Mine Ventilation

→ NVP (Natural Ventilating Pressure)

→ $D(\rho_d - \rho_u)g$

→ $\frac{DgB}{287.1}\left[\frac{T_u - T_d}{T_u \times T_d}\right] \times 10^3$ Pa

T_u → Temp. of UC shaft (K)

T_d → Temp. of DC shaft (K)

B → Barometric pressure of UC & DC shaft (kPa)

D = Depth of DC shaft (m)

ρ_d & ρ_u = Air densities of UC & DC shaft

→ Motive column → i) $H = \frac{\rho_d - \rho_u}{\rho_d} \times D$

ii) $\frac{T_u - T_d}{T_u} \times D$

iii) $\frac{NVP}{\rho_d g} = H$

$\rho_{dry\ air} = \frac{B \times 10^3}{287.1T}$ kg/m³

→ $\rho_{moist\ air} = \left[\frac{B - 0.378e}{287.1T}\right] \times 10^3$ kg/m³

e = Vap. pressure (kPa)

→ Rate of auxiliary ventilation required to bring down conc. of nitrous fumes to permissible level

$$\phi = 2.3 \left(\frac{V_m}{t}\right) \log \left[\frac{C_o}{C_p}\right] + \left[\frac{V - V_m}{t}\right] m^3/min$$

C_o → Initial conc. of nitrous fumes (%)

C_p = permissible conc. of nitrous fumes (%)

V = Vol. of working place

V_m = Vol. of mixing zone

$15 \times w \times h$

t = Safetime = 30 min

→ Air power → $P \cdot Q \times 10^{-3}$ kW

P in Pa, Q in m^3/s

1 hp = 746 W

$1 W = \frac{1}{746}$ hp

→ Critical pressure of Booster fan

$$= \frac{P_{fan} \times R_{split}}{R_{Td}}$$

(in which booster fan is installed)

Conventions

i) 1 mm wg → 9.81 Pa
ii) 1 mm Hg → 133.3 Pa
iii) 1 Weisbach → 9.81 Ns^2m^{-8}
iv) 1 mwg → 0.00981 Ns^2m^{-8}

v) R → Weisbach
P → mm wg

0.001 = 0.00981 × 1000 / 1000000 9.81

→ Thearitical depression (H)

For Backward bladed fan

$$H = \frac{V^2 - uv\cot\alpha}{g}$$

For Forward Bl. fan $H = \frac{V^2 + uv\cot\alpha}{g}$

For Radial fam → $H = \frac{V^2}{g}$

u = Radial velocity of air leaving the fan (flow velocity m/s)
V = Peripherial speed of blade tips (m/s)
$= \frac{\pi D N}{60}$

α = ext. angle b/w fan blade and tangent at the periphery

→ Relative Quantity Method

$$P = \frac{KSQ^2}{A^3}$$ (Atkinson eq.)

If pressure is same & nature of lining is same

$$\frac{KSQ_1^2}{A_1^3} = \frac{KSQ^2}{A^3}$$

$$Q = \sqrt{\frac{PA^3}{KS}}$$

$$Q = \sqrt{\frac{A^3}{KS}}$$

Relative quantity in splits $(q_1) = \sqrt{\frac{A_1^3}{L_1 P_1}}$

$$q_2 = \sqrt{\frac{A_2^3}{L_2 P_2}}$$

$$Q_1 = Q\left[\frac{q_1}{q_1+q_2}\right], \quad Q_2 = Q\left[\frac{q_2}{q_1+q_2}\right]$$

→ Relative humidity

$$= \frac{\text{Mass of water vapour / } m^3 \text{ of air}}{\text{Mass of water vapour required to saturate } 1m^3 \text{ of air}}$$

→ Rate of diffusion

Amount of gas passing through an area / unit time

→ Equivalent Orifice

$$A = \frac{1.2\,Q}{\sqrt{P}} \text{ or } \frac{1.2}{\sqrt{R}}$$

A ↓ Or area (m^2)

P ↓ Pa; Q ↓ m^3/s; R ↓ Ns^2m^{-8}

→ $P = RQ^2$ or $\frac{KSQ^2}{A^3}$ or $\frac{KSV^2}{A}$

P ↓ Pr. absorbed (Pa)

$R = \frac{KS}{A^3}$

K ↓ Friction coeff (Ns^2m^{-4})

S = Rubbing (m^2) surface ($L \times P$)

$K = \frac{f\rho}{8}$ → f = darcy weisbach resistance coeff.

→ $R_{series} = R_1 + R_2 + \dots R_n$

$R_{parallel} = \frac{1}{\sqrt{R_{eq}}} = \frac{1}{\sqrt{R_1}} + \frac{1}{\sqrt{R_2}} + \dots \frac{1}{\sqrt{R_n}}$

↳ $\frac{R}{n^2}$ (in case of equal resistance) → n = no. of splits

→ Reynald number

$Re \leq 2000$ [Laminar flow of air]

$Re \geq 4000$ [Turbulent flow of air]

$2000 \leq Re < 4000$ [Transitional flow of air]
(Mixture of both flow)

→ ~~Pr~~ recovery by Evasee

= ~~$\left[\frac{v_1^2 - v_2^2}{2g} \times \eta\right] \rho g$ Pascal~~

V_1 = air velocity at base

V_2 = air vel. at outlet

$$\frac{\left(\frac{1}{2}\rho V_1^2 - \frac{1}{2}\rho V_2^2\right)\eta}{g}$$

$$\left(\frac{V_1^2 - V_2^2}{2}\eta\right)\rho/g \text{ mm wg}$$

Shock Loss

→ Shock factor for bend

$$X = \frac{0.25}{r^2\sqrt{a}} \times \left(i/90\right)^2$$

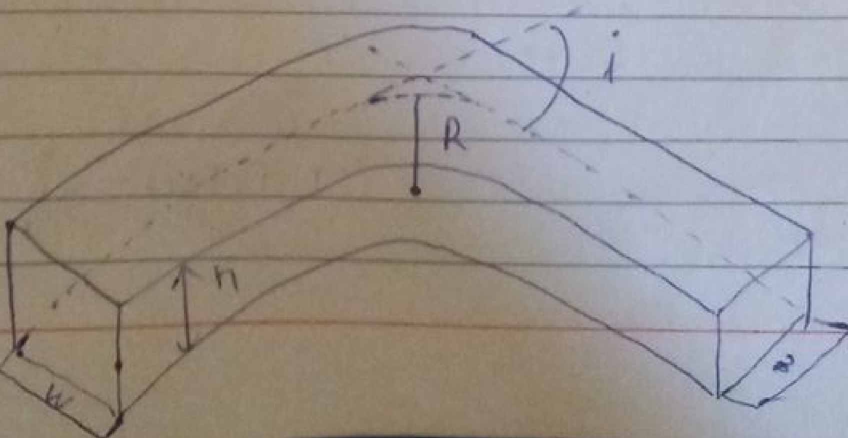
where

r = radius ratio $\left(\frac{R}{w}\right)$

a = aspect ratio $\left(\frac{h}{w}\right)$

i = deflection angle (degrees)

X = shock factor



Shock pressure loss

$X \frac{1}{2} \rho v^2$ or $X P_v$ (vel. pressure)

ρ = density of air

v = air velocity

Reynolds' number

→ Shows nature of flow of fluid (laminar (streamline) / turbulent)

$$= \frac{VD\rho}{\mu}$$

μ = Viscosity of fluid (kg/ms)

ρ = fl. density

D = Duct dia (m)

V = mean vel. of flow (m/s)

Eq. dia = $\frac{4A}{P}$ A → cr sectional area, P → Perimeter of airway

Kinematic viscosity (ν) = μ/ρ m^2/s

$$Re = \frac{VD}{\nu}$$

→ Manametric efficiency

The actual depression produced by a fan is somewhat less than the calculated theoritical depression

→ Man. η = $\frac{\text{Actual dep. prod.}}{\text{Thearitical dep.}}$

~~η = Output~~ produced

→ η = $\frac{\text{Output}}{\text{Input}}$ → air power

↳ a) Power given to the fan shaft

b) Power given to the to the driving motor or engine

→ When fan shaft hp is considered as input η is called mechanical efficiency of the fan or fan efficiency

= $\frac{\text{Air hp}}{\text{Fan shaft hp}}$

→ When power given to the motor is considered as input, η is called overall mech. eff.

OM η = $\frac{\text{Air power}}{\text{Power input to fan motor or engine}}$

H = head generated by fan (m)

$\frac{(H \rho g)\, Q \times 10^{-3}}{\text{Power input to fan motor (kW)}}$

ρ = Air density

Fan motor power

$$= \frac{\text{Air power}}{\eta_{Fan} \times \eta_{motor}}$$

Q. A sealed area known to have blackdamp has 18% O_2, 2% CO_2, 78% N_2. Fresh air has 21% O_2, 0.03% CO_2, 78.97% N_2. Percentage of Blackdamp in the area is?

Sol

$21\% \, O_2 \rightarrow 78.97\% \, N_2$

$1\% \, O_2 \rightarrow \frac{78.97}{21}\% \, N_2$

$18\% \, O_2 \rightarrow \frac{78.97}{21} \times 18 = 67.688\% \, N_2$

$21\% \, O_2 \rightarrow 0.03\% \, CO_2$

$1\% \, O_2 \rightarrow \frac{0.03}{21}\% \, CO_2$

$18\% = \frac{0.03}{21} \times 18\% \, CO_2 \rightarrow 0.0257\% \, CO_2$

Blackdamp:

Excess N_2 + Excess CO_2

$= (78 - 67.688) + (2 - 0.0257)$

$= 12.286\%$ Ans

Q/ Find percentages of blackdamp, whitedamp, firedamp and air in a mine air sample having

O_2 - 19.11%
N_2 - 79.04%
CO_2 - 0.25%
CO - 0.02%
CH_4 - 1.58%

What is the composition of Blackdamp?

Sol

Atmospheric air entering the mine has:-

$O_2 \rightarrow 20.93\%$ (vol %)
$N_2 \rightarrow 79.04\%$
$CO_2 \rightarrow 0.03\%$

~~O_2 20.93% → N_2 79.04%~~

$20.93\%\ O_2 \rightarrow 79.04\%\ N_2$

$1\%\ O_2 \rightarrow \frac{79.04}{20.93}\%\ N_2$

$\rightarrow 19.11\%\ O_2 \rightarrow \frac{79.04}{20.93} \times 19.11$

$= 72.166\%\ N_2$

$20.93\%\ O_2 \rightarrow 0.03\%\ CO_2$

$1\%\ O_2 \rightarrow \frac{0.03}{20.93}\%\ CO_2$

$= \frac{0.03}{20.93} \times 19.11 = 0.0273\%\ CO_2$

Blackdamp :- exc. N_2 + exc. CO_2

= (79.04 − 72.166) + (0.25 − 0.0273)

Bl (sample) = 7.1%

~~Atmosphere~~ air

19.11 + 72.166 + 0.0273

= 91.3%

Q The evasee chimney of a fan has an area of $4 m^2$ at base and $14 m^2$ at outlet.

i) Calculate the saving of water gauge

Sol:

Alternative ⇒ $V_{pi} = \frac{1}{2} \rho v^2 = \frac{1}{2} \rho \left(\frac{Q}{A}\right)^2 = \frac{1}{2} (1.2) \left(\frac{100}{4}\right)^2$

= 375 Pa

V_{po} = 30.612 Pa

Sav. in wg = $\frac{375 - 30.612}{9.81}$

= 35 mm wg

⇒ Gain of Pr

$= \frac{V_1^2}{2g} - \frac{V_2^2}{2g}$

$= \frac{25^2}{2 \times 9.81} - \frac{7.14^2}{2 \times 9.81}$

= ~~31.855~~ 29.256 m × 1.2

Head of air

= 35 mm wg

AKGoral

Q92 sol

$$R = \frac{P}{Q^2} = \frac{1000}{150^2}$$

$= 0.044 \; Ns^2m^{-8}$

Gain in pr due to evasee

$A_o = 18 m^2$

$\frac{A_i}{A_o} = \frac{1}{4} = A_i = 4.5 m^2$

$$V_{pi} = \frac{1}{2}(1.2)\left(\frac{Q}{A}\right)^2$$

$$= \frac{1}{2}(1.2)\left(\frac{150}{4.5}\right)^2 = 666.66 \, Pa$$

$$V_{po} = \frac{1}{2}(1.2)\left(\frac{150}{18}\right)^2 = 41.66 \, Pa$$

Pr. gain $= 625 \eta$

Change in pr due to inst. of evasee

$= 0.044 \times 120^2$

$= 640 \, Pa$

$= 1000 - 640 = 360 \, Pa$

$$\eta = \frac{360}{625} \times 100 = 57.6\%$$

Relative Humidity

$$= \frac{\text{Mass of water vapour / } m^3 \text{ of air}}{\text{Mass of water vapour req. to saturate } 1 m^3 \text{ of air}}$$

Relative Humidity

$$= \frac{\text{Act vapour Pressure}}{\text{Saturated vap. pressure}} \times 100$$

Humidity Ratio:

Specific humidity or Humidity ratio of an air sample is the ratio of the weight of water vapour contained in the sample compared to weight of dry air in the same sample

$$622 \left(\frac{\text{partial vapour pressure}}{\text{Baro pressure} - \text{Saturated vap. pr}} \right)$$

g/m³

→ Lightest gas → H_2
SG → 0.070

Heaviest gas → SO_2
→ 2.264

Blackdamp is also called stythe or choke damp
(mix. of CO_2 & N_2)

Firedamp → Mix. of CH_4, N_2, Heigher hydrocarbons (C_2H_6, C_2H_4), CO_2

White damp → CO

Stinkdamp → H_2S

Afterdamp → prod. by coal dust or firedamp explosion

13/1/21

Metal mining methods

Applications

- Room & pillar

- Development is followed by depillaring for max. extraction of coal from the standing pillars supporting the roof over the coal seams

- Ore is extracted in wide rooms sep by pillars of in situ are provided in a regular manner for support of the hanging wall

→ Ore & hanging wall should be strong
→ Dip should be mild or nearly flat
→ Thick ore body

Advantage :- i) Low pr. cost (production)
ii) Very little dilution of ore
iii) High level productivity (high OMS of face workers)

Disadvantage :- i) Loss of ore in pillars

Open stopes

→ No filling or timber is used to support walls and only simple form of scatered timbering is used as temporary support.

- Both the wall rocks & the ore should be strong
- Dilution of ore is minimum.

i) Overhand Stoping

- The two levels enclosing an ore block are connected to raise at intervals and stoping starts from the raise.

- St. operation proceeds from lower main level towards the upper main level & ore is extracted in step like faces or benches in a ascending order

- Commonly emp. in steeply dipping narrow veins and in bedded dep. of 2-3 m thickness.
- Strong ore & wall rocks

ii) Rill Stoping

Geology - Minerals, rocks & their origin, classification,
Rock types - ore genesis, structural geology

Igneous rocks

- Igneous rocks form when molten lava or magma cools and solidifies

1) Granite
2) Peridotite
3) Pegmatite
4) Diorite
5) Gabbro

Minerals. Quartz, feldspar, mica, olivines, pyroxenes

these are the ex of intrusive igneous rocks :-
these rocks crystallize below earth surface and slow cooling allows large crystals to form

extrusive igneous rocks

→ erupt onto the surface & cool quickly to form small crystals

i) andesite
ii) basalt
iii) rhyolite

Sedimentary Rocks

→ ~~Sea~~ sed rocks are formed by the accumulation of mineral particles or sediments at the earth surface followed by cementation

examples:-

i) Sandstone
ii) Shale
iii) Siltstone
iv) Conglomerate
v) dalamite
vi) coal
vii) limestone

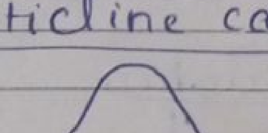
Metamorphic Rocks

i) Phyllite
ii) Schist
iii) Gneiss
iv) Quartzite
v) marble

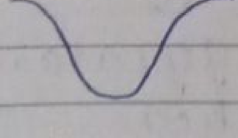
→ Hade of fault
→ angle between vertical plane & fault plane

Fold

- Undulation in stratified rock masses
- anticline convex upward



- syncline concave upward



Monocline

Comminution

→ Reducing the raw ore to size required for mechanical separation.

→ Coarse size reduction stage – Crushing

Fine size reduction stage – Grinding

Objectives of Comminution

i) Reduction of large lumps to small pieces

ii) Production of solids of desired size ranges or specific surfaces for direct metallurgical treatment

iii) Breaking apart valuable minerals from gangue material

Crusher – i) Primary crusher
ii) Secondary crusher

- Cone, roll crusher
- Gyradisc crusher

- Jaw & gyratory crusher
  - Blake crusher
  - Dodge crusher
  - Universal crusher

Gyratory crusher:

i) Suspended spindle type
ii) Supported spindle type
iii) Fixed spindle type.

Grinding → Breaking down the relatively coarse material (prod. by crushing) to the ultimate fineness

i) Ballmill is used

Critical Speed of Mill

D = Dia. of mill
d = dia of ball
g = ac. due to gr.
r = rad. of ball

→ The min speed at which a mill charge will centrifuge is known as critical speed of mill

$$N = 54.19/\sqrt{R} \quad \text{or} \quad * \frac{\sqrt{2g}}{(D-d)} \times \frac{60}{2\pi} \text{ rpm}$$

N = Critical speed (rpm)
R = rad of mill (ft)

or * $\sqrt{\frac{g}{r}}$ m/s

Faster the mill is rotated higher the balls will be lifted, when speed of rotation is great enough, gravity is powerless to the centrifugal force & the balls will be carried through a complete circle. When this happens, the charge is said to be centrifuge.

Gate 2013

→ For spherical charge,
max. allowable ratio of dia to charge length is 1:6

~~Rift stoping method~~
Phreatic Surface

→ Also called water table
→ defined as the surface at every point of which the pressure in the water is atmospheric
→ Also defined as level in the soil where the hydraulic pressure of the water in soil pores = atmospheric pressure

→ $P(A) = 0.5$
$P(B) = 0.4$

If A and B are independent

$P(A \cap B) = P(A) \cdot P(B)$
$P(A \cup B) = P(A) + P(B) - P(A \cap B)$

$P(A \cap B) = 0.20$
$P(A \cup B) = 0.5 + 0.4 - 0.2$
$= 0.7$

→ Impedance of rock = P wave velocity × ρ_{rock}

→ Imp_{exp} → $VOD \times \rho_{exp}$

→ Standard Pressure

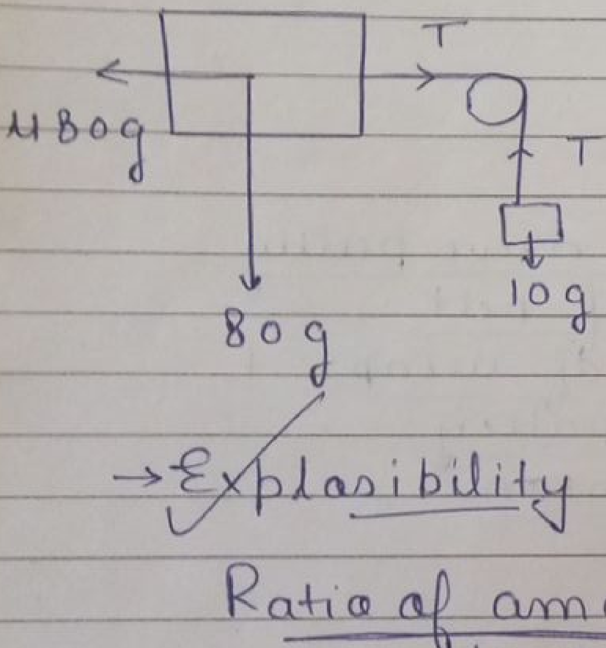
= 101.33 kPa

1 bar = 1.013×10^5 Pa

760 mm of Hg

$P = \rho g h$

$\frac{1.013 \times 10^5}{1000 \times 9.81} = h = 10.32$ m



$10g - T = 10a$

$+ T - 3.2g = 80a$

$6.8g = 90a$

$a = 0.74$ m/s²

(coal dust)

→ Explosibility factor :-

Ratio of amount of inert dust to amount of coal dust

Am. of Limestone dust (Inert dust) = 3g

Am. of coal dust = 2g

Exp. factor = 3/2 = 1.5

→ $L_1 = 20$ m

Actual length
= 19.8 m

$\left(\frac{19.8}{20}\right)^3 \times 4000 = 3881 \, m^3$

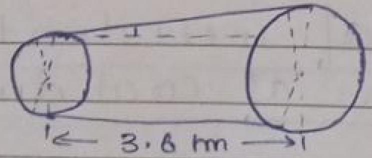
→ Respiratary Quotient / Young's ratio :-

= CO_2 pr / O_2 consumed

$= \frac{3.83}{20.93 - 16.65} = 0.89$

→ Snub Pulley

- Mounted close to drive pulley on return side of belt
- Increases angle of wrap around the drive pulley
- Reduces belt tension

→ 

angle of contact of sm. pulley = 180 − 2α

$\tan\theta = \frac{1.2 - 0.8}{3.6}$

$\theta = 6.3°$

∠ of contact

$= 180 - 2(6.33)$

$= 167.34°$

Enviromental Studies of Mine

→ Sound wave propagation

Point Source

$$L_p = L_w - 20 \log_{10}(r) - 11 \text{ dB}$$

L_p ↓ Sound Pr. level of the point source at a radial distance of r from the noise source

L_w ↓ So power level of the point source

Rel. b/w two sound pr. level at a radial dist. of r_1 & r_2

$$L_{p1} - L_{p2} = 20 \log \left[\frac{r_2}{r_1}\right]$$

L_{p1} & L_{p2} → sound pr. level at a radial distance of r_1 and r_2.

Line Source

$$L_{p1} - L_{p2} = 10 \log \left[\frac{r_2}{r_1}\right]$$

→ Illumination

→ Intensity - relative amount of luminous energy given by any source.
(bright, shining)

→ measured in candles or candle power or in candelas.

→ Mean spherical candle power ∝ total light given by the lamp

→ Illumination of a surface (meter candle) = candela of source × $\frac{\cos\theta}{dist^2}$

θ = angle between normal to the surface & direction of light rays

→ Lumen (lm) → Unit of light emitted by a light source

Lumen emitted by a lamp → Mean sph. cand. power × 4π

1 lux = 1 lumen/m^2

→ Footcandle (Fc) → unit of measure of illumination

1 Fc = 1 lumen/ft^2

1 Fc = 10.764 lux

→ Ultimate BOD

= amount of oxygen required to decompose all the biodegradable organics in a given vol. of water.

$$BOD_t = BOD_L(1 - e^{-kt})$$

t = time (day)

BOD_t = BOD at any time t, mg/L

BOD_L = Ult. BOD (mg/L)

k = const. representing rate of BOD reaction

COD > BOD

↓

Chemical Ox. demand

→ In COD test, $K_2Cr_2O_7$ is used to oxidize the organics

→ Total suspended solids in the filter

$$TSS = \frac{A - B}{C}$$

B → Weight of clean filter

A → Weight of filter + retained solids

C → Volume of sample filtered

→ Equivalent Noise level

Constant sound pr. level which would have produced the same total energy as the actual level over the given time

$$L_{Aeq,T} = 10 \log_{10} \left[\sum_{i=1}^{n} t_i \, 10^{L_i/10} \right]$$

$L_{Aeq,T}$ → eq. SPL

t_i → fraction of time for SPL L_i

L_i → SPL for time duration t_i

Day night equivalent noise level

$$L_{dn} = 10 \log_{10} \left[15/24 \left(10^{L_d/10} \right) + 9/24 \left[10^{(L_n+10)/10} \right] \right] \text{ dB(A)}$$

L_d = day equivalent noise level (6am - 9pm) dB(A)

L_n = night eq. noise level (9pm - 6am) dB(A)

Measurement of Noise

→ Audible frequency range of human ear
= 20-20000 Hz

Sound Pr Level - magnitude of val. as a sound expressed in decibals

$$SPL = 20 \log_{10}\left[\frac{P}{P_{ref}}\right]$$

↓
Sound Pr. Level (dB)

P = Meas. SPL (N/m^2)

P_{ref} = Ref. SPL (N/m^2)
($2 \times 10^{-5} N/m^2$)

Sound Power level -

$$L_w = 10 \log_{10}\left[\frac{W}{W_{ref}}\right] dB$$

W = acoustic power of interest (Watt)
↓
~~Related to sound~~

W_{ref} = reference ac. power
(10^{-12} W)

Electrostatic Precipitator

→ It is a air pollution control device that uses electric charge to remove certain impurities, like solid particles or liquid droplets from air or other gases in smokestacks.

Acoustic power → Sound power or acoustic power is the rate at which sound energy is emitted, reflected or transmitted or received, per unit time.

→ TWA → Time weighted avg noise level

It shows worker's daily exposure to occupational noise, taking into account the avg. levels of noise & time spent in each area

Noise Dose

$$= 100 \times \left(\frac{C_1}{T_1} + \frac{C_2}{T_2} + \cdots \frac{C_n}{T_n}\right)$$

C = Time sp. at each noise level

$$T_n = 8/2^{(L-90)/5}$$ (Power)

L = Meas. sound level

$$T_1 = 8/2^{(-4/5)} = 13.93$$

$$T_2 = 8/2^{(0.4)} = 6.06$$

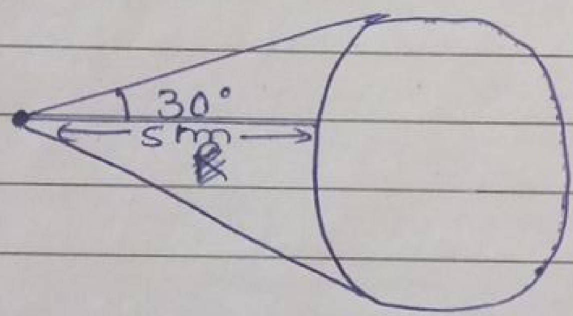
$$\text{Dose} = 100 \times \left(\frac{6}{13.93} + \frac{3}{6.06}\right)$$

$$= 92.57\%$$

$TWA = 16.61 \log [D/100] + 90$

$16.61 \log 0.9257 + 90$

$= 89.4 \text{ dB}$



For plane angle

Surface area covered by beam $= \omega R^2$

For solid angle

$SA = 2\pi (1 - \cos\theta) D^2$

2π

$2\pi(1-\cos\theta) \rightarrow$ Solid angle (rad.)

Drilling

→ Top hammer drilling

→ Used in medium to hard rocks
~~for hole dia upto 230mm~~
top hammer drills include

i) Hand held rock drills
(stopers, jacklegs, sinkers)
ii) light drills mounted on a feed device and boom for tunneling (jumbos)
iii) medium to heavy crawler based or wheel based drill rigs. ~~with boom~~.

THD combines → Percussion - feed - rotation - flushing

→ In THD, hammer produces a percussive force on drill rods which is transmitted to the drill bit

Benefits

i) Faster penetration rate as comp. to DTH
ii) Highly mobile
iii) Lower fuel consumption
~~iv)~~

Detonator

→ Used to trigger explosive device
→ commercial use of explosives uses electrical detonators or the capped fuse which is a length of safety fuse to which an ordinary detonator has been crimped.

Explosives

ANFO (Ammonium Nitrate Fuel Oil)

i) Ammonium nitrate (94%) mixed with diesel oil (6%), is used on a large ~~circle~~ scale for blasting in quarries of coal and metal mines.

VOD → 3.2 km/s Form = All solid.
Size → 2 mm

ii) Slurry explosives — Gelling agents - starch; Fuel sensitisers → TNT, PETN

Slurry explosives are with jelly like consistency & are water gels

Ingridients :- Oxidisers → ammonium, sodium or calcium nitrates
Cross linking agents → potassium or sodium dichromates

→ Slurry & Emulsion exp. are highly water resistant.
→ Size → 0.2mm
Form → Solid / Liquid
VOD → 3.3 km/s

Emulsion explosives

→ Contains AN solution comb. with diesel oil
→ depend entirely on presence of voids for initiation & propagation
Change in amount of void effects a change in density
→ More fluid than slurry exp. & therefore create problems when a loading a blast hole with fissures or cracks
→ Size → 0.001mm
→ Form → Liquid
→ VOD → 5-6 km/s

Advantages of Nonel System

i) Extreme resistance to accidental initiation
ii) Noiseless in character
iii) eleminates the need for complicated electrical circuit testing & shotfiring equipment

iv) eliminates the risk of misfire
v) Full exploitation of explosive energy & minimizes flyrock
vi) Reduced vibration & better rock fragmentation is achieved

Blown out shot

→ A shot or blast is said to be blown out when it come out of the blast hole & does not shatter the rock
→ Such shots dissipate the explosive force by blowing out of the stemming instead of breaking down the coal

→ Occurs due to improper stemming, excessive burden or spacing b/w shotholes,
improper use of delay detonators
incorrect sequence of firing of shot holes.

→ Seismic Waves

Waves of energy caused by sudden breaking of rock within the earth or an explosion

Recarded an Seismagraphs

Types of Seismic Waves

- Body Waves → Travels through earth inner layer
  - P wave
  - S-wave
- Surface Waves → Move along the surface like ripples on water.
  - Love wave
  - Rayleigh waves

Body Waves travelling through the interiar of the earth, body waves arrive befare the surface waves emitted by an earthquake

P wave → also called primary, compressional and longitudnal waves

- → In salid, travels fast as twice as S waves.
- → Fastest seismic wave
- → can travel through any type of material.
- → Pushes & pulls the rock it moves

S-Wave → also called Secandary, transverse or shear waves

- → Slower than P wave
- → Move through salid but not liquid medium

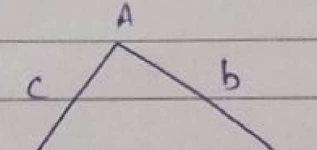
→ ground is displaced perpend. to the direction of propagation.

Love waves → Fastest surface waves
→ causes horizontal shearing of ground
→ travel faster than rayleigh waves

Rayleigh waves → also called ground roll
→ travels as ripples with motions similar to waves on water surface

Trilateration

Trilateration, method of surveying in which the lengths of the sides of a triangle are measured and from this inf., angles are computed



Cosine rule

$a^2 = b^2 + c^2 - 2bc \cos A$

$b^2 = a^2 + c^2 - 2ac \cos B$

$c^2 = a^2 + b^2 - 2ab \cos C$

Sin Rule $= \frac{a}{\sin A} = \frac{b}{\sin B} = \frac{c}{\sin C}$

Traingulation

→ It is the process of measuring the angles of a chain or network of traingles formed by stations.

Mine Machinery

→ Centre to centre distance b/w two spur gears

$$\frac{PDG_1 + PDG_2}{2}$$

$$\frac{(N_1 + N_2)m}{2}$$

PDG_1 & PDG_2 = Pitch Dia of Gear 1 & 2

m = module of gear

N_1 & N_2 = No. of teeth in Gear 1 & 2.

→ Centrifugal Ratio

$$= \frac{\text{Cent. force}}{\text{Weight of vehicle}}$$

$$= \frac{mv^2/R}{mg} = \frac{v^2}{gR}$$

→ Input power (W)

$$= \sqrt{3} \times PF \times VI$$

PF = Power factor $= \cos\phi$

V & I = Voltage & current

$P = F \times v$ W or J/s

$F \times \frac{D}{T}$

= Work done per unit time

Mine Pumps

Mine pumps works on bernoulli's thearm which states

'When a fluid flows through a passage of varying cross section, total energy of moving stream remains constant, assuming no friction losses

Pump laws

i) $Q \propto N$ (rpm) or D (imp. dia)
↓
Quantity of water

ii) $P \propto N^2$ (rpm) or D^2
↓
Pressure / Head developed by each impeller

iii) (BHP) $P \propto N^3$ or D^3
↓
Power required

→ η (Efficiency) = $\frac{\text{Power Output}}{\text{Power Input}}$

→ Mechanical efficiency of pump = $\frac{\text{HP in water}}{\text{HP input to pump shaft}}$

→ Darcy formula for head loss due to friction

$$= H_f = \frac{4fLV^2}{2gd}$$

V = vel. of flow (m/s)

f = coeff of friction

d = pipe dia (m)

L = pipe length (m)

Head

Head is a measurement of the height of a liquid column that the pump could create from the kinetic energy imparted to the liquid

$$Q = AV \rightarrow m/s$$

Q → m^3/s; A → cr. sec. area (m^2)

Specific speed

- Identifies geometric similarity of pumps

$$N_s = \frac{N Q^{0.5}}{H^{0.75}}$$

Q = capacity at best efficiency point at max. imp. diameter (gpm)

N → pump speed (rpm)

H = Head per stage at BEP at max. imp. dia (ft)

Manometric efficiency

$$= \frac{\text{Mano. Head (H)}}{\text{Work head imparted by the rotor on the fluid / Euler head}}$$

$$= \frac{H}{(VU/g)}$$

U = tangential velocity of the imp. at outlet $\left(\frac{\pi D N}{60}\right)$

V = whirl velocity at outlet

~~V = U = V~~

Man. eff. = $\frac{\text{Actual Head}}{\text{Theoritical head}}$

Locomotive

→ Limiting gradient against the load for the loco transportation is 1 in 15 but generally adapted on gradients milder than ~~1 in 25~~ gradients.

Diesel Locomotives

- Locos used in u/g coal mines have the power unit in a flame proof enclosure as a

Safeguard against ignition of firedamp

- Max. permissible CO in exhaust gases is 0.2%
- Max. permissible angle for conveying coal in PVC belt conveyor is 16°

Gear Calculation

Diametral pitch → describes gear tooth size

$$P = \frac{N}{D}$$

N → No. of teeth

D → pitch dia.

Pitch dia → Dia. of pitch circle

$$D = \frac{N}{P}$$

N → No. of teeth

P → diametral pitch

Gear Ratio = $\frac{\text{No. of teeth on driven gear}}{\text{No. of teeth on drive gear}}$

Velocity of Gear

$$= \frac{\pi D N}{60} \text{ (m/s)}$$

→ Circular pitch

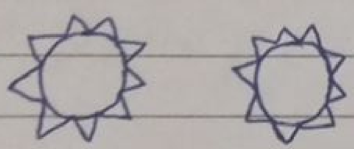
$$= \frac{\text{Circumference of pitch circle}}{\text{No. of teeth}}$$

No. of teeth $= \frac{\text{Circumference of pitch circle}}{\text{Circular pitch}}$

Circular pitch →
dist. between two consecutive teeth, centre to centre, as measured along the pitch circle

Module → $\frac{D}{N}$ (Pitch Dia)
N → No. of teeth

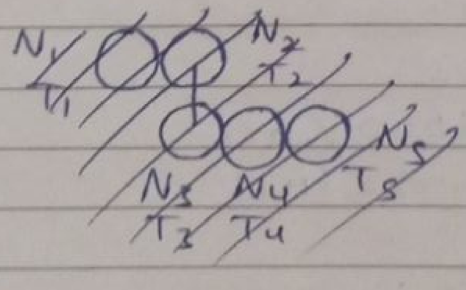
Gear trains



D_2, T_2, N_2

$\frac{N_1}{N_2} = \frac{D_2}{D_1} = \frac{T_2}{T_1}$

D_1, t_1, N_1 → speed (rpm)
↳ No. of teeth
Pitch dia



$= \frac{N_1}{N_2} \times \frac{N_3}{N_4} \times \frac{N_4}{N_5}$

$= \frac{T_2}{T_1} \times \frac{T_4}{T_3} \times \frac{T_5}{T_4}$

$N_2 = N_3$

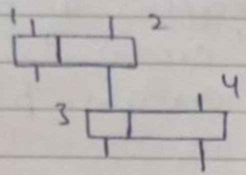
$\frac{N_1}{N_5} = \frac{T_2}{T_1} \times \frac{T_4}{T_3} \times \frac{T_5}{T_4}$

Vel. Ratio $= \frac{\text{Sp. of follower wheel}}{\text{Sp. of driver wheel}}$

$$\frac{N_2}{N_1} = \frac{T_1}{T_2} \qquad \frac{N_3}{N_1} = \frac{N_2}{N_1} \times \frac{N_3}{N_2}$$

$$\frac{N_3}{N_2} = \frac{T_2}{T_3} \qquad = \frac{T_1}{T_2} \times \frac{T_2}{T_3} = \frac{T_1}{T_3}$$

~~Comp~~ train



$$\frac{N_2}{N_1} = \frac{T_1}{T_2}$$

$$\frac{N_4}{N_3} = \frac{T_3}{T_4}$$

$$\frac{N_2}{N_1} \times \frac{N_4}{N_3} = \frac{T_1}{T_2} \times \frac{T_3}{T_4}$$

$$N_2 = N_3$$

$$\frac{N_4}{N_1} = \frac{T_1}{T_2} \times \frac{T_3}{T_4}$$

Ak Gorai Imp. Questions

Q3 sol

$GR_{first\ gear} = 3.81:1$

$TC \rightarrow 2.72:1$

$Diff. \rightarrow 4.11:1$

$$V = \frac{\pi DN/60}{R_1 R_2 R_3}$$

$$= \frac{3.14 \times 1.2 \times 1000/60}{3.81 \times 2.72 \times 4.11}$$

$$V = 5.3 \text{ km/h}$$

~~Q4 Sol~~

$\rightarrow$ rms torque $= \sqrt{\dfrac{\sum_{i=1}^{n} \tau_i^2 t_i}{\sum_{i=1}^{n} t_i}}$ → Torque (Nm); t_i ↳ Time period

= 196 mm

30) Drag force

$$F_d = C_d \frac{1}{2} \rho V^2 \times A$$

- C_d → drag coeff.
- ρ → fluid density
- V → speed of object relative to fluid
- A → CS area

drag force is the resistance force caused by motion of a body through a fluid, such as water or air.

Mine Pumps

→ Head - The main reason for using head instead of pressure to calculate cent. pump energy is that the pr. changes if sp. gravity of liquid will change but head remains constant

$$Pr (P) = \frac{W}{A} = \frac{\rho \times A \times H}{A}$$

$$H_{feet} = \frac{P_{(psi)} \times 231}{\rho}$$

Head is the measurement of pr. req. to successfully deliver the fluid pumped at the desired flow

$$\underset{(ft)}{H} = \frac{V^2}{2g} \, (ft/s)$$

$2g \rightarrow 32.2 \, ft/s^2$

→ Total Suction Head

Read. of gauge on the suction

i) ~~Watery place & corrosive atmos~~ Locos

→ Limiting gradient for loco transportation is 1 in 15 but generally adopted for milder than 1 in 25

Deisel Locomotives

→ Locos used in u/g mine have the power unit in a flame proof enclosure as a safeguard against ignition of firedamp.

→ Max. permissible CO in exhaust gases is 0.2%

Electric Battery Locomotive

→ Less powerful than diesel and trolley wire locomotive

→ Quiet in operation and produces no objectionable fumes & generates much less heat as comp. to deisel locos.

Mining Methods

- Max. possible faces = $3n-2$
- Min. faces available = n
- avg. no. of faces avail. = $2n-1$

n = no. of headings

Conditions for Longwall mining

Depth → Moderate to very deep

Dip → (Flat & uniform) (Dip should be less than 12°)

Deposit Shape → Tabular

Ore Strength → It should be crush under roof pressure rather than yield, preferably material that is weak and can be cut by continuous miner.

Rock Strength → Weak to moderate, must break and cave

Overhead wire locomotive

→ DC current is supplied through the overhead wire to supply power

→ Max. efficiency, high overload capacity, simple maintainence, & good reliability as compared to others.

Mining Methods - Applications

Room & Pillar

i) Ores with horizontal or flat dip
ii) Stable ore & hanging wall
iii) ~~12 Orebody upto 12m thick~~ Thick orebody

Long Hole / Blast Hole stoping

i) Steeply dipping (> 50°) or vertical deposit
ii)

Shrinkage stoping

i) Ore body of steep dip
ii) ~~thickness of orebody b/w 3-12m~~ (Thick orebody)
iii) Ore must not decompose, oxidise or close in storage for longer duration in stope
iv) Regular ore boundaries
v) Stable hangwall and footwall
vi) Ore should be of free flow characteristics

Sublevel Stoping

i) Steeply dipping ore
ii) Thick ore body (4-60m)
iii) Stable Hang wall and Footwall
iv) Competent ore body
v) Regular ore boundaries

Breast Stopping

i) Preferred for low grade ore deposits
ii) Ore & wall rocks should be strong
iii) Suitable for horizontal or mild dip & ~~upto 5m thick~~ lying at moderate depths & thickness.

Cut & Fill Stoping

i) Steeply dipping but can be applied to mildly dipping also
ii) Wall rock can be loose, fissured and almost any type.
iii) It is suitable where ground surface should be protected from subsidence
iv) High grade ore to compensate extra cost of filling
v) Ore should be weaker but should sustain the rock bolting during stoping.

Sublevel Caving

i) Olb and ore should be of weak nature
ii) Steeply dipping
iii) Massive ore body

Sublevel top slicing

i) Thick deposits of horizontal extent (Suit. thickness 6-8 m)
ii) Weak walls & olb
iii) Ground surface allowed to subside
iv) Soft ore, which is weak enough to stand without support only for a short period

Block caving

i) Steeply dipping & massive ore body
ii) Fairly regular ore bodies
iii) Surface is allowed to cave
iv) Low grade ore
v) Weak ore to cave under its own weight

* Resuing

i) Steep vein
ii) Vein is valuable enough to pay for the unprofitable work of blasting down a sizeable quantity of batten wall

Square set stoping

- Where the walls of the ore body and back of the stope are weak & do not stand without support
- High grade ore

Block caving

→ In block caving the ore is devided in large blocks & at the bottom the block is completely undercut i.e. a horizontal slot is blasted, which removes the support of the over lying ore.

→ The undercutting creates a series of fractures in the ore body which gradually affects the whole body block.

→ The ore at the lower part of the block is crushed by cracked upper portion & gives a fragmentation which allows the ore to be drawn through a network of finger raises & draw point

→ Dr. & Blasting is required only in lower portion of ore body. The upper portion caves down.

Post pillar method

- It is a variant of cut & fill deposit
- Applies to inclined deposits
- Narrow pillars are left in the ore at regular intervals, to give additional strength to prevent hanging wall to footwall closure after stoping.

→ While filling material surrounding the narrow pillars provides them with lateral support & prevents them failing through buckling under load operations.

Unbiased estimator

→ An unbiased estimator is an accurate statistic that's used to approximate a population parameter

→ If the estimator (i.e the sample mean) equals the parameter (i.e the pop. mean) than its an unbiased estimator

$$E(\mu) = E(\bar{x})$$
$$E(\mu - \bar{x}) = 0$$

~~Sub level caving~~

→ ~~Ore body is devided into~~

Sub level top slicing

→ This method is classified as a caving method as the ore caves in In top slicing the stoping unit is termed as slice

→ Ore is mined out in a series of horizontal slices by drilling & blasting beginning at the top of the orebody. Immediatly beneath ore as the ore from first slice is being taken out timber supports are erected on the floor of slice.

Properata

~~Greasta~~

Mine cars

Granby cars

- → Provided with side discharge doors hinged at the top. ~~and~~
- → Can handle heavier ore
- → One side dumping
- → Feas. for small gauge curvatures.
- → Greater capacity
- → One of these side doors are fitted with a roller at mid height and when the car passes by the side of a sloping

ramp, the roller passes over the latter, this resulting in gradual opening of the side doors and the contents are discharged.

~~Geate~~ <u>Minepumps</u>

<u>Geate 2017</u>

Mine Survey

Avg. photo Scale

Scale → $\frac{\text{Photo dist.}}{\text{Ground dist.}}$

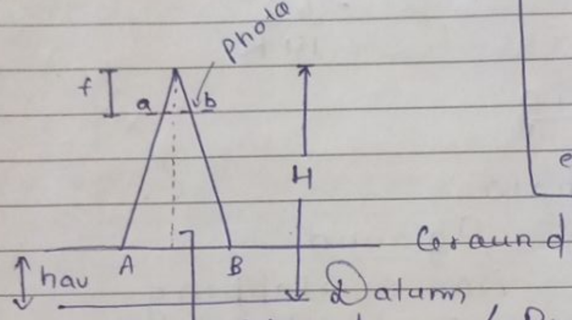
→ Correction for curvature of earth

→ Correction is subtractive

$\left(\frac{D^2}{d}\right) = 0.0785 D^2$

D → dist. b/w I.S. & S.S.

True staff Reading = Observed staff reading $- 0.0785 D^2$



Optical axis / Pr. axis of camera

Scale of photograph / Avg. photo scale $= \frac{f}{H - h_{av}} = \frac{ab}{AB}$

Scale depends upon ground elevation & it increase with elevation

If a line on the ground have diff. elevations

avg. elevation

$= \frac{h_1 + h_2 + \dots h_n}{n}$

$h_1, h_2, h_3 \dots h_n$ → ground height at n no. of points

→ Correction for Refraction :- Correction is additive $= 0.0112 D^2 \left(\frac{D^2}{7d}\right)$

Tr. Staff Reading = Observed Staff Reading $+ 0.0112 D^2$

Combined Correction

- Correction of Refraction + correction of refraction

$$= \frac{-6D^2}{14R}$$

$2 \times 1^2 \times 0.4$

$2 \quad 3^2 \quad 3 \quad 6$

Overlapping

→ No. of photos

$$= \frac{a_{ground}}{\text{Non overlapping area of photo}}$$

Ex:-

Scale of photograph = 50m → 1 cm

Size photograph = 18 cm × 18 cm

$O_L = 60\%$ $A = 110 km^2$

$O_s = 30\%$

No. of photograph

$$= \frac{110}{(1-0.6)\frac{50\times18}{1000} \times (1-0.3)\frac{50\times18}{1000}}$$

$$= 485$$

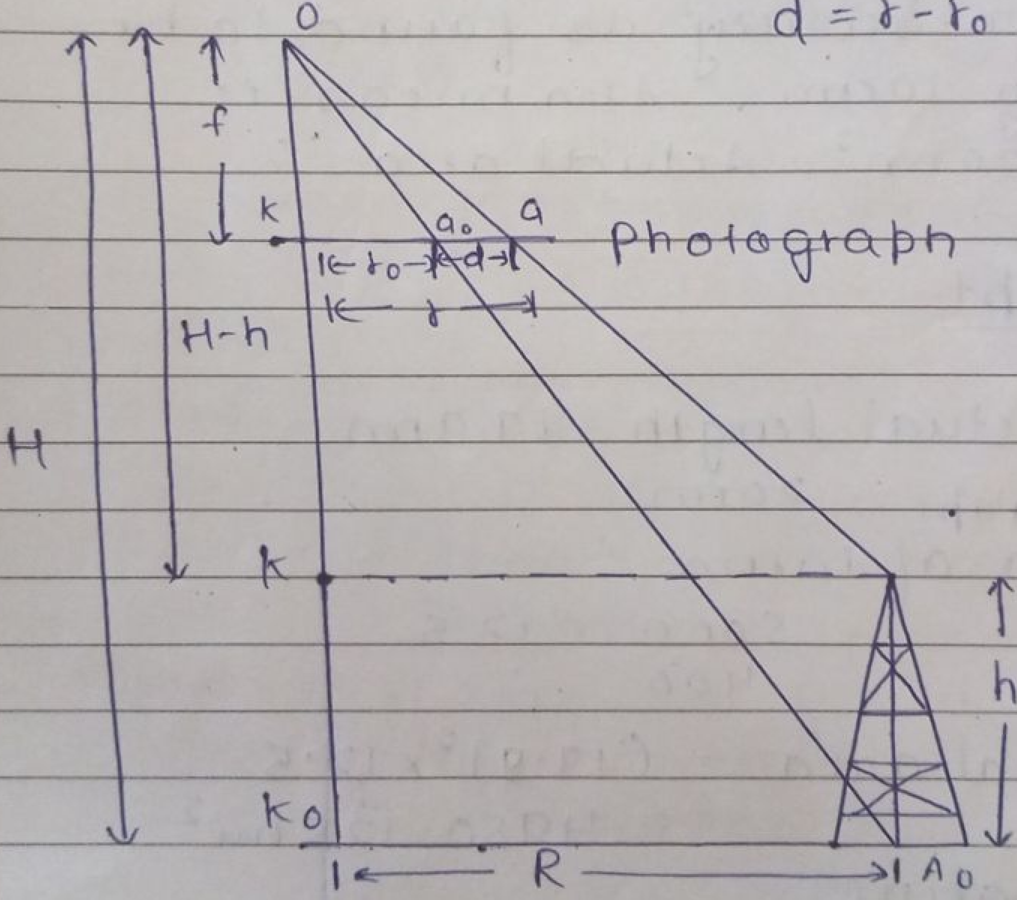
→ ~~L = Length of photo (1 − L_o) / Scale~~

~~W =~~

Relief Displacement

→ Shift in an object image position caused by its elevation above a particular datum.

$d = r - r_0$

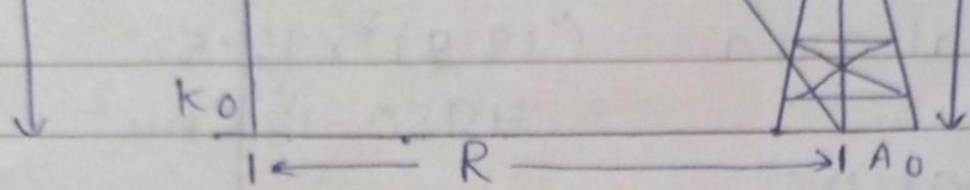


$360° \rightarrow 24\,h$

$1° \rightarrow 4$

$1' = \frac{1}{15}'$

$1'' = \frac{1}{15}''$



$1'' = \frac{1}{15}''$

Q Convert 5° 15' 25'' to hrs, min & sec

Sol

$360° \rightarrow 24 \text{ hrs}$

$(360 \times 60 \times 60)'' \rightarrow (24 \times 60 \times 60) \text{ sec}$

$5° 15' 25''$

$(5 \times 60 \times 60)'' + (15 \times 60)'' + (25)''$

$= 18925''$

$(360° \times 60 \times 60)'' = (24 \times 60 \times 60)''$

$18925'' = x''$

$x = \frac{24 \times 60 \times 60 \times 18925}{360 \times 60 \times 60}$

$x = 1261.66 \text{ sec}$

$x = \frac{1261.66}{60} = 21.027$ min

$= 0.0277 \times 60$

$= 1.66$ s

→ 21 min 1.66 s

Q A 20m long steel tape used for survey is found short by 10cm. Area meas

Rock Mechanics

→ Radial stress on the excavation boundary of circular tunnel is always zero.

→ Texture → Grain size distribution of rock.

→ In longwall caving, the thickness of immediate roof is calculated from

$d \leq d_0$

$H - d = h_{im}(k-1)$

- H → mining height
- d → Sagging of lowest uncaved strata
- h_{im} → thickness of immediate roof or caving height
- k → Bulking factor

d_0 = max. allowable sagging

$$h_{im} = \frac{H-d}{k-1}$$

if $d = d_0 = H$, the $h_{im} = 0$

if $d = d_0 = 0$

$$h_{im} = \frac{H}{k-1}$$

→ Seam thickness & Bulking factor

⇔ Harmonic mining

Bulking factor - $\dfrac{\text{Vol. of a quantity of moist granular material}}{\text{Vol. of same quantity when dry}}$

→ Bulk modulus → $\dfrac{\text{Hydrostatic pressure}}{\text{Volumetric strain}}$

Q system → Rock mass quality

$Q = \dfrac{RQD}{J_n} \times \dfrac{J_r}{J_a} \times \dfrac{J_w}{SRF}$

$\dfrac{RQD}{J_n}$ → sizes of joint rock

J_n = Joint set number (size of intact rock blocks in the rock mass)

J_a = Joint alteration number (shear strength along the discontinuity planes)

SRF = Stress reduction factor (Stress env. on the intact rock blocks)

J_r = Joint roughness number

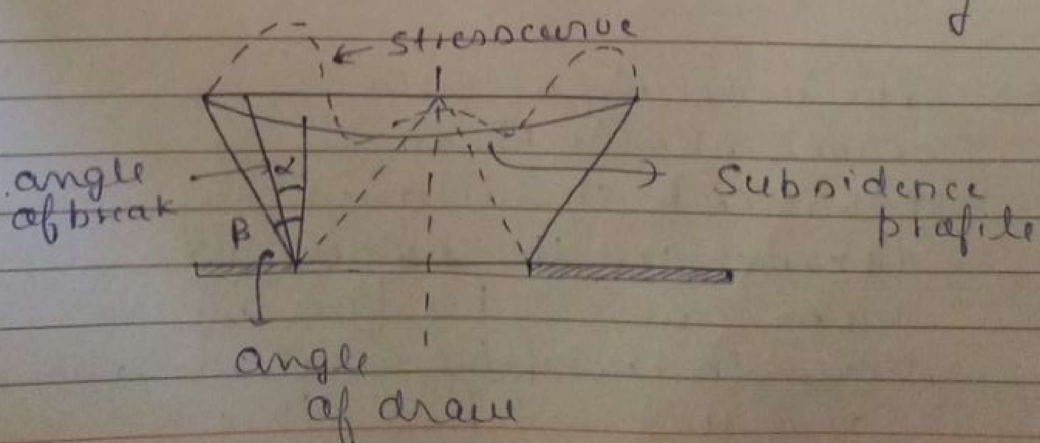
RQD = Rock quality designation

J_w = Joint water parameter

→ RMR = 9 log (Q) + 44

Q → Rock mass quality

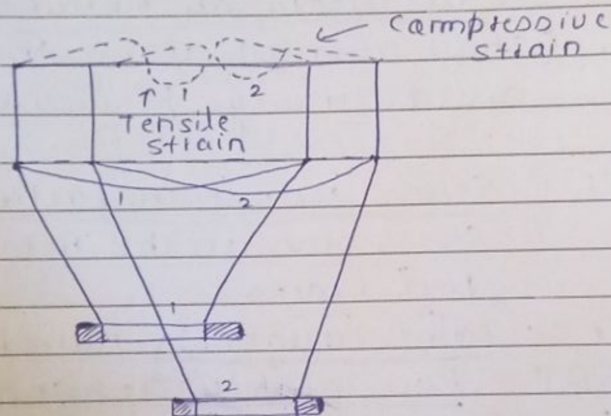
RMR → Rock mass rating



Angle of break in Subsidence

→ The vertical line at the panel edge & line connecting the panel edge and the point of max. tensile strain on the surface

Harmonic method of extraction



→ Extraction of a panel causes tensional & compressive strain at the surface

→ The working of two seams should be so advanced simultaneously to cancel out the balance of strain caused by face by the strain induced by another ~~face~~ at a different level this is called harmonic mining.

→ By using this method, subsidence is minimized or avoided
the width of extraction is decided on basis of depth & geological factors. hence a single seam can have different extraction widths at different elevations
→ bottom working has less width of extraction and topmost working has comparatively more width.

→ Dilatancy → Increase in volume due to cracking of rock

→ Bieniawski's RMR
considers the following parameters

i) RQD
ii) Spacing of Joints
iii) Condition of Joints
iv) Ground water condition
v) Uniaxial comp. strength

→ Rock bumps → Violent burst of coal pillars
due to sudden release of elastic strain energy stored in the pillars

→ Flat Jack → Meas. of stress acting

Borehole deformation gauge → meas. of insitu rock stresses using overcoring technique

Tape extensometer → Roof convergence

Borehole extensometer → used for measurement of deformation of rock mass

Borehole penetrometer - Bed separation resistance

→ The load at which an axially loaded prop reaches its elastic limit or at which it begins to buckle is called 'load bearing capacity'

→ The load on a prop when upper member begins to slide is 'yield load'

→ Freshly exposed roof after blasting is effectively & quickly supported ~~to~~ by 'Safari Supports'

→ ~~Residual stress is the stress due to weight of strata.~~

→ ~~Roof is easily cavable at the platodykonov index is 7 to 8~~

→ The fall which takes place soon after withdrawal of supports is called 'local fall'.

→ Volumetric Strain

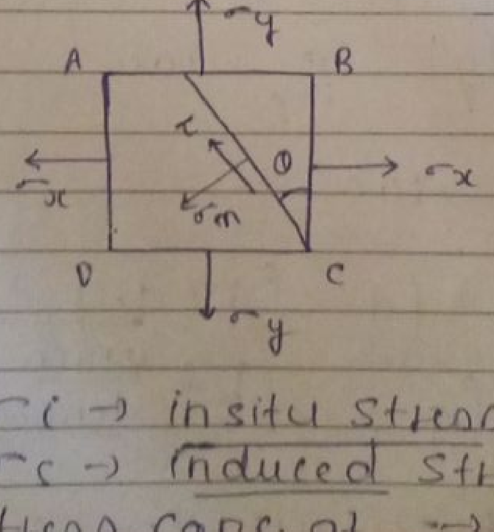
$$\varepsilon_v \text{ or } \frac{\Delta V}{V} = \varepsilon(1-2\nu)$$
$$= \frac{(\sigma_1+\sigma_2+\sigma_3)(1-2\nu)}{E}$$

ν = poisson's ratio
$\sigma_1, \sigma_2, \sigma_3$ = principal stresses

→ If principal stresses are given

$$\sigma_n = \left[\frac{\sigma_x+\sigma_y}{2}\right] + \left[\frac{\sigma_x-\sigma_y}{2}\right]\cos 2\theta$$

$$\tau = \frac{\sigma_x-\sigma_y}{2}\sin 2\theta$$



Angle of max. shear = 45 + ϕ

Angle of principle plane = $\tan 2\phi = \frac{2\tau_{xy}}{\sigma_x-\sigma_y}$

$$\phi = \frac{1}{2}\tan^{-1}\frac{2\tau_{xy}}{\sigma_x-\sigma_y}$$

→ If σ_i → insitu stress
σ_s → induced stress
Stress conc. at that point → $\frac{\sigma_s}{\sigma_i}$

→ adherence Strength (Shear Strength) $= \frac{P}{\pi D L}$ (Load)

→ Borehole extensometer

→ Borehole extensometer is a type of geotechnical instrument that is used to measure the stability, movement and behaviour of rock masses & soil within and adjacent to mines.

→ Void Ratio (e) → $\frac{V_{void}}{V_{solid}}$

$e = \frac{n}{1-n}$ (Porosity)

$e = \frac{wG}{S}$ → w = water content, G = Sp. gravity, S = Degree of Saturation

→ η (Porosity) $= \frac{e}{1+e} \times 100$

→ $\eta = \frac{V_v}{V} \times 100$

→ $\eta = \frac{\rho_g - \rho_d}{\rho_g}$ ρ_d = dry density, ρ_g = grain density

dry density → $\gamma_d = \frac{\gamma_b}{1+m}$ → γ_b = bulk density

→ Permeability

$q = kIA$

q = Flow through area A (m^3/day)

I = hyd. gradient $\left(\frac{dh}{dL}\right)$

k = Coeff. of permeability or hyd. conductivity (m/day)

A = Cr. sectional area (m^2)

$V = \frac{ki}{\eta}$ η → Porosity

→ Sp. gr = $\frac{W}{W-S}$

W = Weight in air
S = Weight in water

→ Elastic strain energy (U) = $\frac{P^2 L}{2AE}$

→ $V_p = \sqrt{\frac{\frac{4}{3}G + K}{\rho}}$

ρ = rock density
G = Shear modulus
K = ~~Bulk~~ modulus

→ $V_s = \sqrt{\frac{G}{\rho}}$

→ Resilience / unit volume (strain energy absorbed / unit volume)

$U = \frac{\sigma^2}{2E}$ & $R = U = \frac{\sigma^2}{2E} \times vol$

→ Vol. strain $= \frac{\Delta V}{V} = \varepsilon(1-2\nu)$

$$\frac{(\sigma_1+\sigma_2+\sigma_3)(1-2\nu)}{E}$$

$\sigma_1, \sigma_2, \sigma_3$ = pr. stresses

→ Mohr-Coulomb failure criterion

Uniaxial comp. strength (σ_c)

$$= \frac{2c\cos\phi}{1-\sin\phi}$$

$$UTS(\sigma_t) = \frac{2c\cos\phi}{1+\sin\phi}$$

c = Cohesive strength

$$\frac{\sigma_c}{\sigma_t} = \frac{1+\sin\phi}{1-\sin\phi}$$

ϕ = friction angle

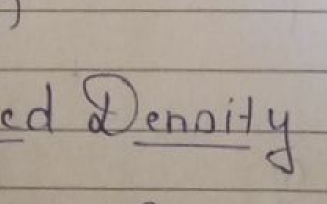
Punch Shear test

↑ Shear strength

$S = \frac{P}{\pi d t}$ (kg/cm²)

P = load applied (kg)

← Pressure



← Plate specimen

t = thickness of disc (cm)

d = dia of Puncher (cm)

Saturated Density

$$\rightarrow \frac{G+e}{1+e}\rho_w$$

$\left(\frac{W_s}{V}\right)$

$$\gamma_d = \frac{G\cdot\gamma_w}{1+e}$$

Dry Density $\left(\frac{M_s}{V}\right)$

$$\rightarrow \frac{G\rho_w}{1+e} = (1-n)G\rho_w$$

~~MPTT~~ Gate

Theory 2012

→ Placer Mining

→ Sluicing is used ~~to~~ in placer mining, is the process of using a sluice box to separate gold from gravel and olb found in placer deposits situated in streams & rivers

→ Sluicing typically follows panning. Panning is used to gather a sample from a placer deposit and sluicing is used to process larger amounts of gravel for maximum gold recovery.

→ Sluicing in its simplest form is accomplished by shovelling material and gravel directly from the bottom of the river or stream into the sluice box in which ~~is placed~~ water is directed & gold gets trapped in riffles affixed to the bottom of the box.

→ Apart from coal, longwall mining method is also used for pyrite & phosphate mining

Camman method of exploration drilling

i) Percussive with rods

Type of Bit used → Chisel shaped
Rock Formation for application → i) Sedimentary rocks of soft or medium hardness
ii) fissured formations

→ Used for lower drill depth

ii) Churn drilling or cable drilling
→ Steel choppy bits are used.
→ In placer deposits

Rotary Drilling:

i) Non coring → Tricone rock roller bits are used
→ Any rock formation except very hard

ii) Diamond drilling → Diam bits of various types
tungsten carbide bits
→ Used for larger depth
→ Any rock formation except fissured

iii) Calyx or Chilled Shot drilling
→ Used for larger dia holes.
→ Calyx bit is used
→ All rocks except the hardest

unsuitable → in soft & fissured formation

→ Nx/> Bx > Ax > Ex
NW

→ Base charge of detonators → PETN
Priming charge → ASA

→ Cast blasting

Casting of o/b by explosive or cast blasting is defined as use of blasting to move o/b directly to the spoil pile without rehandling. It is used for controlled throw of o/b.

Mine Economics

Payback period

→ It ignores time value of money

→ It is length of time required to recover the cost of an investment

→ Project is only choosen when NPV is +ve

→ For acceptance of investment project

Payback period < cutt off period

↓

predetermined length of time for an investment to be recovered

CPM - Activity Oriented technique

PERT - Event Oriented technique

Profitability Index $= \dfrac{\sum_{t=1}^{n} \frac{CF}{(1+i)^t}}{I_0}$

If PI > 1
accept the project

If PI < 1
Reject the project

→ Positive slack - indicates ahead of schedule
Negative slack - behind schedule
Zero slack - on schedule

→ In perpetual inventory system ~~is intended as an~~ material is checked when it reaches its minimum value.

Depletion Method

Capacity → 200000 T
Cost → 200000 Rs
extraction

year → 1	8000 T
2	7000 T
3	5000 T

For 1 year $200000 \times \frac{8000}{20,000} = 80000$ Rs

For 2nd year $200000 \times \frac{7000}{20,000} = 70,000$ Rs

For 3rd year $200000 \times \frac{5000}{20,000} = 50,000$ Rs

30) Depletion allowance for 1st year

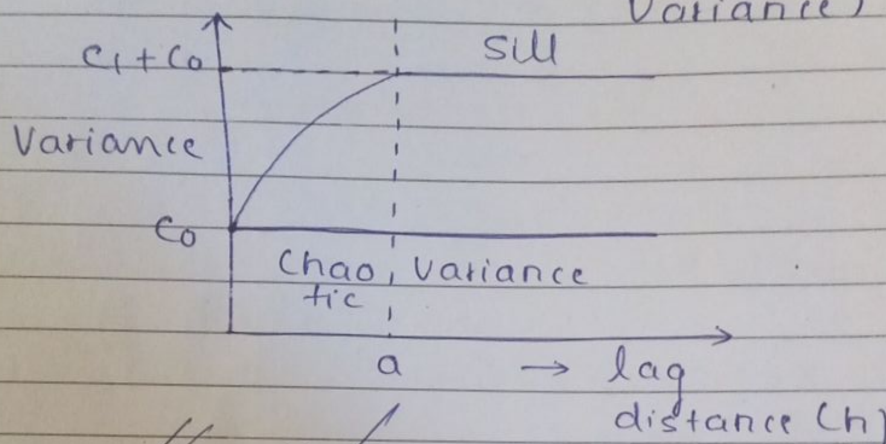
$= \frac{75000}{500000} \times 2000000$

= Rs 300,000

Applicable for waste assets.

Geostatistics

→ Sill Value → $C_0 + C_1$ (Chaotic Variance + Structured Variance)



→ Corellogram

$\rho(h) = \frac{\text{Co-variance}}{\text{Sill value}}$

→ Semi Variance

= Sill Value − Covariance

$\gamma(h) = \text{Sill} - C_0$

→ $Q = \mu \pm z \frac{\sigma}{\sqrt{n}}$ → St. deviation
(μ ↓ Mean)

$z = \frac{x - \mu}{\sigma}$
↓
St. normal variate

→ Coeff. of correlation $= R$

Coeff. of determination $= R^2$

↓

$\frac{\text{explained variation}}{\text{Total variation}}$ of the regression model b/w two variables

→ Sample covariance

$$= \sum_{i=1}^{n} \frac{(x_i - \bar{x})(y_i - \bar{y})}{n}$$

12/1/21

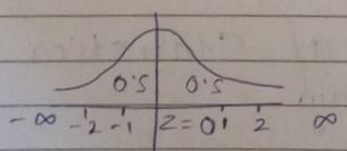
Standard normal Variable

Normal Dist. $f(x) = \frac{1}{\sigma\sqrt{2\pi}} e^{-1/2\left[\frac{x-\mu}{\sigma}\right]^2}$

In standard normal variable

$$z = \frac{x-\mu}{\sigma} \quad \mu = 0, \ \sigma = 1$$

$$f(z) = \frac{1}{\sqrt{2\pi}} e^{-z^2/2}, \quad -\infty < z < \infty$$



Confidence interval

→ The confidence interval constructed from sample data is the range of values that is likely to include the population parameter at same specified confidence level

→ Conf. interval for a popul. mean is determined by taking the sample mean, the point estimate and subtracting & adding margin of error to it.

$$\bar{x} \pm E$$

If σ is known ⟶ Critical value

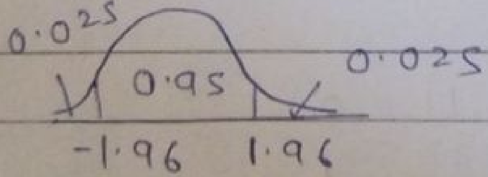
$$E = z_{\alpha/2} \frac{\sigma}{\sqrt{n}}$$

α = Significance level (1 - confid. level)

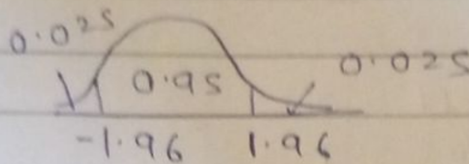
$CL = 1 - \alpha$

$CL = 0.95\%$

$\alpha = 1 - 0.95 = \frac{0.05}{2} = 0.025$



$z_{\alpha/2} = z_{0.025} = 1.96$



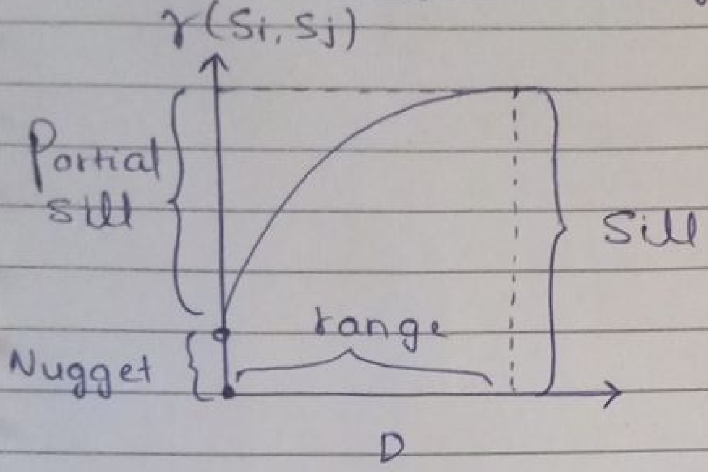
$z_{\alpha/2} = z_{0.025} = 1.96$

→ Applications of Statistics in our body

Semivariogram →

Semivariogram depicts the spatial autocorrelation of the measured sample points

→ Once each pair of locations is plotted, a model is fit through them.



→ If two locations s_i & s_j are close to each other in terms of distance meas. of $d(s_i, s_j)$ they are likely to be similar, so the diff. in their values $z(s_i) - z(s_j)$ will be small

→ If s_i & s_j are far apart, they become less similar $z(s_i) - z(s_j)$ will become larger

→ Var. of diff increases with distance

$$\gamma(s_i, s_j) = \frac{1}{2} \text{var}\left[z(s_i) - z(s_j)\right]$$

Sill → Height of semivariogram reaches when it levels off
↓
nugget effect + Partial sill

Range → dist. at which the semivariagram levels off to the sill is called range.

80 Ps = Ps + 60 + 60 − 40 Pa $(1.2) 10^2$

Mensuration

Square = Area = a^2

$P = 4a$

$D = a\sqrt{2}$

Rectangle = Area = lb

$P = 2(l+b)$

$D = \sqrt{l^2 + b^2}$

Parallelogram

$A = b \times h$

Circle = Area $= \pi r^2$

$P = 2\pi r$

Trapezium

area $= \frac{h}{2}(a+b)$

Semi circle = $A = \frac{\pi r^2}{2}$

$P = \pi r + 2r$

equilateral triangle

area $= \frac{\sqrt{3}}{4} a^2$

Right angle triangle

Area $= \frac{1}{2} \times b \times h$

Cube

$V_{cube} = a^3$

$SA = 6a^2$

(Surface area)

$diag = a\sqrt{3}$

Cuboid → $V \rightarrow lbh$

$SA = 2(lb + bh + hl)$

$diag = \sqrt{l^2 + b^2 + h^2}$

Cylinder →

$V = \pi r^2 h$

$CSA = 2\pi r h$

$TSA = 2\pi r h + 2\pi r^2$

$2\pi r(h + r)$

Cone → $V = \frac{1}{3}\pi r^2 h$

$CSA = \pi r l$

$TSA = \pi r l + \pi r^2$

$\pi r(l + r)$

Sphere → $V = \frac{4}{3}\pi r^3$ ☆

$SA = 4\pi r^2$

Hemisphere

$V = \frac{2}{3}\pi r^3$

$CSA = 2\pi r^2$

$TSA = 3\pi r^2$

Frustam

$V = \frac{1}{3}\pi h\left[R^2 + r^2 + Rr\right]$

gate 2022

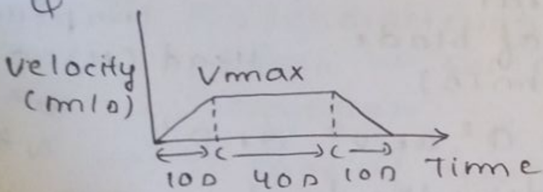
Formula sheet

i) Max. possible coal faces = $3n-2$
Min. faces available = n
avg. no. of faces avail. = $2n-1$
n = no. of headings

ii) FOS = $\frac{\text{Strength}}{\text{load}}$

iii) $180° = \pi$ rad

Q



A minewinder cage travelling 450m from pit bottom to pit top is following a 3 period duty cycle. Vmax (cage) = ?

Sol. Area of velocity time graph given distance

$$\frac{1}{2} \times 10 \times V_{max} + 40 \times V_{max} + \frac{1}{2} \times 10 \times V_{max} = 450\text{ m}$$

$$V_{max} = \frac{450}{50} = 9\text{ m/s}$$

Superelevation

(centrifugal ratio) $\frac{V^2}{gR}$ (on road)

$\frac{GV^2}{gR}$ (on railway)

V = train velocity
R = radius of curve (m)
g = acc. due to gravity
G = track gauge (dist. b/w centres of rails)

Centrifugal ratio / Impact factor = centrifugal force / weight

$$= \frac{F}{W} = \frac{\frac{mv^2}{r}}{mg}$$

$$= \frac{v^2}{gr}$$

→ law related to Q, P, N

i) $Q \propto N$ or V or D^3
V = peripheral speed of blade tips (m/s)

(H)
ii) $Wg \propto N^2$ or Q^2 or V^2 or D^2

iii) HP (req. to drive the fan) $\propto N^3$ or V^3 or Q^3 or D^5
N = rpm of fan / impeller

Pump
$Q \propto D$ or N (Dia of Impeller)
Head (H) $\propto D^2$ or N^2
BHP $\propto D^3$ or N^3

Motive column:-

$$H = \frac{NVP}{\rho d \cdot g} \quad (K)$$

$$H = \frac{\rho d - \rho u}{\rho d} \times D$$

$$H = \frac{Tu - Td}{Tu} \times D$$

% Utilization = $\frac{\text{Total hrs of actual work}}{\text{Total available hrs of work}} \times 100$

→ Properties of Eigen values

$|A - \lambda I| = 0$

In diagonal matrix,

Eigenvalues are leading diag. elements

☆ → $|A|$ = Product of E.V.

Trace = $\Sigma(\lambda)$

→ Skew symm. matrix (odd order)

$|A| = 0$ (Singular)

atleast 1 E.V = 0

→ E.V/characteristic roots/latent roots → sol. of characteristic equation

→ In case of traingular matrix, leading diagonal elements are eigen values

Q $A = \begin{bmatrix} 3 & 2 & 5 \\ 2 & 2 & 1 \\ 1 & 5 & 4 \end{bmatrix}$

Product of EV $= 3(8-5) - 2(8-1) + 5(10-2)$

$= 3(3) - 2(7) + 5(8)$

$= 9 - 14 + 40 = 35$

⇒ Range kutta method (4th order)

Given ODE

$\frac{dy}{dx} = f(x, y)$

$y(x_0) = y_0$

To find $y(x_1)$

RKM → $y_{n+1} = y_n + \frac{1}{6}[k_1 + 2k_2 + 2k_3 + k_4]$

$k_1 = h f(x_n, y_n)$

$k_2 = h f\left(x_n + \frac{h}{2}, y_n + \frac{k_1}{2}\right)$

$k_3 = h f\left(x_n + \frac{h}{2}, y_n + \frac{k_2}{2}\right)$

$k_4 = h f(x_n + h, y_n + k_3)$

$x_1 = x_0 + h$

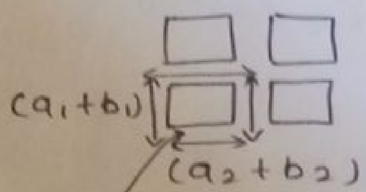
$x_2 = x_1 + h$

→ Elastic Potential energy of spring $= \frac{1}{2} k x^2$

↓ k = spring stiffness (N/m)

→ Percentage of extraction

$= \dfrac{\text{Vol. of coal extraction during development}}{\text{Total insitu coal}}$



$R = \dfrac{(a_1+b_1)(a_2+b_2) - a_1 a_2}{(a_1+b_1)(a_2+b_2)}$

$R = 1 - \dfrac{a^2}{(a+b)^2}$

Tributory area $= (a_1+b_1)(a_2+b_2) - a_1 a_2$

⇒ load acting on pillar :-

$P = \dfrac{\gamma D}{1-R}$ or $\gamma D \dfrac{(a+b)^2}{a^2}$

R ↓ Percentage of extraction

D = Depth of pillar from surface

a = length & breadth of pillar

b = width of galleries

γ = weight / unit vol. of superincumbent rock

Strength of pillar :-

$$S = k \frac{w^{\alpha}}{L^{\beta}}$$

$\alpha = 0.46$

$\beta = 0.66$

S → lb/in²

k → 1320 lb/in²

1 lb/in² = 0.0703 kg/cm²

1 m = 3.281 ft

w = Width of pillar (ft)

L = Height of pillar (ft)

→ $FOS = \frac{\text{Strength of pillar (S)}}{\text{load acting on pillar (P)}}$

Rock Mechanics

$$\gamma_d = \frac{\gamma_b}{1+m}$$

γ_d → Dry unit weight

γ_b → bulk unit weight

m → moisture content

→ $RMR = 9 \ln(Q) + 44$

Q → Rock mass quality

→ $\text{Sp. gravity} = \frac{W}{W-S}$

W → weight in air

S → weight when suspended in water

→ $n = \frac{e}{1+e}$

n (Porosity), e (void ratio)

→ $\rho_d = \frac{G\rho_w}{1+e} = (1-n) G \rho_w \quad \left(\frac{M_{solid}}{V}\right)$

$\gamma_d = \frac{G\gamma_w}{1+e} \quad \left(\frac{W_{solid}}{V}\right)$

$\rho_{saturated} = \frac{G+e}{1+e}\rho_w$

$\gamma_{sat} = \frac{G+e}{1+e}\gamma_w$

ρ_w, γ_w → density / unit weight of water

✪ Simpson 1/3 rule

$$\int_{x_0}^{x_0+nh} f(x)\,dx = \frac{h}{3}\left[(y_0+y_n) + 4(y_1+y_3+y_5+\cdots) + 2(y_2+y_4+y_6+\cdots)\right]$$

Q $\int_{0.5}^{1.5} \frac{dx}{x}$ using simpson's rule with 3 point function exceeds exact value by

Point function:- $x_0 \to y_0$, $x_1 \to y_1$, $x_2 \to y_2$

sub interval = 2 ↓ even (simpson 1/3 rule)

exact value

$$I = \int_{0.5}^{1.5} \frac{dx}{x} = (\ln(x))_{0.5}^{1.5} = 1.0986$$

$y = \frac{1}{x}$

x	0.5	1	1.5
y	2	1	2/3
	y_0	y_1	y_2

$$= \frac{0.5}{3}\left[(2.66) + 4(1)\right] ~~\cancel{+2}~~$$

$= 1.11$

$= 0.0114$ Ans

Q $I = \int_0^4 \sqrt{x}\,dx$ with 2 subintervals

$y = \sqrt{x}$

x	0	2	4
y	0	1.414	2
	y_0	y_1	y_2

$$= \frac{2}{3}\left[(2) + 4(1.414)\right]$$

$= 5.104$

Queuing theory

M/M/1 Queuing system

$\rho = \frac{\lambda}{\mu}$ → arrival rate / → service rate

Traffic density (proportion of time the server is busy)

$P_0 =$
(Prob. of 0 units in the system (service unit is idle))

$1 - \frac{\lambda}{\mu}$

Prob. of more than k units in the system

$P_{n>k} = \left(\frac{\lambda}{\mu}\right)^k$

W_s (expected waiting time / customer in the system)

$= \frac{1}{\mu - \lambda}$

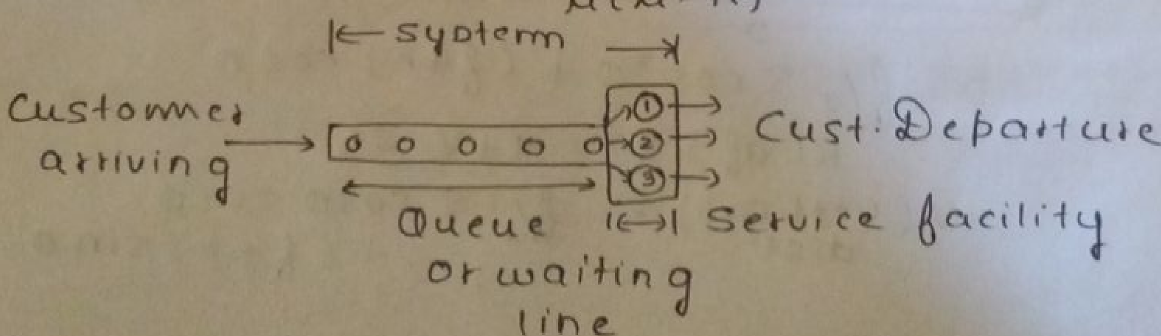
W_q (exp. waiting time / customer in the queue)

$= \frac{\lambda}{\mu(\mu - \lambda)}$

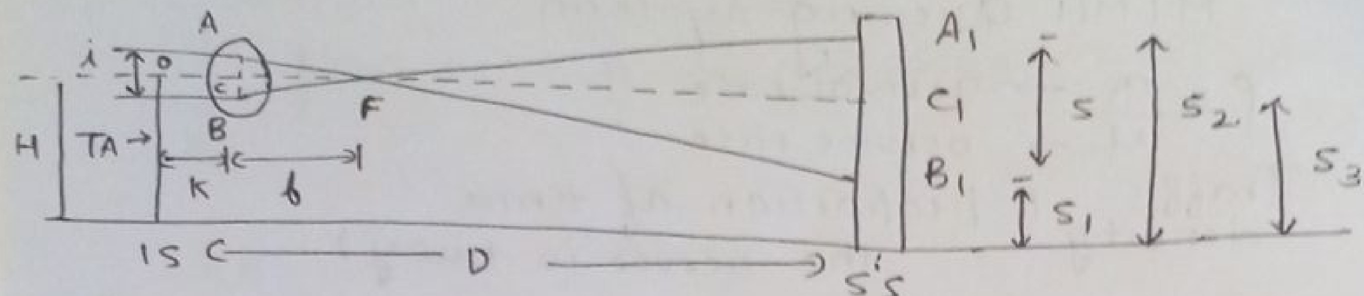
L_s (expected no. of customers in the system / length of system)

$= \lambda \cdot W_s = \frac{\lambda}{\mu - \lambda}$

$L_q = \lambda \cdot W_q = \frac{\lambda^2}{\mu(\mu - \lambda)}$



Tacheometric Surveying



f = focal length of object glass

i = dist. b/w upper & lower stadia hairs

S = staff intercept $(S_2 - S_1)$

D = horizontal dist. b/w IS & SS

S_3 = axial hair reading

K = Dist. of trunnion axis from object glass

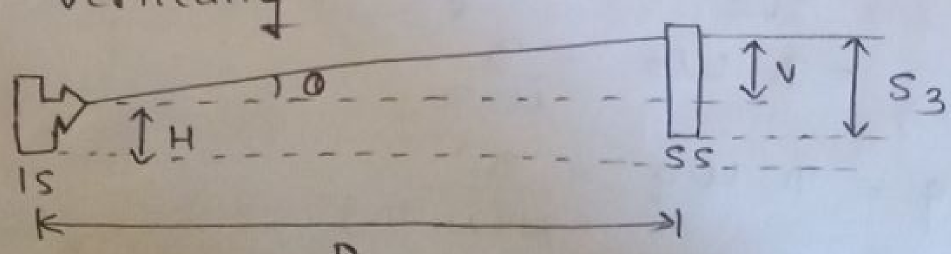
$$D = \frac{f}{i} S + (f + K)$$

f/i = multiplying constant = 100

$(f+K)$ = additive constant = 0 (for anallatical lens)

RL of SS = HOI − S_3

i) when line of sight is inclined to the horizontal and the staff is held vertically



$$D = \frac{f}{i} S \cos^2\theta + (f + K)\cos\theta$$

RL of SS = H + V − S_3

V (vertical dist.) = $\frac{f}{i} S \cos\theta \sin\theta + (f + K)\sin\theta$

(1000) SG

Cancentration af sand by weight (%)

$$= \frac{0.91V}{0.91V + 0.65V} \times 100$$

Imp $= 58.33\%$ Ans

* Conversions

1 Bar $= 10^5$ Pa

1000 lit $= 1\ m^3$

1 Gallon $= 0.00379\ m^3$

1 Darcy $= 9.86 \times 10^{-9}\ cm^2$

1 mm wg $= 9.81$ Pa

1 mm Hg $= 133.32$ Pa

Standard atmospheric Pressure $= 1.013 \times 10^5\ N/m^2$

1 Weisbach/

1 kilomurg = $9.81\ Ns^2m^{-8}$

Match factor (Truck shovel comb.) = $\frac{\text{Truck arrival rate}}{\text{loader service rate}}$

(1) On an old plan of scale 1 : 1000, lead hold area of a mine is now meas as 802 cm^2 using a planimeter

$\frac{d}{dx}(e^x) = e^x$

$\frac{d}{dx}(a^x) = (\ln a)\, a^x$

$\frac{d}{dx}(\sin^{-1}x) = \frac{1}{\sqrt{1-x^2}}$

$\frac{d}{dx}(\tan^{-1}x) = \frac{1}{1+x^2}$

$\frac{d}{dx}(\sec^{-1}x) = \frac{1}{|x|\sqrt{x^2-1}}$

$\frac{d}{dx}(uv) = u\frac{dv}{dx} + v\frac{du}{dx}$

$\frac{d}{dx}\left(\frac{u}{v}\right) = \frac{v\frac{du}{dx} - u\frac{dv}{dx}}{v^2}$

Integration

$\int x^n dx = \frac{x^{n+1}}{n+1} + c \; (n \neq -1)$

$\int \sin x\, dx = -\cos x + c$

$\int \cos x\, dx = \sin x + c$

$\int \sec^2 x\, dx = \tan x + c$

$\int \csc^2 x\, dx = -\cot x + c$

$\int \sec x(\tan x)\, dx = \sec x + c$

$\int \csc x \cot x\, dx = -\csc x + c$

$\int \frac{1}{x} dx = \ln|x| + c$

$\int e^x dx = e^x + c$

$\int a^x dx = \frac{a^x}{\ln a} + c \; (a > 0, a \neq 1)$

$\int \frac{1}{\sqrt{1-x^2}} dx = \sin^{-1}x + c$

$\int \frac{1}{1+x^2} dx = \tan^{-1}x + c$

$\int \frac{1}{|x|\sqrt{x^2-1}} dx = \sec^{-1}x + c$

A.P.

Sum of first n terms $= \frac{n}{2}[2a + (n-1)d]$

n = no. of terms

a = first term

d = common difference

General form (AP)

$a, a+d, a+2d, \ldots$

→ n^{th} term of AP series

$T_n = a + (n-1)d$

d = common diff

n = n^{th} term

→ If last term l is given, sum of n terms → $S_n = \frac{n}{2}(a+l)$

→ Sum of first n natural numbers $= \frac{n(n+1)}{2}$

n = no. of natural numbers

→ Sum of squares of first n natural no. $= \frac{n(n+1)(2n+1)}{6}$

→ Sum of first n odd no $= n^2$

→ Sum of first n even numbers $= n(n+1)$

→ No. of terms in AP

$n = \left(\frac{l-a}{d}\right) + 1$

l = last term
a = first term
d = common diff

→ If a, b, c is given in an AP series

$b = \frac{a+c}{2}$, $b - a = c - b$

GP

n^{th} term $T_n = ar^{n-1}$ r = common ratio

$a, ar, ar^2, \ldots$

2 4 8 16 32 $r = 2$

→ S_n (sum of n terms)

$= \frac{a(r^n - 1)}{r - 1}$

a, b, c

$b^2 = ac$

$b = \sqrt{ac}$

→ Infinte GP $(S_\infty) = \frac{a}{1-r}$

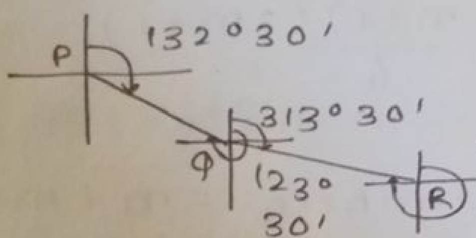
Q

Line	FB	BB
PQ	132°30'	313°30'
QR	123°30'	303°30'
RS	182°30'	2°15'
ST	288°45'	108°0'

Stations free from local attraction

<u>Concept</u>

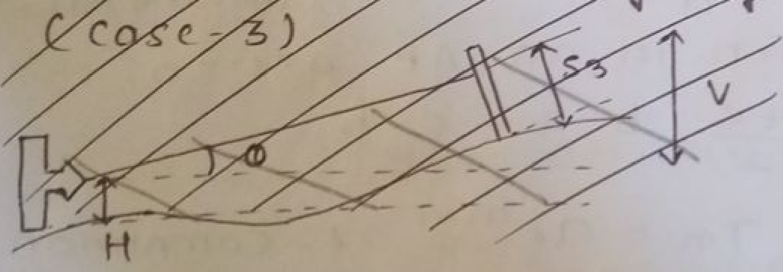
If the difference b/w FB & BB of a line is 180°, the two end stations of that line are free from local attraction



QR = 303°30' − 123°30'
= 180°

Q & R are free from local attraction

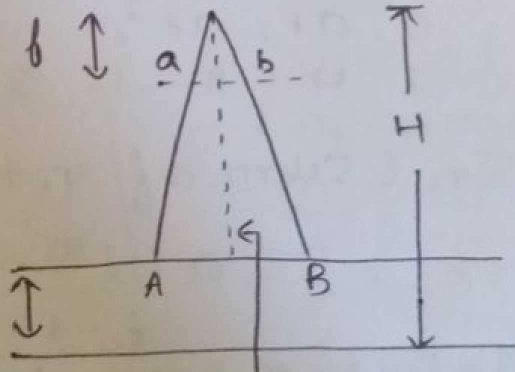
Tacheometric Surveying
(case-3)



Aerial Photogrammetry

Scale = $\frac{\text{Photo dist.}}{\text{ground dist.}}$

$\frac{ab}{AB} = \frac{f}{H-h}$



Scale of photograph/
Avg. photo scale
(depends on ground elevation) $= \frac{f}{H-h_{av}}$

Optical axis / Principal axis of camera

avg. elevation of ground $= \frac{h_1+h_2+\cdots h_n}{n}$

n = ground height at n no. of points

No. of Photographs

No. of Photos = $\dfrac{A_{ground}}{\text{Non overlapping area of photo}}$

Ex:-

Scale of Photograph = 1 cm → 100 m

Size photograph = 23 × 23 cm

$A = 150\ km^2$

$O_{long.} = 60\%$

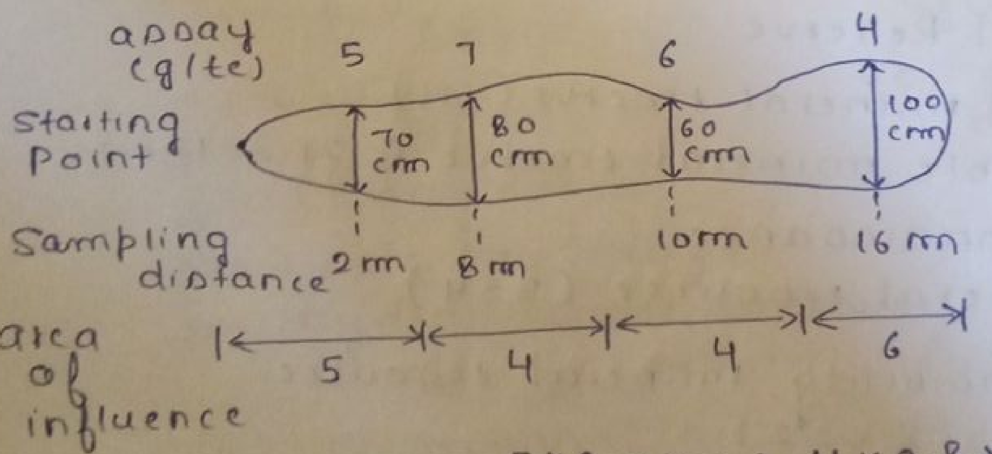
$O_{side} = 30\%$

$$N = \frac{150}{(1-0.6)\frac{100\times 23}{1000} \times (1-0.3)\frac{100\times 23}{1000}}$$

$$= \frac{150}{0.92\times 1.61} = 101.269$$

Q The chip sampling data, spaced irregularly for a gold vein deposit

mean assay value (g/te) = ?



$$= \frac{5\times 0.7\times 5 + 4\times 0.8\times 7 + 4\times 0.6\times 6 + 6\times 1\times 4}{5\times 0.7 + 4\times 0.8 + 4\times 0.6 + 6}$$

= 5.18 g/te

→ Expected total error
(same error)
$$= \sqrt{e^2+e^2+e^2} = \sqrt{e^2 n} = e\sqrt{n}$$

→ Correction for curvature of earth
Correction is subtractive
$0.0785 D^2$
↓ Dist. b/w IS & SS
True staff reading = observed staff reading $- 0.0785 D^2$

→ Correction for refraction
Corr. is additive
$0.0112 D^2$
$TSR = OSR + 0.0112 D^2$

→ Combined correction = − corr. of curvature + corr. of refraction
$= \frac{6D^2}{14R}$ → Radius of earth
(subtractive)

→ Correlation Coefficient
r is the corr. coeff. It is always b/w 1 & −1
−1 → Points are on a perfect straight ~~lines~~ line with −ve slope
+1 → Points are on a perfect straight line with +ve slope

→ coeff. of determination $= r^2$
X, Y (variables) → degree of relationship
↳ Corr. coeff. r

→ Types of correlation
+ve corr. → x↑ y↑
−ve corr. → x↑ y↓, y↑, x↓
no corr. → No change in y with x

Perfect correlation → r = 1

→ Methods of Correlation

i) Coeff. of correlation (Karl Spearman's)

ii) Rank correlation

x	y
x_1	y_1
x_2	y_2
⋮	⋮
x_n	y_n

$$\bar{x} = \frac{\Sigma x_i}{n} \qquad \bar{y} = \frac{\Sigma y_i}{n}$$

n = no. of data points

$$r = \frac{\Sigma(x-\bar{x})(y-\bar{y})}{\sqrt{\Sigma(x-\bar{x})^2}\sqrt{\Sigma(y-\bar{y})^2}}$$

$$r = \frac{n\Sigma uv - \Sigma u \times \Sigma v}{\sqrt{n\Sigma u^2 - (\Sigma u)^2}\sqrt{n\Sigma v^2 - (\Sigma v)^2}}$$

$u = x - A$ A = assumed mean for x values

$v = y - B$ B = assumed mean for y values

assumed mean = no. closest to arithmetic mean

Correlation

→ ~~Corellation~~ coeff.

$$= \frac{E(xy) - (E(x) \times E(y))}{\sqrt{(E(x^2) - (E(x))^2 (E(y^2) - (E(y))^2)}}$$

⇒ Relative humidity = $\dfrac{\text{mass of water vapour/m}^3 \text{ of air}}{\text{mass of water vapour required to saturate 1 m}^3 \text{ of air}}$

⇒ Reynold's number

$Re \leq 2000$ (laminar flow of air)

$Re > 4000$ (turbulent air flow)

$2000 < Re < 4000$ (Transistional flow of air)

Mean value / expected value
$= \sum x \, \cancel{f(x)} \; p(x)$

Mean sq. value $= \sum x^2 \, \cancel{f(x)} \; p(x)$

UNFC (United Nations Framework classification)

UNFC consist of following 3 axis :-

1) Geological assessment
ii) Feasibility assessment
iii) economic viability

→ heighest category of resources under UNFC system will have code (111) & lowest category code (334)

Total mineral resouice

(1) Measured mineral resource (331)
(ii) Indicated mineral resource (332)
(iii) Inferred mineral resource (333)

Mineral Reserve

(i) Proved mineral reserve (111)
(ii) Probable mineral reserve (121 & 122)
~~(iii)~~ Reconnaissance mineral resource (334)

Prefeasibility mineral resource (221 & 222)

Feasibility mineral resource (211)

→ k = coeff. of thermal conductivity
c = Specific heat of rock sample
ρ = density of rock sample

$$\text{Thermal diffusivity} = \frac{k}{\rho c}$$

→ Coeff. of variation = $\frac{\sigma}{\mu}$

$$\text{Sample St. deviation} = \sqrt{\frac{\Sigma (x-\mu)^2}{n-1}}$$

→ $$\frac{\sigma_H}{\sigma_v} = \frac{\nu}{1-\nu}$$

→ Swell factor = Swell factor of soil is the amount of volume increase from the undisturbed to the excavated state due to air pockets created

Rock mechanics

→ $$\text{Bulking factor of rock } (K) = \frac{\text{Vol. of rock after blasting}}{\text{Vol. of rock before blasting}}$$

→ $$\phi = KA\left(\frac{dh}{dL}\right)$$ (Darcy's law)

↓ Flow rate (m^3/day)

K = hydraulic conductivity (coeff. of permeability) (m/day)

A = Cr. sectional area (m^2)

$\frac{dh}{dL}$ = hydraulic gradient

~~→ Moisture content = $\frac{W_{water}}{W_{solids}} \times 100$~~

→ $$\text{Void Ratio } (e) = \frac{V_{voids}}{V_{solids}}$$

γ_d (dry unit weight)

$= \frac{W_d}{V}$ (weight of solid)

→ Centrifugal tension in belt

T_c ~~drive~~ $= mv^2$

Q An air reciever of a compressor having vol. 0.5 m^3, supplies air for charging ANFO in drill holes. During charging process the absolute pressure of the air reciever falls from 900 to 700 kPa. Assuming the entire process is isothermal, the vol. of air supplied by the reciever at 100 kPa ambient pressure (m^3) is

Sol

Boyle's law

$V \propto \frac{1}{P}$ if temp. is constant

$V_1 P_1 = V_2 P_2 = C$

initial / final absolute pressure & volume

Charle's law

$V \propto$ absolute temp.

if pressure is constant

$$\frac{V_2}{V_1} = \frac{t_2}{t_1} = \frac{273 + c_1}{273 + c_2} = c$$

t ↓ absolute temp.

Graham's law of diffusion

$R \propto \frac{1}{\sqrt{D}}$ → Relative densities of gases

↓

Relative rates of diffusion of gases

mmwg = kg/m²

$\frac{R_A}{R_B} = \sqrt{\frac{D_B}{D_A}}$

As per boyle's law

$P_1V_1 = P_2V_2$

let initial vol. be x

$(0.5 - x)\,900 = 700 \times 0.5$

$= 0.111$

Now at $P = 100$ kPa

$900 \times 0.111 = V \times 100$

$V = 1\,m^3$ Ans

Friction / koepe winding

$\frac{T_1}{T_2} = e^{\mu\theta}$

θ = angle of wrap

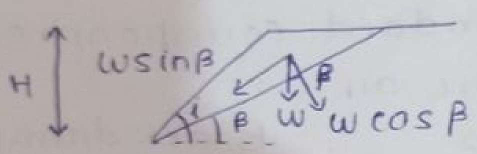
μ = coeff. of friction b/w rope & wheel

T_1, T_2 = Tension in the rope entering / leaving the wheel

Q A conveyor belt consumes 60 kW power

Determination of factor of safety of the slope

case i) when there is no tension crack & water Pressure



H = Bench height
i = overall slope angle from horizontal
β = angle of discontinuity from horizontal

W = Weight of block
c = Cohesion
ϕ = friction angle
A = Area of failure plane
(Shear strength of sliding surface is expressed in terms of cohesion (c) & friction angle (ϕ))

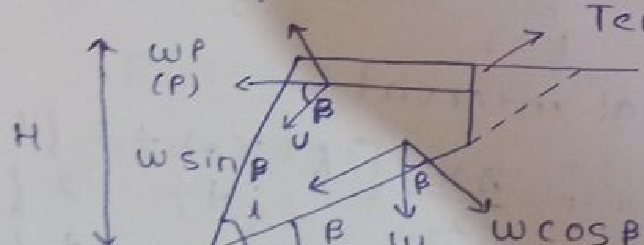
Force tending to induce sliding = W sin β

Force to resist sliding of the block = W cos β. tan ϕ + c . A

FOS = Force to resist sliding / Force tending to induce sliding

$$= \frac{W \cos\beta \tan\phi + cA}{W \sin\beta}$$

case ii) Tension crack & Water Pressure Present



Force tending to induce block sliding = W sin β + U

Water Pr. along sliding Plane (U) = P cos β

Water Pr. normal to the sliding plane (V) = P sin β

Force resisting block sliding = (W cos β − V) tan ϕ + C . A

$$FOS = \frac{C \cdot A + (W\cos\beta - V)\tan\phi}{W\sin\beta + U}$$

⇒ Absolute Weight Strength (AWS) :- theoritical absolute energy available based on ingredients of explosive (MJ/kg of exp.)

Relative weight Strength :-

$$RWS_{exp} = \frac{AWS_{exp}}{AWS_{ANFO}} \times 100$$

Absolute bulk Strength :- Energy available in unit vol. of explosive

$$ABS_{exp} \, (cal/cc) = AWS_{exp} \times \rho_{exp}$$

Relative bulk Strength :- $\frac{ABS_{exp}}{ABS_{ANFO}} \times 100$

Q Relative strength of an unknown explosive is 1.5 with respect to ANFO.

$+ 100 \times 100 \times 8$

$+ 200 \times 100 \times 12$

$= \frac{120000 + 20000 + 25600 + 84000}{800{,}000} \times 100$

$= 31.2\%$

Q Assignment Problem

T \ G	T_1	T_2	T_3	T_4
G_1	6	10	5	4
G_2	4	100	6	4
G_3	6	9	6	2
G_4	3	7	6	4

minimum cost of assignment = ?

Sol
1. Subtract the minimum value of each row from the entries of that row
2. Sub. the min. value of each column from the entries of that column

2	6	1	0
0	96	2	0
4	7	4	0
0	4	3	1

2	2	[0]	0
[0]	92	1	0
4	3	3	[0]
0	[0]	2	1

3. Row scanning/~~col~~

→ Is there exactly 1 zero in that row?
- if no, skip that row
- if yes, mark a square around that 0 and draw a vertical line passing through that 0

if all zeroes are not covered, do column scanning

4. if no. of square marked ≠ no. of rows

Identify minimum value of undeleted cell values

a) Add min. undeleted cell value at the intersection points of matrix

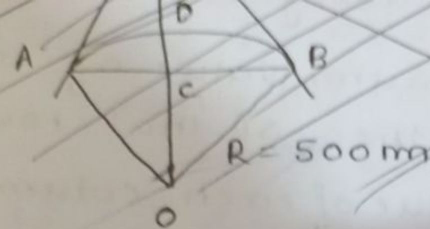
b) Subtract it from all undeleted cell values

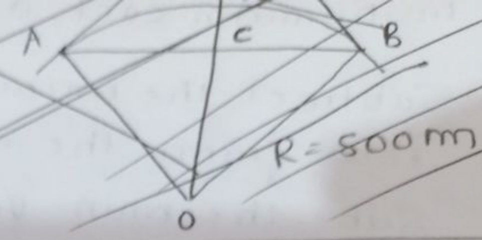
and do row/ column scanning again

= 5+4+2+7 = 18

**

Q Value of mid ordinate of curve ADB (mm) = ?





Stress Distribution around circular excavation

Radial Stress

$$\sigma_r = \frac{P}{2}\left[(1+k)\left[1-\frac{a^2}{r^2}\right] - (1-k)\left[1-\frac{4a^2}{r^2}+\frac{3a^4}{r^4}\right]\cos 2\theta\right]$$

Tangential stress:

$$\sigma_\theta = \frac{P}{2}\left[(1+k)\left[1+\frac{a^2}{r^2}\right] + (1-k)\left[1+\frac{3a^4}{r^4}\right]\cos 2\theta\right]$$

Shear Stress

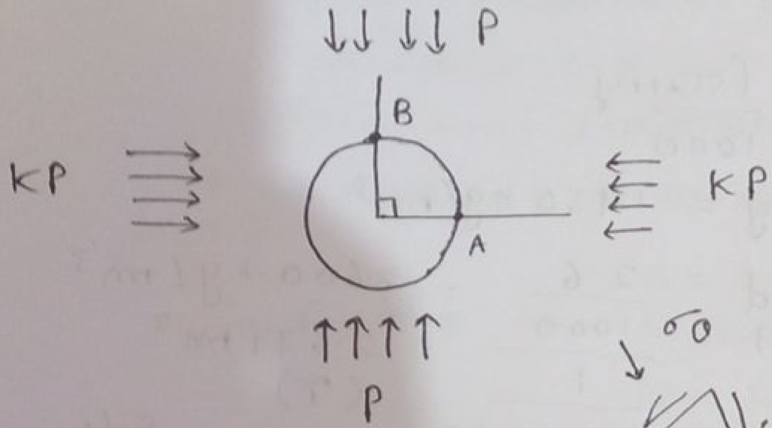
$$\sigma_{r\theta} = \frac{P}{2}\left[(1-k)\left[1+\frac{2a^2}{r^2}-\frac{3a^4}{r^4}\right]\sin 2\theta\right]$$

Stresses on the excavation boundary ($r = a$)

$\sigma_r = 0$

$\sigma_\theta = P[(1+k) + 2(1-k)\cos 2\theta]$

$\sigma_{r\theta} = 0$



Boundary stresses

(A)

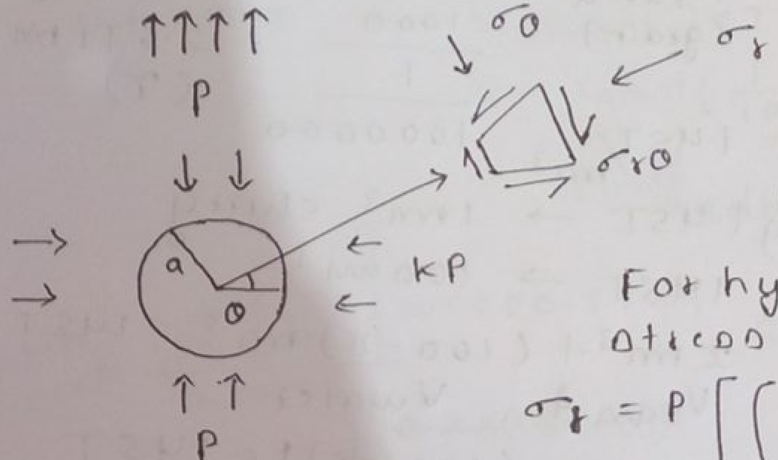
$\theta = 0$

$(\sigma_\theta)_A$

$= P(3-k)$

$B \to \theta = \frac{\pi}{2}$

$(\sigma_\theta)_B = P(3k-1)$

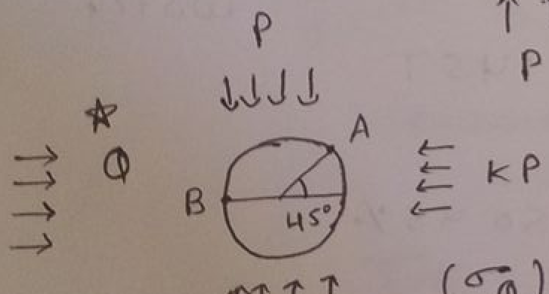


For hydrostatic stress field ($k=1$)

$\sigma_r = P\left[\left(1 - \frac{a^2}{r^2}\right)\right]$

$\sigma_\theta = P\left[\left(1 + \frac{a^2}{r^2}\right)\right]$

* Q



$(\sigma_\theta)_A = 3(\sigma_\theta)_B$

$$\frac{P[(1+k) + 2(1-k)\cos 90]}{P[(1+k) + 2(1-k)\cos 360]} = 3$$

$$= \frac{1+k}{[(1+k)+2(1-k)]} = 3$$

$= 1+k = 3(1+k+2-2k)$

$= 1+k = 3+3k+6-6k$

$= 1+k = 9-3k$

$= 4k = 8$

$k = 2$