



Energy auditing and recovery for dry type cement rotary kiln systems—A case study

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Abstract

Cement production has been one of the most energy intensive industries in the world. In order to produce clinker, rotary kilns are widely used in cement plants. This paper deals with the energy audit analysis of a dry type rotary kiln system working in a cement plant in Turkey. The kiln has a capacity of 600 ton-clinker per day. It was found that about 40% of the total input energy was being lost through hot flue gas (19.15%), cooler stack (5.61%) and kiln shell (15.11% convection plus radiation). Some possible ways to recover the heat losses are also introduced and discussed. Findings showed that approximately 15.6% of the total input energy (4 MW) could be recovered.

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1. Introduction

Cement production is an energy intensive process, consuming about 4 GJ per ton of cement product. Theoretically, producing one ton of clinker requires a minimum 1.6 GJ heat [1]. However, in fact, the average specific energy consumption is about 2.95 GJ per ton of cement produced for well-equipped advanced kilns, while in some countries, the consumption exceeds 5 GJ/ton. For example, Chinese key plants produce clinker at an average energy consumption of 5.4 GJ/ton [2].

The energy audit has emerged as one of the most effective procedures for a successful energy management program [3]. The main aim of energy audits is to provide an accurate account of energy consumption and energy use analysis of different components and to reveal the detailed

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information needed for determining the possible opportunities for energy conservation. Waste heat recovery from hot gases [4] and hot kiln surfaces [5] in a kiln system are known as potential ways to improve overall kiln efficiency. However, it is still fairly difficult to find a detailed thermal analysis of rotary kiln systems in the open literature. This paper focuses on the energy audit of a horizontal rotary kiln system, which has been using in the Van Cement Plant in Turkey. A detailed thermodynamic analysis of the kiln system is first given, and then, possible approaches of heat recovery from some major heat loss sources are discussed.

2. Process description and data gathering

Rotary kilns are refractory lined tubes with a diameter up to 6 m. They are generally inclined at an angle of 3–3.5°, and their rotational speeds lie within 1–2 rpm. Cyclone type pre-heaters are widely used to pre-heat the raw material before it enters the kiln intake. In a typical dry rotary kiln system, pre-calcination gets started in the pre-heaters, and approximately one third of the raw material would be pre-calcined at the end of pre-heating. The temperature of the pre-heated material would be of the order of 850 °C. The raw material passes through the rotary kiln towards the flame. In the calcination zone (700–900 °C), calcinations, as well as an initial combination of alumina, ferric oxide and silica with lime, take place. Between 900 and 1200 °C, the clinker component, $2\text{CaO} \cdot \text{SiO}_2$, forms. Then, the other component, $3\text{CaO} \cdot \text{SiO}_2$, forms in a subsequent zone in which the temperature rises to 1250 °C. During the cooling stage, the molten phase, $3\text{CaO} \cdot \text{Al}_2\text{O}_3$, forms, and if the cooling is slow, alite may dissolve back into the liquid phase and appear as secondary belite [6]. Fast cooling of the product (clinker) enables heat recovery from the clinker and improves the product quality.

The data taken from the Van Cement Plant have been collected over a long period of time when the first author was a process engineer in that plant. The plant uses a dry process with a series of cyclone type pre-heaters and an incline-kiln. The kiln is 3.60 m in diameter and 50 m long. The average daily production capacity is 600 ton of clinker, and the specific energy consumption has been estimated to be 3.68 GJ/ton-clinker. A large number of measurements have been taken during 2 yr, and averaged values are employed in this paper. The raw material and clinker compositions are given in Table 1. The rotary kiln system considered for the energy audit is schematically shown in Fig. 1. The control volume for the system includes the pre-heaters group, rotary kiln and cooler. The streams to and from the control volume and all measurements are indicated in the same figure.

3. Energy auditing and heat recovery

3.1. Mass balance

The average compositions for dried coal and pre-heater exhaust gas are shown in Fig. 2. Based on the coal composition, the net heat value has been found to be 30,600 kJ/kg-coal.

It is usually more convenient to define mass/energy data per kg clinker produced per unit time. The mass balance of the kiln system is summarized in Fig. 3. All gas streams are assumed to be ideal gases at the given temperatures.

Table 1
Raw material and clinker components and their percentages

Component	Raw material (%)	Clinker (%)
SiO ₂	13.55	20.55
Al ₂ O ₂	4.10	5.98
Fe ₂ O ₃	2.60	4.08
CaO	40.74	64.58
MgO	2.07	2.81
SO ₃	0.56	1.30
K ₂ O	0.30	0.50
Na ₂ O	0.08	0.20
H ₂ O	0.5	–
Organics	0.9	–
Ignition loss	34.6	–
Total	100	100

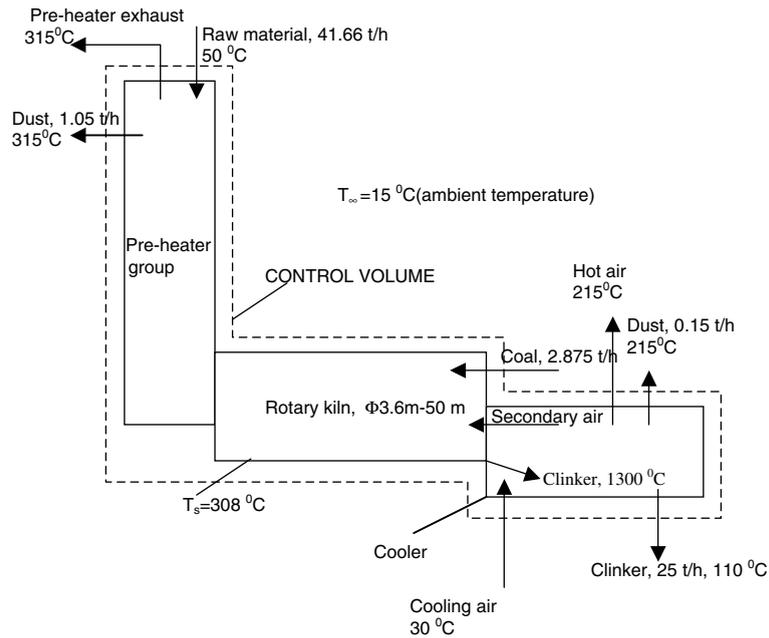


Fig. 1. Control volume, various streams and components for kiln system.

3.2. Energy balance

In order to analyze the kiln system thermodynamically, the following assumptions are made:

1. Steady state working conditions.
2. The change in the ambient temperature is neglected.
3. Cold air leakage into the system is negligible.

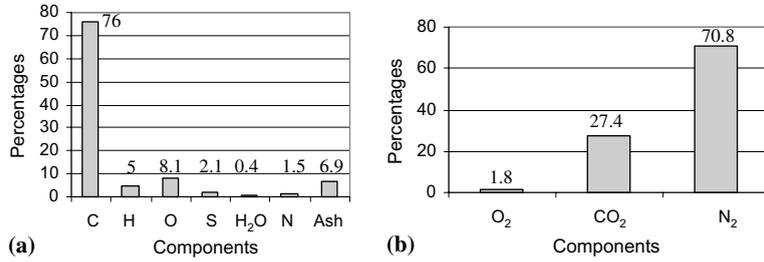


Fig. 2. (a) Coal composition, (b) pre-heater exhaust gas composition (by weight).

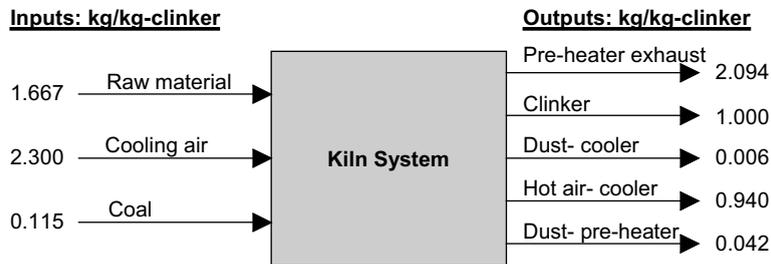


Fig. 3. Mass balance of the kiln system.

4. Raw material and coal compositions do not change.
5. Averaged kiln surface temperatures do not change.

Based on the collected data, an energy balance is applied to the kiln system. The physical properties and equations can be found in Peray’s handbook [7]. The reference enthalpy is considered to be zero at 0 °C for the calculations. The complete energy balance for the system is shown in Table 2. It is clear from Table 2 that the total energy used in the process is 3686 kJ/kg-clinker, and the main heat source is the coal, giving a total heat of 3519 kJ/kg-clinker (95.47%). Also, the energy balance given in Table 2 indicates relatively good consistency between the total heat input and total heat output. Since most of the heat loss sources have been considered, there is only a 273 kJ/kg-clinker of energy difference from the input heat. This difference is nearly 7% of the total input energy and can be attributed to the assumptions and nature of data. The distribution of heat losses to the individual components exhibits reasonably good agreement with some other key plants reported in the literature [2,5,7].

3.3. Heat recovery from the kiln system

The overall system efficiency can be defined by $\eta = Q_6/Q_{\text{total input}} = 1795/3686 = 0.487$ or 48.7%, which can be regarded as relatively low. Some kiln systems operating at full capacity would declare an efficiency of 55% based on the current dry process methodology. The overall efficiency of the kiln system can be improved by recovering some of the heat losses. The recovered heat energy can then be used for several purposes, such as electricity generation and preparation

Table 2
Complete energy balance of the kiln system

Description	Equations used	Data	Result (kJ/kg-clinker)
<i>Heat inputs</i>			
Combustion of coal	$Q_1 = m_c H_c$	$m_c = 0.115$ kg/kg-clinker, $H_c = 30,600$ kJ/kg	3519 (95.47%)
Sensible heat by coal	$Q_2 = m_c h_c, h_c = CT$	$m_c = 0.115$ kg/kg-clinker, $C = 1.15$ kJ/kg °C, $T = 50$ °C	7 (0.19%)
Heat by raw material	$Q_3 = m_{rm} h_{rm}, h_{rm} = CT$	$m_{rm} = 1.667$ kg/kg-clinker, $C = 0.86$ kJ/kg °C, $T = 50$ °C	72 (1.95%)
Organics in the kiln feed	$Q_4 = FK h_{os}$	$F = 0.10, K = 0.9\%, h_{os} = 21,036$ kJ/kg (Ref. [7])	19 (0.52%)
Heat by cooling air	$Q_5 = m_{ca} h_{ca}$	$m_{ca} = 2.300$ kg/kg-clinker, $h_{ca} = 30$ kJ/kg ($T = 30$ °C)	69 (1.87%)
Total heat input			3686 (100%)
<i>Heat outputs</i>			
Formation of clinker	$Q_6 = 17.196(\text{Al}_2\text{O}_3) + 27.112(\text{MgO}) + 32(\text{CaO}) - 21.405(\text{SiO}_2) - 2.468(\text{Fe}_2\text{O}_3)$	(Clinker composition is given in Table 1)	1795 (48.70%)
Kiln exhaust gas	$Q_7 = m_{eg} C_{p-eg} T_{eg}$	$m_{eg} = 2.094$ kg/kg-clinker, $C_{p-eg} = 1.071$ kJ/kg °C, $T = 315$ °C	706 (19.15%)
Moisture in raw material and coal	$Q_8 = m_{\text{H}_2\text{O}}(h_{fg(50\text{ }^\circ\text{C})} + h_{315\text{ }^\circ\text{C}} - h_{50\text{ }^\circ\text{C}})$	$m_{\text{H}_2\text{O}} = 0.008835$ kg/kg-clinker (in coal + raw material), $h_{fg(50\text{ }^\circ\text{C})} = 2384$ J/kg, $h_{50\text{ }^\circ\text{C}} = 2591$ J/kg, $h_{315\text{ }^\circ\text{C}} = 3104$ J/kg	26 (0.71%)
Hot air from cooler	$Q_9 = m_{\text{air-co}} h_{\text{air-co}}$	$m_{\text{air-co}} = 0.940$ kg/kg-clinker, $h_{\text{air-co}} = 220$ kJ/kg ($T = 215$ °C)	207 (5.61%)
Heat loss by dust	$Q_{10} = (m_{\text{dust-preheater}} + m_{\text{dust-air cooler}}) h_{\text{dust,ave}}$	$m_{\text{dust-preheater}} = 0.042$ kg/kg-clinker, $m_{\text{dust-air co}} = 0.006$ kg/kg-clinker, $h_{\text{dust,ave}} = 275$ kJ/kg (Ref. [7])	11 (0.30%)
Clinker discharge	$Q_{11} = m_{\text{cli}} h_{\text{cli}, T=110\text{ }^\circ\text{C}}$	$m_{\text{cli}} = 1$ kg/kg-clinker, $h_{\text{cli}} = 86$ kJ/kg (Ref. [7])	86 (2.33%)
Radiation from kiln surface	$Q_{12} = \sigma \varepsilon A_{\text{kiln}} (T_s^4 - T_\infty^4) / (1000 \dot{m}_{\text{clinker}})$	$\sigma = 5.67 \times 10^{-8}$ W/m ² K ⁴ , $\varepsilon = 0.78$ (oxidized surface [8]), $A_{\text{kiln}} = 565.48$ m ² , $T_s = 581$ K, $T = 288$ K, $\dot{m}_{\text{clinker}} = 6.944$ kg/s	386 (10.47%)
Convection from kiln surface	$Q_{13} = h_{\text{con}} A_{\text{kiln}} (T_s - T_\infty), h_{\text{con}} = \frac{k_{\text{air}}}{D_{\text{kiln}}} Nu$	$Re = 373, 110$ ($V_{\text{air}} = 3$ m/s), $Nu = 733$ (Hilbert's equation, Ref. [8]), at $T_f = 161$ °C (film temp.)	171 (4.64%)
Radiation from pre-heater surface	$Q_{14} = \sigma \varepsilon A_{\text{ph}} (T_s^4 - T_\infty^4) / (1000 \dot{m}_{\text{clinker}})$	$\sigma = 5.67 \times 10^{-8}$ W/m ² K ⁴ , $\varepsilon = 0.78$ (oxidized surface, Ref. [8]), $A_{\text{ph}} = 150.8$ m ² , $T_s = 348$ K, $T_\infty = 393$ K, $\dot{m}_{\text{clinker}} = 6.944$ kg/s	7 (0.19%)

Table 2 (continued)

Description	Equations used	Data	Result (kJ/kg-clinker)
Natural convection from pre-heater surface	$Q_{15} = h_{\text{ncon}} A_{\text{ph}} (T_s - T_{\infty})$, $h_{\text{ncon}} = \frac{k_{\text{air}}}{L_{\text{ph}}} Nu$, $Nu = 0.1(Ra)^{1/3}$	Pre-heater is modeled as a vertical cylinder with $D = 4$ m, $L_{\text{ph}} = 12$ m, $Ra = 0.649 \times 10^{13}$, $Nu = 1865$ (Ref. [8]), at $T_f = 45$ °C (film temp.)	5 (0.14%)
Radiation from cooler surface	$Q_{16} = \sigma \varepsilon A_c (T_s^4 - T_{\infty}^4) / (1000 \dot{m}_{\text{cli}})$	$\sigma = 5.67 \times 10^{-8}$ W/m ² K ⁴ , $\varepsilon = 0.78$ (oxidized surface, Ref. [8]), $A_c = 64$ m ² , $T_s = 353$ K, $T_{\infty} = 328$ K, $\dot{m}_{\text{clinker}} = 6.944$ kg/s	4 (0.11%)
Natural convection from cooler surface	$Q_{17} = h_{\text{ncon}} A_c (T_s - T_{\infty})$, $h_{\text{ncon}} = \frac{k_{\text{air}}}{L_c} Nu$, $Nu = 0.1(Ra)^{1/3}$	Cooler surface is modeled as a vertical plate 8 m × 8 m, $Ra = 0.7277 \times 10^{12}$, $Nu = 899$ (Ref. [8]), at $T_f = 70$ °C (film temp.)	9 (0.24%)
Unaccounted losses			273 (7.41%)
Total heat output			3686 (100%)

of hot water. There are a few major heat loss sources that would be considered for heat recovery. These are heat losses by: (1) kiln exhaust gas (19.15%), (2) hot air from cooler stack (5.61%) and (3) radiation from kiln surfaces (10.47%). In the following section, we discuss some possible ways for recovering this wasted heat energy.

3.3.1. The use of waste heat recovery steam generator (WHRS)

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. The exhaust gas from the kilns is, on average, 315 °C, and the temperature of the air discharged from the cooler stack is 215 °C. Both streams would be directed through a waste heat recovery steam generator (WHRS), and the available energy is transferred to water via the WHRS. The schematic of a typical WHRS cycle is shown in Fig. 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.

$$Q_{\text{WHRS}} = Q_{\text{available}}\eta$$

where η is the WHRS efficiency. Because of various losses and inefficiencies inherent in the transfer of energy from the gas stream to the water circulating within the WHRS, not all of the

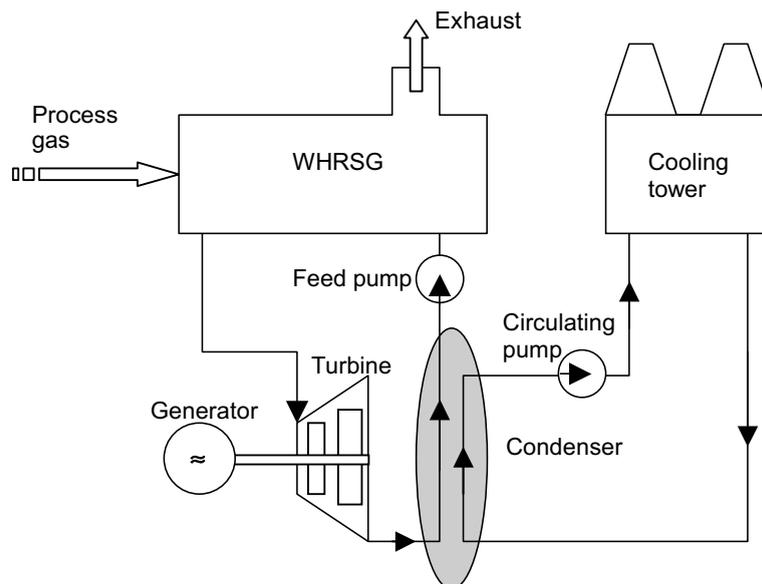


Fig. 4. Process schema of a typical WHRS application.

available energy will be transferred. A reasonable estimate on the efficiency of the WHSRG must be made. We assume an overall efficiency of 85% for the steam generator. 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 °C. As a limiting case, we assume the exit temperatures to be 170 °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. According to the final temperatures of both streams, the final enthalpies have been calculated to be $h_{\text{air}} = 173$ kJ/kg, and $h_{\text{eg}} = 175$ kJ/kg. Therefore, the available heat energy would be:

$$Q_{\text{available}} = [m_{\text{eg}}(h_{\text{eg1}} - h_{\text{eg2}}) + m_{\text{air}}(h_{\text{air1}} - h_{\text{air2}})] \times m_{\text{cli}}$$

$$Q_{\text{available}} = [2.094 \times (337 - 175) + 0.94 \times (220 - 173)] \times 6.944 \approx 2662 \text{ kW}$$

Therefore, the energy that would be transferred through the WHSRG is

$$Q_{\text{WHSRG}} = 0.85 \times 2662 = 2263 \text{ kW}$$

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 10 kPa, it can be shown that the net power, which would be obtained from the turbine, is almost 1000 kW. If we assume that the useful power generated is 1000 kW, then the anticipated savings will be based on the load reduction of 1000 kW. Assuming 8000 h of usage, we find

$$\text{Energy saved} = (\text{Power generated}) \times (\text{hours of usage})$$

$$\text{Energy saved} = (1000 \text{ kW}) \times (8000 \text{ h/yr}) = 8 \times 10^6 \text{ kW h/yr}$$

The average unit price of electricity can be taken as 0.07 USD/kWh, and therefore, the anticipated cost savings would be

$$\text{Cost savings} = 0.07 \times 8 \times 10^6 = 560,000 \text{ USD/yr}$$

If we assume that labor and maintenance costs averaged out to 20,000 USD annually, the savings becomes 540,000 USD/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. 4, based on our calculations, we were able to determine a budget estimation between 700,000 and 750,000 USD, including shipping and installation. Hence, we can make a rough estimate for a simple pay back period:

$$\text{Simple pay back period} = (\text{Implementation cost}) / (\text{Annual cost savings})$$

$$\text{Simple pay back period} = \frac{750,000 \text{ USD}}{540,000 \text{ USD/yr}} = 1.38 \text{ yr} \approx 17 \text{ months}$$

The energy savings by using a WHSRG system would also result in an improvement in the overall system efficiency. It should be noted that these calculations reflect a rough estimation and may vary depending upon plant conditions and other economic factors.

3.3.2. Use of waste heat to pre-heat the raw material

One of the most effective methods of recovering waste heat in cement plants would be to pre-heat the raw material before the clinkering process. Directing gas streams into the raw material just before the grinding mill generally does this. This would lead to a more efficient grinding of the raw material in addition to increasing its temperature. However, in most plants, the fresh raw material taken from the mill is not directly sent to the kiln, and therefore, the temperature increase of the raw material does not generally make sense because it will be stored in silos for a while before entering the clinkering process. On the other hand, some plants may have only kiln systems rather than grinding systems. In such cases, this may not be possible unless some additional modifications are made in the plant.

There is a grinding mill in the plant considered in this study, and the following calculations are made to show how much energy could be saved through pre-heating the raw material. The main advantage of the pre-heating of raw material in the mill is to dry the material, since it is heavily moist in nature. For the plant considered, the moisture content of the raw material was about 6.78%, which indicates a mass flow rate of water of 0.7845 kg/s (0.113 kg/kg-clinker) coming into the mill. Mixing the two main hot gas streams would lead to a single gas flow at about 280 °C, as shown in Fig. 5. Applying the conservation of mass principle and conservation of total energy law for the mill (ignoring heat losses) results in an increase of temperature of the raw material by 85 °C, while the gas stream cools down to 150 °C. It is clear that the majority of the useful energy must be used to heat the water from 15 to 100 °C, and to vaporize it at this temperature completely. A basic energy balance for the mill system would give

$$Q_{\text{gas flow,in}} + Q_{\text{moist raw material}} = Q_{\text{water}} + Q_{\text{gas,out}} + Q_{\text{dry raw material}}$$

The related enthalpies are shown in Fig. 5. Therefore, we can write

$$6.944 \times (3.034 \times 300 + 1.780 \times 12) - 0.7845 \times (419 - 63 + 2257)$$

$$- 6.944 \times 1.667 \times 88 = Q_{\text{gas,out}}$$

$$Q_{\text{gas,out}} \approx 3400 \text{ kW}, \quad h_{\text{gas,out}} \approx 156 \text{ kJ/kg}, \quad T_{\text{gas,out}} \approx 150 \text{ }^\circ\text{C}.$$

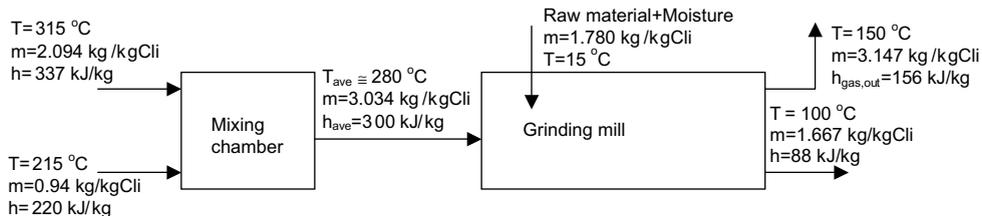


Fig. 5. Mass and energy balance for grinding mill.

3.3.3. Heat recovery from kiln surface

The hot kiln surface is another significant heat loss source, and the heat loss through convection and radiation dictates a waste energy of 15.11% of the input energy. On the other hand, the use of a secondary shell on the kiln surface can significantly reduce this heat loss. Since the kiln surface needs to be frequently observed by the operator so as to see any local burning on the surface due to loss of refractory inside the kiln, we would not consider insulating the kiln surface. The basic principle of the secondary shell is shown in Fig. 6.

For the current rotary kiln, $R_{kiln} = 3.6$ m, and a radius of $R_{shell} = 4$ m can be considered. Since the distance between the two surfaces is relatively small (40 cm), a realistic estimation for the temperature of the secondary shell can be made. We assume $T_2 = 290$ °C = 563 K. We would consider stainless steel for the material of the secondary shell since it has relatively low surface emissivity and thermal conductivity. The heat transfer rate by radiation is then calculated using the following equation [8]:

$$Q_r = \frac{A_{kiln} \sigma (T_1^4 - T_2^4)}{\frac{1}{\epsilon_1} + \frac{1 - \epsilon_2}{\epsilon_2} \left(\frac{R_{kiln}}{R_{shell}} \right)}$$

where $\sigma = 5.67 \times 10^{-8}$ W/m² K⁴, $T_1 = T_s = 581$ K, $\epsilon_1 = 0.78$ (for oxidized kiln surface) and $\epsilon_2 = 0.35$ (lightly oxidized stainless steel).

$$Q_r \approx 147 \text{ kW}$$

This heat loss is to be transferred through the insulation on the secondary shell. Therefore, assuming a reasonable temperature for the outer surface of the insulation, we can determine the required thickness of insulation. For glass wool insulation, the thermal conductivity is taken as 0.05 W/mK. Therefore, the resistance of the insulation layer would be

$$\text{Resistance of insulation} = \frac{\ln \left(\frac{R_{ins}}{R_{shell}} \right)}{2k_{ins}\pi L_{kiln}} = \frac{\ln \left(\frac{R_{ins}}{4} \right)}{15.708}$$

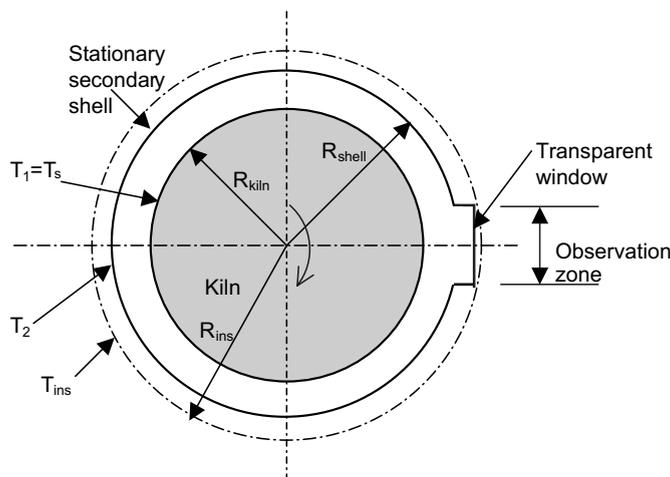


Fig. 6. Secondary shell application to the current kiln surface.

Assuming a temperature difference of $\Delta T_{\text{ins}} = 250$ °C (which means an outer surface temperature of 58 °C), R_{ins} can be determined:

$$\Delta T_{\text{ins}} = Q \times (\text{resistance of insulation})$$

$$250 \text{ °C} = 147,000 \times \frac{\ln(R_{\text{ins}}/4)}{15.708}$$

We found $R_{\text{ins}} = 4.108$ m, and the thickness of insulation would be

$$e = R_{\text{ins}} - R_{\text{shell}} = 0.108 \text{ m} \approx 11 \text{ cm}$$

It should be noted that when the secondary shell is added onto the kiln surface, the convective heat transfer would presumably become insignificant. This is because of the fact that the temperature gradient in the gap would be relatively very low, e.g., 0.45 °C/cm. Therefore, the total energy savings due to the secondary shell would be

$$(386 \text{ kJ/kg-clinker}) \times (6.944 \text{ kg-clinker/s}) - 147 = 2533 \text{ kW}$$

from the radiation heat transfer and

$$(171 \text{ kJ/kg-clinker}) \times (6.944 \text{ kg-clinker/s}) = 1187 \text{ kW}$$

from the convective heat transfer. Therefore, we can safely conclude that the use of a secondary shell on the current kiln surface would save at least 3 MW, which is 11.7% of the total input energy. This energy saving would result in a considerable reduction of fuel consumption (almost 12%) of the kiln system, and the overall system efficiency would increase by approximately 5–6%.

4. Conclusions

A detailed energy audit analysis, which can be directly applied to any dry kiln system, has been made for a specific key cement plant. The distribution of the input heat energy to the system components showed good agreement between the total input and output energy and gave significant insights about the reasons for the low overall system efficiency. According to the results obtained, the system efficiency is 48.7%. The major heat loss sources have been determined as kiln exhaust (19.15% of total input), cooler exhaust (5.61% of total input) and combined radiative and convective heat transfer from kiln surfaces (15.11% of total input). For the first two losses, a conventional WHRSG system is proposed. Calculations showed that 1 MW of energy could be recovered. For the kiln surface, a secondary shell system has been proposed and designed. It is believed that the use of this system would lead to 3 MW of energy saving from the kiln surface. Hence, the total saving for the whole system has been estimated to be nearly 4 MW, which indicates an energy recovery of 15.6% of the total input energy. The pay back period for the two systems is expected to be less than 1.5 yr.

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