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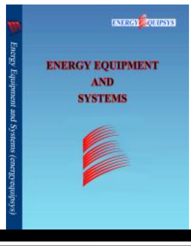

Energy auditing in cement industry: A case study

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ABSTRACT

Industrial energy consumption lies between 30% and 70% of the total energy consumed in selected countries. Cement production is one of the most energy intensive industries all around the world. This paper deals with an energy audit analysis in a cement plant in Iran. In all recent works, after performing an energy audit, different strategies are offered to reduce energy losses. Generally, these strategies differ from the viewpoint of economics and their extent of loss reduction, which makes it difficult to choose one of them. In this paper, a decision-making procedure such as an analytic hierarchy process (AHP) after an energy audit process is proposed to help the decision maker in this process.

Keywords: Cement Industry; Energy Audit; Heat and Electricity Balance; Decision-Making Procedure.

1. Introduction

Industrial energy consumption lies between 30% and 70% of total energy consumed in selected countries [1-8]. A notable amount of energy is used in the cement industry. Therefore, considerable attention is needed for the reduction of energy and energy-related environmental emissions, locally or globally [9-13]. It is reported that this industry consumes about 15% of total energy consumption in Iran [14].

Being an energy intensive industry, this segment of industry typically accounts for 50–60% of total production costs [15]. The typical electrical energy consumption of a modern cement plant is about 110–120 kWh per ton of cement [16].

It has been proven that a thermal energy saving potential of 0.25–0.345 GJ/t, an electrical energy saving potential of 20–35 kWh/t and an emission reduction potential of 4. 6–31. 66 kg CO_2/t [17-22] is feasible in this industry.

Due to their widespread use, efficient strategies for controlling motors are of the essence. Up to 700 electric motors can be found in a cement plant with various power ratings [23]. A number of functions are performed by electric motors and drives in a cement factory, including fan movement, grinding, kiln rotation and material transport. Motors can be rewired (which is often preferred to replacement) when necessary [23]. Fujimoto [24] and Hendriks [25] found the energy saving to be 3–8% with high-efficiency motors.

Variable speed drives (VSD) appear in the fans of coolers, pre-heaters, kilns and mills among other items [26]. Better control strategies for motor drives are crucial as they consume a large portion of power in the cement industry. Although most motors are fixed speed models, partial or variable load operation is common, especially considering the load variations that often occur in cement plants [27].

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In a typical cement industry, energetic and exergetic analysis of waste heat (mainly by flue gases and the ambient air stream used for cooling down the clinker, about 35% to 40% of the process' heat loss) recovery systems has been performed by S. Karellas and coworkers [28] in which two different cycles have been investigated; a water-steam cycle and an Organic Rankine Cycle (ORC) with isopentane as the working fluid. Energy and exergy analysis proved that the water steam cycle shows better performance with a system efficiency of 23.58% compared to 17.56% for the ORC. Finally the water steam cycle can be further improved, reaching 24.58% system efficiency by utilizing the high exhaust temperature of the cooling air in order to preheat the condensation before the inlet of the feed tank.

This paper focuses on the energy audit in the Momtazan cement plant in Kerman, Iran. The limestone obtained from quarries is transported to the crusher. Under the crusher, the primary riddled exists that the small broken stone in crusher shed on it. The suitable pieces of small stones that pass through the primary riddled are transferred to the materials depot, and the coarse pieces remain on the screen, again returning to the crusher. In the preparation of the raw materials for the cement production process, this material must be made entirely into powder; to this end, the bullet mill is used. At this time it is preparation of the kiln feed occurs. This procedure is performed in four ways: wet, semi-wet, semi-dry, and dry. When the kiln feed is prepared for each of these four methods, it is entered into the kiln. In the first step of the kiln, the materials are completely dried at about 800°C. At about 1000°C, the limestone is calcined: in other words, the carbon dioxide is removed. At the bottom of the kiln, approximately 25% of the materials melt at temperatures over 1400°C. This phenomenon, accomplished with the kiln's evolution, will lead to sticking other materials together and clinker production. The clinker is removed from the bottom kiln as the final product.

A preheater is installed above the kiln entrance and materials are entered into it. The output warm air from the top of the kiln enters the preheater, leading to the warming of the raw material in it. This makes both relatively drying material and their warming, and therefore, the same amount of kiln length can be reduced. A significant portion of heat energy is lost at the bottom of kiln due to the output hot clinker. On the other hand, the clinker obtained from the kiln (with a temperature over 1400°C) cannot be used when hot, and must be cooled before the continuation of the cement-making process. These two points will lead to the application of a cooler system to provide the both aim.

The clinker is then milled by the bullet mills. The powder obtained is sieved by riddling. Coarser particles from the mesh are returned to the mill. The final product is cement powder.

The paper is organized as follows. In Section 2, the heat energy balance is described and then the heat recovery from the kiln system is explained. In Section 3, the electrical energy analysis is described, and, finally, improving energy efficiency in the industrial motor system is described. In Section 4, a multi-criteria decision-making method is described. A brief review of the paper is described in Section 5.

Nomenclature

A_{ch}	Total effective area of cooler hood, m^2
A	Surface area, m^2
AIH	Air infiltrated at hood
C	energy cost per kilowatt-hour
CAB_{Pa}	Primary air at cooler, kg/h
CAB_{Ex}	Excess air vented at cooler stack, kg/h
CAB_{CO}	Total air flow into cooler, kg/h
Cl	Percent of a specific type of molecule in clinker
CP	A specific type of molecule from fuel combustion
c_j	Mean specific heat, kJ/kg. °C
D_{cooler}	Cooler width, m
D_{Ig}	Percent ignition loss in kiln dust
$D_{Preheater}$	Preheater diameter, m
Dl	Amount of feed wasted as dust, kg/kg clinker
dl	Percent dust loss
EA	Excess air percent in the kiln
F	Percent of a specific type of molecule in natural gas
F_{HV}	The heat value of natural gas, kJ/m^3
FR	Theoretical amount of feed required to produce one kilogram of clinker, kg/kg clinker
G	Percent of a specific type of molecule in kiln exit gas
h	Convection heat transfer coefficient, W/m^2

I	Current (A)
I_0	No load current (A)
KF	Percent of a specific type of molecule in kiln feed
L_1	Kiln length, m
L_2	Kiln diameter, m
L_3	Effective burner tip orifice area, m^2
L_4	Refractory thickness, mm
L_5	Kiln shell thickness, mm
L_6	Kiln slope, degrees
L_{cooler}	Cooler length, m
$L_{Preheater}$	Preheater height, m
P_h	Hood draft, mm H_2O
P	Electrical power, kW
Q	Heat energy, kW
q_j	Heat transfer coefficient, kJ/m^2. °C
$\dot{q}$	Percent calcination of the kiln dust
t	Operation time , h/yr
T_{amb}	Ambient air temp, °C
T_{KF}	Feed interring kiln temp, °C
T_{Sa}	Secondary air temp, °C
T_{Pa}	Primary air temp, °C
T_G	Kiln exit gas temp, °C
T_{St}	Cooler stack temp, °C
T_{Cl}	Clinker temp at cooler exit, °C
T_F	Fuel temp, °C
T_{Z_1}	Average temp of shell, lower third, °C
T_{Z_2}	Average shell temp of, middle third, °C
T_{Z_3}	Average temp of shell, upper third, °C
T	Kiln room temp, °C
$T_{preheater}$	The surface temperature of the preheater, °C
T_{cooler}	The surface temperature of the cooler, °C
TC_{dust}	Total carbonates in the kiln dust, kg
V_{Be}	Air volume of kiln exit, m^3/s
V_{CO}	Air volume of total air into cooler, m^3/s
V_{Ex}	Air volume of cooler vent stack, m^3/s
V_{Pa}	Air volume of primary air flow, m^3/s
W_A	Fuel rate, m^3/kg clinker
W_{Cl}	Kiln output, kg/h
W_{dF}	Dry feed rate, kg/kg clinker
WGF_{CO_2}	CO_2 from feed, kg/kg clinker
$WGF_{H_2O_{free}}$	H_2O_{free} from feed, kg/kg clinker
$WGF_{H_2O_{chem}}$	H_2O_{chem} from feed, kg/kg clinker
W	Total weight of a specific type of molecule in kiln exit gas, kg/kg clinker
w_{rev}	Reversible work, kJ/kg
ε	Emissivity
γ	Rated load
α	Load current parameter
η	Efficiency

Subscripts

Ig	Ignition loss
M	Moisture
N	Nominal
R	Real

Abbreviation

AHP	Analytic Hierarchy Process
ESV	Energy saved value
IEE	Improvement in energy efficiency
M[illegible]V	Motor investment value
ORC	Organic Rankine Cycle
QES	Quantity of energy saved
SPB	Simple payback
VSD	Variable speed drive
WHRSG	Waste heat recovery steam generator

2. Heat energy balance

In order to perform the energy balance in the cement factory, information about several parameters such as temperature, dimension, and energy consumption of the utility equipment is required. These data may be gathered from existing factory laboratories or by using installed measurement equipment. The required data for this case study is outlined in Table 1.

In order to analyse the kiln system thermodynamically, the following assumptions were made:

1. Steady state working conditions.
2. The change in the ambient temperature is neglected.
3. Cold air leakage into the system is negligible.
4. Raw material compositions do not change.
5. The average kiln surface temperatures do not change.
6. The pre-heater is modelled as a vertical cylinder.
7. The cooler surface is modelled as a vertical plate.

Based on the collected data, an energy balance is applied to the kiln system. The physical properties can be found in Peray's handbook [29]. 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 and 3. It is clear from Table 2 and 3 that the total energy used in the process is 3,658.1 kJ/kg clinker, and the main heat source is natural gas, giving a total heat of 3,278.4 kJ/kg-clinker (89.62%).

The energy balance given in Table 2 and 3 indicates relatively better consistency between the total heat input and total heat output. Since most of the heat loss sources have been considered, there is only 76.65 kJ per kg clinker of energy difference of the input heat. This difference is nearly 2.1% of the total input energy and can be attributed to the assumptions and nature of the data.

The kiln system considered for the energy audit is schematically shown in Fig. 1. The control volume for the system includes the pre-heater group, rotary kiln, and cooler.

Table 1. Information required

Parameter	Unit	Value	Parameter	Unit	Value	Parameter	Unit	Value
A_{ch}	m^2	9.62	G_{CO}	by weight	3.5	T_{amb}	°C	20
Cl_{SiO_2}	by weight	22.277	G_{N_2}	by weight	68.4	T_{KF}	°C	100
$Cl_{Al_2O_3}$	by weight	5.024	KF_{SiO_2}	by weight	14.287	T_{Sa}	°C	1,000
$Cl_{Fe_2O_3}$	by weight	4.074	$KF_{Al_2O_3}$	by weight	3.355	T_{Pa}	°C	22
Cl_{CaO}	by weight	64.714	$KF_{Fe_2O_3}$	by weight	2.693	T_G	°C	330
Cl_{MgO}	by weight	1.319	KF_{CaO}	by weight	41.561	T_{St}	°C	315
Cl_{SO_3}	by weight	0.305	KF_{MgO}	by weight	0.952	T_{Cl}	°C	210
Cl_{Ig}	by weight	1.1389	KF_{Na_2O}	by weight	0.4	T_F	°C	42
D_{cooler}	m	8	KF_{K_2O}	by weight	0.35	T_{Z_1}	°C	390
D_{Ig}	by weight	7.3	KF_{SO_3}	by weight	0.133	T_{Z_2}	°C	330
$D_{preheater}$	m	4	KF_{Ig}	by weight	36.269	T_{Z_3}	°C	120
F_{CO_2}	Volume percent	0.35	KF_M	by weight	0.4	T	°C	29
F_{H_2}	Volume percent	6.01	L_1	m	52	$T_{preheater}$	°C	75
F_{CH_4}	Volume percent	87.81	L_2	m	4.4	T_{cooler}	°C	80
$F_{C_2H_6}$	Volume percent	3.34	L_3	m^2	12.57	V_{Pa}	m^3/s	130.5
$F_{C_3H_8}$	Volume percent	1.38	L_4	mm	200	V_{Ex}	m^3/s	54.55
$F_{C_4H_{10}}$	Volume percent	0.48	L_5	mm	28	V_{CO}	m^3/s	79.6
$F_{C_5H_{12}}$	Volume percent	0.11	L_6	degrees	3.5	V_{Be}	m^3/s	55.55
F_{HV}	kJ/m^3	29,803.92	L_{cooler}	m	8	W_{Cl}	kg/h	150,000
G_{CO_2}	by weight	28.1	$L_{preheater}$	m	12	W_{dF}	kg/kg Clinker	1.6
G_{O_2}	by weight	0	P_h	mm H_2O	0.5	W_A	m^3/kg Clinker	0.11

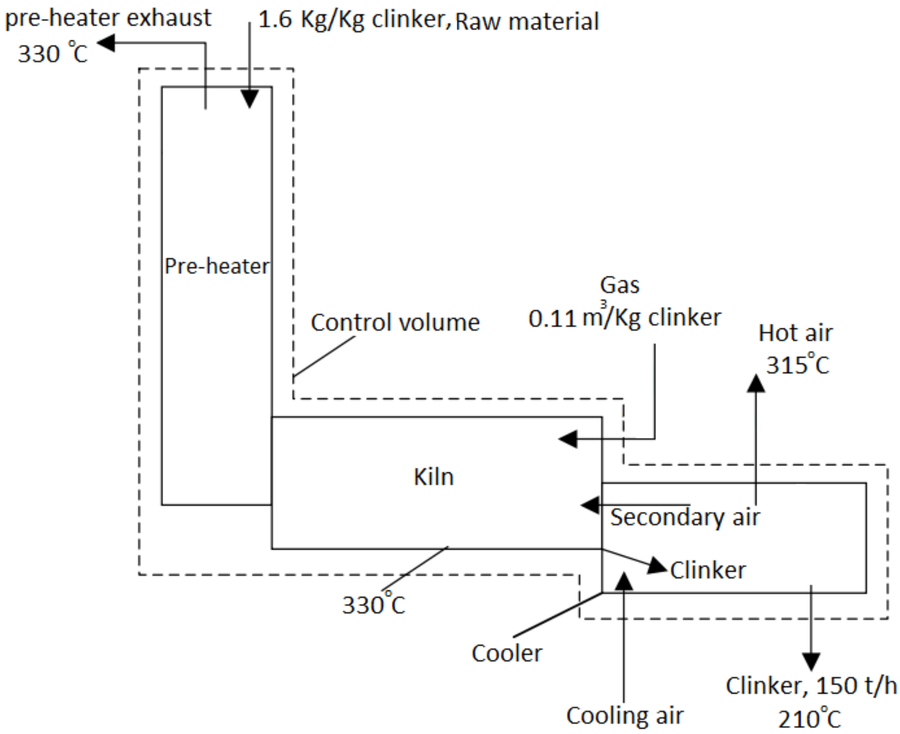

Fig. 1. Control volume, various streams and components for kiln system

Table 2. Total heat input of the kiln system

Heat Inputs	Formulation	Result	Percent
Combustion of fuel	$W_A \times F_{HV}$	3278.4	89.6
Sensible heat in fuel	$W_A \times c_{j_{Fuel}} \times T_F$	79.75	2.18
Sensible heat in kiln feed	$(W_{dF} \times c_{j_{Feed}} \times T_C) + ((WGF_{H_2O_{free}} + WGF_{H_2O_{chem}}) \times T_C \times 4.184)$ $WGF_{H_2O_{free}} = \frac{100 \times W_{df}}{100 - KF_M} - W_{df}$ $WGF_{H_2O_{chem}} = (1+dl) \times \left(0.00075 \times KF_{SiO_2} + 0.0035 \times KF_{Al_2O_3}\right)$ $dl = \left(W_{dF} - FR\right) / W_{dF}$ $FR = [(0.01784 \times KF_{CaO}) + (0.0209 \times KF_{MgO}) + (0.0135 \times KF_{Al_2O_3})$ $+(0.01075 \times KF_{SiO_2}) + (0.01 \times KF_{Fe_2O_3})] \times (\frac{100 - Cl_{Ig}}{100})$	160.31	4.38
Cooler air sensible heat	$\frac{CAB_{CO} \times c_{j_{air}} \times T_{amb}}{W_{Cl}}$ $CAB_{CO} = 4,654.44 \times V_{CO}$	46.92	1.28
Primary air sensible heat	$\frac{CAB_{Pa} \times c_{j_{air}} \times T_{amb}}{W_{Cl}}$ $CAB_{Pa} = 4,654.44 \times V_{Pa}$	76.97	2.1
Infiltrated air sensible heat	$\frac{AIH \times c_{j_{air}} \times T}{W_{Cl}}$ $AIH = 11,720.3 \times A_{ch} \times (1.157 \times P_h)^{0.5}$	15.75	0.46

Table 3. Total heat output of the kiln system

Heat Outputs	Formulation	Result	Percent
Clinker formation	$(4.11\times Cl_{Al_2O_3})+(6.48\times Cl_{MgO})+(7.646\times Cl_{CaO})$ $-(5.116\times Cl_{SiO_2})-(0.59\times Cl_{Fe_2O_3})$	1705.5	46.62
Kiln exit gas	$(W_{CO_2}\times c_{j_{CO_2}}\times T_{Be})+(W_{H_2O}\times c_{j_{H_2O}}\times T_{Be})+(W_{SO_2}\times c_{j_{SO_2}}\times T_{Be})$ $+(W_{N_2}\times c_{j_{N_2}}\times T_{Be})+(Add\ excess\ air\times c_{j_{air}}\times T_{Be})$ $W_{CO_2}=CP_{CO_2}+WGF_{CO_2}$ $W_{H_2O}=CP_{H_2O}+WGF_{H_2O_{free}}+WGF_{H_2O_{chem}}$ $W_{SO_2}=0.5\times CP_{SO_2}$ $Add\ excess\ air=\frac{EA}{100}\times(CP_{CO_2}+CP_{H_2O}+CP_{N_2})$ $CP_{CO_2}=W_A\times\begin{bmatrix}(1.97\times F_{CH_4})+(3.94\times F_{C_2H_6})+(5.9\times F_{C_3H_8})+\\(8.33\times F_{C_4H_{10}})+(9.64\times F_{C_5H_{12}})+(1.97\times F_{CO_2})\end{bmatrix}$ $WGF_{CO_2}=(1+\frac{dl}{2})\times\{(0.0078\times KF_{CaO})+(0.0109\times KF_{MgO})\}$ $CP_{H_2O}=W_A\times\begin{bmatrix}(1.6\times F_{CH_4})+(2.4\times F_{C_2H_6})+\\(3.14\times F_{C_3H_8})+(4.04\times F_{C_4H_{10}})+(5.05\times F_{C_5H_{12}})\end{bmatrix}$ $W_{N_2}=CP_{N_2}=W_A\times\begin{bmatrix}(9.55\times F_{CH_4})+(16.70\times F_{C_2H_6})+(23.86\times F_{C_3H_8})+\\(31.02\times F_{C_4H_{10}})+(38.19\times F_{C_5H_{12}})+(1.25\times F_{H_2})\end{bmatrix}$ $EA=189\times\left[(2.0\times G_{O_2})-G_{CO}\right]/\left[G_{N_2}-\left[1.89\times\left[(2.0\times G_{O_2})-G_{CO}\right]\right]\right]$	150.88	4.1
Moisture in feed or slurry	$(WGF_{H_2O_{free}}+WGF_{H_2O_{chem}})\times 2500.8$	94.14	2.5
Dust in the kiln exit gas	$Dl\times c_{j_{dust}}\times T_{Be}$ $Dl=W_{dF}-FR$	189.47	5.1
Clinker at cooler discharge	$c_{j_{clinker}}\times T_{Cl}$	172.2	4.7
Cooler stack	$\frac{CAB_{Ex}\times c_{j_{air}}\times T_{St}}{W_{Cl}}$ $CAB_{Ex}=4,654.44\times V_{Ex}$	501.2	13.69
Radiation on kiln shell	$\frac{A_{kiln}}{3}\times\frac{(q_{j_1}\times(T_{z_1}-T))+(q_{j_2}\times(T_{z_2}-T))+(q_{j_3}\times(T_{z_3}-T))}{W_{Cl}}$	123.84	3.3
Calcination wasted kiln dust	$\dot{q}\times TC_{dust}\times 1,592.5$ $\dot{q}=\frac{KF_{Ig}-D_{Ig}}{KF_{Ig}}$ $TC_{dust}=\frac{(0.01784\times KF_{CaO})+(0.0209\times KF_{MgO})}{FR}\times W_{dF}\times dl$	619.29	16.92
Convection from kiln surface	$h_{con}\times A_{kiln}\times(T_s-T_{Pa})$ wind speed = 3 m/s	21.96	
Radiation from pre-heater surface	$\varepsilon\times\sigma\times A_{preheater}\times(T_{preheater}{}^4-T_{Pa}{}^4)$	1.13	
Natural convection from pre-heater surface	$h_{ncon}\times A_{preheater}\times(T_{preheater}-T_{Pa})$	0.55	3.07
Radiation from cooler surface	$\varepsilon\times\sigma\times A_{cooler}\times(T_{cooler}{}^4-T_{Pa}{}^4)$	0.53	
Natural convection from cooler surface	$h_{ncon}\times A_{cooler}\times(T_{cooler}-T_{Pa})$	0.77	

The results of energy balance for layout of the curing part of the cement industry, including all input and output items, are calculated based on the above-mentioned formula in Table 2 and 3 and illustrated in a Sankey diagram in Fig. 2.

As can be seen, the bulk of the input energy comes from fuel combustion. The thermal energy consumption in the factory is 3,658.1 kJ per kg of clinker produced. The efficiency of the system is equal to 46.62%, which is relatively low.

2.1 Heat recovery from the kiln system

The kiln system efficiency is 46.62%, which is relatively low. The overall efficiency of the kiln system can be improved by recovering some of the heat losses. The recovered heat energy can be used for several purposes, such as electricity generation and preparation of hot water. There are a few major heat loss sources that would be considered for heat recovery: these are heat losses by the kiln exhaust gas (4.12%), and hot air from the cooler stack (13.7%). In the following, we discuss some possible methods of recovering this wasted heat energy.

There are opportunities in such a plant to capture waste heat to the environment and utilize this heat to generate electricity. The most feasible and, in turn, the most cost-effective waste heat losses available for such a purpose are the clinker cooler discharge and kiln exhaust gas. The exhaust gas from the kilns is, on average, 330°C, and the temperature of the discharged air from the cooler stack is 315°C. Both streams would be directed through a waste heat recovery steam generator (WHRSG), and the available energy is transferred to the water via the WHRSG. 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 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. Having the steaming rate and pressure, the size of the generator can be determined. The following calculations were used to find the size of the generator:

$$Q_{WHRSG}=Q_{available}\times\eta \quad (1)$$

where, η is the WHRSG efficiency.

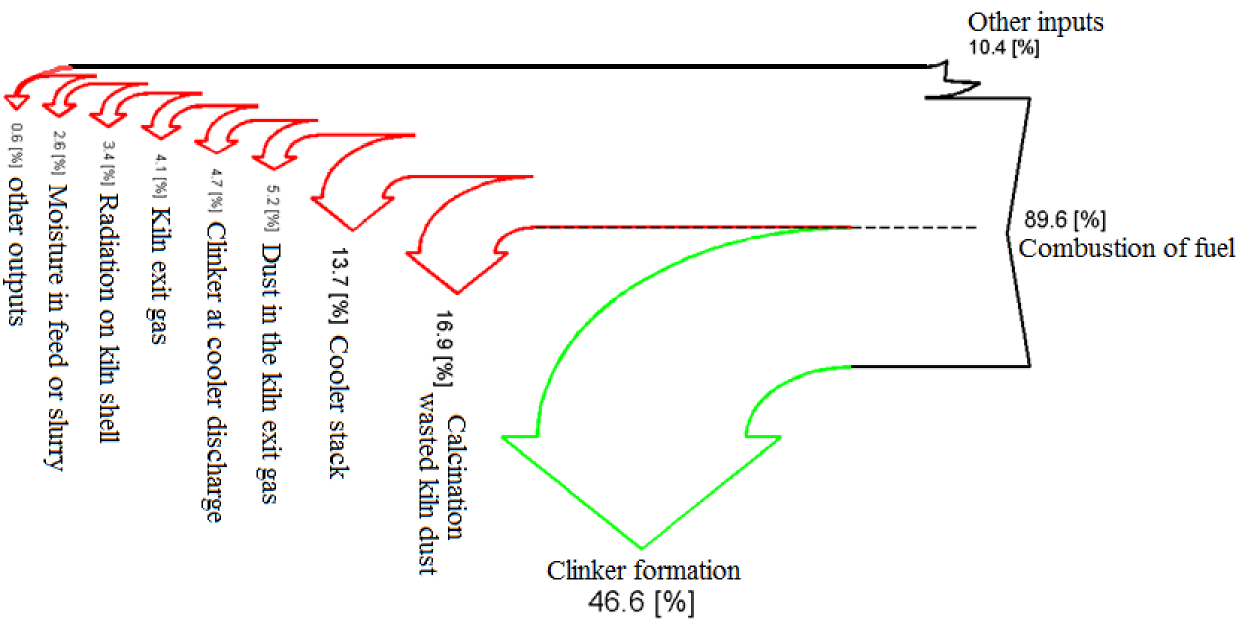

Fig. 2. The energy balance diagram (Sankey Diagram) for the layout of the curing part of the cement industry

Because of various losses and inefficiencies inherent in the transfer of energy from the gas stream to the circulating water within the WHRSG, not all of the available energy will be transferred. A reasonable estimate of 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. The WHRSG has 5 kg/s water at 800Pa and 40°C flowing through it, being heated from two sources. This control volume has a single inlet and exit flow with two heat transfer rates coming from reservoirs different to the ambient surroundings. The characteristics of the exit water are 800Pa and 180°C. The reversible work is obtained from (2) [30]:

$$w_{rev} = T_0(s_e - S_i) - (h_e - h_i) + q_1(1-\frac{T_0}{T_1}) + q_2(1-\frac{T_0}{T_2}) \quad (2)$$

From the steam tables, the inlet and exit state properties are h_i=167.54 kJ/kg, h_e=719.2 kJ/kg, s_i=0.5724 kJ/kg °K, s_e=2.0418 kJ/kg °K. The reversible work is:

$$w_{rev} = 293.2 \times (2.0418 - 0.5724) - (719.2 - 167.54) + (6{,}286.66/5) \times (1-\frac{293.2}{603.2}) + (20{,}883.33/5)(1-\frac{293.2}{588.2}) = 2{,}740.86 kJ/kg$$

Therefore, the available energy is:

$$Q_{available} = (2{,}740.86 kJ/kg) \times (5 kg/s) = 13{,}704.32 kW$$

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

$$Q_{WHSRG} = 0.85 \times 13{,}704.32 = 11{,}648.672 kW$$

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 5,000 kW. If we assume that the useful power generated is 5,000 kW, then the anticipated savings will be based on the load reduction of 5,000 kW. Assuming 8,000 h of usage, we find:

$$Energy\ saved = (5{,}000 kw) \times (8{,}000 h/yr) = 40 \times 10^6 kWh/yr$$

The average unit price of electricity can be taken as 0.15 USD/kWh, and therefore, the anticipated cost savings would be:

$$Cost\ saving = (4 \times 10^6) \times 0.15 = 6{,}000{,}000\ USD/yr$$

If we assume that labour and maintenance costs average out to 20,000 USD annually, the saved amount becomes 5,980,000 USD/yr. The cost associated with the implementation of this additional system would be the purchase price of the necessary equipment and its installation. An additional cost would be the required maintenance of the power generation unit. We estimate the required budget at between 5,600,000 and 6,000,000 USD, including shipping and installation. Hence, we can make a rough estimate for a simple payback period:

$$Simple\ pay\ back\ period = \frac{5{,}600{,}000\ USD}{5{,}980{,}000\ USD/yr} \approx 1\ yr$$

The energy savings made through 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 estimate and may vary depending upon plant conditions and other economic factors.

3. Electrical energy balance

Among the ways in which to improve energy efficiency in motor systems, the replacement of a low-efficiency motor with a high-efficiency one is recommended [31-34]. Before the determination of energy savings, it is necessary to know the real values of load and efficiency for each motor. The mathematical model [35] used for estimating the motor load has presented a correlation coefficient of 99.3% with real motor curves [34]. From the real measured current (I_R), nominal current (I_N) given by the manufacturer, and no load current (I_0), measured or given by manufacturer, the real load (γ) is determined by:

$$\gamma = 1 + \frac{1}{\alpha} Ln(\frac{I_R}{I_N}) \quad (3)$$

where the load current parameter is calculated by:

$$\alpha = -Ln(\frac{I_o}{I_N}) \quad (4)$$

The efficiency is the relation between output power and input power, including energy losses [36]. Thus, the real efficiency is given by:

$$\eta_L = \frac{P_{out}}{P_{in}} = \frac{P_N \gamma}{P_R} \quad (5)$$

where P_N is the nominal output power; P_R is the real input power; γ is the rated load (%); and η_L is the low efficiency (%).

The technical areas – production and maintenance – were selected for four studied motors operating at 6 kV, and 23 motors at 400 V. Motors' nominal data were collected and electric current and power measurements were taken at the motors' input line using a precision meter. The firm intends to replace the motors for others with the same power. The motors' data are shown in Table 4 and 5.

Table 4. Data from low efficiency motors at 400 V

Motors	Name	P_N	P_R	I_N	I_R	I_0	γ	η_L
M1.400	raw material separation fan 2	284	167	512	301.3	205	42.05	71.51
M2.400	cement separation fan 2	284	218	512	393.3	205	71.14	92.67
M3.400	kiln exhaust fan	144	139	260	250.8	104	96.14	99.60
M4.400	raw material separation 2	131	55	236	99.2	95	5.28	12.59
M5.400	cement electro filter fan 4	115	50	207	90.2	83	9.09	20.93
M6.400	the main kiln fan	110	84	198	151.5	79	70.57	92.41
M7.400	raw material separation 1	86	76	155	137.1	62	86.51	97.89
M8.400	cement filtering fan 2	81	45.5	146	82.1	58	37.06	65.97
M9.400	material stack filter fan 3	81	56.6	146	102.1	58	60.88	87.13
M10.400	primary kiln air fan	75	51.3	135	92.5	54	58.55	85.6
M11.400	raw material Q-pump compressor 1	160	85	288	153.3	115	30.97	58.29
M12.400	raw material Q-pump compressor 2	160	100	288	180.4	115	48.71	77.93
M13.400	feeding kiln Q-pump compressor	160	100	288	180.4	115	48.71	77.93
M14.400	feeding raw material Q-pump compressor 1	119	85	215	153.3	86	63.28	88.59
M15.400	feeding raw material Q-pump compressor 2	119	80	215	144.3	86	56.66	84.28
M16.400	air pole kiln compressor 1	119	80	215	144.3	86	56.66	84.28
M17.400	air pole kiln compressor 2	119	82	215	147.9	86	59.36	86.14
M18.400	reserved air pole kiln compressor	110	87	198	156.9	79	74.4	94.07
M19.400	air lift cement compressor	105	89.6	189	161.6	76	82.69	96.9
M20.400	air pole cement compressor 1	81	53	146	95.6	58	53.71	82.08
M21.400	air pole cement compressor 2	81	56.8	146	102.5	58	61.27	87.37
M22.400	material grinding compressor	68	58.5	123	105.5	49	83.58	97.15
M23.400	reserved material grinding compressor	68	30	123	54.1	49	10.69	24.24

Table 5. Data from low efficiency motors at 6kV

Motors	Name	P_N	P_R	I_N	I_R	I_0	γ	η_L
M1.6000	raw material grinding fan	1,000	700	120	84.2	48	61.07	87.25
M2.6000	cement grinding suction fan	280	200	34	24	13	63.28	88.59

3.1 Improving energy efficiency in an industrial motor system

The improvement in energy efficiency (IEE) indicates the percent of energy saved after the replacement of a low efficiency motor (η_L) with a high efficiency motor (η), and is calculated as follows:

$$IEE = (1 - \frac{\eta_L}{\eta}) \times 100 \qquad (6)$$

As the real low efficiency (η_L) and the real rated load (γ) were previously calculated using Eqs. (1) to (3), the real high efficiency was determined from the performance curve of the motor given by the manufacturer, considering the high efficiency (η). The improvement in energy efficiency (IEE) is determined by Eq. (6).

The real high efficiency is not necessarily the nominal efficiency of new motors, because this depends on real load, which varies as a function of the electric current.

The quantity of energy saved (QES) can be calculated as follows [32]:

$$QES = P_N \gamma t (\frac{1}{\eta_L} - \frac{1}{\eta}) \qquad (7)$$

where t is the operating time (h/yr.).

By the calculation of QES and considering the energy cost per kilowatt-hour (C), the energy saved value (ESV) is derived by the following formula[32]:

$$ESV = QES.C \qquad (8)$$

Considering the motor investment value (MIV) and the calculated ESV, the simple payback (SPB) is given by:

$$SPB = \frac{MIV}{ESV} \qquad (9)$$

The results are shown in Tables 6 and 7.

As can be seen, the payback period is less than one year for the 400 V motors, while for the 6kV motors it is more than two years. This is because the amount of investment in the 6 kV motors is high.

Table 6. Motor results at 400 V

Motors	η	IEE	MIV	QES	ESV	SPB
M1.400	94.5	24.32	47,340	355,868.13	53,380.22	0.88
M4.400	93.6	86.55	33,894	417,003.7	62,550.55	0.54
M5.400	93.6	77.63	21,882	340,059.33	51,008.89	0.42
M8.400	93.6	29.51	12,567	117,656.08	17,648.41	0.71
M11.400	93.6	37.72	29,802	280,855.65	42,128.35	0.71
M23.400	91.7	73.57	12,030	193,337.62	29,000.64	0.41

Table 7. Motor results at 6kV

Motors	η	IEE	MIV	QES	ESV	SPB
M1.6000	94.5	7.67	219,320	470,533.23	70,580	3.1
M4.6000	94.5	6.25	38,170	109,561.56	16,434.23	2.3

4. Multi-criteria decision method

In most recently published works, after performing an energy audit programme, different strategies are offered to reduce energy losses. However, generally these strategies are different from the perspective of economics and the amount of loss reduction, which makes it difficult to make a decision between them.

To make a decision in an organized way, so as to generate precedence, we need to separate the decision into the following steps [37].

1. Define the problem and determine the kind of knowledge sought.
2. Structure the decision hierarchy from the top with the goal of the decision, then the objectives from a broad perspective through the intermediate levels (the criteria on which subsequent elements depend) to the lowest level (which usually is a set of alternatives).
3. Construct a set of pairwise comparison matrices. Each element in an upper level is used to compare the elements in the level immediately below with respect to it.
4. Use the priorities obtained from the comparisons to weigh those in the level immediately below. Do this for every element. Then, for each element in the level below, add its weighed values and obtain its overall or global priority. Continue this process of weighing and adding until the final priorities of the alternatives in the bottom-most level are obtained.

To make comparisons, we need a scale of numbers that indicates how much more important or dominant one element is over another with respect to the criterion or property to which are compared. Table 8 exhibits the scale.

In this paper, some criteria used have been defined as follows:

1. Quantity of energy saved
2. Energy saved value
3. Payback period
4. Investment cost

Therefore, after consultation with specialist personnel in the plant, the initial weight of the criteria has been selected according to Table 9.

Table 8. The fundamental scale of absolute numbers [37]

Intensity of Importance	Definition
1	Equal Importance
2	Weak or slight
3	Moderate importance
4	Moderate plus
5	Strong importance
6	Strong plus
7	Very strong or demonstrated importance
8	Very, very strong
9	Extreme importance
Reciprocals of above	If activity i has one of the above non-zero numbers assigned to it when compared with activity j, then j has the reciprocal value when compared with i
1.1–1.9	If the activities are very close

Table 9. Initial weights for the selected criteria

criteria	1	2	3	4
1	1	1	0.5	0.2
2	1	1	0.5	0.2
3	2	2	1	0.33
4	5	5	3	1

By using the analytic hierarchy process (AHP), the final weights of the criteria were obtained according to Table 10.

When the weights of the various criteria were obtained, the weight of suggestions in terms of each criterion should be obtained. In Table 11, the weights of suggestions based on the criteria are presented.

Finally, using the AHP method, the weights of suggestions based on criteria are presented in Table 12.

Therefore, based on the results of Table 12, the priority of executing the suggestions is presented in the Table 13.

As is shown in table 13, the priorities of executing suggestions based on the mentioned criteria (quantity of energy saved, energy saved value, payback period, investment cost) are derived.

Table 10. The final weight criteria

criteria	1	2	3	4
weight	0.1092	0.1092	0.2155	0.5661

Table 11. The weight of different strategies for each criteria

Strategies	**WHRSG**	**M1.400**	**M4.400**	**M5.400**	**M8.400**	**M11.400**	**M23.400**	**M1.6000**	**M4.6000**
first criteria	0.4632	0.0834	0.0976	0.0797	0.028	0.066	0.0456	0.11	0.0262
second criteria	0.4632	0.0834	0.0976	0.0797	0.028	0.066	0.0456	0.11	0.0262
third criteria	0.0993	0.0874	0.05362	0.0417	0.0705	0.0705	0.0407	0.3078	0.2284
fourth criteria	0.0091	0.0853	0.1021	0.1344	0.2019	0.1102	0.209	0.0523	0.0955

Table 12. Weight of each strategy by using the AHP method

Strategies	**WHRSG**	**M1.400**	**M4.400**	**M5.400**	**M8.400**	**M11.400**	**M23.400**	**M1.6000**	**M4.6000**
weight	0.1277	0.0853	0.0906	0.1025	0.1356	0.0919	0.137	0.1199	0.1090

Table 13. The priority list

Rank	**Strategies**
1	M23.400
2	M8.400
3	M1.6000
4	WHRSG
5	M4.6000
6	M5.400
7	M11.400
8	M4.400

5. Conclusion

A detailed energy audit analysis that can be directly applied to any dry kiln system has been formulated 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 as to the reasons for low overall system efficiency. According to the results obtained, the system efficiency is 46.62%. The major heat loss sources have been determined as hot air from the cooler stack (13.7%) and kiln exhaust gas (4.12%). A conventional WHRSG system is proposed for the losses: calculations showed that 5 MW of energy could be recovered. Then, based on defined criteria, the priority of the suggestions was obtained using the analytic hierarchy (AHP) process.

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