OCCASION

This publication has been made available to the public on the occasion of the 50th anniversary of the United Nations Industrial Development Organisation.

DISCLAIMER

FAIR USE POLICY

CONTACT

Please contact publications@unido.org for further information concerning UNIDO publications.

For more information about UNIDO, please visit us at www.unido.org

UNITED NATIONS INDUSTRIAL DEVELOPMENT ORGANIZATION
Vienna International Centre, P.O. Box 300, 1400 Vienna, Austria
Tel: (+43-1) 26026-0 · www.unido.org · unido@unido.org

RESTRICTED

DP/ID/SER.B/558
31 July 1986
ENGLISH

TECHNICAL ASSISTANCE TO THE CEMENT INDUSTRY

SI/DRK/86/006

DEMOCRATIC PEOPLE'S REPUBLIC OF KOREA

Terminal report

Prepared for the Government of the Democratic People's Republic of Korea
by the United Nations Industrial Development Organization,
acting as executing agency for the United Nations Development Programme

Based on the work of A. R. Marei, special adviser

Backstopping officer: C. Rydeng, Chemical Industries Branch

United Nations Industrial Development Organization
Vienna

V.86-58871
3310T

Explanatory notes

Figures for operating costs in this report are given in deutsche mark (DM). At the time when the figures were produced (1979), the value of the deutsche mark in relation to the United States dollar was $US 1 = DM 2.51.

The following technical abbreviations are used in this report:

LOI	loss on ignition
RSP	reinforced suspension preheater

Mention of firm names and commercial products does not imply the endorsement of the United Nations.

ABSTRACT

This project (SI/DRK/86/006), "Technical Assistance to the Cement Industry", has been carried out for the Government of the Democratic People's Republic of Korea by the United Nations Industrial Development Organization (UNIDO) acting as executing agency for the United Nations Development Programme (UNDP). This report covers the one-month mission of a special adviser in cement technology, who was based in Pyongyang from 28 February to 1 April 1986.

The purpose of the project was to find ways of conserving energy and of increasing the production capacity of some of the cement production lines, which currently use the wet-process system, by converting them to semi-wet, semi-dry or dry process systems. The expert was also asked to evaluate and advise on other problems facing the cement industry, on the granite extraction and processing in the country, and on training of national personnel for the cement industry.

The report covers all the project activities and contains a preliminary study of the possibility of converting some of the wet-process production lines in the 8th February Cement Joint Complex to a semi-wet process. Included is an evaluation of the existing production system, criteria for conversion, description and specification of the new proposed machinery and a calculation of the energy balance. Installation of the new machinery is intended to guarantee a decrease in energy consumption and increase in production capacity of approximately 30 per cent. A proposal for the new design based on these calculations was prepared.

The problems and difficulties facing the cement industry in the country in general were investigated, recommendations made and advice given to counterparts on the spot. Examples of these production problems are given in the report.

The On Chun Granite Company was visited. The extraction and processing of granite was studied and recommendations made.

Training of national cement designers, engineers and chemists to enable them to understand the new design, the production system and its technology was given. Two sessions of lectures on energy conservation and on clinker production, followed by technical discussions, were offered to many of the technical personnel in the cement industry.

CONTENTS

INTRODUCTION

This project, "Technical assistance to the cement industry" (SI/DRK/86/006), has been carried out for the Government of the Democratic People's Republic of Korea by the United Nations Industrial Development Organization (UNIDO), acting as executing agency for the United Nations Development Programme (UNDP), as a Special Industrial Services project. Under this project, a special adviser in cement technology was based in Pyongyang for about one month from 28 February to 1 April 1980. This report describes his mission.

General background

Whereas previously energy was seen as an abundant resource, it is now viewed as a scarce commodity and efforts are constantly being made today not only to explore new sources of energy but also to conserve traditional sources. This applies most importantly to the use of energy in industry and to the cement industry in particular. What can be done to improve the efficiency of cement plants? What are some of the considerations involved in designing and introducing energy-saving in cement plants? Energy balance, involving both fuel and electric energy, is the important concept and the goal to aim at. Cement production traditionally involves a high energy consumption, especially in plants using wet-process technology. In view of the constantly rising cost of energy since 1973, the semi-wet, semi-dry and dry processes for cement production, which require less energy, have been gaining more and more importance. The conversion of wet process into semi-wet or semi-dry process is based on the assumption that the cost of the conversion is off-set by substantial savings in fuel consumption.

It should be noted that more than 40 per cent of the world's cement industry is still using the simple wet process. During the last few years, many technical and economic studies have been carried out on converting existing cement plants from wet process to semi-wet, semi-dry or dry process. It was found that a general solution did not exist and that in many cases the wet process continued to be preferred because of some special circumstances or conditions prevailing in the area or because of the available raw materials.

A wet-process kiln can be converted, but the cost-to-savings ratio must be worthwhile. It may be more economical to abandon and rebuild, but this can only be determined by a cost analysis which includes the life expectancy of the existing installation. Future developments in energy prices may also change the picture again.

Project background

The Democratic People's Republic of Korea is in a period of intensified industrial development. The country is already producing approximately 10 million tonnes of cement per year to serve both the domestic and export markets. The country has an urgent need to increase the quantity of cement produced and to decrease its fuel consumption as well. Most of the cement plants use the wet-process method in cement production where fuel consumption is from 50-100 per cent higher than for semi-wet or semi-dry or dry process plants of similar capacity. The Government has therefore decided to ask for assistance from UNIDO to examine the possibilities of converting some of the production lines now using the wet process to production lines using semi-wet, semi-dry or dry process methods in order to conserve energy and to increase the production and if possible to reduce the production costs.

The cement industry of the Democratic People's Republic of Korea consists of the following plants.

<u>Sun Chon Cement Complex</u>, in Sun Chon City, 70 kilometres from Pyongyang.
Production of 3 million tonnes a year, three kilns (dry process)

<u>8th February Cement Joint Complex</u>, near Sari Wan City, 70 kilometres from Pyongyang.
Production of 1,450,000 tonnes a year (fully described in the report)

<u>Chon Nae Ri Cement Company</u>, in Kang Wan Province, 300 kilometres from Pyongyang and 80 kilometres from Won Son City.
Production of 600,000 tonnes a year, four kilns (dry process)

<u>Hae Jon Cement Plant</u>, South Hwang Hae Province in Hae Jon City, 150 kilometres from Pyongyang.
Production of 700,000 tonnes a year, four kilns (dry process)

<u>Sung Ho Cement Plant</u>, 40 kilometres from Pyongyang.
Production of 850,000 tonnes a year, six kilns (wet process)

<u>Other small cement plants</u> producing together about 3.5 million tonnes a year.

In March 1985, UNDP received a letter from the Ministry of Building Materials Industry which suggested some subjects to be investigated in the industry and included the nomination of investigating groups to be submitted to UNIDO. A meeting was held about this on 19 March 1985 between representatives of the Ministry of Building Materials Industry and the Resident Representative of UNDP in Pyongyang. At this meeting, the Ministry requested technical assistance from UNIDO in the field of cement and building materials. The following four project proposals, in order of priority, were discussed:

(a) Introduction of semi-wet, semi-dry or dry process systems in cement production;

(b) Techniques of cutting and processing granite;

(c) Production of board furniture and quality furniture goods;

(d) Improvement of kraft paper bags.

The reasons for deciding to proceed with the first proposal through UNIDO is that the Government considers energy-saving, especially in the cement industry, as one of the most important subjects for the economy now that the Democratic People's Republic of Korea has entered a stage of intensive industrial development. Letters and telexes were circulated between UNIDO and UNDP, and the Government agreed to the UNIDO proposal of a one-month consultancy service under the Special Industrial Services programme.

The main objectives of this one-month mission were agreed at a meeting at UNIDO headquarters on 1 March 1986, attended by representatives of the Government of the Democratic People's Republic of Korea and by officers of UNIDO and UNDP. They were as follows:

(a) To investigate the possibility of introducing the semi-wet, semi-dry or dry techniques into those cement plants in the country which are still using a wet process for cement production, especially in the case of the

8th February Cement Joint Complex. The general purpose of introducing any of these techniques is:

(i) To increase the production capacities of the kilns by 20-30 per cent;

(ii) To decrease the fuel consumption in the converted production lines to below 1,100 kcal/kg of clinker;

(b) To recommend technical solutions to the problems faced by the Chon Nae Ri Cement Plant, especially the fact that, in its production line which uses the dry process method, only 70 per cent of the rated capacity of the reinforced suspension preheater (RSP) kiln can be achieved;

(c) To investigate the main technical production problems facing both the above-mentioned cement plants and give advice and recommendations;

(d) To evaluate the quarrying and processing techniques used in granite production and advise on the technical characteristics of the equipment and machinery used;

(e) To visit Nam Po Port and investigate the cement-dispatching equipment installed under a UNDP project and to advise on it in accordance with a request from the UNDP Resident Representative at a meeting held in the UNDP office on 29 February 1986.

(See also the expert's job description in annex I.)

Project activities

On 1 March 1986, the Adviser accompanied by Kim Jae Ok (Director of the Bureau for Scientific and Technical Leadership) and Zo Ho Sung (Interpreter) left Pyongyang to visit the 8th February Cement Joint Complex, 70 kilometres south of Pyongyang city and 8 kilometres south of Sari Wan city. The visit lasted for five days until 5 March and the Adviser met the following personnel:

Kang Yong Gol (General Manager of the Complex)
Min Ho Yong (Chief Engineer of the Complex)
Mun Jae Gun (Member of the Research Institute in the Cement Complex)
Han Hi Chan (Chief of Technical Section in the Cement Complex)

The last three officials accompanied the Adviser throughout the visit to the 8th February Cement Joint Complex. The last two officials accompanied the Adviser throughout the mission period until he left the country on 1 April 1986.

The Adviser studied the processes of cement production and the machinery and equipment used. He discussed on site all the technical problems facing the cement complex and made various recommendations on avoiding or overcoming such technical problems. All aspects of the proposed conversion were discussed and several meetings were held between the Adviser and the personnel of the 8th February Cement Joint Complex. In these meetings, the Adviser explained in detail the different types of conversion that can be adopted in converting a cement plant from wet process to semi-wet or semi-dry or dry process methods. He answered all the technical questions raised at these meetings. In the end, all those taking part in these meetings agreed on conversion of the wet-process production lines of the 8th February Cement Joint Complex to a semi-wet process by the addition of a filtration unit, dryer and two-stage pre-heaters. They decided also to convert the kiln grate cooler. These decisions were based on the technical and economic factors and criteria indicated later.

The Adviser calculated the original capacities of the machinery used at present in the wet-process production lines and also the capacities of all the new equipment necessary to carry out a process conversion, as shown in part one of this report. Detailed production data supplied by the Cement Joint Complex are given in annex II of this report.

From 6 March 1986, the Adviser continued his work in Pyongyang, giving the government cement designers the technical information on production-line conversion, and how this conversion can be carried out, and explaining the need for a feasibility study which will include the following information as a prerequisite for designing the conversion:

(a) The capacity of the existing rotary kiln;

(b) Heat requirement for drying without waste-heat utilization;

(c) The heat balance of the existing rotary kiln;

(d) Thermal calculations and kiln heat balance for the kilns after conversion;

(e) Heat balance of new machinery such as the dryer, two-stage pre-heaters and meal cyclones;

(f) Heat balance of cooler.

These calculations were done by the Adviser and are given in chapter three. Full descriptions and specifications of the proposed new equipment were also compiled, as well as an account of the changes and modifications to existing machinery and equipment, which would be necessary to suit the new process. (See chapter V.) This description will help the Government to prepare the tender documents when they have decided to proceed with the project.

After many meetings, discussions and corrections, a design for a new production layout, drawn up by the Adviser, came to its final stage as shown in figures 5(a) and (b).

On 16 March, the Adviser, accompanied by Kim Jae Ok (Director of the Bureau for Scientific and Technical Leadership), Li Song U (UNDP) and Zo Ho Sung (Interpreter), left Pyongyang for Won San City in Kang Won Province, about 240 kilometres from Pyongyang. There they visited the Chon Nae Ri cement plant and met Kim Ho Il (Vice Chief Engineer), Ue Yong Mok (Chief of the Technical Section) and Kim Kyo Un (Chairman of the Research Institute). A meeting was held with these persons to discuss the reasons for the failure of the dry-process kiln to achieve more than 70 per cent of its rated capacity. (The kiln is equipped with four suspension pre-heaters and a precalciner.) Many aspects were thoroughly discussed and analysed. The following emerged as some of the problem areas:

(a) Heat balance of the kiln, the suspension preheaters and especially the precalciner and the high specific heat consumption;

(b) The cooler problems and how to avoid clogging of the pipe conveying cooled hot-air gases to the precalciner;

(c) The alkali problems and their effects on the quality of cement;

(d) By-passing a part of the exit gases to overcome the alkali problems;

(e) Adjusting the flame properly in the burning zone;

(f) The low degree of precalcination;

(g) Dust problems;

(h) Fineness of the raw meal;

(i) The free CaO content of the clinker produced;

(j) Segregation of clinker along the cooler grates;

(k) Quality of clinker produced.

Technical explanations, advice and recommendations were given. The visit lasted until 19 March 1986.

A meeting was held with Yong Tu Long (UNDP Deputy Resident Representative) on 24 March. This was also attended by the Adviser and the following:

Kim Jae Ok (Director of Bureau for Scientific and Technical Leadership)

Han Hi Chan (Chief of Technical Section, 8th February Cement Joint Complex)

Mun Jae Gun (Member of the Research Institute, 8th February Cement Joint Complex)

Chai Nam Ri

Kim Chung En

Zo Ho Sung (Interpreter)

Li Song U (Programme Officer, UNDP)

At this meeting, the Director of the Bureau expressed the Government's satisfaction with the work carried out on this mission, which had included training of national engineers (from the Government as well as from the different cement plants) and lectures presented to them on energy conservation and clinker production and on alkali problems and how to avoid them. The Director submitted to the UNDP Deputy Resident Representative a letter outlining their proposal for carrying out the conversion. This proposal indicated the stages of the conversion, the changes to be made and the investment which would be needed. A detailed list of the new equipment required was attached and UNDP was asked to assist in providing the country with the imported equipment. The Deputy Resident Representative informed the meeting that UNDP can only help in providing technical assistance such as study tours to developed countries to familiarize personnel with the new semi-wet system, training of the nationals and providing some special instruments to the laboratories to develop the quality control of the cement produced. The Deputy Resident Representative asked that another tripartite meeting should be arranged to discuss the main technical assistance fields where UNDP and UNIDO can help and that this meeting should take place before the Adviser's departure.

This meeting was held on 31 March 1986 with all the same personnel who attended the tripartite meeting on 24 March. At the meeting, the Director of the Bureau submitted a letter indicating the Government's revised request to UNDP for assistance in carrying out the conversion. The request was as follows:

(a) That a study tour should be prepared for four Korean staff to visit one or two developed countries (e.g. Switzerland (Liesberg Cement Plant) or France) for one month to get acquainted with the semi-wet process;

(b) That six Korean staff should be trained for one month in the Cement Industry Design Institute in Tienjing, China, to become acquainted with new developments of the suspension preheater calciners and the dry process used in China;

(c) That the 8th February Cement Joint Complex should be provided with some laboratory instruments and equipment to develop the quality control in the plant;

(d) That, after the conversion had been carried out, a UNIDO expert should be sent to inspect and evaluate the work carried out.

The Deputy Resident Representative agreed to these requests and indicated that he would contact the Foreign Trade Ministry and UNIDO. Mr. Kim informed those present that his Ministry had already contacted the Foreign Trade Ministry and that the Government was giving more priority to the cement projects than any other UNDP projects and that they would support the above-mentioned requests.

Meetings were also held with Mr. Kim to investigate and study problems faced by some other cement plants within the country. Advice and recommendations on solving these problems were submitted at these meetings.

During the course of two sessions held on 9 and 16 March 1986, attended by the main designers and engineers from the Bureau (Government) and the different cement plants, seven hours of lectures were presented on energy saving in the cement industry and how conversion of wet process into semi-wet, semi-dry or dry process can be carried out, and on clinker production and the alkali problems and how to avoid such problems in cement plants. After the lectures, many technical questions were asked and problems discussed.

On 26 March 1986, the On Chun Granite Company was visited by the Adviser, accompanied by Jo Yong Huei (Director of Korea Cement and Granite Export and Import Corporation) and Zo Ho Sung (Interpreter). The technology used in extracting and processing granite was studied and evaluated and the necessary advice and recommendations concerning the new technology were presented.

The expert's job description also included the elaboration of terms of reference for a feasibility study for possible use in follow-up activities, with or without UNIDO assistance (see annex I). It was understood that no final decision could be taken on process conversion without a really thorough feasibility study which would include all the results of the raw-material tests and an evaluation of all the economic consequences of various alternatives. A draft of the terms of reference was prepared in June 1986 and excerpts from this draft are given in annex III.

FINDINGS AND RECOMMENDATIONS

Findings

Part one: The conversion of 8th February Cement Joint Complex

1. At the technical meetings between the Adviser and the counterpart technical personnel, it was agreed that conversion of the production lines should start with kilns 5 and 6 followed, at a later stage, by kilns 3 and 4.

2. It was decided to opt for conversion to the simple semi-wet process, i.e. the long wet kiln plus a filtration unit plus a two-stage preheater and raw meal cyclone. This decision was based on the desirability of continuing to use the existing production departments, stores, raw mills, slurry basins etc.

3. The production capacity is expected to increase by a minimum of 30 per cent.

4. The clinker produced will contain less alkalis and chlorides, i.e. it may be possible to produce low-alkali cement.

5. It is expected also that the man-hour input per tonne of clinker will only increase by approximately 0.04 per cent for filtration and drying, plus approximately 0.01 per cent for the kiln system.

6. Assessment of the production-capacity efficiency of the machinery used in this plant in the production lines 3, 4, 5 and 6 before and after conversion, showed that the following machinery has to be installed to achieve the desired increase in the production capacity of the converted plant:

A new hammer crusher of a capacity of 150 tonnes per hour (t/h)

A new raw mill of 75 t/h capacity

Three corrective slurry silos of 400 m^3 capacity each

Two slurry pumps of 80 m^3/h capacity each

A closed-circuit cement mill of 50 t/h capacity

Three cement silos of 1,000 t capacity each

A rotary packing machine of 100 t/h capacity
or 2 stationary packing machines of 60 t/h capacity each

A compressor of 40 m^3 per minute capacity

7. Calculations of the capacities of the existing kilns 3, 4, 5 and 6 showed that these kilns are only capable of producing 32 tonnes per hour or, 744 tonnes per day each, instead of the designed capacity of 850 tonnes per day each, due to the incorrect specific kiln capacity which was given as 0.557. In fact, the specific kiln capacity was found to be 0.49.

8. The rotary-kiln heat-balance calculations showed that the energy consumption of the new system, after conversion to the semi-wet process, may reach a maximum in the range 1,050-1,100 kcal/kg clinker. As the energy consumption of the existing rotary kiln using the wet process is in the range 1,620-1,670 kcal/kg clinker (average 1,645 kcal/kg clinker), the energy savings may be approximately 500 kcal/kg clinker, or roughly 30 per cent. The fuel needed may therefore decrease from 0.32 tonnes of coal per tonne of clinker, to 0.21 tonnes of coal per tonne of clinker.

9. The heat-balance calculations of the dryer and the first and second preheaters showed that the gas temperatures leaving them are 466 °C, 718.6 °C and 931.2 °C respectively. The heat content of the clinker discharged from the kiln will be 213.84 kcal/kg clinker and the thermal efficiency of the cooler is 77 per cent.

10. Dewatering of the slurry can be carried out either by using vacuum filters or pressure filters. As the filtration process depends on many factors, such as the chemical, physical and mineralogical constituents of the slurry as well as the shape and grain-size distribution of grains in the slurry, the slurry, filter dusts, fuel (coal) and water used in the 8th February Cement Joint Complex have to be carefully studied and tested before selecting the filtration unit.

11. Due to lack of a suitable area along the axis of kilns 5 and 6 on the electrostatic filter side where the filter unit with its accessories to serve kilns 5 and 6 could be installed, it was agreed that the area situated along the north-eastern side of the water channel (parallel to the kilns' axis) and nearest to the electrostatic filter of kiln 6 would suit the requirements. On the other hand, a suitable area does exist along the axis of kilns 3 and 4 where the filtration unit and its accessories could be installed later on for these kilns.

12. The grate coolers of the existing production lines need to be modified or replaced in order to fit in with the rest of the process conversion.

13. According to the information received from the technical personnel of the 8th February Cement Joint Complex about the availability of raw materials and their reserves, it is expected that the raw material reserves can supply the cement plant for at least 50 years, even with the expected increase in production. The reserves were estimated on the basis of a reliable geological investigation carried out by the company.

14. The results of the technical feasibility study were evaluated in the light of their economic implications, and it was found that the conversion of the production lines from wet process to semi-wet process would succeed in conserving energy and in increasing the production capacity by 30 per cent at least. Accordingly, at two tripartite meetings, the Government of the Democratic People's Republic of Korea asked UNDP for assistance in carrying out the conversion.

Part two: Difficulties and problems facing the cement industry

15. The consumption of the grinding media in the 8th February Cement Joint Complex as well as of the internal fittings and lining plates of the mills (raw and cement) is abnormally high. This can be attributed to the fact that the grinding media are made of cast iron and their shape is irregular rather than round or cylindrical. Besides this, no special care or attention is given to the dimensions of the grinding media neither in the raw mills nor in the cement mills.

16. The consumption of refractory bricks per kilogram of clinker in this plant is rather high (four to six kilograms per tonne of clinker). This can be attributed to the low quality of the refractory bricks used in lining the kilns and especially in the burning zone. It was also found that the nature, shape and position of the kiln flame are in need of adjustment. The abnormal thermal shocks due to the continuous stoppages are accelerating the deterioration of the refractory bricks.

17. The process of clinker formation cannot be considered to be complete when the clinker leaves the hottest area of the kiln's burning zone. The cooling of clinker is also considered to be a part of the process of clinker production since it affects the quality of the clinker produced and its grindability. It is clear that in the 8th February Cement Joint Complex, the temperature of the clinker leaving the grate cooler is very high (300-400 °C) and therefore the grindability and the quality of the clinker were affected.

18. The clinker produced in 8th February Cement Joint Complex is characterized by its very low silica modulus. This leads to the disturbance of the burning zone as well as the increase in the liquid phase percentage which was found to be more than 30 per cent. The percentage of calcium fluoride added to the raw meal was also found to be high and its addition creates many problems in the burning zone and has an effect on the quality of the clinker and cement produced.

19. The sulphate content of the cement produced in the 8th February Cement Joint Complex ranges from 1.5 to 1.68 per cent, which is low. Gypsum ($CaSO_4 - 2H_2O$) is added in a limited quantity to Portland cement not only to regulate its setting time but also to influence the other properties such as grindability, sensitivity to storage, volume stability and strength. The percentage of gypsum is limited in this case due to the fact that gypsum is imported from China and the Union of Soviet Socialist Republics.

20. All the machinery, equipment and instruments used in the laboratories of the 8th February Cement Joint Complex are of an old type and make possible only the ordinary quality-control tests. The sand used in testing the compressive strength is very fine and does not comply with the standard for such sands. Consequently, the results obtained are below the limits.

21. At Chon Nae Ri cement plant, the reinforced-suspension-preheater (RSP) kiln, designed and manufactured in the Democratic People's Republic of Korea, is incapable of reaching its rated capacity of 60 t/h or 1,440 t/d. Instead, it produces only 70 per cent, i.e. 42 t/h or 1,008 t/d. The Adviser attributed this to many factors such as the false air entering the kiln. The coal supplied to the kiln represents 52.38 per cent of the fuel while that supplied to the precalciner is 47.61 per cent which is considered to be low. The temperature of the air recouped from the cooler and blown into the precalciner as well as the secondary-air temperature are low, and the length of the kiln is so short that the calcination of carbonate and the formation of clinker phases do not have enough time to take place properly in the 60-metre length of this kiln.

22. The alkali content of the clinker produced in Chon Nae Ri cement plant (and in some other cement plants in the country) is 1.2 per cent which is a high percentage. This percentage causes many problems in the quality of clinker produced as well as in the parts of the process where condensation of alkali vapours occurs, e.g. in the kiln-inlet region, the bottom stage of the raw-meal preheater and the calcining chamber of the grate preheater. The condensation phenomenon causes local accumulation of alkali salts that melt to produce liquid phases which make the particles of clinker sticky and these adhere together to form balls of clinker and coating.

23. The tertiary duct in use in the Chon Nae Ri reinforced-suspension-preheater kiln, which conveys the cooler's air to the precalciner, is always having problems with clogging. This was attributed to the absence of a cyclone as well as to the precipitation of dust along the walls of the horizontal duct.

24. Mixing of the slurry, especially in the slurry basins of the 8th February Cement Joint Complex, was not carried out properly although the slurry basins are equipped with a mechanical as well as a pneumatic mixing system. This tends to create some trouble and disturbance in the burning zone.

Part three: Extraction and processing of granite in the On Chun Granite Company

25. The machinery and equipment used in the On Chun Granite quarry are all simple and are mainly locally produced. The processing of granite here uses the technology normally used in most developed countries. The efficiency of cutting the blocks could be said to be below standard.

26. The extraction of granite blocks in the On Chun granite quarry is carried out using explosives. These explosives destroy much of the production due to the effect of the pressure of the gases from the explosions on the surrounding granite rocks.

Recommendations

Part one: Conversion of the 8th February Cement Joint Complex

1. It is recommended to carry out the conversion of the production lines of the 8th February Cement Joint Complex step by step. The first conversion would be of the newest kilns, 5 and 6. The second stage would include kilns 3 and 4. The process chosen would be the simple semi-wet system (long wet kiln plus filtraton unit plus dryer plus two-stage preheater and cyclone). This system will enable the company to utilize the old equipment (stores, slurry mills, corrective silos, basins etc.) and should make it possible to conserve energy and increase the production capacity of the existing equipment.

2. It is absolutely necessary to carry out the tests and investigations indicated in the report on representative samples of the raw materials (raw-meal slurry, fuel, water, electrostatic-precipitator-filter dust etc.) taken from the 8th February Cement Joint Complex production lines. This information is needed in order to select the proper system for dewatering the slurry, either vacuum filters or pressure filters, before any final decision on conversion is made. These tests and investigations can be carried out in a specialized laboratory and should be confirmed by pilot-plant tests which can be evaluated by the supplier so that he can submit suitable proposals for modification of the equipment.

3. As the Democratic People's Republic of Korea is a cement-equipment producing country and capable of producing some parts of the main equipment and machinery needed, it is recommended that the Democratic People's Republic of Korea should indicate, in the international tender documents, its willingness to manufacture some specific parts of the equipment according to the designs and drawings which will be received from the supplier after agreement has been reached and the contract signed.

4. It is recommended that the new machinery and equipment which the evaluation shows to be necessary in order to achieve the desired increase in production, should be manufactured and erected locally according to the specifications of the cement equipment required.

5. The tender documents, including the present estimates, calculations, specifications and the design proposal have to be re-checked and re-evaluated by the supplier in the light of the pilot-plant test results. The supplier

should be requested to submit his final conversion proposal to the Government and to confirm his own willingness to carry out the conversion and to guarantee the increase in production figures and savings in energy of at least 30 per cent.

6. It is recommended that the coolers should be completely modified to fit in with the conversion. The tender documents have to include everything necessary for the proposed modifications, as described in the Adviser's proposal.

7. It is recommended that UNIDO enters into a technical-assistance agreement with the Government (through UNDP) to monitor and control on behalf of the Government all the steps, from the beginning of the project until the taking-over of the new machinery, which are necessary to guarantee the success of the conversion. (This idea was discussed with the authorized personnel and it was appreciated and welcomed.)

8. In accordance with the Government's request at the tripartite meeting on 31 March 1986, a study tour for one month for four Koreans to visit one or two developed countries such as Switzerland or France should be arranged through UNDP so that the Koreans can become acquainted with the semi-wet process in situ.

Part two: Difficulties and problems facing the cement industry

9. It is recommended to use highly wear-resistant grinding media made of white cast iron with a chromium content ranging from 12-18 per cent and with tungsten or molybdinum. These types of grinding media are much more economical for the cement plant than the normal grinding media as the consumption will not exceed 50 grams per tonne of cement.

10. At the same time as changing the grinding media, it is also recommended to change the mill lining plates. The lining plates and the internal fittings should be geared to suit the new grinding media. A cost/benefit analysis should be made of the effect of implementing this recommendation.

11. In resolving problems in the kiln areas, it is necessary:

(a) To select the proper refractory bricks for any of the kiln zones and especially for the burning zone, where it is recommended to install magnesite or magnesite-chrome bricks of good quality;

(b) To give special attention to the lining, especially in the deformed areas of the kiln shell and to check and correct the alignment of the kiln with appropriate intervals;

(c) To adjust the position and direction of the flame inside the burning zone of the kiln according to the explanations included in this report;

(d) To increase the amount of oxygen introduced to the burning zone and to reduce the number of kiln stoppages through changing the deformed parts of the kiln shell.

These are all recommended as ways to decrease the consumption of refractory bricks and to conserve energy.

12. It is recommended that both the silica and alumina moduli should be increased to reach the recommended ranges of 2.3-2.7 for the silica modulus and 1.3-1.6 for the alumina modulus. This can be achieved by adjusting the raw mix, i.e. by increasing the amount of the siliceous stone and decreasing

the amount of iron ore and calcium fluoride added to the raw meal. The percentage of calcium fluoride added to the raw meal has anyhow to be rechecked. This adjustment will lead to a decrease in the percentage of liquid phase in the clinker and will help to avoid problems in the burning zone and in the quality of clinker produced.

13. It is recommended that the amount of gypsum added to the clinker in the cement mills should be increased to approximately 2.5 per cent of the sulphate content of the cement produced. The amount of gypsum needed to achieve this percentage in all the cement produced in the Democratic People's Republic of Korea should be imported from the Union of Soviet Socialist Republics or China regardless of the cost of the foreign currency to be paid. This will ultimately operate in favour of the country as the cement produced will comply with the International Standard Specification, which is important in the case of cement exported to other countries.

14. It is recommended that the new technology should be applied in testing and investigating the raw materials and raw mixes, and in controlling the quality of the clinker and cement produced. This new testing technology would require the introduction of some modern laboratory equipment and instruments (as indicated in this report). This would help develop the quality of the cement produced and would make it possible to get tests results in a very short time compared to the classical methods now in use.

15. In order to reach the rated capacity of the Chon Nae Ri kiln, it is recommended that:

(a) Air should be prevented from entering the kiln incorrectly (false air), by regulating the temperature of both the secondary air and the air recovered from the cooler and blown into the precalciner, and by regulating the precalciner temperature so as to achieve a 90-95 per cent precalcination of the carbonate contained in the raw meal;

(b) The ratio of fuel supplied to the kiln and the precalciner should be changed on an experimental basis, increasing the percentage of fuel supplied to the precalciner. (The proportion of fuel supplied to the swirl burner can be in the range 55-70 per cent.);

(c) Evaluation and recalculation of the rate frequency of the induced draft fan (IDF) is recommended, and an increase in its capacity if necessary;

(d) The shape and position of the flame inside the kiln has to be adjusted as indicated in this report;

(e) If the above-mentioned advice and recommendations do not lead to any success in reaching the kiln's rated capacity, then the length of the kiln has to be increased to at least 70 metres.

16. In order to avoid the direct effects of alkalis on the quality of the clinker as well as on the kiln burning zone, it is recommended to install one of the types of by-passes described in this report, or to reduce or to cut down the cycle processes, or to completely get rid of the dust produced. Coating and rings can be eliminated by following the advice included in this report.

17. It is recommended to pay attention to repairing and maintaining the machinery, which mixes the slurry inside the slurry basins. It is necessary to increase the compressed air for mixing the slurry inside the correcting silos as well as inside the slurry basins. This will overcome some problems of quality created by not mixing the slurry properly.

Part three: Extraction and processing of granite at the On Chun Granite Company

18. Modern technology and equipment for extracting granite should be introduced as indicated and described in this report (i.e. comb-drilling and using wire saws). The use of explosives in the granite quarry should be avoided.

19. The company's engineers should visit countries which are using this modern technology in cutting and processing granite (e.g. Federal Republic of Germany, Greece or Italy) to become familiar with the process.

Part four: Training

20. It is recommended that some Korean engineers, chemists and technicians should be trained in other countries. Especially suitable would be cement institutes and research centres in China, India or Turkey. This would help the country to accelerate the process of cement industrialization and to achieve the greatest degree of technological self-sufficiency which is possible within present limitations.

21. Training should aim at making possible full utilization of the existing cement production facilities, as well as at familiarizing the trainees with the new technology used elsewhere in the cement industry. It is, therefore, recommended that UNDP should assist the country in training national personnel in the cement and granite industries. The training of cement personnel in China, as suggested at a tripartite meeting with UNDP, is a recommendation fully supported by the Adviser.

Part One

CONVERSION OF THE 8TH FEBRUARY CEMENT JOINT COMPLEX

(A pre-feasibility study on energy conservation through conversion from wet-process to semi-wet or dry process)

Introduction

During the visit paid to the 8th February Cement Joint Complex from 1 to 5 March 1986, the Adviser concentrated on the following:

(a) The wet-process system used in the plant;

(b) The machinery, equipment and instruments used in each production department, their design, capacity (rated and actual), and production achieved in the last few years;

(c) The performance of the equipment and especially any problems arising with it;

(d) The systematic study of the production cycle through all the departments from reception of the raw materials to the dispatching of the cement produced;

(e) The plant's layout to assess handling of the production between the different departments so as to see how to design and carry out the conversion without causing any bottlenecks in the future;

(f) Study of the raw-mix designs and the characteristics of the raw materials, the clinker and the cement produced as a preliminary step in taking a decision on the conversion;

(g) Evaluation of the quality control carried out in the plant's laboratories;

(h) The equipment-operating systems in all the production departments.

I. REASONS FOR CHOOSING A SEMI-WET PROCESS

The results of the evaluation of the existing process and the criteria for choosing a new process were explained and discussed at many meetings and may be presented under two main headings: energy-saving factors and economic considerations.

A. Energy-saving factors

The filtration process

A semi-wet process depends mainly on filtration equipment. There are two filtration methods, which can be used, vacuum filtration or pressure filtration. The first method can only be used with easily filterable slurries and produces filter cakes of a higher water content (from 3 to 5%) than those produced by pressure filters. The increase in water content corresponds to an average increase in heat consumption of 75 kilocalories per kilogram of clinker. (18 kcal/kg clinker is required to evaporate each additional 1% of water.)

Dewatering of slurry was first done in the cement industry using centrifugal force. This process was not widely accepted due to the high power consumption involved as well as to the problems and difficulties that occurred, especially in the heterogeneity of the slurry constituents, which was reflected in the quality of clinker and cement produced. In the United States of America, about 25 per cent of the cement plants with wet-process technology use filtration to dewater the slurry.

The filtration process can be defined as separating out the solid constituents from the liquid in the slurry by passing the slurry through cloth partitions on which suspended particles of solid matter will be deposited while the water passes through. The liquid is known as the filtrate while the solid materials deposited on the surface area of the textile discs are known as filter cakes.

Filter presses are usually available with a plate size of 2,300 mm diameter or 2,000 mm by 2,000 mm. The number of plates and their diameter varies according to the type and capacity of the filter press. The number ranges from 70 to 150 plates. For example, a filter press of 150 plates with a cake thickness of about 50 mm, will have a total filter area of 850 m^2 corresponding to a filter-cake volume of 20 m^3. A filter press of this type will produce a compact filter cake with minimum residual moisture content as well as clear filtrate.

Experience of such filter presses gained in recent years shows that this type is reliable and little affected by wear due to its simple form of construction. A filter press of 100 chambers, processing a moderately filterable slurry, can supply a kiln with a capacity of 800 tonnes per day of clinker.

The performance of a filtration process depends on many factors, which affect the behaviour of slurry during filtration. These can be summarized as follows:

Chemical composition of slurry
Physical and mineralogical constituents of slurry
Shape and size distribution of grains within the slurry
Pressure applied
Filtration media (type of textile used)
Compressibility of filter cakes
Viscosity of slurry
Temperature of slurry

Korseny-Carman and Darcy applied the following formulae to correlate all these factors and the effect of each factor on the others:

(a) Korseny-Carman formula

$$Q = K_1 \frac{\Delta P}{\eta S^2} \frac{E^3}{(1-E)^2} \frac{A}{e}$$

(b) Darcy formula

$$Q = \frac{\Delta P}{e}$$

where:
- Q = Volume of filtrate
- A = Filter area
- S = Specific surface area of solid material
- K = Constant
- ΔP = Pressure of filterpress
- e = Thickness of filter cake
- η = Viscosity of suspension
- E = Porosity

Applying these formulae, it can be seen that, if the filtration process has to be done in a short time, then the working pressure applied (ΔP) from the press has to be increased (up to 25 bar under the industrial conditions prevailing) and that the filter area has to be made as large as possible. The viscosity (η) has to be low at the normal temperature. It is also preferable to reduce the specific surface area of the suspension as far as possible since this is possibly the most important factor.

We shall deal in detail with the factors affecting the filtration process as follows:

Pressure. Pressure is one of the main factors in the filtration process. The filtration cycle time is reduced to the minimum when the pressure applied reaches its maximum. The residual moisture in the filter cake will depend greatly on the pressure applied and on specific quantity of the suspension in the slurry. If the chambers are well filled with solid materials in suspension, the cake will, under high filtration pressure, be compressed to almost twice its original density and become relatively dry. It is clear, therefore, that the pressure in the filtration process affects the percentage of moisture content in both the filtrate and the filter cakes, as well as the density and the shear stresses acting against the cakes. The pressure applied ranges from 15 to 25 bar. Two systems are available for filling the filter press and for building up the necessary pressure. In both systems, rapid filling is achieved by using centrifugal pumps.

Above the top pressure of the centrifugal pump (about 5 bar), the final filtration pressure (about 15 bar) is achieved by means of piston-diaphragm pumps or, exceptionally, by the use of air-ejector vessels. This last method is one of the more expensive in filtration processing as the compressed air costs a lot. Piston-diaphragm pumps are reliable in operation and subject to relatively low wear as the conveying medium will not come into contact with the actual pump casing.

Filter-cake thickness. The thickness of filter cakes is normally 50-60 mm but this will depend on the operational conditions surrounding the filtration process.

Moisture content of slurry. The cycle time required to complete the filtration depends on the original moisture content of the slurry. The normal average moisture content of slurry is between 30 and 40 per cent. An increase in this percentage will not greatly affect the duration of the filtration cycle as long as most of the excess water in the slurry is pressed out in the first few minutes of the filtration process. However, if the slurry contains more than 40 per cent water, the filtration process can have problems, especially where the suspension contains a high percentage of clayey minerals capable of absorbing more water, such as montmorillonite.

Residual water in filter cakes. The percentage of residual water in the filter cakes is around 17-19% under normal conditions and with the pressure normally used in the filtration process. If the filter cakes contain more water, this can usually be attributed to the presence of some specific clayey minerals, as stated before.

The following are examples of the percentage of residual water likely to be found in the filter cakes of some specific raw mixes after filtration:

Raw mix	Percentage of residual water
Chalk and fine clay	18-22
Limestone and marl	16-18
Limestone, slag, sand and pyrite ash	15-16

Fineness of slurry's suspension. As a rule, the greater the fineness achieved in the grinding process, the longer the filtration cycle time will be. The residual water content in the filter cakes will also increase, according to the Carman equation. For this reason, it is advisable to test the raw-mix components since characteristics differ from one raw material to the other and some clayey materials such as montmorillonite contain very fine particles.

Filtration media. There are two types of textile used in the filtration process in the cement industry as filter media. The first is a many-layered fabric and the other is a one-ply fabric. From past experience in filtration techniques, it has been established that the cycle time required for filtration is reduced by about two minutes per cycle when one-ply fabric is used. Depending on the quality of the fabric employed, the service life of the filter cloth is on average four to five months, corresponding to 3,000-6,000 pressings. This in turn corresponds to a clinker output of 350-500 tonnes per filter cloth, depending on the cake thickness.

The energy consumption of a pressure filter plant is 2-3 kWh/t of clinker or 5-6 kWh/t clinker when further processing of the filter cake is included.

The number of operatives per filter press diminishes with increasing number of presses and is between one and two per shift, corresponding to 0.027-0.045 man-hours per tonne of clinker.

Slurry thinners used in filtration. In many cases it was proved that addition of slurry thinners improves the filtration process. The addition of slurry thinners will reduce not only the time required to complete the filtration cycle but also the pressure needed. Addition of calcium hydroxide, kiln dust and gypsum proved to have a direct effect on reducing the percentage of water in the slurry. The addition of 0.1-0.3 per cent of these materials in a dry state was most effective.

On the other hand, in some cases the addition of such materials will create problems as the pH value of the slurry can rise from 7-8 to 10-12 and this necessitates the addition of neutralizing agents to reduce the pH value.

Filtration cycle time:

(a) Filtration period. The filtration time can be defined as the time required to complete filling of the filter press with slurry and pressing the slurry to dewater it. This takes 15-30 minutes;

(b) Processing period. This includes the time required for discharging, opening valves and transferring the filter cakes. This takes a further 10-15 minutes;

(c) Washing period. This period depends on certain conditions. For example, if there is clogging of the filter media, this will need a period of time to unblock the filter. This is known as the washing period. This rarely happens in the case of the cement industry;

(d) Total cycle time. This can be defined as the time required to complete a total filtration cycle including filtration period, processing period and washing period (if any). Total cycle time is 25-45 minutes.

Temperature. Acceleration of the filtration-cycle time can be achieved by raising the slurry temperature. The temperature mainly affects the viscosity of the slurry which in turn affects the speed at which the slurry passes through the filter press.

Filtrate. The filtrate usually contains a very small percentage of remaining suspended materials, especially when starting up the filtration process, i.e. in the first few minutes. After these minutes, the filtrate should be clear with no suspension appearing at all. The appearance of suspended material in the filtrate depends on the following factors:

(a) Type of slurry. If the slurry is treated with slurry thinners its filtrate will usually contain more suspension than the filtrate released from a slurry not treated with any thinners;

(b) Types of filter media (filter textile). If the fabric is multi-layered, the residual suspended materials will be less than in the case of a one-ply fabric through which more suspended materials can pass with the filtrate;

(c) High pressure. This also increases the residual suspended materials in the filtrate.

It is clear that the residual suspended materials in the filtrate derive from the cake remnants of the last filtration cycle and that these suspended materials or grains represent the very fine grains included in the slurry. The dissolution of some soluble materials is nil in the cement raw mix, except for the salts originally associated with the raw materials. Therefore, the semi-wet process deals best with raw materials originally rich in alkalis (salts) as the filtration process will wash out most of the salts, especially the chlorides, and it is thus possible to produce low-alkali cement from alkali-rich raw materials. If thinning additives are used, such as sodium hydroxide, then the pH value of the slurry produced will increase to around 8-8.5 and, for this reason, it is necessary to add a certain percentage of acids to neutralize the pH value of the slurry. (Examples are HCl, HNO_3, H_2SO_4, $H_3P_2O_5$.) Since these acids are expensive, they can be replaced in the cement industry by introducing into the slurry the carbon dioxide of the exit gases which will act as a weak acid H_2CO_3 which reduces the pH value and precipitates the colloidal materials. In the end, the filtrate passing the filter cloth will contain a very low percentage of residual suspended materials.

Production and storage of filter cakes. Production of filter cakes from various kinds of filters can be summarized as follows:

(a) Suction filters. Filtration by this method depends on using a barrel-shaped cylinder and another filtration cylinder coated with filter cloths. Suction is applied here and because of the difference in pressure, water will leave the slurry and pass through the filter cloth;

(b) Pressure filters. In this process, pressure is applied along the filter-cloth area, pressing the water through the cloth. The filter press is formed of a set of thin plate-shaped cylinders equipped with filter cloths and arranged one after the other like the pages in a book. These plates are enclosed in a metallic structure which accelerates their pressing by using a screw shaft. The pipe which feeds the filter press with slurry using a pump is located in the centre of the metallic structure. As the chambers enclosed between these plates and the main enclosing metallic structure are filled with slurry, the pumps will stop work and the slurry will be pressed against the plates and then released. The resulting filter cakes will be taken out mechanically or manually.

These filter cakes will pass to a belt conveyor lying underneath the filter press, then to round bunkers arranged downstream, and are continuously withdrawn from these by a circumferentially-running screw conveyor. A single-roll belt weigher situated in the filter-cake conveyor belt can control the operation of the round bunkers and screw conveyors together. It establishes a withdrawal rate of a constant weight and, at the same time, the weigh-feeder will control the speed of the belt conveyor. The main belt conveyor will pass the filter cakes to the dryer according to its capacity and this can be controlled from the central control room.

B. Economic considerations

The major economic factors that have to be taken into consideration before taking any conversion decision can be summarized as follows:

Extra capital investment for conversion
Capital cost effect (interest, depreciation etc.)
Additional energy consumption
Additional staff required to operate the additional machinery
Efficiency of fuel (energy) in the existing plant
Expected efficiency of fuel (energy) with the new machinery
Cost of electric power
Water content in the present slurry
Water content in the natural raw materials
Water content in the filter cake, if a filtration process is to be used

All these variables will directly affect the financial viability of conversion. It is worth stating that for a conversion decision to be feasible, it is not enough to aim at the system which promises lowest fuel consumption. The cost of the extra capital investment, the fuel price, the cost of stopping the plant production during conversion, the additional cost of electric power and possible extra staff, are all just as important factors. For a conversion decision to be taken and the converted plant to be just as profitable as the existing wet-process plant, these additional costs have all to be taken into consideration.

The following equation can be used to give a quick estimate:

$$\frac{\text{Additional operating cost}}{10^3 \times \text{fuel price}} + \text{new fuel efficiency} \leq \text{old fuel efficiency.}$$

Extensive research has been carried out by F. L. Smidth and Company, Copenhagen, Denmark, to determine the result of this equation for each of a great number of possible conversion methods. The main conclusions of this research are that a simple conversion with low capital investment is more likely to retain profitability than a complicated conversion, and that an increase of plant output is essential for the overall profitability of a conversion.

From the economic point of view, there are many logical reasons for maintaining the processing of raw materials as in the present wet process. The semi-wet process is not complicated to add and can be combined with the existing installations very smoothly. Abandoning slurry production altogether could involve extensive alterations in the raw-material handling and transport systems within the plant. It would also mean abandoning the original investment in these parts of the machinery and would incur expensive production losses through very long stoppages of the equipment during conversion.

At a series of meetings with personnel of the 8th February Cement Joint Complex and the Government representative, extensive explanations and discussions took place on the various alternatives for converting their wet-process production lines to semi-wet, semi-dry or dry processes. A basic fact to bear in mind is that the raw materials, after their extraction from the quarry, contain more than 15 per cent moisture. Three main possibilities for conversion emerged:

<u>First possibility (simple conversion)</u>

<u>Raw material preparation</u>

Raw materials Slurry	Existing installation
Filtration Filter cake handling and feeding	New installation, operating costs (energy, manpower, maintenance, filter cloth, equal to 100-150 kcal/kg clinker)

<u>Kiln department</u>	<u>Scheme A</u>	<u>Scheme B</u>	<u>Scheme C</u>
Kiln	Unchanged	Unchanged	Shortened
Additional equipment	Chain system	1-stage or 2-stage preheater + dryer + crusher induced-draft fan	Grate preheater + cooler modification + inducted draft fan
Expected heat consumption (kcal/kg)	1 200	1 050	950
Additional energy consumption (kWh/t)	0	7	15

Kiln department	Scheme A	Scheme B	Scheme C
Production increase (%)	0	15-25	25
Cost of conversion	cheap	medium	expensive
Production loss during conversion	small	small	large

Second possibility (complex conversion)

The second possibility would involve a more complex conversion of the wet process to a semi-wet or dry process of raw materials containing more than 15 per cent moisture as follows:

Raw-material preparation

Raw materials Slurry	Existing
Filtration Dryer Raw meal homogenization and feeding	Operating costs equal to 100-150 kcal/kg Supplementary heat 200-250 kcal/kg

Kiln department	Scheme D	Scheme E
Kiln	Shortened	Shortened + increased speed
Additional equipment	4-stage preheater, fan + precipitator, cooler modification	4-stage preheater, calciner, hot-air duct, fan, precipitator + cooler + clinker transport
Expected heat consumption (kcal/kg)	870	800
Additional energy consumption (kcal/kg)	7	15-20
Production increase (%)	30-50	150-200 (if sufficient slurry available)
Cost of conversion	Expensive	Expensive
Production loss during conversion	Large	Large

Third possibility (simple and complex conversion)

The third possibility would be suitable for converting a wet-process cement production line to one using the dry process method so long as the raw materials used have a moisture content of less than 15 per cent.

Raw-material preparation

Raw materials
Material stores (preblending stores)
Dry raw mill (plus furnace depending on moisture; old slurry mill can be used for cement grinding)
Raw-meal homogenizing and storing silos and feeding

Kiln department	Scheme F	Scheme G	Scheme H	Scheme J
Kiln	Unchanged	Unchanged	Shortened	Shortened kiln, increased speed
Additional equipment	Chain system	2-stage preheater + fan	4-stage preheater + fan + precipitator + cooler etc.	Calciner + air duct + fan + precipitator + cooler etc.
Expected heat consumption (kcal/kg)	1 100	950	780	800
Additional energy consumption (kcal/kg)	0	2	7	15-20
Production increase (%)	0	15-25	30-50	150-200
Cost of conversion	Cheap	Fairly cheap	Expensive	Expensive
Production loss during conversion	Small	Small	Large	Large

The following table gives some specific figures based on experience in actual plants to explain clearly the difference between kiln conversions which have been carried out. In addition to the fuel and energy prices given, a wage figure of 33,000 deutsche mark (DM) per man/year is assumed, which corresponds to the current level of wages in the Federal Republic of Germany.

This table gives energy and cost figures for the conversion of small- to medium-size wet-process kilns to the semi-wet process. The systems compared in the table are described in the footnotes at the bottom of the table.

Comparison of energy, time and c. st figures of kilns converted to a semi-wet process (with 18% filter cake moisture content)

Factors for comparison	Process stage	Type of process a/ 1	2	3	4	5
(a) Performance and energy						
Increase in output (t/d)	From	-	500-	500-	1 200-	1 200-
	To	1 200	600	750	1 600	2 500
Number of filter press		-	1	1	2	4
Heat consumption: (kcal/kg)	For the drying system	-	-	50	80	135
	For the kiln	1 330	1 100	950	870	780
	Total	1 330	1 100	1 000	950	915
Electricity consumption: (kWh/t clinker)	Preparation + filtration + kiln system	22.5	22.5	25	30	36
(= kcal/kg clinker)		90	90	100	120	145
Total energy consumption (kcal/kg clinker)		1 420	1 190	1 100	1 070	1 060
(b) Manpower requirement						
Additional man-hour input for semi-wet process:	(i) filtration + drying	-	0.05	0.04	0.04	0.03
	(ii) Kiln system	-	-	0.01	0.01	0.01
Total additional input (man-hour/t clinker)		-	0.05	0.05	0.05	0.04

continued

(continued)

Factors for comparison	Process stage	Type of process a/ 1	2	3	4	5
(c) Cost b/ (DM per tonne of clinker)						
Total energy (@ DM 20/gallon calorie)		28.80	23.8	22.00	21.40	21.2
Total additional costs in wages (@ DM 17/h)		-	0.85	0.85	0.85	0.7
Filtration maintenance		-	0.75	0.75	0.75	0.75
Filter cloth		-	0.40	0.40	0.40	0.40
Hydrated lime (@ 3% per tonne of clinker)		-	0.30	0.30	0.30	0.30
Lining consumption		0.40	-	-	-	-
		0.40	0.30	0.20	-	-
Steel wear		0.40	0.8	1.00	0.15	0.15
Total cost (DM/t)		30.00	27.20	25.50	23.85	23.50
Saving over wet-process (percentage)		-	9.5	10.0	21.0	22.0
Capital cost (million DM)		-	3.0	5.0	12.0	variable
Redemption period at 15% (years)		-	7	6	4.5	variable

Source: E. A. Niemeyer and Th. Lang, "Criteria for the choice of a semi-wet process", in Process Technology of Cement Manufacturing (Wiesbaden, Bauverlag, 1979).

a/ Type of process:
1. Normal wet-process kiln, slurry 35%.
2. Long kiln, cake-fed, with chains and crosses.
3. Long kiln with crosses and two cyclones, operating with up to 50% drying of the material.
4. Kiln with 2-stage preheater, filter-cake drying system.
5. Kiln with 3-stage preheater, filter-cake drying and precalciner.

b/ Basic cost assumptions: Fuel oil @ DM 190 per tonne; electricity @ DM 8 per 100 kilowatt hours (kWh); energy @ DM 20 per kilocalorie (kcal); wages @ DM 33,000 per annum or DM 17 per hour.

C. Taking the final decision

After discussing in detail all the above possibilities and studying all the 8th February Cement Joint Complex data, it was decided by all the parties to the discussion (personnel of the cement plant, the government representative and the Adviser) that Scheme B (page 28) would presumably be the most suitable type of conversion for this cement plant, providing that most of the existing machinery can continue to be used and will give an increased output. The decision to choose this simple-conversion possibility, starting with the conversion of two out of the four production lines from wet to semi-wet process technology, was based on the following considerations:

(a) It would be possible to continue using the existing raw-materials stores, slurry raw mills, correcting silos, slurry basins and slurry pumps, thus retaining much of the old investment to serve the converted production lines;

(b) It would be possible to use the exhaust air from the clinker cooler as well as the kiln exit gas in the drying system. This means that all the energy which would otherwise be emitted and lost would be utilized in the production;

(c) The filter cake would contain fewer chlorides and alkalis than the original raw mix and therefore the production lines could possibly produce low-alkali cement, i.e. the quality of cement will be improved;

(d) The production capacity is expected to increase 15-25 per cent (see page 28, scheme B), and possibly more, and the additional man-hour input per tonne of clinker for filtration and drying will increase only by approximately 0.04 per cent, plus approximately 0.01 per cent for the kiln system.

After this decision was taken the Adviser started to (a) evaluate the performance of the existing equipment and machinery and to look at the plant's layout; (b) calculate the capacity of the existing equipment; (c) draw up the specifications of the proposed new machinery which would be needed to carry out the conversion; (d) calculate the heat balances and requirements for the old as well as the new machinery; (e) work out the tests on the slurry and raw materials which ought to be done to provide necessary information for the conversion.

II. CAPACITY AND PERFORMANCE REQUIRED IN VARIOUS SECTIONS OF THE 8TH FEBRUARY CEMENT JOINT COMPLEX

Daily clinker production (after conversion of kilns nos. 3, 4, 5 and 6)*

Kilns 1 and 2, production capacity per hour each = 27.5 t/h
Kilns 3, 4, 5 and 6, production capacity per hour each = 42 t/h
Working hours per day = 24

(27.5 t/h x 2 + 42 t/h x 4) x 24 hours

(55 t/h + 168) x 24

= 5,352 t/d

Limestone crushing

(a) Jaw crusher

Working hours per day = 15
Production required = 5,352 t/d x 1.35 = 7,225.2 t/d
Present production = 300 t/h x 15 = 4,500 t/d
Difference = -2,575 t/d

Another two jaw crushers should be installed in the new quarry with a capacity of 250 m^3/h (in addition to the present capacity of 300 t/h). These crushers should be sufficient for the production increase.

(b) Hammer crushers

There are four hammer crushers.

The quantity of limestone required = 5,352 t/d x 1.35 = 7,225.2 t/d
Present production per crusher = 150 t/h
Expected working hours per day = 12
Expected production = 150 t/h x 12h x 4 = 7,200 t/d

It is recommended to install a new hammer crusher of the same capacity (150 t/h) after the conversion of the four kilns has been carried out.

Raw-material storage

Raw materials	Storage capacity (tonnes)	Daily consumption (tonnes)	Supply in store (days)	Distance between quarries and cement plant (kilometres)
Limestone	38 520	7 225.2	5.3	2 & 16
Clay	10 250 (Clay and Silica)	1 129.0	7.7 (Clay and Silica)	6
Silica		192.0		50
Iron ore	4 080	171.0	23.8	24
Fluorspar	1 910	107.0	17.8	16

*Expert's estimation.

Although the limestone storage capacity is only sufficient for 5.3 days' production, one quarry is so near to the cement plant that it can be considered as nearby storage for the cement plant.

Raw mix

Raw materials in the following quantities are used to produce one tonne of clinker:

	kg/t clinker
Limestone	1 350
Clay	211
Silica	36
Iron ore	32
Fluorspar	20
	1 649

Raw mills

Slurry required for the production:

5,352 t/d x 1.6 = 8,563.2 t/d or 407.8 t/h (21 hours daily)

Actual production of the raw mills during 1985:

	Raw mills						
	1	2	3	4	5	6	7
Dry production (tonnes)	216 243	210 790	193 115	197 244	200 856	218 811	204 869
Working time (hours)	6 057	5 806	5 149	5 434	5 690	6 095	5 723
Hourly production (tonnes)	35.7	36.3	27.5	36.3	36.3	35.9	35.8

Average hourly production per mill in 1985 = 36.1 tonnes

There are two raw mills under construction with the same capacity as all the other raw mills.

The average number of working hours per mill in 1985 = 5,708 hours

The average number of working days per mill in 1985 = $\frac{5,708}{21}$ = 272 days

The actual production of the raw mills could be increased to 300 days per year. In this case, the production capacity of the seven present raw mills plus the two new mills under construction would be:

= 300 x 9 x 36.1 x 21 = 2,046,870 t/a

However, the designed capacity of the two new raw mills is 40 tonnes per hour which is likely to raise the average hourly rate to 38 tonnes per hour, which would bring the annual production up to 2,154,600 tonnes per year.

Slurry required for kilns

Average hourly production of kilns 1 and 2 in 1985 = 26.5 t/h

Average annual production of kilns 1 and 2 each = 26.5 x 24 x 300 = 190,800 t/a

Production of both kilns 1 and 2 = 190,800 x 2 = 381,600 t/a

Slurry required for kilns 1 and 2 = 381,600 x 1.6 = 610,560 t/a

Average hourly production of kilns 3, 4, 5 and 6 in 1985 = 32.3 t/h

Total annual production of kilns 3, 4, 5 and 6 = 32.3 x 24 x 300 x 4 = 930,240 t/a

Slurry required for kilns 3, 4, 5 and 6 = 930,240 x 1.6 = 1,488,384 t/a

Clinker production of kilns 3, 4, 5 and 6 after conversion:

42 x 24 x 300 x 4 = 1,209,600 t/a

Slurry required for kilns 3, 4, 5 and 6 after conversion:

1,209,600 x 1.6 = 1,935,360 t/a

Slurry required for all kilns per year after conversion:

610,560 + 1,935,360 = 2,545,920 t/a

The difference between the present raw mill capacity and the slurry which will be required for the clinker production:

2,154,600 - 2,545,920 = -391,320 t/a

Therefore, there will be a shortfall in slurry and a further raw mill has to be installed to provide the converted kilns with sufficient slurry.

The daily capacity of the mill has to be:

391,320/300 = 1,304.4 t/d or 62.1 t/h (working 21 hours a day)

In fact, it is recommended to install a raw mill with an hourly capacity of 75 tonnes, with the following specifications, which can be produced in the Democratic People's Republic of Korea:

Diameter	3.4 m
Length	13.5 m
Motor	2,000 kW
Capacity	75 t/h

This raw mill will produce 75 x 21 x 300 = 472,500 t/a

Slurry silos

Slurry required to satisfy production = 8,563.2 t/d

Storage space required (1.1 t/m^3) = 8,563.2/1.1 = 7,785 m^3/d

Storage space for three days slurry = 3 x 7,785 = 23,355 m^3

Actual capacity:

(a) Slurry basins 2,500 m^3 x 6 = 15,000 m^3

(b) Slurry corrective silos:

450 m^3 x 14 = 6,300 m^3

400 m^3 x 4 = 1,600 m^3

7,900 m^3

Total = 22,900 m^3

It is recommended to install three corrective slurry silos with a capacity of 400 m^3 each, so that there is an additional 1,200 m^3 of storage capacity.

Pumps

Pumps take the slurry from raw mills to the corrective slurry silos. Additional capacity will be required for the two mills under construction plus the new recommended raw mill as follows:

(38 t/h x 2 + 75 t/h)/1.1 t/m^3 = 138 m^3/h

Capacity of one pump = 80 m^3/h

Pumps already at site = 2

It will be necessary to install two further pumps to transfer slurry from the slurry basins to the filter press.

150 t/h/1.1 t/m^3 = 136.4 m^3/h

pump capacity = 80 m^3/h

pumps required = 4 (2 working + 2 standby)

Cement mills

Capacity required to deal with the quantity of clinker produced after converting kilns 3, 4, 5 and 6:

= 1,209,600 t/a + 10% additives (5% gypsum + 5% slag)
= 1,330,560 t/a

Actual capacity:

Two mills with a capacity of 24.5 t/h

Two mills with a capacity of 46.3 t/h

One mill with a capacity of 50 t/h

Total actual production:

(24.5 x 2) + (46.3 x 2) + (50 x 300 x 21) = 1,207,080 t/a

Deficit in grinding = 1,207,080 - 1,330,560 = -123,480 t/a

Deficit per day = 123,480/300 = 411.6 t/d

Deficit per hour = 411.6/21 = 19.6 t/h

It is, therefore, recommended to install a closed-circuit cement mill with a capacity of approximately 50 t/h and dimensions as follows:

Diameter 3.4 m
Length 13.5 m
Number of chambers 2
Capacity of this mill per year = 50 x 21 x 300 = 315,000 t/a

This mill will also help in developing the quality of cement produced.

In addition to the mills already mentioned, there are three cement mills with a capacity of 24.5 t/h each serving kilns 1 and 2. They are producing 3 x 24.5 x 300 x 21 = 463,050 t/a.

Total mill capacity will therefore be:

1,207,080 + 315,000 (new) + 463,050 = 1,985,130 t/a

Clinker storage (for kilns 3, 4, 5, 6)

Material	Storage capacity (tonnes)	Daily consumption (tonnes)	Supply in store (days)
Clinker	34 700	4 032	8.6
Slag	2 350	201	11.7
Gypsum	2 300	201	11.4

The gypsum comes from China and the Soviet Union and can be stored in nearby open stores. The slag comes from 60 kilometres away.

Cement silos

Serving cement mills 4, 5, 6, 7, 8 and 9 (new)

Daily cement production = 42 t/h x 4 x 24 x 10% = 4,435 t/d

Storage capacity required for storing 18 days production (in accordance with the factory's request):

4,435 t/d x 18 = 79,830 t

Actual capacity of cement silos:

3,000 t x (12 + 4) = 48,000 t (4 new silos are in the planning stage)

Shortfall in capacity:

48,000 t - 79,830 t = -31,830 t

It is, therefore, recommended to install three cement silos with a capacity of 10,000 t each to store the cement produced by the cement mills.

Packing machine

Daily cement production = 4,480 t/d

New daily storage capacity of cement silos:

$$= 4{,}480 \text{ t/d} \times \frac{3{,}000 \times 4 + 10{,}000 \times 3 \text{ (new cement silos)}}{3{,}000 \times 16 + 10{,}000 \times 3 \text{ (all cement silos)}} = 2{,}412 \text{ t/d}$$

According to the cement plant's information, 50% of the cement is to be packed in bags.

Required packing capacity per hour: 2,412 t/d/2/12 h/d = 100.5 t/h

It is, therefore, recommended to install a rotary packing machine of 100 t/h capacity or to install two stationary packing machines with a capacity of 60 t/h each.

Compressors

Compressors are needed to transport cement from the cement mills to the silos.

The new cement mills have a production capacity of 50 t/h each.

16 kg cement requires one kg of air for conveying.

50,000 kg/h/16 kg/kg air = 3,125 kg air/h

3,125 kg air/h/1.2/60 = 43.4 m^3/min, say 40 m^3/min

It is recommended to install one compressor with a capacity of 40 m^3/min.

Total capacity of coal storage (for kilns 1, 2, 3, 4, 5, 6)

Item	Storage capacity (tonnes)	Daily consumption (tonnes)	Storage (days)	Distance mine-cement plant (kilometres)
Coal	16 830	1 254	13.4	120

Coal mills (serving kilns N . 3, 4, 5, 6.)

The existing kilns, us a wet process, consume 0.32 t coal/t clinker. After conversion, it is expected that the heat consumption will be 1,050 kcal/kg clinker, i.e. 0.21 t coal/t clinker. Therefore, the coal required to supply the kilns after conversion will be: 42 t/h x 4 x 24 x 300 x 0.21 = 254,016 t/a

Actual capacity:

There are four existing coal mills at present, plus one standby. The production capacity of each mill is 11 t/h. The efficiency of grinding is:

11 t/h x (5-1) x 300 x 21 = 277,200 t/a

Difference between actual production and consumption after conversion:

277,200 t/a - 254,016 t/a = 23,184 t/a

There will, therefore, be a surplus coal-grinding capacity of 23,184 t/a due to conversion of the process which will require only 0.21 t coal/t clinker instead of 0.32 t coal/t clinker. Therefore, four mills will work and one will be a standby mill.

III. DETAILED CALCULATIONS AND ESTIMATES OF THE CAPACITY AND EFFICIENCY OF THE PRESENT AND THE PROPOSED MACHINERY

A. Capacity of the existing rotary kilns

The Chodorow formula (which is used for production-planning purposes by the Cement Industry Institute of the Soviet Union) has been used to calculate the capacity of the kilns in this cement plant.

Chodorow formula:

$$Q = \frac{DL\ [45 + k\ (\frac{D}{L} - 0.02)]}{1{,}000\ [1 + \frac{(W-40)\ 1.6}{100}]}$$

where

Q = Rotary kiln capacity (t/h)
D = Mean kiln diameter (m) = an average of approximately 3.8 m
L = Length of rotary kiln (m) = 150.0 m
W = Water content of slurry (%) = 36.0% approximately
K = Characteristic index of the rotary kiln = 1,150 for long kilns with a L/D ratio of 25-50.
1.6 = Clinker factor

The capacity of the 8th February Cement Joint Complex kilns 3, 4, 5 and 6 can be calculated as follows:

$$Q = \frac{3.8 \times 150\ [45 + 1{,}150\ (\frac{3.8}{150} - 0.02)]}{1{,}000\ [1 + \frac{(36-40)}{100}\ 1.60]}$$

$$Q = \frac{570\ [45 + 1{,}150\ (0.025 - 0.02)]}{1{,}000\ [1 + \frac{(-4)\ 1.6}{100}]} = \frac{570\ [45 + 1{,}150\ (0.005)]}{1{,}000\ [1 + \frac{(-6.4)}{1{,}000}]}$$

$$Q = \frac{570\ (45 + 5.75)}{1{,}000\ [1 + (-0.064)]} = \frac{570\ (50.75)}{1{,}000\ (0.936)} = \frac{28{,}927.5}{936}$$

= 30.905 t/h

= 31 t/h

= 744 t/d per kiln

Although many estimates show that rotary kilns of this type with the same dimensions (4 m diameter x 150 m length) can produce 850 t/d (24h), in this case, and in agreement with the calculations using Chodorow's formula, these kilns cannot produce this quantity unless the K value, the specific clinker factor, the slurry water content, the revolutions per minute and the inclination of the kiln are changed and improved.

The specific capacity of the rotary kiln is a function of the kiln's absolute size. In this case, the specific kiln capacity is approximately 0.49 t/m^3 and not 0.557 t/m^3 as indicated for this type of kiln in some references which give the following:

$\frac{L}{D}$ = 37.5 : 1

Kiln's volume = 1,525 m^3

Specific kiln capacity = 0.557 t/m^3

However, using a specific kiln capacity of 0.49 t/m^3, we get:

1,525 x approximately 0.49 = 747.25 t/d

This figure is nearer the actual capacity and agrees with the calculation based on Chodorow's formula.

According to the data obtained from the cement factory, in 1985 the kilns produced the following:

	Kiln		
	3	4	5
Rated annual capacity (t)	250 000	250 000	250 000
Actual annual production (t)	213 886	193 417	157 939
Working hours per year (h)	6 663	5 988	4 875
Capacity of kiln (t/h)	32.1	32.3	32.4

Many sources indicate that kiln systems of different dimensions, including ones similar to these, have not achieved the rated capacity given for them. The following are examples:

Diameter and length of the kiln (metres)	Designed rated capacity (t/d)	Operational capacity as percentage of designed capacity (%)	Operational capacity (t/d)
4.0 x 150	850	95.8	814.3
4.5 x 170	1 200	81.6	979.2
5.0 x 185	1 700	67.6	1 149.2

B. Heat requirement for drying without calculating waste-heat utilization

Calculations of the heat requirement for the drying process have shown that one has to reckon with an additional heat requirement of from 180 to 250 kcal/kg clinker, providing that the kiln waste gas and cooler waste air are used continuously for raw material drying, and given that the raw materials used have an average moisture content of around 20 per cent.

The heat consumption required can be calculated roughly and easily if the waste-heat values of the kiln for drying purposes are left out of consideration for the time being.

In the case of the 8th February Cement Joint Complex, it is expected that the moisture content in the filter cakes produced by the filter presses will reach 20% and that the clinker factor will be 1.6. Therefore, the percentage of water content in the filter cakes which must be evaporated can be calculated as follows:

$$1.6 \times \frac{20}{80} \times 1,000 = 400\text{g } H_2O\text{/kg clinker}$$

If we assume a low evaporation value of 915 kcal/kg H_2O, then the heat consumption required in drying the filter cakes will be:

400 x 0.915 = 386 kcal/kg clinker

If we take away from this amount about 200 kcal/kg clinker, representing the heat which will be available from the kiln waste gas and the cooler waste air, then the additional heat required for the two-preheater dry kiln system is:

386 - 200 = 186 kcal/kg clinker

The entire waste air produced from the two-cyclone preheater at a temperature of 400-500 °C and with a heat content of about 85 kcal/kg clinker will be used in a counter current to pre-dry the filter cakes.

This leads to a total heat requirement for clinker burning, including raw-material drying, of:

850 kcal/kg + 186 kcal/kg = 1,036, or approximately 1,050 kcal/kg clinker

The electric power requirement for the kiln, including the further processing of the filter cakes, is expected to be 20 kWh/t clinker, and for the travelling grate cooler to be 0.5 kWh/t clinker.

C. <u>Calculating the heat balance of the rotary kiln</u> (semi-wet process)

In considering all the factors which affect a decision on conversion, it is necessary to know how the heat is used and to calculate approximately the heat balance of the kiln.

The principal sources of heat loss (heat consumption) are:

(a) In drying slurry or raw meal, the water is evaporated and heated to the exit-gas temperature;

(b) Carbonates of lime and magnesia are decomposed;

(c) The carbon dioxide (CO_2) evolving through carbonate conversion to lime (CaO) + CO_2 is heated to the exit-gas temperature;

(d) The gases formed from the combustion of the fuel still contain heat when discharged from the kiln at the exit-gas temperature;

(e) Any air used in excess of that required for consumption will be heated to the exit-gas temperature;

(f) Any carbon monoxide (CO) gas occurring due to imperfect combustion represents a loss of heat;

(g) The moisture contained by the coal is evaporated during burning and is heated to the exit-gas temperature;

(h) The clinker formed by reaction between the constituents is heated to the temperature at which it leaves the kiln;

(i) Heat losses from the clinker cooler are calculated as equal to (heat in clinker leaving kiln) - (heat in clinker leaving cooler) - (heat in preheated air), provided this is used in the kiln system. If it is used for coal drying or filter-cake drying, there may be additional heat losses;

(j) The air used for combustion always contains varying quantities of water vapour, depending on relative humidity and temperature, and this water vapour is heated to the exit-gas temperature;

(k) Dust carried out of the kiln by the gases carries out heat according to the temperature of these gases. There may also be some loss due to partial decarbonation of this dust in addition to that in (b);

(l) There are heat losses through the kiln shell and hood by convection and radiation due to its temperature being higher than that of the surroundings;

(m) Certain constituents of the raw materials which contain combined water are dehydrated and decomposed, the water being vaporized and heated to the exit-gas temperature;

(n) Where organic matter is present in the raw material this is consumed in the kiln, requiring air for combustion. A residue of nitrogen is left, which must be heated to the exit-gas temperature.

The following are the sources from which heat is supplied to the kiln:

(a) Combustion of the fuel used;

(b) Heat arising from the clinker after leaving the kiln and contained in the preheated air;

(c) Combustion of any organic matter in the raw materials;

(d) The exothermic reaction during combination between the constituents of the raw materials after decarbonization has taken place.

The best base from which the bulk of these heat values should be calculated is, for the sake of simplicity, the average temperature of the atmosphere, which in Pyongyang may be taken as between 15.5 °C and 20 °C.

In (i) above, there is the equation:

Heat losses from the clinker cooler = (heat in clinker leaving kiln) - (heat in clinker leaving cooler) - (heat in preheated air).

This can be replaced by:

Heat in clinker leaving kiln = (heat in clinker leaving cooler) + (heat in preheated air) + cooler losses.

Cooler losses and kiln-radiation losses are difficult to estimate accurately. Measurements of the shell temperature at a considerable number of points throughout its length, and calculation based on the laws of radiation and convection, will allow of their approximate determination, if the "emissivity coefficient" of the surface and the influence of the surroundings can be taken into account. However, these items, together with other small items where values are not known with sufficient accuracy (e.g. losses (j) to (n)) may be grouped under the heading "losses unaccounted for".

The simplified heat balance will then appear as follows and it will be found to answer very well for comparative purposes:

1. Heat required to evaporate and heat water vapour.	1. Heat due to calorific value of fuel.
2. Heat required for decomposition of carbonates.	2. Heat available from exothermic reaction of clinker formation.
3. Heat loss in CO_2 from raw materials.	3. Heat available from combustion of organic matter etc. in raw material.
4. Heat loss in combustion gases	
5. Heat loss in excess air	
6. Heat loss in CO due to imperfect combustion	
7. Heat loss in coal moisture	
8. Heat loss in clinker leaving cooler	
9. Heat losses not accounted for (obtained by difference)	

The two sides must balance and the heat balance may then be used as desired. As indicated, item 9 includes: radiation losses from kiln and cooler (also convection losses); heat loss through dehydration and dissociation (of clayey materials) of raw meal; heat loss through moisture in air used for combustion being heated to exit-gas temperature; heat loss through dust in gases at exit-gas temperature; heat loss through nitrogen from air used for combustion of organic matter being heated to exit-gas temperature (not however allowed for under organic matter); heat loss through products of combustion of organic matter being heated to exit gas temperature; heat loss through decomposition of carbonates included in dust losses. Where any of these values can be calculated with any accuracy, they may be included as separate items. Approximations may well be made for the last three as they are small amounts and considerable errors will have relatively little effect.

1. Thermal calculations

Caloric value of coal

For coal, the constituents per kilogram are indicated as follows:

C = carbon, H = hydrogen, O = oxygen, N = nitrogen, S = sulphur, W = water and A = ash. Average chemical commposition of coal supplied to the kiln:

(Percentage)

C	H	O	N	S	A	W
68.5	0.80	3.40	0.50	0.60	25.00	1.20

With this information, it is possible to calculate the (net) calorific value of the coal by using the following formula:

$$H = 33{,}900\ C + 121{,}400\ (H - 1/8\ O) + 10{,}500\ (O-S) - 2{,}500\ W \quad kJ/kg\ coal$$

or

$$H = 8{,}100\ C + 24{,}600\ H - 2{,}600\ (O-S) - 600\ W \quad kcal/kg\ coal$$

$$H = 8{,}100 \times 0.685 + 24{,}600 \times 0.008 - 2{,}600\ (0.034 - 0.006) - 600 \times 0.012$$

$$= 5{,}665.3\ kcal/kg\ coal.$$

Air requirement

The air required per kilogram of coal is:

$$I_k = 11.6\ C + 34.78\ H - 4.35\ O + 4.35\ S$$

$$= 11.6 \times 0.685 + 34.78 \times 0.008 - 4.35 \times 0.034 + 4.35 \times 0.006$$

$$= 8.102\ kg/kg\ coal$$

The air requirement in m^3/kg of coal (m^3 under standard temperature and pressure condition) is as follows:

$$L_{V} = 8.102\ kg/kg\ coal/1.293 = 6.266\ m^3/kg\ coal$$

Exit gas from coal combustion

The stoichiometric combustion of a kilogram coal produces a quantity of exit gas at least equal to $G_k = 1 + L_k$ kg/kg (theoretical).

For complete combustion, the purely stoichiometric combustion quantity of air is insufficient. However, in practice, a certain excess air has to be provided, usually expressed as an excess-air factor (n). Excess-air factor n = 1.2 and the quantity of exit gas formed per kilogram of coal is then:

$$G_k = 1 + nL_k.$$

$$G_k = (1 + 1.2 \times 8.102) = 10.722\ kg\ gas/kg\ coal$$

Exit gas from cement-burning process

In cement burning, the carbon dioxide and water vapour from the raw materials are additional to the combustion gases arising from the fuel.

Assumptions:

1 kilogram of raw mix contains (x kg water) + (y kg $CaCo_3$). To produce 1 kilogram of cement clinker requires (z kg raw meal) + (k kg coal). The exit gas per kilogram of clinker will be composed as follows:

From combustion of fuel = $(1 + nL_k)$ k kg

From raw mix (carbon dioxide) = zy x 0.44 kg

From water content (vapour) = zx kg

<u>Exit gas from fuel combustion</u>

$= (1 + nL_k)\ k = 10.722$ kg gas/kg coal

k - to produce 1 kg clinker requires 0.21 kg of coal

<u>Exit gas from raw mix</u> (carbon dioxide)

If 1 kg raw mix contains y kg $CaCO_3$ and to produce 1 kg of clinker requires z kg of raw mix:

$$zt = \frac{100 - kA\psi}{100-I}, \qquad z = z_t \frac{100}{100 - D}$$

zt - to produce 1 kg of clinker requires a theoretical z_t kg of raw meal.

A - Coal ash content (%)

D - Rate

I - Loss on ignition of raw mix

ψ - 0.9 (absorbed quantity of coal ash in the raw mix on firing while 0.1 is evolved with kiln-exit gases)

k = 0.21, the quantity of coal applied in burning the clinker.

$$z_t = \frac{100 - 0.21 \times 25 \times 0.9}{100 - 36} = 1.489 \text{ kg raw mix/kg clinker}$$

$$z = 1.489 \frac{100}{100-6} = 1.584 \text{ kg raw mix/kg clinker}$$

Therefore, exit gas from raw mix (carbon dioxide):

$$z_t \frac{44}{56} CaO + (z - z_t) \frac{44}{56} \frac{(CO_2)}{CaO} CaO$$

Percentage of CaO in raw mix = 0.4551

β = 0.3 (the percentage of CO_2 leaving the kiln)

$$1.489 \frac{44}{56} 0.4451 + (1.584 - 1.489) \frac{44}{56} 0.4451 \times 0.3 = 0.530 \text{ kg } CO_2\text{/kg clinker}$$

<u>Exit gas from water content</u>

If 1 kg of raw mix contains x kg of water,

therefore, $x = 1.0 \times \frac{20}{80} = 0.25$ kg H_2O/kg meal

$xz = 0.25 \times 1.584 = 0.396$ kg H_2O/kg clinker

Total G_k = 10.722 + 0.530 + 0.396

Combustion of fuel CO_2 H_2O

G'_k = 10.772 k + 0.926 kg/kg clinker

1 kg of kiln-exit gas corresponds to about 0.76 m^3

Hence $G_{\bar{v}}$ = 0.76 (10.722 k + 0.926)

= 8.149 k + 0.706 m^3/kg clinker

2. Heat consumption of clinker-burning process

Heat balance:

Heat of clinker formation 420 kcal/kg clinker (a)

Exit-gas heat loss from fuel

$(1 - n L_k) k C_g (t - 20)$ t = 150 °C   (I)

From exit gases

C_g = 0.23 - 0.00005 t = 0.23 - 0.00005 x 150 = 0.238 kcal/kg K (Kelvin)

Then applying formula (I)

10.722 k x 0.238 x (150 - 20) = 331.74 k/kcal/kg clinker (b)

Exit-gas heat loss from raw mix (carbon dioxide)

0.53 C_{CO_2} (t - 20)

For carbon dioxide:

C_{CO_2} = 0.19 - 0.00011 t = 0.19 - 0.00011 x 150 = 0.2065 kcal/kg K

0.530 x 0.2065 x (150 - 20) = 14.23 kcal/kg clinker (c)

Exit-gas heat loss from water vapour

0.396 x C_{H_2O} (t - 20)

For water vapour:

C_{H_2O} = 0.42 + 0.000185 t = 0.42 + 0.000185 x 150 = 0.448 kcal/kg clinker

therefore, 0.396 x 0.448 x (150 - 20) = 23.06 kcal/kg clinker (d)

Water evaporation

0.396 x 586 kcal/kg H_2O = 232.06 kcal/kg clinker (e)

<u>Waste heat in clinker</u>

$1.0\ C_k\ (t_k - 20)$

t_k = clinker temperature = 150 °C

For clinker:

$C_k = 0.18 + 0.00007\ t = 0.18 + 0.00007 \times 150 = 0.1905$ kcal/kg K

therefore, 1.0 x 0.1905 x (150 - 20) = 24.765 kcal/kg clinker (f)

<u>Radiation losses</u>

	Kiln				1-stage cyclone	2-stage cyclone	Dryer	Silo	Cooler
Temperature of surface (°C)	150	270	130	100	100	70	50	60	-
Length (m)	25	20	30	58	dia 5.6	dia 2.3	dia 1.6 x 40		30
Partial surface area (m^2)	345	276	414	801	364	200	145	135	I a/
Heat loss per m^2/h (kcal/m^2h °C)	29.7	31.0	28.9	27.0	27.0	25.0	24.4	24.4	II b/
Heat loss per kg clinker (kcal/kg clinker)	31.7	50.9	31.3	42.7	18.7	6.0	2.5	2.4	10.67

Total heat loss per kilogram of clinker =

31.7 + 50.9 + 31.3 + 42.7 + 6.0 + 2.5 + 2.4 = 211.2 kcal/kg clinker (g)

Radiation losses can be calculated from the following formula:

$$\frac{\Sigma \alpha_i\ s_i\ (t-20)}{G} \text{ kcal/kg clinker}$$

G = Kiln capacity (kg/h)

α = Heat loss per m^2 and hour (kcal/m^2ch °C) II (table) b/

t = Temperature of surface area (°C)

S = Surface area (m^2) I (table) a/

<u>Dust heat loss</u>

$z - z_t \times C$

(1.584 - 1.489) x 0.21 kcal/kg K (150-20) (h)

= 2.6 kcal/kg cl

Dust heat loss from limestone dissociation

$CaCo_3 \rightarrow CaO + CO_2$

$(z - z_t)$ [heat of reaction + heat of dissociation of $CaCO_3$ + heat of dissociation of $MgCO_3\beta$]

$(z - z_t)$ [5.68 Al_2O_3 + (7.08 CaO + 4.09 MgO)β]

= (1.584 - 1.489) [5.68 x 2.77 + (7.08 x 44.51 + 4.09 x 1.31)0.3]

= 10.6 kcal/kg clinker (i)

Ignition loss of coal (approximately 3%)

5,665.3 k x 0.03 = 169 k (j)

Excess air heat loss

0.50 m^3/kg clinker x 0.317 x (150-20) = 20.60 kcal/kg clinker (k)

Total heat loss

= 420 (a) + 331.74 k (b) + 14.23 (c) + 23.06 (d)
+ 232.06 (e) + 24.765 (f) + 211.2 (g) + 2.6 (h)
+ 10.6 (i) ÷ 169.90 k (j) + 20.68 (k) = 959.11 + 501.64k

Calorific value of coal = 5,665.3 k

959.11 + 501.64 k = 5,665.3 k

k = 0.1856 kg coal/kg clinker

specific heat consumption:

kH = 0.1856 x 5,665.3 = 1,050.35 kcal/kg clinker

Summary of kiln heat balance (semi-wet process)

	kcal/kg clinker
Heat of clinker formation	420
Exit-gas heat loss from coal	61.50
Exit-gas heat loss from raw meal	14.23
Exit-gas heat loss from water vapour	23.06
Exit-gas heat loss from excess air	20.60
Water evaporation	232.06
Waste heat of clinker	24.76
Radiation loss	211.20
Dust heat loss	2.60
Dust heat loss from limestone dissociation	10.60
Ignition loss of coal	30.92
Total	∿ 1 050

3. Heat balance of the dryer and the cyclone

Material balance

If rotary kiln dust is	Y	=	15%
First preheater and second preheater	η_1	=	80%
Raw-meal cyclone	η_2	=	90%

Raw meal is coming from the exit pipe, and gas and 80 per cent of the raw meal are circulating in the ducts and preheater.

i. Contents of dust in kiln exit gases

$$\frac{z_t}{1-y} - z_t = \frac{1.489}{1-0.15} - 1.489 = 0.263 \text{ kg/kg clinker} \qquad (a)$$

ii. Material input to the kiln (raw meal)

$$\frac{z_t}{1-y} = \frac{1.489}{1-0.15} = 1.752 \text{ kg/kg clinker} \qquad (b)$$

iii. Material input to second-stage preheater

$$\frac{1.752}{n} = \frac{1.752}{0.8} = 2.190 \text{ kg/kg clinker} \qquad (c)$$

iv. Dust emitted with exit gases from second-stage preheater

$$c - b = 2.190 - 1.752 = 0.438 \text{ kg/kg clinker} \qquad (d)$$

v. Material from first-stage preheater

$$c - a = 2.190 - 0.263 = 1.927 \text{ kg/kg clinker} \qquad (e)$$

The same method of calculation was applied to the other cyclone as shown in figure I. This figure shows the raw meal input to the cyclones and dryer.

Heat balance of dryer and cyclone

Heat input (Q)

i. Heat of exit gases from first-stage preheater (Q_1)

as calculated before.

$$C_g = 0.23 + 0.00005\ t$$

$$C_g = 0.23 + 0.00005 \times 500 = 0.255 \text{ kcal/kg K}$$

$$C_{CO_2} = 0.19 + 0.00011\ t$$

$$C_{CO_2} = 0.19 + 0.00011 \times 500 = 0.245 \text{ kcal/kg K}$$

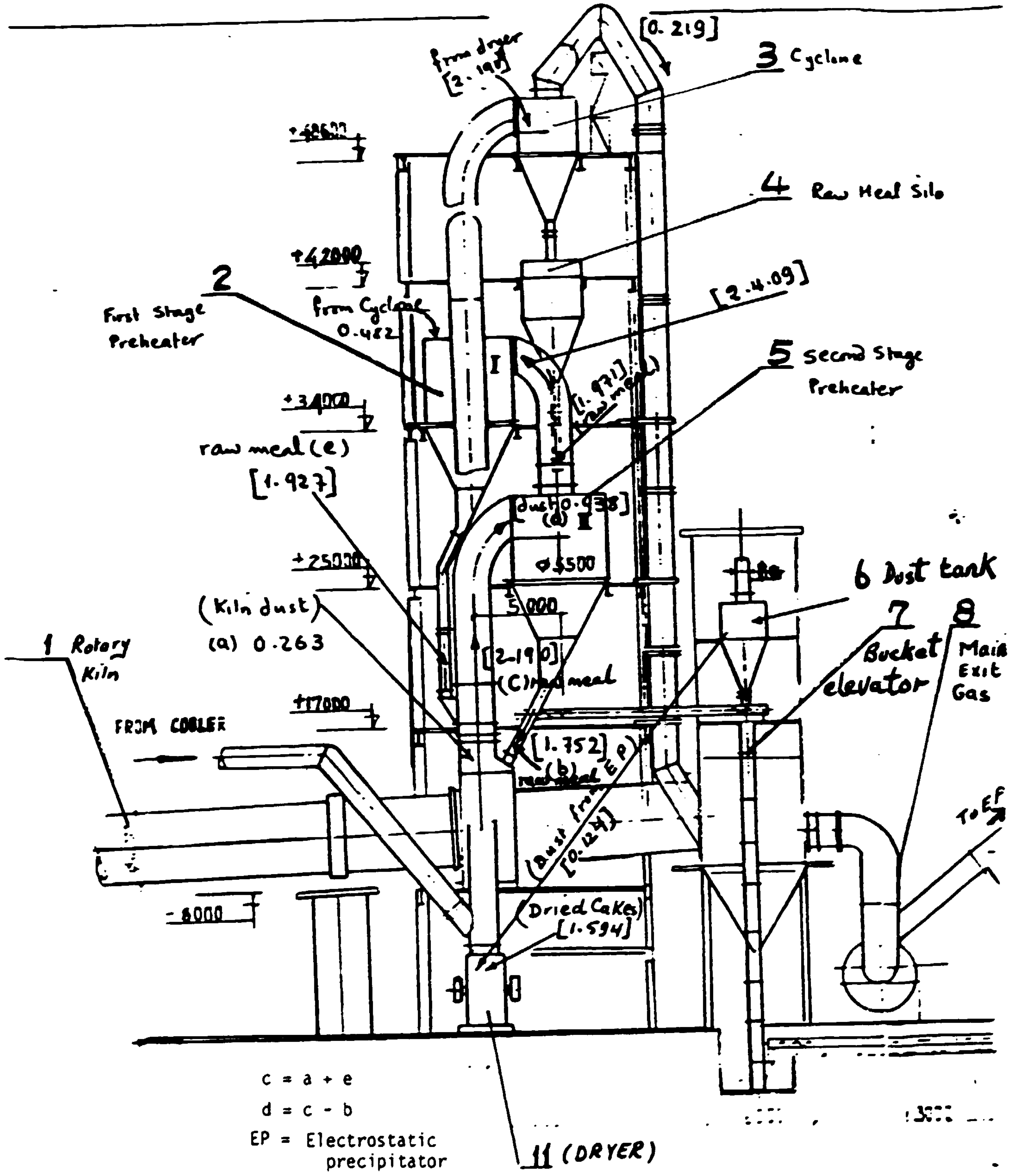
t_1 = temperature of exit gases from first-stage preheater as calculated before

$$Q_1 = (10.722\ kC_g + 0.53\ C_{CO_2}) \times (t_1 - 20)$$

$$Q_1 = (10.722 \times 0.1856 \times 0.255 + 0.530 \times 0.245) \times (t_1 - 20)$$

$$= 0.647\ t_1 - 12.735$$

Figure 1. The raw-meal input to cyclones and dryer



ii. **Heat of dust emitted from first-stage preheater** (Q_2)

$Q_2 = 0.482\ C_m(t_1 - 20) = 0.48 \times 0.253 \times (t_1 - 20) = 0.122\ t_1 - 2.439$

where C_m = specific heat of raw meal

iii. **Heat of air coming from grate cooler** (Q_3)

$Q_3 = 0.5\ m^3/kg\ clinker \times C_{air}\ (t_{air} - 20)$
$= 0.5 \times 0.322 \times (350-20) = 53.130$ kcal/kg clinker

C_{air} = specific heat of air $C_{air} = 0.322\ kcal/m^3\ K$

t_{air} = temperature of air from cooler $t_{air} = 350\ °C$

Total $Q = Q_1 + Q_2 + Q_3$
$= (0.647\ t_1 - 12.735) + (0.122\ t_1 - 2.439) + 53.130$
$= 0.759\ t_1 + 37.956$

Heat output (Q')

i. **Water evaporation** (Q'_1)

$Q'_1 = 232.06$ kcal/kg clinker

ii. **Exit gases heat loss** (Q'_2)

$Q'_2 = 119.39$ kcal/kg clinker

iii. **Dust heat loss** (Q'_3)

$Q'_3 = 0.219 \times C_m \times (150 - 20)$

$= 0.219 \times 0.21 \times (150 - 20)$

$= 3.219$ kcal/kg clinker

iv. **Radiation loss** (Q'_4)

$Q'_4 = 8.4$ kcal/kg clinker

v. **Material heat from cyclone** (Q'_5)

$Q'_5 = 1.971 \times 0.21 \times [150 - (60 + 20)] = 28.974$ kcal/kg clinker

60 = (gas temperature out of cyclone) - (material temperature out of cyclone) = $t_g - t_m$ °C

20 = air temperature

Total $Q' = Q'_1 + Q'_2 + Q'_3 + Q'_4 + Q'_5$
$= 232.06 + 119.39 + 3.219 + 8.4 + 28.974$
$= 392.043$ kcal/kg clinker

$t_1 = 466.518$ °C $Q = 0.759\ t_1 + 37.956 = 392.043$

therefore, $Q = Q'$

4. Heat balance of first-stage preheater

Heat input (Q)

i. **Heat of exit gas from second-stage preheater** (Q_1)

where

$$Q_1 = (1.988 \times C_g + 0.530 \times C_{CO_2}) \times (t_2 - 20)$$

and

$$C_g = 0.23 + 0.00005 \times 700 = 0.265 \text{ kcal/kg K}$$

$$C_{CO_2} = 0.19 + 0.00011 \times 700 = 0.267 \text{ kcal/kg K}$$

t_2 = temperature of gas from first cyclone

$$Q_1 = (1.988 \times 0.265 + 0.530 \times 0.267) \times (t_2 - 20)$$
$$= 0.668\ t_2 - 13.367 \text{ kcal/kg clinker}$$

ii. **Heat of dust from first-stage preheater** (Q_2)

$$Q_2 = 0.438 \times C_m \times (t_2 - 20) = 0.438 \times 0.283\ (t_2 - 20)$$

$$= 0.124\ t_2 - 2.479 \text{ kcal/kg clinker}$$

iii. **Heat of material from raw-meal cyclone** (Q_3)

$$Q_3 = 28.974 \text{ kcal/kg clinker}$$

$$\text{Total } Q = Q_1 + Q_2 + Q_3$$

$$= (0.668\ t_2 - 13.367) + (0.124\ t_2 - 2.479) + 28.974$$
$$= 0.792\ t_2 + 13.128$$

Heat output (Q')

i. **Heat losses of exit gas from first-stage preheater** (Q'_1)

$$Q'_1 = 0.637\ t_1 - 12.735 = 0.637 \times 466.5 - 12.735$$
$$= 284.437 \text{ kcal/kg clinker}$$

ii. **Heat losses of dust from first-stage preheater** (Q'_2)

$$Q'_2 = 0.122\ t_1 - 2.439 = 0.122 \times 466.5 - 2.439 = 54.476 \text{ kcal/kg clinker}$$

iii. **Radiation heat losses** (Q'_3)

$$Q'_3 = 2.5 \text{ kcal/kg clinker}$$

iv. **Heat of dehydration of kaolin** (Q'_4)

$$Q'_4 = z \times 5.68 \times Al_2O_3 = 1.584 \times 5.68 \times 2.77 = 24.922 \text{ kcal/kg clinker}$$

v. **Heat of material from first stage preheater** (Q'_5)

$$Q'_5 = 1.927 \times 0.283\ [t_1 - (50 - 20)] = 216.237 \text{ kcal/kg clinker}$$

50 = (gas temperature out of preheater) - (material temperature out of preheater) = $t_g + t_m$ °C

Total $Q' = Q'_1 + Q'_2 + Q'_3 + Q'_4 + Q'_5$

$= 284.437 + 54.476 + 2.5 + 24.922 = 582.273$ kcal/kg clinker

$t_2 = 718.617$ °C

$0.792\ t_2 + 13.128 = 582.273$

Therefore, $Q = Q'$

5. <u>Heat balance of second-stage preheater</u>

<u>Heat input</u> (Q)

i. <u>Heat of exit gas from kiln</u> (Q_1)

$Q_1 = (1.988 \times C_g + 0.530 \times C_{CO_2}) \times (t_g - 20)$

$C_g = 0.23 + 0.00005 \times 900 = 0.275$ kcal/kg K

$C_{CO_2} = 0.19 + 0.00011 \times 900 = 0.289$ kcal/kg K

t_g = temperature of exit gas from kiln

$Q_1 = (1.988 \times 0.275 - 0.530 \times 0.289)\ (t_g - 20) = 0.670\ t_g - 13.997$

ii. <u>Heat of dust from kiln</u> (Q_2)

$Q_2 = 0.263 \times C_m \times (t_g - 20) = 0.263 \times 0.283\ (t_g - 20) = 0.074\ t_g - 1.489$

iii. <u>Material heat from first-stage preheater</u> (Q_3)

$Q_3 = 216.237$ kcal/kg clinker

Total $Q = Q_1 + Q_2 + Q_3$

$= (0.670\ t_g - 13.997) + (0.074\ t_g - 1.489) + 216.237$

$= 0.744\ t_g + 200.751$

<u>Heat output</u> (Q)

i. <u>Heat losses of exit gases</u> (Q'_1)

$Q'_1 = 466.669$ kcal/kg clinker

ii. <u>Heat losses of dust</u> (Q'_2)

$Q'_2 = 86.629$ kcal/kg clinker

iii. <u>Radiation heat losses</u> (Q'_3)

$Q'_3 = 18.7$ kcal/kg clinker

iv. Material heat input to kiln (Q'_4)

$Q'_4 = 1.752 \times 0.283 \times (t_2 - 20) = 1.752 \times 0.283 \times (718.6 - 20)$

$= 321.595$ kcal/kg clinker

Total $Q' = Q'_1 + Q'_2 + Q'_3 + Q'_4$

$= 466.669 + 86.629 + 18.7 + 321.595 = 893.593$ kcal/kg clinker

Exit kiln gas temperature = t_g = 931.239 °C

$0.744\ t_g + 200.751 = 893.593$

Therefore, $Q = Q'$

(See figure 2 for the estimated temperatures and volumes of the new semi-wet kiln system.)

6. Heat balance of cooler

Heat input (Q)

i. Heat content of clinker discharged from kiln (Q_1)

$Q_1 = 1.0\ C_k\ (t_1 - 20)$
$C_k = 0.18 + 0.00007\ t_1 = 0.18 + 00007 \times 900 = 0.243$ kcal/kg K
t_1 = clinker temperature discharged from kiln = 900 °C
$Q_1 = 1.0 \times 0.243\ (900 - 20) = 213.84$ kcal/kg clinker

Heat output (Q')

i. Heat content of clinker discharged from cooler (Q'_1)

$Q'_1 = 1.0\ C_k\ (t_2 - 20)$
$C_k = 0.18 + 0.00007\ t_2 = 0.18 + 0.00007 \times 150 = 0.190$ kcal/kg clinker
t_2 = clinker temperature discharged from cooler = 150 °C
$Q'_1 = 1.0 \times 0.190 \times (150 - 20) = 24.70$ kcal/kg clinker

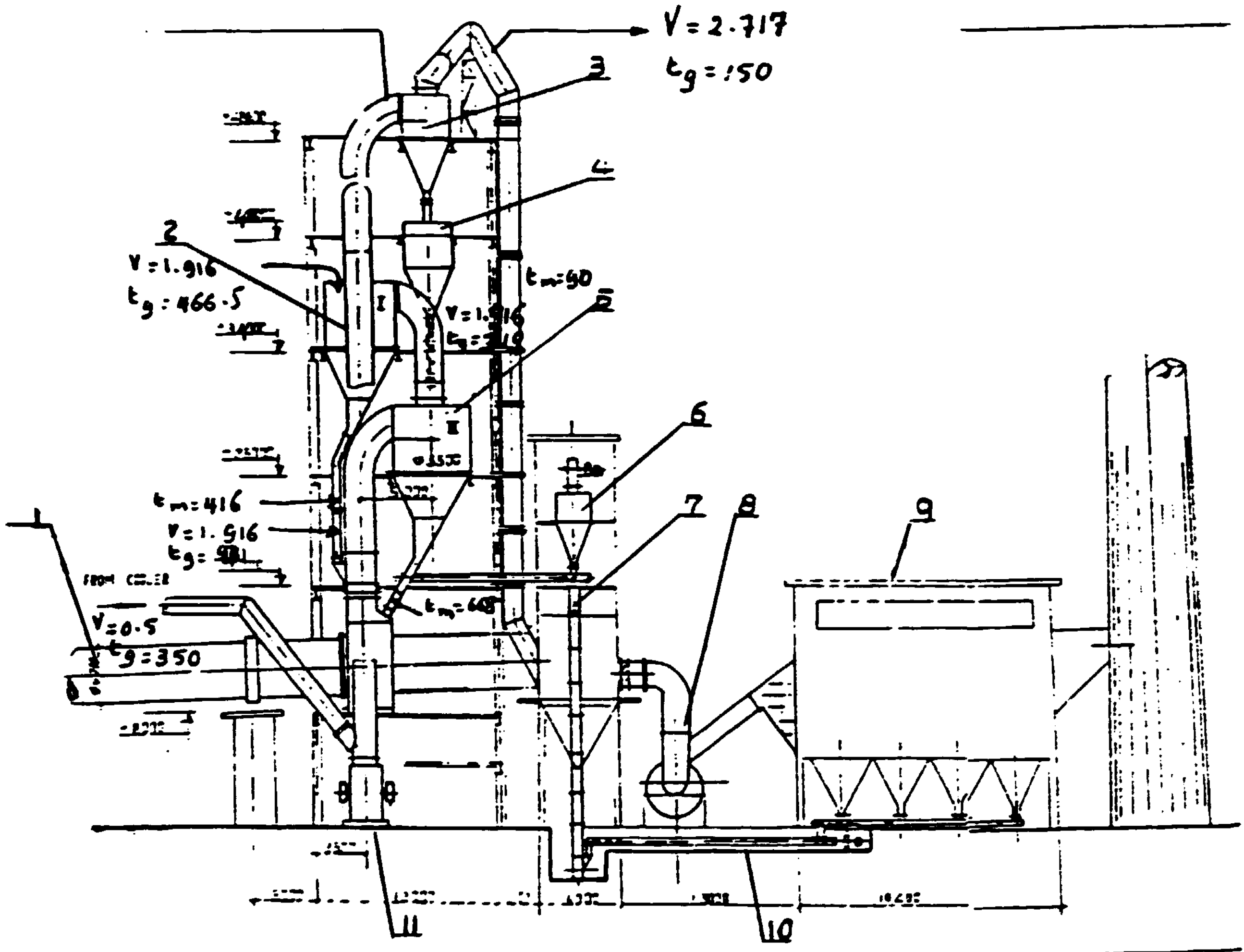
ii. Heat losses due to radiation and convection (Q'_2)

approximately 5% of heat content of clinker discharged from kiln
$= Q'_2 = 213.84 \times 0.05 = 10.67$ kcal/kg clinker

iii. Heat content of exhaust air from cooler (Q'_3)

If:
cooling air = 1.8 m^3/kg clinker
secondary air is 85% of all air of kiln
secondary air = $L_V.K \times 0.85 = 6.266 \times 0.182 \times 0.85 = 0.969$ m^3/kg clinker
air from cooler to dryer = 0.5 m^3/kg clinker
exhaust air = $1.8 - (0.969 + 0.5) = 0.313$ m^3/kg clinker

Figure 2. Estimated temperatures and volumes of the new semi-wet kiln system



Key: V = Volume (Nm3/kg clinker)
t_g = gas temperature (°C)
t_m = material temperature (°C)
Nos. 1-11 see figure V(a).

Therefore:

$$Q'_3 = 0.313 \times C_{air} (t_{air} - 20) = 0.313 \times 0.317 \times (150 - 20)$$
$$= 13.64 \text{ kcal/kg clinker}$$

__Thermal efficiency of the cooler__

$$\eta = \frac{\bar{Q}_1 - (\bar{Q}'_1 + \bar{Q}'_2 + \bar{Q}'_3)}{\bar{Q}_1} \times 100$$

$$= \frac{213.84 - (24.70 + 10.67 + 13.64)}{213.84} \times 100$$

$$= 77.1\%$$

IV. TESTS REQUIRED PRIOR TO SELECTING NEW PROCESS

A. Information needed

In principle, either vacuum or pressure filters can be used to de-water the slurry produced in 8th February Cement Joint Complex if the present wet process is converted to a semi-wet process. The filtration process depends on many factors such as the chemical, physical and mineralogical constituents of the slurry, the viscosity, and the shape and grain-size distribution of grains in the slurry, as well as the amount of pressure used, the type of filter media, the compressability of the filter cakes and the temperature of the slurry. It is necessary to try and quantify all these factors. Therefore, the slurry, filter dusts, fuel and water in the 8th February Cement Joint Complex have to be carefully tested and analysed before selecting a filtration system. The following information would be needed.

Fineness of raw-material slurry. An analysis of the particle size of the materials in the slurry has to be carried out which gives the percentage of the particles remaining on 500, 250, 125, 63 and 32 micron sieves.

Chemical analysis of the main constituents of slurry. The percentages of SiO_2, Al_2O_3, Fe_2O_3, CaO, MgO, K_2O, Na_2O, Cl, SO_3 and loss on ignition (LOI) should be ascertained.

Water content of slurry. The water content, the density and the pH value have to be determined.

Electrostatic precipitator filter dust. The dust produced by the electrostatic precipitators in the converted kilns must be subjected to both chemical and physical tests to determine its characteristics, as follows:

(a) Fineness. Residue on sieve: sizes 20, 30, 45, 63, 90 and 200 micron;

(b) Chemical analysis. This has to show the following constituents of the dust: LOI, SiO_2, Al_2O_3, Fe_2O_3, CaO, MgO, K_2O, Na_2O, Cl and SO_3.

These analyses will throw light on the possibility of returning from 5-10 per cent of this electrostatic precipitator dust as slurry thinner to the wet process. In case the filter dust is not applicable, CaO (burnt lime) or $Ca(OH)_2$ (slaked lime) may be tested/considered to be added in limited quantities by a dosing system.

Fuel. Both types of the solid fuels (low- and high-volatility coal) used in the cement production (in the proportion 88.85:11.15 per cent) have to be subjected to analysis for calorific value, specific gravity, sulphur content, ash content after burning and content of the fly ash deposited after burning.

Industrial water. The water in the cement plant, which is used for cooling, for washing the filter belts and for preparing the flocculant has to be analysed.

Filtration test:

(a) The slurry used has to be tested for its filterability by either suction or pressure (suction filtration on a rotary-drum, filter and belt press);

(b) The filtration has to be carried out in both cases with and without the addition of hydrated lime (electrostatic filter dust) as a filtration aid;

(c) A chemical analysis of the dried filter cake and sludge samples has to be carried out and the alkali and chlorine contents determined;

(d) The specific dry-product capacity (specific filter capacity kg/m^2h) of both filtration systems (suction and pressure) has to be determined, and also the residual moisture content, amount of suction or pressure applied, the amount in percentage of filter aids (dust or lime) added and the filtration period necessary;

(e) The alkali and chlorine content and the pH value of the dried original sludge and the dried filter cake must be compared to see how much these have been reduced in the filtration process.

<u>Guarantee figures</u>

After these tests have been carried out in the laboratory and specific figures obtained, the contractor should subject the sludge to tests in a pilot plant to determine the exact figures under operational conditions. The contractor will also establish whether the existing equipment is in good working condition.

The performance guarantee data for the slurry filtration plant and the filter-dust washing plant are subject to confirmation after the pilot-plant test results have been evaluated.

If the pilot-plant test results differ from the laboratory test results, the contractor should submit suitable proposals for modification of the equipment to be delivered under the contract. These will be submitted to the company to study, evaluate and give a final decision.

B. <u>Materials needed for the tests</u>

Representative samples of raw materials:

(a) For test purposes, at least 50 kilograms of representative samples of the raw materials which make up the slurry will be needed, in the proportions in which each component is used in the slurry. The minimum amount for any representative sample is 1 kilogram. The average analysis of each component as well as the average moisture content of each representative sample has to be submitted with these samples;

(b) At least one litre of the water that is used for washing must be supplied;

(c) A representative sample of at least 2 kilograms of the dust produced from each kiln must be supplied with an indication of the average quantities that can be recycled after the conversion of the kilns.

<u>Slurry</u>:

(a) In order to carry out the conversion tests, the minimum quantity of slurry to be supplied is 100 litres. This slurry must be free from any material used as slurry thinners;

(b) If the raw material components originate in different quarries, or from different parts of the quarry, this will lead to slurries whose moisture content could vary by 10 per cent or more from the average. In this case, it is necessary to provide a sample of these slurries and of all the constituting raw materials.

Fuel:

A representative sample of each type of coal (low and high volatility) has to be sent to the laboratories carrying out the conversion tests. Each representative sample must be at least 1 kilogram and in the same proportion in which each type is used in burning. The average analysis of each type has also to be submitted.

Necessary precautions in packaging and sending the test samples

The wet-product samples have to be packed in waterproof containers previously cleaned and dried. The moisture content of the sample must not change during the period of collecting and packing. If the sample is not at the ambient temperature at the moment of collection, its temperature must be specified.

V. ILLUSTRATIVE DESCRIPTION OF THE PROPOSED NEW MACHINERY AND EQUIPMENT WITH SOME SPECIFICATIONS

All the specifications and figures for the proposed new machinery and equipment have already been calculated in order to select the guarantee figures of the process and to help the Government in preparing the tender documents. Conversion will start with kilns 5 and 6 and conversion of the other kilns will follow at a later stage.

The supplier or contractor is expected to recalculate and to recheck the figures and the proposed conversion and if necessary to modify the conversion approach so as to be able to take responsibility for fulfilling the necessary guarantee figures.

A. Precipitator dust storage and dosing equipment

Dust silo

This silo will store dust produced by kilns 5 and 6. It consists of a steel silo, complete with inlet supports, manhole, two outlets without frame structure for mounting and it will include:

(a) Capacity-level indicator;

(b) Pipes inside and outside the silo up to the air-distribution station as well as between the air-distribution station and the compressed-air plant, as semi-finished goods, including flanges and seal bolts;

(c) Fittings for emptying the silo. A one-tonne loading chain hoist has to be supplied and to be mounted above the dust silo as described later.

		(approximately)
Technical data:	Diameter	4.0 m
	Base diameter	2.7 m
	Height	9.0 m
	Useful capacity	230.0 m^2

(or as specified and recommended by the supplier)

Silo aeration with homogenizing and special dosing device

For the silo base, including:

(a) Air slides;

(b) Two rotary piston blowers for the air supply (one stand-by), complete with three-phase motor, base plate, silencer and various fittings.

		(approximately)
Technical data:	Flow	8.2 m^3/min.
	Pressure difference	600 mbar
	Installed power	22 kW

(or as specified and recommended by the supplier)

Silo dedusting

This consists of:

(a) Fully-automatic compressed-air filter with pneumatic cleaning device;

(b) Two exhaust-air ventilators (one stand-by), including motor (7.5 kW) and accessories;

(c) Various dedusting pipes between the dust sources and the filter, and their fittings.

Air slide

This includes all fresh- and exhaust-air ducts, various seals, throttle valve and cleaning lid, as well as supporting structure.

		(approximately)
Technical data:	Through-put	18 t/h
	Conveying path	8.5 m
	Inclination of slide	12°

(or as specified and recommended by the supplier)

Two high-pressure ventilators (one stand-by)

Impeller vane mounted on motor shaft including three-phase drive and suction filter, as well as mounting elements for ventilator, and pipes to the air slides.

		(approximately)
Technical data:	Flow	7 m^3/min.
	Pressure difference	50 mbar
	Connect load	2.2 kW

(or as specified and recommended by the supplier)

The necessary numbers of screw conveyors will be supplied according to the supplier's design and drawings to convey dust outside the dust silo and the dosing points.

The screw conveyors

The screw conveyors necessary to transfer the precipitator dust from outside to inside the silo or vice-versa or to the various dosing points will be supplied according to the supplier's design. The number, capacity, diameter, length, motor power etc. of these screw conveyors will be given in the supplier's tender with drawings and flow sheet diagrams. These screw conveyors should be manufactured in accordance with the following specifications:

(a) Screw trough in flanged form, with the necessary couplings; bolted-on end plates with built-in bearings and carrier feet; shaft outlets at the end plates will be fitted with cast-iron stuffing boxes; continuous dust-tight flanged plate covers will be supplied; inlet connection and outlet connection with flanges together with a drive plate to carry the driving elements; inspection and cleaning doors;

(b) Screw shaft; a solid shaft with welded-on full-screw flight made of steel plate, intermediate bearings made of cast iron with hinges and lubrication devices; intermediate bearing stations fitted with counter flanges, keys and bolts; end bearings fitted with roller bearings and one bearing will be installed to accept axial loading.

Bulk-flow meters

These are for measuring through-put of precipitation dust and consist of:

(a) Bulk-flow meter with continually-operating flow recorder including measuring transducer in dust-tight aluminium casing, measuring range approximately 0.6-6 t/h;

(b) Standard baffle plate in a suitable casing made of sheet steel;

(c) Through-put recorder with digital reading;

(d) Linearizator for voltage supply of the through-put recorder;

(e) Integrator with counter;

(f) Flexible connection to the subsequent screw conveyor.

B. Lime-dosing equipment

Lime silo

This is a steel silo, complete with inlet supports, one outlet, and without frame structure for mounting:

		(approximately)
Technical data:	Diameter	2.5 m
	Height	7.0 m
	Useful capacity	10.0 m^2

(or as specified and recommended by the supplier)

Top filter (with automatic rapping device)

		(approximately)
Technical data:	Filter surface	15 m^2
	Connect load	0.25 kW

Silo aeration station

Mounted in the outlet funnel with compressed-air supply from the dust-silo aeration station.

Lime dosing station

With rotary lock feeders including speed adjustable DC drive and the necessary screw conveyors.

(approximately)

Technical data:	Rotary lock diameter	2 mm
	Capacity	200-2,000 kg/h
	Drive	0.4 kW

(The supplier is asked to recommend any additional or stand-by equipment that might assist smooth and efficient production.)

C. Filtration, cake and slurry handling for kilns 5 and 6

Slurry mixing and storage tanks

These are made of steel with overflow connections. The internal part of the tank has to be covered with rubber to resist abrasion. There should be a metallic platform on top for mixing support. On the bottom, one outlet with flanges is provided for pump connections.

(approximately)

Technical data:	Diameter	2.5 m
	Height	4.5 m
	Useful height	4.0 m
	Total volume	22.0 m^3
	Useful volume	20.0 m^3
	Thickness of steel wall	4.0 mm

(or as specified and recommended by the supplier)

Mixers for slurry storage

The number of mixers is to be recommended by the supplier. Each consists of a cast-iron support base and roller bearings, directly coupled to a gearbox, and a three-phase motor to drive the agitator, which is fitted with three tapered agitator blades. The surface is hardened steel, the shaft is rubber-covered.

(approximately)

Technical data:	Impeller	85 rpm
	Diameter	1,300 mm
	Shaft length	4,100 mm
	Installed power	14 kW

(or as specified and recommended by the supplier)

Centrifugal pumps

The number is to be recommended by the supplier. These are for slurry feeding to the dust-slurry mixers, designed as an armoured pump with speed-adjustable DC drive, mounted on base plate.

(approximately)

Technical data:	Flow	30.3 m^3/h
	Head	15.0 m liquid column
	Installed power	11.0 kW

(or as specified and recommended by the supplier)

Screw mixer

The number to be recommended by the supplier. These are for mixing dust or other slurry thinning additives with the slurry. Screw mixers have removable mixing blades, special adjusting DC drives, assembled with supporting steel structure.

		(approximately)
Technical data:	Flow	30.0 m^3/h
	Installed power	40.0 kW
	Dimensions	5,000 x 1,500 x 1,000 mm

(or as specified and recommended by the supplier)

Filtration unit, vacuum or pressure filters

It is hoped that with the laboratory and pilot-plant tests on the slurry it will be possible to design an optimally-dimensioned filtration plant, especially with regard to filtration time, cake thickness and working pressure, even when dealing with difficult slurries.

If this filtration unit uses a pressure filter, it will be supplied complete with adjustable DC drive, slurry distribution device (including drive) with all the regulator valves and switches required for operating the machine, pneumatic piping and electric cabling, a suitable set of filter cloth, special cantilever device and walkways as well as central lubrication system for automatic greasing of the individual bearing points.

The leading principle in designing the filter is to achieve a trouble-free unit suitable for continuous operation during a long service life.

The frame has to be kept very simple to provide for easy access and cleaning and to avoid dead corners. All the facings on the frame have to be milled to an accuracy of at least 0.02 mm to ensure that the rolls are placed parallel. Even if the rolls are to be exchanged after some years, difficult fitting work for spare parts is obviated. The wear on the filter cloth has to be reduced to the minimum (any displacement of the rolls creates some form of distortion in the fabrics and, subsequently, the formation of creases).

The lubrication of the unit(s) has to be carried out by a central lubrication plant. All necessary lubrication points on the machines will be supplied with grease by means of a grease distributor from a central pumping station consisting of a pneumatically-operated pump with all the fittings required to function smoothly, including a central switch-board to control the de-watering plant.

If necessary, the grease supply to the individual presses can be interrupted by shut-off valves. In order to facilitate drawing in and drawing out of both felts, the complete heavy frame bottom is lifted up by an overhead crane and a special lifting beam. The whole surface of the machine's frame has to be sand-blasted and coated with glass fibre reinforced polyester (minimum thickness of coat: 1.5 mm) which results in an impact-resistant, anti-corrosion protective layer. On connecting faces, this layer is up to 5 mm thick. All other parts of the machine not coming into direct contact with the medium being de-watered, not coated with synthetic material and not made of stainless steel or plastic, are sand-blasted and given two coats of epoxy-resin prime coat and finishing coats (thickness of coat: 150-170 micron).

The roll structure, the breast rolls (or tension rolls), the drive rolls, perforated rolls, de-watering rolls, tension rolls for high-pressure zone and the tracking rolls for the filter cloth have to be designed in such a way that the materials, the function and the performance of these parts can serve for at least 15 years (coated, good and easy coupling, uniform tension, very low wear).

Function of the filter press. Both felts with the sludge cake in between them are wrapped round the press rolls. The sludge cake is subjected to surface pressure (which is directly proportional to the felt tensions and inversely proportional to the roll radius) by the felts under variable tensile stress. Due to the reducing roll diameter in the running direction of the felts, the surface pressure increases according to the above relation. This ensures that, in each case, the maximum possible pressure is exerted corresponding to the progress of the de-watering process and resulting in raising the stability of the sludge. The geometrical arrangement of the rolls leads to a minimal relative speed of both felts against each other. The shearing thus caused in the sludge cake compresses the solids and presses out more interstitial water.

In order to improve stripping of the water at the bottom, S-tension rolls with the belts wound around the water ducts consisting of a beam with rubber stripper are installed. (See diagrammatic view of filter press in figure 3.)

Press roll for the high-pressure zone. This press roll is pressed against the last roll in the S-tension zone by means of a pneumatically-operated lever system, with a line pressure of up to 30 kg/cm roll width. Due to this line pressing which follows the surface pressing at high pressure in the S-tension zone, the sludge cake is further de-watered and the correct dryness obtained.

Table rolls. These rolls serve as back-up rolls in the gravity zone and as support for the felts in the wedge zone.

Deflection rolls and spreader rolls. The number of the spreader rolls have to be determined by the supplier for each top and bottom felt, in order to prevent the felt shrinking across the breadth and to prevent folds forming. All rolls have to be supported by spherical roller bearings or self-aligning ball bearings. The extreme operating loads encountered at all bearing points in sludge-de-watering machines are handled by bearing seals developed especially for the filter-press machine and consisting of a grease-filled multiple labyrinth with rubber seals. The upper and lower felts have to be provided with:

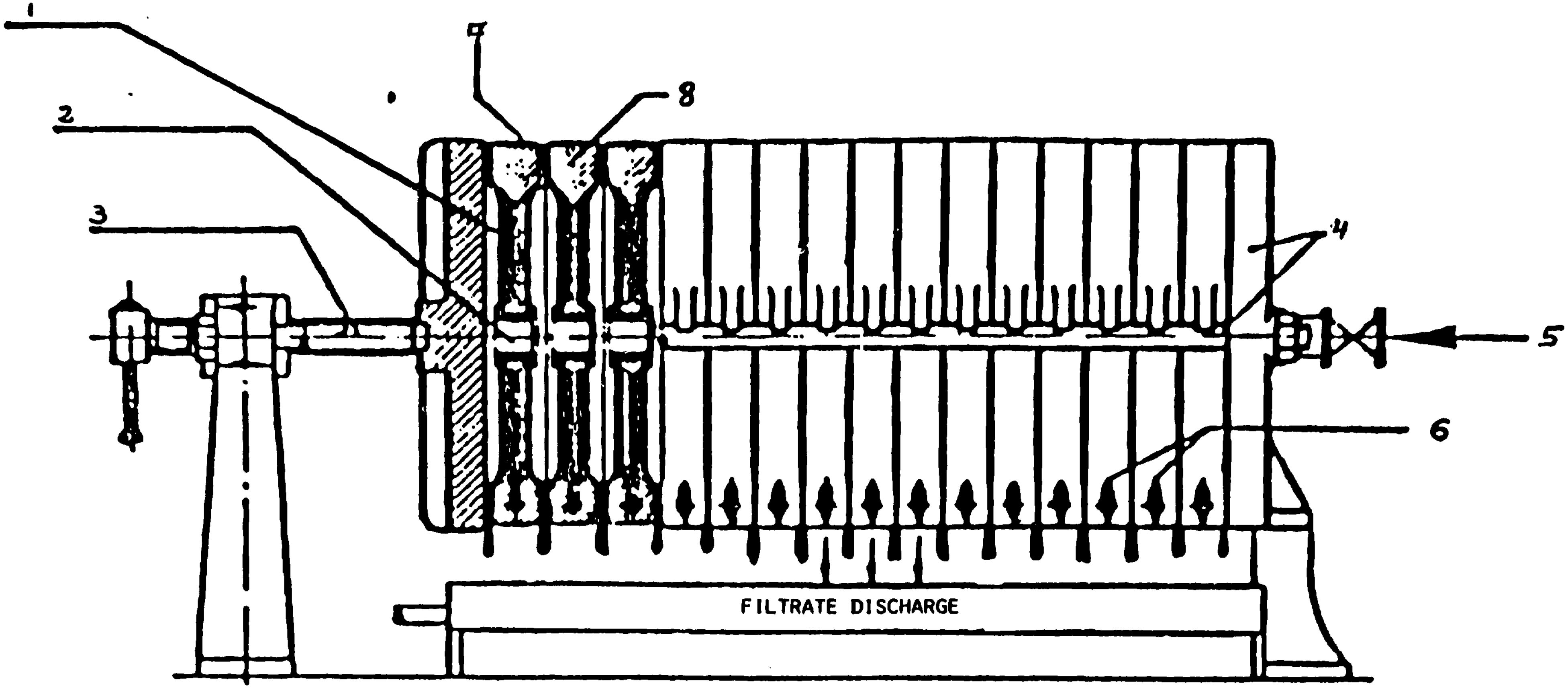
(a) A tracking device to control the swinging movement of the tracking rolls;

(b) Controlling devices with feeler plates and adjustable mounting bracket. The position of the felt edge is transmitted by the feeler plates to the switchboard, which regulates the air pressure of the regulating element. Standard pressurized air is sufficient for feeding the switchboard.

Limit switch for felt tracking. If an unexpected fault occurs in the felt-tracking system, limit switches shut the machine down without the material of the filter felts being destroyed by running up against the machine frame. In addition to that, the machine has to be safeguarded against possible accidents on the tender side as well as on the drive side by so-called cable-hoist safety switches.

(The number, design and specifications of the filter units and their accessories have to be recommended by the supplier.)

Figure 3. Diagrammatic view of filter press



Key:
1 FILTER CHAMBERS
2 INLET CHANNEL
3 PRESSURE SPINDLE
4 SUPPORTING FRAME WITH FIXED HEADPIECE
5 SLURRY FEEDING
6 FILTRATE DISCHARGE COCKS
7 FILTER CLOTHS
8 FILTER PLATES

Source: HOLDERBANK Training Course.

Felt cleaning devices. The top and bottom felts each pass through a closed cleaning box where the felts are cleaned by means of spray water. The belt wash boxes are open on the cantilever side in order to facilitate felt changing. The showers are fitted with non-clogging flat jet nozzles. For cleaning the nozzles, the showers are equipped with brushes on the inside which can be operated during felt cleaning by means of an automatically-operated activating drive. The nozzles and pipes have to be made of stainless steel and all other parts of red bronze or brass. The belt spray box is sealed by fatigue-resistant rubber profiles.

(The number, design and specifications to be recommended by the supplier.)

Devices for high-pressure cleaning. In addition to the felt cleaning devices mentioned above, spray boxes are mounted for the top and bottom felts, whose showers are fitted with flat and needle jet nozzles. They are switched on for short periods, as required, in order to remove deposits inside the felts with high-pressure jets. This device differs from the felt-cleaning devices described above in the design of its nozzles.

(The number, design and specifications to be recommended by the supplier.)

Pipe extractors. These are for demoisturizing and additional cleaning for top and bottom felts.

(The number, design and specifications to be recommended by the supplier.)

Take-off doctors for drive rolls. These are for de-watering the bottom and top felts, consisting of doctor blades, which rest on a special blade holder thus allowing quick changing of the blades. Blades are resistant to wear and made of special material.

(The number, design and specifications to be recommended by the supplier.)

The machine has to be provided with a felt-changing device.

Filtrate trays. The filtrate trays are made of stainless-steel plates and welded to the machine frame. In these trays, the filtrate from the gravity, medium- and high-pressure zones is collected separately and led to the bottom tray through fall chutes. The wash water obtained after cleaning of the top and bottom felts is also collected separately and led through a channel to a central collection tank. The machines have to be provided with distribution screws for the headbox in order to be able to utilize the whole working width of the gravity zone right from the beginning. The screw has to be fitted with screws running in opposite directions starting from the centre, so that the suspension fed in at the centre is distributed evenly up to the gravity zone. Medium-contact parts have to be made of stainless-steel material or other appropriate material recommended by the supplier so as to have a maximum lifetime.

Machine drive. The machine is driven via top and bottom felts by infinitely-variable DC gear motors, each with remote control from the switch board. The DC gear motors, which stand on a separate bracket, are directly coupled with the drive rolls. One of the two DC drives is designed as the leading drive and serves to adjust to a preselected speed. The second drive motor is wired electrically in such a way that independent of its speed, it takes on the same torque as the leading drive.

The supplier will recommend, according to the requirements the design of the equipment most suited for the job with all specifications for auxiliary machinery for instance including:

Belt conveyors (length, width, drive power, capacity)
Electro-mechanical conveyor-type weigher (for indication of the flow and the total weight of the material on the belt conveyor)
Vacuum pumps for suction tubes for cleaning the felts, including drive(s) for
Centrifugal pumps
Multi-stage centrifugal pumps (wear-resistant design)
Overhead travelling crane
Trolley (with chain hoist and built-in steel hand pull)
Pipes (suitable to fulfil the guarantee figure)

Valves and measuring devices (density measuring, radiometric density measuring device, measuring range 1.55-1.75 kg/m^3, corresponding to 57-67% weight of dry matter in the suspension)
Flow-metering of the suspension (inductive flow meter, measuring range 0-30 m^3/h
Level metering in the slurry tanks (ultra-sonic echo-sounding apparatus, measuring range 0-45 m)
Troughed belt conveyors (complete with belt-supporting frame, connecting chute, supporting feet, horizontal arrangement with rope switches, belt-alignment control)

D. Drying system

The filter cakes produced by the presses are expected to have a moisture content of around 20 per cent. These filter cakes need drying before they enter the kiln so that the final moisture content of the dried raw mix is lower than 1 per cent. The supplier may recommend one of the following types of dryer systems to suit his design and to guarantee production figures. The choice should be based on the findings from laboratory and pilot tests regarding physical properties of the material to be dried as well as on the supplier's experience from similar conversions. The supplier may take the following into consideration:

(a) Particle size;

(b) Tendency of the raw material components, especially clays, to change their structure during drying;

(c) Behaviour in relation to hot gases;

(d) Required drying "residence time";

(e) Type of moisture content in the raw materials, whether free moisture, capillary water or absorption moisture.

There are several types of dryers.

Drum dryers

Drying of the filter cakes is done by using the waste gases from the rotary kiln as well as the hot air from the grate clinker coolers. The combination of the rotary kiln, clinker cooler and drum dryer saves fuel costs but complicates the plant operation. It is safe and reliable and will save the cost of at least one dust-collector system.

The production of the drum dryers depends on many factors such as length, diameter, revolution of the drum per minute, residence time of material, angle of repose. These factors are expressed in the following formula:

$$t = \frac{1.77\ l\ \sqrt{\theta}\ F}{p\ d\ n}$$

where t = residence time of material (minutes)

l = length of dryer (metres)

p = slope of dryer (degrees)

d = diameter of dryer (metres)

n = revolutions per minute

θ = angle of repose of material in dry condition; the angle of repose for limestone and argillaceous materials is approximately 36° and $\sqrt{36}$ = 6 which is sufficient for practical purposes.

F = Corrective factor. Flights and lifters may decrease the passage through the cylinder by half. Thus resulting in F = 2.

Rapid dryers

Drying is also carried out by using the waste heat from the rotary kiln and from the grate clinker cooler. The gas inlet temperature to the dryer can be up to 600 °C. The rapid dryer allows the use of up to 2 m^3 gas/kg of dryer feed. The heat balance of the rapid dryer is:

	Percentage
Heat used for evaporation of moisture (useful work)	68.5
Heat loss in dryer-exit gases	9.5
Heat in dried material	11.3
Unaccounted for	10.3
Total	100

This shows that the heat efficiency of the rapid dryer is 68.5 per cent. This deals with the 20 per cent moisture in the filter cakes entering the dryer.

The energy requirement depends on the efficiency of the evaporation of the water content of the filter cake and it will vary between 40 kW and 100 kW for 2,200 kg and 1,000 kg respectively.

The dust content of the gas is from 50-250 g/m^3, with a maximum particle size of 0.2 mm. The proportion of dust less than 0.01 mm (10 microns) is approximately 50-70 per cent. For this reason, dust collection by using a cyclone is recommended.

Impact dryers

Here, wet filter cakes (20% moisture content) will pass from the round bunkers used as filter-cake stores to the impact crusher where they will pass between two impact cylinders. While the material is passing through the impact pressing, it will be affected by heat recycled from the rotary kiln as well as hot air from the grate clinker cooler.

Some of the ground dried material will be carried out of the impact dryer in the dryer exit gas. Because of the high dust content of the exit gas of the impact dryer, preliminary dust collection in cyclones is absolutely necessary. Then the partially-cleaned exit gas can be purified either in fabric filters or through the main electrostatic precipitators.

Flash-dryer tube (an alternative to be considered by the supplier)

Consisting of:

The connecting branch is made out of mild steel and is arranged between the kiln housing and the dryer tube, with connecting flanges and compensator. The branch is designed to have a lining of refractory material. The dryer tube is made out of mild steel, with the required expansion joints connecting flanges and supporting brackets. The top bend is made out of mild steel, arranged at the end of the dryer-tube duct, with the required reinforcing bars and connecting flanges, designed for easy dismantling. Frames are made of specially hardened steel, the sides are reinforced and replaceable wear plates are provided on faces subject to abrasion.

The connecting elbow between top bend and cyclone separator is made out of mild steel. The final moisture content of the raw meal leaving the flash dryer's connecting elbow to the cyclone separator has to be less than 1% H_2O.

(The design, material and lining to be recommended by the supplier.)

Round bunkers for storing filter cakes. These round bunkers serve as buffer stores between the filter-cake press and the dryers. These bunkers have to be supplied with a circumferentially-running screw conveyor. A single-roll belt weigher, situated in the cake conveyor belt can control all the bunkers serving as filter cake stores (or otherwise as recommended and specified by the supplier). Such a system could establish a withdrawal rate at a constant weight to supply the dryer. The speed of the belt conveyor has to be controlled from the weigher. The design, number, capacity and dimensions of the round bunkers and accessories or alternative equipment is to be worked out according to the recommendations of the supplier.

Feeding device. The device for feeding material into the flash-dryer tube will consist of the tube shaft with shaft bearings, the welded-on beaters, bottom plate and housing with connecting branch to the dryer tube. The device is driven by a V-belt drive with gearing. The design, capacity and dimensions of the feeding device as well as the motor power have to be recommended according to the supplier's design and the results of the tests.

E. Cyclone separators

Cyclone separators are made out of mild steel and consist of the cyclones, the raw-gas inlet branch, the material outlet branch with compensator (including supporting brackets and connecting flanges) and clean-gas branch with flanged connection and dust hopper. The cyclone diameter is approximately 3.0 metres or as recommended by the supplier.

Rotary gate valves

These are designed as an air seal underneath the dust hopper of the cyclones of the separator, with outside bearings, designed for direct drive, including the flanges, bolts and all accessories. Couplings and sectional steel bars for fixing the rotary valves and the drives as well as guards have to be provided. The capacity, valve diameter and the motor power is to be recommended by the supplier according to his calculations and design.

Intermediate bin

The bin will act as a buffer store for the material coming from the cyclone separator before going on to the preheater. The bin is set in a heavy steel construction complete with feed socket, three discharge spouts, vent socket, bin-inspection door and brackets. It has a flat, aerated bottom with a special synthetic fabric support, air socket and throttle valves.

The intermediate bin should also be provided with a roots-type blower with free-shaft extension, base frame for blower and motor, side rails for motor, anti-vibration pads including motor bolts, intake silencer, air-intake filter with weather hood, pressure silencer with attached valve and bracing, non-return valve, flanged pipe with rubber sleeve and clips, with V-belt drive and guard. The capacity, pressure and motor power is to be recommended by the supplier according to his design.

Shut-off gates

These are used for an emergency shut-off. They are activated by a pneumatic cylinder-switch system. Manual operation is normally possible in case of systems failure. Compressed-air with 6-bar pressure is needed for the pneumatic system.

Flow-control gates

This should have an activating device and a switch-system with stop and start contact switches, as well as potentio-meter and flow-control gate suitable for infinitely variable quantity control with the necessary outlet and inlet spouts or an alternative solution according to the supplier's recommendations. From behind the flow-control gates, an air slide may be installed (for raw-meal return) and equipped with the necessary high-pressure fan with air filter and control valve.

Further details are described by the expert as follows:

The upper slide casing is a steel-plate construction with the necessary inspection and cleaning holds. The lower slide casing has an air socket, connecting flanges, slide-feeding sockets, discharge housing, air-piping between slide and ventilation fan. The slide fabric is made of polyester suitable for a temperature of 150 °C. There is a sampler. The capacity, pressure, torque motor and range of material flow will be recommended by the supplier. The flow-control gate will be provided with the necessary measuring instruments and devices, such as an impact-type flow meter etc.

Rotary-gate valve

This includes a scraper for keeping the sections clean. The housing is made of cast iron. The rotary valve is of welded design, with shaft bearing, connecting girth gears between rotary valve and scraper shaft, drive gearing, valve housing with flanged connection at both sides as an air lock. The rotary-gate valve will be attached to the connecting chute and belt-type weigh feeder for raw meal to supply the preheater with the required quantity of raw mix.

The belt-type weigh feeder should have the following characteristics:

(a) Conveyor frame, welded side parts with cross-ties and bracings, drive pulley/return pulley with bolted, flanged shafts, convexly turned, running-in pillow blocks, adjustment of the return pulley via two thrust-bearing supports with pressure spindle;

(b) Carrying idlers/weighing idlers, machined with continuous shafts and maintenance-free bearings. The idlers are mounted on their supports allowing adjustment of height and distance. The measuring unit is composed of:

(i) Maintenance-free supported weigh arm;
(ii) Weighing idler;
(iii) Strain-gauge load cell with overload protection;
(iv) Test weight to simulate the belt load;

(c) The design of the measuring unit avoids zero-point variations. Strain-gauge load cell with fitting parts:

(i) Sensitivity 2 mV/V supply;
(ii) Compensated temperature range - 10 °C...... + 70 °C, protection IP 67 h 7;

(d) Belt drive composed of directly mounted gear-box on which the torque converter support its flanges which again is supported by means of a rubber and metal connection on the side of the feeder frame. The DC shunt motor, linked to the torque support, drives the gear box via a flexible coupling;

(e) The complete drive is easily dismounted. There is a digital tachometer for the contactless pick-up of the belt speed. The unit is mounted on the second shaft of the drive motor and it is maintenance-free and wear-resisting;

(f) Pulley scrapers for the drive and return pulley, guided parallel to the two pulleys. Outer belt scraper, mounted on two bearings, spring-loaded with adjustable force, with easily-exchangeable scraper blades. Inner belt scraper, plough-shaped, mounted on two bearings, weight-loaded with cover pulleys, with easily-exchangeable scraper blades. Belt tracking, monitored by two adjustable limit switches mounted on both sides of the belt;

(g) Continuous belt, suited for weighing process. The belt can be exchanged in a very short time without dismounting the pulley idlers, measuring unit or feed device. The lateral support opposite the drive is designed to accommodate the relief bar. The support or suspension opposite the drive can be released after propping up the relief bar. After removal of the support and shifting the return pulley, the belt can be removed or fitted. A belt-tensioning device with a parallel guided weight-loaded idler provides a constant belt tension. A mechanical indicator facilitates checking the belt tension;

(h) Protection covers mounted at the discharge side and on the return pulley as protection against dust and accidental contact;

(i) The cover on the discharge side is to be supported on the guide chute or similar. Chutes of low weight can be fitted on the weigh feeder together with the protection cover. All electrical components of the weigh feeder are completely wired. The cables have to be connected to a junction box;

(j) The height of the material layer on the belt can be adjusted by a vertical gate. A manually-operated isolating gate is provided to interrupt the material flow, e.g. for belt change.

The connecting chute, flow-control gate, air slides with high-pressure fan, the rotary-feed valve and set of pipe connections, their capacities, width, length, range, accuracy and motor power have to be recommended by the supplier in accordance with the production guarantee figures.

The bag filters

These have to be designed and supplied as suction filters in an enclosed construction and are composed of the following:

(a) Housing with inspection door on top, mounted on bins;

(b) Filter insulation with mineral-wool mats on steel wire-netting base frame;

(c) Filter bags of fine-fibre polyester needle felt;

(d) Bag suspension removable from above;

(e) Piping and service kit;

(f) Control valves, one for each row of bags;

(g) Sectional steel supporting frame, including foundation bolts;

(h) Back-protected ladder;

(i) Air fan, sucking from one side, with mounted brackets for the drive unit and statically and dynamically balanced impeller;

(j) Switchboard to control the bag-cleaning system;

(k) One set of piping and stack.

The bag filters are connected to the compressed-air system with magnet valves, 6-bar pressure.

		(approximately)
Technical data:	Air volume	20,000 m^3/h
	Air pressure	200 mm W6
	Motor fan power	30 kW

F. Preheaters

Dimensioning of the preheater stages I and II should be calculated, designed and manufactured by the supplier according to the general rules covering this part. The preheaters as well as the gas ducts will be lined with refractory bricks and castable. The pipes for conveying raw-meal from stage I to stage II and then to the rotary kiln will be lined with castable (the minimum possible). The pressure drop of the raw-mix suspension preheater cyclone will be kept within the range of 550-600 mm W6, or as recommended by the supplier.

The dust load in the exit gases must not exceed 100 g/kg clinker i.e. less than 10 per cent of the clinker weight, and, if the clinker weight is 50-65 g/kg raw meal, then must not exceed 6 per cent of the weight of the raw meal. The exit-gas volume of the raw-meal suspension preheater will be within the range of 1.4-1.5 m^3/kg clinker.

The content of the exit gases of the two-stage preheater plus the hot air produced from the grate cooler will be returned for drying the slurry cakes, which have a water content not exceeding 22 per cent. With an exit-gas temperature of 330 °C, it is possible to dry the total raw meal, with up to 8.5 per cent moisture, fed into the preheater kiln. In this case, according to the preliminary calculations, it is expected that the exit-gas temperature will be 460 °C and that air coming out of the cooler will be 350 °C.

Therefore, the temperature of these gases can be used to dry out the water content of the filter cakes. During the retention time in the two stages, the raw mix will be preheated from 90 °C to 660 °C, whereas the ascending kiln gases will be cooled from 931 °C to 460 °C. The gas and material velocity in the gas duct will be in the range of 21-23 m/sec.

<u>Waste-gas system</u>

<u>Waste-gas ducts</u>. The ducts will connect the cyclone separators and electrostatic precipitator with expansion joints.

<u>Electrostatic precipitator</u>. This is to clean the waste gases from the rotary kiln and dryer or flash dryer. The present electrostatic precipitator has a capacity of 400,000 m^3/h at 250 °C. The waste-gas volume to be cleaned is 350,000 m^3/h at a maximum temperature of 200 °C.

<u>Suction pressure</u>. The supplier is requested to design the discharge screw conveyors, the rotary-feed valves, the collecting screw conveyors, pneumatic-transport surge hopper, pneumatic pump, piping and waste gas ducts and to specify the number, material and dimensions of these according to the requirements of the production lines which are being converted.

<u>Waste-gas fan</u>. The waste-gas fan will be the radial-flow type, sucking on both sides, and will be composed of a multi-sectional, welded spiral casing, base frame with base plate, statically and dynamically balanced impeller with steel shaft, supported on pedestal bearings at both ends of the casing, with wearing-plate lining to exposed points of the casing and elastic coupling between fan and motor. In fact, there will be no need to change the waste-gas fan. The following table gives a comparison between the fan required and the present one and from the figures it is clear that the capacity of the existing one will be sufficient for the expected requirements, but this has to be rechecked and recalculated by the supplier.

<u>Technical data</u>:

	Approximate capacity required	Present capacity
Waste-gas rate (m^3/h)	350,000	400,000 (rated) 300,000 (actual)
Statistical pressure (mm WG)	450	150
Gas temperature (°C)	150	200
Maximum rated temperature (°C)	200	250
Motor power (kW)	650	660

G. Modification of kilns

<u>Rotary kilns</u>

There are four wet-process kilns with a diameter of 4 metres, and a length of 150 metres, with grate coolers. The conversion anticipated and described start on two kilns, nos. 5 and 6. The two kilns will be shortened to approximately 140 metres in length. The modification will also require inlet housing to the rotary kiln, a welded-steel plate construction with the necessary sectional steel stiffenings composed of:

(a) Steel-plate housing with access door;

(b) Connecting flange for the kiln inlet seal between the stationary housing and the rotary kiln;

(c) The necessary assembly sockets for the measuring and control devices;

(d) Connecting flanges to the rotary feed valve.

Inlet chute

This is made of heat-resistant steel with a split construction.

Kiln-inlet seals

These are the segmental type with mechanical pressure holding and have:

(a) A stationary part at the kiln-feed housing;

(b) Split segments;

(c) Split retainer including asbestos packing;

(d) Split sealing ring to the feed housing;

(e) Adjustable retainer for the seal at the feed housing;

(f) Any other necessary modifications suggested by the supplier.

H. Modification and conversion of the grate coolers

The grate coolers of the 8th February Cement Joint Complex, as already mentioned, need to be modified and converted to fit in with the conversion of the rest of the two production lines to a semi-wet process (increased production). The modifications will be based on the following considerations.

Air chambers

Air chambers, divided into compartments, have to be provided below the grate according to the design and calculations of the supplier. These chambers have to be installed in the lower part of the existing cooler as a sturdy steel structure to support the moving grate frames. A new running axle at the front grate for supporting the moving frame with external bearings, slide rails and wheel flanges is necessary. The grate plates have to be made in such a way that one row is moving and the other is stationary and have to be made of heat- and wear-resistant steel. The moving grate frames have to be driven on both sides via the present drive axle, by crank drives mounted on both sides. The grate plates in the recuperation zone have to be made out of highly heat- and wear-resistant special cast chromium-nickel steel alloy. The grate plates in the after-cooling zone are made of heat-resistant cast chrome steel. The support brackets and top brackets have to be made of varying special cast steel alloys depending on the temperatures to which they are subjected. The speed of the grates, drives, shunt motors have to be calculated by the supplier (approximately 4-22 strokes/min).

The grate riddlings

These have to be collected in hoppers located underneath the cooler and have to be extracted when a certain amount has accumulated. The hoppers have to be provided with probes to determine the level of riddlings in the hopper. The probes are controlled by timers. This prevents the removal units from overflowing and at the same time ensures that the column of the material is always available to act as a sealing column against the chamber pressure in the hopper. Acoustic and optical and local switches have to be provided to actuate the gates.

The drag chain

This is located underneath the cooler to remove the grate riddlings. It consists of a drive sprocket and tail wheel with shaft and bearings, chain links, sliding pieces, idler stations, roller chain drive and drive. It is laid in a concrete trough (not encapsulated). The drag chain has to be equipped with a central grease lubricator system consisting of a grease pump (with integrated gear and drive motor, including the required pipes), distributors, control and monitoring system, complete air seal of the running axle bearing consisting of compressed-air generator (including drive motor), the required connecting pipes mountings and throttle elements.

Cooling air fans

These are required for the air chambers. They have an air intake on one side and must be of a heavy, welded construction suitable for industrial operation. They are designed to be connected on the pressure side directly to the inlet ducts. The high-efficiency impeller with backward-inclined blades has to be dynamically balanced and equipped with housing (for removal of the impeller). This is subdivided with a cleaning opening and bearing trestle to take the drive motor. The impeller runs on roller bearings in the pedestal bearing housing. The fans have to be equipped with an air-flow control nozzle with piezometer and inlet dampers. Square or rectangular inlet ducts, with the required bracing and flange connections and any necessary expansion joints, have to be provided between the air fans and the cooler housing.

Clinker crusher

This consists of breaker housing, breaker rotor, drive motor etc.

Measuring and control system

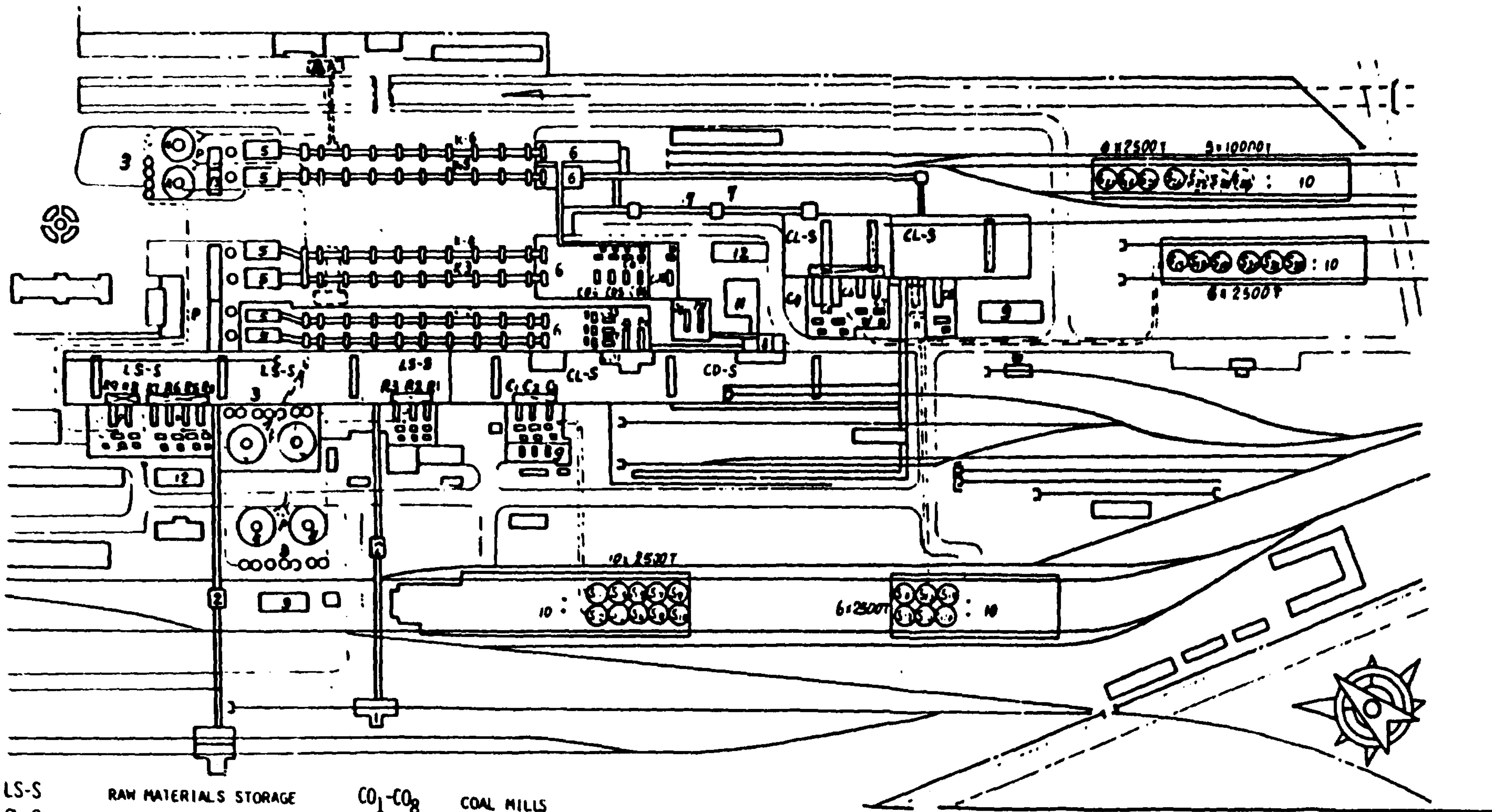
This gives automatic measuring of the differential pressure from the inlet nozzle of the fan and controls the flow of air (i.e. air-volume control per chamber and measurement of grate-plate temperature).

I. Design proposal for the conversion of the existing production lines to semi-wet production lines

After all the information had been assembled on the production capacity of the present equipment and machinery in the 8th February Cement Joint Complex and all the calculations made as to the possibility of converting the existing wet-process production lines to semi-wet production lines, a design proposal was drawn up. The whole layout of the plant is shown in figure 4 and the proposed new installations are shown in figures 5a and 5b. This design proposal was finally arrived at after many trials and testing out various alternatives and after many visits to the site where this conversion should take place.

It has to be said that the plant is already overcrowded with machinery and it was impossible to find an area where the new equipment could be installed in a continuous series with the kilns, especially in the case of kilns 5 and 6. It was necessary, therefore, to choose an area, as indicated in the layout in figure 5a, on the other side of the channel (north-east side of kiln 6). For kilns 3 and 4, as can be seen in the layout, a suitable area exists for the necessary installation.

Figure 4. Layout of the cement plant

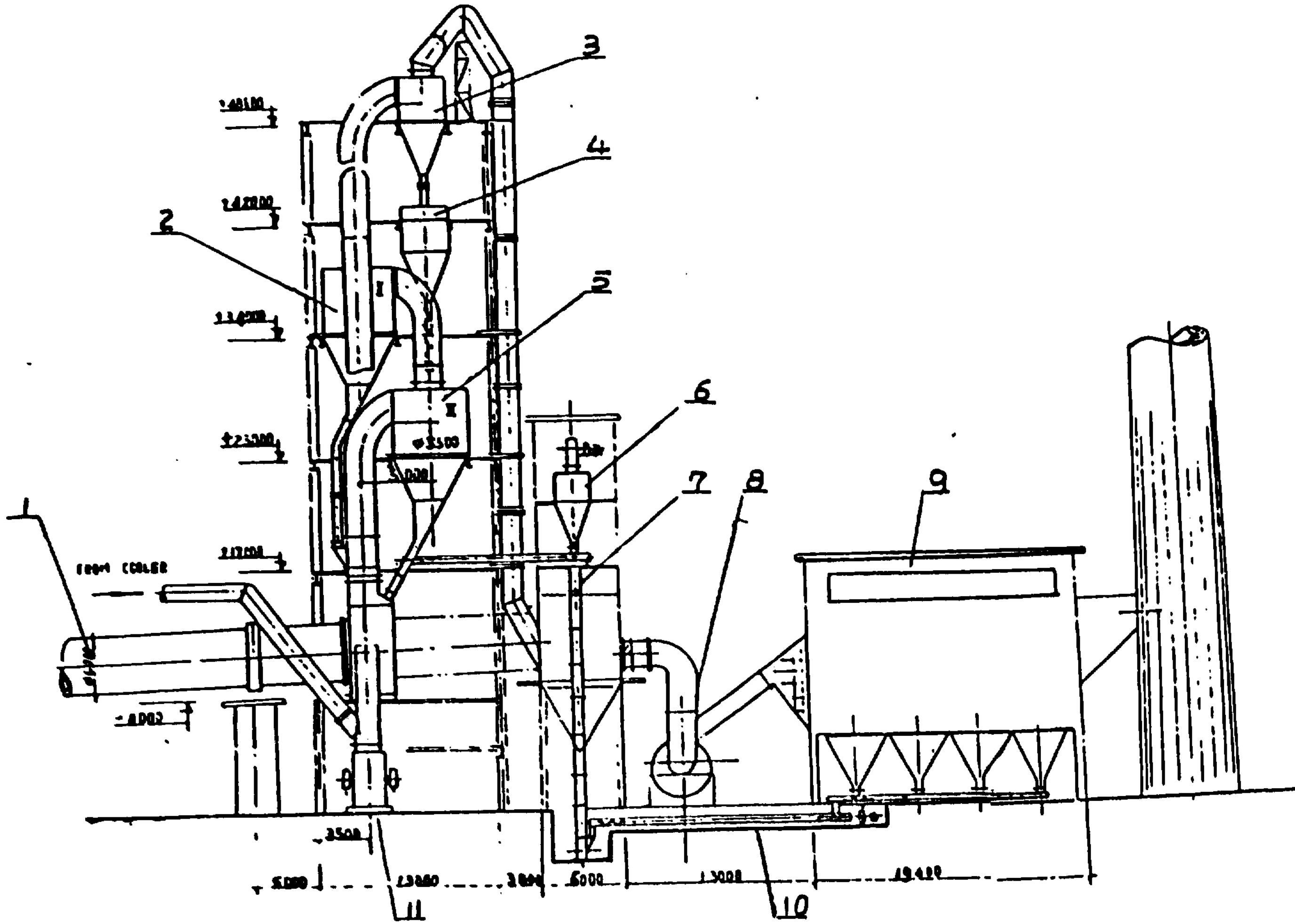


Key:

LS-S	RAW MATERIALS STORAGE	CO_1-CO_8	COAL MILLS	4	SLURRY BASINS	11	BOILERS
CL-S	CLINKER STORAGE	D_1 - D_4	COAL DRYERS	5	ELECTROSTATIC PRECIPITATOR	12	POWER STATION
CO-S	COAL STORAGE	S_1 - S_{29}	CEMENT SILOS	6	CLINKER COOLER	AA	NEW MACHINERY (FILTER PRESS & ACCESSORIES)
K_1-K_6	ROTARY KILNS	1	JAW CRUSHER	7	CLINKER CONVEYOR		
R_1-R_9	RAW MILLS	2	HAMMER CRUSHER	8	COAL HAMMER CRUSHER		
C_1-C_8	CEMENT MILLS	3	SLURRY SILOS	9	COMPRESSOR HOUSE		
				10	PACKING MACHINES		

Figure 5. Design proposal for converting production lines 5 and 6 from wet to semi-wet process by the addition of filter presses, dryers and two-stage preheaters

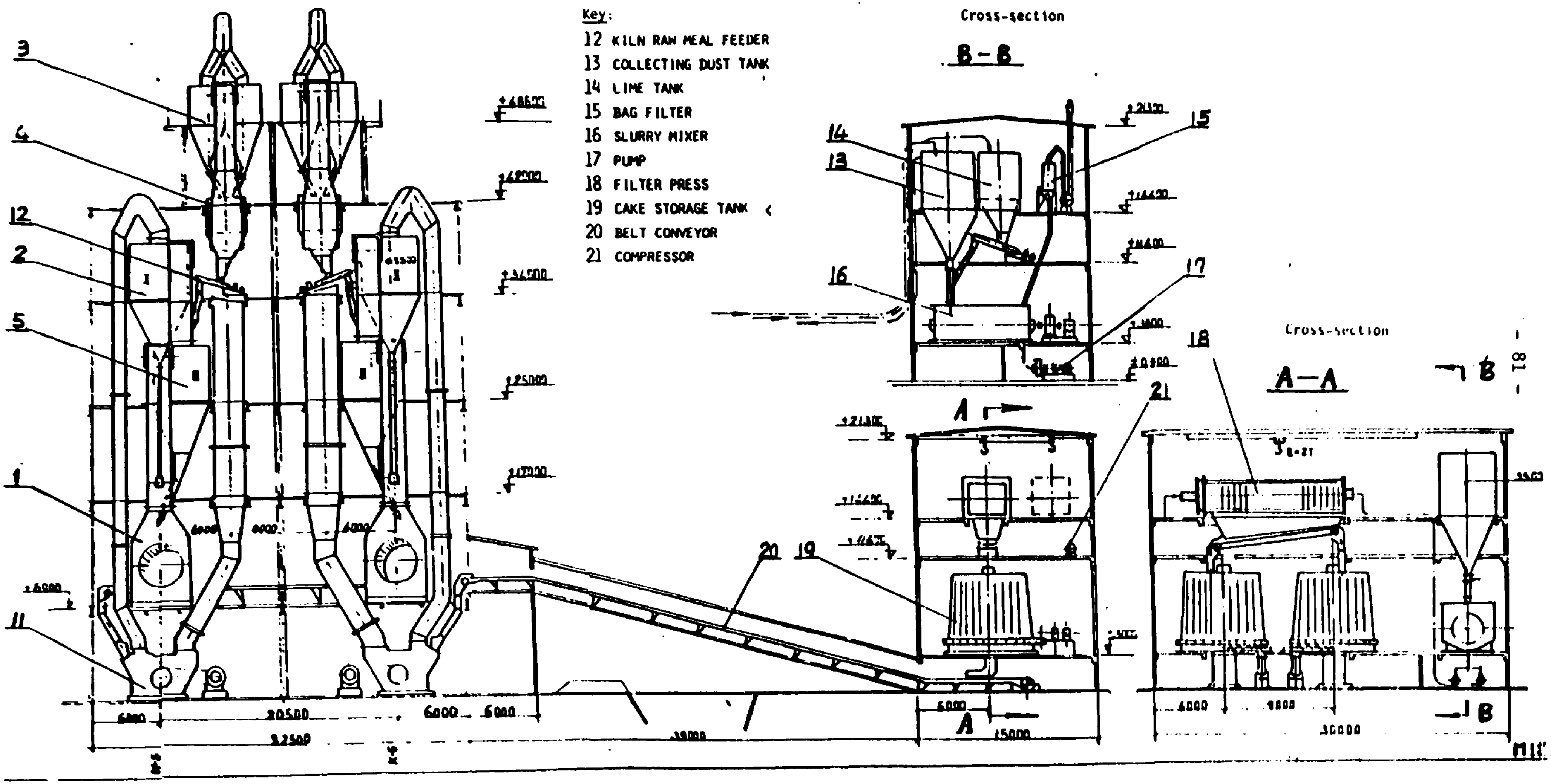
(a) Layout of the drying and firing sections



Key:

1. ROTARY KILN
2. FIRST-STAGE PREHEATER
3. CYCLONE
4. RAW MEAL SILO
5. SECOND-STAGE PREHEATER
6. COLLECTING DUST TANK
7. BUCKET ELEVATOR
8. MAIN EXIT GAS FAN
9. ELECTROSTATIC PRECIPITATOR
10. SCREW CONVEYOR
11. DRYER

(b) Layout of whole process showing the raw-meal preparation and filtration sections



Part Two

SOME DIFFICULTIES AND PROBLEMS AFFECTING CEMENT PRODUCTION AT VARIOUS CEMENT PLANTS

Introduction

A part of the Adviser's task was to evaluate conditions in the cement industry in general and to make recommendations for dealing with any production problems. The examples of problems investigated given in this part are taken mainly from the 8th February Cement Joint Complex and the Chon Nae Ri cement plant. This is only a selection of the problems dealt with, full discussions having taken place with all the technical counterparts at the time of the mission.

VI. PROBLEMS ENCOUNTERED AT THE 8TH FEBRUARY CEMENT JOINT COMPLEX

A. Grinding media and internal fittings

Comprehensive technical and production data were supplied by the 8th February Cement Joint Complex (given in annex II) and, from this data, it is clear that the consumption of the grinding media and lining plates of the mill per tonne of clinker is very high (1.8 kg/t clinker).

The Adviser tested the grinding media used in the raw mills and in the cement mills and observed the following:

(a) The grinding media as well as the liners are manufactured from normal cast iron;

(b) The casting is not done well and, therefore, the shape of the balls and of the cylpebs is irregular and deviates from the required roundness of the balls and the cylindrical shape of the cylpebs;

(c) No special care or attention is given to the size of the grinding media in the mills.

Discussion and comment

Most of the cement plants in developing countries started off using the highly wear-resistant grinding media made of white cast iron with a chromium content of between 12 and 18 per cent. Despite their substantially higher price, they are much more economical for cement plants than the cast iron grinding media mentioned above. The greater economy is not just due to the lower wastage by wear. Because of the slower and more uniform wear of the balls and cylpebs, there is also a slower decline in the mill performance and slower change in ball grading.

Research has shown that, for grinding media with 12-13 per cent chromium content and a Rockwell hardness value of 55-65 (Brinell 555 to about 660), the measurement revealed wear rates of 17 grams per tonne for shaft furnace cement and 45 grams per tonne for slag cement. In the case of normal Portland cement, wear rates were found to be intermediate between these two.

The mill lining must also be changed at the same time as the grinding media and the material used must be harder in order to resist the abrasive action of both the new grinding media and the raw mix and clinker processed.

It is also advisable to use liner plates which have two different angles of slope. This type of plate will help to develop the quality of the material to be ground, i.e. it will improve the grain-size distribution of the material.

Recommendations

1. The diaphragms (intermediate plates) have to be changed and it is recommended to install the double-walled intermediate diaphragm.

2. It is recommended to examine without delay the economic benefits to be derived from introducing the above-mentioned improvements in mill lining and grinding media.

B. Refractory bricks

Rate of consumption

It is clear from the figures for refractory bricks, as shown in annex II, that the consumption of refractory bricks per tonne of clinker is abnormally high. The figure for this plant is 4.6 kilograms per tonne of clinker as compared with a normal rate for a wet-process kiln of 1.1 kilogram per tonne of clinker.

The adviser attributes this high consumption to the following factors:

(a) The operational life of the rotary kilns depends mainly on the quality of the refractory bricks used in lining the different zones of the kiln. The bricks used here can be considered to be of low quality;

(b) The shape and position of the flame inside the burning zone was found to be in need of adjustment;

(c) The flame was such as to adversely affect the quality of clinker produced, as well as reducing the melting points of the raw mix and of the surrounding refractory bricks;

(d) Continuous thermal shocks due to abnormal stoppages of the plant also affect the refractory bricks;

(e) The kiln shell, especially in the burning zones, was found to be deformed, which leads to the instability of the refractory bricks and, in turn, their falling leads to red spots and ovality in the kiln shell.

NB: Also the correct alignment of the kiln is of importance for keeping the consumption of refractories at a reasonable level.

Discussion and comment

As a matter of fact, the linings of rotary-kiln plants deserve particular attention. The effective production time of any rotary kiln and, therefore, the degree of capacity utilization attained, depends first and foremost on the durability of the refractory lining.

The rate of consumption of refractories is affected by the following factors:

Mechanical strength
Refractoriness
Resistance to temperature changes (thermal shocks)
Resistance to chemical attack
Thermal expansion or stability of volume
Thermal conductivity
Resistance to abrasion
Porosity
Burning process and type of lining

The choice of the right refractory bricks is no simple matter. decision should be based on practical tests and observations and res during and after lining, should be carefully recorded. It is usually ought that the presence of calcium fluoride seriously increases the rate of deterioration of the refractory lining in the kiln, but it has been claimed (P. F. Konovalev and F. R. Skue, Tsement 14 (5), 14 [1948]; 18 (3), 14 [1952]) that with an addition of 0.5 per cent calcium fluoride in the dry mix, a stable coating is formed, though this is not the case with appreciably higher or lower contents. In this plant's raw mix, about 0.88 per cent of calcium fluoride is added, which approaches the amount likely to have a deteriorating effect on the refractory lining, besides causing difficulties with the setting time, increased dust in the clinker, and a reduction in the early setting strength.

Recommendations

1. It is recommended to test the addition of different percentages of calcium fluoride to the raw mix in order to investigate the effect of each percentage on the deterioration of the refractory lining, the setting time, the strength of the cement and the dustiness of the clinker. After careful testing, the proper percentage to add can be selected or it may be decided not to add any amount to the raw mix to avoid further problems in the future. (Additional background information on this topic is given below.)

2. It is necessary to select the proper quality of refractory bricks for each of the kiln zones and especially the bricks to be installed in the burning zone. These bricks must be 80 per cent magnesite or magnesite chrome and must be installed with special care, especially in the deformed part of the kiln shell. The kiln itself should also be correctly aligned in order to avoid excessive mechanical impact on the lining. The kiln may in such a case chew the (refractory) lining in exposed positions.

3. It is very necessary to adjust the position and direction of the flame inside the kiln burning zone as described in detail to the authorized technical counterparts at the 8th February Cement Joint Complex. (This is also given below.)

4. It is recommended to increase the amount of oxygen necessary for burning the raw mix to clinker in an oxidizing atmosphere. The quality of primary air can be increased to some extent.

5. It is advisable to try to reduce the number of kiln stoppages to the minimum.

6. It is advisable to change, if possible, the [illegible]ly deformed sections of the kiln shell in the burning zone.

Adjustment of the flame position and direction in the burning zone

Most rotary kilns are equipped with some controls for adjusting the position and the direction of the flame and, to some extent, changing its shape. Any alteration in the primary air pipe or the burner position will be reflected in the flame characteristics and accordingly will affect conditions throughout the burning zone.

Because of the many variables influencing the flame direction, it is necessary to determine the position of the primary air pipe inside the burning zone according to the actual conditions in each individual kiln. The flame characteristics can vary considerably from one cement plant to another, sometimes even from one kiln to another in the same plant. The reason for this is that a flame must always be tailored to the actual kiln design and prevailing operating conditions. Clinker quality, refractoriness [illegible]sence of rings, and kiln-equipment problems all influence the operator to try and obtain a certain flame that best fits the actual conditions in the particular kiln under consideration.

As a general rule in flame control, the aim is to reach the shortest possible fire and the highest possible flame temperature without adversely influencing the clinker quality, formation of coating and rings, and refractory life, or causing damage to the kiln equipment in the discharge area. Furthermore, the flame must not cause overheating in the burner hood, kiln-discharge end, or cooler. Once the ideal flame characteristics have been obtained, the operator should make every effort to operate the kiln in such a fashion as to cause a minimum of disturbance to the flame. A flame should not willfully be changed during the course of kiln operation, unless specific conditions necessitate a change. The operator should keep the flame in the burning zone under observation and pay particular attention to the position of the flame body at a point about seven meters away from the beginning of the primary air pipe (air-pipe nozzle). It is generally assumed that the best heat exchange between the flame and th[illegible] takes place when the flame is pointed slightly towards the feed bed. [illegible] can be achieved when the position of the primary air pipe nozzle is shifted slightly away from the axial centre towards the corner (nearer to the axial centre) of the quadrant where the feed bed is moving continuously up during the rotation of the kiln. The outer part of the flame must barely touch the feed bed and not be in any direct contact with it. It should also be noted that, if the flame is directed too close to the feed bed (i.e. the position of the primary air nozzle is too far from the apex of the corner and nearer to the feed bed) there is a danger that part of the unburned fuel (especially coal and oil) could enter the feed bed, a condition that is highly undesirable. However, a flame directed too much towards the kiln wall (i.e. away from the axial centre on the opposite side of the quadrant containing the feed bed) could result in flame impingement upon the coating, which will shorten the life of the refractory lining.

The following rules can be applied to all flames, regardless of what type of rotary kiln is under consideration:

(a) When the primary air pipe nozzle has accidentally been warped, resulting in an erratic flame shape and direction, immediate steps should be taken to repair this condition;

(b) A flame should never be allowed to impinge upon the coating or bare refractory bricks for a prolonged length of time;

(c) A flame should never be allowed to strike too hard upon the feed bed;

(d) Flame direction should be adjusted only when the kiln is in a stable operating condition and the temperature, fuel-pressure, and air-flow rates are at a normal level. Flame direction changes can be caused by unusual operating conditions. If an attempt were made to adjust the flame direction at such times, there would most likely be an unsatisfactory flame once the kiln returns to normal operating conditions;

(e) It is better to carry out the desired adjustments in the flame direction in several smaller steps instead of in one large one in order that the operating stability of the kiln is not adversely affected;

(f) Once the ideal flame direction has been obtained, the primary air pipe position should not be changed unless there is a definite reason, such as to combat ring formation or hot shell conditions;

(g) To protect the primary air pipe from possible damage during a shut-down, a certain amount of the primary air flow must be maintained until the temperature inside the kiln is low enough (approximately 300 °C) so that the pipe is not liable to damage. If the primary air fan stops because of a power failure, the primary air pipe must be immediately removed from the burner hood.

The Adviser explained to the technical personnel of both the 8th February Cement Joint Complex and Chon Nae Ri Cement Plant how to adjust the primary air pipe before starting burning, as follows:

(a) First establish the centre line of the kiln, especially in the area where the primary air pipe has to be located;

(b) Install a flash or light at the centre line of the primary air pipe;

(c) Shift the position of the primary air pipe nozzle from the centre line of the kiln burning zone somewhat towards the corner of the quadrant where the feed bed is moving up;

(d) Direct the light beams so that they fall on the centre of the feed bed at a distance of about 60-70 meters;

(e) According to the observations made during kiln operation, this distance can be increased or reduced in order to achieve the best heat exchange in the burning zone. This change in distance can be carried out when the kiln is stopped.

Adjustment of the percentage of the mineralizer (calcium fluoride) to be added to the raw meal

It is worth giving some special background information about the use of mineralizers as a possible means of saving energy in cement manufacture. A mineralizer is a substance that accelerates the reaction rates at all or some of the stages of clinkerization. Thus, mineralizers can act both as a flux and as a catalyst (flux is an additive that decreases the melting point of the liquid phase formed in any clinkerization process). The effects of mineralizers can be stated as follows:

Accelerating the decarbonization and sintering reactions
Lowering the clinker temperature
Broadening or narrowing the sintering temperature range
Modification of liquid properties, such as viscosity, surface tension of clinker etc.
Promoting the crystallization of the liquid phase
Promoting clinker balling and ring formation
Promoting clinker refractory interactions
Altering the overall burnability and volatility conditions inside the kiln

Mineralizers like calcium fluoride (CaF_2) are used in the cement industry to save energy thermally and electrically:

(a) To reduce heat consumption (including kiln shell losses) by 60-120 kcal/kg clinker due to lowering the burning-zone temperature by about 100 °C;

(b) To give a corresponding energy economy by reducing the volume of exit gas needing to be handled and also causing some reduction in the quantity of air required for clinker cooling.

Comments

Apart from the obvious advantages in fuel economy, it is clear that in lowering the clinkering temperature by approximately 100 °C, the high percentage of liquid phase present in the clinker due to the presence of high percentages of C_3A and C_4AF will lead to some disadvantages (as described before) and that the percentage of CaF_2 has to be tested under very well-controlled conditions in order to select the proper proportion.

C. Cooling of clinker

What is meant by clinker cooling?

The process of clinker formation is not complete when the clinker has gone through the hottest area of the kiln's burning zone. The cooling of the clinker has to be seen as an integral part of clinker production as it affects the quality of the clinker produced. Once the clinker phases have been formed and the clinker has reached a temperature of about 1,370 °C, some of the compounds are in a liquid phase and others (about 75%) are in a solid state. The cooling of the clinker starts a few meters short of the discharge end of the kiln in what is considered to be still part of the burning zone. Depending on the location of the flame, cooling can be slow or fast. This depends also on the difference in the time the clinker remains in the kiln after it has been burned. The cooling time, that is, the time required until all liquid in the clinker has solidified, is important. It is generally true that rapid cooling is beneficial to the quality of the clinker, resulting in better grindability. After emerging from the kiln, the clinker passes through a cooler. On dis- charge from this, its temperature will not exceed 70 °C.

It is clear from the data received from the 8th February Cement Joint Complex that the clinker temperature leaving the grate coolers is around 400 °C for kilns 1 and 2, and around 300 °C for kilns 3, 4, 5 and 6. These temperatures are very high and will directly affect the quality of clinker produced.

Discussion and comment

This problem was thoroughly discussed with the technical managers and engineers and the government counterpart. It was agreed that:

(a) Clinker cooling has an effect on the crystallographic structure and the mineralogical content of the clinker and consequently influences the grindability and the quality of the cement produced;

(b) It is necessary to achieve the correct cooling of clinker because:

(i) Conveying and handling hot clinker is difficult;

(ii) Hot clinker has a negative effect on the performance of the cement mills;

(iii) The heat content and its loss will be reflected in the final cost of the cement produced;

(iv) If properly cooled, the clinker will improve the quality of the cement produced.

One aspect is that fast cooling prevents the growth of crystals and leads to the transformation of a part of the liquid phase to solidified glassy materials. It increases the soundness of the resulting cement, particularly in relation to the action of magnesium oxide (periclase crystals formed during clinker cooling). Magnesium oxide (MgO), solidified to glass, does not impair cement soundness and therefore rapid clinker cooling will allow for a higher MgO content in the clinker whereas slowly cooled clinker should have a rather low MgO content.

Slow clinker cooling encourages the growth of crystals of the clinker materials. The size of alite crystals not only influences the clinker grindability, it also affects energy consumption, since large crystals need additional energy for comminution, which is clearly shown in the production data in annex II. With proper burning and rapid cooling, alite crystals are kept small, which results in higher cement strength. Of two cements with identical chemical compositions, the one with smaller alite crystals (15 microns) was stronger, its 28-days compressive strength was 391 kg/cm^2, whereas the compressive strength of cement with 40 micron diameter crystals was only 293 kg/cm^2. Cooling also increases the sulphate resistance (i.e. resistance to sodium and magnesium sulphates) of the cement.

It is important, also, to point out that savings in heat consumption can be achieved only if as much high-temperature secondary air as possible is recovered from the heat of the clinker. With a properly-functioning grate-type cooler and kilns with a low heat consumption (suspension preheater) this temperature is probably of the order of 900 °C. Such temperatures are quite often arrived at in calculations for the heat balances of coolers. Even higher temperatures are conceivable as a result of using new coolers with higher efficiencies. In the case of this plant, as the figures in annex II show, the temperature of the secondary air recovered is 300-400 °C. Conversion of the grate cooler at the same time as converting the kilns from wet process to semi-wet process is one of the most important issues to be decided in planning the changes.

D. Mineralogical constituents of the clinker (chemical moduli)

As shown in annex II, the chemical moduli of the clinker produced in the 8th February Cement Joint Complex are as follows:

Lime-saturation factor (LSF)	0.90
Silica modulus (SM)	1.9
Alumina modulus (AM)	1.4

Discussion and comment

These moduli have great effect of the burnability of raw meal in kilns as well as on the grindability of clinker and the quality of cement produced. It was found that the SM has to be in the range 2.3-2.7 to produce clinker of good quality. Likewise, the AM has to be in the range 1.3-1.6 and the LSF in the range 0.92-0.96. There are some other components that also have a direct effect on the quality of clinker and its grindability and on the quality of cement produced. These include magnesia, sulphur, alkalis, chlorides and the liquid phase produced in clinker through burning.

Comparing the ranges desired for the lime-saturation factor and the silica and alumina moduli with the figures actually characterizing the clinker produced in the 8th February Cement Joint Complex, it is clear that the LSF is somewhat low and that the SM is very low and that the liquid phase in such clinker will increase leading to some problems in the burning zone.

The alumina and ferric oxide contents in a cement need to be considered together, since, although they are by no means equivalent to one another, their effects are closely interconnected. It must be said that cements with a high content of alumina and ferric oxide, such as the cements produced in the 8th February Cement Joint Complex (6.53% Al_2O_3 + 4.66% Fe_2O_3 = 11.19 total R_2O_3), are easily sintered and, unless carefully burnt, tend to cause ring formation in the kiln. In this respect, the two oxides act somewhat simmilarly, but in most other respects, they cannot be treated as similar. They form the compounds C_3A and C_4AF and the relative proportion of these two compounds depends on the ratio of alumina to ferric oxide present. An increase of alumina with no change, or with a reduction, in the ferric-oxide content, hastens the setting of a cement and a point is eventually reached at which it becomes impossible to control the setting time adequately. The substitution of ferric oxide for alumina, or an increase in ferric-oxide content reduces the proportion of C_3A and increases that of C_4AF in the cement. The latter has less rapid-setting properties. Thus, if the iron-oxide content is raised along with the alumina, an increased alumina and ferric-oxide content can be carried without setting troubles arising.

The formation of clinker takes place very rapidly in a kiln as the temperature rises from 1,100-1,200 °C to 1,300-1,350 °C. While clinker formation implies an increase in the liquid content sufficient to make the mix cohere into small balls (sintering), it does not necessarily mean that compound formation has proceeded with the same rapidity. Since clinker is dependent on liquid formation, the minimum temperature at which liquid forms in raw mixes is of importance. It was shown that liquid formation can start at minimum temperatures of 1,250-1,280 °C. While almost all cement raw mixes are likely to show this same minimum temperature of liquid formation, the amount of liquid formed at this temperature will vary a lot with the total proportions, and individual amounts, of the minor components such as ferric oxides, alumina, magnesia, alkalis. In normal clinker, the amount of liquid present at the sintering stage is 20-30 per cent. The sintering range will be a function not only of the amount of liquid formed but of the viscosity measured at various temperatures. The calcium fluoride used in the 8th February Cement Joint Complex also has some effect. If it is added in low concentrations (0.7%), it markedly decreases the viscosity of the clinker. If it is used in larger amounts, this substance can act as a mineralizer, promoting crystallization of the liquid. When used as a flux, therefore, the sintering temperature is lowered and the liquid is initially more fluid, but the rapid onset of crystallization stiffens the mass and the risk of formation of rings is much reduced.

The liquid expected to be formed at temperatures 1,340 °C, 1,400 °C and 1,450 °C at the 8th February Cement Joint Complex can be calculated from the following formulae:

Percentage of liquid phase expected to be formed at a temperature of 1,340 °C:

$= 6.1\ Fe_2O_3 + MgO + R_2O$ (alkalis)

$= 6.1 \times 4.66 + 1.72 + 0.59$

$= 28.426 + 1.72 + 0.59$

$= 30.736$

Percentage of liquid phase expected to be formed at a temperature of 1,400 °C:

$= 2.95\ Al_2O_3 + 2.2\ Fe_2O_3 + R_2O\ (alkalis) + MgO$

$= 2.95 \times 6.53 + 2.2 \times 4.66 + 0.59 + 1.72$

$= 19.26 + 10.252 + 0.59 + 1.72$

$= 31.822$

Percentage of liquid phase expected to be formed at a temperature of 1,450 °C:

$= 3.0\ Al_2O_3 + 2.2\ Fe_2O_3 + R_2O\ (alkalis) + MgO$

$= 3.0 \times 6.53 + 2.2 \times 4.66 + 0.59 + 1.72$

$= 19.59 + 10.252 + 0.59 + 1.72$

$= 32.152$

The weight ratio of alumina to iron oxide is $\frac{6.53}{4.66} = 1.4$

The first of the formulae above is used to determine the percentage of liquid phase at a temperature of 1,338 °C when the weight ratio of alumina to iron oxide exceeds 1.38. If this ratio is lower than 1.38, the following formula can be used:

$8.5\ Al_2O_3 - 5.22\ Fe_2O_3 + R_2O + MgO$

It was also clear from these investigations that the colour of both clinker and cement is not normal. The cement is black and greenish black in colour. The clinker produced is surrounded by a black shell and internally it contains a brown core or a yellowish colour. The dark brown colour of clinker is due to slow cooling of the clinker in a reducing atmosphere from immediately below the clinkering temperature. As a result, the colour of the cement also undergoes changes. This phenomenon can be attributed to the partial reduction of the iron oxide in the burning zone followed by a subsequent re-oxidation which has not penetrated beyond the surface. Over-burning can be observed, causing partial melting of the clinker below its normal melting point, which is assisted by the presence of a high percentage of liquid phase and the presence of calcium fluoride.

Recommendations

1. It is recommended to increase the silica modulus and alumina modulus to reach the ranges mentioned before. This can be done by increasing the amount of siliceous stone and decreasing the amount of iron oxide added. This will lead to a decrease in the amount of liquid phase in the clinker produced.

2. As suggested earlier, it is important to test the amount of calcium fluoride which should be added to the raw mix or to determine, after careful investigation, to avoid adding any.

3. The measurement of the oxygen concentration in the rotary-kiln exit gases is recommended. This is considered to be an index of the operational performance of the kiln from a fuel utilization stand-point. Regardless of

what type of fuel is being used, the oxygen content of exit gases is a reliable indicator of combustion efficiency, and its continuous measurement provides an indication of the efficiency of kiln operation.

E. Sulphate content of the cement produced

The data from the 8th February Cement Joint Complex shows that the sulphate (SO_3) content of the cement produced is between 1.5 and 1.68 per cent, which is considered to be low.

Discussion and comment

Gypsum ($CaSO_4.2H_2O$) is added in a limited quantity to Portland cement, mainly for the purpose of regulating its setting time. Gypsum prevents flash setting and keeps concrete from Portland cement workable for hours. The gypsum influences not only the setting time but also other cement properties such as grindability, sensitivity to storage, volume stability and strength. Systematic studies of the influence of gypsum content have shown that there is an optimum content which produces the highest strengths and the lowest drying shrinkage. This amount varies with the contents of alkalis and tricalcium aluminate (C_3A) in the cement. For very low contents of C_3A (below 0.5%), the optimum content of SO_3 is as low as 2%. As the alkali content rises to 1.0% or more, the optimum amount of SO_3 rises to 3-4%.

It is to be noticed that the chemical composition of the gypsum ore plays the basic role in the optimum gypsum content of cement. It also plays a great role in cement grinding. It is advisable to use a high proportion of gypsum in the clinker as this will result in lower power consumption in grinding.

The SO_3 content of cement also affects the compressive strength, shrinkage and expansion when stored under water of the resulting concrete and mortars.

Figures 6(a-c) show the variation in lowest optimum SO_3 content with C_3A and equivalent alkali contents (results obtained by Lerch).

From all this, the following recommendations can be formulated.

Recommendations

1. The percentage of gypsum to be added to the clinker has to be increased so that the percentage of SO_3 in the cement produced reaches at least 2.5%.

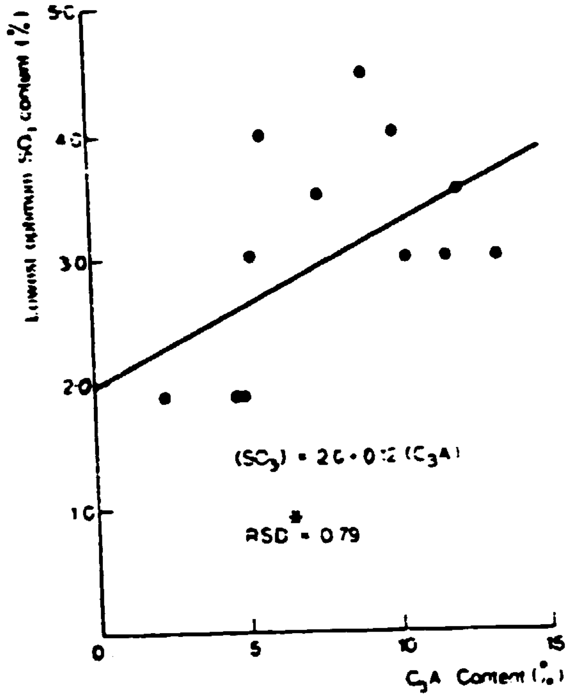
2. Although the raw gypsum used in the cement industry has to be imported either from China or from the Soviet Union, and although an increase in gypsum imports will increase the foreign-currency budget to be paid by the Government, it is, nevertheless, recommended to consider increasing gypsum imports so that sufficient is available to implement the first recommendation. One benefit would be that the cement so produced would comply with the international standard specification, an advantage for potential future cement exports.

F. Laboratory equipment

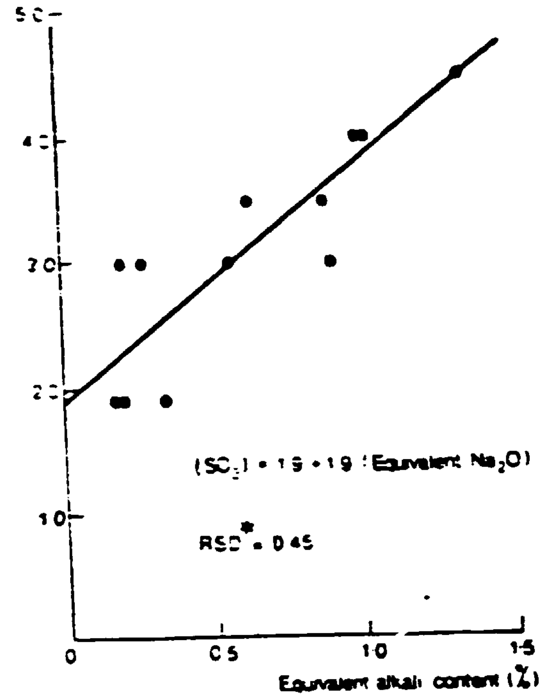
The laboratories of the 8th February Cement Joint Complex were also visited. All the equipment, machinery, instruments and apparatus are of an old-fashioned type for carrying out classical and routine quality-control tests. The laboratories (chemical, physical and quality-control), which deal

Figure 6. Lowest optimum SO_3 content in relation to other factors

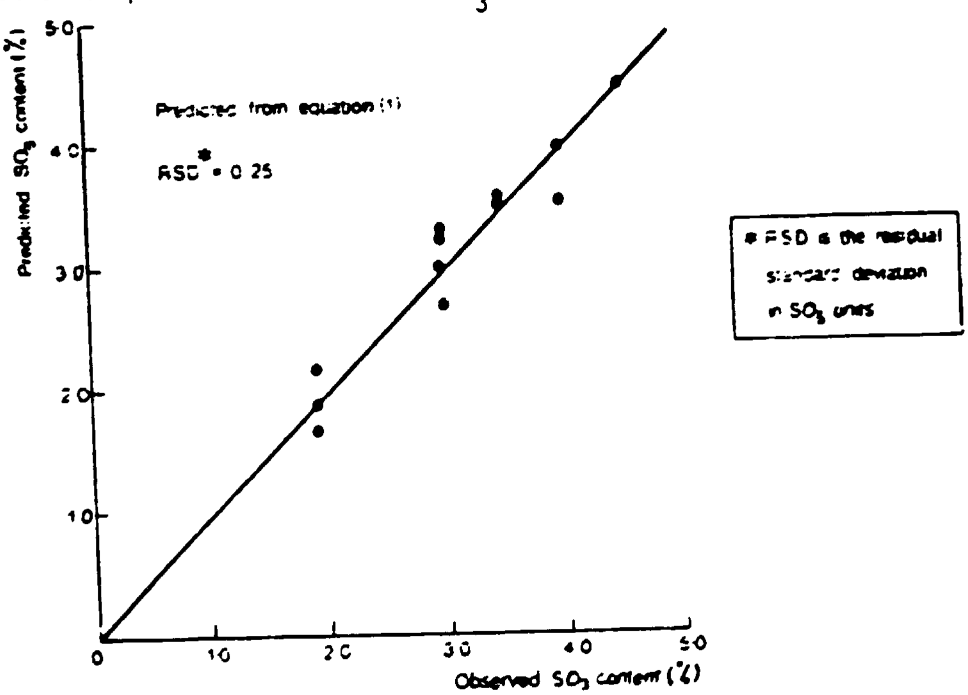
(a) To C_3A content



(b) To equivalent alkali content



(c) Relation of predicted to observed SO_3 content



with the raw materials, raw meal, clinker and cement produced in the company, need to be supplied with some modern equipment capable of carrying out tests in accordance with the most up-to-date cement technology. This equipment would help production, especially by obtaining chemical and physical test results in a very short time. A list of recommended equipment follows. The selection will be made by the plant's management and personnel. The modern equipment recommended is designed to cope with urgent production problems as well as providing all the test facilities needed to assess materials and products.

Proposed equipment and laboratory apparatus

Leco apparatus (for determination of SO_3 content)
Flame photometer
Equipment for testing in accordance with one of the foreign cement specifications (preferably BS)
X-ray fluorescence (and equipment for flux-sample preparation, together with several reference samples)
Sample splitter
Equipment for analysing the exhaust gas of the kiln (dew-point, oxygen content, dust loading etc.)
Differential thermal analysis (DTA) and thermogravimetry (TG) apparatus
Furnaces, including a high-temperature furnace (1,600 °C)
Equipment for vibration analysis
Different sizes of sieves with shaker
Moisture-content determination dryer
Mechanical testing machines
Balances
Automatic vicat (determination of setting time) machine
Electric thermometer
Laboratory crushers, small scale ball mill
Automatic absorption equipment

G. Slurry mixing

During the visit to the 8th February Cement Joi' ' Complex, it was found that the mixing of slurry, especially in the slurry basins, was not properly carried out, although the slurry basins are equipped with mechanical as well as pneumatic mixing systems. Neglecting this factor can lead to problems in burning the raw meal inside the burning zone. This is due to the lack of homogeneity of the slurry so that the raw meal enters the kiln in batches containing different percentages of calcium carbonates ($CaCO_3$) and other components, which makes the burning process uneven and less efficient. There is no need to go into this in great detail, but the following recommendations are made.

Recommendations

1. It is recommended to repair the mechanical mixing system.

2. It is necessary to increase the compressed-air capacity in the pneumatic system so as to improve the efficiency of mixing the slurry properly inside the slurry basins before injecting it into the kiln.

VII. PROBLEMS ENCOUNTERED AT THE CHON NAE RI CEMENT PLANT

A. How to attain the plant's rated production capacity

In the Chon Nae Ri Cement Company there is a reinforced suspension preheater (RSP) kiln which was designed and manufactured in the Democratic People's Republic of Korea. This kiln only produces at 70 per cent of its rated capacity. It is worth looking at some data about this kiln, before considering solutions.

Technical data and specifications

Rated production capacity	60 t/h or 1,440 t/d
Actual production capacity	42 t/h or 1,008 t/d
Kiln diameter	3.75 m
Kiln length	60 m
Kiln inclination (slope)	4°
Filling percentage	6%
Fuel used	Low volatile coal
Moisture	1.4%
Fly ash	34.7%
Volatility	5.52%
Calorific value	4,790 kcal/kg clinker
Oxygen in exit gas	5.0%
Percentage of calcination in precalciner	70%
Heat of gas leaving the precalciner	800 °C
Precalciner gas temperature (inside)	1,100 °C
Temperature of raw meal going into the kiln	800-850 °C
Quantity of dust in exit gases	0.169 kg/kg clinker
Specific heat consumption	1,053 kcal/kg clinker
Quantity of air to the kiln	49,000 m^3
Quantity of air to the precalciner	15,000 m^3
Total air	64,000 m^3
Exit-gas quantity	105,000 m^3
Residence time of raw meal	40 minutes
Secondary-air temperature	800 °C
Coal supplied to the kiln	4.62 t/h (52.38% of the coal)
Coal supplied to the precalciner	4.2 t/h (47.61% of the coal)
Free CaO in clinker produced	2.0%
Fineness of raw meal	13-14% residue

The problem of this kiln was discussed in detail with the Director of the Bureau for Scientific and Technical Guidance and with representatives of the cement plant and of UNDP. The following is a brief summary of the discussion.

Discussion and comment, including some recommendations

The RSP preheater system is a joint development of the Onoda Cement Company Ltd. and Kawasaki Heavy Industries Ltd., Tokyo. Combustion in the heating shaft is maintained with the hot exit air of the clinker cooler. The raw mix coming from the upper stage of the preheater distributes itself evenly in the combustion gas of the heating shaft, into which it drops down after meeting the hot kiln-exit gases. The raw meal is then lifted in the ascending gas duct, called the calcining shaft, from which it enters the cyclone.

The raw meal has to be 90-95 per cent calcined before entering the rotary kiln. In this case, only 70 per cent calcination is achieved, which is a low figure, especially considering that the kiln length is only 60 metres and there is no space to continue further calcination inside the kiln after the raw meal leaves the last preheater stage. In the experts' opinion, the calcination of the raw meal ought to be at least 90 per cent before it enters the kiln so that reaction and production of the clinker phases can begin immediately inside the kiln. That the free lime of the clinker produced is at least 2 per cent confirms this. The decrease in the precalcination performance of the precalciner to 70 per cent of capacity can be attributed, at least partly, to the following factors.

Oxygen in the exit gases. It is clear from the data that the percentage of oxygen in the exit gases is 5 per cent, which indicates that a larger quantity of air is entering the kiln than should be the case, i.e. more false air is being drawn in by the main fan. Therefore, the temperature of the gases inside the precalciner will decrease leading the temperature of the raw material to drop. The performance of the precalciner 11 be diminished and only 70 per cent calcination instead of 90-95 per cent will be achieved. It is clear that the exit-gas temperature leaving the precalciner shaft is about 800 °C whereas it should be approximately 950 °C, which is higher than in the regular suspension preheater kilns.

Kiln/preheater fuel ratio. The amount of coa' supplied to the kiln is 4.62 tonnes/hour which represents 52.38 per cent .e coal used by the production line. The coal supplied to the precalciner is 4.2 tonnes/hour, i.e. 47.61 per cent of the coal supply. As the coa' d is of low calorific value, the percentage of coal supplied to the prec ought to be increased to at least 55 per cent of the total coal supply to help the precalciner temperature to increase so that the gas temperature leaving the precalciner shaft also increases. The change fuel ratio between kiln and precalciner must be under test control and the results have to be recorded and evaluated until the proper fuel-ratio distri' tion can be determined. (The proportion of fuel supplied to the swirl burner can vary from 55-70 per cent.)

The temperature of the air recouped from the cooler and blown into the precalciner is low. It has to be adjusted to within the range 650-800 °C to give optimum conditions for controlling the combustion of coal in the precalciner.

The secondary air temperature is also low (800 °C). This temperature should be as high as 1,100-1,150 °C which can be secured with the stability of the kiln.

The low calorific value of the feed coal will theoretically produce a larger volume of exhaust gas (approximately 4 per cent) at a constant heat consumption. Therefore, the rotational frequency of the induced draft fan (IDF) should be re-calculated and increased or the production capacity of the kiln will be likely to decrease as a result.

The position of the kiln burner should be adjusted in the same way as explained before in the case of the 8th February Cement Joint Complex. In addition to this adjustment, experiments have to be made with the insert length of the kiln burner to determine the best length. The volume of the primary air of the burner has also to be re-calculated and adjusted.

The heat consumption of the kiln is increased from 800-1,053 kcal/kg clinker because of the decrease of the production capacity of the kiln and because of increased radiation losses (due to the decrease in clinker output) and the deterioration of the thermal efficiency of the cooler based on the change in kiln/precalciner fuel feed rates (47.61 per cent/52.38 per cent).

Complete combustion must be possible in the precalciner in the RSP system as pulverized coal is combusted with the aid of air recovered from the cooler which has a high temperature and a high oxygen content. The combustion of pulverized coal can also be assisted in the precalciner by air recovered from the cooler and blown into the swirl burner. It must be noted that, when low-calorific coal is used, the precaliner's temperature distribution will tend to be slightly lower than when using high-calorific coal, but the temperature in the swirl calciner can be controlled by adjusting the swirl burner's air recovered from the cooler. As has already been shown, one way of preventing the lower temperature in the precalciner due to low-calorific coal is to adjust the kiln/precalciner fuel ratio. At the same time, the burner and the cooler have to be adjusted appropriately in order to obtain an excellent quality of clinker.

B. Problems of chemical composition

Alkalis and their effects

The alkali content of the clinker produced in the Chon Nae Ri Cement Plant is 1.2 per cent which is a high percentage. The alkalis originate from the clayey minerals such as mica and feldspar, and from the salts found in the raw materials used in the raw mix. The alkali cycle and its effect will be explained in detail as it causes many problems to the Chon Nae Ri Cement Plant and other cement plants in the country. In general, the quantity of alkali contained as salts in the limestone, clay or any other raw-material component, and the quantity introduced with the fuel oil are of minor importance.

Heating causes the alkalis sodium oxide (Na_2O) and potassium oxide (K_2O) to vaporize, the latter more than the former. Vaporization starts with the mica, and more particularly with the mica-like clay mineral named illite, at lower temperature than it does with feldspars. The escape of the alkali vapours from the clinker is obstructed by the liquid phase, which reduces the porosity of the nodules and thus increases the diffusion resistance. Hence, in general, an additive such as iron oxide, for example, which facilitates the formation of liquid phase will have an inhibiting effect on alkali vaporization. The volatility of the alkalis is also reduced by the presence of sulphates or other sulphur compounds in the raw mix and by sulphur dioxide in the kiln gas.

The alkali vaporization is intensified if the kiln gas contains water vapour as this apparently gives rise to the formation of alkali hydroxides. For this reason and based on experience in the cement industry, it was suggested that water should be sprayed into the clinker cooler and the quantity of cooling air reduced. On evaporation, this water enters the kiln as vapour along with the secondary air and increases the water vapour content of the kiln gas.

The alkali compounds which escape from the charge material are present both in dissociated and in non-dissociated form in the hot kiln gas. The dissociated alkalis react in colder parts of the kiln, under oxidizing conditions, with the sulphur dioxide, carbon dioxide and chloride which are likewise present in the kiln gas. As a result of such reaction, alkali sulphates, carbonates and chlorides are formed. The sulphates are formed preferentially and already at high gas temperature, since their formation enthalpies are greater and their dissociation pressures are lower than those of the carbonates and chlorides. Therefore the alkali carbonates and chlorides are not formed until there is practically no more sulphur dioxide available and the gas temperature has dropped to values below approximately 1,200 °C.

The vapours of alkali compounds condense in the less highly heated kiln zones mainly on the cooler material. The first to condense are the alkali sulphates followed at a lower temperature by the alkali carbonates and finally the alkali chlorides. The condensation regions undergo displacement, however, if the temperature distribution in the kiln changes. The condensation phenomena may cause local accumulations of alkali salts that melt to produce liquid phases which make the particles of material stick together and thus form coatings. Regions where such condensation particularly tends to occur are the kiln-inlet region, the bottom stage of a raw-meal preheater and the calcining chamber of the grate-type preheater.

The alkali compounds which condense on the material will again travel through the kiln and thus establish an internal cycle. Since the alkali sulphates are less volatile, they undergo practically no further vaporization in the sintering zone and are discharged from the kiln with the clinker. The more volatile alkali carbonates and chlorides are, however, vaporized again and re-enter the internal cycle. A closed external cycle is formed by the alkalis which are discharged from the kiln plant with the dust, and precipitated in the gas-cleaning plant and are fed back to the kiln, mixed with the raw meal.

Sulphur

Sulphur is present mainly as sulphates and sulphide compounds in the raw materials as well as in fuels and organic compounds. At temperatures above 1,000 °C, with excess air, sulphur dioxide will be formed which enters the kiln gas. The sulphur dioxide formed in the burning zone during the burning process reacts first with the vaporized alkalis, but also with the alkalis that are still present in the solid form in the charge material. Alkali sulphate is formed as a result of these reactions. The accelerating effect of water vapour likewise plays a significant part. A sulphur dioxide cycle is formed as a result of the reactions of sulphur dioxide with the alkalis, with calcium oxide and calcium carbonate in the kiln, in the preheater and during grinding and drying. If there is an excess of alkalis, the internal sulphur cycle is caused primarily by the reaction with these, which takes place almost entirely in the rotary kiln. The alkali sulphates formed in this reaction are, insofar as they are present in vapour form, precipitated on the material. This occurs mainly in the rotary kiln itself, but partly also in the preheaters. The precipitated alkali sulphates thus travel through the kiln again, pass through the sintering zone, and are discharged from the kiln with the clinker. The reaction of the sulphur dioxide with the calcium carbonate takes place also in the preheater and more particularly in the drying plant or in the evaporative cooler. The calcium sulphate formed as a result of the reaction is subsequently decomposed in the sintering zone, sulphur dioxide again being formed. If the alkali content in the material is not high enough to combine the whole of the incoming sulphur as alkali sulphate, high values for sulphur-dioxide content occur in the kiln gas. In such circumstances, calcium sulphate or other sulphate compounds formed therefrom, may pass undecomposed through the sintering zone.

Some research results on sulphate have indicated that, in kilns equipped with grate-type preheaters, substantially larger quantities of sulphur can be removed in the dust collected in the electrostatic precipitators and in the intermediate dust collection plant than in the case of raw-meal preheater kilns, even though the actual dust quantities are smaller in the former case. The SO_3 content of raw gas from kilns with raw-meal preheaters may be anything up to 1,200 mg/m^3, but it goes down to values under 140 mg/m^3 if the exit gas is re-used for drying the raw materials or is moistened to improve the dust precipitation in an evaporative cooler.

Chlorides

The chloride content in the raw meal is always in the range 0.01-0.1 per cent (by weight) if the meal contains no kiln dust. However, even within the same rock deposit, the chloride content may vary considerably and may even exceed 0.3 per cent. There is a little or no chloride present in the oil or natural gas used as fuels. In general, coal contains between 0.01 and 0.1 per cent (by weight) but in high-ash coal there may be as much as 0.4 per cent of chloride.

The chloride liberated during heating of the material and combustion of the fuel reacts with the alkalis to form alkali chloride. This reaction takes place either in the charge material or, after vaporization, in the kiln gas. The alkali chloride, which is in vapour form at high temperatures, condenses on the material in the cooler parts of the kiln, more particularly in the vicinity of the kiln inlet and also in the preheater, since the vapour pressure of the chloride is almost zero at around 800 °C. Research results on chlorides have shown that 10-53 per cent (on average about 21-25 per cent) of the chloride introduced into the kiln is discharged with the clinker. This corresponds to an average chloride quantity of about 0.1 gm/kg clinker. The amounts of chloride contained in the dust are about twice as large for the grate-type preheater kilns as for the raw-meal preheater kilns. A reduction of the external chloride cycle by removal of the chloride-rich dust is therefore economically justifiable only for grate-type preheater kilns. The chloride cycle was significantly reduced in both types of kiln by means of a by-pass which takes 5-7 per cent of the exit gas in a raw-meal preheater kiln and 15 per cent of the exit gas in a kiln equipped with a grate-type preheater. It is nearly always the case that the amounts of chloride discharged with the clinker and dust will be substantially smaller than the amounts of chloride entering the kiln with the raw materials and fuel. The amounts of chlorides, which remain in the kiln and form what is called the chloride cycle, will increase continuously by further intakes of chloride, resulting in chloride accumulations and trouble arising from coating and ring formation.

Fluorides

Fluoride participates in the cycle processes in the cement kiln in much the same manner as the other volatile constituents, but does not generally give rise to operational disorders. The fluoride content of the raw meal is 0.02-0.07 per cent. Coal contains up to 0.02 per cent and fuel oil up to 0.002 per cent of fluoride (by weight). About 88-99 per cent of the fluoride introduced into the kiln is discharged from it with the clinker. This is so high a proportion that the internal fluoride cycle can not give rise to troublesome accumulations. Residual amounts of fluoride are present in the dust, i.e. the greater the external fluoride cycle (depending on the external dust cycle), the smaller is the proportion of fluoride (in relation to the total balance) that is discharged with the clinker. From this, it can be inferred that the fluoride in the cement kiln is always combined with solids and the discharge of gaseous fluorine compounds cannot therefore occur.

VIII. GENERAL PROBLEMS

A. Avoiding cyclic phenomena

The cyclic phenomena of alkalis, sulphur, chlorides and fluorides is one of the major problems throughout the cement industry in the Democratic People's Republic of Korea, as was illustrated in the case of the Chon Nae Ri Cement Company. This problem and how to deal with it was fully explained to the technical engineers of both cement plants visited as well as in a special lecture held in the Pyongyang Hotel on 23 March 1986 which lasted three hours and was attended by many technical engineers from different cement plants as well as from the Commission of Construction and Building-Material Industry and the Bureau for Scientific and Technical Guidance. The suggested ways and means of reducing the cyclic phenomena are summarized here.

Reducing the cyclic processes of dust and volatile constituents also reduces the tendency to form coatings and rings, which adversely affect the performance of the kiln. In some cases, the quality of the clinker and of the cement made from it, can also be improved by curbing these cycles. The dust cycle can be reduced by improving the flow conditions in the kiln-inlet region. This can be achieved more particularly by repairing any damage to the refractory lining and by the removal of coatings, as the latter are liable to cause turbulance in the material entering the kiln. As a result of such precautions, the effectiveness of a by-pass can also be enhanced, since the volatile constituents to be removed will then be less diluted with dust. Besides this, if the dust cycle in the preheater is reduced, kiln performance is improved, while heat consumption and electric power consumption are decreased.

The magnitude of the alkali, sulphur and chloride cycles depends essentially on the content of these substances in the raw material. Generally speaking, that content can be varied only within very restricted limits. Intervention in the external cycle by the removal of dust in which these substances have become concentrated is economically justifiable only if the dust quantities involved are small and their content of such substances is high. This will, in general, be the case only in kilns equipped with grate-type preheaters. From the balance investigations, it emerges that, for these kilns, with precipitated and discarded dust quantities not exceeding 20-25 g/kg clinker, it is possible on average to get rid of 11 per cent of the alkalis, 20 per cent of the sulphur and 42 per cent of the chloride that enters the system. These amounts are substantially increased if the bed of material on the preheater grate is highly permeable to the passage of gas. In general, the dust removed from the system is not utilized. In special cases, however, it may be economically worthwhile to extract the water-soluble volatile constituents and to return the residual dust to the kiln.

Intervention in the internal cycle by means of a by-pass installation is necessary if, due to fairly high alkali and chloride contents, objectionable coatings occur at the kiln inlet and in the bottom cyclone stage in the raw-meal preheater (as in the kiln system considered here) or in the calcining chamber of kilns with grate-type preheaters, and if a reduction of the external cycle is not economically justifiable or would not be adequate. In general, this is likely to be the case if the total amounts of alkali that go into the kiln are approximately 20 g K_2O/kg of clinker and the chloride amounts exceed about 0.2 g/kg clinker. The limiting values become higher if intervention in the external cycle is possible.

A by-pass is particularly necessary if the chloride content is high. In that case, the chloride cycle constantly increases i.e. a progressively larger amount of chloride per kilogram of clinker remains in the kiln. The increase in this cycle is found as the difference between the chloride introduced with the raw meal and the fuel and the chloride discharged with the clinker and the dust. With the increase of the chloride cycle the content of chloride in the material at the kiln inlet progressively rises.

According to the available research results on this subject, it becomes essential to use a by-pass system (especially for dry process preheater kilns) if the chloride content of the raw meal is more than 0.01-0.015 per cent (by weight), corresponding to about 0.2 g chloride per kilogram of clinker. The by-pass should then be arranged so that the kiln gas extracted through it contains as little dust but as much chloride as possible. The most favourable point of gas extraction will depend on the flow conditions in any particular case. It can be determined only by trial and error. The by-pass, with its associated dust precipitator (designed for a high precipitation efficiency of 99 per cent), must be so designed that up to 10 per cent (in extreme cases up to 15 per cent) of the kiln gas can be drawn off and that this percentage can be modified during the operation of the plant in order to adjust the flow of by-passed gas to possible changes in the chloride content of the raw material.

A by-pass system for lowering the internal alkali cycle or the sulphur cycle is necessary, from an operational point of view, only if the alkali balance of the kiln exceeds about 20 g K_2O/kg clinker. However, available experience with by-pass systems indicates that it is hardly practical economically to produce a low-alkali clinker from high-alkali raw materials by adding a by-pass. This is still not possible even if the alkali vaporization is substantially increased by the addition of calcium chloride to the raw meal, by increasing the water content of the kiln gas or by reduction of the sulphur balance.

B. Design models of by-pass systems

There are four models of by-pass system.

1. One has a preheater by-pass arrangement where the alkali-laden dust of the by-pass gas is separated in cyclones and the cleaned gas is then reunited with the main gas stream. The alkali-laden by-pass dust is usually either discarded or leached.

2. In the second arrangement, the by-pass gas is first precleaned in cyclones and is then led into a separate electrostatic precipitator for final precipitation. In this case, the gas stream is more suitable for raw-material drying than in the first arrangement.

3. In the third design, the gas is introduced directly into a separate dust collector.

4. Fourthly, it is possible to install a variable by-pass for 0-100 per cent of the kiln gas. There is an adjustable throat for the bigger variations.

The choice of one of these arrangements depends on the chemistry of the raw materials as well as on the volume of the by-pass gas to be handled.

After much discussion, it was finally decided, at a meeting with Kim Jae Ok, Director of the Bureau of Scientific and Technical Leadership, to start designing a by-pass system based on the first model described above.

C. Coating and rings

Occurrence

Coating and rings can constrict the cross-section of the kiln at certain places, thus obstructing the passage of the material and increasing the gas-flow resistance. As a result, the discharge rate of the exit-gas fan is reduced and kiln performance is correspondingly impaired. The results of research studies show the frequency of the occurrence of objectionable coating in the rotary kiln as follows:

Frequency	Percentage
Between once a week and once a month	40
Once a month	10
Less often than once a month	19
Irregular	19
No trouble at all	12

Coatings liable to impair the operation of the kiln plant are most likely to occur in the bottom stages of the raw-meal preheater, on the walls and on the roof of the calcining chamber of the grate-type preheater and in the inlet chute of the clinker cooler. In addition, coatings are often observed to form on the blades of the exit-gas fans and intermediate fans. Rings may form in any part of the rotary kiln, meal rings in the calcining zone, sinter rings at the beginning and clinker rings at the end of the sintering zone (transitional zone). In the long wet-process kilns, slurry rings may also develop in the drying zones. Occasionally, balling of the material in the kiln is observed. Such balls, which may be up to 1 metre in diameter, can apparently be formed in the calcining zone of any kiln.

Composition of coatings

The coatings in those parts of the kiln system in which the gas has already cooled to temperatures below about 500 °C consist mainly of the kiln charge material which has not yet been significantly transformed and in which more or less large accumulations of alkali sulphates and alkali chlorides are present. This applies more particularly to the coatings formed on the fan blades and to the slurry rings in long wet-process kilns.

The coatings in the preheaters and in the kiln-inlet regions associated with them occur at gas temperatures of up to about 1,100 °C. Such coatings still contain raw-meal constituents, but these have already become transformed by heating, usually containing substantial amounts of alkali sulphates and alkali chlorides as well as calcium sulphate in the form of anhydrite and a complex potassium-calcium sulphate with the formula $2CaSO_4.K_2SO_4$. In addition, spurrite was shown to be present, which is a calcium carbonate silicate with the formula $2(2CaO.SiO_2)$. $CaCO_3$, and also the so-called sulphate spurrite, which is the calcium sulphate silicate corresponding to spurrite and having the formula $(2CaO.SiO_2).CaSO_4$. These coatings are usually porous at first, but become dense and hard as time goes by, while their content of alkali chloride may also increase.

The ring formations which occur at the end of the calcining zone, at gas temperatures of 1,100-1,400 °C and material temperatures of 800-1,200 °C, for the most part contain no raw-meal constituents except some calcite. The content of CO_2 may be up to 10 per cent (by weight), free CaO sometimes occurs. The content of alkalis generally does not exceed the usual values for clinker except in kilns with low thermal loading. Accumulations of sulphates are however often found to occur.

Ring formations are generally porous and soft, though sometimes, at the end nearest the flame, they are dense and hard. In large rotary kilns with preheaters, usually only the end of such rings nearest the kiln inlet is soft whereas the main part of the ring (which may be up to 20 metres long) is very hard.

The composition of the sinter and clinker rings is, generally speaking, similar to that of the clinker itself. A special case is that of the sinter rings in kilns fired with high-ash coal. Their chemical composition is intermediate between that of the clinker and that of the coal ash. Consequently they may contain gehlenite ($2CaO).Al_2O_3.SiO_2$) instead of tricalcium silicate. Sinter rings and clinker rings are always rather hard. If they are composed of fine-grained particles, they are generally dense, in which case a strati- fied structure is discernible. On the other hand, if they contain a high proportion of coarse clinker particles, they usually have large pores.

Where balls of material form in the kiln, these are similar in composition to the clinker, but often have a higher silica modulus and a higher content of CaO.

Causes of coating formation

Coating, or the adhesion of solid particles to the walls, is attributed to such factors as:

Moisture adhesion
Fusion adhesion due to the surface tension of thin liquid films
Surface adhesion due to the forces of attraction acting at the surface
Adhesion due to electrostatic charges
Adhesion due to matting of fibrous or lath-shaped materials.

Methods used to prevent or dissolve rings and coating

All or some of the following methods may be applicable:

(a) Changing the raw-mix design and the fineness of the raw meal;

(b) Variation or displacement of the flame;

(c) Lining some specific areas with refractory bricks to counteract coating formation;

(d) Replacing the deformed sections of the kiln shell;

(e) Knocking the rings off with rods and rams;

(f) Hosing the rings off with cold water;

(g) Shooting the rings off with guns;

(h) Blasting the rings off, using high-pressure water jets (WOMA pump);

(i) Destroying the rings by vibration;

(j) Applying the BINBLOW method (a paper was supplied describing this method).

Part Three

INVESTIGATION OF THE EXTRACTION AND PROCESSING OF GRANITE

IX. VISIT TO THE ON CHUN GRANITE COMPANY

The Adviser visited the On Chun Granite Company, 45 kilometres south-west of Pyongyang, on 26 March 1986. He was accompanied by Jon Yong Huei, Director of Korea Cement and Granite Export and Import Corporation, and by Zo Ho Sung, Interpreter.

The Adviser visited both the granite quarry and the granite processing plant where explanations were given by the Director of the processing plant. A summary of these explanations is given here.

Production and methods

Production: Granite, raw blocks and polished sheets

Colour: Grey granite, with black tint after polishing

Method applied in extracting the granite blocks: Explosives

Methods applied in processing the granite: Mechanical saws, normal methods used in cutting and polishing the granite. Sawing is carried out with very fine steel balls. The old saws are imported from Italy while the new ones are locally-manufactured according to the same specifications.

Types of products

Raw granite blocks used for monuments and memorials
Production capacity: 50,000 m^2 a year

Polished granite sheets of 2-2.5 cm thickness
Production capacity: 20,000 m^2 a year

Granite gravels (crushed granite)
Production capacity: 30,000 m^2 a year

Small blocks for paving roads
Production capacity: 50,000 m^2 a year

Medium blocks for covering walls of dams
Production capacity: 40,000 m^2 a year

Comment

The company promised to supply data on the machinery used in the processing plant and its production capacity, but unfortunately these were not received before the Adviser left the country on 1 April 1986. These data would be a help in evaluating the production capacity and in giving advice and making recommendations.

As far as the machinery and equipment used in the quarry are concerned, it is all of a simple kind and all produced in the Democratic People's Republic of Korea. The granite extraction requires more effort than when modern machinery is used. It would be advisable to install modern machinery in the granite quarry.

The following chapter gives a general introduction to granite, its properties and uses, and guidelines for granite extraction and processing.

X. INTRODUCTION TO GRANITE EXTRACTION AND PROCESSING

Definition of granite

In industry, the term "granite" is used in a much broader sense than it is in geology. In industry, the term covers true granite, granite gneiss, and the intermediate members of the granite-gabbro series. Gabbro, diabase, anorthosite and pyrozenite are called black granite when used for polished, cut stone. Notwithstanding the broad use of the term, most of the commercial granite has a composition in the range that includes true granite and granodiorite, such as the granite produced in the On Chun granite quarry.

Uses

The main uses and applications of the granite produced in Korean granite quarries are:

(a) Dimensioned granite for monuments and memorials;

(b) In buildings as ornamental polished paving stones to cover the ground, as wall tiles, or for use as steps, columns or foundation blocks;

(c) As curbstones and paving blocks;

(d) Crushed granite as aggregate in concrete, road metal, railway ballast, as riprap in breakwaters, dam spillways and similar structures.

The methods of quarrying and milling granite differ markedly according to the use to which the granite will be put.

Properties of crushed granite

The properties of greatest importance for crushed stone are its toughness (resistance to impact), hardness (resistance to abrasion) and soundness (resistance to disintegration from repeated freezing and thawing). The first two affect the cost of production: toughness determines the case or difficulty of blasting and crushing; hardness affects the wear on crushers and other machinery. Soundness is important in determining the suitability of the stone for a certain use. Nearly any fresh fine- to medium-grained granite possesses these properties in favourable ratios for use as crushed stone.

Properties of dimensioned granite

The property requirements of granite for use as dimensioned stones or slabs are considerably more rigorous than for crushed stone. The colour of the granite produced here is grey and it is controlled largely by the ratio of feldspar and quartz to biotite and hornblende. Uniformity of colour in the huge mountain deposits is well observed and is highly desirable as Korean granite is widely known for its peculiar colour. This has been the most important factor in its choice by users over a long period of time, especially since the reconstruction of the country started.

The granite rock here is hard, and is in fact the hardest rock used for structural purposes. This not only affects quarrying costs, but the cutting and dressing costs are also relatively high.

The texture of the granite is medium and uniform. The grain size is well distributed all over the granite body. In some rare cases, "knots" (either due to mineral segregations or to inclusion of other rocks) are observed and have to be avoided in the dimensioned granite.

Origin

The formation of granite is usually attributed to the solidification of liquid magma. An alternative hypothesis (granitization theory), which is supported by much field evidence (for example, the On Chun quarry) is that some granite is produced from pre-existing solid rock, without passing through a liquid stage. The process of granitization is essentially one of metasomatism or replacement perhaps through pervasive "soaking" by plutonic solutions. Arguments and supporting evidence for both magmatic and metasomatic views have been presented by several geologists. In this quarry, the granite appears to be metasomatic in origin.

Structural features of the granite mass

Most granite bodies, regardless of their origin, are found to possess certain structural features in common. These features are of little interest to the producer of crushed stone, but they are of great importance in quarrying granite for slabs etc. It has been found that most granites have directional properties. The rif. and the grain are the directions of easiest and second-easiest parting. In most deposits, one is vertical or nearly so and the other is horizontal or nearly so. The direction at right angles consists of minute cracks 0.09 to 1.3 millimetres apart, crossing the quartz grains and extending into the feldspar, producing an obscure fissility into two directions at right angles. Rift is parallel to the mica plate and is the direction of predominant feldspar cleavage. (Skilled stonecutters in any granite quarry can detect the rift blindfold, by the feel of the surface). Rift and grain fractures are approximately parallel to, or coincide with, sheets of extremely minute fluidal cavities in the quartz grains. Secondary minerals (sericite, limonite) form within the rift or grain cracks. Rift and grain were apparently produced in granite that crystallized under crystal stress, and are inherent features which the rock has had since its formation.

Joints occur in granite masses in the same way as in other rocks, and are thought to have been produced by regional torsional stress. They commonly occur into two sets, in vertical planes and at right angles to each other. Intermediate curved and irregular joints may also occur. Two aspects of joint spacing and staining, are important to the producer of cut stone. If the two major sets are spaced 3 to 10 metres apart, rectangular blocks may easily be removed and a quarry developed systematically. Closely-spaced joints render the rock worthless for dimensioned stone. This is not observed in this quarry.

Staining of the rock along the joints was observed. These stains have taken place due to oxidation and other chemical processes, since the joints make natural passageways for ground water.

Quarrying

The extraction of granite blocks differs very much from the methods used for the extraction of other raw materials and minerals, where those rocks and minerals are broken into small pieces with explosives. In this granite quarry, explosives are used to extract all the different types of granite required, whether it is to be dimensioned or not. The effect of an explosive detonation in the granite rock, where the explosive is closely confined, is to send out a high pressure pulsation which initially crushes and fractures the rock. When the stress wave meets the free face, it is reflected back through the rock causing more crushing and fracturing. This phenomenon can easily be seen in the On Chun granite quarry and much of the granite has to be rejected. The reason given for using explosives in the quarry, although this method destroys much of the granite in the quarry, is that the deposit of granite in this ridge contains immense reserves of this granite, sufficient for all the country's needs. The Adviser's view is that, whatever the size of the reserves, it is not desirable or economical to use explosives in extracting granite blocks as the main purpose is to produce these blocks as large as possible, which is difficult to achieve when the rock is fractured by the explosives. The extraction of granite blocks needs highly technical methods. There are various processes that can be used. In some cases, manual methods and, in others, mechanical methods will be more suitable.

Some information on both types of method are given here.

Manual method

Granite blocks can be extracted in considerable quantities by using the following manual method. A working trench is made along the quarry face, to relieve strain in the rock and to provide working space on vertical faces. Such a trench may be made by drilling a row of core holes and removing the webs between them, or by making two parallel channels, one to two metres apart, by drilling small holes close together, then cutting the webs with a broaching tool, and removing the granite stone between the channels. Vertical separation of blocks from the parent rocks (ledge) may be done by:

(a) Utilizing natural joint planes, if present, or making cracks to separate the blocks by hand drilling;

(b) Exploding light powder charges in several deep drill-holes.

Which method to use depends on the jointing and the rift and grain fractures through the granite. If the jointing and rift and grain fractures are near together or slightly apart from each other, the first method is used. If, on the other hand, the jointing and rift and grain are far apart from each other, the second method is used.

Method (a). In this method, small sockets, scattered throughout the main granite block are drilled manually. These sockets are V-shaped, the length and the depth of these sockets depends on the volume of the block to be extracted. The sockets are drilled in one line on the faces of the block required for extraction, either horizontally or vertically. The sockets are drilled fairly near each other, at distances 10-15 centimetres apart. After finishing these sockets, rectangular steel plates of a special type have to be inserted in the sockets beside each other. Chisels are inserted firmly in these sockets between the steel plates. After the chisels have been arranged, horizontally and vertically, workers are grouped together to hammer on the heads of these chisels, all hammering at the same time, leaving one and knocking on the next until they reach the last one in the group and then repeat the knocking in, one by one, until the chisels penetrate the socket and find their way inside the granite block. After several knockings, a crack or joint will start to appear, making a sharp line between the sockets. The vertical and horizontal cracks meet each other at right angles, thus separating a block out. Smaller blocks and slabs can be separated one by one from the main block by using the same method. The surface of the blocks can then be smoothed by chisels.

Method (b). In this method, explosives are used to extract and separate the granite blocks. This is appropriate where the joints are far apart from each other and the thickness of the main block is very large and, therefore, the first method can not be applied. In this method, a borehole is made at the required distance from both sides of the block and is filled with black gunpowder which is then ignited. The block is withdrawn or removed from its place with a manual winch. The block's surfaces are then smoothed with chisels.

With both types of manual extraction, a fair amount of reject stones are produced.

Mechanical method

(a) Wire saw. This method has been used for approximately 100 years. Its use spread at the end of the last century. Here, a fast, helical-wire type Telecomp is used. This method has not really changed since its introduction in marble and granite production a century ago, but it has become more efficient in modern times. The helicoidal saw is very simple in its structure. It consists of a motor equipped with an axis upon which some pulleys are arranged. These pulleys and wheels carry the wire. The motor is erected not far from the granite or marble quarry. Some tubular posts are mounted on an orienting unit on bearings and a balanced pulley. These pulleys carry the wire from the axis attached to the motor to the rock face. The motor is equipped with clutch, four-speed-plus-reserve gearbox, reduction unit and driving wheel(s). This is called the propulsion unit. The gearing unit consists of three idle alu-rubber pulleys (900 mm diameter), mounted on ball bearings and protected by a sealed screen, complete with oil seals and felts. The gearing battery is made of a frame attached or anchored to the foundation which carries the idle pulleys. These are special slide-ways on which can be mounted and fixed two slides bearing orienting pulleys. The wire is raised and lowered by an automatic cutting apparatus which is moved directly by the helicoidal wire and operates the cutting carriage with rollers. The cutting rate can be adjusted, even during operation, by using this apparatus.

All the wheels carrying the three-strand helicoidal wire are rotated by the action of the motor and when the wire penetrates the granite face, it cuts through the granite between the two orienting posts. The wire cuts through the granite by using water and minute iron balls or iron filings or sand (coarse sand grains are used in cutting marble). Here, the iron balls or filings or sand grains are captured between the grooves of the helicoidal wire and act as saw teeth cutting through the granite. The wire can be raised or lowered around the face by the automatic device mentioned above. An automatic drilling and boring machine is located at the face. The wire saw cuts automatically. This means that an operator is not required. The cutting rate can be adjusted during operation, and it can be regulated according to the specific drilling or cutting needs. Connected with the Telecomp, an abrasive unit capable of feeding several cuts has to be provided. This unit is capable of supplying the cutting wire with abrasive slurry, also without any need of an operator.

The advantages of this method can be summarized as follows:

(i) Time, space and installation costs are reduced to a minimum by the set-up of the circuit;

(ii) Supervision, control and maintenance are reduced to a minimum as the wire cannot come out from the pulleys, the distance between the drive unit and the cutting area is limited and lubrication is almost permanent;

(iii) The output or cutting rate per hour has doubled as the wire speed and tension have been increased considerably. This has become possible only through the new set-up of the circuit; the new design of balanced over-sized wheels running on tyred tracks, the rational abrasive-feeding system, the new uprights etc.

Technical data:

Power	20-25 hp
Diameter of the driving pulley	700-800 mm
Wire-operating speed:	
Normal	6.5-8 m/min
Fast	12-14 m/min
Wire tension	250-300 kg/cm^3
Number of driving pulleys	3 + 1
Number of gearing pulleys	6 + 2 + 1
Diameter of pulleys	800 mm

In this quarry, the wire cuts after a distance which equals the quarry face length. This distance was fixed by the technical staff. The two posts carrying the wire, which cuts through the granite, are placed in trenches at both sides of the quarry face. The height of the quarry face is also fixed by the technical staff of the quarry. After cutting the quarry face vertically, horizontal cutting at the quarry's foot can be carried out. After this, the main block is subdivided into the required dimensions by using the wire or by manual operations as described before, or this subdivision can be done by using power-drive pumps.

(b) Comb drilling. In this method, comb drilling is carried out by hammer-drilling or channelling machines. There are two pneumatic saddles at variable angles from 0° to 90°. The effective vertical stroke is 1.8 m/sec.

The equipment consists of a three-metre-wide, five-rail truck with screws for fast clamping to the ground, a carriage with swivelling turret for the pneumatic saddles and wheels to run on the rail track. There are two toothed wheels which enable the carriage to run gradually on the rail track. The high-speed hammer drill weighs 20-25 kilograms. There is an air distributor for adjusting the speed of descent. There are flexible control pipes with a line lubricator.

The comb-drilling machine is erected on the quarry where cutting and drilling can be carried out as required. The machine has to be aligned and adjusted to the required place. Then the drilling rods can penetrate through the ground to the desired depth. In the same manner, drilling can start again horizontally, at right angles to the previously-drilled vertical holes so as to connect with them.

After drilling, there is a special "plug-and-feather" procedure. Feathers are strips of steel, flat on one side to fit the wedges and curved on the other to fit the wall of the drill hole. Two are inserted in a drill hole and a "plug" or steel wedge chisel is driven between them. The plugs are inserted gradually, in successive steps, until a fracture or crack appears. Feathers are tapered to match the holes, so that when the plug is driven in, the feathers are forced apart the same distance at every point along the hole. Knocking and hammering on these chisels will create a crack which joins all the boreholes. This main block can again be divided in the same way. Blocks are removed from the quarry by means of derricks.

These are the main methods used in the extraction of granite or marble blocks. The mechanical methods can be used satisfactorily in the On Chun granite quarry to improve the existing methods which are not very satisfactory.

There is all sorts of very modern technology which is currently coming into use in the developed countries for cutting main blocks i.e. ultrasonic waves and water jets, jet-piercing burners etc. In these systems, drilling and wedging are used for making secondary cuttings and for freeing blocks from the floor.

Processing

Cutting off blocks into the desired sheets and dimensions is carried out using mechanical saws. Very fine steel balls 1-3 millimetres in diameter are used to cut the granite blocks into sheets according to the thickness required. This method is used in most countries and there is no need to comment on the process, except that the efficiency of cutting the blocks is below the average rated capacity. There is also no need to comment on the polishing and finishing of the granitic sheets as, here again, the plant is using the same processes as are used in most countries.

Part Four

TRAINING OF NATIONALS IN THE CEMENT INDUSTRY

Introduction

It is the ultimate objective of every country to master the process of industrialization, that is to say, to achieve the greatest degree of technological self-sufficiency of which it is capable within its present limitations. The sophisticated and complex techniques of a modern cement plant cannot be mastered without suitable preparation and training. The larger the plant and the larger the distance separating men and stations, the more important it is to ensure that proper training is given, since the cost of human error becomes greater. It has been found necessary to train manpower before establishing and running a new cement plant, which means creating close links between the educational and training systems and industry. These links were sometimes weak in the developing and least developed countries.

At the present time, the Democratic People's Republic of Korea is facing difficulties in developing their cement industry and in introducing up-to-date technology. Training of nationals will provide the knowledge, skills and attitudes needed for modern production. Training will raise the trainees' productivity, increase the profits of their undertakings and stimulate progress in science and new technology, thereby contributing to general advancement. Industrial training, especially in the field of cement, has to be looked upon as an integral part of national strategies for development of the industry, also encouraging employment promotion, income distribution and popular participation in development.

XI. SPECIAL CONSIDERATIONS AND PROBLEMS OF THE CEMENT INDUSTRY IN THE DEMOCRATIC PEOPLE'S REPUBLIC OF KOREA

The Democratic People's Republic of Korea has been self-reliant in producing cement machinery and equipment. However, during the last ten years, the technology of the cement industry in the developed countries has progressed much faster than that which was used in the Korean cement industry. Here, most of the cement is produced using the wet process and the old-fashioned shaft kiln. Dry-process preheater and precalciner kilns have been introduced but unfortunately up-to-date technology is still unavailable. Because of the need of the country to introduce modern technology in its cement production, it is necessary to prepare the designers, engineers, chemists and technicians to be able to introduce and use the new technology. These personnel have to be trained both academically and practically, in cement plants which use the modern technology.

Place and Objectives of training

Training of the national personnel could be carried out in one of the following institutes:

(a) The Cement Industry Institute in Tianjin, China;

(b) The Turkish Cement Research and Development Centre (Ankara) and the Associated Turkish Cement Plants;

(c) The Cement Research Institute of India.

For the Democratic People's Republic of Korea, it is probably preferable to carry out training in the first institute. The immediate objectives of the training programme are to:

(a) Achieve full utilization of the existing cement-production facilities through in-plant training;

(b) To enable personnel to become acquainted with the up-to-date technology used in the cement industry in the developed countries before introducing it in their country.

The long-term objective is to train the national technicians each one in his own special branch of activity.

Lectures presented by the Adviser

The Adviser gave two lectures. The first one, on 9 March 1986, which lasted for four hours, was intended to explain the new semi-wet process technology which is proposed to be introduced into the Korean cement industry to conserve energy and increase production capacity. The modern technological processes were explained in detail, with special concentration on the methods used to convert high-energy-consuming cement plants using the wet-process method to semi-wet, semi-dry or dry process, low-energy-consuming plants.

In the other lecture, on 16 March 1986, which lasted three hours, the Adviser presented a paper on the flow of raw materials inside the kiln, with special reference to alkalis and the problems created by them in the kiln system. He also discussed how to reduce these problems, especially the problems affecting the quality of the cement produced.

Both lectures were followed by discussions and by answering technical questions for at least two hours more.

Annex I

JOB DESCRIPTION
SI/DRK/86/006

Post title: Special Technical Adviser for the cement industry

Duration: Two weeks

Date required: As soon as possible

Duty station: Pyongyang, with travel within the country

Purpose of project:

Duties: The Cement Adviser will be attached to the Government of the Democratic People's Republic of Korea and delegated to the cement plants to assist in examining the possibilities of converting some of the wet-process plants into semi-wet, semi-dry or dry-process cement plants. The Consultant is particularly expected to:

1. Evaluate the situation of the cement industry in general;

2. Advise on all aspects related to the possibility of converting some cement plants which have a high-energy consumption (wet process) into lower-energy consumption processes (semi-wet, semi-dry or dry);

3. Elaborate terms of reference for a wet/dry conversion feasibility study for possible use in follow-up activities with or without UNIDO assistance;

4. Evaluate the raw-material situation in general and in relation to dry-process operation;

5. Elaborate a plan for possible energy conservation in the existing cement industry;

6. The expert is also expected to prepare a final report setting out the findings and recommendations of the mission.

Qualifications: Graduate chemical engineer with at least 20 years' experience in the cement industry

Language: English

Background information: The Democratic People's Republic of Korea has now entered the stage of intensive industrial development. Conscious of the economic potential of the natural mineral reserves, the Government has already given attention to their proper exploitation.

The Democratic People's Republic of Korea has a large production of cement serving both the domestic and export markets but the country urgently needs to increase the quantity produced and to decrease the energy consumption in the cement industry. Nearly all installations in the country are still based on the wet-process technology, where fuel consumption is from 50% to 100% higher than for dry-process plants of similar capacity.

The Government has therefore decided to ask for UNIDO assistance in examining the possibility of converting some of the wet-process plants into dry, semi-dry or semi-wet process plants, in order to conserve energy and, if possible, to reduce the production costs.

Annex II

TECHNICAL AND PRODUCTION DATA SUPPLIED BY 8TH FEBRUARY CEMENT JOINT COMPLEX

RAW MATERIALS CHEMICAL COMPOSITION

	Ig.loss	SiO_2	Al_2O_3	Fe_2O_3	CaO	MgO	)	K_2O	Na_2O
Limestone	42.27	3.23	0.23	0.34	52.47	0.96	99.45	0.01	0.002
Clay	4.66	62.82	19.45	8.09	1.72	1.44	98.18	1.45	0.25
Silica	1.82	85.52	6.48	2.92	1.30	1.02	99.06		
Iron ore	9.81	23.80	6.98	54.79	2.14	1.44	98.87		
Slag		37.89	13.52	0.96	40.31	4.72	97.44		
Gypsum		4.57	0.24	0.28	31.07	1.23	SO_3 40.58		
Coal ash	0.38	58.17	28.77	7.40	2.56	1.04	0.78		

CLINKER AND RAW MIX CHEMICAL COMPOSITION

CLINKER

Ig.loss	SiO_2	Al_2O_3	Fe_2O_3	CaO	MgO	)	K_2O	Na_2O
0.03	21.23	6.53	4.66	64.51	1.72	98.68	0.32	0.27

LSF	SM	AM
0.90	1.90	1.40

RAW MIX

$CaCO_3$ content 80.41%

Water content 34.89%

Fineness + 88 16.26%

Chemical Component

Ig.loss	SiO_2	Al_2O_3	Fe_2O_3	CaO	MgO	)	CaF2
36.03	11.72	2.78	2.60	44.38	0.98	98.48	0.35

*This annex has not been formally edited.

TECHNICAL DATA OF THE EXISTING ROTARY KILNS

Rotary Kiln	Kiln 5	Kiln 6
Type	Long wet-process rotary kiln	
Size	dia. 3.6/3.3/3.6 x 150 m (diameter x length)	dia. 4 x 150 m
Inclination	4%	4%
Power	220kW	320kW
Revolutions	0.9 - 1 rpm	0.9 - 1 rpm
Temperature		
Primary air	50°C	20°C
Secondary air	~300°C	~400°C
Exit gas	200°C	230°C
Volume of air applied		
Primary air	260 m^3/min	170 m^3/min
Secondary air	660 m^3/min	920 m^3/min
Exit gas	3640 m^3/min	4490 m^3/min
Capacity	670 t/d	780 t/d
Specific capacity	0.52 t/m^3	0.48 t/m^3
Specific heat		
Combustion	1670 kcal/kg clinker	1620 kcal/kg clinker
Clinker Cooler		
Type	Grate cooler	Grate cooler
Size	--	2.4 x 16 m
Cooling air	530 m^3/min	750 m^3/min
Clinker Temperature		
Inlet	900 - 1000°C	900 - 1000°C
Outlet	~400°C	~300°C

Precipitators

Type	Electrostatic precipitator	Electrostatic precipitator
Dust content of vent gas		
inlet	35 g/Nm^3	33 g/Nm^3
outlet	1.5 g/Nm^3	1.2 g/Nm^3
Voltage	40 KV	40 KV
Capacity	6670 m^3/min	4800 m^3/min

Fuel

Used anthracite coal and brown coal

	Water content %	Ash %	Volatile matter %	High calorific value KJ/kg
High volatile coal	5.81	18.16	3.95	26340
Low volatile coal	15.44	53.91	21.11	10870

Coal Dust Applied for Clinker Burning

Water content %	Fineness + R888%	Ash %	Volatile matter %	Heating value KJ/kg
1.21	3.80	26.14	6.00	23110

Technical Data on Raw & Cement Grinding Mills

Raw Grinding Mills

Type	Wet tube mill (open circuit)
Size	dia 2.6 x 13m
Number of compartments	2
Revolutions	21.5 rmp
Power	1000 KW
Ball charge	70 t
Feed size	25 mm)
Capacity	38 t/a
Product	
Fineness	14% R88
Water content	32 - 38%

Cement Mills

Type	Tube mill (open circuit)	
Size	dia. 2.6 x 13 m	dia. 3.4 x 13.5 m
No. of compartments	2	3
Revolutions	21.5 rpm	17.5 rpm
Power	1000 KW	2000 KW
Ball charge	80 t	155 t
Feed size	25 mm)	25 mm)
Capacity	25 t/h	47 t/h

Product

Fineness	14% > R88	10% > R88

Size of Grinding Media Applied in Raw Mill

1st compartment	100 mm	16 t	40 t	
	80 mm	10 t	5 t	
	60 mm	4 t	5 t	
2nd compartment	20x40 mm	40 t	80 mm	5 t
	(	Σ 70 t	60 mm	10 t
			50 mm	10 t
			40 mm	5 t
			20x40	65 t

Total 155 ton

Planned and Actual Consumption of
Raw Material Components/kg Clinker

		Planned kg/t clinker	Actual (1985) kg/t clinker
1.	Raw materials		
	Limestone	1350	1361.9
	Clay	211	189.9
	Silica	36	45.3
	Iron ore	32	36.6
	CaF_2	14.5	14.4
	Total	1643.5	1648.1
2.	Coal Consumption	314.1	320.2
	Low volatile	267	284.5
	High volatile	47.1	35.7
3.	Bricks consumption	4	4.6
	Chamot	1.5	2.5
	Chrome magnesia	2.5	2.1
4.	Gypsum	36	36.4

5.	Grinding media	1.8	1.5
	Ball	0.8	0.6
	Cylpebs	1.0	0.9
6.	Power consumption per kg cement	123 kWh/t cement	125.2 kWh/t cement

Water analysis

Turbidity	3
pH	7.6
Chloride	20 PPM
SO_4	12.5 PPM
Total hardness	max.10
At certain period of time the total hardness	max. 7.5

Quarry Equipment

	Number	Type and supplier	Capacity Nominal	Actual	Annual working hrs in 1985	
Drilling machines	6	Made in DPRK	1 m/h	1.1 m/h	282	9.4
Shovels	3	Made in DPRK	4 m^3	4 m^3		
	2		3 m^3	3 m^3	585	1.95 2 h/d
	8		1 m^3	1 m^3		

Transport Equipment from Extraction Sites to Factory

	Number	Raw materials transported	Type and supplier	Capacity Nominal	Actual	Annual working hours in 1985
Lorries	8	Clay stone	Made in DPRK	25 t	20 t	3432 x 8
Railway	8	Limestone	Made in DPRK	1800 t	600 t	5148 x 8 300 days 17.16 h/d

Crushing Equipment

	Number	Raw materials transported	Type and supplier	Capacity Nominal	Actual	Annual working hours in 1985
Jaw crusher	1	Limestone	Made in USSR	250 m^3/h	300t/h	4131 13.77 h/d
Hammer "	4	Limestone	Made in USSR and DPRK	150 t/h	150t/h	4131 13.77 h/d
Jaw crusher	1	Gypsum	Made in DPRK	50t/h	40t/h	1800 6 h/d
Jaw crusher	2	Limestone	Made in DPRK	250 m^3/h	300t/h	Under construction in the new L.S. Quarry (16KM from the factory)

Power consumption of crushing department: 2.8 KWG/t L.S. - 1985 for Jaw and Hammer Crushers.

Capacity of the Raw Material Storage for Older Two Kilns

1st Storage

No.		Dimensions		Consumption 1 day/6 kilns	Days	Remarks
1	Limestone	162 x 33	38520 t	5840 t	6.6	For all kilns
2	Claystone and sand	42 x 33	10250 t	1070 t	9.6	For all kilns
3	Iron ore	24 x 33	4080 t	138 t	29.6	For all kilns
4	Fluorspar	12 x 33	1910 t	63 t	30.3	For all kilns
5	Clinker	99 x 33	18000 t	1270 t	14.17	For two kilns (old)
6	Slag	24 x 33	2720 t	130 t	20.92	For two kilns (old)
7	Gypsum	12 x 13	700 t	45 t	15.55	For all kilns (old)
8	Coal	126 x 33	16830 t	1380 t	12.195	For all kilns
	TOTAL	489 x 33				

Capacity of Clinker, Slag, Gypsum Storage for 4 Kilns including the new

2nd Storage for Clinker, Slag & Gypsum

No.	Material	Dimensions	Capacity t	Daily consumption	Days	Remarks
1	Clinker	114 x 33	34700	3050	11.37	For 5 cement mills
2	Slag	21 x 33	2350	350	6.7	For 5 cement mills
3.	Gypsum	9 x 33	2300	95	24.2	For 5 cement mills
	Total	144 x 33				

Raw Material Grinding Mills

	Mill No.1	Mill No.2	Mill No.3	Mill No.4	Mill No.5	Mill No.6	Mill No.7	Mill No.8	Mill No.9
Type of mill (Manufactured locally)	w/o dryer	w/o dryer	w/o dryer	w/o dryer	w/o dryer	w/o dryer	w/o dryer	w/o dryer	w/o dryer
Length (m)	13	13	13	13	13	13	13	13	13
Diameter (m)	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6
Speed rev./min	21.5	21.5	21.5	21.5	21.5	21.5	21.5	21.5	21.5
Nominal capacity dry ton/h	40	40	40	40	40	40	40	40	40
Actual production dry ton/h	35.7	36.3	37.5	36.3	35.3	35.9	35.8	(under construction)	
Mill motor in kW	850	850	850	1000	1000	1000	1000	1000	1000
Dimension of mill compartment in metres and charges									
1	6.5 m 30 t	6.5 m 30 t	6.5 m 30 t	6.5 m 30 t	6.5 m 30 t	6.5 m 30 t	6.5 m 30 t	6.5 m 30 t	6.5 m 30 t
2	6.5 m 40 t	6.5 m 40 t	6.5 m 40 t	6.5 m 40 t	6.5 m 40 t	6.5 m 40 t	6.5 m 40 t	6.5 m 40 t	6.5 m 40 t
Power consump./t produced in kWh/t (including slurry silos and basin pumps)	22.89	22.89	22.89	22.89	22.89	22.89	22.89	(under construction)	
Consumption of grinding media and liners per tonne produced	0.63kg/t								
Annual production in 1985	216243	210790	193115	197244	200856	218811	204869	-	-
Working hours	6057	5806	5149	5434	5690	6095	5723	-	-

Working days/year as average for all mills are 272 days/year (average 5708 h/y)

Raw Mix Preparation Equipment from Mills to Silos

	Number	Type and supplier	Capacity Nominal	Actual	Annual working hours in 1985
Compressors (applied for all the cement plants machinery)					
100 m^3/m	3	Made in G.D.R.	100	100	5840 19/47 h/d for each compressor
40 m^3/m	2	Made in DPRK	40	40	4500 15 h/d/unit
20 m^3/m	4	"	20	15	2250 7.5 h/d/unit
Slurry silo	14		450m^3	400m^3	6300 m^3
Slurry silo	34		400m^3	400m^3	1200 m^3
Slurry basin	6		2500m^3	2500m^3	15000m^3
Pump	6	Made in DPRK	80m^3/h	80m^3/h	4380 14.6h/d

Raw Mix Transport Equipment from Slurry Basins to Kilns

	Number	Type and supplier	Motor power in KW	Capacity m^3/h	m^3/h
Pumps	6	Made in DPRK	75	80	-

Power Consumption of kilns Department

Type of energy	1984	1985
KWH/t-clinker	20.83	21.47
Fuel (tons)		303996 t/y

Kiln Department

	Kiln No.1	Kiln No.2	Kiln No.3	Kiln No.4	Kiln No.5	Kiln No.6
Manufacturing process	Wet	Wet	Wet	Wet	Wet	Wet
Supplier	USSR	USSR	DPRK	DPRK	DPRK	DPRK
Daily clinker production (t/d)	600	600	840	840	840	840
Length of the kiln and of its different diameters (m)	3.6/3.3/ 3.6x150	3.6/3.3/ 3.6x150	4x150	4x150	4x150	4x150
Fuel consumption(kg/t cl.)	320kg/t	320kg/t	320kg/t	320kg/t	320kg/t	under construction
Brick lining consump.(kg/t)	4.6 kg/t					
Filter	E	E	E	E	E	E
Cooler	G	G	G	G	G	T
Clinker transport to storage	Bucket conveyor					
Annual rated capacity (t)	200000	200000	250000	250000	250000	250000
Annual production (t)	175686	177167	213886	193417	157939	-
	6506 h	6814 h	6663 h	5988 h	4875 h	

Cement Mills

	Mill No.1	No.2	No.3	No.4	No.5	No.6	No.7	No.8
Type of mill	Open	Open	Open	Open	Open	Open	Open	Open
Length (m)	13	13	13	13.5	13.5	13	13	13.5
Diameter (m)	2.6	2.6	2.6	3.4	3.4	2.6	2.6	3.4
Speed (rev/min)	21.5	21.5	21.5	17.5	17.5	21.5	21.5	17.5
Nominal capacity (t/h)	25	25	25	60	60	25	25	60
Actual production (t/h)	24.7	24.3	24.9	45.4	47.2	24.6	24.7	--
Dimensions of mill compartments (in metres and charges)								
1	5.5m30t			3.5m	45t	5.5m	30t	
2	7.5m50t			2.7m	30t	7.5m	50t	
3				6.5m	80t			
Power consumption per tonne produced (in kWh/t)	42.47							
Consumption of grinding media and liners per tonne produced (kg/t)	0.54							
Annual capacity (t/a) (in thousands)	150	150	150	350	350	150	150	350
Annual production (t/a) (in thousands)	133	122	144	231	201	101	108	-

Specifications of Cement Manufactured

	Year	Prod. (1,000 t)	KWh/t	Fineness Mesch (+88u)	Fineness Surf. (Bl.)	Soundness	SO_3	Compressive strength 3	7	28	Bending 3	7	28
Normal Portland Cement	1985	904	125.2	7.87	2800	1.2	1.68	80	148	230	26	42	59
Low-heat Cement	1985	10	125.5	7.57		1.0	1.54	73	135	215	25	40	57
Slag Portland Cement	1985	156	111.9	7.25		0.8	1.50	75	135	201	23	38	55

Cement : Sand : Water
1 : 2 : 0.65

Packing and Dispatching Department

No. of packing machine	Type and supplier	Capacity (t/h)
No.1	Stationary (Made in Japan)	60
No.2	"	60
No.3	"	60
No.4	"	60

Dispatched Bulk Cement

Packed Cement	:	Bulk Cement
50%		50%

Ply 5 or 6 plies

Cement Silos

Serial No.	Quantity	Capacities in tonnes
1	10	3000 x 10 = 30000
2	6	3000 x 6 = 18000
3	6	3000 x 6 = 18000
4	6	3000 x 6 = 18000
Total		84,000

Transportation Equipment from Cement Silos to the Packing Section

Type	Quantity	Type and supplier	Capacity t/hour
Bucket Elevators	4	Made in DPRK	100
Screw Conveyors	4	"	100
Air Slides	4	"	60

Transport Equipment from Cement Mills to Cement Silos

Type	Quantity	Type & supplier	Capacity t/hour
Pneumatic Pump (Sera Pump)	2	Made in DPRK	80
" "	6	Made in USSR and DPRK	40

Power Consumption of Clinker Grinding Department

Type of energy	1984	1985
Electric kWh/t	42.59	42.47

Annex III

EXCERPTS FROM THE DRAFT TERMS OF REFERENCE FOR CONTRACT SPECIFICATION

Subject: Feasibility study for conversion of wet-process cement plant (Democratic People's Republic of Korea)

Background information

The Democratic People's Republic of Korea has more than 20 years' experience in the cement industry and has, over the last 15 years, become increasingly independent in the development and building of cement plants.

The present total cement-production capacity is about 10 million tonnes per year from a number of factories. Among these, dry-process installations count for nearly half of the yearly output.

The second largest cement factory (8th February Cement Joint Complex) is a wet-process plant presently operating with five kilns and with a sixth kiln under construction. When kiln number six goes into operation, the total wet-process capacity will be 1.4 million tonnes and it is now being considered whether to convert some and possibly all the production lines to a less energy-intensive production process. The Government has, therefore, requested technical assistance and UNIDO has fielded a cement consultant, who was expected inter alia, to:

> "analyse the existing facilities and technological process used in the cement industry, and make recommendations on how to improve the overall performance in the cement industry in order to obtain a higher quality of cement products as well as examine the possibilities of converting existing wet-process cement plants into dry-process plants, and draft project documentation for a conversion feasibility study".

***

(Details of the cement consultant's mission are given in the Introduction to this report.)

The main recommendations of the expert in relation to the wet/dry conversion can be summarized as follows:

1. The slurry (liquid raw mix), filter dusts, fuel (coal), and water have to be carefully studied and tested before selecting the filtration system.

***

In this connection, the expert specifically states that the contractor must carry out all the suggested laboratory tests and must afterwards subject the slurry to a pilot-plant test on site to confirm the findings.

The contractor will guarantee that the existing equipment is in good working condition.

2. All proposals estimates, calculations, specifications and design proposals have to be rechecked and re-evaluated by the consulting company selected to make the conversion feasibility study for UNIDO. It must be kept

in mind that the training of local experts in preparation for operating the proposed new equipment, is as important as the conversion feasibility study itself.

The supplier should be requested to submit his final conversion proposal to the Government and should confirm his own responsibility to carry out the conversion and to guarantee the increased production figures as well as the energy saving of at least 30 per cent.

(The expert's recommendations are given in full following the introduction to the main report.)

The above recommendations are incorporated into the following.

Scope of the subcontracting services

General conditions

1. The scope of the work to be done is described below. Each subparagraph represents a specific area of an integrated conversion feasibility study of which the findings from one subject lead to the next until the feasibility study is complete.

(a) Raw material evaluation.

(b) Evaluation of the existing installations.

(c) Drawing up of conversion proposal(s).

(d) Preparation of a list of the new equipment required, with technical specifications and estimated costs.

(e) Preparation of a list of modifications required to the existing installations, with specifications, material requirements and costs.

(f) Preparation of a workplan for the conversion with an indication of requirements in engineering support, manpower, tools and equipment as well as cost estimates.

(g) Preparation of a plan for training local personnel before, during and after the conversion, with an estimation of training costs.

(h) Preparing a techno-economic feasibility study, presenting all the investment costs necessary for the selected conversion, including:

- Mechanical equipment
- Electrical equipment
- Spare parts
- Insurance and transportation
- Erection and start-up costs
- Civil engineering work
- Estimated reserve for unforeseen costs

This study should also give all normal operating costs (fixed and variable) expected for the converted production line(s), thus making possible a feasibility calculation comparing the situation before and after the conversion and examining the return on the investment in financial terms as well as in energy terms.

(i) Elaboration of tender specificat' « for international competitive bidding for the execution of the version, to include local production of as much equipment as possible and training of personnel.

2. The subcontractors selected for the execution of the conversion feasibility study will submit monthly progress reports and invoice UNIDO monthly in accordance with their delivery and the agreed work plan.

3. All companies interested in assisting UNIDO with this conversion feasibility study should note that they are expected to use their own judgement as to what is necessary for a comprehensive evaluation of the situation and as to how to carry out the conversion feasibility study.

If, for example, a company finds it logical to examine the possibility of a full conversion to dry process and compare this with the presently-proposed conversion to a semi-wet process, they should include this in their offer to UNIDO and the Government. It would be possible to submit two alternative proposals, one examining only the conversion to a semi-wet process, and another where both the possible conversions to semi-wet and to dry processes are examined and compared.

4. The present terms of reference should be used only together with the report DP/ID/SER.B/558 based on the work of A. R. Marei, UNIDO consultant, and dated 31 July 1986, since the additional information, ideas, and calculations contained therein may be of interest to the potential subcontractor.

Details of requirements under (1) above

(a) Raw-material evaluation

During the preliminary fact-finding, it was stated by the technical personnel of the 8th February Cement Joint Complex that the raw-material reserves available are sufficient to serve the cement plant, even with the expected increase in production, for at least 50 more years.

The expert, who undertook the preliminary fact-finding, specifies the tests which must be carried out on the raw materials in the case of a conversion to the semi-wet process.

(For the full text of these specifications, see chapter IV in part one of this report)

***

(b), (c), (d), (e) and (f)

- Evaluation of the existing installations
- Presentation of conversion proposals
- List of new equipment
- List of modifications required
- Work plan

For a company experienced in the engineering involved in converting cement plants from wet to dry or semi-wet processes, the above titles speak for themselves, and the details of the findings are important elements of the conversion feasibility study.

(See also the information given on these subjects in chapter V of part one of this report.)

In this connection, it should be noted that the potential subcontractor should judge the situation from his experience and propose what he finds necessary, whether more or less than that quoted in the report.

Also, in this part of the conversion feasibility study, the potential subcontractor is expected to make his own estimates for the scale and the cost of the evaluation and development work he finds necessary. The selected subcontractor will be expected to work out all aspects of the design which he has proposed and on the basis of which he was selected for the job.

(g) Training of personnel

Training should, as far as possible, be part of the subcontracting services. On the one hand, this makes it possible to make full use of local experts to prepare for and later on to carry out the proposed conversion at the lowest possible cost per tonne of additional capacity installed. On the other hand, it also initiates and promotes as much knowledge as possible about conversions in general and develops a suitable degree of self-reliance in managing conversions of other wet-process plants in the Democratic People's Republic of Korea.

The subcontractor should, therefore, in his own proposal, include training of and co-operation with national experts, including the training to be done by the subcontractor selected to manage the conversion as well as the estimated costs of the training.

(h) Techno-economic feasibility study

The subcontractor should, in accordance with his own experience and in accordance with the UNIDO manual for feasibility studies, carry out a feasibility study on the suitability of the proposed conversion from an economic as well as a technical point of view.

The conversion study should compare the present operation (wet process) with the operation of the converted plant. If the subcontractor finds it logical to examine more than one conversion possibility in order to find the most economic conversion among several possibilities, he should feel free to propose his services for several versions of new plant processes, up to a complete modernization with precalciners or any other solution he can justify.

(i) Elaboration of international tender specifications

Based on the findings from the conversion feasibility study (including the pilot-plant tests), the subcontractor is expected to elaborate tender specifications for the most economic type of conversion and to give all the necessary conversion plans and details, including training, so that competitive bidding can be organized among interested contractors. The subcontractor should also give guidelines on how to evaluate the incoming offers or should offer their own services not only to elaborate the tender specification, but also to assist UNIDO and the Government to evaluate the incoming tenders and to negotiate and select a final offer.

Estimated time and cost requirements

The complete conversion feasibility study including training of counterpart personnel is expected to require, as a minimum, the following services from a consulting company:

	Time required (man/months)
(a) Raw-material evaluation	
Geologist	1
Chemical process engineer	1
(b) Evaluation of existing installation	
Mechanical engineer	1
Electrical engineer	1
Process engineer	1
Civil engineer	1
(c) Elaboration of conversion proposal(s)	
Mechanical engineer	2
Draughtsman	2
Electrical engineer	1
Process engineer	1
(d) Preparation of list of new equipment	
Mechanical engineer	0.5
Electrical engineer	0.5
(e) Preparation of list of modifications	
Mechanical engineer	0 5
Electrical engineer	[illegible]
Process engineer	0.5
Civil engineer	0.5
(f) Preparation of work plan for conversion	
Mechanical engineer	0.5
Electrical engineer	0.5
Civil engineer	1.5
(g) Preparation of training plan	
Mechanical engineer	0.5
Process engineer	0.5
Electrical engineer	0.5
(h) Elaboration of techno-economic feasibility study	
Economist	3
Engineer	3
(i) Elaboration of tender specifications	
Economist	0.5
Legal adviser	0.5
Engineer	1.0
Total man/months	26.5

Estimated costs at $8,000 per m/m Total $US 212,000

Bibliography

Bates, Robert. Geology of the industrial rocks and minerals. New York, Harpers, 1959.

Duda, Walter H. Cement data book. Wiesbaden, Bauverlag, 1976.

Jensen, Oliver. Cost of new cement plants and conversions. Beijing, Interregional Seminar on Cement Technology, 1980.

Labahn, O. and B. Kohlhaas. Cement engineer's handbook. Wiesbaden, Bauverlag, 1983.

Lea, F. M. The chemistry of cement and concrete. London, Arnold, 1970.

Marei, Abd El Rahim. Selecting the proper raw materials is the key consideration to the success of any cement project as well as the decisive factor in energy conservation. Paper submitted to the 4th meeting of the Cement Arab Union, Tripoli, December 1984.

Peray, Kurt E. and Joseph J. Waddell. The rotary cement kiln. New York, Chemical Publishing, 1972.

Process technology of cement manufacturing. Plymouth, Macdonald and Evans, 1979.

Takemoto, K., Y. Kukuda and Y. Ueda. Low-calorie coal combustion successfully used in RSP kiln. Paper prepared for the All India Seminar on Cement and Allied Building Materials, Cement Research Institute of India, 3-6 January 1984.

United Nations Industrial Development Organization. Clinker coolers and their effect on energy savings. [Prepared by Abd El Rahim Marei] 1983.*

______ Fuels "types, characteristics, uses" and the different systems applied in cement industry to conserve energy. [Prepared by Abd El Rahim Marei] 1983.*

______ Gypsum mining and processing of marble and silica sands. [Prepared by Abd El Rahim Marei] 1977.

______ Influence of the quality of cement raw materials on energy savings. [Prepared by Abd El Rahim Marei] 1983.*

______ The influence of using hydraulic materials in cement production and energy savings. [Prepared by Abd El Rahim Marei] 1983.*

______ The trip of cement raw materials (mixes) in rotary kilns. [Prepared by Abd El Rahim Marei] 1985.

______ Transformation of existing cement plants to reduce energy consumption. [Prepared by Abd El Rahim Marei] 1983.*

______ Additives and their role in energy conservation. [Prepared by Abd El Rahim Marei] 1983.*

______ The role of precalciners in energy savings. [Prepared by Abd El Rahim Marei] 1983.*

______ The world-wide strategy of energy conservation. [Prepared by Abd El Rahim Marei] 1983.*

---

*These papers were prepared in Arabic for the Seminar on the Role of Governments and Personnel in Energy Saving in the Cement and Lime Industries, Benghazi, 10-15 December 1983.