Mineral Processing

Module - 1

(2)

3.1.19

Mineral benefication :→

Mineral	Ore
(i) Natural occuring substances of metals present in the earth crust are known as mineral.	(I) Mineral which can be used to obtain the metal profitably are known as Ores.
(ii) All Minerals are not ore.	(II) All ores are generally minerals.
e.g - clay is mineral of Aluminium.	e.g - Bauxite is ore of Al.

gangue - unwanted particle.

Liberation - To release the locked mineral part or metal part.

$$\text{Degree of Liberation} = \frac{\text{Amount released by liberation}}{\text{Total taken amount}} \times 100$$

example :- In Fe_2O_3 10 gm Fe is taken.
3 gm Fe is released

$$D.L = \frac{3}{10} \times 100 = 30\%$$

Size Reduction :→

- Crushing (Size range is high) (coarser part)
- Grinding (finer size)

Mineral Beneficiation :→ It is the 1st step of extraction of metals from natural resources. It is a process by which valuable constituents of an ore are concentrated by means of a physical separation process.

Economic Justification of mineral dressing :→

1. To purify and upgrade the ore.
2. Enhance their chemical purity and physical properties.

smelting = Melting + Roasting

3. Making Melting practice easier,
4. Savings and Freight - waste product are not transported from the mines area and hence reducing the transportation cost / freight.

5. Reduces losses of metal ~~at~~ the smelter - slag volume during the smelting process decreases. ~~slag volume during the smelting process.~~ and results in a lesser loss of metal into the slag.
6. Deduction of the total smelting cost :- lesser amount of upgrade ore is to be smelted for a particular output capacity. This reduces the fuel and energy consumption cost per unit ton of metal smelted.
7. Enhancing ~~the~~ by efficiency of unit process - ore is separated into one of more valuable products & tailing. This leads to separation of certain minerals that interfere during the smelting or leaching process.

Objective of Mineral benefication :→ 5/1/19

1) To increase the metal grade of an ore.

2) To reduce the amount of gangue content.

3) To decrease the thermal energy requirement, for the separation of metal from the gangue content.

4) To decrease the requirement of aquaous sol^n in hydrometallurgical extraction of metal.

- Specific gravity of any metal = $\frac{\rho_{metal}}{\rho_{H_2O}} = \frac{M_{metal}/V_{metal}}{M_{H_2O}/V_{H_2O}}$

$$= \frac{M_{metal} \times V_{H_2O}}{M_{H_2O} \times V_{metal}}$$

- Apparent specific gravity of any metal = $\frac{(\rho_{metal})_{Air}}{(\rho_{metal})_{H_2O}}$

Difference between Mineral grade and Metal grade :→

→ For any metal ore mineral grade is 100%, provided there is middling (50% gangue + 50% Metallic) and tailing (low metal + high gangue)

→ Metal grade of Fe_2O_3 is 70%. -Fe and rest 30% is oxygen.

- Cu grade in pure chalcopyrite ($CuFeS_2$) is ≈ 34.1%.

$$\frac{\text{Mass}_{Cu}}{\text{Mass}_{chalcopyrite}} \times 100 = \frac{63.5}{63.5+56+64} \times 100 = 34.1.$$

- Fe grade in Fe_2O_3 ≈ 70%.

$$\frac{2 \times 56}{2 \times 56 + 48} \times 100 = 70\%$$

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Zn grade in ZnS $\simeq$ 67%.

$$\frac{65.38}{65.38+32} \times 100 = 67.1\%$$

Pb grade in PbS $\approx$ 86.6%.

Al grade in Al_2O_3 $\approx$ 52.94%

$$\frac{2\times27}{2\times27+48} \times 100 = 52.94$$

e.g.- Consider a feed of 1000 kg out of which
500 kg → chalcopyrite. ($CuFeS_2$)
200 kg → Hematite (Fe_2O_3)
150 kg → Al_2O_3.
50 kg → ZnS (sphalerite).
PbS (Galene).

Find out the recovery of copper in the conc of Cu, Fe, Al, Zn, Pb etc.

Cu —— $\frac{0.341 \times 500}{1000} = 0.1706 = 17.05\%$.

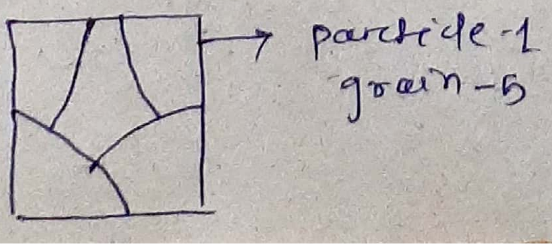
Fe —— $\frac{70 \times 200}{1000} = 14\%$.

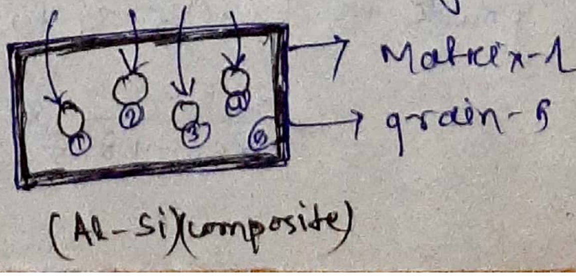
Al —— $\frac{52.94 \times 150}{1000} = 7.9\%$.

Zn —— $\frac{67.1 \times 50}{1000} = 3.35\%$.

Pb —— $\frac{86.6 \times 100}{1000} = 8.6\%$.

Qn:- What is the difference between a grain and a particle.
A grain is classified as a single Mineral whereas a particle is made of one or more mineral grains.





Importance of liberation :→

1) To liberate the locked mineral.

2) Study of the free surface area available.

P > 10 nm

Surface are more
bonding will more

with decrease in particle size, availability of surface area will increase.

3) Study of percentage of ore mineral, that can be liberated.
(% of locked coarser minerals particle)

4) Study of bond strength betn mineral and gangue.

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Unit operations of Mineral processing :→

There are 4 different unit operations.

→ Comminution.
→ Sizing
→ concentration.
→ Dewatering.

Comminution – particle size reduction either by crushing or grinding.

Sizing – Separation of particles having different sizes by screening or classification.

e.g. – If the range of size for Iron ore is 8-12 mm then the particle greater then 12mm are known as over size particle & particle having less than 8 mm is known as under sized particle.

Concentration - Separation of metals from the gangue containing on the basis of its physical, chemical and surface properties.

Dewatering - solid-liquid separation.

Sampling →

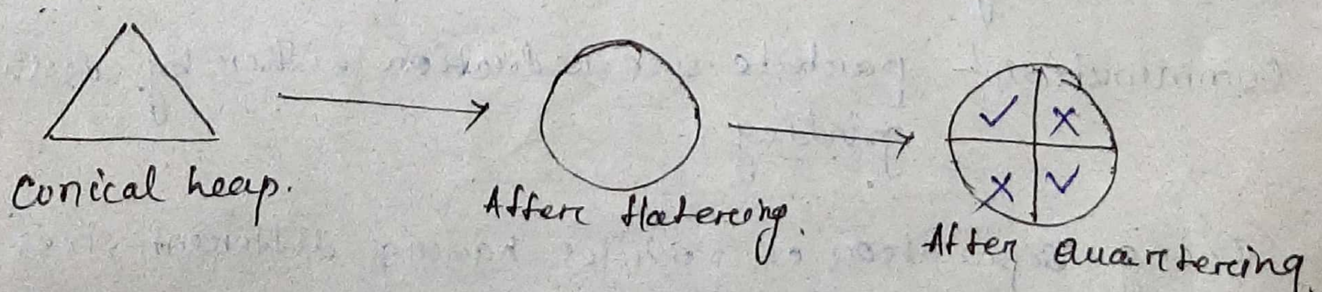
It is the process of selecting a part of a whole such that the measured value for the part (sample) is an unbiased estimate for the whole.

→ There are different methods of sampling.

1. Grab sampling :- It is done by homozening in the materials thoroughly by mixing of materials on a rolling sheet (provided the sample will not stick with the rolling sheet).

This method uses least equipment and as a higher variation from the other sampling methods.

2. Coning and Quartering :-

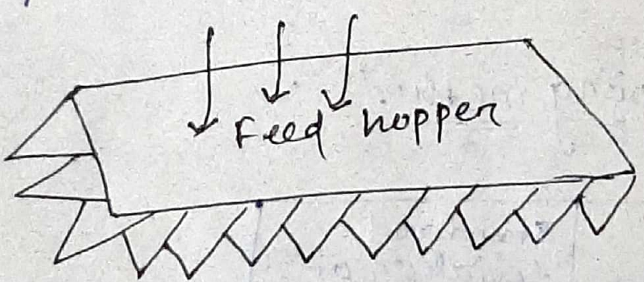


In this methode, first the material is mixed and shoveled into a uniform conical pipe/heap. Then the cone is spread from the center to form a flatened disc. Then the disc is divided into four equisized Quarter using a perpendicular board. One pair of opposite quarters is removed and the other pair is used as the sample.

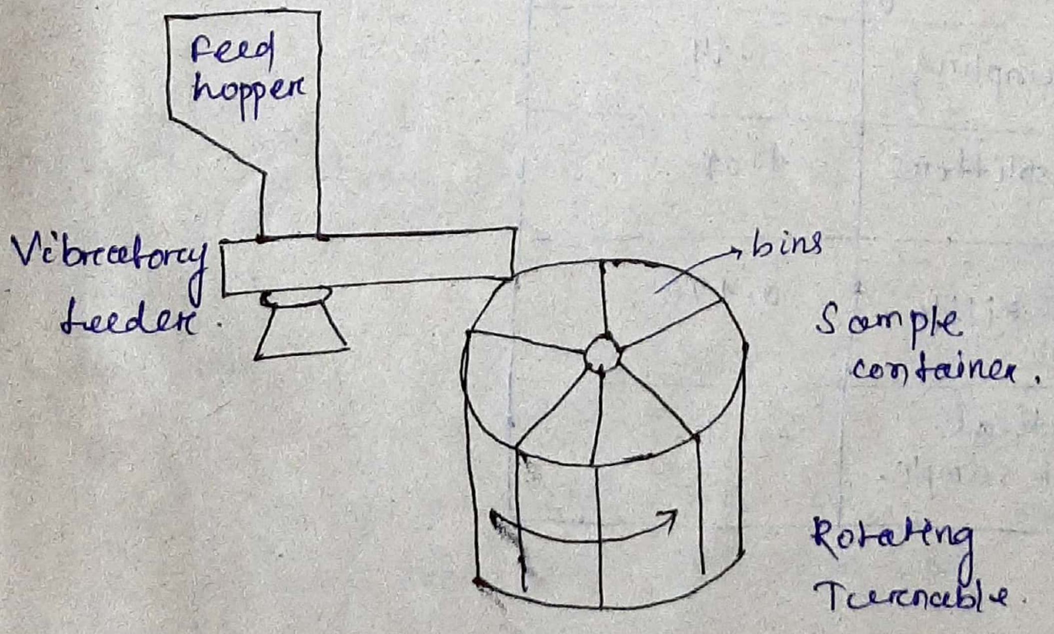
It there is large amount of materials present then sampling is coned and Quartered again and again until the desired sample sized is obtained.

3. Sample splitters :-

This is consist of a series of splitters that run in alternating directions so that when the material is poured into the top of the splitters (Feed hopper), it flows through the splitters and equally randomly divided into two equal sized fractions. One fraction can again split into two portions until a desired sample sized is obtained.



4. Rotary Riffle :-



The rotary or spinning riffles is the best methode to used for dividing the material into representative samples because it produces the lower variance between the samples and can produce a large no. of sample in a single operation.

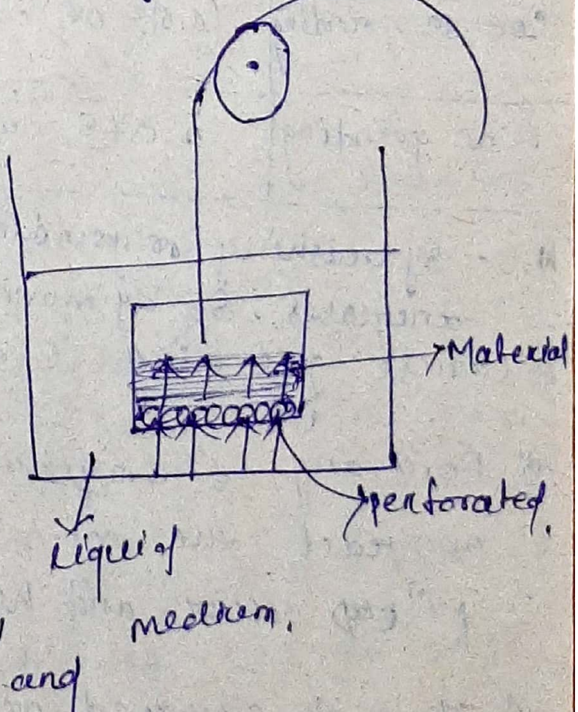
In this riffles the material to be sampled is fed from the a feed hopper to a feeder (vibratory feeder). The feder drop the material at a uniform rate into a series of bins on a rotating table. The turnable speed is set, so that each sample container will pass under the end of the feeder numerous times, before the feed hopper is emptied (generally the speed is varies from 10 to 75 rpm).

Comparision of sampling methods ->

Sampling method.	standard deviation of samples.
Cone & quartering	6.81
Grab sampling	5.14
Sample splitters	1.01
Rotary Riffle.	0.125
Theoritical perfect sample.	0.076

Historical developments ⇒

1. Hand sorting - choosing valuable ore lumps from the worthless lumps (gangue) basing the physical appearence.
2. washing - water exerts a cleaning action and remove the slimes. e.g - washing & cleaning of coals and Iron ores.
3. Crushing - Separated the mineral from the gangue by crushing them into smaller fractional parts with the help of the crusheres.
4. Grinding - Produce fine ores required for gravity concentrations & froth floatation.
5. Tabling and gravity separation - The perticular use of specific gravity of the ore particles for concentrating them.
6. Jigging - ~~Along with~~ Both jigs and shaking tables were developed simultaneously

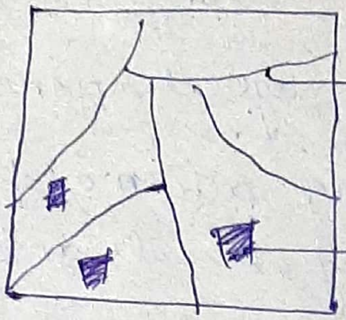


7. Classification - To separate the fine sized particle classifier came in to the picture.

~~8. developm~~

8. development in recent years - In recent years magnetic separators, electrostatic separator, flotation and agglomeration techniques have been developed to operate the ores.

Size Reduction Methods ⇒



* objective is to liberate or release the valuable portion of the ore.

Size parameter for different comminution process ⇒

Process.	Feed size	Product size	Avg. energy (KWh/ton) consumption
Coarse crushing	ROM (Run of Mines) (4-150) cm	(0.5-5) cm	(0.2-0.5)
Intermediate crushing	(0.5-5) cm	(0.01-0.5) cm	(0.5-2)
Coarse grinding	(0.01-0.5) cm	About 75 micron	1-10.
Fine grinding	0.075 mm	0.01 mm	2-25

N.B - By crushing or grinding the ores, the availability of surface area increases. So by moving from large lumps to fine grain/mineral particles more energy get consumed.

* Coarse grinding is also known as fine crushing.

* For any strengthing mechanism, the main objective is to obstract the moment of dislocation, so that dislocation pileup occur and hence strength get increased.

* This is carried out in different ways.
  1. By dispersion of secondary particles in to the primary

phase. (dispersion strengthing).

2. By precipited out a complex phased in the primary phase. (precipitation strengthing /hardening).

3. By Introducing other dislocations to increase, dislocation - dislocation interaction. and hence increase in the hardening. (work hardening).

4. By Increase in number of grain boundries in the initial structure.

e.g - Fe, Ni ⟶ Y_2O_3 (Dispersion hardening)
Al, Ni ⟶ Ni_3Al (precipitation hardening)

Mechanism of size Reduction :⇒

The theory of size Reduction for solid is quite complex But can be activate to the action of following forces acting on the particles.

1. Compressive force. (It happens in case of Jaw crusher, Gyratory crusher & Roll crusher).

2. Impact force - { This is largely utilized in hammer and ball mills.

3. Attrition ; Rubbing action & frictional → This is largely responsible for crushing in attrition mill, tube mill, ~~tabbe~~ pebble, mill.

4. Cutting force → we generally using in soft material i.e - Mica, Asbestos. This is utilised in knife edge mill. To reduce the size of fibrous material like Mica & Asbestos.

* At least one or combination of the above forces is always involved in size reduction in any crushing equipments.

Basic requirements of crushing equipments :→

1. It should have large capacity.
2. It should require a small energy input per unit weight of the production.
3. It should yield a product of uniform size or in required size grains.

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Classification of size Reduction equipments :→

1. Primary crusher
   i) Jaw crusher
   ii) Gyratory crusher
   iii) Toothed Roll crusher
2. Intermediate crusher
   i) Roll crusher
   ii) Cone crusher
   iii) Disc crusher
3. Fine crusher / Coarse Grinder.
   i) Ball mill
   ii) Planetary Ball mill.
4. Fine Grinder.
   i) Rod mill ~~v) Planetary Ball mill~~
   ii) Pebble mill
   iii) Tube mill
   iv) Hammer mill.

Hall-petch Relationship :→

$$\sigma_i = \sigma_0 + K d^{-1/2}$$

Primary Crusher :→

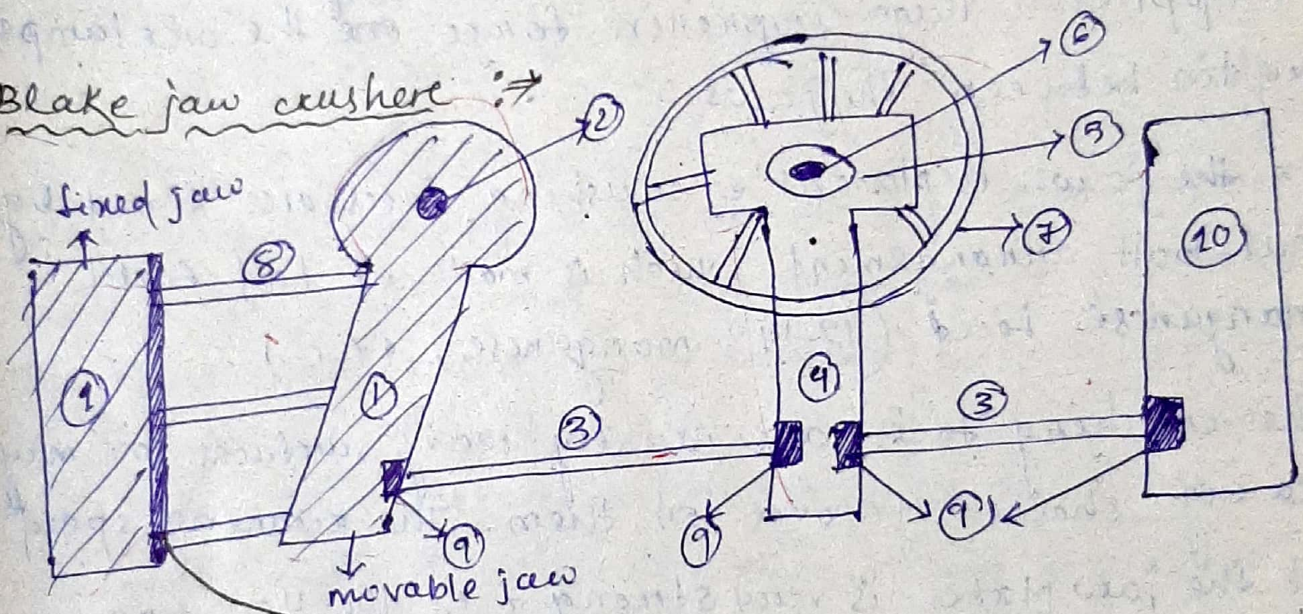
i) Jaw crusher :

1. Blake crusher – In this the movable jaw is pivoted at the top.

2. Dodge – In this case it is pivoted at the bottom.

3. Universal – The movable jaw is pivoted at the centre.

Blake jaw crusher :→



1. Jaw plates → Hadfield Mn steel (12-14% Mn, 1% C).
2. Top hinge
3. Toggles.
4. Pitman
5. Eccentric.
6. Main shaft
7. Fly wheel.
8. Check plates.
9. Bearings.
10. Main Frame.

* Toggles – It is the weakest part. It restrict the damage of jaw plates or crushing plates when clogging occurs.

Constructional featural :→

1. A jaw crusher as two jaws set in the form of V-shape at the top through which feed is admited into the jaw space.
2. One of the jaws is fixed ~~to the~~ at one end almost vertically while the other one is movable.
3. The movable jaw is driven by an accentric, reciprocates in a horizontal plane that makes an angle of 20-30 with the fixed jaw.
4. It applies a huge compressive force on the ore lumps caught in between the jaws.
5. On the jaws, replaceable crushing faces are fixed by nut-bolt arrangement which is made of had-field manganese steel. (12-14% manganese, 1%C).
6. The crushing faces are usually wavy surfaces or may carry shallow grooves on them. the running speed of the jaw plates is very strong i.e- 100-400 rpm.

Importent feature :→

1. The moving jaws is pivoted at the top.
2. The amplitude of the movement is maximum at the bottom.
3. Maximum distance the moving jaw travels is known as throw of the crusher and it is varies from 1-7 cm.
4. Jaw crusher is rated according to the receiving area. it means the length of width the jaw plate and the gape (distance between two jaws) Gape is defined as

the distance betn the jaw plate at the feed opening end.

The distance betn the jaw plates at the discharge opening end is know as set.

e.g - for a jaw crusher having dimension (1830 X 1220)mm.

__Working principle__ :→

The circular motion of the main shaft / ~~eccentric~~ is converted to up and down motion of the pitman by accentric.

The up and down motion of the pitman is converted to to & fro motion of the moving jaw plate through toggle.

Due to clogging of material (ore) or mineral their is huge stress get developed. As long as the process is going on the stress level is going on increase and it may lead to a situation where the jaw crusher would be ~~serio~~ seriourly damaged. To avoid such situation the tuggle fails beyond a particular stress level being the weakest part of the blake jaw crusher.

During crushing their is a uneven / load occur which later on creates problem in crushing operation. To equalise the load throughly ~~out the receiving area~~ ~~by~~ ~~which~~ is used, in one or more fly wheel are the main shaft

__characteristic of Blake Jaw crusher__ ⇒

1. Reduction Ratio : It is the ratio of average feed size with average product size.

R.R > 1 (crushing or grinding).

From crushing to grinding it increases at higher rate.
Reduction Ratio lies between 4-7.

R.R = 1 (no crushing)

2. Capacity : where 'T' is the capacity that express as tons/hour
L = Length or width of the receving opening in inches.

T = 0.6 LS → Taggart's formula.
↓
Tons/hr.

1 inch = 2.54 cm.

T = capacity.
S → set or width of discharge opening in inches.

example-3

The capacity of a jaw crusher may be 725 t/hr for (2250×1680) mm jaw size.

→ $\frac{2250 \times 1680}{25.4 \times 25.4}$

T = 0.6 × 2250 × 1680 = 0.6 × 88.5 × 66.14 = 3512.034 t/hr.

3. Energy consumption and Efficiency :
Energy consumption depends upon the following factors :
→ size of feed.
→ size of product.
→ capacity of machine
→ properties of feed ore = specific gravity, hardness, strength. compressive strength mainly.

capacity increase, o/p decrease, efficiency decrease.
→ capacity is inversely proportional with efficiency.
→ The energy i/p is utilised.
* Elastic & plastic deformation of the feed materials.
* Elastic deformation of the equipment.
* Energy loss in various forms.
(a) heat loss, (b) frictional loss, (c) Noise, (d) Vibration
→ About 10-20% of the total energy i/p is consumed for size reduction and rest is lost in various ways. Proper lubrication and

reduction in frictional losses can increase the efficiency of the crusher.

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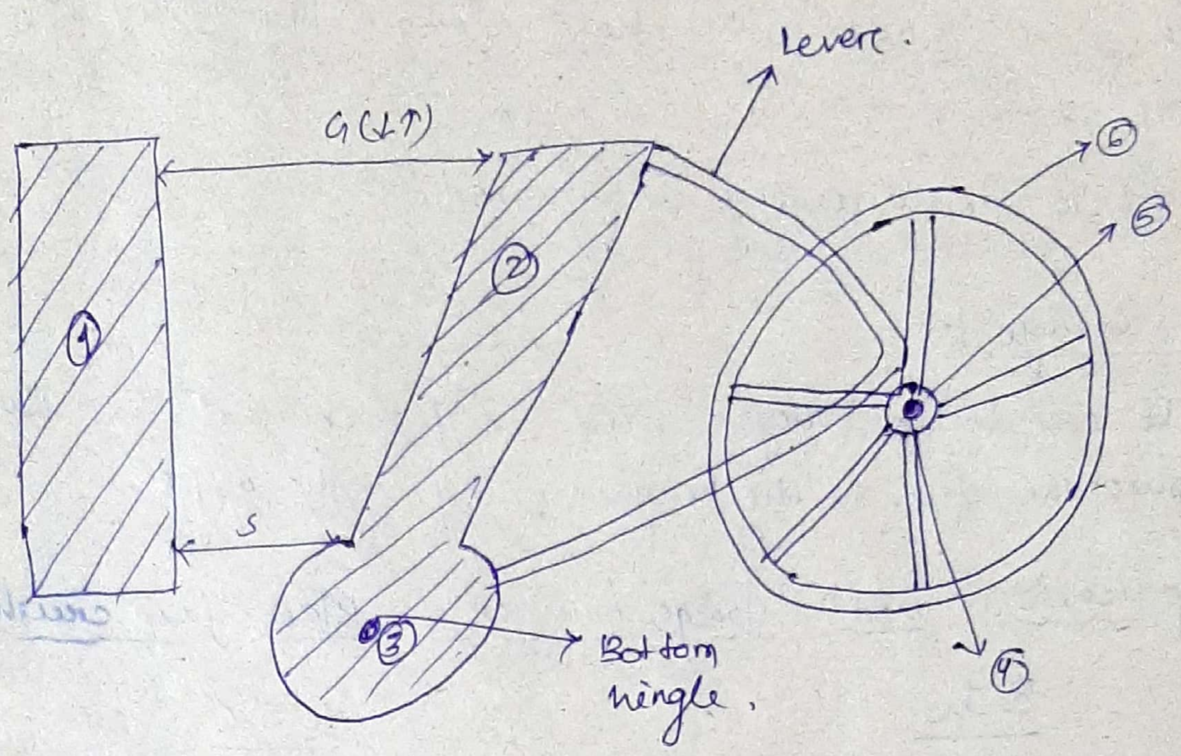
Dodge Crusher :⇒



① fixed jaw
② moveable jaw
③ Bottom hinge
④ drive shaft.
⑤ Accentric
⑥ fly wheel.

Important feature :⇒

1- pivoted at the bottom
2- gape is varying & set is fixed.
3- produced uniform sized product.
4- There is not peddmenn

Lever arrangement

* Lever arrangement comprises a bearing pin. It is simple arrangement that helps a rigid bar to rotates about a fixed point is known a fulcrum.

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Limitations :⇒

* It has a tendency to choke frequently, why the bottom one is fixed

* It is mechanically unbalanced.

Advantage :→

* It can take larger sized feed material, for the reduction purpose due to the large opening in the gape.

Comparision betn dodge crusher & Blacke jaw crusher :⇒

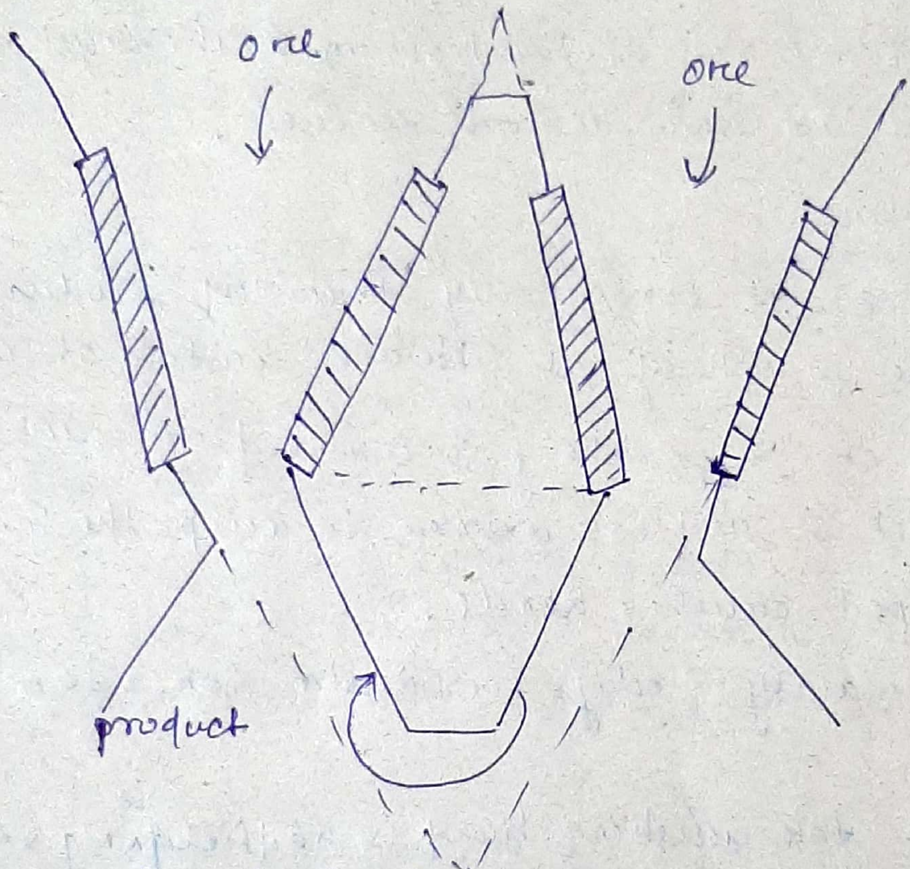
BJC	DJC
i) It have two toggles.	ii) one toggle in the form of Lever.
ii) It has one pitmann	ii) no pitman.
iii) pivoted at the top.	iii) pivoted at the bottom.
iv) No chocking takes place	iv) chocking is a very common problem.
v) Mechanically more balanced and has fewer breakdowns	v) Mechanically the design is interiore and more breakdowns as compare to the black jaw crusher.
vi) large capacity	vi) lower capacity.
vii) Product size distribution is large & produces more fines.	vii) product size distribution is more uniform.
viii) Higher cost for same output	viii) cheaper for same output.
ix) Yields a coarser product.	ix) Yields a much finer product.
x) preferable for industrial purpose/skell	x) prefer to lab scale.

Gyratory Crusher :→

Gyratory crusher is mainly classified into two categories

1- Suspended spindle gyratory crusher. (not in course)

2. parallel pinch or Telsmith gyratory crusher.



Constructional features :→

→ It consist of two substantially vertical ~~truncated~~ trurcated conical shapes.

→ The outer shape has its apex pointing downward where as inner cone has its apex pointing upward.

→ The outer chonical shape is fixed rigidly to the main frame while inner cone (crushing cone) is mounted on a heavy central shaft is known as spindle.

working principle :→

→ The upper end of the shaft is held in a flexible bearing while the lower end is driven by an accentric so as to describe as a circle. Because of this eccentric rotation, the inner cone rotates inside the outer cone alternatively approaching and receding from all the points on the inner periferi of the outer shell. The solids or the feed material caught in the V-shaped space betn the crushed heads and are broken repeatedly until they pass at the bottom. The crushing action takes place all over the cone surface.

Important features:

→ The machine operates continiously throwing product all around the periphery at different instant of time.

→ when one point of the perphery is involved in crushing the opposite point is said to maximum to accept the feed into the V-shaped crusher heads.

→ The crusher mainly employs compressive force for size reduction.

→ The materials for crushing head is hadfield's Mg steel in cast iron.

HW columnar to equiaxed transition.

→ The gyration speed is varied from (125-425) rpm.

Comparison between blake Jaw crusher and Gyratory Crusher

BJC	Gyratory
→ The loading on the machine components intermettent (Discontinuous) and the power draft/power supply is irregular.	→ Uniform loading over the machine components and regular power draft.

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BJC	Gyratory
→ crushing action is intermittent.	→ crushing action is continuous.
→ For a particular gape size the capacity is less as compare to gyratory crusher.	→ same gape size, the capacity is larger.
→ Feed acceptance size is much larger incomparison to gyratory crusher.	→ less acceptance size comparison to jaw crusher for the same capacity.
→ The product size distribution varies widely and the reduction ratio is less than that of the gyratory crusher.	→ with larger reduction ratio and more uniform product size.
→ power consumption is higher for jaw crusher for a particular reduction ratio and capacity.	→ with the same reduction ratio and capacity the gyratory crusher requires less power.
→ less efficient comparison to gyratory and as an efficiency (10-20%).	→ More efficient with a efficiency range (30-50%).
→ wear on the jaw plates is not uniform.	→ quite uniform.
→ Low cost of installation.	→ High cost of installation.
→ Low production rate.	→ High production rate.

characteristics of gyratory crusher :→

→ The capacity of the gyratory crusher is much greater than that of the jaw crusher having equivalent gape size.

→ It has more regular power draft due to continious crushing action.

→ Both jaw and gyratory crushers are equivalent interms of reduction ratio, capacity and power consumption.

→ The rule of installing a gyratory crusher or jaw crusher is given by Taggart formula.

$$\frac{T}{(gape)^2} > 0.115 \Rightarrow \text{select gyratory crusher.}$$

$$\frac{T}{(gape)^2} < 0.115 \Rightarrow \text{select Jaw crusher.}$$

→ T is in tons/hr. and gape in inches.

Intermediate crusher →

1. Cone crusher

→ The construction of the cone crusher is much similar with gyratory crusher.

→ Feed size is much smaller and product is much finer.

→ Both the rotating inner cone and stationary outer cone apex point upward.

→ The outer stationary cone is fixed on the main frame while the inner crushing head in mounted on a heavy control shaft rotating eccentrically.

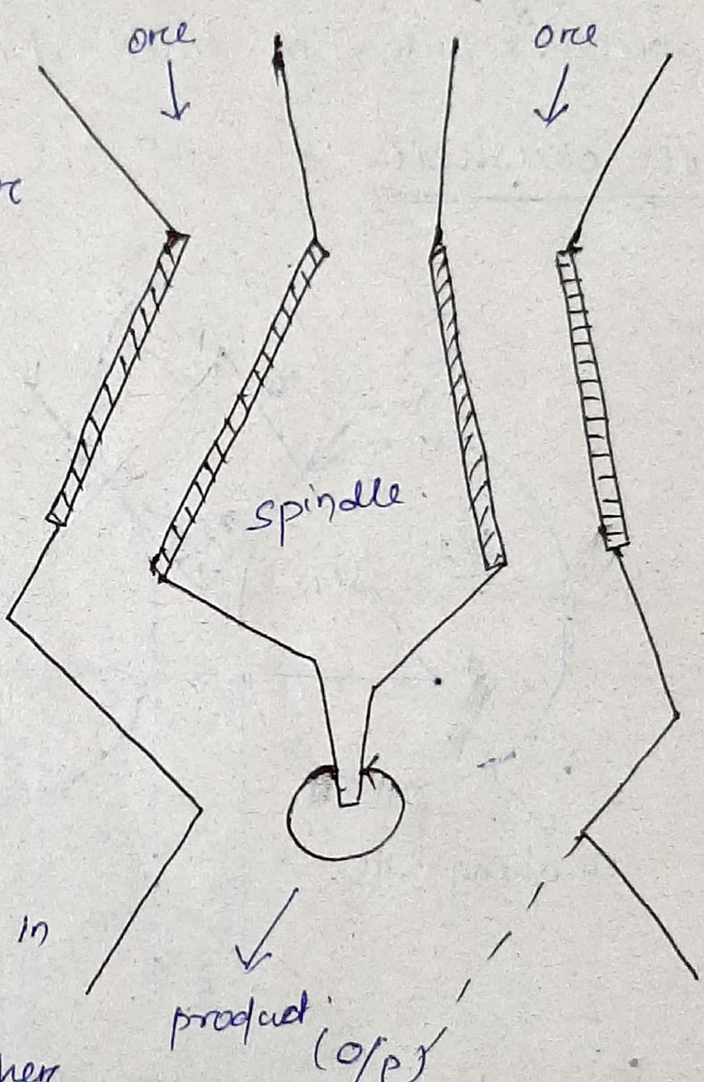
→ The material used as crushing head is hadfield Mn steel in cast iron.

→ The central shaft is fixed with an adjustable bearing and is mounted on an eccentric. Due to the adjustable bearing on the central shaft the position of the inner cone can be altered so as to provide a variable discharge

opening as per the requirement.
This arrangement also takes care of the wear of the crushing faces which may be enlarge the discharge opening (set).

→ The essentric performs the same work as does in case of gyratory crusher.

→ Due to this inner cone alternatively approaches and recides from a particular point on the periphery of the outer cone resulting in continious crushing action. This contenious crushing action results in regulare power draft and much finer product at a greater efficiency.



→ The crushing forces are compressive and frictional in nature.

→ To operate the cone crusher more efficiently, a dry feed free from fines are preferable.
If wet ore is used then it may clog and the problem of clogging in the cone crusher makes it necessary to use efficient screen in closed CKT with them.

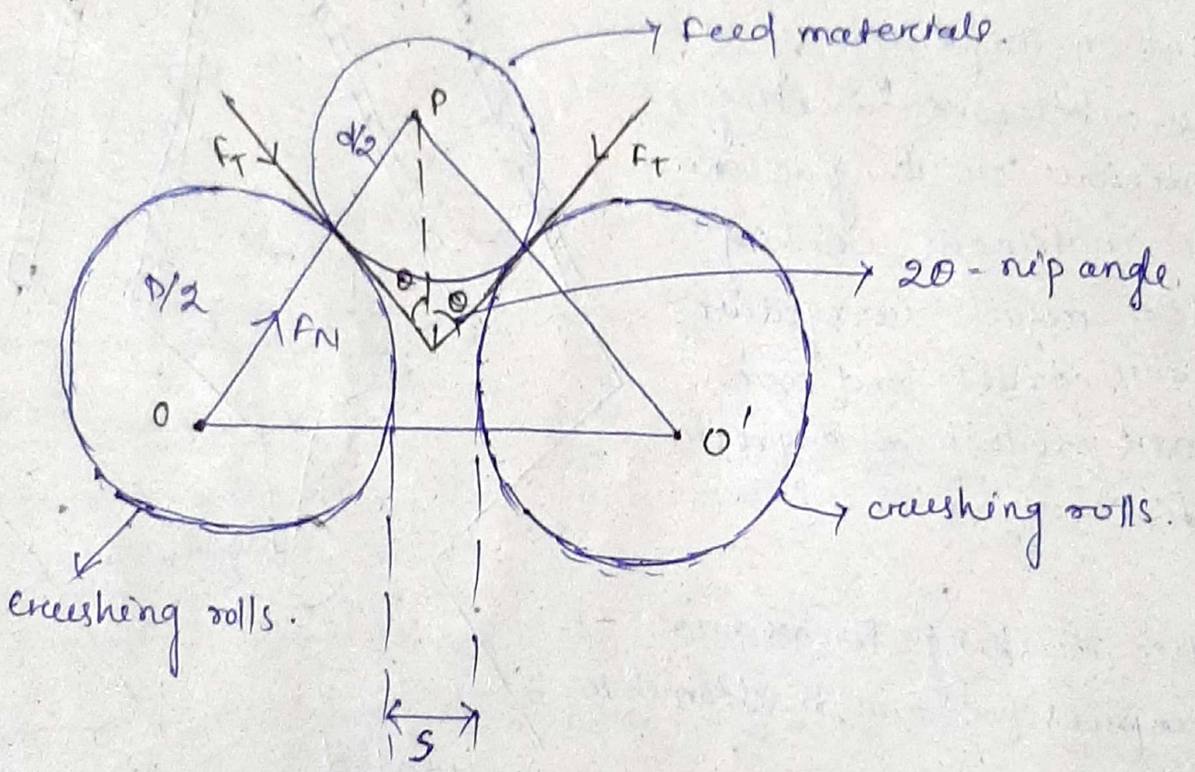
(+5) → over sized product

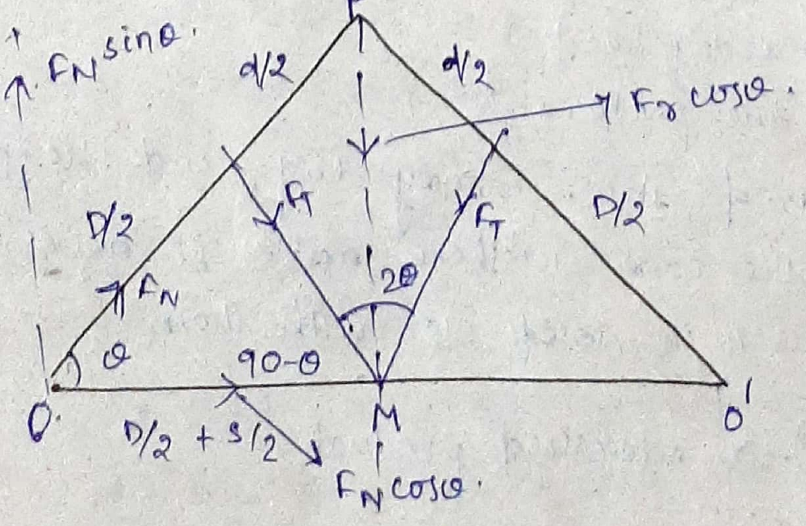
(−5) → undersized product.

Limitations of cone crusher :→

→ Operates only on closed sized brittle materials.
→ Low reduction ratio.
→ Needs extensive lubrication of all its moving parts regularly.
→ Operates based in closed circuit grinding.

Roll crusher :→





$(F_T \cos\theta \geq F_N \sin\theta)$ (Necessary and sufficient) (cond^n for crushing)

$\Rightarrow \frac{F_T}{F_{N1}} \geq \frac{\sin\theta}{\cos\theta}$

Here $F_T \geq \mu F_N$

$\Rightarrow \frac{F_T}{F_N} \geq \tan\theta$ $\Rightarrow \mu \geq \tan\theta$.

Now taking ΔOMP $\cos\theta = \frac{|OM|}{|OP|} = \frac{D/2 + S/2}{D/2 + d/2} = \frac{D+S}{D+d}$

$\Rightarrow \theta = \cos^{-1}\frac{(D+S)}{(D+d)}$

$\Rightarrow 2\theta = 2\cos^{-1}\left(\frac{D+S}{D+d}\right)$

↓

Nip angle.

where D = diameter of the roll.

d = diameter of the feed material.

S = set (discharge opening)

2θ = Nip angle.

θ = half nip angle.

F_T = Tangential force.

F_N = Normal force.

μ = co-efficient of friction.

→ If $F_t \cos\theta < F_N \sin\theta$ (then particle will fly off and crushing doesnot happen).

→ In most of the cases the limiting size of the particle that can be nipped is estimated by:

$$d_{max} = (0.04R + S/2)$$

[R = roll radius]

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→ The co-efficient of friction between steel and most of the ore particle is remain in the range of $\mu = (0.2 - 0.3)$.

→ So the angle of nip 2θ should never be above 30° else the particle will slip / fly.

$$2\theta = 11.31 \times 2 = 22 \text{ to (minimum)}$$

$$2\theta = 16.7 \times 2 = 34 \text{ (maximum)}.$$

→ The kinetic friction (μ_K) between particles and moving rolls can be computed using the following formulas.

$$\mu_K = \left[\frac{1+1.2V}{1+6V}\right]$$

where V = is the peripheral speed which is around 1 m/sec. for small roll and 15 m/s for larger roll.

(having diameter about 1800 mm or more = D)

* Roll crusher consists of pair of heavy cylindrical rolls revolving towards each other. So as to nib a falling ribbon of rock and discharge it cross below the rolls.

<u>Mechanical design</u> :→

The two rolls are heavy and rigid one. Both the rolls are positively driven towards each other by motors. The heavy rolls are turn on parallel horizontal plane having the roll center at the same height separated by a distance S.

The feed caught between the rolls are broken by compressive force and drop down below. The rolls turn towards each other at the same speed. Typically rolls are 600 mm in length & 300mm diameter. The roll speed varies from (50-300) rpm. The feed size varies from (12-75)mm and product size varies from (12-20)mm. The product size mainly depends upon the rolls separation distance, i.e-('s). The operation is quite continuous. At a lower reduction ratio, the crossing rolls produces less fines as compared to the other crushers. However, the crossing roll have large capacity at lower reduction ratio.

The roll clearence is adjustable and depends on feed size and product size requirement. The machine is protected against damage from very hard material by the spring loader mounted onto the rolls. When a hard material having breaking strength is higher then that of the strength of the spring loader, it is simply allows the hard rock to drop down without being crushed.

The most Important characteristics of roll crusher, is the controls of the crossing activity by means of angle of nip or angle of bite.

Angle of Nip (2θ)

It is defined as the angle subtended between two tangents drawn at the points of contact of the rolls and the feed particle to be crushed.

→ crushing is perform only when the ore particles are nipped properly by rolls.

→ The particles that can be nipped by crushing rolls depending upon the following factor.

* Roll diameter (D)
* feed size (d)
* separation between two Rolls (S)
* co-efficient of friction (μ)

Characteristics of roll crusher :→

→ The reduction ratio is varies from 3 to 4.

→ cenet or misize product.

→ Product of the crushing rolls contains pure fine as the fines is limited and no repeated crushing takes place.

→ Capacity of roll crusher depending upon the factor,

(i) speed of the revolution (N)

(ii) width roll faces (W)

(iii) Diameter of the rolls (D)

(iv) Inter roll distance (S)

(v) specific gravity of the feed material ρ)

tons/hr ← $\boxed{C = 0.0034\ DWS\rho}$

where DWS all are expressed in inches.

$\rho \rightarrow lb/inch^3$

$\boxed{C = 1.885\ NDWS\rho}$

↓ kg/hr.

DWS all are expressed in meter.

$\rho = kg/m^3$

The actual capacity is around 10-30% of the theoritical capacity.

$$C_{Actual} = (10-30)\% \cdot C_{theo}$$

If the set(s) is nill or zero, then the capacity of the rolls is nile.

→ Rolls can be operated either in wet ~~weight~~ or dry condition

→ It is best operated on choke feeding for maximum output.

Uses :→

→ Most suitable for smaller size reduction in a single operation.

→ It is common to employ a number of pair of rolls in series to achieve higher reduction ratio.

→ Used to cross oil seeds, gun powder and coal, because of lower residence time reduces the heating effect on the feed material.

Qn:-1 - What should be the diameter of a ~~set~~ pair of rolls to take feed of size 38.1 mm (d) and to cross it to 12.7 mm (s). If $\mu = 0.35$.

D-??

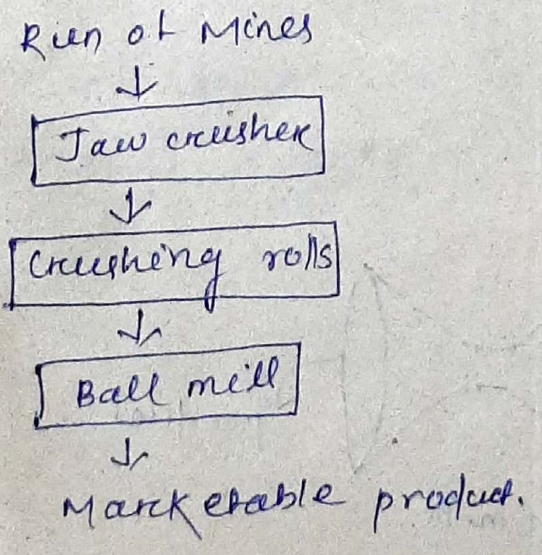
Qn:-2 - What is the ~~minimum roll diameter~~ nib angle of the crusher to reduce 1.5 inch piece of rock to 0.5 inch, provided the diameter of the rolls is 12.5 inch?

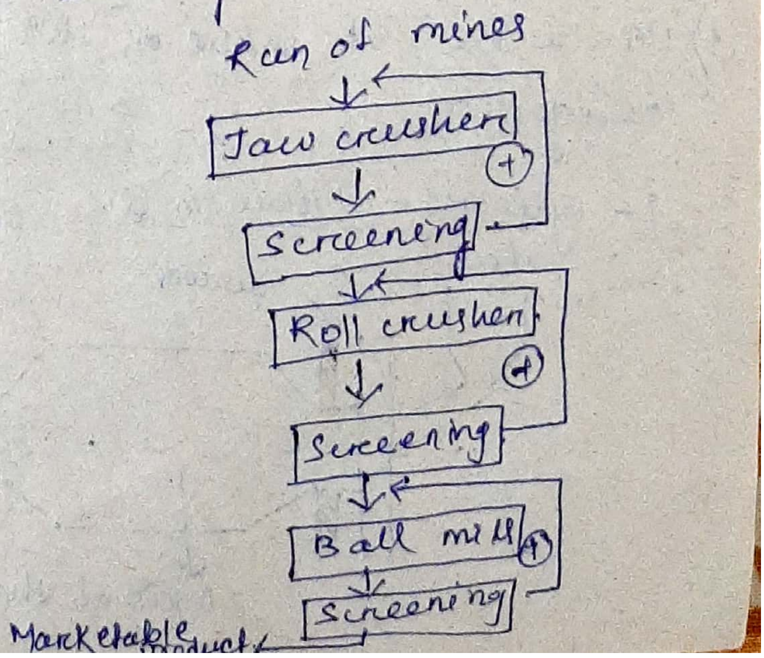
Feeding system in communition equipment :⇒

Free feeding	Choke feeding
→ Low rate of charging followed by discharging.	→ High rate of charging followed by low rate of discharging
→ Less residence time and produce less fines.	→ Higher residence time and produces more fines.
→ Reduces the chances of clogging.	→ frequently clogging happens.
→ less energy consumption in crushing.	→ Higher energy consumption in crushing.
→ Less wear on the crushing faces.	→ High wear on the crushing faces.
e.g - Blake jaw crusher.	e.g - Dodge crusher.

Difference between open circuit Grinding and close circuit Grinding :→

Open CKT Grinding	Close CKT Grinding
→ In many grinding mills the feed is broken into particles of satisfactory size by passing it once through the mill, when no attempts to return the over size particle into the product to the grinder. For further size Reduction the product simply passes of to the next stage of the size reduction, such a methode of size reduction is known as open ckt grinding.	→ In this methode the partially crushed material is screened and the over sized material is returned back to the crusher for further crushing/grinding and the under sized product is given as feed to the machine for further size reduction.
→ less efficient, less effective, less capacity, less energy consumption, less time (crushig/grinding) consume.	→ High efficient, high effective, high capacity, high energy consumption, High grinding/crushing time consume.





Fine crushing or Coarse Grinding :-

* The fine crushing or coarse grinding means the product size not less than 200 mesh screen (#) (74 μm) or 0.074 mm or not above/more than 6mm mesh size.

* Any combination process that can produced less than 6 mm is termed as "fine crusher or coarse grinder.

* Grinding is a slower process is usually carried out in a ball mill for tumbling mill or any other equipment like tube mill, rod mill or pebble mill.

* These mills perform size reduction in a closed chamber consist of grinding media and the material.

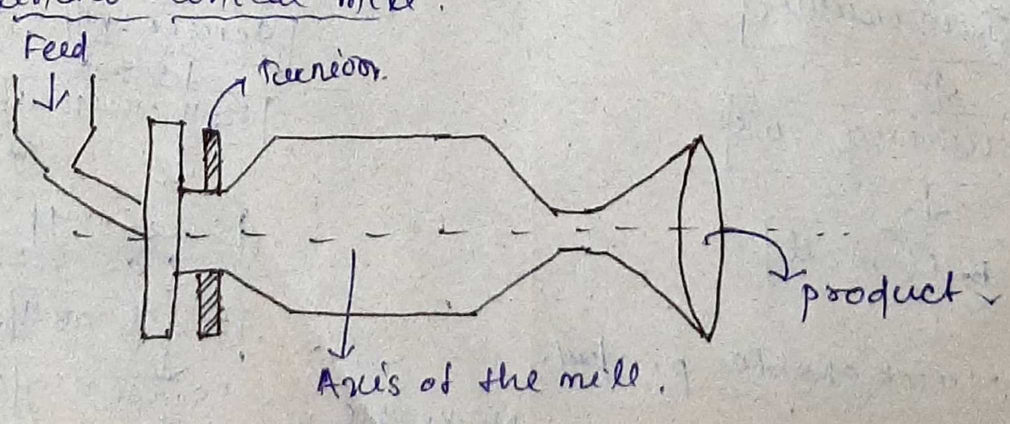
classification of Ball Mills :-

* On the basis of 1) shape of the mill
  2) Methods of discharge.
  3) whether grinding carried out in wet or dry medium.

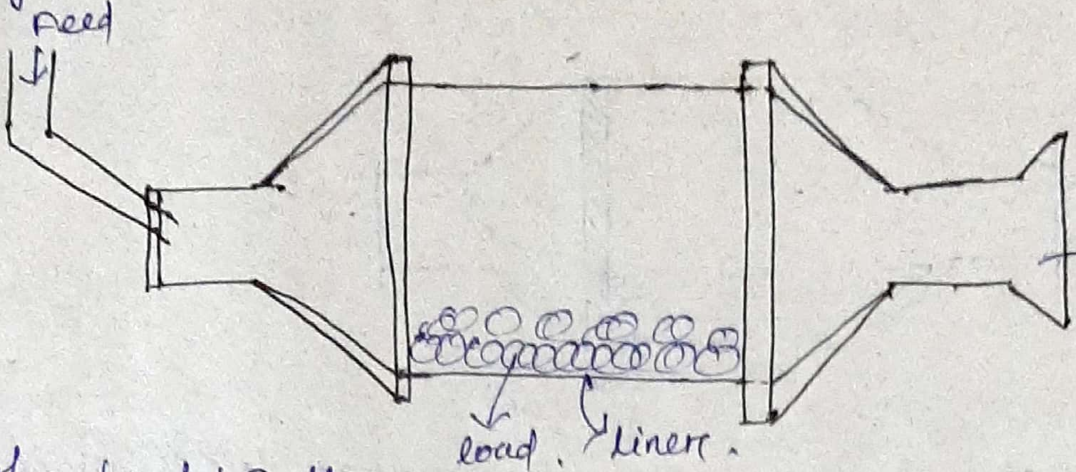
ball mills are classified into different categories.

1) On the basis of shape of, the mills are classified into 2 categories.

i- cylindro - conical mill.



ii - cylindrical



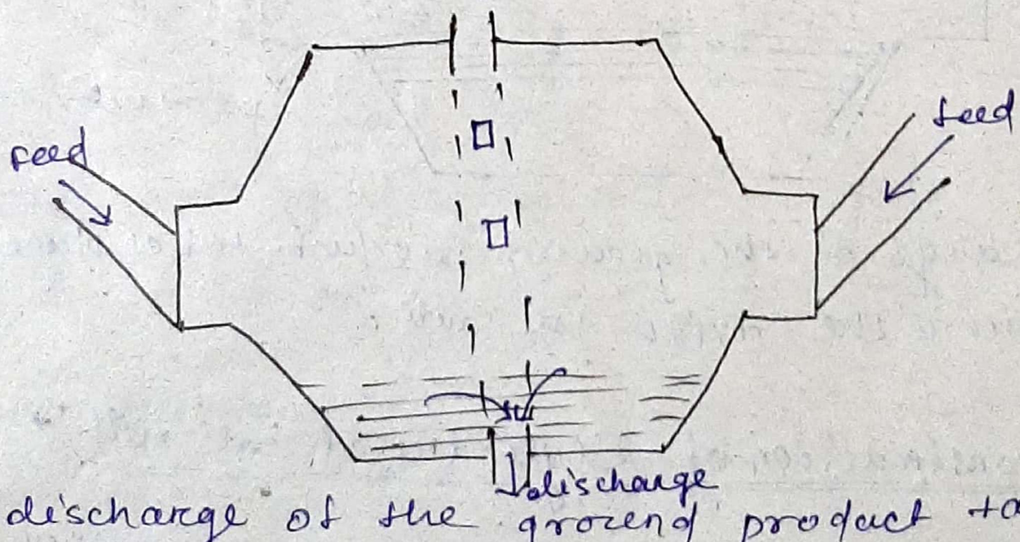
Load = feed + Balls.

* Generally harden still or stainless stell are used for resistance to corrosion (chromium % high)

* Tungsten carbide (WC) used for harder material.

2) On the basis of methods of discharge ball mills are classified into 3 categories.

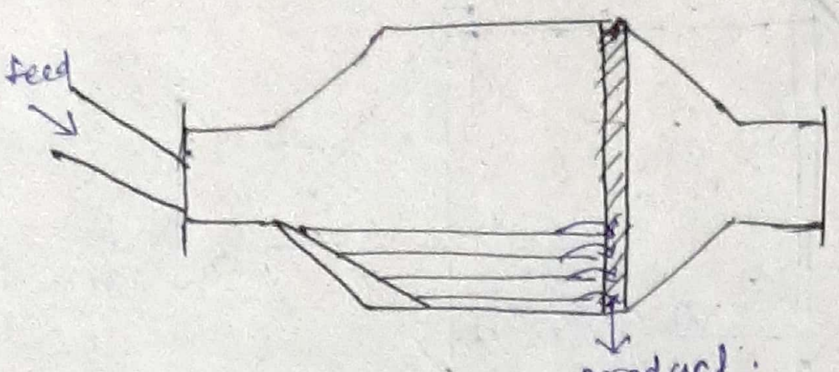
i - Peripheral Discharge mill.



discharge of the ground product takes place through meshed cylindrical shape.

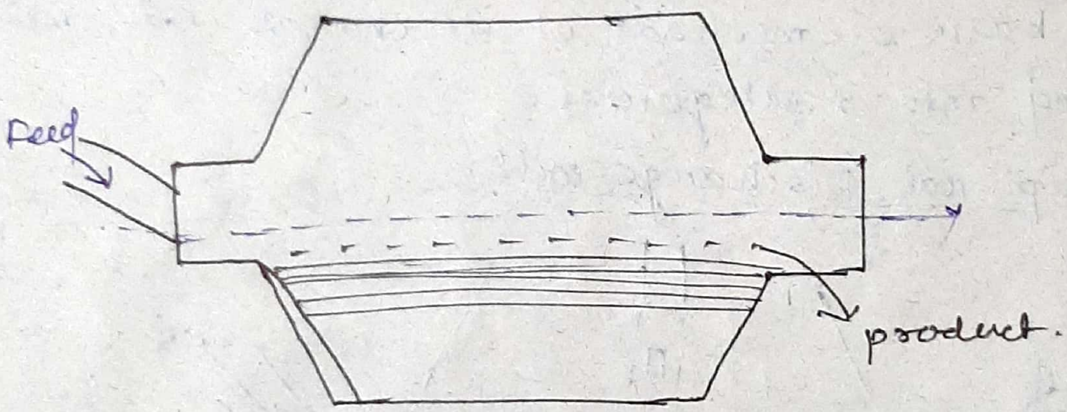
(ground - past tense of grind)

ii - Grate mill



Discharge of the ground product takes place through a screen extending as a diaphragm across the full section of the mill at the discharge end.

iii - Overflow mill



discharge of the ground product takes place free overflow from the axis of the mill.

Mechanical construction of a cylindrical Ball mill :-

The important components of a ball mill is given by.

→ cylindrical cell.

→ Inner surface or liners.

→ Balls or grinding media.

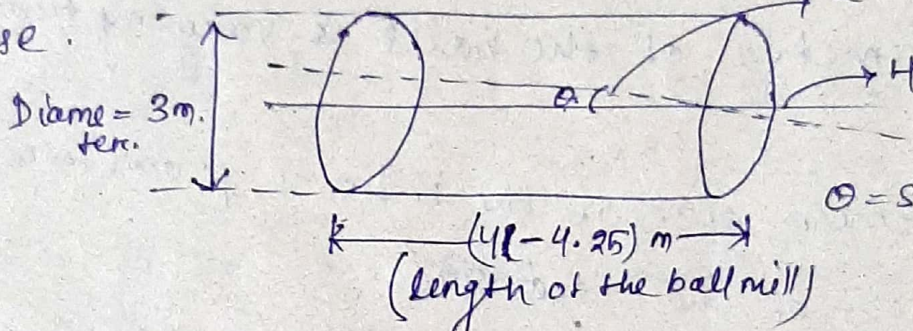
→ Motor drive.

1) Cylindrical cell :→

It is a rotating hollow cylinder partially filled with the balls. The ore to be crushed is fed through hollow torranion (it is a cylindrical projection used as a pivoting point). At one end and the product is discharged through a similar torrnnion at the other end.

The material of construction for this hollow shape is usually high strength stell.

The cell axis is either horizontal or at a small angle to the base.



Ball mills have generally length of 4-4.25 m & diameter is about 3m.

The harden stell balls having a size varying from (25-125) mm. or (2.5 -12.5) cm.

2) Inner surface / Liner :→

As the grinding process involves impact and attrition the interior of the ball mill is lined with replaceable wear resisting liners that usually made of high manganese alloy steels, stones or rubber:

least wear takes place on rubber line interior as compare with steel lined interior. (Due to coefficient of friction between steel liner ball is higher that the rubber and ball) ($\mu_{sl-b} > \mu_{r-b}$)

3) **Balls or grinding media :→**

→ The balls are usually made of cast steels, tungsten carbide (WC), stainless-steel & Hardned steel., flint ball (shiny grey or black stone i.e-like glass with a higher %. of sillica (SiO_2))

→ The diameter of the balls is varying from 1 to 5 inch.

→ the optimum size of the balls is proportional to the square root of the feed size.

→ The bulb and liner wear are usually in the range of (450 - 1250) and (0.5 - 250) gms/ton. of ore ground

4) **Motor drive :→**

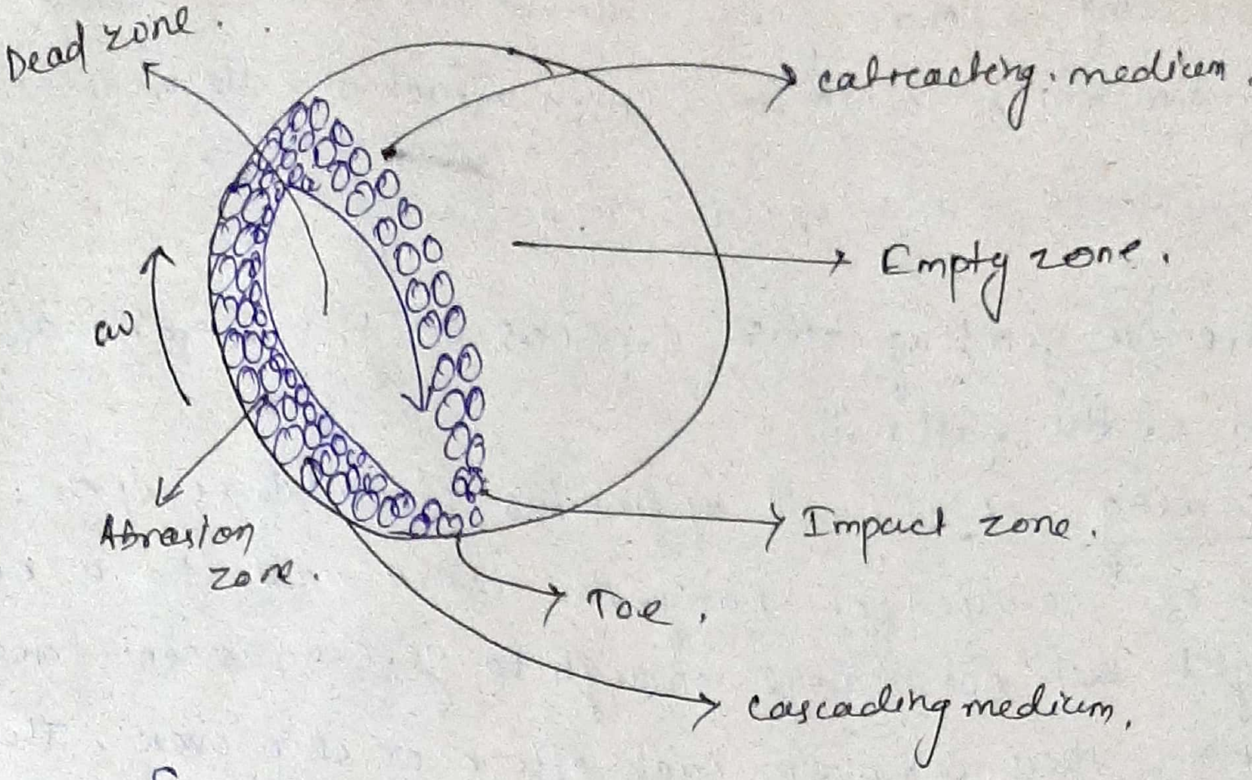
→ The mill is rotated by electric motors connected through gear box → ring gear.

Theory of Ball mill operation :→

→ Cascading → Lower speed (Attrition)

→ cataracting → Medium " (High impact).

→ centrifuging → High speed (No grinding)



[Different stages and zones of ball mill]

* Ball mills may be contineous or batch type in which grinding media and the ore to be ground are rotated around the axis of the mill. Due to the friction between liner balls & liner - ores, both the ore and balls are carried of along the inner wall of the cell nearly to the top from where the grinding media falled down on the ore particles below creating a heavy impact. of the ball mill. This usually happen at the toe of the ball mill.

The energy expanded in the lifting of the grinding media is utilised in reducing the size of the particle as the rotation of the ball mill is contineous.

The grinding process is attributed to 3 different stages of ball mill working and they are

→ cascading → Lower speed (Attrition betn ball & particles)

→ cataracting → Medium speed (High impact of balls over particles).

→ centrifuging → High speed (No grinding).

Effective grinding ~~takes~~ depends on the rotational speed of the ball mill.

<u>cascading</u> - If the mill operates at a lower speed, balls will be carried of along the inner wall to a certain height but not large enough to give an impact force. rather they roll over each other or slip over. This type of operational condition is known as cascading of the ball mill, where grinding takes place due to attrition between balls and feed materials particle.

<u>cataracting</u> - If the speed is rised, the ball start moving up along the inner wall and suddenly fall from a greater height impacting with a high impact force at the toe of the ball mill. This impact is largely responsible for the most of the grinding and this operational condn is known as cataracting.

<u>centrifuging</u> - If the speed of the rotation becomes too high, the balls are carried over and over again all along the inner lining as if they are sticking to the inner wall and there is hardly any grinding. This operational condition is known as centrifuging.

If the speed of the ball mill is too low, cataracting doesn't occur. Rolling down of balls and particles lead to particle rubbing and limited grinding is possible. At the other extreme due to very high speed of the ball mill centrifuging occur leading to little or no grinding. So the mill is operated in between this two extreme.

$$N_o = (0.65 - 0.75) N_c$$

N_o → operational rotational speed.

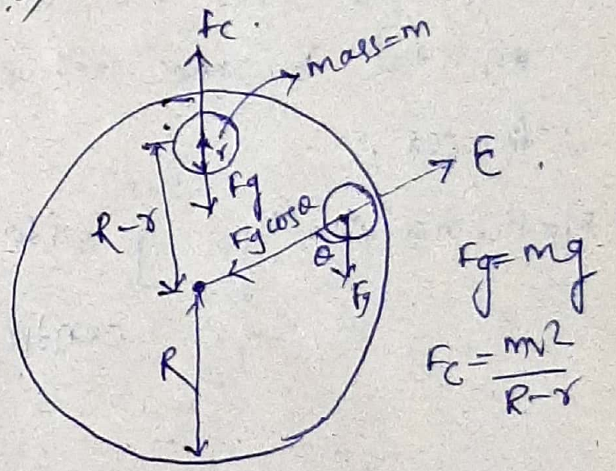
N_c → critical rotational speed.

Critical speed of Ball mill :→

$F_g = mg \cos\theta$

$F_g = F_c \Rightarrow mg\cos\theta = \frac{mv^2}{R-r}$

$\Rightarrow g\cos\theta = \frac{v^2}{R-r}$



→ The minimum rotational speed after which centrifuging occur.

→ No grinding takes place in the ball mill when it centrifuges, so the operational speed should be less than that of the critical speed.

Derivation of critical speed :→

→ Assumption

1. Let the radius of the cylindrical ball mill is R.
2. Only single sized media of radius r is used in the mill.

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→ During the rotation of the ball mill, the grinding media is carried of along the inner wall of the ball mill.

→ At any perticular instant the process working on the media are generally centrifugal force, symbolically it is represent as (F_c). working radially away from the center of the mill.

→ Gravitational force(F_g) acting vetically downward from the center of the particle.

→ The speed at which the outer most bulbs may loose contacts with the inner wall of the ball mill depends on the balance between gravitational & centrifugal forces.

Mathematically gravitational force = F_g --- (1)

centrifugal force is given by $F_c = \frac{mv^2}{R-r}$ ---(2)

where m = mass of the ball

v = linear speed

R-r = effective Radius of rotation.

At top most position of the ball mill

$F_g = F_c$

$\Rightarrow mg = \frac{mv^2}{R-r}$ ---(3)

for centrifuging occur the ball should be exceed top most position of the ball mill. so by considering any position except top most position.

Let us it is making an angle θ with respect to center of the ball mill. Then towards center, the component of gravitational force is acting and i.e. -

$$F_g \cos\theta = mg\cos\theta \quad \text{——— (4)}$$

Now balancing the gravitation force with centrifugal force, we have

$$F_C = F_g \cos\theta$$

$$\Rightarrow mg\cos\theta = \frac{mv^2}{R-r}$$

$$\Rightarrow g\cos\theta = \frac{v^2}{R-r} \quad \text{——— (5)}$$

Again the relationship between linear speed and rotational speed is given by

$$v = 2\pi(R-r)N. \quad \text{- - - - (6)}$$

Now replace the eqn (6) in eqn (5), we have

$$g\cos\theta = \frac{4\pi^2(R-r)^2N^2}{R-r} = 4\pi^2(R-r)N^2 \quad \text{- - - - (7)}$$

Now, at top most position of the ball mill $\theta = 0$.

$\Rightarrow \cos\theta = 1$

and we have critical speed i.e. - N_c.

Now using this condition, in eqn (7), we have

$$g = 4\pi^2(R-r)N_c^2$$

$$\Rightarrow N_c^2 = \frac{g}{4\pi^2(R-r)} \Rightarrow \boxed{N_c = \frac{1}{2\pi}\sqrt{\frac{g}{R-r}}}$$

In different units critical speed of the ball mill can have following values.

$$N_c = \frac{42.3}{\sqrt{D-d}}$$

D = Diameter of the Ball mill
d = Diameter of the Ball
D, d are expressed in meter.

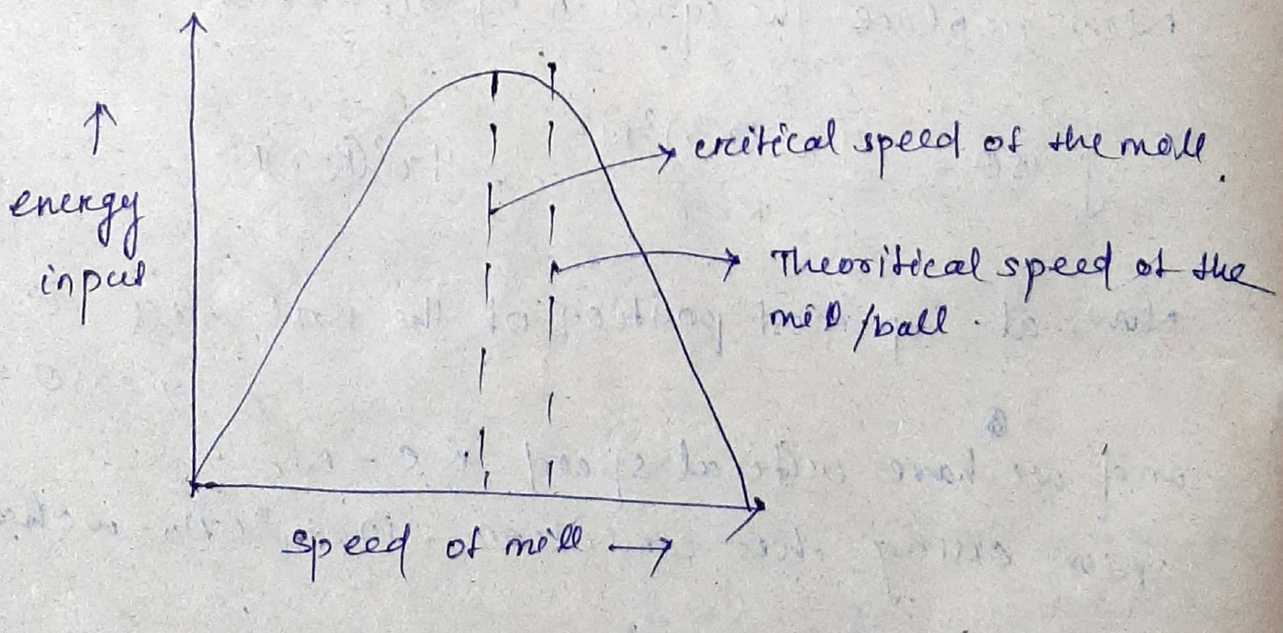
$$N_c = \frac{76.65}{\sqrt{D-d}}$$, D, d are expressed in feet.

Characteristics of Ball mill working :→

1. Speed & energy input interrelation in Ball mill :→

i) Speed of the Ball mill should be as high as possible without centrifuging.

ii) The work input increases steadily as the speed of the ball mill increases. it reaches a peak value at a perticular speed and thereafter the work input decreases rapidly with increase in speed.



2. Ball load. :→
(excluding load of ore).

It is defines as the volume i.e. occupied by the grinding media out of the total volume of the ball mill without ore or water in it.

i) The ball load should be slightly more that 30% of the total volume of the ball mill, and during general operation, it remains 30-50%.

ii) When a mill is operated for the 1st time, the balls of the various sizes is taken rather than single sized the reason is, If balls of definite size are charged, the interstitial voids are created and the particles if caught in that interstitial void region, then it will not grind. To overcome these problems various sizes balls are taken.

iii) During grinding, the balls themselves get worn of which reduces the ball load. The reduced ball load is replaced at regular intervals with new balls of larger size only. In fact larger sized balls grind the feed materials more effectively, whereas the smaller ones are producing fines.

iv) The energy input of the ball mill is a function of speed of the Ball mill, Ball load, specific gravity of the ore, dilution of the pulp.

→with increase in energy input, the speed of Ball mill increase upto maximum & then decreases.

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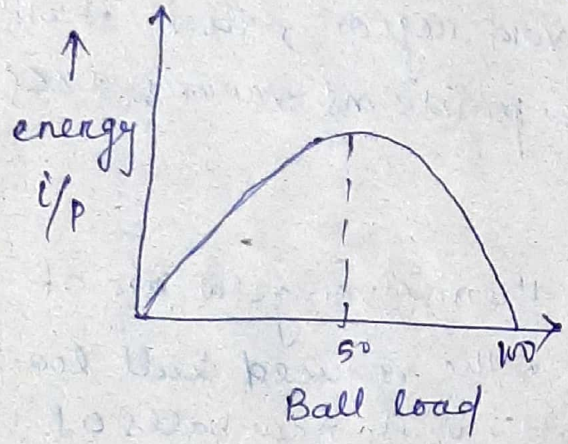
→ With increase ~~upto~~ in ball load energy input increases upto 50%, & then decreases.

(when the Ball load increases the center of gravity of load comes nearer to the axis of rotation of the mill which decreases the energy input of the mill.)

→ Dense material require more energy as intermolecular force is higher

→ With increase in dilution, energy input decreases.
(A pulp density of 60 - ~~70~~ 75% of solid results in maximum energy input)



3. Reduction Ratio ⇒
(Input size / out put size)

The reduction ratio that can be obtained in the ball mill is large compared to reduction ratio obtained in primary & seconadry crusher, and It remains in the range of (50-100).

→ If the reduction ratio is high along with capacity then it will be more economical.

4. Capacity :→

i) It depends upon the size, hardness of the feed ore & the R.R.

ii) Ball mills yield (1-50) ton/hr of ore fines with 90% passing through 200 #.

5. Energy consumption :→

i) The average energy input of the ball mill is 60 Kwh/ton of ore ground.

Factors affecting the size of the product in a Ball mill :→

1- Rate of feed.
higher the rate of feed, lesser is the size reduction, since the residence time of the ore particles in the mill is reduced.

2- Properties of the feed ore.
→ under the same operating conditions, larger the feed larger will be the product, greater will reduction ration lower.

→ for hard particles it results lower Reduction ration.

3- weight of the ball.
→ heavier balls produce finer products. The weight of the balls is normally altered by the use materials of different specific gravity.

4- Diameter of the Ball.

→ smaller sized balls produces fines and larger sized balls are responsible for effective grinding.

→ The limiting size of the reduction is obtained with a given size of the balls is known as free grinding.

5. Slope of the mill.

→ Increase in slope of the mill, increases the capacity of the mill, but a coarser product is obtained as the resident time of the feed in the mill is reduced due to higher slope.

6. Discharge of freedom.

It has same effect as like slope of the mill.

7. Speed of rotation.

The mill should be operated at a speed less then Nc. ; $N_o < N_c$

$$N_o = (0.65 - 0.75) N_c.$$

8. Lable of material in the mill.

It should be 60% maximum out of which 30% is ball load.

<u>Advantages of the Ball mall</u> ⇒

1. It used for both wet and dry grinding.
2. cost of instillation of ball mill is low.
3. It can be used inert atmosphere to grind explosive material.

4. The grinding media used is relatively cheap.

5. It is suitable for grinding of materials with any degree of hardness.

6. It is operated in batches or continiously.

7. It is used for both open and closed circuit grinding effectively.

Dry and wet grinding :-

1. Ball mill is operated in both dry and wet condition.

Advantages of wet grinding over dry grinding

i) wet grinding facilitates better removable of product, eliminates dust problem, lesser the noise and heat produced through the wear may increase by 20%.

ii) power consumption is lowered by (10-30)% over dry grinding per ton of the product.

iii) The capacity increases per unit volume of the mill.

iv) The grinding makes wet screening possible for producing materials in narrow size bench.

v) Surface quality is more uniform.

vi) easier to handle and transport the product.

Disadvantages of wet grinding over dry grinding

i) formation of emulsion residue on the surface (emulsion - A mixture that results when one liquid is added to another but donot dissolve into it)

13/02/19

Laws of crushing :→

The most important consideration in any size reduction is the energy it consumes in performing the activity. The empirical relations between energy consumption and size reduction are termed as laws of crushing.

Rittinger's law of crusher :→

It states that the energy expanded during communition is proportional to the new surface area created due to the result of particle fragmentation.

Mathematically :- $E \propto S \propto \frac{1}{d}$

$$\Rightarrow E = K_R (S_2 - S_1)$$

$$= K_R \left(\frac{1}{d_2} - \frac{1}{d_1} \right)$$

S_2 = product surface area.
S_1 = feed surface area.
K_R = Rittinger constant.
d_2 = size of the product.

It is used for the fine grinding in the range of (10 - 1000) μm

Bond's law of crusher :→

It states that the total amount of work Input represented by a given wet or crushed or ground product is inversely proportional to the square root of the product particle diameter.

mathematically $W_b \propto \frac{1}{\sqrt{D_p}} \Rightarrow W_b = 10\, W_i \left[\frac{1}{\sqrt{D_p}} - \frac{1}{\sqrt{D_f}} \right]$ —(2)

W_b = energy used as work input.

= Bond's work input during crushing

W_i = Bond's work index.

work Index is the combination parameter that expresses the resistance of the material to crushing or grinding.

$$\Rightarrow W_i = \frac{W_b}{10} \left[\frac{\sqrt{D_p D_f}}{\sqrt{D_p} - \sqrt{D_f}} \right] \quad \text{—— (3)}$$

* It is used for intermediate crushing.

Kick's law :→

It states that the energy consumption is directly proportional to the logarithm of size reduction ratio. (D_1/D_2).

$$E \propto \log(d_1/d_2)$$

$$E = K_K \log(d_1/d_2) \quad \text{—— (4)}$$

* Kick's Law is successful in predicting the energy consumption during coarse crushing.

i.e - in the range of 1 cm and above.

Differential form of crushing laws :⇒

The energy requirement during size reduction can be represented in the form of a differential equation which is given by

$$dE = -C \frac{dx}{x^n} \quad \text{——— (5)}$$

It can be written as.

The energy necessary to cause a small change in an object size is proportional to the object size rest to the power (n).

In eqn (5) dE = Small change in the energy requirement to bring a small change in the object size.

x = object size.

dx = small size in object size.

n = 2

$$\int_0^E dE = -C \int_{d_1}^{d_2} \frac{dx}{x^2}$$

$$\Rightarrow E = C\left[\frac{1}{x}\right]_{d_1}^{d_2}$$

$$= C\left[\frac{1}{d_2} - \frac{1}{d_1}\right] \text{ - - - - (6)}$$

using n = 1.5 in eqn (5) we have

$$\int_0^E dE = -C \int_{d_1}^{d_2} \frac{dx}{x^{1.5}} \Rightarrow E = C\left[\frac{1}{\sqrt{x}}\right]_{d_1}^{d_2} = $$

$$= \frac{C}{0.5}\left[\frac{1}{\sqrt{d_2}} - \frac{1}{\sqrt{d_1}}\right]$$

using $n = 1$.

$$\int_0^E dE = -C \int_{d_2}^{d_1} \frac{dx}{x} = -C[\ln x]_{d_1}^{d_2}$$

$$= -C \ln(d_2/d_1)$$

$$= C \ln(d_1/d_2)$$

$$= 2.303\, C \log(d_1/d_2) \quad \text{------ (8)}$$

Ore Grindability :→

Refers to that parameter with which the material can be crushed or ground the most widely used parameter for measuring grindability in the Bonds work Index (wi)

$$\text{Bond's work index } (W_i) = \frac{W_b}{16}\left(\frac{\sqrt{d_1 d_2}}{\sqrt{d_1} - \sqrt{d_2}}\right)$$

Criteria of selecting commminition equipment :→

The choice of machine or a equipment for a given crushing operation will affected by the following factors.

(i) Size of the product. (fine or coarse or intermediate

• along with size of the feed:
↳ shape.

(ii) Quantity of the material that can be handled.

(capacity = i/p − o/p).

(iii) physical properties of the feed material to be crushed

Significat physical properties of the feed ore :→

1. Hardness → The hardness is a function of

f (power consumption.
compressive stress
wear on the m/c.
chocking.

2. Structure →

Normal granular material like coal, ore, or rocks can be effectively crushed by employing normal compressive and impact forces. (e.g - coal)

For fibrous ~~mineral~~ material it is necessary to apply tearing action for size reduction.

e.g - mica and asbestos.

3. Moisture content →

It is found that minerals with higher moisture content (~ 5% - 50%) do not flow effectively and it tends to clog the crusher.

To overcome these difficulties, generally wet grinding for crushing is applied.

4. Crushing strength →

The power required for crushing is directly proportional to the crushing strength of the minerals.

5. Friability →

It is the tendency of mineral to fracture during normal handling. Generally crystalline materials will break along well defined planes. So less power required to crush.

6. Stickiness :→

A sticky mineral will tends to clog the equipment so should be ground in a mill, that can be readily cleaned.

7. Friction factor :→

Coefficient of friction is directly proportial with the crushing action.

$$\mu(\uparrow) \propto \text{crushing action} (\uparrow)$$

If the coefficient of friction is low, then it will be difficult to crush and vice versa.

8. Explosive mineral :→

These mineral must be ground in wet condition or in the presence of an inert atmosphere. Otherwise it will cash fire or exploed.

9. Mineral ~~processing~~ producing heavy dust :→

For these type of minerals wet grinding or crushing is better than dry grinding or crushing.

Particle size determination (Module-II)

Introduction ⇒

particle size plays an important role in the following cases.

(i) To determine the reaction behaviour betn S-L, S-G, calcination, smelting etc.

(ii) To evalute the energy consumption, where the energy consumption ∝ available specific surface area

$$\propto \frac{1}{\text{particle size}}$$

particle size and shape ⇒

The shape of the particle is an important factor in the determination of particle size. Generally the particle size is define uniquely by its diameter however it has no specific dimension. The term most often used to describe an irregular particle is the equivalent diameter ($\bar{d}$). Their can be various shapes of the particles are used to describe a particle as discussed below.

1. Accicular - needle like particles.
2. Angular - sharp edged polyhedrons.
3. crystalline - particles of regular geometric shape.
4. Fibrous - Regular or irregular thread like particles. e.g. - asbestous, mica.
5. Dentritic - particles having branched crystalline structure

6. Flaky - plate like particles (disc shape).

7. Granular - It is irregular in nature but having 3-D. (equidimensional irregular shape particle)

8. Irregular - lack of any symmetry in the particles.

9. Nodular - particles having rounded irregular shape.

10. spherical - Globular like particles.

Related to size :-

for regular shape particles it is quite easy to define the size of the particle whereas for irregular shape particles it is difficult to define the size of the particle.

Regular
spherical → dia

cube → body diagonal, face diagonal.

Irregular

particle size is determine by taking the mean of diameter along all the axis.

Methods of particle size determination :-

There are following methods are available for particle size determination.

1. microscopy → optical, Electron

2. Illutration.

3. sieve analysis.

Illustration :→

It is a process for separating particles based on their size, shape, density using a stream of gas or liquid flowing in a counter dr? with respect to sedementation. This method is mainly used for particles smaller then 1 micrometer.

Sieve analysis :→

This is the most important methode of particle size determination. It is done by manual sieving or mechanical sieving.

Mesh no –	8	16	30	52	72	150
Mesh size – (aparature size)	2.326	1.000	0.500	0.300	0.208	0.104

British	ASTM
→ The apparature size of two succesive screen varies with a factor of $2^{1/4} = 1.189$	→ The apparature size of two successive screen varies with a factor of $2^{1/2} = 1.414$.
→ It indicates the screens with theire mesh no.	→ It indicates the screens with their mesh size (apparatus size).

★ It is better to indicate the screens with their apparature size, rather then theire mesh no. because screens with same mesh no. may have different apparature depending upon the thickness of the wire used to manufacture such screen.

* The materials used to manufacture the screens are

Bronze (88% cu + 12% Sn),

Brass (Metal alloy of cu & Zn) and

stainless steel.

* ~~minimum~~ fine size
* maximum fineness - 0.037 mm (400 mesh no.) &

200 mesh no. is used as the reference screen.

ASTM standard sieve series :-

Sieve or screen analysis :-

The product consist of particle of various sizes. So it is impossible and impractical to determine the size of each particles. Hence the average size of the product is determined by sieve analysis.

1. Sieve analysis for determination of average size of the large particles.

i) A sample is taken from the bulk of material by using coning and quartering technique and the sample may consist of ten to 100 no. of particles.

ii) Measure out the dimensions of each particle in 3 perpendicular dr^n(s) - that is reflect the dimension of the particle along x, y & z axes.

iii) The data recorded for each particle is made into a table as shown below.

particle No.	Dimension			
	x_i	y_i	z_i	d_i
1	x_1	y_1	z_1	d_1
2	x_2	y_2	z_2	d_2
3	x_3	y_3	z_3	d_3
⋮	⋮	⋮	⋮	⋮
10	x_{10}	y_{10}	z_{10}	d_{10}

iv) Then find out maximum and minimum of value

If $\bar{d}_{max} / \bar{d}_{min} < 1.5$, then the average size of the product can be calculated by using the arithmetic mean, geometric mean, harmonic mean.

a) Arithmetic mean $(\bar{d}) = \dfrac{\bar{d}_1 + \bar{d}_2 + \cdots + \bar{d}_{10}}{10}$.

b) Geometric mean $(\bar{d}) = \sqrt[10]{\bar{d}_1 \cdot \bar{d}_2 \cdot \bar{d}_3 \cdots \bar{d}_{10}}$

c) Harmonic mean $\left(\frac{1}{\bar{d}}\right) = \frac{1}{d_1} + \frac{1}{d_2} + \frac{1}{d_3} + \cdots \frac{1}{d_{10}}$

If $\bar{d}_{max}/\bar{d}_{min} > 1.5$, then we use two methode.

a) In terms of sp. surface area.

$$\bar{d} = \frac{\sum_{i=1}^{n} n_i d_i^3}{\sum_{i=1}^{n} n_i d_i^2}$$

n_i = no. of particle under consideratio
d_i = mean diameter of the particle.

$$= \frac{(1 \times \bar{d}_1^3) + (2 \times \bar{d}_2^3) + \cdots + (10 \times \bar{d}_{10}^3)}{(1 \times \bar{d}_1^2) + (2 \times \bar{d}_2^2) + \cdots + (10 \times \bar{d}_{10}^2)}$$

Here 1 no. of particle having $\bar{d}_1$

b) In terms of total weight.

$$\bar{d} = \frac{\sum w_i}{\sum \frac{w_i}{\bar{d}_i}}$$

w_i = It is the individual weight of each particle with diameter d_i. usually in laboratory siev analysis we use 2nd formula to evaluate the average size of the particle.

2. Average size determination for small size particle using sieve analysis.

The feed is kept in the top sieve, after closing the lead of the top screen the entire set is kept on ro-tap sieve shaker and then allowed to be shaken for 15 minutes, then removed.

Ro-Tap sieve shaker :→

It consist of a movable ~~change~~ cage with a base (a) and a top plate (b) between which 13 half height or 7 full height sieves with pan and cover lid can be mounted.

The mounted sieves are subjected to rotary shifting motion while at the same time the lever (c) strikes the top plate once ~~often~~ per revolution. A timer switch with the motor is used to control the time duration of screening. The machine is designed in such a way that operates within a specified time period.

Industrial Screening :→

purposes of screening :→

(i) To prevent the entry of under sized material to the crossing machine so as to increase the capacity and efficiency of communition.

(ii) To prevent over sized material from passing to the next stage in closed circuit grinding or crushing.

(iii) To prepare closely sized feed for the next stage of concⁿ operation such as gravity concⁿ.

(iv) To prepare closely sized end product ~~for~~ as per specification and requirement.

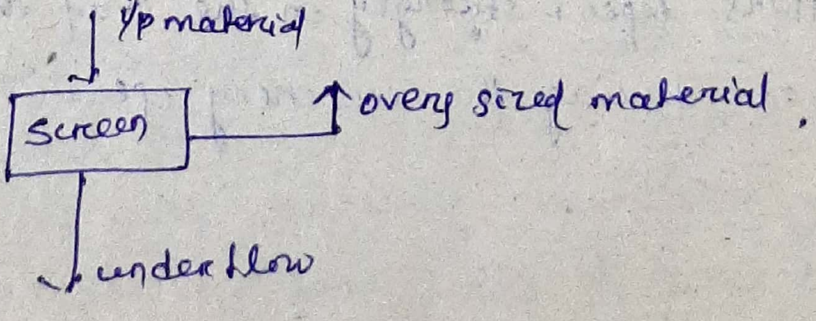
Mechanism of screening :→

(i) The material passing through the screen opening is known as underflow or undersized material, whereas the material retained over the screen was known as overflow or oversized material.

So the basic fact attach to screening is the passage under sized material through the screen.

The factors that affecting the passage of material are given by

1. The absolute size of screen opening.
2. The relative size of the particle to that screen aperture.
3. percentage of open area available on the screen.



4. The angle at which particle strikes to the screening surface.

5. The speed at which the particle strikes the screening surface.

6. The moisture content of material to be screened.

7. Probability of the particles to strike on the screen surface

1) Absolute size of the screen opening —
passage of undersized particle is inversely proportional to apparature size.

2) effect of relative particle size —
The relative particle and apparature size control the passage of particle through the screen, larger sized particle with larger aparature get easily screen as compare to the smaller size particles on finer screens.

3. percentage open area available on the screen —
If large nos. of openings are available on the same screen area, automatically quantity screen would go, but there is a limit in extend of open area due to the fact that the screens are made of the materials such as rods, wires. with having a definite dimensions and this dimensions depend strongly on the load that the screen is going to bear during the screening operation.

4. Angle at which particle strikes the screening surface -

A rod like particle gets through an aperture which is little above its diameter, if the particle hits the screen, with its long axis perpendicular to the screen surface however the same particle will not able to pass through a screen of larger aperture, when the particle reach the screen surface with its long axis parallel to the screen surface. Most efficients results are obtained when the particle hits the screen surface at an angle of (45 - 60)°.

5. Speed at which the particles strike to screen surface —

Effective screening is negligible when the speed of the particle is zero on the screen surface. With an increase in particle speed, the effectiveness of screening will increase however if the speed is excessively high, then the particle passes off to the over flow before it gets a change to go through any particular aperture of the screen.

If the screen doesn't vibrate properly it may clog completely by the over sized particles and thereafter no screening would take place. For effective screening Both vibratory and circular motions are simultaneously employ.

6. Effect of moisture in the field -

When little moisture is present in the feed material to be screened, the screening efficiency get reduced significantly. The moisture tries to bind few smaller particle into larger aggregates and such aggregates are large enough to pass through smaller screen opening. It is found that either totaly dry or ~~weight~~ wet pulps can be screened with relative easyness.

7. probability of particle to strike on the screen structure —

It the particle is given more than two upto eight chances of striking the screen surface, then the probability of screening is increased and quantitatively screening increases with increase in no. of chances of striking.

<u>Screening surface</u> :⇒

Screening surfaces are classified into different catagories such as

1. <u>Parallel Rods</u> - such a surface is usually made up of steel bars, channels, rails etc.

2. <u>punched plate</u> - The surfaces are punched steels sheets or plates of various patterns i.e - circular, hexagonal, rectangular, square & slot like..

3. <u>Woven wires</u> - The screening surfaces are woven carefully by gauged wires. These wires are generally made up of steel, bronze, copper & monels.

Monels - (63-67%) Ni, (28-34)% Cu, (upto 2.5)% Fe ([illegible])

Types of screen :→

The screens are classified as :

1. Stationary
2. Moving .

stationary screen :→

→ These screens are grizzlies . They consists of parallel rods, bars or woven wire mesh at an angle to the ground .

→ They have heavy screening surfaces. The bars are usually held togetcher at right angles to their length and are spaced at the desired distance sleeves on the bolts .

→ They are usually employed incase of porer screening .

→ A slope in generally provided so that the matercial failed onto the screen surface . would rolled down facilited in between screenerng .

→ A typically grizzly is used for stationary screening .

diagram :→

Moving screens :

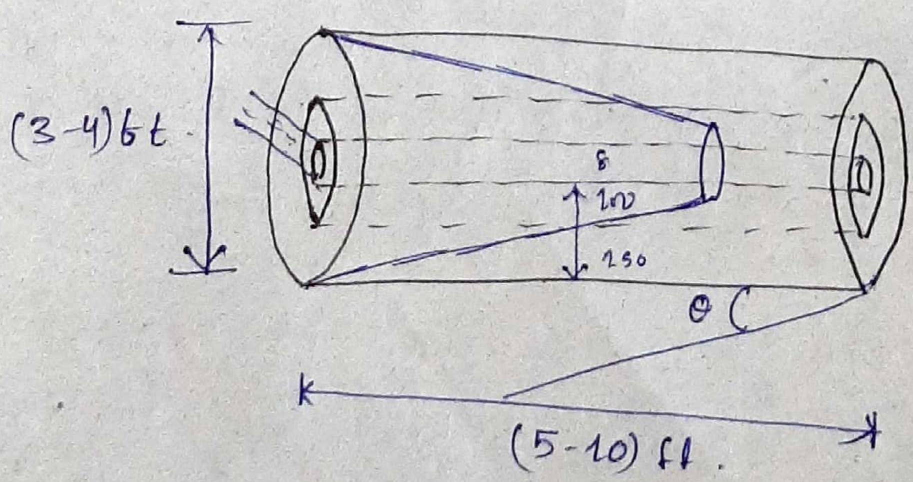
i) Moving grizzlies

- The grizzlies is made up of rods and bars but have movements as compared to the stationary screens in this case alternate bars or rods, alternatively raise and subside so that the feed material moved forward direction with sufficient turning over (flip over)
- There are different grizzles are used.
  - a) moving bar grizzlie.
  - b) chain.
  - c) Travelling
  - d) Disc / Roller type.
  - e) vibrating
  - f) shaking.

Advantage :

- Low floor space is required.
- act as a feder to the intermediate crusher.
- Better screening as compare to the stationary screening.

ii) Trommels / Revolving screen



→ It is used more widely than other type of moveable screens. It consists of rotating cylindrical, prismatic, conical or pyramidal shells or punched plates or thick weven wires.

→ A trommel has one or more shells which are arranged in a concentric manner. When the trommel has only one shell is known as simple trommel and more than one is known as compound trommel.

→ In case of compound trommel the screen opening decreases from innermost shell towards the outermost shell.

→ The trommel is commonly (3-4) ft diameter and (5-10) ft. length.

→ The trommel is driven by a central shaft attached to (electrical energy to mechanical energy) them by 4-6 armspiders, so that the material inside the trommel in revolved and get screened due to the rotation of the trommel.

→ The under sized materials comes out of the trommel all along the peripheri and oversized materials comes out at the other end.

→ The central shaft of the trommel is made to inclined on the horizontal to facilated automatic flow of the material from feed end to discharge end due to force of gravity.

Advantage :⇒

→ It requires smaller floor space.
→ It has a large capacity per unit screening area.
→ Cheap to operate.

→ Several fractions are obtained in the operation.
→ screening operation is quite efficient and can utilise both dry and wet screen.

iii) Shacking screen :→

→ It essentially consist of a shallo/swallo rectangular box where the length is atleast (2-4) times then the width.

$$l = (2-4)w$$

→ It is open at the one end and it feated a screen at the bottom.
→ Speed, slope, length of stroke should be adjusted to produce rapid stratification of the feed. with the forward motion, so that minimum blinding of the screen surface is resulted.

ex- coal screening.

iv) Vibrating screen :→

→ when one screen is used in the vibrating setup i's known as single deck vibrating screen, and if more than one screen is used that it is known as multi deck vibratory screen.
→ The screening operation is almost same as the trommel the coarset screening is carried out in the topmost screen and fine one is carried out at the bottom most.
→ In vibratory screen a large flate plane screening surface is attached rigidly with the frame which moves up and down in a reciprocating way by

electrically or mechanically.

→ In this case a no. of size fractions of materials is carried out in a single operation.

Advantage :→

→ It requires smaller floor surface.

→ It operates continiously.

→ less chances of screen blinding.

→ The screen surface can be repare easily as compare to trommels.

Disadvantage :→

→ Heavy wear occur between material & screen.

Comparesion → shacking & vibrating screen :→

shacking

Advantage :→

→ less cost of installation and operation.

→ It can be set almost parallel or flat during operation.

Disadvantage :→

→ More prove to heavy wear and required more frequent and expensive repaires compare to vibrating screes.

Vibrating

Operating characteristics of screens :→

The operating characteristics of any industrial screens are capacity, efficiency, operating cost.

Capacity :- Capacity of the screen depends upon the

(i) Area of the screening surface.

(ii) ~~Area~~ size of the opening.

(iii) characteristics of input ore such as specific gravity, moisture content, temp., proportion of fines.

(iv) Type of screening mechanism used.

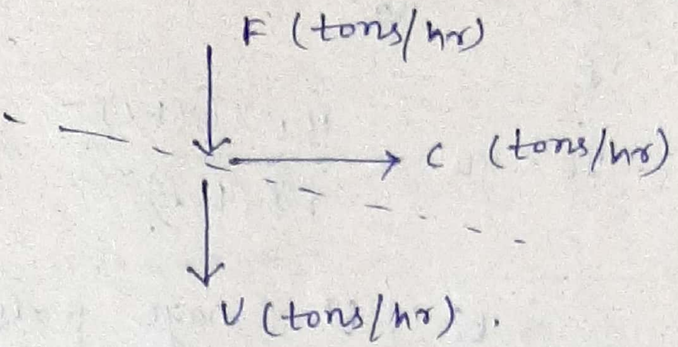
T (↑)	η (↓)
↓	↓
i/p − o/p	$\frac{o/p}{i/p}$

* Capacity (T) ∝ screening area (A)

$\propto \frac{1}{\text{efficiency}}$ $\left(\frac{1}{\eta}\right)$

Type of screen	Capacity Range (Tons/sq.ft area or mm aparture per day)
Grizzly	1 - 5
Trommel	0.3 - 2
shaking	2 - 8
vibrating	5 - 20

efficiency :- It is difficult to quantity the screen efficiency and it is defined as the effectiveness of screening operation as compare to perfect screening operation.



F → mass flow rate of solid in feed

C → mass flow rate of solid in overflow.

U → mass flow rate of solid in underflow.

f → fraction of undersized material in the feed.

c → fraction of undersized material in the overflow.

u → fraction of undersized material in the underflow.

1-f → fraction of oversized material in the feed.

1-c → fraction of oversized material in the overflow.

1-u → fraction of oversized material in the underflow.

$$F = C + U \quad \text{—— (1)}$$

$$Ff = Cc + Uu \quad \text{—— (2)}$$

$$F(1-f) = C(1-c) + U(1-u) \quad \text{—— (3)}$$

$$\eta_{\text{over sized material}} = \frac{\text{oversized material in the overflow}}{\text{oversized material in the feed}}$$

$$= \frac{C(1-c)}{F(1-f)} \quad \text{—— (4)}$$

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$$\eta_{\text{under sized material}} = \frac{\text{under sized material in the under flow}}{\text{under sized material in the feed}}$$

$$= \frac{U_u}{F_f} \quad \text{—— (5)}$$

Net efficiency (η) = $\eta_o \times \eta_u = \frac{U_u \times C(1-c)}{Ff^2(1-f)}$

$$\boxed{\eta = \frac{10,000\,W}{uF}}$$

W = tunage passing through the screen for each F. (tons of feed)

u = percentange of undersized material in the feed.

Operating cost :→

The operating cost of the screen is mainly depends upon the power consumption. attendent (Labour cost), replacement and repairing.

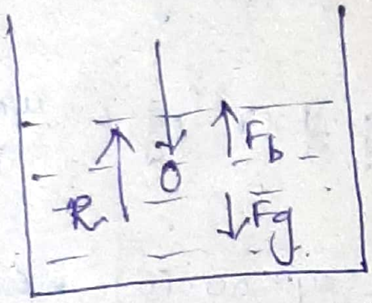
→ For stationary screen power cost is nill.

Movement of Solids in fluid

Introduction :→

The movement of solids in the fluid plays an important role in various classification processes such as gravity concⁿ, heavy media separation, jigging, tabling, thickening, & filtration.

Fluid Resistance and Terminal velocity :→



Net force acting $F_N = F_g - F_b$.

When a solid particle is emerged in a fluid it is acted upon two types of forces in stationary condition.

(i) gravitational force (F_g) = mg —(1)

(ii) buoyancy force (F_b) = m'g —(2) (m' = wt. of the liquid get replaced)

Net force acting in the downward dirⁿ $F_N = F_g - F_b$ —(3)

If $F_g > F_b$, then $F_N(\downarrow)$

If $F_g < F_b$, then $F_N(\uparrow)$

If $F_g = F_b$, then $F_N = 0$.

When the solid moving through the fluid medium it experience a another type of force. i.e. known as viscous force.

So, In this condition, the net force $F_N = F_g - F_b - R$

$$= mg - m'g - 6\pi\mu r v \quad \text{—(4)}$$

r = radius of the particle.

v = velocity of the particle.

μ = coefficient of viscosity

$$\Rightarrow F_N = \frac{4}{3}\pi r^3 \cdot \rho_{\text{particle}} g - \frac{4}{3}\pi r^3 \cdot \rho_f g - 6\pi\mu r v g.$$

$$\Rightarrow m \cdot \frac{dv}{dt} = \frac{4}{3}\pi r^3 \rho_p \cdot \frac{dv}{dt} = \frac{4}{3}\pi r^3 \rho_p g - \frac{4}{3}\pi r^3 \rho_f g - 6\pi\mu r v \quad \text{—(5)}$$

dividing $\frac{4}{3}\pi r^3 \cdot \rho_p$ through out the eqn no. (5) we have

$$\frac{dv}{dt} = \left(\frac{\rho_p}{\rho_p} - \frac{\rho_f}{\rho_p}\right) g = \left(\frac{\rho_p - \rho_f}{\rho_p}\right) g - \frac{9}{2}\frac{\mu v}{r^2 \rho_p} \quad \text{—(6)}.$$

$$- \frac{9}{2}\frac{\mu v}{r^2 \rho_p}$$

ρ_p = sp gravity of particle.

After certain time the velocity of the particle remains constant. So $\frac{dv}{dt} = 0$.

Hence the eqn (6) becomes

v_t = terminal velocity.

$$\left(\frac{\rho_p - \rho_f}{\rho_p}\right) g = \frac{9}{2}\frac{\mu v_t}{r^2 \rho_p}$$

$$\Rightarrow \boxed{v_t = \frac{2}{9}\frac{(\rho_p - \rho_f)\, g \cdot r^2}{\mu}.}$$

This eqn is known as Stoke's law of settling.

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Validity of stoke's law.

The assumptions are :-

i) The particle settling in the fluid is completely spherical.

ii) The container walls do not affect the settling of the particle.

iii) The fluid doesn't interact with the particle either chemically or physically to affect its settling.

iv) The presence of other particles doesn't affect the settling of the particle.

stoke's law only valid for settling of small sized spherical particles (< 60 micron for quartz particle) within the laminar or viscous flow condition. for ~~too~~ higher ~~terminal~~ velocity and larger particle the terminal velocity. So stoke's law fail to credit the terminal velocity. So newton's law of settling is used under that condition.

The settling under stoke's law, is known as free ~~sto~~ settling and under newton's law is known as Hindered settling.

General principles of free settling :⇒

(1) Specific gravity :⇒

If two particles of same ~~size~~ size and shape having different specific gravity, then the particle having high specific gravity will settle faster.

(ii) Size :⇒

Two particles of same specific gravity and shape, the larger one will settle faster, then the smaller one.

(iii) shape :⇒

spherical particles settle faster than, narrow, flat, and rod shape particle.

(iv) specific gravity of the fluid :⇒

In two different fluid of two different specific gravity, the particle will settle faster in the lighter fluid.

$$\eta_{\text{lighter fluid}} < \eta_{\text{heavier fluid}}.$$

$$\mu_{\text{lighter fluid}} < \mu_{\text{heavier fluid}}.$$

Hindered settling :⇒

→ Due to the mutual Interference motion of the particle, the resultant velocity of the particle get achieved is less then that of the computed velocity under free settling condition. Such type of settling is Known as Hindered settling.

Newton's law of settling :⇒

→ Under stoke's law the conditions are highly ideal or practically not visible since when a particle moves in the liquid, it is bound to create some turbulence and the following activities are bound to takes place :-

i) Inter particle collision and mutual interference.

ii) Defect of the container wall on the particle movement.

iii) The shape of the particle can never be totally spherical and the shape of the particle has a strong effect on the settling speed.

Taking all the above factors into the account the modified law of settling is given by the modified terminal velocity equation, i.e -

$$v_t = \sqrt{\frac{8}{3Q} g \frac{(S_p - S') r_p}{S'}}$$

S_p = specific gravity of the particle.

S' = specific gravity of the fluid

Q = coefficient of fluid Resistance which varies with the shape of the particle and with the orientation of the particle. with the dr^n of relative motion.

example :- The Q is about 0.4 for $r_p > 0.2$ cm

$V_t \propto r_p^2$ (stoke's law)

$V_t \propto \sqrt{r_p}$ (Newton's law

pulp or slurry :-

when fine particles are added to the fluid in large quantity they get suspended in the fluid and forms a psudo fluid with an apparent specific gravity i.e- higher than that of a pure fluid is known as pulp / slurry.

In such cases the specific gravity of the fluid medium must be replaced by the pulp or slurry under turbulent flow (newton's law), turbulent condition. Hence terminal velocity under hindered settling condition given by

$$v_t = \sqrt{\frac{8}{3Q} g \frac{(\rho_p - \rho'') r_p}{(\rho'')}}$$

$$\boxed{\rho' < \rho''}$$

Equal settling particles

The particle are said to be Equal settling if they have same terminal velocities in same fluid under the same field of force.

The free settling ratio, i.e- In sort . F.S.R is calculated by applying the stoke's law as follows

$$v_{t_1} = v_{t_2}$$

$$\Rightarrow \frac{2r_1^2(\rho_{p_1}-\rho_f)g}{9\mu} = \frac{2r_2^2(\rho_{p_2}-\rho_f)g}{9\mu}.$$

$$\Rightarrow \left(\frac{r_2}{r_1}\right)^2 = \frac{\rho_{p_1}-\rho_f}{\rho_{p_2}-\rho_f}$$

$$\Rightarrow \boxed{\frac{r_2}{r_1} = \sqrt{\frac{\rho_{p_1}-\rho_f}{\rho_{p_2}-\rho_f}} = FSR}$$

Similarly the hindered settling ratio can also be written as

$$HSR = \frac{r_2}{r_1} = \left(\frac{\rho_{p_1}-\rho_f}{\rho_{p_2}-\rho_f}\right)^m.$$

$$0.5 \le m \le 1.$$

The concept of free and hindered settling ratios (FSR & HSR) can be suitably employed to segregate particles in classifiers according to their sizes and specific gravities.

Application of FSR & HSR :-

Let us imagine two different mineral of two different specific gravities are in a mixture form and to be segregated as per their specific gravities.

Those the particles are almost of equal size they will have different terminal velocities due to the difference in their specific gravities.

Let us consider two particles having specific gravities $S_{P_1} = 5$ and $S_{P_2} = 2$ respectively.

when allowed to settle freely in water medium, they provide a free settling ratio under stoke's law.

Here the fluid medium is water, $S_f = 1$.

$$FSR = \sqrt{\frac{5-1}{2-1}} = \frac{2}{1} = 2.$$

$$HSR = \left(\frac{5-1}{2-1}\right) = 4.$$

In case of FSR (under stoke's condition) the particle '1' is moving two times faster then the particle '2' where as in HSR (newton's law) particle '1' is moving ~~two~~ four times faster then the particle '2'.

If closely sized feed is allowed for settling then it will results in segregation of heavier and lighter particles into two fractions as two layers.

In general for closely sized feed newton's law of settling is more preferable.

Assignment-2

1) consider a suspention of sand-water. The apparent specific gravity of suspention is ℓ'' = 1.66. In the suspention two particle of specific gravity ℓ_{P_1} = 5, ℓ_{P_2} = 2 is allowed to settle. Then Find HSR ?

2) Consider a suspention of sand-water containing 40% solid, 60% water. (f_s = 40%, f_w = 60%) by volume. The apparent specific gravity of suspention is 1.66. In this suspention pure anthracite coal is treated for specific classification. Find out the HSR if specific gravity of pure coal is 1.7. (ℓ_c = 1.7) & specific gravity of sand = 2.65. and also suggest which one will be better or effective (HSR & FSR) with proper explanation.

16/3/19

Classification :→

→ It is a process by which particles of various sizes, shapes and specific gravities are separated into separate groups by allowing them to settle in a fluid medium.

→ The coarse and heavier grains settle faster than the finer and lighter grain.

→ usually air and water are used as the fluid medium.

→ The classification is primarily based on stoke's law of settling.

Factors affecting the classification

(i) specific gravity :- If two particles of same size and shape having different specific gravity, then the particle having high specific gravity will settle faster.

(ii) size :- Two particles of same specific gravity and shape the larger one will settle faster, then the smaller one.

(iii) shape :- spherical particles ~~sto~~ settle faster than, narrow flat

(v) Air bubbles :- Atherence of the air bubbles to the solid particle would decrease the settling speed.

Types of classification processes

Sizing classification	Sorting classification
1) classification is based on the size of the particles.	1) classification is based on specific gravities of the particle
2) solid content is about (3-4)% by weight of the total suspension.	2) Solid content is about (40-70 by weight of the total suspension.

Classifiers :-

- On the basis of nature of the solid particle there is two type of classifier is used.
  1. Sorting classifier. (Based on specific gravity of the particle)
  2. Sizing classifier. (Based on size and shape of the particle).

- On the Basis of type of fluid medium the classifier is again divided into two categories.
  1. Classifier using dilute aquous suspension as the fluid medium for classification.
  2. Classifier using air as the fluid medium for classification.

Sorting classifiers :-

i) Hindered settling takes place.

ii) Separation of particle is Based on specific gravities of the solid particles.

iii) The dense suspension of 40-70 % of solid particles by weight is used in sorting classifier.

The types of sorting classifiers are

1. Evan's classifier
2. Richard's hinder settling classifier.
3. Richard's pulsator classifiere.
4. Hydrotator classifier.

%.

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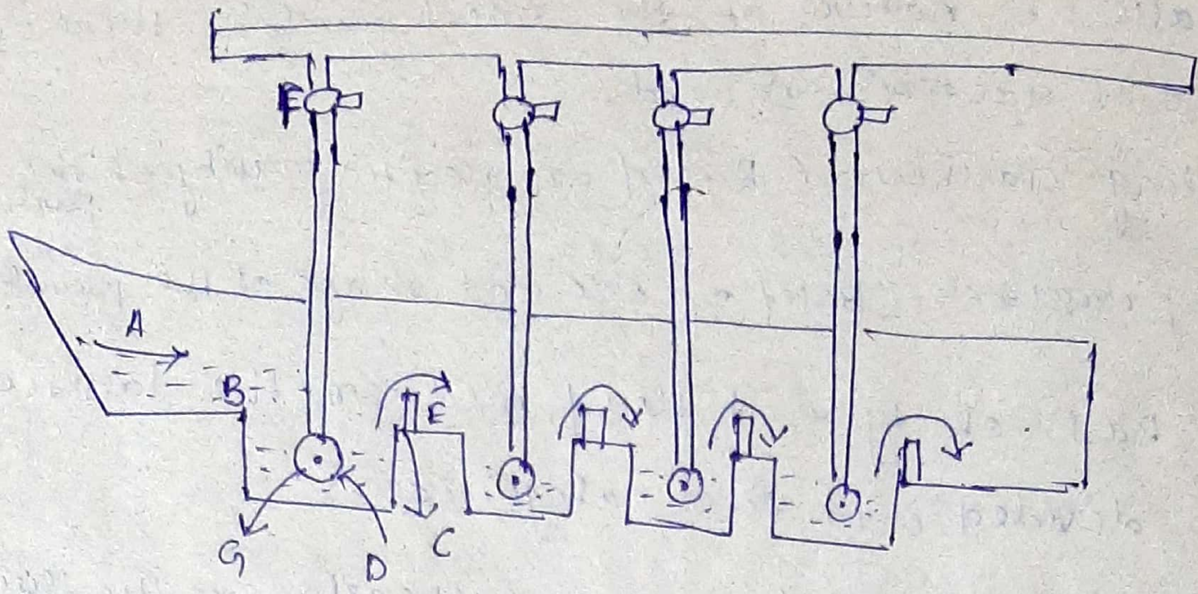
(i) Evan's classifier



BC → Rectangular box

G → Spigot.

D → Discharge.

E → Baffle.

F → controlling valves.

Evan's classifier consist of the following component,

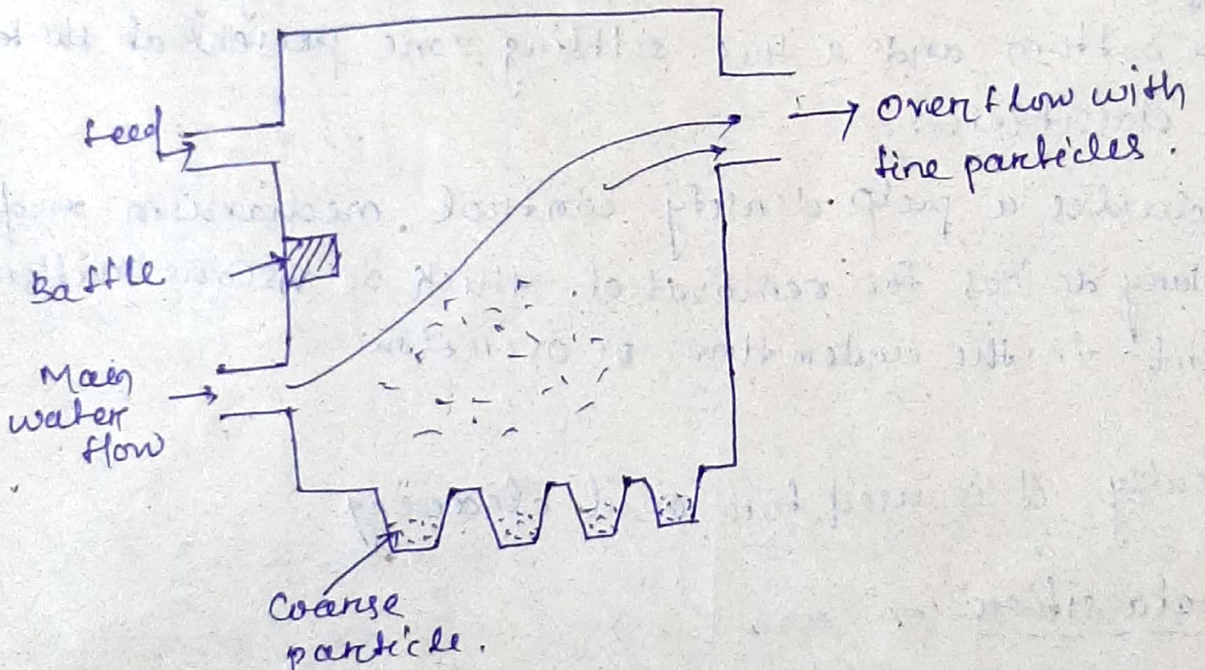
→ A slopping launder (A) with which several rectangular boxes (BC) are attached and that capable of discharging the discharge (D) through spigots. (G). water is introduced into the rectangular boxes through the pipes from the main pipe line. the water supplier is controlled by the controlling valves (F). The faster settling particle discharge through the spigot and the slower particles overflow at baffles (E) to the next rectangular box.

→ Baffles are the flat plates that controles the flow of water. So that to restrict the returning of solid particles to the same rectangular Box from where they have been taken away as overflow.

→ Depending upon the no. of rectangular Boxes and spigot several fractions of solid products are obtained.

→ Water flow rate in each successive pipe is reduced as the sizes of the particles settling get reduced successively.

iii) Richard's Hinder settling classifier :→



→ It is a modified version of evan's classifier. In this classifier cylindrical centering column replace the boxes of Evan's classifer.

→ The water used for settling purpose is supplied from buttom of the classifier.

(iii) Richard's pulsator classifier :→

Richard's pulsator classifier is characterised by the use of an ~~interepretent~~ intermittent or pulsating upward current of water designed to make hindered settling.

(iv) Hydrotator classifier :→

→ In hydrotator classifer a hindered settling zone present at the bottom and a free settling zone present at the top of the classifier.

→ It includes a pulp density control mechanism and auxillary devices for removal of thick or viscous matter present in the underflow or overflow.

→ Generally it is used for coal cleaning.

Sizing classifier :→

→ Sizing classifiers utilise free settling condition to impart sizing as much as possible being unaffected by specific gravity and shape of the particle.

→ It do not require any additional water besides i.e. present in the suspension under going classification.

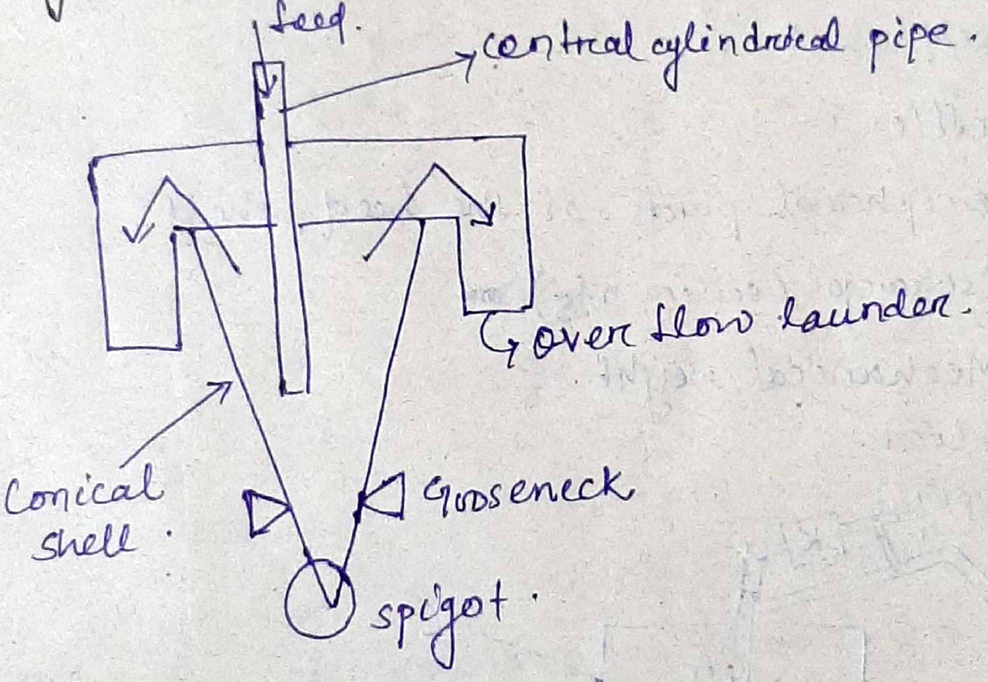
→ sizing classifier may be subdivided into two types.

a) settling cones having no moving parts.

b) Mechanical classifier having moving parts.

→ classifier using air is known as pneumatic classifier, where the settling speed is around 100 times faster than the settling speed of water classifier.

a) Settling cones



→ Settling cones are the conical sheet metals shells with apex at the bottom and a periferal overflow launder at the top.

→ Feed is charged through the central cylindrical bottom less pipe to prevent the by passing of the feed to the ore.

→ spigot at the bottom of the chonical shell discharge the sediment.

→ A Gooseneck pipe of adjustable height is provided to guide the sediment away from the tank.

Mechanical classifier → Allen-cone-classifier

→ It involves the moving parts.

→ It associated with automatic discharge of the classified materials.

→ The important parts or components of Allen-cone classifier are.

A → feed shell

B → Baffle

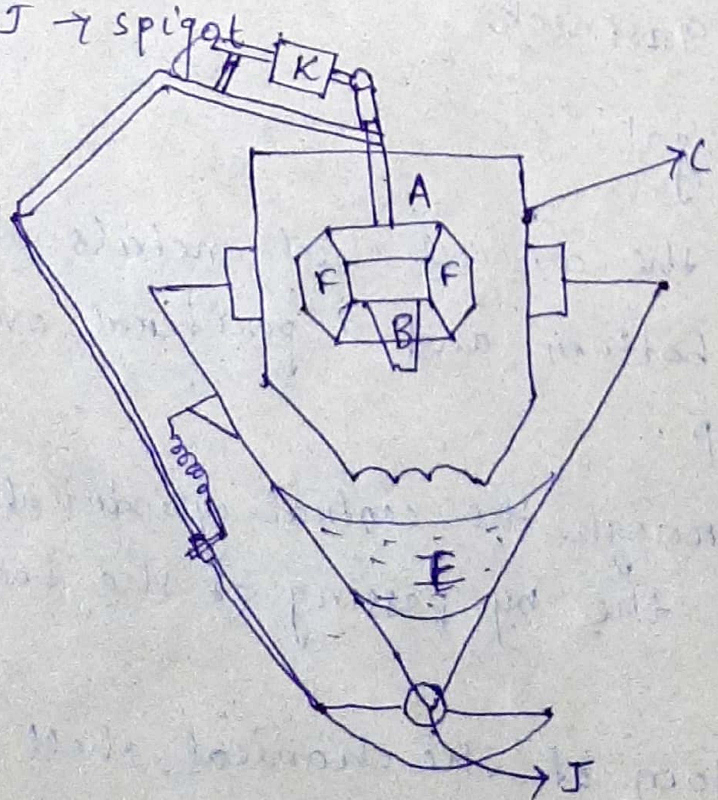
C → peripheral parts of the feed shell.

E → Discharge (sediments)

K → Mechanical weight.

F → float

J → spigot



- A float (F) is situated within the cylindroconical shell (C), which surrounds the feed shell (A). The baffle (B) is working against a spring to keep the spigot (J) closed.

Working principle:→

→ when the level of sediments (E) rises sufficiently in the cone, it prevents the passage of ~~both~~ pulp/slurry from the feed shell to the body of the classifier. Then the float is raised and it opens the spigot allowing the discharge to takes place automatically. Discharge will continue until the float is brought back to its pre-determined initial position. A mechanical weight (K) is used to regulate the density around ~~the itself~~ of feed shell (A).

Performance of classifiers:→

capacity (C)

$$C = a A v r s$$

where a = constant having value 1.875

C is expressed in tons/hr.

A = crosssectional area of the sorting column in square feet.

v = upward velocity or rising velocity of the fluid in the sorting column, expressed in feet/min

r = percentage of solid in the classifier intake or feed expressed by volume.

s = specific gravity of the solid.

Q:- The cross sectional area of the classifier is 10 feet^2. The rising velocity of the fluid in the sorting column is 1.5 ft/min. The solid content in the suspension is 8% by volume. Calculate the classifier capacity if the specific gravity is 2.65.

$a = 1.875$ $A = 10\ ft^2$. $B = 1.5$ ft/min. $\delta = 2.65$

$Y = 8\%$ by volume.

$= 0.08$

$C = 1.875 \times 10 \times 1.5 \times 2.65 \times 0.08$

$= 5.9625$ tons/hr.

Efficiency :-

The usual method consists of screening of the classifier overflow and underflow, and then efficiency can be calculated using the formula.

$$E = 100 \times \frac{c(f-t)}{f(c-t)}$$

where E = efficiency expressed in %.

t, f, c = are the content of ~~[illegible]~~ -x material in the overflow, feed, underflow. respectively
(c) (f) (t)

If sum feed is by passed the overflow then the efficiency increased theoritically, but practically there is no increase in the efficiency and hence the proposed formula to calculate the efficiency is

$$E = \frac{10,000\,(c-f)(f-t)}{f\,(100-f)\,(c-t)}$$

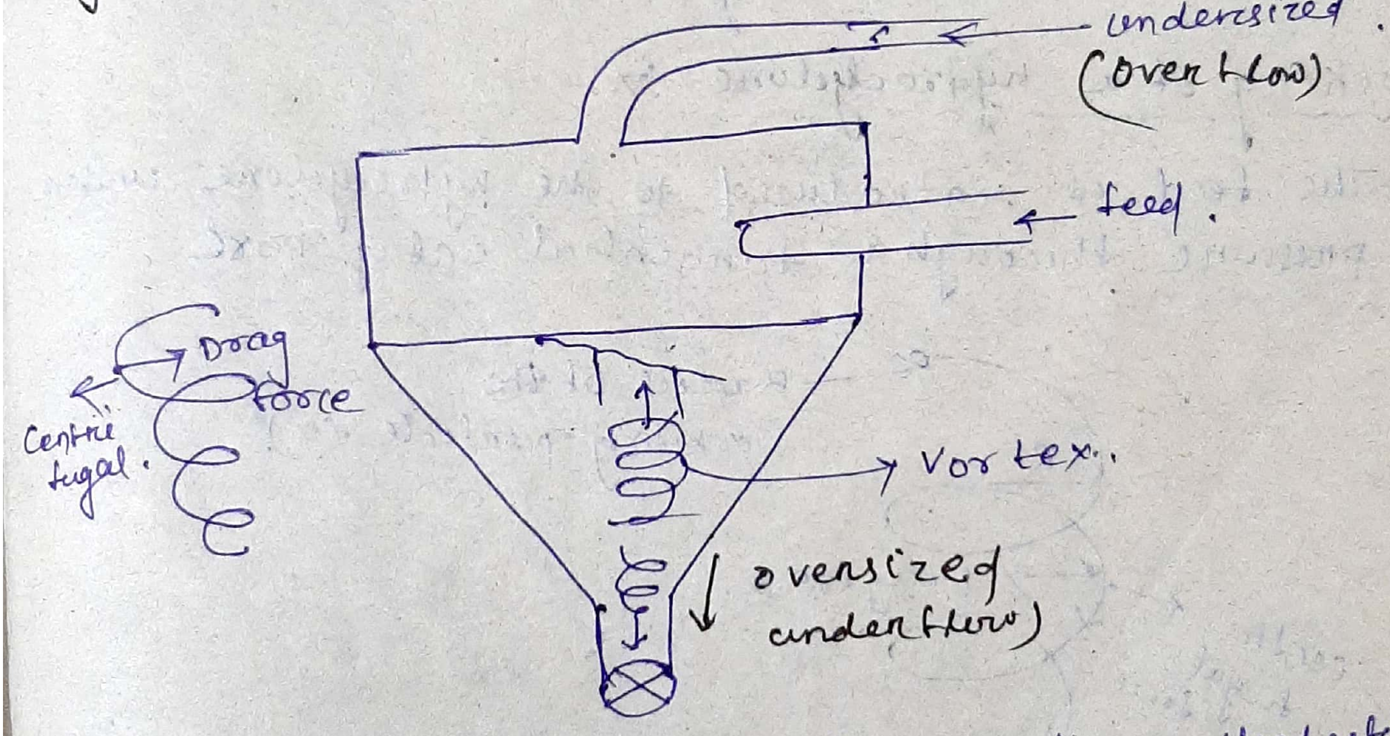
efficiency of classification generally varies from (50-80) %.

cost of operation :→

→ low cost except for fine size materials.

→ It largely depends on capital cost & inventory cost

cyclone separator :→ (Hydro clone)



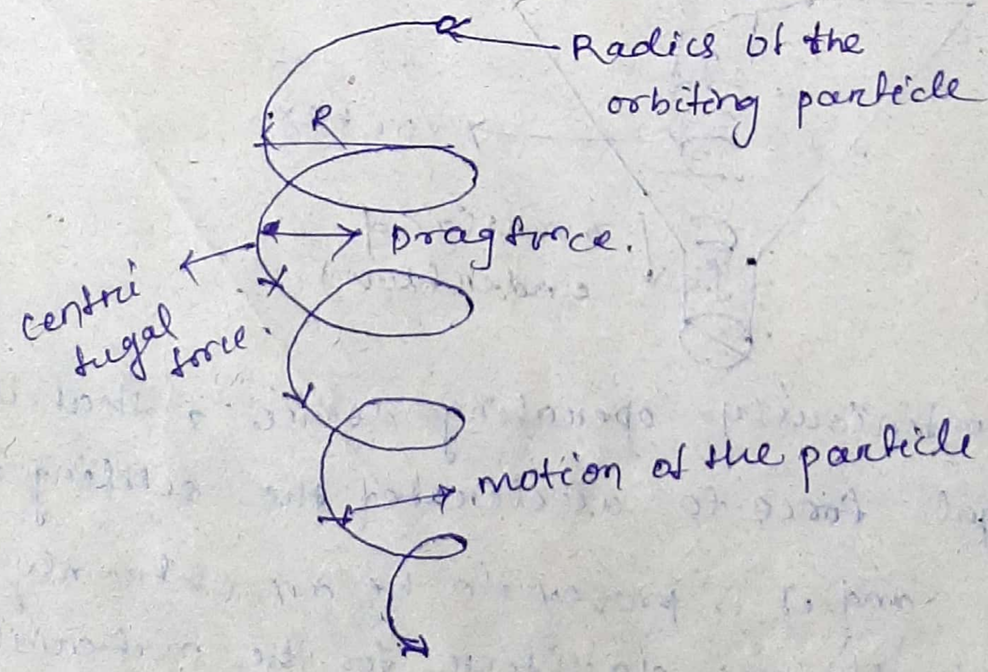
It is a continuously operating device, that utilises the centrifugal force to accelerated the settling rate of particle and it is proved to be an extremely important and efficient classifier for the materials in the range of 5 - 150 micrometer.

construction :-

A typical hydrocyclone consist of a conical shaped vessel. open at its apex, or underflow. A conical vessel is joined to a cylindrical section having a tangential feed inlate. the top of the cylindrical section is closed with a plate having an axially mounted pipe for overflow. The pipe is extended into the body considered as a removal section that known as vortex finder, which prevents the short circuit of the feed directly into the overflow.

working of a hydrocyclone :-

The feed is introduced to the hydrocyclone under pressure through a tangential entry port.



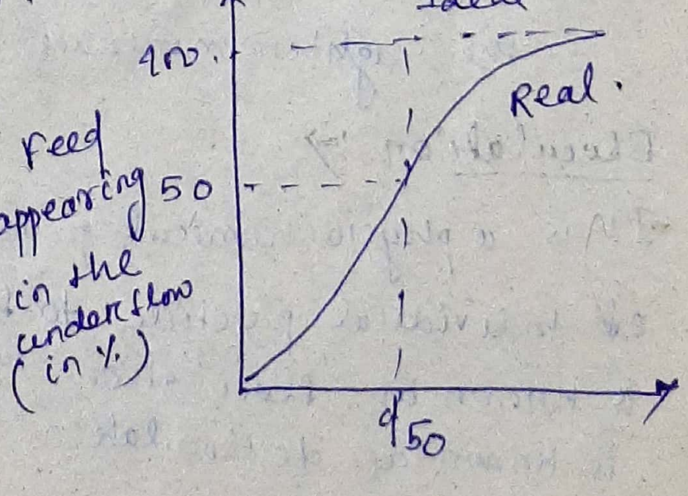
The tangential entry impacts a ~~swelling~~ swirling motion to the pulp that generates a votex with a low pressure zone along the vertical axis. when particles are introduced into the cyclone, they are

subjected to two types of opposing forces such as.
(i) outward centrifugal force.
(ii) Inward drag force.

The centrifugal force accelerates the settling rate of the particles and thereby separate the particles according to their sizes and specific gravity. Faster settling particles move to the periferal wall of the cyclone where the velocity is lowest and migrade to the apex opening. Due to the action of drag force, slower settling particles move towards the center of the cyclone, A low pressure zone and are carried upward through the vortex fined through the overflow.

cyclone efficiency :→

It is represented by a performance or partition curve. The cut off point or separation size of the cyclone is defined as that point on the partition curve for which 50% of the particles in the feed of the size report the underflow.



This means particles at this size having an equal chance going either with the underflow or overflow. This point is usually referred as d_{50} size.

Similarly d_{75} & d_{25} can be find out. The efficiency of separation or imperfection (I) is given by

$$I = \frac{d_{75} - d_{25}}{2 d_{50}}$$

Classification by means of concentration :→

Sorting operation along with gravity concentration as become a important methode of concentrating ore. This is done in two ways:

i) on the basis of size where the value able constituents of the crushed ore are in one size range and the waste materials are in another size range.

ii) on the basis of specific gravity, where the stratification is occure as per the theory of hindered settling. In this case the lower stratum consist of the heavier minerals and the upper stratum consist of the lighter minerals.

Floculation :→

It is a physiochemical process which increases the tendency of individual particle to form the flocx and settle down is known as floculation. The reverse process of floculation is known as defloculation.

Basic phenomena associated with the slime control :→

1) Selective attachment of ions.

This will promote dispersion as it causes ion, covered particles to repel each other (or attract each other).

2) Brownian movement

The spontaneous movement of fine particles in a suspension is known as brownian movement. It is attributed to the bombardment of fluid molecules on the solid particles having concise surfaces, i.e. the fundamental region of flocculation, or deflocculation.

Natural tendency for surface energy to change to kinetic energy :→

(i) Flocculation of discrete particles reduces the interfacial area and the surface energy.

(ii) Flocculation can be induced by the addition of electrolytes

(iii) In case of negatively charged dispersion, cation of the electrolyte is more important and vice versa.

(iv) the examples of flocculating reagents are -

starch → $(C_6H_{10}O_5)_n$.

H_2SO_4 ; ferric alum → Ammonium Iron sulphate → $NH_4Fe(SO_4)$

(v) examples of deflocculating agents – cyanides, gelatin, glue, alkali sillicate, carbonates, sulphides, hydroxides.

glue → polymer of vinyl acetate
($C_4H_6O_2$).

gelatin → $C_{102}H_{151}N_{31}O_{39}$.

Heavy media separation :→ (Module-3)

Introduction :- If a fluid is available whose specific gravity is intermediate between two solids which are to be separated, then as per the law of buoyancy, one of the solid will float ~~as~~ on the surface of the fluid, where the other one will sink to the bottom of the vessel. (in the fluid).

Basic principle of heavy media separation :→

→ Basic principle involve in the gravity ~~separation~~ concⁿ process which is carried out by using a fluid whose specific gravity is remains in betⁿ the specific gravities of the two mixed up minerals in the crossed ore. This is also known as float and sink phenomena. Most useful heavy fluid is acetylene tetra-Bromide whose specific gravity (ρ_{f_1}) is 2.96

→ The fluid can be diluted with carbon tetrachloride whose specific gravity (ρ_{f_2}) is 1.59.

This fluid can be diluted with any extent that will result a series of fluids, whose specific gravity varies from 2.96 to 1.59.

e.g:- solution of $ZnCl$ & $CaCl_2$

Gravity concentration technique/Method :⇒

It has lost its importance due to the Introduction of forth floation method. But again reintroduced due to the following advantages.

i) high capacity
ii) high efficiency.
iii) Low installation cost.
iv) Low power input.
v) enviroment friendly. (Less impact on the enviroment)
vi) No use of expensive chemicals.

Different technique of gravity separation methode

1. Jigging 2. Tabling 3. Flow film concentration,

principle of gravity separation Technique :⇒

It is based on separation of two minerals on the basis of their relative specific gravity as well as othere forces which acting upon it like viscous & buyoncy force.

Concentration criteria :→

provides ~~concentrat~~ the idea of amenability (possibility) of separation of two minerals and it is expressed as

$$\text{concentration criteria} = \frac{(\text{specific gravity of heavy mineral}) - (\text{specific gravity of fluid})}{(\text{sp. gravity of light mineral}) - (\text{sp. gravity of fluid})}$$

Jigging

The process of with stratification of minerals into different layers, on the basis of their specific gravities and sizes

principles of jigging :→

The three physical factors, that responsible for stratification of minerals during jigging are i) differential Initial Acceleration,

ii) Hindered settling.

iii) consolidation Trickling.

Differential Initial Acceleration :→

The heavy particles have a greater Initial acceleration & speed than the lighter particle. If the fall is repeated for sort duration then the total distance travelled by the particles bears a similarity to its initial acceleration than to its terminal velocity.

Mathematical derivation :-

Applying the law of sedimentation we have

$$m \frac{dv}{dt} = (m - m')g - R(v) \quad \text{---} (1)$$

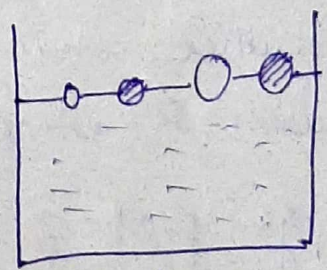
$$\Rightarrow \frac{dv}{dt} = \left(1 - \frac{m'}{m}\right) g \quad \text{---} (2)$$

where $R(v) = 0$ in the begining of the fall, where the velocity of the particle in the fluid is zero.

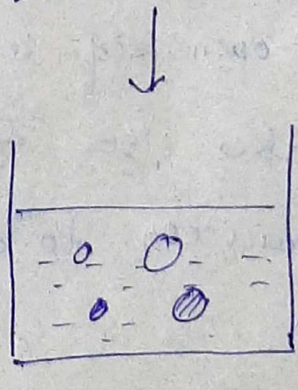
$$\frac{dv}{dt} = \left(1 - \frac{\rho_f}{\rho_s}\right) g \quad \text{---} (3)$$

The initial acceleration is independent on the size of the minerals but depended on the specific gravities of the mineral.

▨ – Heavy minerals.
▯ – light minerals.



If the minerals particles are allowed for longer time or duration during jigging, then they will attain terminal velocities, but for short duration, They are settle down according to their initial acceleration.



Q:- A mixture of quartz ($\rho = 2.65$), Galena ($\rho = 7.5$) (PbS) is rigged in a fluid having specific gravity $\rho_f = 2.0$. Find the ratio of their initial acceleration.

$$\frac{\partial v}{\partial t} = \left(1 - \frac{\rho_f}{\rho_s}\right) g$$

$$= \left(1 - \frac{2.0}{7.65}\right)$$

$$\frac{a_q}{a_g} = \frac{\left(1 - \frac{\rho_f}{\rho_q}\right)}{\left(1 - \frac{\rho_f}{\rho_s}\right)}$$

$$= \frac{\left(1 - \frac{2.0}{2.65}\right)}{\left(1 - \frac{2.0}{7.5}\right)} = \frac{0.25}{0.75}$$

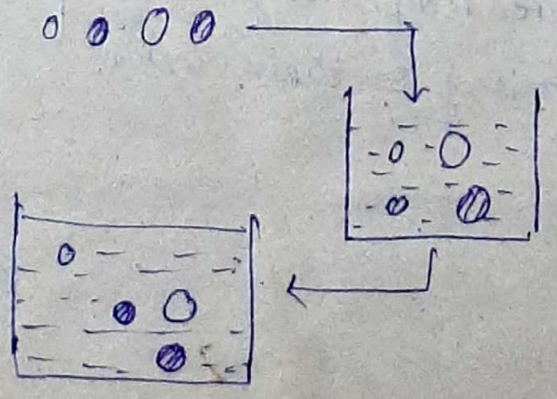
$$= \frac{1}{3}$$

Hindered settling classification :-

→ The difference between hindered settling in jigs and classifiers is interstitial fluid flowing through the particles rather than the fluid carrying the solid particle with its attence in case of classifiers.

→ As the jigs produce a fluidised bed for few second, it offers an open bed alternatively and the particle rearrangement takes place during that time period only.

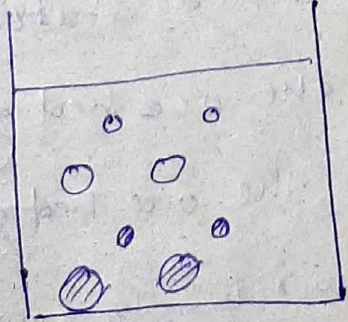
→ As comparision to classifier jigs provides higher settling ratio



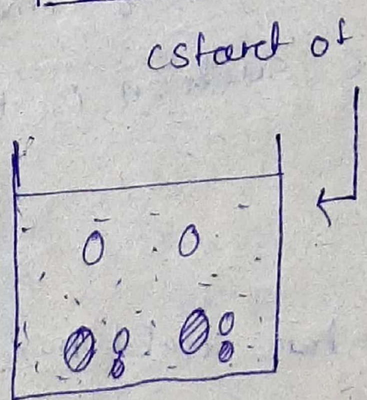
Consolidation Trickling :→

* It is fact that different particles of either same or different specific gravities do not travel the same distance during the settling period. Hence finer particle may appear at the top of the bed of coarse particles or the finer particles may run down through the interstitial pore spaces available in the bed of coarse particles under the influence of gravity and vibration.

* Hindered settling and the initial differential accln put the coarse-heavy grains at the bottom, fine-heavy and ~~coarse~~ coarse-light grains in the middle and fine-light at the top of the stratifier ore bed.. consolidation trickling somehow reverses the jigging process.

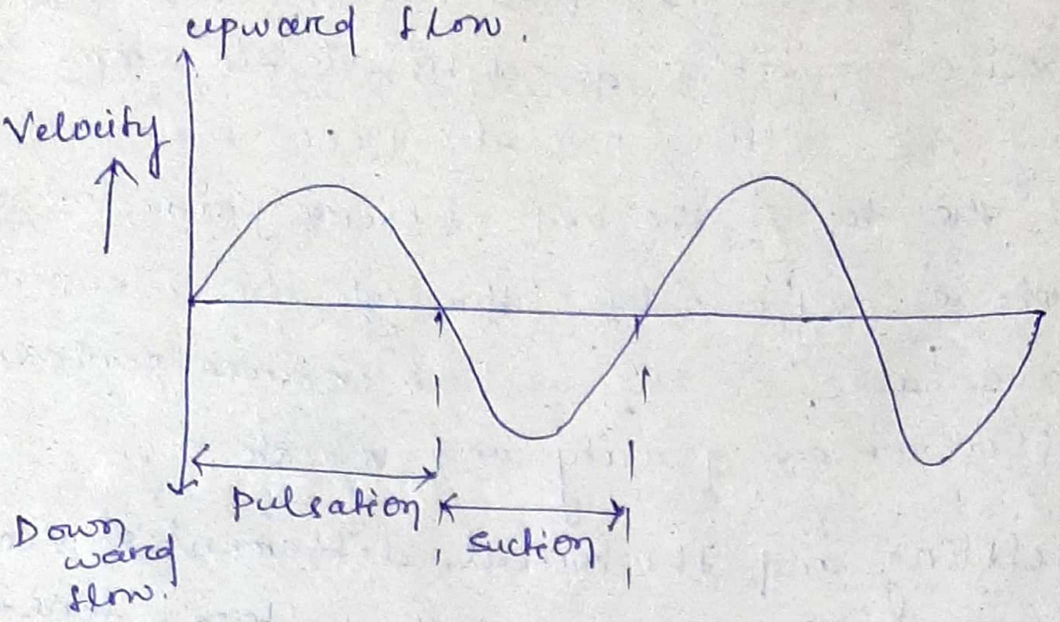


(start of consolidation trickling)



(End of consolidation trickling)

Jigging cycle :→



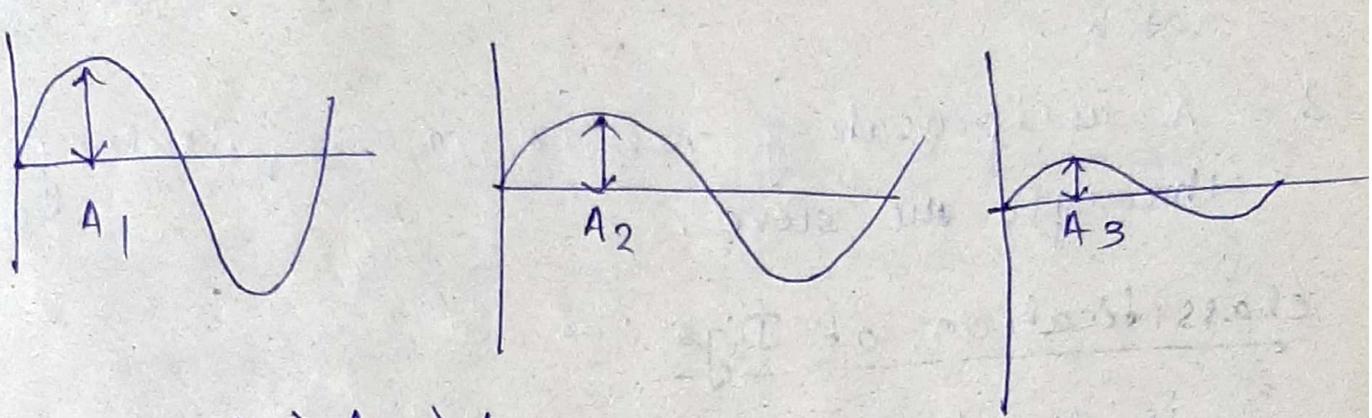
1. Jigging cycle is obtained by pulsation & suction of water or any other fluid through a bed of ground ore held on a perforated grade or sieve.
2. During pulsation & suction the fluid moves up & downward respectively with a statiuntenaly point. During pulsation the ore bed gets expand and while during suction the ore bed gets compact. The pole of fluid velocity w.r.t. to time describing a full cycle of pulsation & suction is teermed as jigging cycle.

JIG :→

→ A Jig is egentially a water field box in which a bed of mineral grains are supported on a perforated surface or a salen.

→ Jigs are usually made of wood or other materials.
The tailings of one compartment works as a feed for the next consecutive compartment in the series.

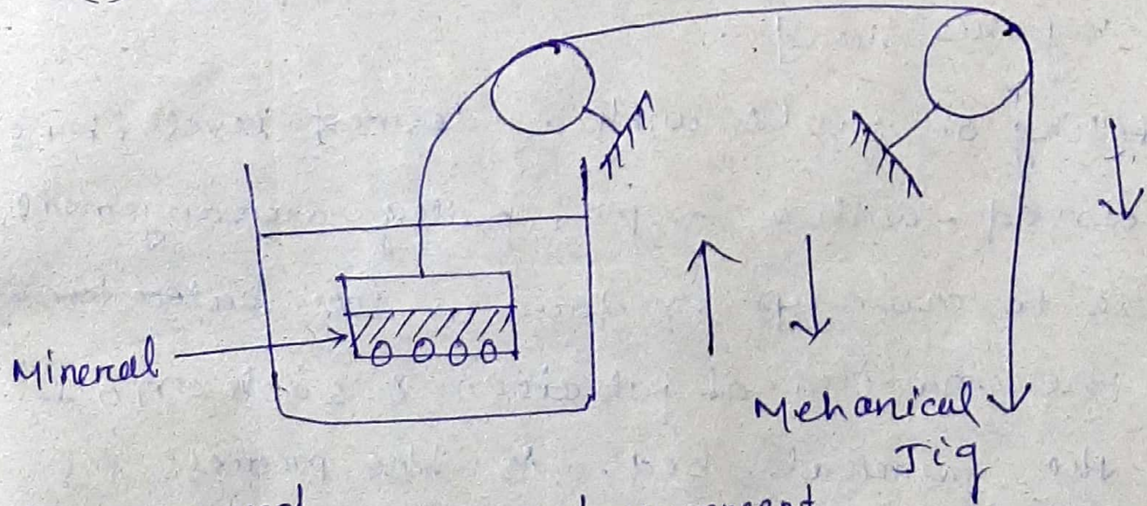
→ The amplitude of jigging is max. in the first cell & minimum in the last cell.



$A_1 > A_2 > A_3$
1st cell, 2nd cell, 3rd cell.

2 types of jig are used :-

(i) Hand
(ii) Mechanical.



Downward movement of rope → upward movement of perforated chamber. → downward movt. of fluid flow. → suction

Basic components of a Jig :→

The major components are :-

1. A shallow open tank containing a screen button on which the ore is supported

2. A hydraulic water chamber is also known as notch.

3. A reciprocating mechanism for pulsating water through the sieve.

Classification of Jigs :-

It is classified into two categories :-

(i) Hand Jigs.

(ii) Mechanical jigs.

Hand Jigs -

1. It consists of a framed sieve held by an operation supported by his hand.

2. After filling of nessle with a desired level it is tightly closed - with a rope, pulley arrangement. It is made to move up or down in free water tank to attain the condition of pulsation & suction of water in the mineral bed. As the process is repeated for several times, complete stratification takes place.

Advantages of Jigs :-

1. It is primarily used to concentrate coarse minerals
2. It is widely used to beneficiate non-magnetic iron ores.
3. It is cheap to operate.

Limitations of Jigs :-

1. It is absolete for beneficiation of sulphide ore.
2. It requires large amt. of water during ore beneficiation.
3. Fines can't be treated in jigs.
4. Jigs do not provide a complete solⁿ to any mineral beneficiation process.

Flowing film concentration & Tabling :-

$$Re = \frac{\rho V D}{\mu} = \frac{VD}{K}$$, Re = Reynold's no. $K = \mu/\rho$

$Re \le 2100$ (laminar flow)

$2100 < Re \le 4000$ (mixed flow)

$Re > 4000$ (Turbulent).

1. Viscosity of fluids defined as the internal friction of fluid which resists the shear force acting on the fluid. It is an intrinsic property of fluid, At a particular temp. & pressure.

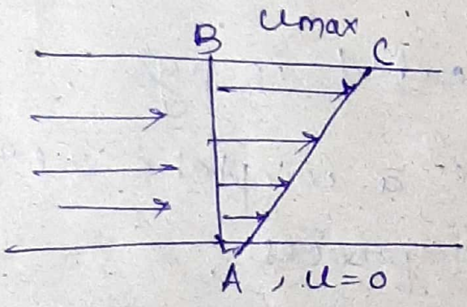
2. For a liquid, a rise in temp lowers viscosity whereas in case of gas rise in temp increase viscosity. Mathematically, $K = \frac{\mu}{\rho}$

(cgs unit is stoke (kinematic viscosity))

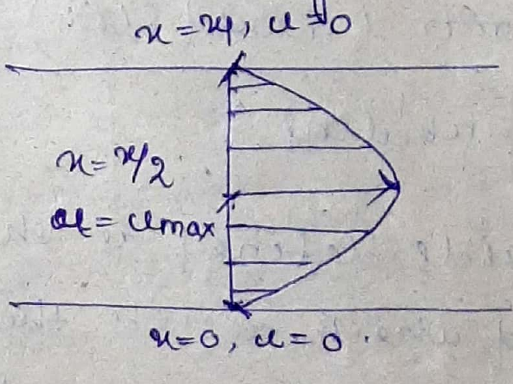
Flowing film concentration :→

→ when velocity of the fluid is not same at all the depth of the film.

→ Let us consider the fluid flow in a rectangular open channel, then the velocity is nill at the bottom and maximum at the top.



→ Similarly in case of a pipe, the flow rate is highest along the central axis and nill at the inner periferi of pipe. In turn the fluid flow is a function of viscosity of the fluid.



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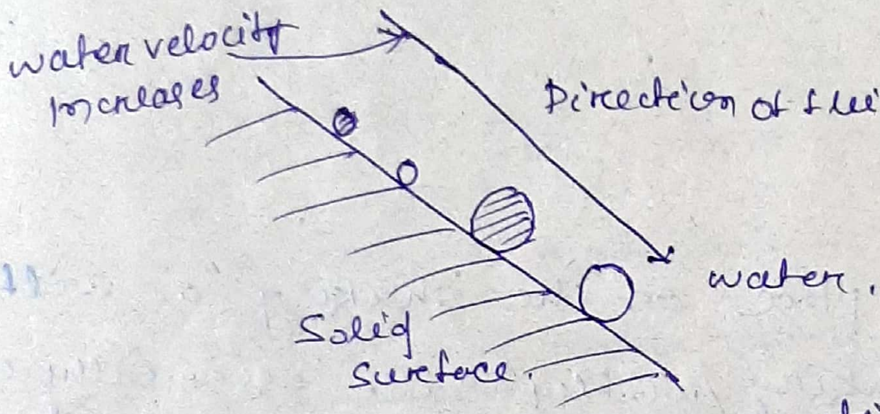
→ The velocity difference during the fluid flow at a different depth can be result in mineral beneficiation. The flowing fluid film can effectively separate coarse, light, small/fine, and heavy particles. The mineral beneficiation carried out by the above principle is known as tabling.

→ The action of the flowing fluid film on the minerals grains is schematically shown in the figure.



● → high sp. gravity particle.

○ → low sp. gravity particle.

Jigging	Tabling
coarse - light	Fine - heavy
Fine - light	Fine - light
Fine - heavy	coarse - heavy.
coarse - heavy	coarse - light.

Tabling :→

Diagram

→ Tabling takes place on the shaking or wilfley Table. The shaking/wilfley table essentially consist of a substaintially plane surface or flate surface known as deck.

→ The table is horizontal from left to right and slightly inclined

Shaken with an assymmetrical motion with in the drn of long axis.

→ Assymmetrical motion makes the stroke of the table faster in the one drn and slower in the reverse. Usually a slow forward with a rapid return causes the mineral particle to scroll along the longitudinal riffles is also known as cleats. that are fixed on the table surface in the drn of the table movement.

→ The wash water flows over the table at right angle to the drn of riffles. A feed of 25% solid by weight is introduced through the feed box at the top corner of the table and as the feed particles heat the deck, and they are fanned out by the combination of differential motion and transversely growing water.

→ The Jolt (certain moment) during the return stoke causes the heavier particles to work down the bed to form the bottom layer.

→ The lighter gangue materials are thrown into the suspension and are discharged out over the edge of the table opposite to the feed box by the wash water.

→ The heavier minerals finally arranged themself on the smooth unriffle portion of the table when they encounter the full force of the wash water.

→ The middling are collected in that portion of the table intermediate between concentrate and tailings

→ The reciprocating speed of the table is usually 200 to 300 stokes per minute with an amplitude of stroke length (12-15) mm.

→ A finer feed requires a higher Reciprocating speed with a smaller stroke length, whereas coarser feed requires larger stroke length with slower / reduced reciprocating speed.

<u>Construction of the wilfley table</u> :-

1. Feed box – use for feeding of the mineral ore.
2. Accentric drive – used for power input as well as the vibrator used for shacking the table.
3. Deck – plain surface / flat surface over which tabling takes place.
4. Riffles / cleats – These are the separators, i.e. covered 2/3 of the table surface which tappered from one end to other end and it required frequents replacement.
5. Water valve and multiple points – These are used for water supplier to washout the tailing parts.

characteristics of wifley table :-

classification :-

(1) The wilfley table can handle the coarse particle max upto 4 # and fine as 200 # . under idealised condition the particles are segrigated into four groups.

i) Light - Large
ii) Heavy - large.
iii) Fine - light.
iv) Fine - Heavy

} bottom → top.

(2) capacity of the table - It depends upon the following factor.

→ Angle of inclination :-
→ Higher the inclination, higher will be the capacity. (ton/hr)
→ Size of the feed :-
→ difference in specific gravities between the minerals.

(3) Cost of operation - It includes the power cost, repairing of the table, cost of the cleats or riffle, deck.

Important use of wifley table :-

(1) It is used to concentrate cassiterite → one of Tin → SnO_2.

(2) used to concentrate free milled gold ores.

③ used for beneficiation of non-metallic like glass and sand.

④ used for beneficiation of chromeatie and tungstan ores.

⑤ used to recover gallena (PbS) & sphalerite (ZnS) in coarse aggregate of Pb-Zn ores.

⑥ used for cleaning of fine coals.

Froth Floatation :→

Floatation method is

i) A weight concentration methode.

ii) A surface phenomena.

iii) Most efficient and complex process.

Floatation proper :→ In the process of floatation addhesion is made betn air bubbles & small mineral particles in such a way that they rise in the pulp. The floating mineralised froth is then skimmed off, while other minerals are retained in the pulp. The above fact is known as floatation proper.

SKIM floatation – In case of floatation the addhesion betn the free water surface & the mineral particle are affected. The particles involve in this process are usually larger than the particle involve in froth floatation. To obtain addherance of the desired mineral particles to the air bubbles, a hydrophobic surface film should be form on the particle surface where as hydrophilic surface film should be form around the particle which are retain in the pulp phase. So the whole phenomena related to surface film is known as skin floatation.

→ Most of the research has been done on floatation process and the results are summerised below.

i) Most of the minerals are suitably protected from contamination adhere to water and not to the air.

ii) Parafin and other hydrocarbons adhere to air in preference to water.

iii) Some minerals are naturally adhere to air due to surface impurities or due to inherent surface properties of the minerals, such a phenomena is known as natural floatability, where the mineral is non-polar in nature.
e.g. - coal, graphite, sulphur, hydrocarbons

iv) for minerals, to be separated by froth floatation, floatability is to be induced on the surface. this is known as aquired floatability.

→ suitable chemical reagents are to be added to the pulp to change their surface properties.

The reagents vary in nature depending on the type of ore to be floated. The group of reagents that used to achieve the acquired floatability is known as collectors or collecting agents.

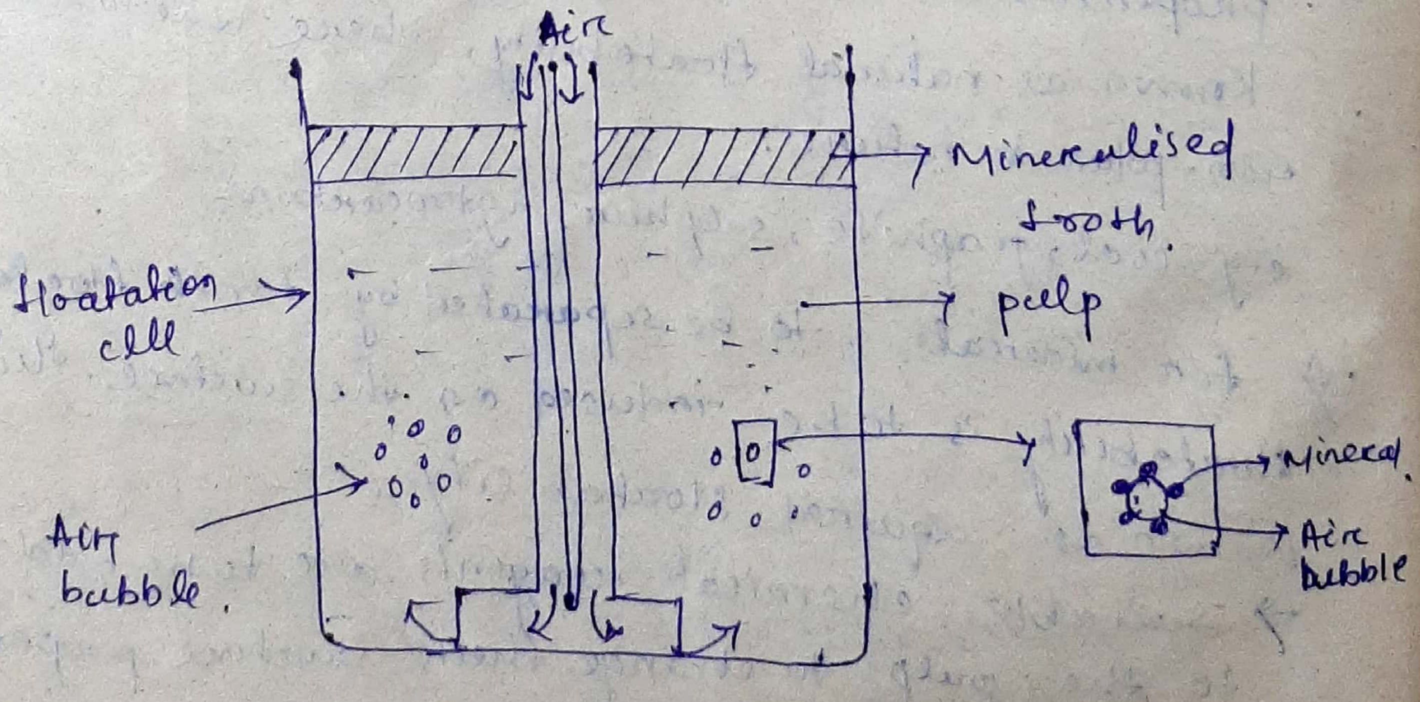
→ Other reagents that are used to selectively to separate out the minerals are known as activators/depressors.

→ Modifiers are the chemical agents that suitably modify the surface properties of the mineral.

v) Almost all minerals can be made of adhere air or water selectively by using suitably chemical reagent but it is not 100% efficient.

vi) The change in surface condⁿ of the mineral due to oxidation will considerably affect the floatability of such minerals.

Floatation Process :→



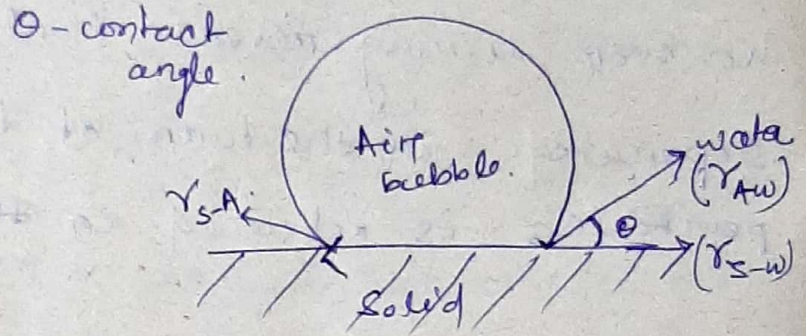
The general floatation depends on a no. of interrelated physico chemical factors. After treatment with reagents, the air bubbles attach with the mineral particles and lift them up to the surface of water. The mineral is usually transfer to the froth leaving behind the gange in the pulp. This is termed as direct floatation; however during reverse floatation, the gangue is separated in the form of froth where as the mineral particles is retained in the pulp.

physico-chemical principles of floatation :-

This is explained in terms of surface tension and surface energy, contact angle, polarity & adsorption.

① surface tension & surface energy, contact angle -

→ At any interface there exist certain amount of energy, that is known as surface energy



$S.T = \frac{F}{L}$

→ when a air bubble attached over a solid mineral part in the presence of water, then at eqn condition, we can write $\gamma_{S-A} = \gamma_{S-W} + \gamma_{A-W} \cos\theta$ — (1)

where γ_{S-A} - Surface energy betn Solid - Air interface.

γ_{S-W} - surface energy betn solid-water interface.

γ_{A-W} - Surface energy betn Air - water interface.

θ - contact angle.

→ The work of adhesion mathematically given by

$W_{S-A} = \gamma_{W-A} (1-\cos\theta)$.

→ greater the contact angle greater is the work of adhesion betn the particle & bubble. The floatability of mineral is therefore i~~ncreased~~ increased.

→ Minerals with higher contact angle are said to aerophilic and minerals with smaller contact angle are said to be aerophobic, or air repelling.

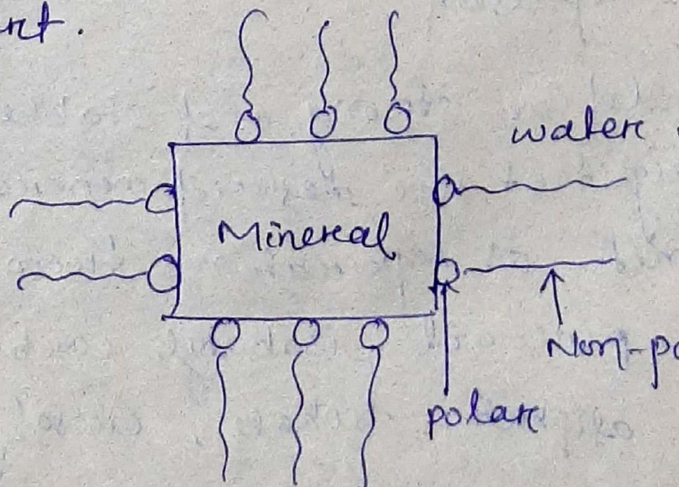
② polarity

→ The non-polar surfaces do not attached radily to the water phase is known as hydrophobic minerals. e.g - graphite, coal. So these are exhibited natural floatability & float over the water.

→ Minerals with polar type are hydrophilic & do not flow naturally on the water. These minerals upto acquired floatability to get floated up.

<u>N.B</u> - collectors are organic compounds which get adsorb on the surface of selected mineral particles & produce a contenious heterocopolar in such a fashion that the non-polar part of the film is oriented away from the mineral part.

e.g -



(3) Adsorption -

Floatation Reagent :→

→ The floatation Reagent are brodly classify into

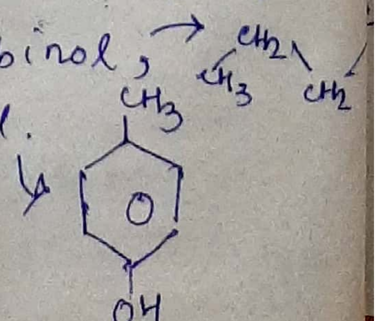
1. Frothers.
2. Collectors.
3. Modifiers.

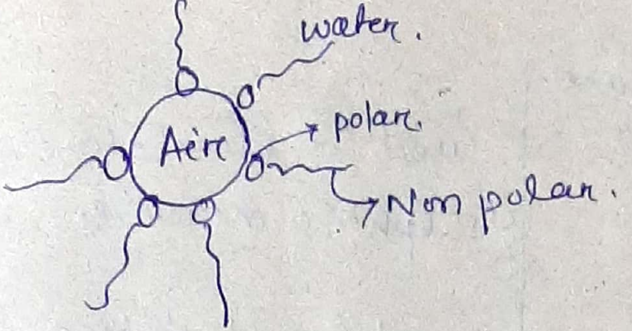
Frothers :→

→ The adsorption of frothers at the bubble - water interface reduces the surface tension and stablises the air bubble. In the froth bubble, the non-polar group is oriented towards the water phase providing necessary water repulsion.

→ The froth should be strong and stable enough to support the weight of the desired mineral attached to it and permits its seperation from the pulp.

e.g of frother - pine oil, isobutyl carbinol, aliphatic alcohols, cresol.





* collectors :→

→ The collector contains a polar group and a non-polar group. The polar group attach with the mineral surface and forms a continious heteropolar film all around the particle, whereas non-polar group is projected towards the water to provide hydrophobicity to the mineral surface.

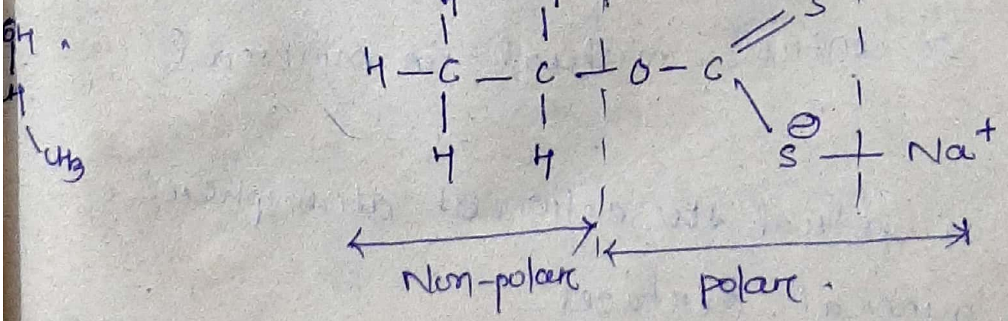
→ The collectors are brodly classified into two category. on the basis of chemical nature of the non polar part.

i) Anionic collectors.

ii) cationic collectors.

Anionic collectors

→ This is the most widely used collectors. If the non polar part of the collector which impacts water repelling behaviour to the mineral surface and carries a negative charge over it, and it's termed as anionic collector.

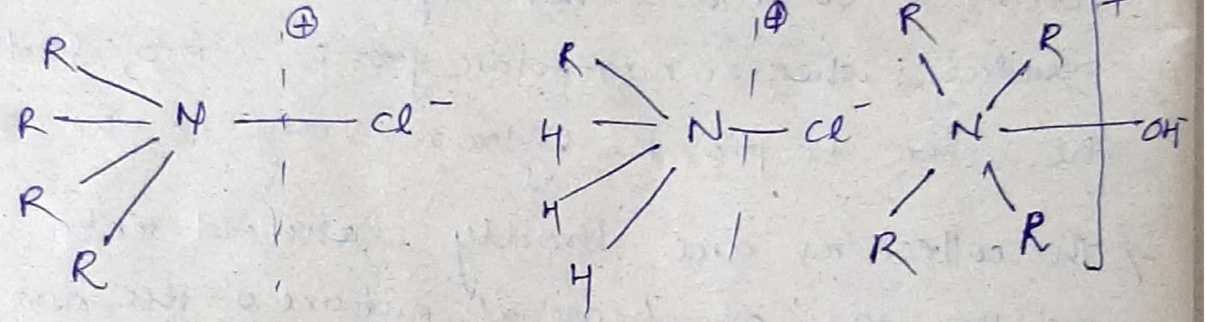


Name of reagent - [Sodium ethyl Xanthate]

e.g- potassium ethyl xanthate
Dithiophosphates.
Thiocarbonates - - - - - - - — used for floatation of sulphide minerals
Fatty acids.
sulphonates - - - - - - - — used for floatation of non-sulphide minerals.

Cationic collectors

→ The polar part carries positive ion.



e.g- fatty amine acitate.
NH_3Cl^-

→ The cationic collector is used for flocetation of oxides, carbonates and sillicate minerals.

Modifiers :→

The modifiers are used.

(i) to utilise collector optimum condⁿ.

(ii) to prevent or control mutual ~~in~~ mineral interaction.

(iii) To prevent or control the action of atmospheric air at the mineral surfaces.

into modify favorably or adversly the ability of some minerals to acquire floatability.

Modifying agents :-

Modifying agents may be clashed into following category.

i) - PH regulator.

ii) - Activator.

iv) - Depressant / Depressor.

v) - Dispersant.

PH - regulator

→ alkaline circuits are used for sulphide ores. The definite range of PH is (7 - 13) to obtain the optimum result.

→ The reagents that used to control the pH and to obtain the desired alkalinity are lime, soda, ash

Activator

→ Sometimes Xanthates are found ineffective in floating spalerite. Then under that condn activator is used to obtain the desired floatability of spalerite.

→ Activator ions are adsorb at the mineral surfaces and enhance the adsorption of collector at the same surface there after.

e.g - cu sulphate ($CuSO_4$) - used to activate spalerite (ZnS)

or H_2S / sulphide = used for galena (PbS)

$CuCO_3$ / lead nitrate – used for non-metallic minerals
$Pb(NO_3)_2$ with fatty acid type collector.

Depressant :-

- ~~used~~ It required to prevent ~~the~~ or supressed the floatation of mineral over another.
- Depressing agents are used only to assist separation of a mineral from another.
- The basic mechanism of this activity is that the depressant gets adsorb at the mineral surfaces and subsiquently inhabit the adsorption of collectors.

e.g - NaCN → along with $ZnSO_4$ used as depressant for sphalerite.

Dichromate salts → PbS.

Dispersant :-

Sometimes the gangue may have the nature of floculation along with the minerals. Then it becomes imperative to use a dispersant.

e.g- Sodium sillicate is used as dispersant.

Starch, casein and glue are used to disperse both gangue and carbonesius material that associated with the mineral.

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Operational principles of floatation :-

Floatation operation depends on the following factors.

1. particle size.
2. surface preparation of the minerals / conditioning.
3. pulp density.
4. Temp of the operation.
5. Time duration.

Effect of particle size in frooth floation

- floation process is most effecient for the particles in the size range of (-20-200) #.
- The failure to float coarse particle arese from :
  - a) Incomplete liberation.
  - b) Too small a contact angle.
  - c) Violent agetion required to form suspension.
- failure to float extremly fine particles are due to
  - a) poorer chance for minerals / bubbles.
  - b) The finer particles have an colder surface then coarse particle. As the surfaces of the particle is affected by the ions derived from other minerals, oxygen and water during fine grinding, they become unresponsive to the reagent and loose their capacity to float.

Conditioning

Conditioning is nothing but mixing of ore with the water and ~~aerasion~~ aeration prior to floation in a cell.

pulp density

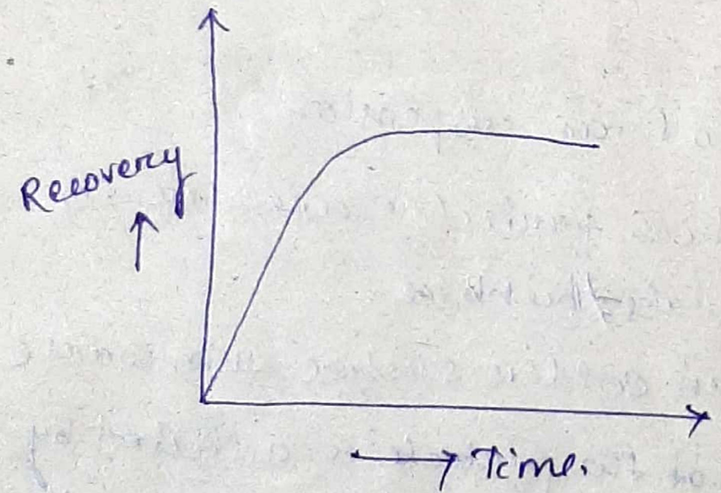
for the mineral and gangue particle to get separated during floation the pulp should be dilute enough, the permit particle rearrangement takes freely. A pulp density of 35% solids by weight shows the best result.

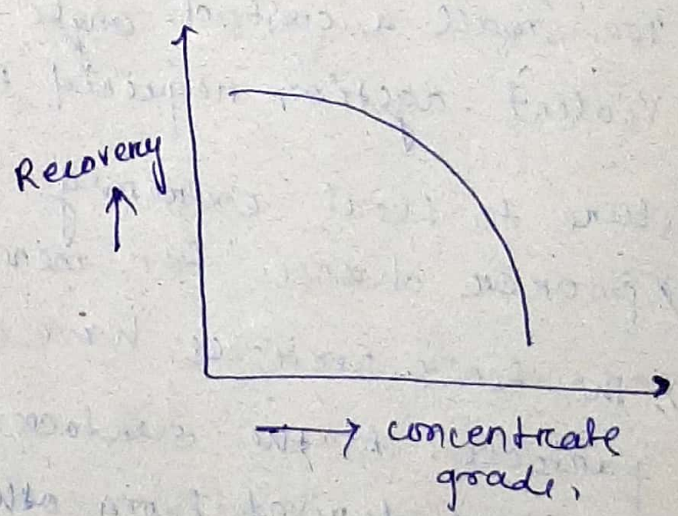
Temp. of the process

The temp. should be maintain 12-20°c.

Time duration of flotation

As the time duration is increases the extend of recovery increases with a fall in grade of the concentrate.



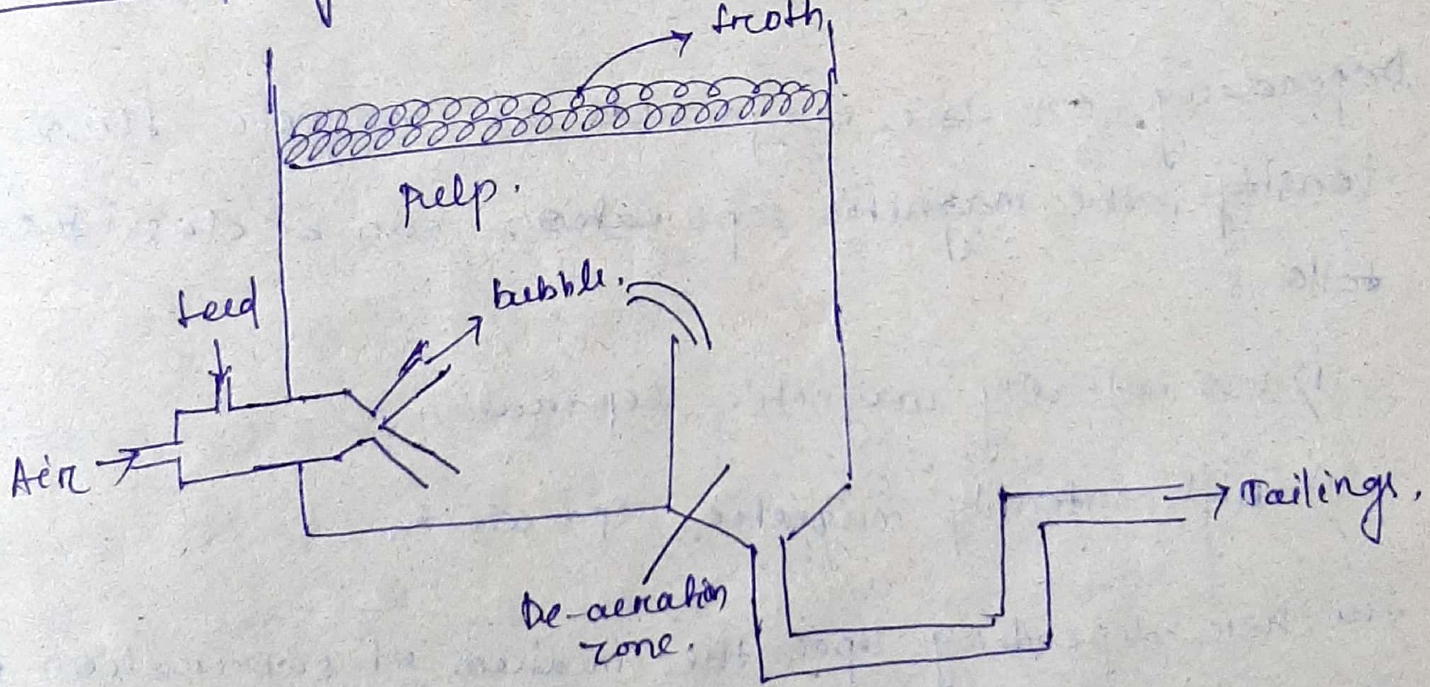


floatation cell

Pneumatic cell :→

In the pneumatic floatation cells compressed air is directly blown in to the pulp while in the suberersion cell a rotating impeller serves as a pump which draws the air through the hollow shaft of the impeller and distributes the same into the pulp to produce the froth.

Mechanically agesitated cell / Sub-aeration cell :→



A series of 10-15 cell connected in series are used simultaneously. they are connected in such a fashion that once a defroth pulp from the preceding cell as its feed. The recovery of such process is usually 90% or more than that.

Magnetic Separation :→

The method of separating different minerals on the basis of their magnetic property is known as magnetic separation, whereas the separation of valuable minerals from the gangue is known as magnetic concentration.

On the basis of magnetic field, the minerals are classified into 3 categories.

(i) Highly magnetic (Ferromagnetic)

(ii) weakly magnetic (paramagnetic) → illmenite, Garnets.

(iii) Non-magnetic (Diamagnetic) → Quartz, calcite.

Types of magnetic separation :→

Depending on the magnitude of magnetic flux density, the magnetic separation can be classified as follows.

i) low intensity magnetic separation.

ii) High intensity magnetic separation.

further depending upon the medium of separation it is again subdivided into two groups or two categories.

i) Dry magnetic separation.

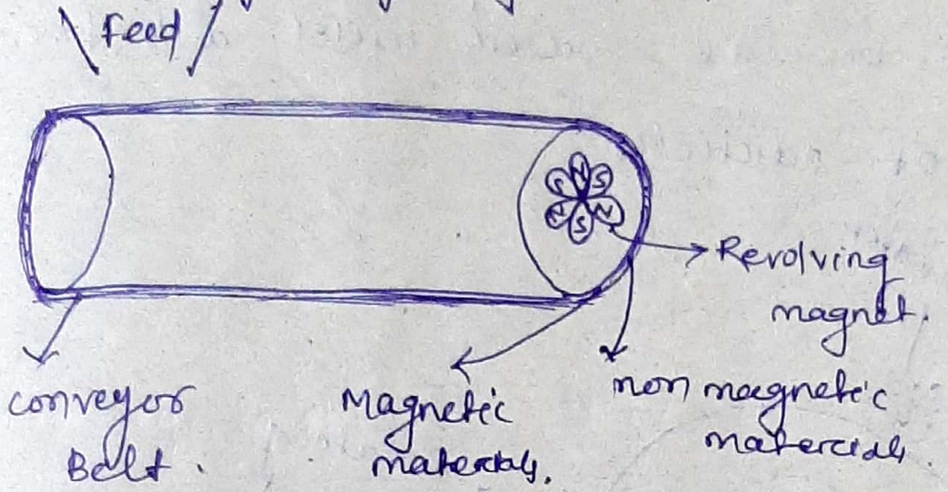
ii) wet magnetic separation

* How magnetic flux is produced ?

Ans :- Opposite poles (north & south pole) of conjucative magnets forms a / develops a strong magnetic field.

Different types of magnetic separators ⇒

1. Low intensity dry magnetic separator.



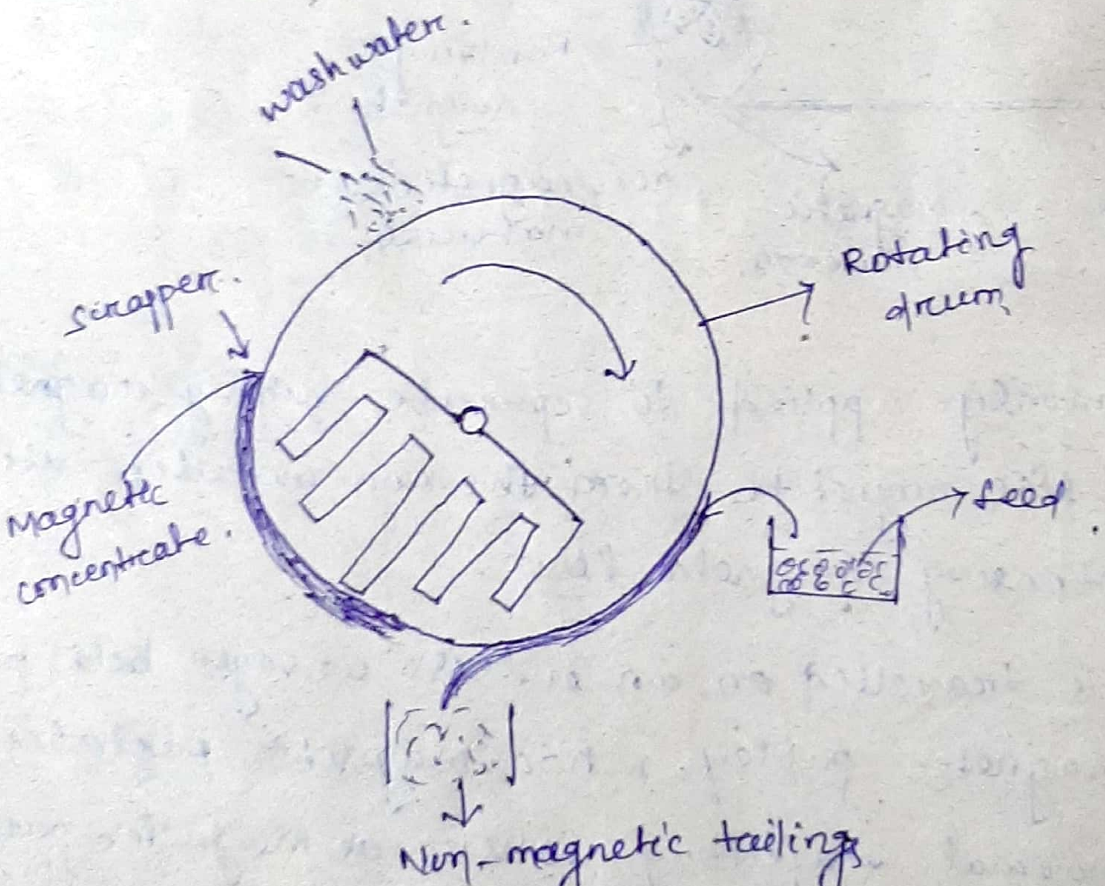
→ It is commonly applied to separate highly magnetic materials like magnetite from the non-metallics utilising a low intensity magnetic flux.

→ When ore is travelled on an endless conveyer belt, passing over a magnetic pulley, non-magnetic particles follows a normal trajectory where as magnetic particles are held farmly to the belt untill it is carried out of the feel and fall down when the belt just leaves the pulley.

2. Low intensity ~~weight~~ wet magnetic separator.

→ This is widely used for concentrating low grade magnetic ore.

→ Advantage - It can treat very fine ores almost in the slurry form, because water causes a better dispersion of particles.



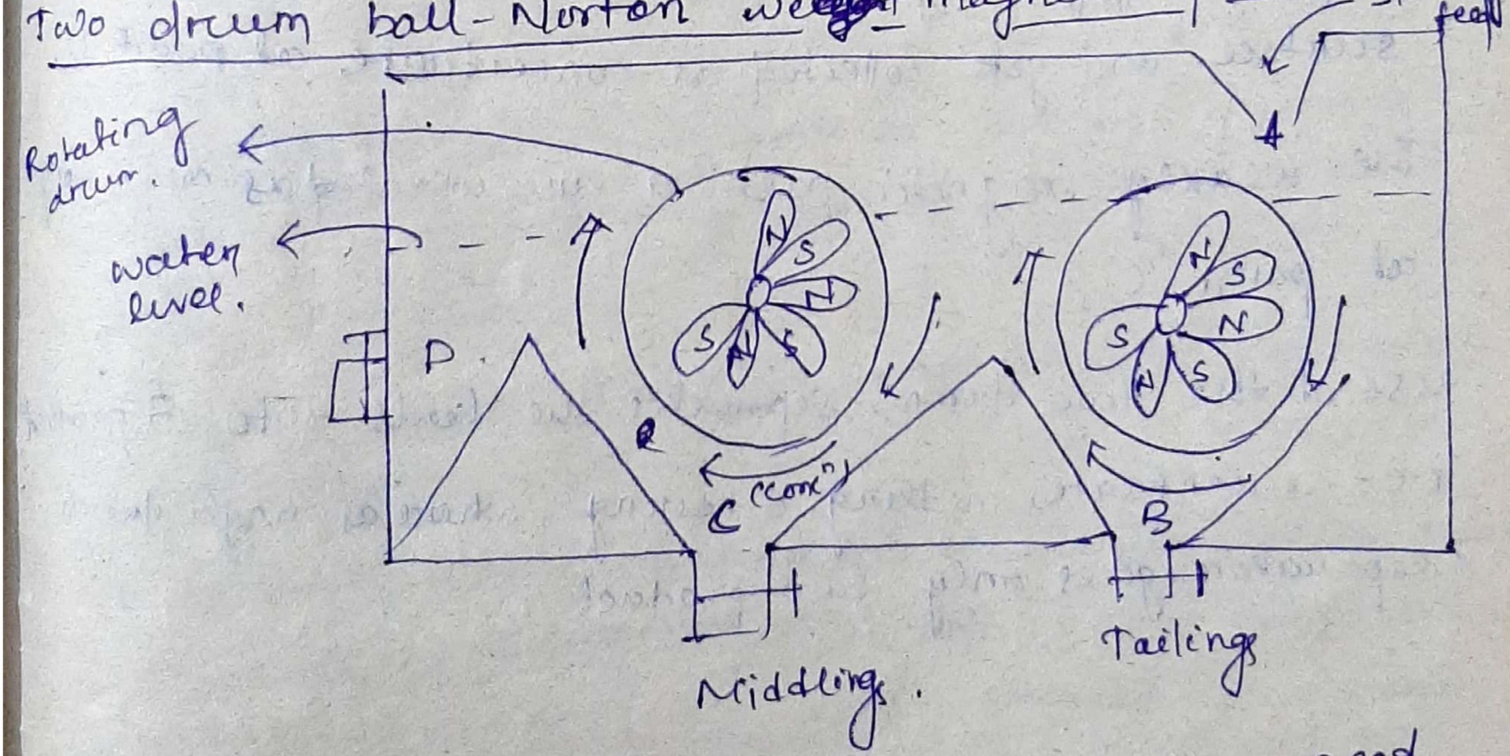
[Single drum weight magnetic separator]

→ The ball - Norton drum separatior consist of one or two rotating drums of non-magnetic metals. In the drum a no. of fixed magnets are arranged in such a fashion that consicutive poles are off opposite nature.

→ Much of magnetic field passes directly from one pole to other inside the drum and there by get wasted.

→ The enough magnetic flux, come out of the drum to attract and hold the magnetic particles strongly. The particles which are magnetic that stick to the surface of the drum and travel along the periferi and finally removed off from the drum surface with the help of a scrapper. The non magnetic particle just fall off at the edge of the drum, during rotation.

Two drum ball-Norton weight magnetic separator



This second drum rotates at a higher speed. and has a weaker magnets inside the drum. From the feed slurry both highly magnetic & weakly magnetic particles get strock to the surface of the drum. in the first compartment. while non-magnetic particles are remove as tailings at B.

Both weakly & highly magnetic particles travel along the surface of the 1st drum and are brought on to a place on the surface below which there is no magnet inside the drum.

From this point they are attracted by the magnets facing them in the 2nd drum and shift onto the surface of the 2nd drum placed in the next compartment.

As the magnetic field strength is low in the 2nd compartment only strongly magnetic particles stick to the drum surface and gets collected as concentrate at point D.

The weakly magnetic particles are collected as middlings at point 'C'.

use of the two drums separates the feeds into 3 product i.e- concentrate, midling & tailing, whereas single drum separator gives only two product

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3. High intensity separators ⇒

very weakly paramagnetic materials can be separated from the ore by employing high intensity magnetic field of 2 Tesla or more.

Cross-belt pick-up separator is very popular separator of this kind.

High intensity induced roll separator are widely used to treat beach sands, wolframite (Fe), tin ores & phosphate rock. It is also known as Dings induced roll separator.

The induced roll dings separator mainly consist of horse shoe magnet – A. An iron keeper i.e – D. Two rolls, one opposite is primary facing the magnet – A. Each pole.

The rolls are laminated to behave as a large assembly of secondary poles. The strength of these poles varies as the rolls ~~[illegible]~~ revolved. It becomes 0, twice per revolution. As the ore particles pass over a roll, the magnetic particles are drawn ~~[illegible]~~ on to the laminated roll and they fall down only when they are at a position, where the magnetic strength of the adjoining secondary pole is zero. This means magnetic particles continue to move along the roll surface to a greater distance compare to non-magnetic particle and fall off much later. So the feed is separated into two fraction as it passes through the rolls. For proper working of dings separator, closely sized feed is required and it operates best ~~over~~ above material size greater than equal to 75 micron. The effectiveness of the separation of fine particles is severely reduced by the effect of air current, particle-particle adhesion and particle rotor adhasion.

This is successfully used on material like mica & MnO_2. After magnetic separation is over the materials written sum amount of residual magnetism. The writtened magnetism is to be remove before the concentrate can be treated further. it means de-magnetisation.

→ Electrostatic separation.

→ Dielectric separation.

→ Electro-osmosis. :- Motion of liquid induced by an applied potential across a poros material, capillary tube, membrane or micro channel.

Electrostatic separation :→

→ It is a method of concentrating or separating minerals from each othere on the basis of their electrical conductivity.

→ The basic principle of electrostatic separation is based on coulomb's law, that implies like charges repel and unlike charges attract each other.

ex:- The electrostatic separation was 1stly employed to separate zinc ore from lead sulphide ore and recently it is used to separate out the non-metallics.

→ Electrical concentration can be broadly classified into two categories.

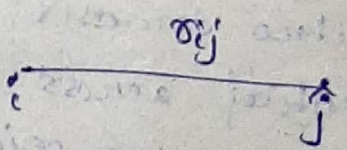
1) Electrostatic separation.
2) High tension separation.

Theory :-

It works on the principle of mutual attraction of unlike charges & mutual repulsion of like charges. In other words it is well known as coulomb's law, which mathematically given by,

$$F = \frac{1}{4\pi\varepsilon_0} \cdot \frac{|q_i q_j|}{r_{ij}^2}$$

$$= k_e \frac{|q_i q_j|}{r_{ij}^2}$$



On the basis of electrostatic charge a particle is said to be positively charge if it is deficient in electrons and is said to be negatively charge if it has excess of electron.

On the basis of electrostatic charge the materials can be classified into 3 categories.

→ conductor – when electrons are highly mobile in nature.
e.g – metals.

→ Insulator – It has no mobility of electrons in there.
e.g – plastic, rubber.

→ Semi conductors – Higher mobility of electrons in them as compare to the insulator and much less conductivity compare to the conductors.

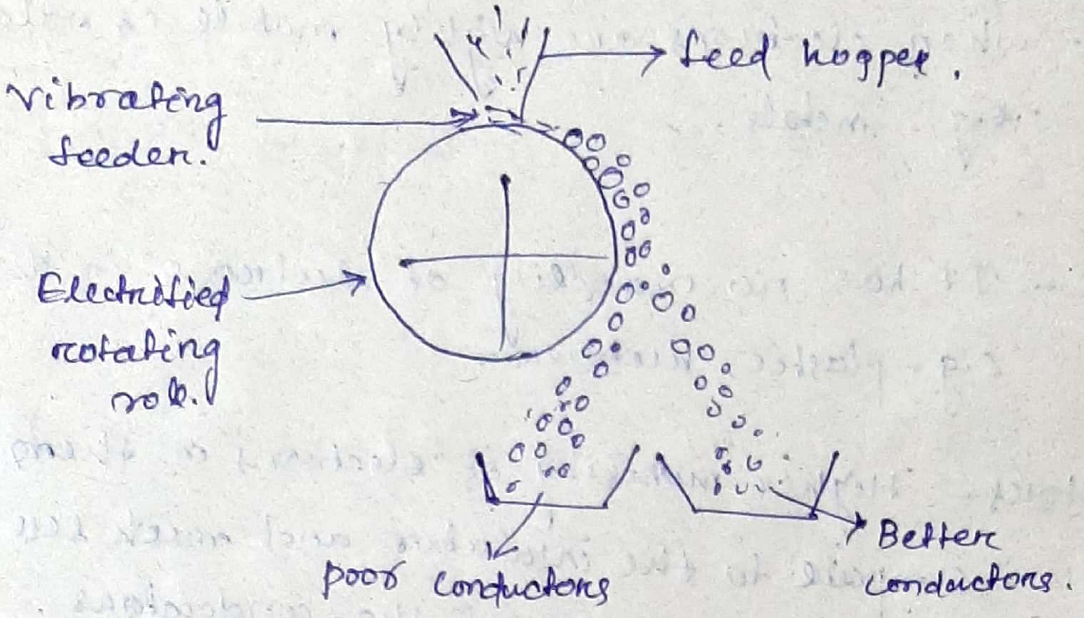
e.g – p-type & n-type semiconductor.

Electron mobility increases in all materials when they are placed inside an electrical field. Due to electrostatic induction. However the extend of ~~electron~~ induction will vary over a large range depending on the material.

Depending on the exchange of Induction the ore particles can be classified into two categories.

conductor →
- Better.
- Poor.

Electrostatic separator :→

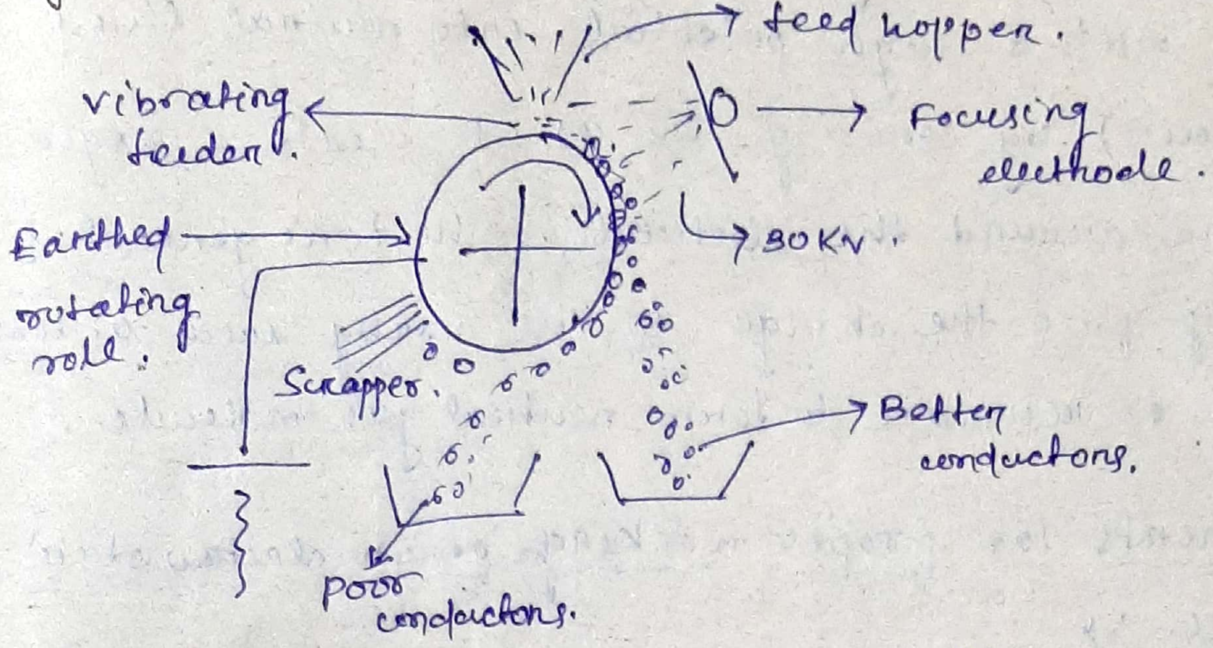


→ In electrostatic separation the feed material is brought near a revolving roll which is either permanently ~~charged~~ electrified or electrified by means of Induction.

→ When the Feed material touches the roll or comes near the electrified roll, it develops a electrostatic charge on its surface by induction, conduction or by friction from charged rotating roll.

→ According to the principle of mutual repulsion of similar electrical charges, better conducting materials are repelled away from the roll surface and form with a trajectory determined by the size and shape of the particle and the speed of the rotating electrified drum. Whereas the poor conducting particles move along the roll surface and have a free fall under the force of gravity.

High tension electrostatic separation :→



High tension roll (HTR) electrostatic separator utilises a grounded roll to transport feed material through the high voltage difference around 30 KV. by ionizing field (corona), where particles are charged by ion bombardments. conducting particles pass their charge to the grounded carrier electrode (rotating roll) and thereby free to be thrown from roll by centrifugal and gravitational force.

Non-conducting particles are pinned to the carrier electrode and are transported further around the roll periferi, where they drag from the electrode surface, either their charge precipitates or by mechanical means (help of scrapper or brush).

Corona Discharge

It is a process by which current flows from an electrode with a high potential into neutral fluid (usually air) by ionizing so as to create a region of plasma around the electrode. The ions generated eventually pass the charge, to the nearby areas of lower potential or recombine to form neutral gas molecule.

Requirements for proper working of an electrostatic separator :-

1. For electrostatic separation, feed material must be dry prior to separation.
2. For effective separations, dry mineral grains are to be fed as a layer of one particle deep at the top of the rotating electrified roll. and this is achieved by using a vibrating feeder.
3. For effective high tension separation feed must be closely sized in the range of (0.1-1) mm with free from fines.

Uses :-

1. It is employed to separate conducting ores and minerals from non-conducting materials in ceramic Industries.

2. this is applied for beneficating rutile beach sand from non-conducting sillica sand in rare earth plants. (rutile: TiO_2)

4th module

Agglomeration

→ The process of converting fines into lumps is known as agglomeration.

→ The basic techniques used in agglomoration process are

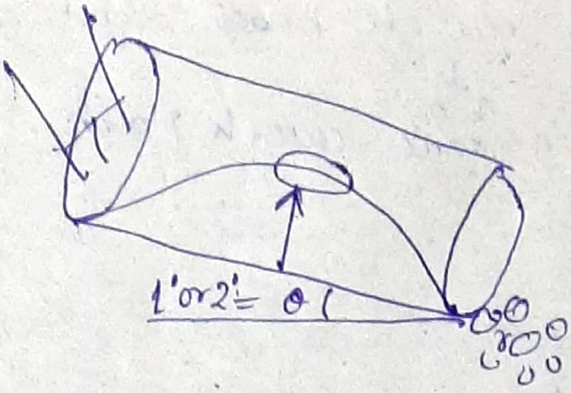
1. Briquetting.
2. Nodulising.
3. vaccum extrusion.
4. Sintering.
5. Pelletization.

Briquetting :→

→ Fine grain iron ores are pressed into pillow shape briquets with the add^n of some water or some binder under high mechanical compressive stress is known as briquetting.

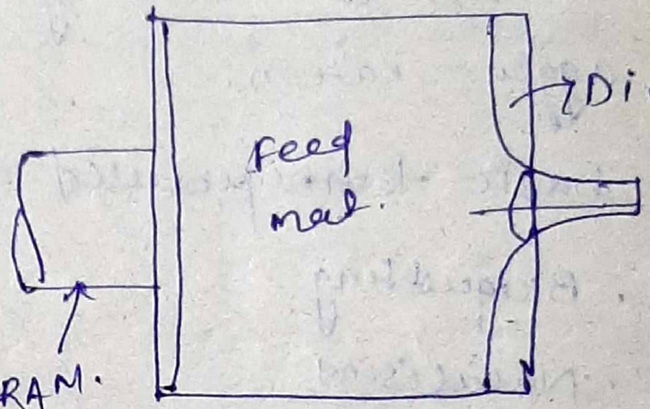
Nodulising :→

→ Fines or concentrates along with carbonesious material are passed through inclined rotary ~~da~~ ~~kiln~~ kiln heated by gas or oil. the temp inside the kiln is sufficient to sinteren but not high enough to fuse the whole.



Vaccum extrusion :→

It the extrusion is done under vaccum condition then it is Known as vaccum extrusion, where extrusion is a process in which a hot or cold semi soft



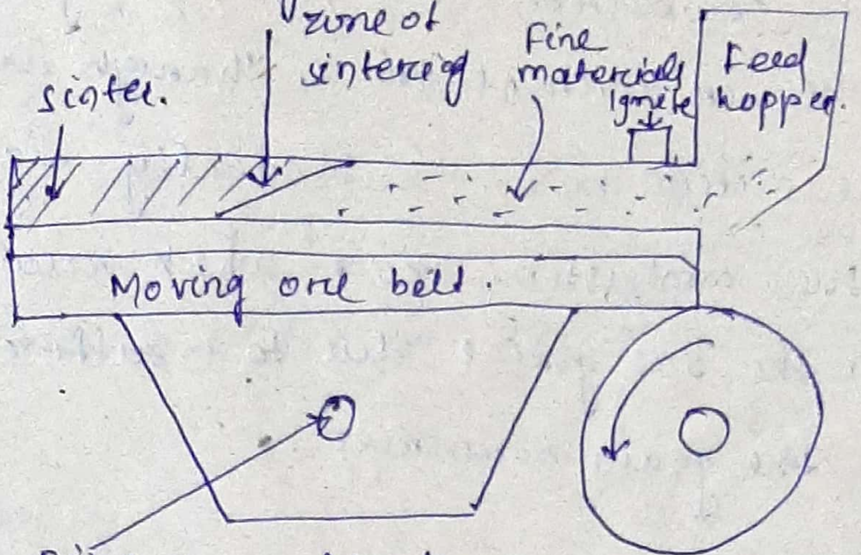
Solid material such as - metal or plastic is forced through the orifice of the die to produce a desire product in a continious manner.

Sintering :→

This process has the largest industrial applications. Sintering may be defined as the agglomoration of fine particles. (above 100 #). to poras lumpy mass by incipient fusion caused by the heat produced by combustion with in the mass it self.

principle of sintering :-

[Dwight-Lloyd sintering Machine]



Iron ores sintering is carried out by putting a mixture of ore fines mixed with solid fuel on a permeable endless moving grate. The top layer of the belt is heated to the sintering temp. around 1200 to 1300 °C by gas or oil fuel burner while air is drawn downwards to the ore bed by the suction box kept below the grate. The narrow combustion zone develop initially at the top layer travels through the ore bed arising the temp. of the bed layer by layer to the sintering temp.

The machine used to carry out the sintering is known as Dwight Lloyd sintering machine.

The fine ore conc^n is charged as a layer of 15-20cm thick on to the endless moving grate at a regulated speed.

The igniter starts the combustion at the surface of the bed. This combustion zone, then propagates through the bed by the air current drawn through the bed of ore by the suction box. Sufficiently high temp. is developed in the combustion zone which results in bonding betn the ore grains due to ~~in suffi~~ incepient fusion along the grain boundries.

Process variables :→

The variables of the sintering process are as follows.

1. The permiability of sintered ~~particle~~ product is decided by the particle size and shape :- Large & irregular shaped particles have high permeability. Sintering is the best suited minerals for +100 # which are unsuitable for palleting.
2. Thickness of the ore bed :. The permeability drop when bed thickness increases.
3. Volume of air blast drawn through the bed :- high is the air-blast, high is the sintering permiabitity.
4. Amount of moisture in the charge: Higher the amount of initial moisture change, Higher is the sinter cake permeability.
5. Amount and quality of the fuel used :- Higher the quantity of the fuel higher will be the permeability.
6. Rate of blast drawn through the bed - Higher the rate of blast drawn higher will be the sinter permiability.

→ The finished sinter is tested for their strength by means of shatter & tumbler index to access its properties and suitablity for the subseiquent use.

→ Apart from Iron ore fines sintering process is also used for sulphide ores of zinc and lead. Sintering of sulphide ores doesn't require any fuel as the combustion is exothermic (in nature.

Pelletization

→ It is best suited for fine grain concentrates all ores which are not suitable for sintering, i.e. generally remains below -100 #.

→ palletization consist of rolling of moist iron ore fines with or without binders to produce the spherical shape particle with a diameter of (10-30) mm

→ Subsequently these moist iron ore are fired at high temp. to eliminate moisture and to develop solid state bonding among the particles.

Parallettrization process consists of following steps.

1. feed preparation.
2. Green ball production and sizing.
3. induration.
4. cooling of hardened pellets.

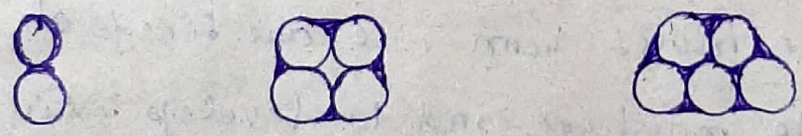
feed preparation

~~Feed~~ preparation of feed mix constitute of Iron ore fines, fuel (coke, coal) → solid fuel, f/c (oil - liq fuel), flux (limestone/Dolomite), water (binder), critical amount of water.

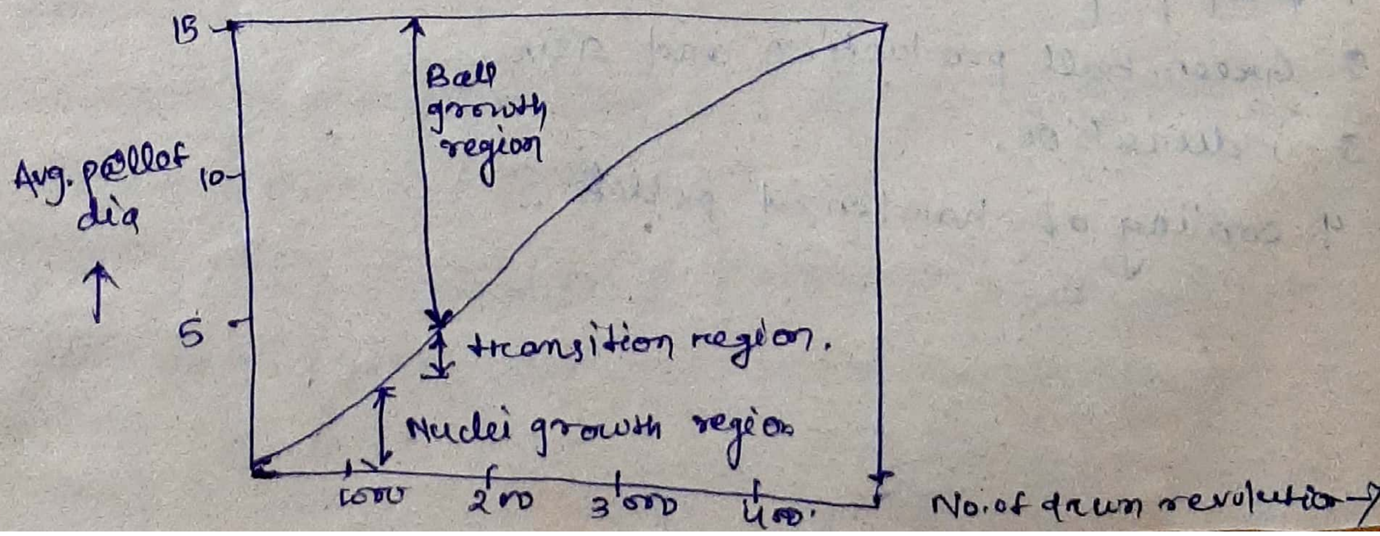
Green ball preparation

→ With the use of critical amt. of water leads to formation of balls with high density, adiquate bonding (due to surface tension, of water on surface of the material/particles) and finally increases the surface area of the bonding. however excess amt. of water is detrimental (causing damage) the green ball making process:

→ Between water & particle the bond develop by 3 stages.



Mechanism of Ball formation :→



The palatte formation is a two stage process.

1. Nucleation or seed formation.

2. Ball growth.

Palate making machine. depends primarily on moisture content and it is around 10% moisture content defined as the critical value to provide the best possible properties to the palate.

The graph bet^n Avg. pellet dia vs. No. of drum revolution from which the final growth of the palate is occure in 3 distinct jones.

1. Nuclei formation - Avg pellet dia increases with a higher rate. respect to drum revolution.
2. Transition zone. - It is somehow remain in bet^n nuclei formation & ball growth zone.
3. Ball growth zone ↳ Avg. pellet dia increases with a lower rate respect to drum revolution.

<u>Machines used for pelletization</u> :→

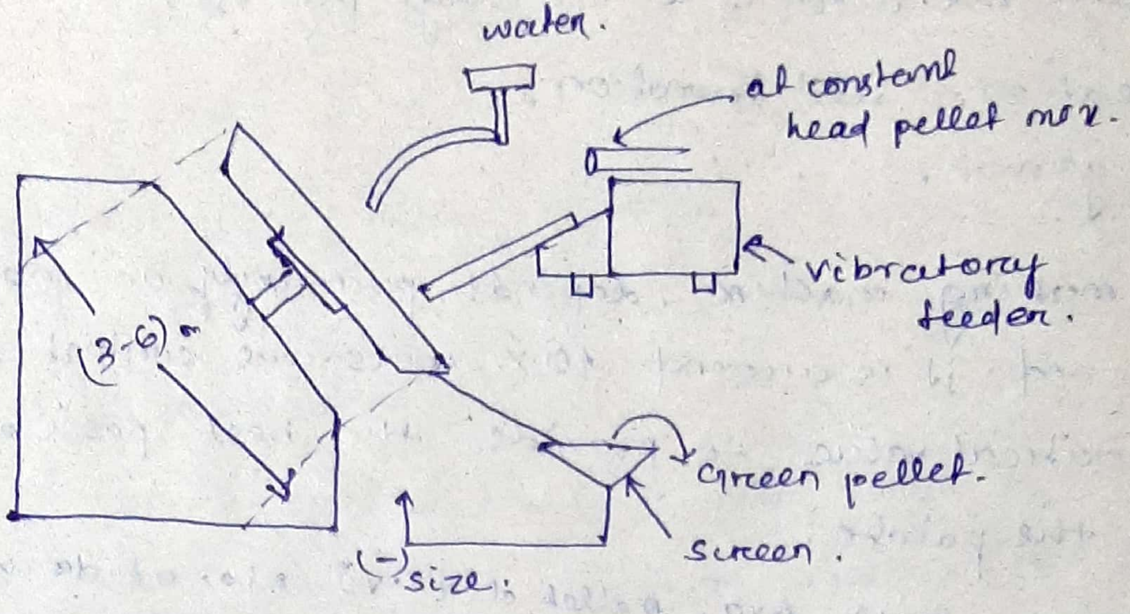
There are two types of pelletisation machines are used.

1. Disc pelletiser.
2. Drum. pelletiser.

<u>Disc pelletiser</u> :→

<u>Important features</u>

→ Diameter of disc is around 3-6 m.
→ inclination angle 45°.
→ speed of rotation (10-15) rpm.
→ Size of the final pellet is (10-30)mm.



Process

→ It is a disc with an outward sloping periferal wall which is rotated around its own center in an inclined position.

→ Materials are fed directly to the disc with water spray arrangement for adding water. Scrapper is used to prevent building of moist materials on the disc surface. So after pelletization process, the pellets are rolled out at 10-30 mm size.

→ The rate of production of green pellets depends on the following factor

1. diameter of the disc.
2. Height of the periferal wall.
3. Angle of inclination of disc w.r.t horizontal.
4. Speed of rotation.
5. Rate of moisture addition.

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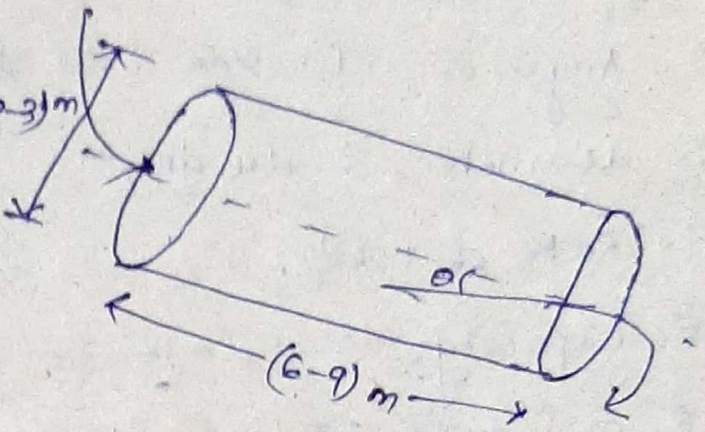
6. Rate of feed mix.
7. Rate of withdrawl of the product.
8. Nature and size of the feed material.
9. Desired size range of the pellets.
10. Use of additives like flux, binder, fuel etc.

Drum Pelletiser ~~Pelletization~~

Features →

→ diameter of the drum (2-3) m. & length of the drum (6-9) m.



~~Ratio drum~~
Diameter to length ration is 1:3.
↓
It can be vary from 2.5 to 3.5.

→ The inclination angle is varies from $(2-10)^\circ$.
→ Speed of rotation 10-15 rpm.
→ Size of the final pellet 10-15 mm.

Process →

→ The feed mix is fed at the upper end of the drum along with water and gets rolled into balls as it moves towards the discharge end.
→ The major difference between drum & disc pelletiser is that the disc pelletiser acts as a classifier unlike the drum pelleting.

→ The size range of the output from the drum pelletiser is larger and it should be operated in closed circuit with a screen.

→ The rate of green ball production depends on the following factor.

1. Speed of rotation.
2. Angle of inclination of the drum w.r.t horizontal.
3. diameter of the drum.
4. Rate of feed.
5. capacity.
6. moisture content.
7. Nature and size of the feed material.

Induration

→ The green balls are send for induration to increase the strength of the green balls. It is carried out in 3 stages.

1. drying
2. preheating
3. firing.

Drying – The green balls are heated upto 120° to remove out the moisture content.

Preheating – further it is preheated to 500–600°c.

Firing – finally the preheated pellets are fire at 1300°–1[illegible] in a shaft kiln / rotary kiln. for developing

solid state bonding
cooling
pelleting :→

It is done in angular cooler to cool down the pellets upto 100°C to transport it for further processing using conveyor belt.

Testing of Agglomerates →

i) Die Pressing Test

It is used to estimate the strength of ceramic agglomorates. The critical nominal pressure (i.e Engg. stress) at which contact areas between the agglomorates start to increase rapidly. The relationship betn change in sample height & applied pressure gives the result of die pressing Test.

ii) shatter test

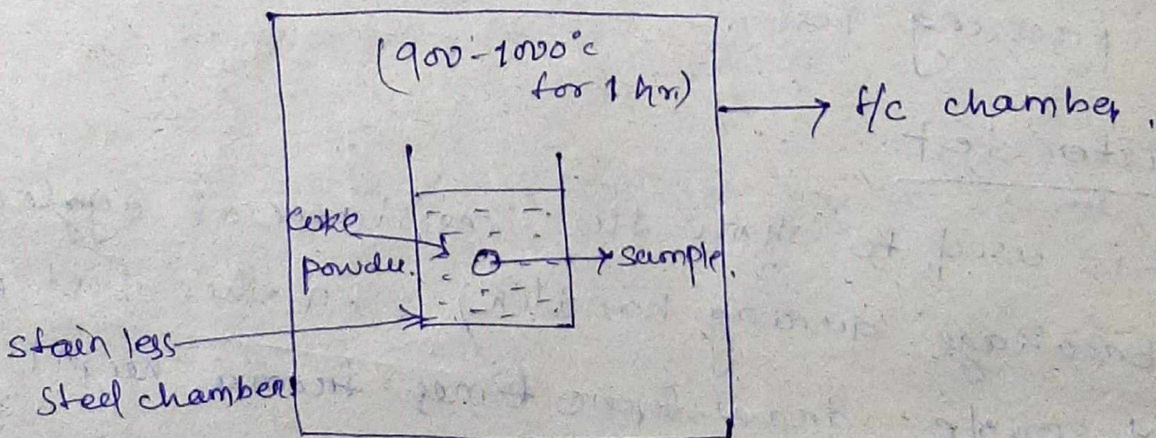
It is used to show the resistance of agglomorates to breakage during handling. In this test 50 pound sample drop four times from a height of 6 feet upon a steel plate. The percentage of original sample which remains on a 2 inch² mesh screen after the treatment is reported as shatter test result or shatter index.

iii) <u>Porosity test</u> :→

It is expressed as the percentage of the volume of the pores. (open & closed pores accessible by fluids) present in the total volume of the material.

iv) <u>Reducibility test</u> :→

It is used to calculate the % of reduction. In this case a known sample weight is taken inside a stainless steel chamber which is surrounded by coke powder, then the whole system is placed inside the furnace at about 900 - 1000 °c. Then the final sample weight is measured after 1 hour. The loss in weight of the sample gives the % in reduction.



v) <u>Tumbler Test</u> ⇒

→ It is use to show the resistance of agglomerates under mechanical impact and attrition (friction).

→ In this test a waited sample of agglomerates is placed in a revolving drum and rotated for a given length of time.

subsequent screen analysis gives a figure said to indicate the friability (describes the ability of solid substance to be reduced to smaller peices with little effort) of the agglomorates.

-> 10 mm sieve was taken.
-> 15 kg material is charged into the drum and rotated around 25 rpm (or 200 revolution).
-> the discharged product from the drum is sieved in a 6 mm screen. Then ~~check~~ by calculating the % of total sample retained on the 6 mm sieve screen is known as tumbler index. further when the same material is sieved on a 500 micrometer sieve screen, then the calculating weight of sample retained on the 500 μm screen is known as abrasion index.

Application of agglomeration Technique in ferrous and non-ferrous industries :->

1) use of Iron ore fines, coke bridge (fine coke), metallurgical waste, lime (CaO), Dolomite ($CaCO_3 \cdot MgCO_3$) for metal production.

2) Recycling of non-ferrous metal ~~from the~~ like Al, Cu, Pb, Zn, precious metal (Au, Ag, Pt) and refractory metal (extraordinary resistance to heat & wear) (Nb, Mo, Ta, W, Re) -> m.p - above 2000°C.

→ Recovered above non-ferrous metals from the process residue and can be ~~xxx~~ returned to the process with out loss of quality in recycling.

Thickening & Dewatering :→

→ Thickening is carried out to obtain the concentrate whereas dewatering is carried out to recover the water from tailing part.