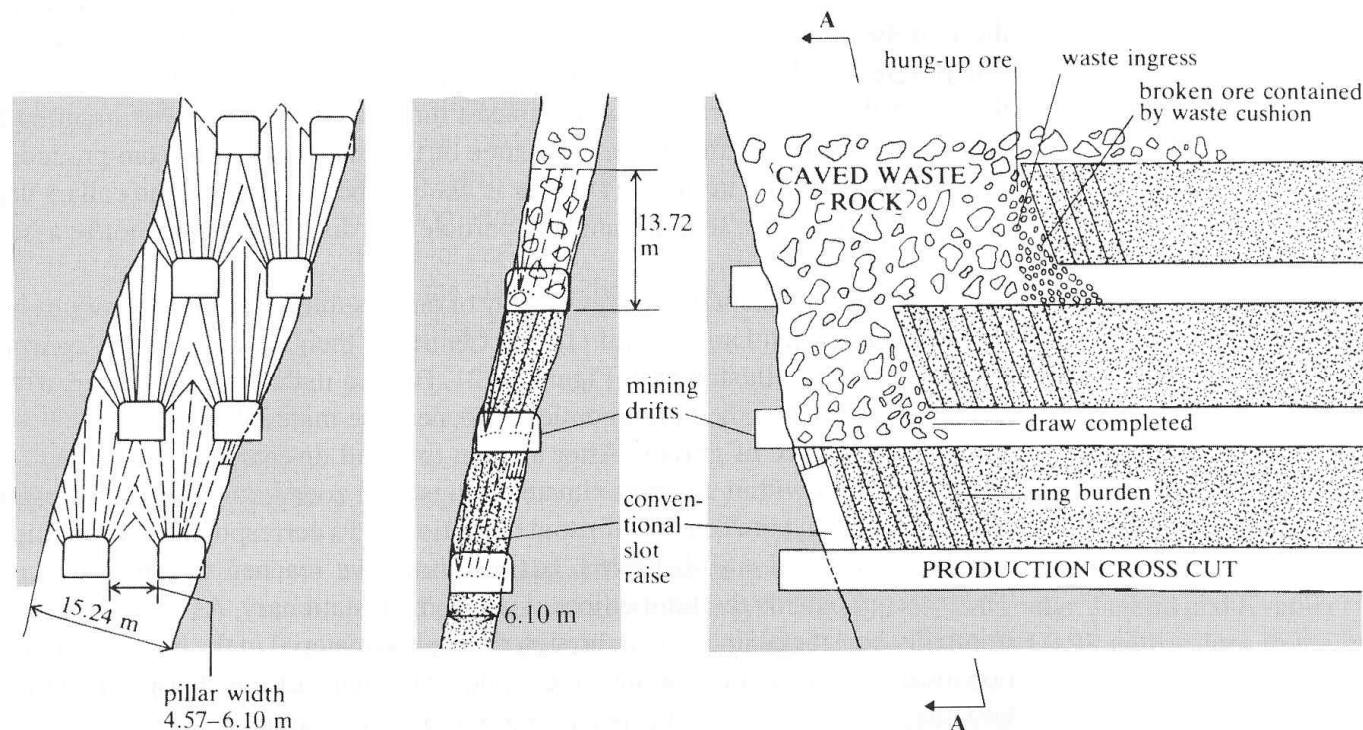
### 15.3.6 Rib and chain pillar design

- Design objectives of chain pillars
  - a) Maintain the serviceability of main and tail gates so that the safety and productivity are unaffected.
  - b) Do not impact adversely on the continuity of extraction of successive longwall panels
  - c) Maximize coal recovery.
- Main stages of chain pillar loading cycle
  - a) Development loading
  - b) Front abutment loading
  - c) Main gate or side abutment loading
  - d) Tail gate loading: pillar experiences its maximum vertical loading.
  - e) Double goaf loading



## 15.4 Sublevel caving

- Longitudinal sublevel caving for narrow orebodies and transverse sublevel caving for wide orebodies.
- Major geomechanics issues involved: gravity flow of blasted ore and caved waste, maximum recovery and minimum dilution while ensuring stability of openings, and surface subsidence.



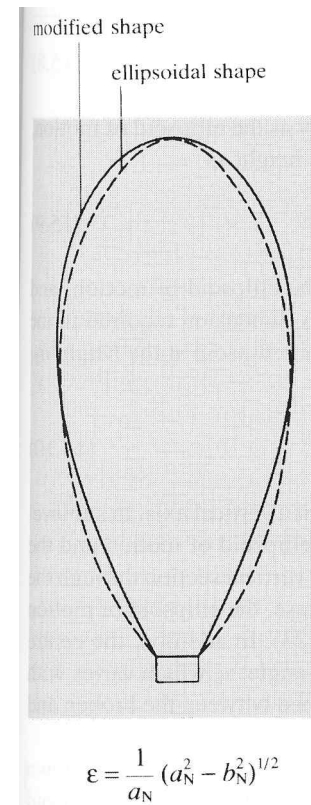
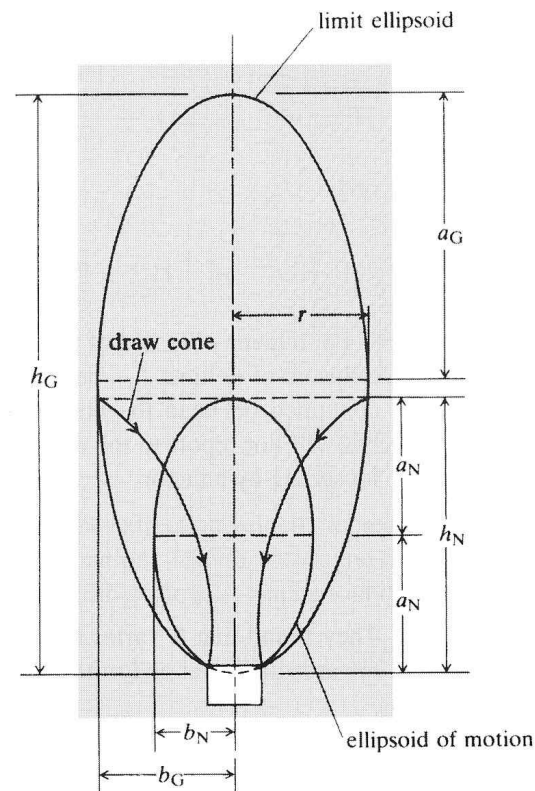
## 15.4 Sublevel caving

### 15.4.1 Gravity flow of caved ore

- Flow of caved ore cannot be described satisfactorily by theories for the flow of granular materials because the particle size, discharge rates, boundary conditions, feature of progressive breakdown of broken ores, and degree of fragmentation of ore.
- Classical concept of the gravity flow of ore in sublevel caving operations is developed by Janelid and Kvapil (1966): When bottom outlet is opened, broken material contained within a bin or bunker will begin to flow out under the influence of gravity
- Ellipsoid of motion: ellipsoidal zone of which broken material is discharged through the outlet.
- Limit ellipsoid: material between the ellipsoid of motion and a corresponding limit ellipsoid will be loosen and displaced, but will not reach the discharge point.
- Draw cone: As draw proceeds, an originally horizontal line drawn through the broken material will deflect downwards in the form of an inverted cone.

## 15.4 Sublevel caving

- Shape of the ellipsoid of motion is a function of the distribution of particle size and the width of discharge opening: the smaller the particle size, the more elongated is the ellipsoid of motion, the upper portion tends to be flattened or broadened particularly large draw heights and irregular particle sizes.



## 15.4 Sublevel caving

- Shape of ellipsoid of motion expressed by eccentricity:

$$\varepsilon = \frac{\sqrt{a_N^2 - b_N^2}}{a_N} : 0.90 \sim 0.98, \text{ most commonly } 0.92 \sim 0.96$$

- Semi-minor axis of the ellipsoid of motion:

$$b_N = \left( \frac{E_N}{2.094h_N} \right)^{1/2} \quad \text{or} \quad b_N = \frac{h_N}{2} \sqrt{1 - \varepsilon^2} \quad (E_N \text{ is volume of an ellipsoid of motion})$$

- Loosening factor of the material contained between boundaries of the two ellipsoids

$$\beta = \frac{E_G}{E_G - E_N} : 1.066 \sim 1.100 \rightarrow E_G \cong 15E_N$$



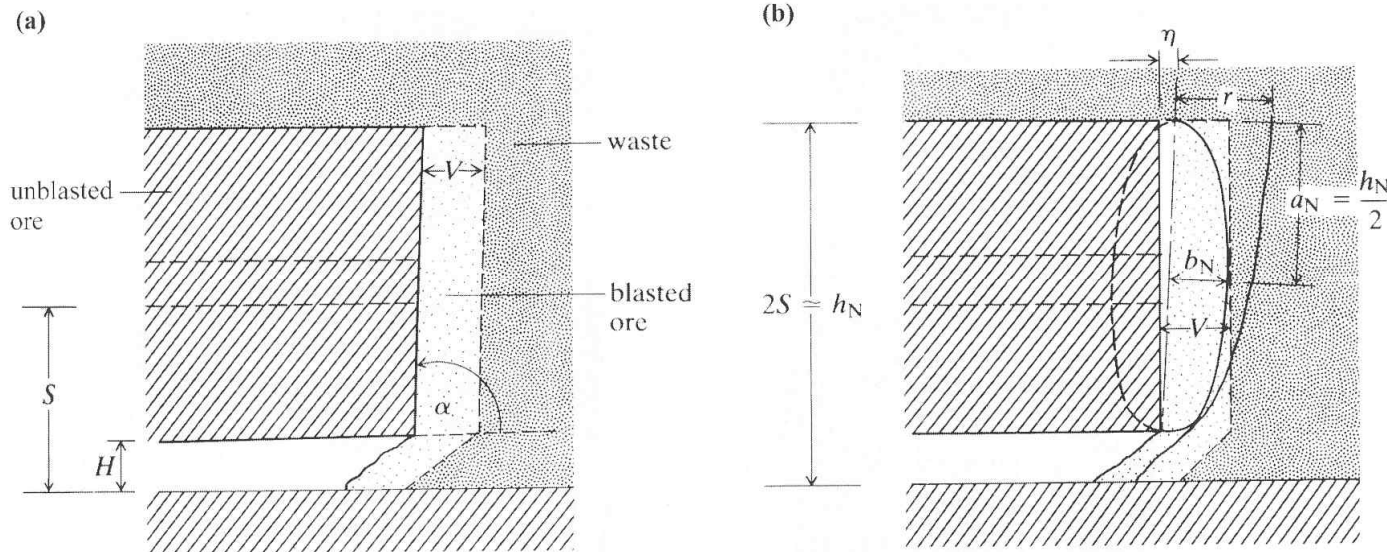
## 15.4 Sublevel caving

- Height of a limit ellipsoid when the eccentricities of the limit ellipsoid and the ellipsoid of motion are assumed to be the same:

$$h_G \cong 2.5 h_N$$

- Radius of the limit ellipsoid:

$$r \cong \left[ h_N (h_G - h_N) (1 - \varepsilon^2) \right]^{1/2}, \quad r \cong S \left[ 6(1 - \varepsilon^2) \right]^{1/2}$$



## 15.4 Sublevel caving

### 15.4.2 Design of sublevel caving layouts

#### 1) Classical gravity flow theory to sublevel caving by Janelid and Kvapil

##### - Nomenclature

$A$ = width of the slice,  $H$ =height of the extraction drift,  $B$ =width of the extraction drift,  $P$ =width of the pillars between extraction drifts,  $V$ =ring burden,  $K$ =factor dependent upon the particle size distribution in the broken ore,  $a_N$ ,  $b_N$  = major and minor semi-axes of the ellipsoid of motion,  $c$ =width of the extraction point,  $r$ =semi-width of the limit ellipsoid at the height  $h_N$ ,  $x$  = digging depth of the loader,  $\alpha$ =gradient of the slice,  $\phi$ =angle of repose of the ore in the draw point,  $\varepsilon$ =eccentricity of the ellipsoid of motion

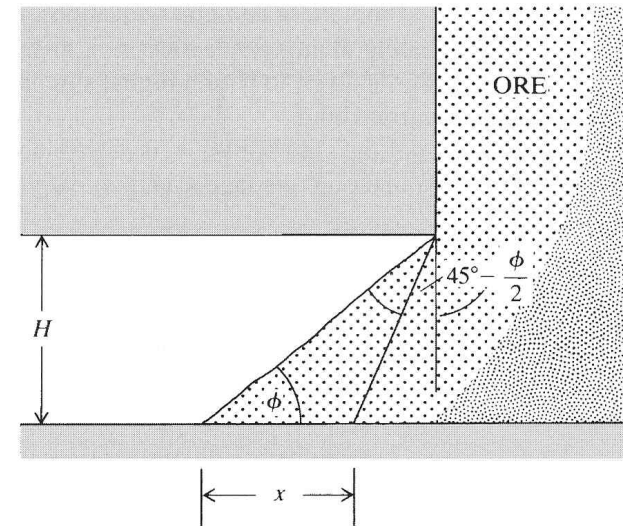
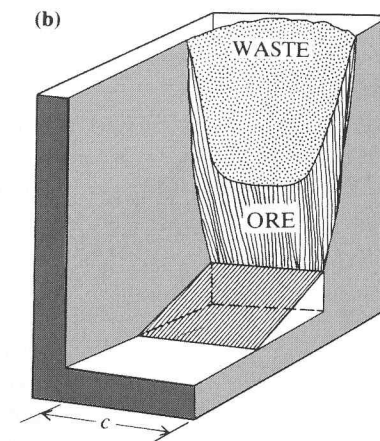
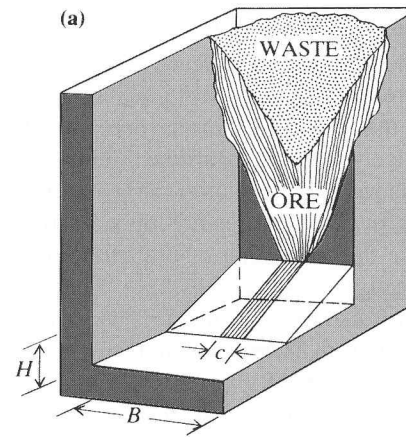
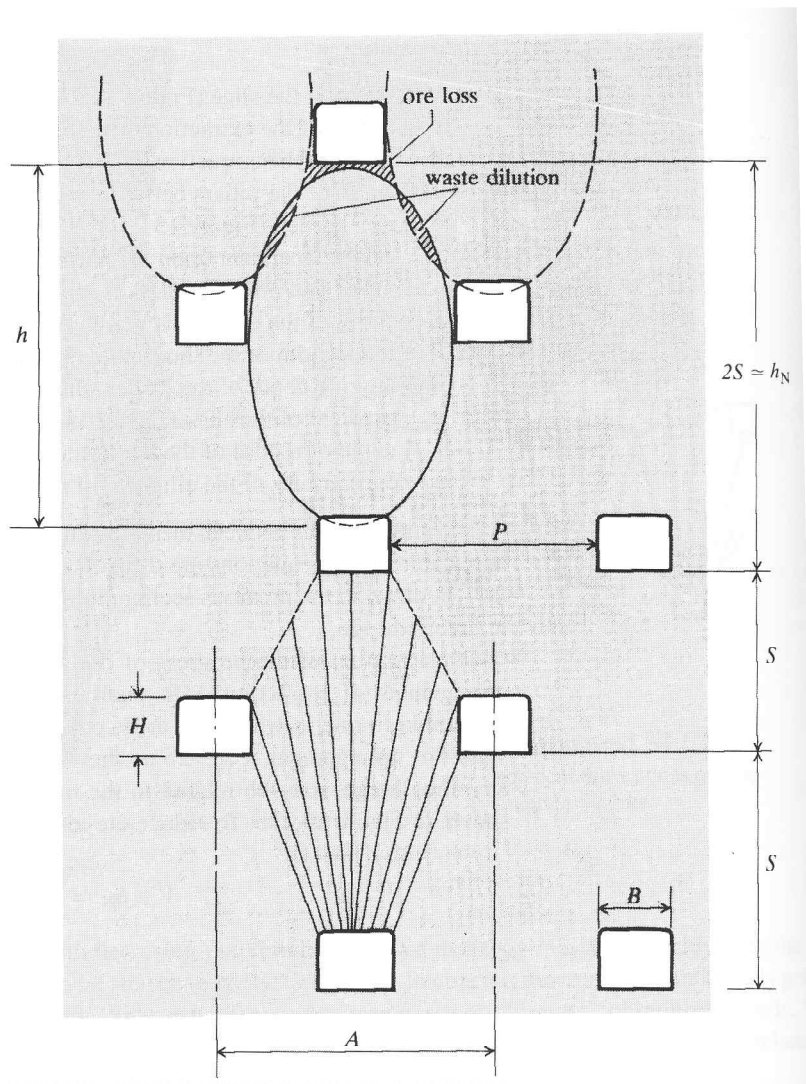
- Staggered arrangement of extraction drifts are generally preferred.

- Parameter determination:

$$V \geq \frac{b_N}{2}(1 - \varepsilon^2), \quad A \leq 2r, \quad P = A - B$$

$$B > 11.2\sqrt{K}D, \quad x = H \left[ \cot \phi - \tan \left( 45^\circ - \frac{\phi}{2} \right) \right], \quad \text{etc.}$$

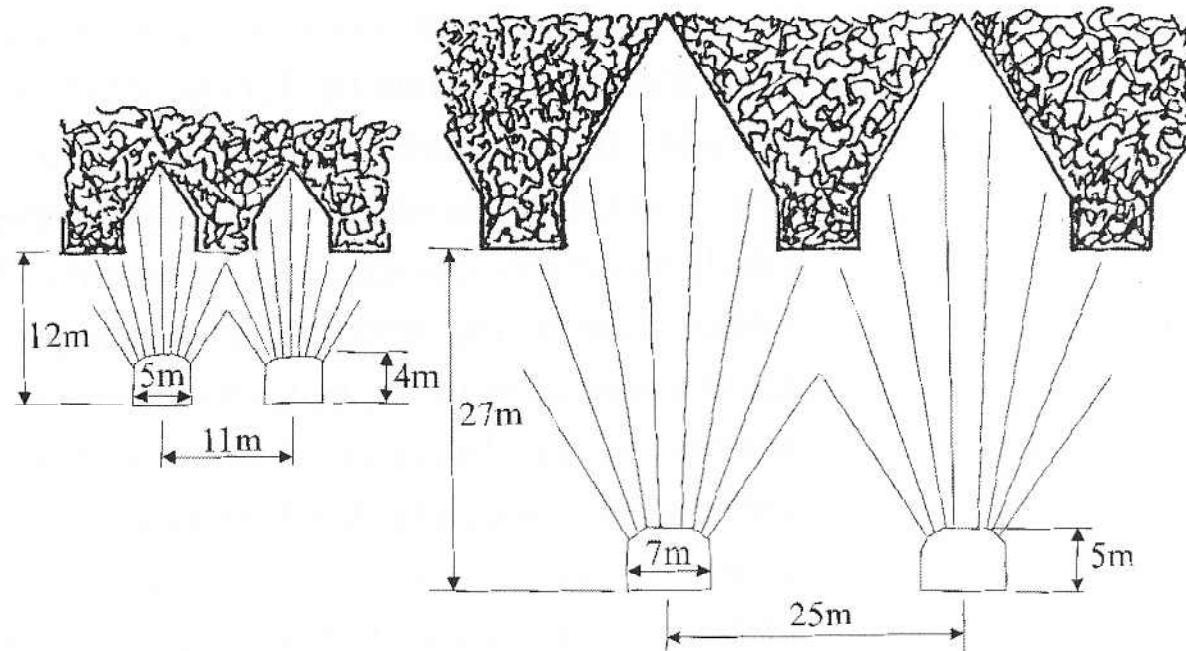
## 15.4 Sublevel caving



## 15.4 Sublevel caving

### 2) More recent sublevel stoping layouts

- Larger sublevel and production drift spacings, and drift sizes have been introduced in large-scale, highly productive sublevel caving operations: relatively small amount of excavating, drilling, and blasting.



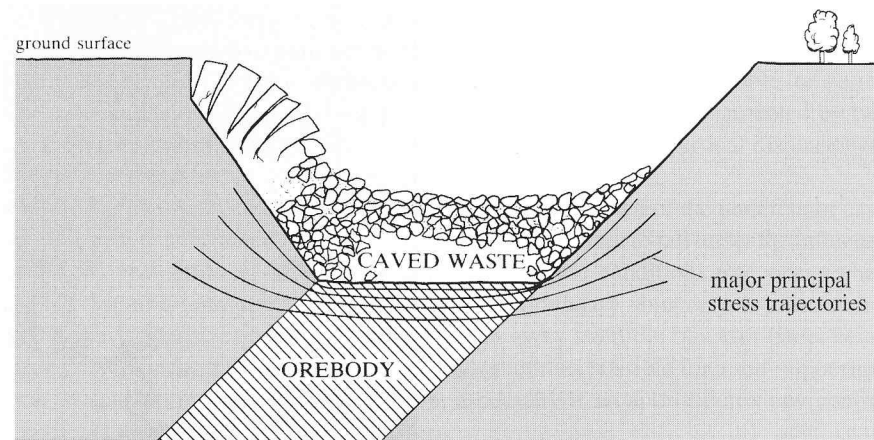
## 15.4 Sublevel caving

### 15.4.3 Support and reinforcement

- Design of sublevel caving layouts are normally based on geometry of the gravity flow of broken ore.
- As mining depth increases the 'honeycomb' effect produced by the relatively closely spaced production openings can lead to stress-induced instability → re-orientation of the mining direction, an increase in sublevel/drift spacing, or reinforcement are required.

#### 1) Pea Ridge iron ore mine, USA

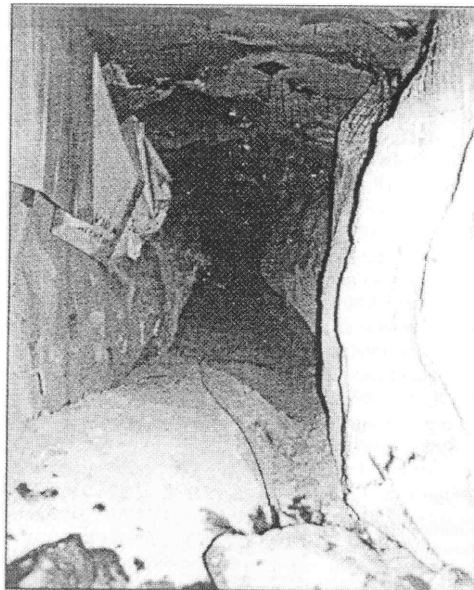
- High horizontal in situ stresses exist.
- Stress-induced instability was significantly reduced when longitudinal sublevel caving was replaced with a transverse method.



## 15.4 Sublevel caving

### 2) Perseverance Mine, Australia

- Weak hangingwall shear zone, swelling minerals, frequent faulting and shearing, high in situ stresses, low rock mass strengths, and inflection at depth of 420 m exist.
- Cross-cut at a depth of 600 m experiences side wall closure of 3 m, more than 1 m of floor heave, and progressive failure of the support and reinforcement.
- New sublevel caving layout was developed for depths greater than 600 m: sublevel spacing of 25 m, drift support and reinforcement with fiber reinforced shotcrete, mesh, Split Set rock bolts, grouted rock bolts are adopted/installed.

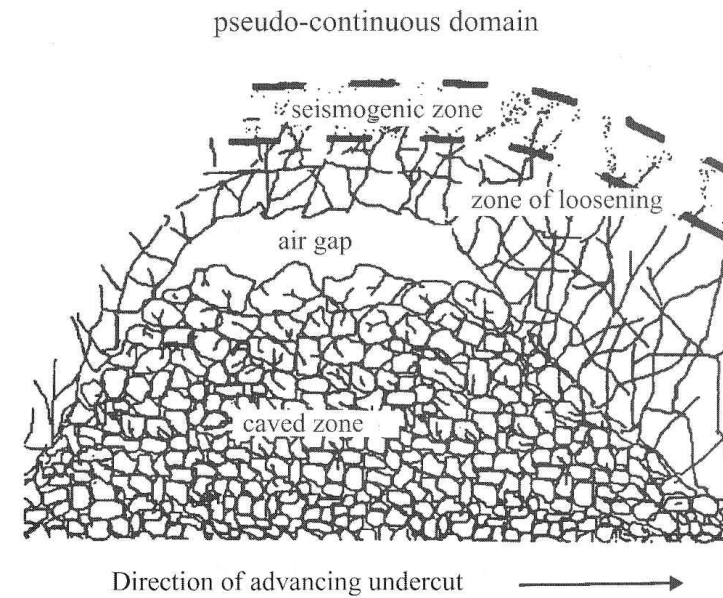
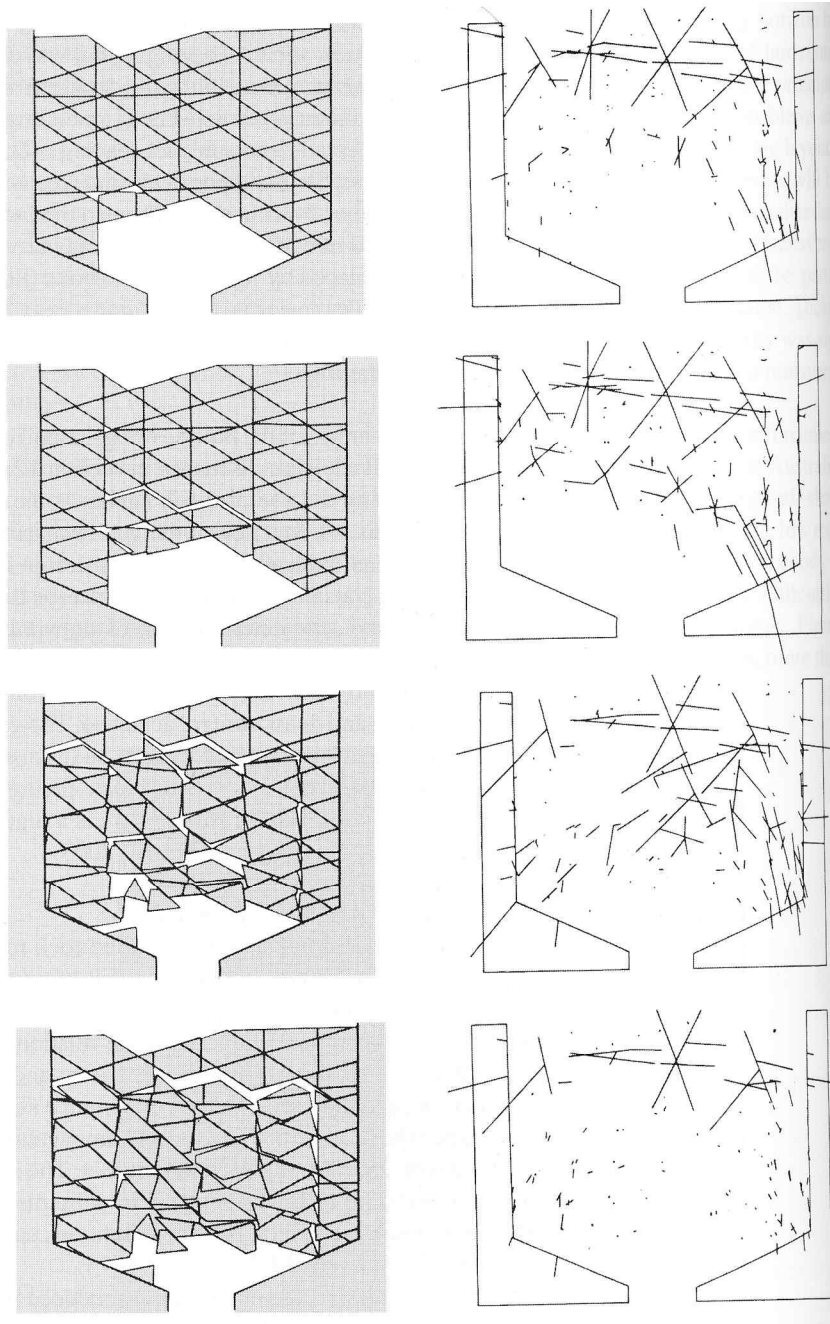


## 15.5 Block caving

- Key issues are cavability, fragmentation, draw control, etc..
- Cavability is a function of geomechanical properties of rock mass and of the in situ/induced stresses.

### 15.5.1 Basic caving mechanics

- Caving occurs by gravity and by stress induced in the crown of the undercut.
- Successful initiation and propagation of caving requires the presence of a well-developed, low-dip discontinuity set.
- Mechanism of caving and arching illustrated by the simple and idealized DEM simulation: refer to Fig.15.32.
- Conceptual model of caving: refer to Fig.15.33
- Compressive tangential stresses in the crown are low/tend to be tensile, blocks may become free to fall under gravity or slide on discontinuities: when low horizontal in situ stresses or boundary slots /previous mining exist.
- Stress caving: tangential stresses are higher than the compressive strength of rock mass and shear strength of discontinuities → brittle failure and slip on discontinuities.



### Five regions:

1. Caved zone – support to the walls
2. Air gap – function of extraction rate
3. Zone of discontinuous deformation
4. Seismogenic zone – slip on joints and brittle failure
5. Surrounding rock mass – elastic deformation



## 15.5 Block caving

- Tangential stresses are high enough to inhibit gravity-induced caving, but are not high compared to the compressive strength of the rock mass: caving may be inhibited, self-supporting arch may develop. Possible damage to installations and injury to personnel when the arch eventually fails to cause impact loads and air blast.
- Case of Northparkes E26 Mine, Australia: hydraulic fracturing and a boundary slot were used to attempt to induce continuous caving after caving of Lift1 had been arrested. A massive air blast resulting from the collapse of the crown pillar into a large void when caving propagated into a weak, gypsum-leached zone led to the death of four men in late 1999.
- Subsidence caving: a large mass of rock subsides rapidly as a result of shear failure in the vertical or near-vertical boundaries of a block. It causes catastrophic consequences. This can be prevented by not allowing a significant air gap to develop below the cave back.

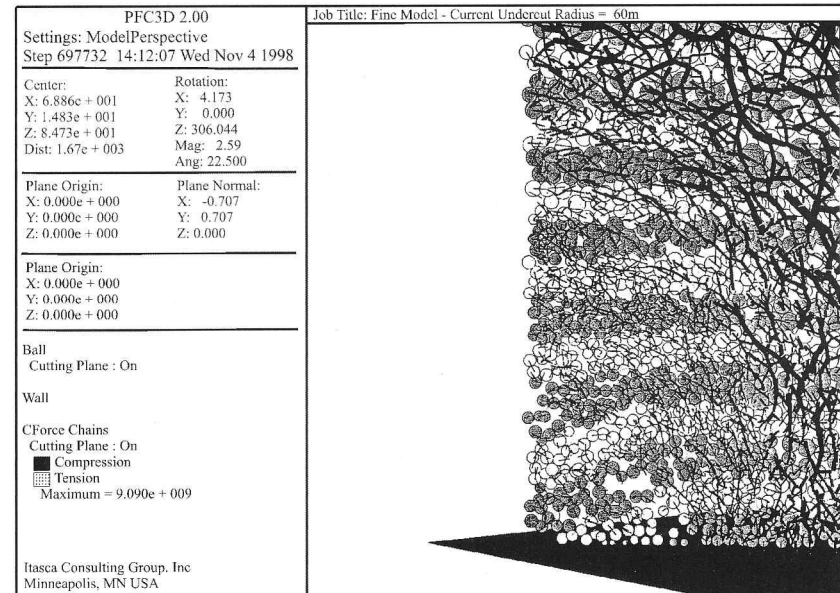
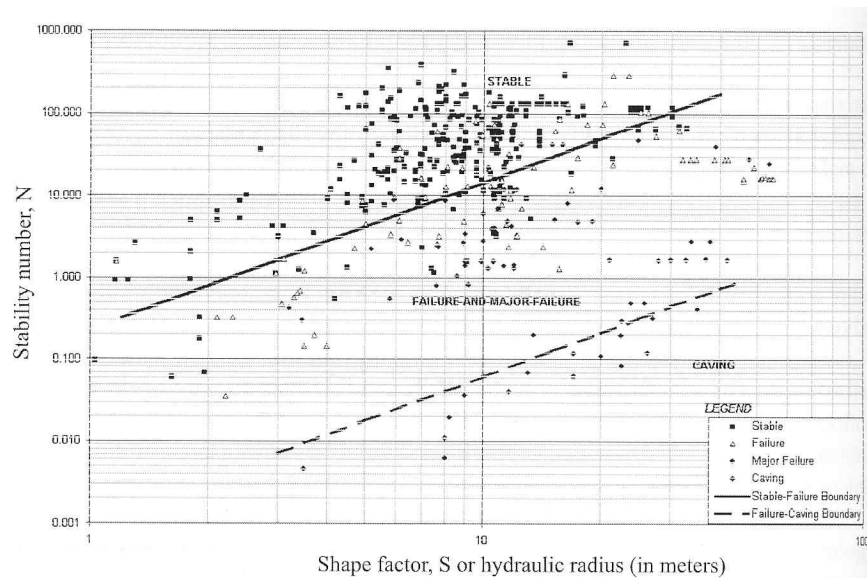
## 15.5 Block caving

### 15.5.2 Cavability

- Affected by discontinuity geometry and strength, rock mass strength, orebody geometry, undercut dimensions, stress induced in the crown of undercut, etc.
- 
- 1) Laubscher's caving chart approach (1980s)
    - Predict cavability by combining measures of rock mass quality, undercut geometry, and induced stresses.
    - Plot values of MRMR (Modified RMR) against hydraulic radius,  $S$  (area/perimeter) of undercut.
    - The chart is divided into three zones – stable, transitional, and caving .
    - Proper to the weak and large orebodies: it may not provide satisfactory result for strong, small and isolated orebodies.
    - Insufficient case studies are available for rock mass having  $MRMR > 50$ .
  - 2) Extended Mathews stability chart graph by Mawdesley (2002)
    - The line separating major failures from continuous caving does not represent a 100% probability of caving.

### 3) Numerical modeling

- PFC3D etc. is available.
- Advantageous in cases for which current experience is lacking or not well developed.



## 15.5 Block caving

### 15.5.3 Cave initiation by undercutting

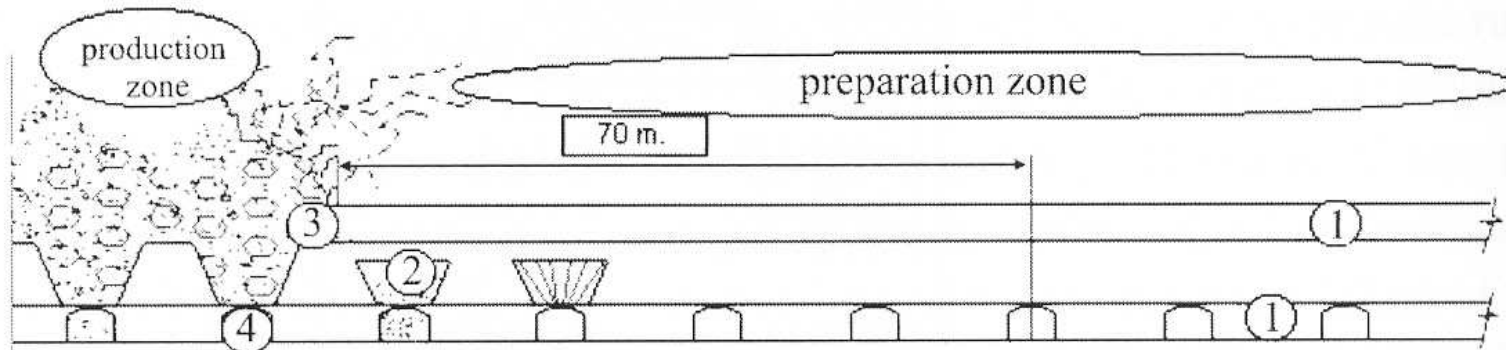
- Factors for successful implementation of undercutting: sequence of undercut and extraction level (undercut strategy), starting point and direction of undercut advance, rate of undercut advance, height of undercut, and shape of the undercut in both plan and vertical section.

#### 1) Undercut strategy

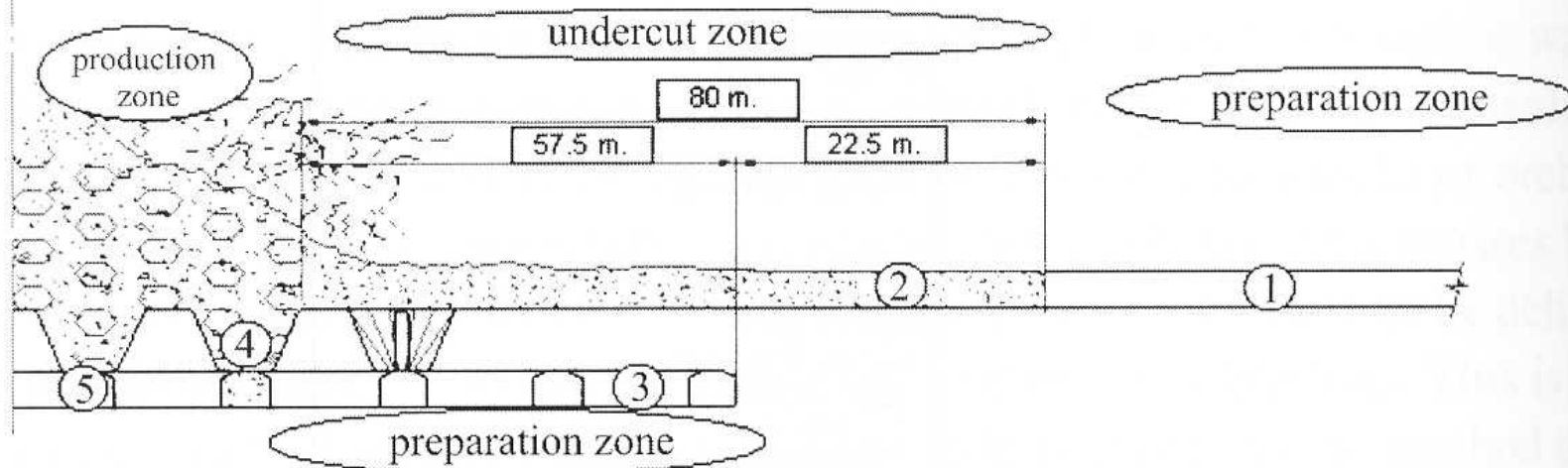
##### a. Post-undercut

- Also referred to as conventional undercutting.
- Undercut drilling and blasting take place after development of extraction level.
- Cones, drawbells or troughs are prepared ahead of undercut.
- Advantage: blocks can be brought into production quickly, and probability of ore compacting is low.
- Disadvantage: rock mass between undercut and extraction level is subjected /to high and variable stress levels → support /reinforcement is required.

## POST - UNDERCUT



## PRE - UNDERCUT



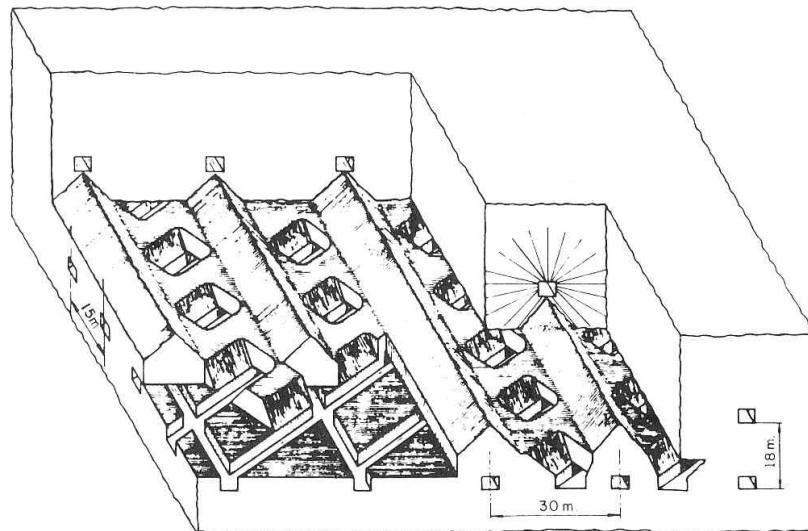
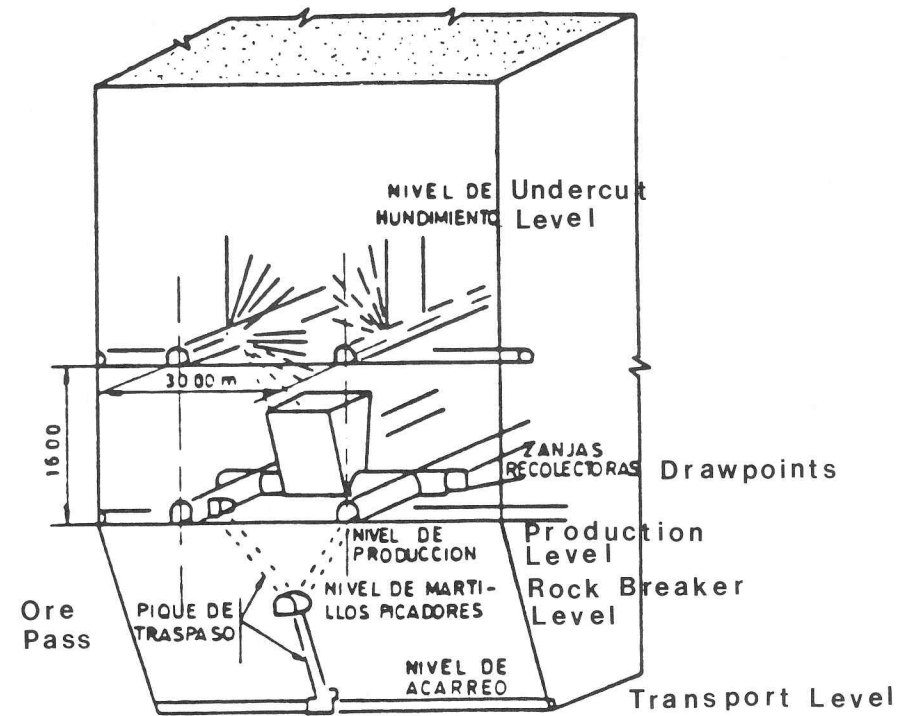
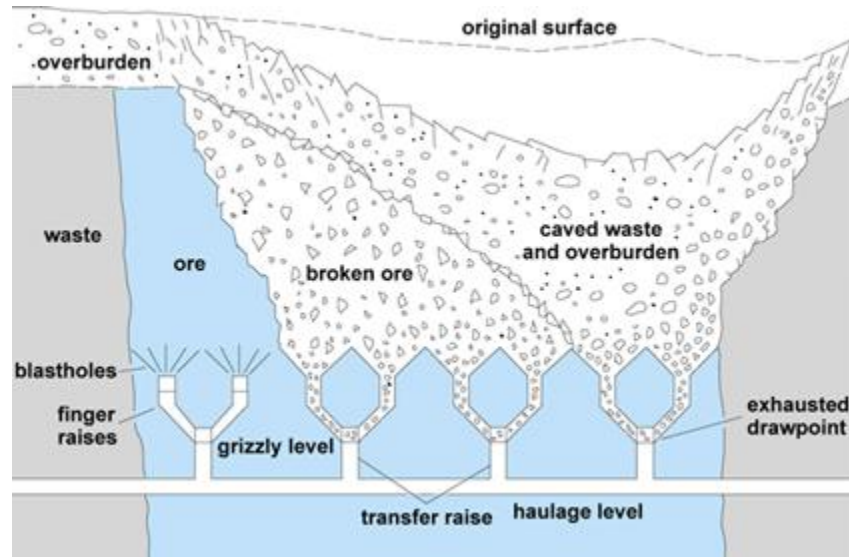
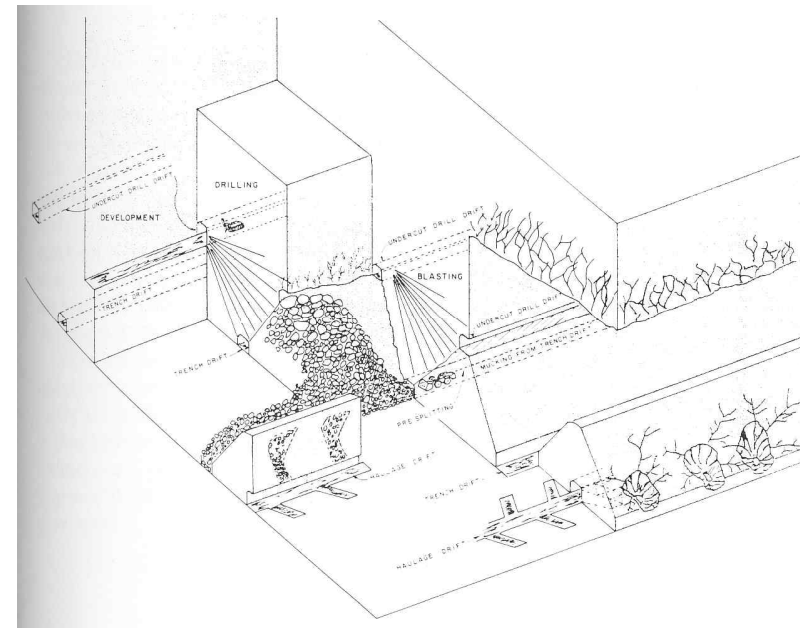


Fig. 20.3.16. Isometric of LHD system — El Teniente.  
Conversion factor: 1 ft = 0.3048 m.



## 15.5 Block caving

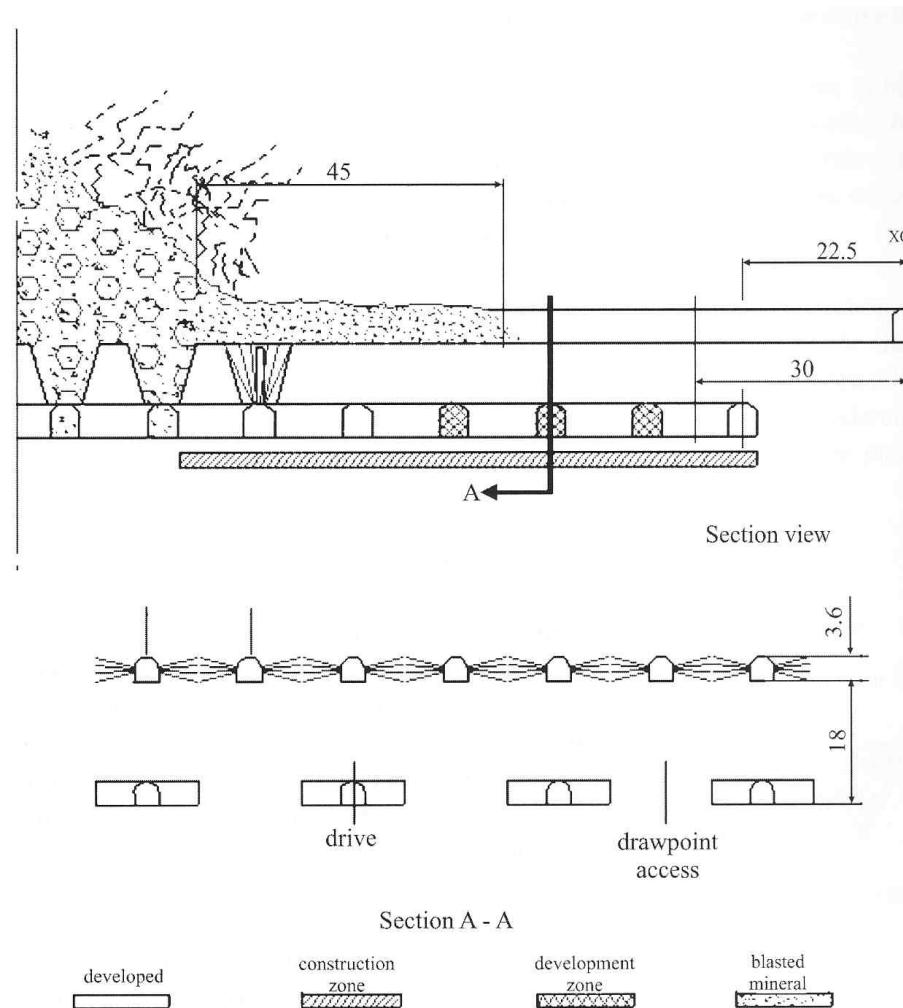
### b. Pre-undercut

- Undercut is mined ahead of extraction level development.
- The min. horizontal distance between the advancing undercut and extraction level is the distance between two levels → “the 45 degree rule”.
- Advantage: the extraction level is developed in a de-stressed environment, support / reinforcement requirements on extraction level are lower than for the post-undercutting method, and the broken ore in the undercut level acts as rock fill.
- Disadvantage: need to develop drawbells from extraction level into broken rock on the undercut level, and drawpoint hang-ups resulting from ore compaction.

### c. Advance (advanced) undercut

- Undercut drilling and blasting takes place above a partially developed extraction level
- Current trend in block and panel cave design
- A compromise between the post- and pre-undercutting in terms of extraction ratio, production speed, and need for the time-consuming/costly repairs to the extraction level drifts

# 15.5 Block caving

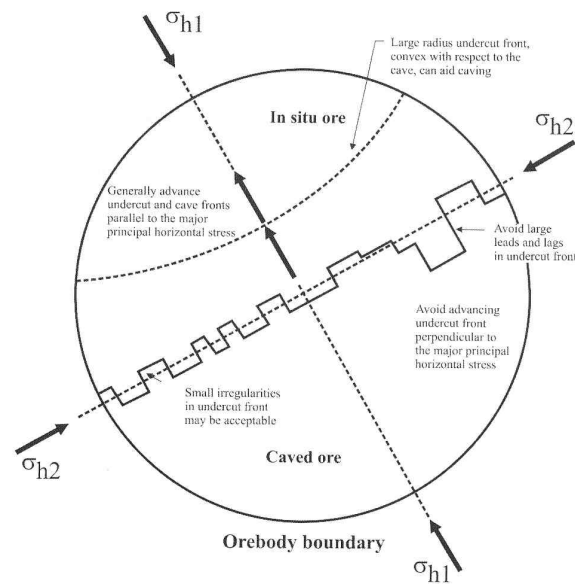






### 3) Shape of the undercut face

- Generally advance undercut and cave fronts parallel to the major principal horizontal stress.
- Sharp changes or large irregularities in the shape of the undercut face should be avoided.
- Lead between adjacent sections of the overall face should be minimized.
- A face that is convex with respect to the cave should cave more readily than one that is concave.



## 15.5 Block caving

### 15.5.4 Extraction level design

- Extraction level design and performance are influenced by the degree of fragmentation and the undercutting strategy adopted.
- Grizzly and slusher systems have been almost completely replaced by mechanized methods using Load-Haul-Dump (LHD) vehicles → production and drawpoint drifts are generally 4 ~5 m wide to accommodate the LHDs.

### 15.5.5 Fragmentation

- Overall success of a block caving operation depend, to a significant extent, on the fragmentation
  - Difficulties in developing and validating caving fragmentation: difficulty of measuring the fragmentation distributions at each stage of fragmentation → Digital image processing systems are now being developed.
- 1) In situ fragmentation
- Fragmentation that are naturally present within the rock mass before mining takes place.
  - Defined by the pre-existing discontinuities.

## 15.5 Block caving

### 2) Primary fragmentation

- Defined when the undercut is mined and caving initiated.
- Most failures can be expected to occur on pre-existing planes of weakness, but under high stress, fracture of intact rock may also occur.

### 3) Secondary fragmentation

- Defined when the caved ore resides in, and pass through, the draw column.
- Mechanism: extension/opening of discontinuities, crushing, compressive/shear/tensile/bending failure of blocks, abrasion of block edges.

### 15.5.6 Draw control

- Objects of draw control:
  - Minimizing overall dilution
  - Ensuring maximum ore recovery
  - Avoiding damage in the extraction level
  - Avoiding air blast or mud rushes

## 15.5 Block caving

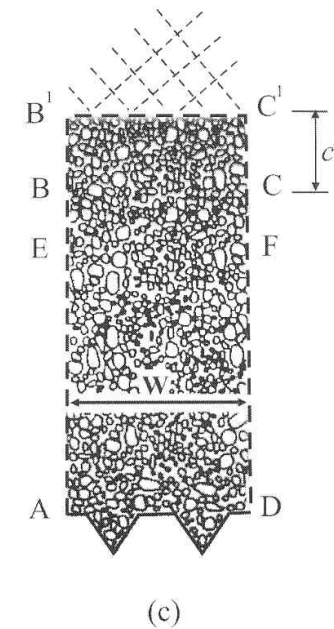
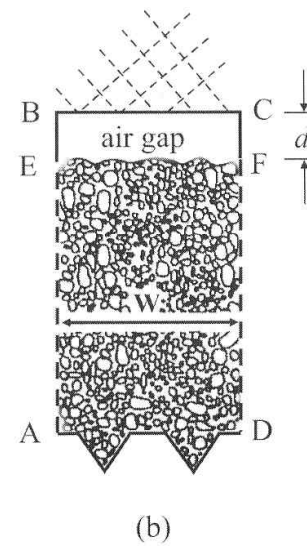
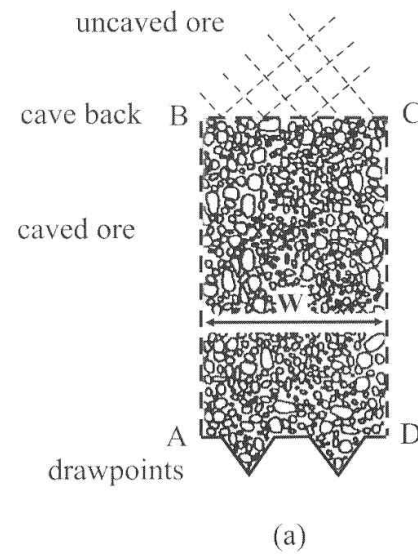
- Draw rate in volume:

$$(c + d)w = c(1 + B)w \quad B = \text{bulking factor (about 0.2 for initial caving)}$$

$$d = cB$$

- Draw rate in tonnage:

$$d_t = \frac{c_t B}{1 + B}$$



## 15.5 Block caving

- Undesirable consequences of excessive draw:  
It could cause an uneven profile in cave back and arrest of caving.  
Chimneying may happen in weak materials and therefore early dilution.  
Air gap may be created leading to the air blast.