# 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 $1450{ }^{\circ} \mathrm{C}$. 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)
Table 2.1: Raw material composition

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

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 (\%) |
| :--- | :--- |
| $\mathrm{C} 2 \mathrm{~S}\left(2 \mathrm{CaO} \cdot \mathrm{SiO}_{2}\right)$ | 17.39 |
| $\mathrm{C} 3 \mathrm{~S}\left(3 \mathrm{CaO} \cdot \mathrm{SiO}_{2}\right)$ | 50.1 |
| $\mathrm{C} 3 \mathrm{~A}\left(3 \mathrm{CaO} \cdot \mathrm{Al}_{2} \mathrm{O}_{3}\right)$ | 10 |
| $\mathrm{C} 4 \mathrm{AF}\left(4 \mathrm{CaO} \cdot \mathrm{Al}_{2} \mathrm{O}_{3} \cdot \mathrm{Fe}_{2} \mathrm{O}_{3}\right)$ | 6.6 |
| Inert (unreacted $\mathrm{CaO}, \mathrm{MgO}$ and |  |
| other impurities) | 16 |

Table 2.3: OPC and PCC composition

| Component | OPC (Ordinary Portland <br> Cement) <br> $(95 \%$ Clinker + <br> $5 \%$ Gypsum $)$ | PCC (Portland <br> Composite Cement) |
| :--- | :--- | :--- |
| $\mathrm{SiO}_{2}$ | 18.25 | $5 \%$ Clinker +1 <br> $5 \%$ |
| $\mathrm{Al}_{2} \mathrm{O}_{3}$ | 4.89 | 23.62 |
| $\mathrm{Fe}_{2} \mathrm{O}_{3}$ | 2.05 | 7.27 |
| CaO | 67.96 | 60.53 |


| 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: $36^{\circ} \mathrm{C}$ (in summer)
Minimum dry bulb temperature: $8^{\circ} \mathrm{C}$ (in winter)

## b. Design conditions for building: HVAC

Summer:
Wet bulb Temperature (max) $30.5^{\circ} \mathrm{C}$
Dry bulb Temperature (max) $36^{\circ} \mathrm{C}$

Relative humidity 84\%
Wet bulb temperature (min) $22^{\circ} \mathrm{C}$
Dry bulb Temperature (min) $25^{\circ} \mathrm{C}$
Dew point $21^{\circ} \mathrm{C}$
Relative humidity 80\%
Winter:
Wet bulb Temperature (max) $21^{\circ} \mathrm{C}$
Dry bulb Temperature (max) $26^{\circ} \mathrm{C}$
Dew point $20^{\circ} \mathrm{C}$
Relative humidity $\mathbf{7 2 \%}$
Wet bulb temperature (min) $7^{\circ} \mathrm{C}$
Dry bulb Temperature (min) $8^{\circ} \mathrm{C}$
Dew point $8^{\circ} \mathrm{C}$
Relative humidity $67 \%$

## c. Meteorological conditions

i. Ambient Temperature:

Absolute max. Temperature: $38^{\circ} \mathrm{C}$
Absolute min. temperature: $12^{\circ} \mathrm{C}$
Designed max. temperature: $38^{\circ} \mathrm{C}$
Designed min. temperature: $5^{\circ} \mathrm{C}$
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 $58 \mathrm{miles} / \mathrm{hr}$ )
v. Rain:

Annual avg. rainfall: 2850mm
Max. Monthly rainfall: 235 mm
vi. Earth Bearing Capacity:

Load bearing capacity: The soil has a bearing capacity of $0.5 \mathrm{~kg} / \mathrm{cm} 2$. Required piling is used.
Corrosive tendency: Non-Corrosive.

### 2.2.3 Utility conditions

## Electric power

$440 \mathrm{~V}-3$ phase, 50 Hz for motors in the range of:
$230 \mathrm{~V}-1$ phase 50 Hz for motors 0.5 KW and below.
$120 \mathrm{~V}-1$ phase 50 Hz for instruments and control.
Normal lighting: $220 \mathrm{~V}-1$ phase, 50 Hz
Air
Instrument air
Pressure: 200 KPa
Temperature: $25^{\circ} \mathrm{C}$
Hot air Temperature: $170^{\circ} \mathrm{C}$
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 $\mathrm{k} / \mathrm{kg}$ of clinker) but the power consumption is typically lower at $110 \mathrm{kWh} /$ ton of cement (Chemical Universe. 2007 <br>)

### 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 $3500-5000 \mathrm{k} / \mathrm{kg}$ of clinker. The power consumption is typically in the range $115-120 \mathrm{kWh} /$ 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 $3500-5000 \mathrm{kj} / \mathrm{kg}$ of clinker. The power consumption is typically in the range $115-120 \mathrm{kWh} /$ 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 $2750-4000 \mathrm{kj} / \mathrm{kg}$ of clinker. The power consumption is in the range of $120-125 \mathrm{kWh} /$ ton of cement 9 Chemical Universe, 2007). However, the most modern, dry process cement plants can have power consumption in the range $80-100 \mathrm{kWh} /$ 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 $750-950 \mathrm{kcal} / \mathrm{kg}$ 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.

Table 3.7 General features of the main types of Portland cement

|  | Classification | Characteristics | Applications |
| :---: | :---: | :---: | :---: |
| Type 1 | General purpose | Fairly high $\mathrm{C}_{3} \mathrm{~S}$ content for good early strength development | General construction (most buildings, bridges, pavements ,precast units etc) |
| Type 2 | Moderate sulfate resistance. | $\begin{array}{lll}\substack{\text { Low } \\(<8 \%)} & \mathrm{C}_{3} \mathrm{~A} & \text { content }\end{array}$ | Structures exposed to soil or water containing sulfate tons. |
| Type 3 | High early strength | Ground more finely may have slightly more $\mathrm{C}_{3} \mathrm{~S}$ | Rapid construction, cold weather concerning |
| Type 4 | Low heat of hydration (slow reacting) | Low content of $\mathrm{C}_{3} \mathrm{~S}$ ( $<50 \%$ ) and $\mathrm{C}_{3} \mathrm{~A}$ | Massive structures such as dams. Now rare. |
| Type 5 | High sulfate resistance | Very low $\mathrm{C}_{3} \mathrm{~A}$ content (<5\%) | Structures exposed to high levels of sulfate ions |
| White | White color | No C3 Af low mgO | Decorative (otherwise has properties similar to type ) |

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




## Chapter 4

## Material Balance

### 4.1 Basis:

$100 \mathrm{~kg} /$ 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.

$$
\begin{aligned}
\text { So, Accumulation } & =0 \text { (Steady state) } \\
\text { Generation \& Consumption } & =0 \text { (No Chemical reaction) }
\end{aligned}
$$

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 |
| $\mathrm{SiO}_{2}$ | 1.88 | 70.69 |
| $\mathrm{Al}_{2} \mathrm{O}_{3}$ | 1.20 | 14.91 |
| $\mathrm{CaCO}_{3}$ | 94.45 | 4.5 |
| $\mathrm{Fe}_{2} \mathrm{O}_{3}$ | .57 | 6 |
| $\mathrm{MgCO}_{3}$ | 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 $=85 \mathrm{~kg} /$ day of limestones
Lost in Crushing operation $=.05 \times 85=4.25 \mathrm{~kg} /$ day
M2 = Amount of limestones crushed $/$ day $=85-4.25=80.25 \mathrm{~kg} /$ day

### 4.3 Dust Purifier



Before Mixing with limestones additives are purified and 5\% loss is assumed in the filter
M3 = Amount fed $=15 \mathrm{~kg} /$ day

Loss $=.05 \times 15=.75 \mathrm{~kg} /$ day
M4 $=$ Additives Recovered $=15-0.75=14.25 \mathrm{~kg} /$ day

### 4.4 Mixer

Crushed Limestone(M2)


Crushed limestone and recovered additives are mixed in a mixer to produce the feed for Grinding.
$\mathrm{M} 5=$ Amount fed in the Raw Mill $=\mathrm{M} 2+\mathrm{M} 4=80.75+14.25=95 \mathrm{~kg} / \mathrm{day}$
(a)

Amount of each component in the feed for raw mill (M5) is calculated in $\mathrm{kg} /$ day
Component amount $(\mathbf{x} 1)=\left(\right.$ weight $\%$ in limestones $\left.\times \frac{M 2}{100}\right)+\left(\right.$ weight $\%$ in clay $\left.\times \frac{M 4}{100}\right)$
$\mathrm{SiO}_{2}=(1.88 \times .8075)+(70.69 \times .1425)=11.59 \mathrm{~kg} /$ day
$\mathrm{Al}_{2} \mathrm{O}_{3}=(1.2 \times .8075)+(14.91 \times .1425)=3.09 \mathrm{~kg} /$ day
$\mathrm{CaCO}_{3}=(94.45 \times .8075)+(4.5 \times .1425)=76.91 \mathrm{~kg} /$ day
$\mathrm{Fe}_{2} \mathrm{O}_{3}=(.57 \times .8075)+(6 \times .1425)=1.32 \mathrm{~kg} /$ day
$\mathrm{MgCO}_{3}=(1.95 \times .8075)+(1.5 \times .1425)=1.79 \mathrm{~kg} /$ day
Inert $=(0 \times .8075)+(2.4 \times .1425)=.342 \mathrm{~kg} /$ day
(b)

Composition of components in M5 is calculated as
Composition in M5 $=\left(\frac{\text { Component } \operatorname{Amount}(x 1)}{M 5}\right) \times 100 \%$
$\mathrm{SiO}_{2}=\left(\frac{11.59}{95}\right) \times 100 \%=12.20 \%$
$\mathrm{Al}_{2} \mathrm{O}_{3}=\left(\frac{3.09}{95}\right) \times 100 \%=3.25 \%$
$\mathrm{CaCO}_{3}=\left(\frac{76.91}{95}\right) \times 100 \%=80.96 \%$
$\mathrm{Fe}_{2} \mathrm{O}_{3}=\left(\frac{1.32}{95}\right) \times 100 \%=1.38 \%$
$\mathrm{MgCO}_{3}=\left(\frac{1.79}{95}\right) \times 100 \%=1.88 \%$
Inerts $=\left(\frac{342}{95}\right) \times 100 \%=.36 \%$

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

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

### 4.5 Raw Mill



M5 $=95 \mathrm{~kg} /$ 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 $=.02 \times 95=1.9 \mathrm{~kg} /$ day
$\mathrm{M} 7=$ Preheater feed $=95-.475=94.525 \mathrm{~kg} /$ day

## (a)

Amount of each component in the preheater feed (M7) is calculated in kg/day
Component amount $(\mathbf{x} 2)=\left(\frac{\text { Component amount }(x 1)}{M 5}\right) \times M 7$
$\mathrm{SiO}_{2}=\frac{11.59}{95} \times 94.525=11.50 \mathrm{~kg} /$ day
$\mathrm{Al}_{2} \mathrm{O}_{3}=\frac{3.04}{95} \times 94.525=3.08 \mathrm{~kg} /$ day
$\mathrm{CaCO}_{3}=\frac{76.91}{95} \times 94.525=76.53 \mathrm{~kg} /$ day
$\mathrm{Fe}_{2} \mathrm{O}_{3}=\frac{1.32}{95} \times 94.525=1.31 \mathrm{~kg} /$ day
$\mathrm{MgCO}_{3}=\frac{1.79}{95} \times 94.525=1.78 \mathrm{~kg} / \mathrm{day}$
Inerts $=\frac{.342}{95} \times 94.525=.34 \mathrm{~kg} / \mathrm{day}$
(b)

Molecular weight of the component in the mixtures are

| Component | Molecular weight <br> $(\mathrm{kg} / \mathrm{kmol})$ |
| :---: | :---: |
| $\mathrm{SiO}_{2}$ | 60 |
| $\mathrm{Al}_{2} \mathrm{O}_{3}$ | 102 |
| $\mathrm{CaCO}_{3}$ | 100 |
| $\mathrm{Fe}_{2} \mathrm{O}_{3}$ | 160 |
| $\mathrm{MgCO}_{3}$ | 84 |
| Inert |  |

Molar amounts of each component in the preheater feed (M7) is calculated by Molar amounts in kmol/day $(\mathbf{y} 1)=\left(\frac{\text { Component amount }(x 2)}{\text { Molecular weight }}\right)$
$\mathrm{SiO}_{2}=\frac{11.50}{60}=.19 \mathrm{kmol} /$ day
$\mathrm{Al}_{2} \mathrm{O}_{3}=\frac{3.08}{102}=.03 \mathrm{kmol} / \mathrm{day}$
$\mathrm{CaCO}_{3}=\frac{76.53}{100}=.76 \mathrm{kmol} /$ day
$\mathrm{Fe}_{2} \mathrm{O}_{3}=\frac{1.31}{160}=.008 \mathrm{kmol} /$ day
$\mathrm{MgCO}_{3}=\frac{1.78}{84}=.02 \mathrm{kmol} /$ day
Inerts $=.34 \mathrm{~kg} /$ day

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

| Component | Preheater Feed (M7) $=94.525 \mathrm{~kg} / \mathrm{day}$ |  |  |
| :---: | :---: | :---: | :---: |
|  | Amount (kg/day) | Weight \% | Molar amount <br> (kmol/day) |
| $\mathrm{SiO}_{2}$ | 11.50 | 12.20 | .19 |
| $\mathrm{Al}_{2} \mathrm{O}_{3}$ | 3.08 | 3.25 | .03 |
| $\mathrm{CaCO}_{3}$ | 76.53 | 80.96 | .76 |
| $\mathrm{FeO}_{3}$ | 1.31 | 1.38 | .008 |
| $\mathrm{MgCO}_{3}$ | 1.78 | 1.88 | .02 |
| Inert | .34 | .36 |  |

### 4.6 Preheater



Preheater is the first stage of pyro-processing of the raw mix . At about $600-700^{\circ} \mathrm{C}$ 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 $=240.63 \mathrm{~mol} /$ day

Kiln Feed $=95.037 \mathrm{~kg} /$ day and composition as follows

## Kiln Feed Composition :

| Components | Mol/day | Kg/day | Weight \% |
| :--- | :--- | :--- | :--- |
| $\mathrm{SiO}_{2}$ | 196.24 | 11.77 | 12.4 |
| $\mathrm{Al}_{2} \mathrm{O}_{3}$ | 31.94 | 3.26 | 3.43 |
| $\mathrm{CaCO}_{3}$ | 765.30 | 76.53 | 80.5 |
| $\mathrm{Fe}_{2} \mathrm{O}_{3}$ | 10.36 | 1.66 | 1.8 |
| $\mathrm{MgCO}_{3}$ | 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



Cooled Clinker (M9)

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

## Desired composition in clinker

| Complexes | Full form | Compound <br> name | Molecular <br> weight | Desired <br> weight\% |
| :---: | :---: | :---: | :---: | :---: |
| C 2 S | $2 \mathrm{CaO} \cdot \mathrm{SiO}_{2}$ | Dicalcium <br> Sillicate | 172 | 20 |
| C 3 S | $3 \mathrm{CaO} \cdot \mathrm{SiO}_{2}$ | Tricalcium <br> Sillicate | 228 | 60 |
| C 3 A | $3 \mathrm{CaO} \cdot \mathrm{Al}_{2} \mathrm{O}_{3}$ | Tricalcium <br> Alluminate | 270 | 12 |
| C 4 AF | $4 \mathrm{CaO} \cdot \mathrm{Al}_{2} \mathrm{O}_{3} \cdot \mathrm{Fe}_{2} \mathrm{O}_{3}$ | Tetracalcium <br> Alluminoferrite | 486 | 08 |

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 $\mathbf{6 0 0}^{\circ} \mathbf{C}$ In the Calcination zone $\mathrm{CaCO}_{3}$ and $\mathrm{MgCO}_{3}$ decomposes and produced $\mathrm{CO}_{2}$ gas is removed from the system at about $900^{\circ} \mathbf{C}$. 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 $\mathbf{1 3 0 0}^{\circ} \mathrm{C}$.

## Reactions in Rotary Kiln

| No. | Reactions | Zone | Temperature |
| :---: | :---: | :---: | :---: |
| i. | $\mathrm{H}_{2} \mathrm{O}(\mathrm{l})=\mathrm{H}_{2} \mathrm{O}(\mathrm{v})$ | Drying | $600^{\circ} \mathrm{C}$ |
| ii. | $\mathrm{CaCO}_{3}=\mathrm{CaO}+\mathrm{CO}_{2}$ | Calcination | $900^{\circ} \mathrm{C}$ |
| iii. | $\mathrm{MgCO}_{3}=\mathrm{MgO}+\mathrm{CO}_{2}$ |  |  |
| iv. | $2 \mathrm{CaO}+\mathrm{SiO}_{2}=2 \mathrm{CaO} \cdot \mathrm{SiO}_{2}$ |  |  |
| v. | $4 \mathrm{CaO}+\mathrm{Al}_{2} \mathrm{O}_{3}+\mathrm{Fe}_{2} \mathrm{O}_{3}=4 \mathrm{CaO} \cdot \mathrm{Al}_{2} \mathrm{O}_{3} \cdot \mathrm{Fe}_{2} \mathrm{O}_{3}$ | Sintering | $>1400^{\circ} \mathrm{C}$ |
| vi. | $\mathrm{CaO}+2 \mathrm{CaO}_{2} \cdot \mathrm{SiO}_{2}=3 \mathrm{CaO} \cdot \mathrm{SiO}_{2}$ |  |  |
| vii. | $5 \mathrm{CaO}+3 \mathrm{Al}_{2} \mathrm{O}_{3}=5 \mathrm{CaO} \cdot 3 \mathrm{Al}_{2} \mathrm{O}_{3}$ |  |  |
| viii. | $4 \mathrm{CaO}+5 \mathrm{CaO} \cdot 3 \mathrm{Al}_{2} \mathrm{O}_{3}=3\left(3 \mathrm{CaO} \cdot \mathrm{Al}_{2} \mathrm{O}_{3}\right)$ |  |  |

$\mathrm{CaCO}_{3}(\mathrm{~s})=\mathrm{CaO}(\mathrm{s})+\mathrm{CO}_{2}(\mathrm{~g})$
Assuming that all the $\mathrm{CaCO}_{3}$ is decomposed
$\mathrm{X} 1=$ Amount of CaO produced $=.765 .30 \mathrm{~mol} /$ day
$\mathrm{Y} 1=$ Amount of $\mathrm{CO}_{2}$ produced $=765.30 \mathrm{kmol} /$ day

$$
\begin{equation*}
\operatorname{MgCO}_{3}(\mathrm{~s})=\mathrm{MgO}(\mathrm{~s})+\mathrm{CO}_{2}(\mathrm{~g}) . \tag{iii}
\end{equation*}
$$

Assuming that all the $\mathrm{MgCO}_{3}$ is decomposed
$\mathrm{X} 2=$ Amount of MgO produced $=21.19 \mathrm{~mol} /$ day
$\mathrm{Y} 2=$ Amount of $\mathrm{CO}_{2}$ produced $=21.19 \mathrm{~mol} /$ day

Total $\mathrm{CO}_{2}$ removed due to decomposition reaction
$=\mathrm{Y} 1+\mathrm{Y} 2=765.30+21.19=786.49 \mathrm{~mol} /$ day
Amount of $\mathrm{CO}_{2}$ removed as a result of decomposition $=.78 \times 44=34.584 \mathrm{~kg} /$ day
$2 \mathrm{CaO}+\mathrm{SiO}_{2}=\mathbf{2 C a O} . \mathrm{SiO}_{2}$ (iv)

In this reaction conversion of all the $\mathrm{SiO}_{2}$ is considered
$\mathrm{SiO}_{2}$ reacted $=196.24 \mathrm{~mol} /$ day
$\mathrm{Y} 3=$ Amount of CaO reacted $=196.24 \mathrm{~mol} /$ day $\mathrm{SiO}_{2}$ reacted $\times \frac{2 \text { mol cao reacted }}{1 \text { mol SiO2 reacted }}$

$$
=392.48 \mathrm{~mol} / \text { day }
$$

X3 $=$ Amount of C 2 S produced $=196.24 \mathrm{~mol} /$ day
$4 \mathrm{CaO}+\mathrm{Al}_{2} \mathrm{O}_{3}+\mathrm{Fe}_{2} \mathrm{O}_{3}=4 \mathrm{CaO} \cdot \mathrm{Al}_{2} \mathrm{O}_{3} \cdot \mathrm{Fe}_{2} \cdot \mathrm{O}_{3}$ $\qquad$
Complete conversion of $\mathrm{Fe}_{2} \mathrm{O}_{3}$ is considered
$\mathrm{Fe}_{2} \mathrm{O}_{3}$ reacted $=10.36 \mathrm{~mol} /$ day
$\mathrm{Z} 4=$ Amount of CaO reacted $=10.36 \mathrm{~mol} /$ day $\mathrm{Fe}_{2} \mathrm{O}_{3}$ reacted $\times \frac{4 \text { mol caO reacted }}{1 \text { mol } \text { Fe2O3 reacted }}$ $=41.44 \mathrm{~mol} /$ day
$\mathrm{Y} 4=$ Amount of $\mathrm{Al}_{2} \mathrm{O}_{3}$ reacted $=10.36 \mathrm{~mol} /$ day
$\mathrm{X} 4=$ Amount of C 4 AF produced $=10.36 \mathrm{kmol} /$ day
$\mathrm{CaO}+2 \mathrm{CaO} \cdot \mathrm{SiO}_{2}=3 \mathrm{CaO} . \mathrm{SiO}_{2}$
Produced C2S reacts with CaO I such proportion so that it maintains a 60:20 0r 3:1 weight ratio of C3S : C2S in the clinker content .

Weight ratio of $\mathrm{C} 3 \mathrm{~s}: \mathrm{C} 2 \mathrm{~S}$ in the clinker $=3: 1$
Mole ratio of C3S:C2S in the clinker $=\left(\frac{3}{(M W) C 3 S}\right):\left(\frac{1}{(M W) C 2 S}\right)$

$$
\begin{aligned}
& =\frac{3}{228}: \frac{1}{172} \\
& =.0131: .0058 \\
& =2.26: 1
\end{aligned}
$$

So for 1 mole of unreacted C2S in the clinker 2.26 mole reacts to form C3S
$\mathrm{Y} 5=\mathrm{C} 2 \mathrm{~S}$ reacted to produce $\mathrm{C} 3 \mathrm{~S}=\frac{2.26}{2.26+1} \times 196.24=136.04 \mathrm{~mol} /$ day
$\mathrm{X} 5=\mathrm{C} 3 \mathrm{~S}$ produced $=136.04 \mathrm{~mol} /$ day
$\mathrm{Z} 5=$ Amount of CaO reacted $=136.04 \mathrm{~mol} /$ day
Unreacted C2S (U5) $=$ X3 - Y5 $=196.24-136.04=60.19 \mathrm{~mol} /$ day
$5 \mathrm{CaO}+3 \mathrm{Al}_{2} \mathrm{O}_{3}=5 \mathrm{CaO} \cdot 3 \mathrm{Al}_{2} \mathrm{O}_{3}$
Excess $\mathrm{Al}_{2} \mathrm{O}_{3}$ which has not reacted in rection no (v) reacts to produce pentacalcium Trialluminate
$\mathrm{Y} 6=$ Amount of $\mathrm{Al}_{2} \mathrm{O}_{3}$ reacted in (vii) rreaction
$=$ Total $\mathrm{Al}_{2} \mathrm{O}_{3}$ in the kiln feed -Y 4
$=31.94-10.36=21.58 \mathrm{~mol} /$ day
$\mathrm{Z} 6=$ Amount of CaO reacted $=21.58 \mathrm{~mol} /$ day $\mathrm{Al}_{2} \mathrm{O}_{3}$ reacted $\times \frac{5 \text { mol cao reacted }}{3 \text { mol Al2O3 reacted }}$ $=36 \mathrm{~mol} /$ day
$\mathrm{X} 6=$ Amount $5 \mathrm{CaO} \cdot 3 \mathrm{Al}_{2} \mathrm{O}_{3}$ produced
$=21.58 \mathrm{~mol} / \mathrm{day}_{\mathrm{Al}}^{2} \mathrm{O}_{3}$ reacted $\times \frac{1 \text { mol } 5 \mathrm{CaO} .3 \mathrm{Al2O3} \text { produced }}{3 \text { mol } \text { Al2O3 reacted }}$
$=7.19 \mathrm{~mol} / \mathrm{day}$
$4 \mathrm{CaO}+5 \mathrm{CaO} \cdot 3 \mathrm{Al}_{2} \mathrm{O}_{3}=3\left(3 \mathrm{CaO} \cdot \mathrm{Al}_{2} \mathrm{O}_{3}\right)$ (viii)
$\mathrm{Y} 7=$ Amount of CaO reacted $=7.19 \times 4=28.77 \mathrm{~mol} /$ day
$\mathrm{X} 7=$ Amount of C 3 A produced $=7.19 \times 3=21.58 \mathrm{~mol} /$ day

Complex product produced which is remained in the clinker
$\mathrm{C} 2 \mathrm{~S}=\mathrm{U} 5=60.19 \mathrm{~mol} /$ day
$\mathrm{C} 3 \mathrm{~S}=\mathrm{X} 5=136.04 \mathrm{~mol} /$ day
$\mathrm{C} 3 \mathrm{~A}=\mathrm{X} 7=21.58 \mathrm{~mol} /$ day
$\mathrm{C} 4 \mathrm{AF}=\mathrm{X} 4=10.36 \mathrm{~mol} /$ day

## Inert Analysis :

Inerts contain unreacted CaO and MgO .
MgO is considered to not react over the whole region of kiln

So , unreacted $\mathrm{MgO}=$ Produced MgO in (ii) +MgO in coal $=21.19+.24 \mathrm{~mol} /$ day

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

Unreacted $\mathrm{CaO}=$ Total $\mathrm{CaO}-$ Reacted CaO

$$
\begin{aligned}
& =\mathrm{X} 1-(\mathrm{Y} 3+\mathrm{Z} 4+\mathrm{Z} 5+\mathrm{Z} 6+\mathrm{Y} 7) \\
& =765.30-(392.48+136.04+41.44+36+28.77) \\
& =130.57 \mathrm{~mol} / \mathrm{day} \\
& =.15 \times 56 \mathrm{~kg} / \mathrm{day} \\
& =7.3 \mathrm{~kg} / \mathrm{day}
\end{aligned}
$$

Total inert $=$ Inert in feed + Unreacted $\mathrm{MgO}+$ Unreacted CaO

$$
=.34+.857+7.3=8.497 \mathrm{~kg} / \text { day }
$$

## Weight of each component in clinker :

Individual weight is calculated by
Weight in kg/day $=$ molar amount in kmol/day $\times$ Molecular weight
$\mathrm{C} 2 \mathrm{~S}=60.19 \times 172=10.35 \mathrm{~kg} /$ day
$\mathrm{C} 3 \mathrm{~S}=136.04 \times 228=31.07 \mathrm{~kg} /$ day
$\mathrm{C} 3 \mathrm{~A}=21.58 \times 270=5.86 \mathrm{~kg} /$ day
$\mathrm{C} 4 \mathrm{AF}=10.36 \times 486=5.03 \mathrm{~kg} /$ day
Inerts $=8.497 \mathrm{~kg} /$ day
Total weight of clinker $=\mathrm{C} 2 \mathrm{~S}+\mathrm{C} 3 \mathrm{~S}+\mathrm{C} 3 \mathrm{~A}+\mathrm{C} 4 \mathrm{AF}+$ inerts

$$
\begin{aligned}
& =10.35+31.07+5.86+5.03+8.497 \\
& =60.81 \mathrm{~kg} / \mathrm{day}
\end{aligned}
$$

Weight \% in Clinker : Calculated by
Weight $\%=\frac{\text { Individual amount }}{\text { Total Amount }} \times 100 \%$
$\mathrm{C} 2 \mathrm{~S}=\frac{10.35}{60.81} \times 100 \%=17.02 \%$
$\mathrm{C} 3 \mathrm{~S}=\frac{31.07}{60.81} \times 100 \%=51.09 \%$
$\mathrm{C} 3 \mathrm{~A}=\frac{5.86}{60.81} \times 100 \%=9.6 \%$
$\mathrm{C} 4 \mathrm{AF}=\frac{5.04}{60.81} \times 100 \%=8.2 \%$
Inerts $=\frac{8.497}{60.81} \times 100 \%=13.97 \%$

## Clinker Analysis

| Component | Molar Amount <br> (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 |

### 4.8 Seperator



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 $=60.81 \mathrm{~kg} /$ 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
$\mathrm{M} 9=\mathrm{M} 11+\mathrm{M} 12+\mathrm{M} 13=\mathrm{M} 11+.95 \times \mathrm{M} 11+.72 \times \mathrm{M} 11$
$\mathrm{M} 11=\frac{60.81}{1+.95+.72}=22.77 \mathrm{~kg} /$ day of clinker produced for sale
$\mathrm{M} 12=.95 \times$ M11 $=21.63 \mathrm{~kg} /$ day of clinker produced for OPC
$\mathrm{M} 13=.72 \times$ M11 $=16.40 \mathrm{~kg} /$ day of clinker produced for PCC

### 4.9 Mixer (OPC)



Now , M14 = Amount of OPC = Clinker produced for sale $(\mathrm{M} 11)=22.77 \mathrm{~kg} /$ day OPC is assumed to be $95 \%$ clinker and $5 \%$ Gypsum .

Gypsum balance around mixer (OPC)
$\mathrm{M} 14=.05 \times$ M15

$$
=.05 \times 22.77=1.14 \mathrm{~kg} \text { Gypsum } / \text { day }
$$

### 4.10 Mixer (PCC)



PCC is taken to be 72 \% Clinker, 18\% Fly Ash and $10 \%$ Slag.
$\mathrm{M} 18=\mathrm{PCC}$ produced $=22.77 \mathrm{~kg} /$ day
$\mathrm{M} 16=$ Fly Ash required $=.18 \times 22.77=4.1 \mathrm{~kg} /$ day
M17 $=$ Slag required $=.1 \times 22.77=2.28 \mathrm{~kg} /$ day

### 4.11 Scaling up

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

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

$$
=1515 \text { ton clinker / day }
$$

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

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
$\operatorname{Loss}(5 \%)=.75 \times 65.53=49.9$ 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 $(.5 \%)=.02 \times 6320=126$ ton/day

## Around Rotary Kiln :

Kiln Feed $=6288$ ton/day
Clinker produced $=60.81 \times 65.53=4045$ ton $/$ day
$\mathrm{CO}_{2}$ 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 $\mathrm{PCC}=1090$ 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

### 5.1 Crusher :



| Component | Limestone $(85 \mathrm{~kg} / \mathrm{day})$ |  |  |
| :---: | :---: | :---: | :---: |
|  | Composition <br> (weight $\%)$ | Amount |  |
|  |  | Kg/day | Mol/day |
| $\mathrm{SiO}_{2}$ | 1.88 | 1.598 | 26.633 |
| $\mathrm{Al}_{2} \mathrm{O}_{3}$ | 1.20 | 1.020 | 10.000 |
| $\mathrm{CaCO}_{3}$ | 94.45 | 80.283 | 802.83 |
| $\mathrm{Fe}_{2} \mathrm{O}_{3}$ | .57 | .485 | 3.031 |
| $\mathrm{MgCO}_{3}$ | 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
$\mathrm{E}_{\mathrm{r}}=\mathrm{E}_{\mathrm{i}} \sqrt{\frac{100}{L 2}}\left(1-\frac{1}{\sqrt{q}}\right) ;$
Here, $\mathrm{E}_{\mathrm{i}}=$ Work index $=12.74 \mathrm{KJ} / \mathrm{kg}$ for dry crushing limestone

$$
\begin{aligned}
& \mathrm{L}_{2}=\text { Final size }=100 \mathrm{~mm} \\
& \mathrm{q}=\frac{\text { Initial Size }}{\text { Final Size }}=\frac{L 1}{L 2}=\frac{1000}{100}=10
\end{aligned}
$$

So, $E_{r}=12.74 \times \sqrt{\frac{100}{100}}\left(1-\frac{1}{\sqrt{10}}\right)$

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

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

## $\mathbf{E}_{\mathrm{k}}=\mathrm{K}_{\mathrm{k}} \times \mathrm{fc} \times \ln \frac{L 1}{L 2} ;$

Here, $\mathrm{f}_{\mathrm{c}}=$ crushing strength

$$
\begin{aligned}
& \mathrm{L}_{1}=\text { Initial Size } \\
& \mathrm{L}_{2}=\text { Final Size }
\end{aligned}
$$

For reducing size from 1000 mm to 100 mm , Kick's Law
$\mathrm{E}_{\mathrm{k} 1}=\mathrm{K}_{\mathrm{k}} \times \mathrm{fc} \times \ln \frac{1000}{100}=8.711$
For reducing size from 100 mm to 25 mm , Kick's Law
$\mathrm{E}_{\mathrm{k} 2}=\mathrm{K}_{\mathrm{k}} \times \mathrm{fc} \times \ln \frac{100}{25}$
(ii) $\div$ (i)
$\frac{\mathrm{Ek} 2}{8.711}=\frac{\ln (4)}{\ln (10)}$
$\mathrm{E}_{\mathrm{k} 2}=5.244 \mathrm{KJ} / \mathrm{kg}$

For, $\stackrel{\circ}{\mathrm{m}}=85 \mathrm{~kg} /$ day feed, total energy required for crushing

$$
\begin{aligned}
\mathbf{E}_{\mathbf{t}} & =\stackrel{\circ}{\mathbf{m}}\left(\mathbf{E}_{\mathbf{r}}+\mathbf{E}_{\mathbf{k} 2}\right) \\
& =85 \times(8.711+5.244) \\
& =1186.260 \mathrm{KJ} / \text { day }
\end{aligned}
$$

Assuming, $30 \%$ of the applied energy $\left(\mathrm{E}_{\mathrm{a}}\right)$ is used in crushing.
$\mathbf{E}_{\mathbf{a}}=\frac{\boldsymbol{E} \boldsymbol{t}}{\mathbf{0 . 3}}=\frac{1186.260}{0.3}=3954.20 \mathrm{KJ} /$ day
Assuming all the lost energy $\left(\mathrm{E}_{1}\right)$ is convert to heat energy.

$$
\begin{aligned}
\mathbf{E}_{\mathbf{l}} & =\mathbf{E}_{\mathbf{a}}-\mathbf{E}_{\mathbf{t}} \\
& =3954.2-1186.26 \\
& =2767.94 \frac{\mathrm{~kJ}}{d a y} \times \frac{1000 \mathrm{~J}}{1 \mathrm{~kJ}} \times \frac{1 \mathrm{cal}}{4.18 \mathrm{~J}} \\
& =662186.285 \mathrm{cal} / \text { day }
\end{aligned}
$$

Now, $\mathrm{E}_{\mathrm{l}}=\sum \boldsymbol{H}=\int_{\boldsymbol{T} \boldsymbol{i}}^{\boldsymbol{T} \boldsymbol{f}} \boldsymbol{n} \times \boldsymbol{C} \boldsymbol{p}(\boldsymbol{T}) \times d \boldsymbol{T} \ldots \ldots \ldots \ldots .$. (iii)
Here, $\mathrm{Cp}_{\mathrm{i}}=$ specific heats of components, $\mathrm{n}=$ molar flow rate $(\mathrm{mol} / \mathrm{day}), \mathrm{T}=$ temperature $(\mathrm{K})$

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

$$
\begin{aligned}
\mathrm{H}_{\mathrm{SiO} 2} & =\stackrel{\circ}{\mathrm{n}}_{\mathrm{SiO} 2} \times \int_{303}^{T}(10.87+0.008712 \mathrm{~T}) d T \\
& =289.5 \times(\mathrm{T}-303)+0.1145 \times\left(\mathrm{T}^{2}-303^{2}\right) \\
\mathrm{H}_{\mathrm{Al2O} 3} & =\stackrel{\circ}{\mathrm{n}}_{\mathrm{Al2O} 3} \times \int_{303}^{T}(22.08+0.008971 \mathrm{~T}) d T \\
& =220.8 \times(\mathrm{T}-303)+0.045 \times\left(\mathrm{T}^{2}-303^{2}\right) \\
\mathrm{H}_{\mathrm{CaCO} 3} & =\stackrel{\circ}{\mathrm{n}}_{\mathrm{CaCO} 3} \times \int_{303}^{T}(19.68+0.01189 \mathrm{~T}) d T \\
& =15799 \times(\mathrm{T}-303)+4.82 \times\left(\mathrm{T}^{2}-303^{2}\right) \\
\mathrm{H}_{\mathrm{Fe} 2 \mathrm{O} 3} & =\stackrel{\circ}{\mathrm{ne} 2 \mathrm{O} 3} \times \int_{303}^{T}(24.72+0.01604 \mathrm{~T}) d T \\
& =74.89 \times(\mathrm{T}-303)+0.024 \times\left(\mathrm{T}^{2}-303^{2}\right) \\
\mathrm{H}_{\mathrm{MgCO}} & =\stackrel{\circ}{\mathrm{n}}_{\mathrm{MgCO}} \times \int_{303}^{T} 16.9 d T \\
& =333.57 \times(\mathrm{T}-303)
\end{aligned}
$$

Now from (iii)

$$
\mathrm{E}_{1}=\mathrm{H}_{\mathrm{SiO} 2}+\mathrm{H}_{\mathrm{Al} 2 \mathrm{O} 3}+\mathrm{H}_{\mathrm{CaCO} 3}+\mathrm{H}_{\mathrm{Fe} 2 \mathrm{O} 3}+\mathrm{H}_{\mathrm{MgCO} 3}
$$

$$
\begin{aligned}
662186 & =16717.76 \times(\mathrm{T}-303)+5.004 \times\left(\mathrm{T}^{2}-303^{2}\right) \\
\mathrm{T} & =336.25 \mathrm{~K}=63.15^{\circ} \mathrm{C}
\end{aligned}
$$

So, temperature of crushed limestone, $\mathrm{T}_{\text {output }}=63.15^{\circ} \mathrm{C}$


| Component | Composition (weight <br> \%) |  | Amount (95 kg/day) |  |  |  |
| :---: | :---: | :---: | :---: | :---: | :---: | :---: |
|  | Kimestone |  | Clay | Limestone | Clay | Limestone |
|  | 1.88 | 70.69 | 1.509 | 10.073 | 25.15 | 167.883 |
| $\mathrm{Al}_{2} \mathrm{O}_{3}$ | 1.20 | 14.91 | 0.963 | 2.125 | 9.441 | 20.833 |
| $\mathrm{CaCO}_{3}$ | 94.45 | 4.50 | 75.796 | 0.641 | 757.96 | 6.41 |
| $\mathrm{Fe}_{2} \mathrm{O}_{3}$ | 0.57 | 6.00 | 0.457 | 0.855 | 2.856 | 5.343 |
| $\mathrm{MgCO}_{3}$ | 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

$$
\mathbf{Q}_{\mathbf{1}}=\mathbf{Q}_{\mathbf{c}}
$$

Here, $\mathrm{Q}_{1}=$ Heat given by limestone $\left(63.15^{\circ} \mathrm{C}\right)$
$\mathrm{Q}_{\mathrm{c}}=$ Heat taken by Clay $\left(30^{\circ} \mathrm{C}\right)$

## Heat given by limestone :

$$
\begin{aligned}
\mathrm{H}_{\mathrm{SiO2}} & =\mathrm{n}_{\mathrm{SiO2} 2} \times \int_{T m}^{336.25}(10.87+0.008712 \mathrm{~T}) d T \\
& =273.38 \times\left(336.25-\mathrm{T}_{\mathrm{m}}\right)+.108 \times\left(336.25^{2}-\mathrm{T}_{\mathrm{m}}{ }^{2}\right) \\
\mathrm{H}_{\mathrm{Al2O3}} & =\text { ñAl2O3 } \times \int_{T m}^{336.25}(22.08+0.008971 \mathrm{~T}) d T \\
& =208.46 \times\left(336.25-\mathrm{T}_{\mathrm{m}}\right)+0.0425 \times\left(336.25^{2}-\mathrm{T}_{\mathrm{m}}{ }^{2}\right) \\
\mathrm{HCaCO3} & =\mathrm{n}_{\mathrm{CaCO}} \times \int_{T m}^{336.25}(19.68+0.01189 \mathrm{~T}) d T \\
& =14916 \times\left(336.25-\mathrm{T}_{\mathrm{m}}\right)+4.55 \times\left(336.25^{2}-\mathrm{T}_{\mathrm{m}}{ }^{2}\right) \\
\mathrm{H}_{\mathrm{Fe2O3}} & =\mathrm{n}_{\mathrm{Fe} 203} \times \int_{T m}^{336.25}(24.72+0.01604 \mathrm{~T}) d T \\
& =70.6 \times\left(336.25-\mathrm{T}_{\mathrm{m}}\right)+0.023 \times\left(336.5^{2}-\mathrm{T}_{\mathrm{m}^{2}}\right) \\
\mathrm{H}_{\mathrm{Mg} \mathrm{CO} 3} & =\mathrm{n}_{\mathrm{Mg}} \mathrm{CO} \times \int_{T m}^{336.25} 16.9 d T \\
& =314.863 \times\left(336.25-\mathrm{T}_{\mathrm{m}}\right)
\end{aligned}
$$

Now, $\quad \mathrm{Q}_{1}=\mathrm{H}_{\mathrm{SiO} 2}+\mathrm{H}_{\mathrm{Al2O}}+\mathrm{H}_{\mathrm{CaCO}}+\mathrm{H}_{\mathrm{Fe} 2 \mathrm{O}}+\mathrm{H}_{\mathrm{Mg} \mathrm{CO}}$

$$
=15783.3 \times\left(336.25-\mathrm{T}_{\mathrm{m}}\right)+4.723 \times\left(336.25^{2}-\mathrm{T}_{\mathrm{m}}{ }^{2}\right)
$$

## Heat taken by clay :

$$
\begin{aligned}
\mathrm{H}_{\mathrm{SiO} 2} & =\stackrel{\circ}{\mathrm{n}}_{\mathrm{SO} 2} \times \int_{303}^{T m}(10.87+0.008712 \mathrm{~T}) d T \\
& =1824.88 \times\left(\mathrm{T}_{\mathrm{m}}-303\right)+0.722 \times\left(\mathrm{T}_{\mathrm{m}}^{2}-303^{2}\right) \\
\mathrm{H}_{\mathrm{Al2O}} & =\stackrel{\circ}{\mathrm{n}}_{\mathrm{Al2O} 3} \times \int_{303}^{T m}(22.08+0.008971 \mathrm{~T}) d T \\
& =460 \times\left(\mathrm{T}_{\mathrm{m}}-303\right)+0.094 \times\left(\mathrm{T}_{\mathrm{m}}^{2}-303^{2}\right)
\end{aligned}
$$

$$
\begin{aligned}
\mathrm{H}_{\mathrm{CaCO} 3} & =\stackrel{\circ}{\mathrm{n}}_{\mathrm{CaCO} 3} \times \int_{303}^{T m}(19.68+0.01189 \mathrm{~T}) d T \\
& =126.15 \times\left(\mathrm{T}_{\mathrm{m}}-303\right)+.038 \times\left(\mathrm{T}_{\mathrm{m}}{ }^{2}-303^{2}\right) \\
\mathrm{H}_{\mathrm{Fe} 2 \mathrm{O} 3} & =\stackrel{\circ}{\mathrm{n}}_{\mathrm{Fe} 2 \mathrm{O} 3} \times \int_{303}^{T m}(24.72+0.01604 \mathrm{~T}) d T \\
& =132.08 \times\left(\mathrm{T}_{\mathrm{m}}-303\right)+0.043 \times\left(\mathrm{T}_{\mathrm{m}}{ }^{2}-303^{2}\right) \\
\mathrm{H}_{\mathrm{MgCO} 3} & =\stackrel{\circ}{\mathrm{n} g \mathrm{CO}} \times \int_{303}^{T m} 16.9 d T \\
& =43.06 \times\left(\mathrm{T}_{\mathrm{m}}-303\right)
\end{aligned}
$$

Now, $\quad \mathrm{Q}_{\mathrm{c}}=\mathrm{H}_{\mathrm{SiO} 2}+\mathrm{H}_{\mathrm{Al} 2 \mathrm{O} 3}+\mathrm{H}_{\mathrm{CaCO} 3}+\mathrm{H}_{\mathrm{Fe} 2 \mathrm{O} 3}+\mathrm{H}_{\mathrm{MgCO}}$

$$
=2586.17 \times\left(\mathrm{T}_{\mathrm{m}}-303\right)+0.897 \times\left(\mathrm{T}_{\mathrm{m}}^{2}-303^{2}\right)
$$

Now, $\quad \mathrm{Q}_{1}=\mathrm{Q}_{\mathrm{c}}$

$$
\begin{aligned}
& =15783.3 \times\left(336.25-\mathrm{T}_{\mathrm{m}}\right)+4.723 \times\left(336.25^{2}-\mathrm{T}_{\mathrm{m}}^{2}\right) \\
& =2586.17 \times\left(\mathrm{T}_{\mathrm{m}}-303\right)+0.897 \times\left(\mathrm{T}_{\mathrm{m}}{ }^{2}-303^{2}\right)
\end{aligned}
$$

$$
\mathrm{T}_{\mathrm{m}}=331.50 \mathrm{~K}=58.40^{\circ} \mathrm{C}
$$

So, temperature of feed for raw mill $\mathrm{T}_{\text {mixer }}=58.40^{\circ} \mathrm{C}$

So, Heat given by limestone $=15783.3 \times\left(336.25-\mathrm{T}_{\mathrm{m}}\right)+4.723 \times\left(336.25^{2}-\mathrm{T}_{\mathrm{m}}{ }^{2}\right)$

$$
\begin{aligned}
& =15783.3 \times(336.25-331.50)+4.723 \times\left(336.25^{2}-331.50^{2}\right) \\
& =89951.15 \mathrm{cal} / \text { day } \\
& =\text { Heat taken by clay }
\end{aligned}
$$

### 5.3 Raw Mill



| Component | Feed for raw mill (95 kg/day) |  |  |
| :---: | :---: | :---: | :---: |
|  | Composition <br> (weight \%) | Amount |  |
|  |  | Kg/day | Mol/day |
| $\mathrm{SiO}_{2}$ | 12.20 | 11.59 | 193.167 |
| $\mathrm{Al}_{2} \mathrm{O}_{3}$ | 3.25 | 3.09 | 30.294 |
| $\mathrm{CaCO}_{3}$ | 80.96 | 76.91 | 769.1 |
| $\mathrm{Fe}_{2} \mathrm{O}_{3}$ | 1.38 | 1.32 | 8.25 |
| $\mathrm{MgCO}_{3}$ | 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 1 mm .
According to Rittinger's Law , required energy
$E_{R}=K_{R} \times f \mathbf{f c} \times\left(\frac{1}{L 2}-\frac{1}{L 1}\right) ;$
Here, $\mathrm{K}_{\mathrm{R}}=$ Proportionality Constant

$$
\begin{aligned}
\mathrm{f}_{\mathrm{c}} & =\text { crushing strength } \\
\mathrm{L}_{1} & =\text { Initial Size } \\
\mathrm{L}_{2} & =\text { Final Size }
\end{aligned}
$$

For reducing size from 100 mm to 25 mm , Rittinger's Law
$E_{R 1}=K_{R} \times f c \times\left(\frac{1}{25}-\frac{1}{100}\right)=5.244$
For reducing size from 25 mm to 1 mm , Rittinger's Law
$\mathrm{E}_{\mathrm{R} 2}=\mathrm{K}_{\mathrm{R}} \times \mathrm{fc} \times\left(\frac{1}{1}-\frac{1}{25}\right)$
(v) $\div$ (iv)
$\frac{E R 2}{5.244}=\frac{\left(1-\frac{1}{25}\right)}{\left(\frac{1}{25}-\frac{1}{100}\right)}$
$\mathrm{E}_{\mathrm{R} 2}=167.80 \mathrm{KJ} / \mathrm{kg}$

For $\stackrel{\circ}{\mathrm{m}}=95 \mathrm{~kg} /$ day feed, total energy required for grinding

$$
\begin{aligned}
\mathbf{E}_{\mathbf{t}} & =\stackrel{\circ}{\mathbf{m}} \times \mathbf{E}_{\mathbf{R} 2} \\
& =95 \times 167.80 \\
& =15941.76 \mathrm{KJ} / \text { day }
\end{aligned}
$$

Assuming, $80 \%$ of the applied energy $\left(\mathrm{E}_{\mathrm{a}}\right)$ is used in grinding .
$\mathbf{E}_{\mathbf{a}}=\frac{\boldsymbol{E} \boldsymbol{t}}{\mathbf{0 . 8}}=\frac{15941.76}{0.8}=19927.20 \mathrm{KJ} /$ day
Assuming all the lost energy $\left(\mathrm{E}_{1}\right)$ is converted to heat energy .

$$
\begin{aligned}
\mathbf{E}_{\mathbf{l}} & =\mathbf{E}_{\mathbf{a} .}-\mathbf{E}_{\mathbf{t}} \\
& =19927.20-15941.76 \\
& =3985.44 \frac{\mathrm{kj}}{d a y} \times \frac{1000 j}{1 k j} \times \frac{1 \mathrm{cal}}{4.18 j} \\
& =953454.55 \mathrm{cal} / \text { day }
\end{aligned}
$$

Now, $\mathrm{E}_{1}=\sum \boldsymbol{H}=\int_{\boldsymbol{T} \boldsymbol{i}}^{\boldsymbol{T} \boldsymbol{f}} \boldsymbol{n} \times \boldsymbol{C} \boldsymbol{p}(\boldsymbol{T}) \times \boldsymbol{d T} \ldots \ldots . . \ldots . .(\mathrm{iii})$
Here, $\mathrm{Cp}_{\mathrm{i}}=$ specific heats of components, $\mathrm{n}=$ molar flow rate $(\mathrm{mol} / \mathrm{day}), \mathrm{T}=$ temperature $(\mathrm{K})$

$$
\begin{aligned}
\mathrm{H}_{\mathrm{SiO} 2} & =\stackrel{\circ}{\mathrm{n} i O 2} \times \int_{331.50}^{T}(10.87+0.008712 \mathrm{~T}) d T \\
& =2099.73 \times(\mathrm{T}-331.50)+0.831 \times\left(\mathrm{T}^{2}-331.50^{2}\right) \\
\mathrm{H}_{\mathrm{Al2O} 3} & =\stackrel{\circ}{\mathrm{n}}_{\mathrm{Al2O} 3} \times \int_{331.50}^{T}(22.08+0.008971 \mathrm{~T}) d T \\
& =668.89 \times(\mathrm{T}-331.50)+0.135 \times\left(\mathrm{T}^{2}-331.50^{2}\right) \\
\mathrm{H}_{\mathrm{CaCO} 3} & =\stackrel{\circ}{\mathrm{n}}_{\mathrm{CaCO} 3} \times \int_{331.50}^{T}(19.68+0.01189 \mathrm{~T}) d T \\
& =15135.89 \times(\mathrm{T}-331.50)+4.62 \times\left(\mathrm{T}^{2}-331.50^{2}\right) \\
\mathrm{H}_{\mathrm{Fe2O} 3} & =\stackrel{\circ}{\mathrm{n}}_{\mathrm{Fe} 2 \mathrm{O} 3} \times \int_{331.50}^{T}(24.72+0.01604 \mathrm{~T}) d T \\
& =203.94 \times(\mathrm{T}-331.50)+0.067 \times\left(\mathrm{T}^{2}-331.50^{2}\right) \\
\mathrm{H}_{\mathrm{MgCO} 3} & =\stackrel{\circ}{\mathrm{n}}_{\mathrm{MCO}} \times \int_{331.50}^{T} 16.9 d T \\
& =360.23 \times(\mathrm{T}-331.50)
\end{aligned}
$$

Now from (iii)

$$
\mathrm{E}_{1}=\mathrm{H}_{\mathrm{SiO} 2}+\mathrm{H}_{\mathrm{Al} 2 \mathrm{O} 3}+\mathrm{H}_{\mathrm{CaCO} 3}+\mathrm{H}_{\mathrm{Fe} 2 \mathrm{O} 3}+\mathrm{H}_{\mathrm{MgCO} 3}
$$

$$
\begin{aligned}
953454 & =18468.68 \times(\mathrm{T}-331.50)+5.65 \times\left(\mathrm{T}^{2}-331.50^{2}\right) \\
\mathrm{T} & =373.96 \mathrm{~K}=100.86^{\circ} \mathrm{C}
\end{aligned}
$$

So, temperature of grinded material, $\mathrm{T}_{\mathrm{RAW}}=100.86^{\circ} \mathrm{C}$

### 5.4 Preheater



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


| Component | Coal (100 kg/day) |  |  |  |
| :---: | :---: | :---: | :---: | :---: |
|  | Weight percent | $\mathrm{Kg} / \mathrm{day}$ | $\mathrm{Mol} / \mathrm{day}$ | Mol <br> fraction |
| Carbon | 70 | 70 | 5833.33 | .904 |
| $\mathrm{Al}_{2} \mathrm{O}_{3}$ | 4.5 | 4.5 | 44.12 | .007 |
| $\mathrm{SiO}_{2}$ | 7.5 | 7.5 | 125 | .019 |
| $\mathrm{Fe}_{2} \mathrm{O}_{3}$ | 9 | 9 | 56.25 | .009 |
| $\mathrm{CaO}^{2}$ | 0.9 | 0.9 | 16.07 | .002 |
| $\mathrm{MgO}^{2}$ | 0.3 | 0.3 |  |  |
| $\mathrm{SO}_{3}$ | 1.5 | 1.5 | 7.5 | .001 |
| Moisture | 6.3 | 6.3 |  |  |
| Total |  |  |  |  |

Average molecular weight of coal $\mathrm{M}_{\text {avg }}=\frac{100}{6451.02} \times \frac{\mathrm{kg}}{d a y} \times \frac{d a y}{m o l} \times \frac{1000 \mathrm{~g}}{1 \mathrm{~kg}}$

$$
=15.50 \mathrm{~g} / \mathrm{mol}
$$

Calorific heat of coal $\mathrm{Q}_{\mathrm{c}}=25.68 \mathrm{Mj} / \mathrm{kg}$

$$
\begin{aligned}
& =25.68 \times \frac{M j}{\mathrm{~kg}} \times \frac{10^{6} j}{1 \mathrm{Mj}} \times \frac{1 \mathrm{cal}}{4.18 j} \times \frac{1 \mathrm{~kg}}{1000 \mathrm{~g}} \times \frac{15.50 \mathrm{~g}}{1 \mathrm{~mol}} \\
& =95224.88 \mathrm{cal} / \mathrm{mol}
\end{aligned}
$$

So, 1 mol coal produces 95224.88 calories by combustion

$$
\mathbf{C}+\mathbf{O}_{2}=\mathbf{C O}_{2} \ldots \ldots \ldots \ldots \ldots \ldots \ldots \ldots \ldots \ldots \ldots \ldots \ldots \ldots ; \quad \Delta H_{\mathrm{r}}=-95224.88 \mathrm{cal} / \mathrm{mol}
$$

Now, $\mathbf{n}_{\text {output }}=\mathbf{n}_{\text {input }}+\mathbf{v}_{\mathbf{i} \times \boldsymbol{\varepsilon}}$
Here, $v_{i}=$ Stoichiometric Coefficient

$$
\varepsilon=\text { Extent of reaction }
$$

$\mathrm{So}, \mathbf{n C}_{(\mathbf{o})}=\mathbf{n c}_{\mathbf{C}(\mathbf{i})}+\mathbf{v C}_{\mathbf{C}} \boldsymbol{\varepsilon}$

$$
0=0.904 \mathrm{P}-(1 \times \varepsilon)[\text { Assuming } 100 \% \text { conversion of carbon }]
$$

$$
\varepsilon=0.904 \mathrm{P}
$$

$$
\mathbf{n O}_{\mathbf{O}(0)}=\mathbf{n O 2}_{\mathbf{O}(\mathrm{i})}+v_{02} \times \varepsilon
$$

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

$$
=1.096 \mathrm{P}
$$



```
\[
=0+(1 \times 0.904 \mathrm{P})
\]
\[
=0.904 \mathrm{P}
\]
```

$\mathrm{SiO}_{2}(\mathrm{~s}), \mathrm{Al}_{2} \mathrm{O}_{3}(\mathrm{~s}), \mathrm{CaCO}_{3}(\mathrm{~s}), \mathrm{Fe}_{2} \mathrm{O}_{3}(\mathrm{~s}), \mathrm{MgCO}_{3}(\mathrm{~s}), \mathrm{C}(\mathrm{s}), \mathrm{CaO}(\mathrm{s}), \mathrm{MgO}(\mathrm{s}), \mathrm{SO}_{3}(\mathrm{~g}), \quad \mathrm{H}_{2} \mathrm{O}(\mathrm{l})$ at $30^{\circ} \mathrm{C}$ and 1 atm
$\mathrm{O}_{2}(\mathrm{~g}), \mathrm{N}_{2}(\mathrm{~g}), \mathrm{CO}_{2}(\mathrm{~g})$ at $25^{\circ} \mathrm{C}$ and 1 atm

| Component | $\mathrm{n}_{\text {in }}(\mathrm{mol} /$ day $)$ | $\hat{H}_{\text {in }}(\mathrm{cal} / \mathrm{mol})$ | $\mathrm{n}_{\text {out }}(\mathrm{mol} /$ day $)$ | $\hat{\mathrm{H}}_{\text {out }}(\mathrm{cal} / \mathrm{mol})$ | Cp's ( $\frac{\text { cal }}{\text { mol } k}$ ) |
| :---: | :---: | :---: | :---: | :---: | :---: |
| $\mathrm{SiO}_{2}$ | 191.67 | $\hat{H}_{1}$ | 191.67+.019P | $\hat{H}_{9}$ |  |
| $\mathrm{Al}_{2} \mathrm{O}_{3}$ | 30.26 | $\hat{\mathrm{H}}_{2}$ | 30.26+.007P | $\hat{H}_{10}$ |  |
| $\mathrm{CaCO}_{3}$ | 765.30 | $\hat{H}_{3}$ | 765.30 | $\hat{H}_{11}$ |  |
| $\mathrm{Fe}_{2} \mathrm{O}_{3}$ | 8.19 | $\hat{\mathrm{H}}_{4}$ | 8.19+.009P | $\hat{H}_{12}$ |  |
| $\mathrm{MgCO}_{3}$ | 21.19 | $\hat{H}_{5}$ | 21.19 | $\hat{H}_{13}$ |  |
| C | .904P | 0 | - |  |  |
| $\mathrm{Al}_{2} \mathrm{O}_{3}$ | .007P | 0 | - |  |  |
| $\mathrm{SiO}_{2}$ | .019P | 0 | - |  |  |
| $\mathrm{Fe}_{2} \mathrm{O}_{3}$ | .009P | 0 | - |  |  |
| CaO | .002P | 0 | .002P | $\widehat{H}_{14}$ |  |
| MgO | .001P | 0 | .001P | $\hat{H}_{15}$ |  |
| $\mathrm{SO}_{3}$ | .002P | 0 | .002P | $\hat{H}_{16}$ |  |
| $\mathrm{H}_{2} \mathrm{O}(\mathrm{l})$ | .054P | $\hat{H}_{6}$ | - |  |  |
| $\mathrm{H}_{2} \mathrm{O}(\mathrm{v})$ | - | - | .054P | $\hat{H}_{17}$ |  |
| $\mathrm{O}_{2}$ | 2P | $\hat{H}_{7}$ | 1.096P | $\hat{H}_{18}$ |  |
| $\mathrm{N}_{2}$ | 7.52P | $\hat{H}_{8}$ | 7.52P | $\hat{H}_{19}$ |  |
| $\mathrm{CO}_{2}$ | - | - | .904P | $\hat{H}_{20}$ |  |

Now, $\mathrm{E}_{\mathrm{I}}=\sum \boldsymbol{H}=\int_{\boldsymbol{T} \boldsymbol{i}}^{\boldsymbol{T} \boldsymbol{f}} \boldsymbol{n} \times \boldsymbol{C p}(\boldsymbol{T}) \times \boldsymbol{d} \boldsymbol{T} \ldots \ldots \ldots \ldots . .(\mathrm{iii})$
Here, $\mathrm{Cp}_{\mathrm{i}}=$ specific heats of components, $\mathrm{n}=$ molar flow rate $(\mathrm{mol} / \mathrm{day}), \mathrm{T}=$ temperature $(\mathrm{K})$

## Enthalpy Calculation:

## Enthalpy of $\mathrm{SiO}_{2}$ at $\mathbf{1 0 0 . 8 6}{ }^{\mathbf{}} \mathbf{C}$ :

$\mathrm{H}_{1} \quad=$ ñ $_{\mathrm{SiO} 2} \times \int_{303.1}^{373.96}(10.87+0.008712 \mathrm{~T}) d T$

$$
=2083.45 \times(373.96-303.1)+.83 \times\left(373.96^{2}-303.10^{2}\right)
$$

$$
=187453 \mathrm{cal}
$$

## Enthalpy of $\mathrm{Al}_{2} \mathrm{O}_{3}$ at $100.86{ }^{\circ} \mathrm{C}$ :

$\mathrm{H}_{2} \quad=\stackrel{\circ}{\mathrm{n}}_{\mathrm{Al2O} 3} \times \int_{303.1}^{373.96}(22.08+0.008971 \mathrm{~T}) d T$
$=668.14 \times(373.96-303.1)+.135 \times\left(373.96^{2}-303.10^{2}\right)$
$=53821 \mathrm{cal}$

## Enthalpy of $\mathrm{CaCO}_{3}$ at $100.86{ }^{\circ} \mathrm{C}$ :

$\mathrm{H}_{3}={\stackrel{\circ}{\mathrm{naCO}} 3 \times \int_{303.1}^{373.96}(19.68+0.01189 \mathrm{~T}) d T}$
$=15061 \times(373.96-303.1)+4.5 \times\left(373.96^{2}-303.10^{2}\right)$
$=1283116 \mathrm{cal}$

## Enthalpy of $\mathrm{Fe}_{2} \mathrm{O}_{3}$ at $100.86{ }^{\circ} \mathrm{C}$ :

$\mathrm{H}_{4} \quad=\stackrel{\circ}{\mathrm{n}}_{\mathrm{Fe} 2 \mathrm{O} 3} \times \int_{303.1}^{373.96}(24.72+0.01604 \mathrm{~T}) d T$

$$
\begin{aligned}
& =202.46 \times(373.96-303.1)+.066 \times\left(373.96^{2}-303.10^{2}\right) \\
& =17513 \mathrm{cal}
\end{aligned}
$$

## Enthalpy of $\mathrm{MgCO}_{3}$ at $100.86{ }^{\mathbf{}} \mathrm{C}$ :

$$
\begin{aligned}
\mathrm{H}_{5} & ={\stackrel{\circ}{\mathrm{n} g \mathrm{CO}} 3 \times \int_{303.1}^{373.96} 16.9 d T} \\
& =358.11 \times(373.96-303.1) \\
& =25376 \mathrm{cal}
\end{aligned}
$$

## Enthalpy of $\mathrm{H}_{2} \mathrm{O}$ at $30{ }^{\circ} \mathrm{C}$ :

$\mathrm{H}_{6}=\stackrel{\circ}{\mathrm{n}}_{\mathrm{H} 2 \mathrm{O}} \times \hat{\mathrm{H}}_{6}$

$$
\begin{aligned}
& =.054 \mathrm{P} \times 40.5 \\
& =2.187 \mathrm{P} \mathrm{cal}
\end{aligned}
$$

Enthalpy of $\mathrm{O}_{\mathbf{2}}$ at $\mathbf{3 0 0 ^ { \circ }} \mathbf{C}$ :
$\mathrm{H}_{7}=\stackrel{\circ}{\mathrm{O}}_{\mathrm{O} 2} \times \hat{\mathrm{H}}_{7}$

$$
=2 \mathrm{P} \times 2026.3
$$

$$
=4053 \mathrm{P} \mathrm{cal}
$$

## Enthalpy of $\mathbf{N}_{\mathbf{2}}$ at $\mathbf{3 0 0}{ }^{\circ} \mathbf{C}$ :

$\mathrm{H}_{8} \quad=\mathrm{n}_{\mathrm{N} 2} \times \hat{\mathrm{H}}_{8}$

$$
\begin{aligned}
& =7.52 \mathrm{P} \times 1942.58 \\
& =14608 \mathrm{P} \mathrm{cal}
\end{aligned}
$$

## Enthalpy of $\mathrm{SiO}_{2}$ at $700{ }^{\circ} \mathrm{C}$ :

$$
\begin{aligned}
\mathrm{H}_{9} \quad & =\AA_{\mathrm{n}}^{\mathrm{SiO} 2} \times \int_{303.1}^{973.1}(10.87+0.008712 \mathrm{~T}) d T \\
& =(191.67+.019 \mathrm{P}) \times\left[10.87 \times(973.1-303.1)+.004 \times\left(973.1^{2}-303.1^{2}\right)\right] \\
& =2051466+203.36 \mathrm{P} \mathrm{cal}
\end{aligned}
$$

## Enthalpy of $\mathrm{Al}_{2} \mathrm{O}_{3}$ at $700{ }^{\circ} \mathrm{C}$ :

$$
\begin{aligned}
\mathrm{H}_{10} & =\stackrel{\circ}{\mathrm{n}}_{\mathrm{Al2O} 3} \times \int_{303.1}^{973.1}(22.08+0.008971 \mathrm{~T}) d T \\
& =(30.26+.007 \mathrm{P}) \times\left[22.08 \times(973.1-303.1)+.0045 \times\left(973.1^{2}-303.1^{2}\right)\right] \\
& =564087+130.5 \mathrm{P} \mathrm{cal}
\end{aligned}
$$

## Enthalpy of $\mathrm{CaCO}_{3}$ at $\mathbf{7 0 0}^{\circ} \mathrm{C}$ :

$\mathrm{H}_{11}=\stackrel{\circ}{\mathrm{n}}_{\mathrm{CaCO} 3} \times \int_{303.1}^{973.1}(19.68+0.01189 \mathrm{~T}) d T$

$$
=765.30 \times\left[19.68 \times(973.1-303.1)+.0056 \times\left(973.1^{2}-303.1^{2}\right)\right]
$$

$$
=13755427 \mathrm{cal}
$$

## Enthalpy of $\mathrm{Fe}_{2} \mathrm{O}_{3}$ at $\mathbf{7 0 0}^{\circ} \mathrm{C}$ :

$$
\begin{aligned}
\mathrm{H}_{12} & =\stackrel{\circ}{\mathrm{n}}_{\mathrm{Fe} 2 \mathrm{O} 3} \times \int_{303.1}^{973.1}(24.72+0.01604 \mathrm{~T}) d T \\
& =(8.19+.009 \mathrm{P}) \times\left[24.72 \times(973.1-303.1)+.008 \times\left(973.1^{2}-303.1^{2}\right)\right] \\
& =191669+210.6 \mathrm{P} \mathrm{cal}
\end{aligned}
$$

## Enthalpy of $\mathbf{M g C O}_{3}$ at $\mathbf{7 0 0}^{\circ} \mathbf{C}$ :

$$
\begin{aligned}
\mathrm{H}_{13} & =\stackrel{\circ}{\mathrm{nggCO}} \times \int_{303.1}^{973.1} 16.9 d T \\
& =21.19 \times[16.9 \times(973.1-303.1)] \\
& =239934 \mathrm{cal}
\end{aligned}
$$

## Enthalpy of CaO at $700{ }^{\circ} \mathrm{C}$ :

$$
\begin{aligned}
\mathrm{H}_{14} \quad & =\stackrel{\circ}{\mathrm{CaO}} \times \int_{303.1}^{973.1}(10.00+0.00484 \mathrm{~T}) d T \\
& =.002 \mathrm{P} \times\left[10.00 \times(973.1-303.1)+.0024 \times\left(973.1^{2}-303.1^{2}\right)\right] \\
& =17.50 \mathrm{P} \mathrm{cal}
\end{aligned}
$$

## Enthalpy of $\mathbf{M g O}$ at $700^{\circ} \mathrm{C}$ :

$$
\begin{aligned}
\mathrm{H}_{15} & =\AA_{\mathrm{MgO}} \times \int_{303.1}^{973.1}(10.86+0.0012 \mathrm{~T}) d T \\
& =.001 \mathrm{P} \times\left[10.86 \times(973.1-303.1)+.0006 \times\left(973.1^{2}-303.1^{2}\right)\right] \\
& =7.79 \mathrm{P} \mathrm{cal}
\end{aligned}
$$

## Enthalpy of $\mathrm{SO}_{3}$ at $700{ }^{\circ} \mathrm{C}$ :

$$
\begin{aligned}
\mathrm{H}_{16} & =\AA_{\mathrm{SO} 3} \times \int_{303.1}^{973.1}(7.7+0.0053 \mathrm{~T}) d T \\
& =.002 \mathrm{P} \times\left[7.7 \times(973.1-303.1)+.0053 \times\left(973.1^{2}-303.1^{2}\right)\right] \\
& =19.38 \mathrm{P} \mathrm{cal}
\end{aligned}
$$

## Enthalpy of $\mathbf{H}_{2} \mathrm{O}$ at $\mathbf{7 0 0}^{\circ} \mathrm{C}$ :

$$
\begin{aligned}
\mathrm{H}_{17} & =\stackrel{\circ}{\mathrm{n}} \mathrm{H} 2 \mathrm{O} \times \hat{\mathrm{H}}_{17} \\
& =.054 \mathrm{P} \times 5962 \\
& =322 \mathrm{P} \mathrm{cal}
\end{aligned}
$$

## Enthalpy of $\mathrm{O}_{2}$ at $700{ }^{\circ} \mathrm{C}$ :

$$
\begin{aligned}
\mathrm{H}_{18} & =\text { no }_{\mathrm{O} 2} \times \hat{\mathrm{H}}_{18} \\
& =1.096 \mathrm{P} \times 5230 \\
& =5732 \mathrm{P} \mathrm{cal}
\end{aligned}
$$

## Enthalpy of $\mathbf{N}_{\mathbf{2}}$ at $\mathbf{7 0 0}^{\circ} \mathrm{C}$ :

$$
\begin{aligned}
\mathrm{H}_{19} & =\AA_{\mathrm{n} 2} \times \hat{\mathrm{H}}_{19} \\
& =7.52 \mathrm{P} \times 4925 \\
& =37036 \mathrm{P} \mathrm{cal}
\end{aligned}
$$

## Enthalpy of $\mathrm{CO}_{2}$ at $\mathbf{7 0 0}^{\circ} \mathrm{C}$ :

$$
\begin{aligned}
\mathrm{H}_{20} & =\stackrel{\circ}{\mathrm{n} O} 2 \times \hat{\mathrm{H}}_{20} \\
& =.904 \mathrm{P} \times 7626 \\
& =6894 \mathrm{P} \mathrm{cal}
\end{aligned}
$$

Total inlet enthalpies :

$$
\begin{aligned}
\sum n_{\text {in }} H_{\text {in }} & =\mathrm{H}_{1}+\ldots \ldots \ldots \ldots \ldots \ldots \ldots \ldots+\mathrm{H}_{8} \\
& =1567279+18663 \mathrm{P}
\end{aligned}
$$

Total Outlet Enthalpies:

$$
\begin{aligned}
\sum n_{\text {out }} H_{\text {out }} & =\mathrm{H}_{9}+\ldots \ldots \ldots \ldots \ldots \ldots \ldots \ldots \mathrm{H}_{20} \\
& =16802583+50573 \mathrm{P} \\
\Delta \boldsymbol{H} & =\sum \boldsymbol{n}_{\text {out }} \boldsymbol{H}_{\text {out }}-\sum \boldsymbol{n}_{\text {in }} \boldsymbol{H}_{\text {in }} \\
& =15235304+31910 \mathrm{P}
\end{aligned}
$$

## Amount of Coal Required in the Preheater:

$$
\begin{aligned}
& 95224.88 \mathrm{P}=15235304+31910 \mathrm{P} \\
& \mathrm{P}=240.63 \mathrm{~mol} \text { of coal } / \mathrm{day} \\
&=3730 \mathrm{~g} \mathrm{coal} / \mathrm{day}=3.73 \mathrm{~kg} / \mathrm{day}
\end{aligned}
$$

Air Inlet $\left(300^{\circ} \mathrm{C}\right)=9.52 \mathrm{P} \mathrm{mol} /$ day $=9.52 \times 240.63 \times \frac{.0821 \times 573}{3.5} \times \frac{1 \mathrm{~m}^{3}}{1000 \mathrm{~L}}=30.79 \mathrm{~m}^{3} / \mathrm{day}$
Exhaust gas:

```
n}\textrm{O}2=1.06 P = 1.06 < 240.63 mol/day = 255.07 mol O2/day
n}\mp@subsup{n}{N2}{2}=7.52\textrm{P}=7.52\times240.63\textrm{mol}/\mathrm{ day }=1809 mol N2 / day
n
```

| Component | $\mathrm{n}_{\text {in }}(\mathrm{mol} /$ day $)$ | $\hat{H}_{\text {in }}(\mathrm{cal} / \mathrm{mol})$ | $\mathrm{n}_{\text {out }}(\mathrm{mol} /$ day $)$ | $\hat{\mathrm{H}}_{\text {out }}(\mathrm{cal} / \mathrm{mol})$ | $\text { Cp's }\left(\frac{c a l}{\operatorname{mol} k}\right)$ |
| :---: | :---: | :---: | :---: | :---: | :---: |
| $\mathrm{SiO}_{2}$ | 191.67 | $\hat{H}_{1}$ | 196.24 | $\hat{H}_{9}$ |  |
| $\mathrm{Al}_{2} \mathrm{O}_{3}$ | 30.26 | $\hat{\mathrm{H}}_{2}$ | 31.94 | $\hat{H}_{10}$ |  |
| $\mathrm{CaCO}_{3}$ | 765.30 | $\hat{\mathrm{H}}_{3}$ | 765.30 | $\hat{H}_{11}$ |  |
| $\mathrm{Fe}_{2} \mathrm{O}_{3}$ | 8.19 | $\hat{\mathrm{H}}_{4}$ | 10.36 | $\hat{\mathrm{H}}_{12}$ |  |
| $\mathrm{MgCO}_{3}$ | 21.19 | $\hat{H}_{5}$ | 21.19 | $\hat{H}_{13}$ |  |
| C | 217.53 | 0 | - |  |  |
| $\mathrm{Al}_{2} \mathrm{O}_{3}$ | 1.68 | 0 | - |  |  |
| $\mathrm{SiO}_{2}$ | 4.57 | 0 | - |  |  |
| $\mathrm{Fe}_{2} \mathrm{O}_{3}$ | 2.17 | 0 | - |  |  |
| CaO | . 48 | 0 | . 48 | $\hat{H}_{14}$ |  |
| MgO | . 24 | 0 | . 24 | $\hat{H}_{15}$ |  |
| $\mathrm{SO}_{3}$ | . 48 | 0 | . 48 | $\hat{H}_{16}$ |  |
| $\mathrm{H}_{2} \mathrm{O}(\mathrm{l})$ | 12.99 | $\hat{\mathrm{H}}_{6}$ | - |  |  |
| $\mathrm{H}_{2} \mathrm{O}$ (v) | - | - | 12.99 | $\hat{H}_{17}$ |  |
| $\mathrm{O}_{2}$ | 481.26 | $\hat{H}_{7}$ | 263.73 | $\hat{H}_{18}$ |  |
| $\mathrm{N}_{2}$ | 1809.54 | $\hat{H}_{8}$ | 1809.54 | $\hat{H}_{19}$ |  |
| $\mathrm{CO}_{2}$ | - | - | 217.53 | $\hat{H}_{20}$ |  |

### 5.5 Rotary Kiln



## Kiln Feed Composition:

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


| Components | $\mathrm{n}_{\text {in }}(\mathrm{mol} /$ day $)$ | $\hat{H}_{\text {in }}(\mathrm{cal} / \mathrm{mol})$ | n $_{\text {out }}(\mathrm{mol} /$ day $)$ | $\begin{gathered} \hat{\mathrm{H}}_{\text {out }} \\ (\mathrm{cal} / \mathrm{mol}) \end{gathered}$ | Cp's ( $\frac{\mathrm{cal}}{\mathrm{molk}}$ ) |
| :---: | :---: | :---: | :---: | :---: | :---: |
| SiO 2 | 196.24 | H11 | 0 |  |  |
| Al2O3 | 31.94 | H2 | 0 |  |  |
| CaCO 3 | 765.30 | H3 | 0 |  |  |
| Fe2O3 | 10.36 | H4 | 0 |  |  |
| MgCO3 | 21.19 | H5 | 0 |  |  |
| CaO | . 48 | Ĥ6 | 130.57 | Ĥ17 |  |
| MgO | . 24 | H7 | 21.43 | H118 |  |
| C | .904Q | 0 | 0 |  |  |
| Al2O3 | .007Q | Ĥ8 | .007Q | H19 |  |
| SiO 2 | .019Q | H9 | .019Q | H20 |  |
| Fe2O3 | .009Q | H10 | .009Q | H21 |  |
| CaO | .002Q | Ĥ11 | 0 |  |  |
| MgO | .001Q | Ĥ12 | 0 |  |  |


| SO3 | .002Q | H13 | .002Q | Ĥ22 |  |
| :---: | :---: | :---: | :---: | :---: | :---: |
| H2O(1) | .054Q | Ĥ14 |  |  |  |
| H 2 O (v) | - | - | .054Q | H23 |  |
| O2 | 2Q | H15 | 1.06 Q | H24 |  |
| N2 | 7.52Q | H16 | 7.52Q | H25 |  |
| CO 2 | - |  | 786.49+.94Q | H26 |  |
| C2S |  |  | 60.19 | Ĥ27 |  |
| C3S |  |  | 136.04 | H28 |  |
| C3A |  |  | 21.58 | Ĥ29 |  |
| C4AF |  |  | 10.36 | Ĥ30 |  |

Now, $\mathrm{E}_{\mathrm{I}}=\sum \boldsymbol{H}=\int_{\boldsymbol{T} \boldsymbol{i}}^{\boldsymbol{T} \boldsymbol{f}} \boldsymbol{n} \times \boldsymbol{C p}(\boldsymbol{T}) \times \boldsymbol{d} \boldsymbol{T} \ldots \ldots \ldots \ldots . .(\mathrm{iii})$
Here, $\mathrm{Cp}_{\mathrm{i}}=$ specific heats of components, $\mathrm{n}=$ molar flow rate $(\mathrm{mol} / \mathrm{day}), \mathrm{T}=$ temperature $(\mathrm{K})$

## Enthalpy Calculation :

## Enthalpy of $\mathrm{SiO}_{\mathbf{2}}$ at $700{ }^{\circ} \mathrm{C}$ :

$\mathrm{H}_{1}=\stackrel{\circ}{\mathrm{n}}_{\mathrm{SiO} 2} \times\left[\left(\Delta H_{f}^{o}\right) \mathrm{SiO} 2+\int_{298.1}^{973.1}(10.87+0.008712 \mathrm{~T}) d T\right.$

$$
=196.24 \times\left[-202620+10.87 \times(973.1-298.1)+.004 \times\left(973.1^{2}-298.1^{2}\right)\right]
$$

$$
=-37.65 \times 10^{6} \mathrm{cal}
$$

## Enthalpy of $\mathrm{Al}_{2} \mathrm{O}_{3}$ at $700{ }^{\circ} \mathrm{C}$ :

$\mathrm{H}_{2}=\stackrel{\circ}{\mathrm{n}}_{\mathrm{Al2O} 3} \times\left[\left(\Delta H_{f}^{o}\right) \mathrm{Al2O}^{+}+\int_{298.1}^{973.1}(22.08+0.008971 \mathrm{~T}) d T\right.$

$$
=31.94 \times\left[-400000+22.08 \times(973.1-298.1)+.0045 \times\left(973.1^{2}-298.1^{2}\right)\right]
$$

$$
=-12.18 \times 10^{6} \mathrm{cal}
$$

## Enthalpy of $\mathrm{CaCO}_{3}$ at $700{ }^{\circ} \mathrm{C}$ :

$\mathrm{H}_{3}=\mathrm{n}_{\mathrm{CaCO} 3} \times\left[\left(\Delta H_{f}^{o}\right) \mathrm{CaCO}^{+}+\int_{298.1}^{973.1}(19.68+0.01189 \mathrm{~T}) d T\right.$

$$
=765.30 \times\left[-298500+19.68 \times(973.1-298.1)+.0056 \times\left(973.1^{2}-298.1^{2}\right)\right]
$$

$$
=-214.60 \times 10^{6} \mathrm{cal}
$$

## Enthalpy of $\mathrm{Fe}_{2} \mathrm{O}_{3}$ at $\mathbf{7 0 0}^{\circ} \mathrm{C}$ :

$\mathrm{H}_{4} \quad=\stackrel{\circ}{\mathrm{n}}_{\mathrm{Fe} 2 \mathrm{O} 3} \times\left[\left(\Delta H_{f}^{o}\right) \mathrm{Fe} 2 \mathrm{O} 3+\int_{298.1}^{973.1}(24.72+0.01604 \mathrm{~T}) d T\right.$
$=10.36 \times\left[-198500+24.72 \times(973.1-298.1)+.008 \times\left(973.1^{2}-298.1^{2}\right)\right]$
$=-1.81 \times 10^{6} \mathrm{cal}$
Enthalpy of $\mathrm{MgCO}_{3}$ at $700{ }^{\circ} \mathrm{C}$ :
$\mathrm{H}_{5} \quad=\stackrel{\circ}{\mathrm{n} g \mathrm{MgO}} \times\left[\left(\Delta H_{f}^{o}\right) \mathrm{MgCO}+\int_{298.1}^{973.1} 16.9 d T\right.$
$=21.19 \times[-261700+16.9 \times(973.1-298.1)]$
$=-5.3 \times 10^{6} \mathrm{cal}$

## Enthalpy of $\mathbf{C a O}$ at $700{ }^{\circ} \mathrm{C}$ :

$\mathrm{H}_{6}=\stackrel{\circ}{\mathrm{n}}_{\mathrm{CaO}} \times\left[\left(\Delta H_{f}^{O}\right) \mathrm{CaO}+\int_{298.1}^{973.1}(10.00+0.00484 \mathrm{~T}) d T\right.$

$$
\begin{aligned}
& =.48 \times\left[-151700+10 \times(973.1-298.1)+.0024 \times\left(973.1^{2}-298.1^{2}\right)\right] \\
& =-68.59 \times 10^{3} \mathrm{cal}
\end{aligned}
$$

## Enthalpy of $\mathbf{M g O}$ at $700^{\circ} \mathrm{C}$ :

$\mathrm{H}_{7}=\stackrel{\circ}{\mathrm{n} g \mathrm{O}} \times\left[\left(\Delta H_{f}^{o}\right) \mathrm{MgO}+\int_{298.1}^{973.1}(10.86+0.0012 \mathrm{~T}) d T\right.$
$=.24 \times\left[-143840+10.86 \times(973.1-298.1)+.0006 \times\left(973.1^{2}-298.1^{2}\right)\right]$
$=-32.63 \times 10^{3} \mathrm{cal}$
Enthalpy of $\mathrm{Al}_{2} \mathrm{O}_{3}$ at $\mathbf{2 5}^{\circ} \mathrm{C}$ :
$\mathrm{H}_{8} \quad=\mathrm{n}_{\mathrm{Al} 2 \mathrm{O} 3} \times\left(\Delta H_{f}^{0}\right) \mathrm{Al2O3}^{2}$
$=.007 \mathrm{Q} \times(-400000)$
$=-2800 \mathrm{Q} \mathrm{cal}$
Enthalpy of $\mathrm{SiO}_{2}$ at $\mathbf{2 5}^{\circ} \mathrm{C}$ :
$\mathrm{H}_{9} \quad=\stackrel{\circ}{\mathrm{n}}_{\mathrm{SiO} 2} \times\left(\Delta H_{f}^{o}\right)_{\mathrm{SiO} 2}$
$=.019 \mathrm{Q} \times(-202620)$
$=-3849 \mathrm{Q}$ cal
Enthalpy of $\mathrm{Fe}_{2} \mathrm{O}_{3}$ at $\mathbf{2 5}^{\circ} \mathrm{C}$ :
$\mathrm{H}_{10} \quad=\stackrel{\circ}{\mathrm{n}}_{\mathrm{Fe} 2 \mathrm{O} 3} \times\left(\Delta H_{f}^{o}\right) \mathrm{Fe} 2 \mathrm{O} 3$
$=.009 \mathrm{Q} \times(-198500)$
$=-1786 \mathrm{Q} \mathrm{cal}$

## Enthalpy of $\mathbf{C a O}$ at $25^{\circ} \mathrm{C}$ :

$\mathrm{H}_{11}=\mathrm{n}_{\mathrm{CaO}} \times\left(\Delta H_{f}^{o}\right) \mathrm{CaO}$
$=.002 \mathrm{Q} \times(-151700)$
$=-303 \mathrm{Q} \mathrm{cal}$

## Enthalpy of $\mathbf{M g O}$ at $\mathbf{2 5}^{\circ} \mathrm{C}$ :

$$
\begin{aligned}
\mathrm{H}_{12} & =\mathrm{n}_{\mathrm{MgO}} \times\left(\Delta H_{f}^{o}\right) \mathrm{MgO} \\
& =.001 \mathrm{Q} \times(-143840) \\
& =143 \mathrm{Q} \text { cal }
\end{aligned}
$$

## Enthalpy of $\mathbf{H}_{\mathbf{2}} \mathrm{O}$ at $25^{\circ} \mathrm{C}$ :

$\mathrm{H}_{14}=$ ñ $_{\mathrm{H} 2 \mathrm{O}} \times\left(\Delta H_{f}^{o}\right)$ H2O
$=.054 \mathrm{Q} \times(-68380)$

$$
=3692 \mathrm{Q} \mathrm{cal}
$$

## Enthalpy of $\mathrm{O}_{\mathbf{2}}$ at $\mathbf{3 0 0}^{\circ} \mathrm{C}$ :

$\mathrm{H}_{15}=\mathrm{n}_{\mathrm{O} 2} \times \hat{\mathrm{H}}_{15}$

$$
\begin{aligned}
& =2 \mathrm{Q} \times 2026.3 \\
& =4053 \mathrm{Q} \mathrm{cal}
\end{aligned}
$$

## Enthalpy of $\mathbf{N}_{\mathbf{2}}$ at $\mathbf{3 0 0}{ }^{\circ} \mathbf{C}$ :

$\mathrm{H}_{16}=\stackrel{\circ}{\mathrm{n}}_{\mathrm{N} 2} \times \hat{\mathrm{H}}_{16}$

$$
\begin{aligned}
& =7.52 \mathrm{Q} \times 1942.58 \\
& =14608 \mathrm{Q} \text { cal }
\end{aligned}
$$

## Enthalpy of $\mathbf{C a O}$ at $1400{ }^{\circ} \mathrm{C}$ :

$\mathrm{H}_{17}=\AA_{\mathrm{n}}^{\mathrm{CaO}} \times\left[\left(\Delta H_{f}^{o}\right) \mathrm{CaO}+\int_{298.1}^{1673.1}(10.00+0.00484 \mathrm{~T}) d T\right.$

$$
\begin{aligned}
& =130.57 \times\left[-151700+10 \times(1673.1-298.1)+.0024 \times\left(1673.1^{2}-298.1^{2}\right)\right] \\
& =-17.16 \times 10^{6} \mathrm{cal}
\end{aligned}
$$

## Enthalpy of $\mathbf{M g O}$ at $\mathbf{1 4 0 0}^{\circ} \mathrm{C}$ :

$\mathrm{H}_{18}=\stackrel{\circ}{\mathrm{n}}_{\mathrm{MgO}} \times\left[\left(\Delta H_{f}^{o}\right) \mathrm{MgO}+\int_{298.1}^{1673.1}(10.86+0.0012 \mathrm{~T}) d T\right.$
$=21.43 \times\left[-143840+10.86 \times(1673.1-298.1)+.0006 \times\left(1673.1^{2}-298.1^{2}\right)\right]$
$=-2.72 \times 10^{6} \mathrm{cal}$

## Enthalpy of $\mathrm{Al}_{2} \mathrm{O}_{3}$ at $1400{ }^{\circ} \mathrm{C}$ :

$\mathrm{H}_{19}=\stackrel{\circ}{\mathrm{n}}_{\mathrm{Al2O} 3} \times\left[\left(\Delta H_{f}^{o}\right) \mathrm{Al2O3}+\int_{298.1}^{1673.1}(22.08+0.008971 \mathrm{~T}) d T\right.$

$$
=.007 \mathrm{Q} \times\left[-400000+22.08 \times(1673.1-298.1)+.0045 \times\left(1673.1^{2}-298.1^{2}\right)\right]
$$

$$
=-2502 \mathrm{Q} \mathrm{cal}
$$

## Enthalpy of $\mathrm{SiO}_{2}$ at $1400{ }^{\circ} \mathrm{C}$ :

$\mathrm{H}_{20}=\stackrel{\circ}{\mathrm{n}}_{\mathrm{SiO} 2} \times\left[\left(\Delta H_{f}^{o}\right) \mathrm{SiO} 2+\int_{298.1}^{1673.1}(10.87+0.008712 \mathrm{~T}) d T\right.$
$=.019 \mathrm{Q} \times\left[-202620+10.87 \times(1673.1-298.1)+.004 \times\left(1673.1^{2}-298.1^{2}\right)\right]$
$=-3359 \mathrm{Q} \mathrm{cal}$
Enthalpy of $\mathrm{Fe}_{2} \mathrm{O}_{3}$ at $\mathbf{1 4 0 0}^{\circ} \mathrm{C}$ :

$$
\begin{aligned}
\mathrm{H}_{21} & =\stackrel{\circ}{\mathrm{n}}_{\mathrm{Fe} 2 \mathrm{O} 3} \times\left[\left(\Delta H_{f}^{o}\right) \mathrm{Fe} 2 \mathrm{O} 3+\int_{298.1}^{1673.1}(24.72+0.01604 \mathrm{~T}) d T\right. \\
& =.009 \mathrm{Q} \times\left[-198500+24.72 \times(1673.1-298.1)+.008 \times\left(1673.1^{2}-298.1^{2}\right)\right] \\
& =-1285 \mathrm{Q} \mathrm{cal}
\end{aligned}
$$

## Enthalpy of $\mathbf{H}_{\mathbf{2}} \mathrm{O}$ at $\mathbf{1 4 0 0}^{\mathbf{}} \mathbf{C}$ :

$$
\begin{aligned}
\mathrm{H}_{23} & =\mathrm{n}_{\mathrm{H} 2 \mathrm{O}} \times\left[\left(\Delta H_{f}^{o}\right) \mathrm{H} 2 \mathrm{O}+\hat{\mathrm{H}}_{23}\right] \\
& =.054 \mathrm{Q} \times[-68380+13456] \\
& =-2965 \mathrm{Q} \mathrm{cal}
\end{aligned}
$$

Enthalpy of $\mathrm{O}_{\mathbf{2}}$ at $\mathbf{1 4 0 0}^{\boldsymbol{\circ}} \mathrm{C}$ :

$$
\begin{aligned}
\mathrm{H}_{24} & =\stackrel{\circ}{\mathrm{n}} 22 \times \hat{\mathrm{H}}_{24} \\
& =1.06 \mathrm{Q} \times 11260 \\
& =11935 \mathrm{Q} \mathrm{cal}
\end{aligned}
$$

## Enthalpy of $\mathbf{N}_{\mathbf{2}}$ at $\mathbf{1 4 0 0}^{\mathbf{\circ}} \mathrm{C}$ :

$$
\begin{aligned}
\mathrm{H}_{25} & =\stackrel{\circ}{\mathrm{n}} 2 \times \hat{\mathrm{H}}_{25} \\
& =7.52 \mathrm{Q} \times 10648 \\
& =80072 \mathrm{Q} \text { cal }
\end{aligned}
$$

## Enthalpy of $\mathrm{CO}_{2}$ at $\mathbf{1 4 0 0}^{\mathbf{\circ}} \mathrm{C}$ :

$$
\begin{aligned}
\mathrm{H}_{26} & ={ }_{\mathrm{n}}^{\mathrm{CO} 2} 2 \times\left[\left(\Delta H_{f}^{o}\right) \mathrm{CO} 2+\hat{\mathrm{H}}_{26}\right] \\
& =(786.49+.94 \mathrm{Q}) \times(-94138+17198) \\
& =-60.51 \times 10^{6}-72323 \mathrm{Q} \mathrm{cal}
\end{aligned}
$$

Heat of formation of clinker content:
$\left(\Delta H_{f}^{o}\right){ }_{\mathrm{C} 2 \mathrm{~S}}=-552033 \mathrm{cal} / \mathrm{mol}$
$\left(\Delta H_{f}^{o}\right)_{\mathrm{C3S}}=-700765 \mathrm{cal} / \mathrm{mol}$
$\left(\Delta H_{f}^{o}\right)_{\text {C3A }}=-858516 \mathrm{cal} / \mathrm{mol}$
$\left(\Delta H_{f}^{o}\right)_{\mathrm{C} 4 \mathrm{AF}}=-1218394 \mathrm{cal} / \mathrm{mol}$
$\left(\Delta H_{f}^{o}\right)_{\text {inert }}=-151700 \mathrm{cal} / \mathrm{mol}$
$\left(\Delta H_{f}^{O}\right)_{\text {Clinker }}=-699952 \mathrm{cal} / \mathrm{mol}$

## Average Molecular weight of Clinker $=220 \mathrm{~g} / \mathrm{mole}$

## Specific heat of Cement:

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

## Enthalpy of Clinker at $1400{ }^{\circ} \mathrm{C}$ :

$$
\begin{aligned}
\mathrm{H}_{27} & =\stackrel{\circ}{\mathrm{n}}_{\text {clinker }} \times\left[\left(\Delta H_{f}^{o}\right){ }_{\text {Clinker }}+\int_{298.1}^{1673.1}(48.42) d T\right. \\
& =380 \times[-699952+48.42 \times(1673.1-298.1)] \\
& =-240.68 \times 10^{6} \mathrm{cal}
\end{aligned}
$$

Total inlet enthalpies:

$$
\begin{aligned}
\sum n_{i n} H_{i n} & =\mathrm{H}_{1}+\ldots \ldots \ldots \ldots \ldots \ldots \ldots \ldots+\mathrm{H}_{16} \\
& =-271.64 \times 10^{6}-3123 \mathrm{Q}
\end{aligned}
$$

Total Outlet Enthalpies:

$$
\begin{aligned}
& \begin{aligned}
\sum n_{\text {out }} H_{\text {out }} & =\mathrm{H}_{17}+\ldots \ldots \ldots \ldots \ldots \ldots \ldots \ldots \mathrm{H}_{27} \\
& =-321.07 \times 10^{6}-174441 \mathrm{Q} \\
\Delta \boldsymbol{H} & =\sum \boldsymbol{n}_{\text {out }} \boldsymbol{H}_{\text {out }}-\sum \boldsymbol{n}_{\text {in }} \boldsymbol{H}_{\text {in }} \\
= & -49.43 \times 10^{6}-171318 \mathrm{P}
\end{aligned}
\end{aligned}
$$

## Amount of Coal Required in the kiln:

$$
\begin{aligned}
-95224.88 \mathrm{P} & =-49.43 \times 10^{6}-171318 \mathrm{P} \\
\mathrm{P} & =649.59 \mathrm{~mol} \text { of coal } / \text { day } \\
& =10068 \mathrm{~g} \mathrm{coal} / \text { day }=10.068 \mathrm{~kg} / \text { day }
\end{aligned}
$$

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

## Exhaust gas:

$$
\begin{aligned}
& \mathrm{n}_{\mathrm{O} 2}=1.06 \mathrm{P}=1.06 \times 649,59 \mathrm{~mol} / \mathrm{day}=688.57 \mathrm{~mol} \mathrm{O}_{2} / \text { day } \\
& \mathrm{n}_{\mathrm{N} 2}=7.52 \mathrm{P}=7.52 \times 649.59 \mathrm{~mol} / \mathrm{day}=4884 \mathrm{~mol} \mathrm{~N}_{2} / \text { day } \\
& \mathrm{n}_{\mathrm{CO} 2}=.94 \mathrm{P}=.94 \times 649.59 \mathrm{~mol} / \text { day }=610 \mathrm{~mol} \mathrm{CO}_{2} / \text { day }
\end{aligned}
$$

### 5.6 Clinker Cooler



For, Coal input in Rotary kiln $\mathrm{Q}=10.068 \mathrm{~kg} / \mathrm{day}=659.75 \mathrm{~mol} /$ day
Combustible Carbon in coal $=.904 \times 659.75=587.23 \mathrm{~mol} /$ day
$\mathrm{O}_{2}$ required $\quad=2 \mathrm{P} \mathrm{mol} /$ day

$$
=1174 \mathrm{~mol} / \mathrm{day}
$$

$\mathrm{N}_{2}$ in air

$$
\begin{aligned}
& =7.52 \mathrm{P} \\
& =4416 \mathrm{~mol} / \mathrm{day}
\end{aligned}
$$

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

$$
=5590 \mathrm{~mol} / \text { day }
$$

For, Coal input in Preheater $\mathrm{P}=3.73 \mathrm{~kg} /$ day $=240.63 \mathrm{~mol} /$ day
Combustible Carbon in coal $=.904 \times 240.63=217.53 \mathrm{~mol} /$ day
$\mathrm{O}_{2}$ required

$$
\begin{aligned}
& =2 \mathrm{Pmol} / \mathrm{day} \\
& =435 \mathrm{~mol} / \mathrm{day}
\end{aligned}
$$

$\mathrm{N}_{2}$ in air

$$
=7.52 \mathrm{P}
$$

$$
=1635 \mathrm{~mol} / \mathrm{day}
$$

Air inlet in Preheater $\left(300^{\circ} \mathrm{C}\right) \quad=435+1635$

$$
=2070 \mathrm{~mol} / \mathrm{day}
$$

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

Total Air inlet in cooler $=2070+5590=7660 \mathrm{~mol} /$ day

$$
\begin{aligned}
& =7660 \frac{\mathrm{~mol}}{\mathrm{day}} \times \frac{28.84 \mathrm{~g}}{\mathrm{~mol}} \times \frac{1 \mathrm{~kg}}{1000 \mathrm{~g}} \times \frac{1 \mathrm{m3}}{1.225 \mathrm{~kg}} \\
& =180 \mathrm{~m}^{3} / \text { day air }
\end{aligned}
$$

Specific Heat of air, $\mathrm{C}_{\mathrm{p}}=6.9 \mathrm{cal} / \mathrm{mol} . \mathrm{k}$
According to Heat balance,

$$
\left[n^{\circ} \times \mathrm{C}_{\mathrm{p}} \times\left(\mathrm{T}_{\text {in }}-\mathrm{T}_{\text {out }}\right)\right] \text { clinker }=\left[\mathrm{n}^{\circ} \times \mathrm{C}_{\mathrm{p}} \times\left(\mathrm{T}_{\text {out }}-\mathrm{T}_{\text {in }}\right)\right] \text { air }
$$

Or, $380 \times 48.42 \times(1400-150)=7660 \times 6.9 \times($ Tout -25$)$
Tout $=460^{\circ} \mathrm{C}$

### 5.7 Scale up value

For Preheater scale up value:
From material balance, scale up factor is $65.53 \mathrm{ton} / \mathrm{kg}$
So, Coal required, $\mathrm{P}=65.53 \frac{\mathrm{ton}}{\mathrm{kg}} \times 3.73 \frac{\mathrm{~kg}}{\mathrm{day}}=244.43$ ton $/$ day
Again, Coal Required $\mathrm{P}=244.43$ ton $/$ day $=\frac{244.43 \text { ton }}{\text { day }} \times \frac{10^{6} \mathrm{~g}}{1 \text { ton }} \times \frac{1 \mathrm{~mol}}{15.5 \mathrm{~g}}=15.77 \times 10^{6} \mathrm{~mol} / \mathrm{day}$ Air Inlet $\left(300^{\circ} \mathrm{C}\right)=9.52 \mathrm{P} \mathrm{mol} /$ day $=150.13 \times 10^{6} \times \frac{.0821 \times 573}{3.5} \times \frac{1 m^{3}}{1000 \mathrm{~L}}=2.018 \times 10^{6} \mathrm{~m}^{3} /$ day Exhaust gas: $\left(700^{\circ} \mathrm{C}\right)$

$$
\begin{aligned}
& \mathrm{n}_{\mathrm{O} 2}=1.06 \mathrm{P}=1.06 \times 15.77 \times 10^{6} \mathrm{~mol} / \text { day }=16.71 \times 10^{6} \mathrm{~mol} \mathrm{O} \\
& 2
\end{aligned} \text { day } 0 \text {. }
$$

$\stackrel{\circ}{\mathrm{n}}_{\mathrm{N} 2}=7.52 \mathrm{P}=7.52 \times 15.77 \times 10^{6} \mathrm{~mol} /$ day $=118.59 \times 10^{6} \mathrm{~mol} \mathrm{~N}_{2} /$ day

$$
=3.32 \times 10^{6} \mathrm{~kg} / \mathrm{day}
$$

$\stackrel{\mathrm{n}}{\mathrm{CO} 2}=.94 \mathrm{P}=.94 \times 15.77 \times 10^{6} \mathrm{~mol} /$ day $=14.82 \times 10^{6} \mathrm{~mol} \mathrm{CO}_{2} /$ day

$$
=.65 \times 10^{6} \mathrm{~kg} / \mathrm{day}
$$

ñsO2 $^{\text {So }}=.002 \mathrm{P}=.002 \times 15.77 \times 10^{6} \mathrm{~mol} / \mathrm{day}=.03 \times 10^{6} \mathrm{~mol} \mathrm{SO}_{2} /$ day

$$
=2.02 \times 10^{3} \mathrm{~kg} / \mathrm{day}
$$

$\check{n}_{\mathrm{n} 2 \mathrm{O}}=.054 \mathrm{P}=.054 \times 15.77 \times 10^{6} \mathrm{~mol} /$ day $=.85 \times 10^{6} \mathrm{~mol} \mathrm{H}_{2} \mathrm{O} /$ day

$$
=15.32 \times 10^{3} \mathrm{~kg} / \mathrm{day}
$$

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

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

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

$$
=4.55 \times 10^{6} \mathrm{~kg} / \mathrm{day}
$$

Molar amount $=\left(16.71 \times 10^{6}+\right.$

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

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

Amount of $\mathrm{CO}_{2}=14.82 \times 10^{6} \mathrm{~mol} \mathrm{CO}_{2} /$ day

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

Amount of $\mathrm{SO}_{2}=0.03 \times 10^{6} \mathrm{~mol} \mathrm{SO}_{2} /$ day

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

Amount of dust $=7543 \mathrm{mg} / \mathrm{m}^{3}$
Total $=111 \times 10^{2} \mathrm{mg} / \mathrm{m}^{3}$

## For Kiln scale up value:

From material balance, scale up factor is 65.53 ton $/ \mathrm{kg}$
So, Coal required, $\mathrm{P}=65.53 \frac{\mathrm{ton}}{\mathrm{kg}} \times 10.068 \frac{\mathrm{~kg}}{\mathrm{day}}=659.75 \mathrm{ton} / \mathrm{day}$
Coal Required $\mathrm{Q}=659.75$ ton/day $=\frac{659.75 \mathrm{ton}}{\text { day }} \times \frac{10^{6} \mathrm{~g}}{1 \text { ton }} \times \frac{1 \mathrm{~mol}}{15.5 \mathrm{~g}}=41.90 \times 10^{6} \mathrm{~mol} /$ day
Air Inlet $\left(300^{\circ} \mathrm{C}\right)=9.52 \mathrm{P} \mathrm{mol} /$ day $=398.88 \times 10^{6} \times \frac{.0821 \times 573}{3.5} \times \frac{1 \mathrm{~m}^{3}}{1000 \mathrm{~L}}=5.36 \times 10^{6} \mathrm{~m}^{3} /$ day

Exhaust gas: $\left(700^{\circ} \mathrm{C}\right)$
$\mathrm{n}_{\mathrm{O} 2}=1.06 \mathrm{P}=1.06 \times 41.90 \times 10^{6} \mathrm{~mol} /$ day $=44.4 \times 10^{6} \mathrm{~mol} \mathrm{O} 2 /$ day
$\mathrm{n}_{\mathrm{N} 2}=7.52 \mathrm{P}=7.52 \times 15.77 \times 10^{6} \mathrm{~mol} /$ day $=315.088 \times 10^{6} \mathrm{~mol} \mathrm{~N}_{2} /$ day
ñ응 $=.94 \mathrm{P}=.94 \times 15.77 \times 10^{6} \mathrm{~mol} /$ day $=39.39 \times 10^{6} \mathrm{~mol} \mathrm{CO}_{2} /$ day
n $_{\mathrm{SO} 2}=.002 \mathrm{P}=.002 \times 15.77 \times 10^{6} \mathrm{~mol} /$ day $=.08 \times 10^{6} \mathrm{~mol} \mathrm{SO}_{2} /$ day
ñ $_{\mathrm{H} 2 \mathrm{O}}=.054 \mathrm{P}=.054 \times 15.77 \times 10^{6} \mathrm{~mol} /$ day $=2.26 \times 10^{6} \mathrm{~mol} \mathrm{H}_{2} \mathrm{O} /$ day

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

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

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

$$
=4.55 \times 10^{6} \mathrm{~kg} / \mathrm{day}
$$

Molar amount $=\left(16.71 \times 10^{6}+\right.$

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

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

$$
=18.37 \times 10^{6} \times \frac{m^{3}}{d a y} \times \frac{1 d a y}{24 \times 3600 \mathrm{~s}}
$$

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

Amount of $\mathrm{CO}_{2}=39.39 \times 10^{6} \mathrm{~mol} \mathrm{CO}_{2} /$ day

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

Amount of $\mathrm{SO}_{2}=0.08 \times 10^{6} \mathrm{~mol} \mathrm{SO}_{2} /$ day

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

Amount of dust $=1711 \mathrm{mg} / \mathrm{m}^{3}$
Total $=9631 \mathrm{mg} / \mathrm{m}^{3}$

## 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 | $\mathrm{H}-241$ | 1 |
| Rotary kiln | $\mathrm{R}-240$ | 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 $\mathrm{CO}_{2}$
- 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 $600 \mathrm{~kg} / \mathrm{cm}$.

Belt width, $l=17000 \mathrm{~m}$

Power drive: Gear motor
$\mathrm{C}=$ Auxiliary resistant coefficient $(1.05$ for long belt and 10 for short belt $)=1.05$
$\mathrm{f}=$ main resistant coefficient $=0.02$
$\mathrm{G}=\mathrm{Weight}$ of belt +weight for idler in the upper and lower strand per meter of belt conveyors $(\mathrm{kg} / \mathrm{m})=10 \mathrm{~kg} / \mathrm{m}$
$\mathrm{V}=$ belt conveyors speed $=1.5 \mathrm{~m} / \mathrm{s}$
$\mathrm{Q}=$ conveyed material (ton/h) =5655 ton/hr
$\mathrm{H}=$ conveying height $(\mathrm{m})=0$
Power equipment:

Power requirement for belt conveyors is
$N=\left(\frac{c \times f \times l}{210}\right) \times(3.6 G \times V+Q)+\frac{Q \times H}{270}(\mathrm{HP})$
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, $\mathrm{m}=223.83$ ton/hr
Work index of Limestone. $\mathrm{W}_{\mathrm{i}}=12.74$

Diameter of the feed, $\mathrm{D}_{\mathrm{pa}}=100 \mathrm{~mm}$

Diameter of the feed, $\mathrm{D}_{\mathrm{pb}}=1000 \mathrm{~mm}$
According to bond's law, $\frac{P}{m}=0.3162 \mathrm{~W}_{\mathrm{i}}\left(\frac{1}{\sqrt{D p b}}-\frac{1}{\sqrt{D p a}}\right)$
Or, $\mathrm{P}=1.95 \mathrm{KW}$

Power requirement for Cone crusher (C121):
Mass flow rate of lime stone, $\mathrm{m}=223.83$ ton $/ \mathrm{hr}$
Work index of Limestone. $\mathrm{W}_{\mathrm{i}}=12.74$

Diameter of the feed, $\mathrm{Dpa}=25 \mathrm{~mm}$
Diameter of the feed, $\mathrm{Dpb}=100 \mathrm{~mm}$
According to bond's law, $\frac{P}{m}=0.3162 \mathrm{~W}_{\mathrm{i}}\left(\frac{1}{\sqrt{D p b}}-\frac{1}{\sqrt{D p a}}\right)$
Or, $\mathrm{P}=2.85 \mathrm{KW}$

### 7.3 Sizing of ESP (H241, H244)

Pollutants in Exhaust gas in Preheater: $11100 \mathrm{mg} / \mathrm{m}^{3}$
Pollutants in Exhaust gas in Kiln: $9600 \mathrm{mg} / \mathrm{m}^{3}$
In ESP if we assume efficiency is $99.995 \%$;
In preheater exhaust gas pollutants will remain: $11100 \times .005=55.5 \mathrm{mg} / \mathrm{m}^{3}$
In Kiln exhaust gas pollutants will remain: $9600 \times .005=48 \mathrm{mg} / \mathrm{m}^{3}$
Let drift velocity of particles $0.1 \mathrm{~m} / \mathrm{s}$
$\eta=1-\exp \left(\frac{-V_{p m} A_{c}}{Q_{g}}\right)$
["Design of ESP in a Cement Plant", available at: www.scribd.com/documents/10405462]
$\mathbf{Q}_{\mathbf{g}}$, preheater $=4 \times 10^{6} \mathrm{~m}^{3} /$ day $=1111 \mathrm{~m}^{3} / \mathrm{s}$
$\mathbf{Q g}$, kiln $=5000 \mathrm{~m}^{3} / \mathrm{s}$
Ac, preheater $=110027 \mathrm{~m}^{2}$
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.
$\mathrm{A}=2 \times 10 \times 3 \times 3=180$
Number of plates needed $=110027 / 180=612(\mathbf{H} 241)$

Ac, kiln = 495174
Assuming the same configuration as above,
Number of plates needed for kiln $=495174 / 180=2750(\mathbf{H} 244)$

### 7.4 Sizing of mixer (F122)

Limestone 85\%

Clay 15\%
volume $=\frac{\frac{m 1}{p 1}+\frac{m 2}{p 2}}{2}$
Volume $=\frac{\frac{0.85 \times 5372 \times 1000}{24 \times 2160}+\frac{0.15 \times 948 \times 1000}{24 \times 22650}}{2}=45.12 \mathrm{~m}^{3}$
(Data Collected from Perry's Chemical Engineer's Handbook)

### 7.5 Sizing of Storage

Storage volume $\left(\mathrm{m}^{3}\right)=$ mass/density
F 331 $=\frac{1515 \times 1000}{1415}=1070.67 \mathrm{~m}^{3}($ for clinker $)$
F $332=\frac{1515 \times 1000}{3150}=480.95 \mathrm{~m}^{3}($ for $O P C)$
F $333=\frac{1515 \times 1000}{2400}=631.25 \mathrm{~m}^{3}$ (for PCC )
(Data Collected from Perry's Chemical Engineer's Handbook)

### 7.6 Sizing of air blower

## Capacity:

G $331: 2 \times 10^{6} \mathrm{~m}^{3} /$ day
G 332: $5.36 \times 10^{6} \mathrm{~m}^{3} /$ day

## Chapter 8

## Individual Equipment Design

# Detailed Design of a Ball mill (C210) 

## Prepared by

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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

Limestone 5372 ton/day
Clay 948 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 $(15-20 \mathrm{~kg} / \mathrm{cm} 2)$ and for bronze $(25-28 \mathrm{~kg} / \mathrm{cm} 2)$

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 <br> bearing | Sliding bearing |
| :--- | :--- | :--- | :--- |
| ease of installation | 30 | 25 | 30 |
| Lowering effect on <br> mechanical stress | 25 | 25 | 20 |
| Effect on mill-tube <br> thickness | 25 | 20 | 23 |
| 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) $=3 \mathrm{~m}$
Length of the second compartment $(66.67 \%$ of the effective length $)=6 \mathrm{~m}$.
["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 1.6 m | 18 mm |
| $1.6-2.0$ | 20 mm |
| $2.0-2.2$ | 25.5 mm |
| $2.2-2.4$ | 28 mm |
| $2.5-3.5$ | 38 mm |
| $3.5-4.25$ | 52 mm |
| $4.25-4.50$ | 58 mm |
| 5 | 63.5 mm |
| 6.4 | $85 / 75 \mathrm{~mm}$ |

Therefore, taking Shell thickness of 38 mm (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
$\mathrm{Nc}=42.3 / \sqrt{ } \mathrm{D}_{\text {eff }}$ [British Chemical Engineering vol. 6 part II]
Where, $\mathrm{D}_{\text {eff }}$ - is the effective diameter of the mill (m)

$$
\mathrm{Nc} \text { - critical speed of the mill (rpm) }
$$

D = outer diameter - liner thickness
$\mathrm{D}=3-2(0.038)$
D $\quad=2.924 \mathrm{~m}$

Therefore, $\mathrm{Nc}=42.3 / \sqrt{ } 2.924$
$\mathrm{Nc}=24.74 \mathrm{rpm}$

### 8.1.8 Mill speed

Mill speed is approximately 70 to $80 \%$ of the critical mill speed. Therefore taking $77 \%$,
Mill speed $\quad=\quad 77 \%$ of critical mill speed
Mill speed $\quad=\quad 0.77 \times 24.74 \mathrm{rpm}$
Mill speed $=19.05 \mathrm{rpm}$

### 8.1.9 Maximum Ball Size:

MBS $(\mathrm{mm})=36 \times F^{0.5} \times\left(\frac{\left(S G \times W_{i}\right)}{\% C \times D_{e f f}{ }^{0.5}}\right)^{0.333}$
Here, 5372 ton/day limestone \& 948 ton/day clay use as feed of the mill.
$\mathrm{F}=$ Feed size (sieve size in mm which retains $20 \%$ material), $\mathrm{mm}=20 \mathrm{~mm}$
$\mathrm{SG}=$ Specific weight of feed material, $\mathrm{gm} / \mathrm{cm} 3=2.60$
$\mathrm{Wi}=$ Bond's work index, $\mathrm{kWh} / \mathrm{t}=10.90$
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]
$\% \mathrm{C}=$ Mill speed as $\%$ critical, $\%=77$
$D_{\text {eff }}=$ Mill effective diameter, $m=2.924$
So, MBS $=\mathbf{9 7} \mathbf{~ m m}$
So, maximum ball size should be $\mathbf{1 0 0} \mathbf{~ m m}$
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 $=112.5-\left(125 \times \frac{H}{\text { Deff }}\right) \quad[$ "Ball mill calculation", sources:
www.scribd.com/documents/30505661]
H = Free height, m
Here, $\mathrm{D}_{\text {eff }}=2.924 \mathrm{~m}$
So, $H=2.62 \mathrm{~m}$
$\mathrm{h}=$ center distance (from mill center to media top layer), m
$D_{\text {eff }}=$ mill effective diameter, $m$
$\mathrm{R}=\mathrm{h} /$ deff, ratio of center distance to mill effective diameter
(h/ Deff used in page A. 4)
$\mathrm{h}=\mathrm{H}-$ Deff $/ 2=1.16 \mathrm{~mm}$
$h / D_{\text {eff }}=H / D_{\text {eff }}-1 / 2=2.62 / 2.924-1 / 2=0.395$

Table 8．1．3：Data for calculating a $\& q$ from $h / D_{\text {eff }}$

| hup | 3 | 4 | HuTD | a | 4 |
| :---: | :---: | :---: | :---: | :---: | :---: |
| 0.000 | 0.425 | 50.0 | 0．200 | 4.647 | 25.2 |
| 0.005 | 0.430 | 49.4 | 0205 | 9．6E3 | 24.6 |
| 0.070 | 0.435 | 48.7 | 0.210 | 0．659 | 24.1 |
| 0.015 | 0.441 | 4B． 7 | 0．215 | 0.665 | 23.5 |
| 0.020 | 0．4．46 | 47.5 | 0.220 | 0.670 | 22.9 |
| 0.025 | 0.452 | 4直，B | 0.225 | 0.675 | 22.4 |
| 0.030 | 0.457 | 46.2 | 0.230 | 0．682 | 21.8 |
| 0.035 | 0.462 | 45.6 | 0.235 | 0.849 | 21.2 |
| 10.040 | － 4 日a | 4.49 | 0.240 | 0．693 | 20.7 |
| 0.045 | 10．473 | 443 | 0.245 | 0.699 | 20.1 |
| －050 | 0.479 | 43.6 | 0.250 | 0.705 | 19.6 |
| 0.055 | 0． 484 | 49.0 | 0.255 | 0.711 | 19.0 |
| 0.060 | 0.450 | 42.4 | 0.260 | 0.717 | 18.5 |
| 0.065 | 0,495 | 41． B | 0.265 | 0.722 | 17.9 |
| 0.070 | 0．501 | 41.1 | 0.270 | 0.728 | 17.4 |
| 0.075 | 10．507 | 40.5 | 0.275 | 0.734 | 16．3 |
| 0.080 | 10．512 | 39.9 | 0.240 | 0.740 | 16.3 |
| －085 | 0．518 | 39.2 | 0.245 | 0.746 | 15.8 |
| 0.090 | 10.523 | 38.6 | 0.290 | 0.751 | 15.3 |
| 0．09S | 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 |
| $0.1+10$ | 0.548 | 38.1 | 0．310 | 0.775 | 132 |
| 0．175 | 0.551 | 35.5 | 0.315 | 07 AT | 127 |
| 0．120 | 0.557 | 34.9 | 0.320 | 0 －7as | 12 L |
| 0.125 | 0.562 | 74．3 | 0.325 | 17．792 | 11.6 |
| 0， 130 | 0．SE日 | 33.5 | 0.339 | 0.798 | 11.3 |
| 0.1 .45 | 0.574 | 39.0 | 0.355 | 0.404 | 10.8 |
| 0.140 | 0.579 | 32.4 | 0.3 .40 | O．810 | 10.3 |
| 0．145 | 0.585 | $31 . \mathrm{B}$ | 0.345 | 0.818 | 9.87 |
| 0.150 | 0.591 | 31.2 | 0.350 | 0．82z | 9.41 |
| 0.155 | D，596 | 30.6 | 0.355 | 0.828 | 8.95 |
| 0.160 | 9．602 | 30.0 | 0.360 | 0.833 | 8．31 |
| 0.165 | 0.608 | 29.4 | 0.365 | 0． 839 | B．9F |
| D． 170 | 9．613 | 2日，日 | 0.370 | $0 . \mathrm{B} 45$ | 7.64 |
| 0．175 | 0.619 | 28.2 | $0.3 \% 5$ | 0．15 1 | 722 |
| 6，180 | 0.625 | 27.3 | 0.360 | 0.857 | 6.80 |
| D．18S | 10.630 | 270 | 9．315 | 0.853 | 6.79 |
| 0.190 | 19.636 | 20.4 | 0.390 | 11．869 | 3.99 |
| 0． 195 | 10.642 | 25.6 | 0.355 | 0.478 | 5.59 |

a can aiso bet calisulaned as：

$$
a=0.009-(96.7-a)
$$

| Material | Grinding | Lining | Internal flttings | Media |  | Torque $\mu$ |
| :---: | :---: | :---: | :---: | :---: | :---: | :---: |
|  |  |  |  | Type | $\underset{\left[U m^{\prime}\right]}{W}$ |  |
| Cement and raw meal (dry) | Coarse and medium | Steel | None | Balls | 4.3 | 0.73 |
|  |  |  | None | Rods | 6.0 | 0.55 |
|  |  |  | Danula | Balls | 4.3 | 0.75 |
|  |  |  | Sonex | Balls | 4.3 | 0.66 |
|  | Fine | Steel | None | Balls | 4.5 | 0.69 |
|  |  |  | None | Cyipebs | 4.7 | 0.73 |
|  |  |  | None | Minipebs | 4.7 | 0.64 |
|  |  |  | Danula | Balls | 4.5 | 0.71 |
|  |  |  | Danula | Cylpebs | 4.7 | 0.75 |
|  |  |  | Danula | Minipebs | 4.7 | 0.66 |
|  | All | Steel | None | Ceramic | 1.9 | 0.75 |
|  |  | Silex | None | Stone | 1.5 | 0.75 |
| Slurry <br> (wet) | Coarse <br> and medium | Steel | None | Balis | 4.3 | 0.66 |
|  |  |  | None | Rods | 6.0 | 0.50 |
|  |  |  | Danula | Balls | 4.3 | 0.67 |
|  |  |  | Sonex | Balls | 4.3 | 0.59 |
|  | Fine | Steel | None | Balls | 4.5 | 0.66 |
|  |  |  | None | Cyipebs | 4.7 | 0.66 |
|  |  |  | Danula | Balls | 4.5 | 0.67 |
|  |  |  | Danula | Cyipebs | 4.7 | 0.67 |
|  | All | Steel <br> Silex | None | Ceramic | 1.9 | 0.85 |
|  |  |  | None | Stone | 1.5 | 0.85 |
| Wash drum |  |  | Lifters | Stone | 1.5 | 0.85 |
| Coal | Coarse | Steet | None | Bails | 4.3 | 0.69 |
|  | Medium |  | None | Ealls | 4.5 | 0.69 |
|  | Medium |  | None | Cyipebs | 4.7 | 0.69 |
|  | Medium |  | Danuia | Cyipebs | 4.7 | 0.71 |

Sources:[https://www.911metallurgist.com/blog/ball-mill-calculator https://www.911metallurgist.com/blog/grinding-ball-diameter-calculation]
$\mathrm{q}(\%)=$ Specific charge
$\mathrm{w}\left(\mathrm{t} / \mathrm{m}^{3}\right)=$ Bulk weight
form Table 8.1.3, $a=0.875$
from Table 8.1.3, $q=5.59$
from Table 8.1.4, $\mathrm{w}=4.3, \mu=0.73$
F = Charge

$$
\mathrm{F}(12)=\mathrm{q} / 100 \times w \times V
$$

$\mathrm{V}\left(1^{3}\right)=$ Volume of compartment $=$ Volume of compartment $1+$ volume of compartment 2

$$
=\pi / 4 \times \mathrm{D}_{\mathrm{eff}} \times \mathrm{L} 1+\pi / 4 \times \mathrm{D}_{\mathrm{eff}} \times \mathrm{L} 2
$$

$\mathrm{L} 1=3 \mathrm{~m}$
$\mathrm{L} 2=6 \mathrm{~m}$
So, $V=60.43 \mathrm{~m} 3$

So, F = 14.52

### 8.1.11 Compartment power consumption:

$\mathrm{a}=$ Arm of gravity in relation to mill diameter
$\mathrm{n}(\mathrm{rpm})=$ Rotational speed of the mill
$\mathrm{g}\left(\mathrm{m} / \mathrm{s}^{2}\right)=$ Acceleration of gravity
$\alpha\left({ }^{\circ}\right)=$ Angle of displacement
$\mathrm{N}(\mathrm{KW})$, net $=$ Power consumption by compartment at mill shell

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

Sin $\alpha$ is the torque factor $\mu$.Standard values from Table 8.1.4

$$
\mathrm{N}=0.514 \times F \times n \times \mu \times D \times a
$$

Where the constant 0.514 is $(\mathrm{g} \times \pi / 60)$
So, $\mathrm{N}=264.8 \mathrm{KW}$

### 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 $1 \mathrm{~m} / \mathrm{s}$ and chute being $25 \%$ for powdery material and $50 \%$ for lumpy material.
$\mathrm{A}=\frac{\mathrm{Q} \times 100}{3600 \times \mathrm{V} \times \mathrm{F}}$
Where,
Q- Volume flow rate in $\mathrm{m}^{3} / \mathrm{hr}$
V - Velocity in m/s
F - Filling level \%, here 28\%

## For Limestone

Mass flow rate of limestone $=5372$ ton $/$ day $=223 \times 10^{3} \mathrm{~kg} / \mathrm{hr}$
$\mathrm{Q}=\frac{\text { mass flow rate }}{\text { density }}$
$\mathrm{Q}=\frac{223 \times 10^{3}}{2.6 \times 10^{3}}$
$\mathrm{Q}=85.77 \mathrm{~m}^{3} / \mathrm{hr}$
Minimum area for clinker
$\mathrm{A}=\frac{85.77 \times 100}{3600 \times 1 \times 28}$
$\mathrm{A}=.09 \mathrm{~m} 2$

## For Clay

Mass flow rate of clay $=948$ ton $/$ day $=39.5 \times 10^{3} \mathrm{~kg} / \mathrm{hr}$
$\mathrm{Q}=\frac{\text { mass flow rate }}{\text { density }}$
$\mathrm{Q}=\frac{39.5 \times 10^{3}}{2.23 \times 10^{3}}$
$\mathrm{Q}=17.7 \mathrm{~m}^{3} / \mathrm{hr}$
Minimum area for clinker
$\mathrm{A}=\frac{17.7 \times 100}{3600 \times 1 \times 28}$
$\mathrm{A}=.02 \mathrm{~m} 2$

## Total minimum chute Area

Total minimum chute Area $=$ Area of limestone + Area of Clay
Total minimum chute Area $=0.11 \mathrm{~m}^{2}$

### 8.1.13 Hot air requirement:

Assuming 28\% media filling for air circulation.
Free area of air flow $=0.72 \times \pi / 4 \times \mathrm{D}_{\text {eff }}$

$$
=4.83 \mathrm{~m}^{2}
$$

Air velocity through the mill $=3 \mathrm{~m} / \mathrm{s}$ at NTP
Air flow rate $=3 \times 4.83 \times 60=870 \mathrm{~m}^{3} / \mathrm{min}$
Providing $10 \%$ extra than capacity $=1.1 \times 870=957 \mathrm{~m}^{3} / \mathrm{min}$
At $100{ }^{\circ} \mathrm{C}=957 \times(273+100) / 273=1309 \mathrm{~m}^{3} / \mathrm{min}=78476 \mathrm{~m}^{3} / \mathrm{hr}$

### 8.1.14 Bag Filter Design:

Flow gas rate $=78476 \mathrm{~m}^{3} /$ hour
Temperature $=100^{\circ} \mathrm{C}$
The filtering surface (SF) given in $\mathrm{m}^{2}$ is estimated using the following expression:
$\mathrm{SF}=0.26 \times \mathrm{F} \times \mathrm{c}^{0.18}+350 \times \varphi^{-0.5}$
$\mathrm{F}=$ gas flow at $25^{\circ} \mathrm{C}$ given in $\mathrm{m}^{3} / \mathrm{min}$
$\mathrm{c}=$ concentration of dust given in $\mathrm{g} / \mathrm{m}^{3}$
$\varphi=$ average diameter of the particle, in $\mu \mathrm{m}$
In case the values of "c" and " $\varphi$ " are unknown, an appropriate expression is:
$\mathrm{SF} \approx .014 \times \mathrm{Q}, \mathrm{Q}$ must be given in $\mathrm{Nm}^{3} / \mathrm{h}$
["Bag House Filter", available at: www.scribd.com/45006735]]

Q at NTP $\left(25^{\circ} \mathrm{C}\right)=78476 \times(273+25) /(273+100)=62700 \mathrm{Nm}^{3} / \mathrm{h}$

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

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

Length of the bags, $\mathrm{L}=3.5$ meter [assumption]
Area of a bag $=\pi \times D \times L+\pi \times D^{2} / 4=1.67 \mathrm{~m} 2$
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 $=53$


Dust Collecting Mode

FRONT VIEW


Bag Cleaning Mode

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




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

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## 8.2

## Detailed Design <br> Of

ROTARY KILN (R-240)


Prepared by
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### 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. $\mathrm{CO}_{2}$ removed $=39.4 \times 10^{6}$ mole per day
d. Coal required $=659$ ton per day
e. Air inlet $=5.36 \times 10^{5} \mathrm{~m}^{3} /$ day

### 8.2.3. Kiln length

Retention time (in minutes) $=\frac{1.77 \times \sqrt{ } \mathrm{Q} \times \mathrm{L} \times \mathrm{F}}{\mathrm{S} \times \mathrm{D} \times \mathrm{n}}$
Where, $\mathrm{Q}=$ Angle of repose (usually $35^{\circ}$ )

$$
\begin{aligned}
\mathrm{L} & =\text { Length of the kiln }(\mathrm{m}) \\
\mathrm{F} & =\text { Multiplying factor }(1) \\
\mathrm{T} & =\frac{1.77 \times \sqrt{35} \times \mathrm{L} \times 1}{2.29 \times \mathrm{D} \times \mathrm{n}}
\end{aligned}
$$

Using retention time 27 minutes

$$
\begin{aligned}
& \mathrm{L}=\frac{2.29 \times 4.5 \times 2.5}{1.77 \times \sqrt{35} \times 1} \\
& \mathrm{~L}=66.01 \mathrm{~m}
\end{aligned}
$$

### 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 2.8 m
$4 \%$ slope for kilns with diameter from 3-3.4m
$3 \%$ slope for kilns with diameter above 3.4 m
Kiln diameter $=4.5 \mathrm{~m}$

### 8.2.4. Kiln slope

It is generally $2^{\circ}$ 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 $=\arctan$ (slope)
Kiln slope $=\operatorname{arc} \tan (0.04)$
Kiln slope $=2.3$ degree

### 8.2.5. Retention time

Retention time (in minutes) $=\frac{1.77 \times \sqrt{\mathrm{Q} \times \mathrm{L} \times \mathrm{F}}}{\mathrm{S} \times \mathrm{D} \times \mathrm{n}}$
Where, $\mathrm{Q}=$ Angle of repose (usually $35^{\circ}$ )

$$
\begin{aligned}
& \text { L = Length of the kiln }(\mathrm{m}) \\
& \mathrm{F}=\text { Multiplying factor }(1) \\
& \mathrm{S}=\text { Slope of kiln (degrees) } \\
& \mathrm{D}=\text { Diameter }(\mathrm{m}) \\
& \mathrm{N}=\text { Speed }(\mathrm{rpm}) \\
& \mathrm{T}=\frac{1.77 \times \sqrt{3} 3 \times 66 \times 1}{2.29 \times 4.5 \times 2.5} \\
& =27.01 \text { minutes }
\end{aligned}
$$

### 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 $(\%)=\frac{1.667 \times T \times Q \times a}{\text { Volume of kiln } \times b}$
$\mathrm{T}=$ retention time in minutes

$$
\mathrm{Q}=\text { Capacity of clinker in tph }
$$

$\mathrm{a}=$ Ratio of raw meal feed/ product

$$
=1.5545
$$

$$
\mathrm{b}=\text { for clinker } 1.2 \mathrm{t} / \mathrm{m}^{3}
$$

Percentage filling $(\%)=\frac{1.667 \times T \times Q \times a}{\text { Volume of kiln } \times 1.2}$

$$
\begin{aligned}
& =\frac{1.667 \times 27 \times 168 \times 1.5}{1049.7 \times 1.2} \\
& =5.83 \%
\end{aligned}
$$

### 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 $35 \mathrm{~cm} / \mathrm{s}$ will now be rotating at 4.2 rpm or a peripheral speed of $87.5 \mathrm{~cm} / \mathrm{sec}$. [1]
Kiln speed $\quad=2.5 \mathrm{rpm}$

$$
=58.9 \mathrm{~cm} / \mathrm{s}
$$

### 8.2.8. Thickness of the kiln shell

Operating pressure $=1 \mathrm{~atm}$
Operating temperature $\quad=350$ degrees
Diameter of the kiln $\quad=4.5$ meters
Design pressure at $5 \%$ above operating temperature
Design pressure

$$
=\frac{105}{100} \times 1.013 \times 10^{5} \mathrm{~N} / \mathrm{m}^{2}
$$

Design pressure $=1.064 \times 10^{5} \mathrm{~N} / \mathrm{m}^{2}$
Therefore the design stress for mild steel at is 841 MPa

| Minimum thickness | $=\frac{\text { design pressure } \times \text { diameter }}{2(\text { design stress })-\text { design pressure }}$ |
| :--- | :--- |
| Thickness | $=284 \mathrm{~mm}$ |

### 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
$\mathrm{D}=$ diameter inside shell
$\mathrm{d}=$ diameter inside refractory
$\mathrm{L}=$ length of kiln
Volume of bricks $=0.785\left(\mathrm{D}^{2}-\mathrm{d}^{2}\right) \mathrm{L} \mathrm{m}^{3}$
If bulk density is $1.9 \mathrm{ton} / \mathrm{m}^{3}$,
Weight of bricks $=1.5\left(\mathrm{D} 2-\mathrm{d}^{2}\right) \mathrm{L}$ tons.

$$
=295 \text { tons }
$$

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.

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

| Sr no | Specification | Hysil blocks | Fireclay Brick 40 \% alumina | High alumuna 70 \% alumina | Castable Firecrete special | Castable White heat ' $\mathbf{A}$ ' | 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. $\max { }^{\circ} \mathrm{C}$ | 950 | 1400 | 1600 | 1350 | 1750 | 1500 |
| 3. | Refractoriness ${ }^{\circ} \mathrm{C}$ min |  | 1700 | 1820 | 1450 | 1800 | 1650 |
| 4. | Bulk density Min. $\mathrm{kg} / \mathrm{m}^{3}$ | 260 | 2300 | 2450 | 2250 | 2800 | 2100 |
| 5. | Linear change Max \% | 2 | $\begin{aligned} & \simeq 0.8 \\ & \text { at } 1400^{\circ} \mathrm{C} \\ & \text { for } 3 \mathrm{hrs} \end{aligned}$ | $\begin{aligned} & \simeq 2 \\ & \text { at } 1500^{\circ} \mathrm{C} \\ & \text { for } 2 \mathrm{hrs} \end{aligned}$ | $\begin{aligned} & \simeq 0.8 \mathrm{at} \\ & 1350^{\circ} \mathrm{C} \\ & \text { for } 2 \mathrm{hrs} \end{aligned}$ | $\begin{aligned} & \simeq 1.0 \mathrm{at} \\ & 1550^{\circ} \mathrm{C} \\ & \text { for } 3 \mathrm{hrs} \end{aligned}$ | $\begin{aligned} & \simeq 1.0 \mathrm{at} \\ & 1550^{\circ} \mathrm{C} \\ & \text { for } 3 \mathrm{hrs} \end{aligned}$ |
| 6. | Cold crushing strength $\mathrm{kg} / \mathrm{cm}^{2}$ at $100^{\circ} \mathrm{C}$ | 120 | 300 | 480 | 400 | 550 | 350 |
| 7. | Porosity max \% | - | 25 | 23 | - | - | - |
| 8. | Refractriness under load $\min ^{\circ} \mathrm{C}$ | - | 1400 | 1480 | - | - | - |
| 9. | Chemical analysis Alumina \% min | Calcium slicate blocks | 40 | 70 | 45 | 90 | 50 |
| 10. | $\mathrm{Fe}_{2} \mathrm{O}_{3} \%$ max |  | 3 | 2.5 | 4 | 0.8 | 1.5 |

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,
$\mathrm{A}=\dot{\alpha} \times \Delta T \times 1$
Where,
$\mathrm{A}=$ Thermal expansion of the kiln shell $\dot{\alpha}$-linear expansion index for steel $(0.000012 \mathrm{~mm} / \mathrm{m})$
L=Length (m)
$\Delta \mathrm{T}=$ Change in temperature (average value)
$\mathrm{A}=0.000012 \times(1000-25) \times 4.5 \times 1000 \mathrm{~mm}$
$\mathrm{A}=51.2 \mathrm{~mm}$

### 8.2.12. Stress Analysis

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

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

Where;
$\mathrm{P}=$ Density of mild steel
$\mathrm{Di}=$ Inner diameter
$\mathrm{t}=$ Thickness
$\mathrm{C}=$ Corrosion allowance
Assuming a corrosion allowance of 3 mm ,
Axial stress $\quad=\frac{4216 \times 2.8}{4(.284-.003)}$
Axial stress $\quad=518.2 \mathrm{~N} / \mathrm{m}^{2}$
$=417.3 \mathrm{~N} / \mathrm{m}^{2}$

Longitudinal stress due to pressure, $\mathrm{S}=\frac{D i \times p}{4 t}$
Where,
$\mathrm{P}=$ Design pressure
$\mathrm{Di}=$ inner diameter
$\mathrm{t}=$ Thickness
$S=\frac{4.2 \times 101325}{4 \times .284}$
Longitudinal stress due to pressure $=3.72 \times 105 \mathrm{~N} / \mathrm{m} 2$

### 8.2.13. Thermal loading

Thermal loading, $\mathrm{TL}=\frac{P \times H}{A \times 24 \times 1000}$
Where, $\mathrm{P}=$ Clinker production (tpd)
$\mathrm{H}=$ Specific heat consumption in kiln $\mathrm{kcal} / \mathrm{kg}$ clinker
$\mathrm{A}=$ burning zone cross sectional area $\mathrm{m}^{2}$
$\mathrm{TL}=\frac{4050 \times 390}{13.9 \times 24000}$

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

### 8.2.14. Volumetric loading

Volumetric loading, $\mathrm{VL}=\frac{P}{A \times L}$
$\mathrm{P}=$ Clinker production, tpd
$\mathrm{A}=$ Burning zone cross-sectional area, $\mathrm{m}^{2}$
L=Kiln length, $m$

$$
\begin{aligned}
& \mathrm{VL}=\frac{4050}{13.9 \times 66} \\
&=4.41 \mathrm{tpd} / \mathrm{m}^{2}
\end{aligned}
$$

### 8.2.15. Girth Gear and Pinion

Kiln Data
4.5 m Diameter

66 m length
Inclination $=4 \%$
Kiln speed $=2.5 \mathrm{rpm}$
a. Gear

Module $=39$
No of Teeth $=148$
Material of construction $=$ CS $640($ Normalized Cast Steel $)$
Dimensions $=550 \mathrm{~mm}$ (width)

## b. Pinion

Module $=39$
No of Teeth $=28$
Material of construction $=30 \mathrm{Cr} \mathrm{Ni} \mathrm{Mo} \mathrm{V8} \mathrm{(Normalized} \mathrm{Cast} \mathrm{Steel)}$
Dimensions $=600 \mathrm{~mm}$ (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 | $4.55 \times 66 \mathrm{~m}$ |
| Kiln length | 66 m |
| Kiln diameter | 4.5 m |
| Kiln slope | $4 \%$ |
| Kiln speed | 2.5 rpm |
| Design pressure | $1.064 \times 10^{5} \mathrm{~N} / \mathrm{m}^{2}$ |
| Operating pressure | 1 atm |
| Operating temperature | $700-1400$ degrees |
| Motor power | 340 kW |
| Gearbox velocity ratio | 284 mm |
| Shell thickness | 4045 tons/day |
| Clinker production output | 6288 tons/day |
| Kiln feed | 659 tons/day |
| Fuel requirements | 711 ton |
| Weight of the kiln | Mefractory brick lining steel |
| Material of lining | Material of construction |

### 8.2.16. Design calculation references

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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) 

## Prepared by

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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 of cooling air per kg of hot clinker

| component | Volume <br> $\left(\mathbf{m}^{3} / \mathrm{hr}\right)$ | Mass <br> $(\mathbf{k g} / \mathrm{hr})$ | Volume <br> $(\mathbf{k g} / \mathbf{k g ~ c k})$ | Mass <br> $(\mathbf{k g} / \mathbf{k g ~ c k})$ | Temperature <br> $\left({ }^{\circ} \mathbf{C}\right)$ | Heat <br> $(\mathbf{K j} / \mathrm{hr})$ |
| :---: | :---: | :---: | :---: | :---: | :---: | :---: |
| Cooling <br> air | 988750 | 1211218.75 | 5.87 | 7.18 | 25 | 0 |
| Hot <br> clinker | 107.35 | 168541.67 | 1 | 1 | 1400 | 231744796 |



Figure 8.3.2: Cooling process of the Grate cooler

Table 8.3.2: General design parameters of cooler

| Column1 | Column2 | Column3 | Column4 | Column5 |
| :---: | :---: | :---: | :---: | :---: |
| Type |  | Grate Cooler (Coolax with fixed inlet) |  |  |
| Total Surface Area | m 2 | 86.8 |  |  |
| Effective Surface Area | m 2 | 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 | m 2 | 7.2 | 34.45 | 45.14 |
| Type of Plates |  | CIS MFR | CFG / RFT | RFT |

Table 8.3.3: General design parameters of Grate plate


Table 8.3.4: Air distribution along clinker cooler

| Clinker production: | 4045 | $\mathrm{~T} / \mathrm{d}$ |
| :--- | ---: | :--- |
| Total air flow: | 988750 | $\mathrm{Nm}^{3} / \mathrm{h}$ |
| Air load: | 7.18 | $\mathrm{Kg} / \mathrm{kg} \mathrm{ck}$ |
| Total cooling area: | 112.85 | $\mathrm{~m}^{2}$ |
| Average blowing density: | 2.4 | $\mathrm{~m}^{3} / \mathrm{s} / \mathrm{m}^{2}$ |

## 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 $\left(m^{2}\right)$
$\mathrm{h}=$ bed $\operatorname{depth}(\mathrm{m})$
$\mathrm{y}=\operatorname{clinker} \operatorname{density}\left(\mathrm{kg} / \mathrm{m}^{3}\right)$
R=Kiln output (kg/hr)
At constant grate speed, the depth of the clinker bed, $\mathrm{h}=\frac{t \times R}{60 \times y}$
In both cases, volume of clinker $\left(\mathrm{m}^{3}\right)$ in the cooler, $\mathrm{V}=\frac{t \times R}{60 \times y}$

## Residence Time Calculation:

Residence Time, $\mathrm{t}=\frac{60 \times \mathrm{a} \times \mathrm{h} \times \mathrm{y}}{\mathrm{R}}$
Here,
Area of cooler grate surface, $a=112.9 \mathrm{~m}^{2}$
Bed depth, $\mathrm{h}=0.18 \mathrm{~m}$

Clinker density, $\mathrm{y}=1570 \mathrm{~kg} / \mathrm{m}^{3}$
Kiln output, $\mathrm{R}=168541.67 \mathrm{~kg} / \mathrm{hr}$
So, $\mathrm{t}=\frac{60 \times 112.9 \times 0.18 \times 1570}{168541.67}=11.35 \mathrm{~min}$
Specific load of coolers $=\frac{4045 t / d}{112.9 \mathrm{~m} 2}=35 \mathrm{t} / \mathrm{d} / \mathrm{m}^{2}$
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 <br> $(\mathbf{T} / \mathrm{hr})$ | Bed depth <br> $(\mathrm{m})$ | Clinker weight in <br> the cooler <br> $(\mathrm{kg})$ | Clinker volume <br> in the cooler <br> $\left(\mathrm{m}^{3}\right)$ | Residence time <br> $(\mathrm{min})$ |
| :---: | :---: | :---: | :---: | :---: |
| 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 | 55043.8 | 54.9 |
| 168.54 | 0.48 |  | 29.01 |  |



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


Table 8.3.7: Specification data of fan

| Fan No | Diameter <br> $(\mathrm{mm})$ | Area <br> $(\mathrm{m} 2)$ | Covered Area <br> $\left(\mathrm{m}^{2}\right)$ | 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 |
| K15 | 1000 | 0.79 | 12.40 | 1477 | 110 |
| K16 | 1250 | 1.23 | 16.50 | 1444 | 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 8.3.8: Nominal \& Extreme condition for design

| Material / Air Stream | Nominal |  | Extreme |  |
| :---: | :---: | :---: | :---: | :---: |
|  | $\mathrm{Nm} 3 / \mathrm{h}$ | ${ }^{\circ} \mathrm{C}$ | $\mathrm{Nm} 3 / \mathrm{h}$ | ${ }^{\circ} \mathrm{C}$ |
| Cooling Air | 988750 | 36 | 998320 | 36 |
| Clinker Outlet | 168 tph | 150 | NA | 200 |

## Reference

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[2] Hans E. Steuch, "clinker cooler design,', 2003
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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 $\mathrm{P}=244.43$ ton $/$ day $=\frac{244.43 \mathrm{ton}}{d a y} \times \frac{10^{6} \mathrm{~g}}{1 \mathrm{ton}} \times \frac{1 \mathrm{~mol}}{15.5 \mathrm{~g}}=15.77 \times 10^{6} \mathrm{~mol} / \mathrm{day}$
Air Inlet $\left(300^{\circ} \mathrm{C}\right)=9.52 \mathrm{P} \mathrm{mol} /$ day $=150.13 \times 10^{6} \times \frac{.0821 \times 573}{3.5} \times \frac{1 \mathrm{~m}^{3}}{1000 \mathrm{~L}}=2.018 \times 10^{6} \mathrm{~m}^{3} /$ day $=50000 \mathrm{acfm}$

## Composition of Preheater feed and outlet content:

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


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

$\mathbf{C}+\mathbf{O}_{2}=\mathbf{C O}_{2} \ldots \ldots \ldots \ldots \ldots \ldots \ldots \ldots \ldots \ldots \ldots \ldots \ldots \ldots ; \quad \Delta H_{\mathrm{r}}=\mathbf{- 9 5 2 2 4 . 8 8 \mathrm { Cal } / \mathrm { mol }}$
Per day clinker produced $=4045$ ton/day
Energy Consumption in Preheater $=95224.88 \mathrm{cal} / \mathrm{mol} \times 15.77 \times 10^{6} \mathrm{~mol} /$ day

$$
\begin{aligned}
& =1.15 \times 10^{12} \times \frac{c a l}{d a y} \times \frac{1 \mathrm{kcal}}{1000 \text { cal }} \times \frac{1 \text { day }}{4045 \text { ton clinker produced }} \\
& =471.25 \times 10^{3} \times \frac{\text { Kcal }}{\text { ton clinker }}=471.25 \mathrm{kcal} / \mathrm{kg} \text { clinker }
\end{aligned}
$$

### 8.4.1 Cyclone Separator portion:

Exhaust gas: $\left(700^{\circ} \mathrm{C}\right)$
$\mathrm{n}_{\mathrm{n} 2}=1.06 \mathrm{P}=1.06 \times 15.77 \times 10^{6} \mathrm{~mol} /$ day $=16.71 \times 10^{6} \mathrm{~mol} \mathrm{O}_{2} /$ day

$$
=.53 \times 10^{6} \mathrm{~kg} / \mathrm{day}
$$

$\stackrel{\circ}{\mathrm{n}} 2=7.52 \mathrm{P}=7.52 \times 15.77 \times 10^{6} \mathrm{~mol} /$ day $=118.59 \times 10^{6} \mathrm{~mol} \mathrm{~N}_{2} /$ day

$$
=3.32 \times 10^{6} \mathrm{~kg} / \mathrm{day}
$$

ñ $_{\mathrm{CO} 2}=.94 \mathrm{P}=.94 \times 15.77 \times 10^{6} \mathrm{~mol} /$ day $=14.82 \times 10^{6} \mathrm{~mol} \mathrm{CO}_{2} /$ day

$$
=.65 \times 10^{6} \mathrm{~kg} / \mathrm{day}
$$

ñ $_{\mathrm{SO} 2}=.002 \mathrm{P}=.002 \times 15.77 \times 10^{6} \mathrm{~mol} /$ day $=.03 \times 10^{6} \mathrm{~mol} \mathrm{SO}_{2} /$ day

$$
=2.02 \times 10^{3} \mathrm{~kg} / \mathrm{day}
$$

$\stackrel{\circ}{\mathrm{n}}_{\mathrm{H} 2 \mathrm{O}}=.054 \mathrm{P}=.054 \times 15.77 \times 10^{6} \mathrm{~mol} /$ day $=.85 \times 10^{6} \mathrm{~mol} \mathrm{SO}_{2} /$ day

$$
=15.32 \times 10^{3} \mathrm{~kg} / \mathrm{day}
$$

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

$$
=60.2 \times 10^{3} \mathrm{~kg} / \mathrm{day}
$$

Total exhaust $=\left(53 \times 10^{6}+3.32 \times 10^{6}+.65 \times 10^{6}+2.02 \times 10^{3}+30.1 \times 10^{3}+15.32 \times 10^{3}\right)$ $=4.55 \times 10^{6} \mathrm{~kg} / \mathrm{day}$

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:

$$
\begin{aligned}
& \check{\mathrm{n}}_{\mathrm{E}}=150.15 \times 10^{6} \mathrm{~mol} / \mathrm{day} \\
& \begin{aligned}
\mathrm{Q}_{\mathrm{E}}=150.13 \times 10^{6} \times \frac{.0821 \times 973}{3} \times \frac{1 \mathrm{~m}^{3}}{1000 \mathrm{~L}} & =3.99 \times 10^{6} \mathrm{~m}^{3} / \mathrm{day} \\
& =3.99 \times 10^{6} \times \frac{\mathrm{m}^{3}}{d a y} \times \frac{1 \text { day }}{24 \times 3600 \mathrm{~s}} \\
& =46.27 \times \frac{\mathrm{m}^{3}}{s e c}
\end{aligned}
\end{aligned}
$$

Assuming inlet velocity $\mathrm{V}=25 \mathrm{~m} / \mathrm{s}$

Area of the inlet duct of cyclone separator $\mathrm{A}=\frac{Q_{E}}{V}=\frac{46.27}{25}=1.85 \mathrm{~m}^{2}$

| Cyclone Separator Dimension [2] |  |  |
| :---: | :---: | :---: |
| Parameter | Dimension | Ratio |
| Body diameter | D | D |
| Length of cylinder | $\mathrm{L}_{\mathrm{b}}$ | 2 D |
| Length of cone | $\mathrm{L}_{\mathrm{c}}$ | 2 D |
| Inlet height | H | $\mathrm{D} / 2$ |
| Inlet width | W | $\mathrm{D} / 4$ |
| Gas exit dia | $\mathrm{D}_{\mathrm{c}}$ | $\mathrm{D} / 2$ |
| Dust outlet dia | $\mathrm{D}_{\mathrm{d}}$ | $\mathrm{D} / 4$ |
| Length of vortex finder | S | $5 \mathrm{D} / 8$ |

From conventional ratio Height of the inlet $\mathrm{H}=\mathrm{D} / 2$

$$
\text { Width of the inlet } \mathrm{W}=\mathrm{D} / 4
$$

Where D is the cyclone diameter

Cross section of the inlet duct is rectangular [2]
So, $\mathrm{H} \times \mathrm{W}=1.85 \mathrm{~m}^{2}$
$\frac{D}{2} \times \frac{D}{4}=1.85$, so $\mathrm{D}=14.8 \mathrm{~m}$
Now calculating other parameters: (according to conventional dimension)
Cut Diameter calculation:
$\mathrm{D}_{\mathrm{cut}}=\left(\frac{9 W \mu}{2 \pi N V \rho_{\text {particle }}}\right)^{\frac{1}{2}} ;[$ Diameter at which efficiency $=50 \%]$
Here, $\mu=$ viscosity of exhaust $=1.83 \times 10^{-5} \mathrm{~kg} / \mathrm{m} . \mathrm{s}$
$\mathrm{N}=$ no. of effective turns
$\mathrm{V}=$ velocity of the inlet $=25 \mathrm{~m} / \mathrm{s}$
$\rho_{\text {particle }}=1500 \mathrm{~kg} / \mathrm{m}^{3}$
Number of Effective turns: $N_{e}=\frac{1}{H}\left[L_{b}+\frac{L_{c}}{2}\right]$
$\mathrm{L}_{\mathrm{b}}=$ Length of the body $=2 \mathrm{D}=29.6 \mathrm{~m}$
$\mathrm{L}_{\mathrm{c}}=$ Length of the cone $=2 \mathrm{D}=29.6 \mathrm{~m}$
So, $\mathrm{Ne}=\frac{1}{7.4}\left[29.6+\frac{29.6}{2}\right]$

$$
\mathrm{N}_{\mathrm{e}}=6 \text { times }
$$

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

$$
=2.08 \times 10^{-5} \mathrm{~m}=20.8 \mu \mathrm{~m}
$$

Fractional Efficiency $\boldsymbol{\eta}=\frac{\left(\frac{D_{p}}{D_{c u t}}\right)^{2}}{1+\left(\frac{D_{p}}{D_{c u t}}\right)^{2}}$ [5]
Efficiency vs Particle dia:

| $\mathrm{D}_{\mathrm{p}}(\mu \mathrm{m})$ | Efficiency $(\mathrm{n}) \%$ |
| :---: | :---: |
| 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 $50 \mu \mathrm{~m}$ is satisfactory. So, Conventional cyclone can be selected [1]

## Gas Residence time ( $\Delta \mathrm{t}$ ) calculation:

$$
\begin{aligned}
\Delta \mathrm{t} & =\frac{\pi D N}{V}[3] \\
& =\frac{\pi \times 14.8 \times 6}{25} \\
& =11.15 \mathrm{Sec}
\end{aligned}
$$

## Particle Drift Velocity ( $\mathbf{V}_{\mathbf{p}}$ ) calculation:

$\mathrm{V}_{\mathrm{p}}=\frac{W}{\Delta t}=\frac{1.86}{5.61}=.17 \mathrm{~m} / \mathrm{sec}[3]$

## Terminal Settling Velocity ( $\mathbf{V}_{\mathbf{t}}$ ):

For a particle of $40 \mu \mathrm{~m}$ dia

$$
\begin{aligned}
\mathrm{Vt} & =\frac{\left(\rho_{\text {part }}-\rho_{\text {air }}\right) D_{p}^{2} V^{2}}{9 \mu D} \\
& =\frac{1500 \times\left(40 \times 10^{-6}\right)^{2} \times 25^{2}}{9 \times 1.83 \times 10^{-5} \times 14.8} \\
& =.62 \mathrm{~m} / \mathrm{s}[3]
\end{aligned}
$$

Pressure drop ( $\Delta \boldsymbol{P}$ ):
$\Delta P=\frac{\alpha \rho V^{2}}{2}[4]$
Here, $\alpha=\frac{K H W}{D_{e}^{2}}$ [where K is a proportionality constant ranging between 13-15]
[ $\mathrm{D}_{\mathrm{e}}$ is the gas exit diameter]

$$
=\frac{14 \times 7.2 \times 3.6}{7.2^{2}}=7
$$

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

$$
=2056 \mathrm{pa}
$$

## Power requirement (W):

$$
\begin{aligned}
\mathrm{W} & =\mathrm{Q}_{\mathrm{E}} \times \Delta P \\
& =46.27 \times 2056 \\
& =96 \times 10^{3} \mathrm{~W} \\
& =96 \mathrm{~kW} \sim 100 \mathrm{KW}[1]
\end{aligned}
$$

Outlet gas velocity ( $\mathrm{V}_{\mathrm{o}}$ ):

$$
\begin{aligned}
\mathrm{V}_{\mathrm{o}} & =\frac{Q_{E}}{\pi r_{e}^{2}} \\
& =\frac{46.27}{\pi \times 3.6^{2}} \\
& =1.14 \mathrm{~m} / \mathrm{sec}
\end{aligned}
$$

Dimensions and parameter of the cyclone selected with $10 \%$ overdesign:
So, actual dia $=\mathrm{D} \times(1+.1)=14.8 \times(1+.1)=16.28 \mathrm{~m}$

| Cyclone Separator Dimension [2] |  |  |
| :---: | :---: | :---: |
| Parameter | Dimension | Value (m) |
| Body diameter | D | 16.28 |
| Length of cylinder | $\mathrm{L}_{\mathrm{b}}$ | 32.56 |
| Length of cone | $\mathrm{L}_{\mathrm{c}}$ | 32.56 |
| Inlet height | H | 8.14 |
| Inlet width | W | 4.12 |
| Gas exit dia | $\mathrm{D}_{\mathrm{c}}$ | 8.14 |
| Dust outlet dia | $\mathrm{D}_{\mathrm{d}}$ | 4.12 |
| Length of vortex finder | S | 10.18 |

Area $=\pi L_{b} D+\pi \times \frac{D}{2} \times \sqrt{ }\left(R^{2}+L_{c}^{2}\right)=2523 \mathrm{~m}^{2}$
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: |  |  |
| ZIP code : |  |  |
| Contact Person: |  |  |
| ii. Gas conditions at cyclone inlet (Air): |  |  |
| Volume: $\underline{50000}$ (acfm) Pressure : $\underline{50.75}$ (psig) |  |  |
| Temperature: $\underline{100}\left({ }^{\circ} \mathrm{C}\right)$ |  |  |
| Moisture Content: $\underline{0.37 \%}$ (by weight) |  |  |
| iii. Dust Condition at cyclone inlet: |  |  |
| Dust description: Grinded Limestone, Raw mix |  |  |
| Specific gravity: $\underline{1.5} \quad$ Bulk density: $\underline{1500} \mathrm{~kg} / \mathrm{m}^{3}$ |  |  |
| Is dust corrosive? Yes Abrasive? Yes | Sticky? No | Toxic? Yes |
| Dust load at inlet: $\underline{2500} \mathrm{~kg} / \mathrm{hr}$. |  |  |
| Dust Concentration: $\underline{0.66 \%}$ (by weight) |  |  |
| iv. Cyclone |  |  |
| Operating Pressure: $\underline{31000 \mathrm{mmH}_{2} \mathrm{O}}$ |  |  |
| Construction Materials: Mild Steel |  |  |
| Is ASME code construction required? Yes |  |  |
| V. Auxiliaries |  |  |
| Cyclone stand: | Clean air out | $\underline{4.12 \mathrm{~m} \mathrm{dia}}$ |
| Dust hopper: 16.8 m with valve |  |  |
| Desired collection efficiency: $\underline{80 \%}$ |  |  |

### 8.4.2 Cyclone Preheater



Figure 8.4.2: Cyclone Preheater

Spray cooling of gas:
$\mathrm{M}_{\mathrm{w}} \times\left(\mathrm{C}_{\mathrm{p}}\right)_{\mathrm{w}} \times\left(\mathrm{T}_{\mathrm{E}}-\mathrm{T}_{\mathrm{w}}\right)=\mathrm{Q}_{\mathrm{E}} \times \rho_{a} \times\left(\mathrm{C}_{\mathrm{p}}\right)_{\mathrm{a}} \times \Delta T[6]$
$\mathrm{M}_{\mathrm{w}}=$ Amount of water required for cooling exhaust ( $\mathrm{kg} / \mathrm{h}$ )
$\left(\mathrm{C}_{\mathrm{p}}\right)_{\mathrm{w}}=$ Specific heat of water $=1 \mathrm{kcal} / \mathrm{kg}^{\circ} \mathrm{C}$
$\mathrm{T}_{\mathrm{E}}=$ Exit Temperature of water $=65^{\circ} \mathrm{C}$
$\mathrm{T}_{\mathrm{w}}=$ Inlet water temperature $=20^{\circ} \mathrm{C}$
$\mathrm{Q}_{\mathrm{E}}=$ Exhaust gas from preheater $\left(\mathrm{Nm}^{3} / \mathrm{h}\right)$

$$
=3.99 \times 10^{6} \mathrm{~m}^{3} / \mathrm{day}=166.25 \times 10^{3} \mathrm{~m}^{3} / \mathrm{hr} .
$$

$\mathrm{Q}_{\mathrm{N}}=$ Normalized gas flow rate

$$
=\frac{T_{N}}{T_{E}} \times \frac{P_{E}}{P_{E}} \times \mathrm{Q}_{\mathrm{E}}
$$

$\rho_{a}=$ Exhaust density $\left(\mathrm{kg} / \mathrm{m}^{3}\right)$

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

$\Delta T=$ gas cooling required $=(700-100)^{\circ} \mathrm{C}=600^{\circ} \mathrm{C}$
$\left(\mathrm{C}_{\mathrm{p}}\right)_{\mathrm{a}}=\frac{1.142+1.005}{2}=1.0735 \mathrm{KJ} / \mathrm{kg} .{ }^{\circ} \mathrm{C}$

$$
=.26 \mathrm{kcal} / \mathrm{kg}^{\circ} \mathrm{C}
$$

$\mathrm{M}_{\mathrm{w}} \times\left(\mathrm{C}_{\mathrm{p}}\right)_{\mathrm{w}} \times\left(\mathrm{T}_{\mathrm{E}}-\mathrm{T}_{\mathrm{w}}\right)=\mathrm{Q}_{\mathrm{E}} \times \rho_{a} \times\left(\mathrm{C}_{\mathrm{p}}\right)_{\mathrm{a}} \times \Delta T$
$\mathrm{M}_{\mathrm{w}}=\frac{166.25 \times 10^{3} \times 1.14 \times .26 \times 600}{1 \times(65-20)}$
$=657.02 \times 10^{3} \mathrm{~kg} / \mathrm{hr}=15.76 \times 10^{3} \mathrm{~kg} /$ day
$=3.85 \mathrm{~kg} / \mathrm{kg}$ clinker

Pressure drop ( $\Delta \boldsymbol{P}$ ):
$\Delta P=\frac{\alpha \rho V^{2}}{2}[6]$
Here, $\alpha=\frac{K A}{D_{e}^{2}}$ [where K is a proportionality constant ranging between 13-15]
[ $D_{e}$ is the gas exit diameter]
$A=$ cross sectional area of inlet
Air Inlet $\left(300^{\circ} \mathrm{C}\right)=9.52 \mathrm{P} \mathrm{mol} /$ day $=150.13 \times 10^{6} \times \frac{.0821 \times 573}{3.5} \times \frac{1 \mathrm{~m}^{3}}{1000 \mathrm{~L}}=2.018 \times 10^{6} \mathrm{~m}^{3} /$ day

$$
=23.35 \mathrm{~m}^{3} / \mathrm{sec}
$$

Assuming a velocity $\mathrm{V}=10 \mathrm{~m} / \mathrm{s}$
$\mathrm{A}=\frac{Q}{V}=2.335 \mathrm{~m}^{2}$
For a circular cross section, $\mathrm{A}=\pi r^{2}$

$$
\text { So, } r=0.86 \mathrm{~m}
$$

Dia of inlet of one-cylinder $\mathrm{D}_{\mathrm{i}}=1.73 \mathrm{~m}$

So, considering conventional arrangement, $\mathrm{D}_{\mathrm{e}}=1.73 \mathrm{~m}$
Body dia $=6.92 \mathrm{~m}$ for each cyclone (total 4 stage cyclone [1])
So, $\alpha=\frac{K A}{D_{e}^{2}}=10.14$
So, $\Delta P=\frac{\alpha \rho V^{2}}{2}=577.98$ pa for each cyclone $=58.9 \mathrm{~mm} \mathrm{H}_{2} \mathrm{O}$
Amount of false air $=8900 \times \mathrm{A}_{\mathrm{p}} \times(\Delta P)^{0.5}\left[\mathrm{~A}_{\mathrm{p}}=\right.$ Aperture area $=.01 \mathrm{~A}$ (assumed) $]$

$$
=2594 \mathrm{~m}^{3} / \mathrm{hr} .
$$

False air $\%=\frac{2594}{46.23 \times 3600}=15.59 \times 10^{-3}$ or $1.55 \%$
As false air is $<5 \%$, These dimensions would be satisfactory

### 8.4.3 Preheater fan design:

Height above mean sea level $\mathrm{H}_{\text {MSL }}=80 \mathrm{~m}$
Preheater feed $=6288$ ton $/$ day
Plant capacity $=1.5 \mathrm{MT} /$ year

$$
=4045 \text {-ton clinker /day }
$$

Air inlet flow rate $\mathrm{V}_{\mathrm{a}}=2.018 \times 10^{6} \mathrm{~m}^{3} /$ day
Air inlet temperature $=300^{\circ} \mathrm{C}$

$$
\text { Elevation factor } \begin{aligned}
\mathrm{K}_{\mathrm{E}} & =\frac{1}{\left(1-\frac{.0065 \times H_{M S L}}{288}\right)^{5.255}} \\
& =\frac{1}{\left(1-\frac{.0065 \times 80}{288}\right)^{5.255}} \\
& =1.009
\end{aligned}
$$

Applying this correction factor,
Corrected temperature $\mathrm{T}_{\mathrm{c}}=(300+273) \times \mathrm{K}_{\mathrm{E}}$

$$
\begin{aligned}
& =573 \times 1.009 \\
& =579 \mathrm{~K}
\end{aligned}
$$

Pressure drop $\Delta P=54$ mbar
Corrected pressure drop $\Delta P_{c}=1.009 \times 54=54.5 \mathrm{mbar}$
Normalized air inlet flow rate calculation:
$\frac{P_{N} V_{N}}{T_{N}}=\frac{P_{C} V_{a}}{T_{c}}$
Here, Corrected pressure $\mathrm{P}_{\mathrm{c}}=\mathrm{P}_{\mathrm{N}}-\Delta P_{c}$

$$
=3545-54.5=3490 \mathrm{mbar}
$$

So, $\mathrm{V}_{\mathrm{N}}=\frac{T_{N}}{T_{C}} \times \frac{P_{C}}{P_{N}} \times \mathrm{V}_{\mathrm{a}}$

$$
\begin{aligned}
& =\frac{273}{593.63} \times \frac{3490}{1013} \times 2.018 \times 10^{6} \\
& =3.21 \times 10^{6} \mathrm{~m}^{3} / \text { day }
\end{aligned}
$$

Normalized air per kg clinker production $\left(\hat{\mathrm{V}}_{\mathrm{N}}\right)$

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

Fan efficiency $\eta_{\mathrm{f}}=80 \%$ (for backward curved fan ) [7]
Dust loading $\mathrm{D}_{\mathrm{l}}=\frac{\text { Dust amount }}{\text { Exhaust gas }}$

$$
\begin{aligned}
& =\frac{60.2 \times 10^{3}}{3.99 \times 10^{6}} \\
& =.01508 \mathrm{~kg} / \mathrm{m}^{3}=15.08 \mathrm{~g} / \mathrm{m}^{3}
\end{aligned}
$$

Air density at inlet $\rho_{a}=\frac{3.5}{.0821 \times 573}$

$$
\begin{aligned}
& =.075 \mathrm{~mol} / \mathrm{L} \\
& =2.163 \mathrm{~kg} / \mathrm{m}^{3}
\end{aligned}
$$

Air density with dust $\rho_{a d}=2.163+.015=2.178 \mathrm{~kg} / \mathrm{m}^{3}$

## Fan power calculation:

Fan Power required without dust $\mathrm{W}_{\mathrm{f}}=\frac{V_{a} \Delta P}{\eta_{f}}$
$\mathrm{V}_{\mathrm{a}}=2.018 \times 10^{6} \mathrm{~m}^{3} /$ day

$$
=23.35 \mathrm{~m}^{3} / \mathrm{sec}
$$

$\mathrm{W}_{\mathrm{f}}=\frac{23.35 \times 5400}{0.8}=157.65 \times 10^{3} \mathrm{~W}=157.65 \mathrm{KW}$
Fan Power required with dust $\mathrm{W}_{\mathrm{fd}}=\mathrm{W}_{\mathrm{f}} \times \frac{\rho_{a d}}{\rho_{a}}$

$$
\begin{aligned}
& =157.65 \times \frac{2.178}{2.163} \\
& =159.1 \mathrm{KW}
\end{aligned}
$$

Margin on shaft power $=5 \%$
Minimum motor power required $=\mathrm{W}_{\mathrm{fd}} \times(1+.05)=167.055 \mathrm{KW}$
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) |
| :---: | :---: |
| $\mathrm{D}_{\mathrm{e}}$ | 6.92 |
| $\mathrm{D}_{\mathrm{o}}$ | 1.73 |
| $\mathrm{r}_{\mathrm{i}}$ | 0.86 |
| $\mathrm{D}_{\mathrm{d}}$ | 1.73 |
| $\mathrm{~L}_{\mathrm{b}}$ | 13.84 |
| $\mathrm{~L}_{\mathrm{c}}$ | 13.84 |
| S | 4.75 |

## List of references:

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## Chapter 9

## Equipment \& Plant Layout

Chapter 10
Economic Analysis

### 10.1 Cost Estimation

| No. | Equipment name | Unit No. | No. of <br> Equipment  | Cost (USD) |
| :---: | :---: | :---: | :---: | :---: |
| 1 | Conveyor belt | J-110 | 1 | 2000000 |
| 2 | Crusher | C-120 | 1 | 20000 |
| 3 | Crusher | C-121 | 1 | 12000 |
| 4 | Storage bin | $\begin{array}{lll} \text { F }-411, & \text { F-331, } & \text { F- } \\ 332, & \text { F-333, } & \text { F- } \\ 122 & \end{array}$ | 5 | 94380 |
| 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 cost |  |  | 8810200 |

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 $\times$ (index value at 2019 / index value at 2014)

$$
=8.81 \times 1.27 \text { million USD }=11.19 \text { million USD }
$$

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 | $\begin{array}{\|lrl} \hline \text { Percent of } & \text { deliver } \\ \text { equipment cost } & \\ \hline \end{array}$ | $\begin{aligned} & \text { Cost } \\ & \text { (million USD) } \end{aligned}$ |
| :---: | :---: | :---: |
| 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 | $\underline{302}$ | 33.80 |
| 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 | $\underline{126}$ | $\underline{14.10}$ |
| Fixed Capital Investment ( FCI ) | $\underline{428}$ | $\underline{47.89}$ |
| Working Capital | 75 | 8.45 |
| Total Capital Investment | 503 | 56.34 |

### 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 $=1.86 \times 10^{6}$ 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) $:=\frac{0.1 \times \text { Fixed Charges }}{0.15}=\frac{0.1 \times}{0.15}=4.60$ 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 $\mathrm{OPC}=120 \mathrm{USD} /$ ton

PCC Production 0.5 ton per year
Price of $\mathrm{PCC}=110 \mathrm{USD} /$ 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

$$
\begin{aligned}
& =(130.5-124.10) \\
& =6.40 \text { million USD }
\end{aligned}
$$

Income $\mathbf{T a x}=10 \%$ 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 $=\mathrm{i}=8 \%$ per year
Plant life $=20$ years
Salvage value $=\$ 4.79$ million

## Cash Flow Diagram

4.79 million \$


Salvage value $=10 \%$ of $\mathrm{FCI}=0.1 \times 47.90=4.79$ million USD
Net PW (8\%) $=\mathrm{I}+(\mathrm{R}-\mathrm{E})(\mathrm{P} / \mathrm{A}, 10 \%, 20)+\mathrm{S}(\mathrm{P} / \mathrm{F}, 10 \%, 20)$

$$
\begin{aligned}
& =-56.34+(130.50-124.10) \frac{(1+0.08)^{20}-1}{0.08 \times(1+0.08)^{20}}+4.79(1+0.08)^{-20} \\
& =7.52 \text { million USD }
\end{aligned}
$$

### 10.5.2 Calculation of Payback Period

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

| End of year | Simple Payback period <br> (at 0\% interest rate) | Discounted payback period <br> (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 |

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 ( $\left.\mathrm{i}^{\prime} \%\right)=0$
Or, $I+(R-E)\left(\frac{P}{A}, i^{\prime} \%, 20\right)+S\left(\frac{P}{F}, i^{\prime} \%, 20\right)=0$
Or, $-56.34+(130-123.6) \times \frac{\left(1+i^{\prime}\right)^{20}-1}{\mathrm{i}^{\prime} \times\left(1+\mathrm{i}^{\prime}\right)^{20}}+4.79 \times\left(1+\mathrm{i}^{\prime}\right)^{-20}=0$
Or, $\mathbf{i}^{\prime}=\mathbf{0 . 0 9 7}$ or $\mathbf{9 . 7 \%}$
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, $\mathrm{PW}(10 \%)=\mathrm{I}=\$ 56.34$ million
Future Worth of positive cash flow, FW $(10 \%)=6.4($ F/A, $10 \%, 20)+4.79=297.67$ million $\$$ ERR calculation,
$(56.34)\left(\mathrm{F} / \mathrm{P}, \mathrm{i}^{\prime \prime}, 20\right)=297.67$
or, $\mathbf{i}^{\prime \prime}=\mathbf{0 . 0 8 6 7}$ or $\mathbf{8 . 6 7} \%$
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)

