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ENERGY AUDIT IN MUGHER CEMENT ENTERPRISE (MCE)

A thesis submitted to the school of Chemical and Bioengineering Presented in Partial fulfillment of the requirements of the Degree of Masters of Science in Chemical Engineering (process engineering stream)

By

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ABSTRACT

Cement factory is energy intensive industry that needs a great attention to optimize its energy usage. Mugher Cement Enterprise utilizes imported Heavy Fuel Oil and domestic electric energy. Therefore, reducing energy dissipation enables the enterprise as well as the country saving of foreign currency and keeps the environment from damage.

The aim of this thesis work is to identify the major energy loss areas in Mugher Cement Enterprise for the third cement line and to present suitable modification to enhance energy utilization performance of the factory.

This thesis first systematically identifies the energy losses areas. This is achieved by conducting Chemical analysis and determining energy consumption (thermal and electrical) of corresponding units. In the second stage, performance assessment of these units is undertaken. Finally, the result is compared with standards and original design.

The result of assessment shows that the Enterprise has an enormous opportunity for improvement without jeopardizing its product quality and reputation. Thermal energy losses take the lion share for poor energy performance of the enterprise. The Eta cooler, fans, compressors and pumice dryer are among extremely inefficient devices that needs immediate modification.

A thorough investigation of MCE’s energy audit reveals that there is an opportunity of saving of around 41 millions birr per annum by reducing amount of fuel burnt (excess air), adapting heat recovery mechanism and improving inadequately operating electrical units.

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AM	Alumina Modulus
AR	Alumina Ratio
C_3S	Alite: - Tricalcium Silicate, (CaO)3SiO2
C_3A	Aluminate: - Tricalcium Aluminate, (CaO)3 Al2O3
C_2S	Belite: - Bicalcium Silicate, $(CaO)_2SiO_2$
Bhp	Brake Horse Power
BF	Burnability Factor
BI	Burnability Index
CI	Coating Index
CV	Caloric Value
EDTA	Ethylene Diamine Tetra Acetic Acid
C_4AF	Ferrite: - Tetracalcium Aluminoferrite, (CaO)4Al2O3Fe2O3
HFO	Heavy Fuel Oil
ID	Induced Draft
LSF	Lime Saturation Factor
MFR	Mechanical Flow Regulator
OPC	Ordinary Portland Cement
PPC	Pozzolanic Portland Cement
SM	Silica Modulus
SR	Silica Ratio
PF	Power Factor
VFD	Variable Frequency Drive
XRD	X-Ray Diffraction
XRF	X-Fay Fluorescence
WHRSG	Waste Heat Recovery Steam Generator

1. INTRODUCTION

1.1 Background

Energy is the hot issue of the twenty first century. It has a great impact on the development and politics of a certain country. Today, factories are utilizing both renewable and non renewable energy sources extensively which is bringing unrecoverable damage to the ecosystem and the human kind. Cement factories are principal energy exhaustive industry in the world.

Cement industry is among the biggest industries in the world producing a wide variety of products. Its product ranges from pure ordinary Portland cement (OPC) to blended cements (like PPC, special cement etc) whose application varies based on the purposes and environmental conditions. Cement is such an important contributor to economic development that all countries aspire to have a domestic cement industry. Cement manufacture is capital intensive which is a barrier to the establishment of cement factories, but once constructed the cement factory becomes a significant "cash cow" generating significant revenues to recover the capital investment. Current world consumption of cement is 2 billon tones and increasing at about 3%. The demand is massively depends on the populations size and Political stability and also the rising GDP can trigger growth in cement demand as experienced in Ethiopia and India. [1]

In Ethiopia, the first cement factory, Dire Dawa Cement and Lime Factory, was established by the Italian occupying forces in 1938. The plant had an initial capacity of 120 tones of clinker (the intermediate product obtained by burning limestone) per day. In response to the increasing demand for cement, the Addis Ababa and Mestewa cement factories were established in 1964 and 1965 respectively, each with a capacity of 70,000 tones of clinker per year.

In 1984, the State-owned Mugher Cement Factory was constructed and commissioned with a capacity of 300,000 tons per year, which created a large increase in capacity of the country's cement supply. Following Eritrea's independence in 1991, the Mestewa cement factory was no longer in Ethiopia and Mugher was the only manufacturer in the sector until Messebo Cement was established in 1996.

Mugher Cement Enterprise is established with a purpose of producing and supplying cement and carrying out related activities that are important for the attainment of its objective. It was established in 1999 through amalgamation of two formerly independent factories: - Mugher Cement Factory and Addis Ababa Cement Factory.

The enterprise is located about 90 km North West of the capital city, Addis Ababa, on the elevation of about 2450 m above sea level. The mother plant of the enterprise has three production lines with production capacity of 5000 tons of clinker per day. The first, second and third lines started operation in 1984, 1990 and 2011 respectively. Besides, it has Addis Ababa cement factory which has stopped production because of obsolesce except cement milling.

The enterprise uses Limestone, Clay, pumice, Sandstone, Gypsum and electric power from local and imported furnace oil and Kraft paper as its major input to produce standardized and job ordered products. The enterprise distributes its products to the local market from the plants premises, through its own branch sales outlets located at Addis Ababa, Tatek and Adama /Nazareth/ city.

The main expenditure of the enterprise is energy expense. A proper management of energy in the enterprise will be focal measure of its performance. Optimizing the energy use in the factory will enable it to be competent and to remain in the market. Besides, it reduces the environmental load and related catastrophes caused by combustion exhaust gases.

The target of this thesis work is to identify the main energy dissipative units and to find a possible solution to tackle the recognized troubles. A thorough chemical, thermal and electrical performance assessment of corresponding units has made to discover inadequately functioning units. Thermal performance of main equipments and processes were assessed for main thermal units including kiln, precalciner, cooler, preheater and pumice dryer. A systematic approach is followed to examine electrical energy consumption of electrical units and compared it with standard and their original design. Furthermore, causes of dissipation are analyzed. Subsequently, based on the evaluation results, economically and environmentally viable measures have been suggested to enhance the competency of the every unit that forms the whole when combined. In addition to unit wise approach, system based approaches has been adopted on units functioning in integrated manner.

1.1 Statement of the Problem

The cement sub-sector consumes approximately 12–15% of total industrial energy use. The cement making process is highly energy intensive accounting for nearly 40 - 50 % of the global cement production costs. Indeed, in Ethiopia energy cost covers 55 – 60 % of total production cost. Therefore, a state of art review on the energy use and savings is necessary to identify energy wastage so that necessary measures could be implemented to reduce energy consumption in this sub-sector. This provides ample opportunities for reducing energy consumption of the cement plants particularly in developing countries where most cement factories are inefficient.

Electrical and fuel (oil or coal) energy are the two energy types used in cement factory. Fuel energy takes the major share. About 60% of fuel is used in pre - calciner for calcination and the remaining 40% is consumed in rotary kiln for ending the calcination process and then for sinterization.

Many researches and industrial practices in cement factory reveal that the pyroprocessing is accompanied with high energy losses and only fraction of the energy obtained from combustion is being effectively used. The practical aspect of our cement factories including the modern one show that more than 50% of the theoretically expected combustion energy is lost due to poor process and equipments performance. According to the study made on 2800 tpd clinker producing plant with five stages SP Kiln that uses coal fuel result indicates about 53% of energy is lost carried by smoke gases from preheater and cooler exhaust as well as from their surface as radiation and convection including from the kiln surface.

Currently Mugher Cement Enterprise is using about one and half fold of the design fuel requirement to produce a ton of clinker. At the same time, its electrical energy consumption has grown by more than 60 % of its expected requirement. Unfortunately, this increase of fuel consumption accompanied with an increase of green house gas release by 0.5 tons for every tons of clinker.

Conducting performance assessment for every thermal and electrical units and investigating their source of poor competency will enable to bring modification that leads to improvement. Therefore, this study will give emphasis on the major energy exhaustive units of the Enterprise's third line cement factor and find solution to improve its energy performance.

1.2 Objective of the Research

1.2.1 General Objective

The general objective of this study is to increase the energy utilization efficiency in the Mugher Cement Enterprise by reducing the energy lost due to poor performance of equipments and processes.

1.2.2 Specific Objective

- To assess energy consumptions of different units in the plant
- To evaluate the performance of different units in the plant
- To determine an additional cost incurred to the plant due to energy losses
- To suggest implementable modifications in order to improve energy utilization efficiency of the plant.

1.3 Significance of the Research

The outcome of the study will have significant importance. The first and the most merit of the study is its economical consequence. Energy is the hot issue of the world in this century. On the other hand, cement factory is energy dissipative. This study stresses on the detection of energy loss units and finding mechanisms to reduce it.

Secondly, cement factory is also known by its enormous green house gas, carbon dioxide disposal. Production of one tone of clinker releases one tone of carbon dioxide. The half of the carbon dioxides is drawn from combustion of fuel. This study will minimize the energy loss which leads the reduction of the production of exhaust gas including CO.

1. LITERATURE REVIEW

2.1 Overview of Portland Cement Production Process

Portland cement is made by heating a mixture of limestone and clay, or other materials of similar bulk composition and sufficient reactivity, ultimately to a temperature of about 1450°C. Partial fusion occurs, and nodules of clinker are produced. The clinker is mixed with a few percent of calcium sulfate and finely ground, to make the cement. [1]

The clinker typically has a composition in the region of 67% CaO, 22% SiO_2, 5% $A1_2O_3$, 3% Fe_2O_3 and 3% other components, and normally contains four major phases, called alite, belite, aluminate and ferrite. Several other phases, such as alkali sulfates and calcium oxide, are normally present in minor amounts. Hardening results from reactions between the major phases and water. [2]

Alite (C_3S) is the most important component of all normal Portland cement clinkers, of which it constitutes about 50-70%. It is tricalcium silicate (Ca_3SiO_5) modified in composition and crystal structure by ionic substitutions. It reacts relatively quickly with water in normal Portland cements & is the most important of the constituent phases for strength development; at ages up to 28 days, it is by far the most important.

Belite (C_2S) constitutes 15-30% of normal Portland cement clinkers. It is dicalcium silicate (Ca_2SiO_4) modified by ionic substitutions and normally present wholly or largely as the β polymorph. It reacts slowly with water, thus contributing little to the strength during the first 28 days, but substantially to the further increase in strength that occurs at later ages. By one year, the strengths obtainable from pure alite and pure belite are about the same under comparable conditions. [3]

Aluminate (C_3A) constitutes 5-10% of most normal Portland cement clinkers. It is tricalcium aluminate ($Ca_3Al_2O_6$), substantially modified in composition and sometimes also in structure by ionic substitutions. It reacts rapidly with water, and can cause undesirably rapid setting unless a set-controlling agent, usually gypsum, is added.

Ferrite (C_4AF) makes up 5-15% of normal Portland cement clinkers. It is tetracalcium aluminoferrite (Ca_4AlFeO_5), substantially modified in composition by variation in Al/Fe ratio and ionic substitutions.

The rate at which it reacts with water appears to be somewhat variable, perhaps due to differences in composition or other characteristics, but in general it is high initially and low or very low at later ages. [4]

2.1.1 Cement Manufacturing Process

The basic process of Cement production as shown in figure 2.1 involves

1. Acquisition of raw materials
2. Preparation of the raw materials for pyroprocessing
3. Pyroprocessing of the raw materials to form Portland cement clinker
4. Grinding the clinker to Portland Cement

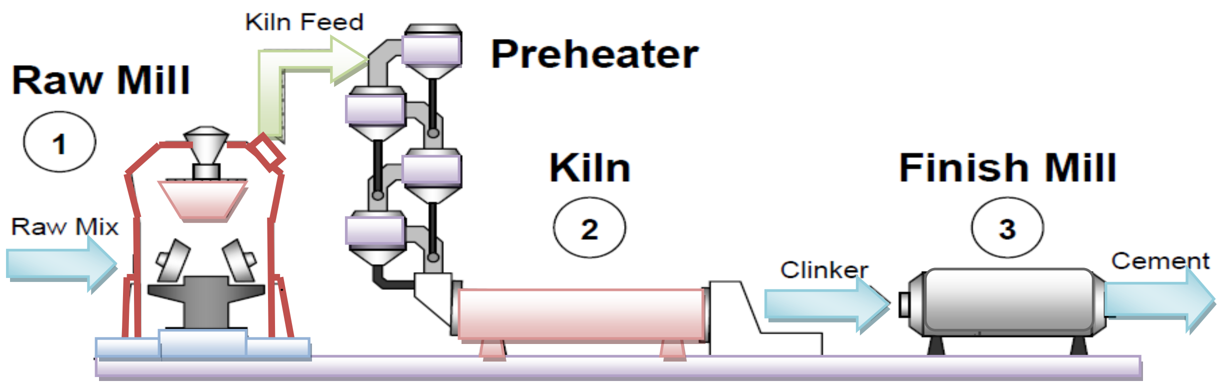

Figure 2.1, Cement production process

Generally, cement manufacturing process comprises of the following main process units.

I. Mining

Limestone, the key raw material is mined in the quarries with compressed air drilling and subsequently blasting with explosives. The mined limestone is transported through dumpers or ropeways to the plant. Surface mining is gradually gaining ground because of its eco friendliness.

II. Crushing

The limestone as mined is fed to a primary and secondary crusher where the size is reduced to 25 mm. The crushed limestone is stored in the stockpile through stacker conveyors.

The crushed limestone, bauxite and ferrite are stored in feed hoppers from where they are fed to the raw mill via weigh feeders in the required proportion. [5]

III. Raw Materials Preparation

Roller mills for grinding raw materials and separators or classifiers for separating ground particles are the two key energy consuming pieces of equipment at this process stage. For dry-process cement making, the raw materials need to be ground into a flowable powder before entering the kiln. After grinding the raw materials composition mixed in homogenization silo.

IV. Pyroprocessing

The function of the kiln in the cement industry is to first convert $CaCO_3$ into CaO and then to react with Silica, Aluminum Oxide and Ferric Oxide to form clinker compounds: alite, belite, ferrite and aluminate. The overall sequence of cement production chemistry is clearly explained in the table 2.1.

Table 2.1, The sequence of reactions occurring in the rotary kiln

Temp (^OC)	Process	Chemical transformation
< 100	Drying, elimination of free water	H_2O (l) → H_2O (g)
100 – 400	Elimination of absorbed water	
400 – 750	Decomposition of claw with formation of metakaolinite	$Al_4(OH)_8 Si_4O_{10} \rightarrow 2 (Al_2O_3 . 2SiO_2) + H_2O$
600 – 900	Decomposition of metakaolinite to a mixture of free reactive oxides	$Al_2O_3 . 2SiO_2 \rightarrow Al_2O_3 + 2SiO_2$
600 -1000	Decomposition of limestone and formation of CS & CA	- $CaCO_3 \rightarrow CaO + CO_2$ - $3CaO + 2SiO_2 + Al_2O_3 \rightarrow 2(CaO.SiO_2) + CaO.Al_2O_3$
800 – 1300	Binding of lime by CA & CS with formation of C_2S, C_3S, C_3A & C_4AF	- $CaO.SiO_2 + CaO \rightarrow 2CaO.SiO_2$ - $2CaO + SiO_2 \rightarrow 2CaO. SiO_2$ - $CaO.Al_2O_3 + 2CaO \rightarrow 3CaO.Al_2O_3$ - $CaO.Al_2O_3 + 3CaO + Fe_2O_3 \rightarrow 4CaO.Al_2O_3.Fe_2O_3$
1250 – 1450	Further binding of lime by C_2S to form C_3S	$2CaO. SiO_2 + CaO \rightarrow 3CaO.SiO_2$

The raw material mix enters the kiln at the elevated end, and the combustion fuels generally are introduced into the lower end of the kiln in a countercurrent manner. The materials are continuously and slowly moved to the lower end by rotation of the kiln. This system transforms the raw mix into clinkers, which are gray, glass-hard, spherically shaped nodules that range from 0.32 to 5.1 centimeters (cm) in diameter.

V. Preheater and Precalciner

Preheaters are cyclones are arranged vertically, in series, and are supported by a structure known as the preheater tower. Hot exhaust gases from the rotary kiln pass counter currently through the downward-moving raw materials in the preheater vessels. Compared with the simple rotary kiln, the heat transfer rate is significantly increased, the degree of heat utilization is more complete, and the process time is markedly reduced owing to the intimate contact of the solid particles with the hot gases. The improved heat transfer allows the length of the rotary kiln to be reduced or in other words for the existing kiln if retrofitted, increases the production. [4]

Additional thermal efficiencies and productivity gains have been achieved by diverting some fuel to a calciner vessel at the base of the preheater tower. This system is called the preheater or precalciner process. While a substantial amount of fuel is used in the precalciner, at least 40 percent of the thermal energy is required in the rotary kiln. Up to 95 % of the raw meal gets calcined before entering the kiln. Calciner systems sometimes use lower-quality fuels (e.g., less-volatile matter) as a means of improving process economics.

From pre-heater and pre-calciner, 60 % of flue gases travel towards raw mill and 40 % to conditioning tower where water injection is used to condition the gases. These gases are ultimately passed through electrostatic precipitator (ESP) for the maximum removal of particulate matters.

VI. Clinker Cooler

The hot clinker is cooled from 1100^0C to about 100^0C in the grate cooler with a series of fans. The cooler has two tasks: to recover as much heat (up to 30% of kiln system heat) as possible from hot (1350^0C) clinker so as to return it to the process; and to reduce the clinker temperature to a level suitable for the equipment downstream.

The hot air from recuperation zone is used for main burning air (secondary air) and precalciner fuel (tertiary air). The remaining air is sent to the stack through multi- clones or ESP. [7]

Once clinker leaves the kiln it must be cooled rapidly to ensure the maximum yield for the compound that contributes to the hardening properties of cement. The main cooling technologies are the reciprocating grate cooler and the tube or planetary cooler. In recent years the design of clinker coolers has changed drastically. The cooler has always been at the heart of efficient clinker production and its impact in terms of kiln fuel consumption, capacity and kiln system run time has always been significant. [5]

VII. Finish Milling

In this final process step, the cooled clinker is mixed with additives to make cement and ground using the mill technologies described above. These materials are then sent through mills which perform the remaining grinding. The grinding process occurs in a closed system with an air separator that divides the cement particles according to size. Material that has not been completely ground is sent through the system again.

Finish milling is the grinding of clinker to produce a fine grey powder. Gypsum ($CaSO_4$) is blended with the ground clinker, along with other materials, to produce finished cement. Gypsum controls the rate of hydration of the cement in the cement-setting process. The cement thus produced is collected in the bag filter and taken to cement silos through a vertical pneumatic pump. Cement fineness is generally measured in a unit called Blaine, which has the dimensions of cm^2 /g and gives the total surface area of material per gram of cement. [10]

The energy required for comminution or size reduction of a material to a certain required fineness will depend on the hardness of the material, its compressive strength, its brittleness (elasticity or placidity), the size, shape of the particle, its moisture and temperature of the material, and on the nature of the comminuting action exerted by the grinding process employed. These factors in combination determine the resistance that the material offers to size reduction.

2.1.2 Process Control

The main controlled variables of a cement kiln are the burning temperature and the exit gas temperature. The manipulated variables are the fuel rate and the kiln feed rate. The oxygen content in these gases is also usually controlled using the exhaust fan speeds. The cement properties like hydraulic, highly depend on the clinker quality which is directly related to the burning temperature profile. Therefore, this temperature profile has to be carefully controlled to produce high quality clinker and to reduce the disturbance effects such as variation in the raw materials composition and in the combustible properties. [8]

2.2 Energy sources for cement production

Cement industry is known to consume a large share of the total industrial use of energy and it is estimated at 30-40% in some countries. Different sources of energy have been utilized either in the form of electrical power or in the form of thermal energy. The amount of energy consumed in the production of Portland clinker ranged from 3 to 6MJ/kg clinker depending on the type of process and raw materials.

Coal, heavy oil & natural gas are used as thermal energy resources for production of cement clinker. Other fuels like oil shale, biomass residues and supplementary alternative fuels may also be used. Although oil shale (Petcoke) is considered one of the largest potential energy resources in the world, its use in the production of clinker is still limited.

2.2.1 Researches and developments on energy optimization in cement factory

A sizeable amount of energy is used in manufacturing cement. Therefore focus should be given on the reduction of energy and energy related environmental emissions locally and globally.

Being an energy intensive industry, this accounts for 50–60% of the total production costs. Thermal energy accounts for about 35–50% of the cement production cost. The typical electrical energy consumption of a modern cement plant is about 110–120kW per tonne of cement.[6]

Many researches and modification have been made to reduce the specific energy consumption. Specific energy consumption in cement production varies from technology to technology. The dry process uses more electrical but much less thermal energy than the wet process.

In industrialized countries, primary energy consumption in a typical cement plant is up to 75% fossil fuel and up to 25% electrical energy using a dry process.

Pyro-processing requires the major share of the total thermal energy use. This accounts for about 93–99% of total fuel consumption. However, electric energy is mainly used to operate both raw materials (33%) and clinker crushing and grinding (38%) equipment. [11]

Electrical energy is required to run the auxiliary equipment such as kiln motors, combustion air blowers and fuel supply, etc. (22%) to sustain the pyro-process.

In the last few decades energy intensive areas have been identified and numerous significant modifications (processes and equipments) have been made. The main modification in the thermal energy reduction is to achieve a good heat transfer and good heat recovery. To achieve these, raw mix chemistry and fineness, preheater, kiln and cooler was the main concern areas. Mills, Fans, pumps compressors, heat exchangers, and kiln and dryer driving units are the electrical energy consumers. To decrease the fan loads, preheater & cooler technology is hastily changing. Mill technology has become research area for many suppliers and has great advance. Today, many old cement factories are replacing the rotary and planetary cooler by ETA cooler for its energy efficiency. Pneumatic transportation has become obsolete. [16]

Cement factories have become research areas to cope with the increasing energy cost and environmental regulation. Researches which have been done so far include new technology development and also modification on the existing technology.

For existing technology, one of trends in energy optimization is modeling. A mathematical model of cement kiln plant without precalciner has been established by Elkjore. The effect of different parameters on the specific heat consumption and preheater exit gas temperature are investigated by the author with the aid of that model. The investigated parameters were the amount of the excess air, primary air, false air at the cement kiln, false air in the pre-heater, wall heat losses and kiln gas bypass ratio. Some operation conditions of the process have been kept at a certain values such as the degree of calcinations in the bottom stage cyclone pre-heater, hot meal temperature at the kiln inlet, kiln gas temperature, dust load in the kiln exit gas.

Results of the calculation showed that the specific fuel energy consumption increases with the increases of excess air, primary air, and false air at the kiln inlet, cooler energy loss, false air in the pre-heater and the kiln gas by pass ratio. On the other hand, the pre-heater exit gas temperature increases with the increase of the excess air, false air at the kiln inlet. It decreases with the increase of the false air in the pre-heater and with the kiln gas by pass ratio. [15]

A balance model of cement kiln plant with pre-calciner with tertiary air has been established by Ghazi. The effect of various factors such as secondary fuel energy proportion, number of stages of cyclone pre-heater, kiln gas by pass ratio and the type of the fuel used on the energy consumption and the pre-heater exit gas temperature were investigated.

Some operation conditions of the process have been specified at certain values, such as the combustion excess air factors, clinkering temperature, wall heat loss and cooler efficiency. By applying the model on different kiln plants in Egypt and other kiln plants in Germany, the result of calculations showed that the difference between theoretical calculated and measured values of fuel energy consumption are in the range of 0.10 to 5.8 %. The change in the energy losses from the individual sections of the system affects on the fuel consumption by different degrees.

A balance model of cement kiln plant with pre-calciner with tertiary air has also been established by Gardeil *et al.* The effect of various factors such which were considered by Ghazi were investigated. Some operation conditions of the process have been specified at certain values, such as the combustion excess air factors, clinkering temperature, wall heat loss and cooler efficiency. The model was based on various assumptions namely, the de-carbonation behavior of the feed material in the calciner and the temperature difference between the materials and the exit gas at the kiln inlet. [14]

The results indicated that the fuel energy consumption and the pre-heater exit gas temperature increases with the increase of the fuel energy proportion. This is in fact due to the increase of the hot meal temperature in the bottom stage cyclone pre-heater.

Thermal analysis of cyclone pre-heater system based on a model has been established by Peng Fei. The model was used to study the effect of dust loads at kiln inlet, precipitation efficiency of the cyclones and number of the stages of cyclone pre-heater on the fuel consumption.

Some operation conditions of the process have been kept constant at certain values, such as kiln exit gas temperature, heat of reaction, wall heat losses, clinker temperature at the kiln outlet, and efficiency of the cooler. The results showed that with the increase of dust load of kiln exit gas by 0.1 kg dust/kg clinker, the specific energy consumption increases by about 13-17 KJ/kg clinker.

The results indicated that the effect of precipitation efficiency of the lower cyclones on the fuel consumption is stronger than of the higher located cyclones.

From industrial experience in cement kiln plants with pre-calciner equipped with tertiary air duct, productivity ranging from 1041 to 3427 ton of clinker/day, found that the energy loss from the pre-heater is about 0.75-1.25 MJ/kg clinker, the loss through the walls of the rotary kilns is about 0.2-0.55 MJ/kg of clinker, loss from the cooler is about 0.4-0.65 MJ/kg clinker. The change in the energy losses from the individual sections of the system affects on the fuel consumption by different degrees. [14]

Furthermore, on applying the mathematical models, the effect of different factors on the apparent degree of calcinations and secondary fuel energy proportion in the calciner have been estimated. Such factors were enthalpy of the kiln exit gas, potassium chloride cycle between kiln and the cacliner and the enthalpy of the tertiary air. The results showed that the secondary fuel energy proportion supplied to the cacliner decreases with the increase of the kiln exit gas temperature and with the temperature of the tertiary air. On the other hand, the degree of calcinations in the calciner increases with the increase of the kiln exit gas temperature and with KCl concentration in the hot meal.

Recent studies on the modeling of cement manufacturing reported in the literature are mostly based on computational fluid dynamics (CFD) and they mainly study aerodynamic behavior of a particles in the preheating system, the shape and the temperature of the flame in the combustion zone, coal combustion itself as well as oxygen enrichment in the burning zone and not so much the thermodynamics and clinker chemistry taking place in the kiln system. [16]

Radwan *et al*, have been studied the modeling, simulation and control of the cement kiln systems. A transient model is derived for cement kiln systems. In this work, two control loops was studied the temperature of the preheater calciner loop and the sintering zone temperature outlet of rotary kiln loop.

The flow rate of the fuel is manipulated variable for the previous two control loops, but the disturbance variables are combustible properties of the fuel and the reaction energy due to the change the composition of the raw materials. The control system using PID controllers, which is robust with good performance in either set point tracking or disturbance rejection cases. [14]

The average energy consumption in Mugher cement factory is significantly higher than the best practice value, indicating that a strong potential for energy efficiency improvements. This paper work is intended to identify the potential energy loss areas and to adapt a mechanism that makes possible energy saving.

To achieve this, energy optimization based on mathematical medaling is the best and the commonly used method. However, it needs depth knowledge of the aerodynamics and the heat transfer phenomena in the kiln. This requires flame temperature measurement, which is the significant factor for heat transfer in the kiln. Moreover, the secondary air temperature has to be measured and known. Contemporarily there is no means to measure these variables in our country.

Alternatively, concentration is given on process optimization by performing mass and energy balance on basic units. Mass and energy balance on every unit provides a clear understanding about a loss areas and types. From the information gathered on every unit, the correct modification on process as well as the equipments will be made. Today most existing plants follow this approach for modification and optimization. This approach is chosen for its better capacity to identify the loss areas and simplicity to conduct. Moreover, it is less ideal than the other approaches.

3. MATERIAL AND METHODS

3.1 Materials and Instruments

The following materials and measuring instruments are used to conduct this study.

- Cement raw materials (Limestone, clay, iron oxides and sandstone)
- Cement Additives (Pumice & gypsum)
- Chemicals (H_2SO_4, EDTA(Ethylene Diamine Tetraacetic Acid), HCl and NaOH)
- Analytical instruments
- Flow meters
- Temperature gauges (infrared camera)
- pressure measuring instruments
- composition analyzers (XRF, X-ray fluorescence spectroscopy)

3.2 Methodology

The audit covered thoroughly the study of all the electrical & thermal equipments as well as a detail assessment of processes and process parameters. Main emphasized equipments were Kiln, Crushers, Raw mills, Grate coolers, Pumps, compressed air systems, Water pumping systems and Fans. The energy audit covered an in-depth study of all the major energy consuming equipment / stages of production. It also carried measurements and analysis covering all major energy consuming sections, to rationally assess losses and potential for energy savings.

To realize the study the following major methodologies has been adopted.

I. Literature Survey

A systematic review of latest documents and studies related to the specific subject was made. The review covered the documents related to raw materials mix design, preheater, calciner, kiln and cooler operation and control.

II. Consultation

Extensive consultation was made with engineers and cement technologists found in the cement factories and review of best practices in the cement factories. Besides, a brief discussion on cement standards and energy were made with Ethiopian Standard Agency and Ethiopian Energy Agency experts respectively.

III. Data Collection

Data for Electrical and thermal energy assessment as well as chemical analysis were collected. Mostly, direct measurement approach was used for thermal and electrical data collection whereas data for chemical analysis generated through both titration and instrumental.

Electrical data collection involved measuring of amperage, voltage, power factor, torque and efficiency of motors, power supply and coupling of different units. Both portable and fixed gauges were used to measure these parameters for different equipments.

Besides, Pressure drop across fans, compressors and pumps was measured using installed pressure probe. Furthermore, sieve analysis was made to determine the feed and product size for mills and Crusher.

Thermal data collection was somewhat complex and tiresome. Measurement made encompasses flow rate and temperature of solid materials (raw materials, clinker and cement), fuel and exhaust gases and temperature of thermal unit surfaces like kiln, cooler, preheater and precalciner. In addition, Composition analysis was made to determine constitutes of raw materials, additives and correctives. Surface and material temperature was determined using portable infrared Camera whereas the preheater gas stream temperature and pressure drop on each stage was taken from the installed thermocouple and pressure probes on every stage. Flow rates were taken from reading of installed flow meter at different location for varies materials.

Chemical data are essential to examine the process performance as well as the quality of raw materials, clinker and cement. It involves determining of composition of each material using different techniques and procedures.

Cement raw materials composed mainly of CaO, SiO_2, Al_2O_3, Fe_2O_3 with little MgO. These raw materials can be those used in the calcinations of the cement clinker and those fed to the finishing mill along with the clinker at time of grinding.

Those used for calcinations are lime components which are rich with $CaCO_3$ & Argillaceous compounds which are rich with Silica, Alumina & ferric Oxide.

Both complete chemical analysis and Instrumental (X-ray fluorescence spectroscopy, XRF) methods were employed to determine the total content (chemistry) of the material out of 100%, the general sequence is LOI, SiO_2, Al_2O_3, Fe_2O_3, CaO, MgO, SO_3, K_2O, Na_2O, TiO_2, Cr_2O_3, Mn_2O_3, P_2O_5, Cl, F.

In Complete chemical analysis methods, Titration techniques with different procedures was used to determine the composition of raw materials, additives and corrective as well as the composition of clinker and cement.

Almost all of titrimetric methods used in determining all of constituents are based on the use of the complexing reagent EDTA (Ethylene Diamine Tetraacetic Acid, usually used as the disodium salt). This reagent has the property of forming soluble octahedral complexes with many metal ions, selectively being achieved through the control of pH and by masking interfering elements by suitable reagents.

CaO, MgO, Al_2O_3 and Fe_2O_3 are determined using this reagent at different pH values. Silica is determined in the latest British Standard by the traditional method of treating with hot strong HCl containing NH_4Cl to dissolve other species, followed by filtration and ignition. Sulfate is determined by the well-tried technique of precipitation with acidified $BaCl_2$ solution, and weighing the dried precipitate. Chloride content is determined by reacting the digested sample with silver nitrate solution and back titrating with ammonium thiocyanate. Carbon dioxide is determined by releasing from the cement with concentrated phosphoric or sulfuric acid and entraining in a gas stream of nitrogen through absorption trains of alkali followed by drying tubes of $Mg(ClO_4)_2$. Spectrophqtometric methods were employed for the estimation of the minor elements in cement, particularly TiO_2, Mn_2O_3 and P_2O_5.

In the XRF methods is a non-destructive instrumental method of qualitative and quantitative analysis for chemical elements based on measurement of the intensities of their X-ray spectral lines emitted by secondary excitation. The primary beam taken from an X-ray tube irradiates the specimens, exciting each chemical element to emit secondary spectral lines having wavelengths characteristic of that element and intensities related to its concentration.

The secondary radiation is analyzed by means of a crystal rotated in the plane of the radiation and its intensity measured using a detector. Once the material is prepared and supplied to it, then it provides all elemental composition.

IV. Data Analysis

1. Thermal energy analysis
   - Investigating the actual thermal energy consumption of equipments
   - Comparing the actual thermal energy consumption with design and standards
   - Identifying the loss areas
2. Electrical energy analysis
   - Examining the actual electrical energy consumption of equipments
   - Comparing the actual electrical energy consumption with design and standards
   - Identifying the loss areas
3. Quality control analysis
   - Exploring the cement quality
   - Comparing the cement quality with design and standards

V. Result Interpretation and Improvements

Based on the data analysis result, proper processes and equipments modification have suggested. Besides the technical viability, the economical feasibility was evaluated and the possible solutions were recommended. The suggested improvements range from modification and go through replacement and include additional installation based on the significance.

4. Thermal energy evaluation of kiln system's performance

4.1 Introduction of kiln system's performance evaluation

The performance of a kiln system with a given production capacity is evaluated from parameters related to the variable production costs of which the most important are

Specific power consumption (SPC)   kWh/ton clinker

Specific heat consumption (SHC)   kcal/kg clinker

Apart from these, availability, maintenance costs, specific refractory consumption, manpower requirements, etc may be evaluated.

A. Specific power consumption (SPC)

Electrical energy is consumed in cement manufacturing primarily for raw material processing (35%), for kiln system driving (22%) and for cement grinding (38%).

The total specific power consumption is calculated by adding the power consumption (kW) of all involved motors and dividing by the production (P),

$$\text{SPC} = \frac{\sum kW}{P} \quad \text{--------------------------------------} \quad (4.1)$$

B. Specific heat consumption

The specific heat consumption is calculated as the consumption of combustible, multiplied by the net caloric value (H_i) and dividing by the production (P)

$$\text{SHC} = \frac{\sum combustible * Hi}{P} \quad \text{---------------------------} \quad (4.2)$$

When considering the possibilities for improving the performance, it is necessary to make a heat balance to investigate how the heat consumption is distributed on losses. Electrical and thermal energy flow in cement plant is shown in fig. 4.1.

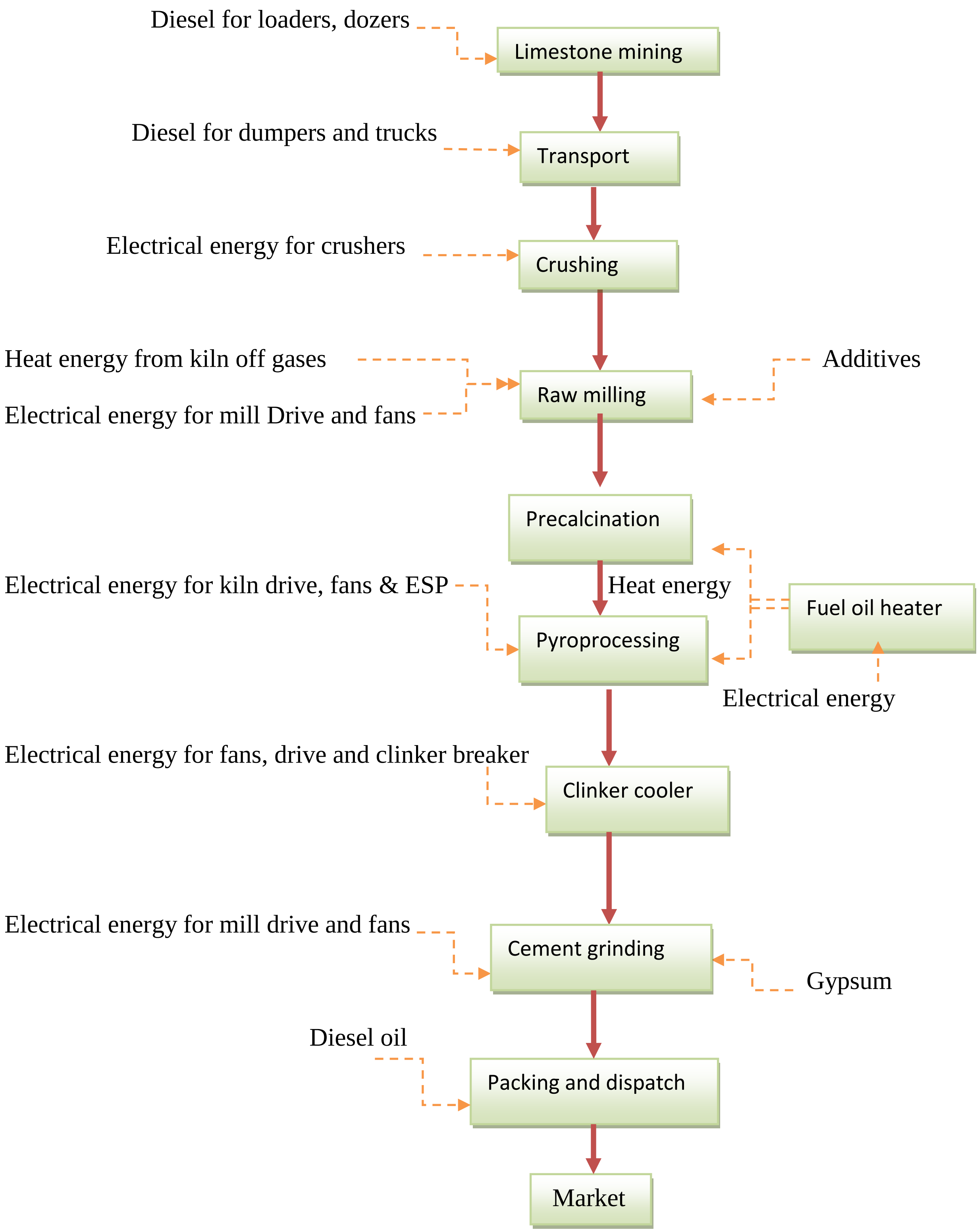

Fig. 4.1, Process flow diagram and type of energy used in cement manufacturing

4.2 Thermal Energy Performance Evaluation

the cement process involves gas, liquid and solid flows with heat and mass transfer, combustion of fuel, reactions of clinker compounds and undesired chemical reactions that include sulfur, chlorine and alkalies. It is important to understand these processes to optimize the operation of the cement kiln, diagnose operational problems, increase production, improve energy consumption, lower emissions and increase refractory life. A heat balance should be constructied for prheater, kiln cooler and the heat output values should be compared with the standard values. The following diagram shows the material flow sheet in the pyroprocessing system. Fig. 4.2 visibly depicts the material flow in cement plant.

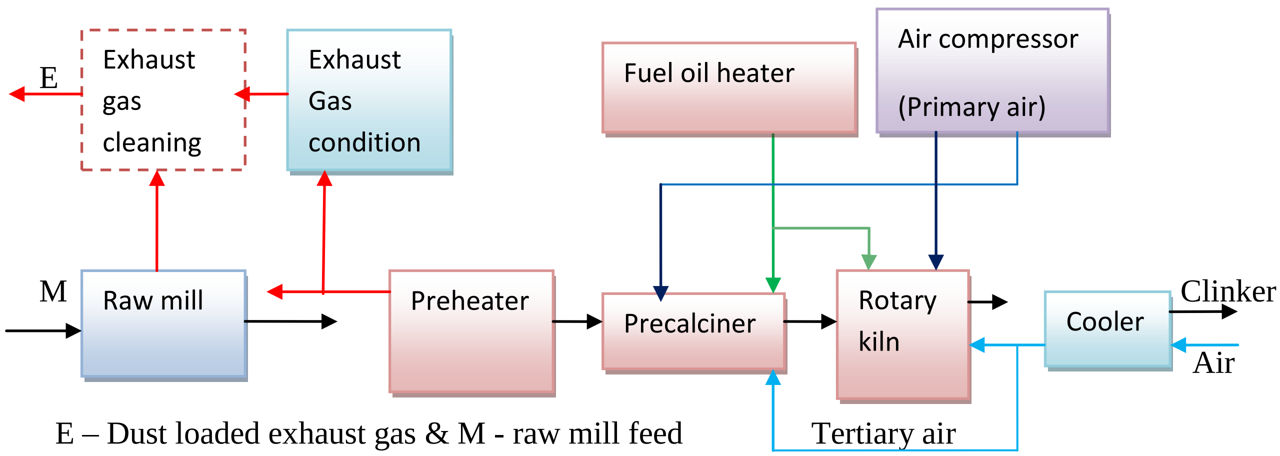

Fig. 4.2, Flow sheet for material flows in kiln system

4.2.1 Mass balance

4.2.1.1 Amount of the recirculated Dust

Since the top cyclone is about 93 – 97% efficient, there will be dust emission. Part of the dust will be recuperated in the subsequent Dedusting units and re- interred with the incoming raw meal. However, the remaining dust will be released to the environment with the exhaust gases.

The dust exiting the preheater in the exhaust gas is not measured, and is not easy to measure. If the raw mill is running, some dust may be collected in the mill product or the dust burden may be increased by the raw mill. When the raw mill is stopped the dust collected in the filters may be able to be diverted out of the process and weighed, but this is an onerous task.

When the raw mill is stopped the dust collected in the filters can also be estimated by dilution techniques, but again this is no small task. The best method to tackle this problem is to carry out mass balance over kiln system. Therefore, to determine the amount of dust recirculated, defining certain boundary and conducting mass balance around the kiln system is critical.

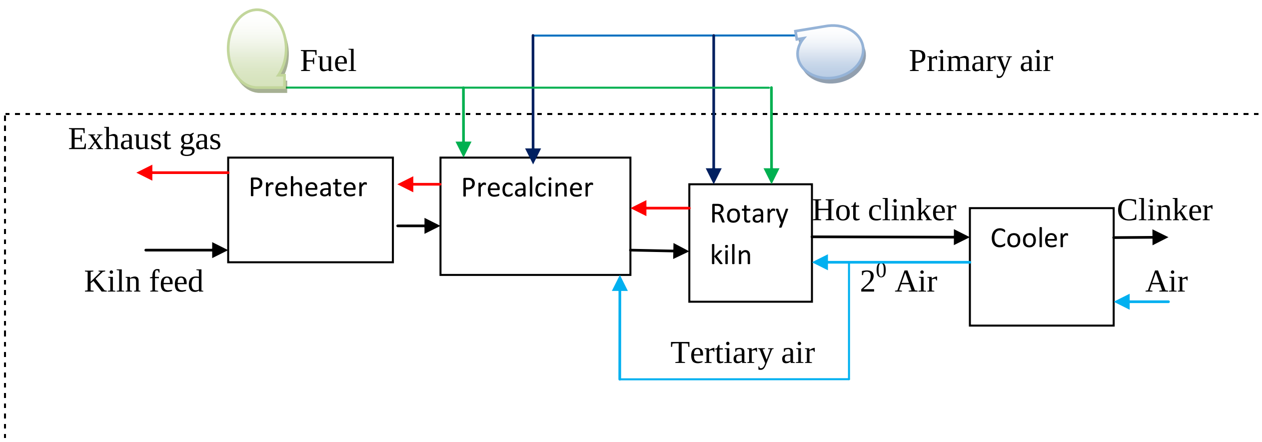

Fig. 4.3, Defined boundary for material flow in pyroprocessing system

Mass of feed (M_F) = $mass\ of\ clinker\ (Mc) + mass\ of\ dust\ in\ exhaust\ gas (Md)$------- (4.3)

$+\ mass\ of\ exhaust\ gas\ from\ calcination\ and\ dehydration$ (Me)

The loss of feed as exhaust gas from calcination and dehydration of raw material is known as *loss of ignition*, LOI_{feed}.

$Tonnes\ clinker\ = (Tonnes\ Feed - Tonnes\ Dust)\ x\ (1 - L.O.IFeed)$ ------------------- (4.4)

Mugher cement factory uses limestone and clay in its raw mix design. However, sometimes when the sand content is low, it complements by adding sandstone as corrective. Therefore, here the loss on ignition is determined by considering the mix that contains only the two raw materials.

Table 4.1, the raw material content of clinker

Feed content	mass flow rate (tph)	LOI (%)	Percentage (mass base)
Limestone	163.149	41.68	79.69
Clay	46.851	9.90	21.31
Total	210	51.58	100

$$\% LOI_{feed} = \frac{M_{limestone} * LOI_{limestone} + M_{clay} * LOI_{clay}}{\text{Total mass of feed}} \text{ --- } (4.5)$$

$$= \frac{163.149*41.68+46.851*9.90}{210} = 34.90$$

This tells us that out of 100% feed 34.90 % is released from the kiln system as exhaust gas. Therefore, out of 210 tph, 34.90 % *210 tph = 73.29 tph is depart as exhaust gas.

Table 4.2, Material and fuel flow rates in the kiln system

Sample	Kiln feed rate (tph)	Kiln fuel feed rate (kg/h)	Precalciner fuel feed rate (kg/h)	Clinker production rate (tph)	Kiln speed (rpm)
1	210	3892	6638	142	3.85
2	210	3892	6638	139	3.86
3	210	3892	6630	133	3.86
4	210	3890	6632	126	3.85
5	210	3894	6630	114	3.86
Average	**210**	**3892**	**6634**	**130.8**	**3.855**

Rearranging equation (4.1) gives

$$Tonnes\ of\ dust\ = Tonnes\ of\ feed - \frac{Tonnes\ of\ clinker}{(1-\%\ LOI)} \text{ ------------------------- } (4.6)$$

= 9.078 tph, 4.32% of feed

4.2.1.2 Amount of exhaust gas

The exhaust gases are originated from fuel combustion and calcination and dehydration of kiln feed. From the above calculation, calcination and dehydration results 73.29 tph of exhaust gas for a production capacity of 130.8 tph clinker. The feed contains about 0.6% moisture, which gives 1.26 tph.

Table 4.3, MCE's HFO specification

Fuel parameters	Unit of measurement	Values	Standard methods of test
Specific gravity at 15 ^{0}C	g/ml	0.89 – 0.96	Factory requirement
Kinematic viscosity	cSt	100 -130	Factory requirement
CV	Kcal/kg	Min 9500	DIN 51603
Flash point	^{0}C	Min 65	DIN 51603
Pour point	^{0}C	Min 15	Factory requirement
Ash content	% weight	Max 0.15	DIN 51603
Carbon residue	% weight	Max 0.15	DIN 51603
Sulfur content	% weight	Max 2.80	DIN 51603
Water content	% volume	Max0.50	DIN 51603
Sediments	% weight	0.50	DIN 51603

Efficient combustion in a cement kiln flame is dependent on the availability of sufficient combustion air, primary air momentum and fuel atomization. The fuels are delivered into the kiln through the main burner pipe together with the primary air, where they ignite as they encounter the kiln temperature in excess of 1400°C.

Part of the combustion air is provided by the primary air, which is blown into the kiln through the burner pipe. The remainder of the combustion air must be entrained into the flame from the preheated secondary air which is drawn into the kiln from the cooler.

The combustion of carbon, C, is completed to CO_2, releasing the full calorific value of the fuel.

$$C + O_2 \rightarrow CO_2$$

Inefficient combustion leads to the carbon not being completely burnt and the formation of CO in the flame.

$$C + \frac{1}{2} O_2 \rightarrow CO$$

Avoiding the formation of CO in the flame is particularly important when sulphates are present in the kiln, as sulphates decompose liberating gaseous SO_2 in the presence of CO.

$$SO_3 + CO \rightarrow SO_2 \uparrow + CO_2 \uparrow$$

The calorific and major components of fossil fuels are hydrocarbons composed of carbon and hydrogen. When the fuels are burnt in a cement kiln these hydrocarbons combine with oxygen provided by the combustion air:

$$C + O_2 \rightarrow CO_2$$

One kg of carbon in the fuel will combine with 2.66 kg (32/12) of oxygen to produce 3.66 kg of CO2 (as the atomic weights of carbon and oxygen are 12 and 16). From the Ideal Gas Laws 3.66 kg of CO_2 occupies 1.86 Nm^3 (3.66 x 22.4/44). Air is 23% oxygen by weight and therefore 8.91 kg (2.66 x 0.77/0.23) of nitrogen, N_2, will be drawn into the kiln with the oxygen in the combustion air, and will need to be exhausted from the kiln. From the Ideal Gas Laws 8.91 kg of N_2 occupies 7.12 Nm^3 (8.91 x 22.4/28). The total combustion product gas volume to be exhausted from the kiln when burning 1 kg of carbon, C, is therefore 8.98 Nm^3 (1.86+7.12).

One kg of hydrogen in the fuel will combine with 8 kg (16/2) of oxygen to produce 9 kg of H_2O.

$$2H + \frac{1}{2} O_2 \rightarrow H_2O$$

From the Ideal Gas Laws 9 kg of H_2O occupies 11.20 Nm^3 (9 x 22.4/18). Air is 23% oxygen by weight and therefore 26.78 kg (8 x 0.77/0.23) of nitrogen will be drawn into the kiln with the oxygen in the combustion air, and will need to be exhausted from the kiln. From the Ideal Gas Laws 26.78 kg of N_2 occupies 21.42 Nm^3 (26.78 x 22.4/28). The total combustion product gas volume to be exhausted from the kiln when burning 1 kg of hydrogen, H, is therefore 32.62 Nm^3 (11.20+21.42). Any increase in the hydrogen content of a fuel, and associated decrease in the carbon content, therefore means a significant increase in the volume of combustion air which must be drawn into the kiln, and combustion product gases which must be exhausted from the kiln.

Table 4.4, Currently used MCE fuel compositional analysis

Elemental analysis (m/m, %)	Ash	Hydrogen	Sulfur	Carbon	Nitrogen	Caloric value (kJ/kg)
Values	< 0.02	11.7	1.35	86.5	0.24	45,005

As it can be seen on table 4.4, presently used HFO in MCE is mainly consists of carbon and hydrogen. Consequently, it is possible to approximate that the HFO used is consists of only hydrogen and carbon without significant deviation. With this assumption, let's determine amount of exhaust gases released by burning 1 kg of HFO.

Amount of exhaust gas from burning hydrogen

= 11.7/98.2 x (9 kg H_2O + 26.78 kg of N_2)

= 4.263 kg of exhaust gases

Or

= 11.7/98.2 * (9/0.84 + 26.78/1.254) = 3.32 Nm^3 of gas

Amount of exhaust gas from burning carbon

= 86.5/98.2 (3.66 kg CO_2 + 8.91 kg of N_2)

= 11.072 kg of exhaust gases

Or

= 86.5/98.2 (3.66 /1.976 + 8.91/1.254) = 7.866 Nm^3 of gas

Amount of stoichiometric air needed

= (11.7/98.2 *26.78 kg of N_2) + (86.5/98.2 * 8.91 kg of N_2)

= 11.039 kg of N_2

Then mass of air = 1 kg of air /0.77 kg of N_2 * 11.039 kg of N_2 =14.34 kg of air.

Hence, in burning 1 kg of fuel, 14.34 kg of air is consumed and 11.072 kg (11.54 Nm^3) of exhaust gases are released from the combustion process.

Nevertheless, for complete combustion to occur, it needs some excess air than stoichiometrically needed amount. Practically, the amount of excess air used in the process is determined from the gas analyzer results at kiln inlet, precalciner and preheater outlet using the following equation.

$$\%\, excess\ air = \frac{79\, O2}{0.21 * (100 - CO2) - O2} \text{ ---------------------------------- (4.7)}$$

$$= \frac{\%O2}{21 - \%\ O2} = 23\%$$

Table 4.5, Content of exhaust gas at the smoke chamber

Gas item, (v/v)	Sample one	Sample two	Sample three	Sample four	Average
NOx (ppM)	211	221	226	229	222
CO, %	0.04	0.05	0.045	0.045	0.045
O_2, %	4.5	4.4	4.4	4.5	4.45

However, currently, only the gas analyzer at a smoke chamber/kiln inlet is working. Therefore, it is impossible to determine the gas linkages at different location. By interpolating the design value of oxygen content with the operating oxygen content value at the kiln outlet, it is possible to determine the precalciner outlet oxygen content but assuming no or insignificant air inleak at preheater and precalciner is very critical.

Table 4.6, Design and actual oxygen content value of exhaust gas at kiln and precalciner outlet

Oxygen content	Design value, %	Actual value, %
Kiln outlet	1.75	3.93
Precalciner outlet	2.39	? =

After interpolating, determining the excess air at the precalciner outlet gives 24%. The total air feed to the kiln and precalciner is the sum of combustion air and the excess air feed. As determined above, to burn 1 kg of fuel, stoichiometrically, 14.34 kg or 10.885 Nm^3 of air is consumed.

To burn 10529 kg of fuel per an hour (Precalciner and kiln), 114608.265 Nm^3/ hr of air is needed. Considering the 24% excess air, 142,114 N m^3/hr of air is being feed to kiln system. By assuming the total content of NOx, Sox, CO and other volatile matters account insignificant, the composition of the exhaust gas can determined.

To get percentage of CO_2 in the exhaust gas, let take one hour bases. This carbon dioxide produced both from the fuel combustion and the raw material calcination.

1. Carbon dioxide from Calcination of calcium carbonate

$$CaCO_3 \xrightarrow{\Delta} CaO + CO_2$$

Decomposition of 1 tone of calcium carbonate gives 22.4 Nm^3 of carbon dioxide at the standard temperature and pressure.

Table 4.7, Percentage of calcium carbonate in clay and limestone

Raw material type	Limestone	Clay
$CaCO_3$ content, %	94.5	7.7

i. For limestone

In an hour operation, 163.149 tones of limestone are consumed. Amount of hourly released carbon dioxide can be determined as follows.

Volume of carbon dioxide produced = $\frac{0.945 * 163{,}149 * 22.4}{100}$ = 34,540 Nm^3 of carbon dioxide

ii. For clay

For every hour, 46.851 tones of clay is used up whose calcium carbonate content is about 7.7%.

Volume of carbon dioxide produced = $\frac{0.077 * 46{,}851 * 22.4}{100}$ = 808 Nm^3 of carbon dioxide

Total volume from calcination = 35,350 Nm^3 of carbon dioxide at standard temperature and pressure. However, this volume of gas at the 1st cyclone occupies (at around 350 ^0C and -4.58 mbar as shown on table 4.8) different volume. To determine the volume of the exhaust gas at these conditions, ideal gas assumption is vital.

Table 3.8, Pressure drops at different stages of preheater

Location	Pressure drop (mbar)	Pressure drop (mbar)	Pressure drop (mbar)	Pressure drop (mbar)	Pressure drop (mbar)	Pressure drop (mbar)	Pressure drop (mbar)	Average p drop (mbar)
1st right stage Cyclone	-4.57	-4.60	-4.56	-4.58	-4.60	-4.58	-4.58	- 4.58
1st left stage Cyclone	-4.51	-4.52	-4.48	-4.54	-4.56	-4.55	-4.54	-4.53
2nd stage Cyclone	-2.61	-2.66	-2.65	-2.60	-2.64	-2.59	-2.6	-2.62
3rd stage Cyclone	-2.51	-2.54	-2.51	-2.50	-2.55	-2.51	-2.56	-2.52
Kiln gas flue chamber	-0.15	-0.25	-0.14	0.15	0.162	0.143	0.166	-0.16
Kiln hood	-0.04	-0.04	-0.03	-0.03	-0.04	0.05	0.041	-0.04

2. Volume of carbon dioxide from burning fuel

Burning a kg of HFO gives 1.63 Nm^3 of carbon dioxide and 1.3104 Nm^3 of water vapor at standard temperature and pressure. In MCE, in average 10526 kg of fuel per hour that can release 17,162.27 Nm^3 of carbon dioxide & 13,797 Nm^3 of water vapor.

Aside with oxygen for combustion, at standard temperature and pressure 112,225 Nm^3 of nitrogen enter and leaves the kiln system. Then, the corresponding oxygen amount in the exhaust gas becomes 5774 Nm^3. The excess oxygen amount = 21/79 x excess nitrogen = 5774 Nm^3/hr.

The total flow rate of carbon dioxide becomes 52,508 (17,158 + 35,350) Nm^3. As mentioned above, about 0.6% of the feed contains water.

The volume of dehydrated water from the feed becomes 1260 kg/hr/0.84 kg/m^3 = 1500 Nm^3. The total moisture in the exhaust gas is the sum of dehydrated water from raw feed and combustion byproduct of fuel, which gives 15,359 Nm^3.

Table 4.9, Composition of the exhaust gas

gas type	Volume flow rate (Nm^3/hr)	% (wet basis)	% dry basis
Carbon dioxide	52,508	28.3	30.85
Nitrogen	112,225	60.3	65.08
Oxygen	5774	3.1	4.06
Water vapor	15,359	8.26	0
Total	185,866	100	100

The air is supplied to kiln system enters both through cooler from fans (secondary air) and compressed air through burner (primary air). About 10% of theoretically needed air is supplied through burner as primary air. That means out of 114,608 Nm^3/hr, 11460.8 Nm^3/hr comes through burner. As a result, the remaining 90 % of stoichiometrically needed air plus the excess air is driven through the cooler fan, 130,635 Nm^3/hr.

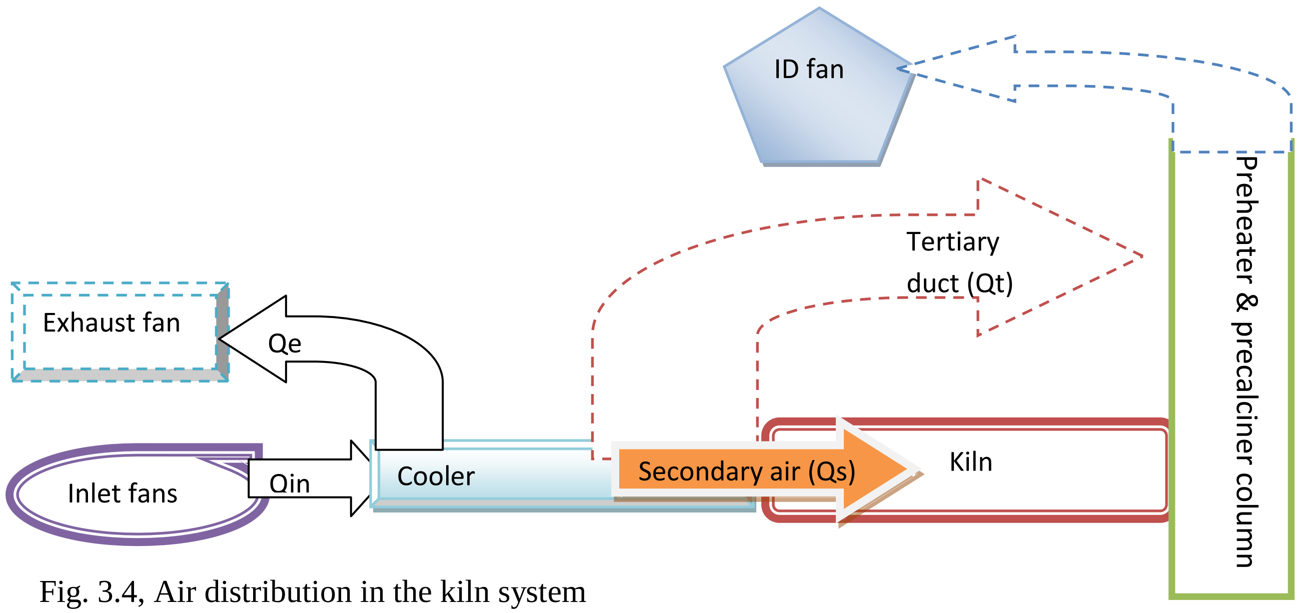

Fig. 3.4, Air distribution in the kiln system

To determine the amount of air in each stream, it is compulsory to conduct mass balance over the kiln system. By taking one hour basis, the following equation can be developed.

Mass of air entering = mass of air to exhaust fan + mass of tertiary air + mass of air 2^0 air

At standard temperature and temperature, density of air for all of the streams is the same.

$$\rho Q_{in} = \rho\, Q_e + \rho Q_t + \rho Q_s \rightarrow Q_{in} = Q_e + Q_t + Q_s \text{ ----------------------------- } \quad (4.8)$$

where, Q_{in}, Q_e, Q_t & Q_s are volume flow rate of entering air, volume flow rate of exit air to exhaust fan, volume flow rate of air through tertiary duct and volume flow rate of secondary air respectively (per unit hour) at standard temperature and pressure.

Table 4.10, The Operating conditions of cooler fans

Fan	Capacity (m^3/hr)	amperage (A)	Motor speed (%)	Head (mbar)
2604	19980	157	82.9	77
2605	18324	120.9	66.6	50.6
2606	15231	138.3	84	105
2607	17420	146	83.9	84.07
2608	29150	187	87.6	Gauge is Not working
2609	40182	204.9	82.1	82.2
2610	32544	69.6	83	2437 =?
2611	31098	163.4	82.6	92.52
2612	45117	291	91.2	79.7
2613	40172	254.4	88.7	74.7

The entering air flow rate is given at room temperature and pressure at Mugher. The density of air varies with altitude and temperature. At standard temperature and pressure air density is 1.293 kg/Nm^3. To get the density of air at other static pressure and temperature, the following equation can be used. Mugher is about 2468 m from sea level.

$$P_t, p = 1.293 \times P/10334 * (273/273 + T) \text{ --------------------------- } (4.9)$$

Where

P - Absolute pressure in mmWC and T is temperature of air in degree centigrade

Absolute Pressure of air at the Mugher is determined by considering pressure variation with altitude as follows.

$$P = Pa\,(1 - Bz/T_0)^{5.26} \text{ --- } (4.10)$$

Where

Pa = pressure at sea level = 101350 Pa

B =lapse rate = 0.00650 K/m

To = average sea level temperature = 288.16 K

Z = altitude from sea level, m

$$P = Pa\,(1 - Bz/T_0)^{5.26}$$

$$= 74{,}966.393 \text{ Pa} = 7398.6 \text{ mmWC} = 0.74 \text{ bar}$$

$$\rho = 1.293 \times 7398.6/10334 * (273/273 + 25)$$

$$= 0.848 \text{ kg/m}^3$$

To convert a given volume of gas at a certain temperature and pressure to standard condition (0 ^{0}C & 1 atm), the following formula is used.

$$Nm^3 = \text{actual volume } (m^3) * \frac{273.15}{273.15+T(C)} * \frac{P(bar)}{1.0132\ (bar)} \text{ --------- } (4.11)$$

$$Nm^3 = (289218\ m^3) * \frac{273.15}{273.15+25} * \frac{0.73986}{1.0132\ (bar)}$$

$$= 193{,}484.5\ Nm^3$$

Mass flow rate of air entering

= density of gas x volume flow rate of air

= 193,484.5 N m^3/hr *1.293 kg/Nm^3

= 250,175.6 kg/hr or 2.31 kg of air/ kg of clinker

As mentioned above, part of the feed air from the inlet fans to cooler passes to cooler exhaust fan and the left over goes to the kiln as secondary air and as tertiary air to precalciner. About the 60% of incoming air to the cooler hood goes through tertiary air duct to be used for fuel burning (60% of fuel consumed here) and calcination (about 95% calcination take place). As a result, 60% of 130,653 Nm^3/hr which is 73,392 Nm^3/hr air is bypassed the kiln to precalciner.

The air that is supplied to the cooler through inlet fans splits into two headed for the cooler exhaust fan and the kiln and precalciner through secondary and tertiary air duct respectively. The total air inlet to the cooler, 193,484.5 Nm^3/hr, is determined by summing up the air flow rates through each inlet fan and adjusting it to standard temperature and pressure. The air flow rate through Cooler exhaust fan (Q_F) is the difference between Air flow rate through Cooler inlet fans (Q_{in}) and Secondary and tertiary air flow rate ($Q_{k\&P}$).

$$
\begin{aligned}
Q_e &= Q_{in} - (Q_t + Q_s) \\
&= (193{,}484.5 - 130{,}635)\ Nm^3/hr \\
&= 62{,}849.5\ Nm^3/hr
\end{aligned}
$$

4.2.2 Heat balance

4.2.2.1 Heat output

i. Heat of formation

Many formulas have been given in the literature for calculating the heat of formation. One of the best is probably H. Zur Strassen empirical equation with slight recent modification

H_F = $32\ \%CaO + 27.1\ \%MgO + 17.2\%\ Al2O3 - 21.4\%SiO2 - 0.25\ \%Fe2O3$, $\frac{kcal}{kg}$ ------- (4.12)

= 369.35 kcal/kg of clinker

From the chemical analysis of MCE clinker, it is found that the average oxide content of the clinker is given in the table below, which is in wet basis.

Table 4.11, The oxide content of clinker for MCE

Sample	Fineness, 90μm	Al_2O_3	CaO	$CaCO_3$	Fe_2O_3	MgO	SiO_2	SO_3	Na_2O	K_2O	Cl
1	12.6	4.02	41.97	74.95	2.15	0.7	14.22	0.27	0.09	0.05	0.04
2	12.8	3.69	42.56	76.0	2.12	0.7	13.13	0.27	0.09	0.05	0.04
3	12.8	3.50	42.73	76.31	2.10	0.7	12.69	0.25	0.09	0.05	0.04
4	12.0	3.36	43.56	77.78	2.11	0.7	11.78	0.26	0.08	0.05	0.04
5	12.4	3.86	42.32	75.56	2.15	0.7	13.67	0.25	0.01	0.05	0.04
6	13.0	3.63	42.42	76.74	2.17	0.7	12.80	0.25	0.09	0.05	0.04
7	13.0	3.60	42.66	76.16	2.18	0.72	12.50	0.26	0.11	0.05	0.04
8	13.2	3.78	42.39	75.69	2.16	0.72	13.42	0.26	0.09	0.05	0.04
Average	12.725	3.68	42.5	76.15	2.14	0.70	13.026	0.25875	0.08125	0.05	0.04

Table 4.12, Main data for cyclone preheater - precalciner kiln

Major design data for preheater - precalciner kiln system	
Kiln system	6-stage single-string preheater with CDC precalciner
Clinker Cooler	Grate cooler
Production capacity	3000 tpd clinker
Kiln feed	Raw meal
Combustible	Heavy fuel oil, HFO
Major equipment dimension	
Kiln feed	Bucket elevator (Belt type), Lift Height = 118 m, Mass flow rate, normal 270 t/h Maximum flow rate 360 t/h Design temperature = 120℃ & Chain Speed = 1.917 m/s
First Cyclones (twins) (C1)	Outside diameter = 5700
Second Cyclone (C2)	Outside diameter = 7800
Third Cyclone (C3)	Outside diameter = 7800
Fourth Cyclone (C4)	Outside diameter = 8100
Fifth Cyclone (C5)	Outside diameter = 8100
Sixth Cyclone (C6)	Outside diameter = 8400
Precalciner	Inside diameter = 6400 mm
Precalciner burner	Capacity: HFO: 3000 kg /h (normal) & 3900kg/h (Max.)
Kiln	Φ 4.4 × 66 m
Inclination	3.5 %
Maximum kiln rpm	4
Cooler	Total grate area = 76.6 m^2, Used grate area = 72.5 m^2 & Stroke = 420 mm Clinker inlet and outlet temperature = 1400 and 65 ^{0}C respectively
Clinker crusher	Hammer crusher (Φ 1300 × 3000 mm), Speed 320 rpm, Outlet size ≤ 25 mm
Kiln burner	Burning capacity = 4000 kg/h(normal) & 5200 kg/h(max)(HFO)

Table 4.13, Parameters for heat and mass balance

Moisture in kiln feed = 0.6%	Clinker temperature (Tc) = 310 ^{0}C
Kiln feed rate = 210 tph	Kiln feed temperature (T_{kf}) = 25 ^{0}C
Clinker = 130.8 tph	HFO feed temperature (T_{HFO}) = 25 ^{0}C
Return dust in preheater @ 4.32% kiln feed rate	Ambient temperature (T_A) = 25 ^{0}C
Kiln feed to clinker factor (gross) = 1.605	Reference Temperature (Tr) = 25 ^{0}C
Kiln feed to clinker factor (net) = 1.536	Primary air temperature (T_{PA}) = 48 ^{0}C
Temperature of fuel feed to burner (T_{fuel}) = 110^{0}C	Cooler vent temperature (Tce) = 450 ^{0}C

Table 3.14, Material temperature variation in cyclone

T cyclone 6	T cyclone 5	T cyclone 4	T cyclone 3	T cyclone 2	T cyclone 1A	T cyclone 1B
856 ^{0}C	691 ^{0}C	567 ^{0}C	529 ^{0}C	486 ^{0}C	325 ^{0}C	332 ^{0}C

i. **Heat in preheater exit dust**

$Q_1 = m_d * C_{pd} * (Te – To)$

Where, m_d = kiln feed to clinker factor x preheater return dust

= 0.0432 * 1.605 = 0.0693 kg/ kg clinker

The exhaust gas exit temperature (Te) is around 350 ^{0}C & C_{pd} = 0.23 kg/ kg ^{0}C.

Q_1 = 0.0693* 0.23 * (350 – 25) = 5.18 kCal/kg clinker

ii. **Heat in preheater exit gas**

$Q_2 = m_e C_{pe} (Te – To)$

Where, m_e = mass of preheater exit gas in kg per kg of clinker (on dry basis). C_{Pe} = specific heat of preheater exit gas in kCal/ kg ^{0}C.

Since, moisture form kiln feed in preheater exit gases is very small, density of gas is estimated on dry basis. Now, at standard temperature and pressure, density of gases at preheater exit is determined as follows.

$$\rho_{stp} = \frac{O2*MO2+CO2*MCO2 + (N2+CO)*M}{22.4*100} \text{ --------------------------(4.13)}$$

$$= \frac{4.06*32+30.85*44 + (65.08)*28}{22.4*100} = 1.4775 \text{ kg/Nm}^3$$

The specific gas volume (Q_e) = volume of gas (Nm^3/hr)/ mass of clinker (kg/hr)

= 185,866/130,800 = 1.421 Nm^3/kg of clinker

$Q_2 = m_eC_{pe}$ (Te – To) = $\rho_e Q_e Cp_e$ (Te – To) ----------------- (4.14)

At the exit temperature and pressure Cpe = 0.247 kcal/kg^{0}C

Q_2 = 1.4775 *1.421*0.247* (350 – 25) = 198.54 kcal/kg of clinker

iii. Heat in clinker from cooler discharge

$Q_3 = m_c * Cp_c * (T_c – T_0)$ & Cp_c = 0.193 kCal/kg^{0}C

= 1 *0.193* (310 - 25) = 99 kCal/kg of clinker

iv. Heat in cooler exhaust air

$Q_4 = m_{ce} * Cp_{ce} * (T_{ce} – T_0)$ & Cp_{ce} = 0.244 kcal/kg^{0}C

Sp. volume of cooler exhaust air = volume of exhaust gas per hour / clinker production per hour

= 62,849.5/130800 = 0.4805 Nm^3/ kg of clinker

Density of air at Mugher, 0.848 kg/m^3

$Q_4 = \rho$* sp. V* $Cp_{ce} * (T_{ce} – T_0)$ = 1.293* 0.4805*0.244*425

= 64.54 kCal/kg of clinker

v. Heat of evaporation of kiln feed moisture

$Q_5 = m_{mr} * [Cp_{kf} (100 – T_{kf}) + L + Cp_s * (T_e – 100)]$ ------------------ (4.15)

Where

Cp_s = 0.457 kcal/kg ^{0}C & Cp_{kf} = 0.213 kcal/kg ^{0}C

m_{mr} = kiln feed to clinker factor * moisture in kiln feed

$= 1.605 * 0.006 = 0.00963$ kg/ kg of clinker

L – Latent heat of water evaporation = 540 kCal/ kg of water

$Q_5 = 0.00963 * [0.213 * (100 – 25) + 540 + 0.457 * (350 – 100)]$

$= 6.454$ kCal/ kg of clinker

vi. Heat loss due to incomplete combustion

$$Q_6 = m_{ic} * 67636 \text{ -- } (4.16)$$

Where, heat of combustion of CO = 67636 kCal/ kg mole and

$$m_{ic} = \frac{\% CO}{100*22.4} * \text{specific volume of preheater exit gases ------}(4.17)$$

By estimating 0.1% CO at the preheater outlet,

$$m_{ic} = \frac{0.1}{100*22.4} * 1.421 = 6.344 \times 10^{-5} \text{ kg mol/kg of clinker}$$

$$Q_6 = m_{ic} * 67636 = 4.2906 \text{ kCal/kg of clinker}$$

vii. Radiation and convection losses

A. Heat losses For surface of kiln

$$\text{Radiation losses } (Q_r) = \delta * \varepsilon * A * (Ts^4 – T_A^4), \text{ kCal/h ------------------------ } (4.18)$$

Where

δ – 4.88×10^{-8} kCal/ m^2 K & ε – surface emissivity = 0.8 for oxidized steel

Convection losses (Q_c), for wind velocity less than 3 meter per second

$$= 80.33 * [(T_A + Ts)/2]^{-0.724} * [(T_s - T_A)/2]^{1.33} * \text{surface area, kCal/h ----------- } (4.19)$$

At Mugher, the wind velocity is about 1.5 meter per second.

Table 4.15, Kiln shell temperature distribution along the kiln length

0 – 6.58 m	6.58 – 16.05 m	16.05 – 26.55	26.55 – 35.25	35.25 – 44.53	44.53 – 53.53	53.53 -58.78
311 ^{0}C	405 ^{0}C	365 ^{0}C	375 ^{0}C	308 ^{0}C	309 ^{0}C	259 ^{0}C
311 ^{0}C	404 ^{0}C	362 ^{0}C	375 ^{0}C	307 ^{0}C	308 ^{0}C	259 ^{0}C
311 ^{0}C	405 ^{0}C	371 ^{0}C	373 ^{0}C	305 ^{0}C	306 ^{0}C	258 ^{0}C

The kiln has length of 66 m and a diameter 4.2 m. throughout the length the kiln diameter is uniform. However, depending on the zone of the kiln, the shell temperature is different.

Surface area of kiln = 2π* D* L --- (4.20)

Where D is the diameter of the kiln and L – length of the kiln

(i) For the length up to 6.58 meter from kiln outlet

The average shell temperature becomes 311 ^{0}C or 584.15 K

Surface area of kiln = 2π* D* L = 173.642 m^2

Q_{c1} = 80.33 * [(298.15 + 584.15)/2]$^{-0.724}$ * [(584.15 – 298.15)/2]$^{1.33}$ * 173.642, kCal/h

= 124,867.4 kCal/h

Q_{r1} = 4.88 x 10^{-8} * 0.8 * 173.642* (584.15 ^{4} – 298.15^4), kCal/h = 735,768.6 kCal/h

Total heat losses = the sum of the two heat losses = 124,867.4 + 735,768.6 = 860,636 kCal/h

(ii) For the length between 6.58 & 16.05 m

Surface area of kiln = 2π* D* L = 249.91m^2

Q_{c2} = 80.33 * [(298.15 + 768.15)/2]$^{-0.724}$ * [(768.15– 298.15)/2]$^{1.33}$ * 249.91, kCal/h

= 303,349.1 kCal/h

Q_{r2} = 4.88 x 10^{-8} * 0.8 * 249.91* (768.15^4 – 298.15^4), kCal/h = 3319763.34 kCal/h

Total heat losses = 303,349.1 + 3319763.34 = 3,623,112 kCal/h

(iii) For the length between 26.55 & 35.25m

Surface area of kiln = $2\pi * D * L = 229.59\ m^2$

$Q_{c3} = 80.33 * [(298.15 + 768.15)/2]^{-0.724} * [(768.15 - 298.15)/2]^{1.33} * 229.59$, kCal/h

= 266,651.2 kCal/h

$Q_{r3} = 4.88 \times 10^{-8} * 0.8 * 229.59 * (748.15^4 - 298.15^4)$, kCal/h = 2,737,275.6 kCal/h

Total heat losses = 266,651.2 + 2737275.6 = 3003,927 kCal/h

(iv) For the length between 35.25 & 44.53 m

Surface area of kiln = $2\pi * D * L = 229.59\ m^2$

$Q_{c4} = 80.33 * [(298.15 + 580.15)/2]^{-0.724} * [(580.15 - 298.15)/2]^{1.33} * 244.9$, kCal/h

= 173406.2 kCal/h

$Q_{r4} = 4.88 \times 10^{-8} * 0.8 * 244.9 * (580.15^4 - 298.15^4)$, kCal/h = 1,007,500 kCal/h

Total heat losses = 173406.2 + 1007500 = 1,180,907 kCal/h

(v) For the length between 44.53 & 53.53 m

Surface area of kiln = $2\pi * D * L = 237.50\ m^2$

$Q_{c5} = 80.33 * [(298.15 + 581.15)/2]^{-0.724} * [(581.15 - 298.15)/2]^{1.33} * 237.50$, kCal/h

= 168,828.6 kCal/h

$Q_{r5} = 4.88 \times 10^{-8} * 0.8 * 237.50 * (581.15^4 - 298.15^4)$, kCal/h = 984362.6 kCal/h

Total heat losses = 168,828.6 + 984362.6 = 1,153,191 kCal/h

(vi) For the length between 53.53 & 58.78 m

Surface area of kiln = $2\pi * D * L = 138.54\ m^2$

$Q_{c6} = 80.33 * [(298.15 + 532.15)/2]^{-0.724} * [(532.15 - 298.15)/2]^{1.33} * 138.54$, kCal/h

= 79,721.04 kCal/h

$Q_{r6} = 4.88 \times 10^{-8} * 0.8 * 138.54 * (532.155^4 - 298.15^4)$, kCal/h = 391,004.78 kCal/h

Total heat losses = 79,721.04 + 391,004.78 = 470,725 kCal/h

(vii) For the length between 58.78 & 66 m

Surface area of kiln = $2\pi * D * L = 190.53\ m^2$

$Q_{c7} = 80.33 * [(298.15 + 473.15)/2]^{-0.724} * [(473.15 – 298.15)/2]^{1.33} * 190.53$, kCal/h

= 78,579.8 kCal/h

$Q_{r7} = 4.88 \times 10^{-8} * 0.8 * 190.53 * (473.155^4 – 298.15^4)$, kCal/h = 314,018.17 kCal/h

Total heat losses = 78,579.8 + 314,018.17 = 1,153,191 kCal/h

Total heat losses from the kiln shell = 11,745,689 kCal/h

B. Heat losses from preheater and precalciner surface

Table 4.16 shows the temperature variation on preheater shell surface that helps to determine the heat losses by radiation and convention from preheater surface.

Table 4.16, Temperature variation on the preheater shell surface

Surface average Temperature	Left 1st stage cyclone	Right 1st stage cyclone	2nd stage Cyclone	3rd stage cyclone	4th stage cyclone	5th stage cyclone	6th stage cyclone
Riser gas line (^{0}C)	69	70	76	100	140	120	120
Conic section (^{0}C)	96	95	117	124	180	200	280
Cylindrical (^{0}C)	63	63	105	180	80	175	235
Material duct (^{0}C)	105	105	107	190	240	300	350

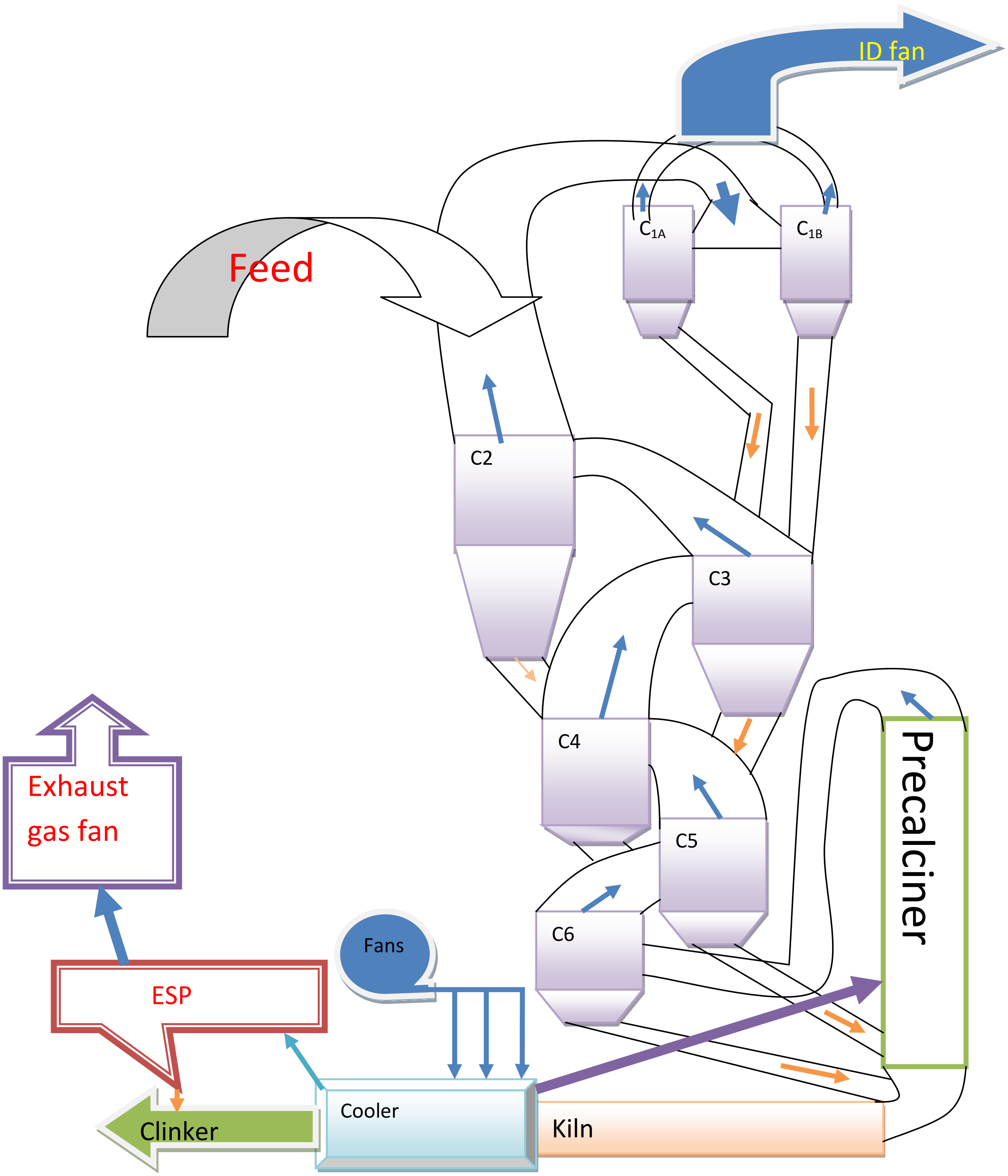

Fig. 4.5, Material flows in kiln system

Unlike that of kiln, the shape of the preheater is not uniform. To determine the surface heat losses, segmenting and approximating the preheater surface is critical. The following tables show the design dimension of different part preheater and precalciner.

Table 4.17, Dimensions of different preheater parts

Measurements	**Cyclone one (C_1)**	**Cyclone two (C_2)**	**Cyclone three (C_3)**	**Cyclone four (C_4)**	**Cyclone five (C_5)**	**Cyclone six (C_6)**
Length of Material duct , m	7	6	8	7.5	17.5	8.66
Height of frustum, m	6	10	9.2	9	11.5	8.9
Smaller Diameter of frustum, m	1.78	1.78	1.78	1.78	1.78	1.78
Bigger diameter of frustum , m	5.7	7.8	7.8	8.1	8.1	8.4
Height of cyclone, m	10.45	7	8.8	8	6	10
Diameter of cyclone, m	5.7	7.8	7.8	8.1	8.1	8.4
Gas duct length, m	Chimney	16.45	13	14	16.19	14
Diameter of gas duct, m		4.12	4.5	4.7	4.96	4.7

Table 4.18, Dimensions of precalciner parts

Measurements	Height (m)	Diameter , (m)	Gas duct length, (m)	Gas duct diameter, m	Frustum top and bottom diameter, m	Frustum length, m
Precalciner	30	7.86	39.5	4.7	7.86 & 1.78 respectively	2.44

The main heat losses mechanisms from the preheater surface are radiation and convection. The same equation used for kiln to calculate radiation and convective heat losses work.

Surface area of frustum = π (r + R) ($\sqrt{((Rr)^2} + h^2) + \pi(R^2 + r^2))$ ----- (4.21)

For rectangular prism guesstimate can be used for material duct line. The gas lines and cyclone surfaces are approximated as cylindrical surface.

Surface area of rectangular prism = 2(wℓ + wh + ℓh) --------------------------(4.22)

However, for squared base area = 2 ℓ (ℓ + 2h)

i. Heat losses from cyclone six surface (Q_{C6})

- Gas duct (Ts = 120 ^{0}C, h = 14 m & Ø = 4.7 m)

Surface area = 2πDℓ = 413.43 m^2

Q_r = 4.88 x 10^{-8} * 0.8 * 413.43 * (393.15^4 – 298.15^4) = 258,068 kCal/h

Q_c = 80.33 * [(298.15 + 393.15)/2]$^{-0.724}$ * [(393.15– 298.15)/2]$^{1.33}$ * 413.43, kCal/h

= 81905 kCal/h

Total heat losses = 81905+ 258,068 = 339,973.4 kCal/h

- Frustum (Ts = 280 ^{0}C, r = 0.89 m & R = 4.2 m, h = 8.9 m) heat losses

Surface area = π (0.89 + 4.2) ($\sqrt{((0.89 * 4.2)^2} + 8.9^2) + \pi(4.2^2 + 0.89^2))$ = 212.3 m^2

Q_r = 4.88 x 10^{-8} * 0.8 * 212.3 * (553.15^4 – 298.15^4) = 710,339.5 kCal/h

Q_c = 80.33 * [(298.15 + 553.15)/2]$^{-0.724}$ * [(553.15– 298.15)/2]$^{1.33}$ * 212.3, kCal/h

= 134,478.6 kCal/h

Total heat losses = 134,478.6 + 710,339.5 = 844818 kCal/h

- Material duct (Ts = 350 ^{0}C, ℓ = w =0.9 m & h = 8.66 m) heat losses

Surface area = 2 ℓ (ℓ + 2h) = 32.8 m^2

Q_r = 4.88 x 10^{-8} * 0.8 * 32.8 * (623.15^4 – 298.15^4) = 182,946 kCal/h

Q_c = 80.33 * [(298.15 + 623.15)/2]$^{-0.724}$ * [(623.15– 298.15)/2]$^{1.33}$ * 32.8, kCal/h

= 27,092 kCal/h

Total heat losses = 182,946 + 27,092 = 210,039.2 kcal/hr

- Cyclone heat losses (Ts = 235 ^{0}C, h = 10 m & Ø = 8.4 m)

Surface area = $2\pi D\ell$ = 527.8 m^2

$Q_r = 4.88 \times 10^{-8} * 0.8 * 527.8 * (508.15^4 - 298.15^4)$ = 1,211,021.5 kCal/h

$Q_c = 80.33 * [(298.15 + 508.15)/2]^{-0.724} * [(508.15 - 298.15)/2]^{1.33} * 527.8$, kCal/h

= 268,634 kCal/h

Total heat losses = 1,211,021.5 + 268,634= 1,479,655 kCal/h

$\mathbf{Q_{C1}}$= 1,479,655 + 844818 + 210,039.2 + 339,973.4 = 2,874,485.6 kCal/h

j. Heat losses from cyclone five (Q_{C5})

- Gas duct (Ts = 120 ^{0}C, h = 16.19 m & Ø = 4.96 m)

Surface area = $2\pi D\ell$ = 504.5 m^2

$Q_r = 4.88 \times 10^{-8} * 0.8 * 504.5 * (393.15^4 - 298.15^4)$ = 314946.4 kCal/h

$Q_c = 80.33 * [(298.15 + 393.15)/2]^{-0.724} * [(393.15 - 298.15)/2]^{1.33} * 504.5$, kCal/h

= 99957.6 kCal/h

Total heat losses = 99957.6 + 314946.4 = 414904 kCal/h

- Frustum (Ts = 175^{0}C, r = 0.89 m & R = 4.05 m, h = 11.5 m) heat losses

Surface area = $\pi (0.89 + 4.05) (\sqrt{((0.89 * 4.05)^2} + 11.5^2) + \pi(4.05^2 + 0.89^2))$ = 241.05 m^2

$Q_r = 4.88 \times 10^{-8} * 0.8 * 241.05 * (448.15^4 - 298.15^4)$ = 305,228 kCal/h

$Q_c = 80.33 * [(298.15 + 448.15)/2]^{-0.724} * [(448.15 - 298.15)/2]^{1.33} * 241.05$, kCal/h

= 82,943 kCal/h

Total heat losses = 82,943 + 305,228 = 388171 kCal/h

- Material duct (Ts = 300 ^{0}C, ℓ = w =0.9 m & h = 17.5 m) heat losses

Surface area = $2 \ell (\ell + 2h)$ = 64.62 m^2

$Q_r = 4.88 \times 10^{-8} * 0.8 * 64.62 * (573.15^4 - 298.15^4)$ = 252,303.8 kCal/h

$Q_c = 80.33 * [(298.15 + 573.15)/2]^{-0.724} * [(573.15 – 298.15)/2]^{1.33} * 64.62$, kCal/h

= 44509.4 kCal/h

Total heat losses = 252,303.8 + 44509.4 = 296,813.2 kcal/hr

- Cyclone heat losses (Ts = 200 ^{0}C, h = 6 m & Ø = 8.1 m)

Surface area = $2\pi D\ell = 305.4\ m^2$

$Q_r = 4.88 \times 10^{-8} * 0.8 * 305.4 * (473.15^4 – 298.15^4)$ = 503,274 kCal/h

$Q_c = 80.33 * [(298.15 + 473.15)/2]^{-0.724} * [(473.15 – 298.15)/2]^{1.33} * 305.4$, kCal/h

= 125939 kCal/h

Total heat losses = 503,274 + 125939 = 629,213 kCal/h

$\mathbf{Q_{C1}}$= 629213+ 296,813.2 + 388171+ 414904 = 1,729,101 kCal/h

k. Heat losses from surface of cyclone four (Q_{C4})

- Gas duct (Ts = 140 ^{0}C, h = 14 m & Ø = 4.7 m)

Surface area = $2\pi D\ell = 413.4\ m^2$

$Q_r = 4.88 \times 10^{-8} * 0.8 * 413.4 * (413.15^4 – 298.15^4)$ = 342,726 kCal/h

$Q_c = 80.33 * [(298.15 + 413.15)/2]^{-0.724} * [(413.15 – 298.15)/2]^{1.33} * 413.4$, kCal/h

= 103,443 kCal/h

Total heat losses = 103,443+ 342,726 = 446,169.8 kCal/h

- Frustum (Ts = 180 ^{0}C, r = 0.89 m & R = 4.05 m, h = 9 m) heat losses

Surface area = $\pi\ (0.89 + 4.05)\ (\sqrt{((0.89 * 4.05)^2} + 9^2) + \pi(4.05^2 + 0.89^2\ ))$ = 204.5 m^2

$Q_r = 4.88 \times 10^{-8} * 0.8 * 204.5 * (453.15^4 – 298.15^4)$ = 273,528 kCal/h

$Q_c = 80.33 * [(298.15 + 453.15)/2]^{-0.724} * [(453.15 – 298.15)/2]^{1.33} * 204.5$, kCal/h

= 73,140 kCal/h

Total heat losses = 273,528 + 73,140 = 346,669 kCal/h

- Material duct (Ts = 240^{0}C, ℓ = w =0.9 m & h = 7.5 m) heat losses

Surface area = 2 ℓ (ℓ + 2h) = 28.62 m^2

Q_r = 4.88 x 10^{-8} * 0.8 * 28.62 * (513.15^4 – 298.15^4) = 68,645 kCal/h

Q_c = 80.33 * [(298.15 + 513.15)/2]$^{-0.724}$ * [(513.15– 298.15)/2]$^{1.33}$ * 28.62, kCal/h

= 14,963 kCal/h

Total heat losses = 68,645 + 14,963 = 83,608 kcal/hr

- Cyclone heat losses (Ts = 80 ^{0}C, h = 8 m & Ø = 8.1 m)

Surface area = 2πDℓ = 407.15 m^2

Q_r = 4.88 x 10^{-8} * 0.8 * 407.15 * (353.15^4 – 298.15^4) = 121,626 kCal/h

Q_c = 80.33 * [(298.15 + 353.15)/2]$^{-0.724}$ * [(353.15– 298.15)/2]$^{1.33}$ * 407.15, kCal/h

= 40,711.15 kCal/h

Total heat losses = 121,626 + 40,711.15 = 189,844.4 kCal/h

Q_{C1}= 189,844.4 + 83,608 + 346,669 + 446,169.8 = 1,066,291.2 kCal/h

l. Heat losses from surface of cyclone three (Q_{C3})

- Gas duct (Ts = 100 ^{0}C, h = 13 m & Ø = 4.5 m)

Surface area = 2πDℓ = 367.6 m^2

Q_r = 4.88 x 10^{-8} * 0.8 * 367.6 * ($373.15.15^4$ – 298.15^4) = 164,821 kCal/h

Q_c = 80.33 * [(298.15 + 373.15)/2]$^{-0.724}$ * [(373.15– 298.15)/2]$^{1.33}$ * 367.6, kCal/h

= 54,316.67 kCal/h

Total heat losses = 164,821 + 54,316.67 = 219,138.3 kCal/h

- Frustum (Ts = 124 ^{0}C, r = 0.89 m & R = 3.9 m, h = 9.2 m) heat losses

Surface area = π (0.89 + 3.9) ($\sqrt{((0.89 * 3.9)^2 + 9.2^2)} + \pi(3.9^2 + 0.89^2)$) = 198.2 m^2

Q_r = 4.88 x 10^{-8} * 0.8 * 198.2 * (397.15^4 – 298.15^4) = 131,093.2 kCal/h

Q_c = 80.33 * [(298.15 + 397.15)/2]$^{-0.724}$ * [(397.15– 298.15)/2]$^{1.33}$ * 198.2, kCal/h

= 41,238.4 kCal/h

Total heat losses = 131,093.2 + 41,238.4 = 172,331.6 kCal/h

- Material duct (Ts = 190 ^{0}C, ℓ = w =0.9 m & h = 8 m) heat losses

Surface area = 2 ℓ (ℓ + 2h) = 30.42 m^2

$Q_r = 4.88 \times 10^{-8} * 0.8 * 30.42 * (463.15^4 – 298.15^4) = 45,261.22$ kCal/h

$Q_c = 80.33 * [(298.15 + 463.15)/2]^{-0.724} * [(463.15– 298.15)/2]^{1.33} * 30.42$, kCal/h

= 11,711.7 kCal/h

Total heat losses = 45,261.22 + 11,711.7 = 56,972.93 kcal/hr

- Cyclone heat losses (Ts = 180 ^{0}C, h = 8.8 m & Ø = 7.8 m)

Surface area = 2πDℓ = 431.3 m^2

$Q_r = 4.88 \times 10^{-8} * 0.8 * 431.3 * (453.15^4 – 298.15^4) = 576,914.3$ kCal/h

$Q_c = 80.33 * [(298.15 + 453.15)/2]^{-0.724} * [(453.15– 298.15)/2]^{1.33} * 431.3$, kCal/h

= 154,263.4 kCal/h

Total heat losses = 154,263.4 + 576,914.3 = 731,177.7 kCal/h

$\mathbf{Q_{C1}}$= 219,138.3 + 172,331.6 + 56,972.93 + 731,177.7 = 1,179,620 kCal/h

m. Heat losses from surface of cyclone two (Q_{C2})

- Gas duct (Ts = 76 ^{0}C, h = 16.45 m & Ø = 4.12 m)

Surface area = 2πDℓ = 425.8 m^2

$Q_r = 4.88 \times 10^{-8} * 0.8 * 425.8 * (349.15.15^4 – 298.15^4) = 115,690.4$ kCal/h

$Q_c = 80.33 * [(298.15 + 349.15)/2]^{-0.724} * [(349.15– 298.15)/2]^{1.33} * 425.8$, kCal/h

= 38,683.4 kCal/h

Total heat losses = 115,690.4 + 38,683.4 = 154,373.8 kCal/h

- Frustum (Ts = 117 ^{0}C, r = 0.89 m & R = 3.9 m, h = 10 m) heat losses

Surface area = π (0.89 + 3.9) $(\sqrt{((0.89 * 3.9)^2} + 10^2) + \pi(3.9^2 + 0.89^2))$ = 209.6 m^2

$Q_r = 4.88 \times 10^{-8} * 0.8 * 209.6 * (390.15^4 – 298.15^4) = 124,620.3$ kCal/h

Q_c = 80.33 * [(298.15 + 390.15)/2]$^{-0.724}$ * [(390.15– 298.15)/2]$^{1.33}$ * 209.6, kCal/h

= 39,827 kCal/h

Total heat losses = 124,620.3 + 39,827 = 164,447.4 kCal/h

- Material duct (Ts = 107^{0}C, ℓ = w =0.9 m & h = 6 m) heat losses

Surface area = 2 ℓ (ℓ + 2h) = 23.22 m^2

Q_r = 4.88 x 10^{-8} * 0.8 * 23.22 * (380.15^4 – 298.15^4) = 11,768.52 kCal/h

Q_c = 80.33 * [(298.15 + 380.15)/2]$^{-0.724}$ * [(380.15– 298.15)/2]$^{1.33}$ * 23.22, kCal/h

= 3834.8 kCal/h

Total heat losses = 11,768.52 + 3834.8 = 15,603.3 kcal/hr

- Cyclone heat losses (Ts = 105 ^{0}C, h = 7 m & Ø = 7.8 m)

Surface area = 2πDℓ = 343 m^2

Q_r = 4.88 x 10^{-8} * 0.8 * 343 * (378.15^4 – 298.15^4) = 169,033.3 kCal/h

Q_c = 80.33 * [(298.15 + 378.15)/2]$^{-0.724}$ * [(378.15– 298.15)/2]$^{1.33}$ * 343, kCal/h

= 54,943.3 kCal/h

Total heat losses = 169,033.3 + 54,943.3 = 222976.6 kCal/h

Q_{C1}= 222976.6 + 15,603.3 + 164,447.4 + 154,373.8 = 557,401 kCal/h

n. Heat losses from surface of cyclone one (Q_{C1})

- Frustum (Ts = 95 ^{0}C, r = 0.89 m & R = 2.85 m, h = 6 m) heat losses

Surface area = π (0.89 + 2.85) ($\sqrt{((0.89 * 2.85)^2} + 6^2) + \pi(2.85^2 + 0.89^2$)) = 104.55 m^2

Q_r = 4.88 x 10^{-8} * 0.8 * 104.55 * (368.15^4 – 298.15^4) = 51,206.2 kCal/h

Q_c = 80.33 * [(298.15 + 368.15)/2]$^{-0.724}$ * [(368.15– 298.15)/2]$^{1.33}$ * 104.55, kCal/h

= 16,743.3 kCal/h

Total heat losses = 16,743.3 + 51,206.2 = 67,949.5 kCal/h

- Material duct (Ts = 105^{0}C, ℓ = w =0.9 m & h = 7 m) heat losses

Surface area = 2 ℓ (ℓ + 2h) = 26.82 m^2

Q_r = 4.88 x 10^{-8} * 0.8 * 26.82 * (378.15^4 – 298.15^4) = 13,136.56 kCal/h

Q_c = 80.33 * [(298.15 + 378.15)/2]$^{-0.724}$ * [(378.15– 298.15)/2]$^{1.33}$ * 26.82, kCal/h

= 4,295.345 kCal/h

Total heat losses = 4,295.345 + 13,136.56 = 17491.93 kcal/hr

- Cyclone heat losses (Ts = 63 ^{0}C, h = 10.45 m & Ø = 5.7 m)

Surface area = 2πDℓ = 374.25 m^2

Q_r = 4.88 x 10^{-8} * 0.8 *374.25 * (340.15^4 – 298.15^4) = 80,140.3 kCal/h

Q_c = 80.33 * [(298.15 + 340.15)/2]$^{-0.724}$ * [(340.15– 298.15)/2]$^{1.33}$ * 374.25, kCal/h

= 26,528.22 kCal/h

Total heat losses = 26,528.22 + 80,140.3 = 106,668.5 kCal/h

Q_{C1}= 106,668.5 + 17,491.93 + 67,949.5 = 192,103 kCal/h

At top, there are twin cyclones which are almost thermally equilibrium. Therefore, the heat losses from them become twice of the single, 384220 kCal/h.

o. Heat losses from surface of precalciner (Q_{PC})

Table 4.19, Temperature variation on the surface of precalciner

Gas line surface temperature	Frustum average temperature	Cylindrical surface temperature
190 ^{0}C	135^{0}C	140^{0}C

- Frustum (Ts = 135 ^{0}C, r = 0.89 m & R = 3.39 m, h = 2.44m) heat losses

Surface area = π (0.89 + 3.39) ($\sqrt{((0.89 * 3.39)^2} + 6^2) + \pi(3.39^2 + 0.89^2$)) = 115.6 m^2

Q_r = 4.88 x 10^{-8} * 0.8 * 115.6 * (405.15^4 – 298.15^4) = 89,569.6 kCal/h

Q_c = 80.33 * [(298.15 + 405.15)/2]$^{-0.724}$ * [(405.15– 298.15)/2]$^{1.33}$ * 115.6, kCal/h

= 27,400 kCal/h

Total heat losses = 27,400 + 89,569.6 = 116,967.6 kCal/h

◆ Cyclone heat losses (Ts = 140 ^{0}C, h = 30 m & Ø = 7.86 m)

Surface area = 2πDℓ = 1481.575 m^2

Q_r = 4.88 x 10^{-8} * 0.8 *1481.575 * (410.15^4 – 298.15^4) = 1,179,773.3 kCal/h

Q_c = 80.33 * [(298.15 + 410.15)/2]$^{-0.724}$ * [(410.15– 298.15)/2]$^{1.33}$ * 1481.575, kCal/h

= 358,988 kCal/h

Total heat losses = 1,179,773.3 + 358,988 = 1,538,761 kCal/h

◆ Gas line heat losses (Ts = 190 ^{0}C, h = 39.5 m & Ø = 7.86 m)

Surface area = 2πDℓ = 1950.7 m^2

Q_r = 4.88 x 10^{-8} * 0.8 *1481.575 * (463.15^4 – 298.15^4) = 2,902,462 kCal/h

Q_c = 80.33 * [(298.15 + 463.15)/2]$^{-0.724}$ * [(463.15– 298.15)/2]$^{1.33}$ * 1481.575, kCal/h

= 751,095.3 kCal/h

Total heat losses = 2,902,462 + 751,095.3 = 3,653,498 kCal/h

Total heat losses from the preheater and precalciner (Q_{Pp}) = 384220 + 557,401 + 1,179,620 + 1,066,291.2 + 1,729,101 + 2,874,485.6 + 3,653,498 = 11,444,617 kCal/h

C. Heat losses from the surface of tertiary air duct

Table, 4.20, Temperature distribution over the tertiary air duct surface

Surface temperature at the inlet	0 – 20 m	20 – 40 m	40 – 60 m	60 – 68 m	Outlet temperature
310 ^{0}C	299	278	256.3	241	237 ^{0}C

◆ Heat losses on the surface for interval of 0 - 20 m (Ts = 299 ^{0}C, h = 20 m & Ø = 3.8 m)

Surface area = 2πDℓ = 477.5 m^2

Q_r = 4.88 x 10^{-8} * 0.8 *477.5 * (572.15^4 – 298.15^4) = 1,850,444.5 kCal/h

Q_c = 80.33 * [(298.15 + 572.15)/2]$^{-0.724}$ * [(572.15– 298.15)/2]$^{1.33}$ * 477.5, kCal/h

= 327,593.3 kCal/h

Total heat losses = 327,593.3 + 1,850,444.5 = 217,803.8 kCal/h

- Heat losses on the surface for interval of 20 - 40 m (Ts = 278 ^{0}C, h = 20 m & Ø = 3.8 m)

Surface area = 2πDℓ = 477.5 m^2

$Q_r = 4.88 \times 10^{-8} * 0.8 * 477.5 * (551.15^4 – 298.15^4) = 1,572,900$ kCal/h

$Q_c = 80.33 * [(298.15 + 551.15)/2]^{-0.724} * [(551.15– 298.15)/2]^{1.33} * 477.5$, kCal/h

= 299,886.6 kCal/h

Total heat losses = 1,572,900 + 299,886.6 = 1,872,787 kCal/h

- Heat losses on the surface for interval of 40 - 60 m (Ts = 256 ^{0}C, h = 20 m & Ø = 3.8 m)

Surface area = 2πDℓ = 477.5 m^2

$Q_r = 4.88 \times 10^{-8} * 0.8 * 477.5 * (529.15^4 – 298.15^4) = 1,317,570$ kCal/h

$Q_c = 80.33 * [(298.15 + 529.15)/2]^{-0.724} * [(529.15– 298.15)/2]^{1.33} * 477.5$, kCal/h

= 271205.6 kCal/h

Total heat losses = 271205.6 + 1,317,570 = 1,588,776 kCal/h

- Heat losses on the surface for interval of 60 - 66 m (Ts = 241 ^{0}C, h = 8 m & Ø = 3.8 m)

Surface area = 2πDℓ = 191 m^2

$Q_r = 4.88 \times 10^{-8} * 0.8 * 191 * (514.15^4 – 298.15^4) = 462,176$ kCal/h

$Q_c = 80.33 * [(298.15 + 514.15)/2]^{-0.724} * [(514.15– 298.15)/2]^{1.33} * 191$, kCal/h

= 100,391.2 kCal/h

Total heat losses = 100,391.2 + 462,176 = 562,567.7 kCal/h

The total tertiary air duct heat losses = 562,567.7 + 1,588,776 + 217,803.8 + 1,872,787

= 4,241,934.5 kCal/h

D. Cooler surface heat losses

The cooler surface average temperature is 90 centigrade and its nearly rectangular shape with 4 * 7 * 30 meter. Then, the total surface area becomes 596 m^2.

$Q_r = 4.88 \times 10^{-8} * 0.8 * 596 * (363.15^4 – 298.15^4) = 220{,}806$ kCal/h

$Q_c = 80.33 * [(298.15 + 363.15)/2]^{-0.724} * [(363.15– 298.15)/2]^{1.33} * 596$, kCal/h

= 73,605 kCal/h

Total radiation and convection heat losses = 220,806 + 73,605 = 294, 411 kCal/h

Radiation and convection losses per kg of clinker = 2.727 kCal/kg of clinker

Table 4.21, summary of heat output for the kiln system

Items	kCal/h	kCal/kg of clinker	Amount in %
Heat output			
Heat of formation of clinker	47,952,000	369	40.75
Heat in dust in preheater exit gases	559,440	5.18	0.57
Heat in preheater exit gases	21,442,320	198.54	21.93
Heat in clinker exhaust air	8,532,000	79	8.7
Heat in clinker leaving cooler	9,130,320	84.54	9.3
Heat for evaporation of moisture of kiln feed	718,632	6.654	0.72
Heat in incomplete combustion	463,384.8	4.2906	0.47
Radiation and convection losses	16,934,400	156.8	17.31
Heat total output	97,783,200	905.4	100

4.2.2.2 Heat input

Heat can enter to the kiln system with kiln feed, cooling air, primary air, leaking air, incoming fuel and from the combustion product of fuel. The heat incoming with cooling air, kiln feed, leaking air and primary air is sensible heat whereas the heat from combustion is a chemical energy of the fuel. Since the reference temperature taken is 25 ^{0}C there will be no heat entering with incoming cooling air, kiln feed and leaking air. As a result, the sensible heat enters only with primary air & fuel. The major heat source is combustion the combustion of fuel.

A. Sensible heat in primary air

$$Q_p = mp * C_{pair} *(T_{pA} - To) \text{ --- } (4.23)$$

Where m_p = primary air to kiln + primary air to calciner

From mass balance it is found that the primary air flow rate is about 11,460.8 Nm^3/h.

$$\text{Mass flow rate } (m_A) = \rho_{STP} * Q_{STP}$$

$$= 1.293 \text{ kg/Nm}^3 * 11{,}460.8 \text{ Nm}^3\text{/h} = 14{,}818.8 \text{ kg/h}$$

$$m_p = \text{mass flow rate of primary air(kg/hr) / production rate of clinker (kg/hr)}$$

$$= 14{,}818.8/130800 = 0.1133 \text{ kg of primary air/ kg of clinker}$$

At 48 ^{0}C, specific heat capacity of air is 1.0065 kJ/kgK or 0.237 kCal/kgk

$$Q_p = 0.1133 * 0.237 *(48 - 25) = 0.6176 \text{ kCal/kg of clinker}$$

B. Sensible heat of fuel

$$Q_{fuel} = m_{fuel} * C_{pfuel} * (T_{fuel} - T_0)$$

$$m_{fuel} = 10529 \text{ kg/h}/130800 \text{ kg/h and } C_{pfeul} = 0.288 \text{ kCal/kgK}$$

$$Q_{fuel} = 0.084 * 0.288 * (110 - 25) = 2.06 \text{ kCal/kg of clinker}$$

C. Heat from fuel combustion

$$\mathbf{Q_{CV}} = m_{fuel} * \text{caloric value of the fuel ----------------------------------- } (4.24)$$

$$= 10760 \text{ kcal/kg of fuel} * 0.084 \text{ kg of fuel/ kg of clinker}$$

$$= 903.8 \text{ kCal/kg of clinker}$$

Table 4.22, Summary of heat input to kiln system

Heat input	kCal/h	Kcal/kg of clinker
Sensible heat in primary air	80,782.08	0.6176
Sensible heat from in fuel	222,480	2.06
Heat from coal combustion	97,524,000	903.8
Total heat input	97,903,900.8	906.52

4.2.2.3 Raw mill heat balance

It will be useful to perform heat balance of the raw mill to optimize the air flow in the mill to improve productivity besides saving energy.

Table 4.23, operating parameters of vertical roller mill

T_h	Hot gas temperature	^0C	240
C_G	Specific heat of hot gases	kCal/Nm^{30}C	0.34
T_0	Reference temperature	^0C	25
T_A	Ambient temperature	^0C	25
$A\ell$	Altitude from sea level	m	2460
m_F	Feed quantity	tph	300
ω_f	Feed moisture content (surface)	%	6
ω_p	Product moisture content (surface)	%	0.6
C_f	Specific heat of raw material	kCal/kg^{0}C	0.21
Fa	False air percentage	%	10
P	Power drawn by mill motor	kW	2318.5
T_{0ut}	Out let gas temperature	^0C	100
S	Surface radiation losses	K Cal/^{0}C difference	50
A	Surface area	m^2	250

A. Heat output

1) Heat to raw material

$$Q_{rm} = m_F C_f (T_{0ut} - T_A - 5) \text{ --- } (4.25)$$

$$= 1{,}470{,}000 \text{ k Cal/hr}$$

Note: the raw material temperature is normally less by 5 centigrade than the exit gas temperature.

2) Radiation losses

$$Q_r = A\, S\, (T_{0ut} - T_A) \quad = 937{,}500 \text{ k Cal/hr}$$

3) Heat loss to evaporate moisture

$$Q_m = W * (540 + C_w (T_{out} - T_A))$$

$$W = \frac{mF * 1000\ (\omega f - \omega p)}{100 - \omega p} = 6122 \text{ kg/hr}$$

$$Q_m = 3{,}765{,}306 \text{ k Cal/hr}$$

Note: The latent heat of evaporation & specific heat capacity of water are 540 kCal/ kg and 4.2 kJ/kg ^{0}C respectively.

4) Heat loss due to false air

$$Q_{fa} = F * 0.1 * C_A (T_{out} - T_0) = 2.25 \text{ F k Cal/hr}$$

Where F is hot gas feed flow rate and C_A is specific heat capacity of ambient air.

Total heat loss at output:

$$Q_{out} = Q_{rm} + Q_r + Q_m + Q_{fa} \text{ -------------------------------------- } (4.26)$$

$$= (1{,}470{,}000 + 937{,}500 + 3{,}765{,}306 + 2.25 \text{ F) k Cal/hr}$$

$$= 6{,}172{,}806 + 2.25 \text{ F, k Cal/hr}$$

B. Heat input

a) Heat from grinding power:

$$Q_p = P * \eta_m * \eta_{\text{gear box}} * 860 \text{ -------------------------------- } (4.27)$$

$$= 0.98 * 2318.5 * 0.94 * 860$$

$$= 1{,}836{,}789.9 \text{ k cal/hr}$$

b) Heat from hot gas:

$$Q_h = F * C_G * (T_h - T_0) \text{ --------------------------------------- } (4.28)$$

$$= 73.1 \text{ F}$$

Total heat input = $Q_p + Q_h$ = 1,836,789.9 + 73.1 F, k cal/hr

C. **Heat balance**

Heat input = heat out put

1,836,789.9 + 73.1 F = 6,172,806 + 2.25 F, k Cal/hr and F = 61,200 Nm^3/hr of hot exhaust gas

The heat needed from the hot gas = 73.1 F = 4,473,716 k Cal/hr

5. Electrical energy evaluation of kiln system’s performance

The major electrical energy overriding equipments in cement factories are mills, fans, pumps comprressors, heaters and driving units. Unless certain criterion is set for performance evaluation of electrical units, consider all units is not easy. Consequently, for this study motors with power consumption of greater than 100kw are considered.

Table 5.1, Summary of electrical units which have power consumption higher than 100 kW

Electrical units	Capacity	Specification	Rated volt (V)	Rated power (kW)
Belt conveyer, 3003	400 tph	315 m, lifting height = 23 m, speed = 1.25 m/s,	pumice and gypsum	
Cement mills, 3401& 3402	105 tph	16.1 rpm, PPC (32.5, 3800 cm^2 /g, grinding media = 254 tones, motor speed = 995 rpm Cement mill size = dia. 4.2 m & 14.5 m.	6000	4000
Separator, 3413, O - SEPA	88 – 125 tph	145 – 190 rpm (separator), 1030 – 1350 rpm, motor, VSD	380	110
Exhaust fan, 3433 & 3434	180,000 m^3/h	TP = 5800 Pa, fan rpm = 1479 & motor rpm = 1485 rpm,	6000	450
Mill fans, 3437 & 3438	82,000 m^3/h	TP = -5400 Pa, fan & motor rpm = 1485 rpm	380	185
Rotary dryer, 4507	75 t /h	Size: Φ 3.5 ×25 m, Moisture inlet and outlet ≤20% & ≤ 5 % respectively, Hot gas temp. at inlet and outlet ≤800℃ & ≤130 ℃ respectively, Rotary speed 3.7 r/min, Inclination = 3.5%		132

Electrical units	Capacity	Specification	Rated voltage (V)	Rated power (kW)	Other specification
Limestone crusher	900 tph	Min. size < 75 mm (95%)	10,000	1400	Max. feed size 1.1 * 1.1 * 1.85
Belt conveyer	1125 tph	0 - 3 m/s, 1166.982 m	2 motors & gear boxes	630	Gear box speed ratio = 31.5
Belt conveyer	1125 tph	0 - 3 m/s, 333.558 m	>>	160	>>
Belt conveyer	1125 tph	170.098 m		185	
Belt conveyer	1125 tph	2.25 m/s, 178.948 m		185	
Stacker	1125 tph	Radius = 23.85 m, speed = 0.08 rpm		190	
Reclaimer	350 tph	Speed of scraper = 0.46 m/s			
Scrape board Reclaimer	100 tph	For sand, high grade limestone, clay		105	
VRM, 2104 (table dia. = 4.5 m, rollers dia. = 2.5 m, 3 rollers)	270 tph	Feeding mat. size = 0 – 25 mm, 95% Feeding mat. Moisture = 4.1% (normal), max = 7.1%	6000 Speed = 993 rpm, 289 A	2450, reducer for main drive cooling water cons = 43 m^3/h	Grinding fineness = ≤ 12 % R 90µm, residual surface moisture ≤ 0.5%
Separator, 2108		Speed = 132 rpm, VFD cooling water = 10 m^3/h	380 V		
System fan/ circulating fan, 2132	643,000 m^3/h	993 rpm (both motor and fan), TP = 9600 pa Gas T = 95 ^{0}C & 120 ^{0}C, max	6000	2400, 289 A	Cooling water cons. = 1.1, m/s max & elect. oil heater = 200 V & 2 kW
ID fan, 2201	780,000 m^3/h	TP = 7800 pa, 993 rpm (motor and fan), gas T = 280 ^{0}C, 430 ^{0}C, max	3300	2400	>>
Exhaust fan, 2211	940,000 m^3/h	TP = 3800 pa, 750 rpm (fan), gas T = 150 ^{0}C, 250 ^{0}C, max, 742 rpm (motor)	3300	1400	Cooling water cons. = 1.1m^3/h max & elect. oil heater = 220 V & 2 kW
Bucket elevator, 2301	270 tph, normal, 360 tph, design	Lift height = 82.9 m, chain speed = 1.78 m/s		160	Raw meal T = 80 ^{0}C, 120 ^{0}C, upset density, dry = 0.8 t/m^3

Electrical units	Capacity	Specification	Rated voltage (V)	Rated power (kW)
Kiln main drive motor, 2501M1	99 – 985 rpm		690	500
Clinker cooler, 2602	3000 tpd	Total grate area =76.5 m^2, Used grate area = 72. 5 m^2 Stroke = 420 mm, clinker inlet T = 1400 ^{0}C, Clinker out let 65 ^{0}C + ambient	380	110
Cooling air fan, 2604	33300 m^3/h	TP = 12100 Pa, 2981 rpm (motor)		185
Cooling air fan, 2605	35300 m^3/h	TP = 11,006 Pa, 2981 rpm (motor)		185
Cooling air fan, 2606	39000 m^3/h	TP = 13668 Pa, 2981 rpm (motor)		185
Cooling air fan, 2607	23900 m^3/h	TP = 12546 Pa, 2979 rpm (motor)		132
Cooling air fan, 2608	31000 m^3/h	TP = 12430 Pa, 2982 rpm (motor)		160
Cooling air fan, 2609	46,400 m^3/h	TP = 11,670 Pa, 1450 rpm (motor)		220
Cooling air fan, 2610	30300 m^3/h	TP = 94813, Pa, 1450 rpm (motor)		185
Cooling air fan, 2611	30300 m^3/h	TP = 13,948 Pa, 2981 rpm (motor)		185
Cooling air fan, 2612	41100 m^3/h	TP = 10,800 Pa, 1486 rpm (motor)		185
Cooling air fan, 2613	35900 m^3/h	TP = 9947 Pa, 1487 rpm (motor)		160
Cooling air fan, 2614	42,800 m^3/h	TP = 8974 Pa, 1486 rpm (motor)		185
Fan, 2616	12,540 m^3/h	TP = 25,000 Pa, 980 rpm (motor & fan)	380	132
Fan, 2621	500,000 m^3/h	TP = 2000 Pa, working T = 250 ^{0}C, Cooling water, 2.5 m^3/h	690	400
Belt conveyer, 2803	500 tph, clinker	Lifting height = 35.4 m, 1.25 m/s, 188.38 m		110

As it is depicted in the table above; mills, crushers, fans, pumps and compressor are the main electrical energy consuming units in the cement factories and they also operate continuously unlike that of belts. Therefore, there is a great opportunity for energy optimization for these units.

5.1 Fans

Almost all fans used in the cement factories are centrifugal types because they are capable of generating relatively high pressures. Moreover, they are frequently used in dirty airstreams (high moisture and particulate content), in material handling applications and in systems operated at higher temperatures.

Centrifugal fans use a rotating impeller to move air first radially outwards by centrifugal action, and then tangentially away from the blade tips. Incoming air moves parallel to the impeller hub and it turns radially outwards towards the perimeter of the impeller and blade tips. As the air moves from the impeller hub to the blade tips, it gains kinetic energy. This kinetic energy is then converted to a static pressure increase as the air slows before entering the tangential discharge path.

Almost all fans regulate the air flow through them by speed control technique rather than damper system. By varying the speed of the fans and keeping the damper fully open, the volume of the air will be controlled. The speed of the fan varies by varying the speed of motor using variable frequency drive (VFD).

5.1.1 Fan efficiency

Fan efficiency is defined as the ratio of power transferred to the airstream to the power delivered to the shaft of the fan. The power of the airflow stream is the product of the pressure and the flow, corrected for consistency of units. The fan efficiency can be expressed in terms of total pressure (Total Efficiency) or in terms of static pressure (Static Efficiency).

Total Efficiency is the ratio of power of the airflow stream (using total pressure) divided by power delivered to the fan shaft in consistent units.

$$\text{Total efficiency} = \frac{total\ pressure * air\ flow}{6362 * \text{bhp}} \text{ ---------------------------------- (5.1)}$$

Where:

- Total Pressure is in inches of water (in. WG)
- Airflow is in cubic feet per minute (cfm)
- bhp is brake horsepower
- 6,362 is the unit consistency factor
- Brake horse power (bhp) - the total power deliver to the fan from the motor.

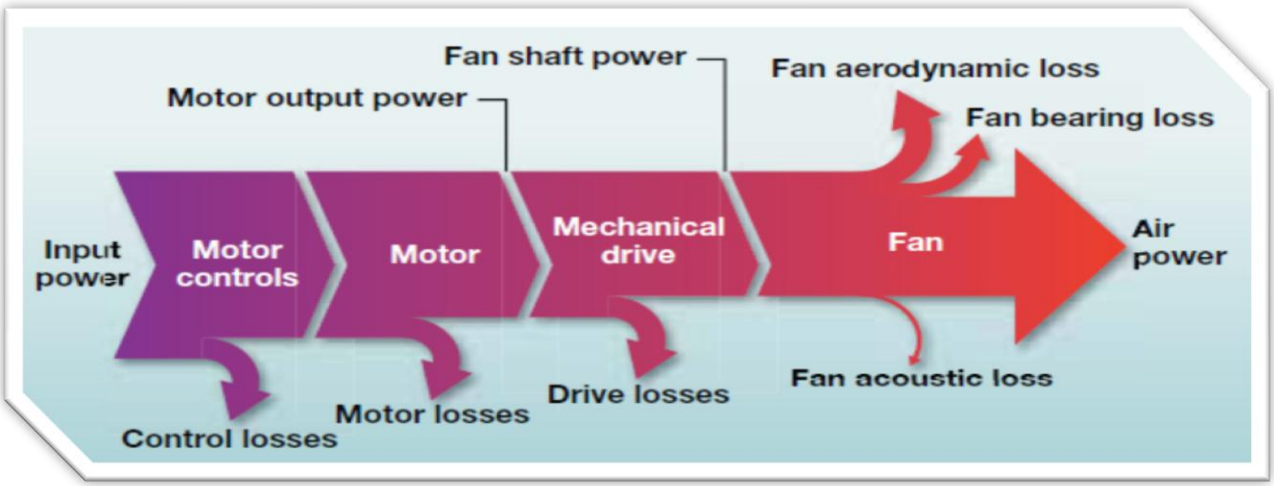

Fig. 5.1, Fan energy flows from left to right through a typical fan system

A. Efficiency of cooler fans

Motor output power (bhp) = $\sqrt{3}$VI Cos Ø η_m -------------------------------------- (5.2)

Where

- V: - measured voltage, volts
- I : - measured amperage, Ampere
- CosØ – power factor of the motor
- η_m - efficiency of the motor

However, the voltage drops of the fans are not known, thus, other alternatives has to be considered. Fan law can be used best alternative.

$$\text{Operating Power} = \text{Power designed} * \left(\frac{oper.rpm}{rpm\ designed}\right)^3 \text{ ---------- (5.3)}$$

The above fan efficiency equation can be transformed into SI unit forms & the equation can be reorganized as

$$\text{Total efficiency} = \frac{0.00089 * total\ pressure\ (mbar) * air\ flow(\frac{m^3}{h})}{\mathrm{P(kW)}}$$

The air flow rate is given in table 3.10.

A. cooler inlet air fans

i. Cooler fan 2604

$$P_{2604} = 185 * \left(\frac{0.829 * \text{rpm designed}}{\text{rpm designed}}\right)^3 = 105.4 \text{ kW}$$

$$\eta_{2604} = \frac{0.00089 * 19980 * 77}{105.4} = 12.98\%$$

ii. Cooler fan 2605

$$P_{2605} = 185 * \left(\frac{0.666 * \text{rpm designed}}{\text{rpm designed}}\right)^3 = 54.65 \text{ kW}$$

$$\eta_{2605} = \frac{0.00089 * 18324 * 50.6}{54.65} = 15.1\%$$

iii. Cooler fan 2606

$$P_{2606} = 185 * \left(\frac{0.84 * rpm\ designed}{rpm\ designed}\right)^3 = 109.65 \text{ kW}$$

$$\eta_{2606} = \frac{0.00089 * 15231 * 105}{109.65} = 12.98\%$$

iv. Cooler fan 2607

$$P_{2607} = 132 * \left(\frac{0.839 * \text{rpm designed}}{\text{rpm designed}}\right)^3 = 78 \text{ kW}$$

$$\eta_{2607} = \frac{0.00089 * 17420 * 80.07}{78} = 16.72\%$$

v. Cooler fan 2608

$$P_{2608} = 160 * \left(\frac{0.876*\text{rpm designed}}{\text{rpm designed}}\right)^3 = 107.55 \text{ kW}$$

$$\eta_{2608} = \frac{0.00089 * 29150*80}{107.55} = 19.3\%$$

vi. Cooler fan 2609

$$P_{2609} = 220 * \left(\frac{0.821*\text{rpm designed}}{\text{rpm designed}}\right)^3 = 121.74 \text{ kW}$$

$$\eta_{2609} = \frac{0.00089 * 40182*82.2}{121.74} = 24.15\%$$

vii. Cooler fan 2610

$$P_{2610} = 185 * \left(\frac{0.83*\text{rpm designed}}{\text{rpm designed}}\right)^3 = 104.78 \text{ kW}$$

$$\eta_{2610} = \frac{0.00089 * 32544*82}{104.78} = 22.45\%$$

viii. Cooler fan 2611

$$P_{2611} = 185 * \left(\frac{0.826*\text{rpm designed}}{\text{rpm designed}}\right)^3 = 104.26 \text{ kW}$$

$$\eta_{2611} = \frac{0.00089 * 31098*92.52}{104.26} = 24.56\%$$

ix. Cooler fan 2612

$$P_{2612} = 185 * \left(\frac{0.912*\text{rpm designed}}{\text{rpm designed}}\right)^3 = 140.33 \text{ kW}$$

$$\eta_{2612} = \frac{0.00089 * 45117*79.7}{140.33} = 22.8\%$$

x. Cooler fan 2613

$$P_{2613} = 160 * \left(\frac{0.887*\text{rpm designed}}{\text{rpm designed}}\right)^3 = 111.66 \text{ kW}$$

$$\eta_{2613} = \frac{0.00089 * 40172 * 74.7}{111.66} = 23.92\%$$

xi. Cooler fan 2614

$$P_{2614} = 185 * \left(\frac{0.912 * \text{rpm designed}}{\text{rpm designed}}\right)^3 = 140.33 \text{ kW}$$

$$\eta_{2614} = \frac{0.00089 * 45117 * 79.7}{140.33} = 22.8\%$$

Table 5.2, Parameters for main fans

Fan items	% motor rpm	Volume flow rate (Nm^3/h)	Altitude, m
Cooler exhaust fan	20	62849.5	2469
ID fan	79	185866	2469
Waste gas fan	72	223992.4	2469
Circulation fan	75	225467	2469

B. Cooler exhaust gas fan, 2621

The total pressure is not known, to tackle this problem, fan law is the available option.

$$\text{Differential pressure (head)} = \text{design differential pressure} * \left(\frac{\text{actual rpm}}{\text{design rpm}}\right)^2 \text{-------} \quad (5.4)$$

$$= (20 \text{ mbar}) * \left(\frac{0.2 * \text{design rpm}}{\text{design rpm}}\right)^2 = 0.8 \text{ mbar}$$

To convert volume flow rate at standard pressure and temperature into actual flow rate, it requires knowing the operating pressure. The inlet pressure& temperature of the gases at the cooler exhaust fan is -350 Pa & 250 ℃ respectively and the cooler is sited at an altitude of 2468 m from sea level. At this altitude, the atmospheric pressure is 74,966 Pa as determined above. Subsequently; the operating inlet gas pressure becomes 74,616 Pa which is equivalent to 0.746 bars.

$$V\ (m^3/h) = \frac{V\ (Nm^3)*(273.15+T(℃))*(1.0132\ bar)}{273.15 * P\ (bar)} \text{ --------------------- } \mathbf{(5.5)}$$

$$= 163487\ m^3/h$$

$$P_{2621} = 690 * \left(\frac{0.2*\text{rpm designed}}{\text{rpm designed}}\right)^3 = 5.52\ kW$$

$$\eta_{2621} = \frac{0.00089 * 163{,}487*0.8}{5.52} = 21.08\%$$

C. ID fan, 2201

$$\text{Differential pressure (head)} = (78\ mbar)*\left(\frac{0.2*\ design\ rpm}{design\ rpm}\right)^2 = 48.7\ mbar$$

The inlet pressure& temperature of the gases at the ID fan is -500 Pa & 290 ℃ respectively and the fan is sited at an altitude of 2469 m from sea level. At this altitude, the atmospheric pressure is 74,966 Pa as determined above. Consequently; the operating inlet gas pressure becomes 74,116 Pa which is equivalent to 0.741 bars.

$$V\ (m^3/h) = \frac{V\ (Nm^3)*(273.15+T(℃))*(1.0132\ bar)}{273.15 * P\ (bar)} = 523{,}962\ m^3/h$$

$$P_{2201} = 2400 * \left(\frac{0.79*\text{rpm designed}}{\text{rpm designed}}\right)^3 = 1183.3\ kW$$

$$\eta_{2201} = \frac{0.00089 * 523{,}962*48.7}{1183.3} = 19.2\%$$

D. Waste gas fan, 2611

$$V\ (m^3/h) = \frac{V\ (Nm^3)*(273.15+T(℃))*(1.0132\ bar)}{273.15 * P\ (bar)} = 628{,}754.4\ m^3/h$$

$$P_{2611} = 1400 * \left(\frac{0.72*\text{rpm designed}}{\text{rpm designed}}\right)^3 = 522.55\ kW$$

$$\eta_{2611} = \frac{0.00089 * 628{,}754.4 *19.7}{522.55} = 19.2\%$$

E. Circulation fan, 2632

$$V\ (m^3/h) = \frac{V\ (Nm^3) * (273.15 + T(°C)) * (1.0132\ bar)}{273.15 * P\ (bar)} = 433{,}026.5\ m^3/h$$

$$P_{2632} = 2400 * \left(\frac{0.75 * rpm\ designed}{rpm\ designed}\right)^3 = 1012.5\ kW$$

$$\eta_{2632} = \frac{0.00089 * 433{,}026.5 * 54}{1012.5} = 20.55\%$$

5.2 Vertical roller mill

The motor drives the grinding table through decelerator. The materials fall down the center of grinding table from feed opening. At the same time, hot air comes into the mill from the air inlet. Due to the centrifugal force, materials move to the edge of the grinding table.

The materials are pulverized by the roller when bypass of the groove on the grinding table. The crushed materials are brought up by vane high-speed airstream, the larger particles fall down to the grinding table for regrinding. The fine powder comes out with the airstream, and is gathered by the dust catcher. The materials content with moisture will be dried when they meet the hot airstream. Through adjusting the temperature of the hot airstream, it can meet different material's requirement, and also through adjusting separator, it can reach proper fineness of materials.

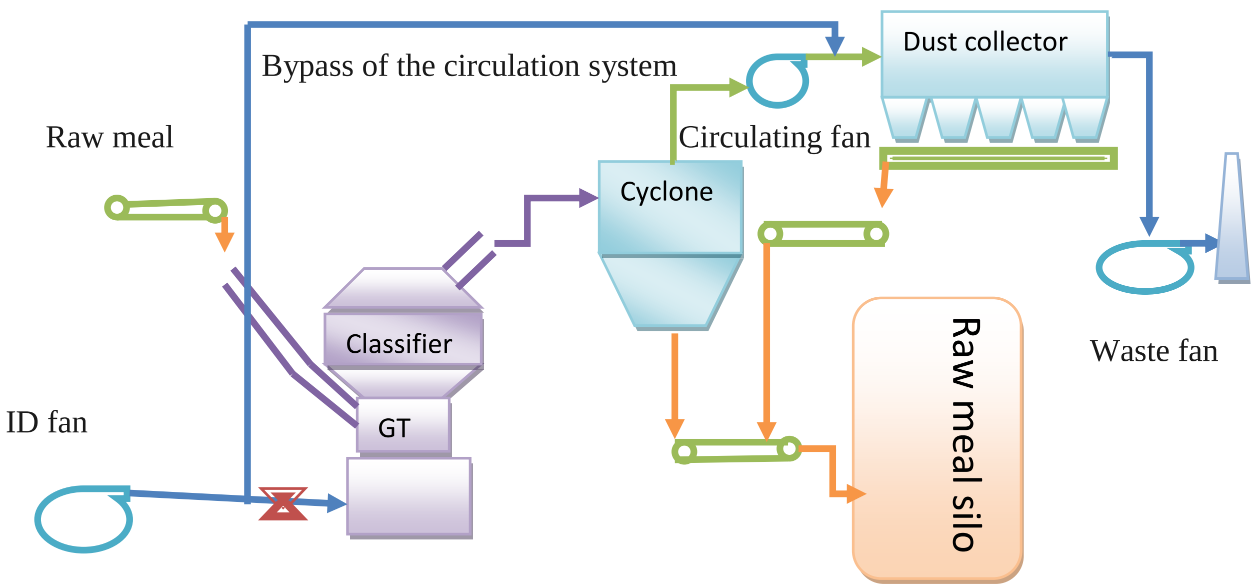

Fig. 5.2, Flow of material in vertical mill with two fans system of MCE

5.2.1 Efficiency of vertical roller mill (VRM)

It Influenced by; Grindability of Raw Material, Grinding Fineness, Classifier Design, Grinding Bed Height / Variations, Dam Ring Height, Air Flow, Temperature Level &Condition of Grinding Elements.

$$\text{Crushing Efficiency (\%)} = \frac{\text{surface energy created}}{\text{energy absorbed by material}} * 100 = \frac{\text{power output}}{\text{power input}} * 100$$

According to Bond, the work done by the mill is half squarely related with the size reduction, which can be stated as follows.

$$E = 10\ E_t \left(\frac{1}{\sqrt{XP}} - \frac{1}{\sqrt{XF}}\right) \text{---} (5.6)$$

Where

- ✓ E_t – is the bond work index or work required to reduce a unit weight from a theoretical infinite size to 80% passing 100 µm.
- ✓ X_F & X_P are the feed and product size in micron

For MCE, after the raw material grinding system operation is stable, the control processing parameter is as follows:

Mill inlet temperature:	222℃	feeding size: 95% ＜ 75 mm
Mill outlet temperature:	95℃	feeding material moisture: ≤ 7%
Pressure differences of raw mill inlet and outlet:		64 mbar
Raw meal size:	90 µm residue 12%	

Power delivered to shaft of motor (kW) = $\sqrt{3} VI\ \eta_m\ Cos\text{Ø}$

Where

V – Voltage of the mill drive, kV

I – amperage of the motor, A

η_m – efficiency of motor

CosØ – power factor of power supply

To determine the efficiency of motor, let use the design value of the parameters. The motor efficiency varies with operating load, winding temperature and others but it is not significant. The average value of the mill motor power factor is 0.92.

$$\text{Motor efficiency} = \frac{power}{\sqrt{3}\text{VI Cos}\text{Ø}} = 0.97$$

Table 5.3, Operating parameters of VRM

VRM motor	Current, A	Power factor	Voltage, kV	Motor efficiency	Power input, kW
Values	250	0.92	6	0.97	2318.5

The mill separator power consumption varies depending on the speed of the separator and the mill feed rate. However, usually it operates at a motor current of 150 Ampere and 380 volts. The power factor and motor efficiency are 0.92 and 0.95 respectively.

$$\text{Power delivered to shaft of motor (kW)} = \sqrt{3}\text{VI } \eta_m \text{ Cos}\text{Ø} = 86.28 \text{ kWh}$$

The specific power consumption of vertical mill system is determined by dividing the total power consumption of mill system (VRM, separator and the mill fan) by the grinding capacity of the mill.

$$\text{SPC of VRM system} = \frac{3417.28}{300} = 11.\ 39 \text{ kWh/tones of raw meal}$$

Or

The corresponding clinker amount becomes = 300/1.607 = 186.6 tones per hour

$$\text{SPC of VRM system} = \frac{3417.28}{186.6} = 18.31 \text{ kWh/tones of clinker}$$

5.3 Limestone crusher

After drilling, blasting and haulage, the limestone crushed in hammer crusher with a capacity of 900 tpd. Oversized blasted slices have to be reduced by other mechanical means before the entry of the crusher which can handle only a size below 1100 * 1100 *1850 mm. the hammer crusher system consists of hopper, apron feeder with its own motor, screw conveyor, hammer crusher with its own motor & screen.

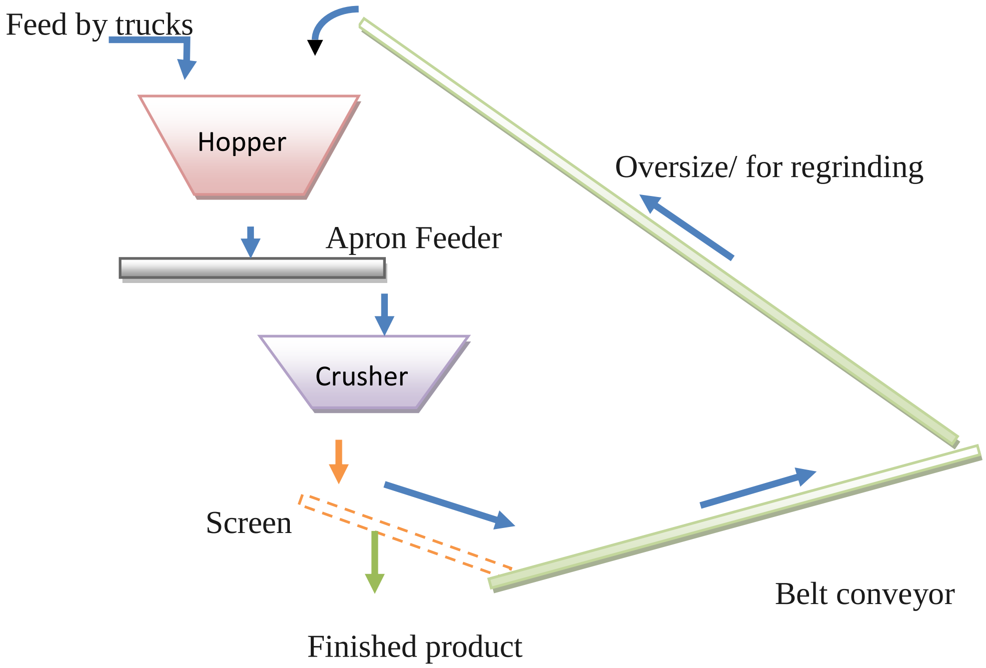

Fig. 4.3, Schematic view of closed circuit crusher system

5.3.1 Crushing efficiency of the hammer crusher

It is expected that 80 % of limestone feed to crusher can pass through 250 mm sieve size and 80% of crusher product can pass through 25 mm mesh size. The crusher usually operates at 95% of its full capacity.

Table 4.4, Working parameters of crusher

Crusher motor	Current, A	Power factor	Voltage, kV	Motor efficiency	Power input, kW
Values	85	0.9	9.5	0.97	1183

$$E = 10 * 11.6 * \left(\frac{1}{\sqrt{25000}} - \frac{1}{\sqrt{250000}}\right) = 0.5 \text{ kWh/t of raw meal}$$

Power consumed for crushing = 0.5 kWh/t of raw meal * 855 tones of limestone

= 429 kW

$$\text{Grinding efficiency of Crusher} = \frac{291}{1183} * 100 = 36.25\ \%$$

5.4 Finish milling system

In cement manufacturing plant, the finish milling system is comprised of four basic components namely; feeders, mill, elevator, and separator.

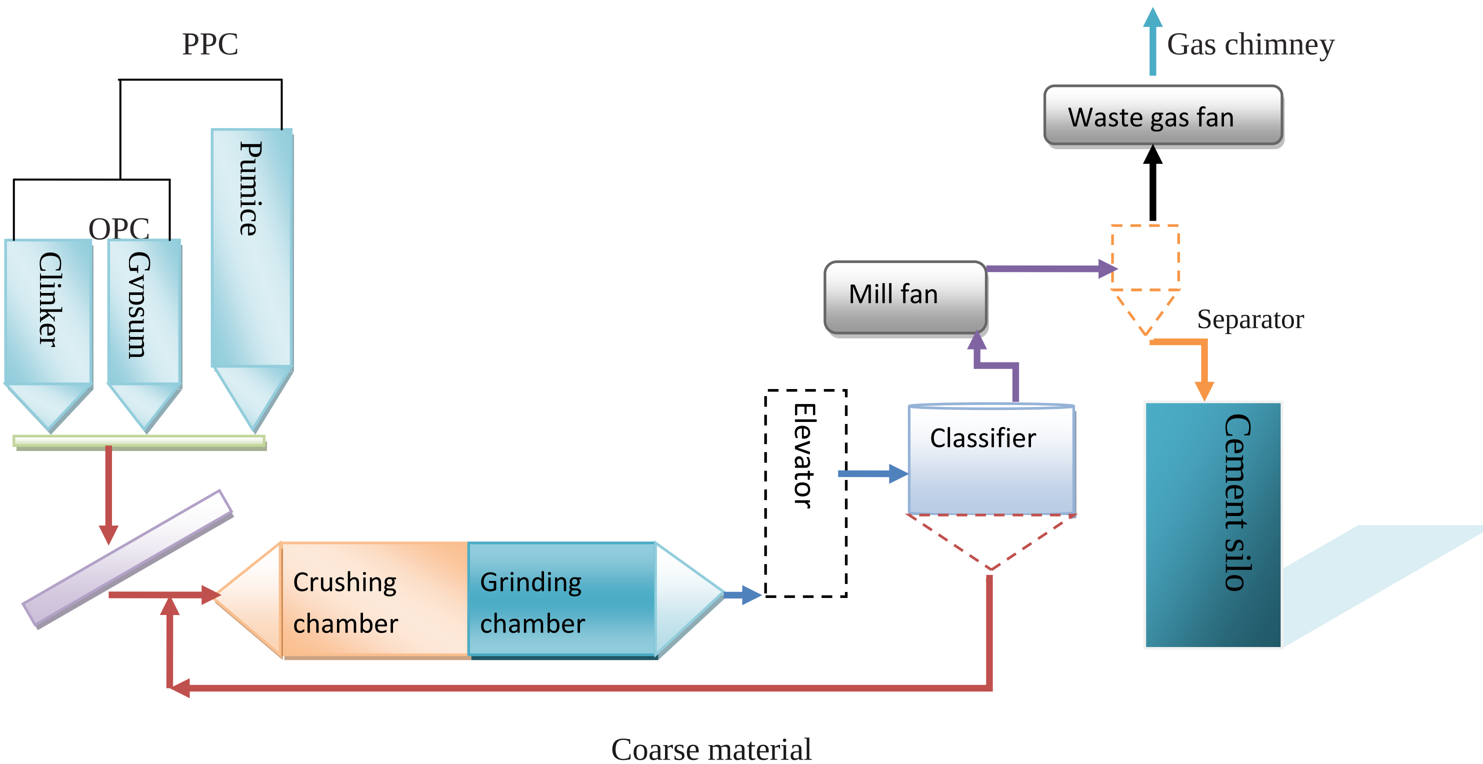

Fig. 5.4, Schematic of finish milling system in a cement plant

5.4.1 Efficiency of ball mill

As the depicted on the above figure, the milling system involves crushing and grinding subsequently in the two chambers. The energy consumption for the pre-crushing and ball milling can be estimated using the following Bond based model:

$$W = Wc + Wm \text{ --- } (5.7)$$

Where,

$$Wc = 10 * Wi \left(\frac{1}{\sqrt{(P_c}} - \frac{1}{\sqrt{(F_c}}\right) \;\&\; Wm = 10 * Wi \left(\frac{1}{\sqrt{(P_{80}}} - \frac{1}{\sqrt{(P_c}}\right) \text{ -------- } (5.8)$$

- Wc - is energy consumed for cement meal crushing in the 1st chamber (kWh/t)
- Wm - is energy consumed for cement meal grinding in the 2nd chamber (kWh/t)
- Wi - is Bond ball mill work index (kWh/t)

- P_c -is sieve size passing 80% of the clinker after crushing (μm),
- F_c - is sieve size passing 80% of the clinker before crushing (μm),
- P_{80} - is sieve size passing 80% of the cement after grinding in the grinding chamber (μm)

The MCE third line has two ball mills. However, due to power shortage only one of them works at a time. From sieve analysis, it is found that 80% of the cement product passes through 32 micron sieve size and the feed size is about 20 mm. The work index of clinker, pumice and gypsum is 13.5, 6 & 8.16 kWh/t respectively. The rated work index becomes 11.93 kWh per tones of feed.

Table 5.5, Working parameters of cement mill II

Parameters	Work index , kWh/t	Feed rate, t/h	Motor power kW	Grinding power consumption	Efficiency of the mill, %
Values	11.93	115	3570	2386	66.9

The MCE third line produces PPC cement with an average composition of 62.5 % clinker, 22.5 % pumice and 5 % gypsum. The average power factor is 0.95.

5.4.2 Fans efficiency

Table 5.6, Operational parameters of mill and exhaust fans

Devices	Flow rate, m^3/h	TP, pa	I, A	Voltage, V	PF	Motor power	Efficiency
Exhaust fan, 3433	117,000	2450.5	35	6000	0.87	284.8 kW	9
Mill fans, 3437	41,000	1350	285	380	0.92	155 kW	3.2

Note that: - the actual gas flow rate and total pressure are determined using fan laws.

Specific power consumption of ball mill system is obtained by dividing the total power consumption of mill system (ball mill, OSEPA separator and mill fan) by actual production capacity of the mill. The OSEPA separator motor power consumption is insignificant but it is high for its fan. The separator fan averagely operates at around 6 kV and 40 Ampere with power factor of 0.87 and a motor efficiency of 0.97. The corresponding power consumption becomes 360 kilowatts.

$$\text{SPC of Ball mill system} = \frac{2901}{115} = 25.22 \text{ kWh/tones of cement}$$

Or

The corresponding clinker amount becomes = 115/1.625 = 69.7 tones per hour

$$\text{SPC of ball mill system} = \frac{2901}{69.7} = 41.6 \text{ kWh/tones of clinker}$$

5.5 Compressed air systems

Cement plants use compressed air throughout their production operations, which is produced by compressed air units. The compressed air can be lost in the form of unusable heat, friction, misuse and noise. For this reason, compressors and compressed air systems are important areas to improve energy efficiency at cement plants. Compressed air systems consist of following major components: Intake air filters, inter-stage coolers, after-coolers, air-dryers, moisture drain traps, receivers, piping network, filters, regulators and lubricators.

A system of distribution pipes and regulators convey compressed air from the central compressor plant to process areas. This system includes various isolation valves, fluid traps, intermediate storage vessels, and even heat trace on pipes to prevent condensation or freezing in lines exposed to the outdoors. Pressure losses in distribution typically are compensated for by higher pressure at the compressor discharge.

For compressors that have load/unload controls, there is an easy way to estimate the amount of leakage in the system. This method involves starting the compressor when there are no demands on the system (when all the air -operated, end-use equipment is turned off). A number of measurements are taken to determine the average time it takes to load and unload the compressor.

Total leakage (percentage) can be calculated as follows:

$$\text{Leakage (\%)} = \frac{T * 100}{T+t} \text{-- (5.9)}$$

System leakage (m3/minute) = Q × leakage (%)

Where

T = on load time (minutes) and t off load time (minutes)

Q = the actual free air being supplied during trial (m^3/min)

Table 4.7, Actual operating parameters of compressors at Mugher site

Compressors	Rated power (kW)	Unloading pressure (bar)	Loading pressure (bar)	Minimum stoppage time (min)	Running (total) hours (hrs)	Loading hours (hrs)
3907	Standby					
3908	160	7.3	6.1	20	18376	16608
3909	160	7.3	6.1	20	18376	16608
3910	160	7.3	6.1	20	18376	16608
3901	250	7.5	6	20	8510	6819
3902	250	7.5	6	20	10,428	7848
3903	250	7.5	6	20	8620	7044

Compressors at Mugher Leakage (%) = $\frac{1768 * 100}{18376}$ = 9.62

Compressor (3901) leakage (%) = $\frac{1691 * 100}{8510}$ = 19.87

Compressor (3902) leakage (%) = $\frac{2580 * 100}{10428}$ = 24.47

Compressor (3903) leakage (%) = $\frac{1576 * 100}{8620}$ = 18.28

5.5.1 Compressor Efficiency

Several different measures of compressor efficiency are commonly used: volumetric efficiency, adiabatic efficiency, isothermal efficiency and mechanical efficiency.

$$\text{Isothermal Efficiency} = \frac{IsothermalPower * 100}{\text{Actual measured input power}} \text{---------------------------- (5.10)}$$

$$\text{Isothermal power (kW)} = \frac{P1 * Q * log\,(r)}{36.6} \quad \text{------------------------------------} \quad (5.11)$$

Where

P_1 = Absolute intake pressure kg/ cm^2 = the ratio of pressure in pa to gravitational acceleration

Q = Free air delivered m^3/h

r = Pressure ratio P_2/P_1

The calculation of isothermal power does not include power needed to overcome friction and generally gives an efficiency that is lower than adiabatic efficiency. The reported value of efficiency is normally the isothermal efficiency. This is an important consideration when selecting compressors based on reported values of efficiency.

MCE uses seven compressors all together and four of them are located at Mugher and the others are found at Tatek site. At Mugher, three of them work together while the other one remain as standby. Each of them provides in average 7.5 bars by sucking air from the atmosphere. Totally the three compressors provide 19 m3/h of air. On the other hand, at Tatek, site only one of the compressor works at a time and provides 41.7 m^3/h of air at a pressure of 7 bars.

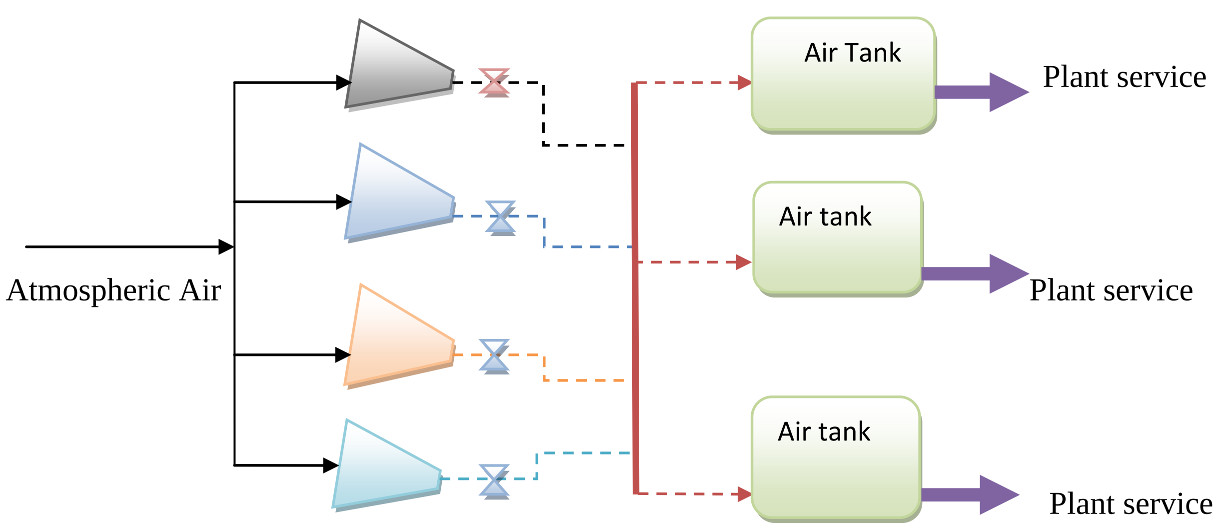

Fig. 5.5, Compressor station at Mugher site

Table 5.8, Performance assessment results of compressor system

Measurement	Flow rate (Q)	Suction pressure P_S, $(\frac{kg}{cm^2})$	Discharge pressure (PD)	Ratio, r	Power	η_P
Compressor at Mugher	1146 $\frac{m^3}{h}$	0.74	7.5 bar	10.13	140 kW	16.5%
Compressor at Tatek	2502 $\frac{m^3}{h}$	0.74	7 bar	9.46	220 kW	22.5 %

5.6 Water pumping System

The transfer of liquids against gravity existed from time immemorial. A pump is one such device that expends energy to raise, transport, or compress liquids. In the cement factory, water pump drives water that used for cooling (motors, gear boxes, lubricating oils, exhaust gases before static electro precipitators, bag filters), grinding units (to stabilize material in the process of grinding), fire control and residence consumption.

If the pump is not doing its job properly, this can increase pumping costs and reduce productivity. To contain costs, it needs to monitor the energy usage regularly and repair and maintain the pump to operate efficiently. Therefore, the aim of careful pump selection and regular pump maintenance is to have the pump performing as efficiently as possible, because this gives the lowest running costs. Pump efficiency measures how well the pump converts electrical power to useful work moving the water.

$$\text{Pump efficiency} = \frac{\text{power output}}{\text{power input}} = \frac{\mathbf{Q}\left(\frac{\mathbf{m^3}}{\mathbf{h}}\right) * \mathbf{H}\left(\frac{\mathbf{kg}}{\mathbf{m^2}}\right) * \mathbf{g}\left(\frac{\mathbf{m}}{\mathbf{s^2}}\right)}{3600 * \text{consummed power (W)} * \eta_m}$$

MCE uses both ground water (Tatek) and river water (Mugher). At Mugher, two big pumps drive the water from the river to the substation (the three lines) and then it distributed into the corresponding plant according to the need. Then at the plant sites, there are twelve pumps that provide plant service (plant running), drinking and fire protection for the third line alone.

Six of them located at the main factory (Mugher) where as the remaining six are found at Tatek, cement milling and packing plant. In both sites, two pumps in parallel connection are used to provide each of the above functions. In addition, at Tatek, there are two pumps that used to bring the water from the ground water location to plant site. However, in both sites fire protection pumps work only occasionally. So, the energy consumption of water pumping from well, service and drinking water pumps is considered. Fig. 5.6 plainly shows this water flow system.

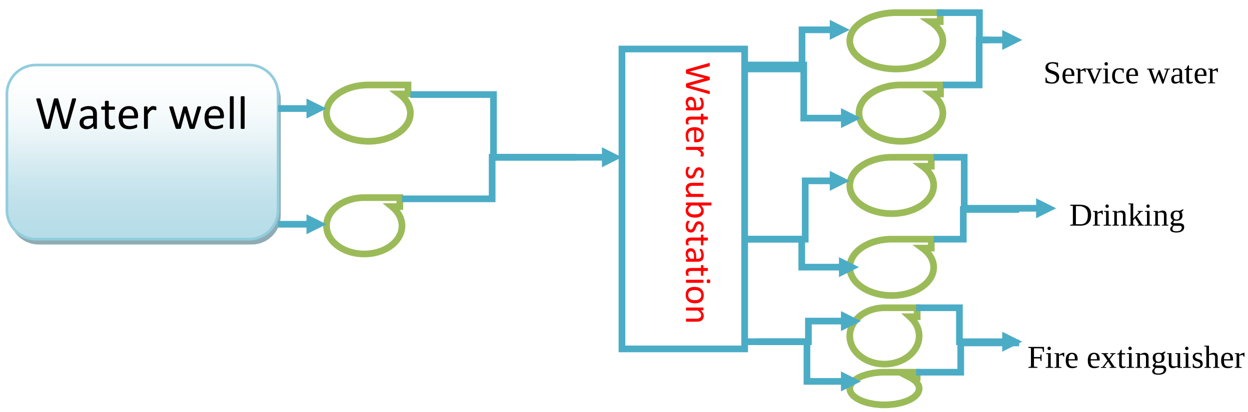

Fig. 5.6, Schematic diagram of water pumping system at Tatek site

Table 5.9, Operating parameters for water pumping system

Measurement	Flow rate (Q)	Suction pressure P_S, $(\frac{kg}{cm^2})$	Discharge pressure (PD)	$H = P_D - P_s$	Power	η_P
Service water pump at Mugher	$190\frac{m^3}{h}$	$0\ \frac{kg}{cm^2}$	$7.5\ \frac{kg}{cm^2}$	$7.5\ \frac{kg}{cm^2}$	62 kW	68 %
Suction pump for well water at Tatek	$453.5\frac{m^3}{h}$	$0\ \frac{N}{m^2}$	$256{,}000\ \frac{N}{m^2}$	$256{,}000\ \frac{N}{m^2}$	78.5 kW	40.5 %
Service water pump at Tatek	$440\frac{m^3}{h}$	$0\ \frac{N}{m^2}$	$250{,}000\ \frac{N}{m^2}$	$250{,}000\ \frac{N}{m^2}$	70.5 kW	43.5 %

Note: - The actual power consumption of the pumps is calculated using operational current and voltage of the pumps. The power consumption of the drinking water is not considered since it is insignificant (doesn't work continuously). Moreover, the pump deriving water from the river at the Mugher site hasn't been considered since it is used commonly with the other two lines.

5.7 Pumice drying system

The pumice coming from Debre Zeit has a moisture content varying from 8 to 20 % depending on the season. MCE utilizes rotary dryer which works with diesel fuel. Before it has to be grounded in the ball mill with other cement materials, the moisture has to be dried about less than five percent, according to the design of the ball mill.

However, from the factory past experience, no need of drying if the coming pumice moisture content is less than ten percent. So, in this situation the dryer doesn't work. It is found that the drier consumes 13.6 liters per ton of clinker averagely.

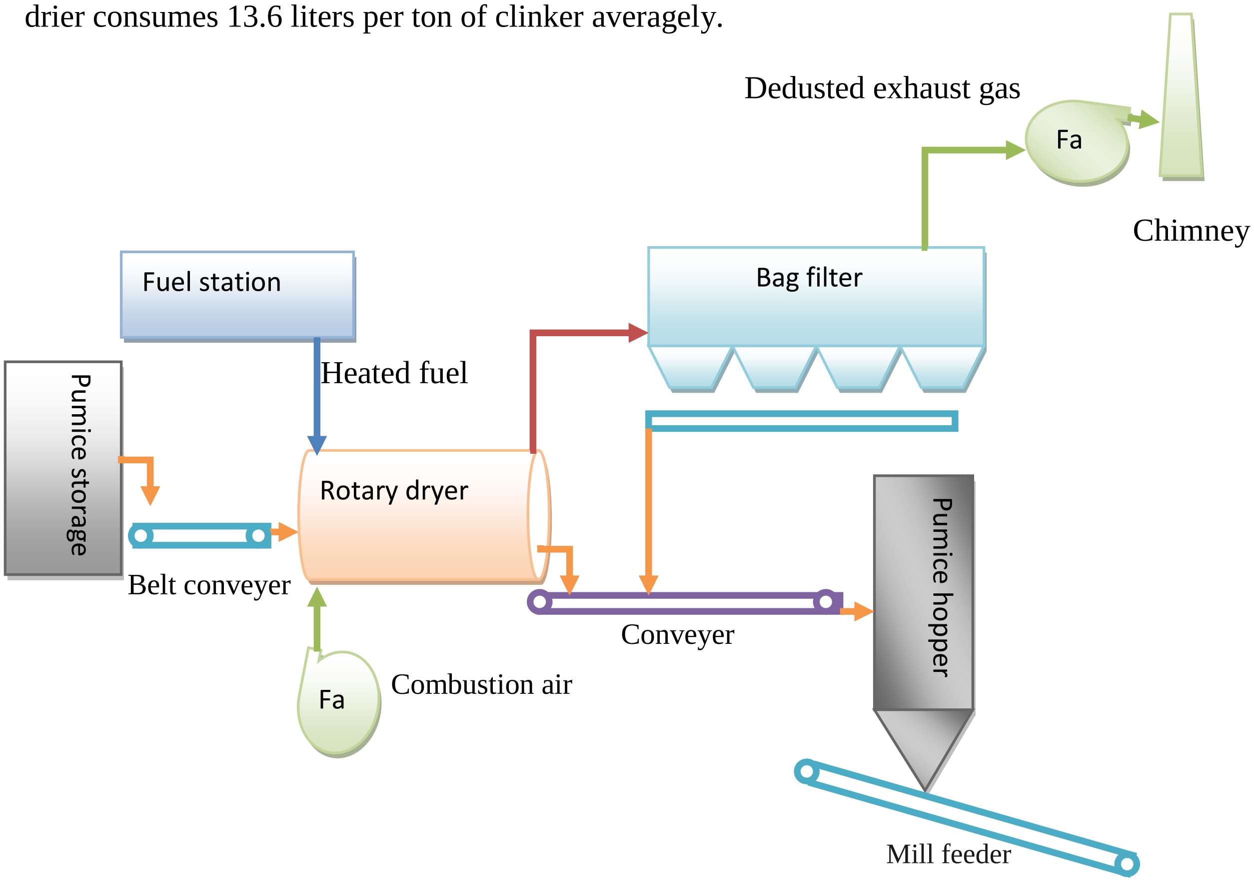

Fig. 4.7, Schematic diagram of Tatek pumice drying system

5.7.1 Pumice dryer energy efficiency

To know the specific power consumption of the dryer, the pumice flow rate has to be known. However, at Tatek the pumice flow meter is not working. The operators manually adjust the valve opening by observing the effect of the moisture content on the ball mill operation.

6. Kiln System Operation and clinker Quality Evaluation

6.1 Kiln System Operation

A cement plant will continue to operate successfully only if it can produce quality product with economic and material efficiency, and meet or exceed the required product performance. To a great extent, operation of the rotary kiln system also determines the quality of the cement produced. Stable operation of the kiln system will result in improved operational and energy efficiency, higher production rates, and better quality clinker. It can also result in reduced environmental emissions and energy consumption a factor of ever-increasing importance.

If it were possible to put exactly the same quantity of kiln feed with a perfectly controlled chemistry, exactly the same amount of fuel with the same heating value, fineness, and ash chemistry, and have perfectly uniform clinker cooler operation, it would be possible to find an operating point where the pyroprocessing system would operate smoothly, with no changes, forever. However, in reality this is not practical. The primary parameters that must be controlled as carefully as possible are:

- Kiln feed chemical composition
- Kiln feed chemical uniformity
- Kiln feed fineness
- Kiln feed rate
- Fuel heating value
- Fuel feed rate
- Clinker cooler operation

6.1.1 Kiln Feed Chemical Composition

The primary goal when designing kiln feed chemistry must be to produce a clinker that when ground with appropriate additives produces a marketable product. Additionally, the kiln feed must have characteristics that will allow the cement plant operator to produce clinker economically.

Kiln feed chemical composition has a large effect on kiln operations and energy consumption. Several of the more important parameters are silica ratio, percent liquid, and C_3S content (or lime saturation factor, LSF)

Table 6.1, Chemical composition of clinker

Chemical composition of clinker	Sample one	Sample two	Sample three	Sample four
Free lime	1.68	0.84	1.12	0.7
CaO	66.23	66.54	66.26	65.67
SiO_2	22.34	22.06	21.84	23
Al_2O_3	5.06	5.05	5.06	4.98
Fe_2O_3	3.32	3.61	3.51	3.42
MgO	1.01	1.01	1.01	1.01
SO_3	1.15	1.29	1.15	1.12
Na_2O	0.12	0.12	0.12	0.12
K_2O	0.42	0.44	0.43	0.42
Cl	0.01	0.02	0.02	0.01

Using data on the table 4.11 and 6.1, the following calculation is made.

- Alumina modulus, AM

The Alumina ratio establishes the relation between alumina and iron to determine the viscosity of liquid phase.

$$AR = \frac{Al2O3}{Fe2O3} \text{ -- } (6.1)$$

$$= 1.45$$

- Silica ratio/ modulus

It is a measurement of the ratio of silica to that of the sum of aluminum and iron oxides. Thus, it is inversely proportional to the liquid phase at burning zone temperatures.

$$SM = \frac{SiO2}{Al2O3+Fe2O3} \text{ --- } (6.2)$$

$$= 2.57$$

- Percentage of liquid phase, Lea and Parker Equation

$$\%\text{ liquid phase} = 3\,Al_2O_3 + 2.25\,Fe_2O_3 + MgO + K_2O + Na_2O \text{ ---- } (6.3)$$

$$= 25.6\%$$

- Lime saturation factor, LSF

Lime Saturation Factor (LSF) – A measurement of the degree of conversion of silica, alumina, and iron oxide to their most highly basic calcium compounds.

$$LSF = \frac{100CaO}{2.8SiO2+1.65Al2O3+0.35FeO3} \text{-------------------------} (6.4)$$

$$= 93.6$$

6.1.2 Kiln feed Fineness

The average residue on the 90 micron is 12.725 %

6.1.3 Kiln feed rate

During start up, kiln feed rate increases with the kiln revolution till the full load rpm is achieved. Then after, both kiln revolution and the kiln feed remains constant.

6.1.4 Cooler operation

Mugher has installed eta grate cooler, which is the latest cooler technology in the cement industry. It is modular types with static horizontal plates (obtain clinker from the kiln) and movable parallel plates (horizontal lanes) which reciprocates back and forth.

The front static part of the cooler has inclination (15^0) that enables material down flow and the reciprocating dynamic part receives from the static plate carry it to cooler crusher. The plates are pocketed for aeration and they are high pressure drop plates to enable uniform air distribution and to avoid the formation of red river. The in front static plates have mechanical flow regulator (MFR) to allow automatic air flow depending on the clinker load. The material transport is controlled by a combination of the stroke speed of the section of grate after the static incline and the air velocity on the static section.

The cooler internal is usually dusty and it involves high clinker outlet temperature. Moreover, there is reoccurrence of red river and dusty clinker. The automatic control system controls the first grate by the undergrate pressure and keeps the speed of the following grates proportional to the speed of the first grate. The air flow rate controlled automatically depending on the thickness of the clinker bed and flow resistance. The hood pressure is maintained around zero (slightly negative) by balancing gas flow rate (pressure) through the ID fan and Waste gas fan manually.

6.2 Clinker Quality Evaluation

The clinker phases, burnability index, burnability factors and the likes are substantial factors in determining the clinker quality. Alite, belite, ferrite and aluminate are the four phases of clinker which can be physically identified using microscope both by their structures and colors. The quantitative phase composition of clinker is widely estimated using a procedure due to Bogue.

Alite, $C_3S = 4.0710\ CaO - 7.6024\ SiO_2 - 6.7187\ Al_2O_3 - 1.4297\ Fe_2O_3$ -- (6.5)

= 53.13%

Belite, $C_2S = 2.8675\ SiO_2 - 0.7544\ C_3S$ ------------------------------------ (6.6)

= 23.85 %

Aluminate, $C_3A = 2.6504\ A_{l2}O_3 - 1.6920\ Fe_2O_3$--------------------------- (6.7)

= 7.41 %

Ferrite, $C_4AF = 3.0432\ Fe_2O_3$ --- (6.8)

= 10.485 %

Others parameters that must be considered in determining cement quality include the following.

Burnability index (BI) = $\frac{C3S}{C4AF+C3A}$ --- (6.9)

= 2.55

Burnability factor (BF) = LSF + 10 SR – 3 (MgO + alkalies)

= 114.9

Coating Index (CI) = $C_3A + C_4AF + 0.2C_2S + 2Fe_2O_3$

= 29.96

The insoluble residue is 0.45

Table 6.2, Free lime content variation with liter weight

Liter weight (g/L)	1263	1330	1335	1313	1320	1312	1300	1311
Free lime (CaO %)	0.84	1.05	1.65	1.12	0.7	0.84	0.7	0.75

7. Result and discussion

7.1 Thermal energy and Chemical analysis

The thermal energy evaluation depicts that the factory has great energy losses and poor energy utilization efficiency. As it can be seen on table 4.21, about 31 % of the heat is lost to surrounding which leaves the kiln system with clinker, cooler waste gases and preheater exhaust gases.

Mugher has installed eta grate cooler, which is the latest cooler technology in the cement industry. It is modular types with static horizontal plates (obtain clinker from the kiln) and movable parallel plates (horizontal lanes) which reciprocates back and forth.

The thermal efficiency of kiln system is exceedingly dependent on the recuperation efficiency of the cooler. The cooler has dual advantage. Firstly, it calms the clinker for stable operation of down customers. At the same time, it warms up the secondary and tertiary air that used up in the kiln and precalciner respectively. The advanced the cooler, it will have the higher heat recuperation efficiency, which mean it has done both well together. The recuperation efficiency of the cooler can be calculated using:

$$\text{Efficiency (\%)} = \frac{\textit{heat of secondary and tertiary air}}{\textit{heat of hot clinker}} * 100\% \text{ ------------------ (7.1)}$$

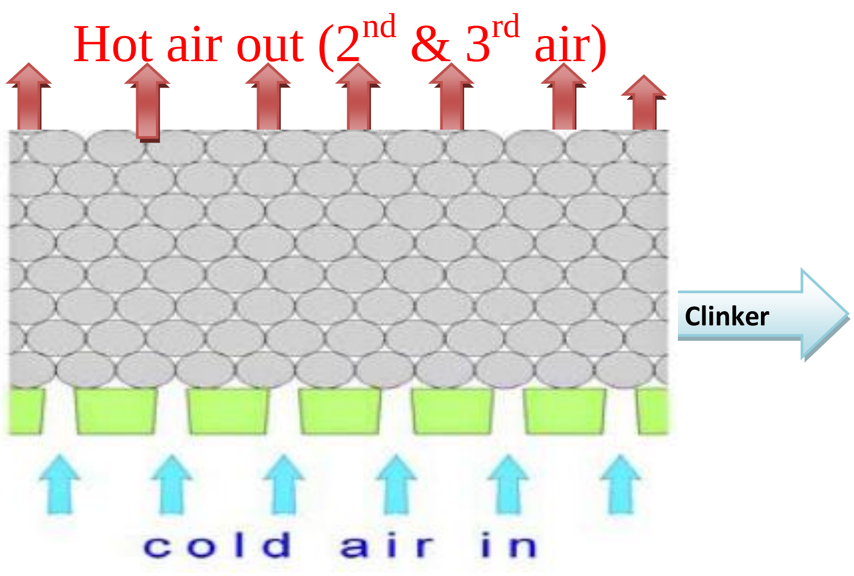

Fig. 7.1 Material movement in cooler

On a cooler, reliable measurement of secondary and tertiary air heat is virtually impossible. Therefore, this value is determined by balance calculation. However, for estimation it is possible to take the kiln hood temperature as the secondary and tertiary temperature.

Assuming, all heat from clinker are recovered to secondary and tertiary air except heat losses with clinker exit, waste gases and surface radiation and convection losses, the following calculation can be valid.

Heat recuperated = heat in clinker at kiln exit – (Heat in clinker exhaust air + Heat in clinker leaving cooler + heat losses from cooler surface) --- (7.2)

= 96,258,548 kJ/h

The heat carried by the clinker when enter the cooler can also be calculated in the same way. The clinker leaves the kiln hood around 1350 ^{0}C. At this temperature the specific heat of clinker is 1.08 kJ/kg K.

Heat in clinker = mass flow rate * specific heat * $(T – T_0)$

= 108,000 kg/hr * 1.08 kJ/kgK * (1350 – 25) K = 154,548,000 kJ/h

Then, the recuperation efficiency becomes 60%

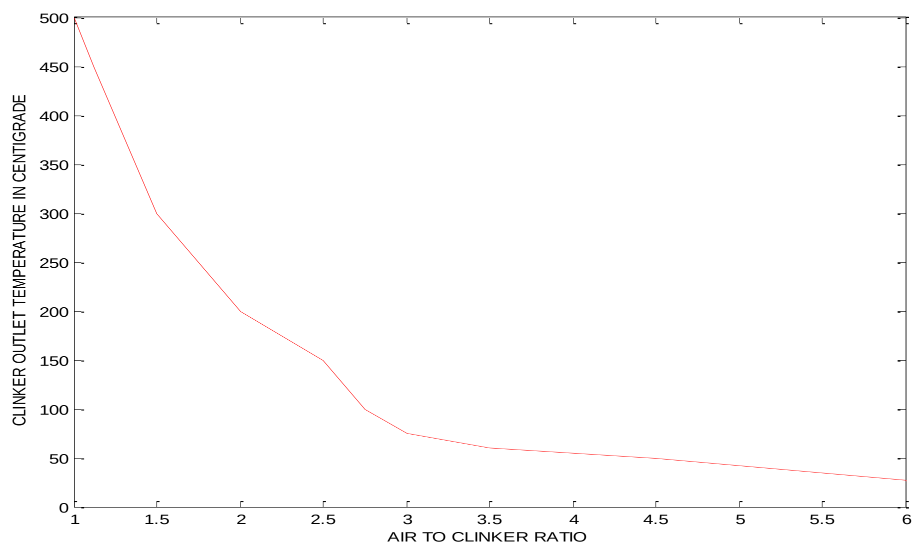

Fig. 7.1 Clinker Temperature variation with Air to Clinker ratio

The eta grate cooler at Mugher operates with specific air consumption of 2.31 kg of air per kg of clinker production.

As figure 7.1 depicts, for 2.31 kg of air per kg of clinker, the clinker outlet temperature is expected to be reduced to around 160 ^{0}C.

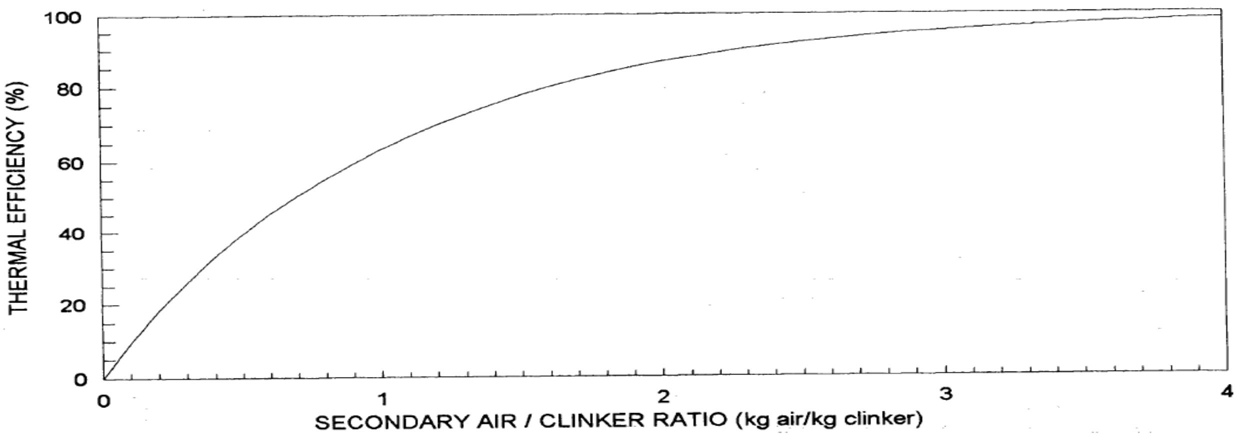

Fig. 7.2, Theoretical performance of grate cooler

Theoretically using 2.31 kg of air per kg of clinker, it is anticipated to have a heat recuperation efficiency of 90%.

The practical experiences of the contemporary coolers which are similar to Mugher confirms as they operate with specific air consumption of 2 kg / kg of clinker and below. Moreover, the design temperature of clinker discharge for Mugher cooler is around 90 ^{0}C though it is operating around 310 ^{0}C.

The low recuperation of heat and high specific air consumption of the cooler indicates that it is not proficient. Then, the subject of dealing on the causes of poor efficiency is critical. Therefore in the subsequent part, the possible causes for inefficiency will be scrutinized and the potential solution will be identified.

At Mugher, the cooler air flow rate and grate speeds are controlled automatically whereas the hood pressure is controlled manually by manipulating the ID fan air flow rate and the cooler waste gas flowrate. However, setting of the automatic air flow meter is elevated which causes more air intake.

As it has been revealed in the chapter four, Mugher operates at around 24 % excess air. The recommended excess air in the kiln system is around 10 % to enable complete combustion and to avoid heat loss with unburned carbon monoxide in the smoke.

At the same time it avoids reducing conditions in the kiln system. However, extra excess air leads heat loss from flame and burning zone to heat up cold air. In addition, it increases ID fan load, by increasing the volume of exhaust gases which in turn intensifies the electrical energy consumption and shorten the life of the fan.

There could be many causes for poor efficiency of the cooler. One of these limitations is design deficiency. Modern coolers have auxiliary diesel driven hydraulic system that drives the cooler lanes when power interruption occurs. Mugher hasn't installed this system. This is a catastrophic phenomenon. Currently a continuous power interruption is happening. During power interruption, the cooler lanes are stopped carrying the hot and waxy clinker. Then it binds together and forms massive blockage as it is cooled and time passes. If the duration of the power interruption is prolonged, the effect becomes intensified. This is the main problem at Mugher; the clinker itself is corrosive that causes lane plate wear.

During blockage removal, digging causes the lanes further to be worn and lanes edge gap increases that assist air inleak from undergrate. This in turn leads clinker segregation, dust entrainment and red river. In addition, it increases the down time of the factory.

MCE raw mix design gives low heat of formation raw meal. At it determined above, the heat of reaction becomes 369 k Cal/kg of clinker. The heat of reaction will normally be in the range of 385 – 410 kCal/kg clinker. This reduction of heat of formation is due to high fluxing agent content in the raw mix design. Low value of LSF proves this condition, for improved clinker quality (less C_2S & high C_3S) it is good to have clinker with LSF of 96 -98 %. Besides, this low LSF value causes the formation of less porous and bally clinker which is hard to grind.

MCE raw mix design relatively has high silica modulus. Averagely, the silica modulus goes up to 2.6 which is far from what it has to be, 2.0 – 2.4. This makes it hard to burn. As a result, it increases fuel consumption and heat losses by radiation. Moreover, it reduces the amount of coating in the burning zone & Produces dusty clinker.

The relationship between the silica ratio and saturation factor indicates that the mix design is appropriate in the perspective of burnability. Figure 7.3, depicts this clearly.

The high heat of reaction due to increased silica modulus is compensated by low heat of formation due to decreased lime saturation factor.

In addition, from the same figure; it is possible to conclude that the clinker composition is in the acceptable region from coating formation viewpoint.

The correlation between liter weight and the free lime (table 6.2) point out as there is a high opportunity to reduce burning of clinker. Because of the ease of burnability of raw mix of raw meal, the free lime content of the clinker is very much lower than the limitation, less than 1.5 %. Therefore, there is an opportunity of saving energy by reducing the retention time of the material in the burning zone or reducing the flame temperature as far as the liter weight is high enough to obtain the intended strength and free lime content is low.

As it is depicted on table 7.1, the clinker is optimized both in quality and energy perspectives except some room for further improvement. The opportunity for improvement arose due to low LSF that impended C_2S combination with lime to form C_3S.

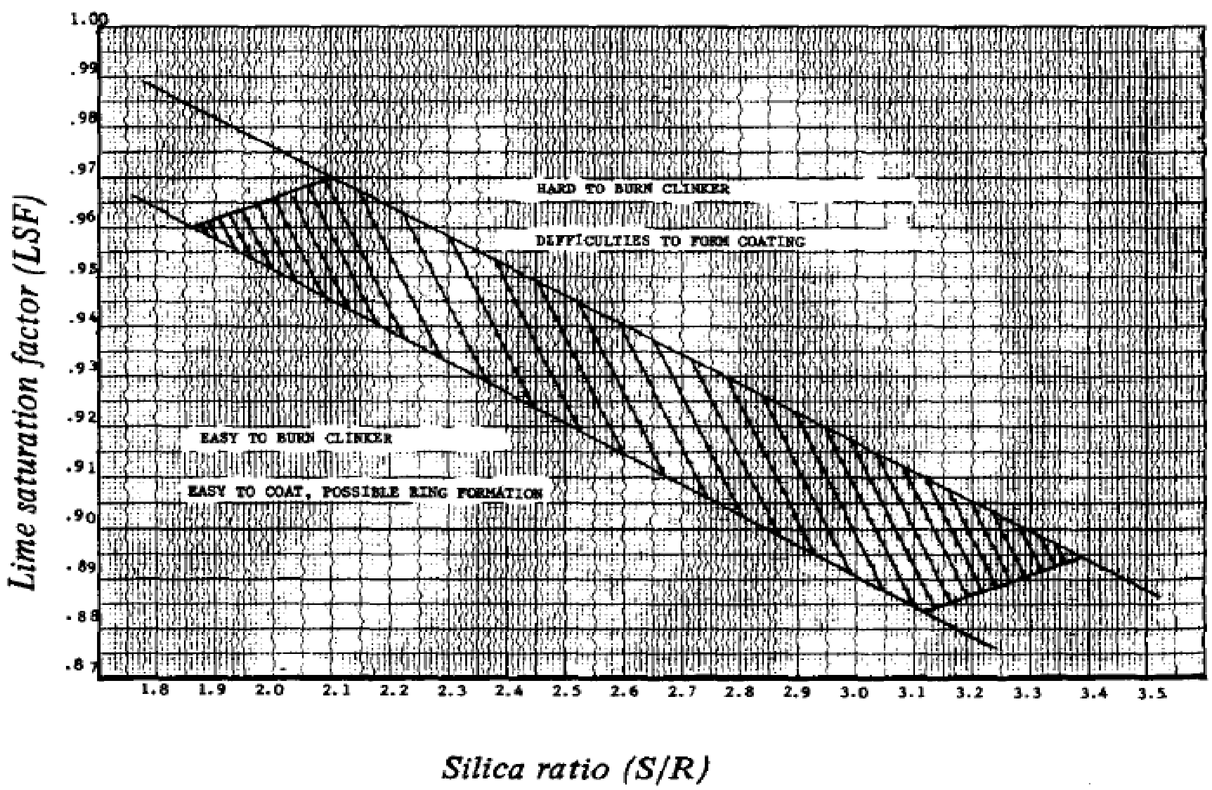

Fig. 7.3, Relationship silica ratio vs. saturation factor

Table 7.1 Clinker quality parameters for MCE

Clinker parameter	Appropriate Range	Actual value
Free lime	$\le$ 1.5 %	0.95
Liter weight	1250 $\pm$ 100 g/ℓ	1312
C3S	52 – 58 %	58.13
C2S	16 – 22 %	23.85
C3A	7 – 9 %	7.41
C4AF	9.5 – 12.5 %	10.485
Coating index	30 $\pm$ 2	29.96
Burnability index	116 $\pm$ 2	114.9
Insoluble residue	$\le$ 0.5 %	0.45
Liquid phase	25 – 27 %	25.7

The heat balance shows more than 40 % of heat obtained by burning fuel is evolved from kiln system with preheater exhaust gases, clinker and cooler waste gases. These are recoverable heat. Out of heat released with preheater exhaust gases (21,442,320 k Cal/hr), about 4, 473,716 k Cal/hr is recovered to remove moisture content in the raw meal. Totally 34,630,924 k Cal/h of heat is pointlessly lost due to poor heat recovery process at MCE. Currently MCE is paying around 15.04 birr/ liter or 16.259 birr/ kg of HFO. With this cost, reducing the recoverable heat losses by a quarter will enables a saving of 38,638,503.64 birr annually (based on the factory 2006 EC nine months production report).

In addition to the recoverable heat losses, there are other losses due to poor insulation and inappropriate bricks type and deterioration. The investigation of preheater surface heat losses verifies this fact. In reality, it is expected that the surface temperature of the preheater decreases as the gas and material temperate decreases as far as there is no blockage. However, the situation is mostly in the reverse and unpredictable for case of MCE.

The first and the prominent action that has to be taken to improve the thermal energy efficiency of the factory is to enhance the heat recuperation competence of the cooler and to reduce the excess air consumption of the kiln system.

Beside the improvement, there has to be further design modification work for waste energy recovery mechanism. Nowadays, for sake of competitiveness and profitability, the technology of heat recovery from waste gases is substantial. For instance Derba MIDROC cement factory utilizes the cooler waste gas heat for drying the pumice. In the contrary, MCE uses heat obtained from diesel to heat the pumice.

As mentioned above, in addition to the cooler improvement, there must be waste heat recovery mechanisms. There are opportunities that exist within the plant to capture the heat that would otherwise be wasted to the environment and utilize this heat to generate electricity. The most accessible and, in turn, the most cost effective waste heat losses available are the clinker cooler discharge and the kiln exhaust gas. Both streams would be directed through a waste heat recovery steam generator (WHRSG), and the available energy is transferred to water via the WHRSG. The schematic of a typical WHRSG cycle is shown in Fig. 7.4.

The available waste energy is such that steam would be generated. This steam would then be used to power a steam turbine driven electrical generator. The electricity generated would offset a portion of the purchased electricity, thereby reducing the electrical demand.

In order to determine the size of the generator, the available energy from the gas streams must be found. Once this is determined, an approximation of the steaming rate for a specified pressure can be found. The steaming rate and pressure will determine the size of the generator. The following calculations were used to find the size of the generator.

$$\mathbf{Q_{WHRSG} = Q_{available}\,\eta} \text{ --- } (7.1)$$

Where η is the efficiency of the WHRSG

Because of various losses and inefficiencies inherent in the transfer of energy from the gas stream to the water circulating within the WHRSG, not all of the available energy will be transferred. A reasonable estimate on the efficiency of the WHRSG must be made. We assume an overall efficiency of 85% for the steam generator.

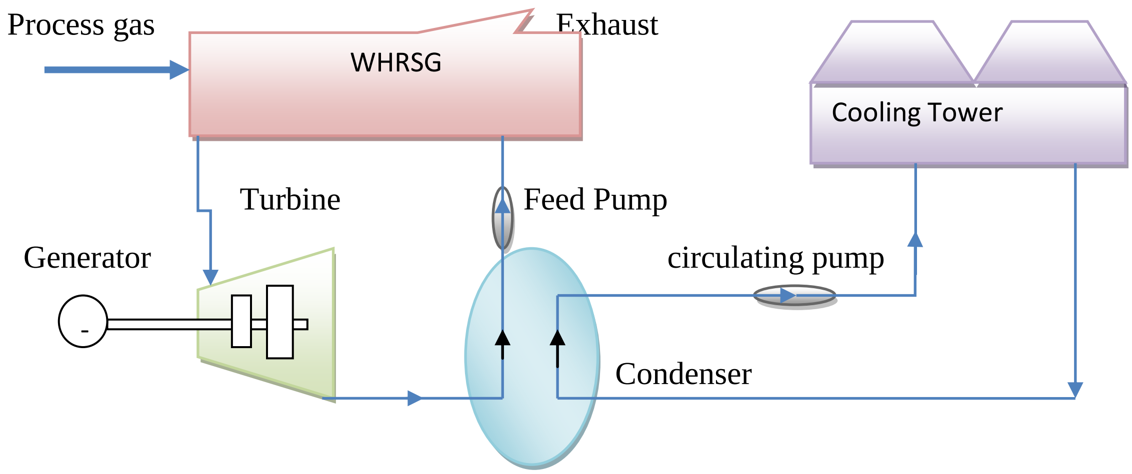

Fig.7.4, Process schema of a typical WHRSG application

As the gas passes through the WHRSG, energy will be transferred and the gas temperature will drop. Targeting a pressure of 8 bars at the turbine inlet, the minimum stream temperature at the WHRSG's exit would be higher than the corresponding saturation temperature, which is roughly 170 ^{0}C. As a limiting case, we assume the exit temperatures to be 170 ^{0}C. After exiting the WHRSG, the energy of those streams can be recovered by using a compact heat exchanger. Hence, the final temperature can be reduced as low as possible, which might be limited by the acid dew point temperature of the stream. At cooler and preheater exit temperature, the corresponding enthalpies become 220 and 337 kJ/kg respectively.

According to the final temperatures of both streams, the final enthalpies have been calculated to be h_{air} = 173 kJ/kg, and h_{eg} =175 kJ/kg.

Currently, part of exhaust gas (61, 200 Nm^3/h) is used to heat up the raw material in the raw mill. Totally about 185,866 Nm^3/h of exhaust gas is being released from the preheater. Therefore, there is an excess of 124,666 Nm^3/h of exhaust gases.

Mass of excess gases (m_{eg}) = gas flow rate * density of the gases

= 1.4925 kg of exhaust gas per kg of clinker

Mass of exhaust gases at cooler (m_{air})

= gas flow rate * density of the gases

= 0.7525 kg of exhaust gas per kg of clinker

Mass flow rate of clinker (m_{cli}) = 108000 kg/hr = 30 kg/s

Therefore, the available heat energy would be:

$$Q_{Available} = \eta * [m_{eg} (h_{eg1} – h_{eg2}) + m_{air} (h_{air1} – h_{air2})] * m_{cli} \text{ ----------------------- } (7.2)$$
$$= 7.067 \text{ MW}$$

The next step is to find a steam turbine generator set that can utilize this energy. Since a steam turbine is a rotating piece of machinery, if properly maintained and supplied with a clean supply of dry steam, the turbine should last for a significant period of time. Considering a turbine pressure of 8 bars and a condenser pressure of 20 kPa, it can be shown that the net power, which would be obtained from the turbine, is almost 2654 kW.

Just by taking the time reference of the last nine months operating hours, the kiln operates for 4000 hrs in its full capacity.

$$\text{Energy saved} = \text{(Power generated) (hours of usage)} \text{ --------------------- } (7.3)$$
$$= (2654 \text{ kW}) (4000 \text{ h}) = 10.616 * 10^6 \text{ kWh/ yr}$$

Using the current average unit price of electricity 0.4086 birr per kWh of energy, and therefore, the anticipated cost savings would be

$$\text{Cost savings} = 10.616 * 10^6 \text{ kWh/ yr} * 0.4086 \text{ birr/ kWh}$$
$$= 4{,}337{,} 698 \text{ birr}$$

If we assume that labor and maintenance costs averaged out to 100,000 birr annually, the savings becomes 4,237, 698 birr /year.

The cost associated with implementation of this additional system would be the purchase price of the necessary equipment and its installation. An additional cost will be the required maintenance of the power generation unit. For the whole system shown in Fig. 6.4, it is found that about 13,992,500 birr investment is required including shipping and installation. Hence, it is possible to make a rough estimate for a simple payback period as follows.

$$\text{Payback period} = 13{,}992{,}500/4{,}237{,} 698 = 3.3 \text{ years.}$$

The rotary pumice dryer is the other thermally incompetent unit. For process and energy optimization, proper functioning of measuring meters is decisive. Its operation is completely controlled manually depending on the operator experience and mill conditions. The fuel, material & primary gases flow meters and temperature and pressure gauges have to work to enable automatic process control.

Once these units become functional, based on the material flow rate and material initial and expected final moisture content; the fuel and the primary air flow rate can be adjusted either automatically or manually. This will enable optimized diesel consumption and also establish stable grinding process in the mill. With some modification, further improvement can be made to utilize alternative fuels which are less costly and better caloric value.

7.2 Electrical Energy Analysis

Electrical energy evaluation result clearly shows that most of the units are less efficient and have open room for modification. It found that size reduction and fluid machine units are intensive electrical energy consuming and dissipative units.

In the last nine months of 2006 EC; 319,223 tones of clinker is produced. Out of this, about 90 % of it is used to produce pozzolanic Portland cement, PPC and the rest 10% is served to make ordinary Portland cement, OPC. The PPC is produced by mixing and grinding clinker, pumice and gypsum averagely with a composition of 72.23%, 22.67% and 5 % respectively. The corresponding amount of PPC produced is 366,796.8 tons. The total electrical energy consumption to produce this amount of PPC is 37,352,662.7 kWh. Then, the specific power consumption becomes 101.8 kWh/tones of cement or 95 kWh/tons of clinker.

In the cement manufacturing process large fans draw air through the kiln, precalciner, mills and filters to an exhaust stack. Many smaller fans push the air into the grate cooler to reduce the temperature of the hot clinker leaving the kiln. For industrial fan applications, it is imperative that the fan be properly sized and designed to meet all the requirements for each application. Because of the unique nature of each system, a unique fan is required that addresses; Aerodynamic performance, Efficiency, Stability, Controllability &Acoustics Mechanical reliability.

The analysis on the performance of fan system indicates as most of them are working with efficiency less than 20%. The efficiency of the fan is the sum of the performance of motor, coupling and the fan as the whole. The motor efficiency falls with a rise in temperature and deterioration of the winding system and it is also a function of its load.

Its performance is high when it operates at its full capacity and falls with a decrease of load on it. Most fans installed at MCE have VFD motors with a direct coupling system.

This system is efficient enough except some energy losses (about 5%) incurred due to addition of VFD system to control the motor speed that reduce an enormous energy losses when the fan is operated at high speed.

The main causes for low efficiency of the fans are their capacity overdesign and high rpm. As mentioned above, the motor efficiency falls as the load on them decreases. Even at around 24 % excess air (appropriate amount is about 10%), most fans are operated averagely 70% of their capacity. This can be clearly depicted by considering the efficiency of fan 2604 and fan 2609. Fan 2604 is operating at 60 % of its capacity and its efficiency is 13% whereas fan 2609 is operating at 86% of its capacity and its efficiency is 24.15%. The second and the most prominent factor is the increase of fan speed due to an increase of gas volume flow rates. According to fan laws, gas flow proportionally increases with an increase of fan speed whereas the power consumption increases cubically.

The Fan system consumes 34.6 kWh of electrical energy per ton of clinker which means 36.42% of electrical energy spent to produce one ton of clinker. If the kiln system is made to operate at the appropriate excess air level, a saving of 10.43 kWh of energy per ton of clinker is possible. With this excess air reduction, in the last nine months, MCE can achieve 3,330,712 kWh of electrical energy reduction. With the current electrical energy cost, 0.4086 birr per KWh, MCE can save 1,360,928.95 birr.

The second electrical energy intensive unit is compressor system. The Compressor system at Mugher consists of four compressors in parallel in which three of them are working at a time while the other one is serving as a standby purpose. This compressor system has poor efficiency due to two main reasons. The first and the critical is the design problem. As it can be clearly seen on compressor performance characteristics, the compressor efficiency falls with an increase of number of compressors for the same volume of gas flows. Secondly, compressor efficiency falls with an increase of inlet gas temperature and dust load. Air enters the compressor suction pore nearly at room temperature and the air is highly dust loaded.

To avoid dust entrance to compressor, there is a dust filter at the intake, which further adds pressure drop to the compressor. Generally these and other factors persuaded the system to have poor efficiency regardless of its low leakage rate.

In the contrast, the compressor at Tatek site has a higher leakage rate but a better efficiency. This proves the preference of single compressor with high capacity over the multiple compressors in parallel. In addition to its proper design, the environment is less dusty, dry (less humidity) and cold (lower intake temperature) which are favorable conditions for better performance. However, even if this efficiency is relatively higher than that of compressor at Mugher, still it is very low. The percentage of lost by leakage never has to be greater than 10%. To compensate the air volume lost by the leakage, additional power has to be consumed to deliver more air. Therefore, to reduce this leakage the air compressed system has to be well maintained.

For compressors at both sites, in addition to the factors stated above, the compressor stoppage time is critical one. During its unloading time (time at which compressor stops working when it is full but the motor runs), compressor consumes about 50% of its full load power. MCE’s compressors have unloading time of 20 minutes which have to be minimized in order to avoid unnecessary energy losses by adjusting its setting.

Reducing the unloading time from 20 minutes to 5 minutes (optimum) enable reduction of electrical energy consumption by around 380,418.75 kWh which is equivalent to 155,439.5 birr for the last nine months.

The next electrical energy consuming unit is size reduction systems. These systems include crusher, vertical roller mill and ball mill. Even if they are relatively better than other units, they are still poor efficient. Oversized feed and high moisture content are the main reasons for lowered efficiency. Moreover, a wide particle size distribution and hard grinding nature of material used has its own contribution for the higher energy consumption. A great modification has been made on the VRM capacity by performing internal structural change without jeopardizing its performance. In the case of the ball mill, its performance can be boosted just by enhancing the competency of the pumice dryer. Having consistent drying system will enable the grinding process to be stable and predictable. Therefore, ball mill effectiveness is indirectly dependant on the pumice dryer performance.

8. Conclusion and recommendation

8.1 Conclusion

The result of the study revealed that MCE has an enormous opportunity to save both thermal and electrical energy. Proper and stable kiln operation can reduce energy consumption and maintenance costs, increase kiln output, and improve overall product quality. Moreover, it can be concluded that the root causes of high energy losses are resulted from improper design of equipments and operational incompetence.

Kiln is the heart of the cement plant, as the critical step of clinkerization takes place here. The kiln is the major consumer of thermal energy and also one of the major electrical energy consumers in the cement plant. So any inefficiency in the kiln section will reflect directly on the whole plant irrespective of the inefficiencies of the other parts of the plant. Hence the kiln section needs to be systematically and seriously concentrated upon to achieve maximum energy efficiency.

Low recuperation efficiency of cooler, large volume of exhaust gases vent from preheater and conventionally operated pumice dryer are key thermal energy loss areas. Low recuperation efficiency is mostly caused by impediment of heat transfer by dusty cooler internals. Regardless of the liquid content in the kiln system, the high air flow rates in the cooler persuades the air to have velocity greater than 5 m/s which entrain the dust from the clinker granule surface and disperse it. Furthermore, the air velocity will be enhanced at static part of the cooler due to narrow cross-sectional area of tertiary air duct just before kiln hood. This further aggravates the dust cloud. In addition to high gas velocity, segregation of clinker causes Red River which is another factor for poor performance of cooler. Generally these and other related factors made the cooler to be less efficient and caused it to release clinker and waste gases at high temperature, 310 ^{0}C and 450 ^{0}C respectively.

Operating at around 24% excess air is unusual in cement factory. However, MCE has accustomed it and losing its energy willingly. This extraordinarily excess air has dual effect. First, it cools the kiln system. So as to compensate this effect, additional fuel has to be burnt. Secondly, it increases the load of down coming fluid machines. This jeopardizes the effectiveness of these machines and increases their electrical energy consumption.

Indiscriminately operated Pumice dryer is the other tricky thermal unit. Systematic operation of this unit will enable an extensive saving of energy and stable ball mill operation. However, achieving this would be possible if and only if measuring gauges become active and conventional controlling system altered to automatic functioning.

A thorough electrical energy evaluation testified that fans and compressors are the chief energy intensive units in MCE which have an immense room for energy optimization. All most all of these fluid machines are operating with poor proficiency due to several reasons. Capacity overdesign and high speed are the root causes of low performance of the fans. On the other hand, improper design of compressors (multiple compressors in parallel instead of single and large) and high leakage rate are hurdles for efficiency improvement of compressors besides to the dusty and moist environment conditions.

In addition of fans and compressors, size reduction units are decisive units for energy utilization efficiency enhancement. Reducing the feed size, selecting easily grindable material composition and reducing moisture content in the feed are vital steps for competency improvement of the grinding system.

Lastly it is possible to conclude that if an effort is made to reduce energy consumption in MCE, there is a vast room for large saving of currency by reducing amount of fuel burnt and improving inadequately operating electrical units. This in turn enables protection of the environmental damage caused by the green house gases released from burning of additional fuel due to poor efficiency. If it were possible to recover a quarter of recoverable energy lost from preheater and cooler exhaust gases and from clinker leaving cooler, it would be possible to save currency about 38,638,503.64 birr in the last nine months. At the same time, if the excess air were reduced from 24% to 10 % (optimum) and the unloading time of the compressor were reduced from 20 minute to 5 minutes it would enable a saving of 1,360,928.95 and 155,439.5 birr respectively.

These all figures prove as the Enterprise has an immense opportunity for saving of energy and money without jeopardizing its performance and further keeping its reputation enhanced.

8.2 Recommendation

Believing thoroughly as this study gives initiatives for the MCE's professionals and administrators would not be wrong. As any fist study, the purpose of this study is to put ground for further study and to create passion for experts inside and outside the Enterprise in the field of area. This study has shown clearly the black spot of the Enterprise in the energy proficiency perspective. After further detail work and feasibility study, there is a possible need of complete modification or partial replacement of units.

At the outset, the critical and most prominent improvement that has to come first is cooler design modification needs. Diesel driven hydraulic system has to be installed to avoid material blockage in the cooler during power interruption. Moreover, the air suction flow rate of cooler fans has to be reduced. This needs recalibration of the automatic system.

In addition to the plan of reducing of energy consumption in cement production process, the recovery waste heats can be achieved in order to produce the electrical energy by utilization cogeneration power plant. The assessment on waste heat recovery visibly shows that it is economically viable and has short payback period to return is investment cost. As indicated above, the third line alone can provide about 3 MW. However, the Enterprise as the whole has an enormous potential to generate electrical energy from waste gases from the three lines.

Waste gases heat recovery for HFO heating, instead of the Electric heater. All of Mugher cement production lines uses electric heater to heat the fuel to reduce its viscosity for ease of atomization and combustion. Replacing the electric heating system by waste gas gives an opportunity of saving of energy about 2.8 millions per annum for the new line alone. The corresponding investment cost of the equipment is around 6.775 millions birr and the payback period is 2.4 years.

For non recoverable energy like heat losses by radiation and convection from surface of precalciner and the like can be reduced by insulating them. This is most common techniques for most optimized cement plants over the world.

To increase the effectiveness of the compressors at Mugher sites, it is better to replace the multi Compressors in parallel by a single compressor of the same capacity of their whole. For both sites, reducing their unloading time has no preference. Moreover, at both sites, the moisture removal system that removes moisture trapped from compressed air has to be made functional. If the moisture carried with compressed air throughout the air distribution lines, it cause wearing and instability for the system.

The rotary dryer flow meters have to be functional. In addition, burning diesel is tricky event for drying of pumice. Therefore, by modifying the burning system and some dryer internals it is better to enable the dryer to be used for all alternative fuels that are cheap and abundant.

Lastly, the Enterprise has to prompt its coal utilization project in order to remain competent in the market and profitable.

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