

**Table 5.2.22 Lighting equipment and utilization factor**

Reflec- tion rate	Ceiling	80%				70%				50%				30%				0%
	Wall	70	50	30	10	70	50	30	10	70	50	30	10	70	50	30	10	0%
	Floor	10%				10%				10%				10%				0%
Room index		Utilization factor ( × 0.01)																
0.6	51	43	38	34	50	42	37	34	48	41	37	33	46	40	36	33	32	
0.8	60	52	47	44	59	52	47	44	56	50	46	43	54	49	45	43	41	
1.0	65	59	54	50	64	58	53	50	61	56	52	50	59	55	52	49	47	
1.25	69	64	60	56	68	63	59	56	66	61	58	55	63	60	57	54	52	
1.5	72	67	63	60	71	66	63	60	68	64	61	59	66	63	60	58	56	
2.0	76	72	68	66	74	71	68	65	72	69	66	64	69	67	65	63	60	
2.5	78	74	72	69	76	73	71	68	74	71	69	67	71	69	67	66	63	
3.0	79	76	74	72	78	75	73	71	75	73	71	69	73	71	69	68	65	
4.0	81	79	77	75	79	77	76	74	77	75	74	72	74	73	72	71	68	
5.0	82	80	78	77	81	79	77	76	78	76	75	74	75	74	73	72	69	
7.0	83	82	80	79	82	80	79	78	79	78	77	76	76	76	75	74	71	
10.0	84	83	82	81	83	82	81	80	80	79	78	78	77	77	76	76	72	

$$K = \frac{A}{H \times (X + Y)} \quad (2-2)$$

where;

K : Room index

H : Height of light source from working face (m) : 6.6 – 0.85 = 5.75 m

X : Width of room (m) : 54 m

Y : Length of room (m) : 72 m

From equation (2-2),

$$K = \frac{3,888}{5.75 \times (54 + 72)} = 5.4$$

When the reflection rates of ceiling, walls and floor is assumed as zero, the utilization factor is 0.69 from Table 5.2.22.

Therefore, the number of lighting equipment is 54 from equation (2-1).

$$N = \frac{170 \times 3,888}{26,500 \times 0.69 \times 0.66} = 54.8$$

It is recommended that these lighting equipment are laid in six rows with nine equipment in each row as matched with the locations of curing presses. The mean illuminance will be about 250 Lx as shown below.

$$E = \frac{F \times N \times U}{A} = \frac{26,500 \times 54 \times 0.69}{3,888}$$

$$= 254 \text{ Lx}$$

As the initial illuminance will drop as the working time elapses, cleaning or replacement of lamps should be made when the illuminance drops to 170 Lx, which is obtained by multiplying the maintenance factor to the initial illuminance.

The power consumption of current fluorescent lamps is as follows.

$$40 \text{ W} \times 3 \times 408 = 49 \text{ kW}$$

It will be as follows after the lamps are changed to high pressure sodium lamps.

$$220 \text{ W} \times 54 = 12 \text{ kW}$$

Therefore, the annual electric power saving amount is as follows.

$$(49 - 12) \times 7,920 = 293,000 \text{ kWh/y}$$

The electric power expenses to be saved are as follows.

$$293,000 \text{ kWh/y} \times 3.69 \text{ Ft/kWh} = 1,081,000 \text{ Ft/y}$$

The expenses required for this improvement are as follows in Japan.

Expenses per lighting equipment

Lamp	¥22,700
Lighting equipment	¥5,400
Stabilizer	¥9,500
Wiring and working	¥20,000
<hr/>	
Total	¥57,600

The total for 54 lighting equipment is 3,110,000 yen (equivalent to 1,555,000 Ft) which can be recovered in about 1.5 years.

(5) Total of effect of improvement

Item	Expected Saving								Investment 1000 Ft	Payback Year	
	Steam				Power						Total
	t/y	1000 Ft/y	%	kWh/y	1000F t/y	%	1000 Ft/y	1000 Ft			y
Curing Press											
Insulation Reinforcement	1,500	1,125	1.6				1,125	1,680	1.5		
Insulation of Pipe	5,000	3,750	5.4				3,750	1,200	0.3		
Steam Trap Maintenance	491	368	0.5				368	0	0.0		
Repairing Steam Leakage	376	282	0.4				282	0	0.0		
Steam Distribution Line											
Insulation of Pipe, Valve	998	749	1.1				749	185	0.2		
Electric Power											
Compressor Press Reduct'n				40,500	149	0.2	149	0	0.0		
High Efficiency Lamp				293,000	1,081	1.7	1,081	1,555	1.4		
Total	8,365	6,274	9.0	333,500	1,230	1.9	7,504	4,620	0.6		



### 5.3 Results of investigation at an alumina factory



### 5.3 Results of investigation at an alumina factory

#### 5.3.1 Outline of the factory

- (1) Company name and factory name : HUNGALU
- (2) Category of business : Chemical industry, alumina production business
- (3) Principal product name and production capacity  
Principal products : Metallurgical alumina and special alumina  
Production capacity : 330,000 t/y
- (4) No. of employees : 1,291
- (5) Location of factory : H-2931 Almásfüzitő Fő út 1
- (6) History of the factory

This company is a large enterprise having 13 production divisions, sales department and engineering department in all the fields related to aluminum ranging from bauxite mining to alumina production, aluminum refining and production of processed goods.

This factory belongs to the alumina production division of the company. A place located along River Donau and close to the bauxite mine was selected for the factory and construction was determined in 1938. But construction was suspended due to the influence of the war, and production was commenced in 1950 with alumina production capacity of 60,000 t/y. The capacity was sequentially increased since then, reinforcing the equipment and improving the production technologies. The capacity reached the level of 330,000 t/y in 1971.

The aluminum trade fell into depression after turning to the 1980's, and sales is stagnant these days because of adjustment accompanying liberalization of the economy. In such circumstances, this factory made efforts in the upbringing of engineers and development of technologies. In addition, the factory improved the equipment to what are competitive in the international market, and made major achievements in the reduction of unit consumption of raw materials and utilities. Recently, the factory added six desiliconizing tanks, two flash tanks and two heat exchangers and improved the steam unit consumption by about 3%.

However, the Hungarian-Soviet Alumina and Aluminium Agreement was expired at the end of 1990, and the factory was forced to deal with trade in the free market. Furthermore, severeness of the economic environment was increased. Accordingly, the production level dropped to the level of 170,000 - 210,000 t/y. The earnings dropped to 1/10 due to drop in the alumina prices, rise of energy prices and drop in the quality of domestically produced bauxite, in addition to the increase of the fixed expenses.

The factory is therefore urged to further reduce expenses including reduction of personnel expenses. As a subject in the future, the factory is planning to increase the ratio of production of special alumina with higher value added.

The company was reformed to a private corporation as the first step of conversion to a private sector in July, 1991. But all the shares of the company are still held by a state-run assets holding company.

(7) Investigation period August 26 - August 30, 1991

(8) Investigators

Mr. Mitsuo Iguchi	Leader
Mr. Teruo Nakagawa	Subleader
Mr. Tatehiro Tanabe	Alumina Process Engineer
Mr. Koichi Inaba	Heat Control Engineer
Mr. Toshiyuki Ochi	Heat Control Engineer
Mr. Kenichi Kurita	Electrical Control Engineer

(9) Interviewees

Dr. Miklós Schlégel	Deputy General Manager
Mr. Ferenc Galba	Section Chief
Mr. Imre Ottohál	Mechanical Engineer
Mr. Péter Gögös	Electrical Engineer
Mr. Lajos Szebenyi	Chief of Power Station
Mr. József B. Tóth	Section Chief of Raw Material
Mr. Imre Domján	Energy Engineer
Mr. Pál Lángfy	Energy Engineer
Mr. Sandor Pirik	Head Foreman, Calciner Section

(10) Progress of production (Table 5.3.1)

Name of Product		1986	1987	1988	1989	1990
Metallurgical Alumina	t	313,208	315,215	317,400	320,710	306,650
Special Alumina	t	6,800	9,000	12,600	13,300	17,512
Total	t	320,008	324,215	330,000	334,010	324,202



(11) Progress of energy consumption (Table 5.3.2)

		1986	1987	1988	1989	1990
Fuel Oil	kt	114.7	116.5	122.3	123.9	118.4
Diesel Oil	t	457	560	540	474	150
Coal	t	15,720	13,369	4,112	100	2,318
Power	MWh	113,188	114,146	116,540	123,975	116,041
Generated Power	MWh	48,197	49,732	49,216	50,573	52,197

(12) Energy unit consumption (Table 5.3.3)

		1986	1987	1988	1989	1990
Fuel Oil	GJ/t	14.9	14.8	15.0	14.9	13.3
Electric Power	kWh/t	354	352	353	370	363
Total Energy	GJ/t	16.2	16.1	16.3	16.2	15.4

(13) Operating hours (Table 5.3.4)

	1986	1987	1988	1989	1990
Annual Operating Hours	8,760	8,760	8,760	8,760	8,760

Reference operating hours for examination of countermeasures

$$24 \text{ hours/day} \times 365 \text{ days/year} = 8,760 \text{ hours/year}$$

(14) Energy prices

Fuel oil                      7,000 - 10,000 Ft/kl

Electric power              3.5 Ft/kWh

(15) Factory layout drawing (Figure 5.3.1)

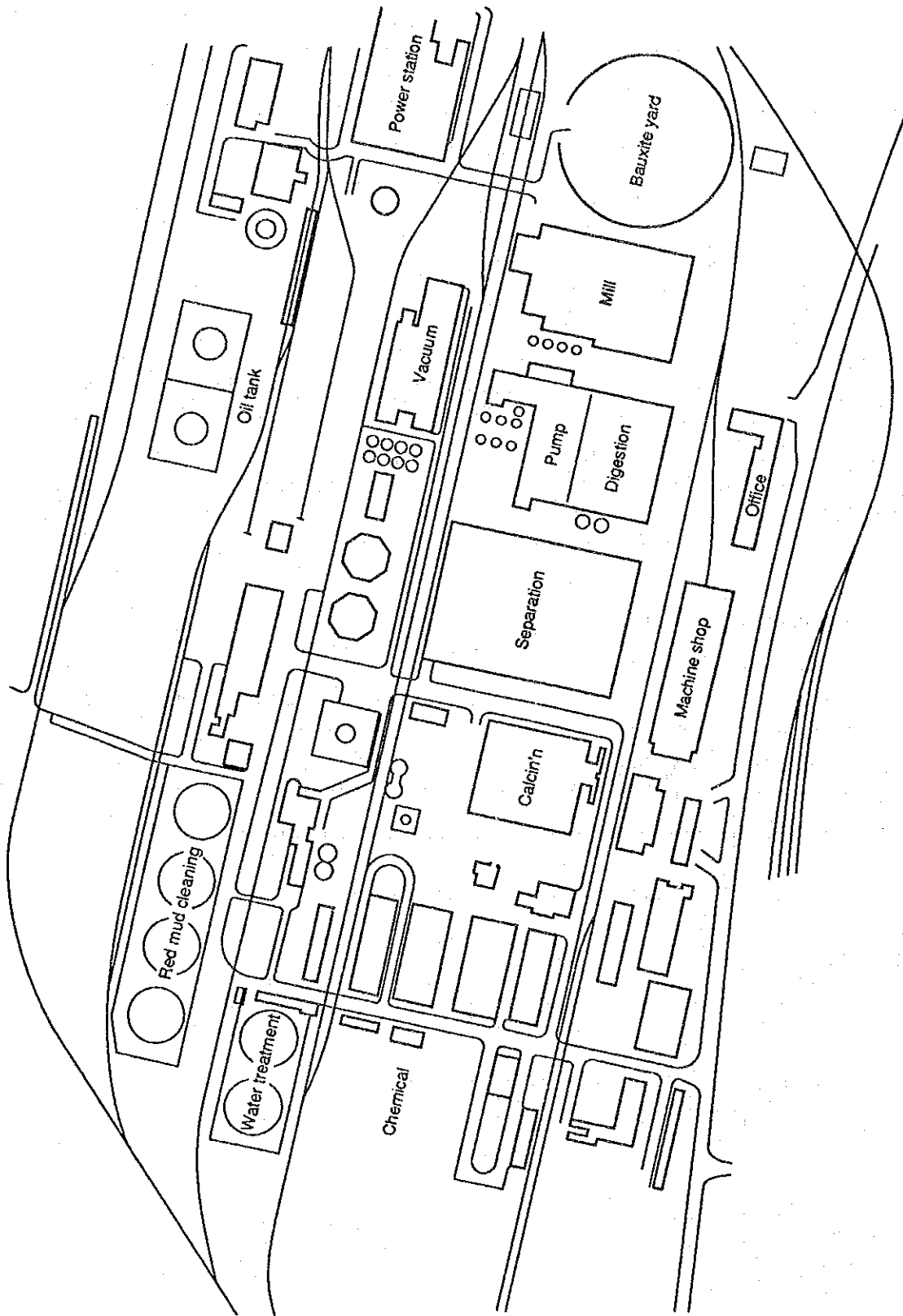


Figure 5.3.1

(16) Manufacturing processes (Figure 5.3.2)

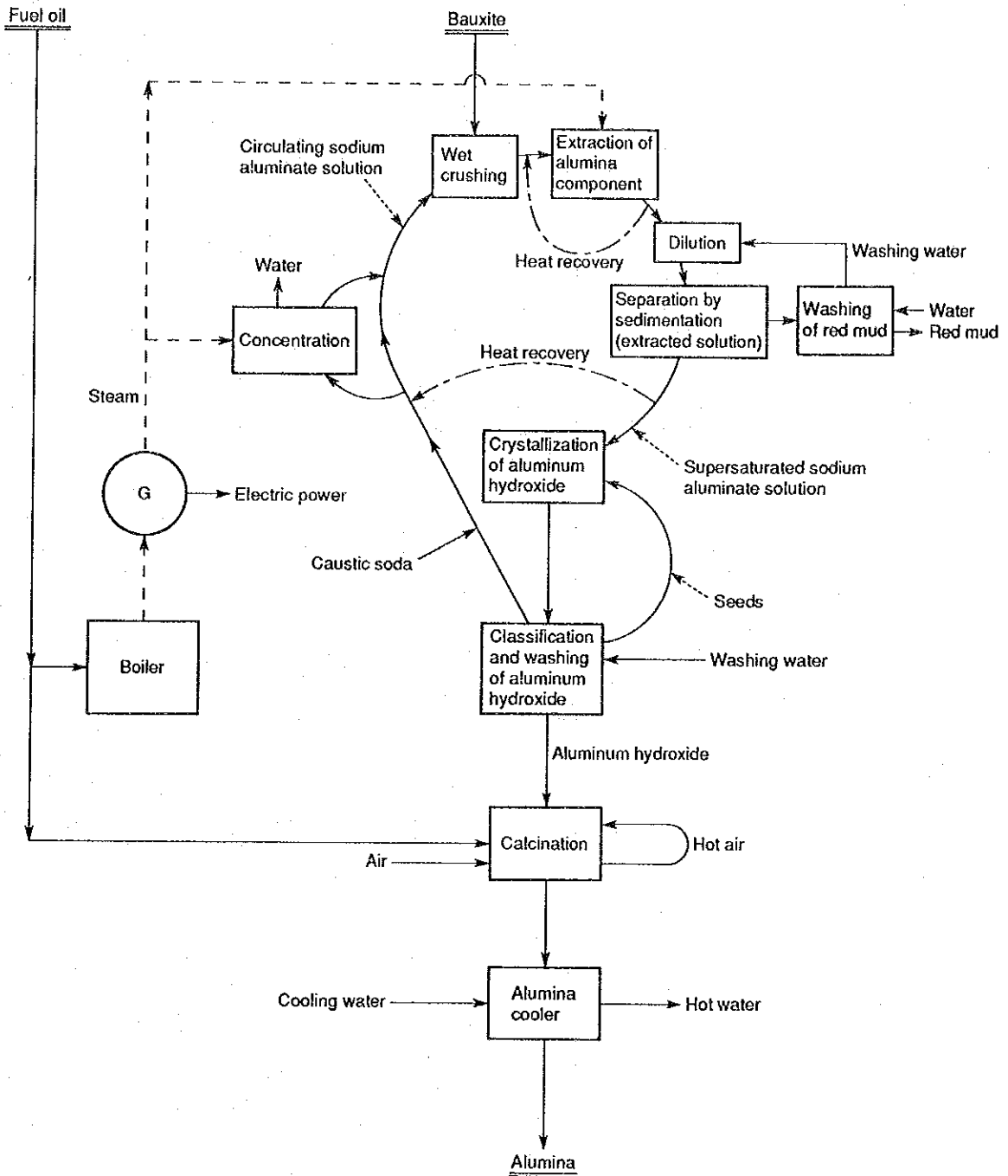


Figure 5.3.2

(17) Electric power one line diagram (Figure 5.3.3)

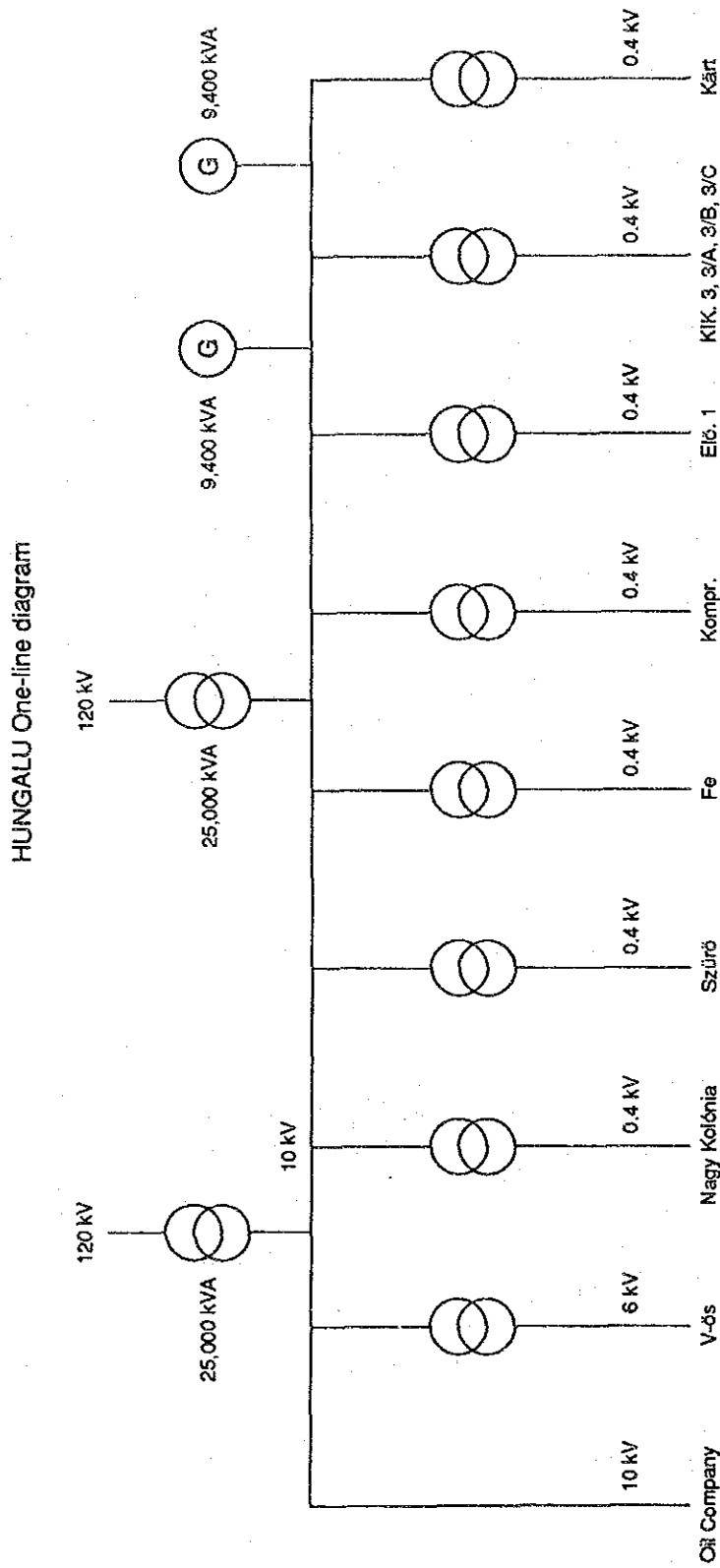


Figure 5.3.3

(18) Outline of principal equipment (Table 5.3.5)

Name	Number	Specification
Digester	4 sets	$22 \text{ m}^3 \times 8 + 50 \text{ m}^3 \times 3$ Heat Transfer Surface Area = $100 \text{ m}^2$
Flash Tank	4 sets	$2.5 \text{ m}\phi \times 4.974 \text{ mH} \times 9 + 3.15 \text{ m}\phi \times 9.0 \text{ mH} \times 2$
Heat Exchanger	4 sets	Heat Transfer Surface Area = $72 \text{ m}^2 \times 7 + 16$
Liquor Thickener	10	$960 \sim 1,500 \text{ m}^3$
Precipitator	104	$400 \sim 600 \text{ m}^3 \times 99 + 2,600 \text{ m}^3 \times 5$
Calciner	2 4	$3.8 \text{ m}\phi \times 50 \text{ mL}$ , 650 t/d, with Cyclone Preheater $2.4 \text{ m}\phi \times 50 \text{ mL}$
Concentrator	9	Quadruple Effect
Boiler	4	$80 \text{ t/h} \times 3 + 40 \text{ t/h} \times 1$

### 5.3.2 Situations of energy management

#### (1) Setting the target of energy saving

This factory has been making efforts in the improvement of unit consumption in correspondence to worsening of the situations of the economy that surrounds the trade. Integration of vacuum pumps was done and flash tanks and heat exchangers were added in the previous year.

The factory officers seize fear of inflation, adaptation to market economy and dealing with drop in the product prices accompanying depression in the aluminum industry, and recognize that further reduction of expenses is required including reduction of personnel for survival.

Probably because the situations are unstable, long-term target of energy saving has not been set up. But the target of unit consumption is established at the time of planning of the annual production program once every quarter. The plan observed during investigation was such that, with cooperation of engineers assumed, the steam unit consumption would be reduced from 3.1 t/t in the third quarter to 2.95 t/t in the fourth quarter. As for the fuel for calcination, a level-off plan of 100 kg/t was established taking into account the condition that the production is reduced to 50%.

It is desirable that unit consumption planned values are indicated in a familiar form to the field employees. It is favorable that this factory also sets up a planned value for each department.

The factory seems to consider that there is little room for improvement unless new investment is made. But if the policy of reducing output is continued for a long time, the attitude is necessary to strive for the optimum operating conditions that correspond to the reduced production and to challenge improvement of unit consumption, regardless of the conventional work standard. Improvement of equipment by investment of a large scale is important for energy saving, but build-up of minor improvements such as review of operating conditions and uplift of the management level is also important. Such actions, therefore, will produce larger effect than expected.

#### (2) Systematic activities

To achieve the target, it is necessary to make the desire of the management side and concrete target values known to the employees. At this factory, cooperation in the energy conservation activities is appealed at the technology meeting held once every quarter as attended by about 50 employees and with the vice factory superintendent acting as the chairman and at the operator's meeting held at each workshop thereafter.

A section that supports and promotes energy conservation activities of each workshop has been organized, and five persons are working in this section. Rearrangement of data related to energy saving also belongs to the scope of work of this section.

However, such a structure has not yet been established in which energy conservation activities are implemented toward a common target with all the employees of the factory.

(3) Management by data

It is fundamentally necessary to seize the realities of energy consumption, to compare it with the plan and to take corrective measures upon troubleshooting if there is any abnormality. The result data of the factory constitute variable information for improvement of the energy unit consumption.

Consumption of fuel and electric energy is recorded with computers daily and is reported to the factory superintendent at this factory. The result values are analyzed by calculation of unit consumption. The results of calculation are collated with the unit consumption reference diagram that corresponds to the operation rate. If any abnormality is recognized, the energy managing department reports it to the applicable workshop with a request to take suitable corrective measures.

No commendation system for the results of energy saving of workshops is provided. But many proposals for improvement are submitted from employees. There also are cases where proposals are submitted from operators. But no commendation system is provided even for these improvement proposals. To promote energy conservation actions by giving incentive and by awakening competition consciousness of workshops is also an effective method.

There were occasions when a heat flow diagram of the entire factory was drawn up spending three months before, and university professors are asked to give advice every year.

(4) Enlightenment of employees

Consumption of energy is often affected by the behaviors of humans. It is therefore necessary to instruct correct working methods to operators and to allow them to observe these methods. Furthermore, when it is intended to allow the operators to improve current working methods and equipment, it is necessary to enlighten them with the trend of the trade, typical cases of improvement and technical knowledge to a certain extent.

The staff in charge of promotion of energy conservation attends the training course held by the Ministry of Commerce and Industry every year. Engineers participate in training courses for one to two weeks held by AEEF and Industrial Reeducation Center. Operators attend courses for training on safe methods for handling oil and gas, operation of burners, compressors and so forth. These training courses do not aim at energy saving only, but these courses are considered to be indirectly useful for promotion of energy saving.

(5) Management of equipment

It is necessary to make efforts in the maintenance of equipment to cause full exhibition of functions of equipment and to minimize energy losses. Drop-off of heat insulation, and leakage of steam, liquid and product solids were observed at this factory. We were explained that it is unavoidable to reduce the repair expenses under the current economic circumstances. But it is desirable that consolidation is made by making use of the leisure personnel with expenditure minimized. Because, the monetary losses caused by such leakage are large.

### 5.3.3 Problems in the use of energy and countermeasures

(1) Energy cycle system and recent results of energy consumption

A) Energy cycle system

Bayer's process is broadly adopted worldwide as a method for production of alumina from bauxite, and it is also adopted at this factory. The points of this process are as follows.

- a) Ground bauxite slurry is steam heated in alkali solution, and sodium aluminate ( $\text{NaAlO}_2$ ) solution is produced by dissolving the alumina component in the bauxite in the alkali solution. Eleven digesters are used and indirect heating is made with superheated steam in order to secure the temperature and time required for dissolving alumina.
- b) Silica acid ( $\text{SiO}_2$ ) in the bauxite is dissolved in the alkali solution, makes reaction with the alkali and alumina component in the solution, and hardly soluble sodalite type compound ( $\text{Na}_2\text{O} \cdot \text{Al}_2\text{O}_3 \cdot 2\text{SiO}_2 \cdot x\text{H}_2\text{O}$ ). It is separated from the sodium aluminate solution as extracted residue, together with insoluble components such as iron oxide and titan oxide.
- c) Crystals of aluminum hydroxide is crystallized by hydrolysis of sodium aluminate solution.
- d) Crystals of aluminum hydroxide are calcined to eliminate crystallization water, and alumina ( $\text{Al}_2\text{O}_3$ ) is obtained.

This process may be roughly divided into two processes, i.e., liquid process (including slurry) (a, b, c) and powder process (d).

The heat energy cycle system in the liquid process is described below.

The steam generated in the boiler is used to heat slurry in digesters for extracting the alumina component from the bauxite. Furthermore, steam is also used for thickening the diluted solution, in order to use the sodium aluminate solution after crystallization of aluminum hydroxide as recycled.



A heat recovery system is incorporated in the liquid process so that heat is used effectively without waste. For example, the heat possessed by the slurry solution of high temperature after extracting reaction is used for preheating slurry of low temperature to be charged to the digesters by the flash vapor generated in the process of cooling by pressure reduction. Thus, a system of good efficiency is constituted as a whole.

Furthermore, the overflow solution from the red mud sedimentation thickener is cooled to the specified temperature of the first Crystallization. Thus heat is recovered using a plate heat exchanger and is used for heating the solution entering the concentration process.

These points are shown in Figure 5.3.2, diagram of manufacturing processes.

B) Recent results of energy consumption

a. Energy consumption (second quarter of 1991)

**Table 5.3.6 Energy consumption in second quarter of 1991**

Period	April-June, 1991
Alumina production	55,000 t
Fuel oil for calcination	103.8 kg/t- $\text{Al}_2\text{O}_3$
Fuel oil for boiler	297 kg/t- $\text{Al}_2\text{O}_3$
Purchased electric energy	264 kwh/t- $\text{Al}_2\text{O}_3$
Fuel oil heating value (H1)	40 GJ/t
Steam for digestion (40 bar 400°C)	1,711 kg/t- $\text{Al}_2\text{O}_3$
Steam for concentration (3.5 bar)	1,363 kg/t- $\text{Al}_2\text{O}_3$

b. Energy unit consumption

**Table 5.3.7 Energy unit consumption in second quarter of 1991**

Energy	GJ/t- $\text{Al}_2\text{O}_3$
Fuel oil for calcination	$0.1038 \times 40 = 4.15$
Fuel oil for boiler	$0.297 \times 40 = 11.88$
Purchased electric energy	$264 \times 3.6 \div 1000 = 0.95$
Total	16.98

c. Energy consumption by process

**Table 5.3.8 Energy consumption by process**

Process	GJ/t-Al <sub>2</sub> O <sub>3</sub>	%
Digestion process	$1,711 \times 3,217 \times 10^{-6} = 5.50$	32.4
Concentration process	$1,363 \times 2,742 \times 10^{-6} = 3.74$	22.0
Calcination process	4.15	24.4
Others	3.59	21.2
<b>Total</b>	<b>16.98</b>	<b>100.0</b>

The results of energy unit consumption in the past were good as shown in Table 5.3.3. But the energy unit consumption in the second quarter of 1991 became 16.98 GJ/t-Al<sub>2</sub>O<sub>3</sub> probably because of reduction of production.

When the breakdown is observed, the energy consumption in the calcination process is relatively good. The reason is thought considerable contribution of the functions of the cyclone preheater.

(2) Digestion process

The majority of the bauxite used at this factory comes from Hungary. Its alumina composition is as shown in Table 5.3.9. (Average value of 1990)

**Table 5.3.9 Bauxite composition**

Boehmite	Al <sub>2</sub> O <sub>3</sub> ·H <sub>2</sub> O	27.2%
Gibbsite	Al <sub>2</sub> O <sub>3</sub> ·3H <sub>2</sub> O	14.4%
Diaspor	H·AlO <sub>2</sub>	1.6%
Alumogoeite	A100H	0.9%
Kaolin		5.7%
<b>Total</b>		<b>49.8%</b>

This bauxite is of large boehmite content, and the digestion temperature should be set at a high level because of this reason. While the digestion temperature for bauxite of gibbsite type is 110 - 150°C, the digestion setup temperature at this factory is as high as 240°C.

The thermal energy used in the digestion process occupies 32.4% of the total energy consumption as stated earlier.

The digestion process is composed of digesters, flash tanks and preheater, and is composed of four systems. The specification and quantity of these components are shown in Table 5.3.5.

The alumina production during the investigation was 11,000 t/month. Since it was about 1/3 of the production capacity, No. 1 system only was running.

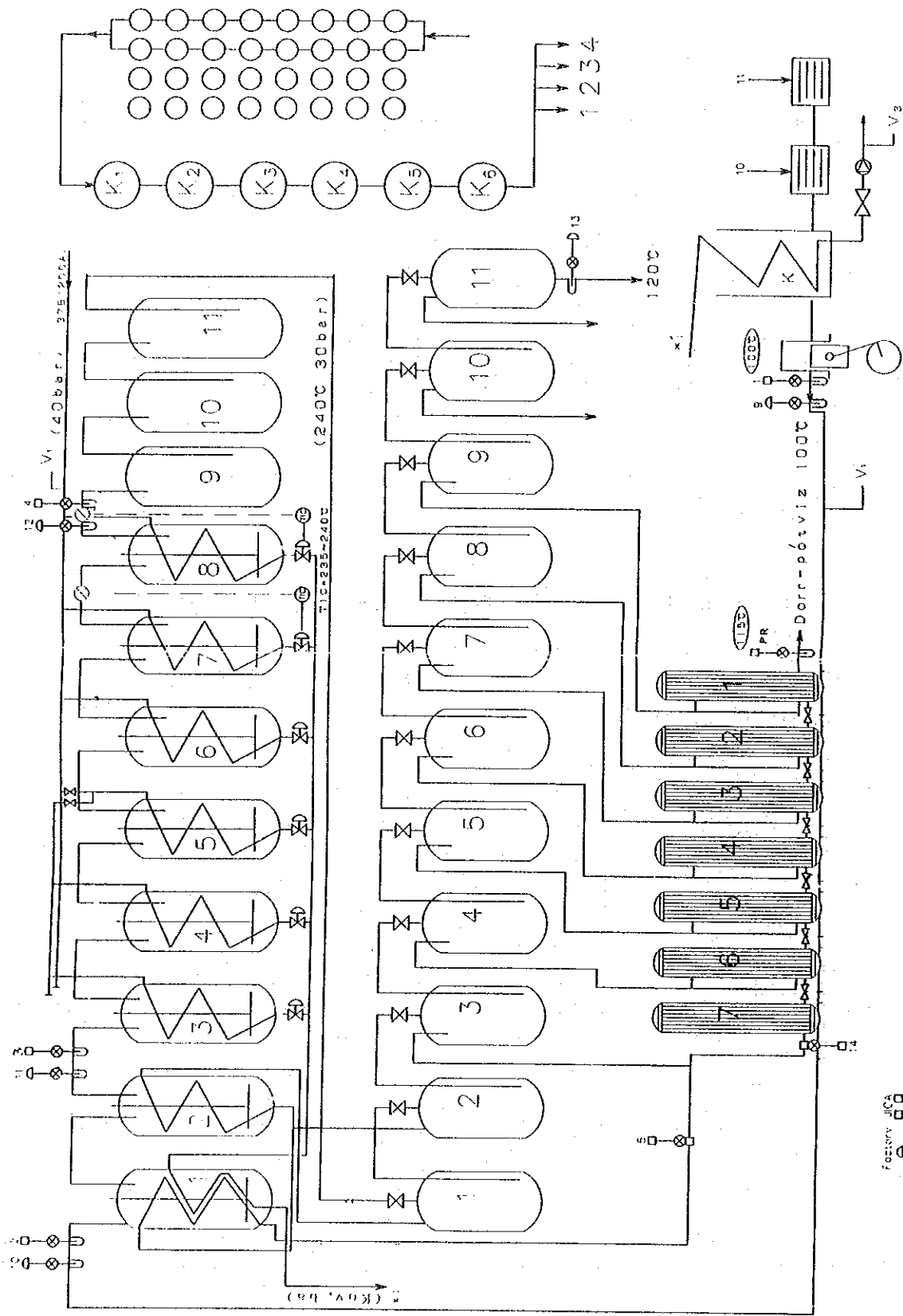
A) Results of measurement of temperature

Measurement of slurry temperature was made at 12:00 of August 28 to 8:30 of August 29, and the readings of the thermometer brought by the investigation team were compared with the readings of the thermometer installed in the factory.

The points of installation of thermometers are shown in Figure 5.3.4 and the measured values are shown in Table 5.3.10.

**Table 5.3.10 Results of measurement of slurry temperature (°C)**

Measuring Point		8/28		8/29		Average
		16:00	21:00	5:00	7:00	
Preheater Inlet	J1	97.9	100.6	99.8	99.2	99.4
	F9	97.1	100.2	99.2	98.7	98.8
Digester Inlet	J2	162.0	162.2	162.3	162.1	162.2
	F10	164.4	164.4	164.1	164.4	164.3
Digester No. 2 - No. 3	J3	176.7	176.2	176.8	175.7	176.4
	F11	179.8	180.0	179.6	179.4	179.7
Digester after No. 8	J4	236.7	236.2	236.8	235.8	236.4
	F12	238.7	238.6	238.5	238.5	238.6



Factory JICA  
 51000  
 5K

AUG 28 12:00-29.5:30

Figure 5.3.4 Points of installation of thermometers

In the results of measurement of temperature, the temperature at the outlet of No. 8 digester was measured as 238.6°C with the thermometer of the factory and was 236.6°C with the thermometer brought by the investigation team, while the set temperature is 240°C. Thus, the measured temperature was lower than the set temperature by 1.4 - 3.4°C. It is necessary to determine the optimum digestion temperature from the dissolving and desiliconizing curve and to execute control to observe this temperature.

#### B) Heat balance

Since this process handles corrosive liquid of high temperature and high pressure, measurement of temperature, pressure and so forth could be made at limited measuring points only. But trial calculation of the heat balance of the system was implemented partly using measured values of outside surface temperature of equipment for reference. The results are shown in Table 5.3.11.

Calculation was made in the following sequence, using Lotus 1-2-3 spreadsheet software, by assigning heat transfer and flash calculation formulas to coordinates. The steam status characteristic used is what was obtained by an approximate expression referred pressure reference. The expressions written at coordinates are enumerated in the latter half of the table.

1. The pressure and temperature of superheated steam (SHS) were set up.
2. Slurry pressure, preheater inlet temperature, heating final temperature, boiling point rise, flow rate, specific heat and specific gravity were set up, and the pressure of SHS condensate was set up.
3. The heat transfer areas of digesters and preheaters were set up.
4. Heat losses at digesters, preheaters, etc. were set up.

The value that is two times the total surface area of principal equipment was determined as the heat radiation area including pipelines in this calculation. It was assumed that equipment is heat insulated with mineral wool of 150 mm, and it was also assumed that heat insulation has dropped off from preheaters by 50% and from others by 20%.

5. The flash tank pressure was input from the lower value so that the preheater temperature becomes the estimated temperature.

Originally, the flash pressure is determined by the heat transfer rate of the preheater, but such a method was adopted for convenience because the overall heat transfer coefficient was not known.

6. The overall heat transfer coefficient (U) of No. 2 - No. 8 digesters was input so that it matches with the temperature rise curve estimated from the results of measured surface temperature of equipment.

7. The SHS value obtained as a result of calculation was input. (This is the measure for preventing circulating calculation.)
8. The surplus heat of No. 1 digester was input to No. 7 preheater. (This is also the measure for preventing circulating calculation.)
9. The sequence of 5 - 8 was repeated until convergence of SHS value, etc. occurs.

There may be incorrect points because the investigation period was short this time. But it is possible to estimate various conditions by changing set values and coefficients in expressions in this table.

Naturally, the steam consumption decreases when the heat transfer rate at preheaters is increased and the digester inlet temperature is elevated.

Table 5.3.11 Heat balance (1/3)

	B	C	D	E	F	G	H	I	J	K	L	M	N	O	P	Q	R	S	T	U
1	Steam										Case									
2	MPa abs	39.1	4.01	390	3193	249.4	1083	3200			Date	82.3.19								
3	h <sup>o</sup> kJ/kg										Note	Set temp. to observed data Present State								
4	Slurry																			
5	Press		3.10 MPa		30 bar															
6	Sat. temp		235.57 °C																	
7	B.P.		245.57 °C																	
8	B.P.R.		10 °C																	
9	Sp.Heat		3.56 kJ/kg.K		0.85 kcal/kg.°C															
10	Flow		210 t/h		145 m <sup>3</sup> /h															
11	Na2O		190 t/h		1.45 kg/l															
12																				
13																				
14	Preheate		HTA=	72 m <sup>2</sup>																
15	Slurry																			
16	Temp		100	74809																
17	1		110.1	82341	8431	23.2	15219	23650	74	26.2	16044	7532	2082							
18	2		118.6	88736	7473	20.5	14245	21717	104	28.2	15219	6895	1739							
19	3		128.3	95855	8774	17.3	12820	21594	120	20.5	14245	7229	1981							
20	4		136.8	102827	8063	14.4	11255	19318	137	17.3	12820	6362	1714							
21	5		145.8	109084	8902	11.3	9255	18157	145	14.4	11255	6757	1818							
22	6		153.3	116179	8753	7.8	6752	16505	154	11.3	9255	7095	1900							
23	7		164.5	123042	7649	5.1	4570	12218	154	7.8	6752	5302	1400							154.474
24																				
25	Reactor		HTA=	100 m <sup>2</sup>																
26	Slurry																			
27	Temp		172.1	128746	4761	28501	3115	24069	205	7102										
28	1		180.1	134736	3485	28501	3115	3115	229	6141										
29	2		197.9	148066					274											
30	3		211.0	157850					309											
31	4		220.7	165128					337											
32	5		227.8	170378					358											
33	6		234.8	175683					379											
34	7		238.5	178494					392											
35	Other Heat Loss			3535																
36	Flash Tank In			174958																
37	Condensate																			
38	MPa abs																			
39	bar		18	1.90	210	2781	389	19508												
40	Sat. Temp																			
41	h <sup>o</sup> kJ/kg																			
42	Heat to Preheater																			1548
																				1560-R29

Table 5.3.11 Heat balance (2/3)

Flash Tank		Slurry		Flash Vapor		h'		h		Sat. Temp		Latent H		Sp.Vol		Vapor Speed		Flash Tank	
bar	Mpa abs	Temp	t/h	MJ/h	t/h	MJ/h	kJ/kg	kJ/kg	kJ/kg.K	kJ/kg.K	kJ/kg	kJ/kg	m <sup>3</sup> /kg	m <sup>3</sup> /h	MJ/h	bar	Calc'd	Obs'd	m <sup>3</sup> /h
43	20.0	2.10	225	206.9	168474	3.4	3485	2808	215	3.12	1855	0.094	317	6870	20				
44	18.2	1.92	220	205.2	160713	1.7	4761	2811	210	3.01	1860	0.102	173	7307	18.2				
45	15.5	1.65	212	202.5	153064	2.7	7648	2812	202	2.83	1916	0.118	322		15.5				110.1
46	12.4	1.34	202	199.0	143312	3.5	9753	2807	192	2.72	1958	0.145	502		12.4				118.6
47	9.9	1.09	193	195.8	134410	3.2	8902	2799	183	2.59	1994	0.176	561		9.9				128.3
48	7.9	0.89	184	192.9	126847	2.9	8088	2782	174	2.49	2028	0.214	619		7.9				136.8
49	6.0	0.70	174	189.8	117573	3.2	8774	2784	164	2.39	2085	0.270	851		6.0				145.8
50	4.6	0.56	165	187.1	110101	2.7	7473	2777	155	2.32	2097	0.335	900		4.6				155.3
51	3.3	0.43	155	184.0	101668	3.0	8431	2767	145	2.25	2133	0.436	1328		3.3				164.5
52	1.5	0.25	138	178.9	87596	5.1	14078	2743	128	2.16	2187	0.725	3721		1.5				
53	0.5	0.15	123	174.7	76272	4.2	11324	2714	113	2.11	2224	1.182	4933		0.5				
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Table 5.3.11 Heat balance (3/3)

	D11=F11+H11
	C18=D18/(SDS11+SDS10)
	D18=D17+L18
No9 FT	E18=I56
	F18=J19
	G18=F18+K55
	H18=E18+G18
	J18=J19+H56
	K18=J18+K56
	L18=H18-I18-K18
	M18=L18+I000/(E56-(C17+C18)/2)*SDS14
	D24=D23+L24+S29
	C29=M29
	D29=SDS11*SDS10+C29
No2 FT	E29=I49
	F29=G26+SHS4
	G29=H30
	H29=(G26+G42)+(H48+H49)+K49
	J29=E29+F29+G29-H29-I29
	N29=(M29+D26*(E48-C24/2)/1000+D24)/(SDS11*SDS10+M29+D26/2000)
	P29=M29+D26*(E44-(C24+C29)/2)/1000
	Q29=D29-D24
	R29=J29+J30-P29-P30
	C30=0.1F(N30+C36, N30, C36)
No1 FT	E30=I48
	H30=K48+H48
	J30=E30-H30-I30
	N30=(M30+SDS26*(E48-C29/2)/1000+D29)/(SDS11*SDS10+M30+SDS26/2000)
UAΔt	P30=M30+SDS26*(E48-(C29+C30)/2)/1000
	K31=L31/(SFS4-SHS4)
	L31=D31-D30+I31
	M36=L36+I000/(D26*((E4-N36)+(G4-N35))/2)
	N36=C36
	K37=0SUM(K31, K36)
	L37=0SUM(L31, L36)
	D38=D36-D37
bar→MPa	D42=C42*0.1+0.1013
TEMP	E42=179.03+D42^0.2452
r+h	F42=2287-453*D42+246*D42^2-80*D42^3+9*D42^4+SGS42
h' kJ/kg	G42=761.758+D42^0.2571
	H42=K37+SGS42
	C48=R48
	E48=L48+I0
	F48=D11-H48
	G48=F48+SDS10+E48
	H48=(D38-SDS11*SDS10+E48)/(J48-SDS10+E48)
	I48=H48+J48
	J48=K48+H48+M48+I0
	M48=2.0325+0.5111458*D48
	O48=0.19181/D48^0.96301
	P48=H48+O48+I000
	Q48=H48*(J48-K48)
	T48=P30
	F49=F48-H49
	H49=(G48-F48+SDS10+E49)/(J49-SDS10+E49)
	Q49=H49*(J49-K49)+G26*(SHS4-G42)+H48*(K48-K49)
	S50=C18
	E63=D17
	E64=SFS4+K37
	E65=0SUM(E63, E64)
	E67=G58
	E68=H42
	E69=K18
	E70=I57+I58
	E71=E72-0SUM(E67, E70)
	E72=E65
	F63=E63/SES65*100

C) Prevention of heat radiation from pipelines and equipment

Heat insulation of digesters was good in general. But there were some points where heat insulation has dropped off, and the total area of points without heat insulation was about 10 m<sup>2</sup>. Heat insulation of flash tanks was good.

The top and bottom covers, as well as flanges of all of No. 1 - 7 preheaters, were without heat insulation. Although covers are removed at the occasion of wash-off, but they can be handled even if they are heat insulated. Permanent heat insulation cannot be applied to flanges, but it is recommended that heat resistant cloth with aluminum film on the surface are suspended from hooks to cover up the body, instead of allowing heat radiation in the bare state. Although it is not complete heat insulation, it is effective to a certain extent. It is also convenient for mounting and removal.

The pipelines for steam, condensate and slurry were heat insulated. But there were some points such as pipelines around steam headers where no heat insulation is made.

The results of calculation of the heat radiation rates of places without heat insulation and of the quantity of heat that can be saved when heat insulation is made are shown in Table 5.3.12 for equipment, in Table 5.3.13 for pipelines and in Table 5.3.14 for valves.

The reducible quantity of heat related to equipment is 1,244.1 MJ/h. It appears as reduction of superheated steam flow and increase of low pressure surplus flash steam flow. The fuel quantity to be reduced and the amount to be saved as converted into superheated steam flow can be calculated as follows.

Here, the enthalpy of the superheated steam is assumed as 3,193 kJ/kg and the evaporation multiple at boilers is assumed as 12.24 kg/kg-Fuel.

$$1,244,100 \text{ kJ/h} \times 8,760 \text{ h/y} \times (1 / 3,193) \text{ kg/kJ} \times (1 / 12.24) \text{ kg/kg} = 278,900 \text{ kg/y}$$
$$278,900 \text{ kg/y} \times 10 \text{ Ft/kg} = 2,789,000 \text{ Ft/y}$$

The expenses required for this heat insulation are estimated as about 5,000,000 yen (equivalent to about 2,500,000 Ft) in Japan. But since the equipment is used by switching, heat insulation is required for two systems, and the total amount is 5,000,000 Ft. Therefore, these expenses can be recovered in 1.8 years.

Also with steam headers, the quantity of heat that can be reduced by heat insulation is 642.1 MJ/h, and the annual fuel saving quantity and amount are as follows.

$$642,100 \text{ kJ/h} \times 8,760 \text{ h/y} \times (1 / 3,193) \text{ kg/kJ} \times (1 / 12.24) \text{ kg/kg} = 143,900 \text{ kg/h}$$
$$143,900 \text{ kg/y} \times 10 \text{ Ft/kg} = 1,439,000 \text{ Ft/y}$$

Table 5.3.12 Heat radiation rates of non-heat insulated portions of equipment and effect of heat insulation

Location	Inner Temp	Surface Area sq.m	Present Heat Loss		Heat Loss after insulation improved				Saved Energy MJ/h	Note		
			Thick mm	kJ/(m <sup>2</sup> h)	MJ/h	Material	Thick mm	Surface Temp			kJ/(m <sup>2</sup> h)	MJ/h
Digester Side	180	10.0	0	9,519	95.2	Mineral Wool	150	47	164.8	1.6	93.5	
Preheater	140	12.3	0	5,894	72.5		150	45	109.3	1.3	71.2	
Flange	140	29.9	0	5,894	176.2		150	45	109.3	3.3	173.0	
Total		52.2			343.9					6.3	337.7	

Method for calculation : See Guideline.

Preconditions for calculation: 1. The room temperature is 40°C.

2. The surface radiation rate is 0.9 at bare portions and is 0.5 at heat insulated portions.

3. The representative length is 5 m.

Table 5.3.13 Heat radiation rates of non-heat insulated pipelines and effect of heat insulation

Location	Inner Temp °C	Pipe		Heat Loss Present		Heat Loss after Insulation				Saved Energy MJ/h	Note
		Dia inch	Length m	kJ/(mh)	MJ/h	Material	Thick mm	kJ/(mh)	MJ/h		
Header A	400	8	0.3	32,115	9.6	Mineral Wool	260	502	0.2	9.5	
	400	6	1	24,932	24.9	Mineral Wool	250	431	0.4	24.5	
	400	5	0.9	21,323	19.2	Mineral Wool	240	402	0.4	18.8	
Header B	400	8	0.5	32,115	16.1	Mineral Wool	260	502	0.3	15.8	
	400	6	1.3	24,932	32.4	Mineral Wool	250	431	0.6	31.9	
	400	5	0.3	21,323	6.4	Mineral Wool	240	402	0.1	6.3	
	400	3	0.5	13,998	7.0	Mineral Wool	210	343	0.2	6.8	
Header A-B	400	3	8	13,998	112.0	Mineral Wool	210	343	2.7	109.2	
Sub Total					227.6				4.8	222.8	
Flash Tank	180	16	9.9	11,223	111.1	Mineral Wool	180	218	2.2	108.9	
Preheater	140	8	14	3,914	54.8	Mineral Wool	150	117	1.6	53.2	
Sub total					165.9				3.8	162.1	
Total					393.5				8.6	384.9	

Method for calculation : See Guideline.

- Preconditions for calculation:
1. The superheated steam temperature is 400°C, the flash tank internal temperature is 180°C, and the preheater internal temperature is 140°C.
  2. The room temperature is 40°C and the outdoor temperature is 20°C.
  3. The heat insulation thickness was determined with the economical thickness in Japan used as a reference.
  4. The surfacre radiation rate is 0.9 at bare portions and is 0.5 at heat insulated portions.

Table 5.3.14 Heat radiation rates of non-heat insulated valves and effect of heat insulation

Location	Inner Temp °C	Valve		Heat Loss Present		Heat Loss after insulation				Saved Energy MJ/h	Note	
		Dia inch	Number	Equ. L m	MJ/h	kJ/(m <sup>2</sup> h)	Material	Thick mm	kJ/(m <sup>2</sup> h)			MJ/h
Header A	400	8	1	1.87	60.1	32,115	Mineral Wool	260	502	0.9	59.1	
	400	6	1	1.78	44.4	24,932	Mineral Wool	250	431	0.8	43.6	
	400	5	3	1.78	113.9	21,323	Mineral Wool	240	402	2.1	111.7	
Header B	400	8	1	1.87	60.1	32,115	Mineral Wool	260	502	0.9	59.1	
	400	6	2	1.78	88.8	24,932	Mineral Wool	250	431	1.5	87.2	
	400	5	1	1.78	38.0	21,323	Mineral Wool	240	402	0.7	37.2	
	400	3	1	1.56	21.8	13,998	Mineral Wool	210	343	0.5	21.3	
Sub Total					426.9						419.3	
Digester Inlet	400	6	6	1.78	266.3	24,932	Mineral Wool	250	431	4.6	261.7	
Condensate	231	4	8	1.58	71.7	5,672	Mineral Wool	160	180	2.3	69.4	
Flash Tank Vap	180	8	11	1.87	137.3	6,672	Mineral Wool	160	180	3.7	133.6	
Slurry	180	8	11	1.87	134.2	6,526	Mineral Wool	160	180	3.7	130.5	
Preheater Vap	140	8	14	1.87	102.5	3,914	Mineral Wool	150	117	3.1	99.4	
Slurry	140	8	7	1.87	51.2	3,914	Mineral Wool	150	117	1.5	49.7	
Sub total					763.2						744.3	
Total					1,190.1						1,163.6	

Method for calculation : See Guideline.

Preconditions for calculation: 1. The room temperature is 40°C and the outdoor temperature is 20°C

2. The surface radiation rate is 0.9 at bare portions and is 0.5 at heat insulated portions.

The expenses required for this heat insulation are 442,000 yen for pipelines, 2,973,000 yen for valves, 3,415,000 yen in total (equivalent to 1,710,000 Ft) in Japan. And the expenses can be recovered in about 1.2 years.

When heat insulation is removed for piping work, it is necessary that restoration is made quickly for sure. If the heat insulation work is performed by the employees with materials only purchased, the cost is less and the length of time required for recovery will be shortened.

#### D) Prevention of steam leakage

Leakage of steam continued by a considerable rate during the entire investigation period of five days from a flange (pipe diameter 125 mm) of a pipe header of a 40 bar steam line to a digester. Leakage of steam at other points was relatively minor. Even minor leakage, when it is left unremedied, will develop to major leakage because wear to the leaking point rapidly makes progress due to the steam of the jetting rate that exceeds the sonic speed. Therefore, repair to leaking points should be made early.

The steam flow rate that leaks through a hole of diameter 1 mm is calculated by the following equation.

$$G = 71.64 \times 10^4 \times A \times \sqrt{P/v''}$$

where;

G : Leaking steam flow rate [kg/h]

A : Area of opening [m<sup>2</sup>]

P : Steam pressure [kg/cm<sup>3</sup>·abs]

v'' : Steam specific volume [m<sup>3</sup>/kg]

As the steam pressure is 40 bar,

$$\begin{aligned} G &= 71.64 \times 10^4 \times 0.785 \times 10^{-6} \times \sqrt{(40 \times 1.0197 + 1.033) / 0.04833} \\ &= 16.5 \text{ kg/h} \end{aligned}$$

the leaking steam flow rate is 16.5 kg/h. It means that steam of the following quantity was wasted during the investigation period of five days.

$$16.5 \times 24 \times 5 = 1,980 \text{ kg}$$

Many hard-handling substances such as alkali solution, slurry, powder and lumps of high and low temperature are handled at the alumina factory besides steam of high temperature. Prevention of leakage is not easy, but leakage of these substances means that the energy applied to them is wasted besides resources. And in addition, there are many disadvantages from the viewpoints of safety, cleanness of the factory and production cost. It is recommended that periodic consolidation is implemented for preventing leakage.

For implementation of periodic consolidation, at first gather required data such as repair interval, scale of repair, repair points as well as service life, specification and quantity of parts, and prepare a periodic consolidation plan.

Based on this plan, perform repair of the scheduled points once every two or three months, with the entire process or a part of the process shut down, in the case of the liquid process. It is desirable that the shut-down time at a time is 12 to 18 hours.

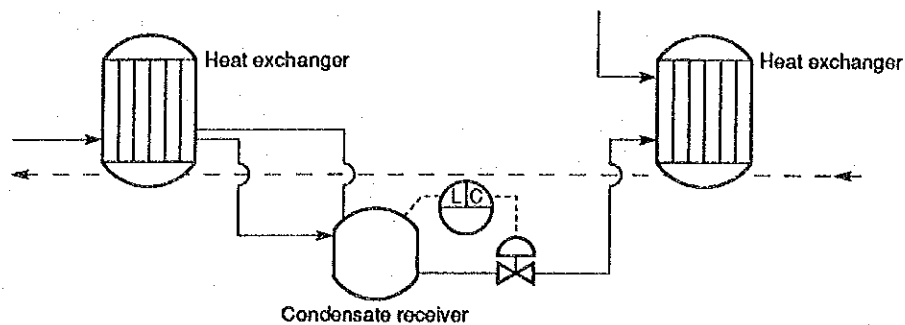
The equipment with which troubles such as sedimentation occur by suspension of operation should be locally kept operated to the minimum extent. With digesters, sedimentation tanks and precipitations tanks, for instance, keep the agitators running. For repair to a process other than the digestion process, discontinue input of bauxite to the digestion process and keep circulation of slurry only. Even for repair in the digestion process, if the repair point permits repair when input of bauxite is discontinued, and further, if repair can be made within a short length of time, perform repair in such a state. If repair cannot be implemented unless the slurry is discharged, switch the system.

What is most important for implementation of periodic consolidation is to establish the scheme in which the repair department and the production department keep close communication and provide mutual cooperation.

#### E) Other improvements

For controlling extraction of condensate from preheaters in the digestion process, float type regulators are provided for No. 1 system and No. 4 system, and hydrostatic regulators are provided for No. 2 system and No. 3 system.

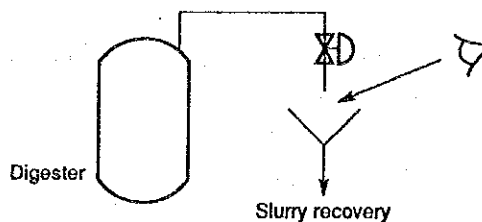
The method of less problem used in general is the method to use a condensate receiver with float type level controller such as what is shown in Figure 5.3.5.



**Figure 5.3.5 Condensate receiver**

Discharge of gases from digesters and preheaters is described next. When digesters and preheaters are filled with  $H_2$  gas and so forth generated by decomposition of organic matter in the sodium aluminate solution, their effective volume is reduced and the effective heat transfer area is also reduced. Therefore, these gases should be discharged.

Attach a gas bleeder to a digester, and discharge the gases once a day by opening the valve manually. It is recommended that a simple device that permits safe identification by eye observation of the fact that discharge of gases was completed is fitted as shown in Figure 5.3.6.



**Figure 5.3.6 Pipe for gas discharge from digester**

Since each preheater is provided with a gas bleeder connecting tube, it is recommended that the gas bleeder pipe of the last one is led into the sodium aluminate solution of low temperature so that the gases are discharged to the atmosphere through the solution. If the valve of the gas bleeder pipe is kept open by a minor extent, gas discharge and heat recovery can be made without manual work.



(3) Calcination process

A) Heat balance

Two large size calciners and four small size calciners are installed. Measurement was taken with No. 2 (large size) calciner. The measuring points are shown in Figure 5.3.7, and the results of measurement are shown in Table 5.3.15.

a. Results of measurement

Table 5.3.15 Results of measurement

Calciner		No. 2	
Measuring time		8/29 16:00 - 8/30 8:40	
Calculated basic data collecting time		8/30 8:00	
Atmospheric temperature	$t_0$	18.7	°C
Aluminum hydroxide			
Charge rate	$F_1$	22.3	t/h (wet)
Rate as converted to alumina	$F_2$	13.14	t/h
Adhered moisture	w	9.87	%
Temperature	$t_b$	60	°C (estimated)
Fuel oil			
Flow rate	f	1,632	l/h
Specific gravity	$\rho_f$	0.929	
Temperature	$t_f$	60.8	°C
Heating value	$H_f$	40.0	MJ/kg
Exhaust gases			
Temperature	$t_g$	148.2	°C
Flow velocity in smoke stack	u	7.88	m/sec
O <sub>2</sub> concentration		5.03	%
Smoke stack bore	D	1.915	m
Product alumina			
Cooler outlet temperature	$t_a$	300.3	°C
Al <sub>2</sub> O <sub>3</sub> content		89.8	%

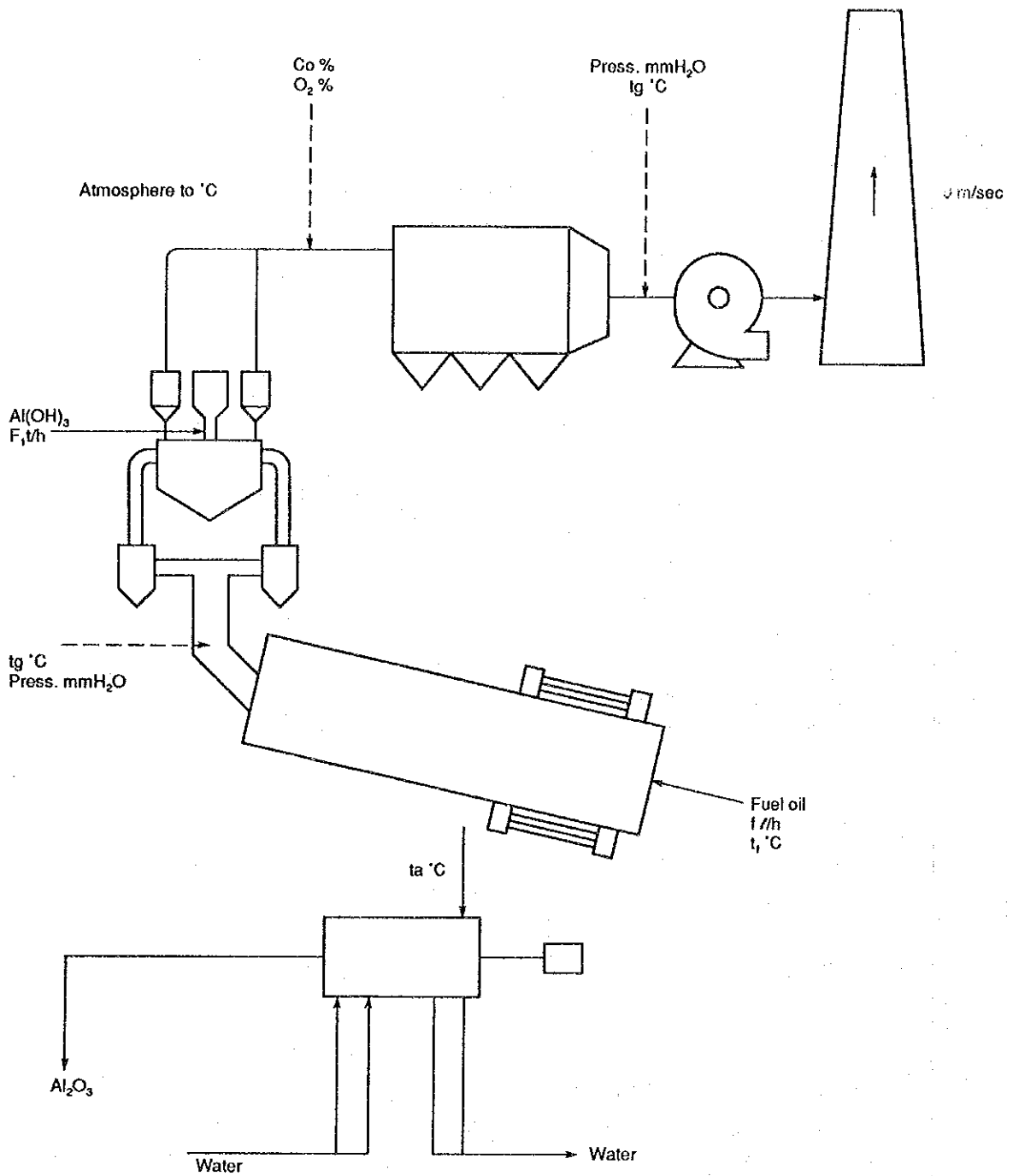


Figure 5.3.7 Calciner measuring points

b. Heat balance chart

Table 5.3.16 Heat balance of calciner

Heat input			kJ/kg-fuel	kJ/kg-Al <sub>2</sub> O <sub>3</sub>	%
1	Fuel oil heating value	H <sub>1</sub>	40,000	4,615	98.5
2	Sensible heat of fuel oil	Q <sub>1</sub>	79	9	0.2
3	Sensible heat of aluminum hydroxide	Q <sub>2</sub>	510	59	1.3
Heat input total			40,589	4,683	100.0
Heat output			kJ/kg-fuel	kJ/kg-Al <sub>2</sub> O <sub>3</sub>	%
1	Theoretical reaction heat	L <sub>1</sub>	14,659	1,691	36.1
2	Exhaust gas loss	L <sub>2</sub>	9,659	1,114	23.8
3	Sensible heat of alumina	L <sub>3</sub>	2,050	236	5.1
4	Radiation and other losses	L <sub>4</sub>	14,221	1,641	35.0
Heat output total			40,589	4,683	100.0

c. Calculation of heat balance

Calculation was made per kg of fuel oil.

For conversion to the value per kg of alumina, it is needed to divide by 8.668.

$$\frac{F_2}{f \times \rho_f} = \frac{13.14 \times 10^3}{1632 \times 0.929} = 8.668$$

[Heat input]

- ① Fuel oil heating value: H<sub>1</sub>

$$H_1 = 40,000 \text{ kJ/kg-fuel}$$

- ② Fuel oil sensible heat: Q<sub>1</sub>

$$Q_1 = (t_r - t_o) \times C_f = (60.8 - 18.7) \times 1.88 \\ = 79 \text{ kJ/kg-fuel}$$

Here, the fuel oil specific heat (C<sub>f</sub>) is assumed as 1.88 kJ/(kg·°C).

③ Sensible heat of  $\text{Al}(\text{OH})_3$ :  $Q_2$

$$Q_2 = \frac{F_1 \times C_{al} \times (t_h - t_o)}{f \times \rho_f} = \frac{22.3 \times 10^3 \times 0.84 \times (60 - 18.7)}{1,632 \times 0.929}$$

$$= 510 \text{ kJ/kg - fuel}$$

Here, the specific heat ( $C_{al}$ ) of  $\alpha\text{-Al}(\text{OH})_3$  is assumed as  $0.84 \text{ kJ}/(\text{kg} \cdot ^\circ\text{C})$ .

Heat input total:  $H_1 + Q_1 + Q_2 = 40,589 \text{ kJ/kg-fuel}$

[Heat output]

① Theoretical reaction heat:  $L_1$

$h$  : Reaction heat of  $\alpha\text{-Al}(\text{OH})_3$  89.8% =  $-172.5 \times 10^3 \text{ kJ/kg mol-Al}_2\text{O}_3$

$$L_1 = \frac{F_1 \times (1 - w/100) \times h}{78 \times 2 \times f \times \rho_f} = \frac{22.3 \times 10^6 \times (1 - 0.0987) \times 172.5}{78 \times 2 \times 1,632 \times 0.929}$$

$$= 14,659 \text{ kJ / kg - fuel}$$

② Exhaust gas loss:  $L_2 = l_1 + l_2$

Combustion exhaust gas loss:  $l_1$

Exhaust gas flow

$$W = \frac{D^2 \times \pi \times u \times 3,600 \times 273}{4 \times (273 + t_g)}$$

$$= \frac{1.915^2 \times \pi \times 7.88 \times 3,600 \times 273}{4 \times (273 + 148.2)} = 52,935 \text{ Nm}^3 / \text{h}$$

$$l_1 = \frac{W \times C_g \times (t_g - t_o)}{f \times \rho_f} = \frac{52,935 \times 1.38 \times (148.2 - 18.7)}{1,632 \times 0.929}$$

$$= 6,240 \text{ kJ / kg - fuel}$$

where;  $C_g$ : Exhaust gas specific heat  $1.38 \text{ kJ}/(\text{Nm}^3 \cdot ^\circ\text{C})$

Moisture vaporization heat:  $l_2$

Attached moisture

$$= \frac{F_1 \times 10^3 \times w / 100}{f \times \rho_f} = \frac{22.3 \times 10^3 \times 0.0987}{1,632 \times 0.929}$$

$$= 1.45 \text{ kg / kg - fuel}$$

$$I_2 = 1.45 \times 2,358 = 3,419 \text{ kJ/kg-fuel}$$

where: Vaporization latent heat of water of 60°C: 2,358 kJ/kg

$$L_2 = I_1 + I_2 = 6,240 + 3,419 \\ = 9,659 \text{ kJ/kg-fuel}$$

③ Sensible heat of alumina:  $L_3$

$$L_3 = \frac{F_2 \times 10^3 \times C_{a2} \times (t_a - t_o)}{f \times \rho_f} = \frac{13.14 \times 10^3 \times 0.84 \times (300.3 - 18.7)}{1,632 \times 0.929}$$

$$= 2,050 \text{ kJ / kg - fuel}$$

where;  $C_{a2}$ : Specific heat of alumina 0.84 kJ/(kg·°C)

④ Radiation and other losses:  $L_4$

$$L_4 = (\text{Heat input total}) - (L_1 + L_2 + L_3) \\ = 40,589 - (14,659 + 9,659 + 2,050) \\ = 14,221 \text{ kJ/kg-fuel;}$$

Unit consumption of fuel oil for calcination =  $f \times \rho_f \times H_1 / F_2$

$$= \frac{1,632 \times 0.929 \times 40,000}{13.14 \times 10^6} = 4.62 \text{ GJ / t - Al}_2\text{O}_3$$

d. Consideration of heat balance chart

It is considered that the share of the bring-out heat of  $\text{Al}_2\text{O}_3$  is small. But it is because of the fact that the alumina temperature at the planetary cooler outlet was measured at a low level of 300.3°C. There is a possibility that this measured value does not represent the true temperature of alumina but the temperature of the air containing alumina is measured. In practice, it is considered that the  $\text{Al}_2\text{O}_3$  bring-out heat is larger and other radiation losses are smaller.

As the O<sub>2</sub>% in the exhaust gases was 5.03%, the air ratio is 1.32, which is in the normal range.

The ratio of the calciners' calcination rate to the production capacity at the time of the investigation was as follows.

$$\frac{13.14 \text{ t/h} \times 24 \text{ h} \times 100}{560 \text{ t/d}} = 56.3\%$$

The fuel oil consumption per ton of alumina was 4.62 GJ/t-Al<sub>2</sub>O<sub>3</sub>. It is learned that the consumption increased due to reduction of production when compared with the figure of 4.15 GJ/t-Al<sub>2</sub>O<sub>3</sub> of the second quarter of 1991.

B) Countermeasures against temperature rise of combustion air

a. Reduction of suction cold air

Holes of diameter about 15 cm were found at two places in the vicinity of the burner of No. 2 calciner. Cold air in the room is sucked at a high speed through these holes. As a result, the portion that should be originally preheated in the planetary cooler is not preheated, but recovery of waste heat is reduced. Calculation of this quantity of heat is indicated below.

Air ratio: *m* When the N<sub>2</sub> content in the exhaust gases is assumed as 79%, the air ratio is as follows.

$$\begin{aligned} m &= 21 / (21 - 5.03) \\ &= 1.32 \end{aligned}$$

Theoretical airflow: *A<sub>o</sub>* From Boie's formula,

$$\begin{aligned} A_o &= 2.957 \times (H_1 - 4,650) / (10,000) \\ &= 2,957 \times (40,000 - 4,650) / (10,000) \\ &= 10.47 \text{ Nm}^3/\text{kg-fuel} \end{aligned}$$

Actual airflow (per kg of fuel): *A*

$$\begin{aligned} A &= mA_o = 1.32 \times 10.47 \\ &= 13.82 \text{ Nm}^3/\text{kg-fuel} \end{aligned}$$

Actual airflow (per hour): *A'*

$$\begin{aligned} A' &= A \times f \times \rho_r = 13.82 \times 1,632 \times 0.929 \\ &= 20,951 \text{ Nm}^3/\text{h} \end{aligned}$$

### Sucked airflow

Hole diameter 150 mm → area 0.0177 m<sup>2</sup> per hole

Internal pressure  $\Delta p = -59 \text{ mmHg} = -59 \text{ kgf/m}^2$

Air density (20°C)  $\gamma = 1.2 \text{ kg/m}^3$

Flow coefficient 0.65

Flow velocity:

$$V = \sqrt{2g \times \Delta p / \gamma} = \sqrt{2 \times 9.8 \times 59 / 1.2}$$

$$= 31 \text{ m/s}$$

Air suction flow =  $0.0177 \times 31 \times 3,600 \times 0.65 = 1,284 \text{ m}^3/\text{h}$  per hole

With two holes;  $1,284 \times 2 \times (273) / (273 + 20) = 2,393 \text{ Nm}^3/\text{h}$

Ratio of sucked airflow  $2,393/20,915 \times 100 = 11.4\%$

### Reduction of heat input

The waste heat recovery rate changes depending on changes in the airflow that passes through the planetary cooler. But when it is assumed that the preheated air temperature remained unchanged at 300°C, reduction of heat input is as follows.

$$13.82 \times 1.30 \times (300 - 18.7) \times 0.114 \\ = 576 \text{ kJ/kg-fuel}$$

Here, the air specific heat is assumed as 1.30 kJ/(Nm<sup>3</sup>·°C).

The ratio to the total heat input is as follows.

$$576/40,589 \times 100 = 1.4\%$$

To eliminate this reduction of heat input, it is recommended that doors or glasses are fitted to the holes. Provisions should be made so that entry of non-preheated air is prevented to the most possible extent and that necessary air is sucked through the planetary cooler.

b. Countermeasures against uneven performance of planetary coolers

The surface temperature of calciner planetary coolers was measured using Thermal Video System "AVIO" on the afternoon of August 29. The results are indicated by color photos. It was found as a result that ten planetary coolers are not of equal state, as every other planetary coolers were of low temperature and high temperature. Due to the fact that alumina is not equally flowing to the planetary coolers through five holes in the calciner, heat exchange between calcinated alumina and air sucked from outside is not equally made at planetary coolers.

The structure is such that alumina falls into two planetary coolers from one hole of the calciner. But it is ideal that entry to one planetary cooler occurs from one hole. Since large expenses are required to make modification to increase the number of holes to ten, a simpler method for modification should be examined.

It is extremely difficult to equally drop alumina into two planetary coolers from one hole because of being in high temperature atmosphere and also because the fluidity largely varies by the  $\alpha$ - $\text{Al}_2\text{O}_3$  content of the calcinated alumina. The idea proposed below will not completely remedy the situations. But it is effective when variation of the calcination rate is small, and the expenses for modification are minor.

Fit a chute with grate made of heat resistant steel to each hole. A typical structure is shown in Figure 5.3.8. It is recommended that the form and size to be adopted in practice are determined through trial and error experience with measurement made using an infrared thermal image processor. Joint use of such a method that a projection or notch made of bricks before the alumina chute can also be considered.





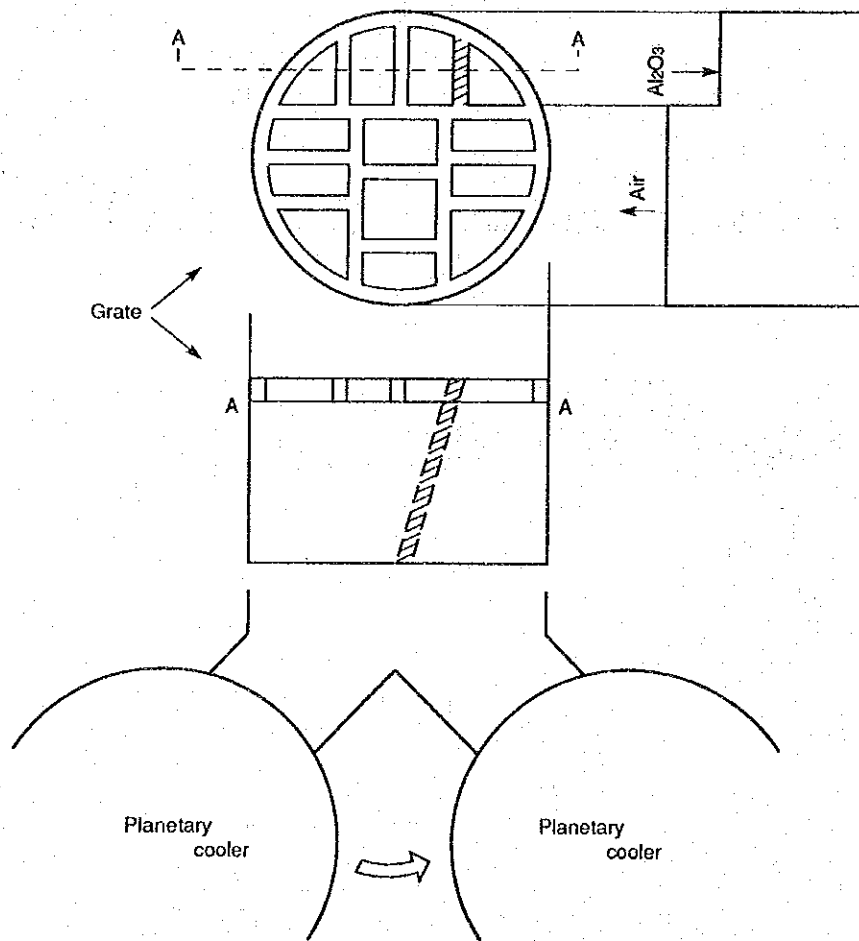
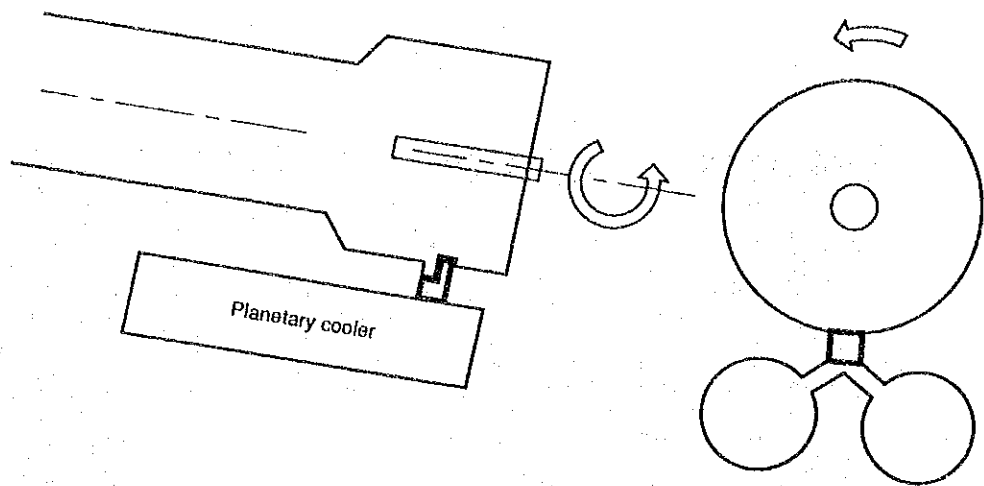


Figure 5.3.8 Grate at alumina chute

(4) Problems in other processes

A) Water balance

After the slurry, out of which alumina component has been extracted, was separated into alkali solution and red mud, the alkali component attached to the red mud is washed off using water. In addition, the aluminum hydroxide is washed with water in order to reduce impurities contained in the product. The water used for such operations is recirculated in the system to prevent alkali loss.

The bauxite used at this factory is of large boehmite content as stated in (2) above, and accordingly, the alkali concentration in the extraction process is relatively high as  $\text{Na}_2\text{O}$  195 g/l. On the other hand, the condition for crystallization of aluminum hydroxide through hydrolysis of sodium aluminate solution in the following process is alkali concentration  $\text{Na}_2\text{O}$  140 g/l. Therefore, dilution should be made after exit from digesters. The water used for washing red mud, water used for washing aluminum hydroxide and other water entering the system are used for dilution. In order to recycle sodium aluminate solution after crystallization of aluminum hydroxide, thickening should be made using steam to vaporize the added dilution water. In the case where water enters the system by a rate that is larger than what is required for dilution, thickening should be made using steam in order to prevent alkali loss, which requires additional steam for vaporization. Not to allow entry of excessive water into the system is important for saving the steam.

The water balance was investigated. The chart indicating water balance of the whole factory during full production in 1990 was well arranged. But there is no data on the water balance of alkali-containing water only. In addition, the water balance in the current state of reduced production is not seized. A chart that indicates entry and exit of water to/from the system should be prepared once a month for suitable management.

Implementation of the following countermeasures began two months ago in order to reduce the water entering the system, and good effect has been obtained.

- a. Lime was conventionally added as  $\text{Ca}(\text{OH})_2$ . But it was changed to make addition as  $\text{CaO}$  to reduce the moisture that makes accompanied entry.
- b. The washing water in the crystallization process was classified from recovered soda solution, and the washing water is used as red mud washing water upon check of the alkali concentration.
- c. The water for cleaning the factory was classified so as not to enter the cycle.
- d. Dry sealing method that does not use sealing water at all was adopted for pumps.
- e. Rainwater, which conventionally entered the drainage pit of the factory, was separated to another pit so as not to allow its entry to the cycle.

Attention should be paid to uplift of the aluminum hydroxide washing efficiency besides the above, in order to reduce the water. At the vacuum rotary drum filters in the crystallization process and also before calciners, washing is performed by such a method that water is caused to make downflow through the holes produced on the bottom side of pipes. But there are places where the holes are plugged and there also are places where the flow rate is excessive, the washing water is not equally distributed and the washing efficiency is inferior. We were informed that the factory has already noticed it and intends to make modification to spray type shortly. It is desirable that this modification is implemented as early as possible.

**B) Energy conservation measures during reduced production**

The alumina production capacity of this factory is 330,000 t/y. But the production program for 1991 was 170,000~210,000 t/y, and the actual production during the investigation period was made at the rate of as low as 130,000 t/y, which is only 40% of the capacity.

The equipment for digestion at this factory are installed in four systems and it was so designed to usually operate three systems. Compared to a factory having one or two systems only, therefore, the ratio of heat loss caused by reduction of output is minor. Furthermore, small size thickeners and precipitators are installed in a large quantity.

At a factory of mass production and high productivity type, the equipment size is increased and the number of systems and number of equipment are reduced to reduce heat radiation loss and electric energy for agitation. In addition, pumps of large capacity are adopted to reduce the number of pumps in an effort to reduce power consumption as a whole. From this viewpoint, the fact that equipment of small size are installed in a large quantity provides such an advantage that drop of efficiency can be prevented while the output is reduced. Although the efficiency is inferior during production at full capacity, it is because operation can be made with equipment of required quantity only with unnecessary equipment disconnected.

All of seven red mud thickeners were operating during the investigation even if the factory was in reduction of output. Because sedimentation becomes slow due to temperature drop caused by reduction of liquid feed rate, it is recommended to use only appropriate number of thickeners during reduction of output and thus heat radiation loss can be reduced and electric energy can be saved. With the counter current multi-stage washers, however, a certain number of stages is required even while the liquid flow rate is small. During production at 130,000 t/y, therefore, operation of four machines (4-stage washing) is considered suitable.

C) Review of desiliconizing process

Desiliconization is made with the desiliconization tanks before the digester process at this factory. Desiliconization should be made before the digester process with low silica bauxite of  $\text{SiO}_2$  content 0.1~1.0%. But it is not necessarily required in the case where the silica content is large.

Desiliconization rate means the crystallization rate of sodalite after silica has dissolved, and it is governed by the surface area of the sodalite seeds generated through natural crystallization. With bauxite of large silica content, seeds are generated at a relatively large rate compared to low silica bauxite, crystallization is induced and reduction of silica concentration in the liquid is accelerated.

It is considered that desiliconization tanks may be bypassed because the bauxite used at this factory is of kaolin content 5.7%,  $\text{SiO}_2$  content 7.24% and also because the factory is under reduction of output. It leads to reduction of heat dissipation from outside surfaces of tanks and reduction of pump power, etc.

(5) Boilers

A) General description of private thermal power generation plant

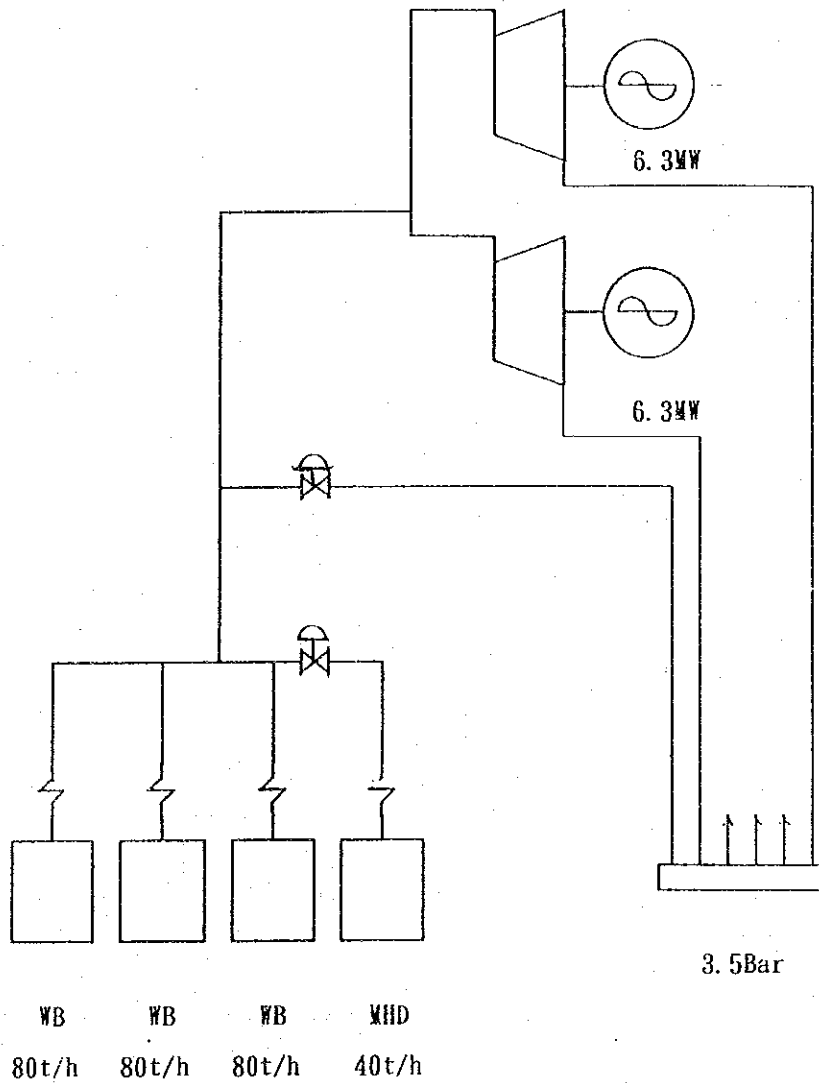
The entire steam and a part of electric energy consumed at this factory are supplied by means of a private thermal power generation plant as shown in Figure 5.3.9. The electric system is linked with the commercial power system, and the steam system is composed of a 40 bar high pressure line and a 3.5 bar turbine back pressure line. Furthermore, the condensate from the liquid process is separated from the condensate from the general process and is recovered as boiler feed water after making ion exchange treatment.

The concentrators run as matched with the peak hour of electric energy consumption to increase the boiler load and to decrease buying power.

Particulars of principal components of the private thermal power generation plant are as shown below.

Table 5.3.17 Particulars of principal components.

<p>① Boiler          Manufacturer          Max. continuous evaporation rate          Normal pressure          Steam temperature          Fuel          Quantity</p>	<p>Wagner-Biro          80 t/h          40 bar(G)          400°C          Fuel oil          3 (No. 1 boiler only was running at the time of data sampling)</p>
<p>② Boiler          Manufacturer          Max. continuous evaporation rate          Normal pressure          Steam temperature          Fuel          Quantity</p>	<p>MHD          40 t/h          70 bar(G)          500°C          Fuel oil (modification from coal firing was completed, and application for permit is pending)          1</p>
<p>③ Turbine generator          Manufacturer          Type          Generator output          Quantity</p>	<p>Ganz          Back pressure turbine          6.3 MW × 10,500 V × 50Hz × 2P          2 (1 is for reserve)</p>



**Figure 5.3.9 Flow sheet of private thermal power generation plant**

B) Heat balance

a. Particulars of boiler

**Table 5.3.18 Particulars of boiler**

Manufacturer's name	Wagner-Biro	
Date of production/modification	1970	
Type	Water tube type	
Max. continuous evaporation rate	80	t/h
Max. working pressure	47	bar(G)
Max. working temperature	425	°C
Normal pressure	40	bar(G)
Normal temperature	400	°C
Boiler heat transfer area	1,327	m <sup>2</sup>
Superheater heat transfer area	380	m <sup>2</sup>
Economizer heat transfer area	920	m <sup>2</sup>
Air preheater heat transfer area	1,260	m <sup>2</sup>
Combustion chamber volume	250	m <sup>3</sup>
Combustion chamber heat generation ratio	985	MJ/(m <sup>3</sup> ·h)
Fuel	Fuel oil	
Burner type	Rotary burner	
Drafting method	Forced draft	
Smokestack (top bore x height)	8 m × 80 m (common to 4 boilers)	
Control system	Main control : Electronic control system Air ratio control : Motor linkage system Water level control : Electronic control system	



b: Flow sheet

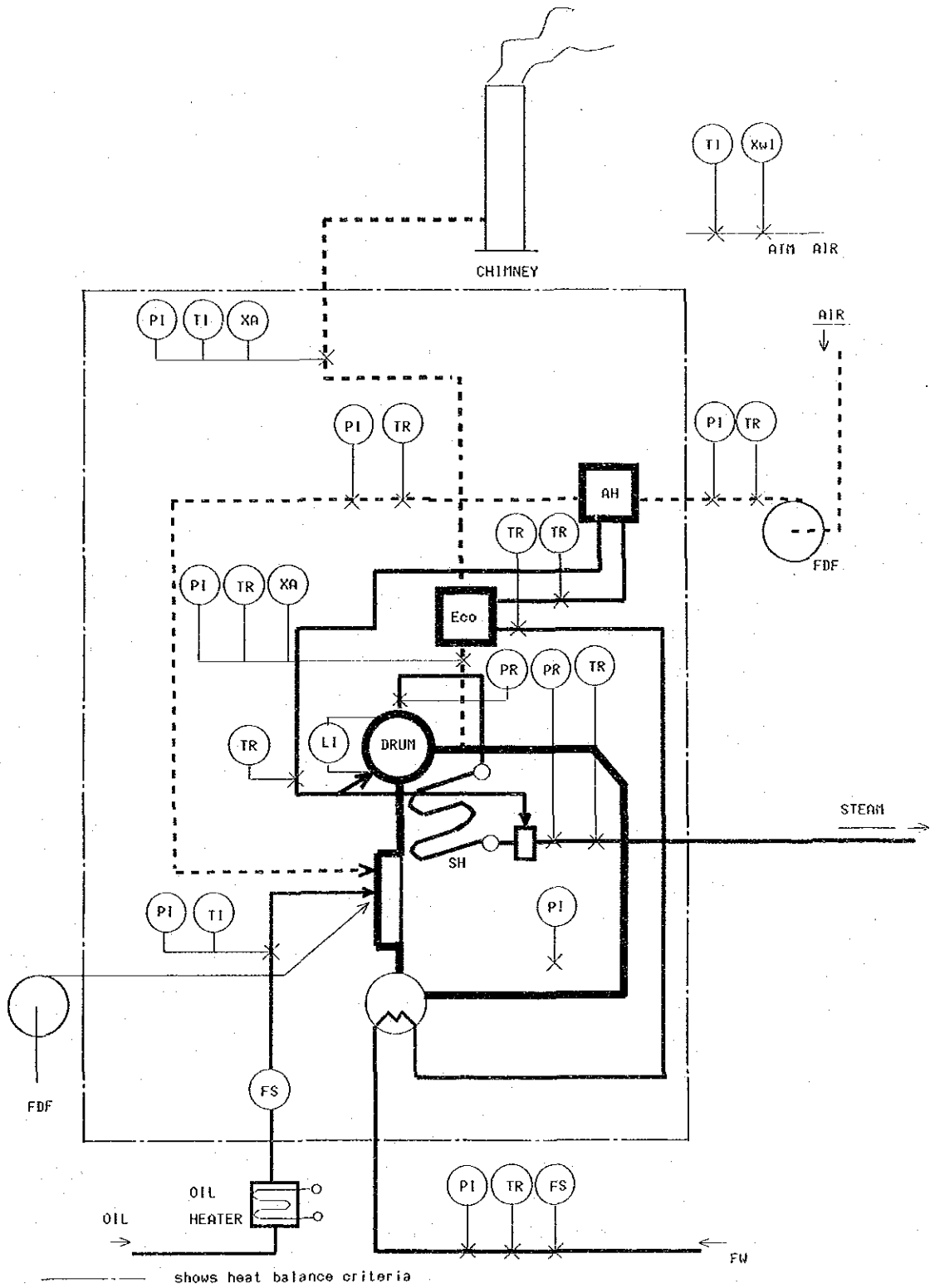


Figure 5.3.10 Boiler flow sheet

c. Results of measurement

Table 5.3.19 Results of measurement

Test date and hour	August 28, 1991 (14:45~16:45)	
Weather and outdoor air temperature	Fine, dry bulb temperature 22°C, wet bulb temperature 17°C	
Load factor	59.2	%
Fuel	Fuel oil (F60/130, F90/160)	
Specific gravity	948.5 at 92°C	kg/m <sup>3</sup>
Working temperature	89.7	°C
Low calorific value	39.23	MJ/kg
Consumption (measured value)	4.07	m <sup>3</sup> /h
Consumption (calculated value)	3,867	kg/h
Feed water		
Water feed rate (measured value)	45,240	kg/h
Water feed rate (calculated value)	47,320	kg/h
Water feed rate per unit fuel	12.24	kg/kg
Lower drum inlet temperature	107.7	°C
Economizer inlet temperature	139.7	°C
Boiler inlet temperature	156.5	°C
Generated steam		
Boiler drum pressure	41.43	kg/cm <sup>2</sup> (G)
Superheater outlet pressure	39.10	kg/cm <sup>2</sup> (G)
Superheater outlet temperature	472.0	°C
Attemperator outlet temperature	423.0	°C
Combustion air		
Air flow per unit fuel	11.42	Nm <sup>3</sup> /kg
Air preheater inlet temperature	28.1	°C
Air preheater outlet temperature	180.6	°C
Air preheater outlet pressure	403	mmAq
Forced draft fan outlet temperature	28.1	°C
Forced draft fan outlet pressure	487	mmAq
Wind box pressure	335.7	mmAq
Air ratio (boiler furnace outlet)	1.097	

**Table 5.3.19 Results of measurement (Cont'd)**

Exhaust gases				
Exhaust gas flow per unit fuel	12.03			Nm <sup>3</sup> /kg
Internal pressure	50.7			mmAq
Boiler furnace outlet temperature	325.7 (343.9)			°C
Boiler furnace outlet pressure	32.7			mmAq
Economizer outlet temperature	176.4 (203.8)			°C
Economizer outlet pressure	-8.8			mmAq
Gas analysis	CO <sub>2</sub>	O <sub>2</sub>	CO	
	13.81	2.01	0.06	%

Note 1: The data recorded with hybrid recorders were used as the basis (except for atmosphere dry bulb temperature and wet bulb temperature) for heat balance. Integrated values were used for water flow rate and fuel flow rate, and mean values were used for other data.

Note 2: The mean value of the values measured on the right and left is indicated as the economizer outlet exhaust gas temperature.

Note 3: The values in ( ) of the exhaust gas temperature are corrected values calculated from the heat balance on the water side.

Note 4: As for gas analyzed values, the air ratio, etc. were calculated assuming the measured values at the boiler furnace outlet as the values at the economizer outlet.

d. Heat balance chart

Table 5.3.20 Heat balance chart

Heat input			kJ/kg	%
①	Heating value of fuel	$H_1$	39,231	99.45
②	Sensible heat of fuel	$Q_1$	127	0.32
③	Sensible heat of air	$Q_2$	91	0.23
Total		$H_1+Q$	39,449	100.00
Effective heat output				
①	Heat absorbed by generated steam		34,455	87.34
	Heat absorbed by boiler furnace	$Q_b$	25,077	63.56
	Heat absorbed by economizer	$Q_{ec}$	2,485	6.30
	Heat absorbed by superheater	$Q_{sh}$	6,833	17.32
	Heat absorbed by lower drum	$Q_d$	1,665	4.22
	Heat discharged to air preheater	$Q_{ab}$	-1,603	-4.06
Sub total		$Q_s$	34,455	87.34
Heat loss				
①	Heat of exhaust gas (including water vapor)	$L_1$	2,561 (3,081)	6.49 (7.81)
②	Heat loss caused by incomplete combustion	$L_2$	81	0.21
③	Dispersed heat loss	$L_3$	315	0.80
④	Other heat losses	$L_4$	2,037 (1,517)	5.16 (3.85)
Sub total			4,994	12.66
Total			39,449	100.00
Boiler efficiency				%
①	Heat input/output method	$y_1 = \{Q_s / (H_1 + Q)\} \times 100$		87.34
②	Heat loss method	$y_2 = \{1 - (L_1 + L_2 + L_3) / (H_1 + Q)\} \times 100$		92.50

Note: The figures in ( ) are of the cases where corrected value is used as the exhaust gas temperature.

e. Calculation of heat balance (per fuel of 1 kg)

[Heat input]

- ① Heating value of fuel (value analyzed by HUNGALU)

$$H_1 = 39,231 \quad \text{kJ/kg-Fuel}$$

- ② Sensible heat of fuel

$$Q_1 = C_f \times (t_f - t_o) = 1.88 \times (89.7 - 22.0) \\ = 127 \quad \text{kJ/kg-Fuel}$$

where;

$C_f$ : Mean specific heat of fuel (1.88 kJ/(kg·°C) in general)

$t_f$ : Temperature of fuel after heating °C

$t_o$ : Ambient temperature °C

- ③ Sensible heat of air

$$Q_2 = A \times C_a \times (t_a - t_o) \quad \text{kJ/kg-Fuel}$$

$$A = m \times A_o \times (1 + 1.61 \times z)$$

$$A_o = 2.957 \times (H_1 - 4,650) / 10,000 = 2,957 \times (39,231 - 4,605) / 10,000 \\ = 10.24 \quad \text{Nm}^3/\text{kg-Fuel}$$

$$m = \frac{21}{21 - 79 \times \left( \frac{(O_2) - 0.5 \times (CO)}{(N_2)} \right)}$$

$$= \frac{21}{21 - 79 \times \left( \frac{2.01 - 0.5 \times 0.06}{84.12} \right)}$$

$$= 1.097$$

$$A = 1.097 \times 10.24 \times (1 + 1.61 \times 0.0105) \\ = 11.42 \quad \text{Nm}^3/\text{kg-Fuel}$$

$$Q_2 = 11.42 \times 1.30 \times (28.1 - 22.0) \\ = 91 \quad \text{kJ/kg-Fuel}$$

where;

A : Airflow per fuel of 1 kg (including water vapor) Nm<sup>3</sup>/kg-Fuel

C<sub>a</sub> : Mean specific heat of air (1.30 kJ/(Nm<sup>3</sup>·°C))

T<sub>a</sub> : Air temperature at inlet of air preheater °C

t<sub>o</sub> : Fresh air temperature °C

m : Air ratio

z : Absolute humidity of fresh air (per dry air of 1 kg) kg/kg-Dry Air

A<sub>o</sub> : Theoretical dry airflow Nm<sup>3</sup>/kg-Fuel

(CO<sub>2</sub>) : Volume % of CO<sub>2</sub> in dry combustion exhaust gas

(O<sub>2</sub>) : Volume % of O<sub>2</sub> in dry combustion exhaust gas

(CO) : Volume % of CO in dry combustion exhaust gas

(N<sub>2</sub>) : Volume % of N<sub>2</sub> in dry combustion exhaust gas  
= 100 - {(CO<sub>2</sub>) + (O<sub>2</sub>) + (CO)}

Note 1: Fresh air absolute humidity value [0.0105] was obtained from the air diagram.

Note 2: The value of A<sub>o</sub> was calculated by Boie's formula because the fuel composition is not known.

[Effective heat]

① Heat absorbed by generated steam

$$Q_s = W_2 \times (h_2 - h_1) = 12.24 \times (3,270 - 455) \\ = 34,455 \quad \text{kJ/kg-Fuel}$$

$$W_2 = W_1 = W/F = 47,320/3,867 \\ = 12.24 \quad \text{kg/kg-Fuel}$$

$$W = 45,240 \times 1.046 \\ = 47,320 \quad \text{kg/h}$$

$$F = 4.07 \times 950.0 \\ = 3,867 \quad \text{kg/h}$$

② Heat absorbed in boiler furnace

$$Q_b = (W_1 - W_3) \times (h_6 - h_5) = (12.24 - 0.50) \times (2,799 - 663) \\ = 25,077 \quad \text{kJ/kg-Fuel}$$

$$W_3 = W_1 \times (h_7 - h_2)/(h_7 - h_5) = 12.24 \times (3,381 - 3,270)/(3,381 - 663) \\ = 0.50 \quad \text{kg/kg-Fuel}$$

③ Heat absorbed at economizer

$$Q_{ec} = W_1 \times (h_4 - h_3) = 12.24 \times (794 - 591) \\ = 2,485 \quad \text{kJ/kg-Fuel}$$

④ Heat absorbed at superheater

$$Q_{sh} = (W_1 - W_3) \times (h_7 - h_6) = (12.24 - 0.50) \times (3,381 - 2,799) \\ = 6,833 \quad \text{kJ/kg-Fuel}$$

⑤ Heat absorbed at lower drum

$$Q_d = W_1 \times (h_3 - h_1) = 12.24 \times (591 - 455) \\ = 1,665 \quad \text{kJ/kg-Fuel}$$

⑥ Heat discharged to air preheater

$$Q_{ah} = W_1 \times (h_5 - h_4) = 12.24 \times (663 - 794) \\ = 1,603 \quad \text{kJ/kg-Fuel}$$

where;

$W_1$  : Water feed rate per fuel of 1 kg kg/kg-Fuel

$W_2$  : Attemperator outlet steam flow per fuel of 1 kg kg/kg-Fuel

$W_3$  : Attemperator spray rate per fuel of 1 kg kg/kg-Fuel

$W$  : Water feed rate per hour kg/h

$F$  : Fuel gas flow per hour kg/h

$h_1$  : Enthalpy of feed water at lower drum inlet 455 kJ/kg

$h_2$  : Enthalpy of steam at attemperator output 3,270 kJ/kg

$h_3$  : Enthalpy of feed water at economizer inlet 591 kJ/kg

$h_4$	: Enthalpy of feed water at economizer outlet	794 kJ/kg
$h_5$	: Enthalpy of feed water at boiler inlet	663 kJ/kg
$h_6$	: Enthalpy of generated saturated steam	2,799 kJ/kg
$h_7$	: Enthalpy of steam at superheater outlet	3,381 kJ/kg

1.046: Density correction factor of water feed rate measuring orifice

$$= (v_d/v)^{0.5} = (0.0011449/0.0010469)^{0.5}$$

$v_d$ : Specific volume of feed water in orifice data m<sup>3</sup>/kg

$v$ : Specific volume of feed water at the temperature  
at the time of data measurement m<sup>3</sup>/kg

950.0: Specific weight of fuel oil at the temperature at the time of data measurement

$$d = d_t + 0.65 \times (t_m - t) = 948.5 + 0.65 \times (92.0 - 89.7)$$

$d$ : Specific gravity of fuel oil at the temperature at the time of data measurement kg/m<sup>3</sup>

$d_t$ : Specific gravity of known fuel oil kg/m<sup>3</sup>

$t_m$ : Temperature at the time of measurement of specific gravity of known fuel oil °C

$t$ : Specific gravity of fuel oil at the time of measurement °C



[Heat loss]

① Heat of exhaust gas (including water vapor)

$$L_1 = G \times C_g \times (t_g - t_o) \quad \text{kJ/kg-Fuel}$$

$$G = G_o + G_w + (m - 1) \times A_o + G_{w1} \quad \text{Nm}^3/\text{kg-Fuel}$$

$$\begin{aligned} G_1 &= G_o + G_w && \text{Nm}^3/\text{kg-Fuel} \\ &= \{3.76 \times (H_1 - 4,605)/10,000\} - 2.18 \\ &= \{3.76 \times (39,231 - 4,605)/10,000\} - 2.18 \\ &= 10.84 && \text{Nm}^3/\text{kg-Fuel} \end{aligned}$$

$$\begin{aligned} G_{w1} &= 1.61 \times z \times m \times A_o = 1.61 \times 0.0105 \times 1.097 \times 10.24 \\ &= 0.19 && \text{Nm}^3/\text{kg-Fuel} \end{aligned}$$

$$\begin{aligned} G &= 10.84 + (1.097 - 1) \times 10.24 + 0.19 \\ &= 12.02 && \text{Nm}^3/\text{kg-Fuel} \end{aligned}$$

$$\begin{aligned} L_1 &= 12.02 \times 1.38 \times (176.4 - 22.0) \\ &= 2,561 && \text{kJ/kg-fuel} \end{aligned}$$

where;

$G$  : Actual exhaust gas flow per fuel of 1 kg  
(including water vapor) Nm<sup>3</sup>/kg-Fuel

$G_o$  : Theoretical dry exhaust gas flow Nm<sup>3</sup>/kg-Fuel

$G_w$  : Water vapor flow generated by combustion Nm<sup>3</sup>/kg-Fuel

$G_{w1}$  : Water vapor flow caused by moisture in the  
combustion air Nm<sup>3</sup>/kg-Fuel

$G_1$  : Theoretical combustion exhaust gas flow containing  
generated water vapor Nm<sup>3</sup>/kg-Fuel

$C_g$  : Mean specific heat of exhaust gases (1.38 kJ/(Nm<sup>3</sup>·°C) in general)

$t_g$  : Exhaust gas temperature at economizer outlet °C

$t_o$  : Ambient air temperature °C

$m, z, A_o$ : Same as before

Note: The value of  $G_1$  was calculated by Boie's formula because the fuel composition is not known.

② Loss caused by incomplete combustion

$$L_2 = 128 \times (G_0 + (m - 1) \times A_0) \text{ (CO)} \quad \text{kJ/kg-Fuel}$$

$$G_0 = G_1 - G_w \quad \text{Nm}^3/\text{kg-Fuel}$$

$$G_w = 1.244 \times (9h + w)/100 = 1.244 \times 9 \times 11/100 \\ = 1.23 \quad \text{Nm}^3/\text{kg-Fuel}$$

$$G_0 = 10.84 - 1.23 \\ = 9.61 \quad \text{Nm}^3/\text{kg-Fuel}$$

$$L_2 = 128 \times \{9.61 + (1.097 - 1) \times 10.24\} \times 0.06 \\ = 81 \quad \text{kJ/kg-Fuel}$$

where;

h : Hydrogen content in the fuel (11% in general)

w : Moisture content in the fuel (0% in general)

$G_0, m, A_0, \text{ (CO), } G_1, G_w$ : Same as before

③ Dispersed heat loss

$$L_3 = I_r \times (H_h + Q)/100 = 0.75 \times (41,718 + 218)/100 \\ = 315 \quad \text{kJ/kg-Fuel}$$

$$H_h = H_1 + 25.12 \times 9 \times h = 39,231 + 25.12 \times 9 \times 11 \\ = 41,718 \quad \text{kJ/kg-Fuel}$$

$$Q = Q_1 + Q_2 = 127 + 91 \\ = 218 \quad \text{kJ/kg-Fuel}$$

where;

$I_r$  : Radiated heat loss rate (=0.75%)

$H_h$  : High heating value of fuel kJ/kg-Fuel

Q,  $Q_1, Q_2$ : Same as before

④ Other heat losses

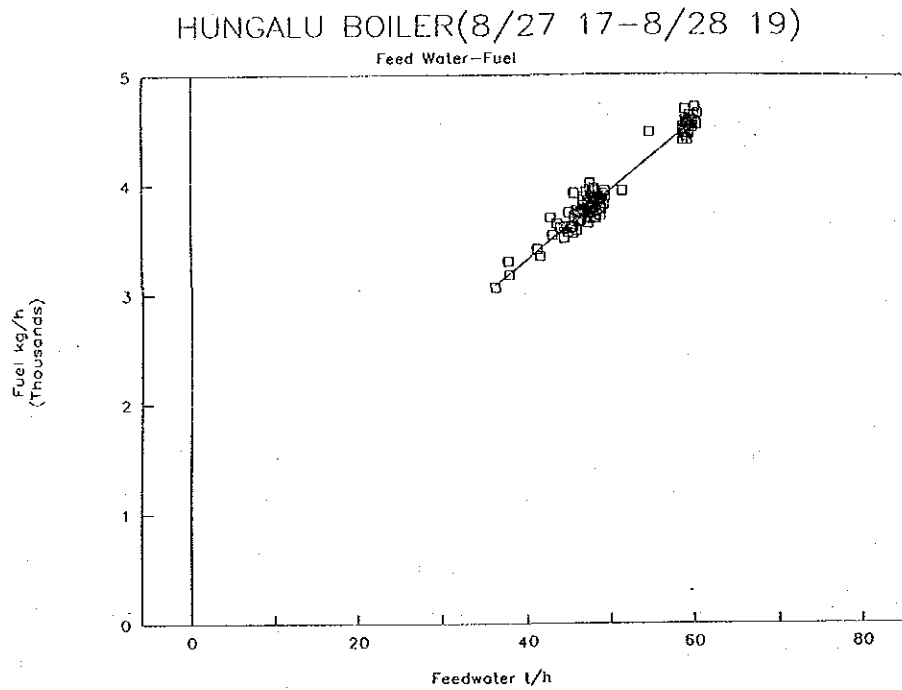
$$\begin{aligned}L_4 &= H_1 + Q_1 + Q_2 - (Q_3 + L_1 + L_2 + L_3) \\ &= 39,231 + 127 + 91 - (34,455 + 2,561 + 81 + 315) \\ &= 2,037 \quad \text{kJ/kg-Fuel}\end{aligned}$$

f. Considerations

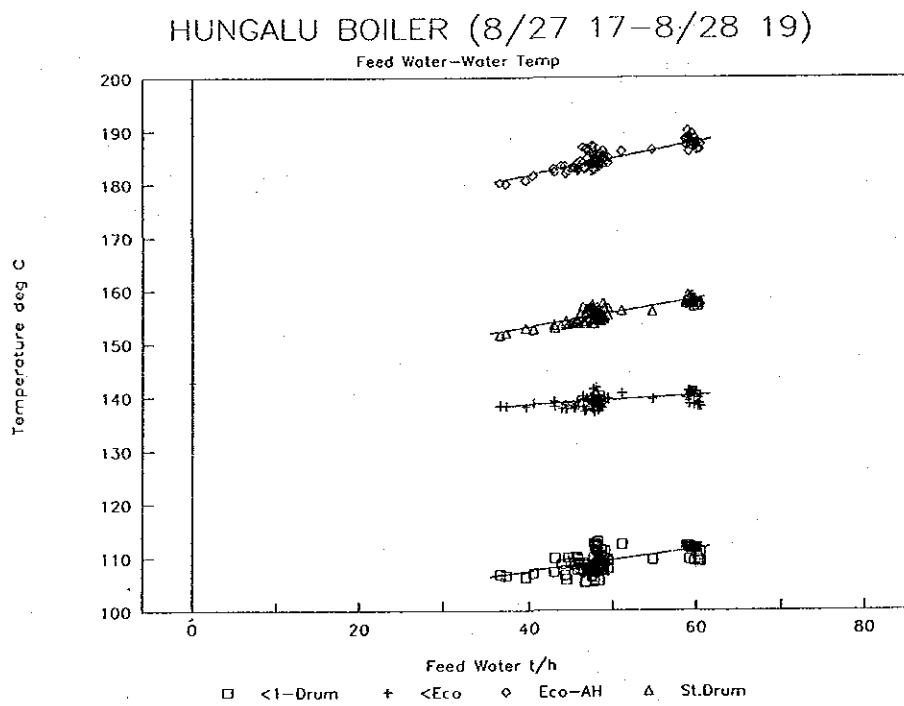
- ① The load variation during the performance test period was minor. The water feed rate was 47.48 t/h at maximum and 41.78 t/h at minimum, and good stability was observed.
- ② The boiler efficiency calculated by heat input/output method was 87.34% at the load factor of 59.2% as a result of calculation of heat balance. It indicates that high efficiency is maintained.
- ③ "Other losses" is rather large in the heat balance chart. The following factors can be raised as its causes.
  - a) "Heat of exhaust gas" was obtained from the average temperature (176.4°C) measured at right and left of the economizer outlet duct. But it is considered that it was measured at a level that is lower than the actual level. 203.8°C is obtained as the exhaust gas temperature at the economizer outlet from the heat balance of the water system. The values inside of ( ) are values of "Heat of exhaust gas" and "Other heat losses" calculated based on this value assuming that the exhaust gas specific heat is 1.41 kJ/(Nm<sup>3</sup>.°C).
  - b) The boiler's continuous blow valve was closed at the time of data measurement. But minor leakage did occur. When this blow rate is assumed as 1% of the water feed rate, the "loss caused by blow" becomes 121 kJ/kg-Fuel (0.3%).

④ Boiler characteristics

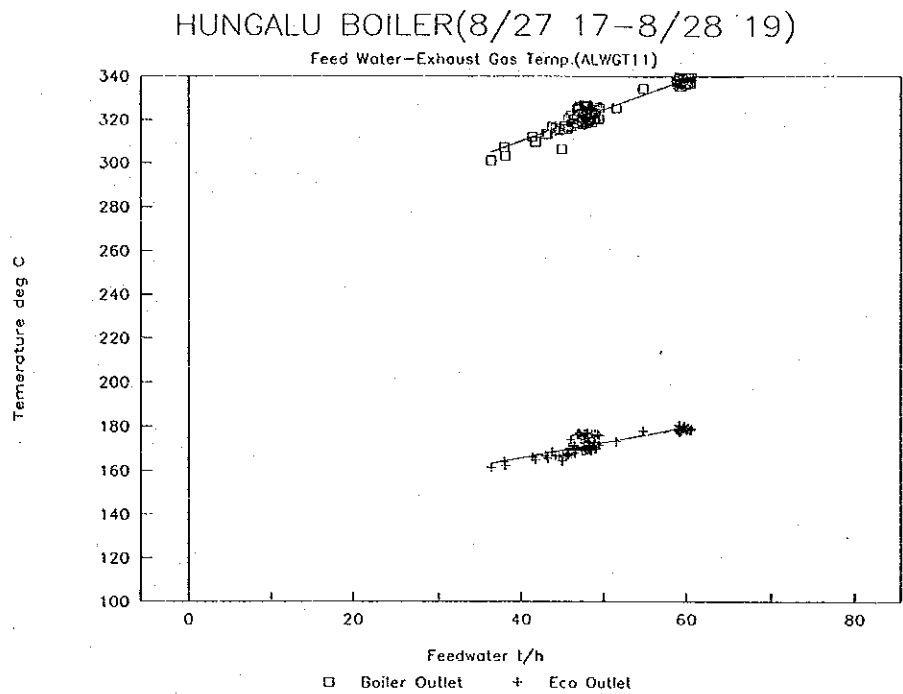
The following boiler characteristics were obtained from the data recorded with hybrid recorders.



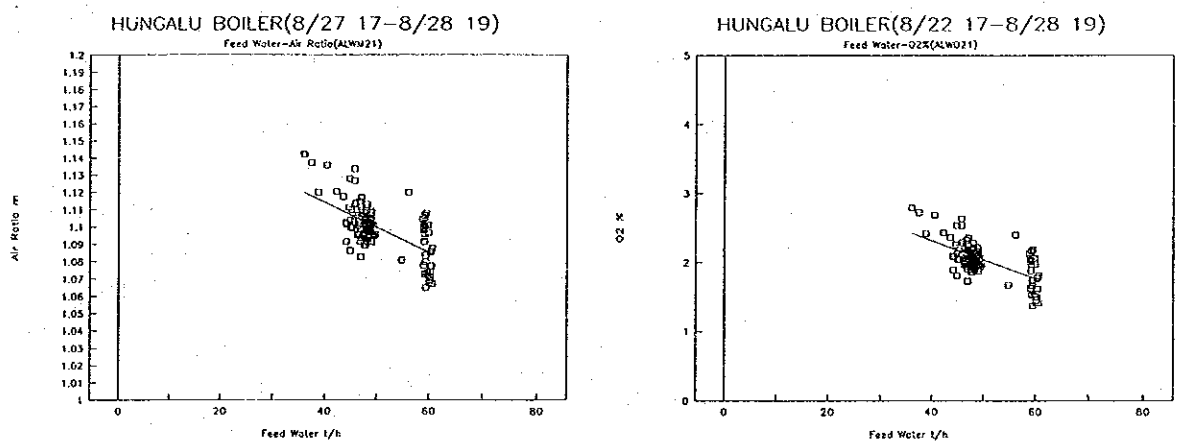
**Figure 5.3.11 Feed water flow rate/fuel flow rate characteristics**



**Figure 5.3.12 Feed water temperature - load characteristics at various points**



**Figure 5.3.13 Exhaust gas temperature - load characteristics at various points**



**Figure 5.3.14 O<sub>2</sub>%, air ratio - load characteristics**

C) Points of improvement related to the boiler and their effect

a. Preheating of air entering air preheater

Preheating of the combustion air for this boiler is made through heat exchange with the feed water at the economizer outlet. But since the forced draft fan has a marginal power and the damper opening is currently reduced. It is therefore possible to add air preheating by an external heat source. The heat taken away from the feed water is reduced as a result of air preheating, and saving of the fuel can be achieved.

① Fuel saving

The fuel saving rate brought by air preheating is given by the following expression.

$$S = \frac{P}{F - Q + P} \times 100$$

where;

S :	Fuel saving rate	%
P :	Quantity of heat brought by preheated air	kJ/kg-Fuel
F :	Heating value of fuel	kJ/kg-Fuel
Q :	Quantity of heat taken away by combustion gas	kJ/kg-Fuel

The effect in the case where the air temperature is increased by 10°C in the conditions of heat balance is trially calculated as follows.

$$P = 11.42 \times 1.30 \times 10 = 148 \quad \text{kJ/kg-Fuel}$$

$$F = 39,231 \quad \text{kJ/kg-Fuel}$$

$$Q = 2,561 \quad \text{kJ/kg-Fuel}$$

$$S = \frac{148}{39,231 - 2,561 + 148} \times 100 = 0.4\%$$

Fuel saving rate

$$3,867 \text{ kg/h} \times 8,760 \text{ h/y} \times 0.004 = 135.5 \text{ t/y}$$

Saving amount

$$135.5 \text{ t/y} \times 10,000 \text{ Ft/y} = 1,355,000 \text{ Ft/y}$$

The following two methods are available for preheating combustion air.

- a) Provision of suction port of FDF in the high temperature portion in the upper part of the boiler.
- b) Installation of a heat exchanger to use as heat source the unrecovered condensate of 60~90°C level or low pressure steam between FDF and air preheater.

With method a), it is possible to increase the temperature by around 10°C with almost no equipment investment.

② Precautions

- a) Increase of the resistance by the duct or heat exchanger should not exceed the margin of the head of the FDF. It is judged that a considerable margin is provided because the suction vane opening was 36% during the investigation.
- b) The margin of the power of the FDF drive motor should not be exceeded.
- c) The burner register passing velocity changes when the combustion air temperature increases as a result of preheating. The change is not too large because preheating by boiler feed water is made in the existing air preheater in the final stage. But it is recommended that control of the combustion air temperature is made with a temperature control valve installed along the heated feed water line in preparation for transient variation of the flow rate.

b. Prevention of heat loss caused by incomplete combustion

CO content of 0.06% at average is observed according to the data during the performance test period. The countermeasures described below will be effective.

① Readjustment of air ratio:

Excessive air ratio should be avoided from the aspect of exhaust gas loss. But check for excessive air reduction during partial load and at the occasions of load change.

② Readjustment of gain of boiler main controller:

The airflow has volume lag characteristics against variation of fuel flow rate at the time of load variation. Therefore, if the fuel flow rate is rapidly increased, air is temporarily short and faulty combustion occurs. Gain of the boiler main controller, D action in particular, requires readjustment.

③ Readjustment of flame form:

Unburnt gases are generated when burner flames are in contact with the boiler heat transfer surfaces in the combustion chamber. Therefore, readjustment of flame form is required by adjustment of the burner cup position and so forth. (The preconditions are same as before.)

Fuel saving rate

$$\begin{aligned} & 81 \text{ kJ/kg-Fuel} \times 3,867 \text{ kg-Fuel/h} \times 8,760 \text{ h/y} / 39,231 \\ & = 69.9 \text{ t/h} \end{aligned}$$

Saving amount

$$\begin{aligned} & 69.9 \text{ t/y} \times 10,000 \text{ Ft/t} \\ & = 699,000 \text{ Ft/y} \end{aligned}$$

c. Shortening of soot blower operating interval

The soot blower operating interval is currently determined as once every two days. But the fuel of this boiler is considerably heavy type as judged from the heating value, and it is considered the contents of carbon residue and ash are relatively high. Since these substances cause external contamination of boiler heat transfer surfaces as so-called soot components, it is wanted that the soot blower operating interval is shortened. It may be considered that soot blow is effective if the economizer outlet gas temperature changes by over 10°C before and after soot blower operation.

Rise of the exhaust gas temperature by 10°C is equivalent to reduction of boiler efficiency of 0.43% with the load at the time of testing of this boiler.

The fuel saving effect is as follows when

- ① the mean exhaust gas temperature is 204°C in the case where the soot blower operating interval is once every two days; and
- ② the mean exhaust gas temperature is 194°C in the case where the soot blower operating interval is once every four hours;

if other preconditions are same as before.

Heat loss by exhaust gas

$$\textcircled{1} L_{11} = 12.02 \times 1.38 \times (204 - 22) = 3,019 \text{ kJ/kg-Fuel}$$

$$\textcircled{2} L_{12} = 12.02 \times 1.38 \times (194 - 22) = 2,853 \text{ kJ/kg-Fuel}$$



Fuel reduction ratio

$$1 - \frac{39,231 - 3,019}{39,231 - 2,853} = 0.0045$$

Fuel reduction rate

$$3,867 \text{ kg/h} \times 8,760 \text{ h/y} \times 0.0045 = 152.4 \text{ t/y}$$

Fuel reduction amount

$$152.4 \text{ t/y} \times 10,000 \text{ Ft/t} = 1,524,000 \text{ Ft/y}$$

The soot blower operating interval is once every four hours in many cases for a heavy type fuel oil firing boiler in Japan.

d. Management by boiler data

Measured data of high accuracy and prompt judgment based on the judging indices calculated based on the measured data are required in order to perform safe running and economic running of boilers. Typical load characteristics of the boiler were indicated earlier, and comparison of daily data with the standard characteristics obtained in such a manner constitutes the first step of computerized management of the boiler. For this objective, it is fundamentally necessary to establish a maintenance work standard and to keep the automatic control devices and measuring instruments always maintained based on this standard.

(6) Electrical equipment

A) Outline of electrical equipment of the factory

The electric power is received at 120kV and is supplied to loads in the factory with stepdown made to 10kV with a transformer of 25,000kVA as shown in the electric power one line diagram in Figure 5.3.3. The contracted power is 9,000kW. Furthermore, two private power generators of 9,400kVA each are provided.

Both of the incoming line for normal use and the incoming line for reserve are of entirely the same two-wire system, and the substations on the load side of the factory are capable of receiving electric power from both of incoming side and private power generator side.

The principal load equipment are shown in Table 5.3.21. The valves and dampers of pumps and blowers may be remote controlled.

Table 5.3.21 Principal load equipment

Workshop name	Application of machine	Motor type	Control /starting mode	Nominal Power kW	I <sub>n</sub> current A	P.f cos θ <sub>1</sub>	Power kW	Cable mm <sup>2</sup>	Specific resist. Ω/km	Cable length m	Loss 3 I <sub>n</sub> <sup>2</sup> r kW
Digestion	1. Slurry pump	IM	No Y/Δ	100	122	0.82	67	Al 185	0.1643	75	0.516
	2. Slurry pump	DC M	Voltage control	320	440	0.95	220	Al 240×3	0.1271	93	1.525
Precipitation	3. NaAl pump	IM	Cascade Auto	132	173	0.85	105	Al 185	0.1643	90	1.327
Sedimentation	4. Hot NaAl pump	IM	(40%) Cascade Auto	160	188	0.75	99	Al 240	0.1271	60	0.809
Precipitation	5. Filtered NaOH pump	IM	No Y/Δ	100	125	0.85	71	Al 185	0.1643	52	0.400
	6. Little vacuum pump	IM	No Y/Δ	95	160	0.72	78	Al 240	0.1271	45	0.439
	7. Little vacuum pump	IM	No Y/Δ	95	188	0.75	94	Al 240	0.1271	57	0.768
Calcination	8. Compressor	IM	Direct	75	74	0.7	34	Al 120	0.2525	30	0.124
	9. Compressor	IM	Direct	75	68	0.66	30	Al 120	0.2525	25	0.088
	10. Little vacuum pump	IM	Y/Δ	90	135	0.73	63	Al 185	0.1643	50	0.449
	11. Compressor	IM	Direct	140	300	0.78	147	Al 185	0.1643	45	0.998
	12. Compressor	IM	Direct	140	305	0.78	157	185	0.1643	55	1.261
Sedimentation	13. Water pump	IM	Y/Δ	132	158	0.83	87	140	0.1271	75	0.714
	14. Water pump	IM	Y/Δ	132	146	0.82	78	95×2	0.3189	20	0.204
	15. Red mud pump	IM	Cascade	200	235	0.42	49	240×2	0.1271	40	0.305
	16. Red mud pump	IM	Cascade	200	220	0.35	48	240×2	0.1271	30	0.277
Precipitation	17. Water pump	IM	Y/Δ	160	240	0.86	130	240	0.1271	65	1.428
	18. Recycled NaOH pump	IM	Y/Δ	100	174	0.91	102	240	0.1271	60	0.693
Sedimentation	19. Flour slurry pump	IM	YΔ	100	70	0.65	29	240	0.1271	85	0.159
	20. Back NaOH pump	IM	Y/Δ	132	222	0.85	120	240	0.1271	65	1.221

Total 12.707

B) Countermeasures for improvement and their effect

a. Airflow control of rotary kiln exhaust gas blower

The results of measurement of electric power with the rotary kiln exhaust gas blower are shown in Table 5.3.22.

**Table 5.3.22 Results of measurement of electric power with rotary kiln exhaust gas blower**

Measuring Time	Active Power	Apparent Power	Reactive Power	Power Factor
16 h 35 m	301.4 kW	353.6 kVA	184.5 kvar	0.853

Motor Capacity: 350 kW

The blower damper opening was 30%, the airflow was 55% of the maximum airflow, and the motor input was about 86% of the maximum value.

Therefore, the maximum value of the motor input is 350.5kW.

$$301.4 / 0.86 = 350.5$$

The shaft power of the blower is proportional to the product of airflow and pressure. Furthermore, since the resistance of a pipeline is proportional to the square of the airflow, the shaft power is proportional to the cube of the airflow. Accordingly, 17% should be ideally sufficient as the shaft power when the airflow is 55%.

$$350.5 \times 0.55^3 = 58.3 \text{ kW}$$

The required power estimated from the exhaust gas flow estimated from the heat balance of the rotary kiln is also of an identical value when the efficiency is assumed as 65%.

$$1,000 \text{ Nm}^3/\text{min} \times (273 + 150)/273 \times 150 \text{ mmAq} / 6,120 / 0.65 = 58.4 \text{ kW}$$

The difference from the current 301.4kW is the loss caused by the reduction in the damper opening. The following methods are available to eliminate this loss.

Case where variation of the airflow is minor

To reduce the outside diameter of the impeller (to such an extent that the air drops by around 20%)

To replace with a small capacity blower

To reduce the number of stages of the multi-stage blower

To reduce the revolution by changing the pulleys or gears because the airflow is proportional to the revolution

Case where variation of the airflow is large

To change the revolution by a variable speed motor

As the variation of the airflow during the measuring period was around  $\pm 15\%$ , the method to change the gear ratio is considered to be most economical when recovery from the current state reduction of output is taken into account.

Reducible electric energy consumption

It is assumed that the revolution is reduced to 70% with a margin taken into account.

$$(301.4 - 350.5 \times 0.7^3) \text{ kW} \times 8,760 \text{ h/y} = 1,587,100 \text{ kWh/y}$$

Power rate saving amount

$$1,587,100 \text{ kWh/y} \times 3.5 \text{ Ft/kWh} = 5,555,000 \text{ Ft/y}$$

In the case where a fluid coupling is used for changing the revolution, the expenses for modification are estimated as ¥ 2,800,000 (equivalent to 1,400,000 Ft) in the case of Japan. This idea should be taken into account if the reduction of output is to continue for a long time.

As the delivery pressure drops in proportion to the square of the revolution, it is necessary to check the delivery pressure required during reduced rate running.

b. Improvement of power factor

The motor load is light and, therefore, the power factor is low in general. Among all, the load of the red mud pump motor is low and the power factor is less than 50%. It is recommended that spare motors in the factory are checked to find a motor of suitable capacity and to replace the red mud pump motor with such a suitable motor.

The results of calculation of the power loss caused by the resistance of the cables to the motors made using the current value that corresponds to the actual load are also shown in Table 5.3.21. The total of the power loss with 20 motors is 12.7kW.

For motors of large cable loss, it is recommended that condensers are connected in parallel with motors. The required capacity of the condenser and of the series reactor is calculated by the following expressions.

$$C = P \times \frac{\sqrt{1 - \cos^2 \theta}}{\cos \theta} \text{ kvar}$$

$$L = C \times 0.06 \quad \text{kvar}$$

When condensers are connected to motors, the cable loss in the factory can be reduced by the reduction of the apparent power. Furthermore, since condensers are opened and closed in correspondence to running of motors, there is no fear of occurrence of lead power factor.

However, to modify the current equipment is not of good economy. It is recommended that such modification is examined at the occasions such as replacement of motors in the future.

c. Improvement of the operation of compressors

Many air compressors are used in the factory. It was not possible to investigate all the compressors this time. But it is recommended that check for excessive compressor delivery pressure against the air pressure required by the working equipment is made and to reduce the pressure in steps. The electric energy can be saved by about 4% when the pressure is reduced by 1 bar. Furthermore, it is recommended that check for leakage and identification of leaking points are made by the pressure drop in the pipelines and by the leaking noise, with the end valves closed during suspension of operation of the factory. Repair can be made when leaking points are found. It is often found that leakage of compressed air is beyond the anticipated level.



## 5.4 Results of investigation at a cement factory





## 5.4 Results of investigation at a cement factory

### 5.4.1 Outline of the factory

- (1) Company name and factory name: Beremendi Cement és Mészipari RT.
- (2) Category of business : Cement
- (3) Principal product name and production capacity  
Principal products : Cement 350-20, Lime  
Production capacity : Clinker 910,000 t/y
- (4) No. of employees : 850
- (5) Location of factory : H-7827 Beremend Pf. 20
- (6) History of the factory

The company was established in 1910, and production of cement had continued at a place that is different from the present site. It was 1973 when a new plant with suspension preheater was constructed at the present site. In 1981, the old plant was withdrawn and the lime calcination plant was newly constructed.

The factory is located at a place near the border to Yugoslavia and delivers the products to South Hungary and Yugoslavia.

The lime mine is located 4.5 km away from the factory, and limestone of 1,500,000 t/y and clay of 350,000 t/y are transported to this factory with dump trucks on a dedicated road. The mine is owned by the national government and the reserve is good for 18 - 20 years.

A computer system was introduced in 1984 for computerization for blending of the feedstock. In addition, the entire situations of the operation are acquired at the central control room to permit centralized management.

The organization was converted to a private corporation from a state-run business in November, 1988. One third of the shares of the capital of 2,000,000,000-2,500,000,000 Ft was acquired by Heidelberger Zement of Germany. The ratio of shares possessed by this company is scheduled to be increased to 80% in November, 1991. The remainder of the shares is owned by the holding organization of the government.

- (7) Investigation period: September 2 - September 6, 1991

(8) Investigators

Mitsuo Iguchi	Leader
Teruo Nakagawa	Subleader, Measuring Engineer
Toshio Onishi	Cement Process Engineer
Toshio Noda	Heat Control Engineer
Tetsuo Oshima	Heat Control Engineer
Kazuo Usui	Electrical Control Engineer

AEEF Member

Mr. János Becz	Team Leader
Mr. Ferenc Pardavi	Electrical Engineer
Mr. Kornél Jonás	Mechanical Engineer
Mr. Endre Slenker	Electrical & Measurement Engineer
Mr. László Szabó	Instrument Engineer

MVMT Member

Mr. Lajos Ropolyi	Mechanical Engineer
Mr. Balács Csovcics	

(9) Interviewees

Dip. Ing. Lajos Katona	Technical Director
Mr. Dezső Varga	Chief Energy Engineer
Mr. József Alföldi	Chief Engineer

(10) Progress of production (Table 5.4.1)

Name of Product	unit	1986	1987	1988	1989	1990
Clinker	1,000 t	625.8	719.7	787.1	738.4	767.5
Cement	1,000 t	747.8	836.8	852.9	794.2	898.9

(11) Progress of energy consumption (Table 5.4.2)

	unit	1986	1987	1988	1989	1990
Fuel Oil	kl	49,939	56,470	57,189	57,603	62,057
Tyre	t	99	95	82	94	65
Power	GWh	87.7	99.8	100.2	96.2	106.2
Water	1,000 t	9,774	10,785	9,757	10,877	7,823

(12) Energy unit consumption (Table 5.4.3)

	unit	1986	1987	1988	1989	1990
No.1 Kiln	kJ/kg-cl	3,820	3,750	3,617	3,727	3,687
No.2 Kiln	kJ/kg-cl	3,623	3,553	3,470	3,528	3,526
Raw Mill	kWh/t-fd	26.00	26.68	24.69	26.88	26.00
Clinker	kWh/t-cl	21.26	21.69	21.11	21.93	28.40
Cement	kWh/t-cm	28.65	30.95	28.32	25.93	29.50

(13) Operating hours (Table 5.4.4)

	unit	1986	1987	1988	1989	1990
Clinker	h	5,667	5,723	6,231	6,374	6,772
Cement	h	4,879	6,340	6,342	5,730	5,810

Reference operating hours for examination of countermeasures

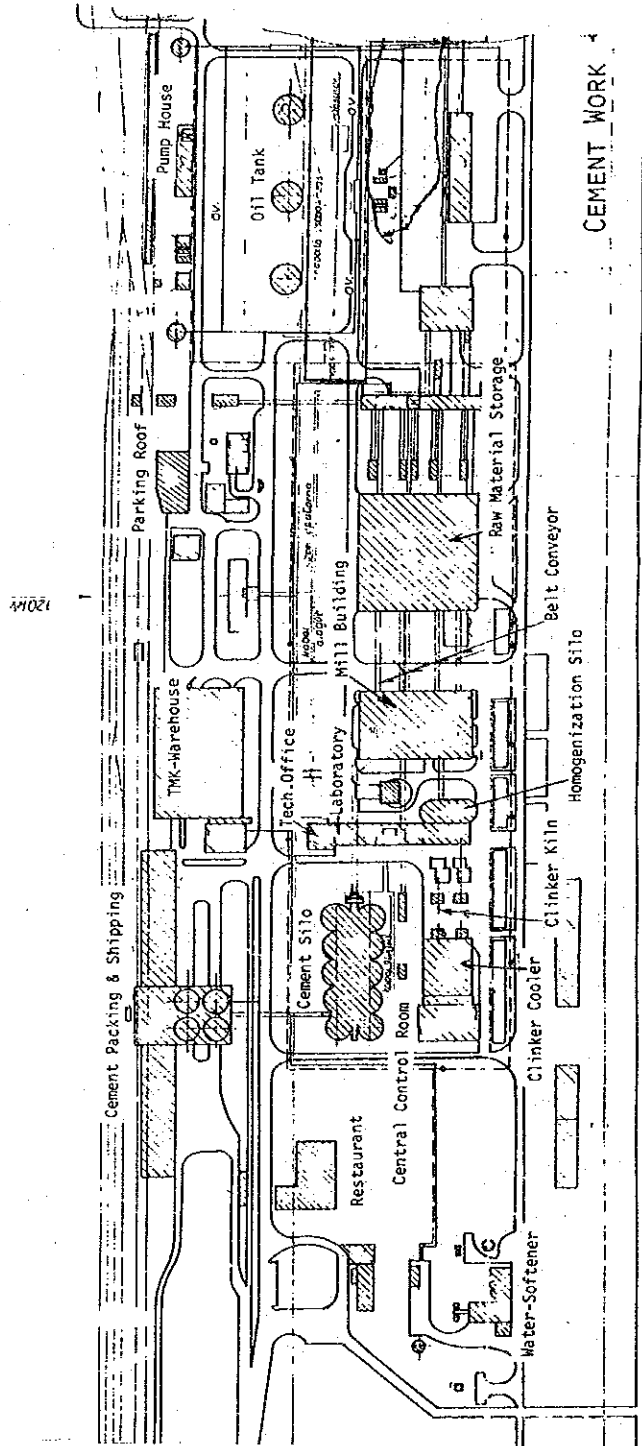
$$24 \text{ hours/day} \times 283 \text{ days/year} = 6,780 \text{ hours/year}$$

(14) Energy prices

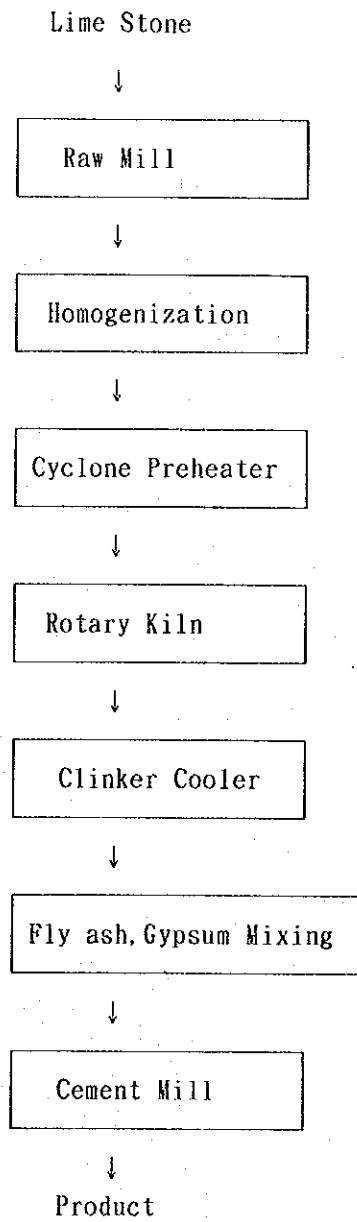
Fuel oil 7,040 Ft/t

Electric power 3.60 Ft/kWh

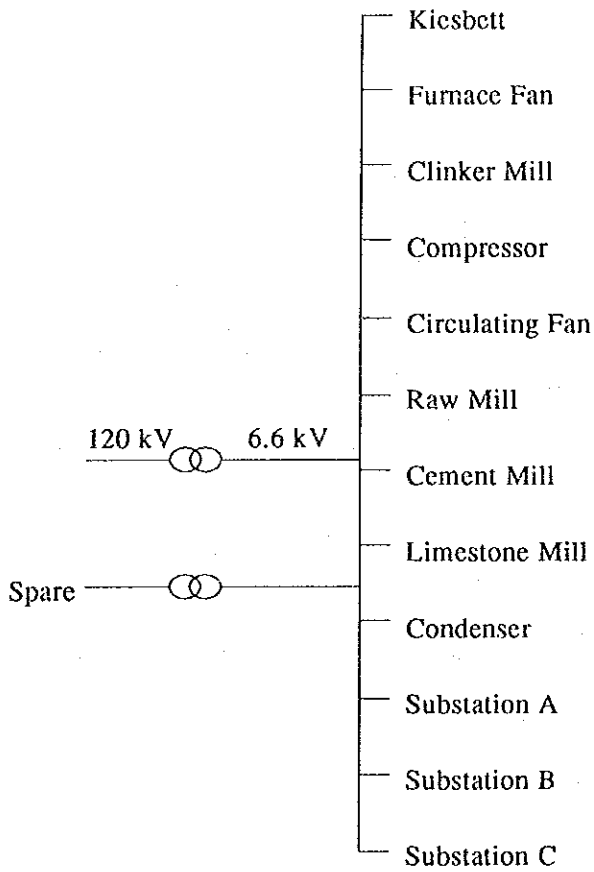
(15) Factory layout drawing (Figure 5.4.1)



(16) Manufacturing processes (Figure 5.4.2)



(17) Electric power one line diagram (Figure 5.4.3)



(18) Outline of principal equipment

Name	Number	Specification
Raw Mill	2	Ball Mill: 4.2 m $\phi$ $\times$ 9 mL, 140 t/h, 1,400 kW $\times$ 2, KHD Hammer Mill: 330 kW, KHD
Rotary Kiln	2	4.4m $\phi$ $\times$ 65mL, 1,500 t/d, 147 kW, 1.85 rpm, KHD Exhaust Gas Fan: 3,900 m <sup>3</sup> /min, 700 kW, KHD
Cement Mill	3	4 m $\phi$ $\times$ 12 mL, 150 t/h, 1,400 kW $\times$ 2, ZAB-DOR

## 5.4.2 Situations of energy management

### (1) Setting energy conservation target

In the aspect of facilities, the factory introduced process control with computers, improved measuring system, used tyres as substitute fuel, reduced compressed air flow rate for silo by the improvement of air distributor and improved classifier of pulverizing. When the achievements of recent five years are observed, the calcination energy unit consumption decreased by about 3%, but the electric energy unit consumption has remained unchanged.

No target of energy conservation has been established because the factory is currently in the process of transition to a private corporation. When the factory becomes a private corporation, a long-term program will be established first of all. At such an occasion, a clear target of improvement should also be established for energy.

In order to allow as many employees as possible to participate in the energy conservation activities to make full use of their capabilities, it is at first necessary that the top management appeals to all the employees that energy conservation is of great significance in the management and that it is a matter that should be targeted in the whole company. It is also important that the top management indicates a concrete target and asks for cooperation to hit this target. Only when the policy of the business is clearly indicated and the target value of each workshop is set up based on the said policy, the employees will search for problems in the works and equipment they are concerned, and will start examination of countermeasures for improvement for energy conservation actions in the whole company including examination of introduction of equipment, technological development and improvement of operation.

### (2) Systematic activities

The materials and energy department is in charge of the operations of energy management. Promotion of energy conservation activities is not done through the whole-company movement, but it is staff-oriented; that is, each staff member of this department determines a theme and promotes improvement. With such a system there is such a trend that the measures tend to be one-sided to countermeasures for equipment and it is not possible to make full use of the capabilities of all the employees. It is therefore desirable that the configuration of whole-company activities to include operators besides the staff is adopted in the future including the production department and the equipment department.

To implement whole-company evolution of energy conservation activities, it is important to have common recognition among departments regarding the situations of energy consumption and other problems. The situations of energy consumption are discussed at the production meeting held every month at this factory as attended by the factory superintendent and officers.

(3) Management by data

It is basically essential to seize the realities of consumption of energy, to compare them with the plan, and to take corrective measures through troubleshooting. The result data of the factory is an important information source for improvement of energy unit consumption because problems in the energy consumption and the effect of energy saving measures become clear only through analysis of data.

Operation of all processes is controlled at the central control room at this factory. With the calcination process and pulverizing process, the energy consumption is measured daily and the data are processed with computers. The entire operation data since 1972 have been recorded in the computer system.

To inform an operator of the situations of energy consumption is desirable as it leads to self-initiated energy saving activities. It is possible for the operators in the central control room to check the energy consumption as required.

Uplift of the general management level including quality control and equipment management is essential for strengthening management of energy. Also in the case of cement, to produce products of stable quality leads to improvement of energy unit consumption.

This factory has as many as 45 inspection staff, and they are mainly engaged in the check of product quality. With the feedstock, however, the invoice only is checked at the time of arrival of the feedstock at the factory and the materials are not checked at all. The method for control, therefore, is of feedback type to adjust the feedstock upon observation of the result of product analysis. This method involves a large time lag at large capacity equipment industries including the cement industry, and good management cannot be expected with such a method.

It is recommended that feedforward control—that is, the quality of the feedstock is controlled and the result values are compared with the product quality anticipated from these figures and operating conditions—is used in order to elevate the level of quality control. If the result values are different from the anticipated values, it indicates occurrence of deviation from standard values due to some factors including method for control of the feedstock, standard values set for processes and method for forecast. It is therefore necessary to make pursuit for the cause in each process and to take corrective measures. Such series of actions constitute the basis of quality control (see the guide line). Uplift of the quality control level can be achieved when such actions are repeated.



(4) Enlightenment of employees

Provision of sufficient information is indispensable to urge self-initiated activities of the employees. It is necessary to motivate the employees by providing information such as trend of energy prices, weight of energy cost in the production cost and cases of success at other factories. It is also essential to enlighten the employees to master the basic technology through training, distribution of manuals and so forth.

Engineers of this factory are trained as dispatched to foreign businesses and by attending training courses held by the State Authority for Energy Management and Safety (AEEF).

Operators are also trained by attending training courses on fundamental technologies such as handling of boilers and transformers.

Furthermore, meetings for exchanging information among businesses in the same trade are held in this trade. Although such meetings are not periodical, they are useful for stimulation for improvement.

(5) Management of equipment

The fuel piping system was well maintained and prompt actions were taken at occurrences of oil leakage; in the pulverizing process or the packing process, however, heavy dispersion of dust and particles came out and also many cases of compressed air leakage were observed. Since raw materials & products scattered and leaking compressed air have resulted from consumption of energy, to prevent these losses will be of great significance in terms of not merely environmental improvements but also energy conservation.

A suspension preheater was partly defective in heat insulation; otherwise, heat insulation situation was generally favorable.

### 5.4.3 Results of cement sample tests

- (1) Cement type : 350-20
- (2) Sampling period : Sampling was made once every hour between 10:00 of September 5 and 9:00 of September 6, and these samples were mixed together to produce one sample.
- (3) Samples  
 for general analysis : Clinker and cement  
 for particle size analysis : Raw materials supplied to two air separators of No. 1 mill as well as fine powder and return powder of this mill
- (4) General analysis  
 a. Chemical analysis (cement) (%)

**Table 5.4.5 Chemical analysis**

Ignition Loss	Insoluble matter	Free-CaO	Cl
1.2	11.4	0.5	0.002

SiO <sub>2</sub>	Al <sub>2</sub> O <sub>3</sub>	Fe <sub>2</sub> O <sub>3</sub>	CaO	MgO	SO <sub>3</sub>	Na <sub>2</sub> O	K <sub>2</sub> O	Total
18.5	5.0	3.1	56.7	0.8	1.9	0.36	0.64	99.60

b. Physical testing (cement, mortar)

Table 5.4.6 Physical testing

Specific Gravity	3.02	g/cm <sup>3</sup>	
Specific Surface	2,400	cm <sup>2</sup> /g-Blaine	
Setting (Water = 26.3%)	initial	2h	54 min
	final	4h	20 min
Flow	225	mm	
Strength (kgf/cm <sup>2</sup> )	3	7	28 [days]
Compressive	86	113	187
Bending	24	32	45

c. Considerations on test results

- a) This sample is considered to be of fly ash cement similar to type B specified in JIS (Japanese Industrial Standards) as judged from its insoluble content. As the values of free CaO and chlorine are sufficiently small, this cement is good as for this point.

[For reference] Japanese Industrial Standard: JIS R-5213 (1979), Fly Ash Cement

Table 5.4.7 JIS R-5213 (1979), Fly ash cement

Type		A	B	C
Fly Ash	(wt %)	5 - 10	10 - 20	20 - 30
Specific Surface Area	(cm <sup>2</sup> /g)	>2,500	>2,500	>2,500
Setting	Initial (min)	>60	>60	>60
	Final (hr)	<10	<10	<10
Stability		good	good	good
Compressive Strength (kgf/cm <sup>2</sup> )	3 days	>70	>60	>50
	7 days	>150	>120	>100
	28 days	>300	>260	>210
MgO	(%)	<5	<5	<5
PO <sub>3</sub>	(%)	<3	<3	<3
Ignition Loss	(%)	<3	—	—

- b) However, the compressive strength and bending strength are considerably low compared to what are specified in Japanese Industrial Standards shown in Table 5.4.7 and the test result values of Japanese fly ash cement on the market shown in Table 5.4.8.

Table 5.4.8 Test results of various cement products on the market

Japan Cement Association (1988)

	Chemical analysis			Physical property							Hydraulic heat					
	ig. loss	MgO	SO <sub>3</sub>	Spec. gravity	Granulometry		Setting			Compr. strength (kgf/cm <sup>2</sup> )		Bend. strength (kgf/cm <sup>2</sup> )				
					Spec. Surface (cm <sup>2</sup> /g)	St. sieve R-90 μ(%)	Initial (h-m)	Final (h-m)	Stability	Flow	3d	7d	3d	7d		
															28d	28d
Ordinary Portland Cement	1.1	1.6	1.9	3.16	3370	0.6	2-29	3-27	good	253	156	264	434	38	54	78
High-Early Portland Cement	1.0	1.5	3.0	3.14	4460	0.1	2-08	3-01	good	247	268 <sup>*1</sup>	363	479	56 <sup>*1</sup>	69	84
Medium-Heat Portland Cement	0.6	1.1	1.9	3.22	3200	0.5	4-07	5-22	good	252	96	150	350	27	37	63
Sulphate-Resistant Portland Cement	0.5	1.0	1.9	3.21	3300	0.6	3-39	5-06	good	255	140	213	335	34	46	65
Blast-furnace Slag Cement A	1.2	3.2	1.8	3.09	3940	0.5	2-22	3-18	good	252	142	229	412	34	49	70
B	1.2	3.6	1.9	3.04	3980	0.5	3-08	4-19	good	250	123	207	419	31	44	73
C	1.2	5.2	2.0	2.91	3790	0.5	4-02	5-34	good	251	71	160	366	21	34	71
Pozzolan Cement A	0.7	1.3	1.9	3.11	3480	0.9	2-40	3-30	good	250	142	237	382	30	48	67
Fly Ash Cement A	0.7	1.1	1.7	3.07	3200	1.6	2-37	3-32	good	276	112	205	357	27	42	66
B	1.1	1.5	1.8	2.98	3430	0.8	3-01	4-12	good	258	126	206	360	32	48	69
C	1.1	1.1	1.5	2.83	3100	2.3	5-28	7-04	good	269	88	163	279	25	38	56

Note: \*1 1 day values of this case, 149 for Compr. S. and 38 of Bend. S.

\*2 C3A of this case, 4%

\*3 C3S of this case, 44%

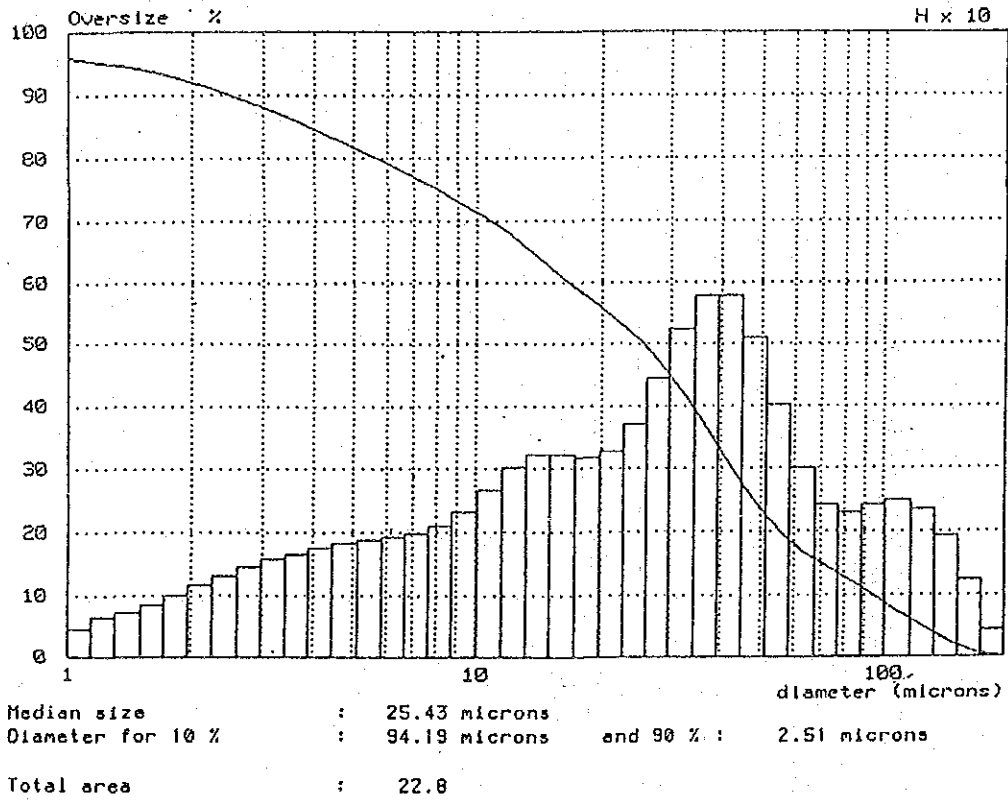
(5) Particle size analysis

Both of particle diameter and Blaine are extremely rough.

Cilas - Alcatel

Granulometer HR 850 119

Suspension fluid : Ethanol -  
 Dispersing agent :  
 Ultrasonic mixer time : 1:8 0s



10	1.0	1.2	1.4	1.6	1.8	2.0	2.2	2.6	3.0	3.5
1 %	95.9	95.1	94.4	93.6	92.9	92.1	91.3	89.7	88.1	86.3
10	4.0	4.5	5.0	5.5	6.3	7.0	8.0	9.0	10.0	12.0
1 %	84.7	83.1	81.7	80.4	78.5	77.0	75.1	73.3	71.5	68.0
10	15.0	18.0	20.0	22.0	25.0	28.0	32.0	36.0	40.0	45.0
1 %	62.6	58.3	55.9	53.7	50.5	47.2	42.4	37.4	32.7	27.5
10	50.0	56.0	63.0	75.0	90.0	105.0	125.0	150.0	175.0	200.0
1 %	23.4	19.7	16.8	13.8	10.8	7.8	4.7	1.8	0.4	0.0

Figure 5.4.4 Particle size distribution

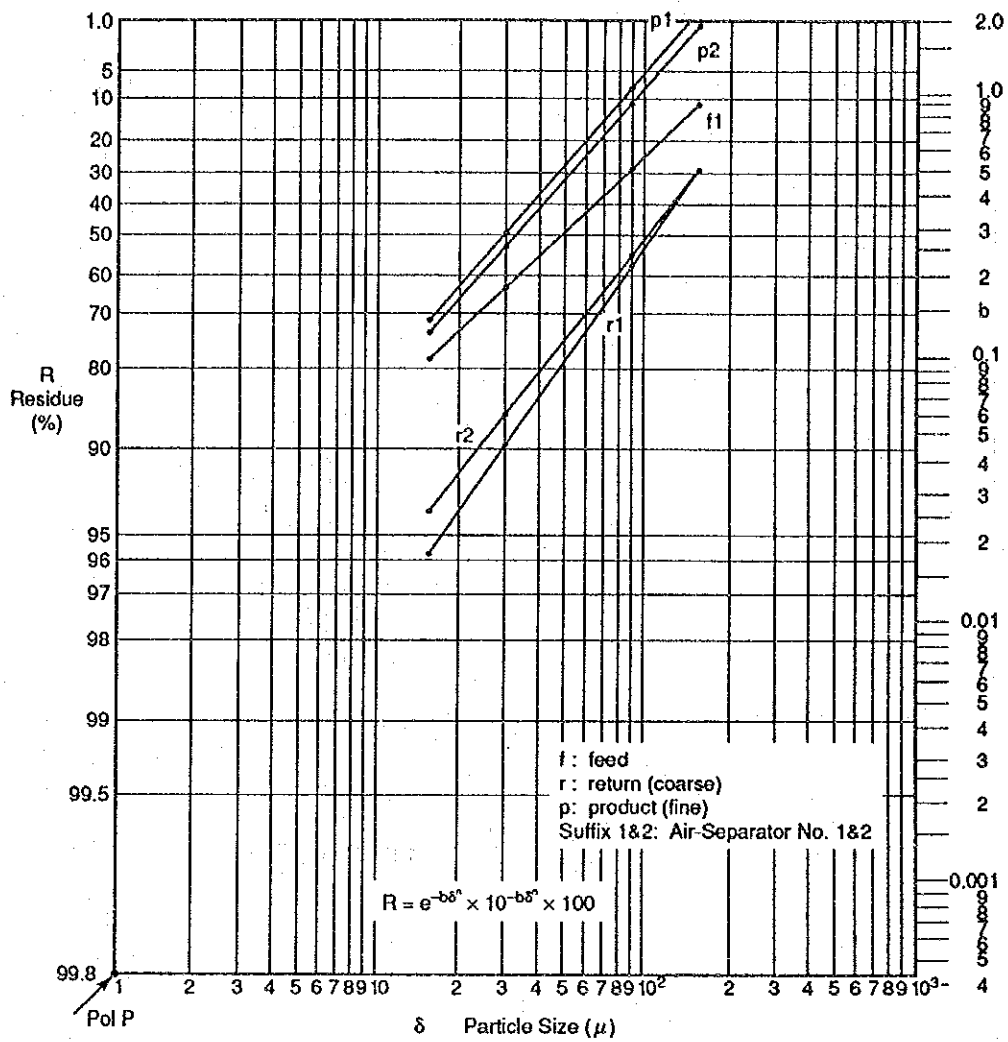
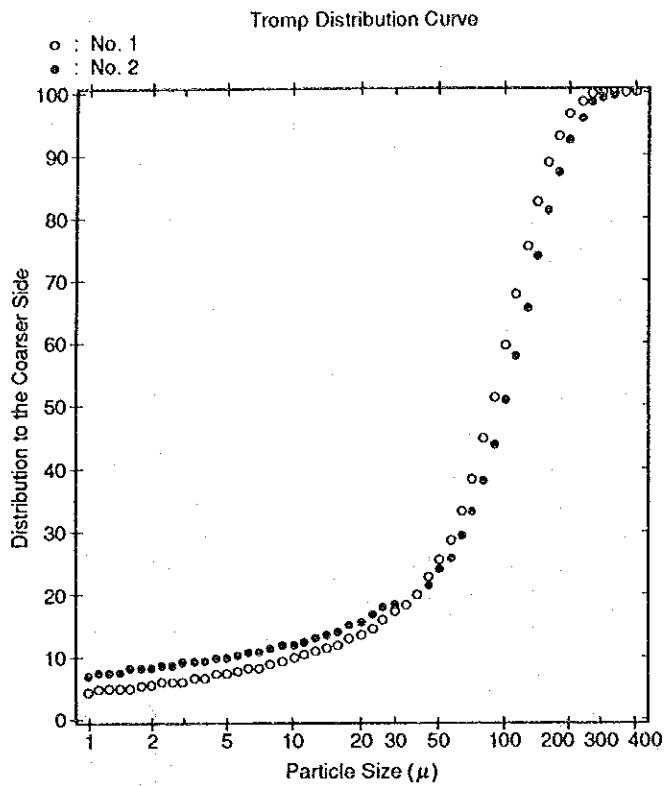


Figure 5.4.5 Rosin-Rammler chart (Granulometry distribution curve)

The Tromp curve indicates considerably good separation efficiency because of the reasons stated below.

- a) The circulation load is as low as 0.55 and 0.60 because the rate of charge to the air separator is extremely low.
- b) The particle size of 50% separation is nearly  $100\mu$ , which is astonishingly rough.



○ : No. 1

Particle Size	15	30	90	150	n
Effective Size	15.0	30.0	90.0	150.0	
f	80.4	64.2	25.8	11.4	1.002
r	95.6	88.7	56.2	30.0	1.427
p	71.9	48.6	6.7	0.7	1.181
$\lambda$	0.3587	0.3890	0.3859	0.3652	0.3747

df	19.6	16.2	38.4	14.4	11.4
dr	4.4	6.9	32.5	26.2	30.0
dp	28.1	23.3	41.9	6.0	0.7
dR	1.6	2.6	12.2	9.8	11.2
dP	17.6	14.6	26.2	3.8	0.4
dY	19.2	17.2	38.4	13.6	11.7
Zr	8.6	15.1	31.7	72.3	96.3

CL: 0.5992 (Circulation Load)

$\lambda$ : 0.37468

● : No. 2

Particle Size	15	30	90	150	n
Effective Size	15.0	30.0	90.0	150.0	
f	80.4	64.2	25.8	11.4	1.002
r	93.9	85.9	54.5	30.8	1.270
p	73.4	51.9	8.9	1.2	1.161
$\lambda$	0.3415	0.3618	0.3706	0.3446	0.3546

df	19.6	16.2	38.4	14.4	11.4
dr	6.1	8.0	31.4	23.7	30.8
dp	26.6	21.5	43.0	7.7	1.2
dR	2.2	2.8	11.1	8.4	10.9
dP	17.2	13.9	27.8	5.0	0.8
dY	19.3	16.7	38.9	13.4	11.7
Zr	11.2	17.0	28.6	62.8	93.4

CL: 0.5494

$\lambda$ : 0.35461

Figure 5.4.6 Tromp distribution curve



#### 5.4.4 Problems in the use of energy and countermeasures

##### (1) Preparation of the feedstock - Prehomogenization

###### a. Effect of prehomogenization

The argillaceous component of the main component (usually limestone) of cement feedstock is relatively homogeneous, but the  $\text{CaCO}_3$  content varies to a considerable extent.

Selective quarrying was made while the feedstock quarrying rate was relatively small. However, it became uneconomical as the quarrying rate increased, and the necessity for prehomogenization increased.

The feedstock prehomogenization method has been used since 1905 for preparation of ore composition, and it has penetrated to almost all cement businesses by this time. Stacking on the blending bed is usually made along the longitudinal axis (longitudinal stacking), and reclaiming is made at the section that is rectangular to the longitudinal axis. The roof type (Chevron method) using a belt conveyor with tripper (Figure 5.4.8) is most frequently used for stacking. (Figure 5.4.7)

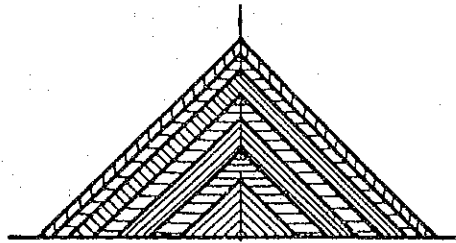


Figure 5.4.7 Roof type storage on blending bed (Chevron method)

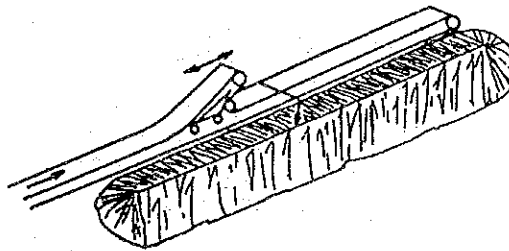
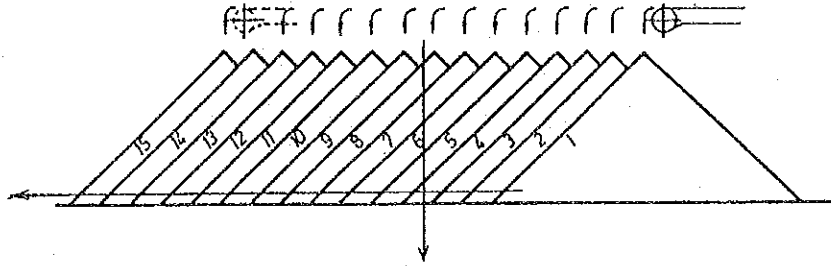


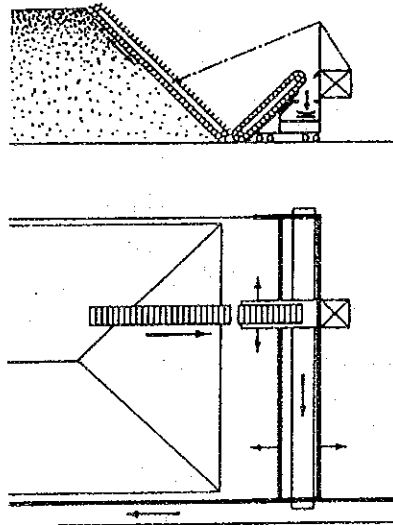
Figure 5.4.8 Longitudinal storage stacking with belt conveyor and tripper

Furthermore, stock piling is also used frequently. (Figure 5.4.9)

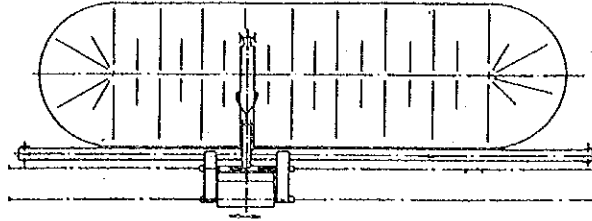


**Figure 5.4.9 Stock piling on blending bed**

Reclamation is made in such a manner that a scraper reclaims the stock pile from the front end or while moving by the stock pile in parallel with the longitudinal axis. Figure 5.4.10 indicates a scraper for reclamation from the stock pile front face, and Figure 5.4.11 indicates a reclaimer scraper that moves by the stock pile.



**Figure 5.4.10 Reclaimer scraper for reclamation from front face of the blending bed**



**Figure 5.4.11 Reclaimer scraper for reclamation along the blending bed**

Regarding the effect of prehomogenization, Duda describes that the variation of the  $\text{CaCO}_3$  content of the limestone drops from 10% to 1.5% or even less and then drops to 1/10 in the prehomogenizing silo in the following process.

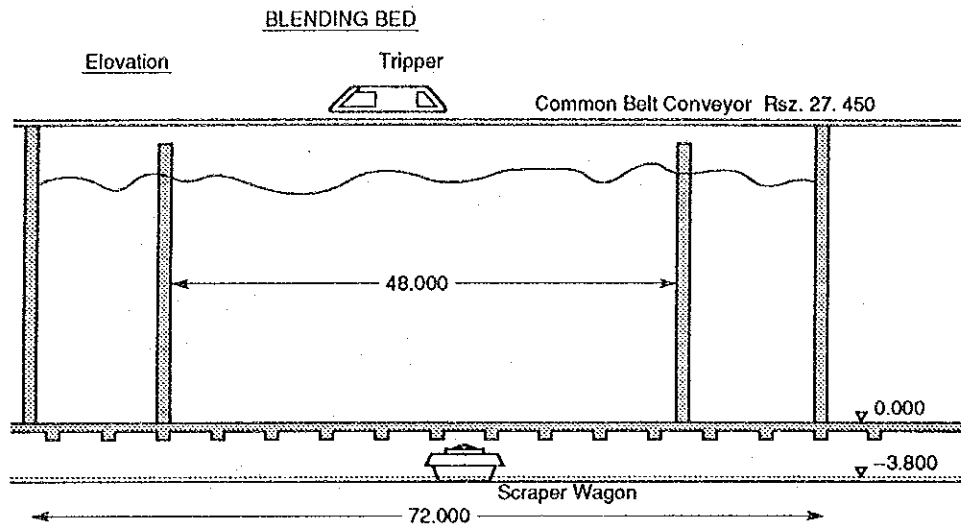
In the announcement made at VDZ Conference (1977), Mr. D. Schmidt indicated the test results shown in Table 5.4.9 regarding the effect of prehomogenization.

**Table 5.4.9 Effect of prehomogenization**

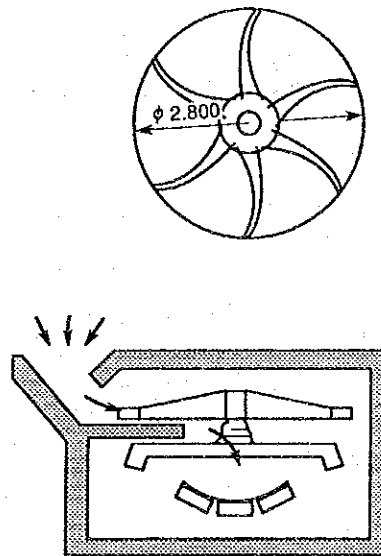
Process	Standard Deviation		Blending Effect $S_\alpha / S_\beta$
	$S_\alpha$	$S_\beta$	
Blending Bed	40.5	15.1	3
Homogenizing Silo	7.1	1.4	5

b. Situations of this factory

An elevation of a section of the blending bed for limestone is shown in Figure 5.4.12. This bed is of upper belt conveyor system with tripper identical to what are shown in Figure 5.4.7 and Figure 5.4.8.



**Figure 5.4.12 Blending bed**



**Figure 5.4.13 Scraper and structure**

Extraction is made from the space under the ground at the bottom of the blending bed using a scraper wagon. However, the wagon is run usually as stopped at one point, and only two or three extraction points are used out of many extraction points. Even if blending may be made to a certain extent, the feedstock is reclaimed at a time in almost unblended state, and it is not a desirable situation at all.

c. Plan for improvement

Such a method is recommended that the wagon slowly and continuously makes round trips along the line under the ground to make collection from No. 1 through No. 15 shown in Figure 5.4.9 almost simultaneously.

Since this wagon is originally movable, it is satisfactory if its electrical or electronic functions are exhibited.

The management of the blending bed is not so good. It is first necessary to implement clean-up and rearrangement of the feedstock, and it is also necessary to keep paying attention to the conditions of the blending bed.

It is recommended that consultation is made with equipment manufacturers and principal contractors regarding the method for improvement of the blending bed. It is estimated that the expenses for modification are as minor as  $50 - 100 \times 10^3$  US\$, although this amount varies by the conditions. The effect in this case is estimated as about one half of that of the case where a blending bed for prehomogenization requiring  $3,000 \times 10^3$  US\$ besides the land cost is newly constructed.

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Vorhomogenisierung (Prehomogenization) [16.a] S. 303-316

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H. Berthold, St. Ingbert-Rohrbach 118-121

Homogenisierungseffekt bei Längs-und Kreismischbetten mit Brüchenkratzern

D. Schmidt, Hardegsen 132-140

Der Vergleichmäßigungseffekt der gesamten Rohmaterialaufbereitungskette des Werkes Hardegsen  
.....

(2) Pyroprocessing - clinker production

a. Situations of operation

Principal running data of the rotary kiln and of peripheral equipment such as raw mill and cement ball mill were recorded during the investigation period. A part of these data was selected to explain typical kiln operation. The variation of four running indices is shown in Figure 5.4.14. The running period is classified into three, and the mean values and differences between maximum values and minimum values of seven indices are shown in Table 5.4.10.

Date : 10:00 on September 4 ~ 9:00 on September 5

Kiln : No. 1 kiln

Classification:	Status	Period	No. of data
SD-1	: Reduced rate operation - 1	9/4 13:18 ~ 9/4 14:49	92
SS-1	: Stable state - first half	9/4 15:00 ~ 9/4 21:30	391
SS-2	: Stable state - second half	9/4 21:31 ~ 9/5 4:00	391
SD-2	: Reduced rate operation - 2	9/5 5:00 ~ 9/5 8:30	211

SEP.04/10 05/09, 1991- BCM #1 KILN

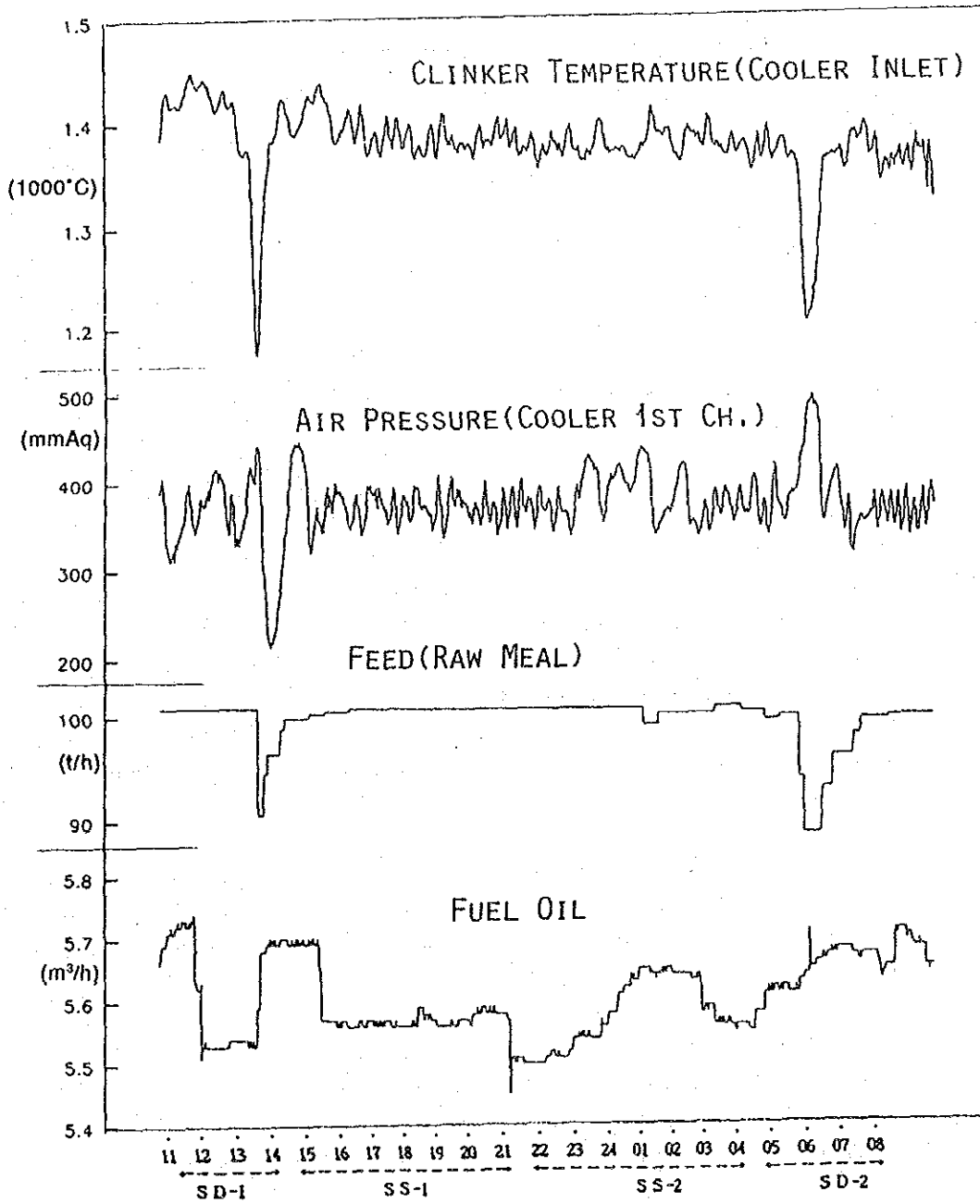


Figure 5.4.14 Situations of kiln operation

Table 5.4.10 Kiln operation data

		SD-1		SS-1 & 2		SD-2	
		$\bar{x}$	R	$\bar{x}$	R	$\bar{x}$	R
Feed	t/h	97.5	9.2	100.6	1.7	96.5	11.3
Fuel Oil	m <sup>3</sup> /h	5.69	0.11	5.57	0.35	5.66	0.10
O <sub>2</sub> -Kiln Outlet	%	2.84	1.61	2.96	2.12	3.13	1.81
-SP Outlet	%	3.80	0.45	3.70	0.54	3.64	0.52
Clinker Temperature	°C	1,365	298	1,381	132	1,341	259
Cooler Air-Pressure	mmAq	346	233	376	233	385	179
Kiln Power	kW	69.8	56.3	81.2	63.9	86.4	61.3
<Number of Data>		92		781		211	

$\bar{x}$ : mean-value      R: [max. – min.]-value

It is considered that two reduced rate periods (SD-1 and SD-2) occurred because the operator reduced supply of the feedstock by 10% upon observation of rapid drop of the clinker temperature which occurred due to collapse of the build up in the kiln. The fuel oil flow rate was changed simultaneously in correspondence to the change in the feedstock flow rate. But this way of doing was not suitable. As the feedstock retention time is long and the thermal capacity is large, a time lag occurs in the changes in the kiln internal temperature. Therefore, control of the fuel oil should have been made in steps, to a minor extent at a time.

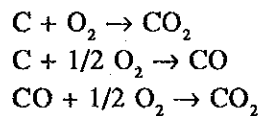
During the intermediate stable state period, the kiln was completely stable in the first half (SS-1). But at around 21 hours the operator rapidly and largely reduced the fuel supply (from 5.58 to 5.45 m<sup>3</sup>/h). After this, the fuel supply was gradually increased and the kiln conditions were recovered in 3.5 hours.

Throughout the entire period, the clinker temperature at the cooler inlet gradually decreased. Clinker temperature is one of the most important indices, and it should be checked frequently. It is extremely important that efforts are made to recover the temperature when it is noticed that the temperature is low.



b. Control of combustion

Fundamental understanding of combustion is required first of all. Whether combustion is satisfactory or not can be judged from the oxygen content and carbon monoxide content in the burnt gases. If it is assumed that the combustible is carbon only for simplification, the reaction of combustion is as follows.



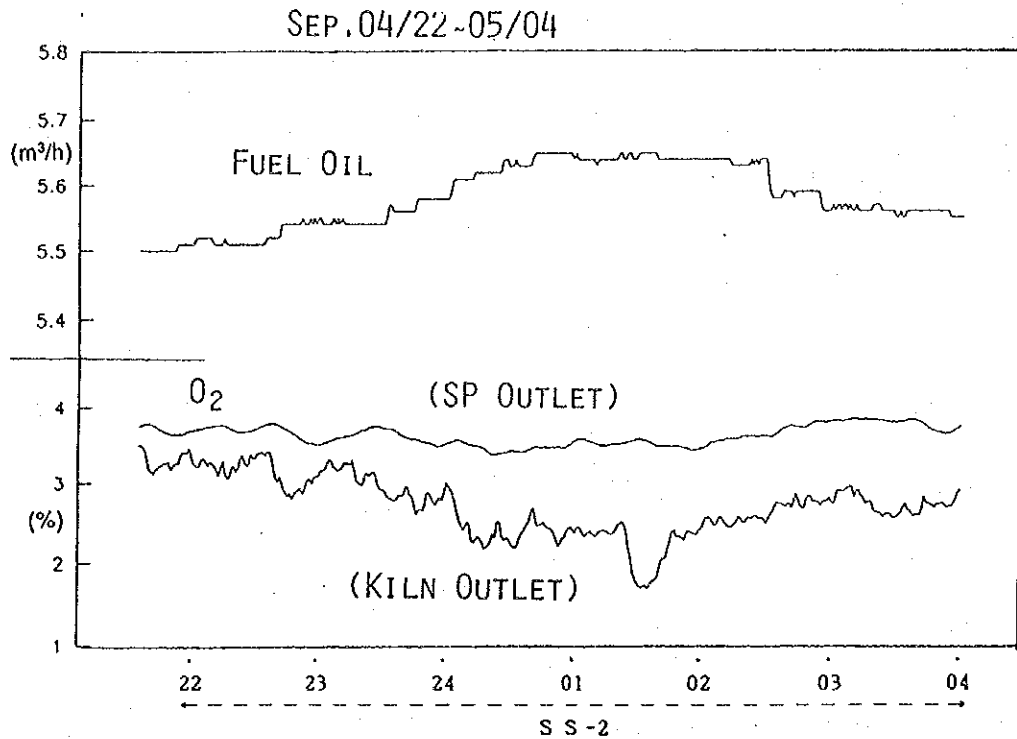
The second reaction to become carbon monoxide occurs when the temperature is low or when the oxygen concentration is low. Ideally, no carbon monoxide should be generated when excess oxygen is present. In the practical combustion, however, because of lag in the mixing, excess oxygen of a certain rate is necessary for suppressing generation of carbon monoxide. The O<sub>2</sub> content and CO<sub>2</sub> content in the exhaust gas are as follows at a rotary kiln of the cement factory with satisfactory combustion.

$$\text{CO} = 0 - 0.1\% \quad \text{O}_2 = 1 - 2\%$$

However, to increase the excess oxygen rate to a level that is beyond the necessary level causes various losses because nitrogen is fed into the kiln simultaneously by a rate that is near four times as much as the oxygen feed rate.

- ① The flame temperature drops and heat transfer by radiation decreases.
- ② The exhaust gas flow rate increases and heat loss and power of IDF increase.
- ③ The freeness of operation is reduced and drop in the capacity of the kiln results.

Figure 5.4.15 indicates changes in the fuel oil flow and oxygen concentration in the exhaust gases. Close correlation is observed between the fuel oil flow and the oxygen concentration. Since good clinker is obtained even when the oxygen concentration is low, efforts should be made to reduce the excess air by controlling the damper of the IDF.



**Figure 5.4.15 Fuel oil flow and oxygen concentration in exhaust gases**

The oxygen concentration in the kiln outlet gas and the oxygen concentration in the IDF inlet gas are also closely related to each other. But there is a large difference between them at the same time. It is because, while the oxygen value in the kiln outlet gas changes directly by combustion of the fuel oil, the IDF inlet gas involves large air leak-in through many feedstock valves appended to the suspension preheater (or cyclone heat exchanger) as the IDF is operated at the full capacity. But this leakage can be easily overcome by checking seals of valves at the time of completion of repair and before starting the running.

The correlation between the oxygen concentration in the kiln outlet gas and the oxygen concentration in the IDF inlet gas is shown in Figure 5.4.16. It is learned from this figure that even when the oxygen concentration in the kiln outlet gas becomes zero, the oxygen concentration in the IDF inlet gas remains at around 3%, meaning that air is leaking in through components of the suspension preheater.

The correlation between the oxygen concentration in the kiln outlet gas and the fuel oil flow rate is shown in Figure 5.4.17. There is such a relation that when the fuel oil flow rate increases by 0.02 m³/h from a certain level, the oxygen concentration in the kiln outlet gas decreases by 0.14%.

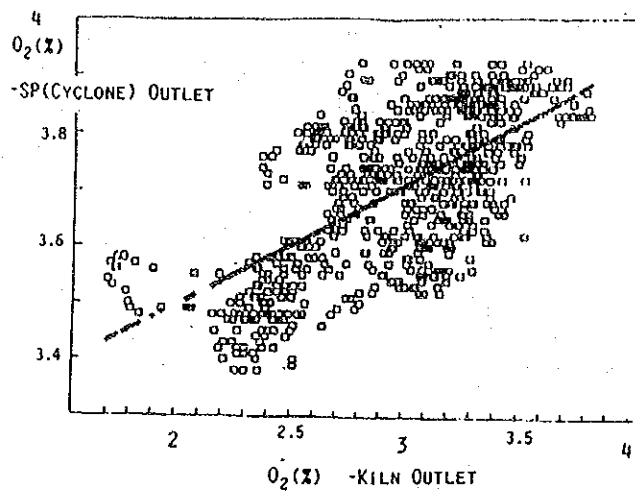


Figure 5.4.16 Correlation between oxygen concentration in the kiln outlet gas and oxygen concentration in the IDF Inlet gas

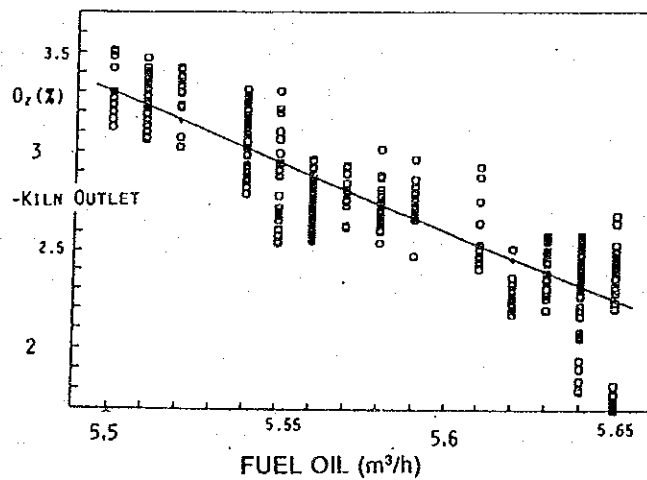


Figure 5.4.17 Correlation between oxygen concentration in the kiln outlet gas and fuel oil flow rate

At any rate, excess air is large and the energy loss is large. It is important to keep the oxygen concentration in the kiln outlet gas at 1~1.5%. Fortunately, the burner is functioning in a considerably satisfactory manner. Therefore, it is possible to hit this target. As the first and effective means, it is recommended to perform running at a lower oxygen concentration level even if carbon monoxide is generated by around 0.1% (assume that the carbon monoxide meter is operating in the good condition).

The temperature of the primary air can be increased to 200 - 250°C when it is supplied from the clinker cooler exhaust. The combustion is also improved if supply of the primary air is reduced to 15% or less, 10% or less if possible, of the entire air flow and if secondary air of high temperature is used at a larger rate that corresponds to reduction of the primary air.

The internal structure of the burner is extremely important for maintaining good combustion. It is therefore recommended that tests are repeated with reference made to relevant papers or with consultation made with the burner supplier.

In the case where the fuel is converted to solid fuel, it is recommended that the oxygen concentration control value is set at 1.5 - 2% because combustibility of solid fuel is inferior.

The fuel saving effect obtained at the time when the oxygen concentration in the kiln outlet gas is reduced to 1.5% from 3.0% is as follows. However, it is assumed that the quantity of CO<sub>2</sub> generated as a result of decomposition is kept fixed, and only the exhaust gases generated as a result of combustion of fuel are considered.

When it is assumed that the fuel heating value (H<sub>1</sub>) is 40,000 kJ/kg, preheater outlet exhaust gas temperature is 365°C, the ambient temperature is 20°C and the exhaust gas specific heat is 1.59 kJ/(Nm<sup>3</sup>-°C), the exhaust gas loss is as follows.

Theoretical airflow: by Boie's formula

$$A_0 = 2.957 \times (H_1 - 4,605)/10,000 = 10.47 \text{ Nm}^3/\text{kg-Fuel}$$

Roughly calculated air ratio:

When oxygen concentration in the exhaust gases is 3.0%

$$m_1 = 21/(21 - 3) = 1.17$$

When oxygen concentration in the exhaust gases is 1.5%

$$m_2 = 21/(21 - 1.5) = 1.08$$

Reduction of exhaust gas flow due to drop in the air ratio

$$(m_1 - m_2) \times A_0 = (1.17 - 1.08) \times 10.47 = 0.94 \text{ Nm}^3/\text{kg-Fuel}$$

Reduction of exhaust gas heat loss due to the air ratio improvement

$$0.94 \times 1.59 \times (365 - 20) = 516 \text{ kJ/kg-Fuel}$$

Approximate fuel saving rate

$$62,057 \text{ t/y} \times \frac{516}{40,000} = 62,057 \times 0.013 = 800 \text{ t/y}$$

Fuel saving amount

$$800 \text{ t/y} \times 7,040 \text{ Ft/t} = 5,632,000 \text{ Ft/y}$$

c. Control of kiln and cooler

1) Kiln power

Figure 5.4.18 indicates the progress of kiln power (kW) throughout the entire period.

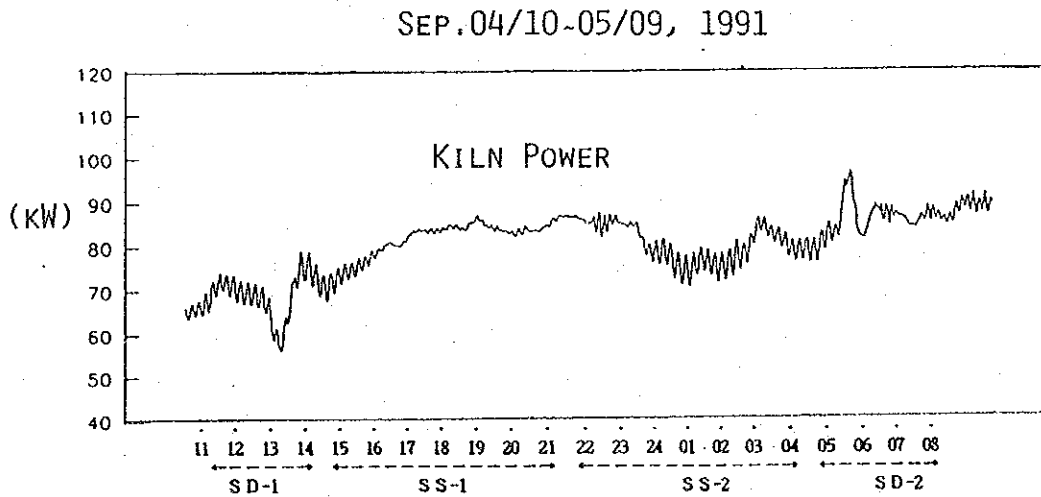


Figure 5.4.18 Progress of kiln power

During the reduced rate period (SD-1, SD-2), the feedstock feed rate and kiln revolution were intentionally changed. In the first half of the stable period (SS-1), the kiln power slowly changed and no clear periodic variation is indicated. In the latter half of the stable period (SS-2), however, stability occurred again and typical periodic variation is indicated. This periodic variation involves a fixed period of about 10 minutes, but no correlation with any other running indices is observed. It is considered to be probably of the following reasons.

- ① Movements of solids are similar to those of breathing phenomena. The majority of such movements are caused by the accumulation-relaxation phenomena by weir effect of refractory structure, ring formation or periodic attachment/decay.
- ② The kiln makes periodic reciprocating motions on the tyres of the kiln drive unit.

2) Clinker temperature and cooler air pressure

The density of the air is inversely proportional to the absolute temperature and the viscosity of the air is proportional to the square of the absolute temperature. Therefore, the air pressure in the clinker layer on the cooler grid varies by the layer thickness and air temperature.

Figure 5.4.19 indicates the progress of clinker temperature and cooler air pressure in the first half of the stable period (SS-1). Periodical variation of 16-minute is observed. It apparently indicates breathing phenomenon of a certain kind, but it is hard to relate it to periodic variation of the kiln power.

SEP.04/15~21, 1991

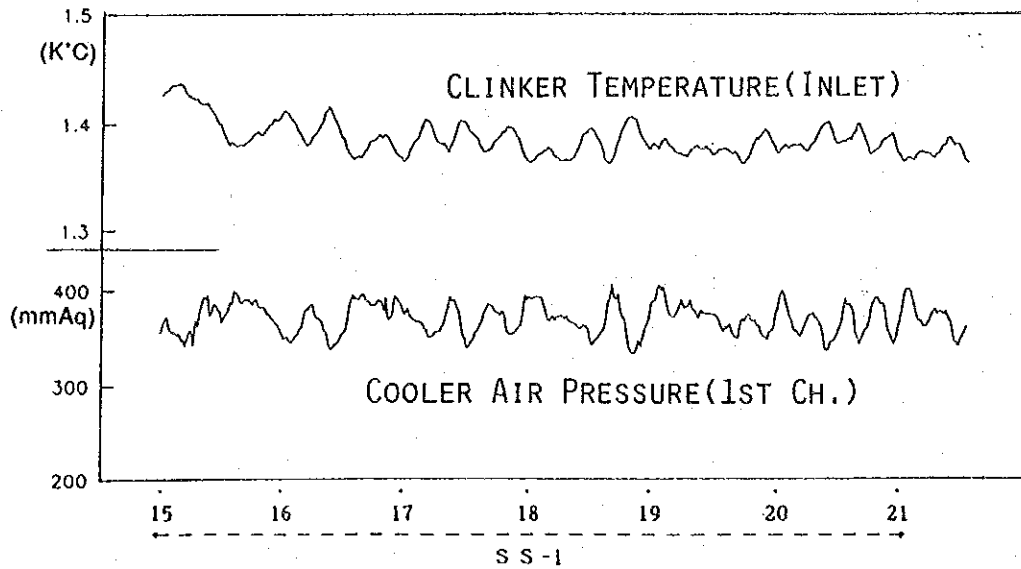
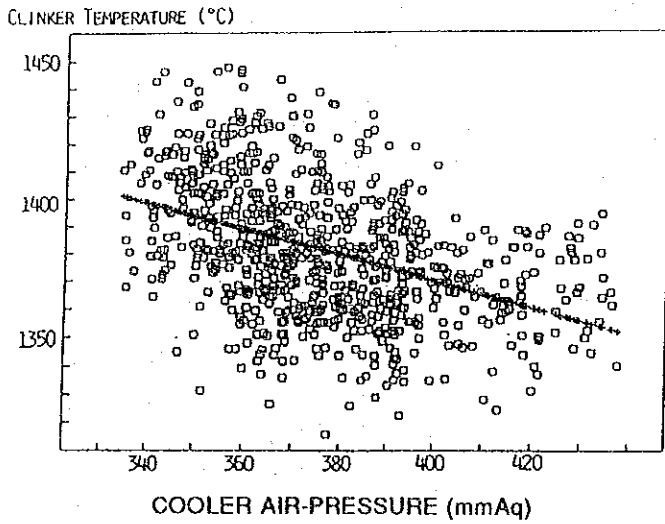


Figure 5.4.19 Progress of clinker temperature and cooler air pressure

Furthermore, it appears that there is relation between the clinker temperature and the cooler air pressure when the graph is observed. Figure 5.4.20 indicates the relation between them for the entire stable period. It is considered that there is a difference of four or five minutes between the time of measurement of two indices, and no clear trend is observed. However, the correlation factor (R) is high compared to the number of data (n), and the measured values are significant from the standpoint of statistics.

From the data mentioned above, the air pressure drops by 20 mmAq when the clinker temperature rises by 10°C.



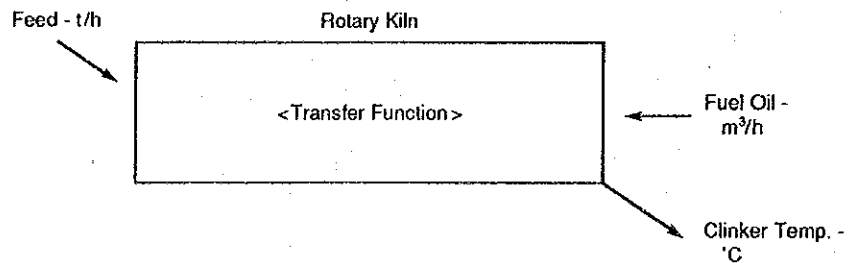
	Cooler Air Pressure(x) v Clinker Temperature(y)
Correlation Index R	0.43
No. of Samples n	781
Regression Line x/y	$y = -0.47x + 1560$

**Figure 5.4.20 Correlation between clinker temperature and cooler air pressure**

The clinker's mean temperature is 1,365°C, which is of a high level that was never observed before. The usual mean temperature is 1,250~1,350°C. To constantly and continuously measure the clinker temperature is not easy. But if it is assumed that the measured clinker temperature is correct, it apparently indicates overcombustion. However, the reason why the cement quality is not good in particular is that there are problems in other processes such as cement grinding process.

### 3) Control

The kiln is a kind of transfer function as shown by the model diagram shown in Figure 5.4.21. It produces an output as a result upon receipt of an input.



**Figure 5.4.21 Concept of control**



Assume that feedstock supply rate (t/h) and fuel oil flow rate (m<sup>3</sup>/h) are input and the clinker temperature is the output and that all the other indices are constant. The target of the control is to keep the clinker temperature constant. When the clinker temperature rises, therefore, correction is made by increasing the feedstock supply rate, by decreasing the fuel oil flow rate or by execution of both of these measures within the range of the equipment restrictions in correspondence to the control guideline. The clinker temperature is called controlled variable and the feedstock supply rate and fuel oil flow rate are called manipulated variables. The equipment restrictions include capacity upper limit and turn down ratio.

The probable methods to control each one or both of feedstock supply rate and fuel oil flow rate mentioned above are as follows.

- ① In the case where the damper of the IDF is fully open and the fuel oil flow rate is fixed, adjustment of the feedstock supply rate should be made in correspondence to minor changes in the clinker temperature, with time lag and retention time taken into account.

When the coating (Ansatz) of the kiln collapses, the feedstock supply rate should be rapidly and drastically reduced, and at the same time, the fuel oil flow rate should be delicately controlled in correspondence to the situations.

- ② In the case where the feedstock supply rate is fixed, variation of the clinker temperature and so forth should be controlled by the means to change the fuel oil flow rate. This method is simple and is excellent when the time lag is taken into account.

There are no problems with the suspension preheater except that drop-off of heat insulation was partly observed. This preheater provides the feature called self-controllability, and it is capable of absorbing turbulence.

Control of the cooler is extremely important to satisfy the following conditions.

- ① To establish cooling conditions that affect the quality of cement minerals, rapid cooling conditions in the high temperature area in particular.
- ② To supply secondary air of high and uniform temperature with little dust to the kiln at a suitable rate that does not disturb drafting in the kiln.
- ③ To cool to low temperature and to spray water for treatment of the clinker.

The clinker temperature at the cooler outlet is 287°C at maximum, 54°C at minimum and 65°C at average according to 781 data during SS-1 and SS-2 periods. Overcooling is rather observed. Overcooling requires drafting at a large rate, and power loss is involved.

If water spraying to crushed high temperature clinker is automated, the clinker temperature may be increased to 100°C, for example, in the range where the clinker will not cause damage to the belt conveyor.

(3) Conversion of fuel

The fuel cost occupies 40% of the production cost in the cement industry. It is an extremely high figure among all the industries. Therefore, all of the concerned parties in the world are looking for more inexpensive fuel and waste substitute fuel. Since the prices of fuel oil, which were most inexpensive before, rose since oil crisis occurred twice in the 1970's, conversion of fuel to coal made progress in the world as shown in Figure 5.4.22. The conversion period, however, is different as two decades in U.S.A., a decade in Europe and 5 years in Japan, in reflection of the differences in the circumstances of energy and in the degree of activeness of the cement industry.

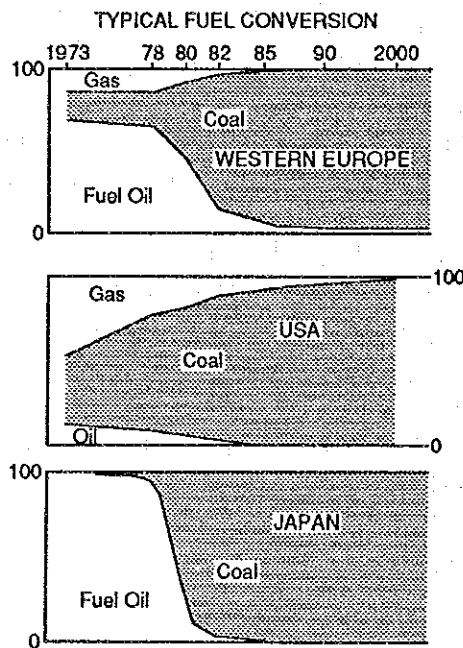


Figure 5.4.22 Situations of conversion of fuel

When the energy resources in Hungary are observed, it is not necessarily true that quality resources are available in abundance. With coal, the share of bituminous coal is less than 10%, and the majority of coal are brown coal or lignite. Upon observation of Figure 5.4.23, it is learned that brown coal is unevenly distributed in the northern part of Hungary. Bituminous coal is mined at two places only, but one is located in the vicinity of Pecs and is not far from this factory.

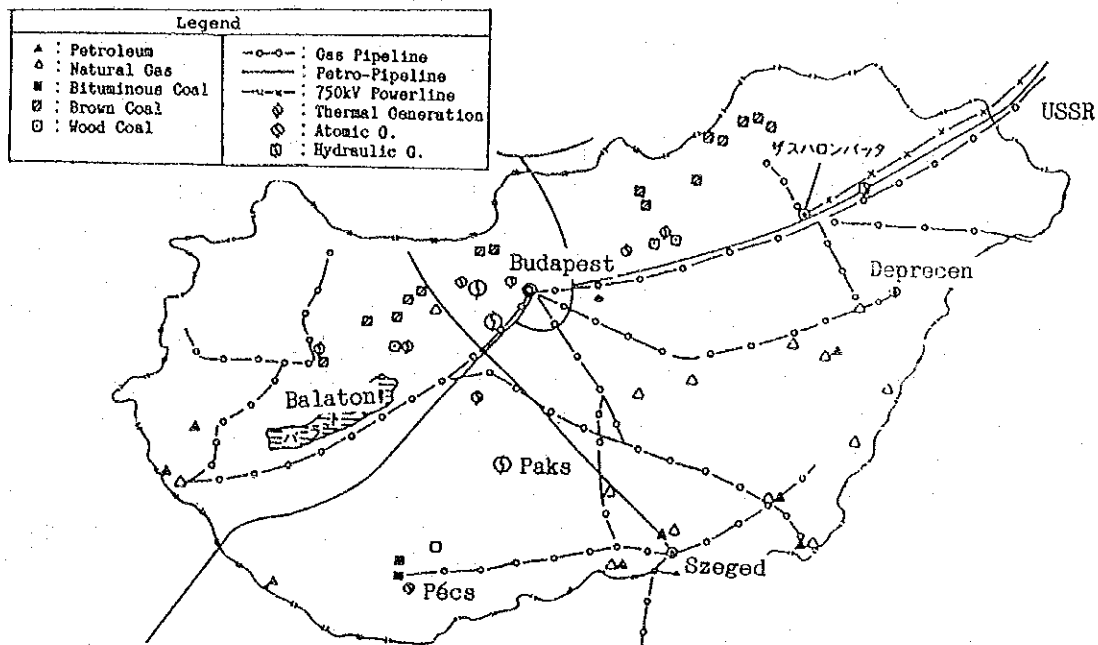


Figure 5.4.23 Distribution of energy resources in Hungary

Brown coal is most inexpensive and is broadly used without problems also in Germany. Relevant magazine called "Braunkohle" is issued, and use of brown coal is technically complete.

Upon observation of Table 5.4.11, it is learned that brown coal is used by a large quantity at thermal power plants in Hungary.

Table 5.4.11 Composition of fuel used at thermal power plants in Hungary (%)

	1970	1980	1985	1990
Bituminous Coal	9.6	6.9	5.3	7.3
Brown Coal	13.3	27.2	26.8	32.7
Wood Coal	12.8	17.1	16.5	15.5
Petroleum	19	13.9	21.2	8.9
Natural Gas	15.2	34.8	30.2	35.6

Origin: "Statistical Pocket Book of Hungary"

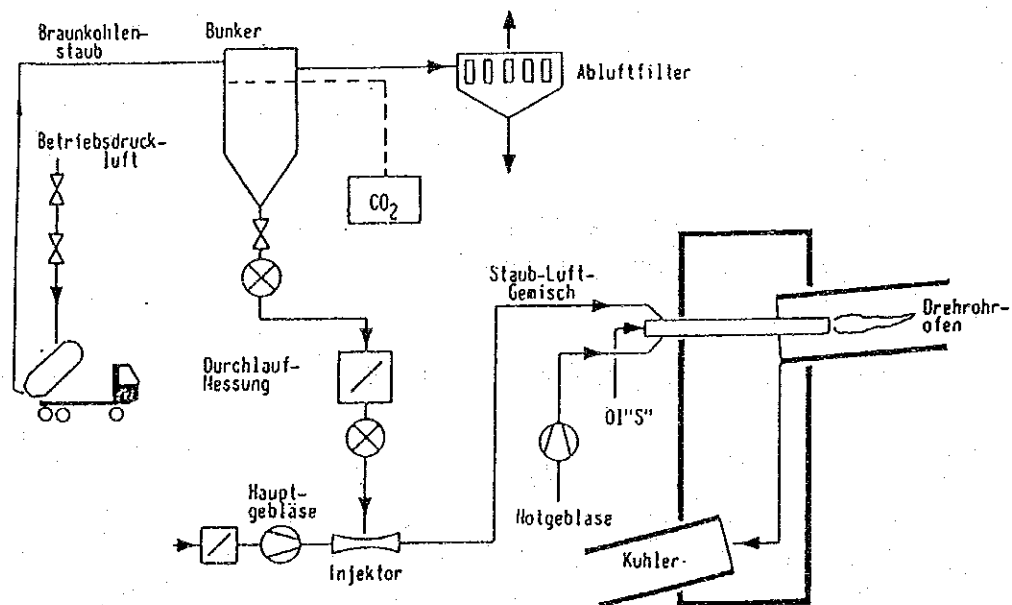
It is necessary to modify the entire fuel equipment for conversion of fuel. But it is considered that the expenses for modification are relatively small, and the cost performance is extremely large. The cement quality and production capacity will remain almost unaffected.

It is therefore proposed to immediately convert to the coal produced in the vicinity of Pècs or to plan use of the brown coal to be transported from the north by railways, with a common storage yard established with cooperation of Pècs power plant.

High temperature combustion gas of about  $1,600^{\circ}\text{C}$  is required for production of clinker, and accordingly, it is necessary that the heating value of the fuel is  $16,000 \text{ kJ/kg}$  or higher. Therefore, in the case where inferior coal is used, good heating value should be secured by mixing it with coal of high heating value.

Since solid fuel is inferior to liquid fuel in the combustibility, selection of the burners is important. In the case of conversion from fuel oil to coal or brown coal, it is recommended that Pillard burner, which is broadly used in the whole Europe, is used.

The flow chart of the fuel supply system at Wulfrather Factory is shown in Figure 5.4.24 for reference. While the brown coal, which was accepted from a railway car or a truck, is carried to the kiln burners, it is completely sealed with inert gas so as not to contact the air.



Braunkohle Heft 7, Juli 1978 S197-201

(Combustion tests with brown coal dust in the cement industry)

**Figure 5.4.24 Flowchart of fuel supply system at Wulfrather Factory**

The following two references are useful for learning fundamental and important points.

- R-1 Z-K-G Nr. 5 Mai 1981 S221-226  
Braunkohlenstaub-Herstellung und verwendung  
(Production and use of pulverized brown coal)

Abstract

Brown coal usually contains moisture by 40~60%. It is dried and is then processed to become briquette or coke or is converted into pulverized brown coal used by a large quantity at thermal power plants. Furthermore, it usually contains volatile matter by 25~40% and its low heating value is 16~24 MJ/kg. The ash content and sulfur content are small. It heats up and natural ignition occurs when it is exposed to the air.

Its use by the cement industry in Germany rapidly increased since the latter half of the 1970's, and the majority of the increased portion of the total consumption of brown coal is consumed by the cement industry.

- R-2 Z-K-G Nr. 8 Aug. 1981 S417-423  
Braunkohlenstaub-Eigenschaften und sicherheitstechnische  
Aspekt bei der Wendung  
(Properties of pulverized brown coal and technology for its safe use)

Abstract

Natural ignition, combustion and explosion are explained in detail.

(4) Finish Grinding with ball mill (Figure 5.4.25)

a. Optimum running of ball mill system

The ball mill and the air separator should be suitably run in order to perform efficiency grinding operation. Fundamental matters of the mill such as hold-up and ball diameter distribution of the mill could not be learned during the investigation of this time. Regarding the operation of the air separator, however, the CL (circulating load) or recycle ratio (F/P) is extremely small. The Tromp curve is extremely sharp because the CL of No. 1 cement mill is as small as 0.55 and 0.60 and the critical separation diameter is as large as 100 $\mu$ .

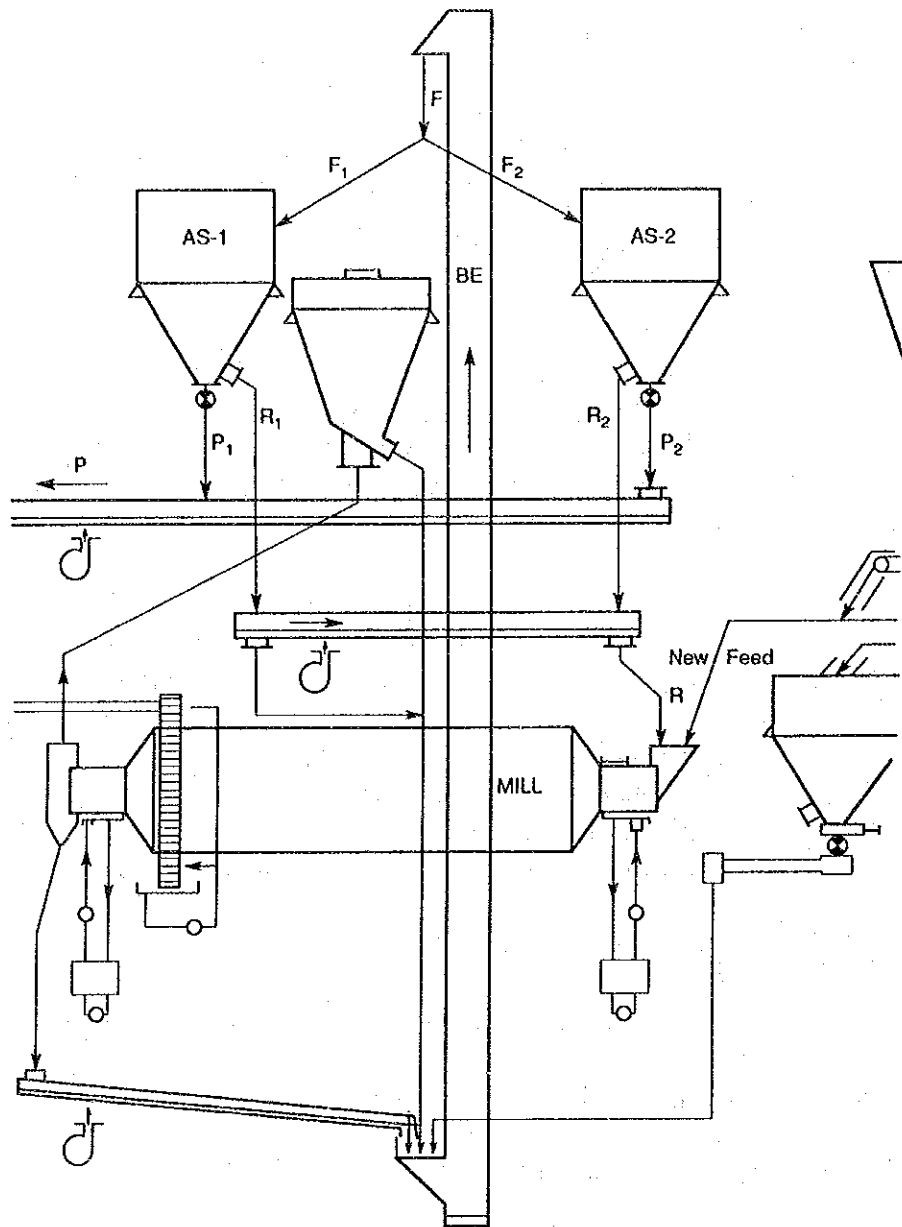


Figure 5.4.25 Cement mill

It is necessary to pay attention to the following points in the running of the ball mill system.

- ① Mill internal adjustment (ball diameter distribution, liner form, grate adjustment, drafting, material level)

The material level should be matched with the ball level. If the ball level is higher, damage to the balls becomes excessive, and if the ball level is lower, the grinding capacity drops.

② Separator adjustment (number of vanes, revolution, airflow)

Fix the number of vanes at the optimum level, and adjust the revolution by changing the V sheave at the time of changcover of the cement type.

③ Circulating load adjustment

Adjust the circulating load at the optimum level between 2 and 4.

④ Control method

Keep the kW or amperage of the bucket elevator at a fixed level and increase or decrease the separator revolution. If deviation from the control range occurs, increase or decrease the material charging quantity.

In order to produce cement of smaller grain size at this factory, it is recommended;

a) to increase the number of separation vanes (or blades) or to increase the revolution of the separation vanes with the V sheave replaced, and

b) to use grinding aids such as TEA to aim at 3,000 cm<sup>2</sup>/g-Blaine or better.

b. "O-Sepa" separator

"O-Sepa" separator (Figure 5.4.26) of the type that combines mechanical type and cyclone drafting type is a new type classifier of high efficiency. Accompanying increase of use of this classifier, interests have become concentrated to the feasibility of improvement of the ball mill circuit for reducing the running cost and for strengthening of inter-business competition of the quality. Without doubt, the method for operation of the ball mill circuit, at least the conception of the operation, has become different from the conventional method or line of thought.

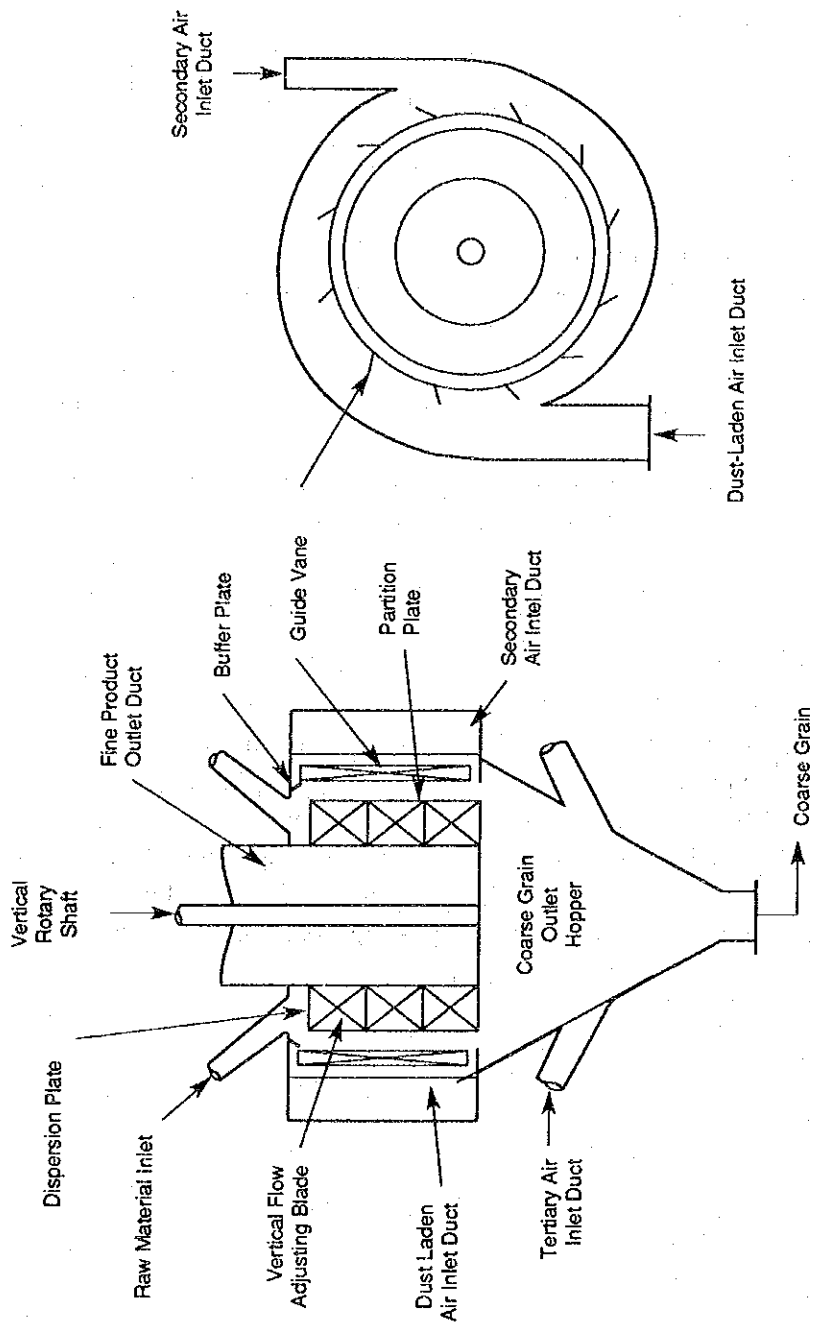


Figure 5.4.26 "O-Sepa" separator



Figure 5.4.27, Figure 5.4.29~Figure 5.4.30 indicate Tromp curves before and after installation of high performance classifier. The classifying efficiency before installation is extremely poor.

In the curve shown in Figure 5.4.27, the fine particle bypass is 25%, which is considerably higher than usual 3%. But it indicates the shape of the Tromp curve is identical regardless of whether the airflow is too low or the feed rate to the separator is excessive.

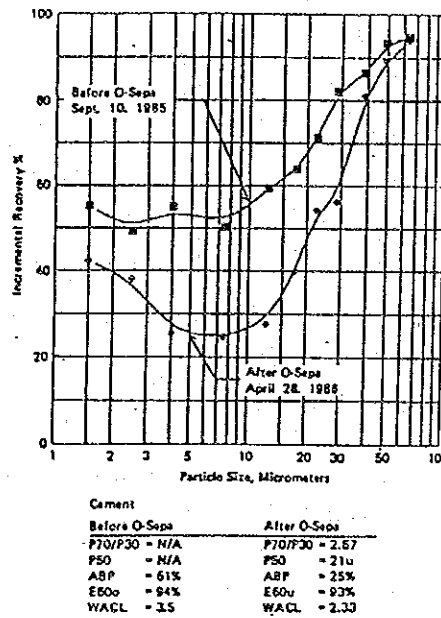


Bild 2: Tromp-Kurven vor und nach Einsetz des O-Sepa-Sichters  
Tromp curve before and after O-Sepa separator

Figure 5.4.27

The optimum feed rate to the separator should be determined upon conduct of tests of a broad range as shown in Figure 5.4.28. The test conditions should be set at a number of different level, and tests should be conducted for at least 24 hours at each one of these levels. It is because correction to problems such as reverse media classification and grit removal are time consuming.

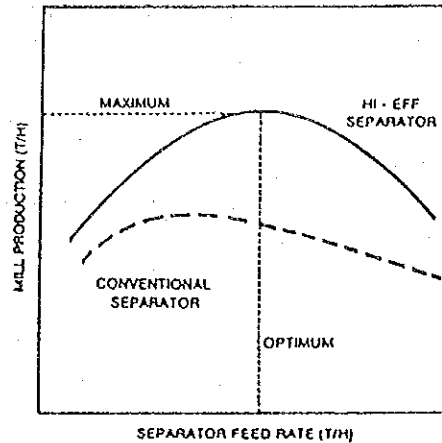


Bild 4: Zusammenhang zwischen Mühlenleistung und Sicherteraufgabemenge bei konstanter Produktfeinheit  
Capacity versus separator feed rate at constant product finess

### Figure 5.4.28 Separator feed rate and mill production

Also with a high efficiency classifier, when the separator capacity is excessive and the feed rate becomes insufficient, the coarse particle classification performance, which is important for the cement strength, may be adversely affected besides drop in the production as shown in Figure 5.4.29.

The curve shown in Figure 5.4.30 indicates good improvement because of increase of the circulation rate. The performance of a high efficiency separator with minor bypass of fine particles and coarse particles and with normal airflow is shown.

Once the separator's airflow and feedstock supply rate are optimized, it is not necessary to change the operating conditions even when the particle size changes between 2,500 and 6,000  $\text{cm}^2/\text{g}$ .