USE OF ALTERNATIVE RAW MATERIALS IN CEMENT INDUSTRY TO REDUCE CO$_2$ EMISSIONS

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CERTIFICATE

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DEDICATION

This thesis is dedicated to all the hardworking faculty, staff and most importantly our parents and friends who have supported us throughout the project.

ACKNOWLEDGEMENT

ABSTRACT

Cement production worldwide is around 4.2 Billion tons as it is one of the most extensively used material in construction and building. It is used in making concrete which is the basis of any building foundation. Throughout the years cement production has increased due to the ever-growing demand. That demand has its own percussions as for every ton of cement produced one ton of carbon dioxide is emitted which is an ever-going menace for the world. As the world is going towards greener energy and solutions, the aim is to develop a strategy and method to overcome the challenges posted by the cement industry. Our project focuses on using alternative raw materials which include using a different fuel, changing the cement composition and proposing an altogether new cement alternative to tackle the ever-going threat of increasing Carbon Dioxide Emissions.

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<div align="right">**CHAPTER 1**</div>

INTRODUCTION

1.1 The Cement Industry

For every ton of cement produced there are 0.927 tons of CO_2 produces as a by-product. This is a huge amount and adversely affects the environment by causing global warming.

Cement industry has a central role in construction and development. A large part of infrastructure is made of cement and the quality of cement is essential to provide safe and durable structures. There are many types of cement to be used for specific purposes. However, the most common type is the ordinary port land cement. In a cement industry, raw materials consisting primarily of lime and silicon dioxide are converted to usable cement along with two main unwanted products;

- Cement by-pass dust
- A large amount (numbers) of CO_2

Cement industry contributes to global CO_2 emission in mainly three ways:

1. Electrical energy generation for running grinders
2. Burning of fuel to heat the raw materials in kiln
3. Process emissions in the calcination process in kiln (60-65% of total emissions)

According to 2018 data, 8% of total carbon emissions were caused by cement production. We aim to minimize the CO_2 expulsion by using alternative raw materials without compromising on the quality and keeping the process as economic as possible.

Following four methods have been considered for achieving the objective:

1. Changing the fuel
2. Heat recycling
3. The new material – Ferrock
4. Limestone Calcined Clay Cement (LC3)

First two methods are concerned with replacement efficient use of fuel. The last one is about reducing the clinker content in cement. However, the third solution is about entire replacement of OPC by a new material called Ferrock.

This project is mainly focused on the first two methods.

1.2 The Cement Manufacturing Process

The raw material used consists mainly of calcium oxide (64% by mass) and silicon dioxide (23% by mass). Other components are iron oxide (3%), aluminum oxide (6%) and gypsum (5%). Raw materials are crushed to form what is called a "raw meal". Then this is heated to a temperature of 1450°C in a kiln. This heating allows a chemical reaction to take place.

$4CaO + 4Al_2O_3 + 4Fe_2O_3$ → $4CaO.Al_2O_3.Fe_2O_3$
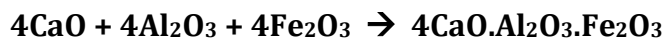
The desired product is clinker which is obtained in the form of nodules of 3 - 25 mm in diameter. Then grinding of clinker to powder and addition of 5% gypsum before final packing. Water is added to make a paste of cement for being used it for construction. While clinker serves as a binder, the purpose of gypsum is to slow down the hardening process so that proper application is possible. Also, it ensures the compressive strength.

Earlier, cement was produced by a process call "wet process". In this process, there is 32-36% water in the mixture that was sent to the kiln. That water needs to be evaporated. however now a days a "dry process" is used that is more energy efficient due to two main reasons:

1. Since the material is dry, energy that had to be used for evaporation is saved.

2. In dry process we use, there are suspension preheaters where raw meal is preheated before sending to the kiln. This provides more efficient heat transfer. The preheaters are provided energy via two ways:

   - The hot gases from the kiln are directed to provide their heat to preheaters. The material passes through a series of cyclones in the preheater tower and remains suspended in air due to the gases. There is a large temperature difference between raw meal and the hot gases. The heat transfer takes place unless the thermal equilibrium is achieved i.e. the gases and material are at the same temperature. About, 30%-40% of the raw meal is decarbonated in this process.

   - The heat energy from the cooling clinker is also used for the preheaters.

<div align="right">**CHAPTER 2**</div>

<div align="center">**LITERATURE REVIEW**</div>

2.1 Main Equipment

2.1.1 Crusher/grinder

The larger rocks are crushed to smaller pieces (80 to 20 mm) that have a relatively homogeneous composition and becomes more manageable. Then it is further grinded to powder form (0.2 mm) called "raw meal" so it can be sent for reaction. Grounded material offers more surface area and necessary for smooth and safe operations. Improper grinding causes the possibility of free lime in product that can adversely affect the quality of lime, strength of concrete and increase the setting time. and offer more surface area for reaction.

The decision of type of crusher is governed by many factors. some of these are; reduction ratio, fragment size, moisture content, hardness, plasticity and abrasiveness of materials being crushed.

2.1.2 Rotary kiln

It is a long cylinder having a length of about 70m and a diameter of 6m. It is made of a steel casing that is lined with refractory bricks so that it can withstand high temperature, reactions and mechanical stresses. The type of lining differs along the length of kiln since It rotates about its axis and it is slightly tilted with a burner at the lower end. The material (at about $900^{o}C$) enters from the back end that is higher and cooler, while leaves at the lower end that is hotter. The rotation of the kiln allows the material to move steadily towards the lower end under the action of gravity.

In the initial stage (about midway between the center and back end) the temperature is about 1000-1200°C. In the burning zone (that is midway between center and lower end) the temperature reaches 1400-1500°C and is called the clinkering temperature. The product is obtained as rounded nodules that need to be cooled and grinded to powder form.

2.1.3 Ball mill

To obtain powder form, cooled clinker nodules are grinded by using a ball mill.

About 66% of the total electrical energy is used in grinding of raw materials, final product or the fuel. Most energy is used in the grinding of concrete nodules.

Now a days, following four cement mills are mostly used:

1. Ball Mill (open circuit, closed circuit)
2. Vertical Roller Mill
3. Roller Press
4. Horizontal Mill

Bestway cement is using ball mill (open/closed circuit) for this purpose.

Following factors greatly affect the performance of ball mills:

- Properties of feed e.g. moisture content; the energy consumption increases more than 10% for every 1% rise in moisture content after 0.5% moisture. The ball mill will not operate when the moisture content has reaches about 3-4% unless it is assisted with a drying chamber.
- Use of clinker extenders (additives); clinker Substitution is achieved by using additives like limestone, slag, fly-ash and pozzolana. In this way;
- Energy can be saved to about 50%.
- The CO_2 emission can be effectively reduced.

However, wet or difficult to grind additives can make the process difficult and less efficient.

- ❑ Use of grinding aids (admixtures)
- ❑ Media filling ratio
- ❑ Fineness required
- ❑ Design of equipment
- ❑ Operations and maintenance
- ❑ Use of grinding aids (admixtures)

2.2 Solutions

For the reduction of carbon dioxide emissions by using alternative raw materials; four solutions are discussed briefly:

2.2.1 Changing the fuel

The fuel has been modified to recycle as much of the waste as possible. However, by analyzing the amount of carbon dioxide emitted per ton of coal, we conclude that natural gas has to be the most feasible fuel regarding the environment. If we take economics into consideration, we have to optimize the quantities of coal, natural gas, tyres, biomass and pet-coke.

2.2.2 Heat recycling

Hot gases from the kiln and waste heat from the cooler can be made well use of by directing them to preheaters. This makes the material to be already very hot before entering the kiln.

Table 1: Reduction of CO_2 using heat recycling

Kiln Type	Heat Input (MJ/ton of clinker)
4-Stage Cyclone Suspension Preheater Plus Calciner	3140
4-Stage Cyclone Suspension Preheater Plus Calciner + High Efficiency Air Cooler	**2996.16**
Difference	143.84
Amount of CO_2 reduced (kg)	**230.144**

As seen in the above table when heat is recycled from the clinker cooler to the cyclones the heat input required is less per ton of clinker. The difference is around 144 MJ/ton of clinker. This energy would have from fuel before and hence there is fuel reduction. 230 kg of carbon dioxide is hence reduced per ton clinker processing by recycling the pre heated air.

2.2.3 The new material – Ferrock

Ferrock is a revolutionary material which tends to absorb carbon dioxide from environment during its hardening process. It appears to be an ideal material since it fulfills the targets of concrete strength as well the environment concerns. Ferrock is mainly composed of (iron oxide) which converts to iron carbonate by absorbing CO_2. Following are some interesting facts about ferrock.

1. It is almost 5 times stronger than ordinary Portland cement.
2. It can be prepared from cheaper materials like steel dust, glass
3. Curing of ferrock is a carbon negative process.
4. It can withstand severe environments and chemicals

Ferrock is an iron-based compound made of 95% recycled materials that have been proven to be less-expensive,stronger and more flexible in its building applications than OPC. Furthermore, this unique material uses compressed carbon dioxide to expedite the curing process and requires no added heat to catalyze its chemical reaction, making it a carbon-negative alternative to OPC. All ingredients necessary for Ferrock production are conventional industrial materials except for iron powder

So far this material is being tested on pilot scale. The current studies have shown great benefits of this material. We aim to prepare the samples in laboratory and test the mechanical properties. If this material proves its properties and free of unwanted affects, it will be a great revolution in the cement industry and a great contribution in preventing the global warming. However, one of the challenges associated with this material is that there may not be enough raw material for its massive production. But still we can make and use the material as much as possible if not completely replacing the Ordinary Portland Cement.

In Ferrock only the some amount of calcination is required because some amount of calcium oxide makes up the final composition of Ferrock so we need calcium oxide and that emits carbon dioxide with the burning of the fuel of course. That amount is lesser than the amount absorbed as 10% by weight Ferrock consists of carbon dioxide and hence we can say carbon dioxide released is lesser than the amount absorbed. Below we can see the composition of Ferrock:

Table 2: Composition of Ferrock

Composition of Ferrock	
Oxalic Acid (C2H2O4)	0.018
Limestone (CaO)	0.09
Flyash/glass (SiO2)	0.252

Iron powder (FeO)	0.54
CO2	0.1

2.2.4 Limestone Calcined Clay Cement

LC3 is relatively new and environmentally friendly compared to OPC and it can reduce up to 30% CO_2 emissions. It is made by blending limestone and calcined clay. This is cost effective and can be manufactured in existing plants. LC3 has about half the amount of clinker compared to OPC. The process emissions for the preparation of clinker are greatly reduced.

LC3 reduces CO_2 emissions in three ways:

1. Less clinker requirement implies lesser decarbonation
2. Grinding of limestone instead of heating makes the process energy saving
3. Calcination at a lower temperature i.e. 800°C (compared to 1450°C for OPC) reduces energy usage to about 55%.

Following are some advantages of LC3

1. No need of capital-intensive modifications in existing plants
2. Lower production costs
3. There is sufficient knowledge about the chemical nature of the hydrates and pore structures
4. Resources efficiency (non-depleting to agricultural soils)

2.3 Energy and Fuel Analysis for CO_2

For one day basis in Bestway cements, mass of clinker produced = 9405 tons

Total Heat input by fuel = 28142216748 kJ = 28142216.75 MJ

Table 3: Calorific Values of Fuel

Types of fuels	MJ/kg	Fuel (kg)/ton clinker
Coal	27.5	108.8
Natural Gas	45	66.5
Tyres	27	110.8
Biomass + Pet Coke	24.27	123.3

Following tables gives the analysis for energy efficiency and CO_2 producing tendency of different fuels:

Table 4: CO_2 emitted by Coal

Coal	
Amount of Fuel (kg)	108.951
Energy Output (MJ)	2996.16
Formula	C
Amount of Carbon Dioxide Emitted (kg)	**399.488**

Table 5: CO_2 emitted by Natural Gas

Natural Gas	
Amount of Fuel (kg)	66.6
Energy Output (MJ)	2996.16
Amount of Carbon Dioxide Emitted (kg)	**183.1**
Formula	CH_4
% Reduction in Carbon Dioxide	**54.2**

Table 6: CO_2 emitted by Tyres

Tyres	
Amount of Fuel (kg)	110.97
Energy Output (MJ)	2996.16

Amount of Carbon Dioxide Emitted (kg)	**328.9**
Formula	$C_{4.05}H_{4.372}O_{0.35}S_{0.0372}N_{0.021}$
% Reduction in Carbon Dioxide	**17.7**

Table 7:CO_2 emitted by Biomass + Petcoke

Biomass + Pet Coke	
Amount of Fuel (kg)	123.5
Energy Output (MJ)	2996.16
Amount of Carbon Dioxide Emitted (kg)	**295.6**
Formula	$CH1.8O0.5N0.2 + C_{7.56}H_3N_{0.3}S_6$
% Reduction in Carbon Dioxide	**26.01**

PROCESS FLOW DIAGRAMS

3.1 Cement Plant PFD
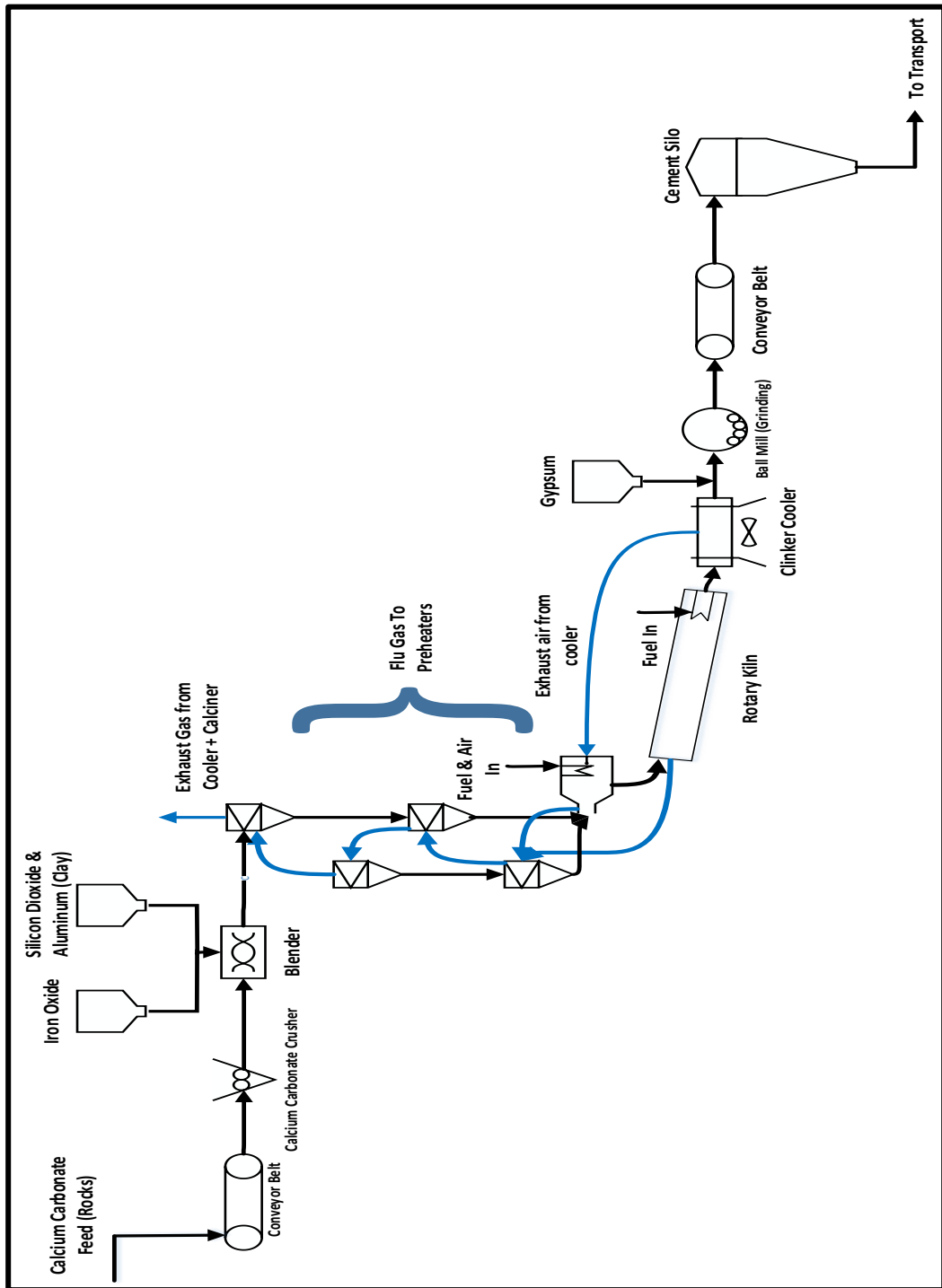
Figure 1:Cement PFD

3.1.1 Cement PFD Description

Calcium Oxide is prepared by decarbonation of Calcium Carbonate. Calcium Carbonate rocks moved by a conveyor belt are crushed and then blended with iron oxide and clay with no chemical reaction involved. The crusher and blender input is combined and sent to preheaters where the temperature is raised to 820°C, and then moved to pre-calciner where 90% conversion of raw materials takes place.

Further conversion takes place in rotary kiln where temperature reaches to 1450°C. In the cooler, the product is cooled and obtained in the form of granules, which is then finally grinded with gypsum in the ball mill and sent to cement silo.

3.2 Ferrock PFD
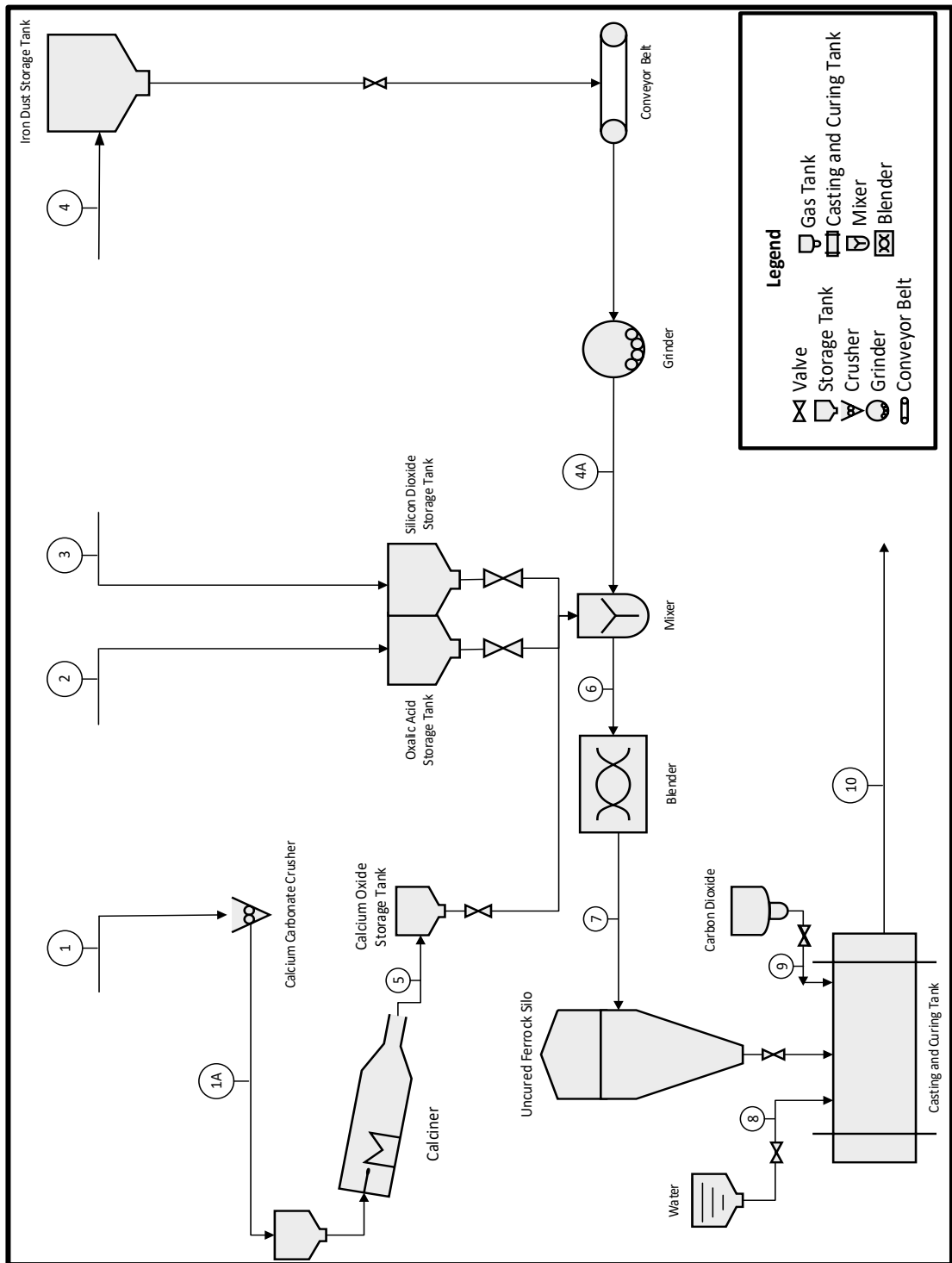
Figure 2: Ferrock PFD

3.2.1 Ferrock PFD Description

Ferrock uses clay and limestone as part of its composition, but the ratio of clay and limestone used is much smaller compared to OPC, eight and ten percent respectively. The majority of the mixture, totaling 80%, is composed of low-value waste products. The main ingredient is metallic iron powder, which is a byproduct of shot blasting, a finishing technique for steel manufacturing. During the shot blasting process, the iron powder is ground to a micro-particle scale ($\sim19.03\mu$m) , which becomes a considerable nuisance to the blasting facility because of its ineffectual applicability and the inherent respiratory hazard associated with working with such a fine material. These ingredients are combined as a dry mix with a source of silica, such as fly ash or recycled glass. Oxalic acid is also added to facilitate the chemical process and then blended to create a uniform mixture. It is necessary to point out that the Oxalic Acid, while small in percentage, represents an important ingredient of the mixture since it promotes the precipitation and mineralization of iron. It is in fact a well-known chemical promoter commonly used in the iron industry due to its characteristics as an iron dissolvent, which prevents oxidization and has the capacity to absorb CO_2 (by creating iron oxalate). While reacting with the Ferrock mixture, it chemically reacts with the compound and transforms it into a bonded carbonate molecule and therefore has no further threat as an emitting greenhouse gas. With the introduction of additional aggregates, water and compressed carbon dioxide, the iron oxide begins to chemically react yielding a new compound, iron carbonate, and emits hydrogen gas as a by-product.

$$Fe\ 0 + CO_2 + H_2O \rightarrow FeCO_3 + H_2$$
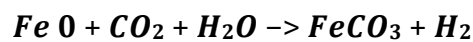
Fully cured samples contain between 8% and 11% of captured CO_2 by weight.

CHAPTER 4

MATERIAL BALANCE

4.1 Material balance of Cement Plant

4.1.1 Composition of Raw material

Table 8: Composition of Cement

Component	Composition
Calcium Oxide	64%
Iron Oxide	3%
Aluminum Oxide	6%
Silicon Dioxide	23%
Gypsum	5%

4.1.2 Basis

At Bestway cements 9900 tons are produces per day.

Table 9: Amount of Raw Meal

Component	Amount
Calcium Carbonate	11252.411
Calcium Oxide	6301.35
Iron Oxide	282.15
Aluminum Oxide	564.3
Carbon Dioxide	4951.06
Silicon Dioxide	2257.2
Gypsum	495
Total	**9900**

4.1.3 Crusher/blender
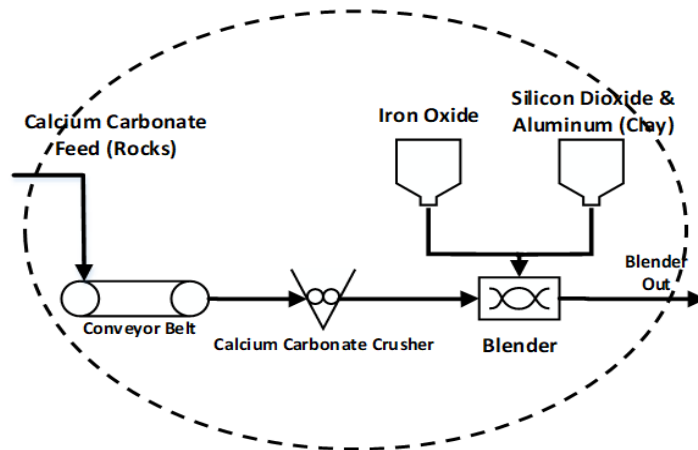
Figure 3: Grinding and Blending Section

Table 10: Material Balance of Grinding and Blending Section

Component	CaCO3 Feed (rocks) to crusher	CaCO3 feed to blender	Fe2O3 + clay to blender	Blender Out
Calcium Carbonate	11252.41071	11252.41071	-	11252.41071
Calcium Oxide	-	-	-	-
Iron oxide	-	-	282.15	282.15
Aluminum Oxide	-	-	564.3	564.3
Carbon Dioxide	-	-	-	-
Silicon Dioxide	-	-	2257.2	2257.2
Gypsum	-	-	-	-
Clinker	-	-	-	-
Cement	-	-	-	-
Total	**11252.41071**	**11252.41071**	**3103.65**	**14356.06071**

4.1.4 Preheater/pre-calciner
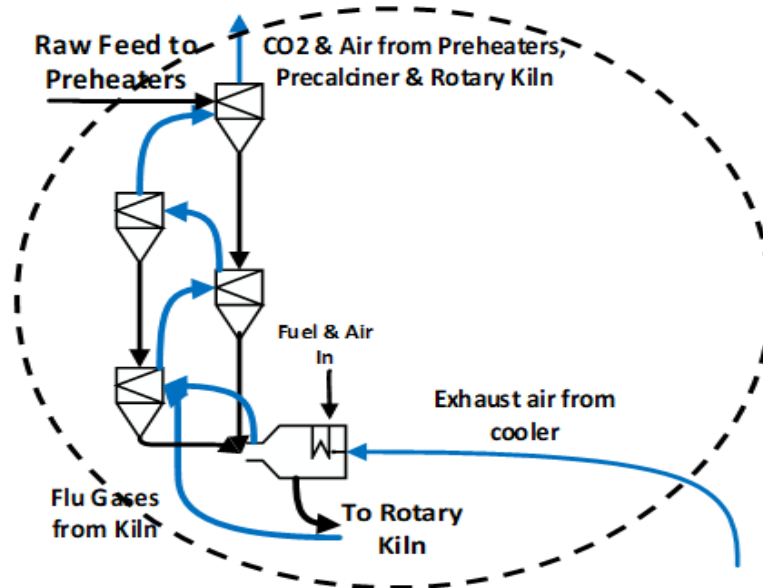
Figure 4: Preheating Section

Table 11: Material Balance of Preheating Section

Component	Raw feed to preheaters	To rotary kiln	Exhaust air from cooler	Flu gases form kiln + pre-calciner	CO2 and air our from preheaters top	Fuel In
Calcium Carbonate	11252.41071	1125.24107 1	-	-	-	-

Calcium Oxide	-	5671.215	-	-	-	-
Iron oxide	282.15	282.15	-	-	-	-
Aluminum Oxide	564.3	564.3	-	-	-	-
Coal	-	-	-	-	-	365.7
Carbon Dioxide	-	-	-	7743.1	7743.1	-
Air	-	-	7700	7638.75	19007.36	4643.81
Silicon Dioxide	2257.2	2257.2	-	-	-	-
Gypsum	-	-	-	-	-	-
Clinker	-	-	-	-	-	-
Cement	-	-	-	-	-	-
Total	**14356.06071**	**9900.10607**	**7700**	**15381.85**	**26750.5**	**5009.51**

4.1.5 Rotary Kiln/Cooler
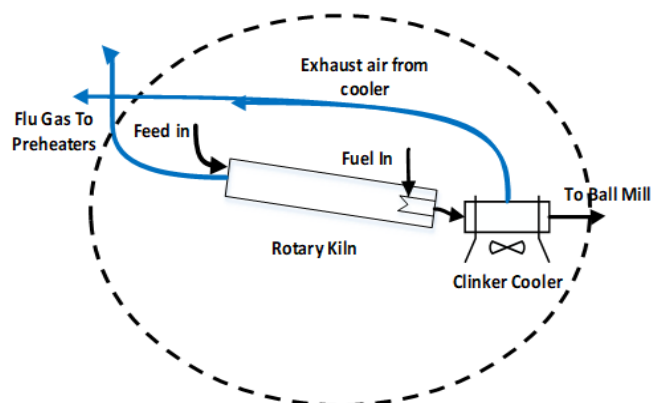
Figure 5: Rotary Kiln Section

Table 12: Rotary Kiln Material Balance

Component	Feed in	Fuel in	To ball mill	Exhaust air from cooler	Flu gas to preheaters	Air in to cooler
Calcium Carbonate	1125.241071	-	-	-	-	-
Calcium Oxide	5671.215	-	-	-	-	-
Iron oxide	282.15	-	-	-	-	-
Aluminum Oxide	564.3	-	-	-	-	-
Coal	-	395.7624	-	-	-	-
Carbon Dioxide	-	-	-	-	1946.13	-

Air	-	5025.5	-	7700	3970.145	7700
Silicon Dioxide	2257.2	-	-	-	-	-
Gypsum	-	-	-	-	-	-
Clinker	-	-	9405	-	-	-
Cement	-	-	-	-	-	-
Total	**9900.10607**	**5421.27**	**9405**	**7700**	**5916.3**	**7700**

4.1.6 Ball Mill
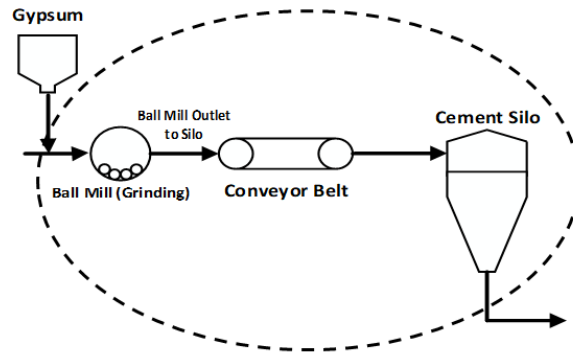
Figure 6: Final Grinding and Storage Section

Table 13: Grinding Section Material Balance

Component	Gypsum In	Clinker In	Ball Mill outlet to Silo
Calcium Carbonate	-	-	-
Calcium Oxide	-	-	6301.35
Iron oxide	-	-	282.15
Aluminum Oxide	-	-	564.3
Carbon Dioxide	-	-	-
Silicon Dioxide	-	-	2257.2
Gypsum	495	-	495
Clinker	-	9405	-
Cement	-	-	9900
Total	**495**	**9405**	**9900**

4.2 Ferrock Material balance

Table 14: Ferrock Material Balance

Flows (Tons/day)				
Line no.	1	1A	2	3
Steam Component	Calcium Carbonate Feed	Crushed Calcium Carbonate	Oxalic Acid Feed	Silicon Dioxide Feed
Oxalic acid ($C_2H_2O_4$)	-	-	115.02	-
Limestone ($CaCO_3$)	575.1	575.1	-	-
Lime (CaO)	-	-	-	-
Fly ash/glass (SiO_2)	-	-	-	1610
Iron powder (FeO)	-	-	-	-
Carbon dioxide (CO_2)	-	-	-	-
Water (H_2O)	-	-	-	-
Hydrogen Gas (H_2)	-	-	-	-
Total	575.1	575.1	115.02	1610
Pressure (bar)	1	1	1	1
Temperature (^oC)	30	1000	30	30

Table 14: (cont'd)

Flows (Tons/day)			
Line no.	4	4A	5	6
Steam Component	Iron Dust Feed	Grinded Iron Dust	Calcium Oxide Feed	Mixed Feed
Oxalic acid ($C_2H_2O_4$)	-	-	-	115.02
Limestone ($CaCO_3$)	-	-	-	-
Lime (CaO)	-	-	575.1	575.1
Fly ash/glass (SiO_2)	-	-	-	1610
Iron powder (FeO)	3450	3450	-	3450

Carbon dioxide (CO$_2$)	-	-	575.1	-
Water (H$_2$O)	-	-	-	-
Hydrogen Gas (H$_2$)	-	-	-	-
Total	3450	3450	1150.2	5750.12
Pressure (bar)	1	1	1	1
Temperature (°C)	30	30	30	30

Table 14: (cont'd)

Flows (Tons/day)				
Line no.	7	8	9	10
Component	Blended Uncured Ferrock	Water Feed	Carbon Dioxide Feed	Cured Ferrock
Oxalic acid (C$_2$H$_2$O$_4$)	115.02	-	-	115.02
Limestone (CaCO$_3$)	-	-	-	-
Lime (CaO)	575.1	-	-	575.1
Fly ash/glass (SiO$_2$)	1610	-	-	1610
Iron powder (FeO)	3450	-	-	3450
Carbon dioxide (CO$_2$)	-	-	71	71
Water (H$_2$O)	-	1278	-	1278
Hydrogen Gas (H$_2$)	-	-	-	122.4
Total	5750.12	1278	71	7221.52
Pressure (bar)	1	1	1	1
Temperature (°C)	30	30	30	30

Note: The "Steam" label appears in the header area above "Component" for the Line no. column.

ENERGY BALANCE

5.1 Energy Balance

This is most important part of any project, to find out the amount of energy that is required for the overall process. As we know from the law of conservation of energy, the first law of thermodynamics as well that energy can neither be created nor be destroyed. It only changes from one form to the other. Keeping that in mind we performed our energy balance on each of the processes.

5.1.1 Methodology

So to start off:

- Average values of Cp were calculated using the formula

$$Cp = a + bT + CT^2$$

  Values of a,b & c for each component were taken from literature and hence Cp was computed for that respective temperature, i.e. 25°C, 900°C & 1500°C.

- Then value of Cp average was calculated and hence that was computed using the formula for each component and then was tabulated in the table.

$$Cp_{avg} = \int_t^T Cpdt \div \int_t^T dt$$

- Then the basic energy balance equation was used for each system i.e.

$$Q + Ws = \Delta H + P.E + K.E$$

  As there was no Ws, P.E. and K.E. in every system the energy balance was reduced to

$$Q = \Delta H = H_2 - H_1$$

- Then heat requirements were calculated for each component separately using the formula

$$Q = \int_{25}^{T} mCpavgdt$$

- And lastly enthalpy of reaction i.e. (the Thermal Decomposition of Limestone) was calculated using Hess's law and is visible in the line diagram made in the above picture.

Basic ΔH equation is, ΔH = Hp – Hr + ΔHrxn

Where ΔHrxn is the enthalpy of reaction taking place at 25°C and Hp is the enthalpy of products at temperature T and Hr is enthalpy of reactants at temperature T.

Our energy balance scheme is summed up in the following diagram below
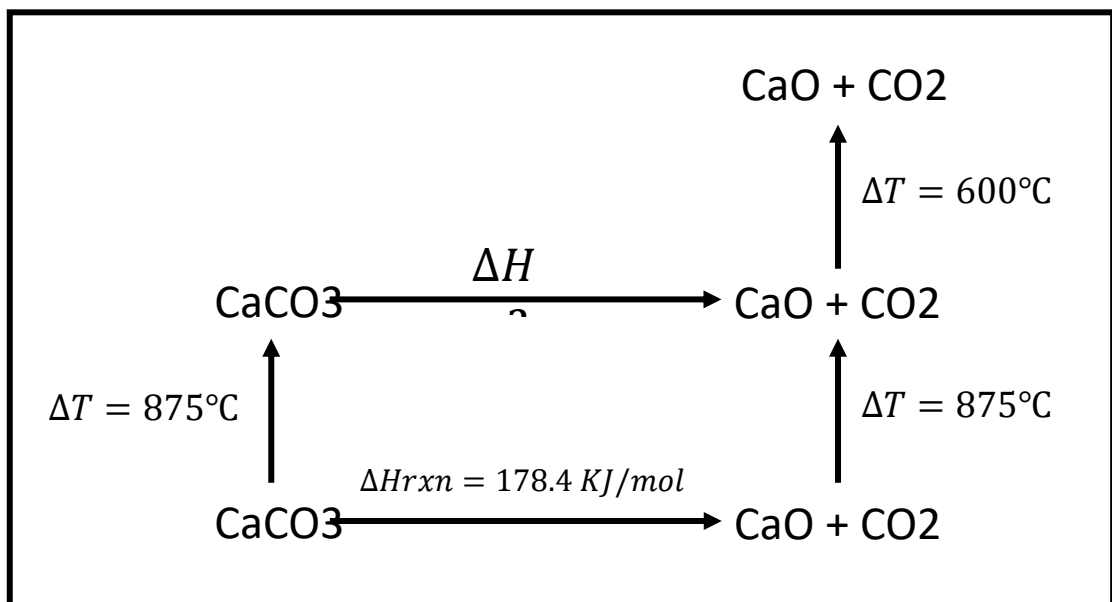
Figure 7: Calculation Scheme of Heat of Reaction

Using the heats of formation, sensible heats of each component and heat of reaction we calculated the heat requirements at each stage and our summed below in the tables.

Component	Cp Avg (KJ/Kg C)
Calcium Carbonate	0.85
Calcium Oxide	0.996
Iron Oxide	0.7815
Aluminium Oxide	0.703
Silicon Dioxide	0.7
Carbon Dioxide	1.443
Gypsum	-

Figure 8: Average values of Cp

5.1.2 Energy Balance on Preheater Cyclones

Feed from Blender @ 25ºC

Flue Gas + Air @1450ºC

For Simplification 4 Preheaters Illustrated as 1

Feed to Pre-Calciner @900ºC
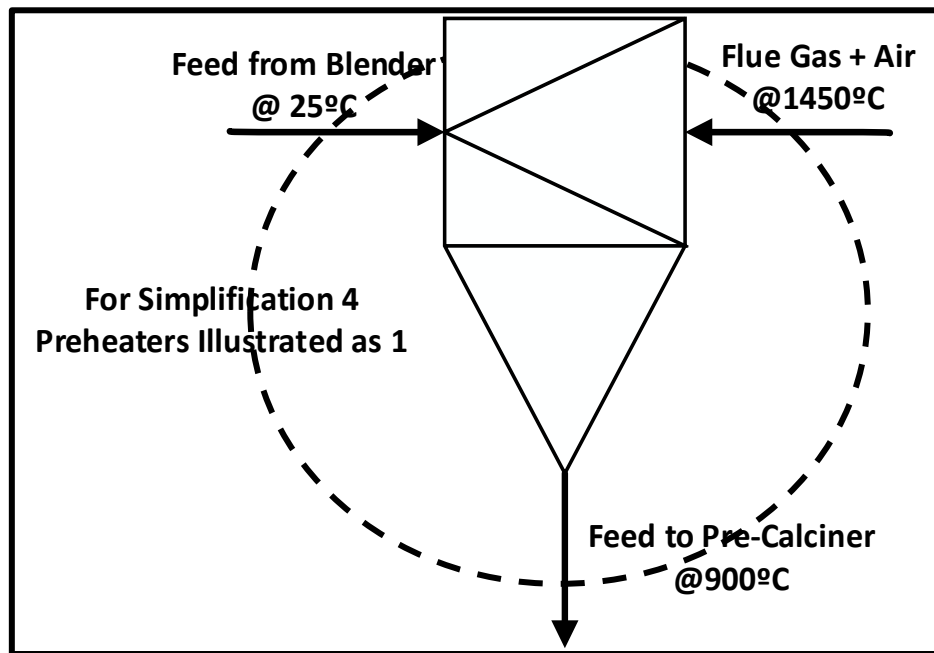
Figure 9: Cyclone preheater Section

Table 15: Enthalpy of Feed

Enthalpy of Feed into Pre-calciner	
Components	ΔH
Calcium Carbonate	7550057669
Iron Oxide	192937696.9
Silicon Dioxide	1382535000
Aluminium Oxide	347115037.5
Calcium Oxide	1098325305
Carbon Dioxide	1250266427
ΔHrxn	232437746.5
Total	**12053674882**

Table 16: Heat input into the Preheaters

Heat Input into 4 Preheaters	
Components	ΔH/KJ
Air	9812880000
Flu Gases	2240794877
Total	**12053674877**

Table 17: Energy Flow

IN (KJ)	OUT (KJ)
12053674881	12053674877

Initially feed is entering at 25°C and hence its enthalpy is 0 into the 4 preheaters that are illustrated a 1 for simplicity.

Feed has to be heated to 900°C hence using,

$$Q = \int_{25}^{T} mCpavgdt$$

for each component we found out the energy requirement and that value is quoted at the end of the table i.e. **12053674881 KJ**

Combustion flu gases and recycled air from the cooler is directed and used to heat up the feed to 900°C, obviously some heat is wasted to the surroundings but majority is utilized.

5.1.3 Energy Balance on Pre-Calciner
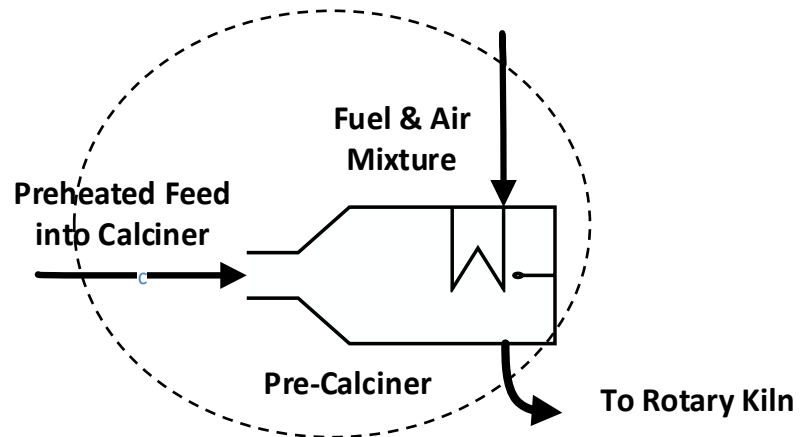
Figure 10: Pre-calciner Section

Table 18: Enthalpy of stream

Enthalpy of Feed Stream into Pre-Calciner	
Components	ΔH/KJ
Calcium Carbonate	7550057669
Iron Oxide	192937696.9
Silicon Dioxide	1382535000
Aluminium Oxide	347115037.5
Calcium Oxide	1098325305
Carbon Dioxide	1250266427
ΔHrxn	232437746.5
Total	**12053674881**

Table 19: Enthalpy of Stream Out

Enthalpy of Feed out of Pre-calciner	
Components	ΔH
Calcium Carbonate	8115198834
Iron Oxide	192937696.9
Silicon Dioxide	1382535000
Aluminum Oxide	347115037.5
Calcium Oxide	4942463873
Carbon Dioxide	6251332133
ΔHrxn	1045969859
Total	**22277552434**

Table 20: Heat input into Pre-calciner

Heat Input into the Pre-Calciner	
Components	ΔH
Fuel (Coal)	10223877552
Total	**13631836736**

Table 21: Energy Flow

IN (KJ)	OUT (KJ)
25685511618	22277552434

So in this section the feed is coming from the pre heater into the pre-calciner, in this system 70% of thermal decomposition takes place CaO and CO_2 being produced again enthalpies of each component were computed. While 20% of thermal decomposition takes place in the pre heaters and the rest is done in the Pre-Calciner along with further heating and sintering processes taking place in the rotary kiln.

Feed is entering at 12053674882 KJ and leaving at 22277552434 KJ. So this makes

$\Delta H = \textbf{1361836736 KJ}$ of energy that is required which is supplied by the fuel i.e. coal as our standard. More energy is supplied than required as burner efficiency is assumed at to be 75%

5.1.4 Energy Balance on Rotary Kiln
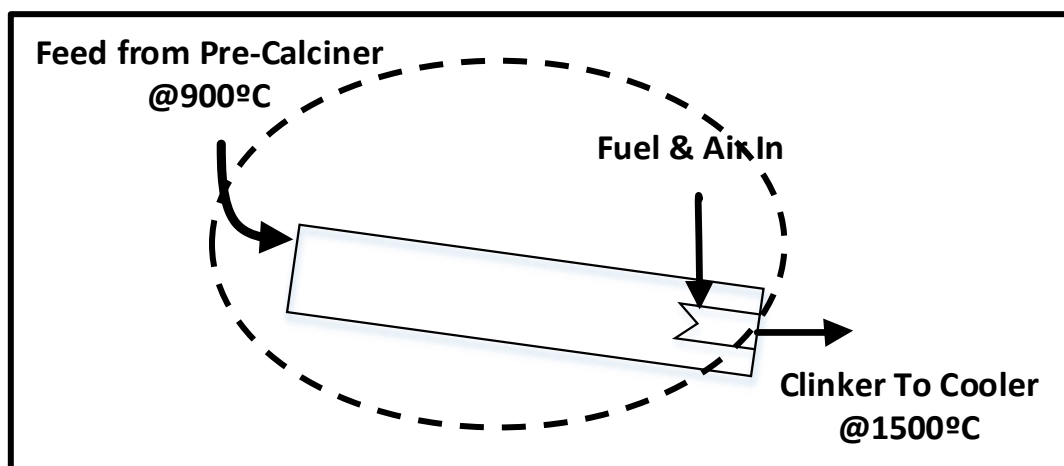
Figure 11: Rotary Kiln Section

Table 22: Enthalpy of Feed into Kiln

Enthalpy of Feed into Rotary Kiln	
Components	ΔH
Calcium Carbonate	8115198834
Iron Oxide	192937696.9
Silicon Dioxide	1382535000
Aluminum Oxide	347115037.5
Calcium Oxide	4942463873
Carbon Dioxide	6251332133
ΔHrxn	1045969859
Total	**22277552434**

Table 23: Enthalpy of Stream out of Kiln

Enthalpy of Feed exiting the Rotary Kiln	
Components	ΔH
Calcium Carbonate	8961941934
Iron Oxide	325237831.9
Silicon Dioxide	2330559000
Aluminum Oxide	585136777.5
Calcium Oxide	9257313286
Carbon Dioxide	10537959881
ΔHrxn	1162188732
Total	**33160337443**

Table 24: Heat Requirements

ΔHrequired	10882785009
Heat Input into Rotary Kiln	
Components	ΔH
Fuel (Coal)	10882785009
Total	**14510380012**

Table 25: Energy Flow

IN (KJ)	OUT (KJ)
36787932446	33160337443

This is the calcination system, 10% of thermal decomposition takes place and the feed is further heated to 1500°C where clinker is formed as the particles due to heating sinter and fuse into each other forming small rock structure.

Enthalpy into the system is **22277552434 KJ** and out of the system **is 33160337443 KJ**

Heat required is the difference between output and input and hence is 10882785009 KJ

It is supplied by the combustion of coal and again 75% burner efficiency is taken which means heat input by the coal would be higher than required to account for losses which is **14510380012 KJ**

Combustion values were taken from literature.

5.1.5 Energy Balance on Clinker Cooler
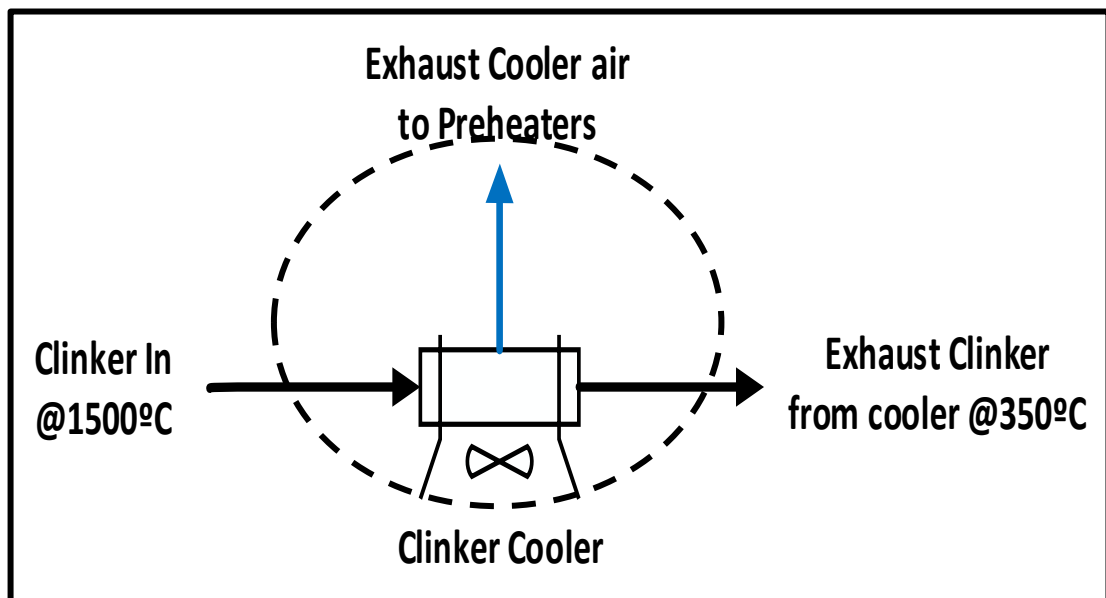
Figure 12: Clinker Cooler Section

Cp of Clinker	Mass (tons)	Temp In/C	Temp Out/C
1.101	9405	1500	350

Figure 14: Air Properties

Q Required	-11908140.8

Figure 13: Heat absorbed by air

Cp of Air	Mass (tons)	Temp In/C	Temp Out/C
1.08	7776.987	25	1450

Figure 15: Clinker final temperature

Finally in the last system the clinker cooler the clinker has to be cooled from 1500°C to 350°C. Air is in at 25°C and using heat balance we compute the Q required that has to be removed from the clinker which turns out to be **(-) 11908140.8 KJ** which is absorbed by the air and hence air outlet temperature is 1450°C whereas the feed's outlet temperature is 350°C.

5.2 Conclusion

. Some is supplied by the fuel some by the recycled air from the clinker cooler and some from the combustion flu gases which exit the calciner and pre calciner stages respectively.

Table 26: Total Heat Requirement

9900 Tons/Day	KJ
Total Heat Required	**32313594341**

CHAPTER 6

PLANT DESIGN

6.1 Design

In a cement plant there are different types of equipment. The equipment types can be categorized into separators, heaters/coolers and size reducers. So we have designed one equipment of each type using different design equations and empirical relations, taking aid from tables and charts as well. Separators include cyclone separators separating solids from the incoming air feed, heaters/coolers include pre-calciner furnace, clinker cooler and rotary kiln although a cyclone also acts as a suspension preheater but we will treat it according to its real use. Lastly size reduces include ball mills and crushers.

The equipment we have designed are

- High Efficiency Conventional Solid Cyclone Separator
- Pre-calciner Furnace
- Ball Mill

6.2 Design of a Cyclone Separator

6.2.1 The operating principle

The operating principle of cyclone is that it employs centrifugal force caused by vortex movement of fluid (gas/liquid)

6.2.2 Description and Salient characteristics of a Cyclone

Cyclone separators as the name suggest are mechanical dust separators which separate solids from the stream they are coming in. Solids can separated from a gas or liquid depending on the carrier fluid. Cyclones in general are easy to

Figure 16: Cyclone Seperation

use and maintain as they contain no moving parts. They require unmanned operation, and all the operations depend on the temperature and pressure they are operating on. They are very efficient when coupled together in series as progressively the efficiency increases as we move from one cyclone to the other one.

Key parameters of cyclone efficiency are dependent upon the:

- Pressure drop (ΔP)
- Dust separation efficiency

ΔP is defined as the difference between average static pressure at the cyclone inlet and outlet.

Dust Separation Efficiency is the ratio of the weight of particles captured to the total mass of solids that were present during inlet.
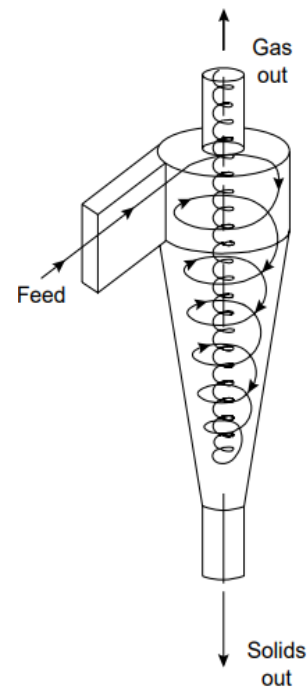
Another important feature is the gas/liquid velocity. It has been seen over time that as velocity of the carriers fluid is increased the separation efficiency increases due to the increase in centrifugal force which aids in the separation of the particles but the cost is high pressure drop so hence a compromise is made between efficiency and pressure drop (ΔP).

In our design we will be designing the cyclone according to Stairmand method and then translating that into Lapple high efficiency cyclone as that type of cyclone is used commonly in cement industries.

6.2.3 Cyclone Design

Stairmand developed two standard designs for gas-solid cyclones: a high efficiency cyclone and a high throughput cyclone. Each method has its own performance curve that is used depending on the method employed. The curves are used to transform the standard cyclone size to other sizes using scaling factors calculated using the following Stairmand equation:

$$d_2 = d_1 \left[\left(\frac{Dc_2}{Dc_1} \right)^3 \times \frac{Q1}{Q2} \times \frac{\Delta\rho1}{\Delta\rho2} \times \frac{\mu2}{\mu1} \right]^{\frac{1}{2}}$$

Where,

d_1 = mean diameter of particle separated at the standard conditions, at the chosen separating efficiency

d_2 = mean diameter of the particle separated in the proposed design, at the same separating efficiency

Dc_1 = diameter of the standard cyclone = 8 inches (208 mm)

Dc_2 = diameter of the proposed cyclone

Q_1 = Standard flow rate

- For High Efficiency Design (223 m³/h)
- For High Throughput Design (669 m³/h)

Q_2 = Proposed flow rate (m³/h)

$\Delta\rho_1$ = solid fluid density difference in standard conditions = 2000 kg/m³

$\Delta\rho_2$ = density difference, proposed design

μ_1 = test fluid viscosity (air at 1 atm, 20°C) = 0.018 mN s/m²

μ_2 = viscosity of proposed fluid

A high efficiency cyclone is used when a high proportion of particle size is below 10 μm. The particle size distribution for particles used in cement are as follows tabulated below:

Table 27: Raw Meal Particle Size Distribution

Size Range (μm)	Mass Percent in Size Range
0-2	4
2-6	16
6-12	20
12-20	20
20-40	20
40-80	18
>80	2

As seen here around 35% of particles are below the size of 10 μm, hence the chart for high efficiency cyclone will be used. Our main focus is to have an efficiency of **around 60-75% per cyclone** which in turn over 4 cyclones gives a **separation efficiency of 85-95%.**

These are the dimensions that will be calculated. Firstly, we will have to find the diameter of the cyclone Dc then all the values will be calculated as given by these proportions.
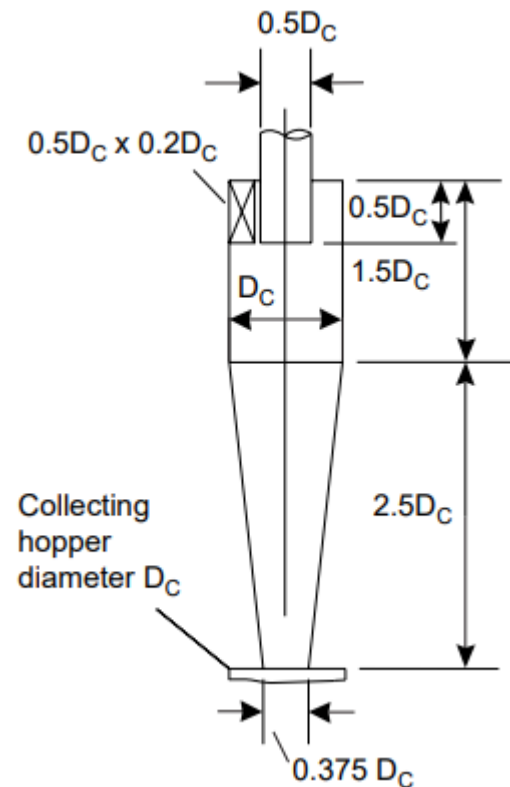
Figure 17: Design of a high efficiency Cyclone

Step 1: Finding the flowrate of the flu gases (Q2)

Mass flow rate of flu gases = 5390 ton/day (average value)

Density of flu gases from literature = 0.8 kg/m3

After using a conversion factor, we convert mass flow rate into volumetric flow rate (m3/s)

Q2 = 78.013 m³/s

Step 2: Finding a suitable diameter of the cyclone

First we take a suitable velocity from which has good separation efficiency and a low pressure drop. The following chart was consulted:
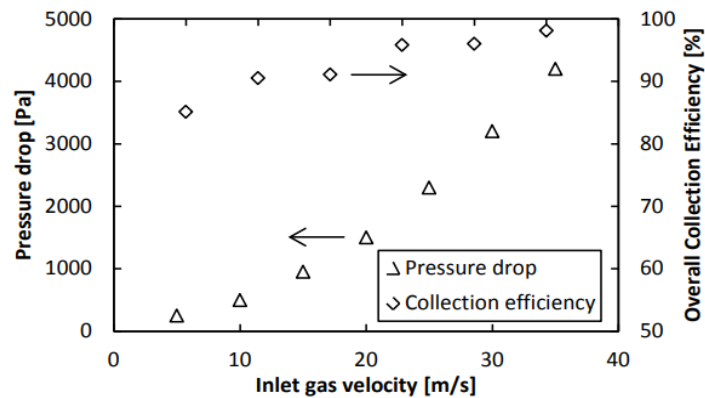
Figure 18: Pressure Drop & Efficiency vs Velocity

From this graph **velocity was taken as 20 m/s.**

Velocity was then used to calculate area of inlet duct

Area of inlet duct @ 20m/s = Volumetric Flow rate/velocity = 78.013/20 = 3.901 m^2

From the figure we can see inlet area of duct = 0.5Dc x 0.2Dc

Therefore,

$0.1Dc_2 = 3.901$ m^2

$Dc^2 = 39.01$,

Taking under root we get

Dc = 6.24 m

Clearly this diameter is very large compared to the Stairmand **standard of 0.203 m** hence the scaling factor comes out to be very large, so a suitable diameter has to be taken so the **scaling factor is not too large** to use the graph properly.

For a better scaling factor we take half of this considering 2 cyclones in parallel which is in reality a common practice and divide the diameter **by 2.**

Therefore,

Dc = 3.12 m

Step 3: Finding flowrate per cyclone in m^3/h (Q2)

Currently, flowrate = 5390 tons/day

Using conversion factor we convert it into m^3/h

Density of flu gases from literature = 0.8 kg/m^3

Q2 = 280729.167 m^3/h

Dividing this flow by 4 cyclones in series to find the flow per cyclone

Therefore,

Q2 = 280729.167/4 = 70182.25 m^3/h

Step 4: Finding the Scaling Factor using Stairmand Equation

We have Q_2, Dc_2

From literature we obtain

$\Delta\rho 2$ & $\mu 2$

$\Delta\rho 2 = 3300 \dfrac{kg}{m^3}$

$\mu 2 = 0.043$ m N/s^2

Plugging values into this equation we get the scaling factor,

$$\text{Scaling factor} = [\left(\frac{Dc_2}{Dc_1}\right)^3 \times \frac{Q1}{Q2} \times \frac{\Delta\rho1}{\Delta\rho2} \times \frac{\mu2}{\mu1}]^{\frac{1}{2}}$$

Therefore,

$$\text{Scaling factor} = [\left(\frac{3.12}{0.203}\right)^3 \times \frac{223}{70,182} \times \frac{2000}{3300} \times \frac{0.043}{0.018}]^{\frac{1}{2}}$$

Scaling factor = 4.089 ≈ 4

Step 5: Use scaling factor to calculate efficiency of the proposed design

Steps to find efficiency

- Find scaled particle size by dividing mean particle size range and scaling factor
- Use the scaled particle size to find efficiency from graph
- Multiply the efficiency and amount of particles in the specific size range
- Add them all up to find the efficiency

The following curve was used to find out the efficiency for each scaled particle size
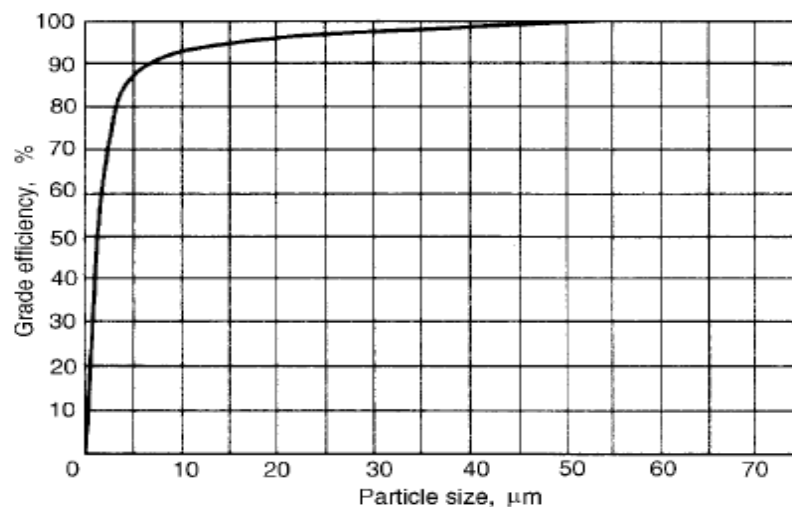
Figure 19: Grade Efficiency vs Particle Size

Performance curves, standard conditions (a) High efficiency cyclone

All the results were tabulated and are as follows:

Table 28: Efficiency Calculation

1	2	3	4	5	6
Size Range (μm)	Mean of Size Range (μm)	Mass Percent in Size Range	Mean Particle Size/ Scaling Factor	Efficiency at scaled size % (from graph)	Collected (3x5)/100
0-2	1	4	0.3	5	0.2
2-6	4	16	1.0	10	1.6
6-12	8	20	2.0	45	9
12-20	16	20	4.0	84	16.8
20-40	30	20	7.5	90	18
40-80	60	18	15.0	95	17.1
>80	80	2	20.0	97	1.94
				Efficiency of Single Cyclone	**65.0**

Thus the efficiency of a Single Cyclone is **65%** which is a modest efficiency value. Had we selected a higher velocity it would have been greater but pressure drop has to be considered as well and our next section deal with the pressure drop of the cyclone.

Dimensions of the Designed Cyclone

The dimensions of the cyclone were tabulated for reference the cyclone image is attached besides it
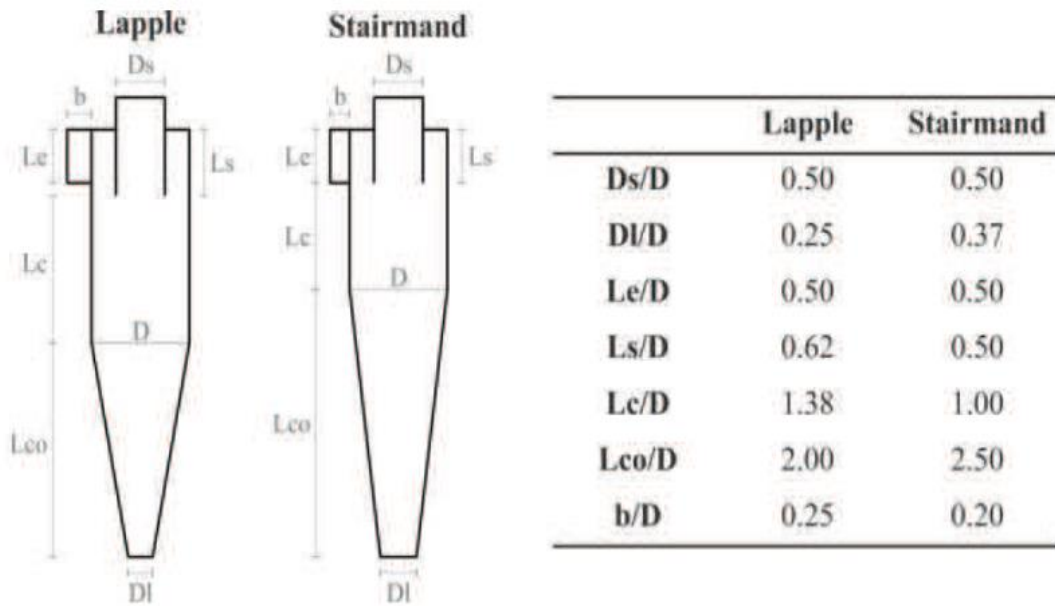
Figure 20: Co-relation between Lapple and Stairmand Cyclones

Table 29: Dimensions interconverted

Dimension	Stairmand	Lapple
Ds	1.56	1.56
Dl	1.17	0.78
Le	1.56	1.56
Ls	1.56	1.9344
Lc	3.12	4.3056
Lco	7.8	6.24
b	0.624	0.78

6.2.4 Pressure Drop in a Cyclone

The pressure drop in a cyclone is due to the entry and exit losses and friction and kinetic energy losses n the cyclone. The following empirical relation developed by **Stairmand** will be used

$$\Delta P = \frac{\rho f}{203} \left\{ u_1^2 \left[1 + 2\emptyset^2 \left(\frac{2r_t}{r_e} - 1 \right) \right] + 2u_2^2 \right\}$$

Where,

ΔP = cyclone pressure drop (mBars)

ρf = Gas Density, kg/m^3

u_1 = inlet duct velocity ($\frac{m}{s}$)

u_2 = outlet duct velocity ($\frac{m}{s}$)

r_t = radius of circle to which the centre line of the inlet is tangential, m

r_e = radius of exit pipe

$\emptyset$ = factor from chart

Another factor is also used which helps find $\emptyset$

$$\gamma = f_c \frac{A_s}{A_1}$$

Where,

f_c = friction factor, taken as 0.005 for gases

A_s = surface area of cyclone exposed to spinning fluid, m^2

A_1 = area of inlet duct, m^2

Calculation for Lapple Cyclone

Table 30: Dimensions of Cyclone

Dimension	Lapple
Ds	1.56
DI	0.78
Le	1.56
Ls	1.9344
Lc	4.3056
Lco	6.24
b	0.78
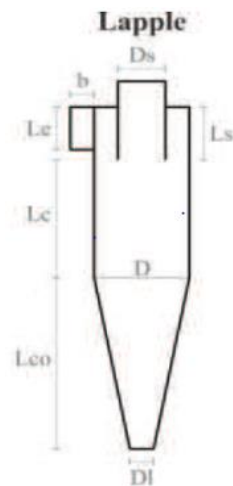
Figure 21: Dimensions of
Lapple Cyclone

Step 1: Find inlet area and Surface Area

A1 = 0.5 x 3.12 x 0.2 x 3.12 = 1.2168 m²

As = 3.12π x (4.306 + 6.24) = 103.37 m²

Step 2: Finding factor γ

$$\gamma = 0.005 \text{ x } \frac{103.37}{1.2168}$$

γ = 0.425

Step 3: Finding the ratio of $\frac{r_t}{r_e}$

$$\frac{r_t}{r_e} = \frac{3.12 - (3.12 \times 0.2)}{1.56} = 1.5$$

Step 4: Finding Ø from the chart

From this chart we get the value of Ø **= 1.1**
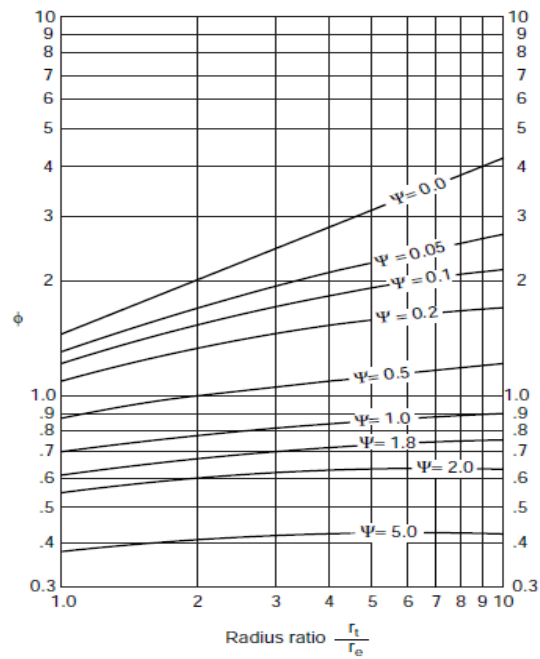
Figure 22: Chart to evaluate phi

Step 5: Find inlet gas velocity

u1= (Q2/3600) x (1/A1)

Therefore,

u1 = (70,182.5/3600) x (1/1.2168) = 16.02 m/s

Step 6: Calculating area of exit pipe

$$\frac{\pi(1.56^2)}{4} = 1.911 \ m^2$$

Step 7: Calculating outlet velocity u2

u2 = (70182.25/3600) x (1/1.911) = 10.201 m/s

Step 8: Calculate Pressure drop from **Stairmand empirical relation**

$$\Delta P = \frac{\rho f}{203} \left\{ u_1^2 \left[1 + 2\emptyset^2 \left(\frac{2r_t}{r_e} - 1 \right) \right] + 2u_2^2 \right.$$

Therefore

$$\Delta P = \frac{0.37 \ (\text{gas density from literature})}{203} \{16.02^2 [1 + 2(1.1)^2(2 \times 1.5 - 1)] + 2(10.201^2) = \textbf{311 pascals}$$

This is a good allowable limit, when we combine all 4 cyclones together our collective pressure drop is,

$$\Delta P = 4 \times 311 = \textbf{1244 pascals}$$

This pressure drop is around 1% of atmospheric pressure which is a good design value as cyclones operate at higher pressures then atmospheric pressure hence it is an acceptable value.

6.3 Design of a Furnace (Pre-Calciner)

Designing a furnace is not exactly the easiest equipment to design. It involves numerous calculations, a lot of assumptions, use of different co-relations and a lot of different graphs to perform those calculations. Even then that is just an approximate and in reality there might be a lot of heat losses and different dimensioning. Here we attempt to design a furnace basically a pre-calciner furnace in which the feed is heated up to 900 – 1100 degrees Celsius at max where 90% of calcination process is complete, then the feed enters the rotary kiln.

6.3.1 Understanding the Design

To understand the design thoroughly a lot of concepts have to be understood in order to fully understand the significance of each calculation at each step and fully grasp the concept.

To start off in every furnace specially is the heating fluid is flu gases, the main medium through which heat is transferred is radiation and convection. Radiation occurs due to refractory bricks and metal tubes in case of boilers (not used in a pre-calciner) and of course the gas itself has its own emissivity and absorptivity which aid in finding out the total heat transfer due to radiation.

The main heat transfer is due to radiation and we will be looking at those concepts first in order to have full comprehension of the design.

6.3.2 Design Pre – Requisites

Stefan Boltzmann Law

The main law governing radiation heat transfer is the Stefan-Boltzmann Law which states that the total power or energy emitted is proportional to the fourth power of the bodies absolute temperature. The law applies to bodies that act as black bodies absorbing all incident radiation within them.

The equation is as follows:

$$Q = F\sigma A'(T_1^4 - T_2^4)$$

Where,

T1 = Temperature of Source, °R

T2 = Temperature of Sink, °R

A' = Effective heat transfer of sink or cold body

σ = Stefan Boltzmann Constant (0.173×10^{-8} Btu/hr ft^2 R^4)

F = factor which allows geometry of the system and the non-black emissivities of hot and cold bodies, dimensionless

6.3.3 The Main Components of a Furnace

A furnace consists of the following essential component

- Heat Source
- Heat Sink
- Enclosing Surfaces

We will be looking at each of these in detail to understand what they really are.

Heat Sink:

So the heat sink is basically the region of low temperature to where the heat flows and further the sink radiates the heat towards the surroundings. Heat sink are usually tubes and refractory bricks on walls of the furnace, roof and floor of the furnace.
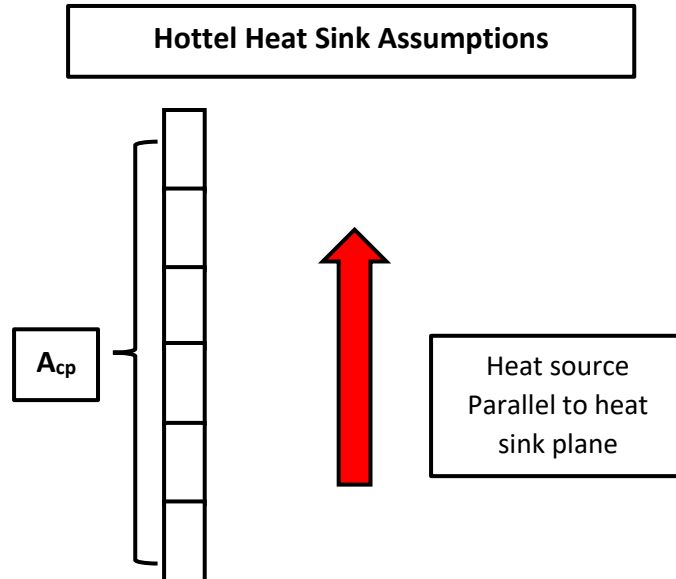
For heat sink the effective heat transfer surface has to be determined in order to further progress in design. The effective heat transfer surface is determined by method of 'Hottel'.

'Hottel' states that to evaluate the effective surface of tubes, assumption is made that the heat source is a radiating plane parallel to the heat sink. All surfaces are assumed to be black.

In short, the heat sink is considered as a straight plane having an emissivity of 1 (black body)

A_{cp} is the plane which basically is evaluated using dimensioning and area calculations.

Heat Source:

The basis of heat is the heat produced from combustion, which produces heat and combustion products carbon dioxide and water.

Carbon dioxide and water radiates the heat, sensible heat of air is also utilized if excess air is used which is most probably the case. Gas fuels provide nonluminous flames.

Oxygen, nitrogen and hydrogen have very poor emissivities as they are non - polar and hence they are not considered in the design work.

Water, carbon dioxide and carbon monoxide all contribute to emitting heat and can be explained due to the polar nature of the molecules, but carbon monoxide is not considered either due to complete combustion taking place as excess air is used.

Radiation of the gas depends upon the volume of gas and the number of radiating molecules and hence to simplify calculations a concept is introduced as the 'Mean beam length'.

Mean beam length: It is the average depth of the blanket of the flue gas in all directions for each point on the bounding surface of the furnace.

The mean beam length is used instead of cubical measure of the volume. Concentration is measured in terms of partial pressure.

Hence the emissivity is calculated using the following formula

$\in_g = \rho L$ in $(atm - ft)$

P = it is the partial pressure obtained at a given temperature and pressure using equilibrium relations.

L = It is the length over which the gas is spreading

If more than one radiating constituent is present the emissivities are additive with a small correction factor. Once the emissivity is calculated the amount of heat transfer to a black body is determined.

In calculating emissivity from a gas mass allowance must be made for the temperature of both the source and sink and hence is evident by the formula.

Heat transfer by a gas to a black body is given by the formula,

$$Q_b = 0.173 F_{bA} \left[\in_g \left(\frac{T_G}{100} \right)^4 - \in_g \left(\frac{T_B}{100} \right)^4 \right] Ab'$$

Where,

T_G = temperature of gas, °R

T_B = temperature of black body, °R

Ab' = Heat transfer surface of black body, ft^2

The formula gets slightly modified when two constituents are present as the emissivity of the other component is also added. As in the case of furnace, emissivities of carbon dioxide and water are considered and the final equation is as follows

$$Q_b = 0.173 F_{bA} \left[\in_g \left(\frac{T_G}{100} \right)^4 - \in_g \left(\frac{T_B}{100} \right)^4 \right] Ab'$$

Where now,

$\in_g = \in_c + \in_w$ found at TG and TB respectively

$$Q_b = 0.173 F_{bA} \left[(\in_c + \in_w) \left(\frac{T_G}{100} \right)^4 - (\in_c + \in_w) \left(\frac{T_B}{100} \right)^4 \right] Ab'$$

The emissivities are calculated at their respective partial pressures, mean beam lengths and temperatures. Temperatures are the gas temperature and the black body temperature.

For convenience there are charts which give the values of the radiant heat transfer flux q_c, q_w and q_b.

q_c is the flux of carbon dioxide at the specific temperature, partial pressure and mean beam length.

q_w is the flux of water vapors at the specific temperature, partial pressure and mean beam length.

q_b is the flux of the black body having emissivity as 1.

These charts are developed by Hottel and Egbert which directly give the value of heat flux and that is further used to calculate the emissivity of the gas including a correction factor. Correction factor is introduced because part of the gas that emits the heat also absorbs it hence the overall heat flux is slightly less than as calculated. It is known as the correction of interference and hence is used when calculating the overall gas emissivity. The following formula is used to calculate the overall gas emissivity:

$$\in_g = \left[\frac{(q_c + q_w)_{T_G} - (q_c + q_w)_{T_S}}{(q_b)_{T_G} - (q_b)_{T_S}} \right] \frac{100 - \%}{100}$$

Where q_b is calculated using the following formula

$$q_b = 0.173 \in_b (\frac{T}{100})^4$$

Once we find the emissivity of the gas we find the total heat transfer by the gas using Stefan Boltzmann equation.

Last but not least the mean beam length is calculated by getting to know what ratio's do the length, height and width exist in. Those ratios are then used to determine the co-relation of mean beam length. The table was again coined up by Hottel and is as follows:

Dimensional ratio*	Mean length L, ft
Rectangular furnaces:	
1. 1-1-1 to 1-1-3	⅔ ∛͞F͞u͞r͞n͞a͞c͞e͞ ͞v͞o͞l͞u͞m͞e͞,͞ ͞ft³
1-2-1 to 1-2-4	
2. 1-1-4 to 1-1-∞	1.0 × smallest dimension
3. 1-2-5 to 1-2-8	1.3 × smallest dimension
4. 1-3-3 to 1-∞-∞	1.8 × smallest dimension
Cylindrical Furnaces:	
5. $d \times d$	⅔ diameter
6. $d \times 2d$ to $d \times \infty d$	1 × diameter
Tube Banks:	
7. As in convection sections	L (ft) $= 0.4P_T - 0.567$ OD, in.

* Length, width, height in any order.

Figure 23: Mean length of radiant beams in various gas shapes

Two important factors to note is that the gas emissivity ($\in_g$) depends on:

- L, the mean beam length
- *P*, the partial pressure of the gas

If the mean beam is large that means the emissivity is large and hence greater heat transfer and if there is high amount of excess air that means more amount of nitrogen which decreases the partial pressure of the flu gases hence decreasing the emissivity and the heat transfer.

Enclosing Surfaces

This is the last essential component of a furnace. The enclosing surfaces consists of roof, refractory walls, arch and the floor of the furnace. The heat is basically

transferred from the gas to the cold surfaces which basically are the enclosing surfaces. Heat is transferred in the form of radiations which is directed from various sections of the gas on a particular point. The gas mass also radiates in other directions which also is directed towards the refractory bricks. The bricks have an emissivity of their own which reradiate or reflect the 'impinging radiation' some directed to the point it came from.

Hence the point to note is that the cold surface receives more radiation than evaluated from the emissivity of the gas. Therefore, high ratio of refractory to cold surface ensures the emissivity is high and near the black body value of 1, of the furnace.

Net radiant heat transfer increases for a given

- Gas emissivity
- Cold surface emissivity
- Gas and cold surface temperature

By increasing the ratio of the refractory to cold surfaces. Hence the area of refractory should be high in order to achieve a good heat transfer per unit area. Ratio of refractory to cold surface may thus be increased but at an expense of decreasing the mean beam length for the furnace so again a compromise has to be made between deciding about ratio of refractory to cold surface and mean beam length of the furnace.

An exchange factor for a furnace has been developed by Hottel which relates the emissivity of the cold surface and furnace cavity.

$$F_c = \frac{1}{\left(\frac{1}{\epsilon_f}\right) + \left(\frac{1}{\epsilon_s}\right) - 1}$$

Hottel defines ϵ_f as a function of gas emissivity, the ratio of refractory surface to cold surface and the angle factor 'F'.

Summary

- The refractory surface is evaluated as an equivalent plane surface with area A_{cp}

- $\in_g$ is a function of ($T_{hotsurface}$, T_{cold} surface, L of furnace, partial pressure of the gas)
- $\in_f$ is a function of ($\in_g$ and ratio of refractory to cold surfaces)
- Overall exchange factor 'F' may be obtained from the furnace and cold surface emissivities for use in the Stefan Boltzmann equation to find 'Q'. (Radiant heat transfer)
- $Q = 0.173 F [\in_g (\frac{T_G}{100})^4 - \in_g (\frac{T_B}{100})^4] A_{cp} \propto$
- α = effectiveness factor, emissivity of the plane surface
- In conclusion it is often satisfactory to consider the temperature of the gas TG, leaving the furnace radiant section as the average temperature when the degree of turbulence of the gases is high.

6.4 The Design

There are 4 main design methods which allow us to calculate the heat absorption in furnace radiant sections.

The 4 methods are:

- Method of Lobo and Evans
- Method of Wilson, Lobo and Hottel
- The Orrok Hudson Equation
- Wohlenburg Simplified Method

We employed the Method of Lobo and Evans because this method covers the type of furnace we are interested to design.

6.4.1 Method of Lobo and Evans

This method makes use of the overall exchange factor *F* and a Stefan Boltzmann type equation. It has a good theoretical basis and is used extensively in refinery-furnace design work. It is also recommended for oil or gas fired boilers.

Given by this method the total heat transfer to a cold surface is given by the following equation which incorporates both heat transfer by radiation and convection.

$$\sum Q = 0.173F\left[\epsilon_g\left(\frac{T_G}{100}\right)^4 - \epsilon_g\left(\frac{T_B}{100}\right)^4\right]A_{cp}\propto + h_cA(T_G - T_s)$$

hc is normally taken as 2 and A of convection is taken as A = $2.0A_{cp}\propto$

Heat balance is as follows

Q = QF + QA + QR + QS − QW − QG

Where,

Q = total radiant-section duty, Btu/hr

QF = heat liberated by the fuel, Btu/hr (lower heating value)

QA = Sensible heat of combustion air above 60 ˚F, Btu/hr

QR = Sensible heat of recirculated flu gases above 60 ˚F, Btu/hr

QS = Sensible heat above 60 ˚F in steam used for oil atomization, Btu/hr

QW = heat loss through furnace walls, Btu/hr (1 to 10 percent of QF, depending on the size, temperature and construction of the furnace. 2 percent is a good furnace design figure)

QG = heat leaving the furnace radiant sections in the flu gases, Btu/hr

As a further simplification QS is considered negligible and the net heat liberation is

Qnet = QF + QA + QR − QW - QG

Where QG is evaluated by the following equation,

QG = W (1 + G')Cav(TG − 520)

Where,

W = fuel rate

(1 + G') = ratio of flu gases leaving the radiant section to fuel fired, lb/lb

G' = ratio of air to fuel, lb/lb

Cav = average specific heat of flu gases between TG ˚R and 520 ˚R, Btu/(lb)(˚F)

Furthermore, in the calculations area of cold plane surfaces are evaluated, total area of furnaces and effective refractory surface

The relation is as follows:

$A_R = A_T - \propto A_{cp}$

Lastly efficiency can be evaluated using the following formula

$$e = \frac{\Sigma Q}{Q_F} \times 100\%$$

In the following section the design will be seen in detail step by step using all the co-relations, equations, tables and empirical equations. We will be calculating the radiant heat transfer flux of the furnace of a given furnace whose dimensions are known.

6.5 Design Calculations for Natural Gas run furnace
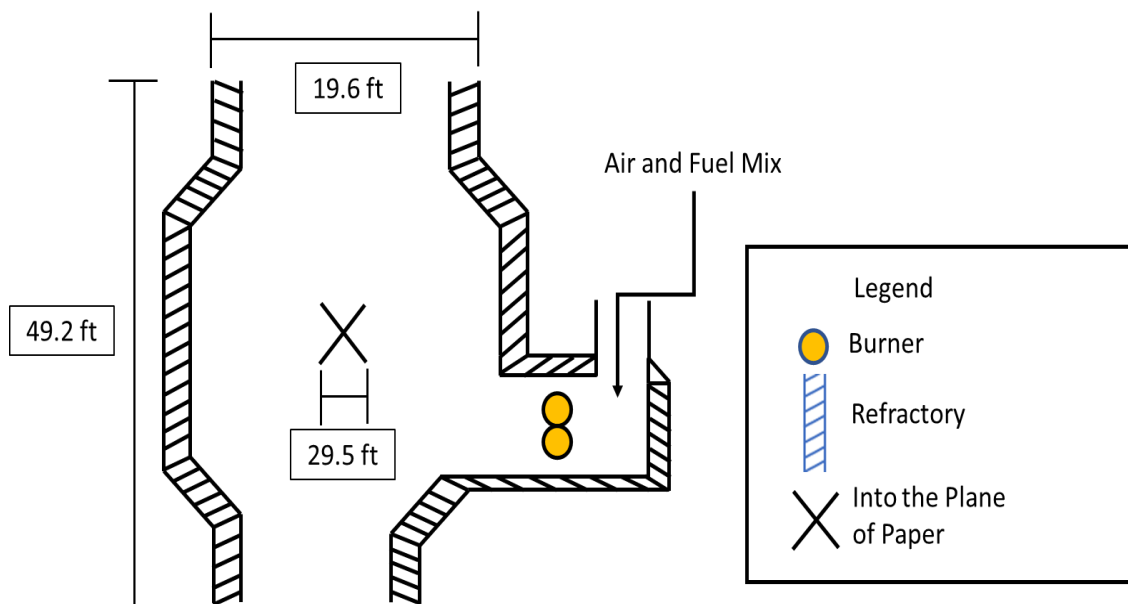
Figure 24: 2D view of the furnace

For the furnace above the heat flux will be calculated using the following steps when using natural gas as a fuel. Of course as natural gas has a higher LHV it is expected that the heat flux will be higher in natural gas run furnace.

Step 1: Finding Q_{net}

Total Duty of Pre-calciner = 10223877550 KJ/day (from energy balance)

After converting it into Btu/hr

Total Duty of Pre-calciner = 404000000 Btu/hr (75% efficiency of furnace)

Natural Gas LHV = 19,500 Btu/lb gas

A/F Ratio = 17.3 lb-air/lb fuel

Inlet Air Temperature = 2642 °F (coming from clinker cooler)

Temperature of Cold Surfaces, Ts = 900 °F

T_{AD} of (CH4) = 3565.4 °F

Exit Temperature of Gases T_G = (3565.4 + 1652)/2 = 2607.8 °F

Heat Liberated by Fuel, Q_F = (404000000/0.75) = 538000000 Btu/hr (75% efficient furnace)

Fuel quantity required = (538000000/19,500) = 27,589.8 lb/hr

Air required = 27,589.8 x 17.3 = 477,303 lb/hr

C_p of flu gases = 0.31

Q_A = 0.31 x 477,302.7 x (2642 – 60) = 376000000 Btu/hr

Q_S = negligible

Q_W = 2% of QF = 0.02 x 538000000 = 10760000 (heat loss through furnace walls)

Q_G = (787.4) (27589.75 + 477302.7) = 400000000

Q_{net} = Q_F + Q_A – Q_W – Q_G = 503240000 Btu/hr

Step 2: Finding Effective Furnace Area

$\propto A_{cp}$ = 1875 ft² , where α = 0.8 for bricks (from literature)

Total Area of Refractory Surface

Area of Side Wall = 32.8 x 29.5 = 967.6 ft^2

Area of top surface = 19.6 x 29.5 = 578.2 ft^2

Area of side Arches = 2 x (1/2 x 8.2 x 19.6) = 80.36 ft^2

Front Back Wall Area = 49.2 x 19.6 x 2 = 1928.64 ft^2

Total Refractory Surface Area = 3554.8 ft^2

Effective Furnace Area given by:

$A_R = A_T - \propto A_{cp}$

A_R = 3554.8 – 1875 = 1680 ft^2

Ratio of Refractory to cold surfaces = 1680/1875 = 0.896 (<1, hence refractory to cold surface ratio less but mean beam length will be high)

Step 3: Mean Beam Length Calculation

First we have to find out ratio of the three largest dimensions, H : W : L

The three main lengths = 49.2 : 29.5 : 19.6

Simplified ratio = 2.5 : 1.51 : 1 $\approx$ 3 : 2 : 1

Using Hottel's Table to find mean beam length equivalent

Dimensional ratio* Rectangular furnaces:	Mean length L, ft
1. 1-1-1 to 1-1-3 1-2-1 to 1-2-4	$\frac{2}{3}$ $\sqrt[3]{\text{Furnace volume, ft}^3}$
2. 1-1-4 to 1-1-∞	1.0 × smallest dimension
3. 1-2-5 to 1-2-8	1.3 × smallest dimension
4. 1-3-3 to 1-∞-∞	1.8 × smallest dimension
Cylindrical Furnaces:	
5. $d \times d$	$\frac{2}{3}$ diameter
6. $d \times 2d$ to $d \times \infty d$	1 × diameter
Tube Banks:	
7. As in convection sections	L (ft) = $0.4P_T - 0.567$ OD, in.

* Length, width, height in any order.

Figure 25: Mean Beam Length for different furnace dimension

We will be classifying this furnace for simplification as a rectangular furnace with ratio 3 : 2 : 1 and hence the mean beam length is equal to,

Mean Beam Length = 2/3 x $\sqrt[3]{\text{Furnace }volume}, ft^3$

Therefore, L = 2/3 x $\sqrt[3]{49.2 \text{ x } 29.5 \text{ x } 19.6}$ = 20.35 ft

Step 4: Calculation of Gas Emissivity

First partial pressures had to be obtained for water vapor and carbon dioxide

Partial Pressures were obtained from online calculators which give the value of partial pressures at equilibrium at the conditions stated.

Partial Pressures

PCO2 = 0.1052 atm

PH2O = 0.1192 atm

PCO2 x L = 0.1052 x 20.35 = 2.14 atm-ft

PH2O x L = 0.1192 x 20.35 = 2.43 atm-ft
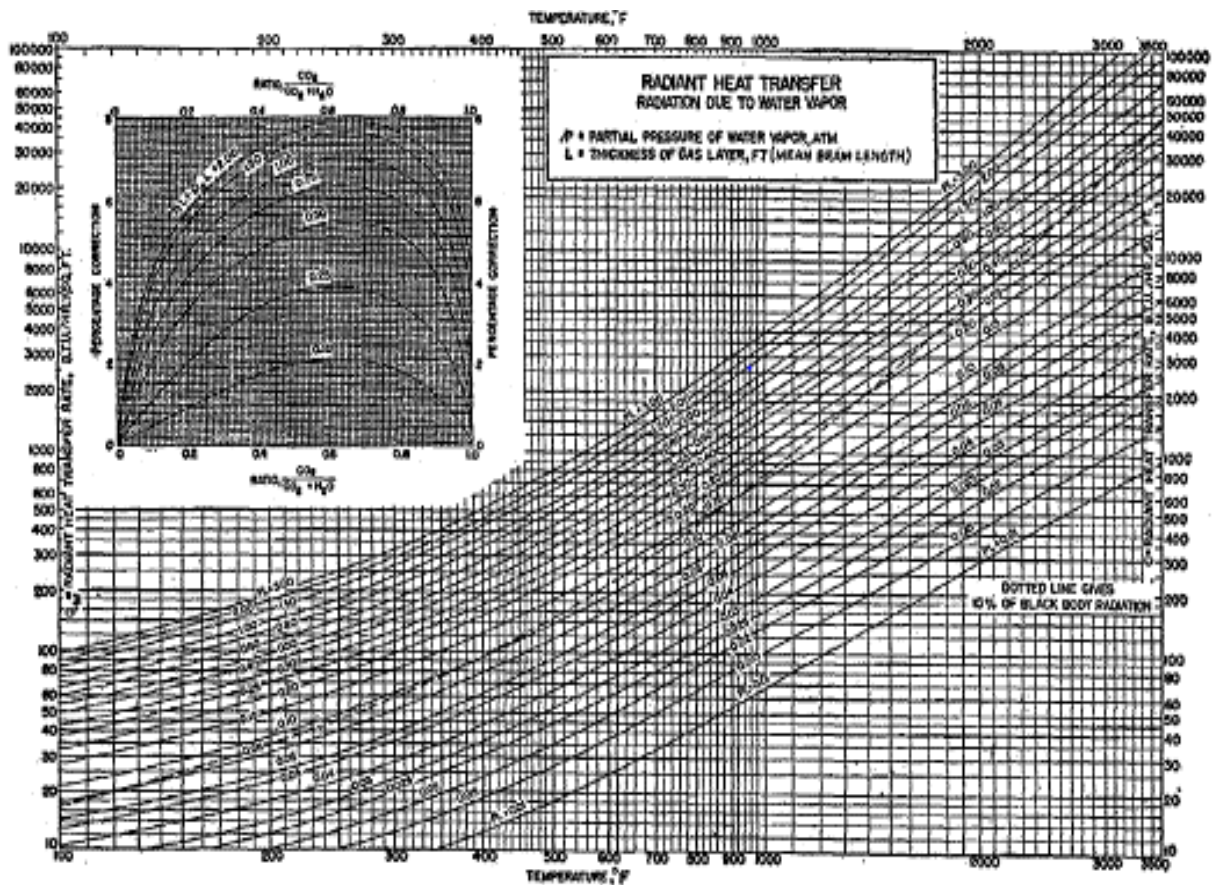
Figure 26: Radiant Heat Transfer for Water

Now Charts developed by Hottel will be used to find radiation by Carbon Dioxide and radiation by water which will help us find the gas emissivity.

Radiation due to Water vapor
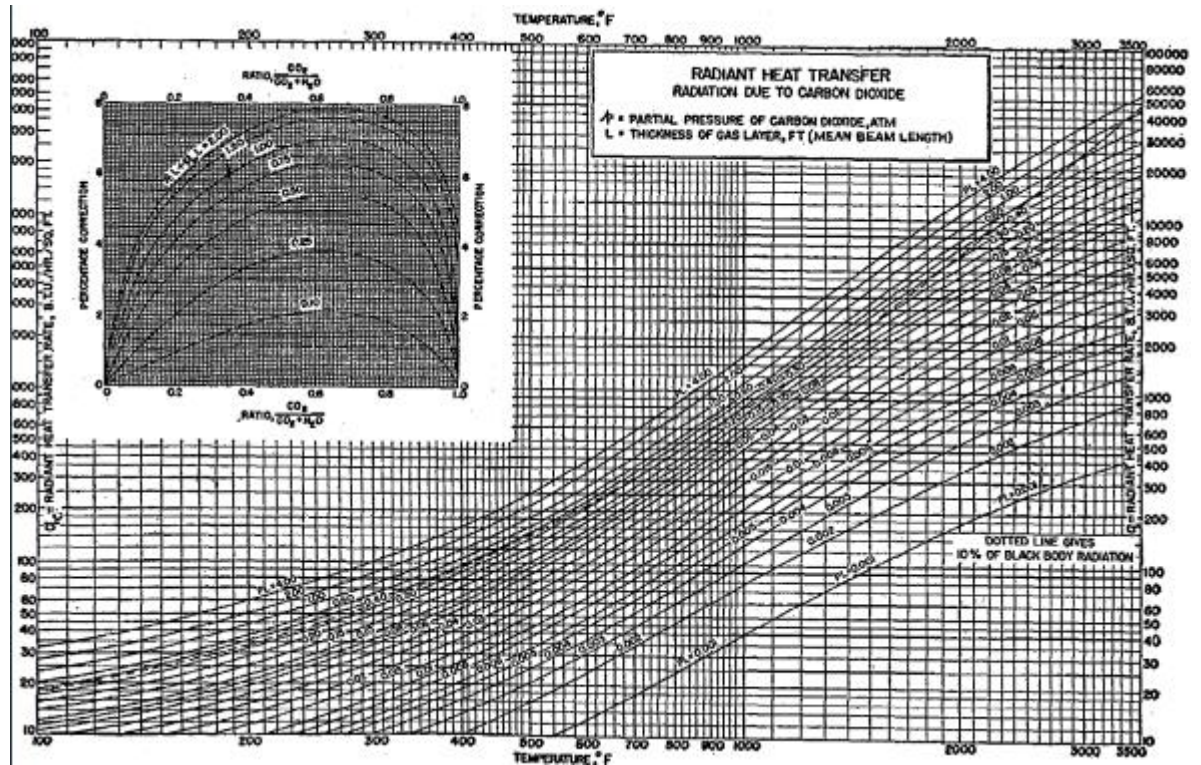
Figure 27: Radiant heat transfer for carbon dioxide

Using the calculated emissivities of carbon dioxide and water, and temperature TG and TS we find the values of q_{CO2} and q_{H2O}

Table 31: Heats at TS

Property	Value
TS	900°F
ε_{CO2}	2.14
ε_{H2O}	2.43
q_{CO2} (from graph)	1000 Btu/hr
q_{H2O} (from graph)	2800 Btu/hr

Table 32: Heats at TG

Property	Value
TG	2600°F
ε_{CO2}	2.14
ε_{H2O}	2.43
q_{CO2} (from graph)	**25000 Btu/hr**
q_{H2O} (from graph)	**30000 Btu/hr**

% correction at $(PCO2) / (PH2O + PCO2)$ = 0.1052 / (0.1052 + 0.1192) = 0.468

$(PCO2$ x L$) + (PH2O$ x L$)$ = (0.1052 x 20.35) + (0.1192 x 20.35) = 2.14 + 2.43 = 4.67 atm-ft

Using this graph we will find the correction factor to be used while calculating the emissivity of the gas.

% correction @ 0.467 and 4.57 atm-ft = 10%

$(q_B)_{TG}$ = 0.173 x 1 x ((2607.8 + 491.67)/100)^4 = 151,483 Btu/hr

Figure 28: Correction Factor Chart

$(q_B)_{TS}$ = 0.173 x 1 x ((900 + 491.67)/100)^4 = 5918 Btu/hr

Gas Emissivity is given by

$$\epsilon_g = [\frac{(q_c + q_w)_{T_G} - (q_c + q_w)_{T_S}}{(q_b)_{T_G} - (q_b)_{T_S}}]\frac{100 - \%}{100}$$

$$\epsilon_g = [\frac{(25000+30000)_{T_G} - (1000+2800)_{T_S}}{151483 - 5918}]\frac{100-10\%}{100} = 0.32$$

Step 5: Finding overall Exchange Factor F

Using Gas Emissivity (0.32) and Ratio of refractory to cold surfaces (0.896) we use the following graph to find the overall exchange factor F.

Overall Exchange Factor F = 0.43 (from graph)
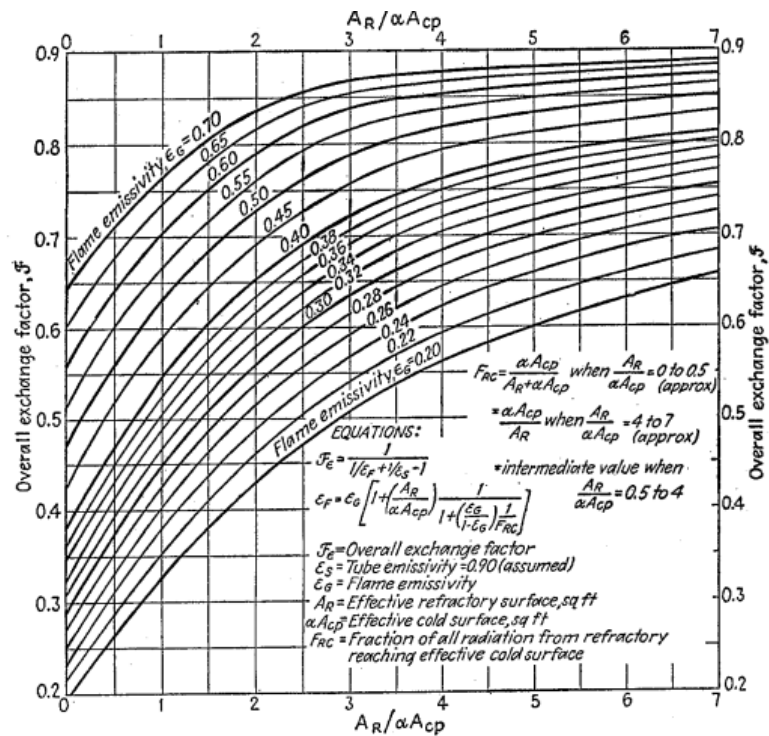
Figure 29: Overall heat-exchange factor in radiant section

Step 6: Finding Radiant Section Average Flux

$$\text{Radiant Section Average Flux} = \frac{\Sigma Q}{\propto A_{cp} \text{ x } F} = \frac{503240000}{1875 \text{ x } 0.43} = 0.624 \frac{\text{MBtu}}{\text{hr} ft^2}$$

6.6 Conclusion

We get a value of 0.624 MBtu / hr ft^2, when the fuel is changed from coal to natural gas   . When using coal a value of 0.54 MBtu / hr ft^2 was reported hence it is clear that using natural gas does actually increase the radiant section quite significantly and hence is the better choice of fuel with regards to consumption, power output and greenhouse emissions.

6.7 Design of a Ball Mill

6.7.1 Introduction

Ball mill is essentially a size reducing equipment which crushes large sized rocks into small sized particles with a particle size distribution depending on the size and the amount of balls in the ball mill.

6.7.2 Operating Principle

It works on the principle of impact and attrition

6.7.3 Working

So basically, a ball mill consists of a hollow cylinder shell rotating about its axis. The cylinder is filled with balls. The balls may be made up of steel or ceramic material. The inner surface is lined up with an abrasion resistant material to prevent wear and tear of the ball mill's internals. Usually the diameter of the ball mill equal to the length of the ball mill.

6.7.4 Critical Speed

The ball mill works on the concept of critical speed. It is the speed of rotation in which the balls move along the radius of the mill and hence no crushing takes place. This is due to the fact that centrifugal force of the balls equal the force of gravity and hence the balls do not fall back. Hence it is the most important design feature when designing a ball mill. In the stages of design critical speed of ball mill is determined and then the RPM is set 70% – 80% of the critical speed.

6.7.5 Derivation of Critical Speed

At critical speed the centrifugal force on balls is equal to force of gravity on the moving balls. Thus, we can write:

$mv^2/r = mg$

$v^2/r = g$

$v^2 = gr$

$v^2 = g(D/2)$

$v^2 = (g/2)D$

Taking square root on both sides

$v = (g/2)^{0.5} (D)^{0.5}$

Putting g= 9.8 m/s^2 = 115718.4 ft/min^2

$v = 240.53 D^{0.5}$

Now for finding the frequency of rotation, we use the relation v = rw = (D/2)w

r = radius of circular path

and w= frequency of rotation (RPM)

So, $(D/2) w = 240.53 (D)^{0.5}$

$w = 481.06/D^{0.5}$ rad/min

Since, 1 rev = 6.28 rad

$w = (481.06/6.28)/D^{0.5}$ rev/min

$w = 76.6/(D^{0.5})$ rev/min

6.7.6 Calculation of RPM

- Ball Mill RPM set according to the critical speed
- RPM is 70 – 85% of critical speed
- Critical speed is speed when centrifugal force equals force of gravity

Critical Speed:

Gravitational force = centrifugal force

$$\omega = \frac{76.6}{\sqrt{D}} \frac{rev}{min}$$

$$\omega = \frac{76.6}{\sqrt{3}} = 44.3 \frac{rev}{min}$$

RPM = 44.3 x 0.8 = 35.4 rev/min

PROCESS SIMULATION & CONTROL

7.1 Simulation

Simulation was one of the most important aspects of our project. We performed simulation to check whether our claims are true or not. Secondly we also wanted to see whether our calculated material and energy balances were correct or not.

7.1.1 Simulation Properties

So to begin with we performed our simulation on Aspen Plus as our plant mainly deals with solid handling hence Aspen Plus had all those solid components we required for our simulation.

Our Fluid package was selected as Solids as seen below:
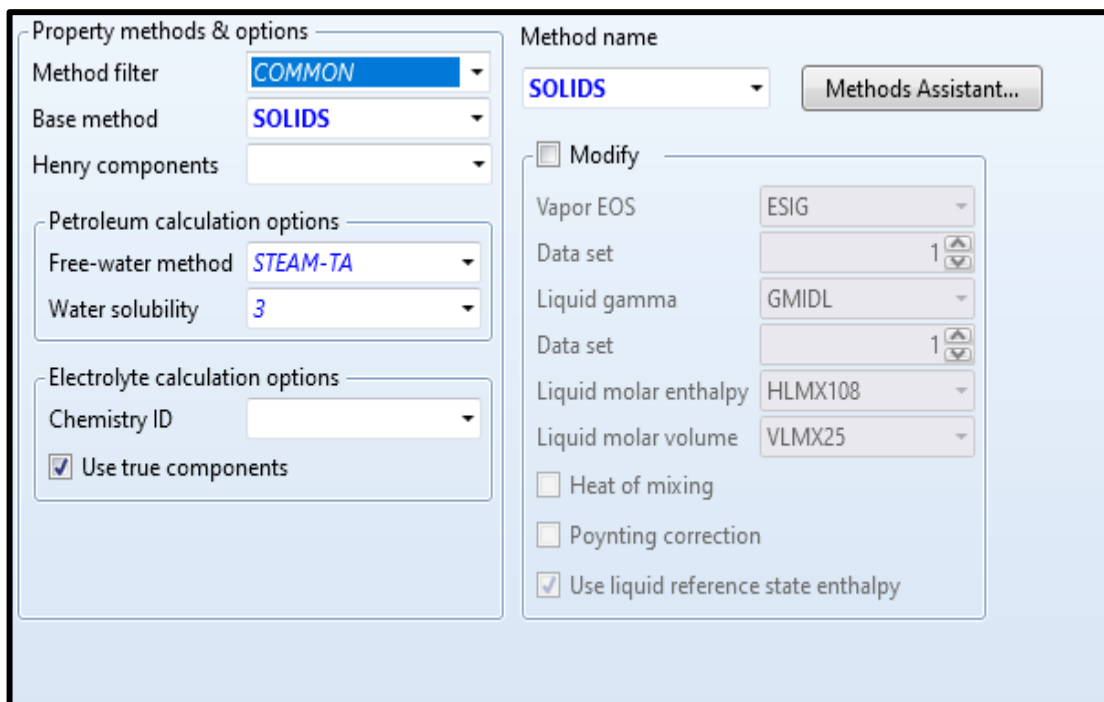
Figure 30: Fluid Package

Next we had to define our component list comprising of all the constituents of cement and the two fuels that we would be comparing against each other for the carbon dioxide emissions. The component list was as follows:

Figure 31: Component List

Once this was defined we had an option to select simulation including solids without particle size distribution or with particle size distribution but as we had the particle size distribution for each of our solids we defined each solid's

Figure 32: Defining Solids

particle size distribution to make the model realistic. The distributions were defined as follows:
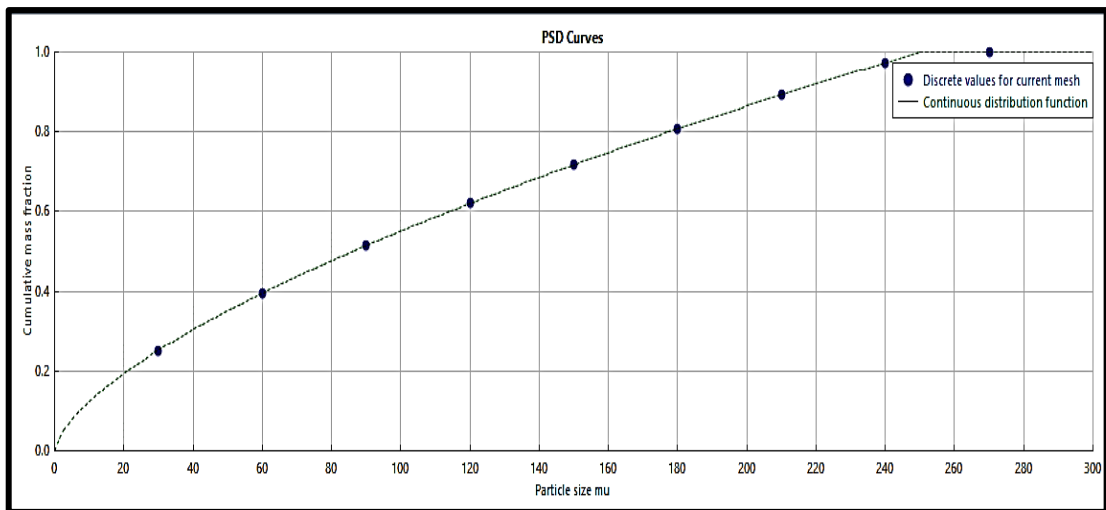
Figure 33: Particle Size Distribution of Solids

The graph shows the size distribution variation over the range of the sizes. This procedure was done for all solids.

Finally, the model was made.

7.1.2 Simulation Model
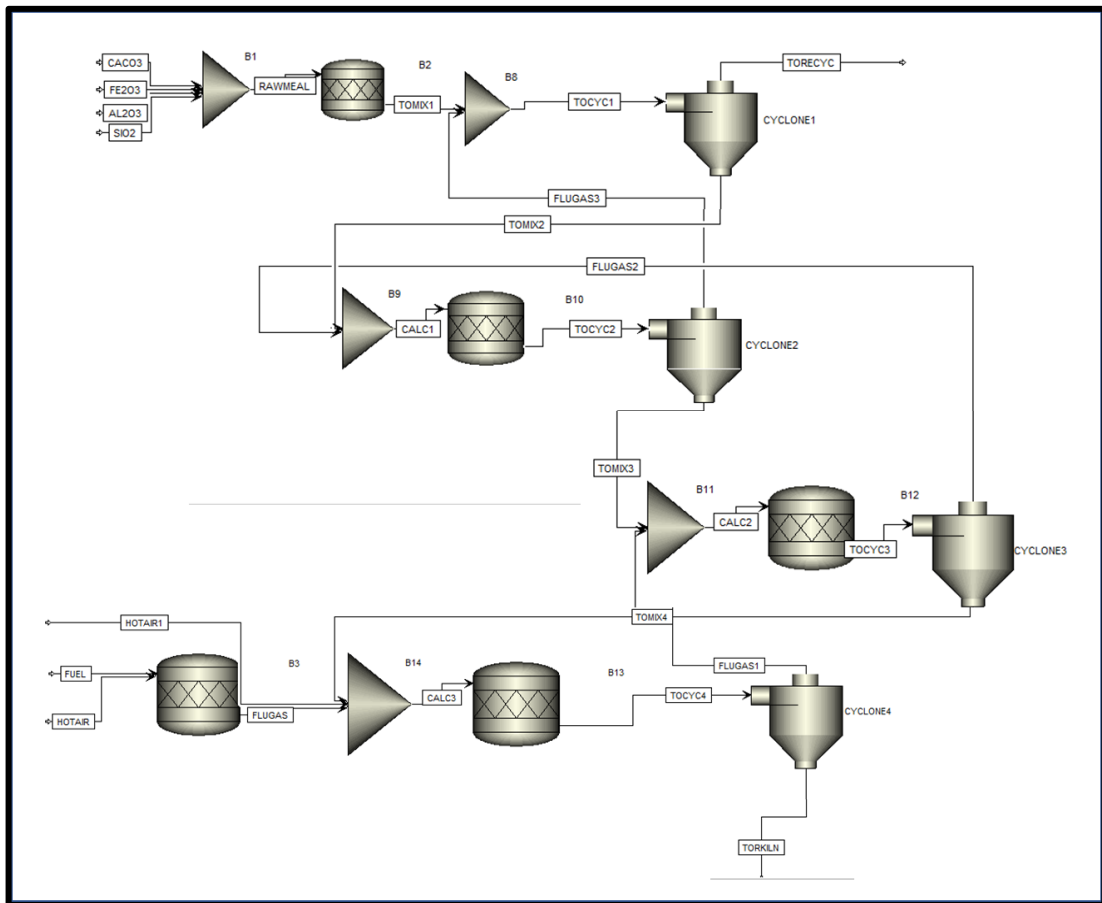
Figure 34: Simulation

So, our main aim was to simulate the suspension preheating section of the cement process. The bottom left reactor is the pre-calciner. There are 4 cyclones and before each cyclone we have set up a reactor to account for all the reactions taking place in the cyclone because on Aspen Plus you cannot carry out a reaction in a cyclone hence a reactor has to be placed just before the reactor. So basically, the main heat source is the heat of the fuel and the heated air which is by-product of the clinker cooler. That heat is utilized to heat the raw meal and carry out the reaction required to form calcium oxide. Hence, a lot of carbon dioxide is involved, and we will be comparing two fuels one by one input into the reactor.

The pre-calciner was set up as follows:

Figure 35: Reaction sets for pre-calciner

First the reactions were input presence of any fuel converged the reactor provided all the conditions were met.

Figure 36: Pre-calciner conditions

The reactor was set to 1 bar pressure and the duty was 0. This meant that no external heating was provided nor there were any heat losses out of the system and thus all heat generated would surely be used to heat the raw meal.
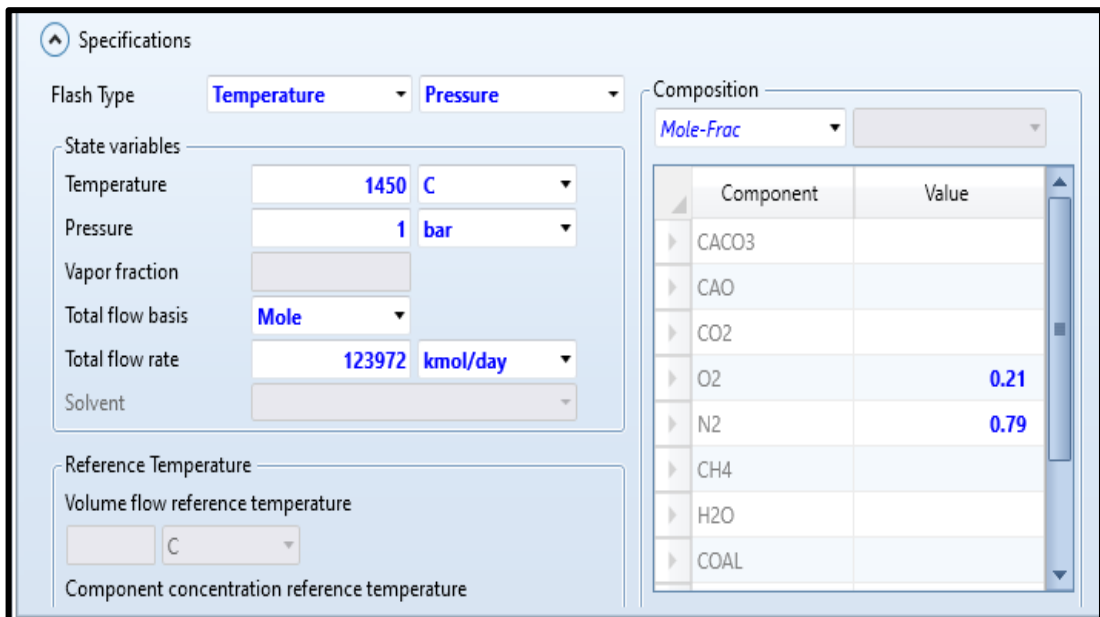
Figure 37: Air flow to pre-calciner

This was the air feed entering into the furnace coming in from the clinker cooler. We converted all our flows into moles to make the calculations easier and the model less complex.
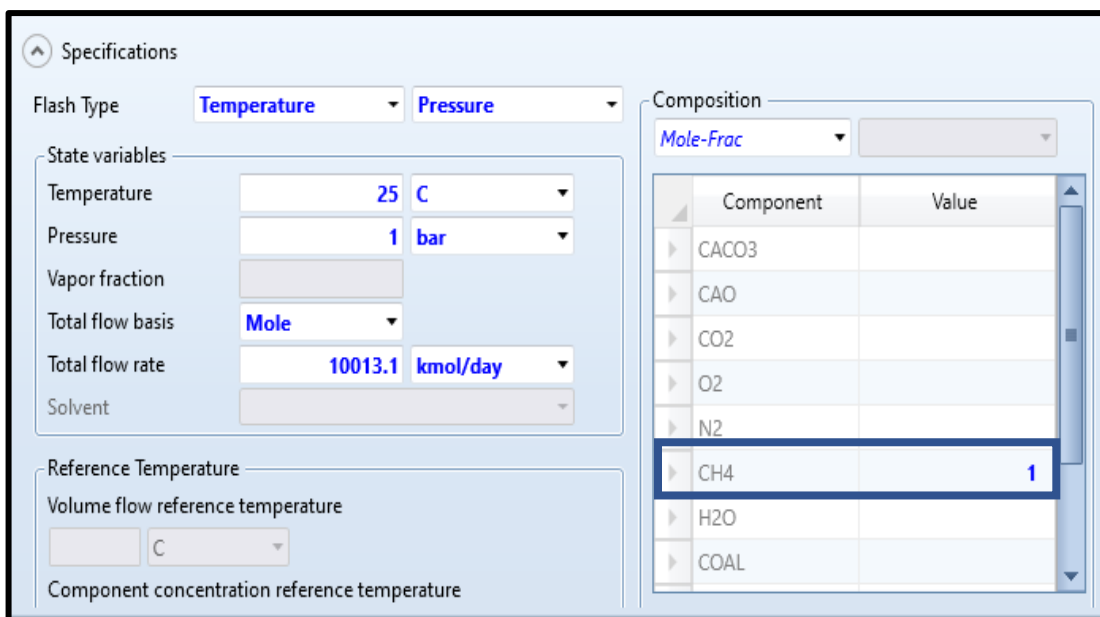
Figure 38: Amount of methane input

From our material balance we found out the value of natural gas fed into the furnace and hence this amount was input. Amount of air was varied for coal of course but aim was that the same final out temperature should be achieved.

The following amount of coal was used:

Figure 39: Amount of coal input

For this again we had to define the particle size distribution as coal is solid in nature.

7.1.3 Reactor and Cyclone Properties

We had to simulate the reactors as well. Conversion reactors were sought because we knew how much conversion was taking place at each stage from literature and Bestway Cement. Hence, we used those conversions at each stage. An example of how reactor properties were defined are given below:

Figure 40: Reactor conditions

Again the pressure was chosen as atmospheric that is the pressure of the cyclone because these reactors were just before the cyclone. The duty was kept as 0 because we are considering adiabatic conditions and all the heated air that is flowing into the reactor is used to heat the feed without any external heat source or losses.

Thermal decomposition reaction set was developed for each of these reactions with different conversions at each stage.
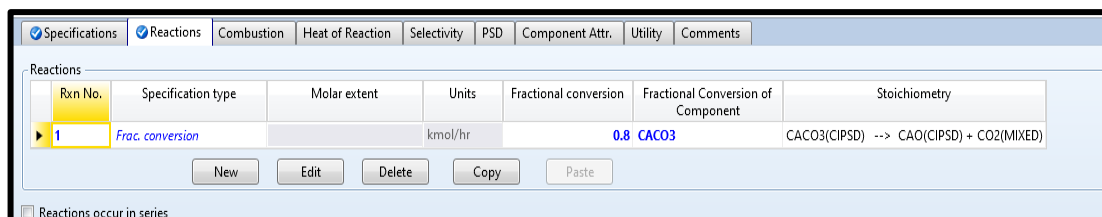
Figure 41: Reaction set for reactor

Finally the cyclone separator was also simulated according to the values we got from our design. Those values were used to simulate the design for our cyclone

The cyclone type selected was Lapple which is used in the cement industry. The diameter was set as 3 meters. We calculated in design as 3.12 meters but for simplification we took it as 3 meters. Then finally we input all our material and energy balance calculated values and ran the simulation. Gladly it did without any errors.
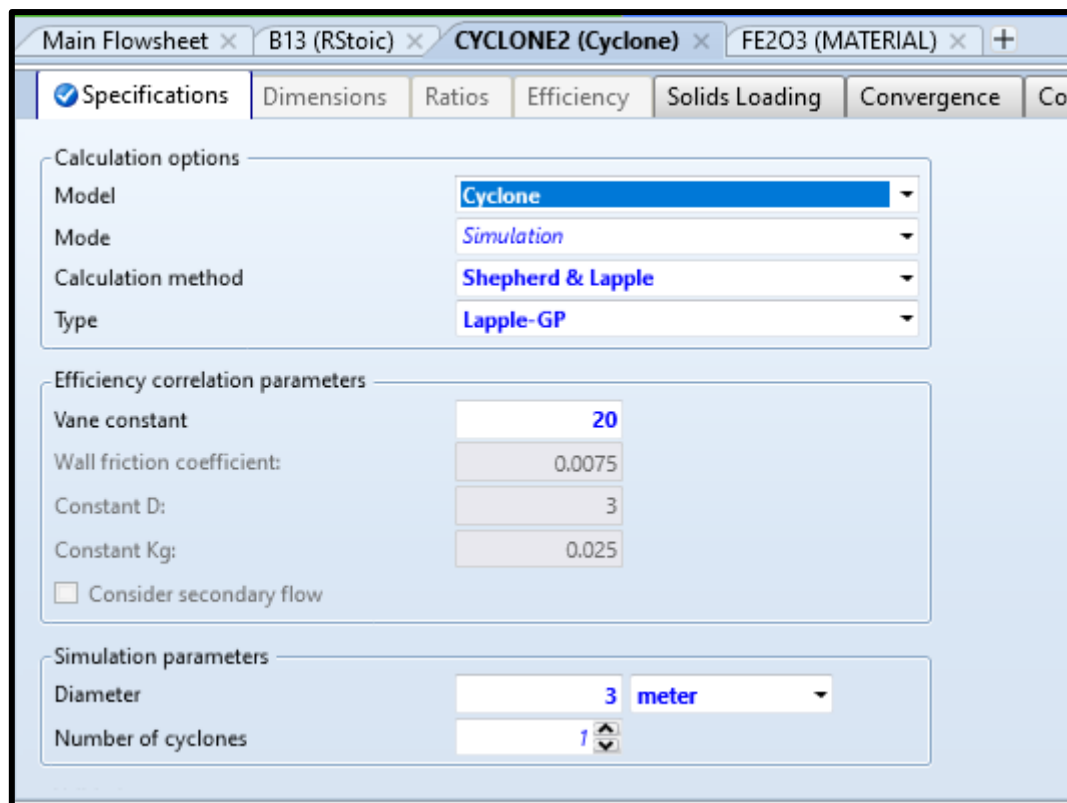
Figure 42: Cyclone specifications

7.2 Simulation Results

First we ran the simulation for coal and our main concern was stream flowing out of the system from the very end which contained all the accumulated carbon dioxide emissions in it. Around 161,000 kg/hr carbon dioxide was being released into the atmosphere which surely is a high amount.

Figure 43: Amount of CO_2 emitted by coal

Moving on we ran the simulation on natural gas and the amount of carbon dioxide in the stream flowing out was as follows:

This is clearly lesser than value obtained from coal which is about 139,000 kg/hr emitted for the same heat load.

Figure 44: Amount of CO_2 emitted by methane

Finally our last focus was to obtain the same outlet temperature of the raw meal entering the rotary kiln. The temperature of the rotary kiln on outlet was obtained as

	Units	TORKILN
Description		
From		B7
To		
Stream Class		MIXCIPSD
Maximum Relative Error		
Cost Flow	$/hr	
− Total Stream		
Temperature	C	1079.3

Figure 45: Temperature of raw meal to kiln

This temperature is slightly higher than what is achieved in the cement plant. In the cement plant the temperature is around 900 – 950 °C. this can be explained due to the fact that we neglected heat losses from the pre-calciner and from the reactors hence the temperature is slightly over other than that the simulation ran without any errors and was a successful one aligned to our aims.

7.3 Control

Process control is an important aspect of any plant as it helps automatically control processes or generate alerts when a system is malfunctioning or requires continuous control throughout. A good control is one which maintains the set point throughout the course of the process or operation and takes necessary actions when there is any deviation in the process. We designed a control for controlling the flowrate of air according to the flow of the fuel and the air/fuel ratio in the pre-calciner furnace.

Control Diagram
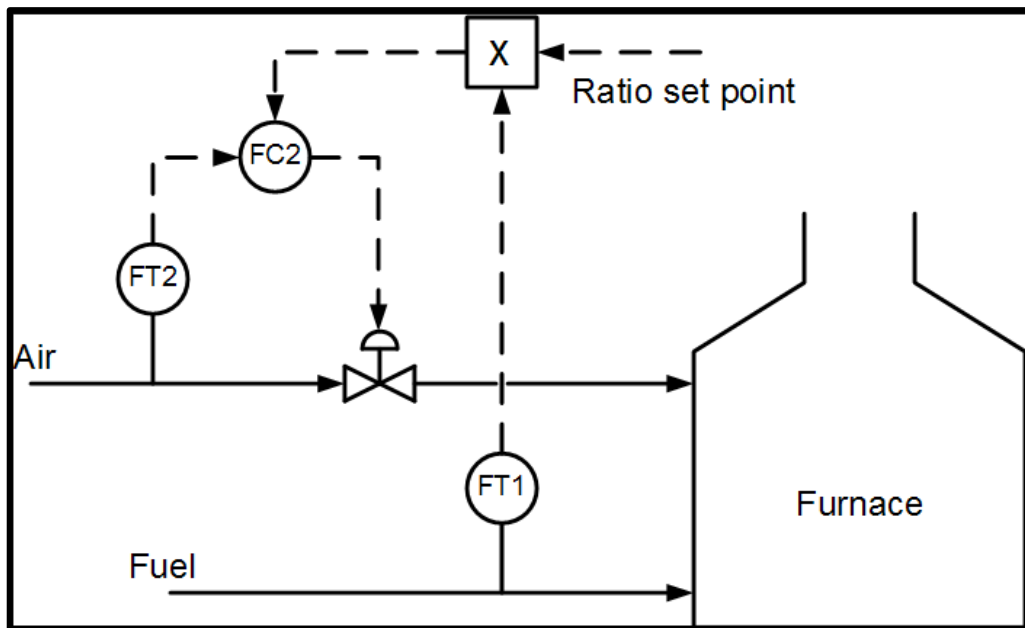
Figure 46: Control of furnace

As seen in this diagram we have a master controller which sends the signal to the air flow controller whose final control element is the gas valve. So what happens is fuel's flow is transmitted to the master controller alongside a ratio set point which is the air/fuel ratio. Combining both of these values a signal is generated which is sent to the flow valve controller. According to the value or set point given by the master controller the flow controller accordingly adjusts the air valve opening and hence this way we make sure that the right amount of fuel and air mix is maintained within the furnace. So basically the first controller's output is setpoint of the second controller and is a type of cascaded control. This way smooth operation is maintained. If too much air enters then the flame tends to have a lower temperature and if too much fuel enters then there are high chances of incomplete combustion hence this control is necessary.

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CHAPTER 8

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HAZARD AND OPERABILITY ANALYSIS

</div>

8.1 Background

A Process Hazard Analysis (PHA) is a key element of a Process Safety Management (PSM) program. It analyzes potential risks to personnel operating in an industrial environment.

A PHA is a thorough detailed systematic approach to finding potential hazards in an industrial plant. It includes an analysis of the equipment, instrumentation, utilities, human actions, and external factors which could be potential hazards.

One of the techniques of a PHA used to identify hazards and operability problems is the HAZOP (Hazard and Operability) study.

It is a study which when applied to an operating plant indicates the potential hazards that may arise from deviations on the intended line or system of study on the design conditions.

8.2 Basic Principles

So, to start off each vessel is assessed one by one, line by line using guide words to help generate thought about the possible deviations that could occur causing hazardous situations.

The following words are also used in a certain way:

Intention: This tells the purpose of equipment and how it is supposed to function under regular operating and design conditions.

Deviations: These deviations are one of the 7 guide words which give us an idea of how things would deviate from the designer's intention.

Causes: Causes tell, why a certain deviation might have taken place and is considered only if the cause is realistic then is the cause taken as meaningful.

Consequences: These are the results of the occurrence of the deviations

Hazards: Any damage caused by the consequences are known as hazards

These are the type of guide words used generally where applicable all may be used

Guide words	Meanings	Comments
No or Not	The complete negation of these intentions	No part of the intentions is achieved but nothing else happens
More	Quantitative increases or decreases	These refer to quantities and properties such as flow rates and temperatures, as well as activities like "Heat" and "React"
Less		
As well as	A qualitative increase	All the design and operating intentions are achieved together with some additional activity
Part of	A qualitative decrease	Only some of the intentions are achieved; some are not
Reverse	The logical opposite of the intention	This is mostly applicable to activities, for example reverse flow or chemical reaction. It can also be applied to substances, e.g. "Poison instead of "Antidote" or "D" instead of "L" optical isomers
Other than	Complete substitution	No part of the original intention is achieved. Something quite different happens

Figure 47: Guide words for HAZOP

8.3 HAZOP in a Cement Plant

A cement plant consists a lot of equipment mainly involving in the handling of solids such as conveyer belts, crushers and cyclones. With that a cement plant also contains the rotary kiln where the main sintering process takes place and involves the burning of the fuel. So we will be looking at each equipment one by one using different guide words to see what the consequences and potential hazards may be.

First, we will briefly apply HAZOP on the Cement Kiln Startup process

8.3.1 Cement Kiln Startup Process
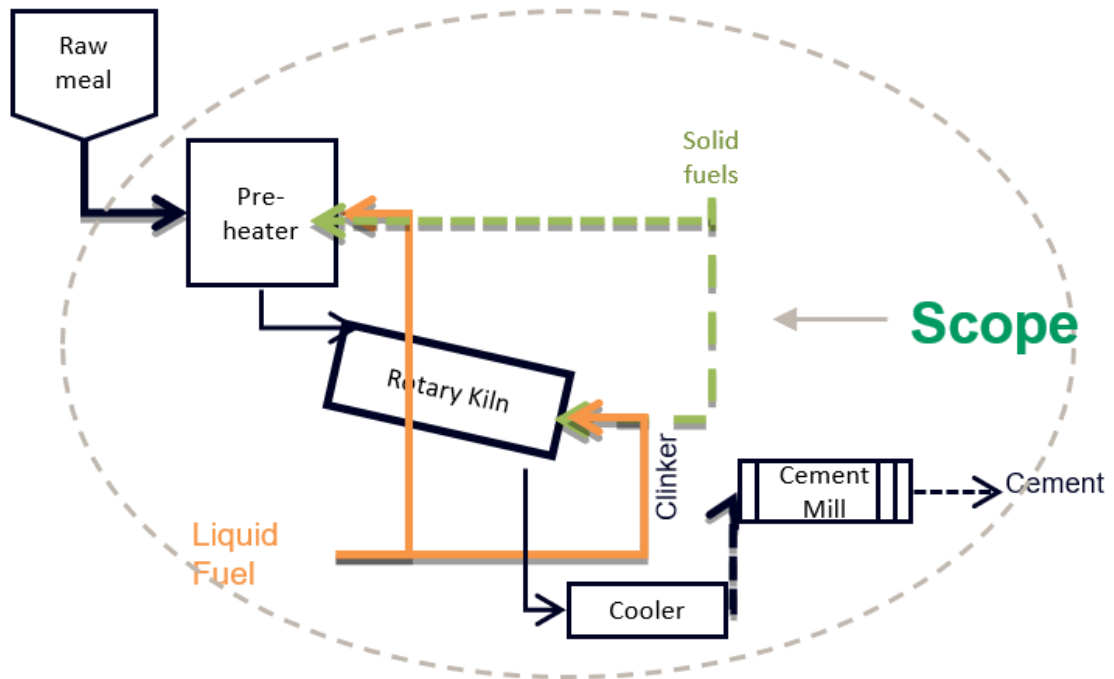
Figure 48: Startup of Kiln

This is concise view of the system we are applying HAZOP on. The figure down below is the general startup procedure for the kiln. From here we will be applying deviations on the variables.
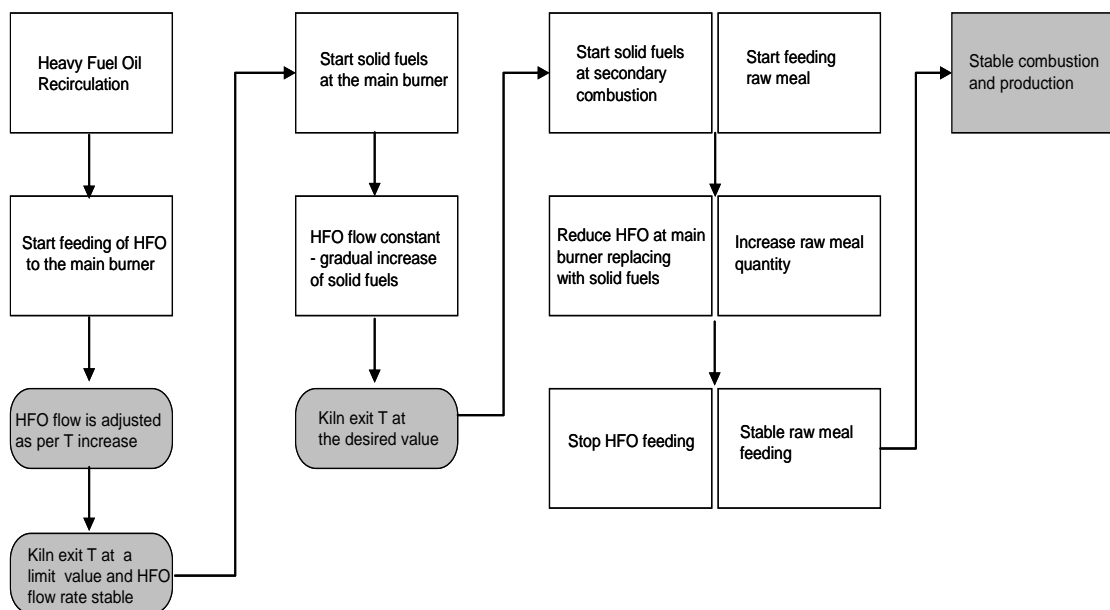
Figure 49: SOP of Kiln

8.3.2 HAZOP for items involved in rotary kiln startup

Table 33: HAZOP of Kiln Startup

Item	Deviation	Cause	Consequences	Safeguards	Recommendations
Silo Outlet to transport pump	No Flow	Failure of the dosing screw due to electrical/mechanical problems or due to agglomeration of the material by CO_2	Risk of fire at the bottom of silo due to idle material. Operational problem, not able to feed dosing silo (postpone feed of raw material)	Use of compressed air	Frequent checks of dosing screws should be carried out
	Reverse flow (air)	Reverse air flow if dampers filling simplex silo are a) stuck opened due to mechanical/electrical failure or, b) opened due to increase flow from the silo to simplex tank	Air is entering the silo risk of fire due to dust explosions and possible ignition of particles	Indication of dampers operation in the control panel. Switch filling of simplex tank from manual and automatic operation to reduce air in-leakage	Regular testing/ inspection of the automatic valves in the dampers system and agitation system

| Line from dosing system to burner | No flow of adjusting air | Failure or radial or axial compressor | Unstable flame may stop/ delay preheating | None | Consider alternative solutions (e.g. spare compressors) |

8.3.3 HAZOP in crusher and material handling of Cement Industry

Table 34: HAZOP of grinding section

Item	**Intention**	**Deviation**	**Cause**	**Consequences**	**Hazard**	**Recommendations**
Hopper	To transfer material from hopper to machine	No Vibration	Motor not working: Vibro feeder not working	Material not forwarded	Loss of system	Provide emergency backup, regular inspection
		Less Vibration	Motor not working: Vibro feeder not working	Material not forwarded with normal speed	Loss of energy	Provide emergency backup, regular inspection

			prope rly			
Electric Power	To provide electricity to the vibro-controller and motors	No electri city	Powe r failur e/ circui t break er trip	Loss of power to operate motor, vibrofeed er	Loss of system operat ion	Backup power should be present
		More electri city	Powe r surge	Trip Circuit breaker	Equip ment damag e	Provide fault detection and isolation
		Less Electri city	Powe r grid fault	Insufficie nt power to operate equipme nt properly	Equip ment damag e	Provide source of emergency backup
Ball Mill (crusher)	Crushing of stone	No crushi ng	Volta ge probl em, vibra tion is less, wear and tear of balls	Motor not working, vibrofeed er, rotor rpm not fully develope d	Equip ment does not operat e proper ly	Regular maintenance , emergency backup

| | | Less crushing | Voltage problem, vibration is less, wear and tear of balls | Motor not working, vibrofeeder, rotor rpm not fully developed | Loss of system operation | Regular maintenance, emergency backup |

8.3.4 HAZOP on Cyclone

Table 35: HAZOP on Cyclone

Item	Intention	Deviation	Cause	Consequences	Recommendations
Cyclone	Separates solids from gas	Low flow/low level	Low solid separation in cyclones from a plant turndown	Baghouse overloaded	Increase gas flow rate check for any valve blockages or leakages
		Very high flow	Valve not adjusted properly	High-pressure drop-in cyclone, although efficiency increase of solid separation	Adjust the valve for an optimum gas flow

8.3.5 HAZOP on Calciner/Reactor

Table 36: HAZOP on Calciner/Reactor

Item	Intention	Deviation	Cause	Consequences	Recommendations
Calciner Reactor	Reaction vessel in which calcination reaction takes place	Less Temperature	Poor Temperature Control	Less conversion of products or no reaction at all	Temperature control to Adjust feed
		More Temperature	Poor material selection	Reactor melting, refinery damaged	Choose Different Material
		Other than the desired composition	Wrong reagent	Possible Reactions	Sample feed before inserting
		Reagents as well as other reagents	Impurity in reagent	Possible Problem in Reactor/development of scales	Check the reactant composition

8.4 Potential Safety Hazards and Solutions

So the cement industry employs a lot of tasks related to solid handling and processing. below are the possible hazards while carrying out these tasks and precautions or steps in order to avoid them.

So the potential hazards are:

- Eye skin and respiratory tract irritation from exposure to cement dust
- Inadequate safety guards on equipment
- Inadequate lockout/tagout systems on machinery
- Overexertion and awkward postures
- Slips, trips and falls
- Chemical burns from wet concrete

Cement Dust

Hazard: Exposure to cement dust can irritate eyes, nose, throat and the upper respiratory system. Skin contact may result in moderate irritation to thickening/cracking of skin to severe skin damage from chemical burns. Silica exposure can lead to lung injuries including silicosis and lung cancer.

Solutions:

- Rinse eyes as soon as dust comes into contact with the eyes
- Use soap and water to wash off skin to avoid irritation and damage
- Wear masks where cement dust exposure is imminent. Masks type include N-95, P-95 or R-95 depending on the availability
- Drinking and eating should be done aside from the cement dust prone areas

Wet Concrete

Hazard: Exposure to wet concrete can result in skin irritation or even first-, second- or third-degree chemical burns. Compounds such as hexavalent chromium may also be harmful.

Solutions:

- Wear alkali – resistant gloves, coveralls with long sleeves and full length pants, waterproof boots and eye protection
- Wash contaminated skin areas with cold, running water as soon as possible

- Rinse eyes splashed with wet concrete with water for at least 10-15 minutes and then go to hospital for further treatment

Machine Guarding

Hazard**:** Unguarded machinery used in the manufacturing process can lead to worker injuries.

Solutions:

- Maintain conveyor belt systems to avoid jamming and use care in clearing jams.
- Ensure that guards are in place to protect workers using mixers, crushers, rollers etc.
- Establish and follow effective lockout/tagout procedures when servicing equipment.
- Be sure appropriate guards are in place on power tools before using them.

Falling Objects

Hazard: Workers may be hit by falling objects from conveyor belt systems, elevators or concrete block stacking equipment.

Solutions:

- Avoid working beneath conveyor belts or any moving machinery.
- Stack and store materials properly to limit the risk of falling objects.
- Wear eye protection when chipping and cleaning forms, products or mixers.

Poor Ergonomics

Hazard: Improper lifting, awkward postures and repetitive motions can lead to sprains, strains and other musculoskeletal disorders.

Solutions:

- Use hand trucks or forklifts when possible.

- Lift properly and get a coworker to help if a product is too heavy.
- Avoid twisting while carrying a load. Shift your feet and take small steps in the direction you want to turn.
- Keep floors clear to avoid slipping and tripping hazards.
- Avoid working in awkward postures.

Confined Spaces

Hazard: Mixers and ready-mix trucks have confined spaces that pose safety risks for workers.

Solutions:

- Follow established procedures for confined space entry and work to assure safety.
- Guard against heat stress when cleaning truck mixer drums.
- Wear appropriate protective equipment to avoid silica exposure when removing concrete residues from inside truck mixer drums.

Hot Equipment

Hazard: Calciners and rotary kilns are high temperature laden, even small exposure can be fatal and injury occurring

Solutions:

- During operation not to come close to equipment with high temperatures
- All safety protocols must be followed when closing in on these high temperature equipment

These are the main hazards while cement processing and handling as specified by OSHA (Occupational Safety and Health Administration). Other few hazards involve while quarrying of the raw material and transportation of the material back to the plant which may involve accidents or cause injuries to the driver.

CHAPTER 9

COSTING & ECONOMIC ANALYSIS

9.1 Costing in a Cement Plant

For setting up any plant costing is one of the main thing a project manager has to keep in mind. Payback period should be such that within 5 years breakeven is achieved and profits start to incur. Selling price should be such that profits are also adequate and the price for the consumer is affordable as well. Costing was done in our plant for 2 different types of fuels, one for coal and one for natural gas due to the nature of our project. As fuel is one the most used raw material in the cement industry there are significant cost savings when using natural gas as a fuel. Fixed capital and other working costs are relatively similar only fuel costs would be varied in the following cases and the effect of using natural gas will be seen.

9.2 Calculating Production Cost

Fixed Capital Costs include cost of equipment, plant setup cost and physical plant cost. For our reference we used the following figure.

To start off we will calculate the PCE I.e. purchased cost of equipment

	Process type		
Item	Fluids	Fluids–solids	Solids
1. Major equipment, total purchase cost	PCE	PCE	PCE
f_1 Equipment erection	0.4	0.45	0.50
f_2 Piping	0.70	0.45	0.20
f_3 Instrumentation	0.20	0.15	0.10
f_4 Electrical	0.10	0.10	0.10
f_5 Buildings, process	0.15	0.10	0.05
*f_6 Utilities	0.50	0.45	0.25
*f_7 Storages	0.15	0.20	0.25
*f_8 Site development	0.05	0.05	0.05
*f_9 Ancillary buildings	0.15	0.20	0.30
2. Total physical plant cost (PPC) $\qquad$ PPC = PCE $(1 + f_1 + \cdots + f_9)$ $\qquad\qquad$ = PCE ×	3.40	3.15	2.80
f_{10} Design and Engineering	0.30	0.25	0.20
f_{11} Contractor's fee	0.05	0.05	0.05
f_{12} Contingency	0.10	0.10	0.10
Fixed capital = PPC $(1 + f_{10} + f_{11} + f_{12})$ $\qquad\qquad$ = PPC ×	1.45	1.40	1.35

Figure 50: Lang factors for different processes

Purchased Cost of Equipment (PCE)

Table 37: Total Equipment

Equipment	**Number**
Cyclone Separators	4
Pre Calciner/Furnace	1
Rotary Kiln	1

Clinker Cooler	1
Conveyer Belt (1m wide)	2
Conveyer Belt (0.5m wide)	2
Ball Mill	5
Storage Tanks	6

These are the Equipment and their amount that will be computed to find the total cost of equipment.

Cyclone:

For the cyclone there was no specific method or literature material available values to compute its cost so the price was directly quoted from a vendor which was **$2200** per cyclone. So total cost of 4 cyclones was as follows:

4 Cyclones = 4 x $2200 = **$8800**

Rotary Kiln:

Rotary Kiln cost was also quoted by a vendor due to being unable to evaluate its cost using the figures provided in literature. It was quoted as $5000000

1 Rotary Kiln = 1 x $5000000 = **$5000000**

Clinker Cooler:

Clinker cooler price was also quoted by a vendor which was quoted as $1040000.

1 Clinker Cooler = $1040000

Rest of the equipment cost was calculated using the following table

Equipment	Size unit, S	Size range	Constant C,£	C,$	Index n	Comment
Agitators						
Propeller	driver	5–75	1200	1900	0.5	
Turbine	power, kW		1800	3000	0.5	
Boilers						
Packaged						oil or gas fired
up to 10 bar	kg/h steam	(5–50) × 10³	70	120	0.8	
10 to 60 bar			60	100	0.8	
Centrifuges						
Horizontal basket	dia., m	0.5–1.0	35,000	58,000	1.3	carbon steel
Vertical basket			35,000	58,000	1.0	×1.7 for ss
Compressors						
Centrifugal	driver	20–500	1160	1920	0.8	electric,
	power, kW					max. press.
Reciprocating			1600	2700	0.8	50 bar
Conveyors						
Belt	length, m	2–40				
0.5 m wide			1200	1900	0.75	
1.0 m wide			1800	2900	0.75	
Crushers						
Cone	t/h	20–200	2300	3800	0.85	
Pulverisers	kg/h		2000	3400	0.35	
Dryers						
Rotary	area, m²	5–30	21,000	35,000	0.45	direct
Pan		2–10	4700	7700	0.35	gas fired
Evaporators						
Vertical tube	area, m²	10–100	12,000	20,000	0.53	carbon steel
Falling film			6500	10,000	0.52	
Filters						
Plate and frame	area, m²	5–50	5400	8800	0.6	cast iron
Vacuum drum		1–10	21,000	34,000	0.6	carbon steel
Furnaces						
Process						
Cylindrical	heat abs, kW	10³–10⁴	330	540	0.77	carbon steel
Box		10³–10⁵	340	560	0.77	×2.0 ss
Reactors						
Jacketed,	capacity, m³	3–30	9300	15,000	0.40	carbon steel
agitated			18,500	31,000	0.45	glass lined
Tanks						
Process	capacity, m³					
vertical		1–50	1450	2400	0.6	atmos. press.
horizontal		10–100	1750	2900	0.6	carbon steel
Storage						
floating roof		50–8000	2500	4350	0.55	×2 for
cone roof		50–8000	1400	2300	0.55	stainless

Figure 51: Costing chart

The formula used was also taken from literature which related size range, equipment cost constant and index to evaluate the cost of the equipment. The following formula was used:

$$Ce = CS^n$$

Where,

Ce = Purchased Equipment Cost ($)

S = Characteristic Size Parameter

C = Cost Constant

n = index for that type of equipment

Pre Calciner:

Pre Calciner is essentially a furnace and from the figure above value values of C, S and n were obtained and then Ce was evaluated,

C = 560

S = 1×10^5 kW (Heating load of furnace)

n = 0.77

Ce = $560 \times (1 \times 10^5)^{0.77}$

Ce = **$3964496**

Hence as there is 1 pre-calciner the cost was

1 Pre-calciner = 1 x $3964496 = **$3964496**

Conveyor Belts:

Values of C, S and n for conveyer belt were as follows,

C = 2900 (for 1 m wide)

C = 1900 (for 0.5 m wide)

S = 25 (length in meters of belt)

n = 0.75

$Ce = 2900 \times 25^{0.75} =$ **$32432** (for 1m wide)

$Ce = 1900 \times 25^{0.75} =$ **$21243** (for 0.5m wide)

In this plant there were 2 of each conveyor belts hence,

Conveyor Belts Cost = (2 x $32432) + (2 x $21243) = **$107331**

Ball Mills:

Value of C, S and n for ball mill from the figure above were,

C = 3800

S = 80 (tons/hour)

n = 0.85

$Ce = 3800 \times 80^{0.75} =$ **$157547**

5 Ball Mills = 5 x $157547 = **$787737**

Storage Tanks:

For storage tanks the values of C, S and n were,

C = 2300

S = 4000 (capacity in m^3)

n = 0.55

Ce = 2300 x 4000$^{0.55}$ = **$220222**

6 Storage tanks = 6 x $220222 = **$1321332**

Therefore total PCE,

= $8800 + $3964496 + $5000000 + $1040000 + $64846 + $42485 + $787735 + $1321332 = **$12229695**

From PCE, Physical plant cost (PPC) will be evaluated and further fixed capital will be evaluated from PPC. So the following table was used to find the values of Lang factor, f, and hence was then summed up and multiplied by PCE. As our plant is concerned mainly with solid processing the Lang factors for solids were used.

Lang factors are basically dependent on the type of processing plant and these factors are determined by multiplying Ce i.e. the purchased cost of equipment and the constant Lang factor dependent on the process f_L which is 3.1 for solid processing plant.

The product is equal to the fixed capital cost and the Lang factor f different for each equipment. These factors are evaluated by different authors and hence can be seen in the table below. The formula is as follows:

$$Cf = f_L Ce \tag{6.6}$$

where Cf = fixed capital cost,
   Ce = the total delivered cost of all the major equipment items: storage tanks, reaction vessels, columns, heat exchangers, etc.,
   f_L = the "Lang factor", which depends on the type of process.

   $f_L = 3.1$ for predominantly solids processing plant
   $f_L = 4.7$ for predominantly fluids processing plant
   $f_L = 3.6$ for a mixed fluids-solids processing plant

Figure 52: Calculation of Lang Factors

We used the literature provided Lang factors.

PPC and Fixed Capital are evaluated and summarized in the table below

Table 38: Calculation of fixed capital

Item	Solids
1. Major equipment, total purchase cost	PCE
*f*1 Equipment erection	0.5
*f*2 Piping	0.2
*f*3 Instrumentation	0.1
*f*4 Electrical	0.1
*f*5 Buildings, process	0.05
*f*6 Utilities	0.25
*f*7 Storages	0.25
*f*8 Site development	0.05
*f*9 Ancillary buildings	0.3
Σ*f*	1.8
PPC = PCE(1 + *f1* + … + *f9*)	$34243146

2. Total physical plant cost (PPC)	
*f*10 Design and Engineering	0.2
*f*11 Contractors Fee	0.05
*f*12 Contingency	0.1
Fixed Capital = PPC (1 + *f*10 + *f*11 + *f*12)	**$46228247**

To break it down PPC = PCE(1 + 1.8(summation of all solid Lang factors))

Therefore, PPC = $12229695(2.8) **= $34243146**

After calculating PPC, utility cost and raw material cost has to be calculated after which we calculate the total production cost using the following relations from the table below:

Variable costs | *Typical values*
1. Raw materials — from flow-sheets
2. Miscellaneous materials — 10 per cent of item (5)
3. Utilities — from flow-sheet
4. Shipping and packaging — usually negligible

Sub-total A   . . . .

Fixed costs
5. Maintenance — 5–10 per cent of fixed capital
6. Operating labour — from manning estimates
7. Laboratory costs — 20–23 per cent of 6
8. Supervision — 20 per cent of item (6)
9. Plant overheads — 50 per cent of item (6)
10. Capital charges — 10 per cent of the fixed capital
11. Insurance — 1 per cent of the fixed capital
12. Local taxes — 2 per cent of the fixed capital
13. Royalties — 1 per cent of the fixed capital

Sub-total B   . . . .

Direct production costs A + B   . . . .
13. Sales expense — 20–30 per cent of the direct
14. General overheads — production cost
15. Research and development

Sub-total C   . . . .

Annual production cost = A + B + C = ...

$$\text{Production cost £/kg} = \frac{\text{Annual production cost}}{\text{Annual production rate}}$$

Figure 53: Production Cost Methodology

First, we will calculate variable costs

Variable costs include

- Raw Material cost
- Miscellaneous materials
- Utilities
- Shipping and packaging

Raw Material Cost

Raw material costs include the materials that are used to manufacture cement. From our basis of 9900 ton/day we calculated the amount required for each material using the compositions provided to us.

The following materials are required in the given capacities:

Table 39: Raw Material Amounts

Components	Amount (tons)
Calcium Oxide	6301.35
Iron Oxide	282.15
Aluminum Oxide	564.3
Silicon Dioxide	2257.2
Gypsum	495

Costs of each raw material was obtained from vendors and are given as follows:

Table 40: Unit cost of Raw Materials

Components	Cost ($)/ton
Calcium Oxide	5
Iron Oxide	7
Aluminum Oxide	6.5
Silicon Dioxide	7
Gypsum	10

Multiplying the Amount and cost we calculated the total raw material cost per day (9900 tons per day) and then this was multiplied by 365 to calculate it per annum.

The table below summarizes the raw material cost

Table 41: Total Cost of Raw Material

Annual Raw Material Costs (Basis 9900 tons/day)			
Components	Amount	Cost/ton	Total Cost
Calcium Oxide	6301.35	$5	$31506.75
Iron Oxide	282.15	$7	$1975.05
Aluminum Oxide	564.3	$6.5	$3667.95
Silicon Dioxide	2257.2	$7	$15800.4

Gypsum	495	$10	$4950
		Total/day	**$57900**
		Annual Cost	**$21133555**

Next from the variable costs we calculate the utility cost.

Utility Cost

The utilities required mainly were

- Coal
- Compressed air
- Electricity

The rates of these utilities were obtained from literature values from the following table:

Utility	UK	USA
Mains water (process water)	60 p/t	50 c/t
Natural gas	0.4 p/MJ	0.7 c/MJ
Electricity	1.0 p/MJ	1.5 c/MJ
Fuel oil	65 £/t	100 $/t
Cooling water (cooling towers)	1.5 p/t	1 c/t
Chilled water	5 p/t	8 c/t
Demineralised water	90 p/t	90 c/t
Steam (from direct fired boilers)	7 £/t	12 $/t
Compressed air (9 bar)	0.4 p/m^3 (Stp)	0.6 c/m^3
Instrument air (9 bar) (dry)	0.6 p/m^3 (Stp)	1 c/m^3
Refrigeration	1.0 p/MJ	1.5 c/MJ
Nitrogen	6 p/m^3 (Stp)	8 c/m^3

Note: £1 = 100p, 1$ = 100c, 1 t = 1000 kg = 2200 ib, stp = 1 atm, 0°C

Figure 54: Utility cost prices

From our energy and material balances we calculated the amount of coal required and air required.

The amounts were calculated as follows:

Table 42: Utility costs (coal)

Utility	Amount
Coal	1023264 tons
Compressed Air	7700000 m^3
Electricity	49.3 MJ

The cost of these utilities were obtained from the fig above

They were as follows

Coal = 0.065 c / MJ

Compressed Air = 0.006c / m^3

Electricity = 0.015c / MJ

The amounts were multiplied by the costs and the total utility cost is tabulated below:

Table 43: Total Utility Costs

Utility Cost (Basis 9900 tons/day)			
Utility	Amount	Cost/unit	Total Cost
Coal	1023264	0.065	$66,512.2

Compressed Air	7700000	0.006	$46,200.0
Electricity	49.3	0.015	$0.7
		Total/day	**$112,712.9**
		Annual Cost	**$41,140,208.3**

After calculating these variable costs, the following table was used to find relations between these variable costs and the other ones. Given below is the table:

Variable costs	Typical values
1. Raw materials	from flow-sheets
2. Miscellaneous materials	10 per cent of item (5)
3. Utilities	from flow-sheet
4. Shipping and packaging	usually negligible
Sub-total A	. . . .
Fixed costs	
5. Maintenance	5–10 per cent of fixed capital
6. Operating labour	from manning estimates
7. Laboratory costs	20–23 per cent of 6
8. Supervision	20 per cent of item (6)
9. Plant overheads	50 per cent of item (6)
10. Capital charges	10 per cent of the fixed capital
11. Insurance	1 per cent of the fixed capital
12. Local taxes	2 per cent of the fixed capital
13. Royalties	1 per cent of the fixed capital
Sub-total B	. . . .
Direct production costs A + B	. . . .
13. Sales expense	20–30 per cent of the direct
14. General overheads	production cost
15. Research and development	
Sub-total C	. . . .
Annual production cost = A + B + C =	. . . .

$$\text{Production cost £/kg} = \frac{\text{Annual production cost}}{\text{Annual production rate}}$$

Figure 55: Production cost figure

Keeping this table in mind all these costs were calculated one by one and Subtotal A, B & C were summed up to find the Annual production cost which was then divided by the annual production rate and hence the Production cost/kg was calculated.

Table 44: Total Production cost (coal)

Production Cost $/Kg	
Fixed Capital	**46228247**
Variable Costs	
1. Raw materials	21133555
2. Miscellaneous materials	462282.4692
3. Utilities	41140208.32
4. Shipping and packaging	0
Subtotal A	**62736046**
Fixed Costs	
5. Maintenance	8321084.446
6. Operating Labor	4320000

7. Laboratory costs	864000
8. Supervision	864000
9. Plant overheads	2160000
10. Capital charges	4622824.692
11. Insurance	462282.4692
12. Local taxes	924564.9384
13. Royalties	462282.4692
Subtotal B	**23001039.01**
14. Sales expense/General Overheads	34294833.82
Subtotal C	**120031918**
Production Cost	205769003
Production Cost $/kg	**0.064**
Production Cost Rs/kg	**10.44**

Profit

The selling price of each 50 kg bag is Rs650. Profit was calculated as follows:

$$\text{Profit} = (650/50) - 10.44 \text{ (production cost)} = \textbf{Rs 2.56}$$

9.3 Payback Period

9.3.1 Payback Period (Coal)

Payback Period is calculated as follows

Payback Period = Annual Production Cost/ Annual Profit

Annual Production Cost (Rs) = 10.44 x 8750 x 1000 x 365 = **Rs 33334578474**

The value of **8750** tons was an average value taken per day incorporating plant shutdowns and downtimes.

Annual Profit was calculated by multiplying profit and amount of cement sold

The table below summarizes it:

Table 45: Profit (coal)

Production Cost Rs/kg	10.44
Selling Price Rs/Kg	13
Kg Sold Per Year	3193750000
Profit	2.56

Profit Per Annum (RS)	**8184171526**

Finally, the payback period was calculated for coal

Payback period (coal) = 3333457874/8184171526 = 4.07 $\approx$ 4.1 years

9.3.2 Payback Period (Natural Gas)

For using natural gas all costs remained constant except the utility cost and that changed the payback period and profits to as follows

Table 46: Utility cost (natural gas)

Utility	Amount	Cost/unit	Total Cost
Natural Gas	626373	0.06	$37,582.4
Compressed Air	7700000	0.006	$ 46,200.0
Electricity	49.3	0.015	$0.7
		Total/day	**$ 83,783.1**
		Annual Cost	**$ 30,580,838.6**

The cost decreased drastically firstly because less fuel is being used and secondly it is slightly cheaper as than coal as well.

The savings done by natural gas are as follows:

Table 47:Savings incurred due to natural gas

Annual cost (coal)	Annual Cost (natural gas)
$41,140,208.30	$30,580,838.60
Savings	**$10,559,369.70**

Payback Period Natural Gas

Payback period calculated as before is evaluated and tabulated below turning out to be **3.6 years.** Price in case of using natural gas is Rs 11.71 /kg hence the payback period is 3.6 years. Price has to be reduced to compare both coal and natural gas at a constant profit in order to make a fair analysis. Of course, if the price is the same then it would take almost half the time as coal did. Pricing obviously depends on the government orders as well so to keep it simple the comparison is made on a constant profit earning. Approximately in the case of natural gas payback period is achieved 4 – 5 months earlier.

Table 48: Payback Period (natural gas)

Selling Price Rs/Kg	11.71
Kg Sold Per Year	3193750000
Profit	2.56
Profit Per Annum (RS)	8169716966
Payback Period (years)	3.6

CONCLUSION

To conclude, cement demand is growing extensively as the years progress due to massive urbanization and modernization. Bestway cement has 4 plants with thousands of tons of production each year. Although they might follow the allowable limit but even that can be reduced further by shifting to a cleaner fuel as we have recommended (natural gas) which has a lower carbon footprint as compared to coal or use alternative fuels such as tires and biomass which when used make the overall process carbon neutral and have lesser emissions compared to conventional coal. Alternative fuels other than natural gas are useful where natural gas is not available, but Pakistan is expected to receive natural gas from Turkmenistan the Turkmenistan Afghanistan Pakistan India (TAPI) pipeline which will surely provide gas security in the upcoming years. Pakistan is expected to get 14 billion cubic meters of this natural gas.

The other prospect Ferrock obviously can not obviously be used in a large scale as is cement used. It can however be manufactured in amounts to meet basic building needs, such as foot paths construction, pavements, small buildings etc. if cements is replaced in smaller applications that will surely decrease the carbon dioxide emissions as well. Hence the prospect of Ferrock is one which will cannot be ignored as eventually one day it will come into use when the emissions will truly be at their highest threat.

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