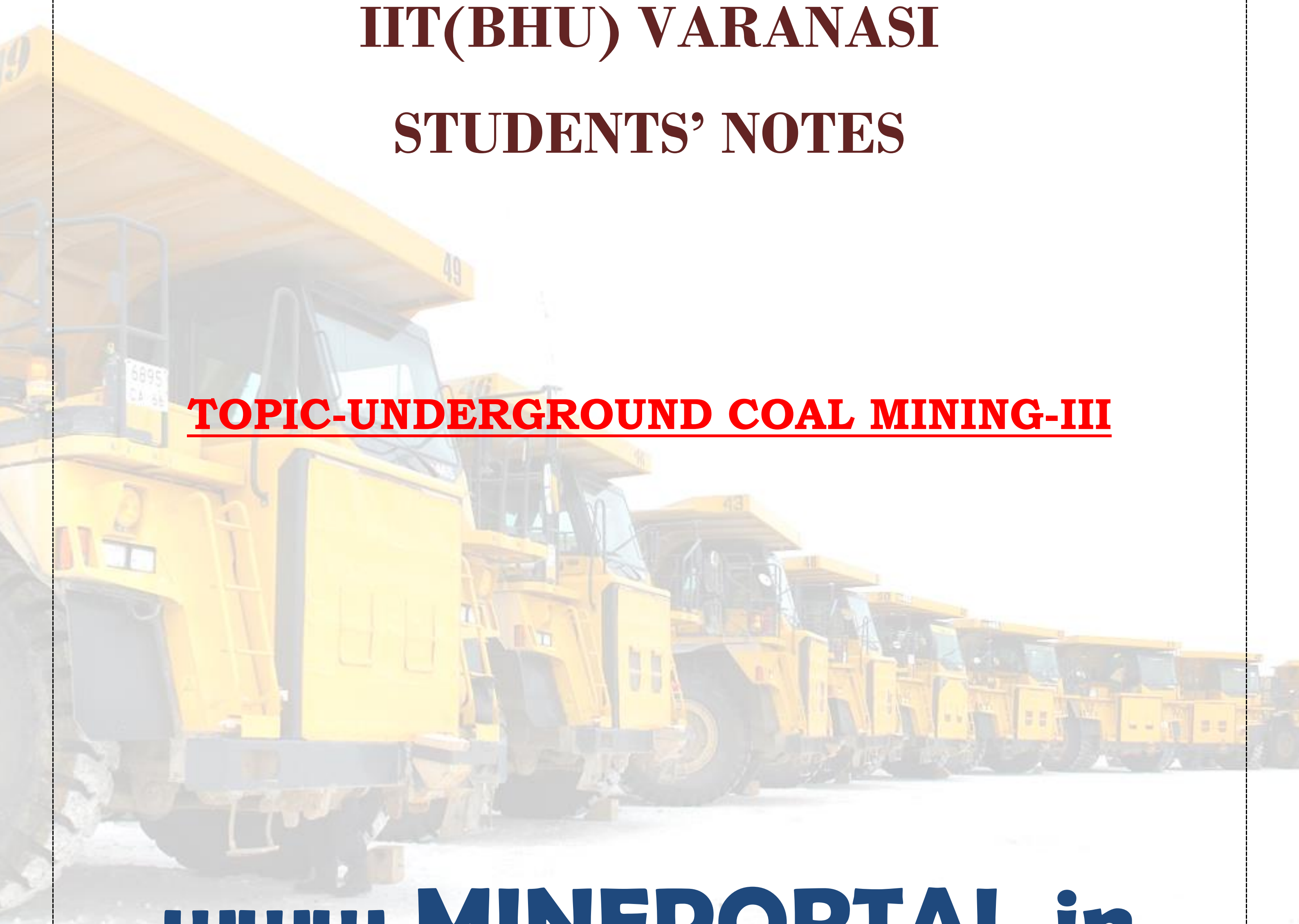
IIT(BHU) VARANASI

STUDENTS' NOTES



TOPIC-UNDERGROUND COAL MINING-III

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SUBJECT → U/G COAL MINING - I

U/G Coal Mining – I (MN 3104) 25/07/2013

Genesis of Coal:-

1.) Insitu theory

2.) Drift theory
used in Indian Subcontinent.

Coal → formed by carbonification of plants. They have a lot of cellulose and under controlled oxidation gets converted into coal.

we had forests in high lying areas and we find coal in low lying area. So over the years plants have been transported from one area to other by mechanical forces

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Ist assignment

Mark all coalfields in a Map of India.

Conditions of Coal formations:-

1.) Sedimentary formation
2.) Always lies as a layer (Seam).
3.) It is underlayn by rocks (roof rock) and overlayn by another series of rocks (floor)
4.) ~~It~~ Coal's lateral ~~exten~~ extent is very large compared to its vertical extent.

Coal ~~major~~ measure rocks → types of rocks found in association with coal seams.

Types of Coal ~~major~~ measure rocks depending on mineralogical composition:-

1. Majority rocks are sandstones. In 65% of coal seams in India 80 to 85% overlying rocks are sandstones

2.) Clay
3.) Shale
4.) ~~Interkellation~~ Intercalation
5.) Sandy shale
6.) Shaly sandstone

In other countries there is some occurrence of limestone in coal ~~major~~ measure rocks (In countries like USA). No limestone occurrence in India yet.

29/07/2013.

Coal measure rocks:-

Rock Type	UCS, (MPa)	Tensile Strength (MPa)	RQD, %
Coal	7-35	0.6-2.8	<45
Shale	20-40	0.8-3.5	>45, <70
Sandstone (sst)	15-50	1.2-5	>45, ≈100
Shaly sandstone	18-40	1.5-3.8	>45, <70
Intercalation	25-50	1.8-4.8	>50, <80
Sandy shale	25-35	2.4-3.2	>50, <80
Clay	15-30	0.4-1.5	<60

⇒ 3 parameters are important to study the characteristics of the rocks bearing coal.

⇒ 2 strength parameters show stability of rocks during excavation

⇒ RQD parameter show the joint distribution in the coal measure rocks

⇒ Generally rock is stronger than the coal but there are exceptions ex:) Godavari coalfields

Inferences from table:-

FACTORS Affecting choice of Mining Methods:-

Strength

⇒ Whenever we have to workout a weaker ~~seam~~ coal then mechanized mining is preferred to drilling and blasting because we have to plan the mine for greater production having same manpower.

Strength

⇒ When working out on a strong coal then drilling ~~and~~ blasting is preferred because if mechanization is opted then the running cost of the machine increases greatly (i.e. the cost of consumable parts is high) so it is not wise to employ mechanization like continuous miner, shearer

$\boxed{\text{PRESENCE OF JOINTS:-}}$

⇒ If coal is highly jointed (i.e. no of cleavage plane per unit exposed area is high) and there is another coal in which the joints are scarce but strength of coal in ~~at~~ both cases are same.

1st case → highly jointed coal.
2nd case → scarcely jointed coal.

presence of cleavage planes favour both drilling and blasting ~~as~~ as well as mechanization

But the extent of favour will be different for both the cases.

In case of mechanization → attack angle is essential. If orientation of joints is such that attack angle is less then mechanization is favoured but if attack angle is large then cutting becomes difficult.

But in drilling and blasting irrespective of the orientation of joints, cleavage planes are favourable.

~~But~~ Hence while planning for cutting of coal then we have to look for the orientation of cleavage planes.

~~depillaring~~ → The process of final extraction of coal. No further scope of mining is there in the area of depillaring.

Goaf → The void (left underground) created after the final extraction of coal. (i.e. after depillaring)

2 types of treatment of goaf:-

1.) fill the goaf → "filling method of goaf treatment" → By planned filling

2.) Caving method of goaf control → Allowing the overlying strata to cave into the goaf area.

Goaf Control →

After replacing the coal block with a void, the rocks (roof, floor, sides) can no longer be in the same state. It becomes unstable. So goaf control is done.

⇒ By filling we are helping nature to maintain stability

⇒ By caving method we are waiting for the nature to become stable.

→ Type of treatment of roof depends on the nature of rocks in the overlying strata.

we want favourable caving i.e. while caving there should be no adverse percussion on the working area where we have men and machinery.

Depth of Coal seam:-

Larger is the depth of coal seam larger is the vertical stress on the coal seam. If vertical stress is more then load bearing capacity of coal is reduced and its workability is increased significantly then for large production mechanized cutting of coal is favourable.
But in case of shallow depth of coal → drilling and blasting is preferred as vertical stress is less.

Gassiness of the coal seam:-

2 criteria for specifying gassiness of coal:-

(I) <0.1%.
(II) 0.1 - 1%.
(III) >1%.

1) % of gas available in the virgin state (Virgin gas content) VGC

2) amount of gas liberated as a result of extraction of coal in m3/ton of coal extracted in the largest working shift.

largest shift in which the production of mine is maximum

0 - 1 m3 (I)

1 - 10 m3 (II)

>10 m3 (III)

Method to known VGC:-

1) Drill core removed from seam, gas evolved from hole is measured.
2) gas from the core is allowed to displace water and this gas evolved is measured.
3) then core is crushed and finally gas in the pores can be measured.

The Virgin gas content is the sum of all these 3 gas volume measurements.

→ drilling and blasting is avoided in case of higher gassy seams so mechanization is to be used.

→ In case of lower degree of gassiness drilling and blasting as well as mechanization can be opted.

30/07/2013

Factors affecting choice of Mining Method

Depth of coal seam:-

↑ depth → ↑ Vertical stress → ↑ workability of coal

As depth of cover ↑ → requirement of minimum size of pillar ↑

larger is size of pillar → lesser is recovery during development

larger size of pillar → larger time of depillaring process for same rate of recovery of coal.

→ Hence time under which we are exposed to danger also increases

As depillaring starts → roof detoriation also starts and rate of degradation of safety of working linearly ~~[illegible]~~ increases, the time for which ~~danger~~ workers are exposed to danger increases with ↑ in size of pillar.

⇒ Working ~~[illegible]~~ in the mine takes place only upto there is no danger involved in it. So whole pillar will not be recovered during depillaring

In deeper coal seams, we should not prefer board and pillar mining. Because in board and pillar mining there is ~~[illegible]~~ greater planning in the pillar ~~&~~ size. and whole of the coal will not be recovered hence other methods like longwall should be prefered.

⇒ whenever we work a coal at greater depth, the cavability of the roof is increased so it is preferred to use roof treatment by caving method so longwall is preferred because cavability of roof is a requirement of longwall

Seam thickness:—

thickness, m	category of the seam
< 1.2 m	unmineable
< 1.5	thin seam
1.5 – 4.5	moderately thick
4.5 – 9	thick
> 9	very thick

If thin seam is to be worked out → longwall is only option

Problem of B and P ⇒
1) It is highly manual, mechanization required (1.5)
2) wherever thickness is less than a limit or more than a limit the recovery of coal decreases if there is (3) no efficient method of mining

~~Between 1.2 and 4.5m thickness → longwall was employed but the capital cost was more and the recovery was not efficient so.~~

moderately thick → B & P and longwall

thick
very thick } Special methods

Gradient of the coal seam:-

In India ⟶ Most coal seams are flattish in nature. But some coal seams have gradient more than 22° and upto 45°.

⟹ 60% of coal seams have gradient less than 10°

⟹ 10 - 15% have gradient between 10 - 15°

→ and remaining have greater gradients

whenever we have a flat coal seam → we have an option to ~~employ~~ deploy ~~a~~ mechanization

But steeper coal seam (> 22°) → we cannot deploy mechanization. we use special methods of mining.

31/07/2013

<u>Proneness to spontaneous heating</u> → (incubation period)

irrespective of quality of coal, all coal seams are prone to spontaneous heating.

For a given rate of extraction (depends on mining method) we have to select a suitable size of the panel so that the deposit that lies in the panel is exhausted ~~before~~ withing the incubation period

For indian coal seams → 5 - 9 months, depending on quality of coal.

if rate of extraction is known, then the size of panel has to be designed depending on the quality of coal.

All coal seams have diffrent proneness to spontaneous heating.

Incubation period → is considered from the start of depillaring process ~~considered from the time of complete development~~

Production and productivity

More than 95% of U/G mines are running in losses.

→ To plan a mine of higher productivity we have to opt mechanization

Mandatory fulfilled OMS → 1.8

To run it on break-even basis → required OMS is 2.2
(no profit or no loss)

⇒ To run a mine in ~~OMS~~ profit → OMS > 2.2

⇒ OMS is for a panel not for the entire mine.

⇒ So to get better productivity and production we should opt for longwall mining

Cost of production and Capital Cost:-

Capital cost → is the cost component of total mine which is required to buy, set up machines and get various infrastructure needed.

operating cost → cost required to be incurred ~~so~~ for running the mine on day to day basis.

If mechanization is opted → Capital Cost is very high

but for manual ~~mining~~ → capital cost is less

only shaft cost, ventilation fan cost, and other basic requirements are needed.

Before planning for mechanization we should first see if we have a proper know-how for using the machines.

ex.) Continuous miners used in India have a utilization of 65% but those used in S.A. and China have a utilization of 95%.

So we have to be prepared for mechanization. Proper training should be given to the workers, operators. We should be able to convert the resource of mechanization into results on a long term basis.

$\boxed{\text{Surface constants:-}}$

When we are extracting the mineral underground, the overlying strata is liable to get damaged. There will be surface subsidence and cracks on the surface.

But before mining the surface damage is to be considered and then depending on the surface constants and the subsidence we get a permission wheather mining is allowed or not.

~~Before starting~~.

During mining if surface considerations is not taken into account then local people will not allow mining if they come to know that their interest is not taken into account. So we have to look for surface stability for u/g mining.

Problem of rehabilitation → ex:) In JHARIA

For sustainable mining practice → caving practice should be allowed in only those areas where there are no surface constants presently and we have to see that there will be no surface constants even in the forseable future.

ex:) Tata steel in Dhanbad is running u/g mines at 350m depth in Conjuction with stowing so they are avoiding any chances of danger to the surface features/constants.

$\boxed{\text{Nature of roof and floor:-}}$

Nature of rock that is forming either roof or floor affects ~~nature~~ method of mining.

If roof and floor and weak then we have ~~to~~ look for methods which have intensive ~~extensive~~ supporting allowed.

If we have competent roof then intensive support is not required and we can opt for other methods.

30 cm thick coal falling freely from a height of 1m is sufficient to cause fatal accidents.

<u>longwall mining</u> → roof and ~~fl~~ floor supports are placed very systematically so it can be used in non-competent roof and floor conditions.

<u>B and P mining</u> → ~~so~~ fit for competent roof. Because the ~~so~~ layout and design for roof support in such method is ~~so~~ not competent enough to avoid fatal accidents due to roof failure.

$\boxed{\text{Presence of dirt/ bands:—}}$

<u>bands / dirt</u> → Geological structure in coal seams (particularly high quality seams). These are basically igneous intrusions in the structure of coal.

When igneous intrusions are there in coal then it starts burning. It turneds into jhama coal (very strong). So dirts and bands are seen as layers of jhama coal.

Bands are visible in subhorizontal seams. They can be in floor or roof.
dirts are subvertical (inclined).

whenever there are dirt/ bands frequently then mining is preferred which does not depend on ~~the~~ mechanization because machine damage will be more ~~and~~ as the Jhama coal is very hard in nature

we should opt for conventional drilling and blasting.

no igneous intrusions → mechanization opted.

Presence of major geological disturbances:-

major geological disturbances are folds and faults.

Minor disturbances like ~~of~~ joints, cracks do not have effect on the method of mining.

If a coal seam encounters a fault then suddenly its horizon changes (depth changes) so we should opt for mining method which allows flexibility in the coal seam horizon.

So if a coal seams has multiple faults so we cannot establish longwall mining in such mines.

Moonidih → 1) Father of longwall mining in India

2) Coal gassification project first started here. It is one of the highest gassy seams in India.

Reason for failure / underperformance of the moonidih longwall project is presence of multiple faults in the coal seam.

Proneness to Coal bump:-

~~Coal bumps~~ whenever we have a seam ~~prom~~ highly prone to coal bumps we should avoid methods which require multiple entries to the coal seam.

So B and P should be avoided in such mines.

It is a regional ground control hazard.

So we should opt for methods which require lesser entries or we should opt for stowing ~~is~~ for goaf control during mining of such seams.

Similar to coal bumps, rock burst is there in metal mines

1.) Phenomenon is same

2.) Degree of damage is same

But ocurrance is different.

01/08/2013

Methods of U/G Coal Mining

- Bord and Pillar/ Room and pillar
- longwall/ shortwall/ olique longwall
- Blasting gallery
- Top Coal Caving
- Special Methods
  - Tipong
  - Bhaska

  } NECL (North eastern Coalfeilds Limited)
- Hydraulic Mining
  - Polish Coal mining technology

Hydraulic mining → Putki mine
~~shortw~~ olique longwall → Balrampur mine

Bord and Pillar Method:-

~~Parall~~ Any mine where we are accessing coal seam by shaft or incline has atleast 3 main dip galleries.

Main dip galleries → not because they have been constructed along the true dip but because their significance is quite high from their operational point of vein. Irrespective of the location of the working galleries at some point of time, they are to be maintained throughout the life of the mine. They start from the beginning of the coal seam (shaft level/zero level) and goes upto the end of the deposit (i.e. upto the pit limit.)

~~Comparision~~ -

Companion dip gallery → constructed along the dip but the significance is lesser than main dip gallery. They do not commence from the zero-level.

Cross-cuts → Connects a dip gallery to a level gallery with the shortest distance is called cross cuts

The pillars which are planned to be formed for extraction later on are called working pillars but pillars which are formed to be left forever are called barrier pillars. If barrier pillars are along the boundary of the mines are called Inter-mine barrier pillars.

⟹ Principle is to divide the total mining entity into ~~sector~~ subdivision for better management of the mine

A mining block $\xrightarrow{\text{Consists}}$ ~~Section~~ Sectors $\xrightarrow{\text{consists}}$ regions $\xrightarrow{\text{consists}}$ panels $\xrightarrow{\text{consists}}$ districts $\xrightarrow{\text{consists}}$ pillars and galleries.

Larger mines are operated because of better operational management.

In any mining area → accessment is started from the centeral point and then development is started further by making main dip, companion dip galleries are made and so on development is made.

Panel → An area always surrounded by a row of Barrier pillars with minimum no of entries to facilitate access into the panel for men, material, machines and ventilation.

It has independent supply of intake air and independent root for return air. Due to their vitality the panels do not have a dependent ventilation network although they are a part of the mine. This independence is possible by having ventilation connection directly with the main ventilation root of the mine. Similarly the return roots are also directly connected to the main return way of the mine. The necessity of the independence is just to ensure that incase there is some problem (fire, water inundation, etc) in the mine, the ventilation in the panel is not compromised and working in the panel area is easily continued. If the ventilation is made dependent then the purpose of having barrier pillars is not fulfilled.

05/08/2013

Applicability of Bord and Pillar Mining:-

1.) Seam thickness → The seam thickness can be 1.5 to 4.5 m (give reason)

2.) Depth of the seam → should not be adopted at higher depth.
depth of working < 360 m
although we have B and P working upto 400m but these are exceptional. They are encountering a lot of problems
ex.) Parascole Colliery, ECL – 370-385m
Churcha East mine – 425m

3.) ~~Nat~~Nature of roof and floor → They should be highly competent because the B and P method does not have a systematic scheme of support.

4.) Gassiness of the seam → Degree I preferred although we have B and P workings deployed for Degree III seams.

5.) Proneness to spontaneous heating → should be low as our rate of extraction is lower so we require a larger incubation period.

6.) Presence of major geological discontinuities → They should be low ~~[illegible]~~ but if we encounter them then the degree of problem faced by B and P method is ~~more~~ less compared to other methods taking benefits of the flexibility of B and P methods.

7.) Scope of selective mining → ~~[illegible]~~ If we intend then we can selectively mine out only those areas of the seam which have a good grade and do not affect adversely the ROM coal of the mine as a result of which the cost of coal reduces. By leaving th

undesirable coal we can get rid of a low grade output.
It is always preferred to plan the mining of the seam using the B and P method.

8.) Production and productivity → If requirement of production is very high then this method is dis~~c~~couraged. The productivity of this method is varying from low to medium (> 600 tPd)

9.) Skill requirement → Those ~~de~~ methods which have ~~dep~~ demand for high mechanization requires high technical expertise for running it effectively. But B and P method required less expertise for a smooth running of mine. The skill requirement ~~f~~ in B and P mining is low and in India ~~it is so~~ ~~unk~~ low skilled ~~laboured~~ labours are easily available so in India it is preferred.

10.) Degree and proneness to coal bumps → ~~Depending on~~ If degree and proneness to coal bumps is high then B and pillar method should be discouraged because problem in mining increases. If proneness to coal bumps increases then roof instability ~~also~~ increases.

~~Two~~ Proneness of coal bumps is due to two reasons:

1.) Contribution of mining method
2.) Nature of formation of deposit.

Adverse condition in mining is due to:-

1.) Due to improper planning, and

2.) improper forecasting of the problem

If we think of applying B and P method in Degree 3 coal seams then

(i) ventilation requirement is very high ~~do~~ because we have to dilute the gasses evolved during mining to permissible levels.

But for this speed of air required is also very high but we have restriction ~~of~~ on the max. speed of the air we can have at different places in the mine

So this ~~~~ results in poor ventilation plan in B and P mining.

(ii) In B and P, drilling and blasting of solid is required but even the use of P_5 (most safest) explosive, we are not allowed to go for solid blasting in degree 3 and 2 seams

So ~~the~~ we are allowed to use free face blasting using auger drill or auger miner. Earlier coal cutting machine was used for ~~f~~ preparing free face.

If solid blasting is done then there is sudden imbalance in gas levels ~~and~~ due to high CH_4 emission from coal and fumes of blasting. ~~S~~ Poor ventilation is improper in such situations and the face becomes ~~~~ vulnerable to explosion. So what we do is we first create free faces in the working face by drilling long ~~~~ holes into the face. Then the emitted CH_4 is removed by ventilation and finally blasting is performed.

Advantage of B and P mining:-

1.) Quick output / low gastation period →

In U/G mines it is very important to minimize the payback period, infact it is a factor which determines the feasibility of the project. (Payback period is the time in which we can return to the bank the loan amount along with the interest.)

So for this the production should start at the earliest time. The advantage that the B and P offers if we ignore the level of production is that it gives us a production as soon as we touch the coal seams and start developing the main dip galleries.

2.) Low capital investment → Because mechanization requirement is low and labour cost is less. (Doesn't not require high skilled labours)

3.) Uniform output throughout the mine life →

Some methods although have capacity of very high production but their consistency of production is low.

Consistency is required to meet the "promise to the consumers".

4.) Minimum unproductive work →

unproductive work → any work which is not converting into production.

Even the depelopment of main dip galleries gives us production

5.) More populer among workers →

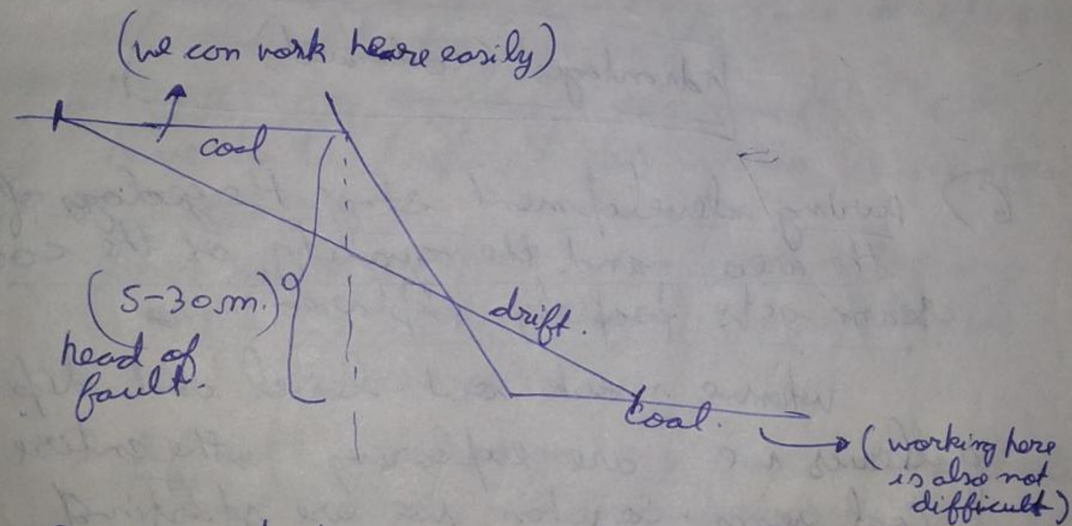
Because the workers are familiar since it is ~~one~~ of the oldest method of mining.

06/08/2013.

B and P is a Flexible Mining Method, why??

B and P is a flexible mining method → if we encounter lot of major geological disturbances then B and P method comes to our help.

Suppose we have encounter a fault the mining horizon changes suddenly. If we have to move from one side of fault to other side ~~we have~~ ~~to~~ we have to negotiate this fault zone.



For negotiation we create a drift at certain inclination to approach the 2nd part of coal seam from the 1st part. Shallow is the gradient of the drift, longer is the drift length ~~of~~ for a given head of the fault. drift length can be from 20m to few hundred metres. We do not have any drilling and blasting design available for drift advance and we do not have a proper ventilation during drivage of drift because it is a blind drivage process. Working for longer hours in such drifts is difficult. We do not have machinery and mechanization for drinage of drift. For a given rate of advance of drift the time of preparing a drift can be upto few months, for longer drifts.

In general, our mining plan does not advocate to negotiate the fault, instead we should limit the size of the panel (working area) and use the fault zone as a barrier pillar (which separates two working areas)

But this method is available in B and P. In other methods sudden deduction in size of panel is not possible. That is the only reason it is called flexible mining method.

Advantages (Continued)

6) During development stage the geology of the area and the quality of the coal seam gets properly explored:-

when we work out level and dip galleries we are exploring the entire coal seam. So when we are starting depillaring then we hardly have any geological surprises. The gallery are widely spaced and give a very good level of exploration and also during final extraction of the coal we are prepared for the different geological problems.

Disadvantages of B and P

1) Sluggish Ventilation → If we go for drilling and blasting we require allowance time to dilute the fumes and dust of blasting and for ventilation of the area. If allowance time is more the cycle time of production is increased so the number of productive cycles ~~increases~~ decreases and so for some level of manpower our production rate will be slow.
So due to multiple interconnected galleries we have a sluggish ventilation.

Depillaring Ventilation:-

The ventilation near the fan is 90%. then near the working face it is reduced to (10-12%).

wherever we employ this method we have to be prepared for poor ventilation system.

Other methods have significant advantage as far as efficiency of ventilation is concerned,

2) Difficult supervision → we ~~cannot~~ should have a very close supervision in U/G mines (the manpower and machinery). Production of the mine is directly related to supervision.

for higher production
- manpower more → more supervision
- more machinery → more supervision

if we have a higher manpower for low production then we ~~have~~ do not require a higher level of supervision.

In B and P → it is difficult to have such a high level of supervision because the man force is very large.

3.) Strata Control problem :→ All problem which are likely to be developed due to unwanted behaviour of the strata is called strata control problem.

We have to have a good strata behaviour in our working areas but this is not possible ~~to~~ because we are working against nature.

Strata control problem is ~~depends~~ due to ~~~~:-

1.) unforvourable stress structure
2.) rock mass ~~behaviour~~ behaviour
3.) Strength of ~~roof~~ rockmass
4.) layout of mine

In B and P → the support pattern is not continuous so we require a competent roof and floor for safe mining operation.

Development time → strata control problem is less so ~~not~~ no DGMS permission is required.

Depillaring time → strata control problem is very high so even for ~~a~~ mining one ton of coal we have to follow the stringent rules of DGMS.

4.) Low extraction %age and high loss of coal →

In case of B and P → overall extraction is not more than 75% if we are working a seam upto ~~4~~ 4.5 m (if thickness is more the losses of extraction is even more.)

5.) More chance of fire → Between ~~dev~~ development and depillaring (end of), a large surface area of coal is exposed to oxidation and ~~for~~ a longer time as extraction rate is low so higher chances of fire is there.

6.) Low output, low productivity and high cost of production → Since level of mechanization is less so output is less and we have a lower profit which directly affects the cost of production.
For a high manforce, output is low by B and P method thus increasing cost of production.

7.) Labour oriented, needs large workforce
↳ Method is highly manual.

8.) Chances of spontaneous heating, premature collapse, over-riding of pillars and air blast.

07/08/2013

Development of B and P working:-

Development can be by:-

1.) ~~open~~ open system (had disadvantages, not scientific so abolished)

2.) Panel system

In case of panel system → ~~total~~ total mining area divided into panel.

open system → no panels as such, whole mining area does not have any subdivision by barrier pilliars. Barrier pillars are only around the boundary of the whole mine.

Problem of open System:-

1.) no ~~inter~~ ~~coma~~ intra mine barries so if any problem in one corner of mine, whole mine was affected. Suppose we have fire in one location, the presence of CO affects whole mine working. Similarly if we have innundation in one part of mine, we have no provision to segregate that part of mine so whole mine is affected

But in panel system in emergency case, the ~~maximum~~ (minimum) number of openings of the panels are sealed out in the minimum possible time.

2.7 Suppose there ~~t~~ is premature collapse or overriding of pillars we have to compromise a lot of ore. We thought that in ~~open~~ panel system barrier pillars are more so ore loss will be more but on practice it was found that due to barrier pillars the ~~occurance~~ of premature collapse or overriding of pillars is reduced to a large extent because barrier pillars serve as local support so for some condition, ~~act~~ overriding is less.

Panel system is advantageous in ventilation. In case of open system we have to force the air from one part to the last part throughout the year and throughout the mine life. Practically it is not possible (theoritically may be) because we do not have active working in the entire mine. We only work in some concentrated area.

Panel system gives us advantage that we can isolate one area where we do not want any ventilation and by putting barriers in the panels, leakage into such (isolation stoppings) ~~part of the sys~~ panel (where working is not performed) is stopped.

During depillaring, before starting we have to isolate the panel by barriers (isolation stoppings) and define its size ~~that~~ so that ~~it~~ the entire coal can be recovered between the incubation period. This is irrespective weather development is by open or panel system.

Advantage of panel :-

you can perform depillaring the coal irrespective of its position in the mine.
In case of open system, depillaring starts from the last corner.
So for final exaction of coal in an area ~~the~~ we have to wait until the entire mine is developed but in panel system irrespective of the position of panel we can start depillaring as soon as development of panel is over. because in case of any problem (fire, explosion, inundation) we can restrict the effect with the panel only. So operation in a panel is very reliable.

Cycle of operation in B and P workings:-

1) winning the coal → To extract the coal and transport it outside the mine.
winning can be by:-
1) drilling and blasting.
2) Mechanised cutting (more productivity)

Before we had safe explosives, we used to win the coal by pick mining but production was less.

⇒ Road headers, Dint headers for cutting of coal in blind headings

⇒ Road headers ~~can be~~ used for excavation of tunnels.

In case of drilling and blasting:—

1.) solid blasting
- instantaneous → drill holes, charge them and blast.
- delayed → the holes are blasted in sequence. 0 delay blasted, it creates free face then 1st delay blasted and so on.

2.) free face blasting {To have better yield of blasting we need to have better utilization of free face wherever available}.

But in case of blind heading there is only one free face ~~so the free face~~ is on the ~~other~~ facing side then the explosive wave or burning wave generated always get retracted from this free face.

Explosive wave is transmitted (in solid only) ~~in~~ either:—

1.) ~~from~~ forward (~~no~~ damage} is less or mild because energy decreases continuously)

2.) backward (~~creates~~ creates damage in ~~rock~~ solid mass

↓

Backward movement is ~~free~~ when it encounters a free face.

So to ~~have~~ a better output of the explosive energy we have the increase the available free face.

⇒ The yield per blast is not greater than 85% in U/G workings (Pull factor)

85% of hole length is the pull of blast.

We have restriction on the explosive used per length of hole (charge density - charge per length of hole)

so to improve the performance of blasting in conditions of 1.) inflammable gas in rock

2.) restriction of charge density.

we switched over to free face blasting.
~~Bet~~ for safer and better blasting.

Benefits of free face blasting:-
1.) Better blast output
2.) Reduced Hazard of blasting due to gas

$\boxed{\text{For creating free faces:-}}$

for drilling in coal.
← ~~coal~~ drill → 48 mm dia. hole

⇒ Formation of professional free face in solid blasting by using delays ?%

auger drill → 42 ~~cm~~ m dia hole
Specially used to create free face.
Free face is always created in the centre.
The hole length is 0.1 m more than the length of the blast hole.

Similarly in coal cutting machine, free face is created on top, bottom, sides or center the depth of free face should ~~b~~ be 0.1 m more than the length of blast holes.

2.) $\boxed{\text{loading of the coal} \rightarrow}$ can be done by:-

1.) Basket loading (totally manual)
2.) Mechanised loading.

SDL

1.) comfortably work on loose ground due to crawler mounted
2.) FEL → Front end loading.
3.) Beneficial in low ht. galleries

LHD

1.) minimum ht of 2.2 m of gallery
2.) tyre mounted is a disadvantage
3.) Adv → speed is more than ~~[illegible]~~ SDL
4.) Strong ground conditions

3.) $\boxed{\text{Transport} \rightarrow}$ <u>we have:-</u>

1.) trackless transportation system (combination of chain and belt conveyor or combination of belt and rope haulage)

2.) Rope haulage system

we have rope haulage for transportation inside the panel.

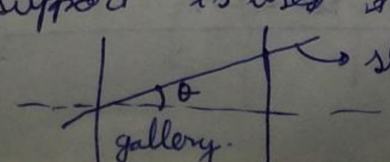
we have other types for transportation from panel to outside (usually large capacity DRH)

⇒ endless haulage for (central) level (gallery) transportation and Dip haulage for dip gallery transportation in a panel.

On ~~using a~~ combination of Belt (continuous transport) and rope haulage (intermitting transport), we have a bunker and the conveyor is switched off as soon as the bunker is empty and then the haulage is allowed to again fill the bunker and once bunker is full the Belt conveyor is started again.

4.) $\boxed{\text{Support} \rightarrow}$ Simplest types are prop supports. Earlier timber was used then due to scarcity of timber we switched to friction props and then we shifted to hydraulic props. But when mechanization was implementing in B and P we discouraged prop support and shifted to ~~to~~ roof bolts. Latest used bolts are raisin bolts.

Wherever we ~~the~~ have a slip plane ~~then~~ intersecting the gallery at an angle < 60° the we face lot of problems so choke or cog support is used there.



12/08/2013

U/G Coal Mining

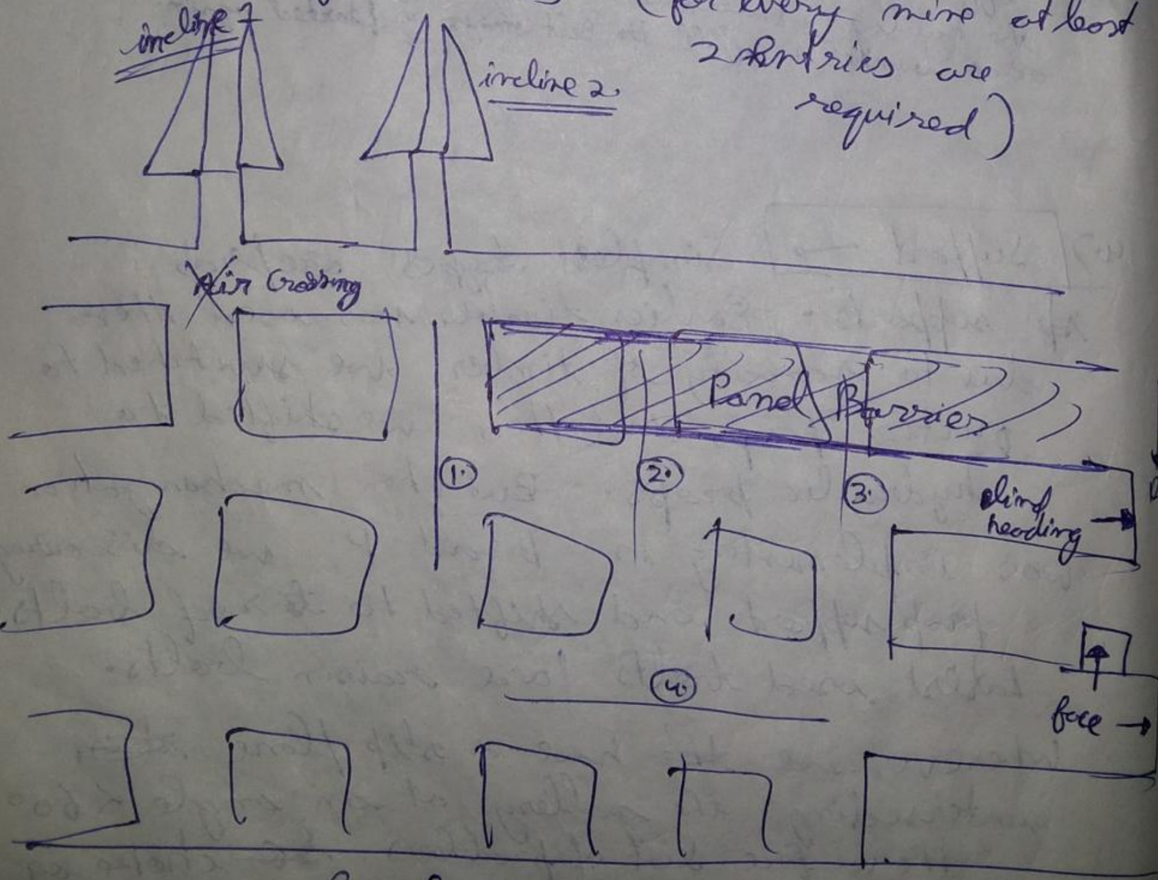
~~Road~~ Road header → A cutter loader. It has 2 units (i) cutting (ii) loading.
An apron has 2 mechanized arms which are rotated to facilitate inward movement of coal. A chain conveyor transmits coal to the back of machine. It is crawler mounted machine.

SDL → specifically a loading machine for dedicated loading purpose

Dint header → also for mechanized cutting. On a sprocket (chain) cutting bits are mounted which cut the rock.

Board and pillar layout :-

⇒ 1 pair of inclines (for every mine atleast 2 entries are required)



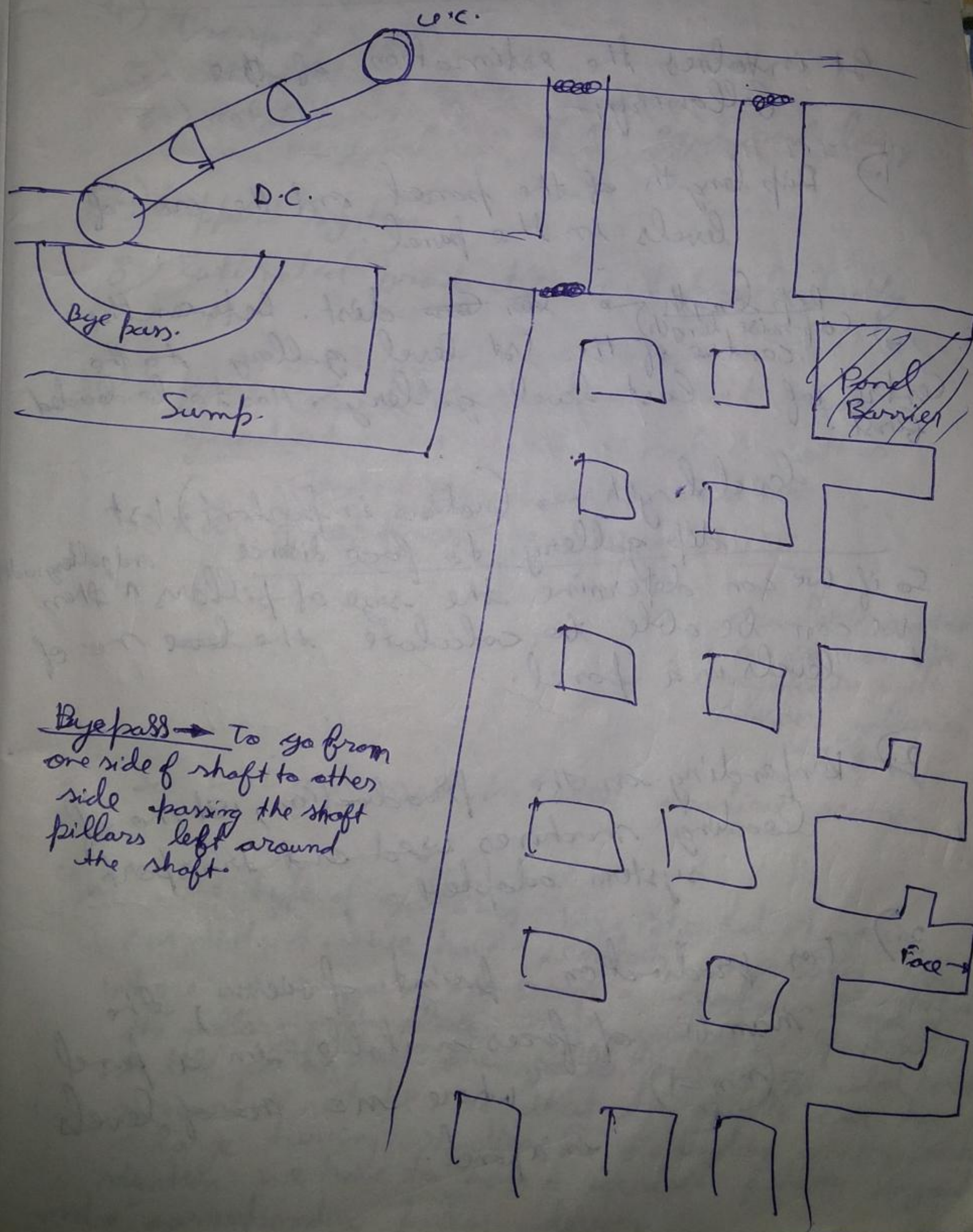
Panel Barrier.

① → main dip galleries

② and ③ → companion. dip galleries

④ → level galleries.

In B and P with 2 shafts:-



Byepass → To go from one side of shaft to other side passing the shaft pillars left around the shaft.

It is a mandatory requirement to join UC and DC shafts by air locks (doors) because if in some condition if we require reversal of ventilation then it can be done easily.

Designing the layout of B and P Development Panel :-

It involves the estimation of the following :-

1.) Dip length of the panel and the no. of levels in the panel.

Dip length (dip-raise length) → dist. between the centre of the 1st level gallery to the centre of last level gallery. Has to be decided first.

level length → (not so important) 1st dip gallery to face distance

So if we can determine the size of pillars and gallery then we can be able to calculate the no. of levels in a panel.

2.) Depending on the production, no. of loading machines used and transport system adopted

3.) For production point of view, the number of faces available in a panel $=(2n-1)$ where $n=$ no. of levels in a panel.

4.) No of loading machines to be deployed
5.) Transport System.

Design requisites:- (Input info)

1.) Dip of the seam and its direction
2.) Entry galleries to the district
3.) Working faces
4.) Transport System
5.) Ventilation System
6.) Relative position of the equipment being used.
7.) Support system
8.) Estimated production and productivity.

1.) Suppose we have a seam with max. gradient is 1 in 6 and in other condition 1 in 10 is there.
we have 2 options → LHD and SDL

SDL ↓ crawler mounted so it can negotiate with steeper galleries easily

So LHD not used in steep gradient seams.

2.) The type of galleries we should have in our district. We have certain mechanization where we require to have a certain orientation of the ~~pillars and galleries~~ entry galleries. If we use shuttle cars (in Band) then the ~~galle~~ pillars should have a rombus shape so that the shuttle car has a large turning radius. So depending on the mechanization we have to have a certain gallery layout and a corresponding pillar design.

3.) Position of working faces and their extent of staggeration?? If working faces are ~~3) Position~~ staggered then we will require more number of loading machines but if we want to reduce the no of loading machines the we have to opt for those loading machine which are more ~~manouverable~~ manouverable (mobility is higher), then only we will be able to efficiently mine out the panel with reduced no of loading machines.

4.) Trackless and Track transportation System. The system should be such that it should be compatible with the loading system. Else Our production will not be as expected.

5.) Depending on gassiness of mine and no of people and machine the ventilation system should be designed so that people can work to an optimum level to give a good production.

7.) The support system should be such that we can have a safe operation of the mine.

8.) Ultimate output → production
So we have to know the estimated production requirement of the mine before designing the layout.

B and P Development Layouts:- using SDL

Q.) A Coal seam 3 m thick is lying at a depth of 200 m. The roof and floor of the seam is Competent and the seam is not ~~very~~ highly gassy. Propose a method for developing this coal seam using B and P method. Also estimate the production and productivity of the B and P panel. The seam is moderately dipping.

⇒ The first parameter to be considered while deciding gallery ht. is seam thickness. $\boxed{\text{gallery ht. max} = \text{seam thickness}}$ but we have to keep CMR in mind.

SDL :-

- A SDL is a crawler mounted loading machine designed as a highly maneuverable loader capable of turning on its own length but not designed for its continuous travel (working faces should be close)

- Consists of a bucket of of 1 m³ capacity fitted ~~supported~~ with ~~by~~ 2 hydraulic ~~g~~ cylinders on a boom which can be
  - Boom can be extended upto 1.8 m and swivel ±20° about horizontal axis.

- discharges either on chain conveyor or standard coal tub.

- Can be used when one way travel distance should not be more than 20 m.
  lead distance

Applicability Condition:-

- Limiting gradient: 1 in 5, ideal 1 in 6 or flatter
- Minimum room height

  ~~1.5~~ 1.5 m for loading onto conveyor and 2.4 m for loading onto standard rool tubs

- Ground pressure - 0.10 MPa (can be used in poor floor conditions)
- good ~~to~~ roof
- Short haul distance (upto one pillar)

Method of development By SDL:-

30m

- Initially, 3-4 main dip headings are driven from the shaft level (zero level) (level of D.C. shaft) along the floor of the seam.

shaft level → The level which is directly connected to the shaft (downcast shaft). It is the zero level. This is the level at which we touch the coal seam.

NOTE ⇒ (Main dip galleries are those which commence from the zero level. others are companion dip galleries.)

- These dip headings are interconnected by level headings at every 30 m interval forming pillars of 30m X 30m centre to centre or any other size depending on the depth of the working and the gallery width.

→ We understand that the access has already been established ~~on which~~ with a suitable method.

→ Then we make 3-4 main dip galleries from the shaft level of the D.C. shaft ~~and~~ along the floor of the seam.

→ The galleries and pillars are to be developed. ~~we tr~~ The size of the galleries is a very important design parameter. If we have a smaller size of gallery then
(a) production during development is less
(b) ventilation is not effective

So we want to have a max. ht of the gallery and max width of the gallery.

The max ht. of the gallery is restricted by CMR to 3m and the minimum height should be such that it is comfortable for men and machinery to work in the galleries.

If we consider width of the gallery then the max. permissible width = 4.8m
min " width = 2.4m.

~~But if we have a~~ in some cases as in Continuous miner width of gallery can be upto 6m (permission taken from DGMS)

But if ~~max~~ width is large then the rock of roof should be competent as stability of roof reduces ~~as~~ width of gallery increases. In case we use a large ~~high~~ width of gallery then our support system should be efficient enough to support the roof until the final extraction from the area. But high support becomes Costly.

Q.) when we require to have 3 - 4 main dip galleries ??

Ans.)

1 is not sufficient because → we should have one dedicated intake and one dedicated return airway or else the air will mix and quality of ventilation will degrade.
But regulation ~~provision~~ says that we must have at least 3 because in case of some problem, if one of the intakes is non functional then the other can efficiently solve our purpose of ventilation and transportation of men and material ~~Some~~ and secondly by the principle of splitting, the effective airway resi

is reduced and so airway pressure loss is reduced and hence better ventilation network is established. So we should have 2 intakes and 1 return gallery.

But it is favourable to have 2 intakes and 2 return galleries in our ~~pa~~ panel.

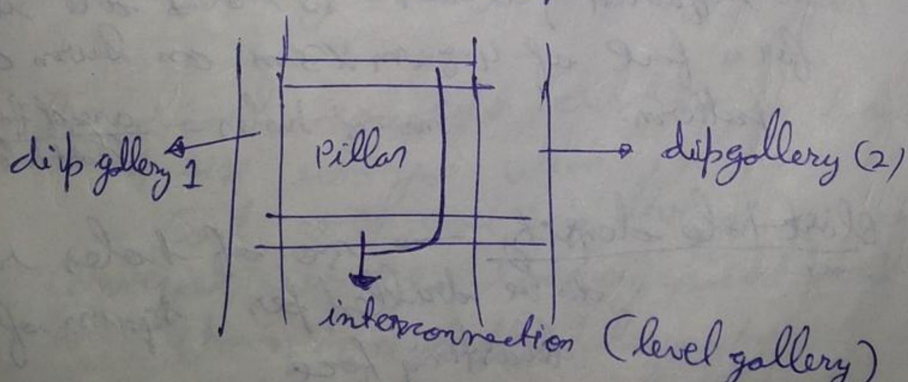
13/08/2013

B and P layout :- (Continued)

⇒ The dip-raise galleries are required to be developed and ~~f~~ then they are interconnected because there is a limitation of such blind galleries if length of such galleries is more than 20 m then it is ~~f~~ a problem to provide a healthy ventilation at the dead end (closed end) because the intake and ~~pillar~~ return airs are mixed.

⇒ Size of pillar depends on (The min. distance at which subparallel galleries are interconnected depends on :-)

1. width of the ~~g~~ galleries
2. Cover depth



- When the dip headings get advanced by about 200 - 250 m, 5-6 no. of level headings are driven along the strike direction.

Preparatory arrangement before depillaring:-

- The following arrangements are necessary before depillaring operation is started:-
  - Determination of panel size taking into account incubation period and rate of extraction, if development has not been done by panel system.
  - Obtaining permission from DGMS for depillaring in all cases, from railway authority if depillaring is to be conducted below railway line and from the district authority if depillaring is conducted below district board roads.
- For depillaring with caving method, dewatering of old water logged working, if any, either by putting in seam pumps or by burn side safety boring machine.

<u>Burn side safety boring machine</u> → very useful when we are moving towards a water ~~loged~~ lodged working and we ~~do~~ do not knows its location. So we move forward carefully and ~~as~~ as soon as we see water seepage from the face holes we can leave the machine. The machine is such ~~the~~ that the drill does not allow water to come out of the hole and hence does not allow puncture of the face thus preventing the danger of inundation. The machine helps in providing safety against mine inundation from old workings by reducing the probability of hazard due to collapse of parting due to high water pressure

- ~~Plans~~ to be updated ~~showing~~ showing all features so that in case of any mine hazard a proper resque plan can be made out.

- Adjocent mines, likely to be affected are informed. A hazard may take place due to any work in one mine, affecting other mine due to lack of coordination of the managements. So a new rule of amalgamation of such mines and bringing them under a single administration is set up.

- Adequate number of ~~all~~ supervisory staffs and workers having experience in depillaring have to appointed.

  In absence of sophisticated equipments → experience counts a lot

- There shall be adequate stocks of supports, fire fighting equipment, stone dust, lime, bricks etc.
  - stone dust: 1.) better illumination 2.) rule out possibility of coal dust explosion
  - lime, bricks: 1.) sealing of galleries/ stoppings

- Completion of preparatory stoppings all around the panel, if development has not been done in panel system

- Adequate provision of pumping arrangement (because ~~in the~~ ~~rainy~~ during depillaring the roof goes considerable degradation and during rainy season problem of water inundation increases and productivity in that period is compromised largely in the absence of proper pumping.)

• Establishment of ventilation network → The already sluggish ventilation degrades further during depillaring, so ventilation plan has to be revised to see how we can properly ventilate the depillaring areas.

• Advance supporting up to 2 pillars, ahead of the pillar under extraction, or 60m whichever is more in all direction as per the approved systematic support rule. (prepared by mine manager and approved by DGMS)

→ Advance supporting upto 2 pillars from the dipmost pillar of the panel depending on the line of extraction is done in accordance to systematic support rule

Bord P depillaring panel:-

Q.) A Coal seam 4m thick, dipping at 1 in 10, occurring at a depth of 130m have been developed on Bord and pillar system along the floor of the seam with 4.5m wide galleries and 30m × 30m C/C pillar. Describe the method of depillaring of the seam and propose a suitable design for such a depillaring panel. Assume your own conditions. Estimate the production and productivity from the panel.

9/09/2013

1.)

Seam thickness = 4m

gradient = 1 in 10

Cover depth = 130m

width of the gallery = 4.5m

Pillar size = 30m × 30m (c-c)

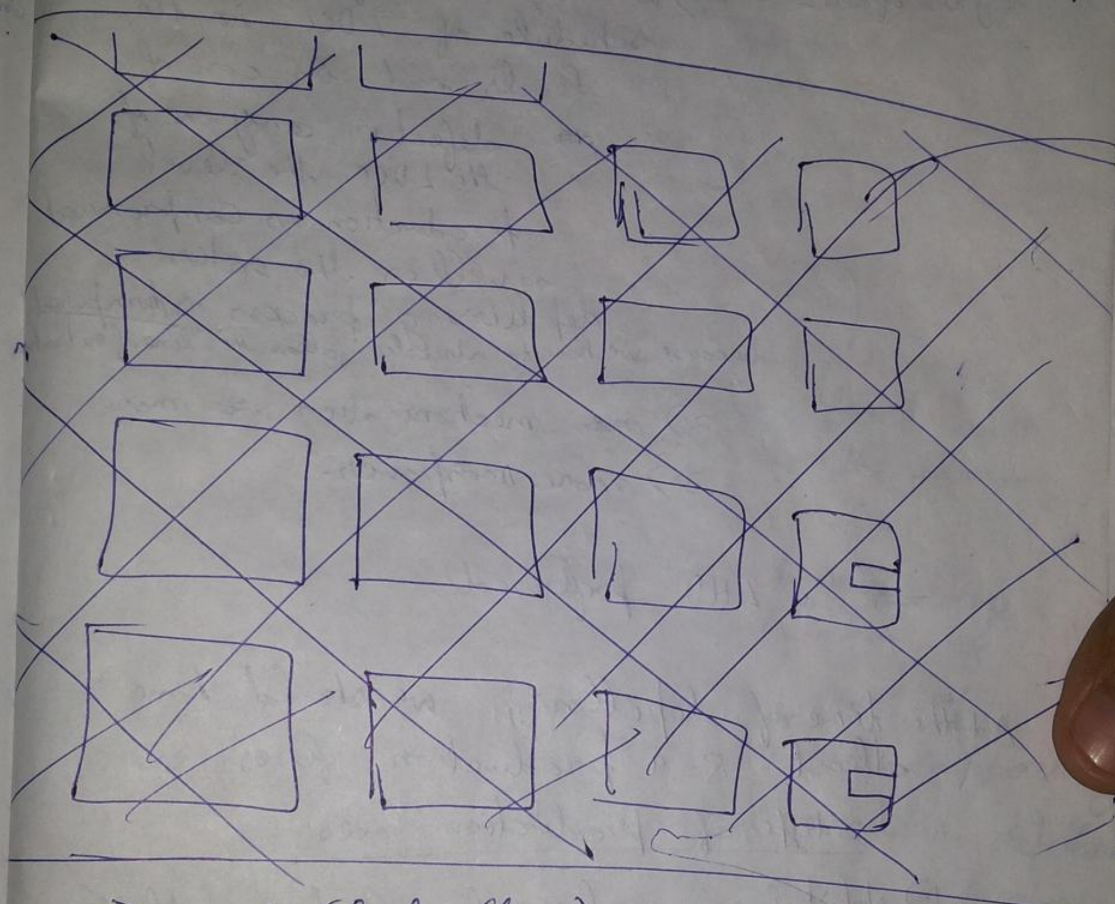
$\left\{\begin{array}{l}\text{max. extraction ht} = 3\,m \\ \text{min extraction ht} = 2.2\,m\end{array}\right.$ (Based on LHD)

Assumptions:-

1.) ht of developed galleries = 3m

Gallery size = 4.5m x 3m

Before depillaring we should have a layout suggesting the no. of pillars in our developed panel.



2.) n=6 (level galleries) i.e. 5 rows of pillars

3.) All the ~~prerequisites~~ prerequisites for starting depillaring operation have been fulfilled

4.) Method of goaf treatment → we assume that there is no surface structure to be protected, it is proposed to depillar the seam by caving method. (Better production and productivity). The roof strata is easily cavable, with the help of LHD as loading machine and a

Combination of chain conveyor and rope haulage as district /panel transport.

By the use of LHD → 1.) mechanization reduced
2.) manpower reduced
3.) greater speed
4.) greater lead distance

By use of SDL → 1.) we require a follow a strict schedule of LDCC in the levels and in case of defect in any one of the LDCC the level production is compromised as well as the entire depillaring process is hampered, because we have to strictly follow the line of extraction
2.) mechanization more
3.) more manpower.

⟹ So LHD preferred

At the time of depillaring we should have atleast 8-9 production faces.

2 types of production faces:-

1.) split faces → faces formed during splitting
2.) slicing faces → faces formed during actual slicing process.

Initially (when no. of pillars under extraction is less), we have lesser no. of ~~split~~ slice faces and greater no. of split faces
But later on when the no. of pillars under extraction increases we have an equal share of split and slice faces.

But we have some restrictions :—

1) no of slices under extraction $\ngtr 1$ at any moment in each pillar.

2) split faces should be restricted upto a distance of 2 pillars from the pillar under extraction

And the above 2 restrictions have to be followed throughout the depillaring process

Statement of the Principle:-

— The layout of the depillaring panel with caving has been shown in the drawing given

— After completing all the preparatory arrangement such as construction of preparatory stoppings, grant of permission from DGMS, advance supporting as per SSR upto 2 pillars etc, depillaring operation is started from the dip most pillar in the panel.

— It will progress towards rise side following diagonal line of extraction as shown in the drawing.

3) Splitting operation →

- Each pillar is divided into 2 equal parts by driving a level split of 4.5m width and 3 m height leaving 1 m coal at the roof.

- The splitting operation is restricted upto 2 pillars or 6 m, whichever is less, ahead of the pillar under extraction.

- However initially when the pillar extraction is about to complete, splitting operation can be extended upto 4 pillars or 120m from the first pillar to be extracted.

- The size of the split galleries is the same as size of level galleries

10/09/2013

Incase of SDL used in Depillaring → each face is to be served by various level LDCC(s), whether it is a split face or a slice face.

But there is a time lag within development and depillaring because development was done 30 years ago so the size of the gallery would be lesser than required now for mechanization during depillaring because earlier the method was manual mining. So before slicing operation we should decide for a slice width and split gallery width compatible with the mechanization deployed for depillaring.

Max width of slice and split = 4.8m

Extraction of the stook:-

- After the pillar is splitted, each stook is extracted by driving a dip slice of 4.5m width and 3 m height, leaving 1m coal at the roof.

- A rib of coal about 1.5m (1.02 – 1.8m) wide is left in between two consequetive slices which is judiciously reduced as much as possible during retreat.

- When we are working within the slice our ht is restricted to 3 m max. but while retreating from the slice we can extract the additional 1m of coal by ~~b~~ roof blasting by increasing the extraction ht. because while retreating all the supports are withdrawn and no more supporting is required. So wherever the support is removed there we can increase the extraction height upto 4.8m max.

- The total area of exposure under a slice in weak roof conditions vary from 60-80 m^2 but under competent roof condition the area allowed is within the range 120-150 m^2

- The total area of exposure under a slice is restricted upto 150 m^2 or the limit specified by DGMS in the permission letter.

- Only one slice in any one pillar is under extraction at any point of time. When the slice is completed, the l/m rod is blasted down in stages during retreat from the slice.

- Before introducing the slices and splits the area upto 2 pillars have been supported but ~~by~~ the newly formed splits and slices are to be supported using the same SSR.

- The entry point of the slice → (mouth of the slice) is to supported by special ~~sp~~ supports because it is very significant during slicing.
  Then we have to define support
  1.) along the roof edge of slice
  2.) along the centre of the slice
  3.) along the other edge of the slice

- The splits and slices are supported during advancing as per SSR, once a slice is completely extracted, the supports from that slice are completely withdrawn with the help of chain sylvester. prop withdrawer and the roof is allowed to cave.

- A new slice is then started in the similar manner after setting out goaf edge supports.

Systematic support Rule:-

Defines it { — Type of support
— Density of support }
to be put along level, dip galleries and along splits and slices.

- Support of original and split galleries
- Support at the slices
- Support at the junction.
- Support at the goaf edge

Support of original and split galleries:-

- Original and split galleries are supported by three rows of supports.

- The central row ~~o~~ is supported by roof bolts placed at 1.2 m interval.

Why??

~~Ans.~~

1) To reinforce the roof and prevent it from bending.
2) free space for movement of machines.

- The rise side of the gallery is supported by steel chock and roof bolts put alternatively at every 1.2 m interval.

- The dip side of the gallery is supported by alternative timber/steel props and roof bolts at every 1.2 m interval.

NOTE:-

Weak roof conditions → supports are denser; i.e. spacing between supports is reduced upto 0.9 m

Competent roof conditions → supports spacing can be increased to 1.5 m

Support at the slices:-

- The mouth of the slice is supported by two steel chocks with a cross bar above these chocks.

- The central portion of the slice is supported by roof bolts at an interval of 1.2 m

- The one side of the slice (generally the goaf edge side) is supported by steel chocks and roof bolts at an interval of alternate 1.2 m.

- The front row of support at the slice must not lag more than 1.8 m from the advancing face.

Physical meaning:-

~~So~~ Pull = 1.5
effective pull = 1.2 m

So after every 1 1/2 cycle of blasting we have a mandatory requirement of supporting the roof to proceeding forward.

Support at the junction:-

Junction → places where level and dip-rise galleries intersect each other.

- All the junctions upto 2 pillars ahead of the pillar under extraction is supported by four number of steel chocks with wooden cross bars or steel girders.

Support at the goaf edge:-

- The goaf edges are supported by steel chocks or wooden ~~chocks~~ props skin to skin all across the width of the gallery.

NOTE. ⟹ ~~The~~ skin to skin means the density of support is such that passing across the supports is not possible

11/09/2013.

Sequence of Operation:-

The sequence of operations during drivage of splits and slices are as follows:-

- Drilling and blasting → Coal winning is effected by drilling 36 mm diameter holes of 1.5 m length on parallel cut pattern using hand held electric coal drill machine

1.) lot of free faces already available
2.) gives better blasting efficiency

- Blasting → solid blasting (Deg I) / → free face blasting (Deg II)
- The face is blasted off the solid using P_3 or P_5 type of explosive in Conjunction with electric delay detonators
- clearance of fumes, face dressing and erection of temporary supports.

- Loading and Transport:-

- The blasted coal from the face is loaded by ~~LHD~~ SDL which discharges it on to LDCC laid along the level split galleries.

NOTE:- LHD does not require LDCC along split and level galleries.

- These conveyors discharge the coal onto MDCC installed along dip-rise gallery.

- Finally coal is loaded onto 1t coal tubs at the loading point through a short length bridge conveyor for onward transport to pit bottom/surface through a series of endless and direct rope haulages.

DRH → used to meet transport requirement by the use of gravity so it is used as main haulage in the dip-rise galleries

endless → used in level galleries for level transport requirement.

- Support

- All the working slices and splits galleries are supported ~~as per DGMS~~ . upto 2 pillars as per DGMS

- The support is as per the Systematic Support Rule.

Estimation of production:-

⇒ In any case we should have atleast 8-9 working faces in a panel.

- with the layout given, it is expected that 4 split faces and 4 slice faces would be available all the time

- The cycle time of operation at any face would be:-

  - Drilling time = 45 min
  - Charging and blasting = 45 min
  - fume clearance, dressing and erection of support - 60 min

– Loading by LHD : 75 min

– Support = 60 min

- Total cycle time = 305 min ≃ 5 hrs

Therefore, 1 blast per face per shift can be achieved easily.

Production per blast from a split face = $4.5 \times 3 \times 1.5 \times 0.85 \times 1.5 = 24$ tons

Similarly production per blast from a slice face = 24 tons.

Therefore, production per shift = $8 \times 24 = 192$ tons

Daily production = $192 \times 3 = 576$ tons

Assuming 75% availability of the faces, expected production = 0.75×576 ≃ 400 tons

⟹ The coal from roof blasting is the additional coal so nowhere we are estimating its tonnage

Estimation of No. of LHD required.

- Assuming hourly productivity of LHD = 16 tons

If the LHD(s) are effectively used for 3 hrs per shift, No. of LHD required = $\frac{400}{(3 \times 3 \times 16)} = 3.1 \simeq 4$

Man power requirement:-

- Assistant Manager = 1X3=3
- Under manager = 1X3=3
- Assistant Engineer = 1X3=3
- overman = 3+1
- Foreman = 3
- Mining Sirdar = 3
- Explosive carrier : 9
- Driller = 18
- Support gang = 27
- LHD operator = 72
- LHD helper = 24
- Roof dresser = 6
- Water sprayer = 3
- Line mistry = 6
- Material supply gang = 6
- Heavy tyndal = 6
- E and M fitter = 6
- E and M fitter helper = 6
- Conveyor operator = 12
- EH operator = 3
- Misc = 20

Total = 183

$OMS = \frac{400}{183} = 2.22$

List of equipment in the panel:-

- LHD, $1.5 m^3 = 4$
- LDCC = 4
- MDCC
- Drill m/c = 4
- Auxiliary fan with duct = 3
- Endless haulage = 1
- Face pump = 1
- Roof bolter = 1

AIR BLAST

- ~~When the roof falls~~. Looking at layout of depillaring panel. the size of galleries is small and they are staggered.

Precaution ~~of~~ against air blast:-

- At the planning stage, depillaring with caving should not be adopted in the area where roof is very hard consisting of massive sandstone
- In any depillaring area, all workmen should be withdrawn to a safe place when the

indiction of roof fall is observed. Sometimes, work may be suspended ~~for~~ for 2 hours before the roof fall occurs.

- More number of openings as far as possible should be kept opened in the panel so that the displaced air gets path to escape with reduced velocity.

- Stooks / ~~Ribs~~ ribs must not be left or at least reduced as far as ~~possible~~ practicable from the depillaring area. (Even when ribs are crushed they continue to take load so affect cavability of even good cavable roofs. Atleast they should be dislodged so that the integrity of ribs is lost.)

- After every blast, there is usually a thick cloud of coal dust in suspension, which may ignite resulting into coal dust explosion. Therefore all the electrical switches should be put off before such air blast is anticipated.

- Construction of few preparatory stopping with small weak zone or opening breakable by air speed. (In the Barrier pillars)

- Sufficient ~~num~~ number of manhole should be provided along the galleries where person can take shelter during air blast.

manhole → an entry of particular dimension as per CMR at regular interval so that in case of derailment of coal tubs, it becomes a life saving area.

- During air blast, the workers should lie down along the floor if it is not possible to take shelter during airblast (major effect of air blast is above the floor)

- After the air blast has taken place all the supports must be reset/ ~~&~~ checked before the work is started.

AIR BLAST

During Depillaring of a panel we face 2 major hazards — air blast, U/G fire

If we look at the layout of the Depillaring panel, the galleries are very small and they are staggered. Goaf is the area where pillars are removed and there is a presence of air in the goaf area. When we allow the roof of the goaf to cave, a certain volume of air is displaced. But if the caving process takes place in a gradual manner then the displacement of air in the goaf is also in a gradual manner and the phenomenon of displacement of air is not felt much. But if the caving process in not regular and as we go on depillaring the area of the roof exposed to goaf goes on increasing then a danger of air blast arises. A point comes when the entire goaf roof collapses or caves. As it caves, the quantum of material which is dislogded from the roof is so large that the quantity /quantum of air displaced in a very small time interval is also large. When a very large volume of air is displaced ~~and~~ a suddenly in a small time interval they move ~~into~~ ~~into~~ through the galleries of the panel at very high pressure.

This high velocity movement of air causes the machines in ~~it~~ the path of air flow to be ~~blow of~~ blown off and whatever supports are encountered by the moving air, they are just dislodged. This phenomenon is called AIR BLAST.

During the occurrence of the AIR BLAST the safety of the mine workers reduces largely and workers trapped ~~by~~ in the event of Air Blast are exposed to a high level of threat to their life.

So to control the event of airblast we have to control the dynamism ~~in~~ behind this process.

Hence to mitigate the ~~d~~ ~~problem~~ danger of air blast, we should ensure that the dynamism of the caving is regular and gradual. Secondary mitigating techniques involve proper training to the workers to act wisely during air blast, ~~withdrawal of machineries and supports in the path of air flow, etc.~~ use of ~~t~~ indicators which gives us a warning for the event of air blast.

12/09/2013

Precautions against fire and spontaneous heating during depillaring

- The coal ~~fel~~ left in the goaf ~~o~~ in the form of rib or stook during depillaring operation with caving is conductive to spontaneous heating as air may leak through the goaf and the heat of oxidation may not be dissipated due to insufficient oxygen. Therefore it is necessary to take steps to prevent spontaneous heating.

- The following steps may be taken for this purpose:

Spontaneous heating:—

% of recovery is not more than 75% in B and P workings in form of (rib ~~pa fo~~, barrier or coal in the roof) so a large ant of coal is ~~little~~ liable to be left in a condition where coal is liable to be subjected to autoxidation.

~~In the goaf:~~

In the goaf ~~@~~ O_2 supply is restricted, it is controlled. We are ventilating the working areas ~~too~~ with fresh air and ~~to~~ some part of it leaks into the goaf area. In the goaf area the ~~coal~~ coal is not in blocks but it is fractured and fragmented and it is easily oxidized by ~~the~~ absorbing the fresh ~~air~~ leakage air into the goaf. Due to auto oxidation, reaction is exothermic and heat is produced ~~and this~~ and as heat is produced the rate of oxidation also increases so the oxidation process takes place in an exponential manner. And as more and more heat is produced finally a U/G fire is set up. During depillaring operation we decide to extract ~~th~~ all the pillars in the panel within the incubation period but our assumption is that the oxidation of coal is a linear process ~~so~~ but actually it is exponential reaction. So in most of the ~~cases~~ depillaring cases the time of ~~the sta~~ interval between the ~~occurrence~~ starting of spontaneous heating and the first occurance of U/G fire in the panel is generally very small. So if we are not ~~care~~ careful and prepared then ~~they~~ there is a high ~~changes~~ chances of fire. And this is the case in most of the depillaring panels where fire is set out before the complete extration of coal from the panel. And once fire is encountered then we have to seal out the panel in the minimum possible ~~min~~ time and then it can only be reopened once the fire is controlled. There is a very ~~lotion~~ lengthy procedure involved in this process and the productivity ~~i~~ of the mine is largely affected.

(i) Depillaring must be done in panel system leaving barriers of coal pillars on three sides. This barriers will prevent the spreading of fire, if it occurs.

(ii) Size of the panel should be selected judiciously so that the panel can be extracted and isolated within the incubation period of the coal. (But this is not true because the oxidation process is exponential in nature so for an incubation period of 6 months, we see that fire is set out in 2 months only)

(iii) If the development has already been done in open system, preparatory isolation stoppings should be erected all around the panel which can be quickly completed in the event of fire / spontaneous heating.

(iv) Depillaring operation should be done from dip to rise so that the goaf can be submerged with water, if necessary.

(v) An attempt should be made not to leave any coal in the rib as far as possible

(vi) In shallow depth workings the subsidence is dynamic in nature (subsidence progresses along with the progress in mining) and cracks are developed due to caving operation on the surface. Prevention of breathing of air through cracks to the surface due to caving of roof. The cracks at the surface need to be blanketed.

(vii) Provision of fire fighting equipment, sand, lime, nitrogen flushing, etc in the panel.

LONGWALL MINING

↳ Method of mass extraction

Introduction:-

- Longwall method of mining consists of laying out long straight face either along the dip-rise direction or along the strike direction of the coal seam from which all the coal is removed in one working section in a series of operations maintaining a continuous line of advance in one direction and leaving behind the void which may be caved or stowed with sand.

14/09/2013

- Longwall
- The gate a ... may be ... direct...
- The face ... to 300 m ... from ... on
App...
- Caving ... Roof ... safer and ... longwall ... adverse ... unless ... solutio... timely

Longwall panel → high extraction panels
(because we are loosing very low amt of coal)

14/09/2013

- Longwall mining consists of formation of a large pillar by driving roadways called gate roads.

- The gate roads are connected forming a longwall panel. The longwall face may be laid either along dip - rise direction and advanced along the strike. Alternatively it may be laid along the strike direction and advanced ~~from~~ along dip - rise side.

- The face length may vary from 80m to 300m ~~to 300m~~ and panel length from few hundred to 2 Km depending on the availability of area / reserve available for longwall mining.

Applicability Conditions:-

- Caving behaviour of strata:-

Roof should be easily Cavable. Consistently safer and productive operation of longwall faces is very difficult in adversely caving strata condition unless proper planning and reasonable solution are not readily available for timely management of the impending problems.

→ Longwall method is the most inflexible System.

~~Ad~~

- Seam Thickness :-

Coal seams having thickness range from 0.6 – 6m can be effectively worked in one section ~~o~~ with powered support and ~~shearer~~ shearer / plough.

~~Thickness~~ Thicker seams (>6m) can be worked in slices of 2-3 m ~~thicke~~ thick either in ascending or descending order.

The thinnest seam that can be worked is about 0.4 m only depending on the quality of coal.

The thickness of the seam should be more or less uniform throughout the panel for effective mining by longwall method.

For thin seam (<1.5m) → we use plough as a cutting ~~machine~~ and loading ~~ma~~ machine.

For thick seams (>1.5m) → we employ shearer ~~of~~ as a cutting ~~machine~~ and loading machine.

Gradient of Seam:—

— Seams dipping upto 1 in 3 can be easily extracted by longwall method. For steeper gradient, special methods like inclined slicing or horizontal slicing method can be used.

Although theorically we can mine a coal seam with gradient upto 1 in 3 but practically we face a lot of problems in terms of what we call face creep. We face problems in a dip of face = 1 in 5.5

face creep → Tendency of supports to get slided in, towards the dip side ~~as~~ as the face advances

Cover depth:—

— Longwall method is the best method for underground exploitation of coal at ~~of~~ depth more than 200m from strata control as well as ventilation points of view.

— Practically longwall can be operated even at a cover depth < 40m

— Deepest longwall ~~is~~ is at a depth of 1200m.

• Strength of Coal:—

— Now-a-days, with increase in available machine power, the strength of coal has little importance. However, mechanised longwall with shearer or plough demand soft or medium hard coal for better workability.

• Geological disturbance:—

— The coal seam should be ideally ~~face~~ free from dirt band, stone, fault, slip planes etc.

— Seams liable to spontaneous heating, coal bumps etc can be worked comfortably using longwall methods

• Gassiness of the Seam:—

— Highly gassy coal seams can also be worked safely maintaining effective level of ventilation which may ~~to~~ not be possible in other methods.

— As ~~as~~ the method of coal winning is strictly based on mechanised cutting hazards associated with use of explosive is not there.

Advantages of Longwall Method

- Potential of high production and productivity

  – A longwall face can produce from 500 – 5000 t per day depending on seam thickness and type of mechanisation.

- Better ventilation and lesser chance of fire and spontaneous heating

- High percentage of extraction, almost 100% within the panel. With single entry system, overall percentage may be 100.

- Better supervision and control of operations due to concentrated working at one place

- Better roof control due to more systematic sequence and line of extraction

- High productivity and less unit cost of production

  – A mine producing same output (say 1000 tpd) by longwall will have productivity of 6–8 t compared to 1–1.5 t in case of Bord and Pillar method.

- Gassy seams and seams liable to spontaneous heating can be worked with least risk.

- wide application under different conditions with respect to thickness, gradient and depth.

Disadvantages of longwall mining

- High Capital investment
  - A heavy duty powered support longwall face 100m long needs an investment of Rs 80 – 100 Crore for production of 2000 – 2500 tpd. A Bord and pillar mechanised district producing ~~$~~ 500 tpd requires about Rs 6 crore of investment.

- High gestation period
  - More time is required for development and installation of a longwall face

- Can not be applied in geologically disturbed areas, therefore the investment decision is subjected to high risk

- High amount of subsidence observed at surface

- Selective mining not permissible

- The method needs highly multi skilled manpower

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- The installed power requirement of a longwall panel is very high requiring dedicated power distribution.

16/9/2013

Longwall advancing VS Longwall Retreating:-

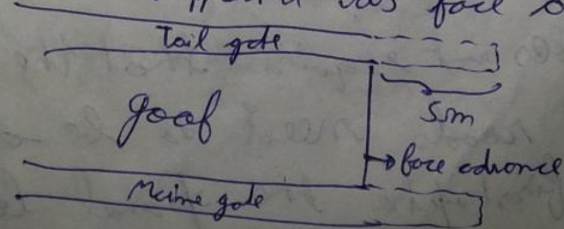
Longwall advancing:-

- It is not required to have a complete development of the panel before working the longwall.

- In longwall advancing system, a longwall face (100 - 150m long) is advanced away from the main transport route either in the dip direction or in the strike direction simultaneously with the gate roads.

- It is a greater concern to maintain and keep the gate roads functional. The strata above the heading, that has to be kept stiff.

- In this system, gate roads are advanced simultaneously with face advance - Gate roads are kept 5m advance from the face and duly supported as face advances.



- Longwall retreating

- In longwall retreating system, gate roads are driven from the main transport route upto the panel boundary and then, they are connected forming a longwall face, which is retreated from the panel boundary towards the main transport route.

- In this system, the gate roads are supported by solid coal ~~coal~~ pillars and can be abandoned as the face retreats

- We have to wait until the drivage of gate roads is completed, so high gestation period.

~~Advantages of~~

- The part of the gate roads in the goaf area does not require to be maintained so maintence of gate roads is easier.

Advantages of longwall advancing:-

- Fully productive operation of longwall mining can be started with little development work

- It does not require that the gate roads need to be driven first upto the panel boundary as in case of retreat fashion. Thus, gestation period is less.

- No investment and separate organisation is necessary for the drivage of the gate roads.

In retreating → we have special machines of high cost for drivage of gate roads at high ~~speed~~ rate

In advancing → gate roads are driven by drilling and blasting.

Disadvantages of longwall advancing:-

— The gate roads need to be maintained throughout the life of the panel in the extracted area or through goaf. It required heavy support which is difficult to maintain

— A much ~~of~~ rigid cycle of operation need to be followed. The gate roads needs to be simultaneously advanced alongwith the advance of the longwall face

— The rate of face advance depends on the rate of advance of gate roads, higher output can not be expected.

- Geology of the area can not be explored in advance as it can be known in retreating system.

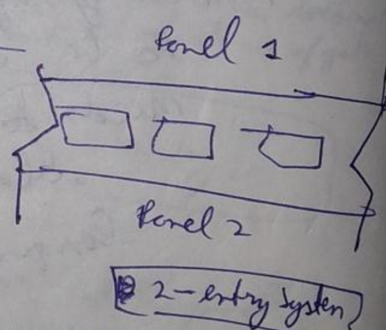
- Drivage of gate roads are to be done with drilling and blasting. Fast heading machine like road header or dint header can not be used

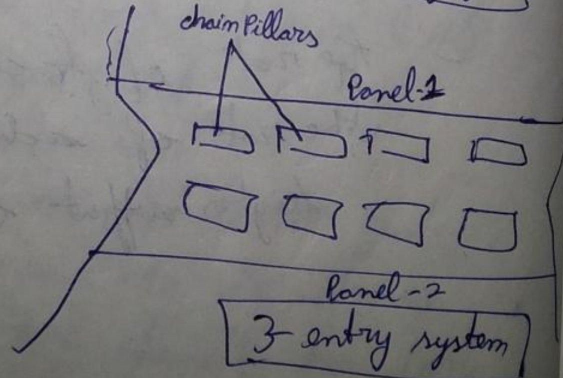
- As the gate roads are driven along the face, more dust is carried onto the face

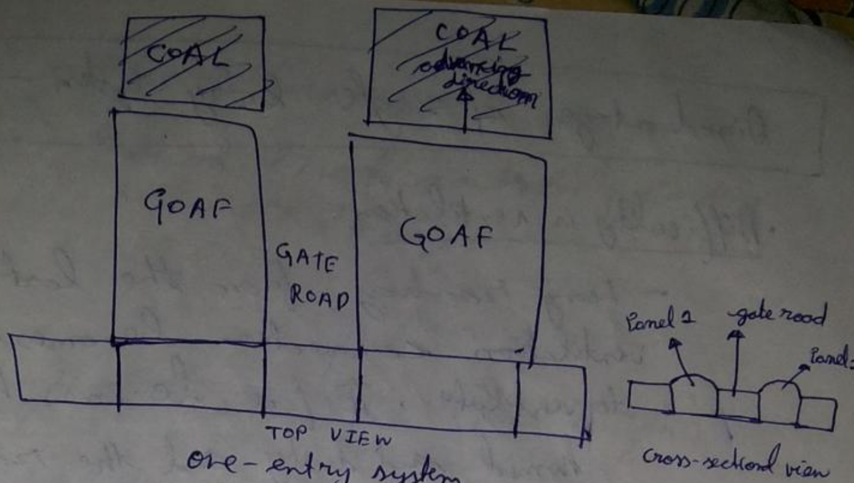
- Ventilation of the face is poor compared to retreat mining as much air is leaked through the ~~goaf~~ goaf in advancing system.

Type of entry system:-

1.) Single entry system
2.) 2 entry system
3.) 3-entry system
4.) 4 entry system
5.) 5- entry system







One-entry system

- No chain pillars in the gate road
- gate roads serves for both panels
- challenge is maintainance of gate roads
- The life of gate roads is very high
- used only in stowed longwall faces

~~In double entry syst~~

Advantages of single entry system:-

- Quick production, panel can be operaed quickly and full production can be obtained involves less development work

- No loss of coal in between the gate roads.

Disadvantages of single Entry System:-

· Difficulty in ventilation

- Long roadway from the last ventilation connection becomes difficult to ventilate. The face becomes hot, humid and dusty and the rate of drainage of roadway reduces.

- The gate roads need to be maintained for the use of the next panel.

Different parameters of longwall mining:-

· Individual variants of longwall mining include:-

- Length of the face
- rate of advance
- Cyclic and non-cyclic operation
- orientation of face, either along strike or along dip - rise
- Method of coal winning, drilling and blasting, cutting by shearer or plough

- Type of face support
- Method of roof control, caving or stowing

• Length of longwall face (face length)

- The length of a longwall face is a problem of optimisation and it depends on lot of factors.

- Variation in any of the factors affect the other.

- It is a common understanding that higher the length of longwall face, higher is the production. But it is not always true because production from a longwall face depends on many other factors while the geo-technical and rock-mechanics properties of the locale governs the technical feasibility of longwall face length.

- The overall economics has a decisive role on the face length.

longest face length — 450m

- The major factors which govern the choice of longwall face length are as follows:-

  - Strata Control (smaller length of face — better strata control)
  - Capacity of longwall face equipment (larger length of face — more is the requirement of high capacity mechanisation)
  - Strata characteristics (better roof characteristics — longer longwall face allowed)
  - Production and productivity (larger face length — more production)
  - Ventilation at the face (larger face requires more ventilation)
  - Human factors (larger is length of face — more is the distance the man have to walk - so more is the fatigue of the person)
  - Investment and cost of production
    ↳ (larger length of face — more support required - better machinery required so more is the investment)

18/09/2013.

Types of longwall mining method:-

- Depending on the type of mechanization in coal winning, support system and goaf treatment, the longwall faces may be of any one of the following types:-

  - Individual support longwall (ISLW) with drilling and blasting the face, loading on armoured face conveyor and goaf treatment either with caving or stowing (ISLW with drilling and blasting): can be used for 1.4-3 m thick seam, gradient upto 20° and daily production upto 500 tonnes.

Comparable to B and P mining (production is limited)
To realize the potential of longwall mining this method should be avoided

- Individual support longwall face with shearer, either stowing or caving (ISLW with shearer) for 3 m thick seam, 700 tpd production

- Powered support longwall with shearer and caving (PSLW with shearer) for seam thickness 2-6 m

- PSLW with coal ~~[illegible]~~ plough and caving for thin seam < 2 m thick

ISLW with Drilling and blasting with Caving:-

• Development of the panel

- A 100 m long LW face is formed along the dip-rise direction by driving tail and main gate roads following double entry system.

- The gate roads are driven with drilling and blasting with chain pillars of 20m×60m. The width and height of the roadways are restricted to 4.8m and 3m respectively

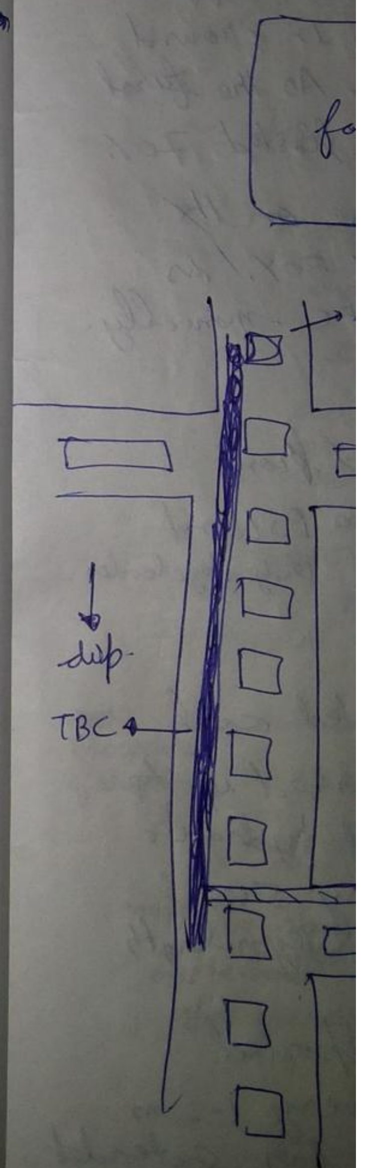
- Height of the roads may be less than 3m if the seam thickness is lesser than 3m

- When gate roads are driven upto the panel boundary, they are connected by driving a 3.6m wide gallery (installation chamber) thus forming a longwall face along the dip-rise direction, which is extracted on retreat system. Face is advanced or retreated along the strike direction

• Commissioning of face:-

- The longwall face is equipped with an armoured face conveyor (AFC) along the face, and stage loader (STL) and gate belt conveyor (GBC) along the main gate and a trunk belt conveyor (TBC) along the trunk roadway outside the panel and necessary <u>electrical power pack</u> as shown in the layout. (A large transformer with all ~~[illegible]~~ control system to provide necessary current and voltage regulation)

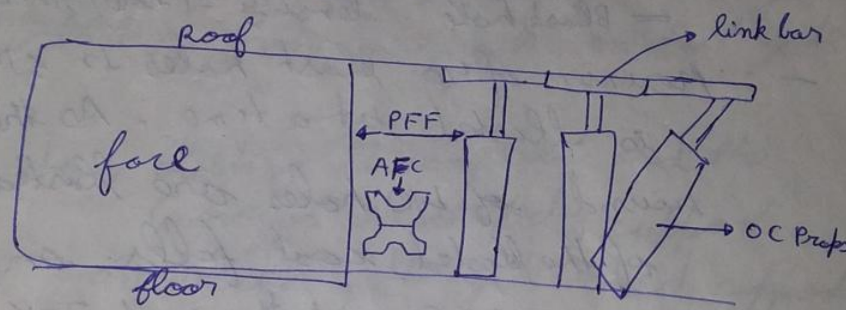
The fluid of prop which is released on yielding of the prop is permanently lost

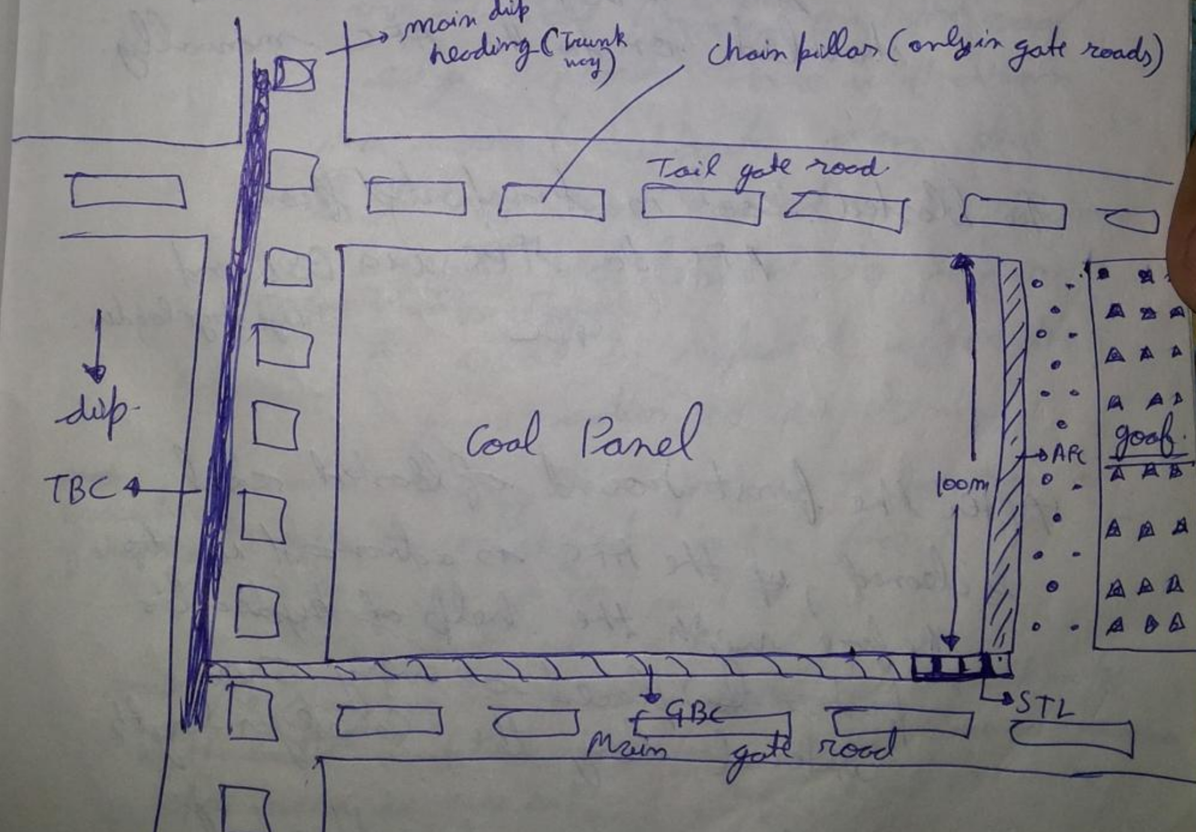


— The face is supported with 40 t open circuit hydraulic prop (OCHP) with triangular system in conjunction with link bar. (1.2m length)

yield load capacity

The fluid of prop which is released on yielding of the prop is permanently lost.





NOTE → The supply to the gate comes through the main gate and return via tail gate.

• Sequence of operation :-

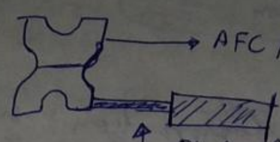
- Coal winning is affected by drilling holes in stages with the help of hand held drilling machine with 36 mm diameter with 1.5 m long holes on ~~fan~~ cut pattern

- The holes are charged with P_5 explosives in conjunction with delay detonators
  - Blast hole density = 1 hole / m^2 of face
- Maximum of 40 blast holes in a round is blasted at a time. As the first round of 40 holes are blasted 70% of the blasted coal falls on the AFC and the rest 30% is shoveled on to the AFC manually.

- The blasted coal is transported from AFC to TBC via BSL and GBC
  BSL: Bridge stage loader.

- After the first round of blasted coal is cleaned, the AFC is advanced up to the face with the help of hydraulic pusher placed at every 5 m interval along the conveyor length

- The exposed area is supported by 40 t OCHP at every 1.2 m interval with link bars extended upto the face.



- When the conveyor / support are being erected in the first round of blasted area, drilling and charging of blast hole, is done in the second round. In this way the complete face is blasted in stages starting from main gate to tail gate

- When, blasting is completed and the conveyor is shifted, supports are erected, the gate road supports are advanced / erected and the goaf edge supports are withdrawn and inclined supports along the last row are erected. The maximum span is limited to 4.2 m (distance between face and goaf edge)
The cycle is repeated after the face advances by 1.2 m.

~~and~~ Procedure of shifting { While shifting the props → The last row of support is shifted to the face

⟹ The span of 4.2 m is decided on the capacity of the ~~load~~ support installed at the face. As the ~~advance increases~~ span increases our support requirement is more.

19/09/2013.

- Support System : -

- 40 t OCHP(s) in Conjunction with link bar is used to support the face as well as gate roads. The gate roads is supported as per SSR. The main and tail roads are supported upto 30m from the longwall face because the front abutment of the face is upto a distance of 30-35m from the face.
- Generally, the triangular system of supports are used for face supports.

- The support resistance depends upon the allowable span and the depth of working. The front row of supports are kept at every 1.2m interval with the link bar projecting 2/3 rd towards the face

- The 2nd row of supports are put at a distance of 0.6m from the first row but in between the two props of the first row, maintaining 1.2 m distance between the rows.

- The goaf edge supports are strengthened with one additional inclined hydraulic prop. The pattern of supporting has been shown in the layout

- In case of stowed longwall ~~...~~, single ~~...~~ entry system can be used. So the gate roads should remain intact and functional even after one panel has been worked out. So we had a provision of Packwalls.

- Packwall is made after every 4-6m of face advance.
  - Man power required = 14
  - Pack wall width = 2.5m
  - Total cycle time = 24hrs
  - Achievement = 4-6m/day
- Coal fines are generally brought from faces
- There are acqua pack, Tekpack system where insted of coal fines, chemicals like aquacem, acquabent and Tekcem are used for quick setting and high strength.

Prop free front face:- (PFF)

~~The introduction of certain~~ :

significance of PFF → it is the distance that is utilizable for persons and machine installation

PFF provides:-

1) Space for manpower movement
2.) installation of AFC
3) installation of Shearer.

From operational point of view → PFF should be as large as possible but

AS the PFF is increased, the distance of the face end and the point of action of the support increases so from strata control side, PFF should be minimum

- The introduction of certain type of coal winning and loading machine and in particular, the application of flexible armoured face conveyor which can be moved forward without dismantling in longwall mining, has led to the elimination of prop between the coal face and the conveyor, yet the face is supported. This distance between the coal face conveyor. is called Prop free front face.

• The best way to support the prop free front face is by powered support, whose roof canopy can be extended upto the face as soon as coal is cut, with ISLW the support ~~of~~ prop free front face can be done with the help of link bar which is extended upto the face when. ~~which~~ the props are outside the AFC

PRODUCTION CAPACITY

Assumptions:-

- seam thickness = 7m
- Maximum no of holes ~~per~~ blasted per round of blast = 40
- Hole density per m^2 = 1

- Thus in one round with 40 blast holes and 3 m thickness of seam, about 13 m of face length is blasted

- Cycle Time estimation:—

— Drilling time for 40 holes @ 3 min per hole with 2 gang of drillers $= \frac{40 \times 3}{(2 \times 60)} = 1$ hour

— charging and blasting time = 30 min

— shoveling and conveyor snaking time = 30 min

— support time
  - This time segment does not hamper the cycle of operation as when the next round is being drilled and charged, support of the previously exposed area can be done.

— Total cycle time for 1 round of blasting along 13 m of the face segment = 2 hours

— Thus, in a shift, three round of blasting covering $13 \times 3 = 39$ m of face length can be blasted. Thus a face of 100 m length can complete one round of blast per day.

- Production per day $= 100 \times 3 \times 1.5 \times 0.85 \times 1.5$
$= 573$ tons

- Thus a production of about 500 tons per day can be realised from such longwall face.

ISLW with D and B and stowing:-

- In case of ISLW with stowing, the face layout, sequence of operation of drilling and blasting, conveyor shifting, support setting etc remains same as in case of caving.

- The only difference in this system is that the face is advanced maximum upto 6 m from the goaf edges, in stage (roof control is better in stowing)

- Supports are erected as the face advances without dismantling the last row of supports.

- After the face advances by 6 m, the last 2 or 3 rows of props i.e. all the props within 2.4-3.6 m distance from the goaf are withdrawn, stowing barricade is made all along the face length and the void is stowed in stages.

- Thus after every 3.6 m of face advance the production of the face is stopped and stowing is done. Therefore the production from the face reduces.

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Designation

under mana

Engineers

Overman

Mining Sirdar

Foreman

Shotfirer

Explosive Carrier

Face crew (for drilling, shovelling, AFC operation, prop withdrawing and setting) @ work load of 5t/person

- Considering that 2 shifts are lost for stowing the area of 100m X 3.6m, after every 3.6m of face advance, 2 shifts are lost in stowing. Thus 3.6m of face advance now requires 11 shifts with stowing.
- Thus average production from the face reduces to $570 \times \frac{9}{11} = 466$ tons
- If seam thickness is less, the production ~~with~~ will further reduce.

Daily manpower requirement

Designation	Caving	Stowing
Under manager	3	3
Engineers	3	3
Overman	3	3
Mining Sirdar	3	3
Foreman	1	1
Shotfirer	6	6
Explosive Carrier	15	15
Face crew (for drilling, shovelling, AFC. operation, prop withdrawing and setting) @ work load of 5 t/person	114	114

Haulage operator	3	3
Mechanical fitter	3	3
Electrical fitter	3	3
Fitter helper	6	6
Conveyor operator	3	3
Stowing gang	nil	12
Casual worker for material supply and miscellaneous job	6	6
Surveyor and chainman	6	6

23/09/2013.

Mechanized Longwall using shearer:-

- Shearer → A cutting and loading machine
- Shearer is a machine for cutting coal at LW face by grinding action. It is mounted on the ~~face~~ AFC installed along the face and is ~~mounted~~ moved from ~~one~~ one end of the face to other either with chain arrangement or by rack-rail arrangement as it cuts the face.
- The main parts of shearer are electrical motor unit, gear head, gear box unit, cutting drum with arm and haulage unit.
- The electrical motor power unit ranges from 150 KW – 1000 KW. It supplies power to the hydraulic pumps

in the haulage unit and the gear head for cutting unit.

- The arm of the cutting drum is attached to the gear head with gear box. Gear head is responsible for cutting and ranging operation of the drum of the shearer.

NOTE:- Ranging → adjusting height of the drum

- The cutting drum of a shearer is laced with spiral Vanes fitted with pick boxes and cutting picks. Cutting picks are made of tungsten Carbide. The drum may be unidirectional or bidirectional (i.e. it may rotate in one direction or in both directions)

- The diameter of the drum gives the cutting height whereas the width of the drum gives the cutting depth or the web (web depth) or the advance of the face per cut.

- Diameter of drum ranges from 0.85m to 3.5m with rotational speed of 30-45 rpm. Thumb rule :→ The drum diameter should be 3/3 rd of seam height.

- The cutting speed/flitting speed/haulage speed is maximum upto 7 m/min with avg speed of 3m/min (TBU)

- The width of drum (web depth) varies from 0.6-0.75m (Now its 1.2 m) (TBW)

- Lifting cylinder → To counteract the drum from toppling and negotiate its height (i.e. used to prevent ranging arm from toppling and used to control the arm height)

- Cowl → To direct the coal cut by drum of shearer and to get it loaded on to the AFC Panel.

- Each bit/pick is having a hole out of which pressurised water comes out to tackle dust problem at the source point. (Spraying mechanism). The dedicated motor for pumping is ~~introduced~~ integrated in the shearer and interlocked with the cutting mechanism i.e. If mechanism of controlling dust is not working the cutting will not take place.

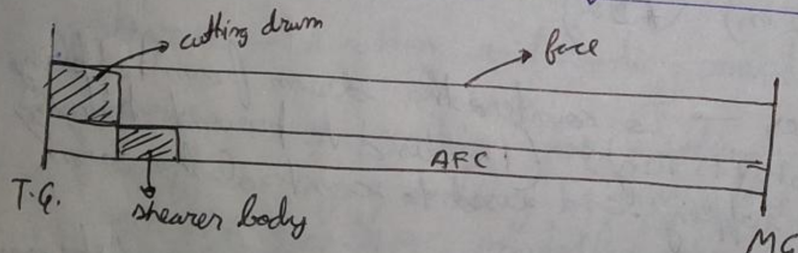
Types of Shearer (Unidirectional / Bidirectional)
(drum rotates in one direction)

- Fixed drum
  - Single ended (one arm with cutting arm)
  - Double ended (2 arms with cutting arm)
- Ranging drum (There is a scope of changing the cutting horizon by changing position of the arm as the drum is pivoted on it.)
  - SERD (Single ended Ranging Drum Shearer)
  - DERD (latest)

SEQUENCE OF CUTTING OPERATION BY SHEARER:-

- There are 2 methods of cutting by shearer
  - (i) Full face method.
  - (ii) Half face method

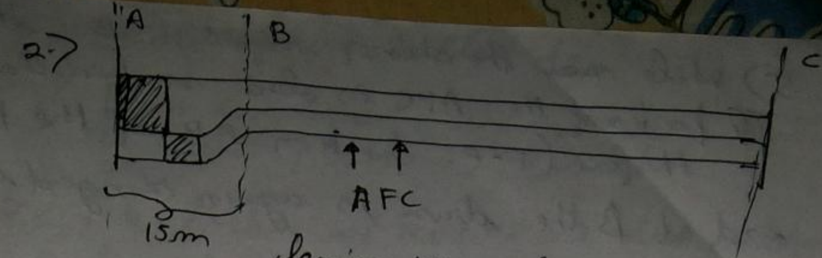
FULL face method of cutting by SERD:-



PLAN VIEW

1) The initial Condition of the ~~face~~ face configuration. The shearer is installed at the face but the rotating drum is not engaged with ~~f~~ the face to cut coal. ~~The~~ The drum is just in contact with the coal face.

2.>

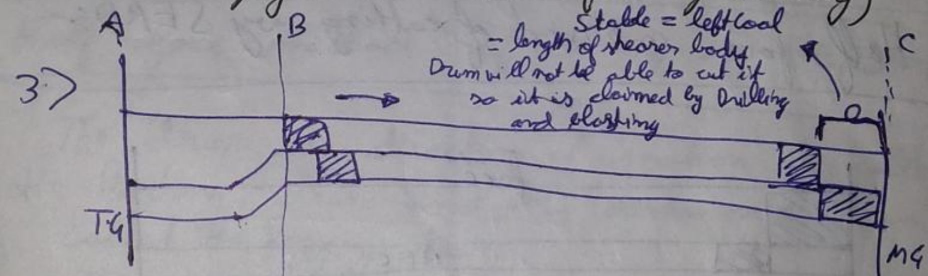


leaving 15 m from the T.G, the Conveyor is snaked upto the face. The shearer is flitted to the snaked portion and as the shearer reaches location B the machine is sumped with the drum in the raised position. The rotating action of the drum can now effectively cut the coal face.

flitting operation ⟶ movement of the drum without cutting operation, from one end to the other.

Sumping operation ⟶ when the drum of shearer gets engaged with the face to initiate cutting operation (operation to allow drum to be engaged with coal face gradually)

3.>



As the shearer moves from B to C (MG) cutting along the upper horizon takes place

4.> On reaching the end C the drum is ranged down (as it is a SERD so only 2/3 rd of the face is cut and remaining has to be cut by ranging the arm downward). The direction of movement of the shearer is reversed and it moves from C to B cutting the lower horizon of coal. While cutting the lower horizon the shearer serves 2 purposes:-

1.> cuts the lower horizon of coal

2.> Engaging already cut coal from upper horizon and loading it on AFC (i.e. it clears the goaf area by tackling coal lying down and pushing it onto the AFC)

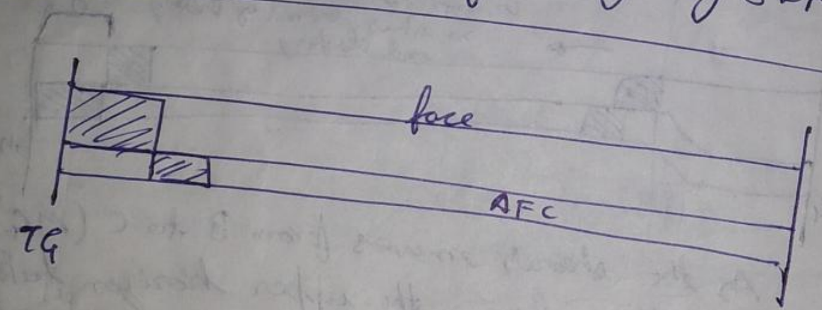
5.) while ~~...~~ the shearer moves to B the TG portion of the AFC is also snaked towards the face (i.e. portion AB of the AFC) and at B the drum is again ranged up.

6.) The remaining upper horizon at AB is cut off creating free face from the lower horizon of AB portion of the face

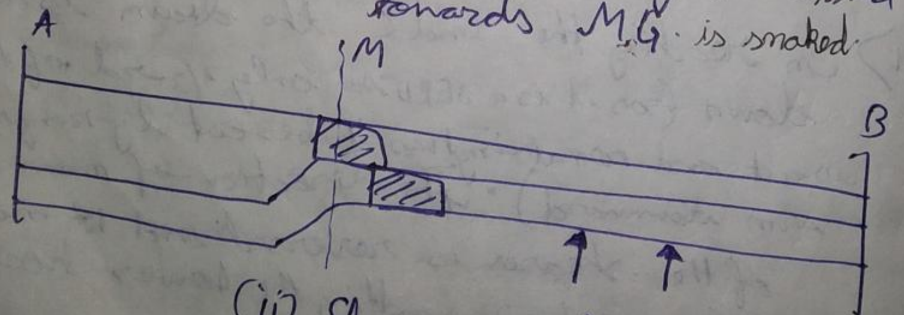
~~7~~ 7.) Then on reaching A the drum is ranged down and the lower horizon ~~...~~ of AB portion is mined as the shearer moves from A to B. ~~and the B...~~ This is one complete ~~...~~ cutting cycle.

8.) The Portion BC of the AFC is again snaked towards the face and the above cycle is repeated.

Half face method of cutting by SERD:-



(i) Half of the conveyor, ~~...~~ towards M.G. is snaked.

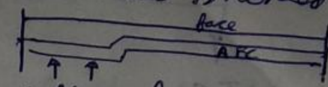


(ii) Shearer is flitted to the snaked portion and the machine is sumped with the drum in raised position. Half portion of face is cut along the roof.

3.) Stable of 2 m is blasted.

4.) Drum is lowered, shearer direction is reversed, Bottom half is cut.

5.) T.G. portion of the AFC is also snaked towards the face



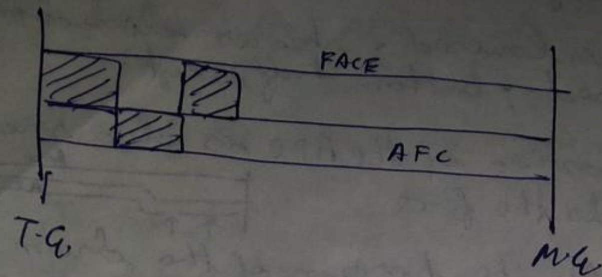
6.) At M the horizon of the drum is raised, the ~~f~~ upper horizon of coal from M to A is cut off

7.) At A the drum is again lowered and direction of motion is ~~do~~ reversed. towards M and lower horizon between A and M is extracted by the shearer. The MTG portion of the AFC is further snaked and the shearer is brought back to the half face configuration as it was initially with the drum in the the raised position.

Half face method of cutting by DERD ⇒

- The drum in direction of motion is called the leading drum and the ~~drum~~ other drum is called the lagging drum.
- With DERD ~~the~~ we have a provision to extract the entire face at one go. The leading drum cut the upper horizon while the lagging drum cuts the lower horizon.

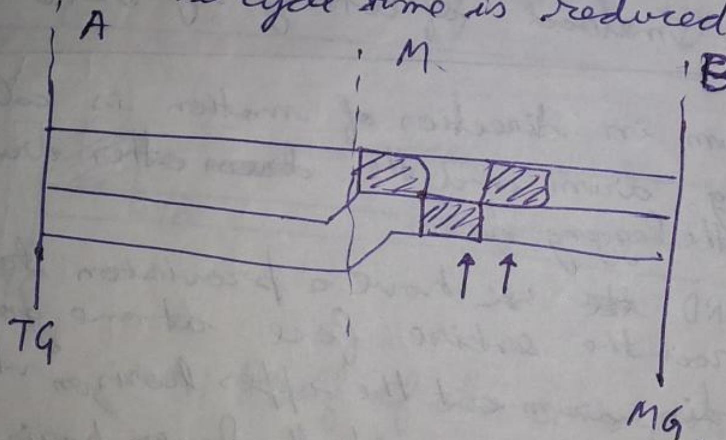
The leading drum cuts the upper horizon as well as creates ~~a~~ free face for cutting the lower horizon while the lower horizon is cut by the ~~lower horizon~~ lagging drum and in addition ~~the~~ it loads the coal previously cut by the shearer's leading drum onto the AFC.



~~As the~~ — The MG drum is raised while th T.G drum is in lower position

— Half portion of conveyor is snaked along MG. Shearer is ~~fltter~~ flitted to snaked position and sumped into the face. Shearer is run towards the MG cutting full face thickness.

— ~~Hence~~ Here no stable coal is left at the MG end so drilling and blasting is not required so cycle time is reduced.



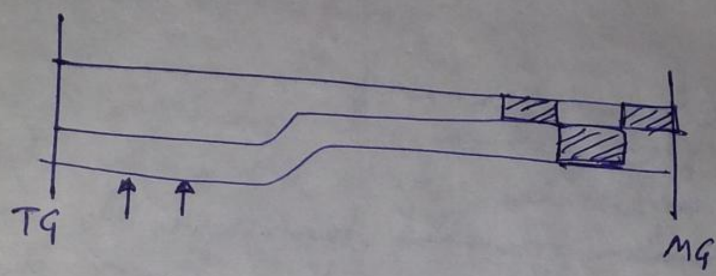
— Drum position is reversed, cutting the roof coal by TG drum and floor coal by MG drum at B. Direction of DERD is reversed ~~as the~~ from B to M. The portion AM of the AFC is also snaked towards the face and the remaining half of the face is also cut off. ~~AdA~~

(TG Portion)

- At A again the ~~drums~~ horizons of the drums are reversed (MG up, TG down) and shearer is moved towards M (direction changed)

- The shearer is brought back to the middle of the face and sumped into the face.

- ~~The unit~~ After one complete ~~mining~~ cycle ~~the~~ of the shearer the initial configuration of the shearer is attained back.



ISLW with shearer and stowing

- In this case, the production will proportionately reduce as after every $\underset{\text{(4 cuts of operation)}}{\underline{2.4/3m}}$ face advance, the goaf is stowed which takes about 2 shift. If stowing is done after every 2.4m of face advance, the production per day $= \frac{810 \times 4}{6} = 540t$

<u>NOTE:</u>- One cut of 0.6 m is obtained in 1 shift (as an average value)

Manpower, Deployment in ISLW faces with shearer:-

Designation	ISLW Caving	ISLW Stowing
Under manager —	3+1	3+1
Engineer —	3+1	3+1
Overman —	3+1	3+1
Foreman —	3+1	3+1
Mining Sirdar —	3+1	3+1
Shearer operator with helper —	6+2	6+2
Hydraulic fitter —	3+1	3+
Hydraulic fitter helper —	3+1	3+1
Conveyor operator —	9+3	9+3
Haulage operator —	3	3
E and M fitter —	6+2	6+2
E and M fitter helper —	6+2	6+2
Prop withdrawer and setter —	45	45
Stowing gang —	0	9
Material supply gang —	4	4
Others —	16	16

OMS = 5-6 for caving; 4-5 for stowing

Installation of equipment at the face and panel:-

- The longwall face is equipped with AFC, DERDS, BSL and GBC is placed along the gate road. The powered supports are placed one after another along the face from the tail gate to the main gate. The gate roads upto 20 m from the face are supported by 2 or 3 rows of 40t OCHP(s) at every 1.2m interval in conjunction with I section steel girders.

26/09/2013.

Sequence of operation:-

- Coal is won by cutting with shearer (DERD) by half face method, a sump is made at centre of the face by snaking the AFC and the shearer is flitted to the sump position.

- As the shearer enters into the coal face, it is moved from centre to the MG thereby cutting the coal. Both the drums are adjusted in such a manner that full thickness of the seam is cut in one sequence.

- When the shearer reaches the MG the cutting horizons of the drums are reversed and the coal left at the main gate is cut by moving the shearer in the reverse direction.

- While the coal face is being cut in the MG side, AFC in the TG side is snaked starting from the middle of the face and the supports are advanced.

- The shearer is brought from the M G side to the centre and the face is cut from the centre up to the T.G.

- The ~~sh~~ DERD cuts ~~in one~~ one half of the face and return to the centre by flitting operation (In each half face). The advantage of this process is to reduce the health hazard of the workers (due to dust and gas generated at the face) by reducing the time of exposure of the workers to cutting operation at the face.

- The sequence of operations of conveyor advancing and support shifting are as follows:-

(a) with the legs of the supports in raised position i.e. having set against the roof, the conveyor is pushed forward with the help of double acting ram pump.

(b) the legs of the supports are lowered down, the supports are then advanced forward by activating the double acting ram pump in reverse direction.

(c) the legs of the powered support is then raised and set against the roof.

(d) All the supports are advanced, the roof on the goaf side caves in.

Transport of coal.

- As the face is cut, coal falls onto the AFC. From AFC coal is transported to outbye TBC through BSL and GBC

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Shifting of BSL, AFC and GBC:-

- AFC is shifted after ~~each~~ every cut of the face i.e. 0.6 ~~[illegible]~~ of face advance
- BSL is shifted after every 3 cut i.e. 1.8 m of face advance
- The length of the stage loader is about 15m. It can overlap by 9m from the tail end of the GBC. Therefor, the GBC needs shifting ~~after~~ every 9m of face advance only by shifting the tail end and storing the extra belt in the loop take up arrangement of GBC. However, GBC needs shortening by cutting out the belt after every 30-35m of face advance.

Estimation of production Capacity:-

- With average cutting speed of 3 m/min and time taken to prepare the sump, flitting of sheares, changing the position of drum, etc, one complete cut takes about 1.5-2hrs. Thus in one shift, 3-4 cuts are possible (NOTE:- In powered support the supports can also be advanced efficiently ~~to~~ as the face is cut and requires no additional time as in ISLW) **ASSUMPTION:-** Utilization factor of the machinery at the face is 40%.
- Production per cut = Face length × web depth × extraction height × solid density of coal = 100 × 0.6 × 2.5 × 1.5 = 225t

• Production per shift = 225×3 = 675t (min)

= 225×4 = 900 t (max)

• Production per day with 2 production shift and 1 maintainance shift
= 675×2 = 1350 t (min),
1800t (max)

• Production with 4 overlapping shifts (i.e. 3 production shift and 1 maintainance shift), production per day = 3×675t = 2025 t (min)

3×900 t = 2700 t (max)

Manpower requirement (4 shift working):—

• Asst Manager	4
• Assistant Engineer	4
• Overman	4
• Mining Sirdar	4
• Foreman	4
• Face crew	20
• Gate road support	24
• Conveyor operator	9
• Hydraulic filler and operator/helper	8
• Power pack operator	4
• E and M fitter	8
• Fitter helper	8
• Miscellaneous for shifting of cables, conveyor cleaning etc	15

Total: 116

01/10/2013

Stowing

The goaf is required to ~~f~~ be filled → For goaf treatment

Introduction and objective:-

- After the mineral has been extracted from underground, stowing operation may be required for filling the mining excavation with the following objectives:-
  - To control ground/roof
  - To minimise damaging effects due to mining on other mine workings and mine surface
  - To dispose off stone obtained from ~~works~~ development workings (Particularly metal mine workings)
  - To control fire (Coal seams having very high susceptibility to fire cannot be worked in conjunction with caving)

Applicability Conditions:-

- Stowing is essential to protect important sensitive surface features
- While working under sea, canal or water bodies
- Working under overlying seam which has not been extracted (Potential of overlying seam to get mined out in future should remain intact)
- Working thick seam in multi-section in ascending order.

- Working seams in which unusually strong methane exudation is feared (During caving → during the time of peak loading the coal seam is going to be crushed and a high amt of CH_4 locked in the pores of the strata is released suddenly and there is a non consistent ~~supply~~ release of CH_4 and in those period our ventilation network is not capable enough so it is safer to work with stowing to avoid such situations)
- Seams with geological disturbances (All folds and faults are naturally weak zones (sheared zones) so when working in the vicinity of weak zones we are going to trigger the instability due to mining (a very less amt. of energy is required to trigger instability). Our conventional supports do not have the capability to control such regional instability problems so we should employ some regional supports. So we have to come up with stowing.)
- Seams with massive sandstone roof (not easily cavable so unsafe)
- Seams liable to spontaneous heating or fire (During caving → we are leaving the goaf with a potential to get oxidized by the leakage of fresh air into the goaf with high concentration of O_2 but by opting for stowing, we are able to ~~to~~ reduce the chances of oxidation by allowing less leakage into the goaf area. This stowing is hence opted in firey coal seams.)
- Seams having high concentration of methane gas.

Advantages of stowing:-

- It protects the surface structures
- Solid packing results in good ~~b~~ strata control and reduces the effect of subsidence
- Roadways and working faces ~~of~~ are maintained in better ways with safety.
- The chances of accumulation of fire damp in goaf and its subsequent expulsion into the working are eliminated / avoided by filling the void by inert material.
- It contributes towards elimination of bump/ burst and air blast.
- It renders possible the thick seam below a fire area or water bearing area, under railway & towns etc.

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- Higher extraction percentage is achieved, an aspect which is very important from mineral conservation point of view.
  with caving → The extraction of pillars is ~~easily~~ difficult
  with stowing → we have a good control on the voids, the loading on our workings will be less, damage of roof will be less so we have a greater time to work out the slice roof coal or rib coal so increasing extraction %age of coal.
- Ventilation planning is made easier as chances of air current leakage into the goaf is eliminated
- Hazards of fire due to spontaneous heating is eliminated.
  (Breathing ~~[illegible]~~ into the goaf from:-
  1.) surface cracks due to subsidence (subsidence reduced)
  2.) U/G due to leakage of fresh air
  is less so the hazards of fire is also reduced.)

Disadvantages of stowing

- Stowing adds extra cost to the mining operation (cost ↑; productivity ↓ so disadvantageous)
- Non availability of stowing material
- Increases the cycle of operation of mining and reduces the production.
- Increases the man power, additional pumping of stowing water and humidity in case of hydraulic stowing. (Wet Bulb temperature ↑ so difference between wet and dry temperature reduces so humidity increases and ventilation efficiency reduces. The problem is pronounced in Deep mines where DBT is already very high due to high geothermal gradient.)
- Increases dust in case of pneumatic / mechanical stowing.

· Increases humidity in case of hydraulic stowing.

Stowing materials:-

· Sand
· Crushed overburden
· Crushed ore
· Washery rejects
· fly ash
· Mill tailings (In metal mining)
· Blast furnace slag (where steel plant is located nearby)
· Boiler ash.

Operational properties of suitable stowing material:-

• The suitability of stowing materials vary with the method of stowing adopted. In general, a stowing material should have the following characteristics:-

- The material should ~~be~~ not stick to the transporting and conveying equipment, storage bunkers and Stowing machines; the stickiness of material depends on the moisture content and the presence of clay fines.
(more is the clay, more is the ability to pick up moisture)

- The material should not crumble down to smaller size during its transportation and it should have certain strength

- The material should contain as small a percentage of fine content as possible so that it does not pose a serious problem with mechanical or pneumatic stowing methods. With pneumatic method, the consumption of compressed air increases, with increasing fine content.

Fines do not settle and so we are not able to project them to the requisite place. By using compressed air, insted of getting projected to the required place, there is formation of air bond dust if % fines is more in the stowing material.

- Moisture content in the stowing material must be low

- The material must have suitable size distribution so that it fills the void completely, yielding / resulting in compact mass that is least porous. So it allows little convergence thereby adressing the problem to be avoided due to roof deformation.
- The material should be capable of being compressed only little so that the % of convergence is less.

- The material should not be abrasive (maintainence requirement of stowing channels and equipment increases)

- The material should not exert considerable pressure on barricades under rock pressure. So the material should be porous enough so that is allows systematic relaxation or systematic seepage of water once it settles down so that the water can be drained out properly.

- The material should have high specific gravity so that it compresses little and exerts pressure on coal stooks.
- The material must not get dissolved when mixed with water in hydraulic sand stowing
- The material must not be harmful to human beings.

17/10/2013.

Classification of Stowing methods:-

- Hand filling
  - Dumping stowing material in basket with the help of human labour into the void.

- Pneumatic stowing
  - Throwing or introducing the stowing material in a stream of compressed air by pneumatic stower / blower.

- Mechanical stowing
  - Throwing the material into the void with high speed belt conveyor.

~~Bo~~ material → (large specific gravity, particle size should be larger)

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• Hydraulic Stowing

– Mixing the sand/stowing material with water into suitable proportion and transporting the mixture in pipes and discharging it into the void. The water percolates through bamboo mating or similar perforated barricade erected at the site.

17/10/2013.

Pneumatic Stowing – (obsolete technology)

• It consists of discharging stowing material under the stream of compressed air. The stowing machine through which pneumatic stowing is accomplished, is called pneumatic stower. It is placed at a convinient point near the roof to be packed.

NOTE:- A greater safety concern arises here as the stower has to work in a close proximity of the area under consideration

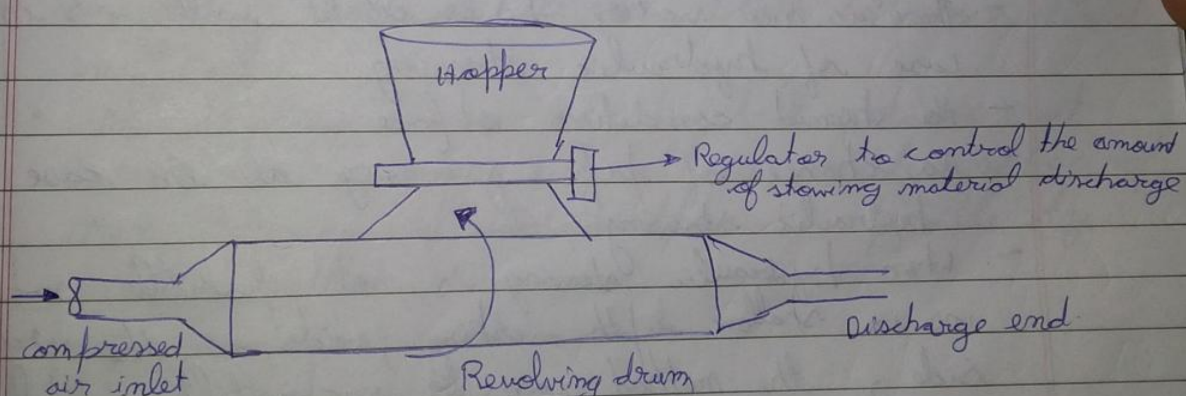


FIG: Pneumatic Stower

Stowing material in pneumatic Stowing :-

- The stowing materials may consist of washery rejects, boiler ashes.
- Sand alone is not used as it is heavy and abrasive
- A mixture of sand and washery rejects may be suitable for this purpose
- Debris containing much clay material is also unsuitable as it clogs the pipe.
- The suitable size of crushed material (65mm apertured) is brought to site either by gunny bags or tubs.
- The required compressed air can be either created at the site by a compressor or it can be conveyed to the stower through pipes from surface.

Advantages of pneumatic Stowing :-

- There is no water to be dealt with as in case of hydraulic stowing
- No humid condition at face
- No chance of pipe jamming as in case of hydraulic stowing
- when hydraulic stowing is not possible such as in shallow depth working, voids on the rise side, the method can be successfully applied
- coarser material can be used.

old data

Disadvantages:-
- High production of dust and nuisance
- High energy consumption
- High velocity of stowing dust is dangerous
- Packing is only 70% as compact as solid
- High Convergence of the order of 20-25%

Mechanical Stowing → Maximum reliability

- In this system, a high speed belt is used to propel the stowing material into the goaf to be packed. The stowing material is transported to the thrower belt either by bags or mine tubs or by means of any other available system.
- The material is deflected on the thrower belt by deflection plate known as scraper plough, which drops the material into a hopper above the high speed stowing belt.

old data
- The stowing belt is mounted on a travelling carriage and a 660 mm belt is considered sufficient.
- The driving drum of the belt rotates at 1000 rpm giving the belt a velocity of 600 m/min. Stowing of 50-60 t/hr is possible by this method.

Advantages:-
- It is the cheapest method, less capital intensive
- Energy cost is about 1/4 of that in case of pneumatic stowing

- Different uneven sizes of material can be used
- Operation is simple, reliable and safe

Disadvantages:-

- Higher % of convergence than pneumatic type
- High wear of throwing belt
- High dust and noise
- cannot be applicable in thick seams

Hydraulic Sand Stowing:-

(a) Hydraulic stowing consists of mixing stowing material, mainly sand with water in definate proportion in a mixed cone/trough and conveying the mixture through metallic pipes up to the void to be filled

(b) The mixture is transported due to gravity and no power is required for stowing.

(c) A modern hydraulic sand stowing plant consists of—

1.) a sand bunker
2.) a mixing chamber which houses
    - slow speed conveyor
    - high speed Conveyor
    - mixing trough
    - Vibrating screen
3.) water level indicator to check slurry level
4°) water discharge pipe/pumps

5.) stowing range
6.) water reservoir
7.) Access to the mixing chamber either by drift or by stable pit.

Steps of hydraulic stowing operation:-

1.) Sand at different rate is fed to the slow speed belt conveyor by opening gate / chute of the sand bunker. The slow speed conveyor recieves sand and feeds to high speed conveyor at uniform speed.

2.) The high speed conveyor discharges sand over the vibratory screen provided at the top of mixing cone. Along with the sand water in required quantity is mixed over the screen. The screen prevents entry of big boulders ($> d/3$ where d = diameter of stowing pipe) into the mixing cone.

3.) Water is fed from reservoir at surface through pipes. The quantity of water to the trough is maintained by regulating the valve connected at the discharge end of the pipe

4.) The water pipe is generally provided all around the trough like a garland and T-opening is taken from the main so that sand is mixed thoroughly with water in the mixing cone.

5.) The big boulders that accumulates in the chamber i.e. over the mixing trough is removed periodically through the access drift.

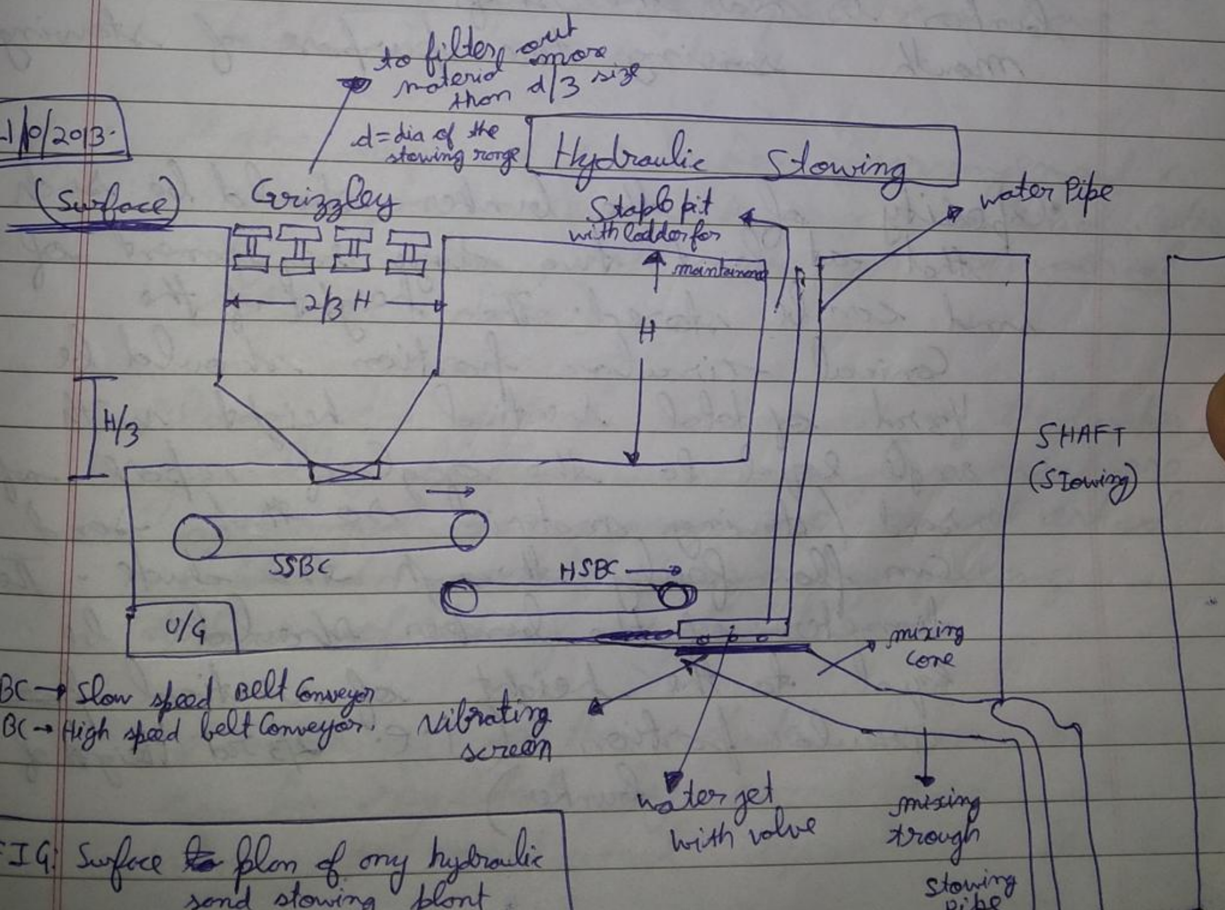
6.) As the mixture falls on the mixing trough the mixture gravitates along the inclined wall of the cone and enters the stowing pipe connected with air tight seal at the back of mixing cone.

7.) Water level indicators consisting of electrodes are placed at different level within the mixing cone to record or display the water level / mixture level in the trough, these electrodes are connected to battery via electric bulbs on a board placed at the mixing chamber.

8.) Each level indicator is connected with an electric bulb, which glows when water level falls down.

21/10/2013.

Hydraulic Stowing



SSBC → Slow speed Belt Conveyor
HSBC → High speed belt Conveyor.

FIG: Surface ~~to~~ plan of any hydraulic sand stowing plant

Design of Stowing plant:-

1) Stowing Bunker (Sand Bunker):-

• A large ~~Bunker (Sand Bunker)~~ ~~circular~~ ~~capacity~~. capacity vertical store with a conical or tapering bottom, furnished with valve or mechanically operated chute having dimensions not less than 0.8m x 0.8m, which controls the discharge of sand into the mixing cone. The bunker is driven or excavated from surface and extend to certain depth below the surface. The ideal location of stowing bunker is near the shaft or incline mouth serving the purpose of stowing.

• Capacity of the bunker should be such that at least two days requirement of sand can be stored. The height of the conical circular portion should be 1/3rd of total vertical height with angle equal to the angle of repose of sand / stowing material so that sand can flow freely through the chute. The diameter of the bunker should be equal to the height of vertical ~~portion~~ circular portion (i.e. 2/3rd Height of bunker)

- The top of the bunker is covered with grizzly of such size having aperture smaller than the dimension of the chute. This will prevent the big boulders from entering into the bunker avoiding jamming of chute gate.

- For prevention against the collapse, the bunker is lined either with bricks or concrete. Lining also reduces the frictional resistance in addition to increasing the life.

- If capacity of bunker is small → instead of lining the bunkers, we can have a steel bunker instead.

- At the top of the bunker, arrangement is made to recieve stowing material either by trucks, belt conveyor or <u>aerial ropeway</u> (older)

- Along the vertical sidewall of the bunker, generally platforms or angle iron are arranged at every 2.0 m of vertical depth to avoid large free falling distance of stowing material.

- The opening of the chute below the bunker can be regulated by compressed air valves or mechanical operation. The amount of sand to be feed into the mixing cone from the bunker can be regulated by the opening of the chute.

Modern technique of regulation

- The alternate arrangement of regulation of the feed is by use of slow and high speed belt conveyor between the sand bunker and mixing cone. By regulating the speed of the conveyors, a particular amount of feed can be fed through the mixing cone.

2. Mixing Chamber/Stowing Chamber:—

- The stowing/mixing chamber is a room normally located vertically below the sand bunker. The chute from the sand bunker, feeder belt conveyor, screen over the mixing cone, water sprinkling arrangement, water level indicator panel, etc. are housed in this chamber. An approach drift or staple pit is driven from surface for access to the stowing chamber.

3.2 Mixing cone / Plant / Trough:–

- Sand - water mixing cone or mixing plant or trough is the most important aspect in hydraulic stowing. The design of the mixing cone should be such that it is always filled upto certain level with slurry. To achieve this, water pipes are arranged above the mixing cone like a garland to inject required quantity of water into the cone.

- The mixing cone is excavated like a funnel. The floor of the cone is sloped towards the shaft or stowing drift at an inclination of 1 in 3 so that slurry ~~gravities~~ gravitates towards the stowing pipe.

- The floor, roof and sides of the mixing Cone is made of concrete. Sometimes, conveyor belt is placed along the floor to facilitate flow of sand and prevent erosion of mixing cone. The capacity of the mixing Cone depends on the stowing rate for a particular pipe range and it ~~should~~ should'nt be less than $10 m^3$

- A screen is provided on the top of mixing cone to screen out over size material (size greater than D/3 ; D = diameter of the stowing pipe). The screen can be made vibratory or stationary with provision for monitoring to clean the aperture of the screen against clogging

- The length of the screen varies from 3-5m, width 1-1.5m gradient 5°. The oversized material after screening out, are removed periodically from the mixing cone.

22/10/2013

4.> Water Jets:-

- water jets are provided on the top of the mixing cone to inject water into the mixing cone for mixing of sand and water. A main water pipe is brought from the surface reservoir and the pipe is arranged like a garland.

- Holes with valves are made into the pipes which discharges water in required quantity. The diameter of the water pipe, head of water and the discharge opening (number and size) are determined depending on the water required to make a slurry of required concentration. Generally sand : water at 1:3 is required for stowing.

5) Water level indicator:-

- The level of sand and water mixture in the mixing cone is very important to obtain full bore stowing. The cone must be filled all ~~t~~ the time at least 2/3 rd of its height, otherwise, cavitation will occur and air will be sucked into the pipe causing pulsatory flow and ultimately causing pipe jam.

- To know the level of slurry in the cone, electrically connected electrodes are dipped at different level/depth of the mixing cone. The electrodes are connected to electric bulbs placed on a control panel through spring loaded solenoid valves. When water level goes below a particular electrode, the corresponding bulb will glow indicating that the water level has gone down. Audiable alarm is also provided which gives alarm in case water level goes below critical limit.

6) Stowing Range:-

- The slurry leaves the mixing cone and enters into the stowing pipe placed along a stowing drift connected to shaft. Stowing range is installed along the shaft with stool pipe and buntons.

- By adding successive pipes, the range is taken upto the place of stowing. A 'Y' pipe is fitted at the junction of the drift coming from mixing cone and the shaft. Immediately below "Y" pipe, a stool pipe is fitted to absorb thrust and reduce vibration in the shaft range.

- Stool pipes are installed after every 12 m length of pipe to divide the shaft column pressure into different section.

- A sound stowing range with robust and long life is naturally the aim of stowing plant installation. This invites choice of proper material for the pipe and mechanical design of the range.

7) <u>Materials for Pipe:-</u>

Cast iron, most commonly used in India, is one of the most unsuitable material for the purpose, having very short life. Ordinarily steel pipes are better but hardened steel with or without tempering and alloy steel pipes are preferable.

- The wear in the pipe depends directly on
  - the resistivity of the pipe against abrasion
  - speed of the slurry
  - concentration of the slurry (sand: water ratio)

- The inner surfaces of the pipe should be very smo

- Slight roughness and undulations in the pipe may cause turbulence in flow and pipe jam.

- Mechanical designing of pipe range are those of proper centering of the pipe against each other, and those of increasing life of bends. The pipe wear in the shaft range may be greatly reduced by inserting a small conical ring at each joint of pipe. The slurry may thus, be directed away from the walls of the pipe. To increase the life of the bends, the most common practice is ~~the~~ to thicken the outer radius wall or lining of the inner bend pipe.

Theory of Hydraulic stowing.

- The sand water mixture leaves the mixing cone with a velocity depending on the cross section of the pipe and quantity of mixture.

- After passing the foot pipe of the shaft range, the sand water mixture moves in a horizontal or slightly inclined pipe range.

- For the flow of the mixture along the horizontal pipe, a pressure head is necessary to be created which must be higher than the frictional head loss, necessary to overcome the resistance in the pipe range and the discharge velocity head. The more the resistance, the higher is the vertical pressure head necessary for stowing.

- Since under normal condition, the discharge velocity does not exceed 3-4 m/s, corresponding to a head of ~~abo~~ about 0.5 m, only the pipe resistance is of importance. This resistance not only depends on the diameter and length of the pipe range, but also on the specific gravity of the slurry, i.e. a higher sand: water ratio will cause a higher resistance.

- The ~~specific~~ frictional head loss or frictional head required to cause flow of sand water mixture is given by:-

$$H = \lambda \frac{l v^2}{2 d g}$$

$$\lambda = \left(a + \frac{0.0018}{\sqrt{vd}}\right) \times \text{spe. gravity of slurry}$$

where:-

$H \rightarrow$ frictional head loss in m

$l \rightarrow$ length of the pipe range in m } equivalent horizontal length

$d \rightarrow$ diameter of pipe range in m

$v \rightarrow$ velocity of slurry m/s

$g \rightarrow$ acceleration due to gravity (m/s^2)

$\lambda \rightarrow$ is a factor.

$a \rightarrow$ is a constant 0.02 – 0.03 for sand-water mixture

For ~~straight~~ horizontal sections: length of pipe = horizontal length

For inclined sections: length of pipe > horizontal length of pipe

It is the horizontal length of pipe which is considered.

• when different diameter pipes are available, it is essential to reduce all the lengths to an equivalent length of pipe to a chosen reference diameter. If L_1 is length and d_1 is diameter of the pipe, then equivalent length L_0 to a base diameter d_0 is given by the relation

$$\underbrace{L_0}_{\text{equivalent horizontal length}} = \underbrace{L_1}_{\text{actual horizontal length}}\left(\frac{d_0}{d_1}\right)^5$$

~~• The hydraulic profile of any hydraulic sand stowing system is a plot or graph depicting the network of stowing pipe and available head at any point of the pipe range. It is obtained by plotting the length of the pipe along X-axis against the head available plotted along Y-axis.~~

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• The specific gravity of the slurry may be obtained by the relation

$$\text{Sp. gravity of slurry} = \frac{Q_S \gamma_S + Q_W \gamma_W}{Q_S + Q_W}$$

where Q_S and Q_W being the quantity of sand and water in m^3/min, and γ_S and γ_W their specific gravity

- Sand water mixture being a heterogeneous mixture, the flow of sand is by means of saltation. A minimum velocity is necessary for maintenance of transportation of the sand-water mixture through the stowing range with available head. This is called critical or shuttling velocity. This minimum velocity to assure flow in the pipe is called critical velocity.

- The head loss corresponding to this critical velocity is the minimum for a given pipe range and mixture for any velocity.

- If the head available is less than that required to maintain this critical velocity, no stowing is possible. Therefore, the knowledge of critical velocity is important.

$$\underset{(m/s)}{V_c} = K\sqrt{\frac{2gD(\gamma_s - \gamma_w)}{\gamma_w}}$$

where $K = 1.34$ for sand,

γ_s and γ_w are specific gravity of sand and water,

D is the diameter of pipe (m)

(equivalent diameter)

Hydraulic profile, hydraulic gradient and H:L ratio:-

- An underground stowing system from surface generally comprise of vertical, horizontal, dipping and rising pipes depending on the pipeline route adopted in the mine

- It is obvious that vertical and dipping pipes develop pressure helping the mixture to flow where as horizontal and rising pipe consume the pressure

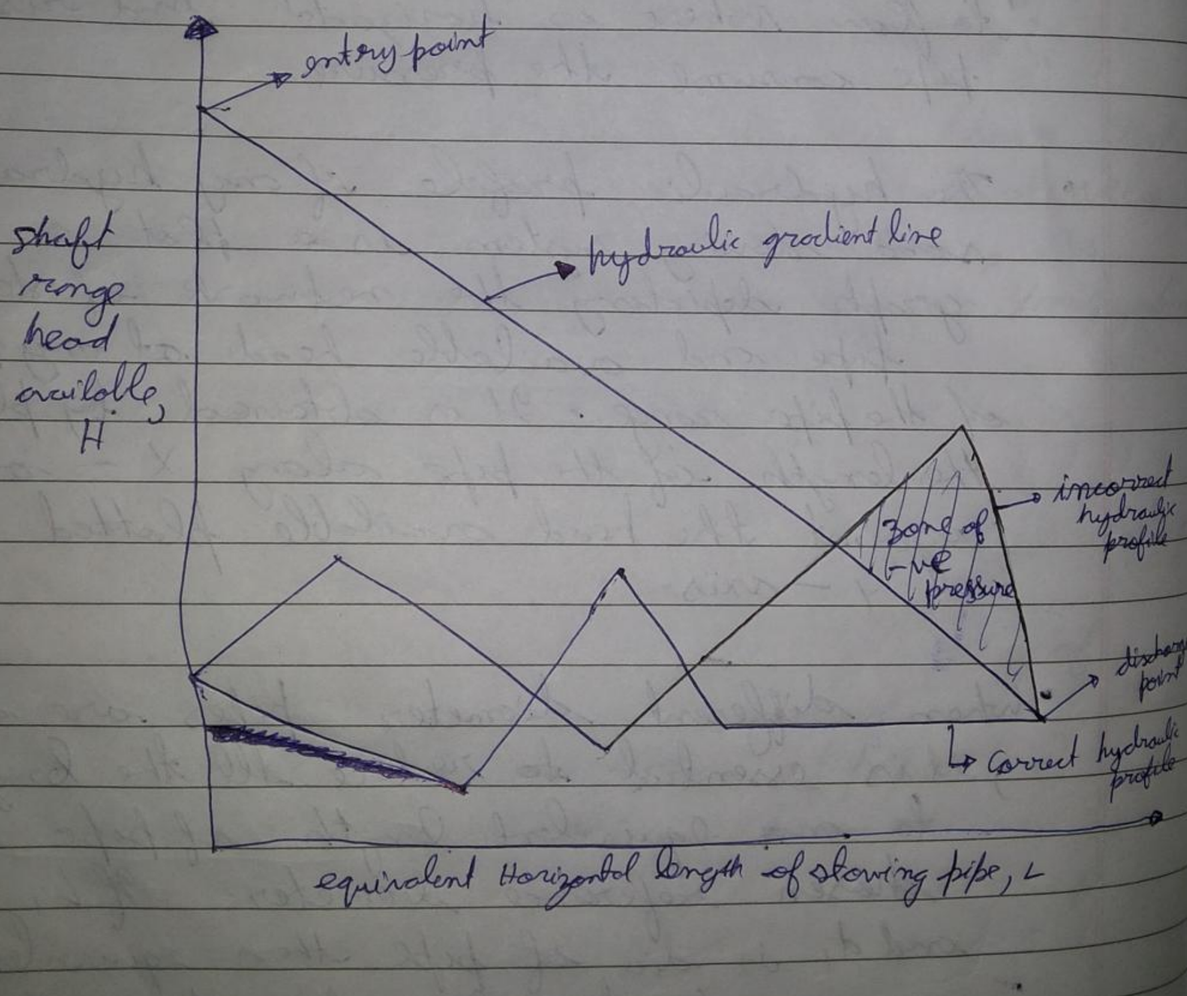
- The hydraulic profile of any hydraulic sand stowing system is a plot or graph depicting the network of stowing pipe and available head at any point of the pipe range. It is obtained by plotting the length of the pipe along X-axis against the head available plotted along Y-axis.

- When different diameter pipes are available, it is essential to reduce all the lengths to one equivalent length of pipe to a chosen reference diameter. If L_1 is length and d_1 is dia of pipe then equivalent length L_0 to a base dia d_0 is given by:

$$L_0 = L_1 \left(\frac{d_0}{d_1}\right)^5$$

- Hydraulic gradient line is the line joining entry point and discharge point of stowing mixture. Virtually, it is the ratio of available head between entry and discharge point and equivalent horizontal length of pipes.

- Distance between hydraulic profile and hydraulic gradient at any point of pipe line as measured along the Y-axis is the pressure available at that point.



Significance of hydraulic gradient and hydraulic profile → By having these plots we can modify the profile such that at no point there is any zone of -ve pressure developed as a -ve pressure ~~is~~ causes a non-feasability of the stowing operation.

- In case, ~~of~~ any point of hydraulic profile is below the corresponding point on hydraulic gradient, pressure at that point is positive

- But if hydraulic profile crosses the hydraulic gradient, the pressure at that point would be negative i.e. air will be sucked and cavitation will occur, which causes pulsatory flow of mixture. It reduces the efficiency and increases possibility of pipe jam and hence it must be avoided.

available head for stowing = pressure at the point on hydraulic gradient line - pressure at a corresponding point (vertically above or below) on the hydraulic profile

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- An incorrect hydraulic profile leads to cavitation and therefore,
  - the full available head is not utilised
  - high local velocities are caused leading to excessive wear of pipes
  - air may be sucked in, leading to pulsatory flow which might cause jamming

H:L Ratio

- It is the ratio between the total available head between the discharge and entry point of sand water mixture (i.e the vertical height) to the total length of the pipe of the same diameter on a stowing range starting from the mixing cone to the discharge end of the pipe. In other words, H:L Ratio is the Hydraulic gradient.

- If pipe of different diameter is used for different length, it has to be converted to equivalent length of a common diameter for this purpose.

- The rate of stowing theoretically depends on H:L ratio. Higher the H:L ratio, higher is the head available to overcome the frictional head loss and higher is the velocity and hence higher is the rate of stowing per hour.

- Theoritically, Q - the rate of stowing in m^3/hr is given by:-

Reasons ... with

(a

(b

(c

(d

(e

(f

(b)

$$Q = 630\sqrt{\frac{H}{L}}$$ for 150 mm diameter pipe

For any other diameter of pipe

$$Q = 630\sqrt{\left(\frac{d}{0.15}\right)^5 \frac{H}{L}}$$

- For efficient stowing without pipe jamming, H:L ratio should be higher than 1:6

Reasons for pipe jamming and measures to deal with such problems:-

(a) incorrect hydraulic profile (H:L ratio)
(b.) concentration of slurry
(c.) saturation level factor
(d.) size of the stowing material
(e.) effect of air pockets
(f.) leakage in the stowing range

(b) Concentration of slurry:-

- The frictional head loss in stowing range not only depends on length and diameter of the pipe but also on the concentration or specific gravity i.e. sand : water ratio

- For a given stowing range with particular H:L ratio, there is a maximum concentration beyond which flow will not occur.

The required head (H, m) to cause flow of mixture can be obtained by:-

$$H = \left(a + \frac{0.0018}{\sqrt{v \cdot d}}\right) \times \text{sp. gravity of mixture} \times \frac{lv^2}{2gd}$$

where :-

- a is a constant (0.02 – 0.03 for sand)
- v is velocity of slurry flow, m/s
- l, d are equivalent horizontal length and base diameter in m
- g is the acceleration due to gravity (m/s^2)

→ Though the sand : water ratio depends on head available, the ratio should be atleast 1:2 in general

(c) $\boxed{\text{Saturation level factor:-}}$

For a given stowing range, there is a certain maximum quantity of stowing mixture of a given concentration which can flow through it. If this quantity is fed through the mixing cone, the total head loss in the stowing range would be equal to the total available head and the stowing range would be full right upto the mixing cone. If greater quantity of mixture is fed, it will lead to overflow. Thus we can call stowing at this maximum level as Saturation level of stowing.