DESIGN OF A CEMENT PLANT PRODUCING 0.5 MMT CLINKER AND EACH OF 0.5 MMT OPC AND PCC CEMENT PER YEAR

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Abstract

This report deals with overall design of a cement plant. The plant is designed based on Dry Process. The plant will produce 0.5 MMT of clinker along with 0.5 MMT of OPC (Ordinary Portland Cement) and PCC (Portland Composite Cement) per year. The necessary mass and energy balance along with detail sizing of four major equipment have been performed. The economic analysis yielded an IRR of 9.7% and ERR of 8.67%, both greater than MARR, considered 8% in the case, proving the feasibility of the designed plant.

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Chapter 1 Introduction

1.1 Background

Cement is an essential component of infrastructure development. It is also the most important input of construction industry, mainly in case of the government's infrastructure and housing programs, which are necessary for the country's socio-economic growth and development. Due to increasing population, various constructional activities are increasing day by day. As a result, the market demand of cement is also increasing continuously but still now most of plants aren't up to the mark technologically. They are very inefficient, not so eco- friendly and have very low production rate.

Cement is a fine powder which sets after a few hours when mixed with water, and then hardens in a few days into a solid, strong material. Cement is mainly used to bind fine sand and aggregates together in concrete. Cement is a hydraulic binder, i.e. it hardens when water is added. There are 27 types of common cement which can be grouped into 5 general categories and 3 strength classes: ordinary, high and very high. In addition, some special type of cements exists like sulphate resisting cement, low heat cement and calcium aluminate cement

The process of manufacture of cement can be divided into three basic steps, preparation of raw materials, pyro processing to produce clinker, and grinding and blending clinker with other products to make cement. The raw materials obtained from the quarry are crushed, ground and mixed as a slurry in the wet process and a powder in the dry process. This mixture is then fed into a calciner and preheater before being fed into the kiln, for pyro processing (clinker formation). The kiln reaches temperatures greater than 1450 °C. The clinker nodules produced, and any additives are then ground to the desired fineness in the cement grinder. Pyro processing consumes 99% of the fuel energy while electricity is mainly used to operate both raw material (33%) and 1 clinker (38%) crushing and grinding equipment.

1.2 Demand and Supply In Bangladesh

The installed production capacity is much higher than the actual production and cement demand of the country, an avenue exists for producers to increase amount of cement export.

Bangladesh has less demand for cement than supply. So we can easily export cement to foreign countries and have a good amount of revenue.



Figure 1.1: Relationship between production and export of cement in Bangladesh



Figure 2: GDP growth rate, cement demand growth rate and population growth rate in Bangladesh.

Some mega projects such as Padma Bridge, Ruppur Nuclear Plant, Rampal Power Project, Metro Rail etc are ongoing. Theses will create an increasing demand for cement.

Cement consumption is increasing rapidly as the economic growth and infrastructure development is showing simultaneous increment. Cement industry is completely dependent on the import for raw materials and the sources of these materials are virgin resources. The supply of these materials is affected by natural stock, geopolitics, inter-governmental relationship and the economic condition of the country. Each and every cement companies in Bangladesh are dependent on different sources for their raw material supply

The export potential will also rise as the production trend of cement in Bangladesh is higher than the demand.

So, we can see, establishing a cement plant is highly profitable.

Chapter 2 Process Definition

2.1 Definition of the Project

2.1.1 Project Definition

A cement plant has to be set up at Chattak, Sunamganj in Bangladesh having a capacity of producing 1515 metric ton of clinker and 1515 metric ton of each of Portland cement (OPC and PCC) per day corresponding to 0.5 MM metric ton of clinker and 0.5 MM metric ton of each of Portland cement(OPC and PCC) per year considering that plant operates 330 days per year, including all offsite, auxiliaries, utilities and supporting facilities using limestone imported from Meghalaya, India , clay and gypsum as raw materials and coal as fuel.

2.1.2 Specification of raw materials and their sources

Limestone (Source: Meghalaya, India) & Clay (Source: local market)

| Component | Weight % | | | |
|--------------------------------|-----------|-------|--|--|
| | Limestone | Clay | | |
| SiO ₂ | 1.88 | 70.69 | | |
| Al ₂ O ₃ | 1.20 | 14.91 | | |
| CaCO ₃ | 94.45 | 4.5 | | |
| Fe ₂ O ₃ | .57 | 6 | | |
| MgCO ₃ | 1.95 | 1.5 | | |
| Inert | 0 | 2.4 | | |

Table 2.1: Raw material composition

Gypsum (Source: local market) is added to Clicker to produce OPC and fly ash, blast slag are added as well to produce PCC.

2.1.3 Specification of Product:

Table 2.2: Clinker composition

| Component | Weight percentage (%) |
|---|-----------------------|
| C2S (2CaO.SiO ₂) | 17.39 |
| C3S (3CaO.SiO ₂) | 50.1 |
| C3A (3CaO.Al ₂ O ₃) | 10 |
| C4AF (4CaO.Al ₂ O ₃ .Fe ₂ O ₃) | 6.6 |
| Inert (unreacted CaO, MgO and other impurities) | 16 |

Table 2.3: OPC and PCC composition

| | OPC (Ordinary | PCC (Portland | | | |
|--------------------------------|----------------|---------------------|-------------------|--|--|
| Component | Cement) | | Composite Cement) | | |
| | (95% Clinker + | (80% Clinker +1 | | | |
| | 5%Gypsum) | 5% Fly Ash+5% Slag) | | | |
| SiO ₂ | 18.25 | | 23.62 | | |
| Al ₂ O ₃ | 4.89 | | 7.27 | | |
| Fe ₂ O ₃ | 2.05 | | 2.53 | | |
| CaO | 67.96 | | 60.13 | | |

| MgO | 1.28 | 0.93 |
|--------|------|------|
| Inerts | 0.55 | 0.56 |
| Gypsum | 5.00 | 5.00 |

2.1.4 Utilities

1. Electricity

2. Air

3. Fuel

4. Water

2.1.5 Plant Location

The plant will be built at Chattak upazilla in the district of Sunamganj.

The plant site is located at the bank of Surma river, so own jetty can be constructed for transporting raw materials and products through marine way. Limestones will be carried by conveyor belts of 17 km long (7 km inside of India).

2.2 Design Basis

Design basis includes:

- 1. Geological data
- 2. Climatic condition
- 3. Utility condition
- 4. Raw materials
- 5. Structural design

2.2.1 Geological condition:

a. Site characteristics

Minimum water level: 8 meters

Maximum water level: 81 meters

During flooding, maximum water level recorded is 17 ft above the plant ground level.

b. Soil type:

Soil of Chattak mainly composed of sand, silt and clay. From base to top, soil can be classified into seven litho-stratigraphic layers beneath which ground water can be found. Soil may also contain coarse sands while ground water consists of clay and silt.

c. Seismic condition:

Situated at earthquake zone I having basic seismic coefficient .08 g.

2.2.2 Climatic condition

a. Design conditions for equipment or facility

Maximum dry bulb Temperature: 36°C (in summer)

Minimum dry bulb temperature: 8°C (in winter)

b. Design conditions for building: HVAC

Summer:

Wet bulb Temperature (max) 30.5°C

Dry bulb Temperature (max) 36°C

Relative humidity 84% Wet bulb temperature (min) 22°C Dry bulb Temperature (min) 25°C Dew point 21°C Relative humidity 80% Winter: Wet bulb Temperature (max) 21°C Dry bulb Temperature (max) 26°C Dew point 20°C Relative humidity 72% Wet bulb temperature (min) 7°C Dry bulb Temperature (min) 8°C Dew point 8°C Relative humidity 67% c. Meteorological conditions i. Ambient Temperature: Absolute max. Temperature: 38°C Absolute min. temperature: 12°C Designed max. temperature: 38°C

Designed min. temperature: 5°C

ii. Atmospheric Pressure:

Annual average atm. Pressure: 0.11 MPa

11Max. Average atm. Pressure: 0.15 MPa

Min. average atm. Pressure: 0.09 MPa

Designed average atm. Pressure: 0.2 MPa

iii. Humidity:

Annual average relative humidity: 80 %

Max. Monthly average relative humidity: 87 %

Min. monthly average relative humidity: 65 %

Designed relative humidity: 90 %

iv. Wind:

Wind direction: Generally, wind flows from the north to the south in the winter season and from the south to north in the summer in our country.

Wind Velocity: 50 knots (50 nautical miles per hour or 58 miles/hr)

v. Rain:

Annual avg. rainfall: 2850mm

Max. Monthly rainfall: 235mm

vi. Earth Bearing Capacity:

Load bearing capacity: The soil has a bearing capacity of 0.5 kg/cm2. Required piling is used.

Corrosive tendency: Non-Corrosive.

2.2.3 Utility conditions

Electric power

440 V - 3 phase, 50 Hz for motors in the range of:

230V-1 phase 50 Hz for motors 0.5 KW and below.

120V-1 phase 50 Hz for instruments and control.

Normal lighting:220 V-1 phase,50 Hz

Air

Instrument air

Pressure: 200 KPa

Temperature: 25°C

Hot air Temperature: 170 °C

Plant effluents and other waste disposal should meet the environmental regulations of the government.

2.3 Selection of process

There are 4 basic types of processes for cement production. These process routes are distinguished by the moisture content of the feed going into the kin and are known as follows:

- Wet process: Raw materials are ground in water to form a slurry which is either fed directly into the kiln or first to a slurry direr.
- Semi wet process: raw slurry is first dewatered in filter process and the filter cake extruded into pellets and feed either to a grate preheater or directly to a filter cake drier for raw mix production.
- Semi –dry process: dry raw mix is pelletised with water and fed into a grate preheater before the kiln or to a long kiln equipped with chains and crosses to facilitate the heat exchange between the kiln feed and the combustion gases.
- Dry process: raw materials are ground and dried into form a dry raw mix in the form of a flowable powder. The dry mix is fed into the preheater or pre calciner kiln or more rarely to a long dry kiln.

Each process requires a slightly modified approach and use of equipment in the first two of the three manufacturing stage, i.e. the raw grinding and pyro processing stages. These differences reviewed in the sections below.

Historically, the clinker manufacturing process has evolved from 'wet' to 'dry' systems with the intermediate steps of the 'semi -wet' and 'semi – dry 'process routes. The first rotary kilns, introduced around 1895, were long wet kilns (CEMBUREAU,1999). 'Wet' kilns have allowed for easier blending and homogenous raw mix using the dry process. The 'dry" production process requires less energy and so has lower fuel costs than the wet process and thus is generally favoured where is application is feasible.

2.3.1 Wet process

In the wet process preparation of the raw mix achieved by grinding the raw materials with water in wash mills, or ball mills with rubber things. The resultant slurry is mixed in slurry basins air is introduced and the slurry is continuously stirred to produce a slurry of consistent composition. In the conventional wet process this thick liquid slurry, which may have water content of 30-40% is fed directly into the kiln where the water is evaporated in the drying zone at the kiln inlet. This drying zone is fitted with curtains of chains which promote heal exchange between the slurry and the exhaust gases from the combustion of fuel in the kiln passing counter current with the slurry feed up the kiln. After passing trough the drying zone the raw material moves down the kiln through the preheating zone to calcined and finally burnt to clinker in the sintering zone. Conventional wet kiln technology has a high heat consumption and produces large volumes of exhaust gases. In more modern wet kiln systems water is driven off the raw slurry to be pumped to the kiln with lower water content. Those more modern wet kiln systems have lower specific heat consumption compared to conventional wet kilns.

In this process the fuel consumption is the highest is the highest (in the region of 5000 to 7000 k/kg of clinker) but the power consumption is typically lower at 110 kWh/ton of cement (Chemical Universe. 2007 \rangle)

2.3.2 Semi Wet process

As in the wet process the raw mix used in the in the semi –wet process is prepared as a slurry. Unlike the wet process is then dried in crusher-dryer before being delivered to a preheater for preheating, calculation and clinker formation in the rotary section of the kiln.

The fuel consumption of the semi-dry and semi-wet process is about 3500-5000 k/kg of clinker. The power consumption is typically in the range 115-120 kWh/ton of cement (Chemical Universe ,2007)

2.3.3 Semi Dry process

In the semi dry process a dry-powder raw mix is made into pellets in a nodulizing pan with the addition of 10-13% water. The pellets are loaded onto a grate preheater which dries ,preheats and partially calcines the raw mix .The pellets then drop into the kiln for completion of the calcination and combination into clinker.

If the semi dry is used the kiln exhaust is used kiln exhaust gaese cannot be utilized in the raw mix drying and grinding due to their lower temperature following their prior use. this combined with the grate preheater maintenance costs that the semi dry process is no longer installed in modern cement plants.

The fuel consumption of the semi dry and semi wet process is about 3500-5000 kj/kg of clinker. The power consumption is typically in the range 115-120 kWh/ton of cement (Chemical Universe ,2007)

2.3.4 Dry Process

In the dry process, the raw materials are dried in a combined drying and grinding plant to reduce the moisture current below 1%. The materials are ground in either a ball or a roller mill with drying of the materials being achieved by sweeping the mill with the exhaust gases from the rotary kiln. The drying capacity of the raw grinding s system maybe supplemented by auxiliary hot air generating furnaces. The dried, finely ground raw mix powder is then blended and homogenized in silos with the mixing being provided by compressed air.

The blended and homogenized raw mix is then usually fed to a gas –suspension preheater which ensures complete calcining of the raw mix before it enters the kiln. Pre-calciner increases kiln capacity and reduce energy consumption.

Fuel consumption is lowest in this process and is in the range of 2750-4000kj/kg of clinker. The power consumption is in the range of 120-125 kWh/ton of cement 9Chemical Universe,2007). However, the most modern, dry process cement plants can have power consumption in the range 80-100 kWh/ton of cement by deployment of consumption of compression grinding equipment for raw and finish milling rather than ball mills. Production of blended cements is also widely used to reduce unit electrical energy consumption.

2.3.5 Reason for Choosing Dry process

Dry process is mostly limited to the use of air suspension preheater. Dry process provides maximum benefits as the heat consumption is an important issue. Development of the dry process, using air suspension preheaters being integrated with pre calcinators. Pre calinators ensure complete calcination of the raw mix before its entry to the kiln. The advantages of this process is that fuel consumption is lowest through existing technologies. The fuel use in this process is in the range of 750-950 kcal/kg of clinker. The chief advantages of the wet process are the low cost of excavating and grinding raw materials, the accurate control of composition and homogeneity of the slurry, and the economical utilization of fuel through the elimination of separated drying operations. On the other hand, the longer kilns, essential in the wet process, cost more and are less responsive to a variable clinker demand than the short kilns which can be used in the dry process. For these advantages we have chosen the dry process for the cement plant.

2.4 Product selection

There are different types of cement. Most used cements are Portland Cements. Others are used for special cases. Portland cements are of various kinds.

| | Classification | Characteristics | Applications |
|--------|--|--|---|
| Type 1 | General purpose | Fairly high C ₃ S content for good early strength development | General construction (most buildings, bridges, pavements ,precast units etc) |
| Type 2 | Moderate sulfate resistance. | Low C ₃ A content (<8%) | Structures exposed to soil or water containing sulfate tons. |
| Type 3 | High early strength | Ground more finely may have slightly more C ₃ S | Rapid construction, cold weather concerning |
| Type 4 | Low heat of hydration (slow reacting) | Low content of C ₃ S (<50%) and C ₃ A | Massive structures such as dams. Now rare. |
| Type 5 | High sulfate resistance | Very low C ₃ A content (<5%) | Structures exposed to high levels of sulfate ions |
| White | White color | No C ₃ Af low mgO | Decorative (otherwise has properties similar to type) |

Table 3.7 General features of the main types of Portland cement

For the case of our country, Type 1 (also known as ordinary Portland cement) is highly preferable [*Cement Industry of Bangladesh vol I*; Department of Research, Emerging Credit Rating Limited]. So we will produce OPC. PCC uses fly ash and blast slag which reduce the cost of production. It has the same strength as OPC [*Durability performance of locally produced OPC & PCC*, Department of Civil, DUET]. It is getting popular to customers day by day. So along with OPC we will produce PCC.

Chapter 3 Process Diagrams

Chapter 4

Process Block Diagram



Process Flow Diagram





Air to

air cooler

Clinker Cooler

Filter

Compressor

Coal stroage

Rotary Kiln

Filter

Clinker

Storage





| J-110 | C-120 | C-121 | F-122 | M-130 | C-210 | H-211 | G-221 | E-220 | C-243 | H-24 |
|------------------|---------|---------|----------------------------|--------------|------------------|-------------------|------------|-----------|--------------|--------|
| Conveyor belt | Crusher | Crusher | Raw material storage | Raw Mixer | Ggrinder Mill | Baghouse Flter | Compressor | Preheater | Coal Mill | Filter |

| G-221 | H-241 |
|------------|-------|
| Compressor | ESP |
| | |

Cement Storage

Cement

Storage

Chapter 4

Material Balance

4.1 Basis:

100 kg/day of raw materials used (Dry basis)

Material out= Material in + Generation – consumption – Accumulation

Until Pyro-processing, no reactions occur. Steady state is assumed in the operation.

So, Accumulation = 0 (Steady state)

Generation & Consumption = 0 (No Chemical reaction)

Raw materials includes limestones and clay. They are mixed in 85:15 weight ratio

| Raw Materials | Weight % | Weight (kg/day) |
|---------------|----------|-----------------|
| Limestone | 85 | 85 |
| Clay | 15 | 15 |

Composition of Raw materials:

| Component | Weight % | |
|--------------------------------|-----------|-------|
| | Limestone | Clay |
| SiO ₂ | 1.88 | 70.69 |
| Al ₂ O ₃ | 1.20 | 14.91 |
| CaCO ₃ | 94.45 | 4.5 |
| Fe ₂ O ₃ | .57 | 6 |
| MgCO ₃ | 1.95 | 1.5 |
| Inert | 0 | 2.4 |

4.2 Crusher



Limestone from quarry is Crushed and 5% loss is assumed in the crusher

M1 = 85 kg/day of limestones

Lost in Crushing operation = $.05 \times 85 = 4.25$ kg/day

M2 = Amount of limestones crushed / day = 85 - 4.25 = 80.25 kg/day

4.3 Dust Purifier



Before Mixing with limestones additives are purified and 5% loss is assumed in the filter

M3 = Amount fed = 15 kg/day

 $Loss = .05 \times 15 = .75 \text{ kg/day}$

M4 = Additives Recovered = 15 - 0.75 = 14.25 kg/day

4.4 Mixer



Crushed limestone and recovered additives are mixed in a mixer to produce the feed for Grinding.

M5 = Amount fed in the Raw Mill = M2 + M4 = 80.75 + 14.25 = 95 kg/day

(a)

Amount of each component in the feed for raw mill (M5) is calculated in kg/day

Component amount (x1) = (weight % in limestones $\times \frac{M2}{100}$) + (weight % in clay $\times \frac{M4}{100}$)

$$SiO_{2} = (1.88 \times .8075) + (70.69 \times .1425) = 11.59 \text{ kg/day}$$
$$Al_{2}O_{3} = (1.2 \times .8075) + (14.91 \times .1425) = 3.09 \text{ kg/day}$$
$$CaCO_{3} = (94.45 \times .8075) + (4.5 \times .1425) = 76.91 \text{ kg/day}$$
$$Fe_{2}O_{3} = (.57 \times .8075) + (6 \times .1425) = 1.32 \text{ kg/day}$$
$$MgCO_{3} = (1.95 \times .8075) + (1.5 \times .1425) = 1.79 \text{ kg/day}$$
$$Inert = (0 \times .8075) + (2.4 \times .1425) = .342 \text{ kg/day}$$

(b)

Composition of components in M5 is calculated as

Composition in M5 = $(\frac{Component Amount(x1)}{M5}) \times 100 \%$

$$SiO_{2} = (\frac{11.59}{95}) \times 100 \% = 12.20 \%$$
$$Al_{2}O_{3} = (\frac{3.09}{95}) \times 100 \% = 3.25 \%$$
$$CaCO_{3} = (\frac{76.91}{95}) \times 100 \% = 80.96 \%$$
$$Fe_{2}O_{3} = (\frac{1.32}{95}) \times 100 \% = 1.38 \%$$
$$MgCO_{3} = (\frac{1.79}{95}) \times 100 \% = 1.88 \%$$
$$Inerts = (\frac{.342}{95}) \times 100 \% = .36 \%$$

Components analaysis for the output from mixer or the feed for raw mill

| Component | Feed for Mill (M5) = 95 kg/day | |
|--------------------------------|--------------------------------|----------|
| | Amount (kg/day) | Weight % |
| SiO ₂ | 11.59 | 12.20 |
| Al ₂ O ₃ | 3.09 | 3.25 |
| CaCO ₃ | 76.91 | 80.96 |
| Fe ₂ O ₃ | 1.32 | 1.38 |
| MgCO ₃ | 1.79 | 1.88 |
| Inert | .342 | .36 |



M5 = 95 kg/day of mixture of raw materials are fed in the Raw Mill .

A 2% loss is assumed from Bag House Filter.

M6 = Amount lost due grinding = $.02 \times 95 = 1.9 \text{ kg/day}$

M7 = Preheater feed = 95 - .475 = 94.525 kg/ day

(a)

Amount of each component in the preheater feed (M7) is calculated in kg/day

Component amount (x2) = $\left(\frac{Component amount (x1)}{M5}\right) \times M7$

$$SiO_{2} = \frac{11.59}{95} \times 94.525 = 11.50 \text{ kg/day}$$
$$Al_{2}O_{3} = \frac{3.04}{95} \times 94.525 = 3.08 \text{ kg/day}$$
$$CaCO_{3} = \frac{76.91}{95} \times 94.525 = 76.53 \text{ kg/day}$$
$$Fe_{2}O_{3} = \frac{1.32}{95} \times 94.525 = 1.31 \text{ kg/day}$$
$$MgCO_{3} = \frac{1.79}{95} \times 94.525 = 1.78 \text{ kg/day}$$
$$Inerts = \frac{.342}{95} \times 94.525 = .34 \text{ kg/day}$$
Molecular weight of the component in the mixtures are

| Component | Molecular weight (kg/kmol) |
|--------------------------------|-------------------------------|
| SiO ₂ | 60 |
| Al ₂ O ₃ | 102 |
| CaCO ₃ | 100 |
| Fe ₂ O ₃ | 160 |
| MgCO ₃ | 84 |
| Inert | |

Molar amounts of each component in the preheater feed (M7) is calculated by

Molar amounts in kmol/day $(y1) = (\frac{Component \ amount \ (x2)}{Molecular \ weight})$

 $SiO_{2} = \frac{11.50}{60} = .19 \text{ kmol/day}$ $Al_{2}O_{3} = \frac{3.08}{102} = .03 \text{ kmol/day}$ $CaCO_{3} = \frac{76.53}{100} = .76 \text{ kmol/day}$ $Fe_{2}O_{3} = \frac{1.31}{160} = .008 \text{ kmol/day}$ $MgCO_{3} = \frac{1.78}{84} = .02 \text{ kmol/day}$ Inerts = .34 kg/day

| Component | Preheater Feed (M7) = 94.525 kg/day | | | |
|--------------------------------|---|----------|----------------------------|--|
| | Amount (kg/day) | Weight % | Molar amount (kmol/day) | |
| SiO ₂ | 11.50 | 12.20 | .19 | |
| Al ₂ O ₃ | 3.08 | 3.25 | .03 | |
| CaCO ₃ | 76.53 | 80.96 | .76 | |
| Fe ₂ O ₃ | 1.31 | 1.38 | .008 | |
| MgCO ₃ | 1.78 | 1.88 | .02 | |
| Inert | .34 | .36 | | |

Component analysis for the raw mix from grinder or Preheater feed (M7)

4.6 Preheater



Preheater is the first stage of pyro-processing of the raw mix . At about 600-700°C raw mix from the grinder is fed in the preheater . This heat is generated by burning fuel and also from directly feeding the hot exhaust gas from kiln .

At this temperature almost no or very little reaction occurs . The main function of preheater is to remove the free moisture from the feed and also reduce the work load of the kiln .

From energy balance around the Preheater,

the amount of coal required = 240.63 mol/day

Kiln Feed = 95.037 kg/day and composition as follows

Kiln Feed Composition :

| Components | Mol/day | Kg/day | Weight % |
|--------------------------------|---------|----------------|----------|
| SiO ₂ | 196.24 | 11.77 | 12.4 |
| Al ₂ O ₃ | 31.94 | 3.26 | 3.43 |
| CaCO ₃ | 765.30 | 76.53 | 80.5 |
| Fe ₂ O ₃ | 10.36 | 1.66 | 1.8 |
| MgCO ₃ | 21.19 | 1.78 | 1.9 |
| CaO | .48 | .027 | .28 |
| MgO | .24 | .01 | .11 |
| | | Total = 95.037 | |

So, Kiln feed (M8) = 95.037 kg/day

4.7 Rotary Kiln & Clinker Cooler



Rotary Kiln is the primary reactor in cement production Industry. The components of the raw mix react in high temperature produced by the combustion of fuel in different regions of rotary kiln and produce complex compounds.

100 % conversion is assumed for the components present in smaller amount. However, reactions occur in such a way so that a desired proportion of complexes is maintained in the clinker

| Complexes | Full form | Compound name | Molecular weight | Desired weight% |
|-----------|---|---------------------------------|---------------------|--------------------|
| C2S | 2CaO.SiO ₂ | Dicalcium Sillicate | 172 | 20 |
| C3S | 3CaO.SiO ₂ | Tricalcium Sillicate | 228 | 60 |
| C3A | 3CaO.Al ₂ O ₃ | Tricalcium Alluminate | 270 | 12 |
| C4AF | 4CaO.Al ₂ O ₃ .Fe ₂ O ₃ | Tetracalcium Alluminoferrite | 486 | 08 |

Desired composition in clinker

In the major 3 regions of a Rotary kiln various transformation of the kiln feed occurs .

In the **Dehydration** or **Drying** zone , the free moisture is removed by evaporation at about $600^{\circ}C$

In the **Calcination** zone $CaCO_3$ and $MgCO_3$ decomposes and produced CO_2 gas is removed from the system at about **900°C**. Also the bound moisture is removed .

In the **Sintering** or **Clinker Formation** zone most of the reactions occur at a very high temperature of more than $1300^{\circ}C$.

| No. | Reactions | Zone | Temperature |
|-------|---|-------------|-------------|
| | | | |
| i. | $H_2O(l) = H_2O(v)$ | Drying | 600°C |
| | | | |
| 11. | $CaCO_3 = CaO + CO_2$ | Calcination | 900°C |
| iii. | $MgCO_3 = MgO + CO_2$ | | |
| iv | $2C_{2}O + SiO_{2} - 2C_{2}O SiO_{2}$ | | |
| 1. | 2000 + 5102 - 2000.5102 | | |
| V. | $4CaO + Al_2O_3 + Fe_2O_3 = 4CaO.Al_2O_3.Fe_2O_3$ | Sintering | >1400°C |
| vi. | $CaO + 2CaO.SiO_2 = 3CaO.SiO_2$ | | |
| vii. | $5CaO + 3Al_2O_3 = 5CaO.3Al_2O_3$ | | |
| viii. | $4CaO + 5CaO.3Al_2O_3 = 3(3CaO.Al_2O_3)$ | | |

Reactions in Rotary Kiln

 $CaCO_3(s) = CaO(s) + CO_2(g)$ (ii)

Assuming that all the CaCO₃ is decomposed

- X1 = Amount of CaO produced = .765.30 mol/day
- $Y1 = Amount of CO_2 produced = 765.30 kmol/day$

 $MgCO_3(s) = MgO(s) + CO_2(g)$(iii)

Assuming that all the MgCO₃ is decomposed

X2 = Amount of MgO produced = 21.19 mol/day

Y2 = Amount of CO₂ produced = 21.19 mol/day

Total CO₂ removed due to decomposition reaction

= Y1 + Y2 = 765.30 + 21.19 = 786.49 mol/day

Amount of CO₂ removed as a result of decomposition = $.78 \times 44 = 34.584$ kg/day

 $2CaO + SiO_2 = 2CaO.SiO_2$ (iv)

In this reaction conversion of all the SiO₂ is considered

 SiO_2 reacted = 196.24 mol/day

 $Y3 = Amount of CaO reacted = 196.24 mol/day SiO_2 reacted \times \frac{2 mol CaO reacted}{1 mol SiO_2 reacted}$

= 392.48 mol/day

X3 = Amount of C2S produced = 196.24 mol/day

$4CaO + Al_2O_3 + Fe_2O_3 = 4CaO.Al_2O_3.Fe_2.O_3....(v)$

Complete conversion of Fe₂O₃ is considered

 Fe_2O_3 reacted = 10.36 mol/day

Z4 = Amount of CaO reacted = 10.36 mol/day Fe₂O₃ reacted $\times \frac{4 \text{ mol CaO reacted}}{1 \text{ mol Fe}_{2O3} \text{ reacted}}$

$$= 41.44 \text{ mol/day}$$

 $Y4 = Amount of Al_2O_3 reacted = 10.36 mol/day$

X4 = Amount of C4AF produced = 10.36 kmol/day

$CaO + 2CaO.SiO_2 = 3CaO.SiO_2 \dots (vi)$

Produced C2S reacts with CaO I such proportion so that it maintains a 60:20 Or 3:1 weight ratio of C3S : C2S in the clinker content .

Weight ratio of C3s : C2S in the clinker = 3:1

Mole ratio of C3S:C2S in the clinker
$$=\left(\frac{3}{(MW)C3S}\right): \left(\frac{1}{(MW)C2S}\right)$$
$$= \frac{3}{228}: \frac{1}{172}$$
$$= .0131: .0058$$
$$= 2.26: 1$$

So for 1 mole of unreacted C2S in the clinker 2.26 mole reacts to form C3S

Y5 = C2S reacted to produce C3S = $\frac{2.26}{2.26+1} \times 196.24 = 136.04 \text{ mol/day}$

X5 = C3S produced = 136.04 mol/ day

Z5 = Amount of CaO reacted = 136.04 mol/day

Unreacted C2S (U5) = X3 - Y5 = 196.24 - 136.04 = 60.19 mol/day

$5CaO + 3Al_2O_3 = 5CaO.3Al_2O_3$(vii)

Excess Al₂O₃ which has not reacted in rection no (v) reacts to produce pentacalcium Trialluminate

Y6 = Amount of Al₂O₃ reacted in (vii) rreaction

- = Total Al_2O_3 in the kiln feed Y4
- = 31.94 10.36 = 21.58 mol/day

 $Z6 = Amount of CaO reacted = 21.58 mol/day Al_2O_3 reacted \times \frac{5 mol CaO reacted}{3 mol Al_2O_3 reacted}$

= 36 mol/day

 $X6 = Amount 5CaO.3Al_2O_3 produced$

 $= 21.58 \ mol/day \ Al_2O_3 \ reacted \times \frac{1 \ mol \ 5CaO.3 \ Al 2O3 \ produced}{3 \ mol \ Al 2O3 \ reacted}$

=7.19 mol/day

4CaO + 5CaO.3Al₂O₃ = 3(3CaO.Al₂O₃).....(viii)

Y7 = Amount of CaO reacted = 7.19 × 4 = 28.77 mol/day

X7 = Amount of C3A produced = $7.19 \times 3 = 21.58 \text{ mol/ day}$

Complex product produced which is remained in the clinker

C2S = U5 = 60.19 mol/ dayC3S = X5 = 136.04 mol/dayC3A = X7 = 21.58 mol/ day

C4AF = X4 = 10.36 mol/day

Inert Analysis :

Inerts contain unreacted CaO and MgO.

MgO is considered to not react over the whole region of kiln

So, unreacted MgO = Produced MgO in (ii) + MgO in coal = 21.19 +.24 mol/day

$$= 21.43 \text{ mol/day} = .857 \text{ kg/day}$$

Unreacted CaO = Total CaO - Reacted CaO

Total inert = Inert in feed + Unreacted MgO + Unreacted CaO

$$= .34 + .857 + 7.3 = 8.497$$
 kg/ day

Weight of each component in clinker :

Individual weight is calculated by

Weight in kg/day = molar amount in kmol/day × Molecular weight

- $C2S = 60.19 \times 172 = 10.35 \text{ kg/ day}$
- $C3S = 136.04 \times 228 = 31.07 \text{ kg/ day}$
- $C3A = 21.58 \times 270 = 5.86$ kg/ day
- $C4AF = 10.36 \times 486 = 5.03 \text{ kg/ day}$
- Inerts = 8.497 kg/ day

Total weight of clinker = C2S + C3S + C3A + C4AF + inerts

$$= 10.35 + 31.07 + 5.86 + 5.03 + 8.497$$

Weight % in Clinker : Calculated by

Weight % = $\frac{Individual \ amount}{Total \ Amount} \times 100 \ \%$

$$C2S = \frac{10.35}{60.81} \times 100\% = 17.02\%$$
$$C3S = \frac{31.07}{60.81} \times 100\% = 51.09\%$$
$$C3A = \frac{5.86}{60.81} \times 100\% = 9.6\%$$
$$C4AF = \frac{5.04}{60.81} \times 100\% = 8.2\%$$
$$Inerts = \frac{8.497}{60.81} \times 100\% = 13.97\%$$

| Component | Molar Amount (mol/day) | MW | Weight (kg/day) | Weight % |
|-----------|---------------------------|-----|-----------------|----------|
| C2S | 60.19 | 172 | 10.35 | 17.02 |
| C3S | 136.04 | 228 | 31.07 | 51.09 |
| C3A | 21.58 | 270 | 5.86 | 9.6 |
| C4AF | 10.36 | 486 | 5.04 | 8.2 |
| Inerts | | | 8.497 | 13.97 |

Clinker Analysis



After cooling clinker is seperated in three portion . One of them for sale which will contain only clinker with no other additives . For OPC 5% Gypsum will be mixed with the Clinker . And for PCC some slag and fly ash are added to produce an adulterated version .

No clinker loss in the cooling process is assumed

So , M9 = Cooled clinker = Total clinker produced = 60.81 kg/day

Clinker for sale contains 100% Clinker, Clinker for OPC contains 95% clinker and clinker for PCC contains 72% Clinker. The final production amount of clinker, OPC and PCC is same.

SO, a clinker balance around the seperator gives

$$M9 = M11 + M12 + M13 = M11 + .95 \times M11 + .72 \times M11$$

M11 = $\frac{60.81}{1+.95+.72}$ = 22.77 kg/day of clinker produced for sale

 $M12 = .95 \times M11 = 21.63$ kg/day of clinker produced for OPC

 $M13 = .72 \times M11 = 16.40$ kg/day of clinker produced for PCC



Now, M14 = Amount of OPC = Clinker produced for sale (M11) = 22.77 kg/day

OPC is assumed to be 95% clinker and 5% Gypsum .

Gypsum balance around mixer (OPC)

$\mathbf{M14} = .05 \times \mathbf{M15}$

 $= .05 \times 22.77 = 1.14$ kg Gypsum / day



PCC is taken to be 72 % Clinker , 18% Fly Ash and 10% Slag .

- M18 = PCC produced = 22.77 kg/ day
- M16 = Fly Ash required = $.18 \times 22.77 = 4.1$ kg/day
- M17 = Slag required = $.1 \times 22.77 = 2.28$ kg/day

4.11 Scaling up

The actual production = 0.5 MMT of each clinker , OPC and PCC per year with 330 working days in a year

Per day production for sale = $\frac{0.5 \times 10^6 \text{ ton clinker}}{\text{Year}} \times \frac{1 \text{ Year}}{330 \text{ days}}$

= 1515 ton clinker / day

Multiplication factor (MF) = $\frac{1515 \text{ ton clinker}}{22.77 \text{ kg clinker}} = 65.53 \frac{\text{ton}}{\text{kg}}$

Raw material required in dry basis = $100 \times 65.53 = 6553$ ton/ day

Scaled up values :

Around crusher:

Limestone feed = $85 \times 65.53 = 5655$ ton / day

Crushed limestones = $80.75 \times 65.53 = 5372$ ton/day

Loss $(5\%) = 4.25 \times 65.53 = 282.67$ ton/day

Around Deduster :

Clay feed = $15 \times 65.53 = 998$ ton/day

Loss(5%) = $.75 \times 65.53 = 49.9$ ton/day

Clay outlet = $14.25 \times 65.53 = 948.1$ ton/day

Around Mixer:

Crushed limestones = 5372 ton/day

Clay Outlet = 948.1 ton/day

Feed for Raw Mill = 5372 + 948.1 = 6320 ton/day

Around Raw Mill :

Raw Mixture = 6320 ton/day

Preheater feed = $.98 \times 6320 = 6194$ ton/day

Loss from ESP $(.5\%) = .02 \times 6320 = 126 \text{ ton/day}$

Around Rotary Kiln :

Kiln Feed = 6288 ton/dayClinker produced = $60.81 \times 65.53 = 4045 \text{ ton/day}$ CO₂ removed = 6288 - 4045 = 2243 ton/day

Around Seperator :

Clinker produced = 4045 ton/day

Clinker for sale = $22.77 \times 65.53 = 1515$ ton/day

Clinker for producing OPC = $.95 \times 1515 = 1439$ ton/day

Clinker for producing PCC = $.72 \times 1515 = 1090$ ton/ day

Around Mixer (OPC) :

Clinker for OPC = 1439 ton/day

Gypsum added = $.5 \times 1515 = 75.67$ ton/day

OPC produced = 1515 ton/day

Around Mixer (PCC) :

Clinker for PCC = 1090 ton/day

Fly Ash added = $.18 \times 1515 = 272.67$ ton/day

Slag added = $.1 \times 1515 = 151.67$ ton/day

PCC produced = 1515 ton/day

Chapter 5

Energy Balance



| Component | Limestone (85 kg/day) | | | |
|--------------------------------|---------------------------|--------|---------|--|
| | | Ame | ount | |
| | Composition (weight %) | Kg/day | Mol/day | |
| SiO ₂ | 1.88 | 1.598 | 26.633 | |
| Al ₂ O ₃ | 1.20 | 1.020 | 10.000 | |
| CaCO ₃ | 94.45 | 80.283 | 802.83 | |
| Fe ₂ O ₃ | .57 | .485 | 3.031 | |
| MgCO ₃ | 1.95 | 1.658 | 19.738 | |

Assuming limestones of 1000 mm was fed to the crusher and crushed up to 100 mm in the primary crusher and up to 25 mm in the secondary crusher

Bond's Law is applied to find energy required for crushing raw materials to 100 mm .

According to Bond's Law, required energy

$$\mathbf{E}_{\mathbf{r}} = \mathbf{E}_{\mathbf{i}} \sqrt{\frac{100}{L2}} \left(\mathbf{1} - \frac{1}{\sqrt{q}} \right);$$

Here, $E_i = Work$ index = 12.74 KJ/kg for dry crushing limestone

$$L_{2} = \text{Final size} = 100 \text{ mm}$$

$$q = \frac{Initial \ Size}{Final \ Size} = \frac{L1}{L2} = \frac{1000}{100} = 10$$
So,
$$E_{r} = 12.74 \times \sqrt{\frac{100}{100}} \left(1 - \frac{1}{\sqrt{10}}\right)$$

$$= 8.711 \text{ KJ/kg}$$

Kick's Law is applied to find energy required for crushing raw materials to 25 mm According to Kick's Law, required energy

$$\mathbf{E}_{\mathbf{k}} = \mathbf{K}_{\mathbf{k}} \times \mathbf{f} \mathbf{c} \times \ln \frac{L1}{L2};$$

Here, $f_c = crushing strength$

 $L_1 =$ Initial Size

 $L_2 = Final Size$

For reducing size from 1000 mm to 100 mm, Kick's Law

$$E_{k1} = K_k \times fc \times \ln \frac{1000}{100} = 8.711 \dots (i)$$

For reducing size from 100 mm to 25 mm, Kick's Law

 $(ii) \div (i)$

$$\frac{\text{Ek2}}{8.711} = \frac{\ln(4)}{\ln(10)}$$
$$E_{k2} = 5.244 \text{ KJ/kg}$$

For, m = 85 kg/day feed, total energy required for crushing

$$\mathbf{E}_{t} = \mathbf{m} (\mathbf{E}_{r} + \mathbf{E}_{k2})$$

= 85 × (8.711 + 5.244)
= 1186. 260 KJ/day

Assuming, 30% of the applied energy (E_a) is used in crushing.

$$\mathbf{E_a} = \frac{Et}{0.3} = \frac{1186.260}{0.3} = 3954.20 \text{ KJ/day}$$

Assuming all the lost energy (E₁) is convert to heat energy.

$$E_{l} = E_{a} - E_{t}$$

$$= 3954.2 - 1186.26$$

$$= 2767.94 \frac{kJ}{day} \times \frac{1000J}{1kJ} \times \frac{1cal}{4.18J}$$

$$= 662186.285 \text{ cal/day}$$

Now,
$$\mathbf{E}_{\mathbf{l}} = \sum H = \int_{T_i}^{T_f} \mathbf{n}^i \times Cp(T) \times dT$$
(iii)

Here, Cp_i = specific heats of components, n = molar flow rate (mol/day), T = temperature (K)

| Components | $Cp's \left(\frac{Cal}{molK}\right)$ |
|--------------------------------|--------------------------------------|
| SiO ₂ | 10.87 + 0.008712T |
| Al ₂ O ₃ | 22.08 + 0.008971T |
| CaCO ₃ | 19.68 + 0.01189T |
| Fe ₂ O ₃ | 24.72 + 0.01604T |
| MgCO ₃ | 16.9 |

$$\begin{split} H_{SiO2} &= \mathring{n}_{SiO2} \times \int_{303}^{T} (10.87 + 0.008712T) \, dT \\ &= 289.5 \times (T - 303) + 0.1145 \times (T^2 - 303^2) \\ H_{AI2O3} &= \mathring{n}_{AI2O3} \times \int_{303}^{T} (22.08 + 0.008971T) \, dT \\ &= 220.8 \times (T - 303) + 0.045 \times (T^2 - 303^2) \\ H_{CaCO3} &= \mathring{n}_{CaCO3} \times \int_{303}^{T} (19.68 + 0.01189T) \, dT \\ &= 15799 \times (T - 303) + 4.82 \times (T^2 - 303^2) \\ H_{Fe2O3} &= \mathring{n}_{Fe2O3} \times \int_{303}^{T} (24.72 + 0.01604T) \, dT \\ &= 74.89 \times (T - 303) + 0.024 \times (T^2 - 303^2) \\ H_{MgCO3} &= \mathring{n}_{MgCO3} \times \int_{303}^{T} 16.9 \, dT \\ &= 333.57 \times (T - 303) \end{split}$$

Now from (iii)

 $E_l \quad = \ H_{SiO2} \, + \, H_{Al2O3} + \, H_{CaCO3} + \, H_{Fe2O3} + \, H_{MgCO3}$

 $662186 = \ 16717.76 \times (\ T - 303\) + 5.004 \times (\ T^2 \text{-} 303^2\)$

T = 336.25 K = 63.15 °C

So, temperature of crushed limestone, $T_{output} = 63.15$ °C



| | Composition (weight %) | | Amount (95 kg/day) | | | |
|--------------------------------|------------------------|-------|--------------------|--------|-----------|---------|
| Component | | | Kg/day | | Mol/day | |
| | Limestone | Clay | Limestone | Clay | Limestone | Clay |
| SiO ₂ | 1.88 | 70.69 | 1.509 | 10.073 | 25.15 | 167.883 |
| Al ₂ O ₃ | 1.20 | 14.91 | 0.963 | 2.125 | 9.441 | 20.833 |
| CaCO ₃ | 94.45 | 4.50 | 75.796 | 0.641 | 757.96 | 6.41 |
| Fe ₂ O ₃ | 0.57 | 6.00 | 0.457 | 0.855 | 2.856 | 5.343 |
| MgCO ₃ | 1.95 | 1.50 | 1.565 | 0.214 | 18.631 | 2.548 |
| Inert | 0 | 2.40 | 0 | 0.342 | 0 | |

In Mixer, assuming no heat loss

$Q_1 = Q_c$

- Here , Q_1 = Heat given by limestone (63.15°C)
 - Q_c = Heat taken by Clay (30°C)

Heat given by limestone :

$$\begin{split} \mathrm{H}_{\mathrm{SiO2}} &= \mathring{\mathrm{n}}_{\mathrm{SiO2}} \times \int_{T_m}^{336.25} (10.87 + 0.008712\mathrm{T}) \, dT \\ &= 273.38 \times (336.25 - \mathrm{T_m}) + .108 \times (336.25^2 - \mathrm{T_m}^2) \\ \mathrm{H}_{\mathrm{A12O3}} &= \mathring{\mathrm{n}}_{\mathrm{A12O3}} \times \int_{T_m}^{336.25} (22.08 + 0.008971\mathrm{T}) \, dT \\ &= 208.46 \times (336.25 - \mathrm{T_m}) + 0.0425 \times (336.25^2 - \mathrm{T_m}^2) \\ \mathrm{H}_{\mathrm{CaCO3}} &= \mathring{\mathrm{n}}_{\mathrm{CaCO3}} \times \int_{T_m}^{336.25} (19.68 + 0.01189\mathrm{T}) \, dT \\ &= 14916 \times (336.25 - \mathrm{T_m}) + 4.55 \times (336.25^2 - \mathrm{T_m}^2) \\ \mathrm{H}_{\mathrm{Fe2O3}} &= \mathring{\mathrm{n}}_{\mathrm{Fe2O3}} \times \int_{T_m}^{336.25} (24.72 + 0.01604\mathrm{T}) \, dT \\ &= 70.6 \times (336.25 - \mathrm{T_m}) + 0.023 \times (336.25^2 - \mathrm{T_m}^2) \\ \mathrm{H}_{\mathrm{MgCO3}} &= \mathring{\mathrm{n}}_{\mathrm{MgCO3}} \times \int_{T_m}^{336.25} 16.9 \, dT \\ &= 314.863 \times (336.25 - \mathrm{T_m}) \end{split}$$

Now, $Q_1 = H_{SiO2} + H_{Al2O3} + H_{CaCO3} + H_{Fe2O3} + H_{MgCO3}$ = 15783.3 × (336.25 - T_m) + 4.723 × (336.25² - T_m²)

Heat taken by clay :

$$\begin{split} H_{SiO2} &= \mathring{n}_{SiO2} \times \int_{303}^{Tm} (10.87 + 0.008712T) \, dT \\ &= 1824.88 \times (T_m - 303) + 0.722 \times (T_m^2 - 303^2) \\ H_{A12O3} &= \mathring{n}_{A12O3} \times \int_{303}^{Tm} (22.08 + 0.008971T) \, dT \\ &= 460 \times (T_m - 303) + 0.094 \times (T_m^2 - 303^2) \end{split}$$

$$\begin{split} H_{CaCO3} &= \mathring{n}_{CaCO3} \times \int_{303}^{Tm} (19.68 + 0.01189T) \, dT \\ &= 126.15 \times (T_m - 303) + .038 \times (T_m^2 - 303^2) \\ H_{Fe2O3} &= \mathring{n}_{Fe2O3} \times \int_{303}^{Tm} (24.72 + 0.01604T) \, dT \\ &= 132.08 \times (T_m - 303) + 0.043 \times (T_m^2 - 303^2) \\ H_{MgCO3} &= \mathring{n}_{MgCO3} \times \int_{303}^{Tm} 16.9 \, dT \\ &= 43.06 \times (T_m - 303) \end{split}$$

Now, $Q_c = H_{SiO2} + H_{Al2O3} + H_{CaCO3} + H_{Fe2O3} + H_{MgCO3}$ = 2586.17 × (T_m - 303) + 0.897 × (T_m² - 303²)

Now, $Q_1 = Q_c$ = 15783.3 × (336.25 - T_m) + 4.723 × (336.25² - T_m²) = 2586.17 × (T_m - 303) + 0.897 × (T_m² - 303²) T_m = 331.50 K = 58.40°C

So, temperature of feed for raw mill $T_{mixer} = 58.40$ °C

So, Heat given by limestone = $15783.3 \times (336.25 - T_m) + 4.723 \times (336.25^2 - T_m^2)$ = $15783.3 \times (336.25 - 331.50) + 4.723 \times (336.25^2 - 331.50^2)$ = 89951.15 cal/day = Heat taken by clay

5.3 Raw Mill



| | 1 | | | |
|--------------------------------|-------------------------------|--------|---------|--|
| Component | Feed for raw mill (95 kg/day) | | | |
| | Amount | | | |
| | (weight %) | Kg/day | Mol/day | |
| SiO ₂ | 12.20 | 11.59 | 193.167 | |
| Al ₂ O ₃ | 3.25 | 3.09 | 30.294 | |
| CaCO ₃ | 80.96 | 76.91 | 769.1 | |
| Fe ₂ O ₃ | 1.38 | 1.32 | 8.25 | |
| MgCO ₃ | 1.88 | 1.79 | 21.315 | |
| Inert | .36 | .342 | | |

Assuming mixer output of 25 mm average size was fed to the Raw mill and grinded up to 1 mm

Rittinger's Law is applied to find energy required for fine grinding up to 1mm .

According to Rittinger's Law, required energy

$$\mathbf{E}_{\mathbf{R}} = \mathbf{K}_{\mathbf{R}} \times \mathbf{fc} \times \left(\frac{1}{L2} - \frac{1}{L1}\right);$$

Here, K_R = Proportionality Constant

 f_c = crushing strength L_1 = Initial Size L_2 = Final Size

For reducing size from 100 mm to 25 mm, Rittinger's Law

$$E_{R1} = K_R \times fc \times (\frac{1}{25} - \frac{1}{100}) = 5.244$$
(iv)

For reducing size from 25 mm to 1 mm, Rittinger's Law

$$E_{R2} = K_R \times fc \times (\frac{1}{1} - \frac{1}{25})$$
(v)

 $(v) \div (iv)$

$$\frac{\text{ER2}}{5.244} = \frac{(1 - \frac{1}{25})}{(\frac{1}{25} - \frac{1}{100})}$$

 $E_{R2} = 167.80 \text{ KJ/kg}$

For $\mathring{m} = 95 \text{ kg/day}$ feed , total energy required for grinding

 $\mathbf{E}_t = \mathring{\mathbf{m}} \times \mathbf{E}_{\mathbf{R}2}$

 $=95 \times 167.80$

Assuming, 80% of the applied energy (Ea) is used in grinding .

$$\mathbf{E}_{\mathbf{a}} = \frac{Et}{\mathbf{0.8}} = \frac{15941.76}{0.8} = 19927.20 \text{ KJ/day}$$

Assuming all the lost energy (E_l) is converted to heat energy .

$$E_{l} = E_{a.} - E_{t}$$

$$= 19927.20 - 15941.76$$

$$= 3985.44 \frac{kj}{day} \times \frac{1000j}{1kj} \times \frac{1cal}{4.18j}$$

$$= 953454.55 \text{ cal/day}$$

Now,
$$\mathbf{E}_{l} = \sum H = \int_{T_{i}}^{T_{f}} \mathbf{n} \times Cp(T) \times dT$$
(iii)

Here, Cp_i = specific heats of components, n = molar flow rate (mol/day), T = temperature (K)

$$\begin{split} H_{SiO2} &= \mathring{n}_{SiO2} \times \int_{331.50}^{T} (10.87 + 0.008712T) \, dT \\ &= 2099.73 \times (T - 331.50) + 0.831 \times (T^2 - 331.50^2) \\ H_{Al2O3} &= \mathring{n}_{Al2O3} \times \int_{331.50}^{T} (22.08 + 0.008971T) \, dT \\ &= 668.89 \times (T - 331.50) + 0.135 \times (T^2 - 331.50^2) \\ H_{CaCO3} &= \mathring{n}_{CaCO3} \times \int_{331.50}^{T} (19.68 + 0.01189T) \, dT \\ &= 15135.89 \times (T - 331.50) + 4.62 \times (T^2 - 331.50^2) \\ H_{Fe2O3} &= \mathring{n}_{Fe2O3} \times \int_{331.50}^{T} (24.72 + 0.01604T) \, dT \\ &= 203.94 \times (T - 331.50) + 0.067 \times (T^2 - 331.50^2) \\ H_{MgCO3} &= \mathring{n}_{MgCO3} \times \int_{331.50}^{T} 16.9 \, dT \\ &= 360.23 \times (T - 331.50) \end{split}$$

Now from (iii)

$$E_1 = H_{SiO2} + H_{Al2O3} + H_{CaCO3} + H_{Fe2O3} + H_{MgCO3}$$

 $953454 = 18468.68 \times (T - 331.50) + 5.65 \times (T^2 - 331.50^2)$

 $T = 373.96 \text{ K} = 100.86 \text{ }^{\circ}\text{C}$

So, temperature of grinded material, $T_{RAW}\!=100.86$ °C

5.4 Preheater



| | Preheater Feed (M7) = 94.525 kg/day | | | | |
|--------------------------------|---|----------|---------------------------|--|--|
| Component | Amount (kg/day) | Weight % | Molar amount (mol/day) | | |
| SiO ₂ | 11.50 | 12.20 | 191.67 | | |
| Al ₂ O ₃ | 3.08 | 3.25 | 30.26 | | |
| CaCO ₃ | 76.53 | 80.96 | 765.30 | | |
| Fe ₂ O ₃ | 1.31 | 1.38 | 8.19 | | |
| MgCO ₃ | 1.78 | 1.88 | 21.19 | | |
| Inert | .34 | .36 | | | |

| ~ | Coal (100 kg/day) | | | | |
|--------------------------------|-------------------|--------------|-------------------|-----------------|--|
| Component | Weight percent | Kg/day | Mol/day | Mol fraction | |
| Carbon | 70 | 70 | 5833.33 | .904 | |
| Al ₂ O ₃ | 4.5 | 4.5 | 44.12 | .007 | |
| SiO ₂ | 7.5 | 7.5 | 125 | .019 | |
| Fe ₂ O ₃ | 9 | 9 | 56.25 | .009 | |
| CaO | 0.9 | 0.9 | 16.07 | .002 | |
| MgO | 0.3 | 0.3 | 7.5 | .001 | |
| SO ₃ | 1.5 | 1.5 | 18.75 | .002 | |
| Moisture | 6.3 | 6.3 | 350 | .054 | |
| Total | | = 100 kg/day | = 6451.02 mol/day | | |

Average molecular weight of coal
$$M_{avg} = \frac{100}{6451.02} \times \frac{kg}{day} \times \frac{day}{mol} \times \frac{1000 g}{1 kg}$$

Calorific heat of coal $Q_c = 25.68 \text{ Mj/kg}$

$$= 25.68 \times \frac{Mj}{kg} \times \frac{10^{6}j}{1\,Mj} \times \frac{1\,cal}{4.18\,j} \times \frac{1\,kg}{1000\,g} \times \frac{15.50\,g}{1\,mol}$$

= 95224.88 cal/mol

So, 1 mol coal produces 95224.88 calories by combustion

 $C + O_2 = CO_2$; $\Delta H_r = -95224.88$ cal/mol

Now, $\mathbf{n}_{output} = \mathbf{n}_{input} + \boldsymbol{v}_{i \times \boldsymbol{\varepsilon}}$

Here , v_i = Stoichiometric Coefficient

 ϵ = Extent of reaction

So, $\mathbf{n}_{C(0)} = \mathbf{n}_{C(i)} + \boldsymbol{v}_{C \times \boldsymbol{\varepsilon}}$

 $0 = 0.904P - (1 \times \epsilon)$ [Assuming 100 % conversion of carbon]

 $\epsilon = 0.904P$

 $\mathbf{n}_{O2(o)} = \mathbf{n}_{O2(i)} + \mathbf{v}_{O2 \times \mathbf{\epsilon}}$

 $= 2P - (1 \times .904P)$ [50 % excess air]

= 1.096P

 $n_{CO2(o)} = n_{CO2(i)} + \upsilon_{CO2 \times} \epsilon$

 $= 0 + (1 \times 0.904 P)$

= 0.904P

 $SiO_2(s)$, $Al_2O_3(s)$, $CaCO_3(s)$, $Fe_2O_3(s)$, $MgCO_3(s)$, C(s), CaO(s), MgO(s), $SO_3(g)$, $H_2O(l)$ at 30°C and 1 atm

$O_2(g)$, $N_2(g)$, $CO_2(g)$ at 25°C and 1 atm

| Component | n ⁱⁿ (mol/day) | $\hat{\mathrm{H}}_{\mathrm{in}}$ (cal/mol) | n _{out} (mol/day) | \hat{H}_{out} (cal/mol) | $\operatorname{Cp's}\left(\frac{cal}{mol\ k}\right)$ |
|--------------------------------|---------------------------|--|----------------------------|---------------------------|--|
| SiO ₂ | 191.67 | \hat{H}_1 | 191.67+.019P | Ĥ9 | |
| Al ₂ O ₃ | 30.26 | \hat{H}_2 | 30.26+.007P | $\hat{\mathrm{H}}_{10}$ | |
| CaCO ₃ | 765.30 | Ĥ3 | 765.30 | \hat{H}_{11} | |
| Fe ₂ O ₃ | 8.19 | Ĥ4 | 8.19+.009P | Ĥ ₁₂ | |
| MgCO ₃ | 21.19 | Ĥ5 | 21.19 | \hat{H}_{13} | |
| С | .904P | 0 | | | |
| Al ₂ O ₃ | .007P | 0 | | | |
| SiO ₂ | .019P | 0 | | | |
| Fe ₂ O ₃ | .009P | 0 | | | |
| CaO | .002P | 0 | .002P | \hat{H}_{14} | |
| MgO | .001P | 0 | .001P | \hat{H}_{15} | |
| SO ₃ | .002P | 0 | .002P | \hat{H}_{16} | |
| H ₂ O(l) | .054P | $\hat{\mathrm{H}}_{6}$ | | | |
| H ₂ O(v) | | | .054P | \hat{H}_{17} | |
| O ₂ | 2P | Ĥ ₇ | 1.096P | $\hat{\mathrm{H}}_{18}$ | |
| N2 | 7.52P | $\hat{\mathrm{H}}_8$ | 7.52P | Ĥ19 | |
| CO ₂ | | | .904P | Ĥ ₂₀ | |

Now,
$$\mathbf{E}_{l} = \sum H = \int_{T_{i}}^{T_{f}} \mathbf{n} \times Cp(T) \times dT$$
(iii)

Here, Cp_i = specific heats of components, n = molar flow rate (mol/day), T = temperature (K)

Enthalpy Calculation:

Enthalpy of SiO₂ at 100.86 °C:

H₁ =
$$\mathring{n}_{Si02} \times \int_{303.1}^{373.96} (10.87 + 0.008712T) dT$$

= 2083.45 × (373.96 - 303.1) + .83× (373.96² - 303.10²)
=187453 cal

Enthalpy of Al₂O₃ at 100.86 °C:

H₂ =
$$\mathring{n}_{Al2O3} \times \int_{303.1}^{373.96} (22.08 + 0.008971T) dT$$

= 668.14 × (373.96 - 303.1) + .135× (373.96² - 303.10²)
= 53821 cal

Enthalpy of CaCO₃ at 100.86 °C:

H₃ =
$$\mathring{n}_{CaCO3} \times \int_{303.1}^{373.96} (19.68 + 0.01189T) dT$$

= $15061 \times (373.96 - 303.1) + 4.5 \times (373.96^2 - 303.10^2)$
= 1283116 cal
Enthalpy of Fe₂O₃ at 100.86 °C:

H₄ =
$$\mathring{n}_{Fe2O3} \times \int_{303.1}^{373.96} (24.72 + 0.01604T) dT$$

= 202.46 × (373.96 - 303.1) + .066× (373.96² - 303.10²)
= 17513 cal

Enthalpy of MgCO₃ at 100.86 °C:

H₅ =
$$\mathring{n}_{MgCO3} \times \int_{303.1}^{373.96} 16.9 \ dT$$

= 358.11 × (373.96 - 303.1)
= 25376 cal

Enthalpy of H₂O at 30 °C:

$$H_6 = \mathring{n}_{H2O} \times \widehat{H}_6$$

= .054P × 40.5
= 2.187P cal

Enthalpy of O₂ at 300°C:

H₇ =
$$\mathring{n}_{O2} \times \widehat{H}_7$$

= 2P × 2026.3
= 4053P cal

Enthalpy of N₂ at 300°C:

$$\begin{array}{ll} H_8 & = \,\,{\mathring{n}}_{N2} \! \times \! \hat{H}_8 \\ & = 7.52 P \times 1942.58 \\ & = \! 14608 P \, cal \end{array}$$

Enthalpy of SiO₂ at 700 °C:

H₉ =
$$\mathring{n}_{SiO2} \times \int_{303.1}^{973.1} (10.87 + 0.008712T) dT$$

= $(191.67 + .019P) \times [10.87 \times (973.1 - 303.1) + .004 \times (973.1^2 - 303.1^2)]$
= $2051466 + 203.36P$ cal

Enthalpy of Al₂O₃ at 700 °C:

H₁₀ =
$$\mathring{n}_{Al2O3} \times \int_{303.1}^{973.1} (22.08 + 0.008971T) dT$$

= $(30.26 + .007P) \times [22.08 \times (973.1 - 303.1) + .0045 \times (973.1^2 - 303.1^2)]$
= $564087 + 130.5P$ cal

Enthalpy of CaCO₃ at 700°C:

H₁₁ =
$$\mathring{n}_{CaCO3} \times \int_{303.1}^{973.1} (19.68 + 0.01189T) dT$$

= 765.30 × [19.68 × (973.1-303.1) + .0056 × (973.1² - 303.1²)]
= 13755427 cal

Enthalpy of Fe₂O₃ at 700°C:

H₁₂ =
$$\mathring{n}_{Fe2O3} \times \int_{303.1}^{973.1} (24.72 + 0.01604T) dT$$

= $(8.19+.009P) \times [24.72 \times (973.1-303.1) + .008 \times (973.1^2 - 303.1^2)]$
= $191669 + 210.6P$ cal

Enthalpy of MgCO₃ at 700°C:

H₁₃ =
$$\mathring{n}_{MgCO3} \times \int_{303.1}^{973.1} 16.9 \ dT$$

= 21.19 × [16.9 × (973.1 - 303.1)]
= 239934 cal

Enthalpy of CaO at 700 °C:

H₁₄ =
$$\mathring{n}_{CaO} \times \int_{303.1}^{973.1} (10.00 + 0.00484T) dT$$

= .002P × [10.00 × (973.1- 303.1) + .0024 × (973.1² - 303.1²)]
= 17.50P cal

Enthalpy of MgO at 700°C:

H₁₅ =
$$\mathring{n}_{MgO} \times \int_{303.1}^{973.1} (10.86 + 0.0012T) dT$$

= .001P × [10.86 × (973.1- 303.1) + .0006 × (973.1² - 303.1²)]
= 7.79 P cal

Enthalpy of SO₃ at 700 °C:

H₁₆ =
$$\mathring{n}_{SO3} \times \int_{303.1}^{973.1} (7.7 + 0.0053T) dT$$

= .002P × [7.7 × (973.1- 303.1) + .0053 × (973.1² - 303.1²)]
= 19.38P cal

Enthalpy of H₂O at 700°C:

- $\mathbf{H}_{17} = \mathbf{\mathring{n}}_{H2O} \times \mathbf{\widehat{H}}_{17}$
 - $= .054P \times 5962$
 - = 322 P cal

Enthalpy of O₂ at 700 °C:

$$H_{18} = \mathring{n}_{02} \times \widehat{H}_{18}$$

= 1.096P × 5230
= 5732P cal

Enthalpy of N₂ at 700°C:

 $H_{19} = \mathring{n}_{N2} \times \widehat{H}_{19}$ = 7.52P × 4925 = 37036P cal

Enthalpy of CO₂ at 700°C:

 $H_{20} = \mathring{n}_{CO2} \times \widehat{H}_{20}$ = .904P × 7626 = 6894P cal

Total inlet enthalpies :

 $\sum n_{in} H_{in} = H_1 + \dots + H_8$ = 1567279 + 18663P

Total Outlet Enthalpies :

 $\sum n_{out} H_{out} = H_9 + \dots + H_{20}$

$$= 16802583 + 50573P$$

$$\Delta H = \sum n_{out} H_{out} - \sum n_{in} H_{in}$$
$$= 15235304 + 31910P$$

Amount of Coal Required in the Preheater:

95224.88P = 15235304 + 31910P

P = 240.63 mol of coal/day

= 3730 g coal/day = 3.73 kg/day

Air Inlet (300°C) = 9.52P mol/day = 9.52×240.63× $\frac{.0821 \times 573}{3.5}$ × $\frac{1m^3}{1000L}$ = 30.79 m³/day

Exhaust gas:

 $\mathring{n}_{O2} = 1.06 \text{ P} = 1.06 \times 240.63 \text{ mol/day} = 255.07 \text{ mol } O_2/\text{day}$

 \mathring{n}_{N2} = 7.52P = 7.52× 240.63 mol/day = 1809 mol N₂ /day

 n_{CO2} = .94P = .94 × 240.63 mol/day = 226.19 mol CO₂/day

| Component | n ⁱⁿ (mol/day) | $\hat{\mathrm{H}}_{\mathrm{in}}$ (cal/mol) | n _{out} (mol/day) | $\hat{\mathrm{H}}_{\mathrm{out}}\left(\mathrm{cal/mol} ight)$ | $\operatorname{Cp's}\left(\frac{cal}{mol\ k}\right)$ |
|--------------------------------|---------------------------|--|----------------------------|---|--|
| SiO ₂ | 191.67 | Ĥ1 | 196.24 | Ĥ9 | |
| Al ₂ O ₃ | 30.26 | Ĥ2 | 31.94 | $\hat{\mathrm{H}}_{10}$ | |
| CaCO ₃ | 765.30 | Ĥ ₃ | 765.30 | $\hat{\mathrm{H}}_{11}$ | |
| Fe ₂ O ₃ | 8.19 | $\hat{\mathrm{H}}_4$ | 10.36 | Ĥ ₁₂ | |
| MgCO ₃ | 21.19 | Ĥ5 | 21.19 | Ĥ ₁₃ | |
| С | 217.53 | 0 | | | |
| Al ₂ O ₃ | 1.68 | 0 | | | |
| SiO ₂ | 4.57 | 0 | | | |
| Fe ₂ O ₃ | 2.17 | 0 | | | |
| CaO | .48 | 0 | .48 | $\hat{\mathrm{H}}_{14}$ | |
| MgO | .24 | 0 | .24 | $\hat{\mathrm{H}}_{15}$ | |
| SO ₃ | .48 | 0 | .48 | $\hat{\mathrm{H}}_{16}$ | |
| H ₂ O(l) | 12.99 | $\hat{\mathrm{H}}_{6}$ | | | |
| H ₂ O(v) | | | 12.99 | Ĥ ₁₇ | |
| O ₂ | 481.26 | Ĥ ₇ | 263.73 | $\hat{\mathrm{H}}_{18}$ | |
| N ₂ | 1809.54 | Ĥ ₈ | 1809.54 | Ĥ19 | |
| CO ₂ | | | 217.53 | Ĥ20 | |



Kiln Feed Composition:

| Components | Mol/day | Kg/day | Weight % |
|--------------------------------|---------|----------------|----------|
| SiO ₂ | 196.24 | 11.77 | 12.4 |
| Al ₂ O ₃ | 31.94 | 3.26 | 3.43 |
| CaCO ₃ | 765.30 | 76.53 | 80.5 |
| Fe ₂ O ₃ | 10.36 | 1.66 | 1.8 |
| MgCO ₃ | 21.19 | 1.78 | 1.9 |
| CaO | .48 | .027 | .28 |
| MgO | .24 | .01 | .11 |
| | | Total = 95.037 | |

| Components | n [°] in (mol/day) | Ĥ _{in} (cal/mol) | nout (mol/day) | Ĥ _{out} (cal/mol) | Cp's $(\frac{cal}{mol \ k})$ |
|------------|-----------------------------|---------------------------|----------------|-------------------------------|------------------------------|
| SiO2 | 196.24 | Ĥ1 | 0 | | |
| Al2O3 | 31.94 | Ĥ2 | 0 | | |
| CaCO3 | 765.30 | Ĥ3 | 0 | | |
| Fe2O3 | 10.36 | Ĥ4 | 0 | | |
| MgCO3 | 21.19 | Ĥ5 | 0 | | |
| CaO | .48 | Ĥ6 | 130.57 | Ĥ17 | |
| MgO | .24 | Ĥ7 | 21.43 | Ĥ18 | |
| С | .904Q | 0 | 0 | | |
| A12O3 | .007Q | Ĥ8 | .007Q | Ĥ19 | |
| SiO2 | .019Q | Ĥ9 | .019Q | Ĥ20 | |
| Fe2O3 | .009Q | Ĥ10 | .009Q | Ĥ21 | |
| CaO | .002Q | Ĥ11 | 0 | | |
| MgO | .001Q | Ĥ12 | 0 | | |

| SO3 | .002Q | Ĥ13 | .002Q | Ĥ22 | |
|--------|-------|-----|-------------|-----|--|
| H2O(l) | .054Q | Ĥ14 | | | |
| H2O(v) | | | .054Q | Ĥ23 | |
| O2 | 2Q | Ĥ15 | 1.06Q | Ĥ24 | |
| N2 | 7.52Q | Ĥ16 | 7.52Q | Ĥ25 | |
| CO2 | | | 786.49+.94Q | Ĥ26 | |
| C2S | | | 60.19 | Ĥ27 | |
| C3S | | | 136.04 | Ĥ28 | |
| СЗА | | | 21.58 | Ĥ29 | |
| C4AF | | | 10.36 | Ĥ30 | |

Now,
$$\mathbf{E}_{\mathbf{I}} = \sum H = \int_{T_i}^{T_f} \mathbf{n}^* \times Cp(T) \times dT$$
(iii)

Here, Cp_i = specific heats of components, n = molar flow rate (mol/day), T = temperature (K)

Enthalpy Calculation :

Enthalpy of SiO₂ at 700 °C:

H₁ =
$$\mathring{n}_{SiO2} \times [(\Delta H_f^o)_{SiO2} + \int_{298.1}^{973.1} (10.87 + 0.008712T) dT$$

= 196.24 × [-202620 + 10.87 × (973.1 - 298.1) + .004 × (973.1² - 298.1²)]
= -37.65 × 10⁶ cal

Enthalpy of Al₂O₃ at 700 °C:

$$H_{2} = \mathring{n}_{Al2O3} \times \left[\left(\Delta H_{f}^{0} \right)_{Al2O3} + \int_{298.1}^{973.1} (22.08 + 0.008971T) dT \\ = 31.94 \times \left[-400000 + 22.08 \times (973.1 - 298.1) + .0045 \times (973.1^{2} - 298.1^{2}) \right] \\ = -12.18 \times 10^{6} \text{ cal}$$

Enthalpy of CaCO₃ at 700 °C:

H₃ =
$$\mathring{n}_{CaCO3} \times [(\Delta H_f^0)_{CaCO3} + \int_{298.1}^{973.1} (19.68 + 0.01189T) dT$$

= 765.30 × [-298500 + 19.68 × (973.1 - 298.1) + .0056 × (973.1² - 298.1²)]
= -214.60 × 10⁶ cal

Enthalpy of Fe₂O₃ at 700°C:

H₄ =
$$\mathring{n}_{Fe2O3} \times [(\Delta H_f^o)_{Fe2O3} + \int_{298.1}^{973.1} (24.72 + 0.01604T) dT$$

= 10.36 × [-198500 + 24.72 × (973.1 - 298.1) + .008 × (973.1² - 298.1²)]
= -1.81 × 10⁶ cal

Enthalpy of MgCO₃ at 700 °C:

H₅ =
$$\mathring{n}_{MgCO3} \times [(\Delta H_f^o)_{MgCO3} + \int_{298.1}^{973.1} 16.9 \, dT$$

= 21.19 × [-261700 + 16.9 × (973.1 - 298.1)]
= -5.3 × 10⁶ cal

Enthalpy of CaO at 700 °C:

$$H_{6} = \mathring{n}_{CaO} \times [(\Delta H_{f}^{O})_{CaO} + \int_{298.1}^{973.1} (10.00 + 0.00484T) dT$$

= .48 × [-151700 + 10 × (973.1 - 298.1) + .0024 × (973.1² - 298.1²)]
= -68.59 × 10³ cal

Enthalpy of MgO at 700°C:

$$H_7 = \mathring{n}_{MgO} \times \left[\left(\Delta H_f^o \right)_{MgO} + \int_{298.1}^{973.1} (10.86 + 0.0012T) dT \right]$$

= .24 × [-143840 + 10.86 × (973.1 - 298.1) + .0006 × (973.1² - 298.1²)]
= -32.63 × 10³ cal

Enthalpy of Al₂O₃ at 25 °C:

H₈ =
$$\mathring{n}_{A12O3} \times (\Delta H_f^o)_{A12O3}$$

= .007Q × (- 400000)
= -2800Q cal

Enthalpy of SiO₂ at 25 °C:

H₉ =
$$\mathring{n}_{SiO2} \times (\Delta H_f^o)_{SiO2}$$

= .019Q × (-202620)
= -3849 Q cal

Enthalpy of Fe₂O₃ at 25 °C:

H₁₀ =
$$\mathring{n}_{Fe2O3} \times (\Delta H_f^o)_{Fe2O3}$$

= .009Q × (-198500)
= -1786Q cal

Enthalpy of CaO at 25 °C:

H₁₁ =
$$\mathring{n}_{CaO} \times (\Delta H_f^o)_{CaO}$$

= .002Q × (-151700)
= -303Q cal

Enthalpy of MgO at 25°C:

H₁₂ =
$$\mathring{n}_{MgO} \times (\Delta H_f^o)_{MgO}$$

= .001Q × (-143840)
= 143Q cal

Enthalpy of H₂O at 25 °C:

H₁₄ =
$$\mathring{n}_{H2O} \times (\Delta H_f^o)_{H2O}$$

= .054Q × (-68380)
= 3692Q cal

Enthalpy of O₂ at 300°C:

$$H_{15} = \mathring{n}_{02} \times \widehat{H}_{15}$$

= 2Q × 2026.3
= 4053Q cal

Enthalpy of N₂ at 300°C:

$$H_{16} = \mathring{n}_{N2} \times \widehat{H}_{16}$$

= 7.52Q × 1942.58
=14608Q cal

Enthalpy of CaO at 1400 °C:

$$H_{17} = \mathring{n}_{CaO} \times [(\Delta H_f^O)_{CaO} + \int_{298.1}^{1673.1} (10.00 + 0.00484T) dT$$

= 130.57 × [-151700 + 10 × (1673.1 - 298.1) + .0024 × (1673.1² - 298.1²)]
= -17.16 × 10⁶ cal

Enthalpy of MgO at 1400°C:

H₁₈ =
$$\mathring{n}_{MgO} \times [(\Delta H_f^o)_{MgO} + \int_{298.1}^{1673.1} (10.86 + 0.0012T) dT$$

= 21.43 × [-143840 + 10.86 × (1673.1 - 298.1) + .0006 × (1673.1² - 298.1²)]
= -2.72 × 10⁶ cal

Enthalpy of Al₂O₃ at 1400 °C:

H₁₉ =
$$\mathring{n}_{Al2O3} \times [(\Delta H_f^0)_{Al2O3} + \int_{298.1}^{1673.1} (22.08 + 0.008971T) dT$$

= .007Q × [-400000 + 22.08 × (1673.1 - 298.1) + .0045 × (1673.1² - 298.1²)]
= -2502Q cal

Enthalpy of SiO₂ at 1400 °C:

$$H_{20} = \mathring{n}_{SiO2} \times [(\Delta H_f^o)_{SiO2} + \int_{298.1}^{1673.1} (10.87 + 0.008712T) dT$$

= .019Q × [-202620 + 10.87 × (1673.1 - 298.1) + .004 × (1673.1² - 298.1²)]
= -3359Q cal

Enthalpy of Fe₂O₃ at 1400°C:

$$H_{21} = \mathring{n}_{Fe2O3} \times \left[\left(\Delta H_f^0 \right)_{Fe2O3} + \int_{298.1}^{1673.1} (24.72 + 0.01604T) dT \right]$$

= .009Q × [-198500 + 24.72 × (1673.1 - 298.1) + .008 × (1673.1² - 298.1²)]
= -1285Q cal

Enthalpy of H₂O at 1400°C:

$$H_{23} = \mathring{n}_{H20} \times [(\Delta H_f^o)_{H20} + \hat{H}_{23}]$$

= .054Q × [-68380 + 13456]
= -2965Q cal

Enthalpy of O₂ at 1400°C:

$$H_{24} = \mathring{n}_{02} \times \widehat{H}_{24}$$

= 1.06Q × 11260
= 11935Q cal

Enthalpy of N₂ at 1400°C:

$$\begin{array}{ll} H_{25} & = \,\,\mathring{n}_{N2} \times \hat{H}_{25} \\ & = 7.52 Q \times 10648 \\ & = 80072 Q \,\, cal \end{array}$$

Enthalpy of CO₂ at 1400°C:

$$H_{26} = \mathring{n}_{CO2} \times [(\Delta H_f^o)_{CO2} + \hat{H}_{26}]$$

= (786.49+.94Q) ×(-94138 + 17198)
= -60.51 × 10⁶ - 72323Q cal

Heat of formation of clinker content:

$$(\Delta H_f^o)_{C2S} = -552033 \text{ cal/mol}$$

 $(\Delta H_f^o)_{C3S} = -700765 \text{ cal/mol}$
 $(\Delta H_f^o)_{C3A} = -858516 \text{ cal/mol}$
 $(\Delta H_f^o)_{C4AF} = -1218394 \text{ cal/mol}$
 $(\Delta H_f^o)_{\text{inert}} = -151700 \text{ cal/mol}$
 $(\Delta H_f^o)_{Clinker} = -699952 \text{ cal/mol}$

Average Molecular weight of Clinker = 220 g/mole

Specific heat of Cement:

$$(Cp)_{\text{clinker}} = .92 \frac{kj}{kg} \times \frac{1000J}{1 \, \text{K}j} \times \frac{1 \, kg}{1000 \, g} \times \frac{1 \, cal}{4.18 \, j} \times \frac{220 \, g}{1 \, mol}$$
$$= 48.42 \text{ cal/mol K}$$

Enthalpy of Clinker at 1400 °C:

H₂₇ =
$$\mathring{n}_{clinker} \times [(\Delta H_f^o)_{Clinker} + \int_{298.1}^{1673.1} (48.42) dT$$

= $380 \times [-699952 + 48.42 \times (1673.1 - 298.1)]$
= -240.68×10^6 cal

Total inlet enthalpies:

 $\sum n_{in} H_{in} = H_1 + \dots + H_{16}$ $= -271.64 \times 10^6 - 3123Q$

Total Outlet Enthalpies:

 $\sum n_{out} H_{out} = H_{17} + \dots + H_{27}$ $= -321.07 \times 10^6 - 1744410$

$$\Delta H = \sum n_{out} H_{out} - \sum n_{in} H_{in}$$
$$= -49.43 \times 10^6 - 171318P$$

Amount of Coal Required in the kiln:

 $\text{-95224.88P} = \text{-49.43} \times 10^6 \text{ - 171318P}$

P = 649.59 mol of coal/day

= 10068 g coal/day = 10.068 kg/day

Air Inlet (300°C) = 9.52P mol/day = $9.52 \times 649.59 \times \frac{.0821 \times 573}{3.5} \times \frac{1m^3}{1000L} = 83.12 \text{ m}^3/\text{day}$

Exhaust gas:

 $\mathring{n}_{O2} = 1.06 \text{ P} = 1.06 \times 649,59 \text{ mol/day} = 688.57 \text{ mol } O_2/\text{day}$

 \mathring{n}_{N2} = 7.52P = 7.52× 649.59 mol/day = 4884 mol N₂ /day

 n_{CO2} = .94P = .94 × 649.59 mol/day = 610 mol CO₂/day

5.6 Clinker Cooler



For, Coal input in Rotary kiln Q = 10.068 kg/day = 659.75 mol/day

Combustible Carbon in coal = $.904 \times 659.75 = 587.23$ mol/day

 O_2 required = 2P mol/day

= 1174 mol/day

 N_2 in air = 7.52P

= 4416 mol/day

Air inlet in Rotary Kiln $(300^{\circ}C) = 1174 + 4416$

= 5590 mol/day

| For, Coal input in Preheater P = | = 3.73 kg/day = 240.63 mol/day |
|----------------------------------|---|
| Combustible Carbon in coal = | $.904 \times 240.63 = 217.53 \text{ mol/day}$ |
| O_2 required = 2 | 2P mol/day |
| = 4 | 135 mol/day |
| N_2 in air = $$ | 7.52P |
| = 1 | 635 mol/day |
| Air inlet in Preheater (300°C) | = 435 + 1635 |
| | = 2070 mol/day |

Air input in cooler must be enough for providing combustion of coal in both preheater and rotary kiln

Total Air inlet in cooler = 2070 + 5590 = 7660 mol/day

$$= 7660 \ \frac{mol}{day} \times \frac{28.84 \ g}{mol} \times \frac{1 \ kg}{1000 \ g} \times \frac{1 \ m3}{1.225 \ kg}$$
$$= 180 \ m^3 / \ day \ air$$

Specific Heat of air, $C_p = 6.9 \text{ cal/mol.k}$

According to Heat balance,

 $[n \times C_p \times (T_{in}-T_{out})]_{clinker} = [n \times C_p \times (T_{out}-T_{in})]_{air}$

Or, 380×48.42×(1400-150) = 7660×6.9×(Tout -25)

Tout=460°C

5.7 Scale up value

For Preheater scale up value:

From material balance, scale up factor is 65.53 ton/kg So, Coal required, P= $65.53 \frac{ton}{kg} \times 3.73 \frac{kg}{dav} = 244.43 \text{ ton/day}$ Again, Coal Required P = 244.43 ton/day = $\frac{244.43 \text{ ton}}{day} \times \frac{10^6 \text{ g}}{1 \text{ ton}} \times \frac{1 \text{ mol}}{15.5 \text{ g}} = 15.77 \times 10^6 \text{ mol/day}$ Air Inlet (300°C) = 9.52P mol/day = $150.13 \times 10^6 \times \frac{.0821 \times 573}{3.5} \times \frac{1m^3}{1000L} = 2.018 \times 10^6 \text{ m}^3/\text{day}$ Exhaust gas: (700°C) $\mathring{n}_{O2} = 1.06 \text{ P} = 1.06 \times 15.77 \times 10^6 \text{ mol/day} = 16.71 \times 10^6 \text{ mol } O_2/\text{day}$ $= .53 \times 10^6$ kg/day \mathring{n}_{N2} = 7.52P = 7.52× 15.77 × 10⁶ mol/day = 118.59× 10⁶ mol N₂ /day $= 3.32 \times 10^6$ kg/day $\mathring{n}_{CO2} = .94P = .94 \times 15.77 \times 10^6 \text{ mol/day} = 14.82 \times 10^6 \text{ mol CO}_2/\text{day}$ $= .65 \times 10^6$ kg/day $n_{SO2} = .002 \text{ P} = .002 \times 15.77 \times 10^6 \text{ mol/day} = .03 \times 10^6 \text{ mol SO}_2/\text{day}$ $= 2.02 \times 10^3$ kg/day \mathring{n}_{H2O} = .054 P = .054 × 15.77 × 10⁶ mol/day = .85 × 10⁶ mol H₂O/day $= 15.32 \times 10^3$ kg/day

Assuming .5% loss as dust, dust = $.005 \times 6194$ ton/day

$$= 30.1 \times 10^{3} \text{ kg/day}$$

Total exhaust = $(53 \times 10^6 + 3.32 \times 10^6 + .65 \times 10^6 + 2.02 \times 10^3 + 30.1 \times 10^3 + 15.32 \times 10^3)$

 $=4.55\times10^6$ kg/day

Molar amount = $(16.71 \times 10^6 +$

 $= 151 \times 10^6 \text{ mol/day}$

$$= 151 \times 10^{6} \times \frac{.0821 \times 973}{3} \times \frac{1m^{3}}{1000L} = 3.99 \times 10^{6} \text{ m}^{3}/\text{day}$$
$$= 3.99 \times 10^{6} \times \frac{m^{3}}{day} \times \frac{1day}{24 \times 3600 \text{ s}}$$
$$= 46.27 \frac{m^{3}}{sec}$$

Amount of $CO_2 = 14.82 \times 10^6 \text{ mol } CO_2/\text{day}$

 $= 103 \times 10^2 \text{ mg/m}^3$

Amount of
$$SO_2 = 0.03 \times 10^6 \text{ mol } SO_2/\text{day}$$

 $= 481 \text{ mg/m}^3$

Amount of dust = 7543 mg/m^3

 $Total = 111 \times 10^2 \text{ mg/m}^3$

For Kiln scale up value:

From material balance, scale up factor is 65.53 ton/kg

So, Coal required, P= $65.53 \frac{ton}{kg} \times 10.068 \frac{kg}{day} = 659.75 \text{ ton/day}$ Coal Required Q= $659.75 \text{ ton/day} = \frac{659.75 \text{ ton}}{day} \times \frac{10^6 \text{ g}}{1 \text{ ton}} \times \frac{1 \text{ mol}}{15.5 \text{ g}} = 41.90 \times 10^6 \text{ mol/day}$ Air Inlet (300° C) = $9.52P \text{ mol/day} = 398.88 \times 10^6 \times \frac{.0821 \times 573}{3.5} \times \frac{1m^3}{1000L} = 5.36 \times 10^6 \text{ m}^3/\text{day}$

Exhaust gas: (700°C)

 $\mathring{n}_{O2} = 1.06 \ P = 1.06 \times 41.90 \times 10^{6} \ mol/day = 44.4 \times 10^{6} \ mol \ O_{2}/day$

 $\mathring{n}_{N2} = 7.52P = 7.52 \times 15.77 \times 10^6 \text{ mol/day} = 315.088 \times 10^6 \text{ mol } N_2 \text{ /day}$

 \mathring{n}_{CO2} = .94P = .94 × 15.77 × 10⁶ mol/day = 39.39 × 10⁶ mol CO₂/day

 $\mathring{n}_{SO2} = .002 \text{ P} = .002 \times 15.77 \times 10^{6} \text{ mol/day} = .08 \times 10^{6} \text{ mol SO}_{2}/\text{day}$

 \mathring{n}_{H2O} = .054 P = .054 × 15.77 × 10⁶ mol/day = 2.26 × 10⁶ mol H₂O/day

Assuming .5% loss as dust, dust = $.005 \times 6288$ ton/day

 $= 31.44 \times 10^3 \text{ kg/day}$

Total exhaust = $(53 \times 10^6 + 3.32 \times 10^6 + .65 \times 10^6 + 2.02 \times 10^3 + 30.1 \times 10^3 + 15.32 \times 10^3)$

 $=4.55\times10^{6}$ kg/day

Molar amount = $(16.71 \times 10^6 +$

$$= 401 \times 10^{6} \text{ mol/day}$$

$$= 401 \times 10^{6} \times \frac{.0821 \times 1673}{3} \times \frac{1m^{3}}{1000L} = 18.37 \times 10^{6} \text{ m}^{3}/\text{day}$$

$$= 18.37 \times 10^{6} \times \frac{m^{3}}{day} \times \frac{1day}{24 \times 3600 \text{ s}}$$

$$= 212 \frac{m^{3}}{sec}$$

Amount of $CO_2 = 39.39 \times 10^6 \text{ mol } CO_2/day$

 $= 9435 \text{ mg/m}^3$

Amount of $SO_2 = 0.08 \times 10^6 \text{ mol } SO_2/\text{day}$

 $= 279 \text{ mg/m}^3$

Amount of dust = 1711 mg/m^3

Total= 9631 mg/m³

Chapter 6

Equipment, Codes

6.1 List of Equipment

| Equipment name | Unit No. | No. of Equipment |
|----------------|------------------------------|------------------|
| Conveyor belt | J-110 | 1 |
| Crusher | C-120 | 1 |
| Crusher | C-121 | 1 |
| Storage bin | F -411, F-331, F-332, F-333, | 5 |
| | F-122 | |
| Ball Mill | C-210 | 1 |
| ESP | H-211, | 2 |
| Preheater | E-220 | 1 |
| Filter | H-241 | 1 |
| Rotary kiln | R-240 | 1 |
| Cooler | E-249, E-228 | 2 |
| Cement mill | C-420 | 1 |
| Air blower | G-311 | 1 |
| Air Blower | G-312 | 1 |
| Compressor | G-247 | 2 |
| Clinker Cooler | E-310 | 1 |

6.2 Auxiliaries and off sites

- Power generation unit
- Cooling tower
- Office air cooling system
- Effluent treatment plant
- Limestone, coal, clinker and cement storage
- Product Packaging unit with storage
- Tree belt around the plant to absorb emitted CO₂
- Office Space
- Garage for vehicles parking
- Maintenance Shop
- Jetty
- Laboratories
- Warehouse

6.3 List of vendors

- Rotary kiln : Zhengzhou Hongji Mining Machinery Co., Honghi, Mainland China
- Ball Mill : Wuxi Rising Machinery Jiangsu ,Mainland China
- Jaw Crusher : Zhengzhou Honjgi Mining Machinery Co., Hongji , Mainland China.
- Cone Crusher Crusher : Zhengzhou Honjgi Mining Machinery Co., Hongji , Mainland China.
- Mixer :WenZhou jhen Ten Machineries Co.Ltd ,Mainland China
- Storage: Henan DI Hai Machinery Co.Ltd., Mainland China
- Preheater : Qingdao Enneng Machinery Co.,Ltd . China(Mainland)
- Clinker Cooler: Zhengzhou DIngli Industrial Machinery & Equipment Co., Ltd.China (Mainland)
- Air Blower: Qingdao Enneng Machinery Co.,Ltd . China(Mainland)
- ESP: Zhengzhou DIngli Industrial Machinery & Equipment Co., Ltd. China (Mainland)

6.4 Codes & Standards

IS 269:1989 - Specification for ordinary Portland cement, 33 grade

IS 383:1970 - Specification for coarse and fine aggregates from natural sources for concrete

IS 455:1989 Specification for Portland slag cement

IS 456:2000 Code of practice for plain and reinforced concrete

IS 457:1957 Code of practice for general construction of plain and reinforced concrete for dams and other massive structures

IS 516:1959 Method of test for strength of concrete

IS 650:1991 Specification for standard sand for testing of cement

IS 1199:1959 Methods of sampling and analysis of concrete

IS 1343:1980 Code of practice for prestressed concrete

IS 1344:1981 Specification for calcined clay pozzolana

IS 1489(Part 1):1991 Specification for Portland pozzolana cement Part 1 Flyash based

IS 1489(Part 2):1991 Specification for Portland-pozzolana cement: Part 2 Calcined clay based

IS 1727:1967 Methods of test for pozzolanic materials

Chapter 7

Equipment Sizing

7.1 Sizing of Conveyor belt (J-110)

Belt specification: Synthetic fiber with a tensile strength of 600kg/cm.

Belt width,
$$l = 17000$$
 m

Power drive: Gear motor

C= Auxiliary resistant coefficient (1.05 for long belt and 10 for short belt) = 1.05

f = main resistant coefficient = 0.02

G= Weight of belt +weight for idler in the upper and lower strand per meter of belt conveyors (kg/m) = 10 kg/m

V = belt conveyors speed = 1.5 m/s

Q =conveyed material (ton/h) =5655 ton/hr

H = conveying height (m)=0

Power equipment:

Power requirement for belt conveyors is

 $N = \left(\frac{c \times f \times l}{210}\right) \times (3.6G \times V + Q) + \frac{Q \times H}{270} \text{ (HP)} \dots \dots (A)$

Or, N=91.8 HP=91.8×746=68482.1 Kw

[https://www.911metallurgist.com/blog/sizing_conveyor_belts]

7.2 Design of Crushers

Power requirement for **Jaw crusher (C120):** Mass flow rate of lime stone, m=223.83 ton/hr Work index of Limestone. W_i=12.74 Diameter of the feed, D_{pa}=100 mm Diameter of the feed, D_{pb}=1000 mm According to bond's law, $\frac{P}{m}$ =0.3162 W_i ($\frac{1}{\sqrt{Dpb}} - \frac{1}{\sqrt{Dpa}}$) Or, P=1.95 KW Power requirement for **Cone crusher (C121):**

Mass flow rate of lime stone, m=223.83 ton/hr

Work index of Limestone. W_i=12.74

Diameter of the feed, Dpa=25 mm

Diameter of the feed, Dpb=100 mm

According to bond's law, $\frac{P}{m} = 0.3162 \text{ W}_{i} \left(\frac{1}{\sqrt{Dpb}} - \frac{1}{\sqrt{Dpa}}\right)$

Or, P=2.85 KW

7.3 Sizing of ESP (H241, H244)

Pollutants in Exhaust gas in Preheater: 11100 mg/m³

Pollutants in Exhaust gas in Kiln: 9600 mg/m³

In ESP if we assume efficiency is 99.995%;

In preheater exhaust gas pollutants will remain: $11100 \times .005 = 55.5 \text{ mg/m}^3$

In Kiln exhaust gas pollutants will remain: $9600 \times .005 = 48 \text{ mg/m}^3$

Let drift velocity of particles 0.1 m/s

$$\eta = 1 - \exp(\frac{-V_{pm}A_c}{Q_g})$$

["Design of ESP in a Cement Plant", available at: www.scribd.com/documents/10405462]

Q_g, preheater = $4 \times 10^6 \text{ m}^3/\text{day} = 1111 \text{ m}^3/\text{s}$

Q_g, kiln= 5000 m³/s

A_c, preheater=110027 m²

Assuming plate type ESP use in a cement plant for removing dust particles consist of 10 equal channels. the plates are 3 m high and 3 m long.

Number of plates needed = 110027/180=612 (H241)

Ac, kiln = 495174

Assuming the same configuration as above,

Number of plates needed for kiln=495174/180 = 2750 (H244)

7.4 Sizing of mixer (F122)

Limestone 85%

Clay 15%

 $volume = \frac{\frac{m_1}{p_1} + \frac{m_2}{p_2}}{2}$

 $Volume = \frac{\frac{0.85 \times 5372 \times 1000}{24 \times 2160} + \frac{0.15 \times 948 \times 1000}{24 \times 22650}}{2} = 45.12 \text{ m}^3$

(Data Collected from Perry's Chemical Engineer's Handbook)

7.5 Sizing of Storage

Storage volume (m³)=mass/density

F $331 = \frac{1515 \times 1000}{1415} = 1070.67 \text{ m}^3 \text{ (for clinker)}$ F $332 = \frac{1515 \times 1000}{3150} = 480.95 \text{ m}^3 \text{ (for OPC)}$ F $333 = \frac{1515 \times 1000}{2400} = 631.25 \text{ m}^3 \text{ (for PCC)}$

(Data Collected from Perry's Chemical Engineer's Handbook)

7.6 Sizing of air blower

Capacity:

G 331 :2×10⁶ m³/day

G 332: 5.36×10⁶ m³/day

Chapter 8

Individual Equipment Design

Detailed Design of a Ball mill (C210)

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8.1.1 BALL MILL DESIGN

It is a horizontal cylinder partly filled with steel balls that rotates on its axis, imparting a tumbling and cascading action to the balls. Materials feed through the mill is crushed by impact and ground by attrition between the balls. It is a two-compartment mill. The first chamber consists of larger balls to crush coarse materials and the second chamber consists of small balls for fine grinding. There are separated by a diaphragm lined with lifter plates.



Figure 8.1.1: Schematic Diagram of Ball Mill



Figure 8.1.2: Open Circuit Ball Mill

Loss in grinding (126 ton/day)



Figure 8.1.3: Material Balance Around Raw Mill

8.1.2 Basic considerations:

Holding the mill length constant, while increasing diameter has

- ✤ Higher power efficiency
- ✤ Less floor space per unit capacity
- ✤ Fewer submicron particles in the mill product
- ✤ More oversize tramp particles
- ✤ Lower steel wear rates per ton of product

Holding the mill length constant, the greater mill diameter provides;

- Lower capital costs per installed horsepower
- ✤ Fewer oversize tramp particles
- ✤ More micron fines in mill product

- ✤ Lower power efficiency
- ✤ Opportunity for partitioning the mill cylinder

8.1.3 Material of construction

The shell is welded structure and manufactured from steel sheets, or from fine-grained structure steel. Boiler plates are also frequently in use.

8.1.4 Mill bearings

Sliding bearings

The resistance generated by friction in the roller bearing is the same as during the normal mill run, this causes a significant reduction in breakaway torque. When sizing the bearing bushes it should be considered that the bearing pressure should not exceed the following limits, for white metal bearing metal (15-20kg/cm2) and for bronze (25-28kg/cm2)

Mill slide shoe bearing

Bearings used are mostly hydrodynamic bearings. Bearings with slide shoes or riding rings are used for mills requiring a throughput of large volumes of hot gases thus avoiding a bottleneck for the gas flow
Table 8.1.1: Decision matrix

| | Percentage rating | Mill slide shoe | Sliding bearing |
|----------------------|-------------------|-----------------|-----------------|
| | | bearing | |
| ease of installation | 30 | 25 | 30 |
| Lowering effect on | 25 | 25 | 20 |
| mechanical stress | | | |
| Effect on mill-tube | 25 | 20 | 23 |
| thickness | | | |
| Cost of installation | 20 | 10 | 15 |
| TOTAL MARK | 100 | 80 | 88 |

Therefore, the decision reached is the use of sliding bearing. [Source: *Grinding machineries handbook*, available at: libgen.io/view/items/654331]

8.1.5 Mill shell size:

Mill should have minimal weight, this results out of predetermined length to diameter ratio of the mill, L/D. For two compartment mill (one partition), ratio = 3. Let assume, Diameter= 3 m. So, Useful mill length minus one partition of 0.3 m will be 9 m. Thus, effective length is 9 m.

Length of the first compartment (33.33% of the effective length) = 3 m

Length of the second compartment (66.67% of the effective length) = 6 m.

["Ball mill design", sources: www.scribd.com/documents/40005661]

8.1.6 Mill shell thickness

It ranges from between (1/100) and (1/75) of the mil diameter depending upon the diameter and length of the mill tube. The following shell thicknesses are being used.

Table 8.1.2: Shell thickness

| Mill-tube diameter | Shell thickness |
|--------------------|-----------------|
| Up to 1.6m | 18mm |
| 1.6-2.0 | 20mm |
| 2.0-2.2 | 25.5mm |
| 2.2-2.4 | 28mm |
| 2.5-3.5 | 38mm |
| 3.5-4.25 | 52mm |
| 4.25-4.50 | 58mm |
| 5 | 63.5mm |
| 6.4 | 85/75mm |

Therefore, taking Shell thickness of 38mm (as mill diameter is 3 m)

["Ball mill design", sources: www.scribd.com/documents/40005661]

8.1.7 Critical mill speed

The critical speed of a tube is the speed of rotation at which the centrifugal power neutralizes the force of gravity which influences the grinding balls and at this situation the grinding balls do not fall therefore they will not perform grinding work.



Figure 8.1.4: Relation between Optimal mill speed and ball filling level

Nc= 42.3 / $\sqrt{D_{eff}}$ [British Chemical Engineering vol.6 part II]

Where, D_{eff} – is the effective diameter of the mill (m)

Nc – critical speed of the mill (rpm)

- D = outer diameter liner thickness
- D = 3-2(0.038)
- D = 2.924m

Therefore, Nc = $42.3 / \sqrt{2.924}$

Nc = 24.74 rpm

8.1.8 Mill speed

Mill speed is approximately 70 to 80 % of the critical mill speed. Therefore taking 77%,

| Mill speed | = | 77% of critical mill speed |
|------------|---|----------------------------|
| Mill speed | = | 0.77×24.74 rpm |
| Mill speed | = | 19.05 rpm |

8.1.9 Maximum Ball Size:

MBS (mm) = $36 \times F^{0.5} \times (\frac{(SG \times W_i)}{\% C \times D_{eff}^{0.5}})^{0.333}$

Here, 5372 ton/day limestone & 948 ton/day clay use as feed of the mill.

F= Feed size (sieve size in mm which retains 20% material), mm = 20mm

SG= Specific weight of feed material, gm/cm3 =2.60

Wi = Bond's work index, kWh/t = 10.90

So, SG of the mixture = $\frac{5372 \times 2.7 + 948 \times 2.23}{6320}$ = 2.6 [British Chemical Engineering vol.6 part II] So, Wi of the mixture = $\frac{5372 \times 11.61 + 948 \times 7}{6320}$ = 10.9 [British Chemical Engineering vol.6 part II] %C = Mill speed as % critical, % =77

 D_{eff} = Mill effective diameter, m =2.924

So, MBS = 97 mm

So, maximum ball size should be 100 mm

For better utilization of the space occupied by the grinding media, grinding balls with

different diameters are used. The first mill compartment, where grinding is performed by impact alone should contain balls of 100 - 60 mm diameter. The second mill compartment, where comminution is performed by impact and friction (mostly by friction) should be charged with balls of 60 - 15 mm diameter.

8.1.10 Calculation of charge:

Let assume, % of filling = 28% (for both compartment) ["Ball mill design", sources: www.scribd.com/documents/40005661]

% filling = $112.5 - \left(125 \times \frac{H}{Deff}\right)$ ["Ball mill calculation", sources: www.scribd.com/documents/30505661]

H = Free height, m

Here, $D_{eff} = 2.924 \text{ m}$

So, H = 2.62 m

h= center distance (from mill center to media top layer), m

D_{eff}= mill effective diameter, m

R=h/deff, ratio of center distance to mill effective diameter

(h/ Deff used in page A. 4)

h = H - Deff/2 = 1.16 mm

 $h/\ D_{eff} = H/\ D_{eff} - \frac{1}{2} = 2.62/2.924 - \frac{1}{2} = 0.395$

| h/D | а | P | h/D | а | P |
|-------|-------|------|-------|-------|------|
| 0.000 | 0.425 | 50.0 | 0.200 | 0.647 | 25.2 |
| 0.005 | 0.430 | 49.4 | 0.205 | 0.653 | 24.6 |
| 0.010 | 0.435 | 48.7 | 0.210 | 0.659 | 24.1 |
| 0.015 | 0.441 | 48.1 | 0.215 | 0.665 | 23.5 |
| 0.020 | 0.446 | 47.5 | 0.220 | 0.670 | 22.9 |
| 0.025 | 0.452 | 46.8 | 0.225 | 0.676 | 22.4 |
| 0.030 | 0.457 | 46.2 | 0.230 | 0.682 | 21.8 |
| 0.035 | 0.462 | 45.6 | 0.235 | 0.688 | 21.2 |
| 0.040 | 0.468 | 44.9 | 0.240 | 0.693 | 20.7 |
| 0.045 | 0.473 | 44.3 | 0.245 | 0.699 | 20.1 |
| 0.050 | 0.479 | 43.6 | 0.250 | 0.705 | 19.6 |
| 0.055 | 0.484 | 43.0 | 0.255 | 0.711 | 19.0 |
| 0.060 | 0.490 | 42.4 | 0.260 | 0.717 | 18.5 |
| 0.065 | 0.495 | 41.8 | 0.265 | 0.722 | 17.9 |
| 0.070 | 0.501 | 41.1 | 0.270 | 0.728 | 17.4 |
| 0.075 | 0.507 | 40.5 | 0.275 | 0.734 | 16.8 |
| 0.080 | 0.512 | 39.9 | 0.280 | 0.740 | 16.3 |
| 0.085 | 0.518 | 39.2 | 0.285 | 0.746 | 15.8 |
| 0.090 | 0.523 | 38.6 | 0.290 | 0.751 | 15.3 |
| 0.095 | 0.529 | 38.0 | 0.295 | 0.757 | 14.8 |
| 0.100 | 0.534 | 37.4 | 0.300 | 0.763 | 14.2 |
| 0.105 | 0.540 | 36.7 | 0.305 | 0.769 | 13.7 |
| 0.110 | 0.546 | 36.1 | 0.310 | 0.775 | 13.2 |
| 0.115 | 0.551 | 35.5 | 0.315 | 0.781 | 12.7 |
| 0.120 | 0.557 | 34.9 | 0.320 | 0.786 | 12.2 |
| 0.125 | 0.562 | 34.3 | 0.325 | 0.792 | 11.8 |
| 0.130 | 0.568 | 33.6 | 0.330 | 0.798 | 11.3 |
| 0.135 | 0.574 | 33.0 | 0.335 | 0.804 | 10.8 |
| 0.140 | 0.579 | 32.4 | 0.340 | 0.810 | 10.3 |
| 0.145 | 0.585 | 31.8 | 0.345 | 0.816 | 9.87 |
| 0.150 | 0.591 | 31.2 | 0.350 | 0.822 | 9.41 |
| 0.155 | 0.596 | 30.6 | 0.355 | 0.828 | 8.95 |
| 0.160 | 0.602 | 30.0 | 0.360 | 0.833 | 8.51 |
| 0.165 | 0.608 | 29.4 | 0.365 | 0.839 | 8.07 |
| 0.170 | 0.613 | 28.8 | 0.370 | 0.845 | 7.64 |
| 0.175 | 0.619 | 28.2 | 0.375 | 0.851 | 7.22 |
| 0.180 | 0.625 | 27.6 | 0.380 | 0.857 | 6.80 |
| 0.185 | 0.630 | 27.0 | 0.385 | 0.863 | 6.39 |
| 0,190 | 0.636 | 26.4 | 0.390 | 0.869 | 5.99 |
| 0 105 | 0.642 | 25.8 | 0.395 | 0.875 | 5 59 |

Table 8.1.3: Data for calculating a & q from h/ D_{eff}

a can also be calculated as:

 $a = 0.009 \cdot (96.7 - q)$

I - I

| Material | Grinding | Lining | Internal fittings | Med Type | Media Type W | | |
|--------------|-------------------------|--------|----------------------|-------------|-----------------|------|--|
| | C | | None | Balls | 4.3 | 0.73 | |
| | Coarse | | None | Rods | 6.0 | 0.55 | |
| | and | Steet | Danula | Balls | 4.3 | 0.75 | |
| | medium | | Sonex | Bails | 4.3 | 0.66 | |
| Cement | | | None | Bails | 4.5 | 0.69 | |
| and | | | None | Cylpebs | 4.7 | 0.73 | |
| aw meal | Teo. | 0 | None | Minipebs | 4.7 | 0.64 | |
| (dry) | Fine | Steel | Danula | Bails | 4.5 | 0.71 | |
| | | | Danula | Cylpebs | 4.7 | 0.75 | |
| | | | Danula | Minipebs | 4.7 | 0.66 | |
| | AIC | Steel | None | Ceramic | 1.9 | 0.75 | |
| | | Silex | None | Stone | 1.5 | 0.75 | |
| | Coarse and medium | Steel | None | Balls | 4.3 | 0.66 | |
| | | | None | Rods | 6.0 | 0.50 | |
| | | | Danula | Bails | 4.3 | 0.67 | |
| | | | Sonex | Bails | 4.3 | 0.59 | |
| Slurry | | Steel | None | Bails | 4.5 | 0.66 | |
| (wet) | - | | None | Cylpebs | 4.7 | 0.66 | |
| | mine | | Danula | Balls | 4.5 | 0.67 | |
| | | | Danula | Cylpebs | 4.7 | 0.67 | |
| Ì | | 1 0 | None | Ceramic | 1.9 | 0.85 | |
| | A.F. | Steel | None | Stone | 1.5 | 0.85 | |
| Wash drum | All | Silex | Lifters | Stone | 1.5 | 0.85 | |
| | Coarse | | None | Balls | 4.3 | 0.69 | |
| Coal | Medium | Charl | None | Balls | 4.5 | 0.69 | |
| | Medium | 21661 | None | Cylpebs | 4.7 | 0.69 | |
| | Medium | | Danula | Cylpebs | 4,7 | 0.71 | |

Sources:[https://www.911metallurgist.com/blog/ball-mill-calculator https://www.911metallurgist.com/blog/grinding-ball-diameter-calculation] q (%) = Specific charge

- w (t/m^3) = Bulk weight
- form Table 8.1.3, a = 0.875
- from Table 8.1.3, q = 5.59
- from Table 8.1.4, w= 4.3, $\mu = 0.73$
- F = Charge
 - $F(12) = q/100 \times w \times V$
- $V(l^3) = Volume of compartment = Volume of compartment 1 + volume of compartment 2$

$$= \pi/4 \times D_{eff} \times L1 + \pi/4 \times D_{eff} \times L2$$

- L1 = 3 m
- L2 = 6 m
- So, V = 60.43 m3
- So, F = 14.52

8.1.11 Compartment power consumption:

- a = Arm of gravity in relation to mill diameter
- n (rpm) = Rotational speed of the mill
- $g(m/s^2) = Acceleration of gravity$
- α (°) = Angle of displacement
- N (KW), net = Power consumption by compartment at mill shell

N = F× $g \times D \times sin\alpha \times \pi \times \frac{n}{60}$ [https://www.911metallurgist.com/blog/ball-mill-calculator]

Since is the torque factor μ . Standard values from Table 8.1.4

N =0.514× F × n × μ × D × a

Where the constant 0.514 is $(g \times \pi/60)$

So, N = 264.8 KW

8.1.12 Inlet chute design

In designing the inlet chute the following aspects were put into consideration

- ✤ Type of material handled by the mill
- ✤ Moisture content during operation
- ✤ Abrasiveness factor
- Chute thickness
- ✤ Material of construction

Material of construction

ASTM A 36 material is used of thickness 5 mm and the wear plate is Hardox material which protects the chute from the abrasive material.

Inlet chute area calculation

It is calculated on the basis of velocity 1 m/s and chute being 25% for powdery material and 50% for lumpy material.

$$A = \frac{Q \times 100}{3600 \times V \times F}$$

Where,

- Q- Volume flow rate in m³/hr
- V Velocity in m/s
- F Filling level %, here 28%

For Limestone

Mass flow rate of limestone = $5372 \text{ ton/day} = 223 \times 10^3 \text{ kg/hr}$

$$Q = \frac{\text{mass flow rate}}{\text{density}}$$
$$Q = \frac{223 \times 10^3}{\text{density}}$$

$$Q - \frac{1}{2.6 \times 10^3}$$

 $Q = 85.77 \text{ m}^3/\text{hr}$

Minimum area for clinker

A =
$$\frac{85.77 \times 100}{3600 \times 1 \times 28}$$

A =
$$.09 \text{ m2}$$

For Clay

Mass flow rate of clay = 948 ton/day = 39.5×10^3 kg/hr

$$Q = \frac{\text{mass flow rate}}{\text{density}}$$
$$Q = \frac{39.5 \times 10^3}{2.23 \times 10^3}$$

 $Q = 17.7 \text{ m}^3/\text{hr}$

Minimum area for clinker

A =
$$\frac{17.7 \times 100}{3600 \times 1 \times 28}$$

Total minimum chute Area

Total minimum chute Area = Area of limestone + Area of Clay

Total minimum chute Area = 0.11 m^2

8.1.13 Hot air requirement:

Assuming 28% media filling for air circulation.

Free area of air flow = $0.72 \times \pi/4 \times D_{eff}$

 $=4.83 \text{ m}^2$

Air velocity through the mill = 3 m/s at NTP

Air flow rate = $3 \times 4.83 \times 60 = 870 \text{ m}^3/\text{min}$

Providing 10% extra than capacity = $1.1 \times 870 = 957 \text{ m}^3/\text{min}$

At 100 °C = 957 × (273+100) / 273 = 1309 m³/min = 78476 m³/hr

8.1.14 Bag Filter Design:

Flow gas rate = $78476 \text{ m}^3/\text{hour}$

Temperature = $100 \text{ }^{\circ}\text{C}$

The filtering surface (SF) given in m^2 is estimated using the following expression:

$$SF = 0.26 \times F \times c^{0.18} + 350 \times \phi^{-0.5}$$

 $F = gas flow at 25 \ ^{\circ}C given in m^{3}/min$

 $c = concentration of dust given in g/m^3$

 ϕ = average diameter of the particle, in μ m

In case the values of "c" and " ϕ " are unknown, an appropriate expression is:

SF \approx .014 ×Q , Q must be given in Nm³/h

["Bag House Filter", available at: www.scribd.com/45006735]]

Q at NTP (25 °C) = $78476 \times (273+25) / (273+100) = 62700 \text{ Nm}^3/\text{h}$

So, total filtering area = $0.014 \times Q = 877.8 \text{ m}^2$

Diameter of the bags, D = 0.15 meter [assumption] [http://fluegasknowhow.com/bag-filters/bag-filter-design-calculator/]

Length of the bags, L = 3.5 meter [assumption]

Area of a bag = $\pi \times D \times L + \pi \times D^2/4 = 1.67 \text{ m}2$

No. of bag required = 877.8/1.67 = 527

Distribution:

Number of rows in the bag house = 10

Number of columns in the bag house = 53



Figure 8.1.5: Bag House Filter



Figure 8.1.6: Sectional view of a ball mill



Figure 8.1.7: Cross Sectional view of a ball mill



Figure 8.1.8: Cross sectional view of a ball mill during rotation



Figure 8.1.9: P&ID of Ball Mill

Table 8.1.5: Ball mil design parameters

| Mill size | 3m x 9m |
|------------------------------|---------------------------|
| Mill diameter | 3m |
| Mill shell thickness | 38mm |
| Critical mill speed | 24.74 rpm |
| Mill speed | 19.05 rpm |
| Mill surface | 60.43m ² |
| Minimum inlet chute area | 0.11 m ² |
| Air flow opening area | 0.380 m ² |
| Mill energy requirement | 265 Kw |
| Clinker entering temperature | 58.4 ⁰ C |
| Cement leaving temperature | 100.86 ⁰ C |
| Maximum Diameter of ball | 100mm |
| Type of material for balls | Stainless steel |
| Lining material | Abrasion resistant rubber |

| | | | | BALL MILL | | | Specification Number | | | Α | | | |
|---------------|----------------|-----------------|-----------------|-------------------------------|-------------------------|-----------|----------------------|------------------------|------------------|--------------|---------|----|---|
| | | | | Process Data | | | | | Ball-efg | -222 | | | |
| | | | | Sheet | | | | | Sheet No | : | 1 of | 1 | |
| Area: | Sylhet | | | Client : | | ont P | lant | | Operating Centre | e: | Jakarta | | |
| No. | 1 | Working | 1 | Service: | Ba | all Mill | lane | | Item No: | | | 1 | |
| of: | | Standby | 0 | | | | | | BM-M | IL-001 | | 2 | |
| Manufactu | rer: | Ν/Λ | | Model Type | and Number: | | Ν/Λ | | | | | 3 | |
| | | | | 0 | PERATING CONDITI | ONS | | | | | | 4 | |
| MATERIAL | HANDLED: | | Crushed Lime | estone & Clay | | | | | | | | 5 | |
| Hazard Co | nsiderations: | | Yes | No | Type: NI// | ٩ | | | | | | 6 | |
| DESIGN C | ONDITIONS | | | FE | ED | - | | | PRODUCT | | | 7 | |
| | | | | | F ₁₀₀ = 40mm | | | | | | | 8 | |
| | | | | | $F_{m} = 20 \text{mm}$ | | | P ₈₀ = 90µm | | | | 9 | |
| | SIZE | IS | | | 1 80 - 2000 | | | - au - cop | | | | 10 | |
| | | | | | | | | | | | | 11 | |
| Feed Desc | ription: | Crushed Lime | stone & Clav | | | | | Bulk | Density (t/m3): | 16 | | 12 | |
| Design Nev | w Feed (dtph): | 76 | | | | | | Buikt | Material SG: | 2.60 | | 12 | |
| Ore Moistu | re (wt%): | - 5 | | | | | | | Material 50. | 2,00 | | 14 | |
| Pobblo roc | volo (tob): | _ 0 | | | | | | | | | | 15 | |
| Crindobility | (Rond work in | | 10.0 | k\A/b/t | | | | | | | | 10 | |
| Grindability | | Jex). | 10.9 | KVVII/L | | | | | | | | 10 | |
| Type of Cir | cuit: | Wet | Dry | Open | Closed | % F | ecirculation | (u/f:mill feed): | 250 | (design max) | | 17 | |
| Duta: | cuit. | Continuoun | Diy | Botoh | Closed | 0/ E | | (u/f:mill food): | 200 | (design max) | | 10 | |
| Duty. | | Continuous | | Datch | | 70 F | | | 750/ | (average) | | 19 | |
| Puip pH: | as received | | | | | Ope | rating Pulp d | iensity (wt%): | 75% | | | 20 | |
| 01 | 0 | Qualance | | Nie andre al | A | • | - | Quere l'a el la se | | | | 21 | |
| Classificatio | on System: | Cyclone | | Nominal | Aperture (mm): | | | Supplied by: | vendor | | | 22 | |
| Method of a | Discharge: | | . An all shares | | | | | | | | | 23 | |
| | Discharge to: | pump nopper | via cnute | | | | | | | | | 24 | |
| 0 1/5 | | | | | | | | | _ | | | 25 | |
| Speed (Fix | ed, variable): | | fixed | | Rotation direc | tion (vie | wed from dis | scharge end): | | | | 26 | ļ |
| Speed (RP | M): | | 19 | | % Critical speed | (based | on inside-sh | ell diameter): | 11% | | | 27 | |
| Max Charg | e (% vol): | | 40% | | | | Operating Ch | arge (% vol): | 40% | | | 28 | ļ |
| Max Ball C | harge (% vol): | | 35% | | | Ope | rating Ball Cl | narge (% vol) | 35% | | | 29 | ļ |
| Load Cell: | | Yes > | No | | | | | | _ | | | 30 | ļ |
| Location: | | Indoor | Outdoor | | | | Climat | tic conditions: | G | | | 31 | |
| | | | | | MECHANICAL DAT | A | | | | | | 32 | |
| GRINDING | MEDIA: | Type: | Balls | s Material: Steel Hardness: * | | | * | | 33 | | | | |
| | Ch | arge (tonnes): | * | | Size Ma | ix(mm): | 100 | | | | | 34 | |
| | | Supplied by: | * | | | | | | | | | 35 | |
| s | Nomi | nal Length (mm) | : 9000 | | Diameter, inside shel | ll (mm): | 3000 | | | | | 36 | |
| Ĥ | | Material | l: * | | Thickness | s (mm): | * | | | | | 37 | |
| E L | MANHOLES: | No |): * | | Size | e (mm): | * | | | | | 38 | |
| L | LINERS: | Туре | e: Rubber | | Materia | al/Spec: | * | | Hardness: | | | 39 | |

| | Nominal Thickness (mm) | : 100 | | | Attachment: | | | | 40 |
|-------------|------------------------|-----------------|--------|----|----------------|--------------|-----|-----------------------|----|
| | Backing Line | r: | Yes | No | Туре: | - | | | 41 |
| | | | 11 | | | | | | 42 |
| | HEAD: | | | | | | | | 43 |
| | Туре: | * | | | Material/Spec: | * | | | 44 |
| | Thickness (mm): | * | | | Attachment: | N/A | | | 45 |
| | ENDPLATE LINER: | | | | | | | | 46 |
| | Type: | * | | | Material/Spec: | * | | Hardness: N/A | 47 |
| | Thickness (mm): | * | | | Attachment: | N/A | | | 48 |
| | Backing Liner: | | Yes | No | Туре: | N/A | | | 49 |
| | SLEEVE LINER: | | | | | | | | 50 |
| Т | Type: | N/A | | | Material/Spec: | N/A | | Hardness: N/A | 51 |
| U | Thickness (mm): | N/A | | | Attachment: | N/A | | | 52 |
| N | BEARINGS: | | | | | | | | 53 |
| I | Type: | N/A | | | Material: | N/A | | ^{Seals:} N/A | 54 |
| O | Diameter (mm): | N/A | | | | | | | 55 |
| S | Pressure (t/m2 pr | ojected area): | N/A | | | | | | 56 |
| | Trunnion S | Surface Finish: | N/A | | | | | | 57 |
| | | | | | | | | | 58 |
| | Temperature Probes: | | Yes | No | | | | | 59 |
| | Manufacturer: | N/A | | | | Туре | N/A | | 60 |
| | Vibration Probes: | | Yes | No | | | | | 61 |
| | Manufacturer: | N/A | | | | Туре | N/A | | 62 |
| | | | | | | | | | 63 |
| | Cooling Water: | | Yes | No | Flo | wrate (m3/h) | : * | | 64 |
| Rev No | | | A | | В | | | | |
| Prepared By | / Date | MS 2 | 9/6/18 | 1 | 8/04/2019 | | | | |
| Checked By | / Date | | | | | | | | |
| Authorised | By / Date | | | | | | | | |

Source: BAICHY machinery limited (Wulong Industrial Park, Zhengzhou, China)

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Detailed Design

Of

ROTARY KILN (R-240)



Prepared by Md. Shehabul Hasan Mamun Student ID: 1402040

8.2. ROTARY KILN

Rotary kiln is the largest single equipment in cement industry in fact in all process industries.

Rotary cement kiln; a cylinder shaped vessel slanted somewhat to the level which is pivoted gradually about its axis. The material (raw mill) to be processed is bolstered into the upper end and may experience a specific measure of blending. Hot gases go along the oven inverse way to the procedure material (counter current). Hot gases are created by a flame inside the furnace.

8.2.1. The design considerations of a rotary kiln are:

- 1. Kiln slope
- 2. Kiln length
- 3. Kiln diameter
- 4. Heat load
- 5. Residence time
- 6. Volume loading
- 7. Material of construction
- 8. Refractory lining material.

Sizing of kilns has been largely empirical.

Therefore sizing and other design calculations are based on empirical value and references of the calculations are mentioned where necessary.

8.2.2. Kiln specifications:

- a. Capacity = 4045 ton per day
- b. Raw feed = 6288 ton per day
- c. CO_2 removed = 39.4×10^6 mole per day
- d. Coal required = 659 ton per day
- e. Air inlet = 5.36×10^5 m³/day

8.2.3. Kiln length

Retention time (in minutes) =
$$\frac{1.77 \times \sqrt{Q \times L \times F}}{S \times D \times n}$$

Where, Q = Angle of repose (usually 35⁰)

L = Length of the kiln (m)

F = Multiplying factor (1)

$$T = \frac{1.77 \times \sqrt{35 \times L \times 1}}{2.29 \times D \times n}$$

Using retention time 27 minutes

$$L = \frac{2.29 \times 4.5 \times 2.5}{1.77 \times \sqrt{35 \times 1}}$$

 $L_{=}66.01m$

8.2.3. Kiln diameter

The following rotary kiln slopes were found in cement plant operation handbook. [2][5] 5% slope for kilns with diameters up to 2.8m 4% slope for kilns with diameter from 3-3.4m 3%slope for kilns with diameter above 3.4m Kiln diameter = 4.5m

8.2.4. Kiln slope

It is generally 2° or 3.5 %. For calciner kiln up to 4% at a reduced speed. To promote longitudinal travel of the solid the rotary kiln is mounted on a slope of 4%.[2]

Slope = tan (Kiln slope) Kiln slope = arc tan (slope) Kiln slope = arc tan (0.04) Kiln slope = 2.3 degree

8.2.5. Retention time

Retention time (in minutes) = $\frac{1.77 \times \sqrt{Q \times L \times F}}{S \times D \times n}$ Where, Q = Angle of repose (usually 35⁰)

L = Length of the kiln (m) F = Multiplying factor (1) S = Slope of kiln (degrees) D = Diameter (m) N = Speed (rpm) T = $\frac{1.77 \times \sqrt{35 \times 66 \times 1}}{2.29 \times 4.5 \times 2.5}$ =27.01 minutes

8.2.6. Percent filling

To keep the difference between temperature at surfaces of material and its core, to a minimum it is desirable to keep degree of filling between 4-6 %. [1]

Percentage filling (%) = $\frac{1.667 \times T \times Q \times a}{Volume \ of \ kiln \times b}$ T = retention time in minutes Q = Capacity of clinker in tph a = Ratio of raw meal feed/ product =1.5545 $b = for clinker 1.2 t/m^{3}$ Percentage filling (%) = $\frac{1.667 \times T \times Q \times a}{Volume of kiln \times 1.2}$ $= \frac{1.667 \times 27 \times 168 \times 1.5}{Volume of kiln \times 1.5}$

$$=\frac{1.667\times27\times168\times1.5}{1049.7\times1.2}$$
$$= 5.83\%$$

8.2.7. Peripheral speed

To keep same or about same degree of filling, retention time is reduced by increasing kiln speed. A kiln that was earlier rotating at 1.67 rpm with a peripheral speed of 35 cm/s will now be rotating at 4.2 rpm or a peripheral speed of 87.5 cm/sec. [1]

Kiln speed = 2.5 rpm

=58.9 cm/s

8.2.8. Thickness of the kiln shell

| Operating pressure = 1 atm |
|----------------------------|
|----------------------------|

| Operating temperature | = 350 degrees |
|-----------------------|----------------|
| Diameter of the kiln | = 4.5 meters |

Design pressure at 5% above operating temperature

| Design pressure | $\frac{105}{=100}$ | \times 1.013 $\times 10^{5}$ N/m ² |
|-----------------|--------------------|---|
| Design pressure | = | $1.064 \times 10^5 \text{N/m}^2$ |

Therefore the design stress for mild steel at is 841 MPa

| | design pressure×diameter | |
|-------------------|--|--|
| Minimum thickness | \pm 2(design stress)–design pressure | |
| Thickness | = 284mm | |

8.2.9. Kiln shell material

This is made from rolled mild steel plate welded to form a cylinder. Mild steel is used because of its good thermal conductivity, resistant corrosion and can withstand high stresses.[3]

8.2.10. Refractory lining

Kiln is a cylinder. Refractory lining is 150, 200 and 250 mm thick according diameter of the kiln. Quantity of bricks can be roughly estimated

D = diameter inside shell d = diameter inside refractory L = length of kiln Volume of bricks = $0.785 (D^2 - d^2) L m^3$ If bulk density is 1.9 ton/m³, Weight of bricks = $1.5(D2 - d^2) L$ tons. = 295 tons

This is also true of all round vessels like calciners, ducts like T.A. ducts.

Ready reckoner tables or graphs can be built up to estimate weight of refractory for kilns of different diameter.

| Sr no | Specification | Hysil blocks | Fireclay Brick 40 % alumina | High alumuna 70 % alumina | Castable Firecrete special | Castable White heat 'A' | Castable White heat 'C' |
|-------|--|---------------------------------|-----------------------------------|------------------------------------|----------------------------------|----------------------------------|----------------------------------|
| 1. | Location | Preheater Cooler calciner | Preheater kiln canciner cooler | Kiln burning zone | Preheater Cooler calciner | kiln | Preheaeter kiln |
| 2. | Service temp. max °C | 950 | 1400 | 1600 | 1350 | 1750 | 1500 |
| 3. | Prefractoriness °C min | | 1700 | 1820 | 1450 | 1800 | 1650 |
| 4. | Bulk density Min. kg/m ³ | 260 | 2300 | 2450 | 2250 | 2800 | 2100 |
| 5. | Linear change Max % | 2 | ≃ 0.8 at 1400 °C for 3 hrs | ≃ 2 at 1500 °C for 2 hrs | ≃ 0.8 at 1350 °C for 2 hrs | ≃ 1.0 at 1550 °C for 3 hrs | ≃ 1.0 at 1550 °C for 3 hrs |
| 6. | Cold crushing strength kg/cm ² at 100 °C | 120 | 300 | 480 | 400 | 550 | 350 |
| 7. | Porosity max % | - | 25 | 23 | - | - | - |
| 8. | Refractriness under load min °C | - | 1400 | 1480 | - | - | - |
| 9. | Chemical analysis Alumina % min | Calcium slicate blocks | 40 | 70 | 45 | 90 | 50 |
| 10. | Fe ₂ O ₃ % max | | 3 | 2.5 | 4 | 0.8 | 1.5 |

Annexure 1 Table 55.1 Properties of refractories commonly used in cement plants.

Figure: Properties of refractories commonly used in cement plant [1]



Figure 8.2.1: Refractory lining inside the kiln shell

Refractory bricks called andalusite based fire bricks are used in lining to insulate the steel shell from the high temperatures inside the rotary kiln, and to protect it from the corrosive properties of the process material

8.2.11. Thermal expansion of the kiln shell

High temperature inside the kiln provides expansion of kiln material to some extents which is given by,

 $\mathbf{A} = \mathbf{\dot{\alpha}} \times \Delta T \times \mathbf{l}$

Where,

A=Thermal expansion of the kiln shell $\dot{\alpha}$ –linear expansion index for steel (0.000012mm/m)

L=Length (m)

 ΔT = Change in temperature (average value)

 $A = 0.000012 \times (1000-25) \times 4.5 \times 1000 \text{mm}$

A = 51.2 mm

8.2.12. Stress Analysis

The stresses that act on the rotary kiln are axial and compressive stresses.

Axial stress =
$$\frac{Di \times p}{4(t-C)}$$

Where;

P=Density of mild steel

Di= Inner diameter

t=Thickness

C= Corrosion allowance

Assuming a corrosion allowance of 3mm,

| Axial stress | 4216×2.8 | | |
|--------------|-------------------------|--|--|
| | 4(.284003) | | |
| Axial stress | $= 518.2 \text{ N/m}^2$ | | |
| | $= 417.3 \text{ N/m}^2$ | | |

Longitudinal stress due to pressure, $S = \frac{Di \times p}{4t}$ Where, P=Design pressure Di= inner diameter t=Thickness $S = \frac{4.2 \times 101325}{4 \times .284}$

Longitudinal stress due to pressure = 3.72×105 N/m2

8.2.13. Thermal loading

Thermal loading, TL= $\frac{P \times H}{A \times 24 \times 1000}$

Where, P= Clinker production (tpd)

H= Specific heat consumption in kiln kcal/kg clinker

A = burning zone cross sectional area m^2

 $TL = \frac{4050 \times 390}{13.9 \times 24000}$

 $= 4.735 \text{ Gcal/hr/m}^2$

8.2.14. Volumetric loading

Volumetric loading, $VL = \frac{P}{A \times L}$

P=Clinker production, tpd

A=Burning zone cross-sectional area, m²

L=Kiln length, m

$$VL = \frac{4050}{13.9 \times 66}$$

= 4.41 tpd/m²

8.2.15. Girth Gear and Pinion

Kiln Data 4.5 m Diameter 66 m length Inclination = 4% Kiln speed = 2.5 rpm **a. Gear** Module = 39 No of Teeth = 148 Material of construction = CS 640 (Normalized Cast Steel) Dimensions = 550 mm (width) **b. Pinion** Module = 39 No of Teeth = 28 Material of construction = 30 Cr Ni Mo V8 (Normalized Cast Steel) Dimensions = 600 mm (width)





Figure 8.2.2: Cross section view of rotary kiln (section A-A')

Figure 8.2.3 : P&ID Diagram for rotary kiln

8.2.15. Rotary cement kiln design specification sheet [5]

| Rotary kiln data required | | | | |
|---------------------------|------------------------------------|--|--|--|
| capacity | 4045 tpd | | | |
| Kiln | 4.55×66m | | | |
| Kiln length | 66m | | | |
| Kiln diameter | 4.5m | | | |
| Kiln slope | 4% | | | |
| Kiln speed | 2.5 rpm | | | |
| Design pressure | $1.064 \times 10^5 \mathrm{N/m^2}$ | | | |
| Operating pressure | 1 atm | | | |
| Operating temperature | 700 -1400 degrees | | | |
| Motor power | 630 kW | | | |
| Gearbox velocity ratio | 34.1 | | | |
| Shell thickness | 284mm | | | |
| Clinker production output | 4045 tons/day | | | |
| Kiln feed | 6288 tons/day | | | |
| Fuel requirements | 659 tons/day | | | |
| Weight of the kiln | 711 ton | | | |
| Material of lining | Refractory brick lining | | | |
| Material of construction | Mild steel | | | |

8.2.16. Design calculation references

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- 2. Lawrence C (2000), "The production of low energy cements", 4th edition, Arnold, London.
- 3. Bogue and Robert Herman (2007), "*The Chemistry of Portland Cement*", 2nd edition; Reinhold Publishing Corporation.
- 4. Lafarge Surma cement industry
- 5. Rotary kiln specification(n.d), available at www.great-wall.co/products/rotary-kiln/Rotary
- 6. Simkom consultants sindhu (I.J.S B tech), cement industry
8.3 Detailed Design of a Grate Cooler (C311)

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COOLER – NOMENCLATURE



COOLER- COMPONENTS

- Fixed inlet
- Grate system
 - Fixed Structural Framework
 - Movable Framework
- Drive Arrangement
- Central Lubrication System
- Hammer Crusher

AIR DISTRIBUTION PLATES

- Fabricated design (MOC-Stainless steel)
- Light weight.
- Built-in Mechanical Flow Regulator (MFR)
- Constant air flow through the clinker bed, independent of particle size distribution & clinker bed height

GRATE PLATE ARRANGEMENT

- Gap between front edges of grate plate to be maintained between 0.8 to 1.8 mm
- In an operating Cooler when the gap exceeds 3 mm adjusting plates are to be welded to the front edge
- High temperature resistant grease to be applied on the grate plate bolts during mounting

FLEXIBLE CONNECTION

- Used to convey air from static structure to movable frame
- Spherical joints are hard-chrome plated maintained between 1 to 1.4 mm lower joint.
- Gap between spherical joints to be maintained between 1 to 1.4 mm.
- Maximum wear usually takes place in the lower joint

Material of Construction:

Plate: Chromium-Vanadium, Side Body: Stainless Steel, Inside Body: Brick lining



Figure 8.3.1: Flow around clinker cooler

| Table 8.3.1: Required amount o | f cooling air per kg of hot clinker |
|--------------------------------|-------------------------------------|
|--------------------------------|-------------------------------------|

| component | Volume (m ³ /hr) | Mass (kg/hr) | Volume (kg/kg ck) | Mass (kg/kg ck) | Temperature (°C) | Heat (Kj/hr) |
|----------------|--------------------------------|-----------------|----------------------|--------------------|---------------------|-----------------|
| Cooling air | 988750 | 1211218.75 | 5.87 | 7.18 | 25 | 0 |
| Hot clinker | 107.35 | 168541.67 | 1 | 1 | 1400 | 231744796 |

Hot Air out



Figure 8.3.2: Cooling process of the Grate cooler

| Column1 | Column2 | Column3 | Column4 | Column5 |
|------------------------|---------|--|-----------|---------|
| Туре | | Grate Cooler (Coolax with fixed inlet) | | |
| Total Surface Area | m2 | 86.8 | | |
| Effective Surface Area | m2 | 85.5 | | |
| Grates | No | Fixed Inlet | Grate 1 | Grate 2 |
| Length | m | 2 | 9.57 | 12.54 |
| Width | m | 3.6 | 3.6 | 3.6 |
| Effective Area | m2 | 7.2 | 34.45 | 45.14 |
| Type of Plates | | CIS MFR | CFG / RFT | RFT |

Table 8.3.2: General design parameters of cooler

Table 8.3.3: General design parameters of Grate plate

| Length | 330 | mm | | No of air slot | in a plate | 3 | |
|-------------|-------|--------|-------|-----------------|-------------|-------|------------------|
| Width | 300 | mm | | Length of air s | slot | 220 | mm |
| | | | | Width of air s | lot | 35 | mm |
| No of row | 'S | 72 | | Total area for | air passage | 19.95 | 8 m ² |
| Plates in a | l row | 12 | | | | | |
| Area of a j | plate | 0.099 | m^2 | | | | |
| Total Area | a | 85.536 | m^2 | | | | |
| | | | | | | | |

Table 8.3.4: Air distribution along clinker cooler

| 4045 T/d |
|---------------------|
| 8750 Nm³/h |
| 7.18 Kg/kg ck |
| 2.85 m ² |
| $2.4 m^{3}/s/m^{2}$ |
| |

Residence Time at Constant Output

Under these circumstances, the bed depth is directly proportional to the grate speed. Knowing the area of the cooler, the bed depth under normal operating conditions for a given kiln output rate, and the density of the clinker, the residence time can then be calculated by, $t = \frac{60 \times a \times h \times y}{R}$

a= area of cooler grate surface (m^2)

h= bed depth(m)

```
y = clinker density(kg/m^3)
```

R=Kiln output (kg/hr)

At constant grate speed, the depth of the clinker bed, $h = \frac{t \times R}{60 \times y}$

In both cases, volume of clinker(m³) in the cooler, $V = \frac{t \times R}{60 \times y}$

Residence Time Calculation:

Residence Time,
$$t = \frac{60 \times a \times h \times y}{R}$$

Here,

Area of cooler grate surface, a =112.9 m^2

Bed depth, h=0.18m

Clinker density, y=1570 kg/m³

Kiln output, R=168541.67 kg/hr

So, t = $\frac{60 \times 112.9 \times 0.18 \times 1570}{168541.67}$ = 11.35 min

Specific load of coolers $=\frac{4045t/d}{112.9 \text{ m2}} = 35 \text{ t/d/m}^2$

Table shows clinker residence times at constant kiln output when Grate speed varies.

COOLER PARAMETERS

Table 8.3.5: Clinker residence times at constant kiln output (Grate speed varies)

| Kiln Output (T/hr) | Bed depth (m) | Clinker weight in the cooler (kg) | Clinker volume in the cooler (m ³) | Residence time (min) |
|-----------------------|------------------|---|--|-------------------------|
| 168.54 | 0.18 | 31891.4 | 20.3 | 11.35 |
| 168.54 | 0.2 | 35434.9 | 22.6 | 12.61 |
| 168.54 | 0.23 | 40750.1 | 26.0 | 14.51 |
| 168.54 | 0.25 | 44293.6 | 28.2 | 15.77 |
| 168.54 | 0.28 | 49608.9 | 31.6 | 17.66 |
| 168.54 | 0.3 | 53152.4 | 33.9 | 18.92 |
| 168.54 | 0.33 | 58467.6 | 37.2 | 20.81 |
| 168.54 | 0.36 | 63782.8 | 40.6 | 22.71 |
| 168.54 | 0.38 | 67326.3 | 42.9 | 23.97 |
| 168.54 | 0.41 | 72641.5 | 46.3 | 25.86 |
| 168.54 | 0.43 | 76185.0 | 48.5 | 27.12 |
| 168.54 | 0.46 | 81500.3 | 51.9 | 29.01 |
| 168.54 | 0.48 | 85043.8 | 54.2 | 30.28 |



Figure 8.3.5: Scheme of the clinker cooler



Figure 8.3.6: Typical view under grate compartment

The typical Fuller grate (the most common) is 1-feet wide and 1.5-feet long. They overlap giving an effective length of 1.1 feet.

Table 8.3.6: Operation parameters of fan

| Reference | 25°C |
|--------------|-----------|
| temperature: | 25 C |
| Altitude: | 246.0 m |
| Atmospheric | 09 41 kDo |
| Pressure: | 90.41 KFa |

| Fan No | m ³ /hr | T(°C) | P (kPa) | kg/h | Compartment grate area/aeration area (m ²) |
|--------------------------|--------------------|-------|------------|---------|--|
| K10 | 29004 | 25 | 11.55 | 35530 | 2.28 |
| K11 | 36185 | 25 | 13.05 | 44327 | 2.84 |
| K12 | 36230 | 25 | 13.07 | 44382 | 3.13 |
| K13 | 66732 | 25 | 4.81 | 81749 | 8.30 |
| K14 | 88438 | 25 | 4.79 | 108338 | 12.40 |
| K15 | 154348 | 25 | 4.26 | 189076 | 16.50 |
| K16 | 143243 | 25 | 5.27 | 175473 | 16.50 |
| K17 | 143243 | 25 | 5.27 | 175473 | 16.50 |
| K18 | 155034 | 25 | 4.96 | 189918 | 16.50 |
| K19 | 136288 | 25 | 4.50 | 166953 | 17.90 |
| Total cooling air: | 988750 | 25 | | 1211219 | 112.9 |

| Fan No | Diameter (mm) | Area (m2) | Covered Area (m ²) | RPM | KW |
|--------|------------------|--------------|--------------------------------|------|-----|
| K10 | 630.0 | 0.31 | 2.75 | 2966 | 90 |
| K11 | 710 | 0.4 | 2.75 | 2965 | 110 |
| K12 | 710 | 0.4 | 2.75 | 2965 | 110 |
| K13 | 900 | 0.64 | 8.30 | 1415 | 90 |
| K14 | 1000 | 0.79 | 12.40 | 1477 | 110 |
| K15 | 1250 | 1.23 | 16.50 | 1444 | 160 |
| K16 | 1250 | 1.23 | 16.50 | 1533 | 160 |
| K17 | 1250 | 1.23 | 16.50 | 1533 | 160 |
| K18 | 1250 | 1.23 | 16.50 | 1540 | 160 |
| K19 | 1250 | 1.23 | 17.90 | 1339 | 200 |

Table 8.3.7: Specification data of fan

| Material / Air Stream | Nominal | | Extreme | |
|-----------------------|---------|-----|---------|-----|
| | Nm3/h | °C | Nm3/h | °C |
| Cooling Air | 988750 | 36 | 998320 | 36 |
| Clinker Outlet | 168 tph | 150 | NA | 200 |

Table 8.3.8: Nominal & Extreme condition for design

Reference

[1] Data from LafargeHolcim Bangladesh Limited

[2] Hans E. Steuch, "clinker cooler design," 2003

[3] D. Touil, H. F Belabed, C. Frances, S. Belaadi, "HEAT EXCHANGE MODELING OF A GRATE CLINKER COOLER AND ENTROPY PRODUCTION ANALYSIS," 2005

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[6] https://www.semanticscholar.org/paper/%22The-originator%22-of-modern-clinker-coolerdesigns%3A-Bentsen-Jespersen/51c1c37dc4be2806e6565cac213fe26ee8fc779b, [Accessed: 10-April-2019].

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8.4 Detailed Design of a Preheater (E220)

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8.4 Preheater



Around Preheater:

Preheater feed = 6194 ton/day

Kiln Feed = 6288 ton/day

Coal Required P = 244.43 ton/day = $\frac{244.43 \text{ ton}}{day} \times \frac{10^6 \text{ g}}{1 \text{ ton}} \times \frac{1 \text{ mol}}{15.5 \text{ g}} = 15.77 \times 10^6 \text{ mol/day}$ Air Inlet (300°C) = 9.52P mol/day = $150.13 \times 10^6 \times \frac{.0821 \times 573}{3.5} \times \frac{1m^3}{1000L} = 2.018 \times 10^6 \text{ m}^3/\text{day}$

= 50000 acfm

| Component | Preheater Feed = 6194 ton/day | | | | |
|--------------------------------|-------------------------------|----------|--|--|--|
| | Amount (ton/day) | Weight % | | | |
| SiO ₂ | 754 | 12.20 | | | |
| Al ₂ O ₃ | 199 | 3.25 | | | |
| CaCO ₃ | 5015 | 80.96 | | | |
| Fe ₂ O ₃ | 85.84 | 1.38 | | | |
| MgCO ₃ | 116.64 | 1.88 | | | |
| Inert | 22.28 | .36 | | | |

Composition of Preheater feed and outlet content:

| Components | ton/day | Weight % |
|--------------------------------|--------------|----------|
| SiO ₂ | 771 | 12.4 |
| Al ₂ O ₃ | 214 | 3.43 |
| CaCO ₃ | 5015 | 80.5 |
| Fe ₂ O ₃ | 109 | 1.8 |
| MgCO ₃ | 117 | 1.9 |
| CaO | 1.77 | .28 |
| MgO | .66 | .11 |
| | Total = 6288 | |

Per day clinker produced = 4045 ton/day

Energy Consumption in Preheater = 95224.88 cal/mol $\times 15.77 \times 10^{6}$ mol/day

 $= 1.15 \times 10^{12} \times \frac{cal}{day} \times \frac{1 \, kcal}{1000 \, cal} \times \frac{1 \, day}{4045 \, ton \, clinker \, produced}$ $= 471.25 \times 10^3 \times \frac{Kcal}{ton \, clinker} = 471.25 \, \text{kcal/kg clinker}$

8.4.1 Cyclone Separator portion:

Exhaust gas: (700°C)

 $\mathring{n}_{O2} = 1.06 \ P = 1.06 \times 15.77 \times 10^6 \ mol/day = 16.71 \times 10^6 \ mol \ O_2/day$

 $= .53 \times 10^6$ kg/day

 $\mathring{n}_{N2} = 7.52P = 7.52 \times 15.77 \times 10^6 \text{ mol/day} = 118.59 \times 10^6 \text{ mol } N_2 \text{ /day}$

 $= 3.32 \times 10^6$ kg/day

 \mathring{n}_{CO2} = .94P = .94 × 15.77 × 10⁶ mol/day = 14.82 × 10⁶ mol CO₂/day

 $= .65 \times 10^6$ kg/day

 $\mathring{n}_{SO2} = .002 \ P = .002 \times 15.77 \times 10^6 \ mol/day = .03 \times 10^6 \ mol \ SO_2/day$

 $= 2.02 \times 10^3$ kg/day

 $\mathring{n}_{H2O} = .054 \ P = .054 \times 15.77 \times 10^6 \ mol/day = .85 \times 10^6 \ mol \ SO_2/day$

 $= 15.32 \times 10^3$ kg/day

Assuming 1% loss as dust, dust = $.01 \times 6194$ ton/day

 $= 60.2 \times 10^3$ kg/day

 $Total \ exhaust = (53 \times 10^6 + 3.32 \times 10^6 + .65 \times 10^6 + 2.02 \times 10^3 + 30.1 \times 10^3 + 15.32 \times 10^3)$

 $=4.55\times10^{6}$ kg/day

Dust concentration (wt.%) = $\frac{60.25 \times 10^3}{4.53 \times 10^6} \times 100\% = .66\%$

Moisture Content = $\frac{15.32 \times 10^3}{4.55 \times 10^6} = .0037 = .37\%$

Exhaust volumetric flow rate:

$$\begin{split} \mathring{n}_{E} &= 150.15 \times 10^{6} \text{ mol/day} \\ \mathring{Q}_{E} &= 150.13 \times 10^{6} \times \frac{.0821 \times 973}{3} \times \frac{1m^{3}}{1000L} = 3.99 \times 10^{6} \text{ m}^{3}/\text{day} \\ &= 3.99 \times 10^{6} \times \frac{m^{3}}{day} \times \frac{1day}{24 \times 3600 \text{ s}} \\ &= 46.27 \times \frac{m^{3}}{sec} \end{split}$$

Assuming inlet velocity V = 25 m/s

| Cyclone Separator Dimension [2] | | | |
|---------------------------------|----------------|-------|--|
| Parameter | Dimension | Ratio | |
| Body diameter | D | D | |
| Length of cylinder | L _b | 2D | |
| Length of cone | Lc | 2D | |
| Inlet height | Н | D/2 | |
| Inlet width | W | D/4 | |
| Gas exit dia | Dc | D/2 | |
| Dust outlet dia | D _d | D/4 | |
| Length of vortex finder | S | 5D/8 | |

Area of the inlet duct of cyclone separator $A = \frac{Q_E}{V} = \frac{46.27}{25} = 1.85 \text{ m}^2$

From conventional ratio Height of the inlet H = D/2

Width of the inlet W = D/4

Where D is the cyclone diameter

Cross section of the inlet duct is rectangular [2]

So, H×W = 1.85 m² $\frac{D}{2} \times \frac{D}{4} = 1.85$, so D = 14.8 m

Now calculating other parameters: (according to conventional dimension)

Cut Diameter calculation:

$$D_{cut} = \left(\frac{9W\mu}{2\pi NV\rho_{particle}}\right)^{\frac{1}{2}}; [Diameter at which efficiency = 50\%]$$

Here, $\mu = viscosity$ of exhaust = 1.83×10^{-5} kg/m. s

$$N = no.$$
 of effective turns

V = velocity of the inlet = 25 m/s

 $\rho_{particle} = 1500 \text{ kg/m}^3$

Number of Effective turns: $N_e = \frac{1}{H} \left[L_b + \frac{L_c}{2} \right]$

 L_b = Length of the body = 2D = 29.6 m

 $L_c = Length of the cone = 2D = 29.6 m$

So, Ne =
$$\frac{1}{7.4} \left[29.6 + \frac{29.6}{2} \right]$$

 $N_e = 6$ times

Now, Cut diameter
$$D_{cut} = \left(\frac{9 \times 3.72 \times 1.83 \times 10^{-5}}{2\pi \times 6 \times 25 \times 1500}\right)^{\frac{1}{2}}$$

= 2.08 × 10⁻⁵ m = 20.8 µm

Fractional Efficiency
$$\eta = \frac{(\frac{Dp}{D_{cut}})^2}{1 + (\frac{Dp}{D_{cut}})^2}$$
 [5]

Efficiency vs Particle dia:

| D _p (µm) | Efficiency (ŋ) % |
|---------------------|------------------|
| 5 | 5.46 |
| 10 | 18.77 |
| 15 | 34.21 |
| 20 | 48.03 |
| 25 | 59.09 |
| 30 | 67.53 |
| 35 | 73.9 |
| 40 | 78.71 |
| 45 | 82.39 |
| 50 | 85.24 |



Figure 8.4.1 : Cyclone efficiency vs average particle dia

Decision: As other secondary separators are used along with cyclone separator efficiency of about 85% for 50 μ m is satisfactory. So, Conventional cyclone can be selected [1]

Gas Residence time (Δt) calculation:

$$\Delta t = \frac{\pi DN}{V} [3]$$
$$= \frac{\pi \times 14.8 \times 6}{25}$$

= 11.15 Sec

Particle Drift Velocity (Vp) calculation:

 $V_p = \frac{W}{\Delta t} = \frac{1.86}{5.61} = .17 \text{ m/sec } [3]$

Terminal Settling Velocity (Vt):

For a particle of $40 \ \mu m$ dia

$$Vt = \frac{(\rho_{part} - \rho_{air})D_p^2 V^2}{9\mu D}$$
$$= \frac{1500 \times (40 \times 10^{-6})^2 \times 25^2}{9 \times 1.83 \times 10^{-5} \times 14.8}$$

= .62 m/s [3]

Pressure drop (ΔP) :

$$\Delta P = \frac{\alpha \rho V^2}{2} \left[4 \right]$$

Here, $\alpha = \frac{KHW}{D_e^2}$ [where K is a proportionality constant ranging between 13-15]

[D_e is the gas exit diameter]

$$= \frac{14 \times 7.2 \times 3.6}{7.2^2} = 7$$
So, $\Delta P = \frac{7 \times 0.94 \times 25^2}{2}$

Power requirement (W):

W=
$$\mathring{Q}_{E} \times \Delta P$$

=46.27 × 2056
= 96 × 10³ W
= 96 kW ~ 100 KW [1]

Outlet gas velocity (V₀):

$$V_{o} = \frac{Q_{E}}{\pi r_{e}^{2}}$$
$$= \frac{46.27}{\pi \times 3.6^{2}}$$

= 1.14 m/sec

Dimensions and parameter of the cyclone selected with 10% overdesign:

So, actual dia = $D \times (1+.1) = 14.8 \times (1+.1) = 16.28$ m

| Cyclone Separator Dimension [2] | | | |
|---------------------------------|----------------|-----------|--|
| Parameter | Dimension | Value (m) | |
| Body diameter | D | 16.28 | |
| Length of cylinder | L _b | 32.56 | |
| Length of cone | Lc | 32.56 | |
| Inlet height | Н | 8.14 | |
| Inlet width | W | 4.12 | |
| Gas exit dia | D _c | 8.14 | |
| Dust outlet dia | D _d | 4.12 | |
| Length of vortex finder | S | 10.18 | |

Area = $\pi L_b D + \pi \times \frac{D}{2} \times \sqrt{(R^2 + L_c^2)} = 2523 \text{ m}^2$

Mild Steel was selected as material of construction [1]

Purchase Cost = $2523 \times 150 = 378.5 \times 10^3$ \$

Cost of Stand = 18900\$ (5%) Cost of Blower = 37800\$ (10%) Cost of construction expenses=94625\$ (25%) Cost of Maintenance = 37800\$ (10%) Total Cost = 567625 \$[4]

Typical Specification Sheet for Cyclone separator Process [9]

| 1. Company name: | | | | |
|--|--|--|--|--|
| Address: | | | | |
| City: State : | | | | |
| ZIP code : | | | | |
| Contact Person: Title: | | | | |
| ii. Gas conditions at cyclone inlet (Air): | | | | |
| Volume: <u>50000</u> (acfm) Pressure : <u>50.75</u> (psig) | | | | |
| Temperature: <u>100(</u> °C) | | | | |
| Moisture Content: 0.37% (by weight) | | | | |
| iii. Dust Condition at cyclone inlet: | | | | |
| Dust description: Grinded Limestone, Raw mix | | | | |
| Specific gravity: $\underline{1.5}$ Bulk density: $\underline{1500}$ kg/m ³ | | | | |
| Is dust corrosive? <u>Yes</u> Abrasive? <u>Yes</u> Sticky? <u>No</u> Toxic? <u>Yes</u> | | | | |
| Dust load at inlet: <u>2500 kg/hr</u> . | | | | |
| Dust Concentration: 0.66 % (by weight) | | | | |
| iv. Cyclone | | | | |
| Operating Pressure: <u>31000 mm</u> H ₂ O | | | | |
| Construction Materials: Mild Steel | | | | |
| Is ASME code construction required? Yes | | | | |
| V. Auxiliaries | | | | |
| Cyclone stand: Clean air outlet: <u>4.12m dia</u> | | | | |
| Dust hopper: <u>16.8 m with valve</u> | | | | |
| Desired collection efficiency: 80% | | | | |

8.4.2 Cyclone Preheater



Figure 8.4.2: Cyclone Preheater

Spray cooling of gas:

 $M_{\rm w} \times (C_{\rm p})_{\rm w} \times (T_{\rm E} - T_{\rm w}) = Q_{\rm E} \times \rho_a \times (C_{\rm p})_{\rm a} \times \Delta T$ [6]

- M_w = Amount of water required for cooling exhaust (kg/h)
- $(C_p)_w$ = Specific heat of water = 1 kcal/kg°C
- $T_E = Exit Temperature of water = 65°C$
- T_w = Inlet water temperature = 20°C
- $Q_E = Exhaust gas from preheater (Nm³/h)$
 - $= 3.99 \times 10^6 \text{ m}^3/\text{day} = 166.25 \times 10^3 \text{ m}^3/\text{hr}.$

 $Q_N = Normalized$ gas flow rate

$$=\frac{T_N}{T_E}\times\frac{P_E}{P_E}\times \mathbf{Q}_E$$

 ρ_a = Exhaust density (kg/m³)

$$=\frac{4.55 \times 10^6 \text{ kg/day}}{3.99 \times 10^6 \text{ m3/day}} = 1.14 \text{ kg/m}^3$$

 ΔT = gas cooling required = (700 - 100) °C = 600°C

$$(C_p)_a = \frac{1.142 + 1.005}{2} = 1.0735 \text{ KJ/kg.°C}$$

$$M_{w} \times (C_{p})_{w} \times (T_{E} - T_{w}) = Q_{E} \times \rho_{a} \times (C_{p})_{a} \times \Delta T$$

$$M_{\rm w} = \frac{166.25 \times 10^3 \times 1.14 \times .26 \times 600}{1 \times (65 - 20)}$$

$$= 657.02 \times 10^3 \, \text{kg/hr} = 15.76 \times 10^3 \, \text{kg/day}$$

$$= 3.85 \text{ kg/kg clinker}$$

Pressure drop (ΔP) :

$$\Delta P = \frac{\alpha \rho V^2}{2} \left[6 \right]$$

Here, $\alpha = \frac{KA}{D_e^2}$ [where K is a proportionality constant ranging between 13-15]

[De is the gas exit diameter]

A= cross sectional area of inlet

Air Inlet (300°C) = 9.52P mol/day = $150.13 \times 10^6 \times \frac{.0821 \times 573}{3.5} \times \frac{1m^3}{1000L} = 2.018 \times 10^6 \text{ m}^3/\text{day}$ = 23.35 m³/sec Assuming a velocity V = 10 m/s

$$A = \frac{Q}{V} = 2.335 \text{ m}^2$$

For a circular cross section, $A = \pi r^2$

So,
$$r = 0.86 m$$

Dia of inlet of one-cylinder $D_i = 1.73 \text{ m}$

So, considering conventional arrangement, $D_e = 1.73$ m

Body dia = 6.92 m for each cyclone (total 4 stage cyclone [1])

So,
$$\alpha = \frac{KA}{D_e^2} = 10.14$$

So, $\Delta P = \frac{\alpha \rho V^2}{2} = 577.98$ pa for each cyclone = 58.9 mm H₂O

Amount of false air = $8900 \times A_p \times (\Delta P)^{0.5} [A_p = Aperture area = .01A (assumed)]$

$$= 2594 \text{ m}^3/\text{hr}$$

False air % = $\frac{2594}{46.23 \times 3600}$ = 15.59 × 10⁻³ or 1.55%

As false air is < 5%, These dimensions would be satisfactory

8.4.3 Preheater fan design:

Height above mean sea level H_{MSL}= 80 m

Preheater feed = 6288 ton/day

Plant capacity = 1.5 MT/year

= 4045-ton clinker /day

Air inlet flow rate $V_a\!=\!2.018\times 10^6~m^3/day$

Air inlet temperature = 300 °C

Elevation factor
$$K_E = \frac{1}{(1 - \frac{.0065 \times H_{MSL}}{288})^{5.255}}$$
$$= \frac{1}{(1 - \frac{.0065 \times 80}{288})^{5.255}}$$
$$= 1.009$$

Applying this correction factor,

Corrected temperature $T_c = (300+273) \times K_E$

Pressure drop $\Delta P = 54$ mbar

Corrected pressure drop $\Delta P_c = 1.009 \times 54 = 54.5$ mbar

Normalized air inlet flow rate calculation:

$$\frac{P_N V_N}{T_N} = \frac{P_c V_a}{T_c}$$

Here, Corrected pressure $P_c = P_N - \Delta P_c$

= 3545 - 54.5 = 3490mbar

So,
$$V_N = \frac{T_N}{T_c} \times \frac{P_c}{P_N} \times V_a$$

= $\frac{273}{593.63} \times \frac{3490}{1013} \times 2.018 \times 10^6$
= $3.21 \times 10^6 \text{ m}^3/\text{day}$

Normalized air per kg clinker production ($\hat{V}_N)$

$$\begin{split} \hat{V}_{\rm N} &= 3.21 \times 10^6 \frac{m^3}{day} \times \frac{1 \ day}{4045 \ ton \ clinker} \times \frac{1 \ ton}{1000 \ kg} \\ &= 0.80 \frac{m^3}{kg \ clinker} \end{split}$$

Fan efficiency η_f = 80 % (for backward curved fan) [7]

Dust loading
$$D_1 = \frac{Dust \ amount}{Exhaust \ gas}$$

$$= \frac{60.2 \times 10^3}{3.99 \times 10^6}$$
$$= .01508 \ \text{kg/m}^3 = 15.08 \ \text{g/m}^3$$
Air density at inlet $\rho_a = \frac{3.5}{.0821 \times 573}$
$$= .075 \ \text{mol/L}$$

 $= 2.163 \text{ kg/m}^3$

Air density with dust $\rho_{ad} = 2.163 + .015 = 2.178 \text{ kg/m}^3$

Fan power calculation:

Fan Power required without dust $W_f = \frac{V_a \Delta P}{\eta_f}$

$$V_a = 2.018 \times 10^6 \text{ m}^3/\text{day}$$

= 23.35 m³/sec
$$W_f = \frac{23.35 \times 5400}{0.8} = 157.65 \times 10^3 \text{ W} = 157.65 \text{ KW}$$

Fan Power required with dust $W_{fd} = W_f \times \frac{\rho_{ad}}{\rho_a}$

$$=157.65 \times \frac{2.178}{2.163}$$

Margin on shaft power = 5%

Minimum motor power required = $W_{fd} \times (1+.05) = 167.055$ KW

Normally fan is designed 15% over capacity, Power requirement would be 192.11 kW

So, a 200 KW fan would be suitable. [1]



Figure 8.4.3: Cross Sectional View of a Cyclone
Dimension of single cyclone in the set of 4 preheater arrangement :

| Dimension | Value (m) |
|----------------|-----------|
| De | 6.92 |
| Do | 1.73 |
| Гì | 0.86 |
| D _d | 1.73 |
| L _b | 13.84 |
| Lc | 13.84 |
| S | 4.75 |

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[9]<u>https://www.globalspec.com/learnmore/manufacturing_process_equipment/air_quality/c</u> <u>yclone_separators</u> Chapter 9

Equipment & Plant Layout

Chapter 10

Economic Analysis

10.1 Cost Estimation

| No. | Equipment | Unit No. | No. of | Cost (USD) |
|-----|-----------------|-------------------|-----------|------------|
| | name | | Equipment | |
| 1 | Conveyor belt | J-110 | 1 | 2000000 |
| 2 | Crusher | C-120 | 1 | 20000 |
| 3 | Crusher | C-121 | 1 | 12000 |
| 4 | Storage bin | F -411, F-331, F- | 5 | 94380 |
| | | 332, F-333, F- | | |
| | | 122 | | |
| 5 | Ball Mill | C-210 | 1 | 20000 |
| 6 | ESP | H-241, H244 | 2 | 1145920 |
| 7 | Preheater | E-220 | 1 | 120000 |
| 8 | Filter | H-211 | 1 | 3000 |
| 9 | Rotary kiln | R-240 | 1 | 280000 |
| 10 | Cooler | E-249, E-228 | 2 | 240200 |
| 11 | Cement mill | C-420 | 1 | 367000 |
| 12 | Air blower | G-311 | 1 | 87300 |
| 13 | Air Blower | G-312 | 1 | 100400 |
| 14 | Compressor | G-247 | 2 | 10000 |
| 15 | Clinker Cooler | E-310 | 1 | 4310000 |
| | Total purchased | | | 8810200 |
| | cost | | | |

So, total purchased equipment cost at 2014, based on US Gulf Coast = 8.81 million USD

According to Marshall and Swift installed equipment indexes for process industry,

The cost index of 2019 / The cost index of 2014 = 1.27

Hence,

Total Purchased Equipment Cost (2019) = Original Cost × (index value at 2019 / index value at 2014)

 $= 8.81 \times 1.27$ million USD=11.19 million USD

Equipment (7), (9), (10), (15) – Price estimation was done from www.alibaba.com

Equipment no (1), (2), (3), (4), (5), (6), (8), (11), (12), (13), (14) – Price estimation was done from – www.matche.com

All of the equipment price [Except-(7), (15)] was checked again from

"Chemical Process Design" by dioneoil.com. Citied- February, 2019. Available at:

https://www.dioneoil.com/uploads/6/8/7/4/6874938/equipment_sizing_and_costing_.pdf

10.2 Calculation of total capital investment (TCI)

| Cost Type | Percent of deliver | Cost | | |
|-----------------------------|--------------------|---------------|--|--|
| | equipment cost | (million USD) | | |
| Direct Cost | | | | |
| Purchased Equipment Cost | 100 | 11.19 | | |
| Equipment Installation Cost | 39 | 4.36 | | |
| Instrumentation and control | 26 | 2.91 | | |
| Piping | 31 | 3.47 | | |
| Electrical Systems | 10 | 1.12 | | |
| | | | | |
| Building and services | 29 | 3.24 | | |
| Yard improvements | 12 | 1.34 | | |
| Service facilities | 55 | 6.15 | | |
| Total Direct Cost | 302 | <u>33.80</u> | | |
| Indirect Costs | | | | |
| Engineering and Supervision | 32 | 3.58 | | |
| Construction Expenses | 34 | 3.80 | | |
| Legal Expenses | 4 | 0.45 | | |
| Contractors fee | 19 | 2.12 | | |
| Contingency | 37 | 4.14 | | |
| Total Indirect Cost | <u>126</u> | <u>14.10</u> | | |
| Fixed Capital Investment | <u>428</u> | <u>47.89</u> | | |
| (FCI) | | | | |
| Working Capital | <u>75</u> | <u>8.45</u> | | |
| Total Capital Investment | <u>503</u> | <u>56.34</u> | | |

10.3 Estimation of product Cost

Manufacturing Cost

Working day basis 330 day/yr

1. Fixed Charges (15% of total product cost)

Depriciation:

Economic life: 20 years

FCI: 47.89 million USD

Salvage value: (10% of FCI) = 4.79 million USD

Depriciation per year = $\frac{47.89 - 4.79}{20}$

= 2.155 million USD

Insurance: (1% of Fixed Capital Investment) = 0.48 million USD

Local Taxes: (3% of FCI) = 1.44 million USD

Financing: (5% of TCI): 2.81 million USD

Fixed Charges = 6.89 million USD

2. Direct Production Cost

Raw material

Limestone: 5655 ton/day = 1.86×10^6 ton/year

Cost: \$22 per ton

Annual cost: 40.92 million USD

Clay: 49.9 ton/day = 16467 ton/year

Cost: \$18 per ton

Annual cost: 0.30 million USD

Gypsum: 75.67 ton/day = 24971 ton/year

Cost: \$40 per ton

Annual cost: 1 million USD

Fly Ash: 272.67 ton/day = 89981 ton/year

Cost: \$20 per ton

Annual cost: 1.80 million USD

Blast Slag: 151.67 ton/day = 50051 ton/year

Cost: \$15 per ton

Annual cost: 0.75 million USD

Coal: 904 ton/ dat = 298379 ton/ year

Cost: \$170 per ton

Annual Cost: = 50.72 million USD

Total Raw Material cost: 95.5 million USD

Operating labor: (10% of Total product cost): $=\frac{0.1 \times Fixed Charges}{0.15} = \frac{0.1 \times}{0.15} = 4.60$ million USD **Direct supervisory and clerical labor:** (15% of Operating labor): 0.69 million USD **Utility:** (10% of Total Product cost): 4.6 million USD **Maintenance and repair:** (5% of Fixed Capital Investment): 2.4 million USD **Operating Supplies:** (10% of Maintenance and repair): 0.24 million USD **Laboratory Charges:** 10% of Operating labor: 0.46 million USD **Patent & Royalties:** (1% of Total Product cost): .46 million USD **Total Direct Production cost:** 108.9 million USD

3. Plant Overhead Cost

50% of Operating labor, supervision and maintenance cost: 3.84 million USD

Total manufacturing Cost: 119.64 million USD

General Expenses

- 1. Administrative cost (2% of Total product cost): 0.77 million USD
- 2. Distribution and marketing cost (5% of total product cost) : 2.30 million USD
- **3.** Research and Development (3% of total product cost) : 1.38 million USD

Total Product Cost = total manufacturing cost + general expense = 124.10 million USD per year

10.4 Estimation of Annual Revenue

Total Sales per Year

Revenue earned from Cement Plant

Clinker for sale Production 0.5 ton per year

Price of clinker = 31 USD/ ton

OPC Production 0.5 ton per year

Price of OPC = 120 USD/ ton

PCC Production 0.5 ton per year

Price of PCC = 110 USD/ ton

Total Sales = $(0.5 \times 30) + (0.5 \times 120) + (0.5 \times 110) = 130.5$ million USD

Gross Earning = Total income – Total product cost

=(130.5-124.10)

= 6.40 million USD

Income Tax = 10% of gross earning = 0.64 million USD

Annual Net Earning = (6.40-0.64) = 5.76 million USD

10.5 Feasibility Analysis

15.5.1 Present Worth

MARR = i = 8% per year

Plant life = 20 years

Salvage value = \$4.79 million

Cash Flow Diagram



Salvage value = 10% of FCI= $0.1 \times 47.90 = 4.79$ million USD

Net PW (8%) = I + (R-E) (P/A, 10%, 20) + S (P/F, 10%, 20)

$$= -56.34 + (130.50 - 124.10) \frac{(1 + 0.08)^{20} - 1}{0.08 \times (1 + 0.08)^{20}} + 4.79(1 + 0.08)^{-20}$$

= 7.52 million USD

10.5.2 Calculation of Payback Period

| End of year | Simple Payback period | Discounted payback period | |
|-------------|-----------------------|----------------------------|--|
| | (at 0% interest rate) | (at interest rate=MARR=8%) | |
| 0 | -56.34 | -54.33 | |
| 1 | -49.94 | -50.42 | |
| 2 | -43.54 | -44.93 | |
| 3 | -37.14 | -39.85 | |
| 4 | -30.74 | -35.15 | |
| 5 | -24.34 | -30.8 | |
| 6 | -17.94 | -26.77 | |
| 7 | 11.54 | -23.04 | |
| 8 | -5.14 | -19.59 | |
| 9 | 1.26 | -16.39 | |
| 10 | | -13.43 | |
| 11 | | -10.685 | |
| 12 | | -8.145 | |
| 13 | | -5.79 | |
| 14 | | -3.61 | |
| 15 | | -1.59 | |
| 16 | | .273 | |

Table: Payout period calculation (Interest rate: 0% and 8%)

Simple Payback period is 9 years and discounted payback period is 16 years.

10.5.3 Internal Rate of Return

MARR= 8% per year

IRR calculation by PW method,

PW (i'%) = 0 Or, I + (R - E) $\left(\frac{P}{A}, i'\%, 20\right)$ + S $\left(\frac{P}{F}, i'\%, 20\right)$ = 0 Or, -56.34 + (130 - 123.6) $\times \frac{(1+i')^{20}-1}{i'\times(1+i')^{20}}$ + 4.79 $\times (1+i')^{-20} = 0$

Or, i' =0.097 or 9.7%

The plant is economically justified because **IRR> MARR**.

10.5.4 External Rate of Return

Assuming,

External reinvestment rate per period, $\epsilon = MARR = 8$ % per year

Annual R-E= 130.50-124.10=6.40 million USD

Present Worth of negative cash flow, PW (10%) = I = \$ 56.34 million

Future Worth of positive cash flow, FW (10%) = 6.4 (F/A, 10 %, 20)+4.79 = 297.67 million \$

ERR calculation,

(56.34) (F/P, i'', 20) = 297.67

or, i'' = 0.0867 or 8.67 %

The plant is economically justified because **ERR** > **MARR**.

Chapter 11 Conclusion

Conclusion

The cement plant was designed selecting dry process. The raw material, limestone, is to be imported from Meghalaya, India. An economic analysis of the plant was performed and the plant was found to be economically feasible. The investment is likely to be recovered within 16 years. The local and global demand of cement is increasing. However, the production being larger than local demand, international market needs to be captured. Because of raw material and fuel availability and good transportation facility, Chatak, Surma can be considered a good choice as the location of the plant.

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Figure 9.1: Plot plan of cement plant



Figure 9.2: cement process plant equipment layout (major units)