www.MINEPORTAL.in

(All Mining Solutions)

DOWNLOAD MINEPORTAL APP FROM GOOGLE PLAY STORE

- **ONLINE TEST SERIES FOR**
  - **DGMS COAL/METAL FIRST & SECOND CLASS**
  - **GATE MINING**
  - **COAL INDIA EXAMS**
  - **OTHER PSUs MINING EXAMS**
- **FREE STUDY MATERIAL-MINEPORTAL NOTES & USER NOTES**
- **MINING VIDEO LECTURES**
- **DISCUSSION FORUM**
- **WHATSAPP CHAT SUPPORT FOR MINING EXAMS & QUERY RELATED TO MINING**

CALL/WHATSAPP- 8804777500

VISITING SITE- **www.MINEPORTAL.in** **Facebook-** **www.fb.com/mineportal.in**

EMAIL- INFO@MINEPORTAL.IN

GROUND CONTROL

Fifth Semester

www.MINEPORTAL.in

⇒ All formulations in Ground control taught are analytical formulas because we want to solve a complex problem by a simpler approach. So analytical formulas has been made on some idealizations which cause some limitations in the analytical formulaes

⇒ Formulaes which are derived from field observations are called Empirical formulaes

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Topics:-

25/07/2013.

1.) Design of Mine openning
2.) Design of pillars
3.) Rock Support
4.) Caving
5.) Subsidence
6.) Slopes
7.) Rock Burst
8.) Load and Deformation Monitoring

Mine opening:-

Any opening, excavation or structure above or below the ground is a disturbance (from a ground control point of view) to the rockmass. This disturbance is significant.

1.) First type disturbance in rocks is the disturbance of insitu stress flux. in the original orientation

~~Depending on the extent of disturbance w.r.t. the ability of rock mass to resist the disturbance~~

Mine opening
- narrow
- wide

These types behave totally differently. wide is less stable than narrow openning.

The terms have nothing to do with size. They are influenced by structure of rocks in which the openings are made.

The same opening can be narrow in one type of rockmass and wide in another type of rockmass.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

In Case of Single openings:-

narrow opening is that type of opening which is not influenced by the presence of joint structure existing in the rockmass. In contrast to this. In case of wide opening the stability or performance of the opening is essentially affected by the presence of joint structure already existing in the rockmass.

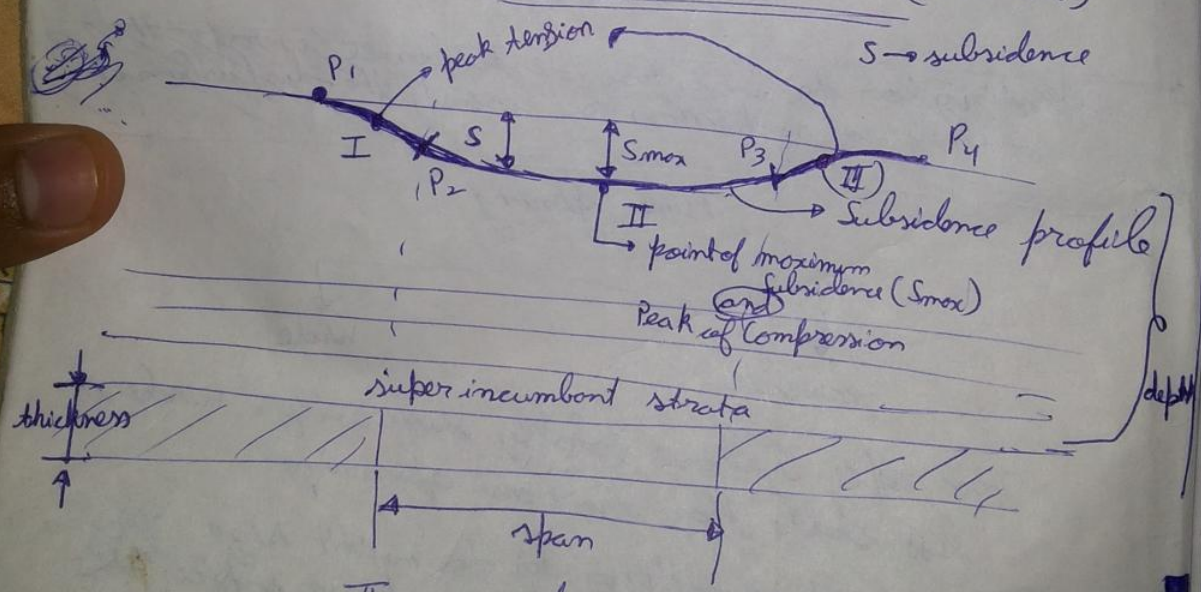
In case of multiple openings:-

an opening is treated as narrow opening only when it is neither affected by the presence of joint structure already existing in the rockmass or by the presence of other structure that has been introduced by mining operations.

If one structure affects the other structure presence then it is wide opening.

29/07/2013

Mine Subsidence (BKS Sir)



The ground movement of the surface (vertical or horizontal) is called mine subsidence

This profile is made up of three curves. (subsidence profile)

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

When seen from above:-

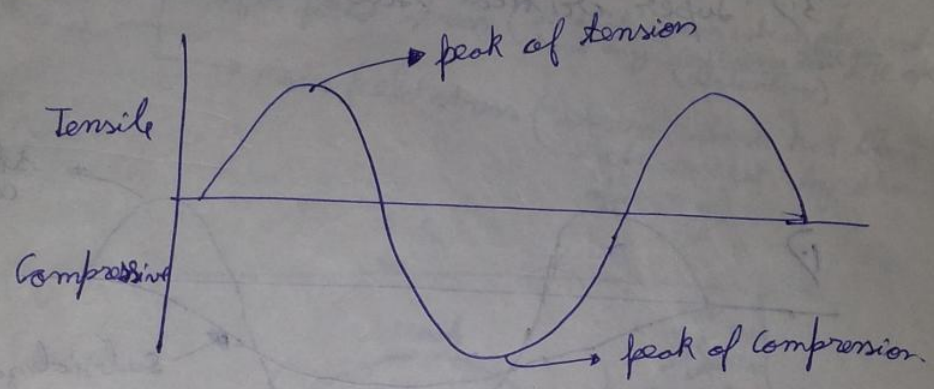
(I) → Convex curve

(II) → Concave curve

(III) → Convex curve

(III), (I) → tensile strain

(II) → Compressive strain



P_1, P_2, P_3, P_4 → Inflexion points of zero strain

Rock is very weak in tension and peak of tension lies outside the panel area so damage is seen more outside the panel area than within the panel area.

The inflexion points P_2, P_3 where there is change from tensile to compressive strain lie directly above panel and slightly inside or slightly outside the compressive strain curve.

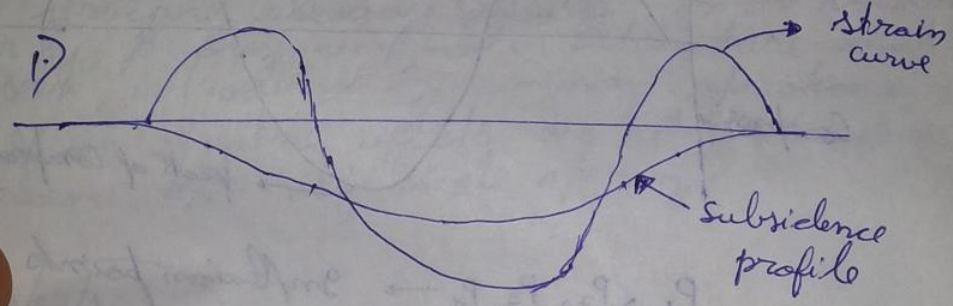
Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

⇒ When span is increased subsidence increases ~~and~~ and the limit of subsidence is the seam thickness but it is never the case.

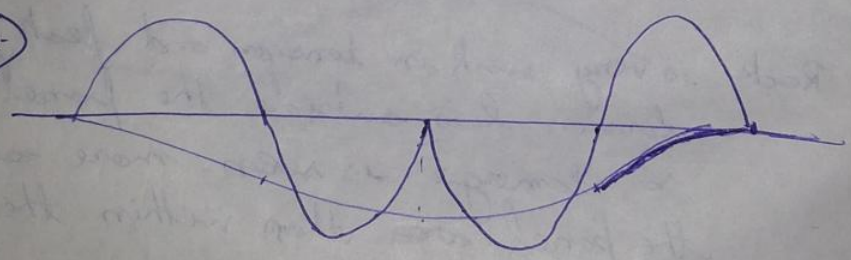
Classification of Subsidence :—

1.) Sub-critical subsidence
2.) Critical subsidence
3.) Super critical subsidence

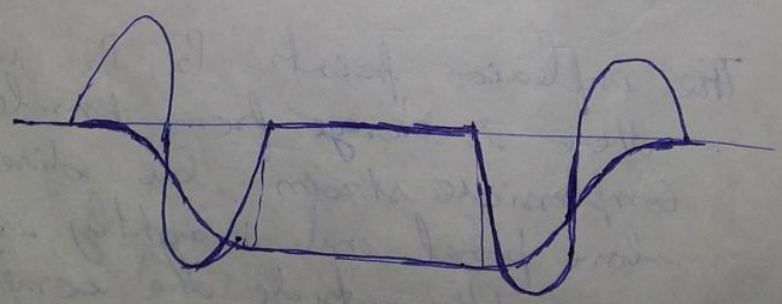
1.)



2.)



3.)



Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

(GSPS Sir) 01/08/2013

Stresses around Opening

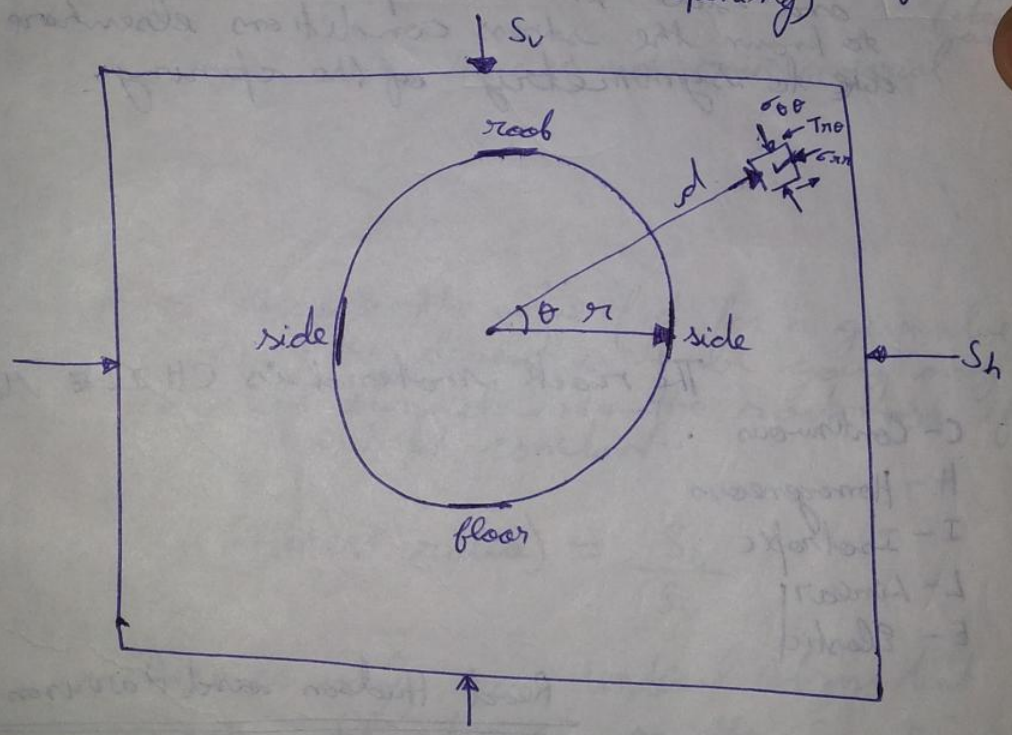
Circular opening

Driven in a rockmass with a vertical stress S_v and horizontal S_h.

S_v and S_h are far field stress (stress are acting at infinite)

Stress — far field stress (is that stress which does not have any influence (disturbance) due to the opening)

— near field stress (stress which has been influenced by the opening)



Both S_v and S_h are compressive in nature

The radius of the opening which demarcates the size is r

We are trying to evaluate the influence of the opening on a rockmass at a distance d from the center of the opening

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

we are calculating that due to the introduction of the opening to what extent the stress have been affected.

we have 3 components of stress:-

$\sigma_{\theta\theta}$ → Tangential stress

σ_{rr} → ~~Cor~~ radial stress

$\tau_{r\theta}$ → shear stress

→ Rather than focussing on the entire periphery of the opening we can focus on ~~f~~ roof and side and extrapolate the ~~to~~ results to know the stress conditions elsewhere due to symmetry of the opening.

The rock material is CHILE Material.

C - Continuous
H - Homogeneous
I - Isotropic
L - Linear
E - Elastic

Read Hudson and Harrison Vol-I

Kirsh Solution equations

$$\sigma_{rr} = \left(\frac{S_h + S_v}{2}\right)\left(1 - \frac{r^2}{d^2}\right) + \left(\frac{S_h - S_v}{2}\right)\left(1 - \frac{4r^2}{d^2} + \frac{3r^4}{d^4}\right)\cos 2\theta$$

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

$$\begin{cases} \sigma_\theta = \left(\frac{S_h + S_v}{2}\right)\left(1 + \frac{r^2}{d^2}\right) - \left(\frac{S_h - S_v}{2}\right)\left(1 + \frac{3r^4}{d^4}\right)\cos 2\theta \\ \tau_{r\theta} = \left(\frac{S_h - S_v}{2}\right)\left(1 + \frac{2r^2}{d^2} - \frac{3r^4}{d^4}\right)\sin 2\theta \end{cases}$$

These set of formulae are known as closed form solution for studying stresses around a circular material.

Minimum value of $d/r = 1$ because for $d < r$ the area is a void/constructed area. There is not rock material.

As we increase the value of d for a given value of r, the element will be going away from the tunnel into the rockmass if θ is kept constant.

$$K \text{ (Stress ratio)} = \frac{S_h}{S_v}$$

There is only one vertical stress but 2 horizontal stresses so the S_h we are considering here is the mean horizontal stress.

$$\left(S_h = \frac{S_{h\,max} + S_{h\,min}}{2} \right)$$

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

$\underset{MPa}{S_v} = \gamma H$ → cover depth (m)

γ → unit weight of rock. MPa/m

$\boxed{Pa/m \equiv N/m^3}$

γ → the amt of the weight the ~~ground~~ rock ~~will have~~ having 1 m^2 surface area of unit thickness

$\gamma = 0.025$ for practical purposes but depends on the density of rock.

we can have value of $\underbrace{K > 1}_{\text{horizontal stress} > \text{Vertical stress}}$ or $\underbrace{< 1}_{\text{horizontal stress} < \text{vertical stress}}$ or $K = 1$

If this condition is encountered then while designing it has to be taken into consideration. This is a very special case. Otherwise it will create safety and maintainance problems throughout the life of the tunnel

if $S_v = P$ then $S_h = KP$

$\frac{\sigma_{rr}}{P}$, $\frac{\sigma_{\theta\theta}}{P}$, $\frac{\sigma_{r\theta}}{P}$

This gives us the extents of intensification of stress concentrations

if ratio $> 1 \Rightarrow$ intensified.

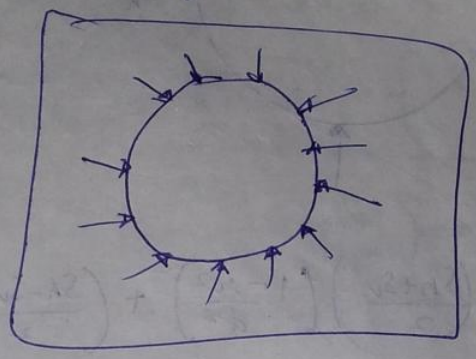
if ration $< 1 \Rightarrow$ reduction in concentration

Lab	K value
1	0.1
2	0.2
3	0.5
4	0.7
5	0.8
6	1
7	1.2
8	1.4
9	2
10	2.5
11	3
12	5

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

when K = 1 → This type of stress condition is known as hydrostatic stress condition.

if K = 1 then the amount of Far field stress that the opening is subjected to is equal.

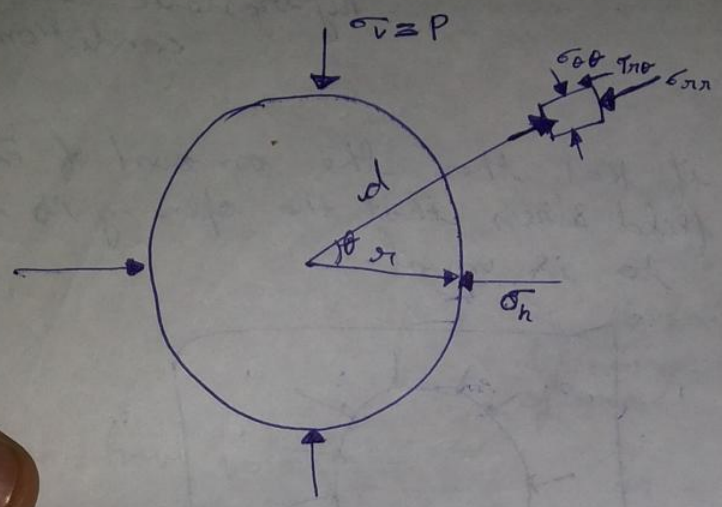


Lab	K values
1	0.1
2	0.2
3	0.5
4	0.7
5	0.8
6	1
7	1.2
8	1.4
9	2
10	2.5
11	3
12	5

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

02/08/2013

GSPS Sir



$$\sigma_{rr} = \left(\frac{S_h+S_v}{2}\right)\left(1-\frac{r^2}{d^2}\right) + \left(\frac{S_h-S_v}{2}\right)\left(1-\frac{4r^2}{d^2}+\frac{3r^4}{d^4}\right)\cos 2\theta$$

$$\sigma_{\theta\theta} = \left(\frac{S_h+S_v}{2}\right)\left(1+\frac{r^2}{d^2}\right) - \left(\frac{S_h-S_v}{2}\right)\left(1+\frac{3r^4}{d^4}\right)\cos 2\theta$$

$$\tau_{r\theta} = \left(\frac{S_h-S_v}{2}\right)\left(1+\frac{2r^2}{d^2}-\frac{3r^4}{d^4}\right)\sin 2\theta$$

Case - I :-

If $S_h = S_v$ (Hydrostatic condition)

$$\sigma_{rr} = S_v\left(1-\frac{r^2}{d^2}\right)$$

$$\sigma_{\theta\theta} = S_v\left(1+\frac{r^2}{d^2}\right)$$

$$\tau_{r\theta} = 0$$

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

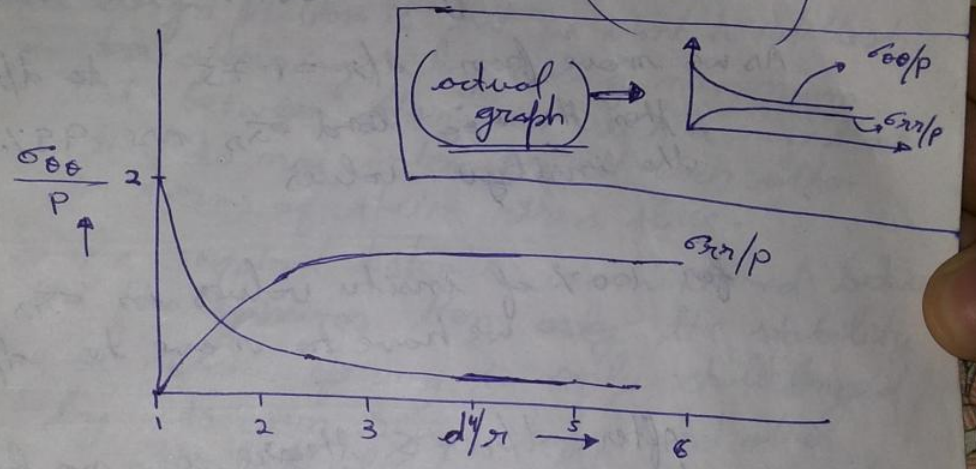
if $r = 2m$ $\quad 2 \leq d \leq 10$

at $\theta = 0$

	$d=2$	$d=4$	$d=6$	$d=8$	$d=10$
$\sigma_{\theta\theta}$	2P	1.25P	1.111P	1.0625P	1.04P
σ_{rr}	0	0.75P	0.889P	0.9375P	0.96P

($\sigma_{\theta\theta}$ (↓) at a ~~non~~ variable ~~constant~~ rate)

(σ_{rr} (↑) at a variable rate)



$\frac{d}{r}$ → indicates how far our rock element is from the periphery of the tunnel.

⇒ The results are independent of the value of θ

At the surface :-

⇒ radial stress is zero at all points on the surface

⇒ tangential stress is 2 times the virgin stress (i.e. the vertical stress)

⇒ In hydrostatic condition only.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

As we go into the rockmass the value of $\sigma_{\theta\theta}$ decreases but with a variable rate and the value of σ_{rr} increases with a variable rate

As we exceed the points of $d/r = 1.75$ the value of $\sigma_{\theta\theta}$ or σ_{rr}, then they are achieving almost the insitu value.

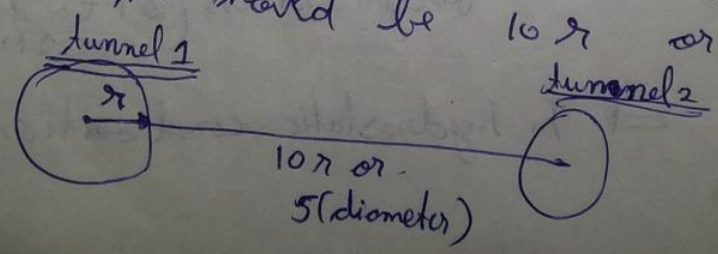
As we move from $d/r = 1.75$ to $d/r = 3$ then the $\sigma_{\theta\theta}$ and σ_{rr} are 99% of the insitu values

for 100% of insitu values in σ_{rr} and $\sigma_{\theta\theta}$ we have to move to $d/r = 5$

After $d/r = 5$ there is no further chances of saturation in the values of σ_{rr} and $\sigma_{\theta\theta}$

In Hydrostatic conditions only:-

If we want to have two companion excavations which remain in proximity of each other without any influence on each other then the minimum distance between the two should be $10r$ or 5 (diameter)



Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Zone of influence → the dist away from the surface of the structure upto which the original state of stress is likely to be disturbed.

The Concept of zone of influence is very important for mine design because it may provide considerable simplification of a design problem.

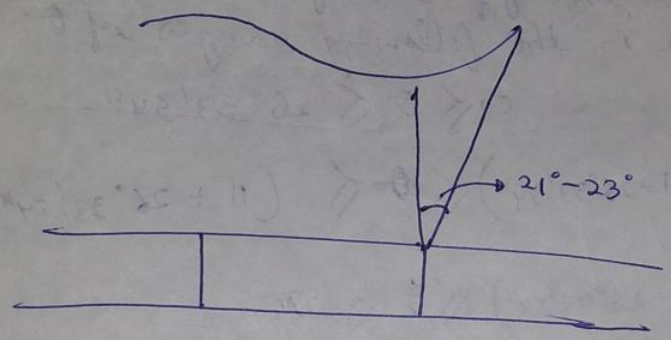
Our design problem is that we want to establish 2 ~~two~~ openings by having minimum dist. between them in such a way that they do not interact with each other in terms of their stress flux.
So if a required distance is maintained between the two openings then ~~only~~ the stability of each opening is only challenged by its own size, shape and other attributable parameters and not by the size or shape of the nearby opening. Then this design problem will become simple.

➡ Our mining structures are not simple openings because they have influence on each other.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

12/08/2013. BKS Sir (GROUND CONTROL)

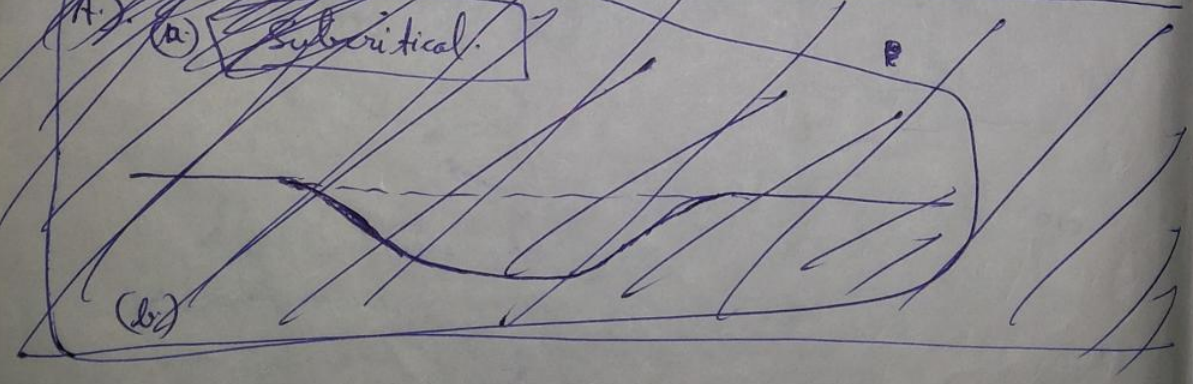
→ angle of the line from edge of the working to the point on the surface having negligible or zero subsidence measured in a vertical plane is called Angle of of Draw



Q.) Draw 2 figures (1 profile, 1 strain) as the span is increasing from sub critical to super critical area.

~~1) Profiles:~~

~~(A.) (a.) Subcritical.~~



~~(b.)~~

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

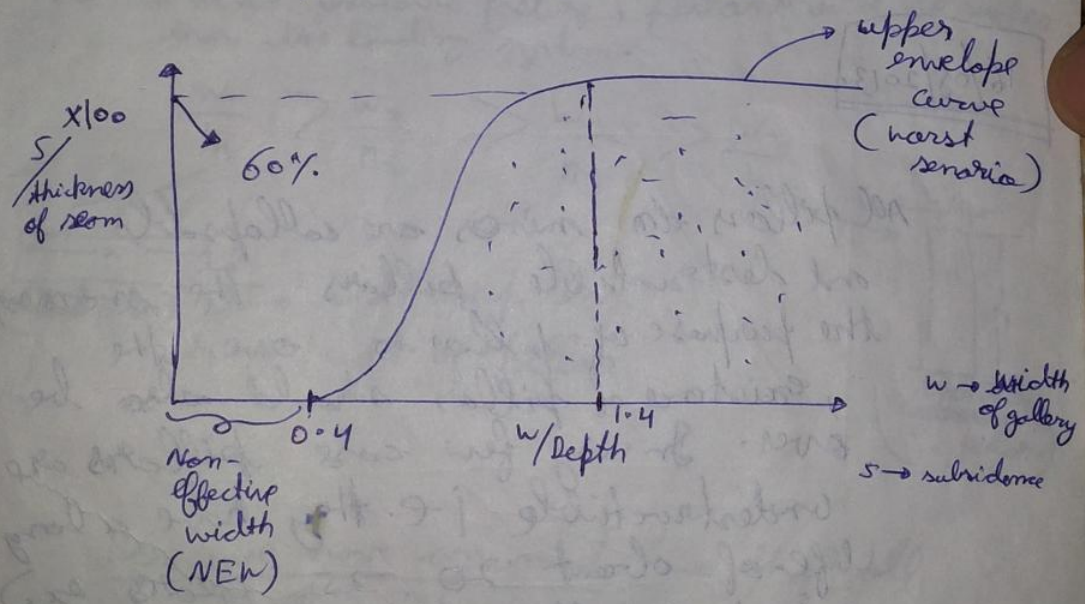
Prediction of Mine Subsidence:-

→ During Mining there is always percussion on the surface. So we have to take care of subsidence so that surface features (roads, rails, houses etc) are protected.

→ Due to subsidence cracks develops and air breathes from these cracks and coal seam gets oxidized and fire is set up.

(i) depth → less then more subsidence
(ii) span → more then more subsidence
(iii) thickness → more thickness of seam then more subsidence

→ subsidence depends on these parameters



if W/d < 0.4 → then ~~subsi~~ subsidence is almost negligible.

if W/d > 1.4 → then supercritical subsidence takes place

~~we can~~ if we know that there is to be supercritical subsidence then we can plan accordingly. In supercritical there is no cracks as strain becomes zero.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

This the empirical method based on field data. The upper envelope curve gives the idea of NEW.

At 60% of $\frac{\text{subsidence}}{\text{thickness}}$ ratio we have the maximum subsidence (supercritical)

But by this method no relationship could be developed between subsidence and (depth, span, thickness)

Profile function method:-

Next Class.

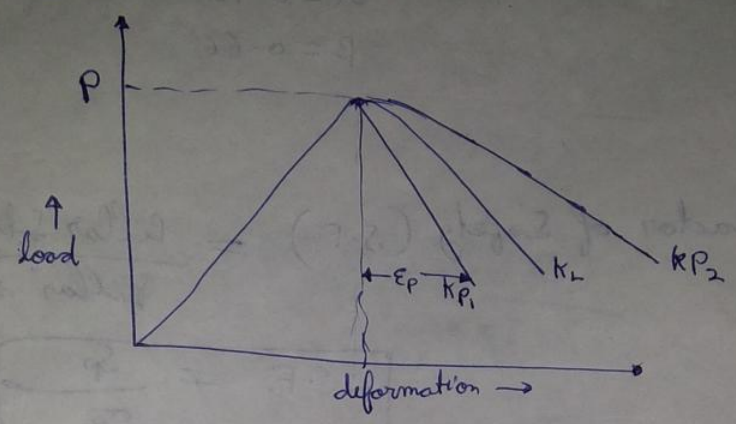
16/08/2013.

All pillars in mines are collapsable and destructible pillars. The moment the purpose of pillar is over the existance of pillar should also be over. In very few case pillars are undestructible i.e. they have a long life of about 20 – 25 years ex. shaft pillar. There is complexity in designing a pillar which undergoes a failure after certain time interval.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

To understand the failure mechanism of the pillar the following cases may be considered.

~~Case I~~



K_L → Post failure stiffness of the LOADING SYSTEM

K_P → Post failure stiffness of the PILLAR

Consider 2 pillars with stiffness K_{P_1} and K_{P_2} under the same loading system which has a stiffness K_L. The stiffness of the pillars and loading system are related as :-

$$K_{P_1} > K_L > K_{P_2}$$

with the w/h ratio of the pillar being in the order $\frac{W_1}{h_1} < \frac{W_2}{h_2}$

The loading system undergoes deformation as result of resettlement of the ~~initially~~ stresses and this deformation loads the pillars. By allowing deformation the loading systems stores strain energy in it but only upto the yield point (Since it is not perfectly elastic). When it fails the stored energy is released. (The energy hence released is in order to attain a stable lower energy state) The energy released on complete failure of the loading system is transmitted to the pillars.

Let

A_P be the capability of the ~~pillar~~ either pillar to accomodate strain energy.

A_L be the potential of the loading system to transfer energy.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

~~Th~~ A_P and A_L can be formulated as:-

$$A = \frac{1}{2} \times P \times \varepsilon_P$$

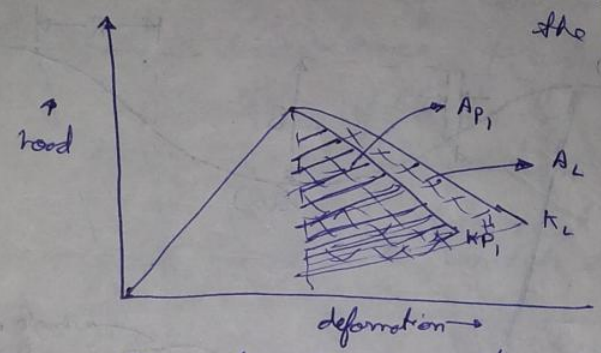
where P = yield point load

ε_P = Plastic strain

A is the area under the post failure deformation curve.

Case 1:-

$K_{P_1} > K_L$ (i.e. the pillar is stiffer than the loading system)



~~Form~~ From the load-deformation characteristic it is clear that $A_L > A_{P_1}$ which thereby ~~bro~~ brings us to the conclusion that:-

~~The pillar designed has~~

The loading system has higher potential transfer capacity (A_L) ~~and~~ while the pillar has a lower energy accomodating capability (A_{P_1}). So the excess energy ~~is released~~, when released, will be turbulent thus resulting into a violent or uncontrolled failure of the pillar.

Case 2:- $K_{P_2} < K_L$ (i.e. the pillar is softer than the loading system)

From the curve:- $A_{P_2} > A_L$

The conclusion here is opposite to the previous case and hence even after the loading systems transfers all its energy to the pillar, there is still ample scope in the pillar to accomodate more energy. This can be seen as a gradual failure of the pillar.

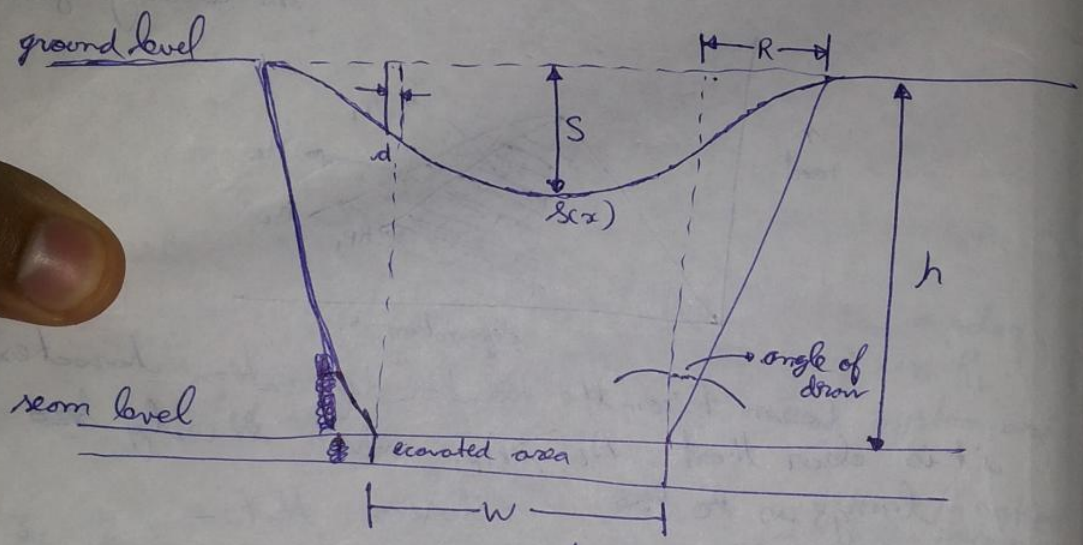
Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

So we can conclude that with the a lower stiffness of the pillar and hence with a higher w/h ratio we can plan for a design of the pillar with gradual or controlled failure with a higher degree of stability.

19/08/2013.

GROUND CONTROL (BKS Sir)

PROFILE FUNCTION METHOD:-



w → span of working

S → max. subsidence on the surface

h → depth of mine

R → horizontal distance between the edge (rib edge) of working to point of negligible subsidence

d → dist between transition (going from compression to tension) point to rib edge (edge of working)

s → subsidence in all other points (varies with x)

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

$$s = f(S, x, R)$$

x → horizontal dist from origin

If the extraction is subcritical then another point $\left(\frac{w}{h}\right)$ also has to be taken into account.

$$s = f\left(S, x, R, \frac{w}{h}\right)$$

A profile function simply defines the shape of the curve and it requires a max. value of subsidence and a no. of constants to calculate the subsidence profile. The method is generally applied by adjusting the constants in the equation until a satisfactory shape is obtained which provides an acceptable match to the observed data.

Consequently using this ediberated equation, it should then be possible to predict subsidence curves for that particular locality.

Profile functions has 2 main advantages:-

1.) As it is defined by an equation it can easily be calculated with the help of a calculator / computer.

2.) When very little measured data are available the method can still be applied using limited data to calculate constants

Disadvantages:-

1.) The profile function shape depends on various external parameters that are not taken into account by the function and it may lead to an error in the shape.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

2.) The profile function may not match exactly the shape ~~given~~ of ~~taking~~ that given by measured data and consequently introduce an error. Such an error may not be significant but it may give significant error when tilt or curvature are calculated from the subsidence profile.

ex.) Hyperbolic function given by King and ~~Kitton~~ whetton :-

$$s = \left(\frac{S}{2}\right)\left\{1 - \tanh\left(\frac{2x}{R}\right)\right\}$$

Based on :- Transition point origin

ex.) By Kumar, Saxena and Singh (1983) (Indian profile function)

for subcritical

$$s = S e^{-\frac{nx^2}{P^2 - x^2}}$$

for critical

$$s = S e^{-\frac{nx^4}{P^4 - x^4}}$$

$n \rightarrow$ constant (1–5)

$x \rightarrow$ distance from extraction center

$R \rightarrow$ radius of critical area of extraction

P = half width of subsidence profile $\left(\frac{W}{2} + R\right)$

This is based on face center origin (center of the subsidence curve)

Assignment 1:-

Search various profile function methods and explain any one profile function method.

NOTE → Strain can be calculated using profile function for a particular location by relating strain to the curvature. And this curvature is the 2nd derivative of the subsidence (s) equation

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

GROUND CONTROL (GSPS Sir)

Design of Pillars (Prediction of width of pillar for a particular extraction height)

- This is a hit and trial method that can be easily applied for square pillars.

If we have a value of K (strength constant)

K = strength of 1 cubic inch sample

then we can easily design the pillar for a particular factor of safety (S.F.) and Extraction height (h).

$$S.F. = \frac{S_p}{\sigma_p}$$

$$S_p = K \frac{W^{\alpha}}{h^{\beta}}$$

$\alpha, \beta \rightarrow$ empirical constants.

$W \rightarrow$ solid width of pillar.

For indian Coalfields:- $\alpha = 0.46$; $\beta = 0.66$

$$\sigma_p = \frac{0.025 H (W + W_b)^2}{(W)^2}$$

$W_b \rightarrow$ width of bord (gallery width)

$$S.F. = \frac{K W^{\alpha} \quad (W)^2}{h^{\beta} \times 0.025 \times H \times (W + W_b)^2}$$

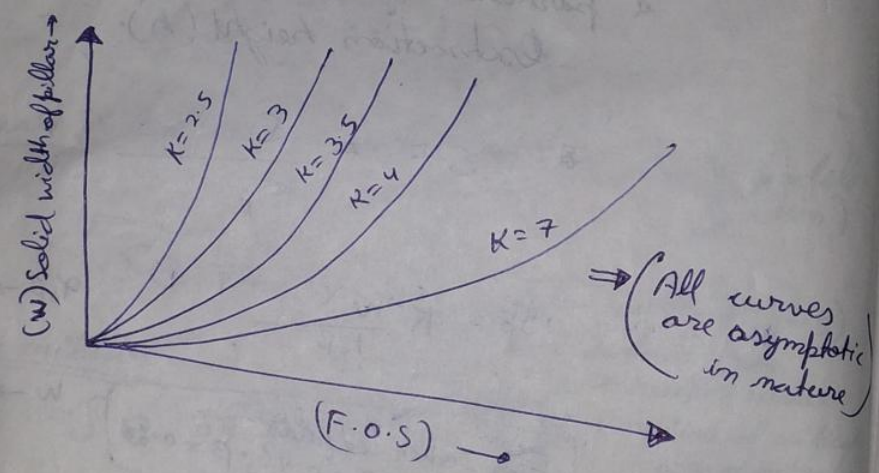
$$\therefore S.F. = \frac{K (W)^{2+\alpha}}{0.025 \times H \times h^{\beta} (W + W_b)^2}$$

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

For a particular S.F. and K (strength constant) and having the values of w_b, h, H, α, β we would have a non-linear equation in (w) which can be solved by hit and trial values of w.

This is an easy approach to guess the the width of pillar for a fixed K (strength constant) and S.F.

Now for the same values of K if the pillar is designed for different (F.O.S.) then we see that:



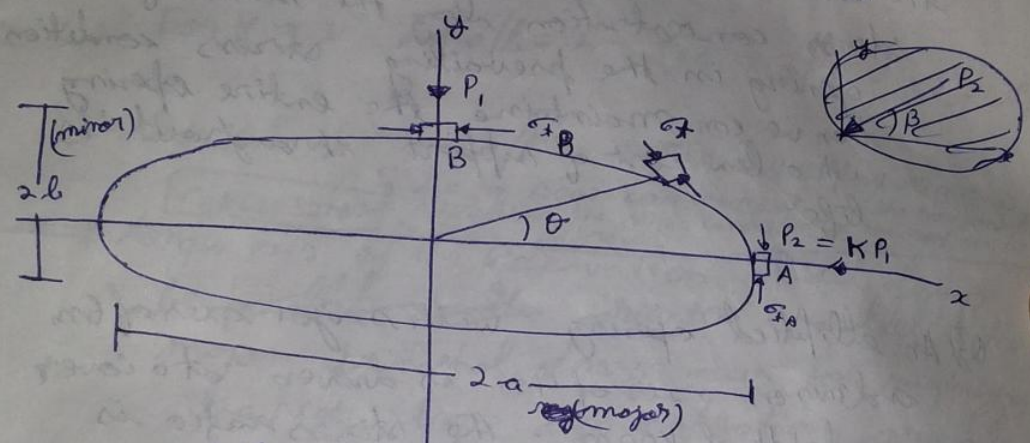
Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

22/08/2013

GROUND CONTROL (FSPS)

Stress around elliptical shaped tunnel:-

for an ellipse → the diameter along 2 perpendicular axis is different.



The radial stress along roof and sides is zero

P_1, P_2 → Vertical and horizontal principle stress (if opening is horizontal)
(principle stress components.)

P_1, P_2 → 2 components of horizontal stress (if opening is vertical)

The tangential stresses at point A on the side of the elliptical shaped tunnel and at point B in the roof are:-

Closed form solution for elliptical opening.

$$\sigma_{tA} = P_1\left(1 - K + \frac{2a}{b}\right)$$

$$\sigma_{tB} = P_1\left(K - 1 + \frac{2b}{a}K\right)$$

NOTE → By changing the orientation of the ellipse by 90° the above formulae remain same except that the ratio of a/b get inversed.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

⇒ Some areas along the periphery may have a high induced stress concentration and may undergo failure. So such areas have to be highly supported throughout the life of the opening.

But

⇒ If we have an opportunity to ~~do~~ modify the orientation of the opening to relax the stress concentration along the sides of the opening in the prevailing stress condition then we can maintain the entire opening with a low cost of support throughout its life

Q.) An elliptical opening with major axis of 6m and minor axis of 4m is driven at a cover depth of 400m. The stress ratio is (0.4). Suggest a suitable orientation of the opening under the given circumstances.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

In case of uniaxial stress direction (assuming $K=0$) and $A/b = 2$ we have ; P_1 is normal to the longer axis of the ellipsoidal shape of the tunnel:-

$$\sigma_{tA} = 5P_1 \quad ; \quad \sigma_{tB} = -P_1 \text{ (tensile stress)}$$

- For P_1 normal to the shorter axis, they are

$$\sigma_{tA} = 2P_1 \qquad \sigma_{tB} = -P_1$$

For $K = 2$ and $a/b = 2$ $\sigma_{tA} = 3P_1$; $\sigma_{tB} = 3P_1$

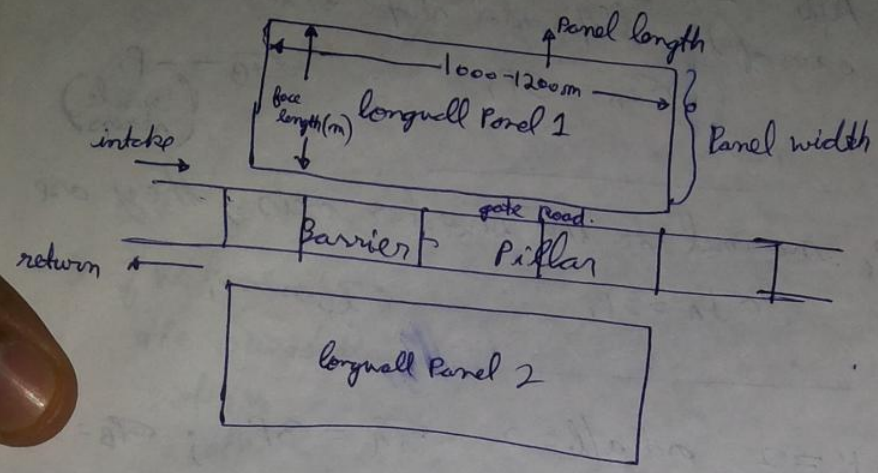
For $K = 0.5$ and $a/b = 2$ $\sigma_{tA} = 4.5P_1$; $\sigma_{tB} = 0$

CONCLUSION:- due to some confinement in the horizontal direction the stress condition at the roof can be improved.

⇒ Longer axis of the Excavation should always be ~~parallel to the~~ oriented parallel to the major principal stress axis for minimizing the high stress concentration around the excavation

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Chain Pillar:-



(90-160m) face length is much smaller compared to the width

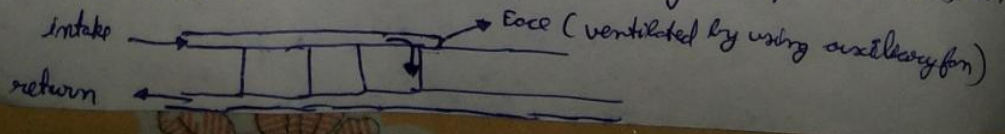
abroad { longest longwall face = 350m
longest panel length = 3500m

Panel is developed by driving gate roads along the boundary of the Panel.

restriction { max. width of gate road = 4.8m
max. height of gate road = 3m

Due to large panel length, such long blind heading cannot be driven without ventilation so there is a requirement for splitting of the barrier pillars but (splitting should be minimum)

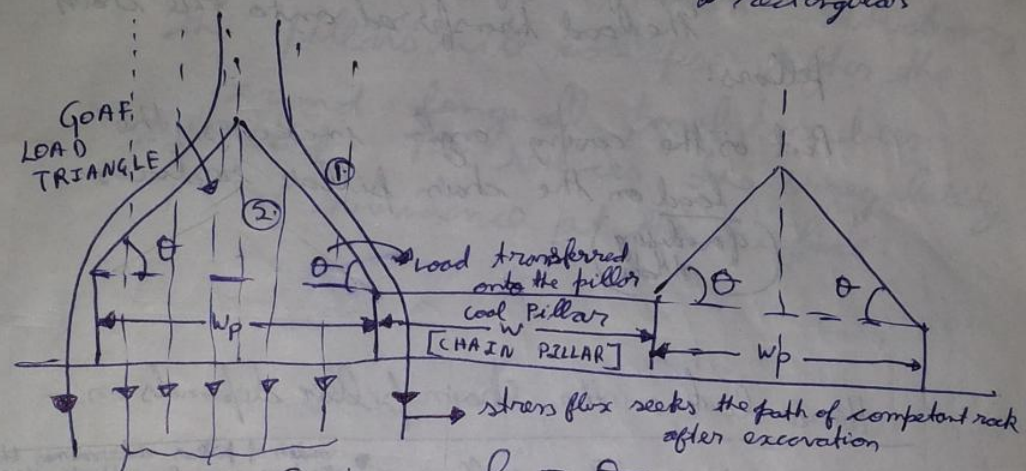
These pillars are called chain pillars because their length is quite large compared to their width.



Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

we can have:-

- Single row of chain pillar. → square / → rectangular
- Double line of chain pillar → square / → rectangular



Caving angle = θ

(direction of stress flux in unexcavated rock)

W_p → width of panel + width of gate road

Rockmass (1) is very very compacted.

Rockmass (2) is very loose rock.

Initially soft capability to accomodate stress is more of material (1) than material (2) due to looseness of material (2)

So majority of the load on the goaf material is shared by the coal pillar but not entirely initially

But later on, on compaction of goaf material the load bearing capacity of goaf increases and it too takes a major amount of load of the goaf roof. Rest is shared by the pillars.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

⇒ Hence the chain pillar is ~~Am~~ more loaded than development pillars in B and P mining due to the large width of the panel.

If we consider the Caving angle → As the caving angle reduces the overhang of the goaf roof is more so more is the load transferred onto the chain pillars.

But as the caving angle increases the load (of overlying strata) on the chain pillar reduces.

Hence load on the chain pillar depends on:-

fundamental parameters
1.) width of pillar → width of pillar determines the effective area for supporting the roof. As width of pillar ↑ → stress decreases
2.) width of Panel → determines the span of the goaf which is to be supported by the pillar on complete extraction of adjacent ~~pillars~~ panels.
3.) Cover depth of seam → vertical stress increases with cover depth so does the load on the pillar
4.) Caving angle of the goaf → As the caving angle decreases a greater amt of load is transferred onto the chain pillars.

other parameters
5.) Strength of Coal seam
6.) Stiffness ratio of coal pillar w.r.t stiffness of loading system

⇒ These affect loading and stability of chain pillars

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

23/08/2013.

GSPS Sir

CHAIN Pillar Design

- Once the second adjacent longwall panel has been extracted, the chain pillars ~~are~~ subjected to extremely high strata loads. In most situations, the pillars will collapse after the second longwall panel has been extracted. This reduces the irregularity of subsidence at the surface.

⇒ Till adjacent panel ~~the~~ is not ~~extracted~~ extracted the load on chain pillar is very near to the in situ value.

exceptions → In 1994 we had started working a longwall at the depth of 180 to 200 m. The extraction ht was 4.2 m. Panel 1 was of a length of 150 m. The width of chain pillar was 34 m.

In the major period of extraction we experienced spalling (failure from side of pillar) and as the face was advanced by 800 m then the spalling was upto 3 m (i.e. $W_{gallery} = 7.8$ m).

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

dependence Bending stress of the roof of gallery is $\propto W_{gal}^2$

NOTE:-

Putting inadequate support is more dangerous than putting no support.

Our knowledge about the rational requirement of the support is very important.

But this spalling was observed locally (at some points)

Realizing the problem the next face length (bond 2) was reduced to 120m

Fortunately the panel extraction was successful.

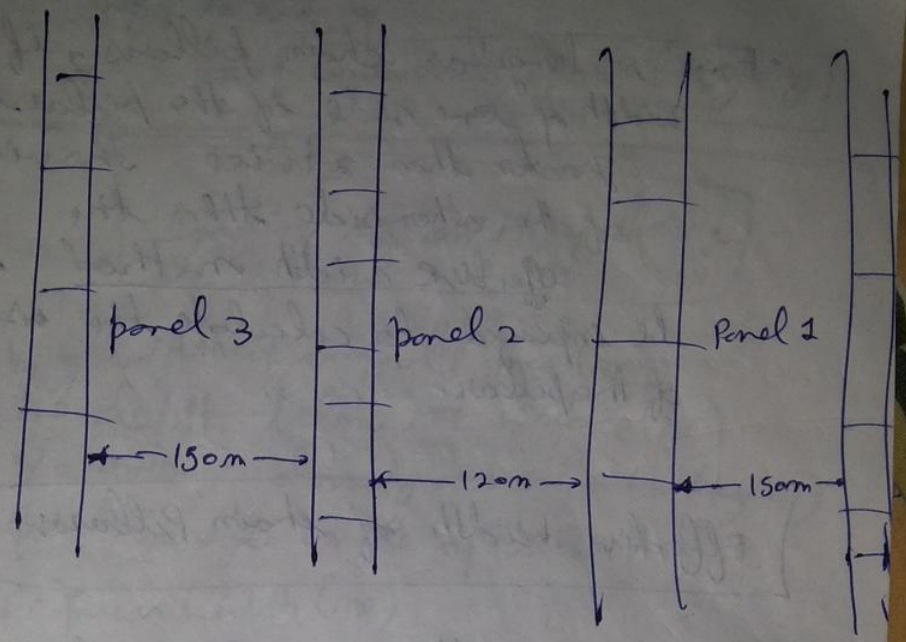
This resulted the case of over confidence and the next panel was again at a face length of 150m.

But when this panel face was advanced by 700 to 800m, supports from 1 to 80 suffered mechanical damage (hydraulic system failed), spacing between the support was 1.5 m. Hence a length of $(80 \times 1.5) = 120$m face length failed.

The major reason of failure was credited to improper design of the chain pillars.

The accident took place in the year 1997

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in



Strength of pillars:-

(Salamon Munro formula)

$$S_p(MPa) = \frac{7.2\, p^{0.46}}{m^{0.66}}$$

7.2 → K (size specific value)

p = pillar width (m) (solid width) m = pillar height (m)

Main pillar are generally rectangular so that the no. of splittings in the Barrier pillar is less.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

• For rectangular chain pillars, if the width of one side of the pillar is greater than 2 twice the width of the other side then the effective width method must be employed to calculate the strength of the pillar.

Effective width of chain Pillars:-

The effective width of a large rectangular chain pillar can be calculated by using either of the formulae:-

$$W_{eff} = (W_1 \times W_2)^{0.5}$$

$$W_{eff} = 4A/C$$

W_1 = Width of one side of the chain pillar, smaller dimension (m)

W_2 = width of other side of chain pillar, larger dimension (m)

A = Area of the pillar (m^2)

C = Circumference of the pillar (m)

h = height of the pillar (metres)

Restrictions on the above formula:
- if $W_2 = 2W_1$, $\frac{W_1}{h} \leq 5$
- if $W_2 > 2W_1$, $\frac{W_1}{h} > 5$

$W_1 = W_2$

$W_2 = 2W_1$

here h = height of pillar

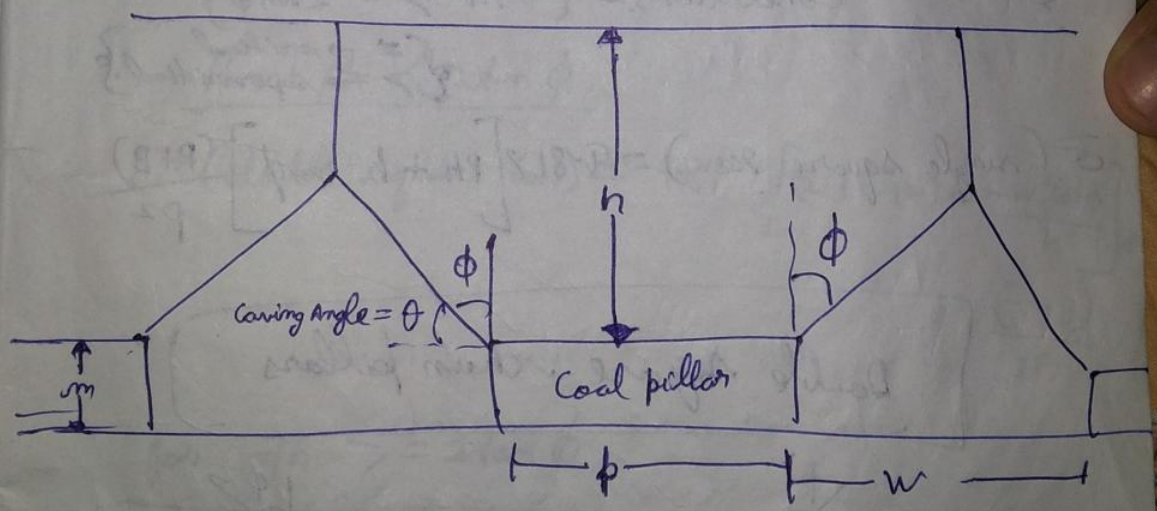
Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

refer to site of

⇒ university of Wollongong

Loading ~~of~~ on the pillar :-

W = width of longwall face (m)
h = depth below ~~the~~ surface (m)
γ = average density of overburden (kg/m³)
ϕ = average angle of shear (°)
P = pillar width (m)
B = Bord (heading) width (m) (width of gate road)



Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

depending on width of extraction w.r.t the cover depth, the different types of subsidence are:-

1.) Subcritical

2.) critical

3.) supercritical.

Single Square chain design:-

For $w/h < 2\tan\phi$ (For subcritical subsidence conditions)

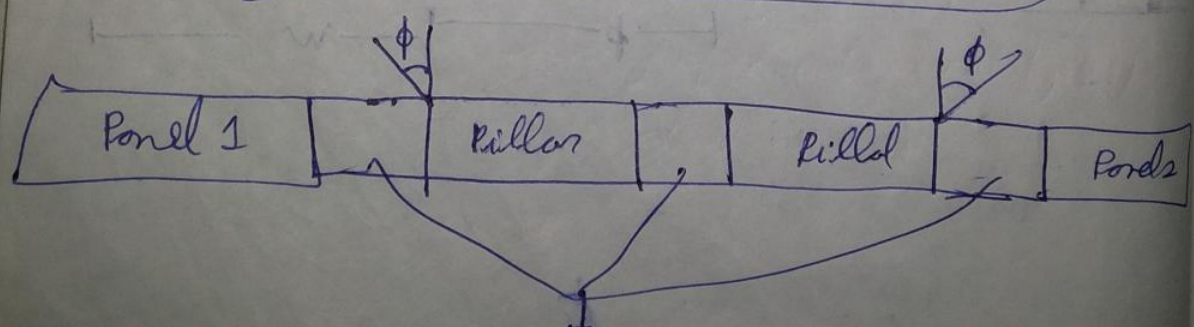
$$\bar{\sigma}\,(\text{single square row}) = 9.81\,\gamma\left[(P+w)h - \frac{w^2\cot\phi}{4}\right]\frac{(P+B)}{P^2}$$

For critical and super critical mining conditions:- ($w/h \geq 2\tan\phi$)

$\{= \rightarrow$ critical
$> \rightarrow$ supercritical$\}$

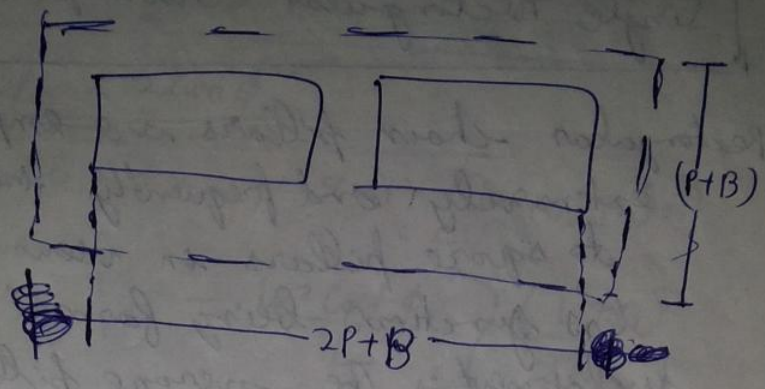
$$\bar{\sigma}\,(\text{single square row}) = 9.81\,\gamma\left[Ph + h^2\tan\phi\right]\frac{(P+B)}{P^2}$$

Double square chain pillars



(Roadway Crossection View)

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in



(Roadway Plan view)

- effective width of the chain pillar support area in this case. (2P+B) and load is borne by two pillars of a plan area $2P^2$

The respective average pillar stresses for this situation are:-

<u>for w/h < 2 tan ϕ</u>

$$\bar{\sigma}\text{ (double square row)} = 9.81\,\gamma\left[(2P+B+w)h - \frac{w^2\cot\phi}{4}\right]\times\frac{(P+B)}{2P^2}$$

<u>for w/h >= 2 tan ϕ</u>

$$\bar{\sigma}\text{ (double square row)} = 9.81\,\gamma\left[(2P+B)h + h^2\tan\phi\right]\times\frac{(P+B)}{2P^2}$$

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Single Rectangular Chain pillars:-

- Rectangular chain pillars are employed extensively and frequently in preference to square pillars in view of less junctions being formed during development. The average pillar stress is given by:-

$$\bar{\sigma}\,(\text{single rect-row}) = 9.81\gamma\left[(P+W)h - \frac{W^2\cot\phi}{4}\right]\frac{(L+B)}{P\cdot L}$$

for $W/h < 2\tan\phi$

$$\bar{\sigma}\,(\text{single rect row}) = 9.81\gamma\left[Ph + h^2\tan\phi\right]\frac{(L+B)}{P\cdot L}$$

for $W/h >= 2\tan\phi$

where:-

W = width of longwall face (m)

h = depth ~~of~~ below surface (m)

γ = average density of overburden (kg/m^3)

ϕ = average angle of shear (°)

P = pillar width (m)

L = pillar Length (m)

B = Bord (heading) width (m)

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Double Rectangular chain pillars:-

for $w/h < 2\tan\phi$

$$\bar{\sigma}\,(\text{double rec. row}) = 9.81\gamma\left[(2P+B+w)h - \frac{w^2\cot\phi}{4}\right]\frac{(L+B)}{2P\cdot L}$$

for $w/h \geq 2\tan\phi$

$$\bar{\sigma}\,(\text{double. rect row}) = 9.81\gamma\left[(2P+B)h + h^2\tan\phi\right]\times\frac{(L+B)}{2P\cdot L}$$

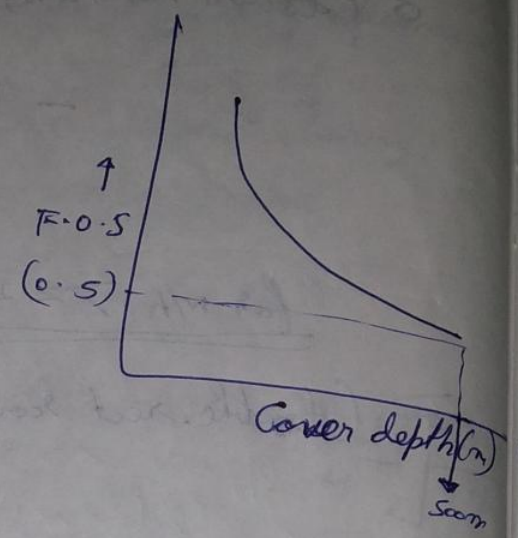
Q.) A longwall mine is planned for extracting 3.2m thick seam for its full thickness under a cover depth of 250m. The planned length of face is 150m. Single row rectangular type chain pillar is proposed with a size of 15m×35m for this purpose. The width of the gate road is 4.6m where as the strength of coal (K) = 7.2 MPa Estimate the F.O.S of the pillar when both the panels intervening Barrier pillar under consideration has been droped out.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Factors affecting Safety factor of a chain pillar :-

• Cover depth :-

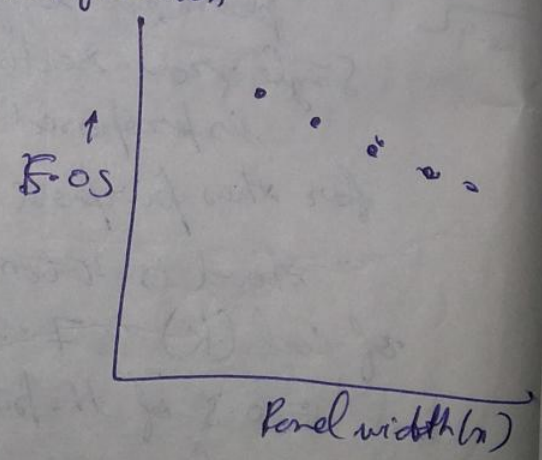
- with increasing depth of cover, the amt. of vertical stress a chain pillar is subjected to, increases.



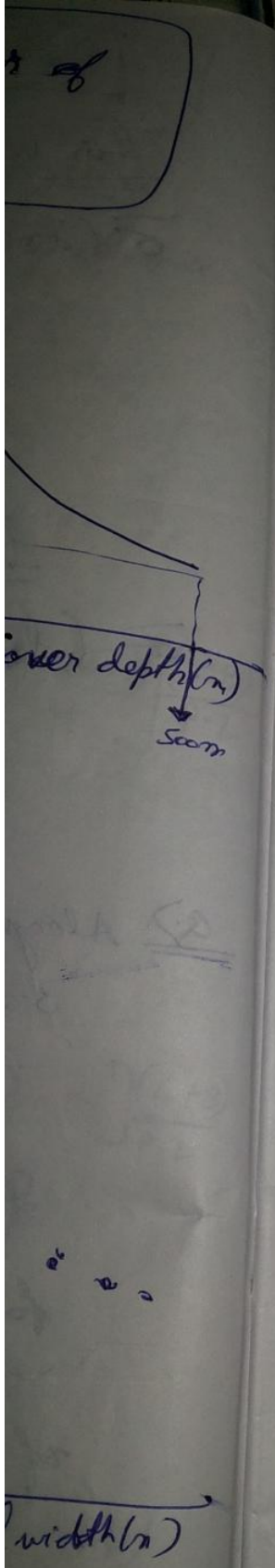
- A numerical modelling study (Chakhani, 2013) reveals that with increasing depth of cover, the S.F. of a chain pillar decreases following a power function

• Panel width :-

- The safety factor of chain pillar reduces as a non-linear function with increase in width of panel.

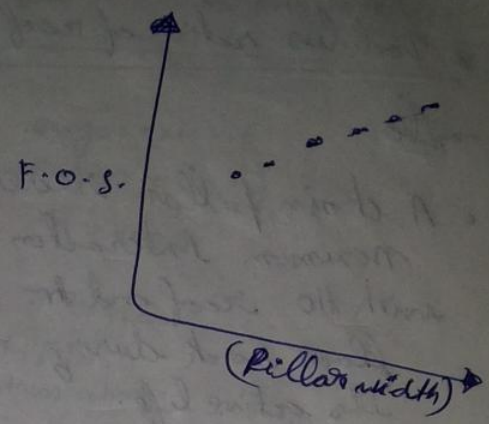


Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in



• Pillar width :-

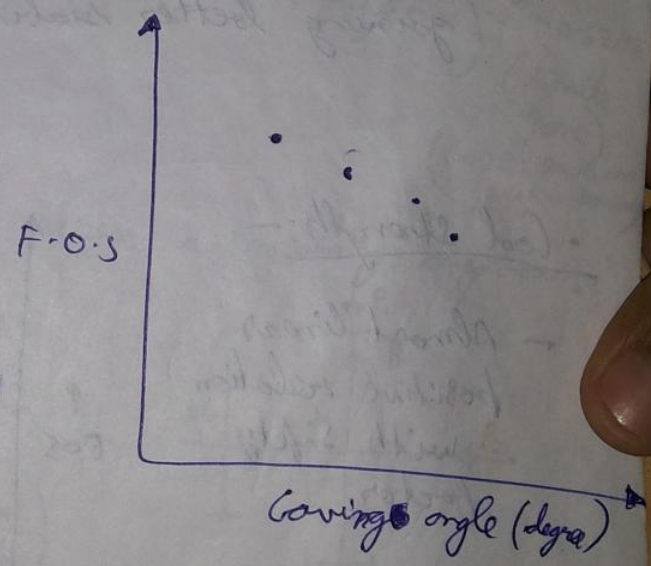
- Numerical modelling study suggests a non-linear positive function of relation between safety factor of the pillar and the width of chain pillar



• Caving angle :-

- Decides the extent of strata involved in induction of abutment loading on the chain pillar

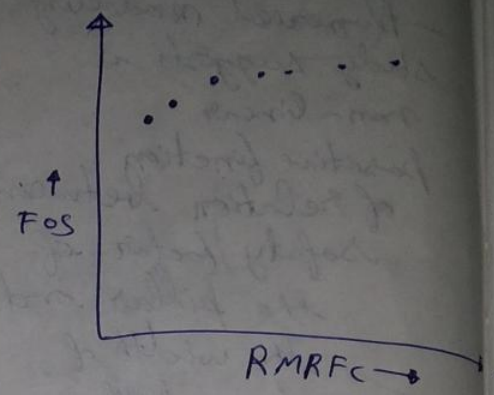
- The F.O.S of chain pillar decreases with increase in caving angle



Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Modulus ratio of roof - floor of coal seam :-

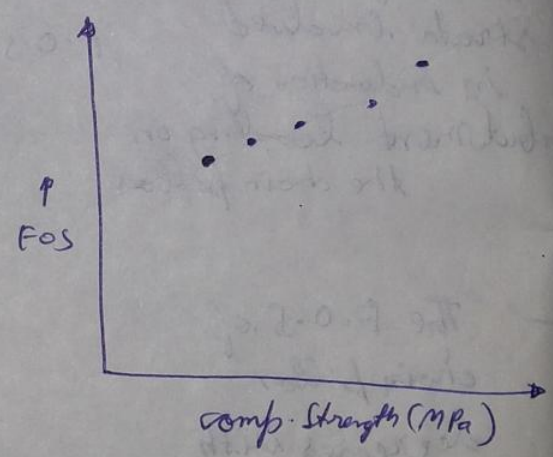
- A chain pillar recieves maximum interaction with the roof and the floor rock during its active life.



- A pillar in stiff loading environment gets better confinement from the roof and floor, gaining better stability

Coal strength :-

- Almost linear positive relation with Safety factor



Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

2/09/2013 BKS Sir

Numerical Methods:-

Important aspect for subsidence:-

1.) Depth of seam
2.) span of working
3.) Thickness of seam

⇒ Physico-mechanical properties of immediate roof controls subsidence to a large extent.
So we have to consider the physico-mechanical properties of the various layers to get the effect of ~~subsi~~ caving on the surface. The present methods do not consider this factor.

⇒ Geometrical features (thickness, etc) of all beds are different so ~~to~~ stresses developed are different. This is also taken into account

⇒ Sequence of arrangment of state by nature also has a significant impact on subsidence

⇒ Interbed strength – Bonding between all layers of strata.
On bending ~~the~~ some tensile stresses are developed within beds also. ~~If~~ ~~the~~ The inter bed stress should be such as the beds should separate on failure of roof so that proper caving takes place.

⇒ After failure the void of mining is filled by broken material which gradually takes the load. So property or behaviour of the ~~broo~~ broken material is ~~not~~ necessary

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

we establish the whole mining system from surface to seam level in a computer by a mathematical model. And by inputting various parameters of mining we get more and more accurate results of subsidence. After making the model it should be compared with a known case and reliability of the model is to be determined and then results are predicted.

Physical Simulation Method:-

- Prior to numerical modelling methods.
- The dimensions of the mine and all the physical mechanical properties and other properties and parameters were scaled down (to lab scale) and a physical model was made which was simulated to ~~yet~~ predict results of subsidence.
- But this method is expensive and highly tedious so obsolete.

6/09/2013 — **Support Design** — (GSPS Sir)

Purpose of Support:

- The structure of the design may not be self sustainable for the time it is desired ~~to be~~ for, so we require to provide proper support to the structure
- The provide safety to the mine personnel and for undisturbed production

The mining excavations are not permanent, they have limited life, so the amount of support we are putting has to be in accordance with the life of the support and the life of excavation.

- The general support requirements of productive excavations differ significantly from those

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

applicable to excavations of a more permanent nature

- The support employed in productive excavations must be simple, robust, effective, easy to install and cheap.

- The actual choice of the support system depends upon the availability of materials, economics of the venture (project) and the method of extraction

availability of material:-

good quality timber is now difficult to find so we shifted to friction & hydraulic supports.

economics of venture:-

The cost of support always adds to the production cost of the mine and affects profit margin.

mining method:-

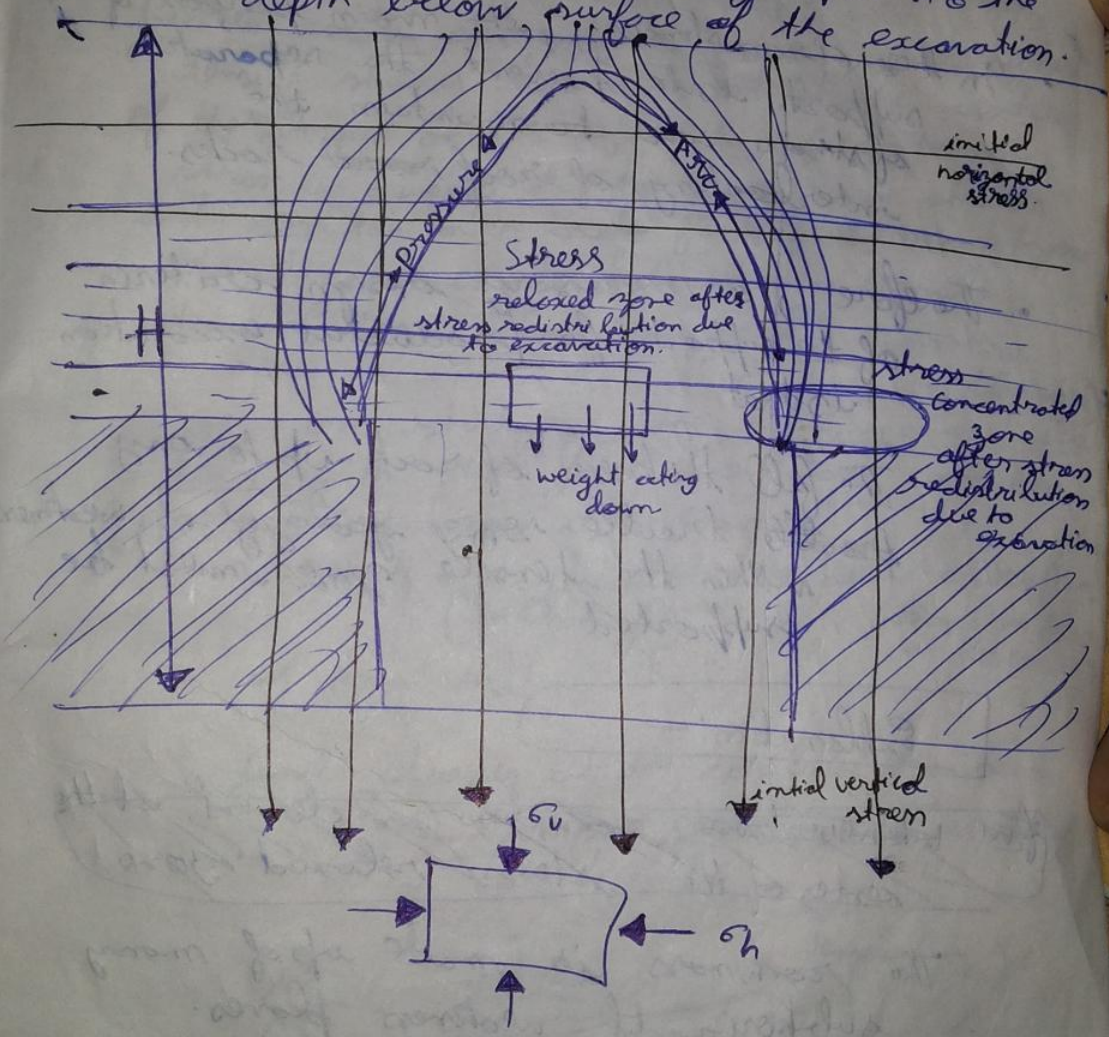
when we had manual method of mining, the span of gallery could be less ~~but~~ so prop support was used but by the use of mechanization we required more span of gallery so ~~the~~ props become obstructions so we shifted to roof bolting.

- The decisive considerations regarding the choice of support system is based on either the experience in similar mining condition or sound rock mechanics principles.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Principle of Roof Control:-

- Mining activities induces vertical and horizontal tensile stresses in the roof strata of tabular excavations. (thickness of the excavation is much less than the length of the excavation)
- The relative magnitude of the tensile stresses and the vertical extent of the tensile zone depend largely on the ratio of the mining span to the depth below surface of the excavation.



On central element $\sigma_v = 0$, σ_h = minimum } So stress relaxed zone because it relaxes both σ_v and σ_h

As we move towards the stress concentrated zone :-

σ_v = increases and at the corner of opening it is very very high, more than initial value

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

σ_h = increases, we can recover σ_H but it is difficult to predict its value, it has complex distribution.

- where weakness occurs within the tensile zone, strata separation can be expected unless adequate support is provided.

- In these circumstances, the main purpose of support is to prevent separation of strata and to maintain the interlocking of roof rocks.

- Therefore, a necessary design criteria of the support for productive excavation is that

  - The full thickness of rock up to any trouble geological weakness within the tensile zone must be supported.

Explanation:-

~~Pick consider an element at the center of the stress relaxed zone~~

The rockmass is made up of many subhorizontal weakness planes.

An element in the stress relaxed zone tends to displace downward due to its weight acting downward and due to lack of confinement (traction free) loading to stress relaxation

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

So the layers of rockmass tend to bend. But due to differente in the elasto-mechanical properties and Bending strength of different layers because they are formed of different material, the Bending extent of different layers is different so a parting develops between the different layers. Bending increases and hence parting increases if the no. of weakness planes in the rockmass increases.

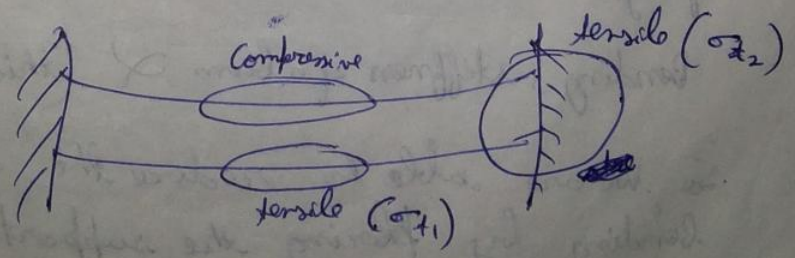
Stresses develop at different points on the roof rock due to bending of the roof (analogous to a beam bending)

At the corners of the opening, (stress concentrated zones), highest tensile stresses develop. At the center of the roof layer on the upper face compressive stresses develop due to compressive strain and on the lower face of the rock layers tensile stresses develop due to tensile strain. (Stress relaxation zone)

On studying the bending of a elastic beam (tensile deformation) with the given boundary condition (i.e. 1.) Confinement at the corners
2.) lack of confinement (to bending) at the center (i.e. traction free at the center)

we know that

tensile stresses at the corner of the beam
= 2 X tensile stresses at the center of the beam



$\sigma_{t_2} = 2\sigma_{t_1}$

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

So we can conclude that the tensile stresses at the center of the roof will be half of the tensile stresses at the stress concentrated zones (corner of gallery)

But tensile strain will be maximum at the center of the roof compared to the corners and so this point is likely to undergo tensile failure due to maximum bending at the center. The area of stress concentrated zones (corners of the opening) are likely to undergo shear failure.

So point of attack (i.e. the point at which support is required to be installed) is at the center of the roof as it is the position of maximum tensile strain. By supporting at the center we can control the tensile stresses developed at the corners of the roof.

By attacking at the center we are increasing the inherent strength of the rockmass (The rockmass has some strength due to its formation). The support reduces the parting between the rock layers. By reducing the parting the rockmass behaves as a massive rock (i.e. a composite block) of higher thickness and we know that

Bending stiffness of a beam $\propto$ (thickness of beam)

So we are able to reduce the extent of bending by placing the support at the point of maximum strain (stress relaxed zone) i.e. at the center of the roof due to an increase in the bending stiffness of the roof rock.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

So the moment of tensile stresses developed at all points reduces so chances of tensile failure of roof reduces.

Q) What should be the ~~vertical~~ Vertical extent of support ??

~~of the support~~

A) The zone of influence of the support should be upto the extent to which tensile stresses are likely to develop. The zone of influence of the support should be able to nullify the zone of tensile stress. The above is true if the support is placed at the point ~~of~~ prone to ~~undergo~~ tensile failure (i.e. at the center of the roof).

If the support is placed as the ~~place~~ ~~point~~ ~~prone~~ point which is likely to undergo shear failure ~~then~~ (stress concentrated zone) then the orientation of the support should be such that it increases the cohesion of the rockmass and the friction of the rockmass. By increasing either of the 2 or both the shear strength of the rockmass increases and its chances of failure (shear) reduces.

⇒ This is the principle of roof bolting and ~~this~~ the above states the reason why the first bolt is placed at the center of the gallery roof.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

9/09/2013

BKS Sir

Factors affecting Mine Subsidence :-

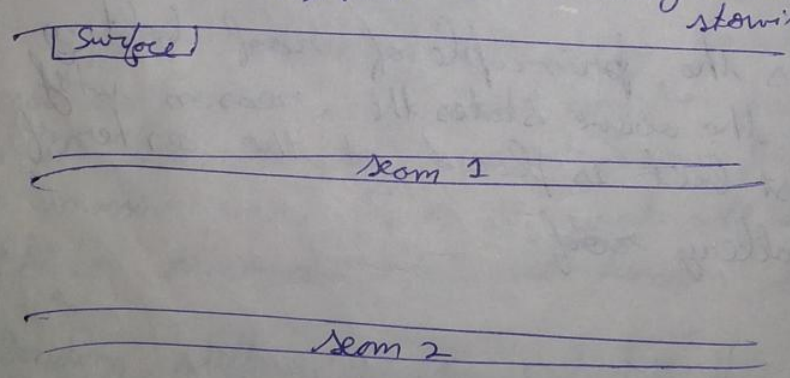
(1.) effective seam thickness
(2.) Multiple seam
(3.) Seam Depth

1.) effective seam thickness → Thickness which is mined out is effective seam thickness

thickness ↑ → subsidence ↑

2.) Multiple seams → we mine in decending order (upper first and then we move down) in Caving. So if we have multiple seam in close proximity then they will have a cumulative effect of subsidence and it will be more than predicted by indivisual seams.

→ we mine in ascending order in stowing



when seam 1 mined → above strata becomes weak and fractured

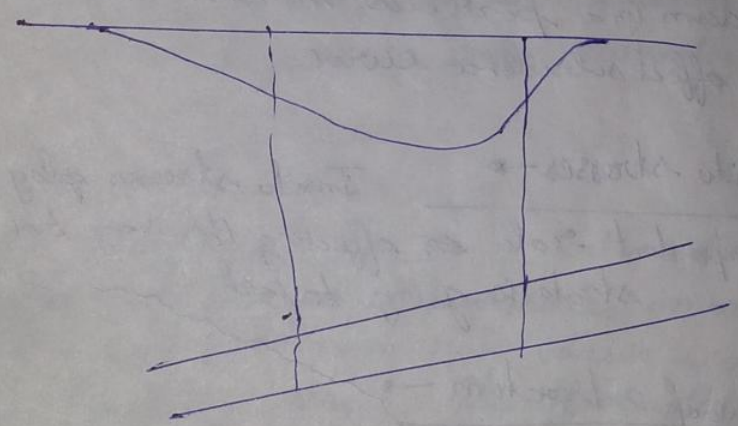
when seam 2 mined → then parting becomes weak and the entire above strata slips down.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

3.) Seam Depth →

more depth → less subsidence

4.) Dip of the seam → when the coal seam being mined is inclined ~~and~~, an asymmetric subsidence trough is formed that is skewed towards the rise. i.e. the limit angle is greater on the dip side of the working, The strains are also smaller towards the dip direction



5.) Competence of Mine roof and floor →

If the immediate roof is competent and massive, it delays subsidence and we get a different subsidence curve.

6.) Nature of overburden → The intermediate layers of rocks should be tovable so that we get a conventional subsidence profile. All layers upto the surface should break properly.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

7) Near surface geology →

Surface has very weak material usually (sand, soil, etc). Ultimate result of subsidence is seen on the surface. If surface is made up of hard rock then it will not crack properly thereby affecting subsidence curve.

8.) Geological Discontinuity →

Presence of folds, faults affects subsidence profile by affecting the pattern of caving.

9.) Fractures → Joints allow material to fall down in a particular manner and hence affect subsidence curve.

10) Insitu stresses →

Insitu stresses play a important role affecting the way the strata is going to fail.

11.) Degree of extraction →

Degree of extraction is less → delays subsidence → ~~affect subsi~~ less subsidence

Coal is strain softening material so it takes some load even after failure and so if pillars are left then they delay subsidence. This is the case in B and P mining.

12.) Surface Topography → tensile strain may become more marked on hill tops and decrease in valleys

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

13.) Ground water →

Removal of water from underground leads to subsidence in a fashion similar to that caused by removal of material.

But we have a benefit :-

Sometimes water is filled in the mines. after extraction and so water pressure supports the strata and delays subsidence.

14.) Span → (Mined Area)

The span of the mine should be greater than the NEW for subsidence to take place.

15.) Method of working →

Board P → less subsidence
longwall → more subsidence

16.) Rate of face advance → Surface subsidence follows the face as it progresses in a panel. If the coal extraction rate varies significantly, the travelling strain also fluctuates & this results in large differential settlement. A fairly rapid even rate of face advance is best.

17.) Backfilling of the goaf

18.) Time lapse

19.) Structural characteristics

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Regional Support.

- There are circumstances when there is a presence of a competent strata in roof rock in small mining depth and the thickness of the bed is more.

— There could be a situation where this competent layer of rock is required to fail in a stable and controlled manner at a given mining span. ~~This~~ If this is not possible, the roof strata is required to be supported by a regular system of pillar (yield pillar). These pillars are designed in a such a manner that they do not fail regardless of the extent of mining.

— By the provision of yield pillar the strata will fail but in a controlled manner providing safety against the entity to be addressed (i.e. uncontrolled failure)

— Under such conditions, the pillars so designed perform the function of regional support.

— Similarly filling out the goaf area with waste rock is also a regional support

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Depending on time for which the supports are designed

1.) Temporary support — Their actual life, usefull time is very very small

2.) Permanent support — Once installed, they can be left at the place forever.

Temporary Vs Permanent Support :-

- ~~For~~ Temporary supports are exclusively local support, ~~designed to be~~ required only for a short period of time ex.) Hyd. Props, ~~powered~~ Powered supports.

- Regional support are generally permanent support. Permanent supports are generally employed for regional stability reasons.

All ~~sup~~ withdrawable supports, irrespective of the load bearing capacity of the support are Temporary supports ex.) chock, log props.

roof bolts → permanent Support

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Depending on ~~mechanism~~ mode of action, supports can be classified as:-

1.) active supports

2.) Passive supports.

Passive Support

- Any support which is installed unstressed is regarded as Passive support. Such support begin to function only in response to strata movement.

- As the movement of unfractured rock is usually small, passive supports often become effective only when rock fails and undergoes some displacement around the excavation.

- Frequently, there is poor contact between the support and the rock, therefore, the potential benefit of passive support is hardly ~~realised until a~~ realised till a considerable time has elapsed allowing considerable movement of failed rock.

- The delayed ~~in~~ built up of support resistance results in considerable deterioration of mining condition which can be prevented by ~~the~~ use of Active Support.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Active Support

- A support structure is active if it exerts and maintains a restraint on the rock immediately after installation

- Essential feature of an active support is that the provision of the initial support force does not depend on rock movement.

- Fall of roof or side is one of the major reasons (61%) of accidents in underground mines

- A large number (45%) of these accidents occurs in freshly exposed roof areas and the thickness of fall is generally less than 1 m (in 80% of such cases)

- Fall occurred in all types of roof with a higher rate of incident in coal or shale (weak roof) roof. Geological disturbances have been reported to be the main reason of such accidents in sandstone roof. (competent)

- A proper assessment of support requirement for different part of the working and its proper implementation is very important for better safety and productivity of mine workings.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

While supporting we should remember that 2 factors affect the level of supporting:-

1.) Type of support

2.) Time lag between exposure of fresh roof and installation of support — lesser is the lag better is the performance of the support.

Support design based on CMRI ~~ISM~~ ISM ~~ISM~~ RMR

• For development headings:-

$$P_r = \gamma B F \left(1.7 - 0.037\,RMR + 0.0002\,RMR^2\right) \text{ ton/m}^2$$

• For junctions:-

$$P_r = 5\gamma B^{0.3} F \left(1 - \left(\frac{RMR}{100}\right)^2\right) \text{ ton/m}^2$$

where :-

γ — unit weight of rock (t/m^3)

B — is the Bord width (m)

F — is the factor of safety, generally taken as 1.5

RMR is the average RMR of immediate roof after adjustment

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Q. Why F is used in the formula??

Ans:- γ and B are explicitly obtained and RMR is the subjective parameter (varies from person to person)

So F is used to magnify the ~~stress~~ support requirement in order to address those inherent restrictions developed ~~by~~ due to the subjectivity in determining the RMR value.

Parameters	Rating
Layer thickness	30
Structural features	25
Rock weatherability	20
Intact rock Compressive strength	15
Ground water seepage	10

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Support design in depillaring workings:-

· Slice junction

$$SLD_{sn} = \frac{\gamma H^{0.5} K^{0.64} W^{1.17}}{R^{0.90}} \ t/m^2$$

· Slice

$$SLD_{sl} = \frac{\gamma H^{0.67} K^{0.84} W^{1.74}}{R^{1.42}} \ t/m^2$$

· Split gallery

$$SLD_{sp} = \frac{\gamma H^{0.52} K^{0.59} W^{1.12}}{R^{1.02}} \ t/m^2$$

· Goaf edge

$$SLD_{ge} = \frac{\gamma H^{0.54} K^{0.49} W^{0.89}}{R^{0.79}} \ t/m^2$$

H → depth of working
W → width of gallery
K → stress ratio
R → RMR
γ → (t/m^3)

SLD → Setting load density (t/m^2)

SLD is defined as amt of load density that the support should provide at the point of ~~suppo~~ setting

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Assignment - III

Q.) Estimate the setting load density for a depillaring working under following geotechnical condition

RMR of the immediate roof = 45

width of gallery = 4.5m

Stress ratio = 1.2

Cover depth = 240m

13/09/2013

GSPS Sir

In 99% of longwall faces, caving is used as goaf control method.

But we can have:-

1.) Caved longwall
2.) Stowed longwall

The challenge of ground control is in caved longwall cases.

Caving Mechanism/Caving Mechanics

Q.) Estimate the span of first local fall, main fall, subsequent local fall and periodic ~~giving~~ caving span in a longwall working with a face length of 150m planned to be operated under set of strata condition as given below:-

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

layer-3 Fine grained sandstone, flex. strength = 5MPa, 3m thick, density = 2500 kg/m³

layer-2 Fine grained sandstone, 8m thick, flex strength = 5MPa, density = 2500 Kg/m³

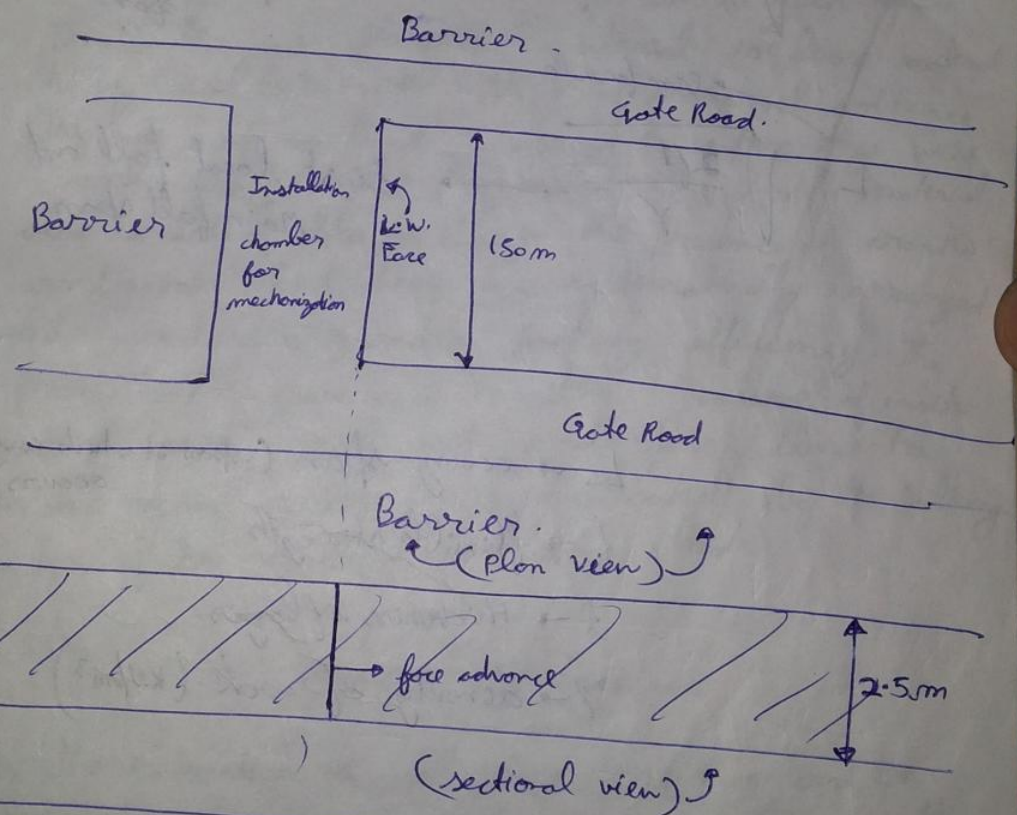
layer 1 Shaly sandstone, 1m thick, flexible strength = 3MPa, density = 2500 kg/m³

Coal seam, 2.5m, full thickness extraction, density = 1400 kg/m³

→ average cover depth of seam = 200m

(Bulking ~~working~~ factor of rock) K = 1.25

sol.)



To estimate the ht. of caving zone:-

$$t = \frac{h}{K-1}$$

t → caving height
K → bulking factor of overlying strata
h → extraction ht.

K → bulking factor

GOAF and GOB are same

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

~~After extraction of coal seam as the span increases the overlying strata is going to fail and it will be become a goaf/gob material.~~

~~The volume of the rock in the goaf will be greater than the the rock in the roof.~~

To estimate spans of roof fall:-

~~First local fall span~~

cantilever formula

$$L = \sqrt{\frac{ft}{3\gamma}}$$ → subsequent local fall span and periodic caving span

Beam formula

$$L = \sqrt{\frac{2ft}{\gamma}}$$ → First local fall and Main fall span.

L → caving span (span at which caving occurs)

f → flexible strength

t → thickness of layer.

γ → density of rock (KN/m^3)

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

The support requirement of longwall with caving is quite complex compared to that required in stowing. The challenge of ground control in caved longwall cases.

On extraction of the coal seam the ~~overlying strata~~ will be redistribution of stresses in overlying strata which will cause displacement of the overlying strata and hence as the span increases the overlying strata will undergo failure at some point of time. The ~~f~~ strata on failure will fill the void created due to mining. The volume of rock ~~of~~ the goaf material will be larger than the volume of insitu rock. ~~hence~~ This is because the goaf material is a loose material having voids between the blocks of material and hence they are not compact compared to where they were ~~in the roof~~ not fragmented. The insitu material also consists of blocks due to the presence of cracks and joints but they are systematically arranged and hence have a greater packing efficiency. The proportional increase in the volume of insitu rock material when it undergoes failure to become goaf material is indicated by the Bulking factor of the rock of overlying strata.

$$V_{goaf} = K\, V_{insitu\ rock}$$

As the extraction of coal seam continues and the span of mining ~~is~~ increases a point comes when the overlying strata fails. This is the time of occurance of the first local fall. This fall involves the failure of the immediate roof. So the first local fall is the process of failure of the immediate layer (layer 1) for the first time. The mining span which is responsible for the first local fall is called the span of first local fall.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

The occurance of first ~~roof span state~~ local fall takes place at the center of the mining span and the heap of goaf material is formed at the center of the mining span.

~~As the sp~~ On further advancement of the face the occurrance of the next local fall takes place and it is called ~~sequen~~ subsequent local fall. ~~And its span~~ And the span for subsequent local fall can be given as.

$$\text{span for subsequent fall} = \left(\text{~~Present~~ span for immediate fall}\right) - \left(\text{span for previous fall}\right)$$

ex:) Suppose the first local fall occurs at 15m and subsequent local fall occurs at 20m of span then the span for subsequent fall is $(20-15) = 5m$.

~~But the failure~~ The subsequent local falls also involve failure of the immediate roof rock. But the volume of material dislodged due to fall of the immediate roof is not capable to fill the entire void created by mining. So the second layers (of overlying strata) also fails and starts caving and as ~~the second layer~~ other layers of overlying strata fails, ~~completely~~ the void is ~~comp~~ entirely filled by goaf material. The point when the entire goaf is filled up by goaf material / gob material is the point when main fall has occured. The span of mine (at which there is the occurance) ~~responsible for~~ of the main fall is called the main fall span. The main fall mainly involves the failure of the second layer which is also called the main roof. The main fall hence involves the failure of more than one layers of the overlying strata.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

The second layer (which is 8m thick) is called the main roof because it has a dominent influence ~~on~~ in characterising the caving mechanism.

~~Failure mechanism of Main roof:-~~

Mainfall → It is the occurance ~~of~~ of caving upto the point, the roof is completely filled.

Failure mechanism of Main roof:-

The main roof bends due to its weight and in addition due to loading by the third layer. The third layer is called the loading roof/strata/layer.

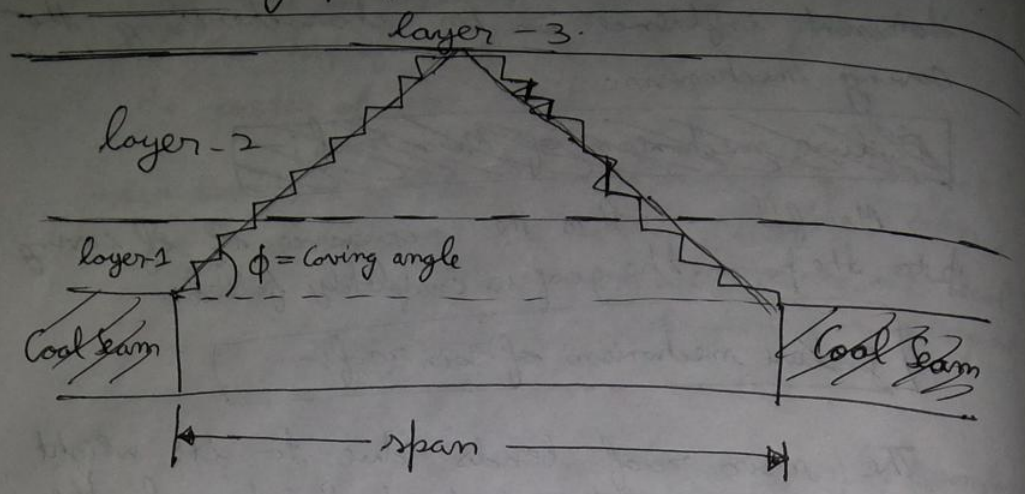
→ To understand the above take the example of 2 beams one thin, one thick, restricted at its ends ~~On application of~~ and free to bend. On application of load on the 2 beam system the thin beam bends ~~less~~ more compared to the thicker beam. This is because the bending stiffness of thicker beam is more compared to the thinner one. Due to this differential bending of the two beams the ~~thicker~~ thicker beam is additionally loaded.

Hence ~~we~~ ~~can~~ due to the excessive bending of layer 3 of the overlying strata, this additional bending further induces loads on the main roof causing its failure.

Hence the failure of Main roof is due to
1.) Bending by virtue of its weight.
2.) Loading due to additional bending of the 3rd layer (loading roof)

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

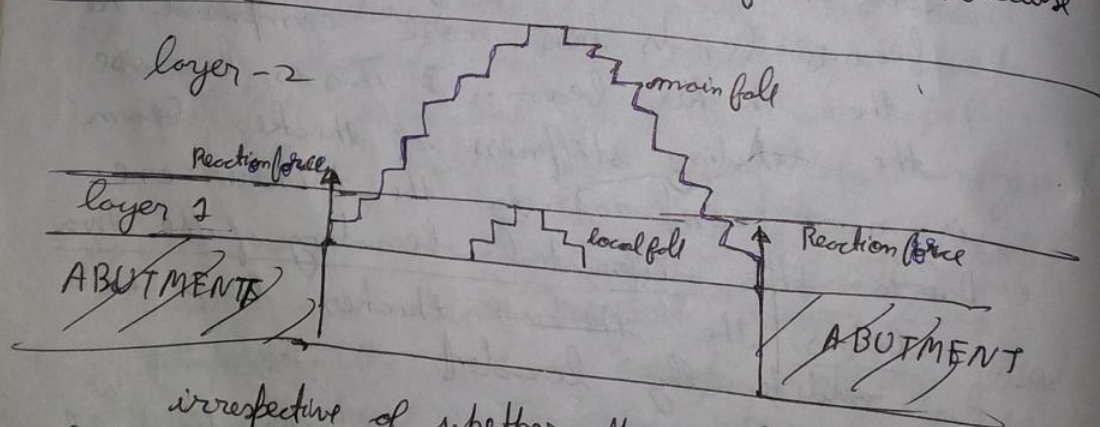
The caving profile of the roof is shown below:-



To estimate the span for first local fall and Main fall:-

$$L = \sqrt{\frac{2ft}{\gamma}}$$

This is called the Beam formula because



irrespective of whether the roof is undergoing first local fall or main fall, the overlying strata behaves as a beam unsupported at the Center (as the underlith material is excavated) and restricted at the corners or boundaries as well as supported at the Corners by abutments (reaction forces). Hence the overlying strata behaves as a restricted as well as simply supported beam.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

To estimate the span for subsequent local fall / periodic caving span :-

$$L = \sqrt{\frac{ft}{3\gamma}}$$

This is called the cantilever formula, because the occurance of the first fall creates a free face / end and the overlying strata behaves as a cantelever beam supported and restricted / clampled at one end and free to move at the other end.

A/Q:-

Span for 1st local fall = $\sqrt{\frac{2 \times (3 \times 10^6 \, Pa) \times 1 \, m}{2500 \, kg/m^3 \times 10 \, m/s^2}}$
(involves failure of immediate roof)

Span for Subsequent local fall = $\sqrt{\frac{(3 \times 10^6 \, Pa) \times 1 \, m}{3 \times 2500 \, kg/m^3 \times 10 \, m/s^2}}$
(involves failure of immediate roof)

$= 6.32 \, m$
$\approx 6 \, m$

Span for Main fall = $\sqrt{\frac{3 \times (5 \times 10^6 \, Pa) \times 8 \, m}{2500 \, kg/m^3 \times 10 \, m/s^2}}$
(involves failure of Main roof)

$= 69.28 \, m$
$\approx 69 \, m$

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Solution of Question by GSPS Sir given on 12/09/2013

~~SLD JN~~

$$\left.\begin{aligned} R &= 45 \\ w &= 4.5\,m \\ K &= 1.2 \\ H &= 240\,m \end{aligned}\right\} \text{(given)}$$

$$\gamma = 2.5\ t/m^3$$

$$SLD_{JN} = \frac{\gamma H^{0.5} K^{0.64} W^{1.17}}{R^{0.9}} = 8.22\ t/m^2$$

$$SLD_{sp} = \frac{\gamma H^{0.67} K^{0.84} W^{1.74}}{R^{1.42}} = 7.05\ t/m^2$$

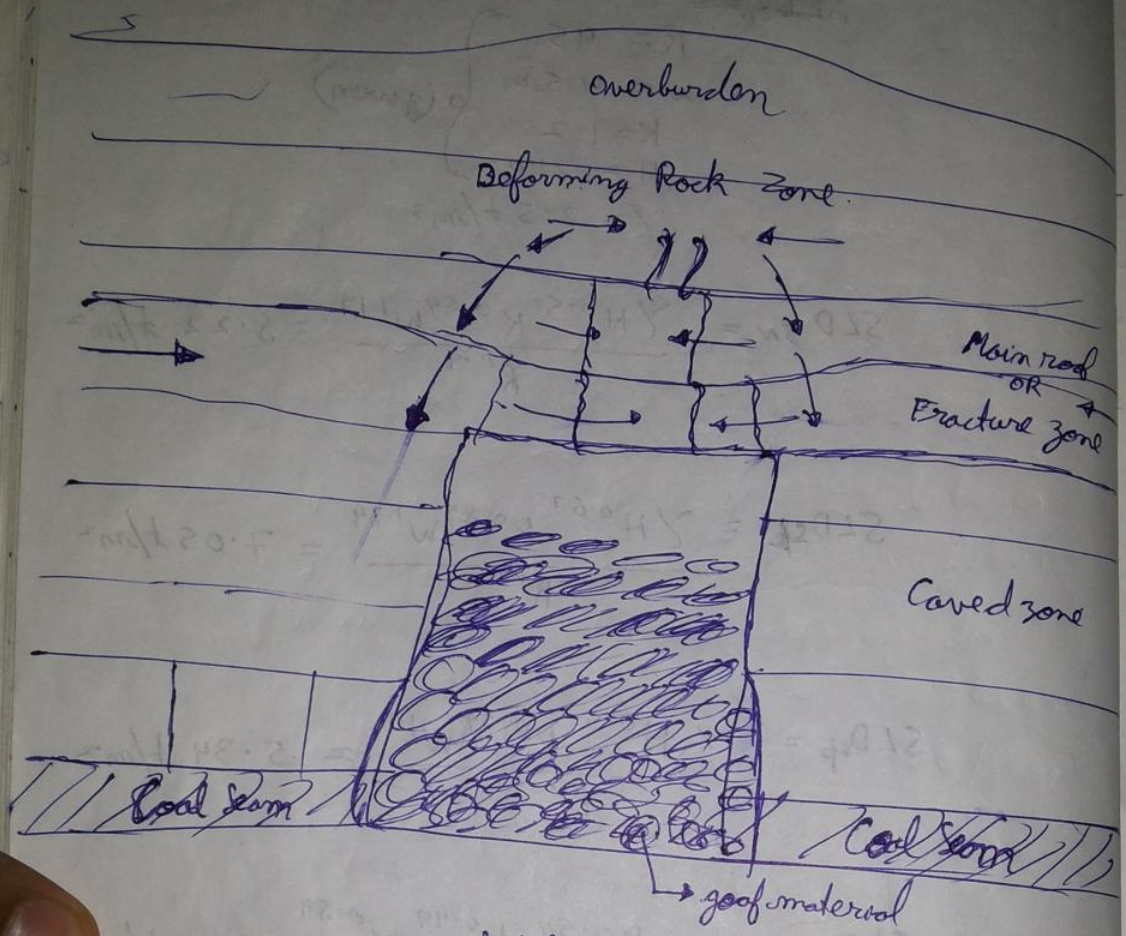
$$SLD_{sp} = \frac{\gamma H^{0.52} K^{0.59} W^{1.12}}{R^{1.02}} = 5.34\ t/m^2$$

$$SLD_{ge} = \frac{\gamma H^{0.54} K^{0.49} W^{0.89}}{R^{0.79}} = 9.94\ t/m^2$$

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

14/09/2013.

GSPS Sir



We have a ~~tabular~~ tabular coal seam and above the deposit we have various layers.

The mineral deposit has a stress domain and by the time, mineral is excavated we have a redistribution of stress and we have a stress relaxed zone. All the layers which are within the stress relaxed zone are free to cave or deform downward under the effect of gravity. The layers undergo bending and there are zones of tensile stresses. when tensile stress > tensile strength

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

then the immediate roof fails, and as the excavation span is increased more and more tensile stresses develop which propogate the process of failure initiated earlier. Then the failure is increased to such an extent that the overlying strata is no longer able to remain in place and it caves.

Then as the mining span increases the main roof also involves in caving with the same mechanism (bending → development of tensile stresses at critical points → tensile failure followed by caving)

Condition of roof during 1st local fall:-

As the span is increased, at a particular span of mining the roof of the mine behaves as a beam restricted at 4 ends (2 ends across the face and 2 ends along the face) and traction free at the center. The rate of increased of span affects the caving mechanism.

Tensile stresses develop on the lower surface of the immediate roof and (max tensile stresses develop at) corners across the face and as the span increases, the immediate roof fails. This is the first local fall. As the span further increases the subsequent local falls take place but since after the ocurence of the first local fall the immediate roof looses its continuum

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

and hence now behaves as a different mechanical structure called Cantilever. Hence the immediate roof behaving as a cantilever is involved in subsequent local falls.

As the span is increased the main roof comes into picture. The main roof is not involved in first local falls and subsequent local falls. The main roof also fails in the same manner as the immediate roof. It behaves as a beam during the first failure of the main roof (i.e. during the main fall) and behaves as a cantilever as the periodic caving process takes place.

The main roof is involved in

1.) main fall
2.) periodic caving process

first local fall (immediate roof) / main fall (main roof)

$$L_B = \sqrt{\frac{2ft}{\gamma}}$$

subsequent local fall (immediate roof) / periodic caving (main roof)

$$L_c = \sqrt{\frac{ft}{3\gamma}}$$

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

$f \rightarrow$ flexible strength / bending strength of layer involved in failure

$t \rightarrow$ thickness of layer involved in failure

$\gamma \rightarrow$ unit weight of layer involved in failure

But the main roof is bended by its own weight as well as due to differential loading / bending from the loading roof. So while using formula for main fall and periodic coving span in place of γ we will use

γ_e (effective value) involving the effect of main roof and loading roofs.

$$\gamma_e = \frac{E_1 t_1^2 \left[\gamma_1 t_1 + \gamma_2 t_2 + \cdots \right]}{E_1 t_1^3 + E_2 t_2^3 + \cdots}$$

$E \rightarrow$ Young's modulus

$E_1 \rightarrow$ Young's modulus of main roof

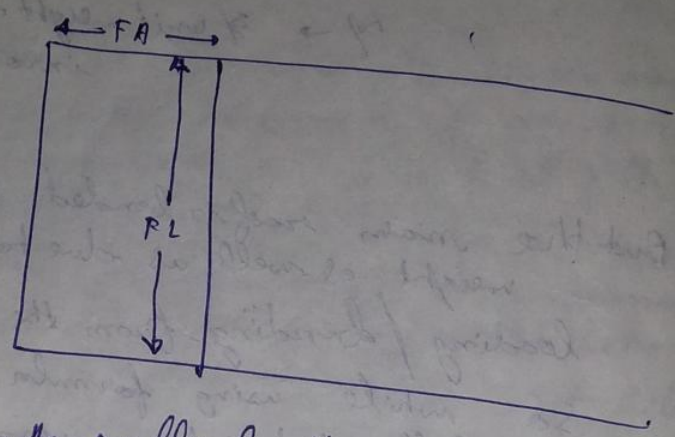
$E_2, E_3 \cdots \rightarrow$ Young's modulus of loading roofs.

But in case the main roof is thinner than the loading roof then there is no loading by the layer above the main roof so we will not use γ_e but only γ of the main roof for main fall span and periodic fall span.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Plate formula:-

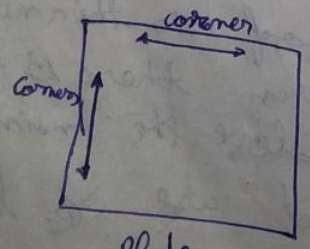
We observe that during the occurrence of the first local fall, the span of face adjacence is lesser than the face length



Taking the smaller length as a and longer length as b

$\frac{b}{a} \geq 2 \rightarrow$ roof behaves as an elastic beam and the classical formula for elastic deformation of beam can be used.

$1 < \frac{b}{a} < 2 \rightarrow$ The roof behaves like a plate and plate formula is used to explain failure of roof



Plate

The plate formula says that the max. stresses developing along the corners of the plate depends on an empirical function β which is a function of b/a

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

$$\beta = f(b/a)$$

$$\sigma_{max} = \frac{6\beta \gamma_e a^2}{t}$$

γ_e is used in case upper layer is also involved in loading of roof.

When σ_{max} exceeds the flexible strength/bending strength of the layer then the layer becomes unstable and is bound to undergo failure.

b/a	1	1.25	1.50	1.75	2	>2
β	0.0513	0.0665	0.0757	0.0806	0.0829	0.0833

Beam Plate formula:-

Closed form solution based on Beam and plate ~~formula~~ theory.

$$\sigma_{max} = \frac{6\beta \gamma_e a^2}{t}$$

$$L_B = \sqrt{\frac{2\sigma_t t}{\gamma_e}} = \sqrt{\frac{2ft}{\gamma_e}}$$

$$L_C = \sqrt{\frac{\sigma_t t}{3\gamma_e}} = \sqrt{\frac{ft}{3\gamma_e}}$$

Caving Mechanism ~~of~~ in Stratified rock formations:-

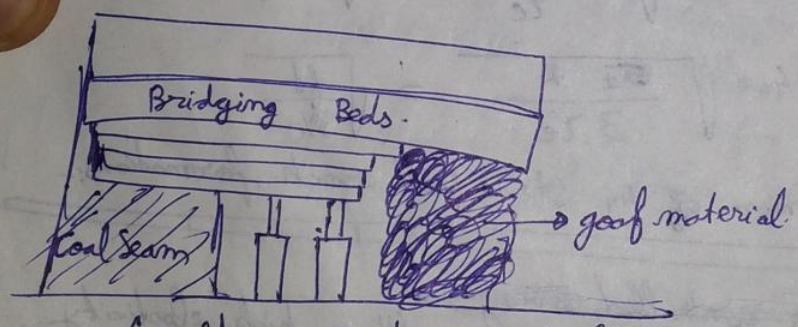
~~Bulking factor controlled caving~~

Bridge Bed → Strata with a high elasticity showing perfectly elastic deformation without fracture of the bed. ex:) Sill

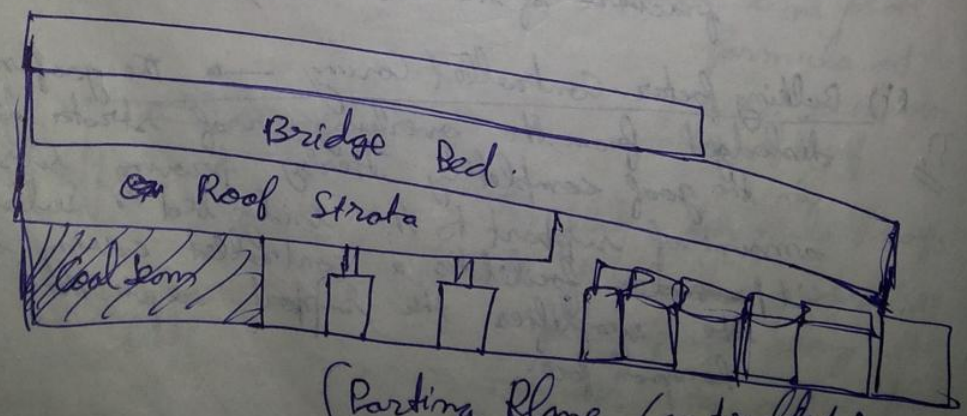
(i) Bulking factor Controlled Caving → The goaf material dislodged from the overlying roof strata fills in the goaf completely thereby proving substantial amount of support to the bridge bed. Subsequently it ~~provides~~ facilitates a controlled caving and ~~the~~ simplifies the support requirement at the longwall face.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

(??) Parting plane Controlled Caving → The quantum of goaf material is insufficient ~~to fill~~ to fill the goaf completely. So There is hardly any contribution from the goaf pile to support the bridging bed. Hence there is significant ~~deformation~~ bending of the bridge bed until it is supported either by the goaf pile or by the floor. This deformation of the bridging bed with the advance of the longwall face is uncontrollable. The rate of deformation of bridging bed with face advance is not uniform. ~~[illegible]~~ So with increasing deformation the load on immediate roof increases (with induction of higher tensile stresses) which further increases the load on the supports at the longwall face. Due to non uniform loading of the supports the role of support ~~at~~ has to be justified at a point of time. This makes the support design ~~at~~ the face very complex depending on the requirement at the face.



(Bulking Factor Controlled Caving.)



(Parting Plane Controlled Caving)

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Factors affecting Cavability:-

1.) Area of excavation → Area of exposure of goaf is essential for caving of the roof layer.

2.) Thickness of the layers → More is the thickness, difficult it becomes to undergo caving.

3.) Bending strength of layer → more the bending strength, more difficult it becomes to cave.

4.) Depth of Cover → Greater depth results in easier cavability (because vertical stress increases and augments the caving process)

5.) Horizontal stress → with its increase, the intrinsic strength of rockmass increases as it increases confinement of layers, So it impedes the caving process.

6.) Presence of joints and parting planes → ~~more the~~ Higher density of joint planes in overlying strata, the effective strength of rockmass decreases and hence caving ~~pro~~ characteristics increases (it augments caving process)

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Empirical Model:-

$$I = \frac{\sigma L^{m} t^{0.5}}{5}$$

(Cavability index)

(σ → compressive strength (kg/cm²))

(We can estimate the caving ability of the rock bed)

L = average core length of rock bed, cm

t = thickness of the rock bed, m

$1.1 < m < 1.2$ (depends on the RQD of the layer)

For RQD < 80 $m = 1.1$

else $m = 1.2$ for RQD ≥ 80

To understand the overall caving behaviour of strata above the coal seam :-

1.) Evaluate caving index of each and every rock bed lying between the caving zone.

2.) Find the rock bed of max. I value. This is the strongest bed. (I_{max})

3.) Put I_{max} in next formula to obtain span of main fall with accuracy.

$$S_{mw} = 0.72\, I_{max}^{0.51}$$

(S_{mw} → Span for main fall (m))

4.) The periodic caving span can be evaluated by the next formula:-

$$S_{pw} = 3.05 + 0.25\, S_{mw}$$

(S_{pw} → span for periodic fall (m))

(S_{mw} (m) → main fall)

Rock Bed	Rock Characteristics	Thickness (m)	RQD (%)	Avg. length of core	σc kg/cm²	I
1	Cg, Mg, SSH	4.4	98	24	210.68	4005
2	T, Lenticular	0.4	79	17	411.13	1558

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

r strength (kg/cm^2)

ge core length, C_m of rock bed, m

ness of the rock bed, m

m < 1.2 (depends on the RQD of the core)

RQD < 80 m = 1.1

m = 1.2 for RQD ≥ 80

haviour of

eavy rock

ering zone.

volume of This

to obtain

can be

t formula:-

main fall

Q7.

Rock Bed	Rock Characteristics	thickness (m)	RQD (%)	Avg. length of core	σ_c kg/cm^2	I
1	Cg Mg SSt	14.4	98	24	210.68	4005
2	Intercalation	0.4	79	17	411.13	1558
3	Cg fg SSt	3.8	93	22	269.17	4284
4	Intercalation	0.76	97	23	411.13	3084
5	Cg Fg SSt	12	87	20	293.91	$\boxed{7413}$ I_{max}
6	Coal	0.67	not possible to find RQD			0
7	laminated bed of shale and intercalation	3.78	80	18	362.29	4520
8	Sandstone	2.4	97	23	439.16	5859
9	Laminated bed of shale and sandstone	3.8	84	19	377.11	5034
10	Sandstone and intercalation	3.9	79	17	396.45	4691

We are moving upward from the Coal Seam

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Roof Category	I_{max}	Expected Caving behaviour.
Cat I	≤ 2000	Easily Caving
Cat II	$2000 < I_{max} \leq 5000$	Moderately Caving
Cat III	$5000 < I_{max} \leq 10000$	Cavable with difficulty
Cat IV	$10000 < I_{max} \leq 14000$	Cavable with substantial difficulty
Cat V	$I_{max} > 14000$	Cavable with extreme difficulty

~~The problem of caving is cavity formation~~

Maximum Convergence (mm/m of face advance) (between roof and floor)	Expected Roof Condition
< 60	Continuity of roof remains intact
60–100	Minor cracks and breaks in the exposed face area
100–160	Prominent fractures for lower range, rock fall occuring at higher range
> 160	Rock fall leading to face collapse

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Ans:- E

The roof is of category III as our $I_{max} = 7.4/3$

Difficulty in Caving → Cavity formation will be there. The roof will try to converge and the support will not be able to bear the convergence of the roof strata at the face and will allow large deformation of the roof at the face without failure. So the roof will develop cracks and fractures at this stage.

BKS Sir

16/09/2013.

H.W.

Methods of Subsidence measurement (Submit on 18/09/2013)

Damages due to Subsidence:-

1) Buildings → Most earth structures are weak in tension. Due to tensile strain the walls are likely to crack. If building is reinforced then damage is less due to a higher tensile strength of iron. If the building are earthen structures then they very easily break up.

Differential ground movement in vertical movement also causes damage of building.

2) bridges → Horizontal tensile strain leads to the damage of bridges.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

3.) Roads and airport runways → Cracks and potholes develop in roads due to subsidence vertical movement of land.

4.) Rail lines → Generally in Indian condition, mining below rail lines is not allowed. Due to subsidence land gradient changes leading to non feasibility for rail lines. A gradient greater than 1 in 300 in not fit for railways.

Remedial measures for mitigating subsidence:-

I.) Partial mining → Techniques involve:-

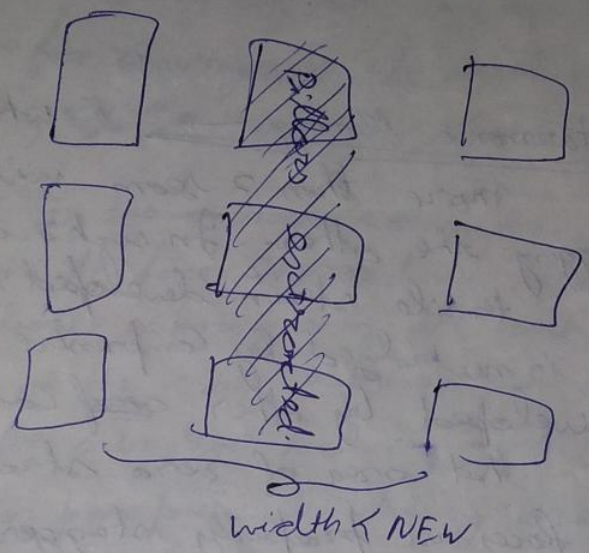
1.) Development only

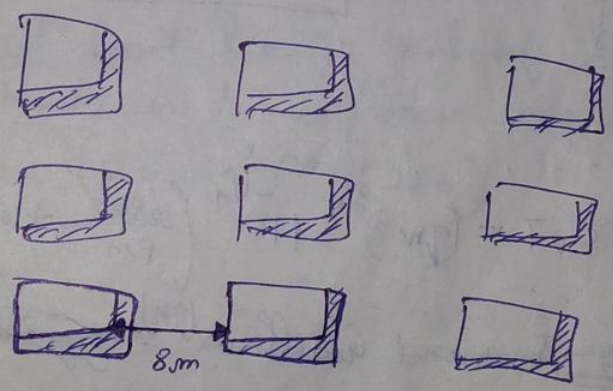
2.) wide stall method → Pillars extracted in L-shape leading to widening of the gallery and the remaining pillars is left intact. The maximum permissible widening of gallery is upto (8 - 10) m. The roof of the gallery is supported by roof bolting, etc. The amt. of widening of gallery depends on the roof condition.

3.) Splitting → Splitting of pillars is done into 2, 4 stooks but the danger is that the size of pillars is reduced to a large extent so their supporting strength reduces so wide stall is preferred but in case of shallow workings splitting can be done.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

4.) NEW → ~~The~~ A row of pillars is extracted such that the width is less than the NEW to ~~stop so~~ reduce chances of subsidence, ~~but~~ but it was not an effective method so discarded. Alternate row are mined out.





Wide - Stall Method.

III.) Back filling →

1.) Material taken from underground followed by filling using sand stowing. (common in coal mines)

2.) Many metal mines take the Mill tailings and use it for filling purpose in method of cut and fill mining.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

3.) fly ash → It is also used, for backfilling but flyash retains water and so remains muddy and flushy ~~and~~ so now flyash is made into pellets (by mixing with cement, etc) on the surface and send ~~is~~ with water.

III.) Harmonic Mining → when we have more than 2 seam with one face lagging the other. In such a case the tensile strain developed by one face is neutralized by compressive strain developed by other ~~coal~~ coal face so that area of zero strain is developed. The faces are properly staggered for this purpose.

19/09/2013.

GSPS Sir

$$C_vT = (qw)^{3/4} H^{-1/4} \left(\frac{6800}{PM} + 66\right)$$

Given by → Josien and Gouillaux (1978) ~~suggested that~~

C_vT → Convergence parameter

PM → function of the support resistance

q → parameter associated with goaf treatment

W → working thickness of seam

H → ~~Cover~~ Cover depth

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Josien and Gouillaux (1978) suggested that a relation between the desired load bearing capacity P, per meter length of face may be obtained using the following equation, by limiting the face convergence to its threshold value, C_{vo} of 40mm/m of face advance.

$$P = \frac{C_{vT} \cdot w \cdot q \cdot H \cdot PM}{1.5}$$

where C_{vT} is the average convergence of the walls in the face expressed in meter per m advance of face; w is the working thickness of the seam expressed in m $(0.8 \leq w \leq 3)$; q is the subsidence factor, 2 for caving and 0.6 for ~~pneu~~ pneumatic stowing and 0.15 for ~~hydrau~~ hydraulic stowing, H is the depth of the mine in m $(100 \leq H \leq 1000)$, PM is the load bearing capacity of the supports in tonnes/linear m of the face $(20 \leq PM \leq 260)$

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Estimation of support requirement in longwall panel:-

$$C_m = \left(\frac{A}{P} + 9.6h\right) + \left(\frac{KI}{K'+1.5} - 23\right)$$

Where :-

⇒ h → extraction height

⇒ C_m is the maximum convergence at face (mm/m)

⇒ P is the mean load density (t/m^2)

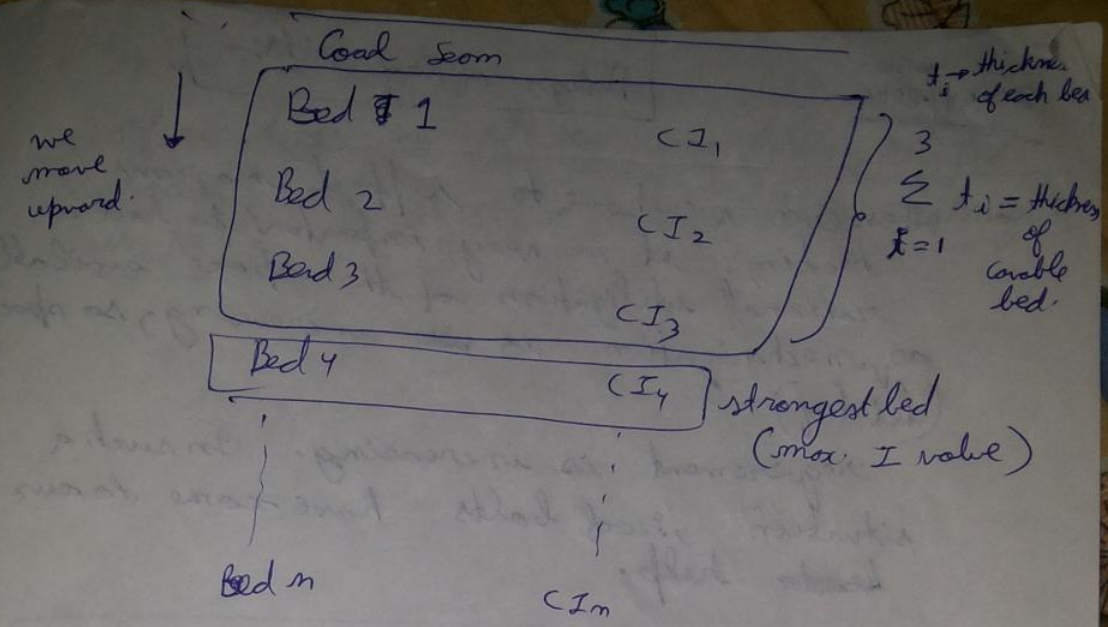
⇒ I is the cavability index of strongest bed

⇒ A is a constant depending on rock type as 1440 in lot I and II, 1700 for lot III and IV and 1900 for lot V rocks

⇒ K' is a factor depending on the ratio of thickness of cavable bed in between the strong bed ~~and the coal seam and~~ extraction height causing the weighting and the coal seam, and the extraction height; its value is 2 for ratio upto 2, 3 for ratio between 2 to 4 and 5 for ratio between 4 to 8 and 10 for values above 8, ~~K is 0.025 for sandstone~~

K is 0.025 for sandstone (strong beds)

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in



<u>For all practical purposes</u> → the convergence at the face is limited to 60mm/m (permissible maximum convergence)

⇒ Then we can obtain the support requirement. P

Then the value of P is magnified or enhanced by 30% to look into the leakages in hydraulic circuit of supports.

we observe that the support capacity specified by the ~~Com~~ manufacturer and the capacity at which supports are used in field ~~have~~ has a variation of around 30%.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

20/09/2013-

Design of Roof bolts:-

- Whenever we have to support narrow headings it is very important to have a rational utilization of the space available as, mechanization is increasing, so space (level of) requirement is increasing. In such a situation, roof bolts have come to our help.

- If we are able to make use of roof bolting technology effectively then we can control all situations of roof failure.

  So we should have.

  1.) knowledge of roof bolt design
  2.) knowledge to select the correct roof bolt for the purpose

- Roof failure has been the primary concern in all kinds of U/G mines

- However for centuries, all support systems were passive and external. They were roof to floor supports, such as timber props or rule supports.

- These supports require a great amount of timber and constant maintainance, further more they are not effective in controlling roof stability

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

narrow
t to have a
pace available
sing, so space

In such a
ome to our

roof
y then
tions of

bolt design
correct
purpose

rn in

ystems
ere roof
rops

nt
rther
lling

- In an attempt to reduce the number of accidents caused by roof fall, USBM advocated the use of roof bolting technique in 1947

Advantages of roof bolting:-

- Reducing storage and material handling requirements
- Decreasing the size of opening that is needed to achieve the same given clearance (bolts provide less hindrance in U/G pathways so size of opening can be reduced so maintaince of opening becomes easier)
- Preventing any appreciable roof deformation by quick installation after opening is excavated.

The maximum intensity of deformation is in the green roof zone so it is necessary to install support as soon as possible, which is the case of roof bolts.

- Improving ventilation by lowering the resistance to the air via elimination of obstruction, such as cribs, posts, girders, etc.

- Providing greater freedom for trackless vehicles without risk of dislodging support.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

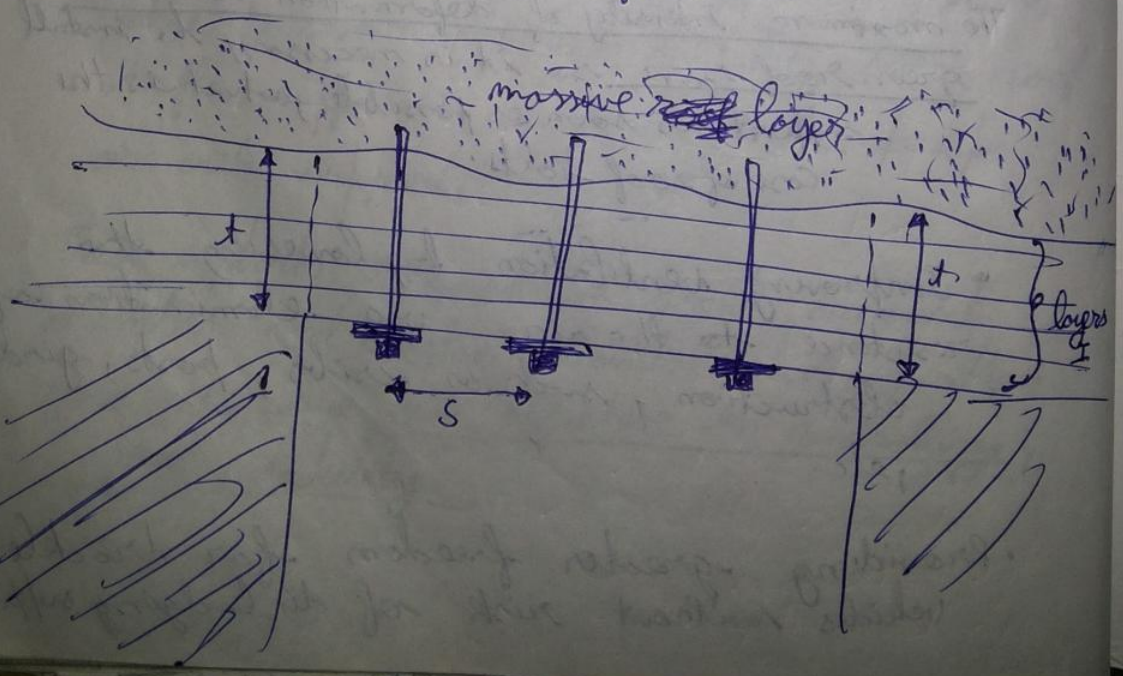
- Providing natural supports for hanging pipes, tubes, electrical cables, etc.

Theories of roof bolting:-

- Suspension theory
- Beam theory
- Keying theory

Suspension theory :-

In a given heading if a roof has a massive roof very close to the roof of the heading then it is possible to bound the layers ~~of~~ between the roof and the massive roof such that these intermediate layers will not fail as they are clamped to a layer which is ~~non fallable~~ massive and will not fail.



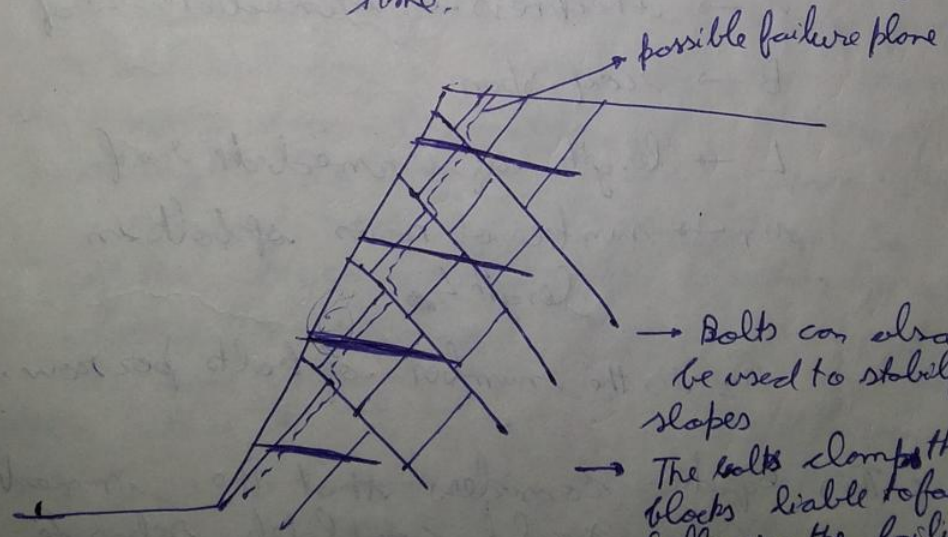
Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

The layers I are liable to fail as their inherent strength is low due to the weakness planes in them. They will not remain intact for a long time after the drivage of the gallery. But their strength can be increased by clamping or suspending them with the massive roof which will not fail easily. The immediate roof will not fail till the massive roof fails.

— when an opening is made, the strata directly overhead tends to sag and if this strata is not supported adequately and properly in time, the laminated immediate roof could separate from main roof and fall out in due course of time

Suspension mechanism of roof bolting :-

- When an underground opening is made, the strata directly overhead tend to sag
- If the strata are not ~~suspended~~ supported adequately and properly in time, the laminated immediate roof could separate from the main roof and fall out in due course of time.



blocks of rock lying at an inner location so slope stability is improved.

→ The orientation of bolts w.r.t the slope face and the failure blocks is very important.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

- In situations, roof bolts anchor the immediate roof to the self supporting main roof by the tension applied to the bolts. In some instances, it appears that the immediate roof is suspended from the main roof by the bolts.

- The bolts have to carry dead weight of the strata between bolt head and the point of anchorage. The load to be carried by each bolt can be calculated as

$$P = \frac{WtBL}{(m_1+1)(m_2+1)}$$

where :-

$W \rightarrow$ unit weight of immediate roof

$t \rightarrow$ thickness of immediate roof

$B \rightarrow$ roof span

$L \rightarrow$ length of immediate roof

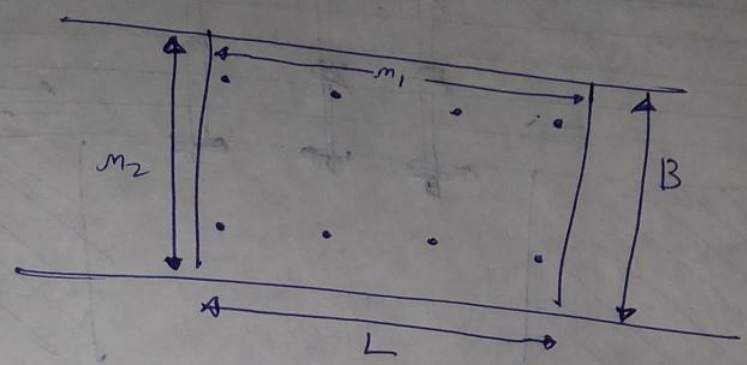
$m_1 \rightarrow$ number of rows of bolts in length L

m_2 is the number of bolts per row.

- This equation considers that the immediate roof would completely separate from the main roof such that it is suspended entirely by the bolts. A portion of the weight supported by abutment

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

on both sides of the opening is also ignored. Hence this formula always gives an overestimated value of the load on the bolts.



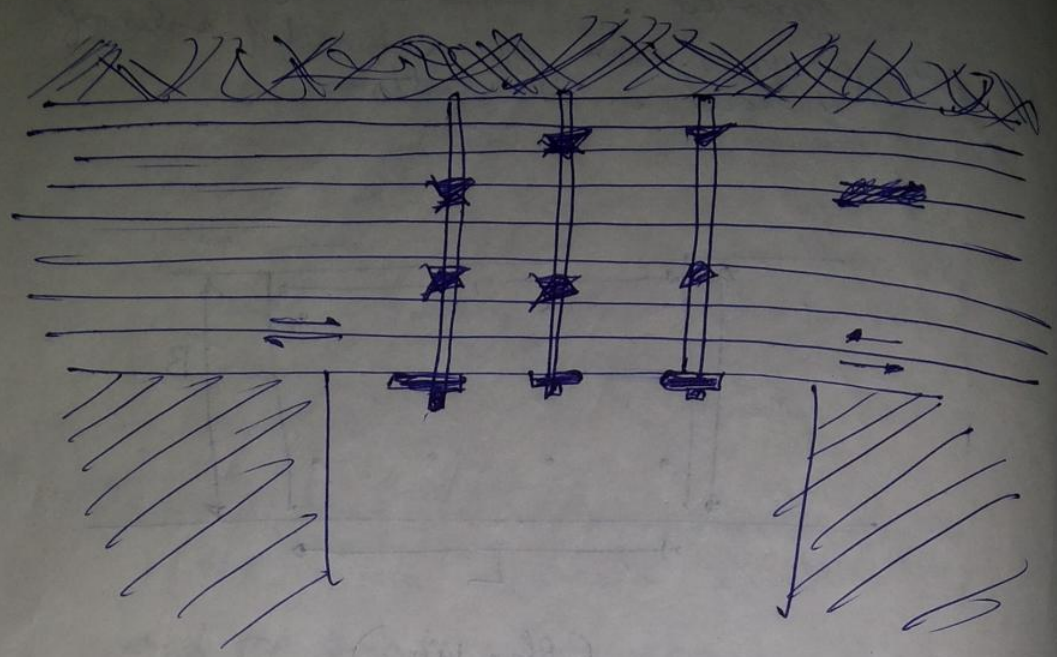
(Plan View)

Beam Building Mechanism:-

If self supporting roof is too away from the roof of the gallery then it is not possible to make use of suspension theory. The roof bolts can be used to support the roof using the Beam theory.

The individual layers between the roof and massive layers ~~then~~ are thin beams which ~~with~~ do not ~~have~~ have the inherent strength to stabilize on their own. They are not self sustaining layers. So if the layers are binded together by bolts then we are able to increase the inherent strength of the beams by making them behave as an individual thick beam/layer. Hence we are able to change the bending behaviour of the immediate thin layers by binding them using roof bolts.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in



- In most cases the strong and self supporting main roof is beyond the reasonable distance that ordinary roof bolts can reach to anchor for suspension. However roof bolts can be applied in such situation with great success.

- In fact, sagging and separation of roof laminations cause ~~the~~ both vertical movement and horizontal movement along the bedding interfaces.

- Bolts installed through these layers can prevent or greatly reduce horizontal movements

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

- Tension applied to the bolts manually on installation or induced by vertical movement of rock, clamps the layers ~~of different~~ together, making them to move with the same magnitude of vertical displacement.

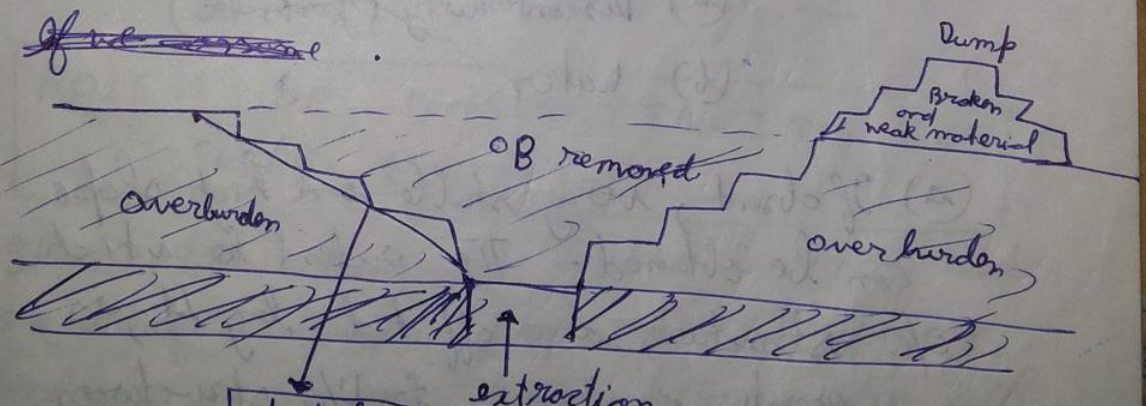
- On the other hand frictional forces (proportional to the bolt tension) are induced along the bedding interface, which also makes horizontal movement difficult.

23/09/2013.

BKS Sir

Slope Stability Analysis:-

It can ~~be~~ broadly be about two things:-
(i) rock mass
(ii) Dump material



pit slope → slope of a line joining the toes of each bench and extending from the surface upto the pit bottom

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

* To extract coal, the upper material (overburden) has to be extracted.
Stability of the slope thus formed is very important. This is stability of rock mass

* The extracted material is kept somewhere outside / inside so a dump is created. It contains broken and weak material. The stability issues are called dump stability problems.

* If we have a flatter slope the the stability is greater but we have to extract huge amts of OB

But

if we have a steeper slope then it is favourable from the economic point of view but large slope instability problems may arise.

Thus an optimum and stable pit slope is needed.

* The two factors which are most important while considering rock mass stability are
(a) Discontinuity (joints etc)
(b) water.

(a) If absent, very stable and high slope can be obtained. The extent to which O/C structures are affected by it, is greater compared to U/G structures.

(b) If present then worsens the situation further.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

(*) While considering a pit slope 2 aspects are to be looked into from the stability point of view.

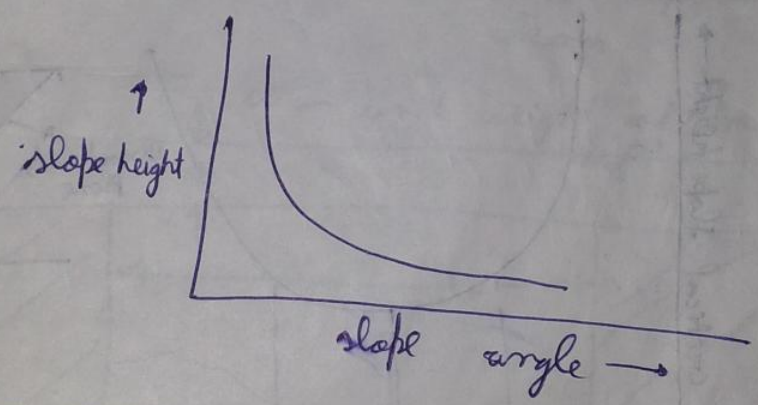
stability → Bench stability
stability → overall pit stability

Individual benches contrive steeper slopes but pit slope stability is a concern and is affected by 2 factors:

(i) Height
(ii) slope angle

Height ↑ , slope angle ↓ for stability

Height ↓ , slope angle ↑ for stability.



ROLE OF DISCONTINUITY IN SLOPE FAILURE:-

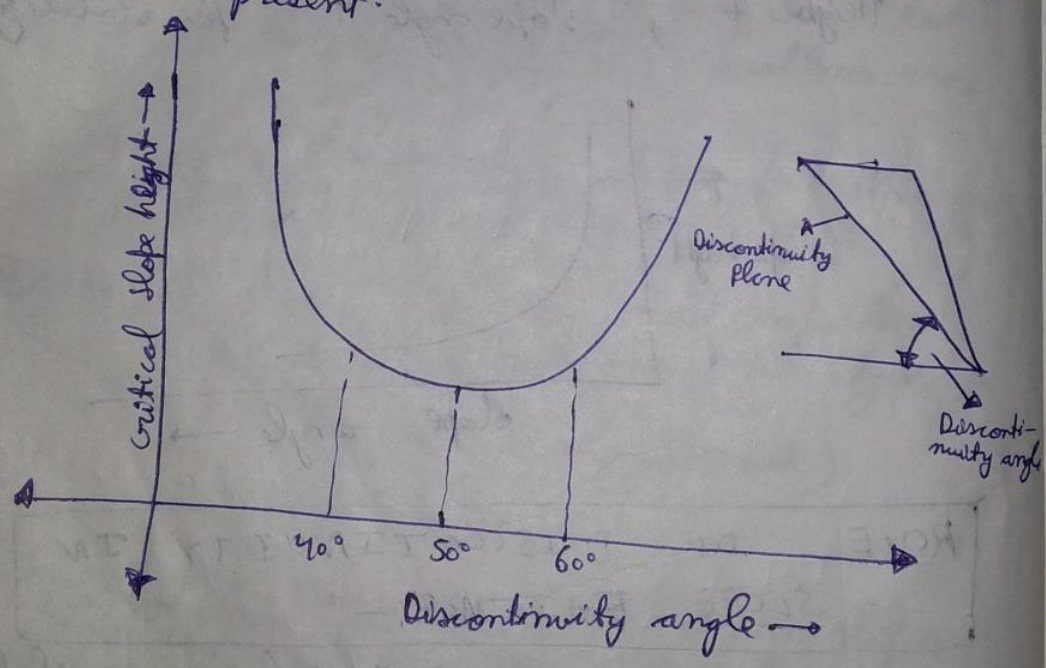
It has been observed that while many slopes are stable at steep angles and at height of hundreds of meters, many flat slopes fail at heights which is considerably low.

This differentle, is due to the fact that stability of rock slope varies with inclination of discontinuity surfaces such as faults, joints and bedding planes within the rock mass. When

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

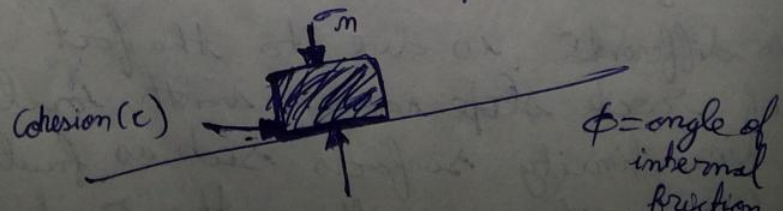
these discontinuity are vertical / horizontal simple sliding cant take place and the slope failure will involve fracture of intact blocks of rock as well as movement along some of the discontinuities.

On the other hand, when the rock mass contains discontinuity surfaces dipping towards the slope face at angles between 30° and 70° simple sliding can occur and the stability of these slopes is significantly lower than those in which only horizontal and vertical discontinuities are present.



Relation between shear and normal stress

$$\tau = c + \sigma_n \tan\phi$$



Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

26/09/2013.

GSPS Sir

Beam theory (Continued)

- If we assume that all the thin layers are clamped to form a thick layer, the maximum bending strain at the clamped ~~ends~~ of the composite beam is given by:-

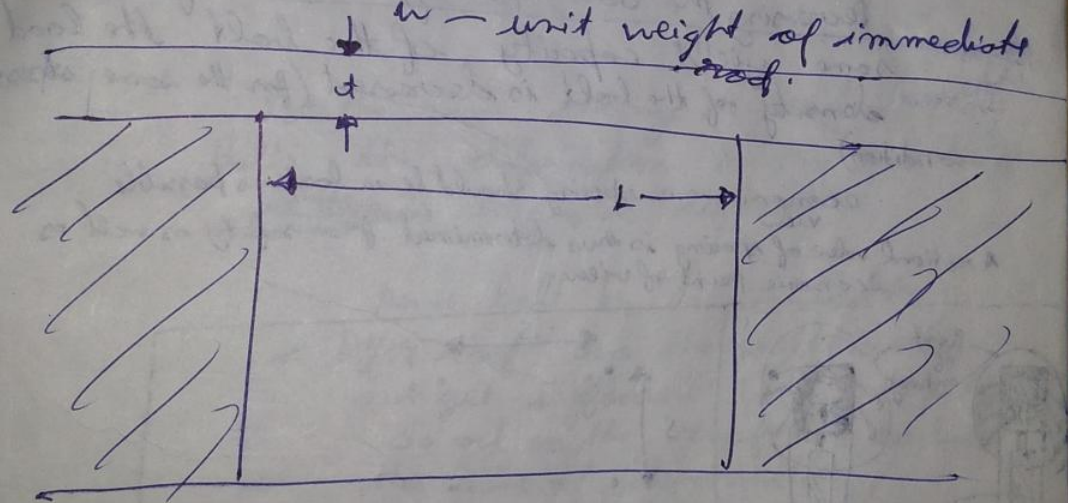
$$\varepsilon_{max} = \frac{WL^2}{2Et}$$ (there is no consideration of insitu stress)

E — Young's modulus

L — length of the immediate roof ~~in the~~ exposed in the gallery

t — thickness of composite beam

w — unit weight of immediate roof.



We are going to have the control on the bending strain by binding the roof layers together (t↑) and hence we are controlling the bending stress and hence we can control the stability of the roof.

Hence by roof bolting we are improving the immediate roof stability.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

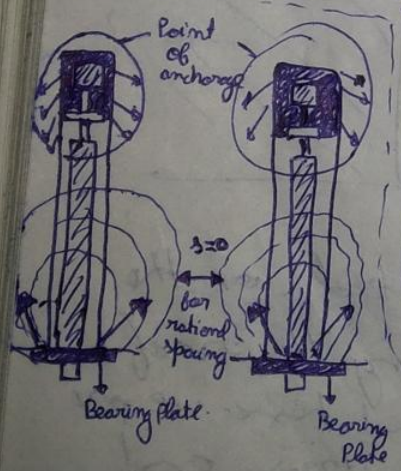
- The above expression indicates that the the thicker the beam, the smaller the maximum strain induced at the clamped ends i.e. the clamping action produces a beam building effect.

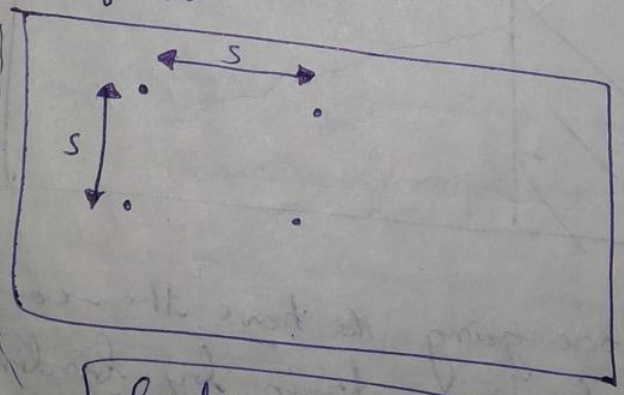
- Findings of some other research indicate that the beam building effect increases with
  - decreasing bolt spacing
  - increasing bolt tension
  - increasing no. of bolted laminal/layers
  - decreasing roof span

Decreasing the bolt spacing → so for the same yield capacity of the bolt the load density of the bolt is decreased (for the same stress condition)

economic view → spacing should be as large as possible

A rational value of spacing is thus determined from safety as well as economic point of view.





loading of the bolt:-

1.) from the dead weight of the immediate roof

2.) from the main roof

→ The anchorage will neutralize the tension in the bolt by development of a compression zone at the zone of anchorage.

→ As the roof descends, the bearing plate try to restrict the downward movement of the bolt, hence a compressive zone of larger extent of influence than at the point of anchorage is established.

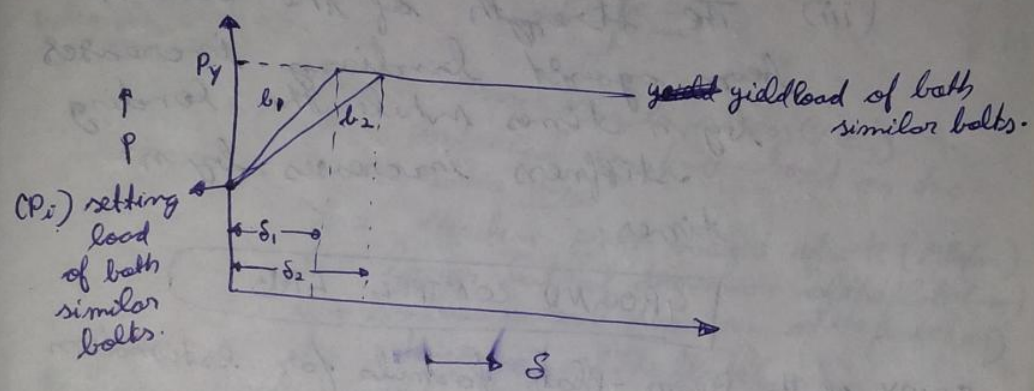
Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

optimum spacing → we should decide for a spacing such that the outer ~~flux of~~ line of the stress flux of two adjacent bolts have a minimum gap between each other (s=0)

increasing the laminae of bolt spacing →

~~thicker~~ thicker layer → stiffer also

A structure which is stiffer in nature induces a larger load for the same amt. of deformation compared to a structure which is softer in nature.



For the same change in load of the bolts the deformation in the two roof ~~bolts is~~ layers is different.

Here bolt b_1 is installed in a stiffer roof. so the bolt does not get sufficient opportunity to act on the roof to reduce the rate of deformation.

Bolt b_2 is installed in a softer roof which allows a larger deformation for the bolt to yield. It gives enough opportunity to the bolt to act on the roof.

Here we should keep in mind that the length upto which the bolt is binding is not so large that the stiffness of the roof is increased to such a large value that the bolt is not getting any opportunity to act on the rate of deformation

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

- The beam building effect has a dual contribution towards the stability of the immediate roof.
  (i) It increases the bending strength of the composite beam and also
  (ii) increases the bending stiffness of the layer
  (iii) The strength of the bolted beam against bending increases by n times while the bending stiffness increases by n^2 times.

GROUND CONTROL LAB

STUDY of the Beam-Plate formula for estimation of roof failure span in longwall workings.

Q) A longwall face = 150 m length is extracted. The strata condition is as follows:-

Fine grained Sandstone = 3 m f = 5 MPa
density = 2500 kg/m3

Fine grained Sandstone = 12 m , f = 5 MPa
density = 2500 kg/m3

Shaly sandstone = 1 m f = 3 MPa
density = 2500 kg/m3

Coal seam = 2.5 m , density = 1400 kg/m3

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Also estimate the effect of reduction of face length on the main fall span if a reduced face length of 80m is implemented.

Ans: THEORY:-

The caving span of roof is estimated using the approach suggested by Obert and Duvall (1967) based on the theory of plate and beams originally developed by Timoshenko et al (1959) for tensile failure for a gravity loaded plate or beam clamped on all edges. The maximum tensile stress likely to develop in the plate under the effect of its bending due to gravity loading is given by the expression below:-

$$\sigma_{max} = \frac{\sigma \beta \gamma_e a^2}{t}$$

where:-

σ_{max} = max. tensile stress (MPa)
β = empirical constant based on the ratio b/a
γ_e = effective unit weight of rock (MPa/m)
b = longer lateral dimension of the plate (m)
a = smaller lateral dimension of the plate (m)
t = thickness of the plate (m)

The effective unit weight of the rock is calculated by:-

$$\gamma_e = \frac{E_1 t_1 (\gamma_1 t_1 + \gamma_2 t_2 + \cdots)}{E_1 t_1^3 + E_2 t_2^3 + E_3 t_3^3 + \cdots}$$

where:-

E_1, E_2 etc are the young's modulus of the rock layers and γ_1, γ_2 -- are the unit weights and t_1, t_2 -- are their thickness values.

The values of β for different b/a values as suggested by Timoshenko is as follows:-

b/a	1.00	1.25	1.50	1.75	2.00
β	0.0513	0.0665	0.0757	0.0806	0.0829

For estimating the failure span of a beam (b/a > 2), the failure span is given by:- (For estimating 1st local fall and Main fall)

$$L_B = \sqrt{\frac{2ft}{\gamma_e}}$$

For estimating the subsequent local fall span and the periodic caving span cantilever formula is used:

$$L_c = \sqrt{\frac{ft}{3\gamma_e}}$$

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

where:- $f \rightarrow$ flexural strength of the layer involved in specific caving event.

$\gamma_e \rightarrow$ effective weight density.

For estimating local fall span and subsequent local fall span, it is the immediate roof which is involved in the process of caving where as main roof is involved in main fall and periodic caving process.

CALCULATIONS:-

→ Face length = 150 m

1st **First local fall span:-**

$t = 1m$

$f = 3 MPa$

$\gamma = 0.025 MPa/m$

(γ_e is not required because no additional loading from ~~the~~ the main roof.)

$$L_B = \sqrt{\frac{2ft}{\gamma_e}} = \sqrt{\frac{2 \times 3 \times 1}{0.025}}$$

$$= 15.5 m$$

2nd **Subsequent local fall:-**

$t = 1m$

$f = 3MPa$

$\gamma = 0.025$

(γ_e is not required because no additional loading from main roof)

$$L_c = \sqrt{\frac{ft}{3\gamma_e}} = \sqrt{\frac{3 \times 1}{3 \times 0.025}} = 6.32 m$$

E_1 = modulus of main roof

E_2 = modulus of loading roof

Assuming $E_1 = E_2$ ∴ (Since main and loading roof are made of same material)

$$\gamma_e = \frac{(12)^2 [(2500)(12) + (2500)(3)]}{[(12)^3 + (3)^3]}$$

$$\therefore \gamma_e = 0.0308 \ MPa/m$$

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

4th **Periodic caving span**

$t = 12m$

$f = 5 MPa$

$\gamma_e = 0.0308 \ MPa/m$

$$L_c = \sqrt{\frac{ft}{3\gamma_e}} = 25.48m$$

3rd **Main fall span:-**

$t = 12m$

$f = 5MPa$

$\gamma_e = 0.0308 MPa/m$

$$L_B = \sqrt{\frac{2ft}{\gamma_e}} = 62.42m$$

With the face length of 15am $b/a > 2$ at all ~~instan~~ instances so beam formula can be used with reasonable accuracy.

when face length = 80m :-

For local fall and subsequent local fall $b/a > 2$ since they occur (at a) quite early span so they remain unchanged as the same beam and cantilever formula is used to calculate their spans.

But with reduction in face length to 80m,

For same main fall span $b/a < 2$ and so the main roof behaves as a plate ~~sq~~ is likely to change. (main fall span)

So for the same Main fall span $b/a = \frac{80}{62.4} \simeq 1.28 < 2$

Now the value of β corresponding to the $b/a = 1.28$ will be 0.0676 (by linear interpolation)

$$\therefore \sigma_{max} = \frac{6\beta \gamma_e a^2}{t} = \frac{6 \times 0.0676 \times 0.0308 \times (62.4)^2}{12}$$

$\therefore \sigma_{max} = 4.05 MPa < 5 MPa$ of Main roof so Main fall will not occur.

Main fall failure will occur only when σ_{max} = flexural strength of the Main roof for a certain value of a and corresponding value of β depending on b/a where a is the new smaller dimension.

The value of a estimates as:-

$$\sigma_{max} = 5 = \frac{6 \times \beta \times 0.0308 \ a^2}{12}$$

$$\beta a^2 = 324.675$$

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

By hit and trial :-

For $a = 80$ m $\quad b/a = 1 \quad B = 0.0513$

$$\sigma_{max} = \frac{6 \times 0.0513 \times 0.0308 \times (80)^2}{12}$$

$= 5.05$ MPa) 5 MPa ~~for~~ of the main roof

So the new main fall span = 80 m

↳ when the face length is reduced to 80 m

27/09/2013

Beam building Mechanism (continued)

- For a beam consisting of n identical layers without bolt (all the layers are independent to have their own bending, there is no mutual interaction between the layers.

$$B = n\frac{bh^2}{6}$$

$$T = \frac{nEbh^3}{12}$$

- For a composite beam consisting of n identical layers with bolts binding them together

$$B_1 = \frac{b(nh)^2}{6} = nB$$ (enhanced by a factor n after installation of bolt)

$$T_1 = \frac{Eb(nh)^3}{12} = n^2 T$$

The proportion by which the bending stiffness (T) is increasing is further n times more than the ~~increase~~ proportional increase in B.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

B = 0·0513

× 0·0308 × (80)²

2

MPa ~~for~~ of the Main roof

= 80 m

is reduced to 80 m

nism (continued)

ical layers independent to interaction between

f n identical them

(enhanced by a factor n after installation of bolt)

T

tiffness (T) is ore than the rease in B.

Where:-

E → Young's modus.

n → no of identical layers

h → thickness of each layer

b → length of the beam (span)

B_0, B → Bending stress of beam (Bending strength of the beam)

T, T → Bending stiffness of the beam.

- The improvement in bending strength is always good for roof stability. But increase in bending stiffness in some cases extra load from the overlying strata acting on the beam.
- Under such ~~a~~ case, the beam may undergo shear failure (in place of commonly expected tensile failure) at the 2 ends once the accumulated shear stresses exceed the shear strength. This type of failure is called Cutter roof failure.
- This kind of failure has the following feature:-
  - The bolted composite beam falls out
  - Failure planes at two ends of the beam are nearly vertical
- The upper failure plane is exactly at the bolted horizon where pretension of the bolts creates a tensile stress area around the anchor of each bolt.
- Sometimes, using longer bolts just creates increased height of roof fall.

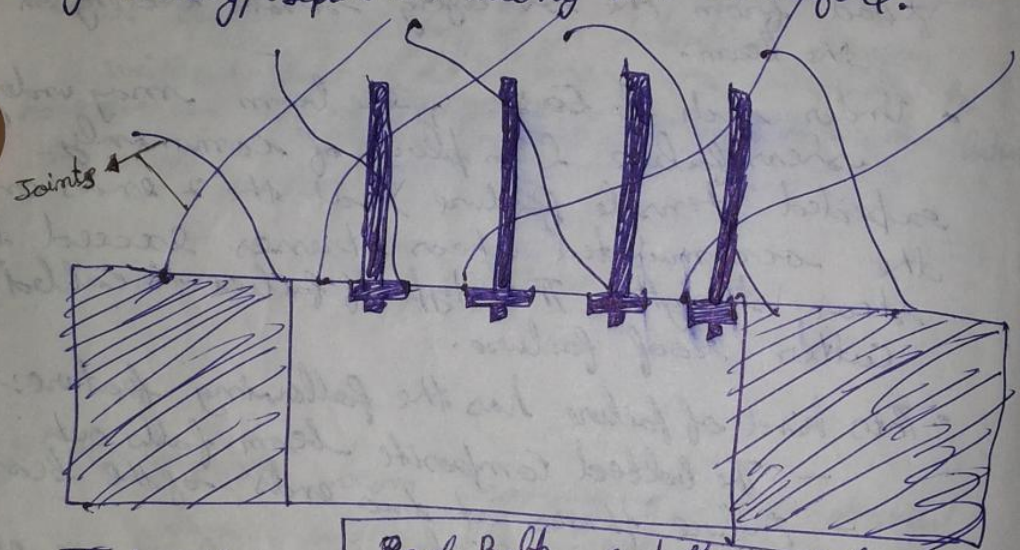
Keying mechanism of bolts:-

If we have to support above which have intersecting joint planes. On creation of galleries there are some blocks which are liable to undergo failure. These are critical rock blocks. The exposed rock ~~bolts~~ blocks have a higher potential to undergo failure. To support these

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

~~bolts~~ → critically unstable blocks we are trying to bind these blocks with other blocks which are stable thus increasing the overall stability of the roof. Otherwise there would be chances of shear failure of these unstable blocks along the joint planes thus resulting in fall of such blocks of rock.

- When the roof strata, are highly fractured and blocky, or the immediate roof contains one or several sets of joints with different orientation to the roof line,
  - roof bolting provides significant frictional forces along fractures, cracks and weak planes, thus preventing sliding/separation along the interface.



~~Types of roof~~ Roof Bolts and their applications:-

Type of bolt	Type of anchor	Suitable strata type	Comments.
Point anchored tensioned bolt	~~Split~~ slot and wedge	Hard rock	used in early stages
	Expansion shell	Medium strength	
	Resin grout	All strata especially weak rock	increasing use recently
Full length grouted (untensioned)	Cement	Most strata	Good anchorage with "no mix" resin
	Perfo		
	Injection		
	Cartridge (quick setting cement in slurry) →	(just after 30 min of setting the bolt gives 30-40% of yield load capacity)	

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Type of bolt	Type of anchor	Suitable strata type	Comments
Roof truss	Expansion shell	Adverse roof	To be used at inter-sections and heavy pressure areas

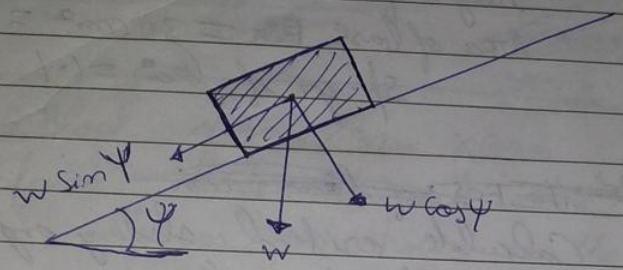
Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

GROUND CONTROL

(BKS)

30/09/2013

Sliding due to gravitational loading:-



$$\sigma = \frac{W\cos\psi}{A}$$

$$\tau = c + \frac{W\cos\psi}{A}\tan\phi$$

Resisting force $= CA + W\cos\psi \cdot \tan\phi$

Sliding force $= W\sin\psi$

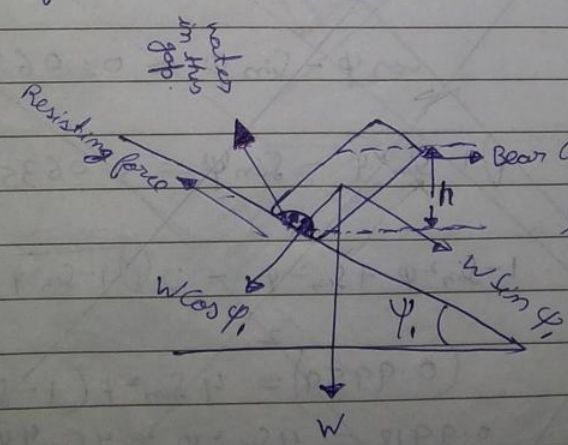
$c \rightarrow$ cohesive forces

$\phi \rightarrow$ angle of internal friction

$\therefore$ FOS $= \dfrac{\text{Resisting force}}{\text{Sliding force}}$

FOS = 1 — limiting condition

Influence of water pressure:-



let water pressure at the base of can $= U = \gamma_w h$

$h \rightarrow$ height of fluid

Let Area of base $= A$

Resisting force $= CA + (W\cos\psi_1 - UA)\tan\phi$

sliding force $= W\sin\psi_1$

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Q.

weight of Can = 500 gm (W)

Coefficient of friction between surface and can = 30° (ϕ)

height of bear = 15 cm. = h

Area of base = 30 cm² = A

Sp. gr. of bear = 1.1

~~Calculate the FOS of bear can~~

Calculate critical sliding angle

(i) when bear can is punctured

(ii) when bear can is not punctured.

→ Ignore cohesion :-

Ans:

(i)

$$450 \cos\psi_1 - \left(1.1 \times 10 \times 1000 \times \frac{15}{100} \times \frac{30}{10000}\right) \times \tan 30^\circ = 450 \sin\psi_1$$

$$\Rightarrow 450 \cos\psi - 3.3 \times 15 \times \frac{1}{\sqrt{3}} = 450 \sin\psi_1$$

$$\Rightarrow 450 (\cos\psi - \sin\psi) = 28.58$$

$$\cos\psi - \sin\psi = 0.0635$$

$$\sqrt{1-\sin^2\psi} - \sin\psi = 0.0635$$

$$1 - \sin^2\psi + \sin^2\psi - 2\left(\sqrt{1-\sin^2\psi}\right)\sin\psi = 4.032 \times 10^{-3}$$

$$(0.9959)^2 = 4\sin^2\psi(1-\sin^2\psi)$$

$$0.9918 = 4\sin^2\psi - 4\sin^4\psi$$

$$\sin^2\psi = \frac{-4 \pm \sqrt{16 - 4 \times (-4)(-0.9918)}}{2(-4)}$$

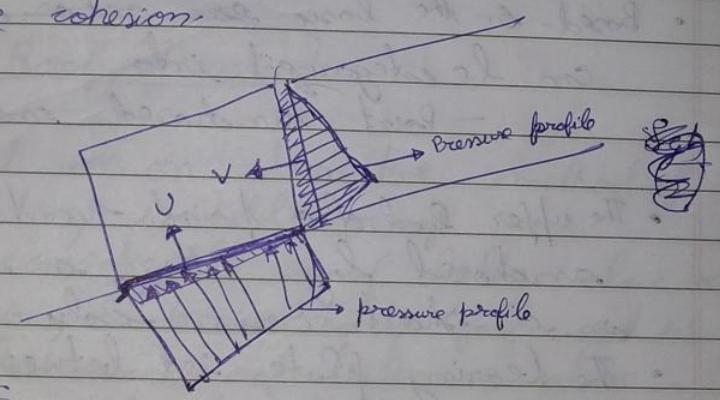
Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Q.2) A Block of 30m X 30m X 30m got detached from its parent rock and is lying on the rockmass as shown in the figure. Considering uplift force and horizontal forces. Calculate the angle of the slope at which it is just going to slide.

Sp. gr. of rockmass = 2.7

coefficient of friction between rock and rockmass = tan ϕ 30°

Ignore cohesion.



Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

03/10/2013

GROUND CONTROL (GSPS)

Types of ~~the~~ roof bolts and variations:-

- Bolts reinforce the rock by binding the stratified ~~or~~ broken roof layers (or blocks) together. The binding effect is achieved through friction forces created by the physical interlocking along the anchor and rock interface.

Friction is ~~Bolts and the covering or anchorage~~ between: (a) Bolt and the grouting material.
(b) grouting material and the rockmass.

- Based on the basic ~~ear~~ ~~anchor~~ anchor types, bolts can be categorized into
  - Point anchored and full length anchored.

- The upper end of a point anchored bolt is anchored by either a mechanical device or a short resin column.
- The bearing plate set between the bolt head and the roof line serves as the other anchor.

Use of Bearing plate:-

When lower end of the roof bolt comes under tension due to downward deformation of the roof as the bolt is trying to resist that. The tensile stress acting at the mouth of the bolt is required to be redistributed (significantly throughout the length of the bolt) or ~~to~~ else there will be failure at the mouth of the bolt and the utility of the bolt is lost. This is the purpose of the bearing plate.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

classmate
Date
Page

03/10/2013

- Usually, a certain amount of tension is applied to the bolt at the time of installation. So bolt acts as an active support. The bolts tends to close the existing weakness planes and the inherent strength of the rockmass increases and the time interval for undergoing a certain amount of deformation by the roof strata increases. This is the benefit of having pre-tension in the bolt.

- A full length grouted bolt is grouted to the borehole by fast setting high strength resin or inorganic cement throughout the full column of the hole. No pretension is applied to the bolt.

Roof truss

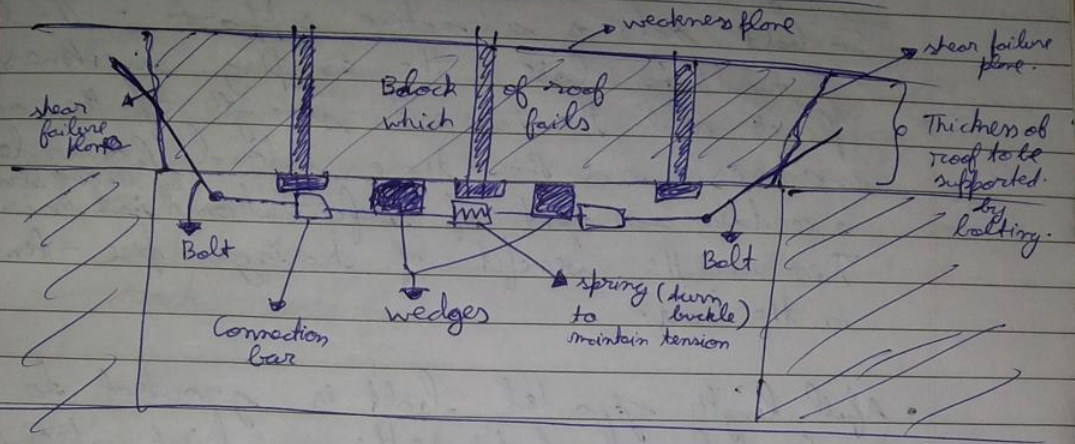
- Introduced as an alternative method of supporting unstable roof
  - when conventional roof bolts alone are not effective, such as when fall out is frequent between roof bolts or where cutter roof failure is encountered.

- Consists of 2 inclined point anchored bolts, a connection bar (provides rigid connection between the bolts), a turn buckle (a spring whose tension can be controlled) to give proper tension to the bar and an adjusting wedge box.
  - supports the roof when there is a tendency of the downward movement of the roof.

- The connection elements utilize and (transmit) the compressive stresses developed at the point of anchorage to support the ~~downward movement~~ roof and prevent its downward movement by creating compression in the roof.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

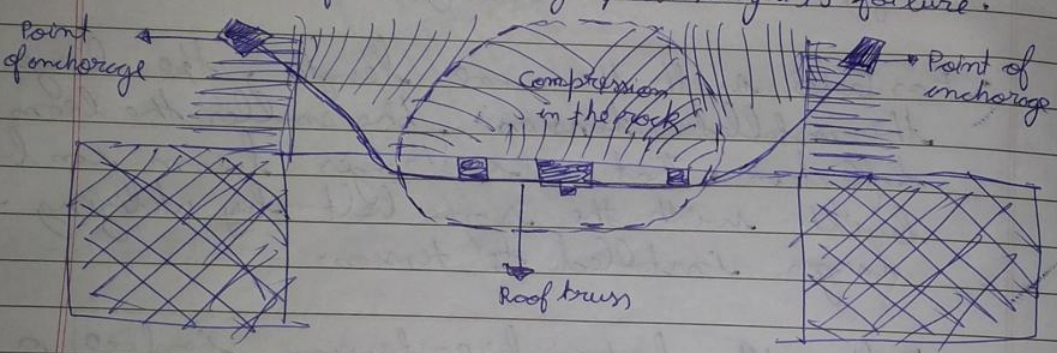
Cutter failure:-



When the roof is so stiff that the roof bolt do not have much opportunity to support the roof (the roof undergoes failure along shear planes (almost vertical) at the corners of the opening) Cracks are developed along the corners of the gallery and these cracks act as shear planes (along which failure takes place) along which the entire block of roof rock exposed to the gallery which is intended to be supported comes down (falls) in one section. The block of rock separated by a weakness plane from above and by shear planes on the corners from the competent roof undergoes shear failure along these shear planes and falls as an intact block of rock. This type of failure of the roof is called cutter failure. This is not the failure of an individual bolt but the failure of the entire roof of the gallery along shear planes. By putting a vertical bolt or inclined bolt in the block mass liable to undergo cutter failure, we will not be able to reduce the chances of roof fall.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

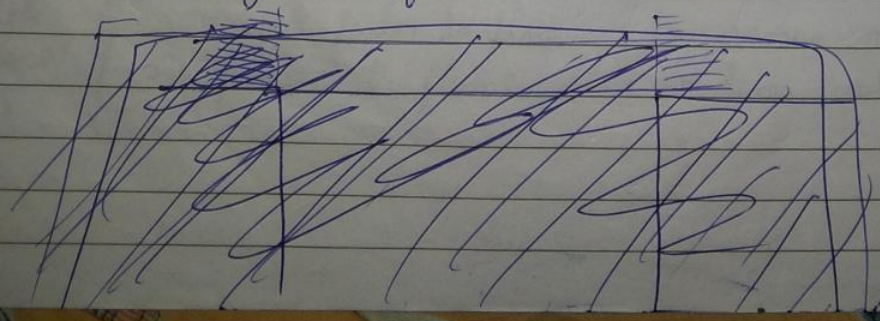
So we install two inclined bolts in such an orientation that the point of anchorage of the bolts still lies in the roof layer but ~~at~~ an a ~~block~~ section of roof rock which is much more competent and is not liable to undergo such failure. This competent block lies in the part of the roof strata not exposed by mining. The two inclined bolts are connected by a series of elements which maintain proper tension the bolts. ~~To~~ By the use of connection elements we are able to translate/transfer the compressive ~~f~~ stresses developed at the point of anchorage in the competent roof section to support the block of roof ~~liable to undergo~~ exposed by mining which is liable to undergo utter failure. This entire arrangement is a roof truss. The failure of the roof of the gallery initially is due to tension but by installation of roof truss mechanism we are able to create compression in the roof rock thereby preventing its failure.



- The tensioned connection bar keeps the roof in compression and can also yield downward as the roof displaces.

- In 3 or 4 way intersections where tensile stresses are very high, trusses can produce compressive forces to offset high tensile stresses at mid intersection roof.

- Trusses are ~~also~~ ~~or~~ very useful for supporting newly exposed roof.



Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Some facts about roof bolts:-

- The longer the bolt, the larger the beam deflection. Alternatively, the shorter the bolt, the smaller the deflection and the stronger the beam.
- The longer the bolt, the larger the load that will be induced in the bolt as the result of in-situ or mining related stresses.
- For bolts with the same length, the larger the installed tension, the smaller the beam deflection. That is a stronger beam can be built with the same bolt by utilizing a larger installed ~~to~~ tension.
- Bolts with higher pre-tension induce a smaller deflection within the bolting range, which in turn, results in a smaller deflection higher up in the roof above the bolting range.

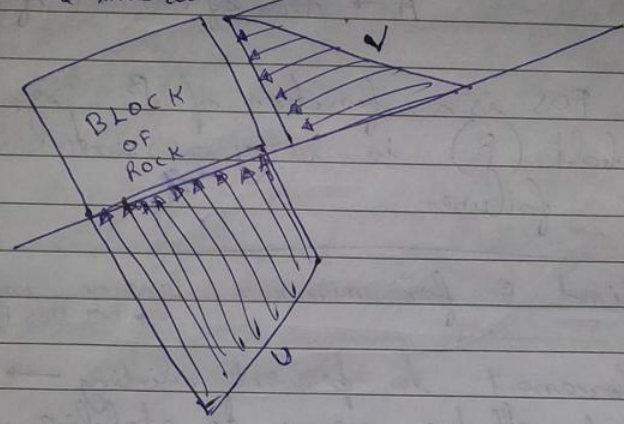
Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

~~solution of questions of BKS Sir given on 20/09/2013~~

21/10/2013.

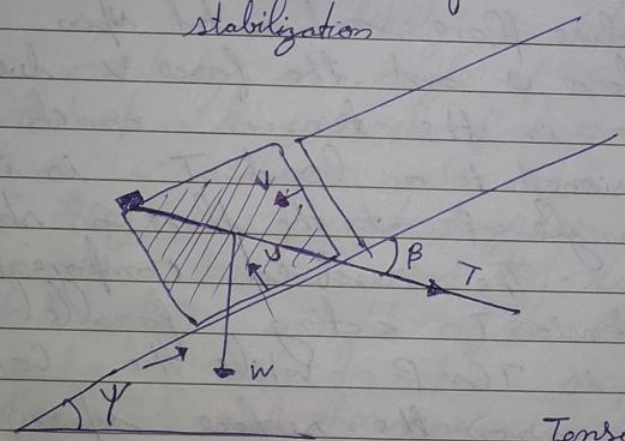
GROUND CONTROL (BKS Sir)

* If Both water columns are interconnected.



Case of rock reinforcement:-

→ Bolt is used for rock slope stabilization



Tension of bolt has 2 Components:-

Both increase FOS.
1.) perpendicular to plane → increases normal stress
2.) parallel to plane → reduces sliding force

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

At equilibrium:-

$\boxed{FOS=1} \Rightarrow W\sin\psi + V - T\cos\beta = CA + (W\cos\psi - U + T\sin\beta)\tan\phi$

where:-

T → tension in Bolt

A → Area of contact of rock block

FOS is a function of β so if orientation of bolt (β) is very essential to control failure.

Find β for minimum Tension in the bolt:-

→ For this FOS = 1 (Similarly β for minimum T can be found out for any value of FOS)

Reinforcement to prevent sliding → One of the most effective means to stabilize slabs of rocks which are likely to slide down inclined discontinuity surfaces is to install tensioned rock bolts. Consider the block resting on the incline plane and acted upon by the uplift force U and the force V due to water pressure in the tension crack. The rock bolt tensioned to a load T is installed at an angle β to the plane as shown in the figure. The resolved component of the bolt tension T acting parallel to the plane is $T\cos\beta$ while the component acting across the surface upon which the block rests is $T\sin\beta$. The condition of limiting equilibrium for this case is defined by:-

$$W\sin\psi + V - T\cos\beta = CA + (W\cos\psi - U + T\sin\beta)\tan\phi$$

It is obvious that the Bolt tension reduces

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

the distributed forces acting down the plane and increases the normal force and hence the frictional resistance between the base of the block and the plane. The minimum bolt tension required to stabilize the block is obtained by rearranging the given equation to give an expression for bolt tension T and minimizing this expression w.r.t angle β and differentiating w.r.t β partially and putting it equal to zero

$\boxed{\text{On Calculating:-}}$ So for minimum T:-

$$\beta = \phi - \psi$$

Q.) ~~A block of rock~~ A block of rock of size (30 × 30 × 30) m³ was detached from the parent rock.

and likely to slide on an incline surface making an angle of 45°. The water has filled up in the crack between the parent rock both in vertical as well as inclined plane and they are joined together. The coefficient of friction is tan 30° and the cohesive forces are 1/5th of the weight of the block. A bolt has to be inserted to ~~stab~~ improve the FOS of this block and the tension in the bolt is equal to 1/10th of the weight of the block.

Calculate the FOS of the block (Sp. gravity of rock = 2.7)

(a) in dry condition

(b) in wet condition (both joints interconnected)

(c) in wet condition (joints not interconnected)

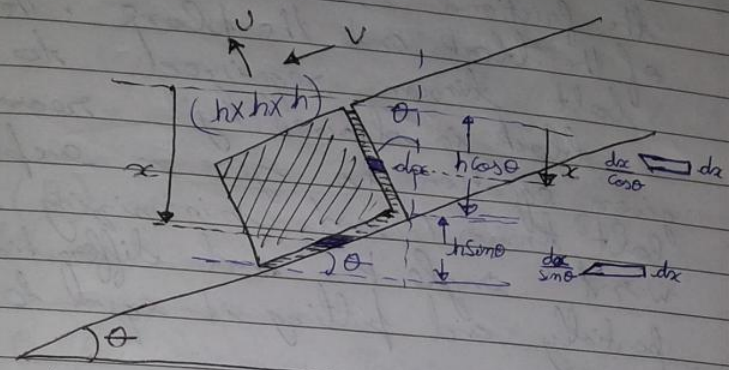
Calculate (d) minimum angle at which the bolt should be inserted to improve the FOS. (i.e. for FOS to be maximum)

$\boxed{FOS = 1}$ →

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

(e) the FOS of the block when the bolt is inserted.

Solution:-



<u>when water columns are interconnected:-</u>

$$dV = \rho g x \frac{dx\, h}{\cos\theta} \Rightarrow V = \int_0^{h\cos\theta} dV$$

$$\therefore V = \frac{\rho g h^3 \cos\theta}{2}$$

$$dU = \rho g x \frac{dx\, h}{\sin\theta} \Rightarrow U = \int_{h\cos\theta}^{h\cos\theta + h\sin\theta} dU$$

$$\therefore U = \frac{\rho g h^3}{2}\left[\sin\theta + 2\cos\theta\right]$$

<u>when water columns are not interconnected:-</u>

$$dV = \rho g x \frac{dx\, h}{\cos\theta} \Rightarrow V = \int_0^{h\cos\theta} dV$$

$$\therefore V = \frac{\rho g h^3 \cos\theta}{2}$$

$$dU = \rho g x \frac{dx\, h}{\sin\theta} \Rightarrow U = \int_0^{h\sin\theta} dU$$

$$\therefore U = \frac{\rho g h^3 \sin\theta}{2}$$

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

$h = 30m$ $\theta = 45°$ sp. gr = 2.7 $\rho_w = 1000\ kg/m^3$ $g = 10 m/s^2$ $\phi = 30°$

(a) $$FOS = \frac{CA + W\cos\theta \tan\phi}{W\sin\theta} = \frac{\frac{W}{5} + W\cos\theta \tan\phi}{W\sin\theta}$$

$$= \frac{(1/5) + \frac{\tan 30°}{\sqrt{2}}}{1/\sqrt{2}} = 0.86$$

(b) $$V = \frac{\rho g h^3 \cos\theta}{2} = \frac{1000 \times 10 \times (30)^3 \times \cos 45}{2}$$

$$\therefore V = 95459415.46 N$$

$$U = \frac{\rho g h^3}{2}\left[\sin\theta + 2\cos\theta\right]$$

$$\therefore U = \frac{1000 \times 10 \times (30)^3}{2}\left[\sin 45 + 2\cos 45\right]$$

$$= 286378246.9 N$$

$$\therefore FOS = \frac{\frac{W}{5} + (W\cos\theta - U)\tan\phi}{(W\sin\theta + V)}$$

$$W = 2700 \times (30)^3 \times 10 = 7.29 \times 10^8 N$$

$$\therefore FOS = \frac{\frac{7.29 \times 10^8}{5} + \left[7.29 \times 10^8 \times \cos 45° - U\right]\tan 30°}{(7.29 \times 10^8 \sin 45° + V)}$$

$$\therefore FOS = 0.455$$

Solution:-

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

(c) $V = \frac{\rho g h^3}{2} \cos\theta = 95459415.46\,N$

$U = \frac{\rho g h^3}{2} \sin\theta = 95459415.46\,N$

$$\therefore FOS = \frac{\left(\frac{7.29\times10^8}{5}\right) + \left[\left(7.29\times10^8 \times \cos 45^\circ\right) - U\right]\tan 30^\circ}{\left(7.29\times10^8 \times \sin 45^\circ + V\right)}$$

$\therefore FOS = 0.636$

(d) Assuming water columns interconnected and wet condition.

$$FOS = \frac{CA + (W\cos\theta - U + T\sin\beta)\tan\phi}{W\sin\theta + V - T\cos\beta}$$

$$FOS = \frac{\frac{W}{5} + \left(W\cos\theta - U + \frac{W}{10}\sin\beta\right)\tan\phi}{W\sin\theta + V - \frac{W}{10}\cos\beta}$$

$\frac{d(FOS)}{d\beta} = 0 \longrightarrow$ for max. FOS

$$\therefore (W\sin\theta + V - T\cos\beta)\, T\cos\beta \tan\phi = \left[\frac{W}{5} + (W\cos\theta - U + T\sin\beta)\tan\phi\right] T\sin\beta$$

$W = 7.29\times10^8\,N$ $\qquad T = W/10 = 7.29\times10^7\,N$

$U = 286378246.4\,N$

$V = 95459415.46\,N$

$\theta = 45^\circ$

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Substituting above values we get the equation:-

$$2.79\sin\beta - 3.53\cos\beta + 0.421 = 0$$

$$\Rightarrow \beta = 46.37^\circ \quad \text{for max. FOS}$$

(e) The maximum FOS is:-

$$FOS = \frac{\frac{W}{5} + \left(W\cos\theta - U + \frac{W}{10}\sin\beta\right)\tan\phi}{W\sin\theta + V - \frac{W}{10}\cos\beta}$$

$$= \frac{\left(\frac{7.29\times10^8}{5}\right) + \left[\left(7.29\times10^8\cos45\right) - U + \left(7.29\times10^7\sin46.37\right)\right]\tan30^\circ}{\left(7.29\times10^8\times\sin45^\circ\right) + V - \left(7.29\times10^7\cos46.37^\circ\right)}$$

$U = 286378246.4\,N$

$V = 95459415.46\,N$

$\therefore FOS = 0.55$

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Solution of problems of BKS Sir given on 30/9/2013

A1) $W\sin\theta = (W\cos\theta - \rho g H A)\tan\phi$

$M = 0.45\,Kg$

$g = 10\,m/s^2$

$A = 30\,cm^2$

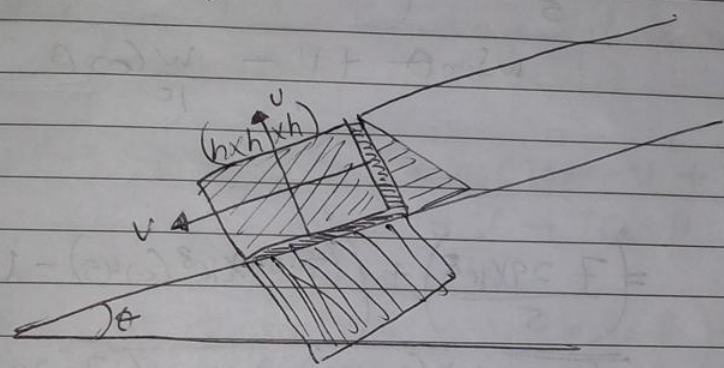
$\rho = 1100\,Kg/m^2$

$H = 0.15\,m$

$\tan\phi = 0.577$

Solve for $\theta = ??$

A2)



Assuming water columns interconnected:-

$$U = \frac{\rho g h^3}{2}\left[\sin\theta + 2\cos\theta\right]$$

$$V = \frac{\rho g h^3 \cos\theta}{2}$$

$h = 30\,m$ $\quad \rho = 1000\,Kg/m^3$ $\quad g = 10\,m/s^2$ $\quad \cancel{\theta = 45^\circ}$

Sp. gravity $= 2.7$ $\quad \phi = 30^\circ$

$W = 7.29 \times 10^8\,N$

$$U = \frac{1000 \times 10 \times (30)^3}{2}\left[\sin\theta + 2\cos\theta\right] = (1.35 \times 10^8)(\sin\theta + 2\cos\theta)$$

$$\therefore V = (1.35 \times 10^8)\cos\theta$$

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

$\therefore (W\cos\theta - U)\tan\phi = W\sin\theta + V$

$\therefore 4.2\cos\theta = 0.78(\sin\theta + 2\cos\theta) = 7.29\sin\theta + 1.35\cos\theta$

$\therefore 1.3\cos\theta = 8.07\sin\theta$

$\therefore \tan\theta = \frac{1.3}{8.07}$

$\boxed{\therefore \theta = 9.15^\circ}$

17/10/2013 GSPS Sir

Design of Roof bolting System:-

For complete and appropriate rock bolting system design, the following parameters must be determined properly:-

1.) bolt type 2.) Bolt length 3.) pattern and spacing of bolts

4.) Bolt diameter and anchor capacity

5.) whether pretension should be applied or not. If yes, what is the appropriate magnitude?

1.) Selection of bolt type:-

→ The selection of an appropriate bolt type to meet specific requirements at the lowest cost and max. effectiveness is a confusing task.

→ Selection of type of bolt also depends on strata condition

→ Careful examination of geological condition and through understanding of performance of different types of bolts under different settings are essential for a successful bolt selection type.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Guidelines for selecting bolt type:-

- Mechanical bolt:-

1. harder rock where the rock properties will not adversely affect the gripping force of the anchor
2. Temporary reinforcement systems
3. where bolt tension can be checked regularly
4. Rock that will not experience high shear forces
5. Rock that is not highly fractured
6. Areas away from blast sites where bolt tension may be lost
7. If the rock is soft. The rock harder has high value of frictional angle. The higher the frictional angle higher the frictional force / gripping force

- Grouted Bolt:-

1. conditions where mechanical bolts are not permitted:-
2. permanent reinforcement systems
3. Boreholes without continuous water run off problems
4. Rock without wide fractures and voids in which significant amount of grout could be lost
5. Untensioned bolts are recommended in rock that is highly fractured and deformable as long as adequate bolt installation is feasible. Generally, bolts in more competent strata often requires shorter grout column than that in less competent strata.
6. Tensioned grouted bolts (point anchored bolt) are recommended for use where additional frictional forces, in combination with a grouted column, may enhance roof stability

⇒ We cannot ~~put~~ pretension in full length grouted bolt that's why we use point anchored bolt.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Advantages of fully grouted resin bolts compared to mechanical bolts:-

1.) Anchorage is virtually guaranteed and independent of strata type
2.) Bolt is permanently effective throughout its full bonding length
3.) The fully grouted bolt prevents both vertical and horizontal strata movement
4.) The hole length is not critical since the bonding can be adjusted according to the type of strata
5.) Fully grouted bolts seal wet holes and exclude air, thus reducing corrosion of the bolt assembly and weathering of rock.
6.) Damage to the bolt head, bearing plate or rock at the collar of the hole does not cause the resin bolts to become ineffective
7.) The resin grouted bolt can absorb blast vibrations without bleed off of the bolt load.
8.) The fully grouted resin bolt shows excellent performance with regard to anchorage creep. When the bolt is subjected to creep loading then the grout bolt or bolt which we have put in the rock should be able to bear that load.
9.) Time and labour for tensioning the bolts are not needed.

<u>A General thumb rule for mechanical bolts:-</u>

- They cannot be reliably installed in critical areas as permanent support, and in places where significant roof deformation is likely to occur. Such bolts may be considered as temporary reinforcement.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

(II) Selection of bolt length :-

- Minimum length of bolt should be greatest than the following three :
  - Twice the bolt spacing
  - Three times the width of critical and potentially unstable rock blocks defined by the average discontinuity spacing in the rockmass
  - For span less than 6 m, bolt length of one half the span, for span between 6-18 m, linearly interpolate between 3-5 m respectively. For excavation higher than 18 m, sidewall bolts are 1/5 of the wall height.

(used mainly for tunnels)

(III) Bolt Spacing :-

(1st)

$$b = \frac{2l}{3} \quad \text{or} \quad b = \frac{2}{9}L$$

} Consider minimum of both

where $b \rightarrow$ bolt spacing

$l \rightarrow$ bolt length (~~maximum~~)

$L \rightarrow$ roof span

- Thumb rule maximum bolt length

(2nd)

- least of :-
  - One half the bolt length zone and one half the width of critical and potentially unstable rock ~~bol~~ blocks (1.8 m generally)

(V) Pretension :-

(1) Although application of pretension is very important for mechanical bolts and resin anchored bolts to create frictional forces between layer ~~of~~ and to reduce interlaminal separation, it is not always true that higher pretension will ensure better stability

(2) Maximum pretension should not exceed 60% of the yield strength or the anchorage capacity of the bolt.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

18/10/2013

Shotcrete Support

(*) Basically it means surface support

Introduction:-

- Shotcrete support has been in use with considerable success in tunnelling projects. In recent years, it has been applied with success for stabilization of vital mining structures like shaft pillars, permanent galleries, and longwall gate roads in coal mines.

Gunite material is used to support

- Basically, shotcrete is a mixture consisting of sand, cement, concrete along with some other additives such as steel fibres and silica fumes.
- The dry mixture is carried through a horoscope by means of compressed air and shot over the rock surface through a nozzle.

Shotcrete technology:-

- shotcreting is the generic name given for cement, sand and fine aggregates which are applied pneumatically and compacted dynamically under high velocity.
- Depending on mixing mechanism, shotcrete can be of 2 types: - wet mix
  - Dry mix

Dry mix shotcrete

- Dry mix shotcrete components, which may be slightly pre damped to reduce dust are fed into a hopper with continuous agitation
- Compressed air is introduced through a rotating barrel or feed bowl to convey the materials in a continuous stream through the delivery hole
- water is added to mix at the nozzle
- Dry mix shotcrete is also known as gunite

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

wet mix shotcrete:-

In this case the shotcrete components and the water are mixed before delivery into a positive displacement pumping unit, which then delivers the mix hydraulically to the nozzle when air is added to project the material onto the rock surface

Difference between dry and wet mix

In case of dry at the point of preparation of mixture water is not required

- The final product of either the dry or wet shotcrete process is very similar
- The dry mix system tends to be more widely used in mining, because of inaccessibility for large transit mix trucks and because it generally uses smaller and more compact equipment
- This can be moved round relatively easily in an U/G mine environment
- Wet mix system is ideal for high production applications in mining or civil engg. where a deep shaft or long tunnel is being driven and access allows the application equipment and delivery trucks to operate on an more or less continuous basis
- Decisions to use the dry or wet mix shotcrete process are usually made on a site-by site basis.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Steel fiber, reinforced microsilica shotcrete

- Of the many development in shotcrete technology in recent years, two of the most significant are the introduction of silica fume, used as a cementitious admixture and steel or polypropylene fibre reinforcement.

- Silica fume or microsilica is a by product of ferro silicon metal industry and is an extremely fine pozzolan.

- Pozzolans are cementitious materials which react with the calcium hydroxide produced during cement hydration

- Silica fume, added in quantities of 8 – 13% by weight of cement, can allow shotcrete to achieve compressive strength which is double or triple the value of plane shotcrete mix.

- The result is an extremely strong, impermeable and durable shotcrete. Other benefits include reduced rebound, improved flexural strength, improved bonding with the rockmass and the ability to place layers of upto 200 mm thick in a single pass because of the shotcrete "stickiness".

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

→ However when using wet mix shotcretes this "stickiness" decreases the workability of the material and superplasticizers are required to restore the workability.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

24/10/2013.

(Shotcrete Support continued) (GSRS Sir)

- Super plasticizers also known as high range water reducers are chemical admixtures used where well-dispersed particle suspension is required. These polymers are used as dispersants to avoid particle aggregation and to improve the flow characteristics of suspensions such as in concrete applications.

- Steel fibre reinforced shotcrete has gained wide acceptance as a replacement for traditional wire mesh reinforced plain shotcrete.

- As we know rock support is only called upon to carry significant loads once the rock surrounding an underground excavation deforms. This means that unevenly distributed non-elastic deformations of significant magnitude may overload and lead to failure of the

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

support system unless that system has sufficient ductility to accomodate these deformations.

- The main role that reinforcement plays in shotcrete is to impart ductility to an otherwise brittle material.

Mechanism Of Shotcrete Working A Support:-

- When the adequately mixed shotcrete is sprayed at the wall to be supported at a high velocity @ 20 – 100 m/s, a layer of cement and sand particles penetrates into pores, cracks, joints in the rock mass and forms a basic foundation for the upper layers to be sprayed.

- As the shotcreting continues, the fines are squeezed into pores, cracks and joints in the rockmass. The shotcrete starts sticking on the initial layer of fines and subsequently, a layer is deposited.

- The shotcrete support stabilises the rockmass by:
  (a) maintaining the existing stability
  (b) acts as a media transferring load from otherwise failure prone weak zone to comparatively stable zone,
  (c) provides additional confinement to the wall against shear / tensile failure.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

(d) a fiber reinforcement provides additional bridging of failure zones apart from incorporating desired ductility to the skin support to cap up the large post-elastic deformation being transferred by the rock surface.

Mesh Reinforced Shotcrete

- While steel fibre reinforced shotcrete has been widely accepted in both civil and mining engineering, mesh reinforced shotcrete is still widely used and is preferred in some applications.

- In very poor quality, loose rock masses where adhesion of the shotcrete to the rock surface is poor, the mesh provides a significant amount of reinforcement, even without shotcrete.

- Therefore when stabilising slopes in very poor quality rock masses or when building bulkheads for underground fill, weldmesh is frequently used to stabilise the surface or to provide reinforcement. In such cases, plain shotcrete is applied later to provide additional support and to protect the mesh against corrosion.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Recommended Shotcrete applications in U/G mining for different rockmass condition:-

Rock mass description	Rock mass behaviour	Support requirement	Shotcrete application
Massive sedimentary rock, low stress condition	Surfaces of some shales, siltstones or slaystones may slake as a result of moisture content change	Sealing surface to prevent slaking	Apply 25 mm thickness of plain shotcrete to permanent surface as soon as possible after excavation, repair shotcrete damage due to blasting
Massive sedimentary rock, high stress condition	Surface slabbing, spalling and possible squeezing in shales and soft rocks	Retention of broken rock and control of squeezing	Apply 75 mm layer of fibre reinforced shotcrete directly on clean rock, rock bolt are also needed for additional support.
Sedimentary rock with a few widely spaced bedding planes and joints, low stress conditions	Potential for wedges or blocks to fall or slide due to gravity loading, bedding plane exposures may deteriate in time	Provision of support in addition to that available from rock bolts or cables, sealing of weak bedding plane exposures	Apply 50 mm of steel fibre reinforced shotcrete on rock's surface on which discontinuity traces are exposed with particular attention to bedding plane traces.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in

Bedded and jointed weak sedimentary rock, high stress condition	Slabbing, spalling and possibly squeezing	Control of rockmass failure and squeezing	Apply 75mm of steel fibre reinforced shotcrete to clean rock surfaces as soon as possible, install rock bolts, with face plates through shotcrete, apply second 75mm shotcrete layer
Highly jointed and bedded sedimentary rock, low stress condition	Bed separation in wide span excavations and ravelling of bedding traces in inclined faces	Control of bed separation and ravelling	Rockbolts to control bed separation, apply 75mm of fibre reinforced shotcrete to bedding plane traces before bolting.
Heavily jointed sedimentary rock with clay coated surfaces, high stress condition	Squeezing and plastic flow of rockmass around opening, clay rich rocks may swell	Control of rockmass failure and dilation	Apply 50mm of steel fibre reinforced shotcrete as soon as possible, install girders with invert struts where required and then more fibre reinforced shotcrete to cover girders. Fore poling or splicing may be required ahead of excavation. Gaps may be left in final shotcrete to allow for movement resulting from squeezing or swelling. Gaps should be closed once opening is stable.

Call/Whatsapp-8804777500 www.MINEPORTAL.in www.fb.com/mineportal.in