Ist Module – Introduction 26/08/2021

Stripping ratio – vol. of overburden to be removed for 1 tonne of coal or any mineral.

$$S.R = \frac{V_1}{V_2 \times \rho}$$

V_1 – OB (vol) m^3

ρ – density

$V_2 \times \rho$ – tonne (vol).

Minerals –

total – 87 (in India)

fuel – 4

metallic – 10

Non-metallic – 47

atomic – 3

minor minerals – 23 (building)

Largest mine –

1. Bingham canion mine (copper)
   - 1210 m depth
   - 7.7 km Area.
   - open-cast.
   - ~~produces~~ ~~copper~~ United States. (Utah)

(world)

2. Gevra ~~op~~ coal mines
   - 220 m depth
   - 19.08 km Area.
   - open-cast.
   - Gevra (chhattisgarh)
   - 43 cu.m shovel & 240 t dumper capacity.

(India)

S.R

Incremental – change in SR year to year ~~[illegible]~~

Overall – ratio of total vol. of waste to the total vol. of ore.

ot on OnePlus

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- Massive – huge deposits
- Bedded
- vein – stockwork
- columnar

- Stripping Index – Instantaneous S.R divided by ore of grade percentage.

(cut off Ratio)
- Economic S.R – Max value of S.R – labour cost.

- Brake even S.R – loss obtained.
  (BESR)

$$= \frac{\text{price} - \text{cost/tonne}}{\text{stripping cost / t ore}}$$

profit/tonnage

* what are the conditions in which open cast has been done preferance to U/G? Advantages + Disadvantages.

→ conditions of obtaining open-cast (pit) mine may change due to following -

1. when deposits are found near the surface.
2. It has high recovery rate
3. Takes less time to extract all material.
4. No need of ventilation system.
5. ~~NO~~ more economic than U/G
6. ~~[illegible]~~

GIS – Geographic Information system.

Advantages –

1. Full visualization of exposed orebody
2. No ore reserve is blocked.
3. Greater concentration of operations.
4. greater safety.
5. Easy draining of subsurface water.
6. No restrictions for any machinery.
7. Quick return capital investment.

Disadvantages –

1. Large area is required.
2. Generates excessive waste rocks from overburden
3. Loss of production due to extreme summer & winter rain & snow.
4. Not suitable for higher depth.

Geological conditions –

- Massive
- Bedded
- vein
- columnar

} Types of deposits in o/c.

Requirements –

- Economic

Mining cycle –

1. reconnaissance
2. prospecting – speciman + mapping
3. Exploration – Drilling, exploratory mining
4. Detailed exploration – mining

⇒ Advantages

O/C	U/G
shallow	Deep
massive	Thin
SR less	SR high
safety	Environment
Efficiency	
production	

* ~~Geological~~ Report. –

1. Reconnaissance report
2. Geological report
3. Feasibility report → Geological angle, economical angle, social feasibility angle.

   i) prefeasibility report.

4. DPR (Detailed project Report).
   - i) Reserve calculation
   - ii) Tech. feasibility.
   - iii) production rate.
   - iv) cost economics. (demand or not)

* UIG to OIC Stripping Ratio →

$$= \frac{\text{₹ UIG mining cost pert. ₹} - \text{OIC mining cost of ore pert}}{\text{₹ OIC stripping cost of waste / t of ore}}$$

It is also called Limiting SR.

* Type of OIC. –

1. Nature of deposit
2. SR
3. Desired (P) & production
4. Dilution (unnecessary things comes like gangue)
5. Recovery (Need 100%)
6. capital available
7. cost of mining
8. surface topography
9. climate
10. Avaibility of skilled labour.

Width
$W = h$ (45°) (If width = height of bench then bench face is 45°)

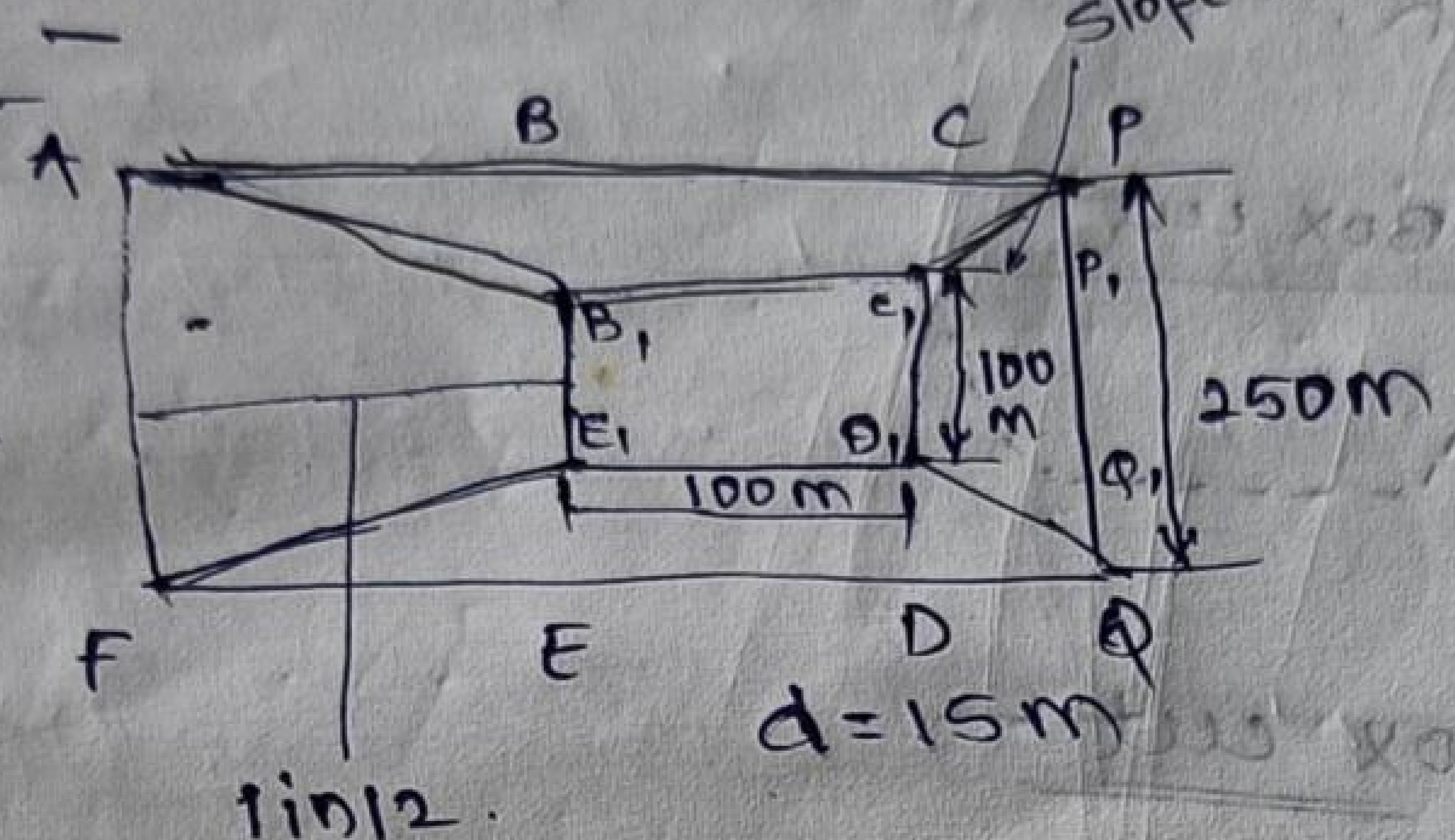
Width = 3 x dumper (width of largest m/c) ~~For transport~~.

Width = Wlm + 5m. for drilling m/c.
└largest m/c.

BOX cut —

Initial (first) extracted burden is called BOX cut.

Calculate volume.
?



~~Height of box cut (H) = 15m.~~
~~Width of box cut (W) = 100 m~~
~~inclination of floor (θ) = 1 in 12 = 4.76.~~
~~Front slope angle (α) =~~
~~side slope angle (β) =~~

$\tan^{-1}(\frac{1}{12})$

=~~[illegible]~~75.

Length = 100 m

bench height (H_b) = 15 m.

Width (m) = 100m.

Ramp slope = 4.76°.

$$Area = \frac{1}{2}(100+250) \times 15$$
$$= \frac{1}{2}(350) \times 15$$
$$= 2625\ m^2.$$

Volume = Area x Total length
= 2625 X
= m³.

Shot on OnePlus

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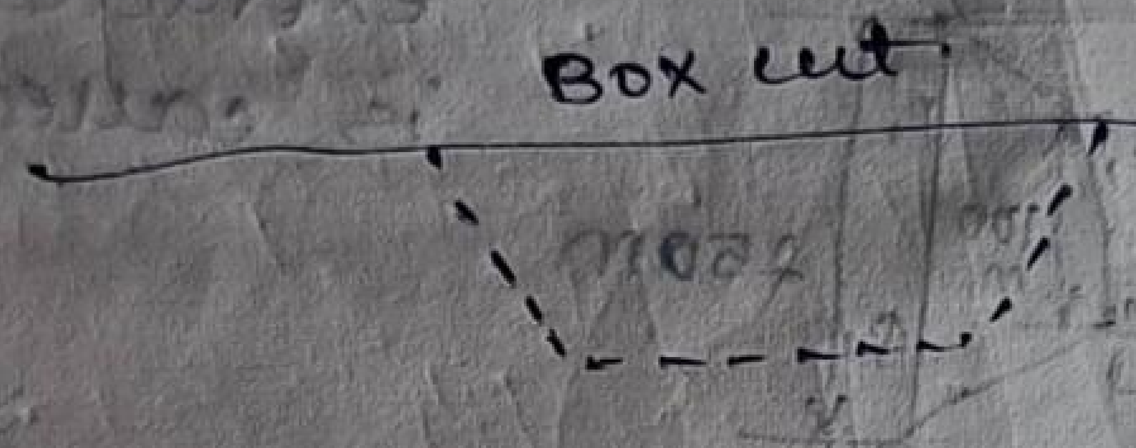
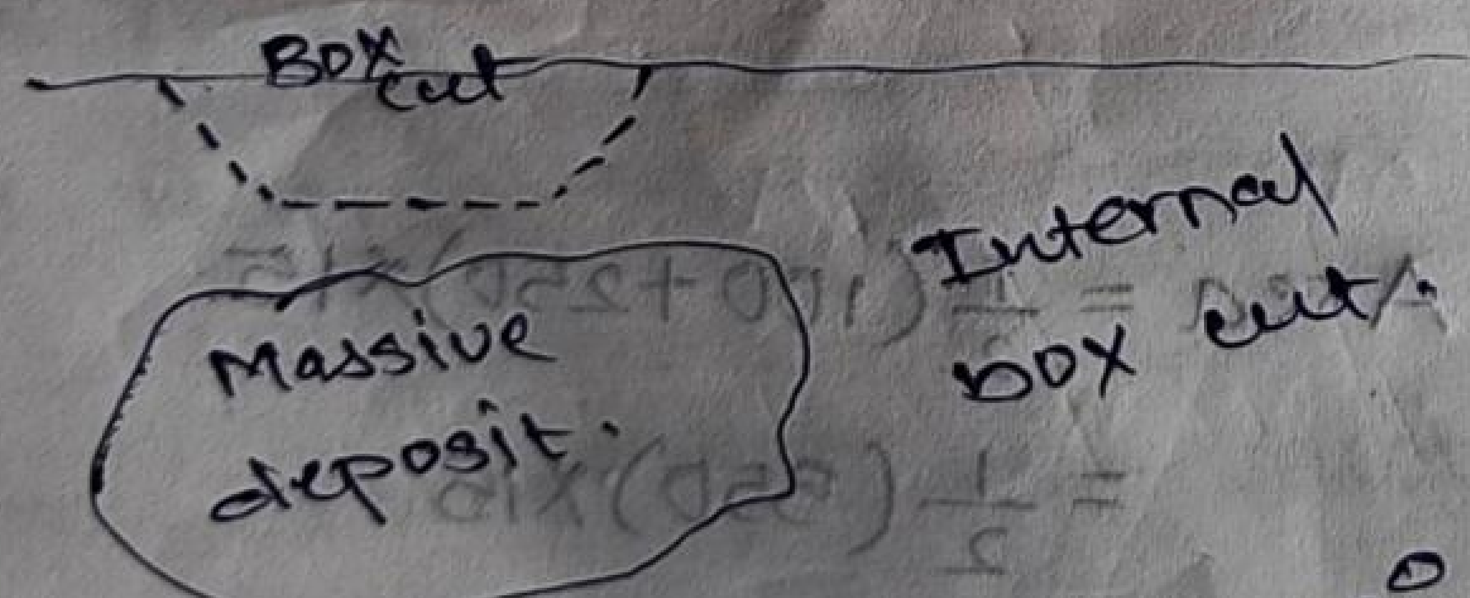
- The first excavation made to open a deposit for surface mining is called box cut.
- The purpose is to form the bench.
- Benches are req to go for horizontal slicing of rock.

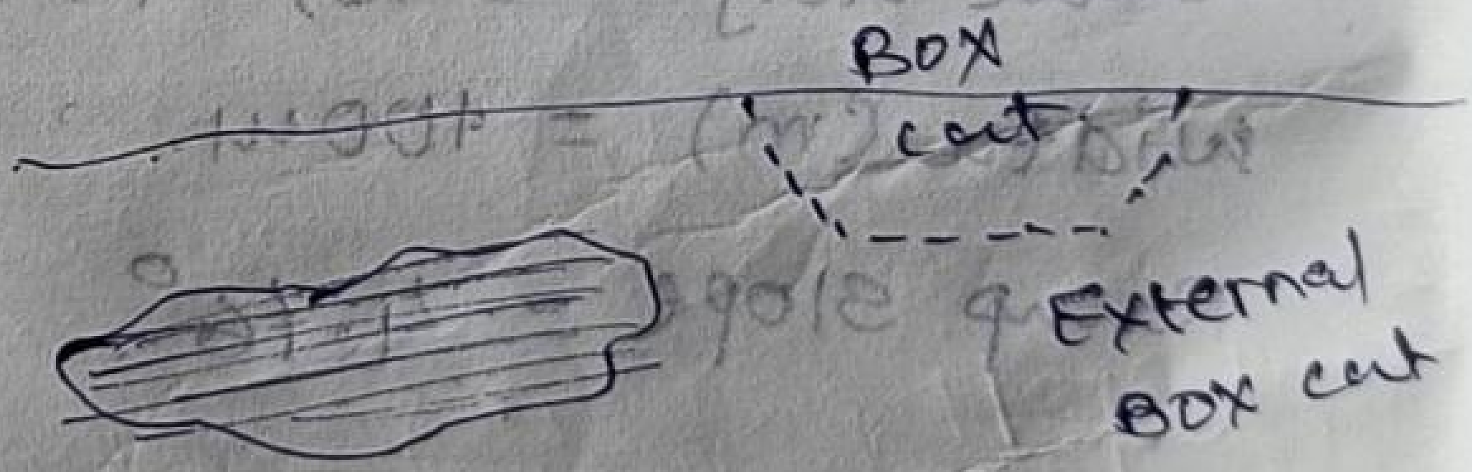


* <u>Location of box cut</u> –

<u>Internal</u> – If the box cut is placed vertically above the deposit then it is internal.

<u>External</u> – If the box cut is placed not above the deposit but within the pit limit.





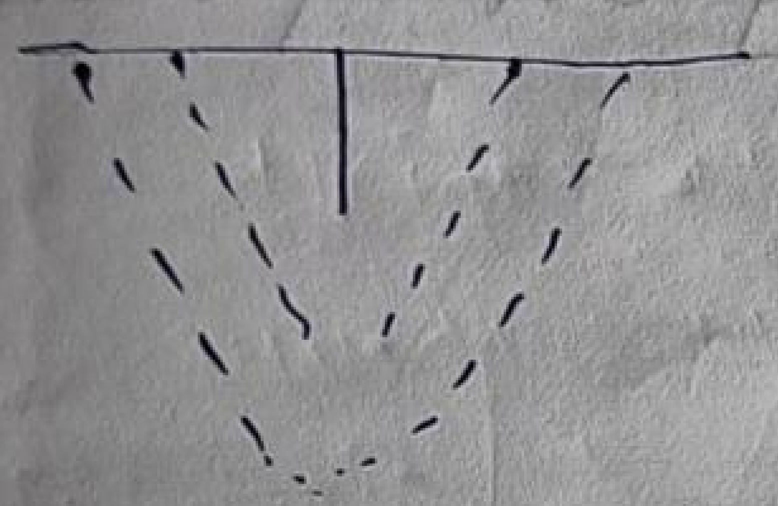
For massive deposit multiple box cut can be used.

* Selection of Box cut

i) Shape of deposit. (Massive/hill) — internal.
ii) Size of deposit (Extensive bedded) — External.
iii) Dip of the deposit (High ~~outside~~) — External.
iv) Thickness of deposit (Thick) — internal.
(thin) — external.

v) Grade of ore.
vi) Economic criteria. (Get back the ore first).

* Drilling & Blasting Pattern —



vertical at centre
Inclined at sides of centre.

Methods —
- Based on system of mining
continuous & Discontinuous.
- Based on mechanisation
Manual & Mechanised
- Based on method & nature of mining
1. open pit — reclamation after mining
2. opencast — Reclamation during mining
3. Quarries — stone, crushed rock
4. Strip mining — removing surface coal in strips
upto 50m wide × 1 km. long.
5. Placer mining.

* Applicability -

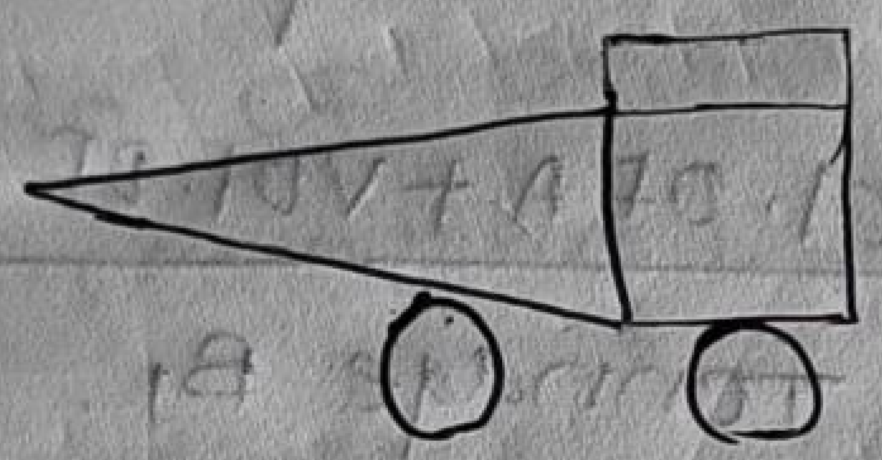
1. Thickness of coal seam > 5m
2. shallow depth - 300m
3. flat gradient - < 1 in 6 — 9.514°.
4. Strike length - > 1 km.
5. coal without stone bands.
6. Free from surface structures / Features
7. preferably non-forest land
8. Availability of adequate place for dumping
9. stripping ratio depending on quality of coal.

* Advantages —

- Roof control & ventilation is not req.
- High OMS (Output per man shift)
- High % of extraction
- Less man power
- Natural lighting
- Gestation period is less (starting ~~stage~~ stage to production stage)
- Quick returns.
- No risk (less)
- Large scale mechanisation possible.

* Mechanised methods –

- Shovel – Dumper combination
- Shovel dumper combination with Dragline
- Input crushing & conveyor technology.
- Surface minor.
- Bucket wheel excavator.
- High wall mining.



* Stripping Ratio

vol. of OB to be removed to get one tonne of coal.

$$SR = \frac{\text{Vol. of 'A'}}{\text{Tonnage of 'B'}}$$

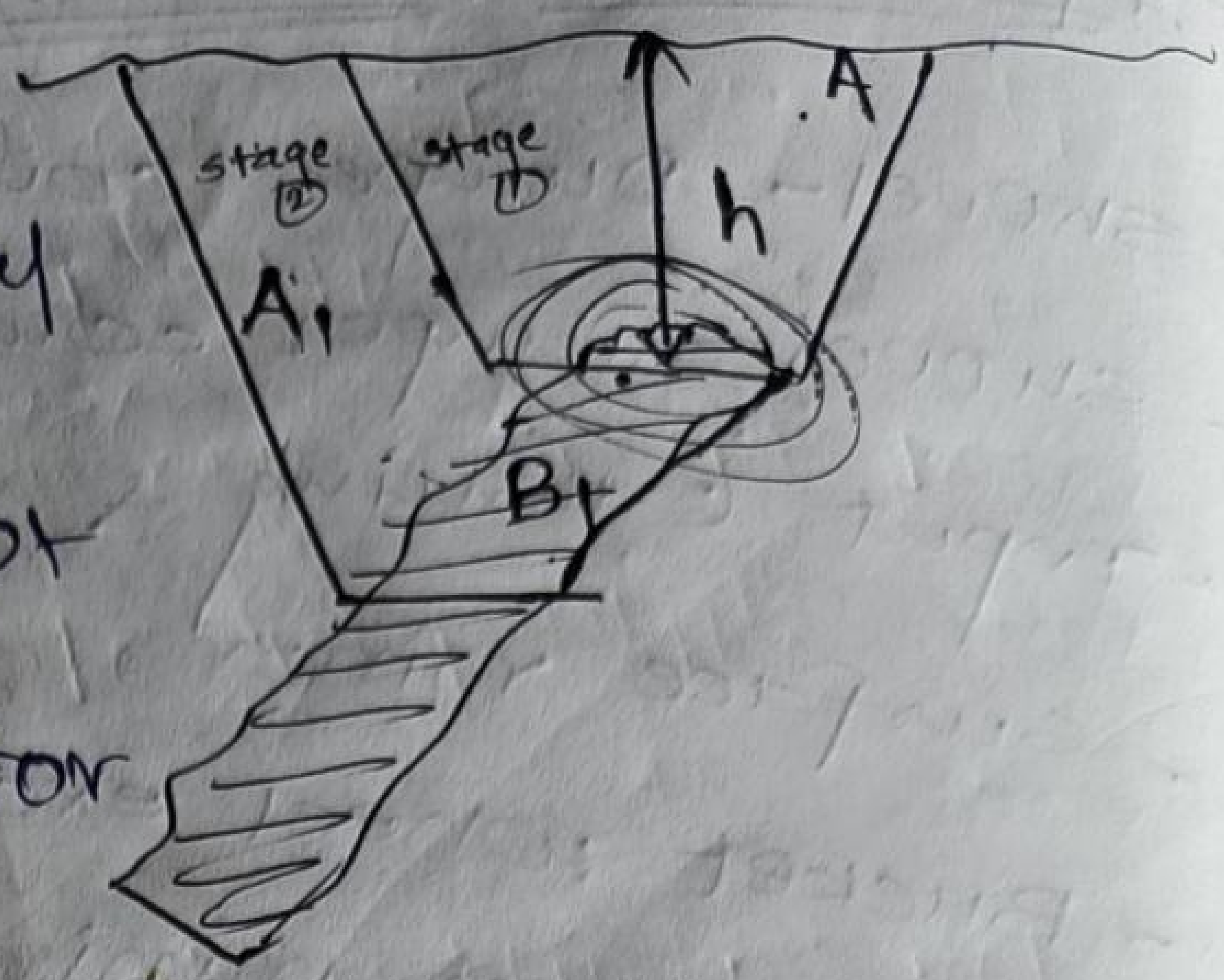
Factors affecting SR –

- cost of stripping (stripping cost).
- Extraction cost of ore/mineral.
- percentage of rejects (impurities drawn)
- cleaning cost of coal or ore dressing cost.
- sale value of clean coal/dressed ore
- Reclamation cost.
- cost of transportation.

Sub outcropped.

* Incremental SR –

* expanding operation. known as Incremental SR.



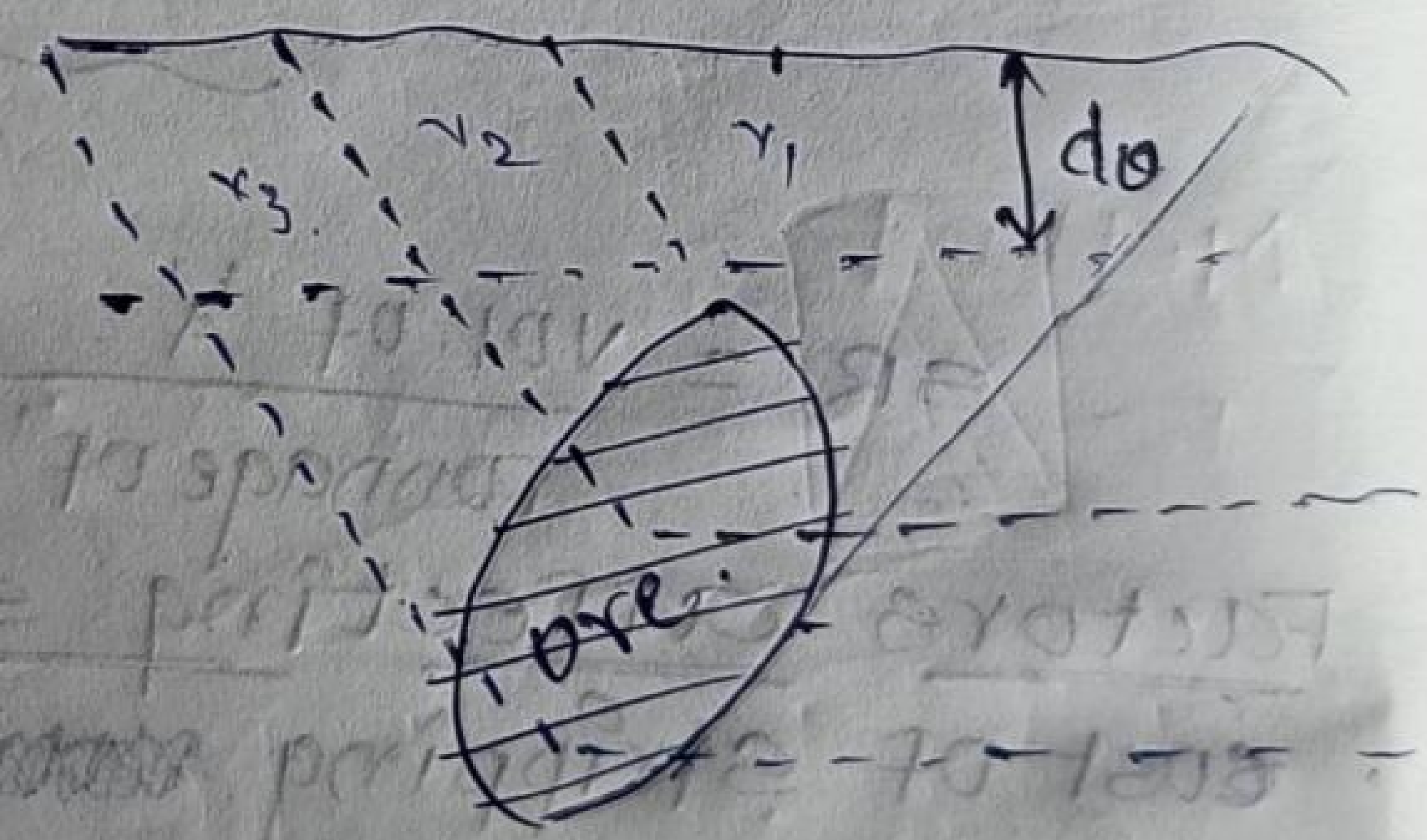
In stage 1: we are not getting ore so we will go for 2nd stage by increasing area.

$$\underline{\text{Incremental SR}} = \frac{\text{Vol. of } A_1}{\text{Tonnage } B_1}$$

$$\underline{\text{Over all SR}} = \frac{\text{Vol. of } A + \text{Vol. of } A_1}{\text{Tonnage } B_1} \quad \frac{m^3}{\text{Tonne}}$$

Incremental = Instantaneous SR.

*



$$\text{BESR} = \frac{\text{Price per tonne of ore} - \text{o/c mining cost}}{\text{stripping ratio cost per tonne}}$$

Breake even SR – The point or value or SR. beyond the which the coal (mineral) can not be economically extracted out called brake even SR.

$$BESR = \frac{\text{Price per ton of ore} - \text{O/c mining cost per tonne (overall work)}}{\text{O/c stripping cost per tonne of ore}}$$

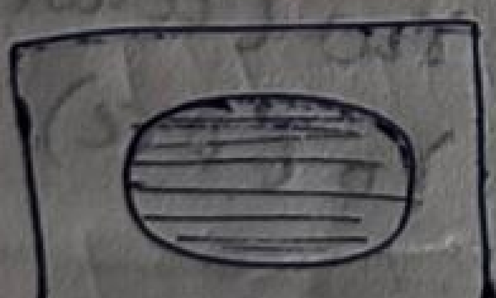
- Ratio of profit in O/c to the cost of removing OB.

$$BESR = \frac{\text{Profit in O/c mining per tonne of ore}}{\text{O/c stripping cost per tonne of ore}}$$

Decide a perticular SR for extraction of 1 tonne of ~~min~~ coal (EX – 4:1)

Quarriable limit – It is the stripping limit ~~ratio~~ upto which surface mining is economic. It is also called as economical stripping ~~ratio~~. limit.

economical stripping ratio.



~~consider~~, ~~5:1~~ m^3 Tonne.

In perticular area we have to do mining limit not more area. No profit

~~5 m³ for~~ / 1 tonne. if we extract 8 m^3 for 1 tonne then it will not economical.

NOTE –

- If instantaneous stripping ratio is equal to BESR NO profit NO-loss.

Instantaneous > BESR – loss

Instantaneous < BESR – profit.

(Shirting limit)

* ~~Limiting~~ **Limiting SR** – If ~~Instantaneous~~ Incremental SR more than limiting SR, U/G is profit.

If ~~Instantaneous~~ Incremental SR is less than limiting SR, O/C is profit.

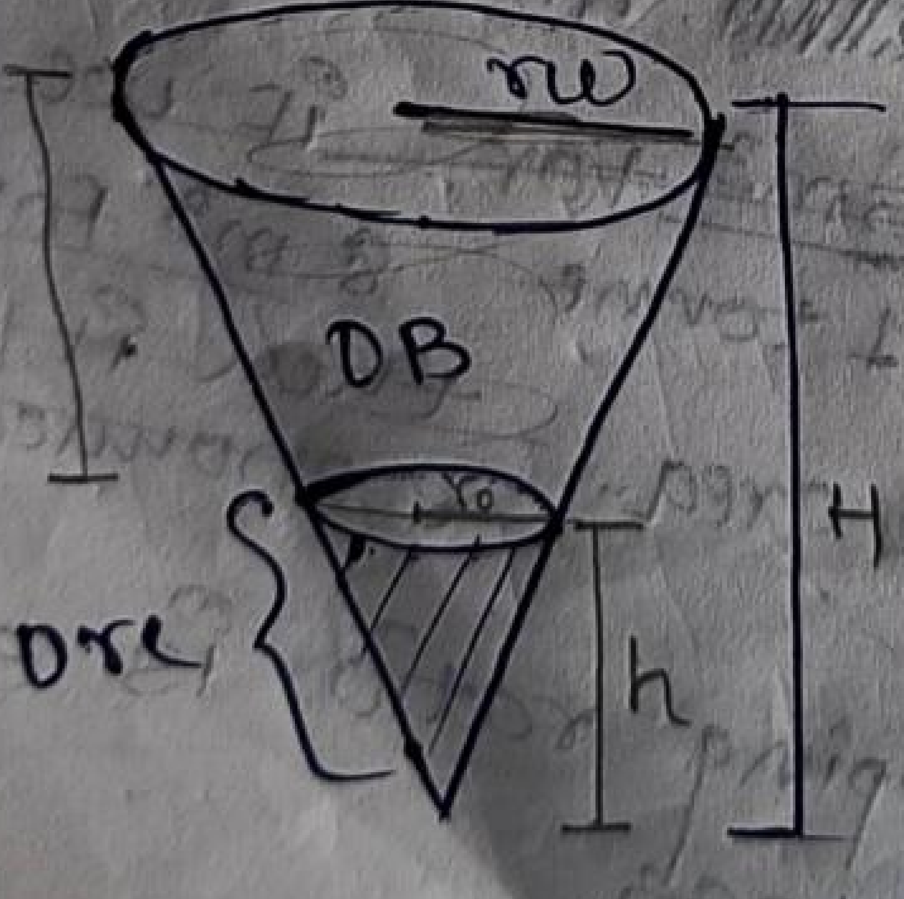
* **Avg SR** – Is defined as ratio of vol. of OB removed over a period of time to the amount of coal extracted over a period of time.
(time duration).

- BWE = 3-4 : 1 (m^3 / tonne)
  (Bucket wheel excavator)
- Shovel-dumper = 4-5 : 1 (m^3 / tonne)
- Dragline = 8-10 : 1 (m^3 / tonne)

Q. Find SR ?

$\pi r^2 h$

cone $= \frac{1}{3}\pi r^2 h$



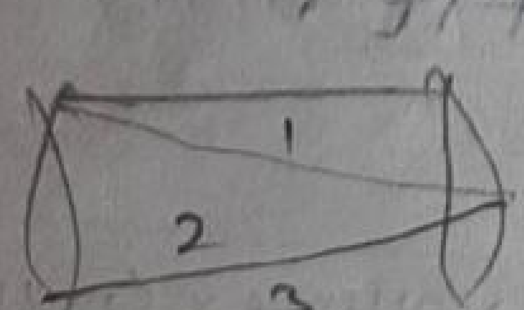
SR = ?

Radius = rw (waste)
Radius = r_o (ore)

$$\boxed{\text{Vol of cone} = \frac{1}{3}\pi r^2 h}$$

$\frac{1}{3}\pi r^2 h$.

$$= \frac{\text{Vol of DB}}{\text{coal extracted (tonne)}}$$



$\pi r^2 \times h$

$\frac{1}{3}\pi r^2 h$.

$$\boxed{\text{tonne} = \text{Sp.gr.} \times \text{volume}}$$

$$= \frac{\frac{1}{3}\pi r_w^2 \times H - \frac{1}{3}\pi (r_0)^2 \times h}{\frac{1}{3}\pi (r_0)^2 \times h}$$

$$= \frac{\cancel{\frac{1}{3}\pi} r_w^2 \times H}{\cancel{\frac{1}{3}\pi} r_0^2 \times h} - \cancel{\frac{\frac{1}{3}\pi r_0^2 \times h}{\frac{1}{3}\pi \times r_0^2 \times h}}$$

$$SR. = \frac{\cancel{\frac{1}{3}\pi} r_w^2 \times H}{\cancel{\frac{1}{3}\pi} r_0^2 \times h} - 1$$ —— In volume.

$$\frac{r_w^2 \times H}{r_0 \times h \times G} - 1 = \frac{\cancel{\frac{1}{3}\pi} r_w^2 \times H}{\cancel{\frac{1}{3}\pi} r_0^2 \times h \times G} - 1$$ —— In tonne.

Q.

Yr.	1st	2nd	3rd	4th
OB (m^3)	2023	2002	1876	1432
coal extracted	502	1321	1272	1567

find SR?

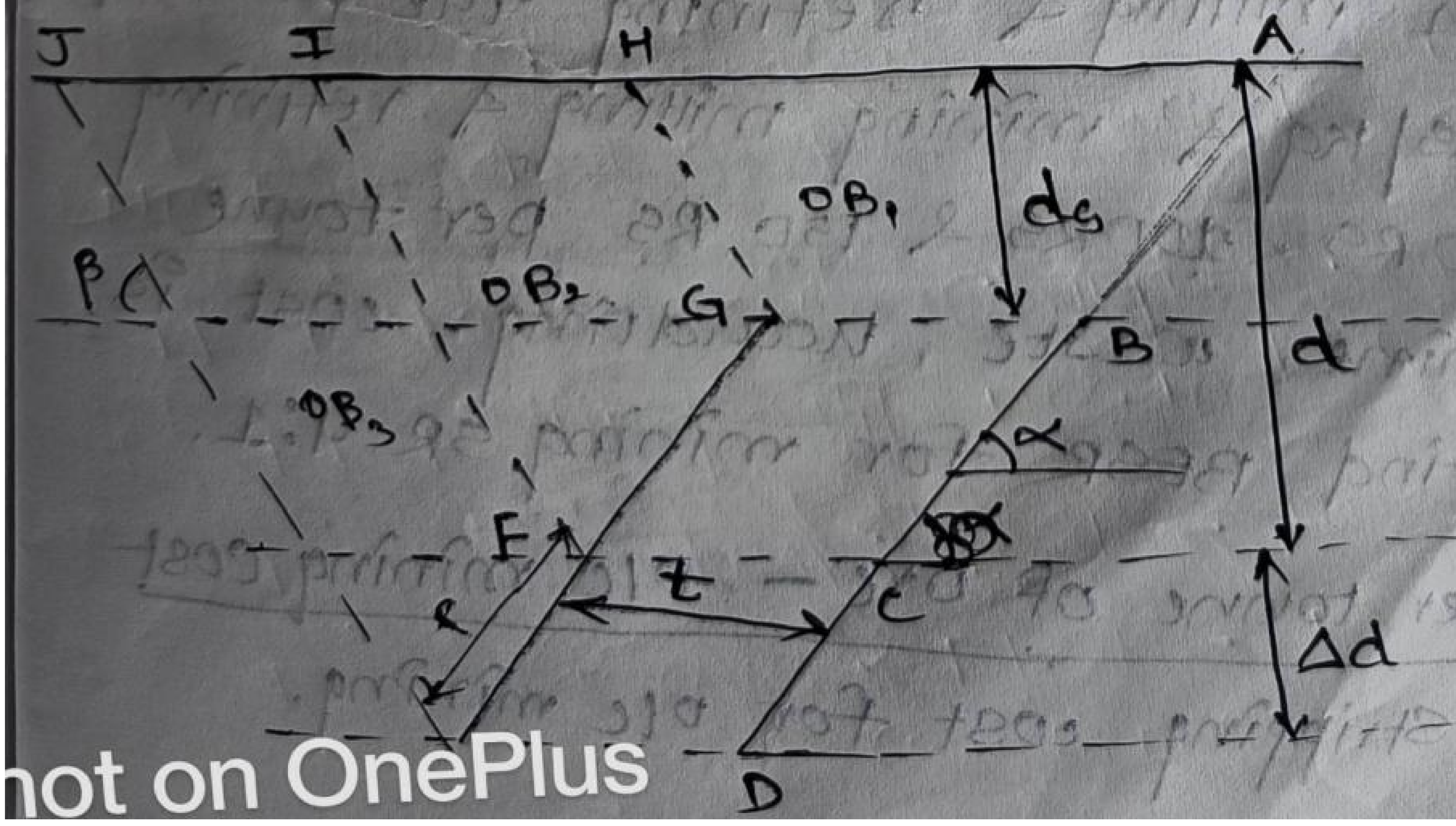
$\frac{2023}{502} + \frac{2002}{1321} + \frac{1876}{1272} + \frac{1432}{1567}$

$$S.R = \frac{2023+2002+1876+1432}{502+1321+1272+1567}$$ first this all, then this all.

$\frac{7333}{4659}$ $\frac{7333}{4662}$

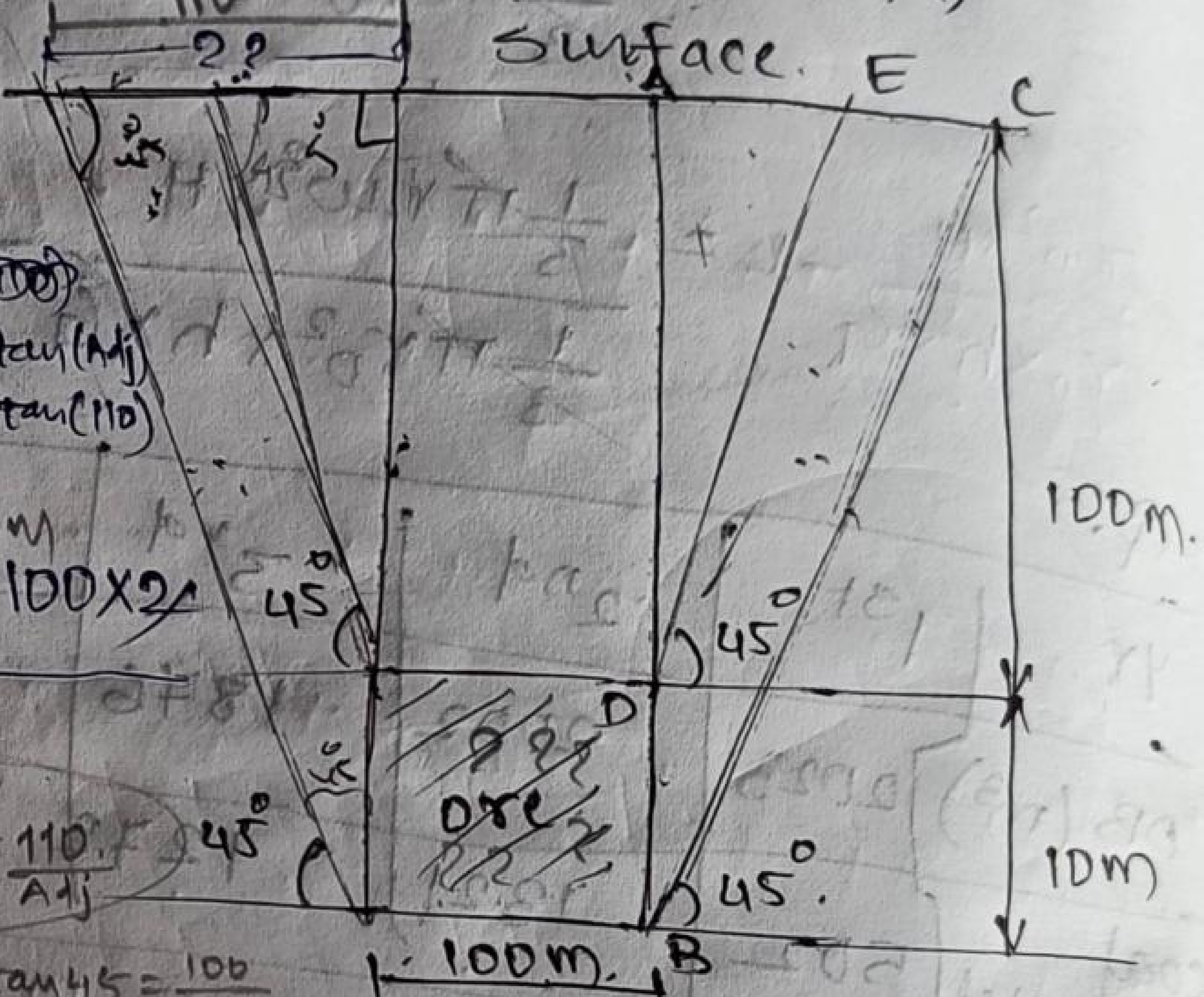
4.02+
1.51+
1.47+

SR = 1.5~~7~~ : 1   1.572 : 1



ds - sub outcroped
t - thickness
β = stability of slope
α = Dipping of OB.
P - pushing increamental

Q. A vertical ~~OB~~ overbody having a consistant width of 100 m ~~has~~ been excavated by the openpit method upto depth of 100 m at a pit slope angle of 45°. The fig shows the cross section of open pit. If the overbody is further exivated by depth of 10 m maintain same angle the ratio of waste rock to ore is how much. (density = 1.4 t/m³)



SR = ?

tan = $\frac{opp}{Adj}$

~~opp = tan(110)~~

opp = tan(Adj)

opp = tan(110)

opp =

$$\frac{\frac{1}{2} \times 110 \times 110 \times 2 - \frac{1}{2} \times 100 \times 100 \times 2}{100 \times 10} \frac{m^2}{m^2}$$

$\tan 45 = \frac{opp}{Adj} = \frac{110}{Adj}$

Adj = 110.

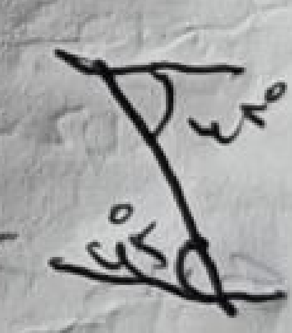
$\tan 45 = \frac{100}{Adj}$

Adj = 100.

$\frac{m^2 - m^2}{m^2 \times t/m^3}$ m^3/t

= 2.1

∴ 2.1 : 1



Q. Consider a mine working with 0.5% of copper ore with overall recovery of copper is 80%. After milling & refining the price of copper 700 Rs/kg & mining milling & refining costs are 200 RS, 100 RS & 150 RS per tonne of run of mine. Waste handling cost is 250 Rs/m³. Find BESR for mining SR 4:1.

$$\text{BESR} = \frac{\text{Price per tonne of ore} - \text{o/c mining cost}}{\text{stripping cost for o/c mining.}}$$

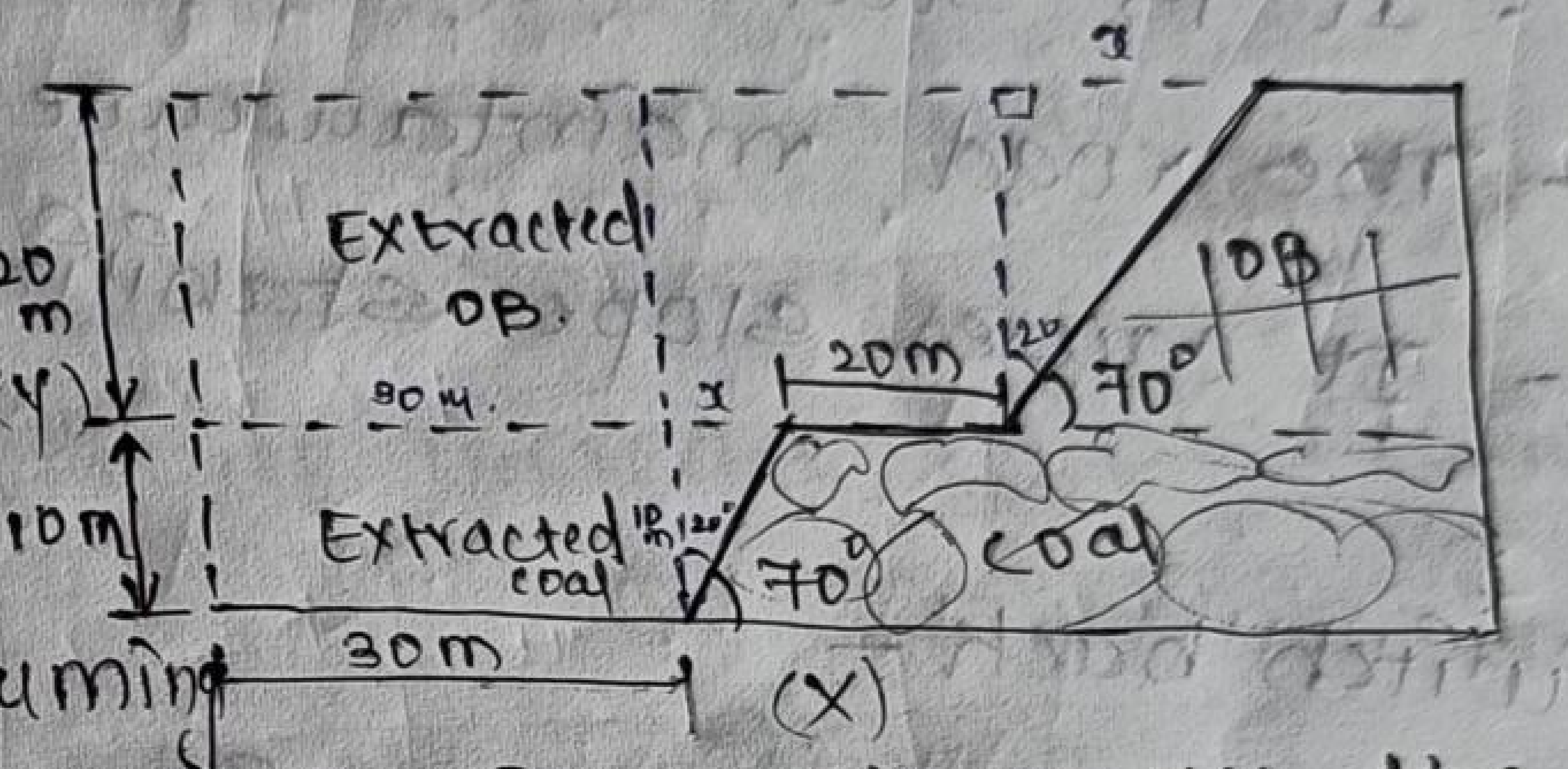
$$= \frac{700 \times \frac{0.5}{100} \times 0.8 \times \underset{\text{for kg-T.}}{1000} - 200 + 200 + 150}{250 \times 4^{\rightarrow SR.}}$$

for $1m^3$.

$$= \frac{2250}{1000}$$

$$= 2.25.$$

Q. In an opencast mine shown in fig, a coal has a density of 1.4 tonne per m^3 ($1.4 t/m^3$) assuming the mining operation started from plane xy the stripping ratio under the given conditions in m^3/t is how much?



→ $S.R = \frac{\text{Vol. of OB to be removed}}{\text{per tonne of coal}} \; m^3/t$

$\tan 20 = \frac{x}{10}$

$x = \tan 20° \times 10$

$x = 3.64.$

$\tan 20° = \frac{x}{20}$

$x = 7.28.$

$\rho_{coal} = 1.4 \; t/m^3.$

$S.R = \frac{\text{Vol. of OB}}{\text{coal extracted}}$

$$= \frac{\left((30+3.64+20)\times 20 + \frac{1}{2}\times 7.28\times 20\right)\times \overset{\text{thickness}}{\cancel{x}}(m)}{\left(30\times 10 + \frac{1}{2}\times 10\times 3.64\right)\times \cancel{x}(m)\times 1.4}$$

$\frac{m \times m \times m}{m \times m \times m \times t/m^3}$

$= \frac{m^3}{t}.$

$$= \frac{1145.6}{445.48}$$

$$= 2.57 : 1$$

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~~*~~ Haul roads —

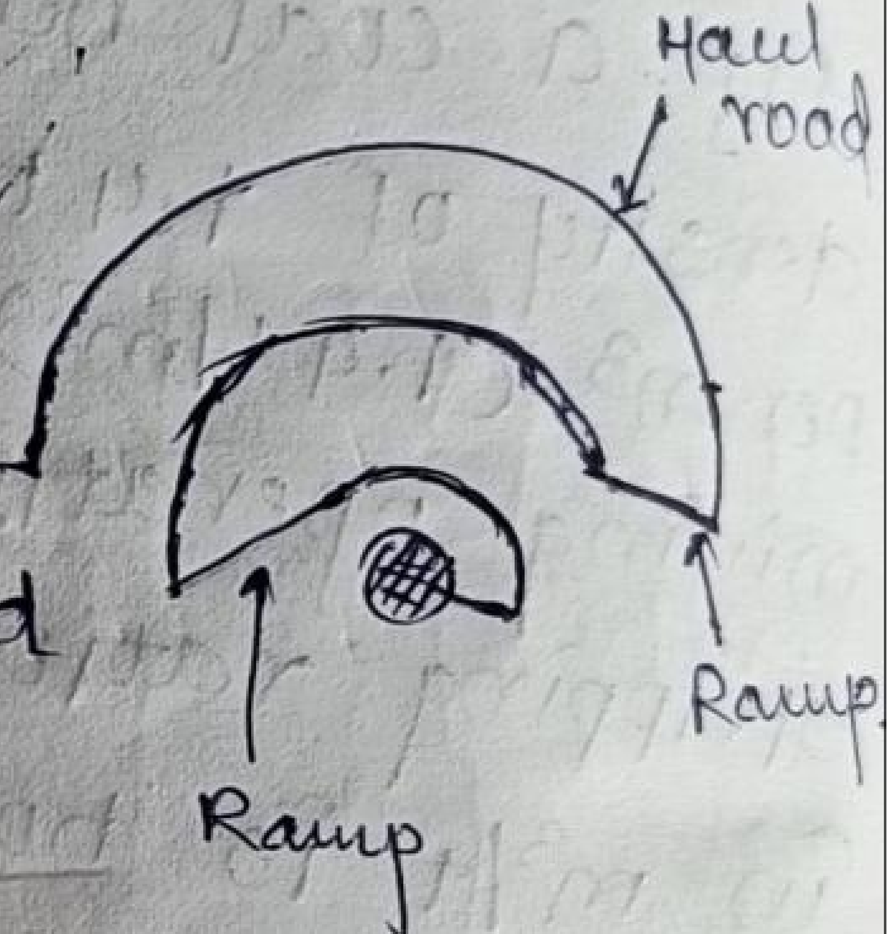
1) spiral — round round.

2) switch back — 180° angle then ramp then 180° road then ramp.

Spiral — ~~the~~ Here roadway which is made around the boundry of mine.

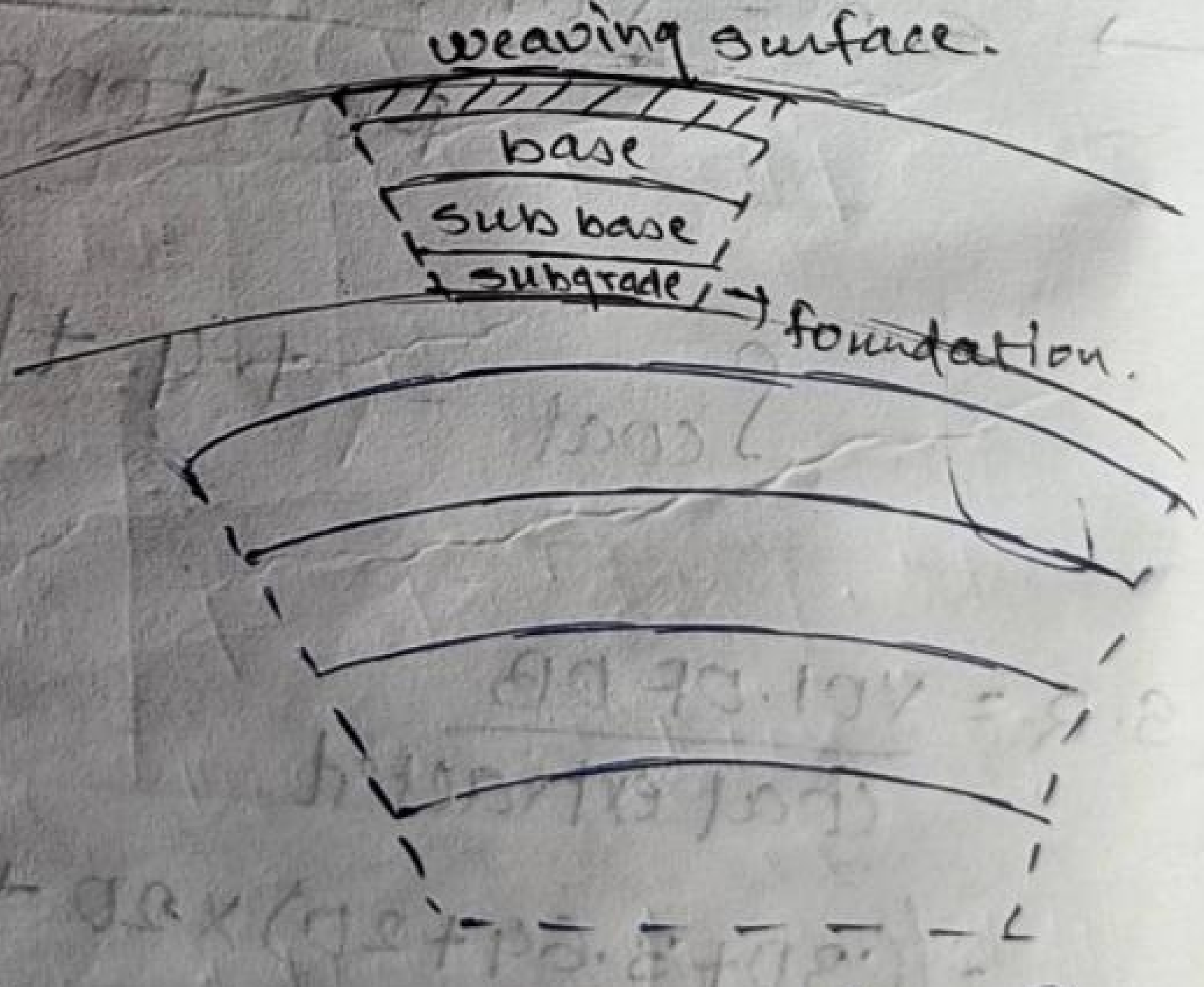
- It is suitable for deep deposite.
- The road maintanance is high.
- It causes slope stability problems.

Switch back —



- The roadway turns ~~such~~ at 180° at the end of extend.
- It is done at any one side of mine.

* Haul road design —



According to california bearing ratio.

* Advantages of good haulroad -

- easy transportation.
- Reduces accidents. / time.
- Less maintainance.
- fuel consumption less. (30% less)
- Reduces cost of maintanance of equipment
- Reduces breakdown of vehicles & provides their better availability.
- Reduction in fuel consumption.
- Acheiving better safety standards.
- Improved working environment.
- A good haul road will increase in production by 30%.
- Increase in tyre life of equipment.

*

* Characteristics of Haul road as per mine statute (Rules). $\boxed{3\times HEMM + 5m}$

- The width of haulroad -

NO haulroad shall be of width of less than 3 times of the width of largest HEMM playing on the road + 5m.

For two way haul roadway

width $\geq$ (3x width largest plying HEMM) + 5m.

For single way traffic - 2xHEMM+3m.

width $\geq$ (2xlargest plying HEMM) + 3m.

Ex - width of 35 t dumper - 4.4 m
85 t dumper - 5.8 m
100 t dumper - 6.6 m.

- Gradient of Haul road -
  At any place on the haul road the gradients should not be more than 1 in 16.
  In case of small ramps over small structures a gradient upto 1 in 10 may be permitted.

- Visibility of Haul road -
  ~~All~~ corners & bends in roads shall have clear view of distance not less than 3 times the breaking distance of largest HEMM working at 40kmph.
  3 X breaking distance (40kmph)
  3 X 70
  210 m.

  Distance of visibility = 3 X (Breaking distance of HEMM at 40kmph)

- Lighting of Haul road -
  Haul road required standard of elumination shall be 0.5 - 3 Lux.
  ~~1 luman = 1 lux~~
  1 luman/m^2 = 1 lux.

The amount of elumination provided when one luman is evenly distributed over an area of 1 m^2

- Traffic signs should be provided in the haul roads (where required) like speed, bends, curves, etc.

—— xox ——

* Super elevation -

The amount by which outer edge of roadway to be raised is known as super elevation

In opencast $h = \frac{v^2}{rg} \times B$ — width of roadway

speed in m/s — v^2; r — Radius of curve; g — gravity

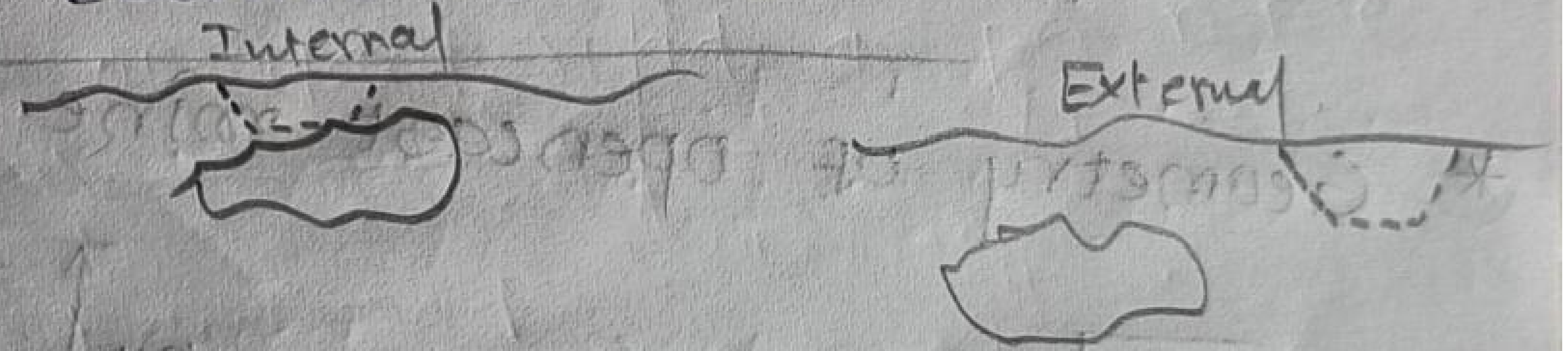
or $\boxed{\frac{v^2}{rg}}$ below / Above

generally $h = \frac{v^2}{rg}$

$h = \frac{v^2}{rg} \times B$ $\frac{v^2}{rg}$ — v^2 → speed in m/s, g → sp gr, r → radius of road,

* Box cut -

Opening up of open pit (or) open cast is done by an opening cut for the development of first working bench. The opening cut is called box cut.



1. Internal
2. External

(selection)
* Location of site for box cut -

1. Minimum cost of haulage within the openpit.
2. If shifting of box cut is necessary it is to be located at the boundry of deposit.
3. In deeping deposite, the location of boxcut should be in the middle of boundry of reserve for minimising haulage cost. or the location of the boxcut should at mineral outcrop or where ratio of OB is to ore is least.

4. In horizontal deposit, box cut should be located in middle of any boundry. depending on location.

5. The ~~slope~~ site of box cut should be free from geological disturbances & should be stable.

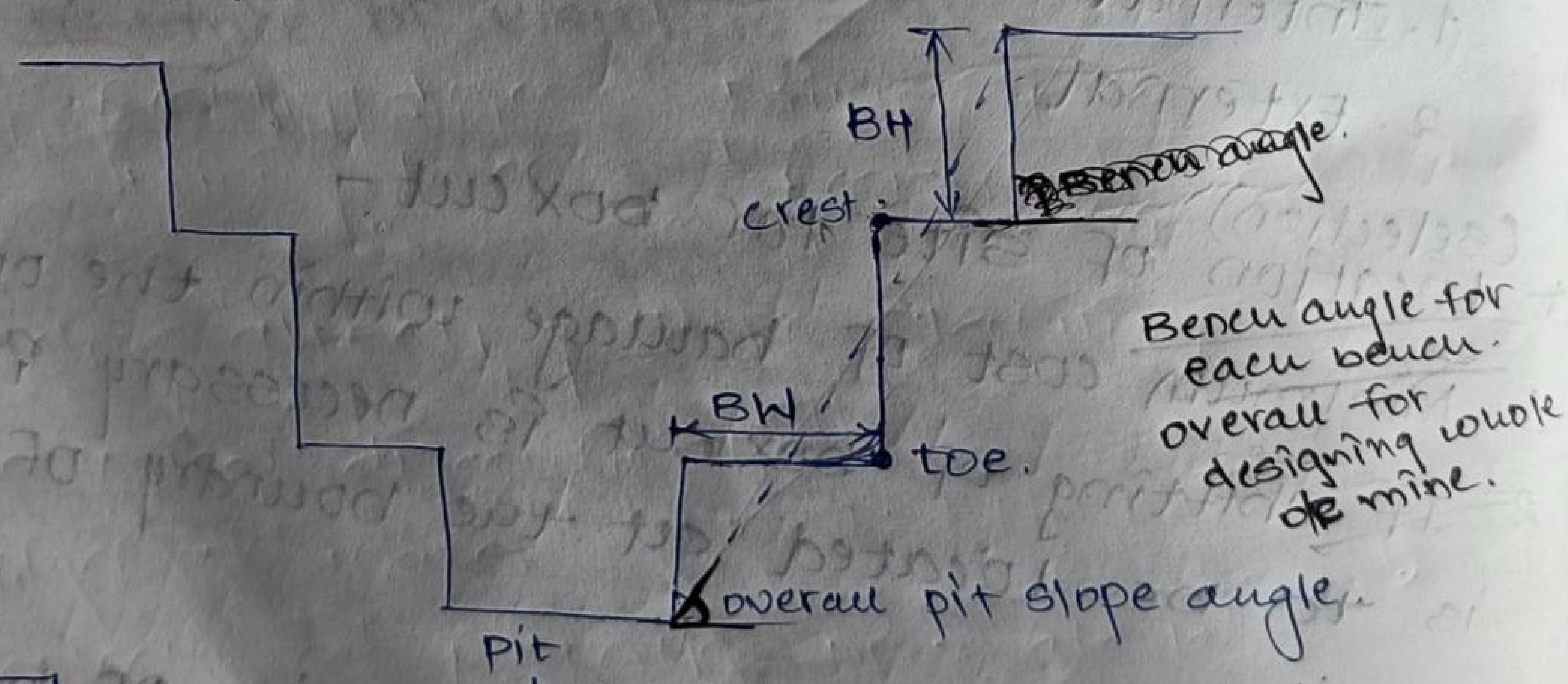
6. The site should be located at rise side to guard against floods.

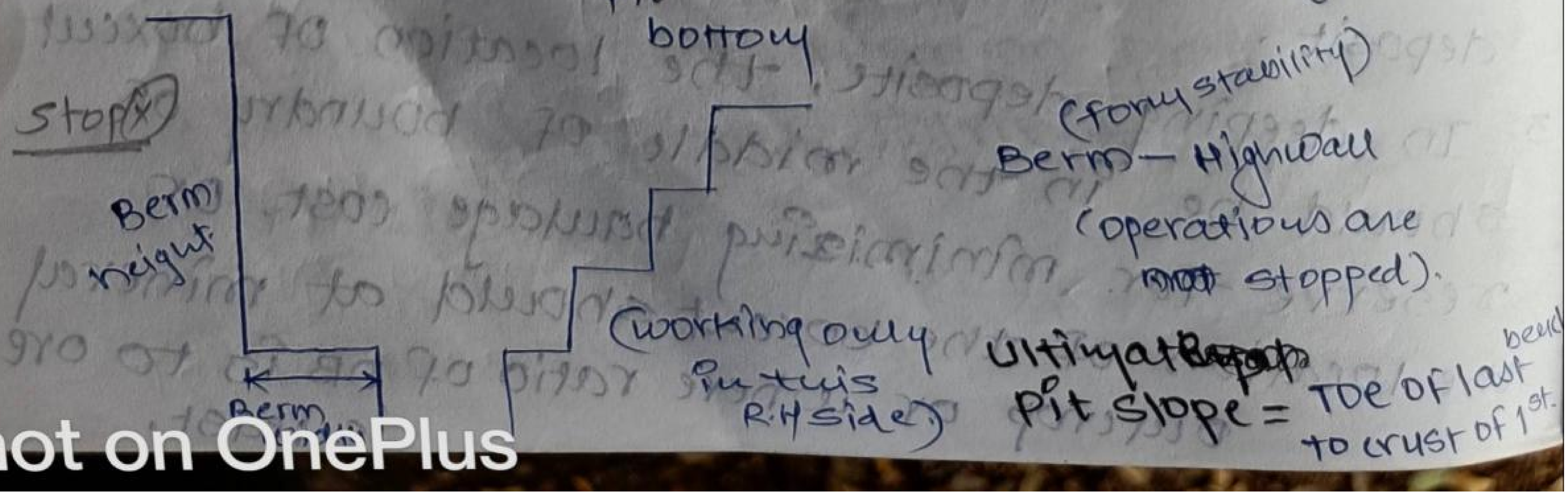
7. Site should be selected where construction of approach road is very convenient.

8. Site should be preferably, where deposit has high grade mineral to compensate the development cost.

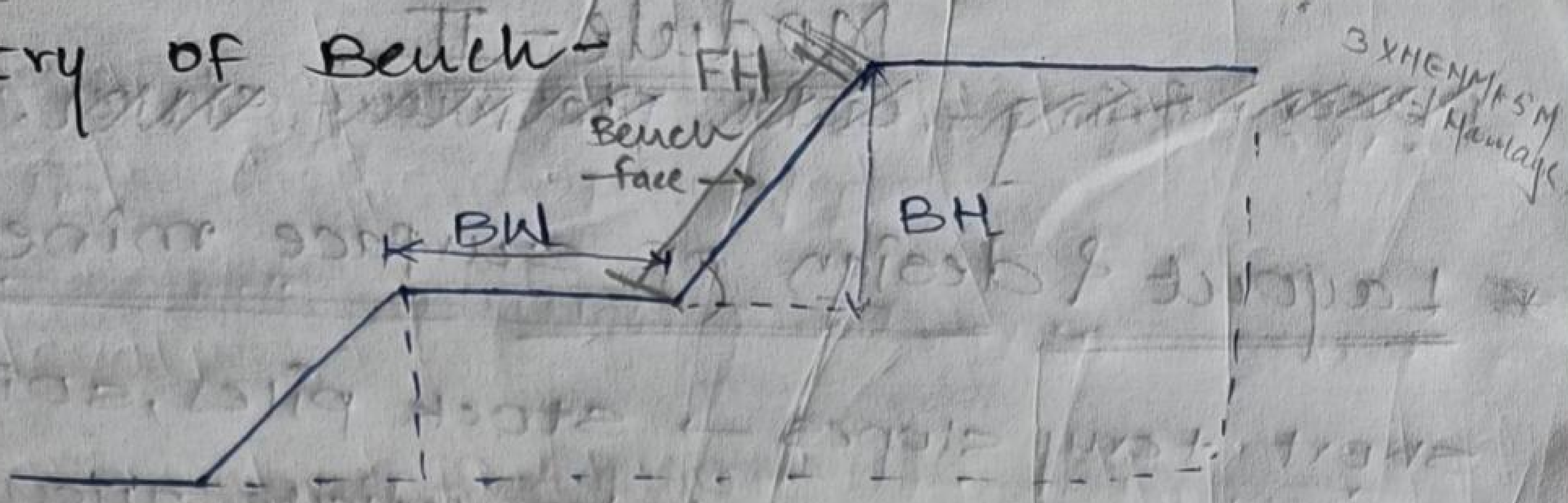
9. Location of box cut should be such that it serves the purpose of max mining area.

* Geometry of opencast mine –





* Geometry of Bench-



Bench height - It is the distance between toe of 2nd bench, to crest of 1st bench.

Larger bench height reduces transportation cost but increases slope stability problems.
(slope stability decreases)

~~General~~ Parameter affecting BH -

1. Type of rock (losse rock), height ↓
2. ~~Heat~~ wetness of rock (Ex-soil), height ↓
3. Thickness of ~~Rock~~ deposit.
4. machinery available (equipment)
5. Bench width   BODM
6. Bench angle.

* Elements of surface mine planning-

Hydraulicking:-

Nozzle → sludge ⟶ Drudge pump

- 10-12 kgf/cm²

Placer mining - The minerals which are formed by weathering.

Gold, platinum, silver.

—xox—

Module - II

~~Planning - Surface mining~~

* Layout & design of surface mine -

Short term slopes → stock piles, soil mounds, intermediate quarry tacks.

Medium ~~term~~ slopes - soil mounds, querry tailes screening

Long term slopes - quarry faces, soil mounds.

* Mechanism of pit slope failure -

- (Mechanism of slope failure - when driving force exceeds the resisting force.)

~~resisting force (upside)~~
~~driving force (downside).~~

- (factor of safety, other ratio of resisting force to driving force if $FS \leq 1$ the slope will fail.
  if $FS \geq 1$ the slope will be stable.)

$$\frac{\text{Resisting force}}{\text{Driving force}} \quad \frac{\text{Driving force}}{\text{Resisting force}}.$$

* Factors affecting slope stability -

- Geological discontinuties
- Effect of water
- Geotechnical properties of material
- mining methods
- State of stress
- Geometry slope.
- Temperature.

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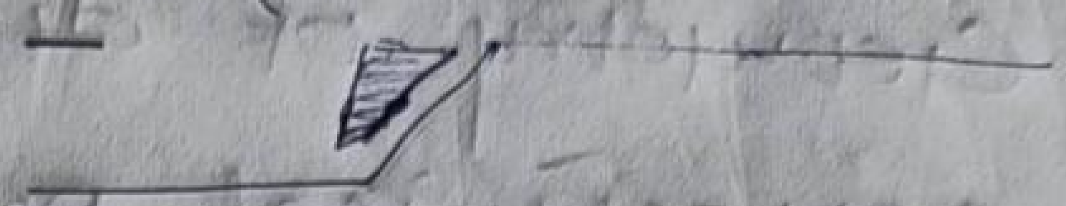
- Erosion
- Sesimic effect
- vegetation
- Dynamic blasting.

*Modes of slope failure - (pdf)



- plane failure
- wedge failure

- circular failure (Rotational)
- Toppling failure

- favorable conditions of plane failure -
* The dip of the planer discontinuety must be less than the dip of the slope.
* The dip of the planer discontinuety must be greater than angle of friction of surface

* Wedge failure -
  occurs in rock with two or more sets of discontinuities whose lines of intersection are approximately $\perp^{er}$ to the strike of slope & dip towards the plane of the slope.

  wedge failure of rock slope results when the rock mass slides along two intersecting discontinuties both of which dip out of the slope at an oblique angle to the face, thus forming wedge.

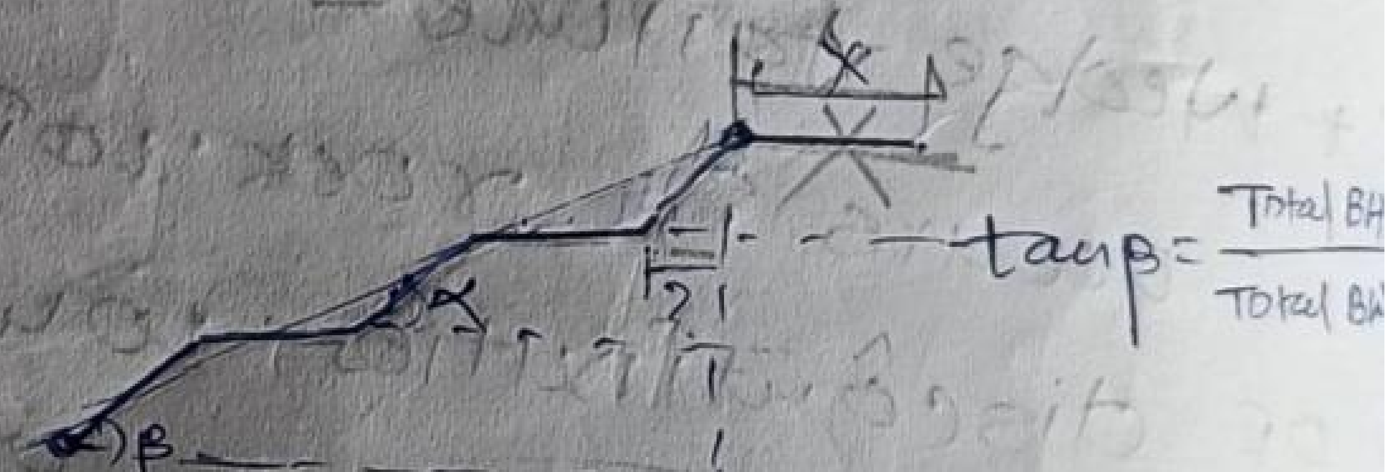
* circular failure – weak rocks (or) soil slopes

* factor affecting Dump slope stability –

- Geometry of dump
- Geotechnical properties of dump material
- Geotechnical properties of foundation (base)
- method of dumping.
- Hydrological conditions.
- static & dynamic forces (blasting, etc).

* Modes of dump failure –

- surface or edge slide
- shallow flow slides
- Rotational circular
- Block translation
- Base failure
- Liquefaction



* Ultimate pit slope – Toe of last bench to crest of 1st bench.

$\tan\beta = \frac{NB \times BH}{(NB-1)BW + RW + \frac{NB \times BH}{\tan\alpha}}$

$\tan^{-1}$

$$\beta = \tan^{-1}\left(\frac{NB \times BH}{(NB-1)BW + RW + \frac{NB.BH}{\tan\alpha}}\right)$$

for equal bench height & equal width.

β = ultimate pit slope angle.
α = bench angle.

If Ramp width (RW) not given then take '0'.

$$\beta = \tan^{-1}\left(\frac{\Sigma BH}{\Sigma BW + \Sigma RW + \frac{\Sigma BH}{\tan\alpha}}\right)$$

Numericals –

Q. A mine has 12 benches, bench height of 12 m, bench width of 15 m, bench angle 55°. Calculate ultimate pit slope angle.

→ $$\beta = \tan^{-1}\left(\frac{N_B \times BH}{\underset{\text{Dim.}}{(N_B - 1)}BW + \underset{\text{ramp width}}{RW} + \frac{N_B \cdot BH}{\tan\alpha}}\right)$$

$$= \tan^{-1}\left(\frac{12 \times 12}{\underset{165}{(12-1)15} + 0 + \underset{101.40}{\frac{12 \times 12}{\tan 55^\circ}}}\right)$$

$$= \tan^{-1}\left(\frac{144}{165 + 100.82}\right)$$

$$= \cancel{\tan^{-1}(2.29)}$$

$$= \tan^{-1}\left(\frac{144}{265.82}\right)$$

degree $= \tan^{-1}(0.541)$

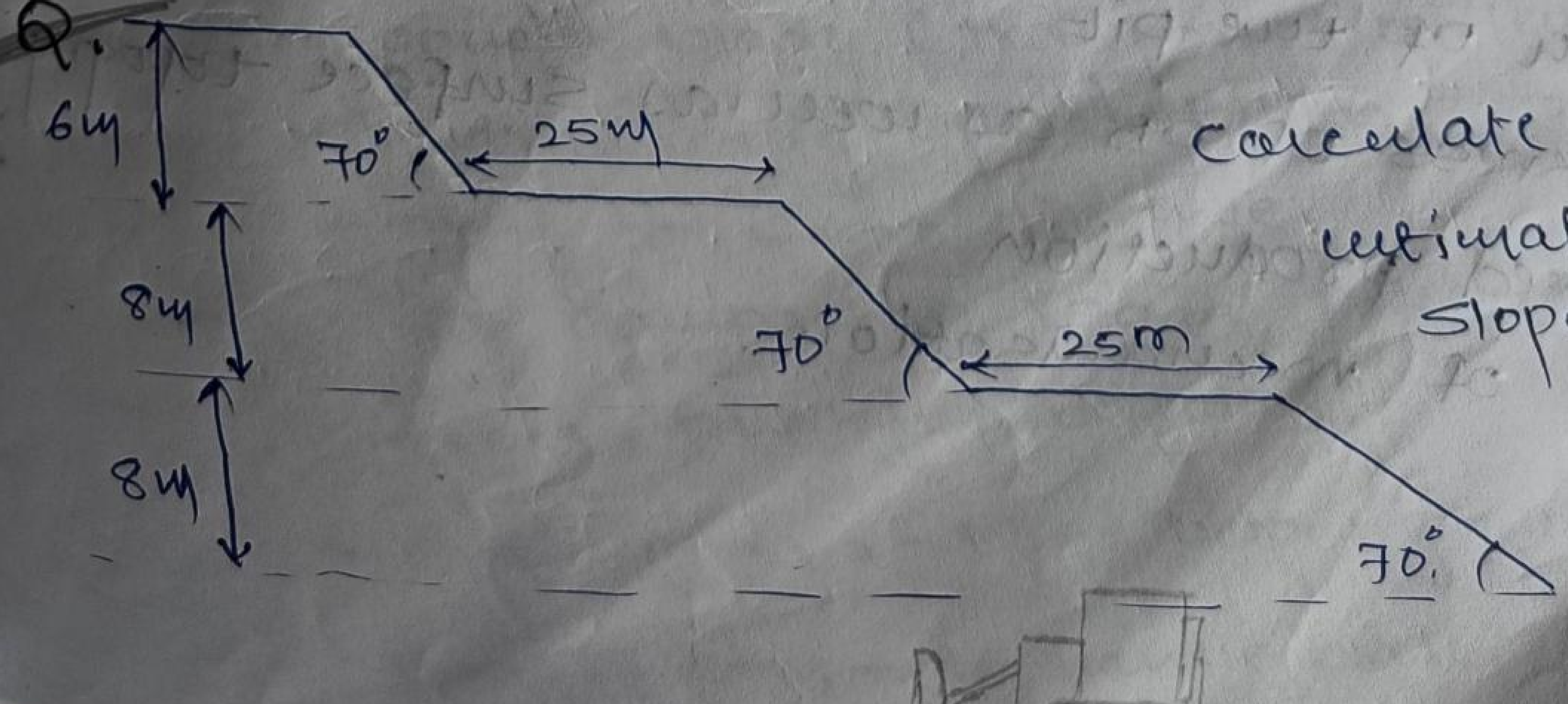
$= 28.41.$

$\frac{144}{266.4} = 0.540$

$\frac{144}{165.42}$ $\frac{144}{101.42}$ 101.90

$\frac{144}{265.84} = 0.5416.$

Q.



Calculate ultimate pit slope angle

$$\rightarrow \beta = \tan^{-1}\left(\frac{\Sigma BH}{\Sigma BH + \Sigma BH + \frac{\Sigma BH}{\tan\alpha}}\right) \quad 8.07$$

2.7474

$$= \tan^{-1}\left(\frac{22}{50 + 0 + \frac{22}{2.74}}\right)$$

$$= \tan^{-1}\left(\frac{22}{50 + 8.02}\right)$$

$$= \tan^{-1}\left(\frac{22}{58.02}\right)$$

$$= \tan^{-1}(0.37)$$

$$\beta = 20~~76~~$$

* Open-pit layout *

***Deposits —**

1. Massive huge body
2. vein incline deposit external dump
3. Bedded . Backfilling. less in thickness but length is more.
4. columnar : external dump same as ~~Vein~~ vein but not more incline

***Layout of an open pit.**

- shape
- size (extent, thickness)
- Dip of deposit
- Depth of the pit.
- Thickness of OB as well as surface topography
- Desired production
- Type of mechanisation.

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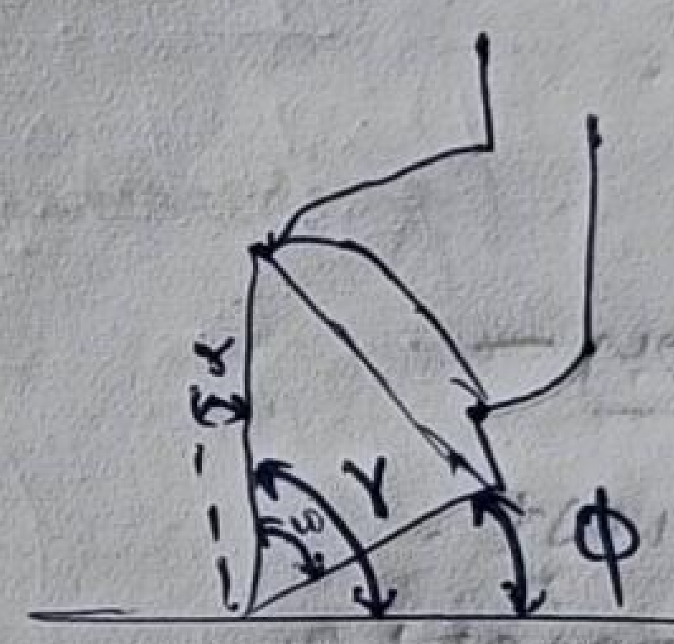
* Mechanical excavation system - Working pit slope. OB.

Blade — Dozer, motor grader, scraper.
Teeth — shovel, dragline, BWE, BCE.
Pick — shearer, CM, road header, CSM (surface).
Disc cutter - TBM, dimensional stone cutting m/c.

Button - TBM

Auger tool - CAM, highwall miner.
(Not only in O/C).

Stages -
1. Preparation of site.
2. Drilling
3. Blasting
4. Excavation.



Dozer -

ϕ - clearance angle
γ - cutting angle.
ω - tip edge angle
r - rake angle. (with vertical axis)

Site preparation - levelling of site.
(Dozer, ripper)

Dozer - crawler (or) tier mounted.
- consist blade. (transported or levelling purpose).
- hydraulically operated blade.
- only for short distance.
- ~~for~~ compacting roads, pulling down trees, pushing boulders,
- dig upto 1.2 - 1.5m below ground

Pushing arm - to handle blade.
hydraulic cy - to operate/tilt blade.

Weight - A tractor can't push more than it weighs., limited by coefficient of traction

major component -
1. undercarriage unit - sprocket.
chain
pocket (pads)

classification of dozer -
1. propel mechanism
2. blade type → straight (S) straight.
→ Universal (U) curved & more weight
→ combination short, less curved small wings.
3. power

Application -
- Road const.
- site preparation.
- face preparation
- Trunk uprooting
- Grading (Levelling)
- Ramp preparation.
- stock pile dressing
- Ground filling
- pushing scrapers, shiftable skid mounted structure
- Demolishing & clearing job.

ditching - like canal type.

selection –

optimizing by using right blade.

- fuel savings.
- higher savings.
- less wear-tear on the tractor.
- better finished product.

Blade is operating tool & needs proper selection considering.

- soil characteristics
- moisture content.
- compaction

Angle dozer – 25°.
- for side casting (unloading)

Ripper –

- 0.6–1m deep furrows in the ground.
- Also compared with farmers plough.
- crawler mounted, heavy duty diesel tractor, with ripper attachment.
- travels along close paths 1.2–1.5 m apart.
- soft, medium, hardness below 5 are suitable for ripping
- If DB is suitable for ripping then D&B can be dispensed.
- Relative ~~repipp~~ reppability can be known as.

1. straight shank
2. curved shank
3.

Types -

1. single
2. double.

* Extraction of developed pillars by opencast method-

safe - parting is ~~other~~more than 15m.

critical - Blasting carried out safely, Back hoe shovel excavation, zone is demarcated with 'yellow zone'. (6-15m).

Danger - area where parting over UG galleries is 4m hard OB or less. No work shall be done in this zone except dozing & drilling for compaction. (Red flags)

compacted - When danger zone is converted to safe zone by compacting UG galleries by drilling & blasting. This zone is permitted for normal extraction of coal.

Multiple seam partition min- upto 6m.

1. safe
2. critical
3. Danger
4. compaction

zon

IV th Module -

* Universal rules -

1. production rate = $\frac{\text{capacity} \times \text{no. of cycles}}{\text{unit time}}$

$= C \times \frac{\text{No. of cycles}}{\text{unit time}}$

2. productivity = production rate x Efficiency.

3. Bucket fill factor = $\frac{\text{fill ability}}{\cancel{\text{swell factor}}\ \text{vol. of bucket}}$

4. swell factor (S) = $\frac{\text{Loose (fragments)}}{\text{Bank (in-situ rock) (before blasting rock)}}$ (when more than 1) (>1) (x)

(or)

$= \frac{\text{Bank}}{\text{loose}}$ (<1) (÷)

5. Availability - $\frac{\text{up time}}{\text{up time + down time}}$

production rate of shovel -

- $Q = Bc \times Bf \times C \times S \times A$
(m^3/time)

$Q = Bc \times Bf \times C \times S \times A$

Q = production rate
Bc = bucket capacity (m^3)
Bf = bucket fill factor.
C = no. of cycle per unit time
S = swell factor
A = Availability

Q. A shovel has bucket capacity of $2.5\,m^3$ having cycle time of 55 sec. If fill factor is 0.8, swell factor 0.75 & Availability of shovel is 69%. Find production of shovel in 8 hours of operation.

→ $B_c = 2.5\,m^3$

$C = 55$ sec. 0.015.

$B_f = 0.8$

$S = 0.75$

$A = 69\% = \frac{69}{100}$

$\frac{8 \times 3600}{55} = 523.63$

hr→sec $\frac{8 \times 3600}{55} = 523.63$

$2.5 \times 0.8 \times 523.63 \times 0.69$

$Q = B_c \times B_f \times C \times S \times A$

$= 2.5 \times 0.8 \times \overset{523.63}{\cancel{C}} \times 0.75 \times 0.69$

$Q = 541.95\ m^3/s.$

Q. Mine has target production of $175000\ m^3/\text{month}$ & if swell factor of bucket fill factor 0.8 & if swell factor of material is 1.23, availibilty of 85% has cycle time of 25 sec. find bucket capacity. Assume productive hour in month 310 hours.

→

$Q = 175000$

$B_f = 0.8$

$S = 1.23 = \frac{1}{1.23}$

$A = 0.85$

$\frac{310 \times 3600}{25} = 44640$

$c = 25$ sec.

$Q = 175000\ m^3/\text{month}$

$B_f = 0.8$

$S = 1.23$

$A = 85\%$

$N = \frac{310 \times 3600}{25} = 44640$ No. of cycle.

more than 1 ∴ divide.

$Q = B_c \times B_F \times C \times S \times A$.

$175000 = B_c \times 0.8 \times \frac{1}{1.23} \times 0.85 \times 44640$.

$B_c = \frac{175000}{0.8 \times 44640 \times \frac{1}{1.23} \times 0.85}$

$B_c = 7.09\ m^3$

Q. A shovel of $5\ m^3$ capacity has cycle time of 60 sec, fill factor 0.8 rock looseness factor 1.5 shovel working utilization 0.6 find hour output of shovel in m^3/hr.

$B_c = 5\ m^3$, $A = 0.6$, $BF = 0.8$, $S = 1.5 = \frac{1}{1.5}$

$\frac{1 \times 3600}{60} = 60$

→ $B_c = 5\ m^3$

$C = 60\ sec = \frac{1 \times 3600}{60} = 60\ sec$.

$BF = 0.8$

$S = 1.5 = \frac{1}{1.5} = 0.6666$.

$A = 0.6$.

$Q = B_c \times BF \times C \times S \times A$.

$Q = 5 \times 0.8 \times 60 \times 0.6666 \times 0.6$.

$Q = 96\ m^3/hr$.

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/2021

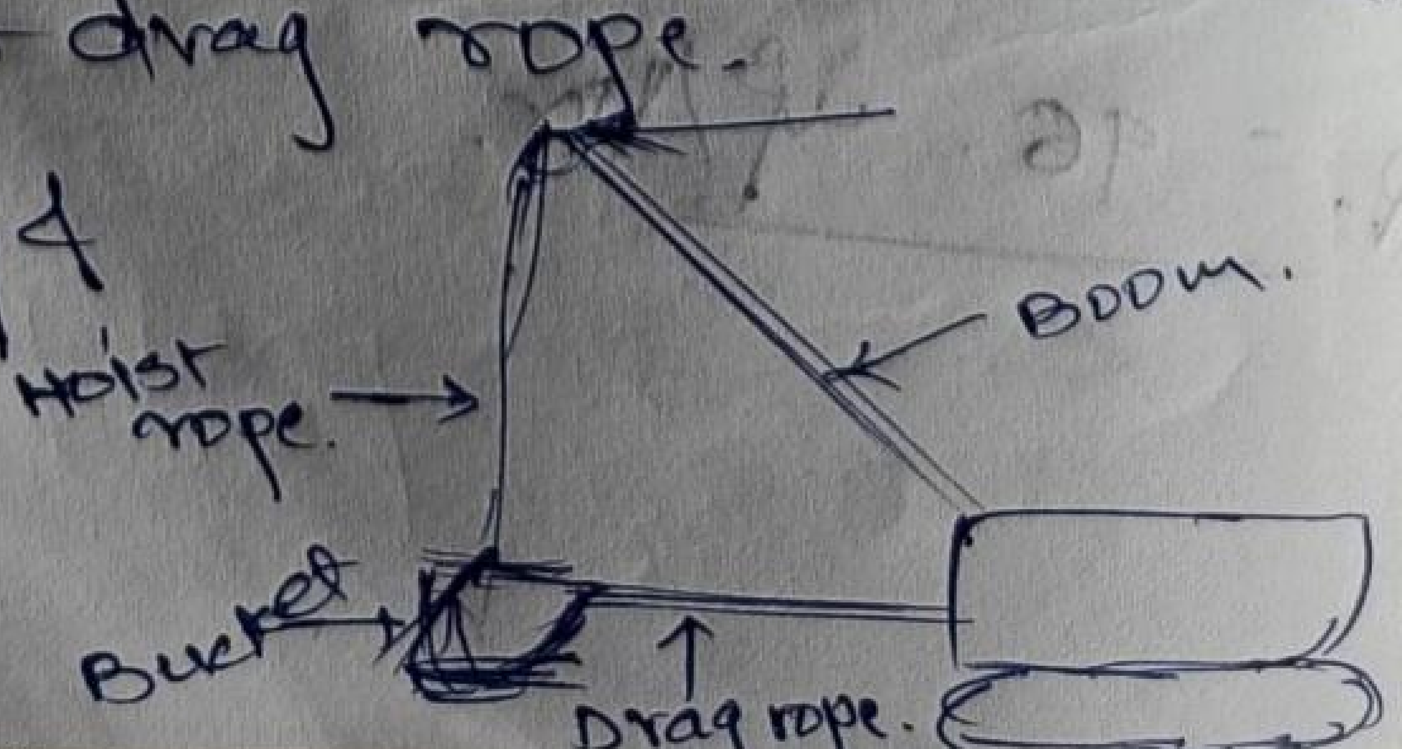
* Draglines -

- 40-50 m^3 (bucket)
- 2 ropes.

Drilling
Blasting
loading
Hauling → Drag
Transportation
processing

- crawler mounted
  a. low travel speed
  b. suitable for soft ground.
- wheel
  a. High travel speed
  b. for firm ground.
- Truck mounted
  a. same as wheel.
- walking is not used now, on crawler.
- cost of OB removal is cheaper than shovel-du
- Bucket is maneuvered by means of no. of ropes & capacity.
- No need of trucks or conveyors.

- 2 ropes - hoist & drag rope.
- 24 m^3 bucket capacity & 99 m boom length.
- Diesel operator



Factors affecting -

1. size & type of bucket -

   Light bucket - for loose, dry, sand & gravel

   Medium - for clays, compacted gravel

   Heavy - for hard, broken rocks.

2. Length of Boom -

3. Type of material.
4. Angle of swing.
5. Depth of cut. - work at optimum depth.
6. size of hauling unit.
7. skill of operator

Application
- flat gradient (1 in 6)
- seams should be free from faults & other disturbances.
- Deposits with larger strike length (>2 km).
- present cost of dragline above 100 cr & life of dragline is 1.50 lakh hours. Life 25 yr. or more.
- A mining property not suitable
- property life should be 15 yrs

ot on OnePlus

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* Dragline - Method
  - simple casting
  - Extended bench
  - horse shoe
  - continuous.

Boom length - 74-94 m.

Boom angle - 30-38°.

capacity ~~26~~-34 cu.m

Dump height - 30 m.

Q. A dragline created a spoil bench, the capacity is 24 m³, fill factor 0.6, swell factor is 0.85 availability 85%., cycle time is 150 sec. Find no. of days required to create this spoil bench, working hours is 22 hours per day.

→ $B_C = 24\,m^3$

$BF = 0.6$

$S = 0.85$

$A = 0.85$

$\dfrac{22 \times 3600}{150} = 528$

No. of cycle(c) $= \dfrac{22 \times 3600}{150}$

$= \underline{\underline{528}}$ No. of cycles.

$Q = 24 \times 0.6 \times 0.85 \times 528 \times 0.85$

$= \underline{5493.3}\ m^3/\text{day}.$

NOW,

$$\cancel{\text{Area}}\ \text{Vol.} = \frac{1}{2} \times 25 \times 15 \times 54$$

$$= 10125\ m^3$$

$$\text{days} = \frac{10125\ m^3}{5493.312\ m^3/\text{day}}$$

$$\text{days} = 1.84\ \text{days}$$

Q. A dragline require to remove 3,00,000 m^3 of rock per month on bank vol. basis. effective working hours of dragline is 450 hours, bucket fill factor is 0.8 & has cycle time of 65 sec, swell factor is 1.25. min capacity in m^3?

→

$Q = 300000\ m^3/\text{month}$

$t = \frac{450 \times 3600}{65}$

$= 24923.$

$B_f = 0.8.$

$S = 1.25 = \frac{1}{1.25}$

$B_c = ?$

$Q = 3,00000\ m^3/\text{month}.$

$\frac{450 \times 3600}{65}$

$= 24923.0$ No of cycles

$B_f = 0.8$

$S = 1.25 = \frac{1}{1.25} = 0.8.$

$Q = B_c \times B_f \times C \times S \times \cancel{A}$

$3,00000 = B_c \times 0.8 \times 24923 \times \frac{1}{1.25}$

$\frac{300000}{0.8 \times 24923 \times 0.8} = B_c$

$\frac{300000}{15950.72} = B_c$

$\therefore B_c = 18.80\ m^3$

$\frac{300000}{24923 \times 0.8 \times \frac{1}{1.25}} = B_c.$

$B_c = 18.80\ m^3.$

(or) $S.F = \frac{\text{loose}}{\text{bank}}$

$1.25 \times 3,00,000 = \text{loose}.$

$\text{loose} = 3,75,000$

$= \frac{375000}{0.8 \times 24923}$

$= 18.80\ m^3$

10/11/2021.

* Surface miners –

- 2 conveyors. (primary & discharge (secondary))
- milling drum in middle. (tungstan carbide)
- 2 set independent crawler.
- No blasting required.

- cutting m/c used in surface mines.
- for extraction of thin seams (not less 2.5m thick)
- crawler mounted m/c with ~~et~~ milling drum in middle.
- used for flat & soft deposits (coal, lignite, gypsum
- 3000 SM has capacity of 1000 t/hr.
- Boom conveyor 12 m length generally.
- Drilling, blasting & face loading no need.
- produce clean & high quality material.
- Reduction in transportation cost.
- Environmental friendly mining.
- Deposit can be exploited fully.
- takes care of both safety & economics.

Benefits –

- without blasting & drilling.
- Insitu (placed same) crushing is feasible
- unique conveying system.
- haul road preparation & maintanace in deposit is reduced.

~~Advantages~~ –
- less coal loss & dilution
- improved coal recovery.
- primary crushing & fragmentation
⊕.

Working –

- drum is lowered & raised by hydraulic system.
- directly conveyed to conveyor.
- discharge boom adjustable (90°).
- depth of cut – 50–60 cm

1. Mainframe unit
2. Conveying unit
3. Drive unit
4. Crawler unit.
5. Cutting unit.

1. M/C with middle drum
2. M/C with front boom cutting drum.
3. M/C with front cutting wheel.

Method of working –

Based on sequence of extraction → wide bench, block mining, stepped cut

Based on mode of travel → Empty travel, b- back travel., c. continuous travel.

Based on ~~level~~ loading
- Direct
- Indirect

Q.

* capacity of surface miner / sec =

$v_m \times h \times b$ m^3/sec.

v_m - cutting speed (m/s)
h - cutting depth (m)
b - milling drum width (m)

Q. A surface miner has a milling drum width of 3500 mm ($b = 3.5$ m), cutting depth of 25 ($h = 0.25$), if cutting speed is 0.5 m/s (v_m), density of coal 1.5 t/m³ ($\rho = 1.5$) Find the capacity of surface miner in tonnes / hour.

→ $b = 3500$ mm $- 350$ cm $- 3.5$ m
$h = 25$ cm $- 0.25$ m
$v_m = 0.5$ m/s.
$\rho = 1.5$ t/m³

capacity $= 0.5 \times 0.25 \times 3.5 \times 1.5$
$= 0.65625$ t/s. $\times 3600$ hr.
$= 2362.5$ t/hour

b = 3 m | V_m = 10 m/min | h = 0.30 m. b = 3 m
L = 200 m. | ρ = 1.4 t/m^3

Q. A surface miner excavates with 3 m cutting drum in wind rowing method for bench length of 200 m (L = 200 m), cutting speed is 10 m/min (V_m = 10 m/min), density of coal is 1.4 t/m^3 (ρ = 1.4), cutting depth is 30 cm (d = 0.30 m), avg. time for each bench cut is 2 hr. 40 min (160 min). find rate of production per hour.

→ b = 3 m

L = 200 m.

V_m = 10 m/min

density = 1.4 t/m^3

h = 30 cm = 0.3 m.

time = 160 min.
 + 20 min

= 180 min

$\frac{L}{V_m} = \frac{200\,m}{10\,m/min}$ = 20 min.

$V_m \times b \times h \times \rho$ (te/m^3)

capacity = 200 m × 3 × 1.4 × 0.3

= 252 te.

→ 180 min = 252 te.

60 min = ?

? = $\frac{252 \times 60}{180}$

rate of production = 84 te/hr

180 min = 252 tone.
60 min = ?

$\frac{252 \times 60}{180}$ = 84

$\frac{200\,m}{10\,m/min}$ = 20 min

2 hr 40 min = 160 min

∴ 160 + 20 m = 180 m.

capacity = $V_m \times b \times h \times \rho$
= 200 × 3 × 0.3 × 1.4
= 252 tonne.

180 min = 252 tonne.
60 min = ?

$= \frac{252 \times 60}{180}$ = 84.

60 min = 84 tonne

30/11/2021

* Bucket wheel excavator —

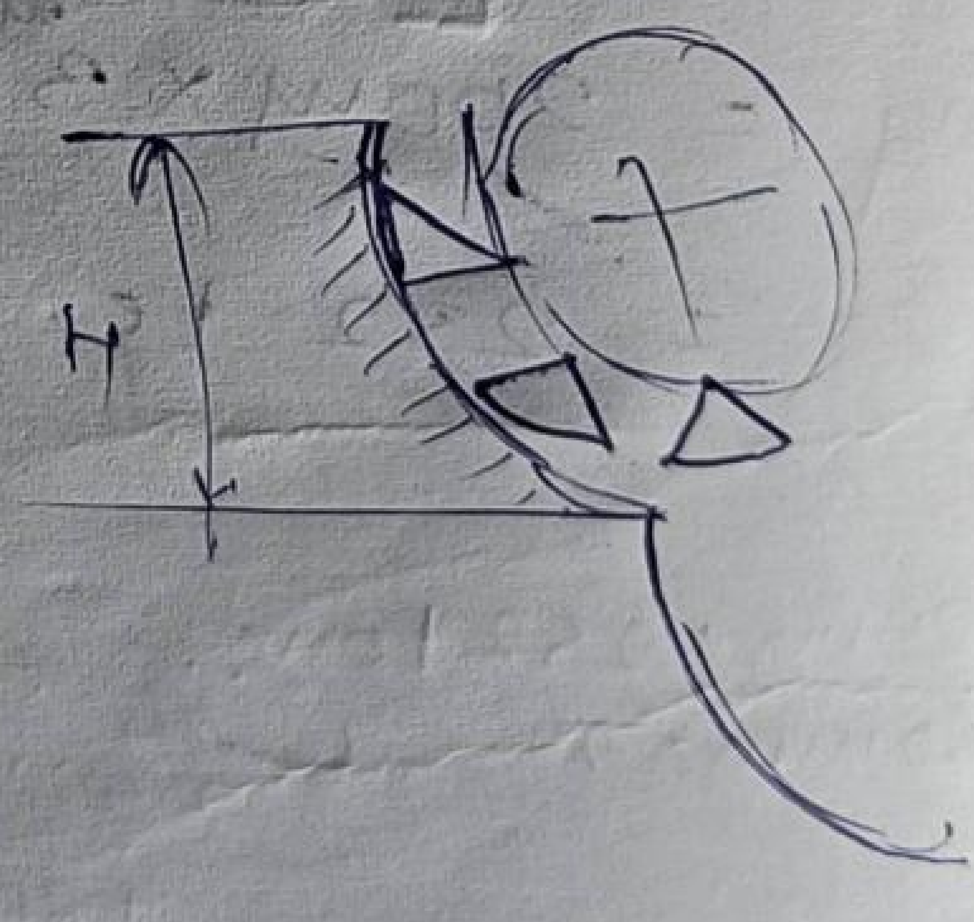
- used in surface mining. (soft rock)
- thousands of OB per day.
- continuous excavation
- 6-8 (10) buckets
- lignite mine (Nayveli) TN
- wheel rotates ; ~~teeth~~, of bucket cut the rock wall .

length - 225m. upto - 76m can work
height - 96m

Parts —
- Bucket .
- Bucket wheel
- Boom
- Discharge boom
- Slewing mechanism
- Hoist cables.

Cuts —

1. Terrace cut.



2. Dropping cut.

System of BW operation –

1. Full block working – by excavating 2–3 benches at a time. (several bench cuts by rising).
2. Face or front working – excavator travels along the working face.
3. Side block working.
   one side shovel working & other side BWE.

Bucket filling %

semi cell type – 90%
cell-type – 87%
cell-less type – 90%.

Demerits

1. High investment
2. Skilled operator. (4–5)
3. If any part gets damaged whole operation.
4. Supervision is difficult.

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* Bucket chain excavator –

– used for surface mining & dredging (sea)

– similar to BWE.

– BCE remove material from below their plane of movement.

– wheel replaced with chain & bucket ladder.

1. Bucket ladder
2. Discharge boom. } components

– Buckets are mounted on a flexible chain. (Bucket fixed – chain move)

– capacity 14,500 m^3/h.

– weight of BCE – 1,150 tons.

– length – 58.5 m.

– bucket ladder – 23.5 m.

– speed – 1.22 m/s.

– digging force – 1170 KN/m^2.

* Front End Loaders (FEL)

- same as LHD.
- wheel or crawler mounted.
- front end bucket.
- open place to another place material.
- & - loading in hopper (like bunker).
- loading & trasporting
- pit clean up.
- lower capital cost.
- one operator.
- large lumps not acceptable.
- unsuitable for hard & dense rock.
- space required.
- Rainy season - performance decreases. because of tyre mounting.
- Reduced stability in load & carry position.
- poor visibility. (for driver)

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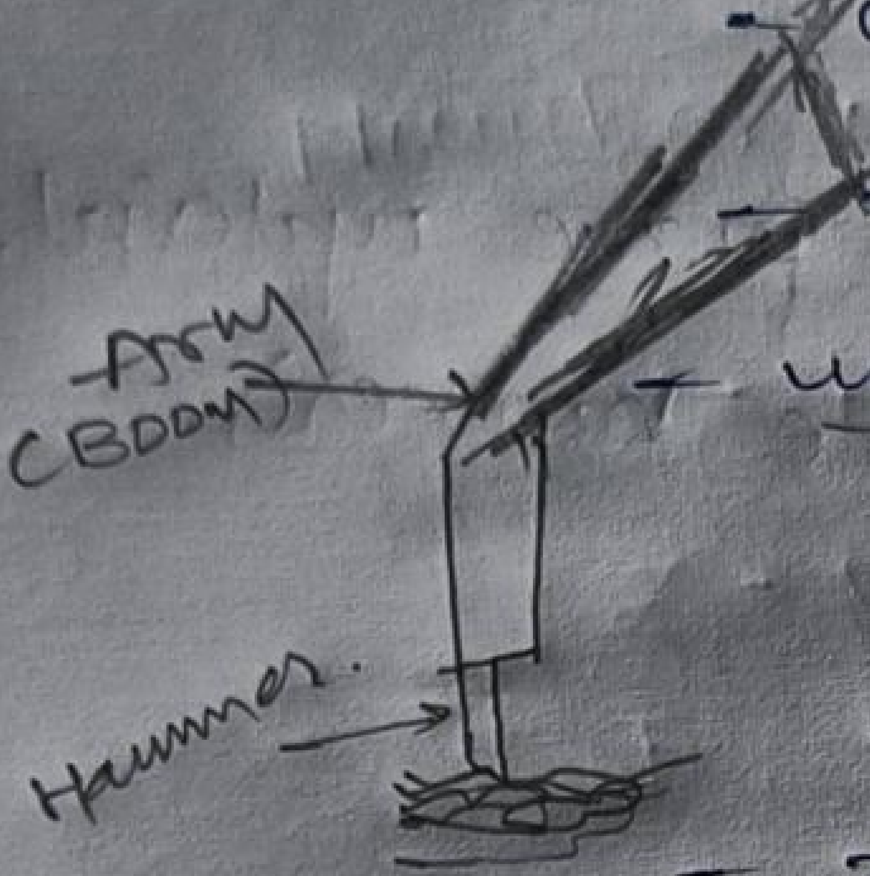
III^rd Module

* Rock Breakers –

- used in surface as well as U/G also.
- designed to manipulate large rocks,
- reducing large rocks to small one.



- major componets –
  - hydraulic hammer (used to break)
  - A boom (arm)
- 2-types of breakers –
  - mobile & → flexible
  - Stationary → stable
- It can be applied for breaking of rock/coal ore even about 500 kg/cm².
- It is used where blasting not possible (near victinity structures & habitations).
- Jack hammer operation (percussion)

3 types –
- pnumatic (compressed air)
- Hydraulic (attached to shovel of bucket)
- electropnumatic

Applications –

1. UG – steel grate (grizzly)
2. Gyratory crusher –
   - fixed type
   - used in crushing plants.
3. Mobile crusher –
   - designed to moved from place to place
   - near crusher
4. stationary crusher –
   - fixed type

EX —

Made — INDECO

Model — 8500 Wp Model

Weight — 4500 kg

Technical parameters of Mega Breaker

dia of tool — 195mm dia of tool

Stokes — 350-450 blows/min (striking rate)

pressure to breaker — 150-160 Bar.

Energy/Blow — 10,000 Jules.

Prodn/hour : 190-200 tons

Prodn/month : 55,000 tons.

diesel : 30 lit/hr.

Life of moil part : 300 hours.

Limitations —

- Not suitable for mining huge reserves
- Have to be dependant on a single m/c.
- Floor level will not be uniform as compared to surface miner.

* Numericals –

BWE

Number of buckets dump per sec = $\frac{V x^2 Z}{D}$

V = cutting speed
Z = no of buckets
D = Dia of wheel.

capacity of bucket wheel excavator (Q/hour) = $I_s \times S \times BF \times 3600$ sec → hour.

I_s = Nominal bucket capacity
S = No. of buckets dumping per sec
Bf = Bucket fill factor.

① Bucket WE has a cutting speed of 12 m/s & wheel dia 10 m & has 30 buckets with bucket capacity of 0.35 m³. If bucket fill factor is 0.65. Find capacity of BWE in hour.

→ cutting speed = 12 m/s
wheel dia = 10 m
No. of buckets = 30

No. of buckets dumping per sec = $\frac{12 \times 30}{10} = 36$

Bf = 0.65
capacity = 0.35 m³

capacity of BWE = $0.35 \times 36 \times 0.65 \times 3600$ (sec → hour)
= 24,484 m³/hour.

(2) ~~A~~ BWE has 20 buckets has bucket capacity of $0.5m^3$ rotates at 5 rpm. If B_f is 0.8. Find capacity of BWE/hour.

→ No of buckets = 20

$B_c \times B_f \times$ No. of bucket × dump/hr

capacity = $0.5 m^3$

speed = 5 rpm.

$B_f = 0.8$

No. of buckets per min dump = 20×5

= 100 buckets/min.

min−hour.

capacity of BWE $= 0.5 \times 100 \times 0.8 \times 60$

$= 2400\ m^3/hr$.

* **cutting resistance** —

$$\frac{Q_1}{Q_2} = \frac{K_2^2}{K_1^2}$$

Q = capacity of BWE

K = cutting resistance in N/mm.

(3) ~~A~~ BWE has capacity of $250 m^3/hr$ has a cutting resistance of 15 N/mm. find capacity if cutting resistance is increased by 150%.

→ capacity = $250 m^3/hr$.

cutting resistance = 15 N/mm.

cutting resistance increased by 150%.

then $(K_2) = \cancel{\frac{150}{100} \times 15} = 22.5$.

$\frac{150}{100} \times 15 = 22.5$

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$$\frac{Q_1}{Q_2} = \frac{K_2^2}{K_1^2}$$

$$\frac{250}{Q_2} = \frac{(22.5)^2}{(15)^2}$$

$$\frac{1}{Q_2} = \frac{(22.5)^2}{(15)^2 \times 250}$$

$$\cancel{= \frac{506.25}{225 \times 250}} \cancel{= \frac{506.25}{56250}}$$

$$Q = \frac{225 \times 250}{506.25} = \frac{56250}{506.25}$$

$$Q = 111.11 \ m^3/hr.$$

* No. of passes required to load dumper = $\frac{\text{Dumper size}}{\text{loader capacity} \times \text{density of mat'l.}}$

(2). A shovel has capacity of $5.5 m^3$. Bucket fill factor of 0.8. Density of coal is $1.4 \ ton/m^3$. Shovel capacity is $250 m^3$/hour. Find the no. of passes required to fill 110T capacity dumper.

→ shovel capacity = $5.5 m^3$

BF = 0.8

density = $1.4 \ ton/m^3$

shovel capacity = $250 m^3/hr$.
per hour.

TO fill = 110T.

$$\text{No. of passes required} = \frac{\text{dumper size}}{\text{loader capacity} \times \text{density}}$$

$$= \frac{110\ T}{5.5\ m^3 \times 0.8 \times 1.4\ T/m^3}$$

$$= 17.85$$

$$= 18 \text{ passes (Approx)}.$$

$$\text{Hour capacity} = 18 \times 5.5$$

$$= 99\ m^3/hr$$

$$\text{No. of dumper required} = \frac{250\ \cancel{m^3/hr}}{99\ \cancel{m^3/hr}}$$

$$= 2.52 \text{ dumper. per hour.}$$

*** capacity of shovel –**

Material hauled by dumper per hour × No. of dumpers.

*** Match factor –**

$$= \frac{\text{No. of hauler (or) Dumpers}}{\text{No. of loaders}} \times \frac{\text{loading time of loader}}{\text{Hauling time of hauler}}$$

$$= \frac{\text{No. of hauler (Dumper)} \times \text{loading time of loader}}{\text{No. of loaders} \times \text{Hauling time of hauler}}$$

Q. Two shovels has 6 dumper shovel has loading time of 3 mins.
Dumper has hauling time of 15 mins.

$$\rightarrow \text{Match Factor} = \frac{6 \times 3}{2 \times 15} = \frac{18}{30} = 0.6$$

spreaders? hopper?

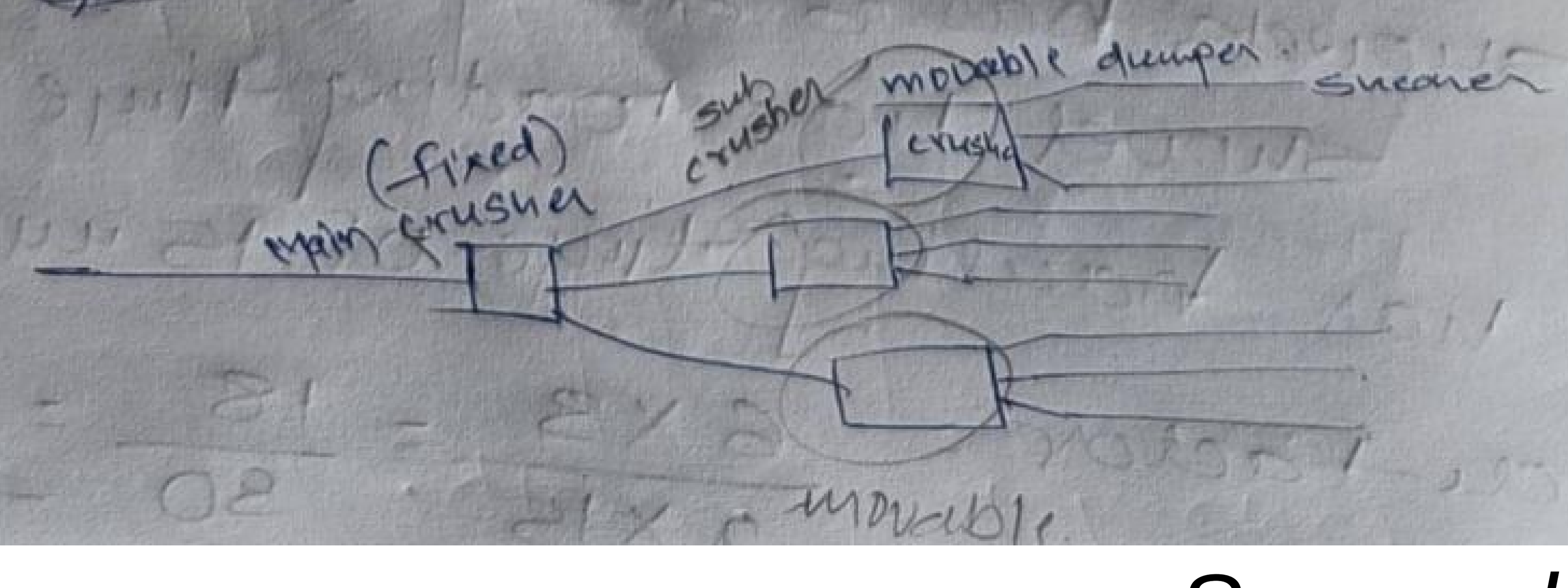
~~*~~ In - pit crushing systems -

- In pit area, the crushing & transportation work will be done.
- The use of dumper system will reduced & high capacity belt conveyors has been used.
- The crushed ~~waste~~ material transportated to plant & waste by belt conveyor.
- CCL, Ramagundam - II.
- The shovel will directly dump into crusher (fully mobile type) or to dumper & dumper to crusher (semimobile)
- crusher can be installed place to other place. (fully mobile)
- It reduces the transportation system

Types of In-pit crushing system
1. Permanent / stationary in-pit crushing system
2. Relocatable / Movable crushing system
3. Mobile in pit crushing system

1 → The place of crushing system should not disturb to working. (fixed foundation)
- fixed (can't move)
- instead it as - It should reduce transportation cost.

2 → - Medium capacity crushers used.
- The production is medium to large.
- working faces are distributed in no. of benches.
- ~~..~~ - dumper to crusher.



3 → located nearer to face. IPCC – Inpit crushing & conveying system.
– fully mobile (mobile in-pit)

– fully mobile – NO trucks, higher crushing & conveying cost

– semi mobile – some trucks & lower crushing & conveying cost.

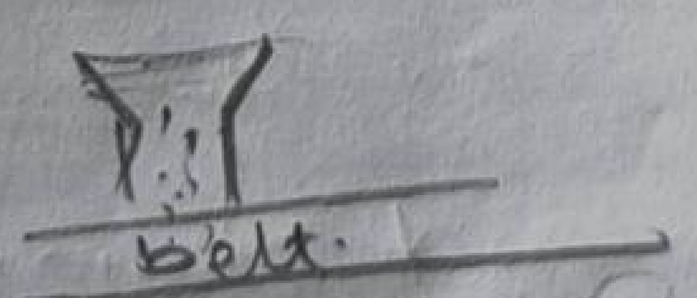
V^th Module Transportation 07/12/2021

* Belt conveyor –
* Dumper –

Belt conveyor –

operating betn 2 or more pulleys.
1. driving
2. driven (tensioning).
its load supported on idlers

- endless belt which carries load & transmits power to move the load.
- belt passes over driving drum & return drum with tensioning arrangement.
- supported by idlers mounted on light steel frames in betn drums.
- material is fed to belt at any point by chute feeders.



- 1. Belt
2. driving mechanism
3. pulleys.
4. Idlers.
5. loading & discharging device

parts

large distance – troughed shape.
small distance – flat belt.

Parameters   Inclination
sized coal - 18°
coal (fines) - 20°
coal sized (>13 mm) 17°
wet sand - upto 27°
~~woda~~ friction

High angle conveyor upto 90°.

Materials -

1. Rubber
   - used for bulk matl.
   - Made up of carcass.
2. Woven cloth
   - Light weight matl.
3. Steel
   - for metal industries

characteristics -
- High strength
- low mass
- low relation elongation

- pulley drive 30-50 kW is reqd.
- we can provide upto 250 kW

pulleys used
1. Head pulley
2. drive pulley
3. Take up pulley.

- dia of idlers. 4-7 inches.

on OnePlus

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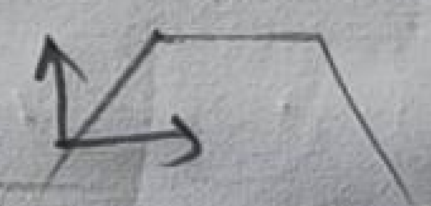
Design (material) of BC (choice of depends on)
- Avl space
- conveying length
- characteristics of matl
- capacity req.

capacity depends on idlers strength.

Factor affecting selection of BC -
1. Terrain condition (ground) - flat terrain / undulated/hilly.
2. Material property.
3. Inclination
4. Angle of repose - 
5. Stickiness
6. Dust proneness
7. Req transportation rate
8. compatibility with primary mining machinery
9. Feeding & discharge req.
10. Belt shifting needs & frequency.
11. power consumption in KWh per te-Km.

Advantages -
- capacity to handle very fine powder materials as well as large lump size.
- Less power consumption i.e. 100 KW/te-Km
- Higher reliability

- lower operating cost
- longer life
- lightest trasporting machine
- Adaptable to special requirements.
- less man power req.
- Easier safety & trouble monitoring easy.
- simple design
- 3 - 4 Km (length can be varified upto)
- High load capacity (30000 t/h)
- Easy maintanance
- Higher reliability for operation
- More flexible.

Disadvantages -

- vertical transportation can't be done
- seperate arrangement for material handling req. (chute & feeder)
- High initial cost for installation

Limitations -

$$B \geq Xa + 200$$

B - Belt width

X - longest diagonal of irregular lump, of

a - factor to account for gradient.

(Belt width should be more or equal to diagonal of irregular lump. of matl. + 200.)

Types -

1. permanent. Belt conv
2. portable. Belt conv - Troughed belt conveyor.
- flat belt conveyor.

High capacity, High speed

- for coal speed - 2m/sec - 457 mm width of belt.

* Dumps

Huge amounts of overburden material are either casted back in to the pits are dumped either inside the mine (internal). or outside area (external)

classification -

1. size can range from less clay size particles to big boulders (less than 0.1 mm to more than 1 m).

- Dumping operation that allows the material fall from high height

Dumping Methods -

1. valley fill - From upstream to downstream. if not filled completely then called as partially valley fill

2. cross valley fill -
3. side fill hill
4. Ridge embankment
5. Heaped - slope all sides (
6. End dumping
7. push dumping (Bull dozer)
8. Free dumping (Anywhere)
9. Dragline dumping

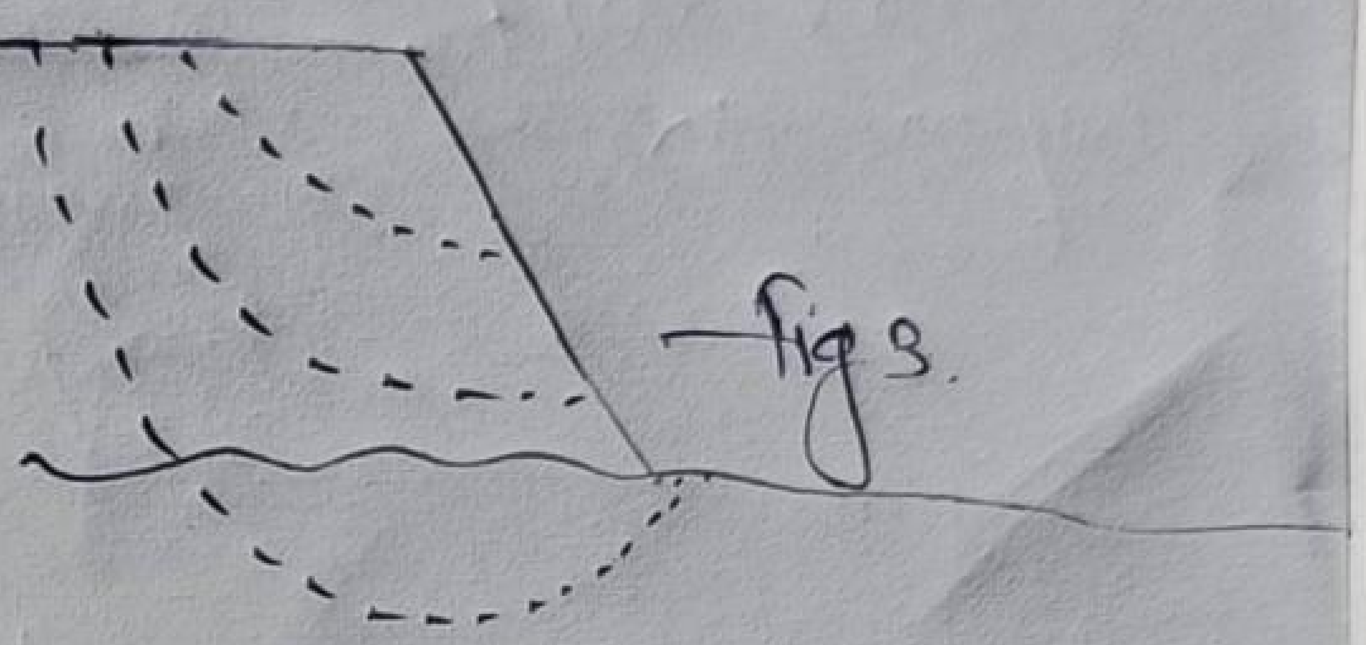
Segregation - coarse material (particles) goes down & fine materials is settles up.

Factors affecting dump slop stability -

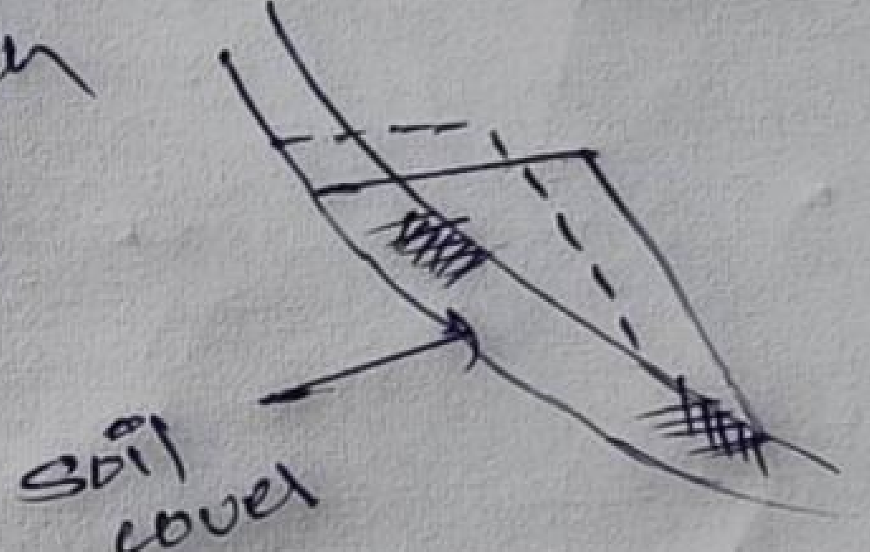
1. Geometry of dump
2. Geotechnical properties of material
3. —— " — of foundation
4. Method of dumping
5. Hydrological condition
6. Static & dynamic forces (due to drilling blasting vibrations).

Modes of dump failure —

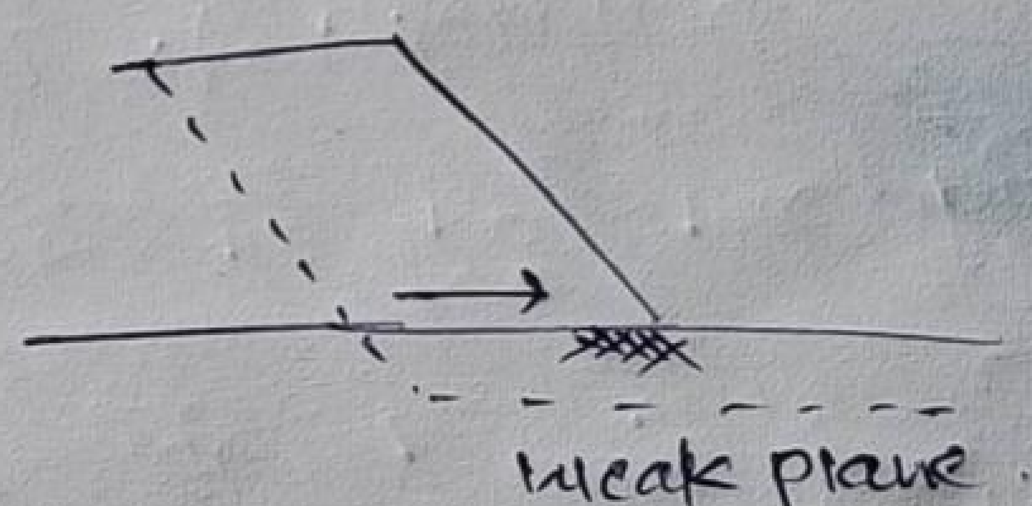
1. surface or edge failure
2. shallow flow slides
3. Rotational circular



4. Block translation



5. Base failure



6. Liquefaction