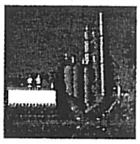
Cement Plant Operations Handbook
5th Edition



Cement Plant Operations Handbook

for Dry-Process Plants

Fifth Edition
October 2007

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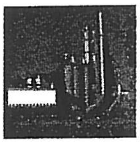
Tradeship Publications Ltd



Acknowledgements

I would like to thank the legions of grey men with whom I have worked and from whom I have learne over the years. Specifically I would like to express my appreciation to James Post, Andrew Jackura, Arthur Chin-Fatt, Michael McCabe and Peter Bergmann who have made significant contributions as the text developed. I am indebted to David Hargreaves of International Cement Review for his continued interest and support in publishing successive editions of the Handbook.

The author, however, retains proprietary rights over all expressions of ignorance or opinion.

Publisher: David Hargreaves, International Cement Review
Author: Philip A Alsop
Editorial: Muriel Bal
Design: Rachel Burgess
Advertising: Gary Morton

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Printed by Emirates Printing Press, UAE.

Preface

For brevity, the objective has been constrained, and whole areas of operations technology and management have been omitted as being inappropriate to address in so limited a compass. It is also appreciated that regulations, specifications, and even operating practices are not universal, and our observations should be discounted accordingly.

The scope attempted comprehends:

- A consideration only of cyclone preheater kiln technology, which comprises more than 80% of world production and virtually all kilns installed since 1970.
- The use only of metric units.
- A review of major plant sub-systems with a proposed list of data which should be available to plant and corporate management, and some suggestions regarding problem areas and possible solutions.
- A summary of cement types and concrete problems.
- Reference to ASTM and EN specifications for cements and for standard methods.
- A collection of process formulae.
- A selection of reference data and notes.
- An outline of plant assessment and plant valuation.
- References to review articles and a limited bibliography.
- Addresses of pertinent organisations.

Philip Alsop
Lonan, Isle of Man.
October 2007

Obiter Dicta

Public discussion of the environmental issue is frequently characterised more by emotion than reason. Much of it proceeds as if the issue is pollution versus no pollution, as if it were desirable and possible to have a world without pollution. That is clearly nonsense.

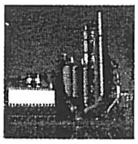
In the case of pollution, the devil blamed is typically "business", the enterprises that produce goods and services. In fact, the people responsible for pollution are consumers. They create, as it were, a demand for pollution……Ultimately, the cost of getting cleaner air and water must be born by the consumer. There is no one else to pay for it. Business is only an intermediary.

Milton Friedman

There are known knowns. These are things that we know that we know. There are known unknowns. That is to say, there are things we know that we do not know. But there are also unknown unknowns. There are things we don't know we don't know.

Donald Rumsfeld

Human respiration is estimated to produce 450 litres CO_2 per person per day equivalent to some 2 billion tonnes/year or about 8% of total CO_2 production. Given that the other 92% is also related to population, this may suggest an effective course for saving the planet.



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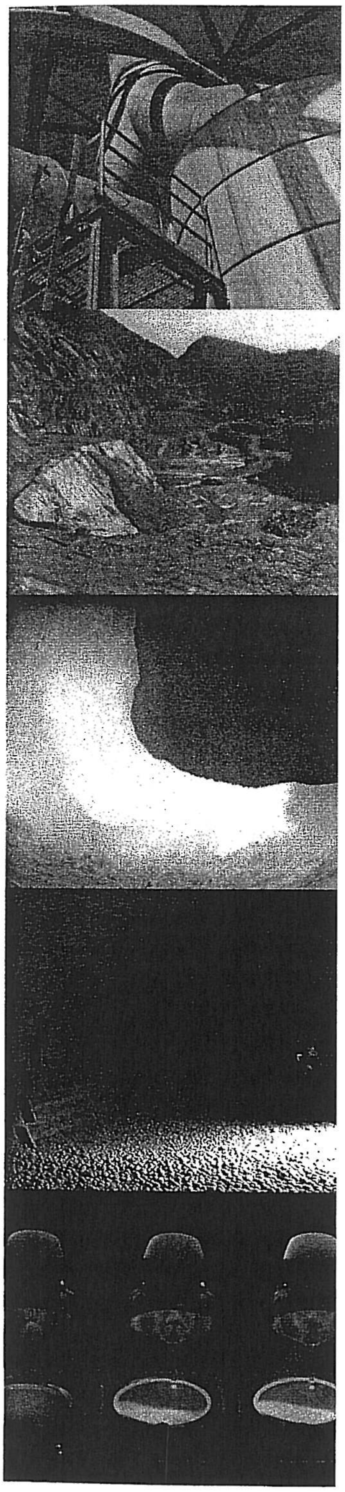

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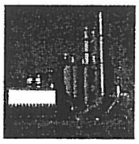

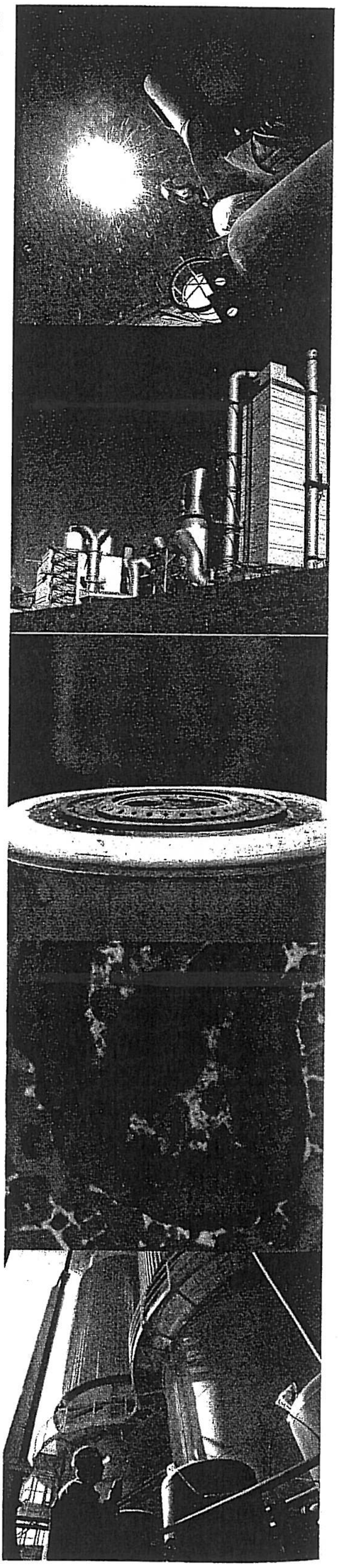

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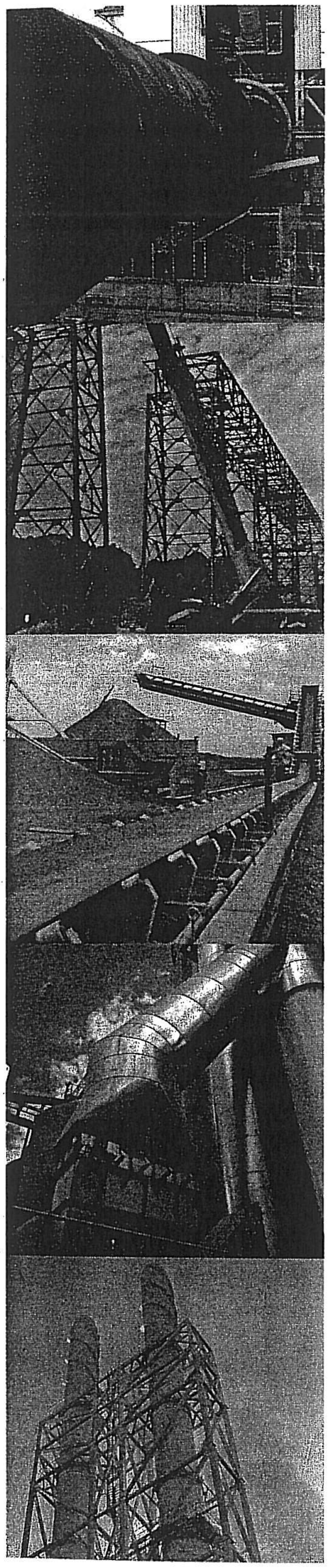

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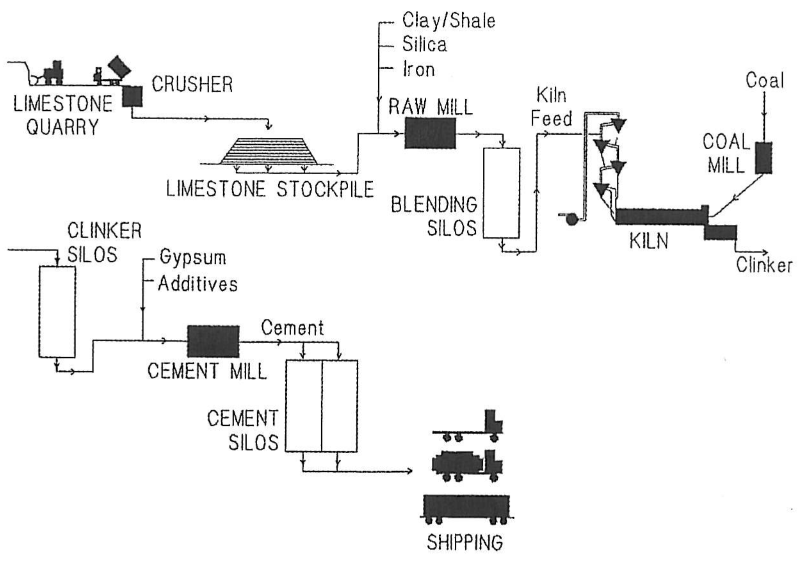
1. Introduction

Cement is "a substance applied to the surface of solid bodies to make them cohere firmly" or, more specifically, "a powdered substance which, made plastic with water, is used in a soft and pasty state (which hardens on drying) to bind together bricks, stones, etc in building" (SOED). ***Portland cement*** **is a calcined material comprising lime and silicates which is mixed with sand and stone and, upon hydration, forms a plastic material which sets and hardens to a rock-like material,** ***concrete.*** **Confusion between cement and concrete is endemic among the uninitiated.**

Portland cement is manufactured in a series of processes which may be represented as shown:

Figure 1.1. – Cement Plant Schematic Process Flow



Limestone (calcium carbonate) and other materials containing appropriate proportions of calcium, silicon, aluminium, and iron oxides are crushed and milled to a fine flour-like *raw meal*. This is heated in a kiln, firstly to dissociate calcium carbonate to calcium oxide with the evolution of carbon dioxide, and then to react calcium oxide with the other components to form calcium silicates and aluminates which partially fuse at material burning temperatures up to 1450°C. The reaction products leave the kiln as a black nodular material, *clinker*. The clinker is finally inter-ground with a small proportion of gypsum (to control the rate of hydration) yielding a fine product which is *cement*.

1.1. History of Cement Manufacture

The ancient history of hydraulic mortars is extensive but becomes appreciable with the widespread use of mixtures of natural pozzolans and burned lime by both Greeks and Romans. The Pantheon in

Rome is the only perfectly preserved building from this period; it was constructed in 27BC and rebuilt 117-125AD and is of pozzolan-lime concrete with an unsupported dome spanning 45m. Portland cement was developed in the 19th century and is so called due to its resemblance in colour and character to the naturally occurring stone of Portland Bill, off the south coast of England.

Following are some of the more significant dates in the development of Portland cement manufacture (Alsop, ICR, 7/2002, p. 37).

1824	Aspdin patented Portland cement.
1845	Isaac Johnson recognised the significance of high temperature to produce C_3S. This was the first cement as we know it.
1880s	Gypsum first added for set control.
1885	Ransome patented the rotary kiln.
1891	The continuously fed ball mill was patented.
1928	Introduction of the grate preheater kiln (Lepol) by Polysius provided the first major improvement in thermal efficiency from the previous long, wet kilns.
1930s	Roller mill first applied to cement manufacture; rapid development after 1960.
1930s	Introduction of the roll press; rapid development after 1980.
1932	Patent of the cyclone preheater kiln with commercial development by KHD dating from 1951.
1937	Introduction by Fuller of the grate cooler.
1950s	Introduction of mechanical separators.
1960	Introduction by KHD of the kiln bypass to allow use of raw materials with high volatiles contents.
1966	Introduction of precalcination which was initially air-through riser-firing.
1970s	Introduction of high-efficiency separators.
1973	Introduction by IHI of the flash calciner with tertiary air duct.

The world consumption of Portland cement has grown:

	Cement Demand	*World Population*	*Per Capita*
1910	30Mta	1.5bn	20kg
1925	150Mta	2.0bn	75kg
1940	400Mta	2.2bn	180kg
1955	600Mta	2.7bn	220kg
1974	1000Mta	4.0bn	250kg
2000	1500Mta	6.0bn	250kg
2005	2300Mta	6.5bn	350kg

This shows a long-term growth rate of 2-3% per year accelerating to a little under 4% for the decade up to 2000 (Portland Bill, ICR, 7/2001, p. 92). Consumption since 2000 has increased by nearly eight per cent per year (ICR Global Cement Report, 7th Ed., p. 8)

Total international cement trade was 122Mt in 2001 with 80Mt sea-borne (Thomas, WC, 11/2003, p. 25). As GDP per capita increases above $3000, cement consumption increases substantially; above $15,000 consumption tends to reach a plateau (Betts, WC, 11/2003, p. 25). World consumption is projected at 3060Mt per year by 2020 (ibid).

Assuming an average selling price of $50/t, the cement industry worldwide has revenues about one third of Walmart's.

2. Raw Materials

2.1. Raw Materials

The composition of Portland cement varies from plant to plant due both to cement specifications and to the mineralogy of available materials. In general, however, an eutectic mix is sought which minimises the heat input required for clinkering and the total cost of raw materials, while producing a cement of acceptable performance.

An approximate analysis for raw mix on ignited basis, or for clinker, is:

CaO	65-68%
SiO_2	20-23%
Al_2O_3	4-6%
Fe_2O_3	2-4%
MgO	1-5%
Mn_2O_3	0.1-3%
TiO_2	0.1-1%
SO_3	0.1-2%
K_2O	0.1-1%
Na_2O	0.1-0.5%

Note that, with a substantial proportion of the raw mix being $CaCO_3$, heating either in a kiln or in a laboratory furnace evolves some 35% by weight as CO_2; this results in a requirement of approximately 1.5t of raw materials to produce 1t of cement, and also requires that analytical data be clearly distinguished between "raw" and "ignited" basis.

Cement manufacture begins in the quarry with the mining of raw materials, primarily limestone, and their transport to the plant. Quarrying may be effected either by ripping or by drilling and blasting. In either case the recovered material needs to be of consistent quality and the necessary level of mine planning is facilitated either by a bore hole survey throughout the mining area or by assaying the cuttings from blast-hole drilling. Quarry management has been greatly facilitated of late by introduction of Global Positioning (GPS) technology (Mercy, ICR, 8/2001, p. 31).

Cement mixes vary from "cement rock", a single component which, as mined, contains appropriate proportions of all the required minerals, to four- or five- component mixes comprising one or two grades of limestone, a shale or clay, and one or more additives to augment SiO_2, Al_2O_3 or Fe_2O_3 levels. Kiln feed typically contains 78-80% $CaCO_3$ so that limestone can only fall close to this level to the extent that it also contains the other ingredients. It is essential to have sufficient flux (Al, Fe, Mg, F) to promote fusion in the kiln, but MgO should not exceed 4-5% or the cement may be expansive. Excess alkalis (K, Na) affect both kiln operation (build-ups) and product quality (alkali-aggregate reactivity). Excess S causes kiln build-ups and limits the addition of gypsum which may result in setting problems. The stoichiometric ratio of alkalis to sulphur is normally kept between 0.8-1.2. Excess Cl (>0.015%) in kiln feed causes serious build-up problems for preheater operation.

Materials, as mined, therefore, are typically proportioned:

Limestone (CaO)	85%
Shale or clay (SiO_2, Al_2O_3 & Fe_2O_3)	13%
Additives (SiO_2, Al_2O_3 or Fe_2O_3)	<1% each

Normally, cement plants are located on limestone deposits while shale or clay is sufficiently ubiquitous for most plants to mine this locally. Limestone comprises 1.5% of the earth's crust (Kolb, ZKG, 5/2001, p. 262). Additives are usually brought in, albeit in small quantities.

The conventional explosive used for limestone quarrying is ANFO (ammonium nitrate activated with ca. 5% of fuel oil). Usage averages about 200g/t with considerable variation.

Mining plans are developed according to the geology of the materials. If the limestone is not homogeneous, it may be necessary to blend rock from different areas in order to maximise recovery, and it may also be necessary to mine selectively in order to avoid low grade material or problems such as alkalis. Mining and hauling are commonly monitored by:

Blasting	grammes explosive /tonne rock
Stripping ratio	tonnes waste removed /tonnes used rock
Loading	tonnes/hour of equipment & loader availability (% of required hours)
Hauling	tonnes/hour per truck & truck availability

All production and inventory records are most conveniently kept in dry tonnes but moisture levels of mined, hauled, and crushed rock must be used for assessing equipment efficiency.

Apart from chemistry, grindability and burnability are also factors in selecting raw materials. In particular, silica additives containing large-grain quartz are very difficult to grind and can result in hard burning, high fuel consumption, and increased equipment maintenance. If quartz silica is employed it should, preferably, have a natural grain size of less than 50μ or should be ground to less than 2-3% coarser than this size.

In recent years, cement kilns have been increasingly employed to utilise industrial by-products (eg mill scale) and to dispose of industrial waste materials (eg water treatment sludge) in return for disposal fees. Such materials include:

Ca agents:	industrial lime	carbide slurry
	lime slurry	water treatment sludge
Si agents:	foundry sand	silica fume
Fe agents:	roasted pyrites	steel slag
	synthetic hematite	converter flue dust
	red mud	mill scale
Si, Al, Ca agents:	coal fly ash	metallurgical slags
	fluidised bed ash	stone working residues
S agents:	desulphogypsum	
F agents:	CaF_2 filter slurry	

Natural raw materials include minor quantities of various elements such as P, Ti, Cr and Mn. Waste materials such as lime, fly ash and slags containing calcined calcium (CaO as opposed to $CaCO_3$) are particularly attractive in avoiding the heat required for dissociating carbonate (Section B5.5.). The use of waste materials for cement manufacture has led to incorporation of a much wider range of trace elements and their effects are reviewed by Bhatty (Role of Minor Elements in Cement Manufacture and Use, PCA, 1995).

Apart from raw materials, gypsum and fuel are required for cement manufacture together with various pozzolanic materials (both natural and by-product) if inter-ground cements are produced.

2.2. Reserves

A knowledge of limestone and, to a lesser extent, shale reserves is necessary, particularly when justifying investment to increase plant capacity. Reserves are classed according to the detail in which they have been explored:

Class A or proven reserves:	Extensive drilling has confirmed quantity, mineralogy, variation, mining and legal availability.
Class B or probable reserves:	Sufficient drilling to allow presumption of quality and availability.
Class C or indicated reserves:	Widely spaced drilling gives extent and some knowledge of quality.
Class D or inferred reserves:	Initial exploration and consideration of geology allow general assumption.

2.3. Crushing

Primary crushers should be capable of accepting shot rock with the minimum of wastage or of preliminary size reduction. Typically feed should be less than 120cm and, either the feed hopper should be protected by an appropriate grizzly, or a hydraulic breaker may be installed to reduce oversized rock. Commonly there are primary, secondary and, occasionally, tertiary crushers in series. Most crushers are operated in open circuit though, frequently, they are also preceded by a screen or grizzly to bypass fine material direct to product.

Crushed rock should ideally be -20mm for feed to ball type raw mills. For roller mills and roll presses, the feed size can be roughly related to roller diameters (Dr):

Roller Mills - Easy-grinding materials <4% +0.06Dr; <20% +0.025Dr
- Hard-grinding materials 0% +0.06Dr; <20% +0.015Dr

Sometimes a simplified rule-of-thumb of not more than 5% of table diameter is used for materials of average grindability.

Roll Press - Maximum feed should not exceed 0.05Dr

Alternatively, a limit of twice the roll gap is used (Liedtke, WC, 9/2000, p. 41).

Location of the crusher may be either at the quarry or the plant and is largely a function of haulage vs conveying costs (Heur, WC, 11/97, p. 34). Mobile crushers are common in aggregate quarries but rare for cement (RP, 9/1994, p. 31).

Hoppers and silos may be designed for:

Mass flow where material at every point is in motion during discharge. This requires smooth walls and steep sides with no abrupt transitions.

Funnel flow where material from the top surface only discharges through a vertical channel above the outlet while surrounding material remains static. This occurs particularly in squat silos and hoppers with insufficiently steep walls. It should be noted that the angle of repose of free material is usually inappropriate for surface slope design. In particular, the handling, crushing, conveying, and storage of wet clay materials prior to drying are prone to difficulties (Maynard, GCL, 4/2001, p. 12).

Hopper and silo flow processes are reviewed by Bresler (WC, 2003/BMH, p. 29).

Crushers may operate by compression or by impact. Compression machines comprise single and double jaw crushers and gyratory crushers (Figure 2.1.).

> **Double Toggle Jaw Crushers** have a simple compression motion with jaw angle of 15-20°. They are effective for hard abrasive materials with low (<5%) moisture and give a reduction ration of about 6:1.
> **Single Toggle Jaw Crushers** also have a measure of vertical jaw motion which adds attrition to compression. Sticky materials can be handled but wear rate is increased.
> **Gyratory Crushers** operate by pressure between a gyrating cone and a stationary or spring-loaded crushing ring. Hard, abrasive materials can be handled with reduction ratios of about 5:1.

Compression type crushers normally produce a cubic product with a low proportion of fines; being choke fed, plugging will result with feed moisture in excess of about 5%. Wear is low but reduction ratios are usually less than 8:1.

> **Double Roll Crushers** such as that supplied by MMD (AC, 11/1998, p. 38) employ a combination of shear and tensile forces. Counter-rotating rolls with low speed and large teeth can accommodate both hard rock and wet, sticky clay. The rotor and tooth configuration also acts similarly to a wobbler feeder allowing under-sized material to pass between the rotors, thereby avoiding the generation of excessive fines.

Impact machines may be either hammer mills or impact crushers. These are usually preferred for limestones with quartz contents of less than 10%. Wear is greater than for compression crushing and there is a larger proportion of fines, but moisture contents up to about 12% can be accepted and reduction ratios up to 80:1 are possible.

> **Hammer Mills (Single & Double Horizontal Shaft)** operate by material falling into the circle of the rotating hammer(s) and being impacted both by the hammer(s) and the breaker plate. The feeder elevation and, therefore, the velocity with which the material enters the circle is critical; if too low, the material bounces on top of the hammers and if too high, it penetrates through the circle and can damage the rotor discs. The discharge is partially or wholly screened by grates against which secondary reduction by attrition takes place. The grate slot size governs discharge top size but this configuration requires relatively dry material to avoid plugging.
> **Impact Crushers (Horizontal or Vertical)** are similar in operation to hammer mills. Some units involve attrition but relatively wet materials can be handled.

Monitoring of crusher operation requires:

- Production rate, tonnes/hour
- Operating hours
- Involuntary downtime hours
- Feed moisture, %
- Product screen analysis

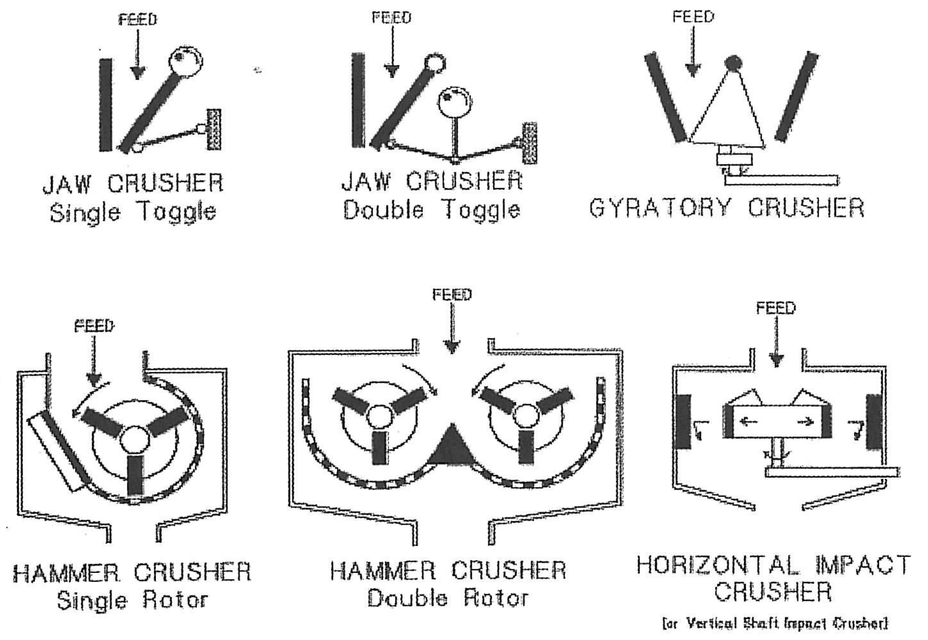


Figure 2.1. – Crushers

2.4. Drying

The handling characteristics of materials relative to moisture content vary widely. In general, drying can be performed in the raw mill for up to 15% aggregate moisture, but pre-drying may be necessary for certain materials to facilitate their handling.

Drying is commonly effected either in combined crusher-dryers, in rotary dryers which can either use dedicated hot gas generators or waste heat from kiln exhaust, or in autogenous mills. In each case, the air flows will involve dust collection while the high humidity of the used gas usually favours electrostatic precipitators over bag houses.

Monitoring includes:

- Production rate, tonnes/hour
- Operating hours
- Involuntary downtime
- Hot gas temperature, °C
- Feed moisture, %
- Discharge gas temperature, °C
- Product moisture, %
- Heat input (for dedicated heat generators), kcal/kg

Heat consumption is most conveniently calculated on equivalent clinker basis so that it may be compared arithmetically with kiln heat to assess total process heat.

If raw materials are pre-dried, they may cause serious handling problems. Steam evolution from the hot material causes build-up and can plug dust collectors. Also the dry, fine fraction is liable to flush if held in intermediate storage. There are advantages in both handling and heat consumption if raw materials can be conveyed up to mill feed and dried in the raw mill.

2.5. Pre-blending

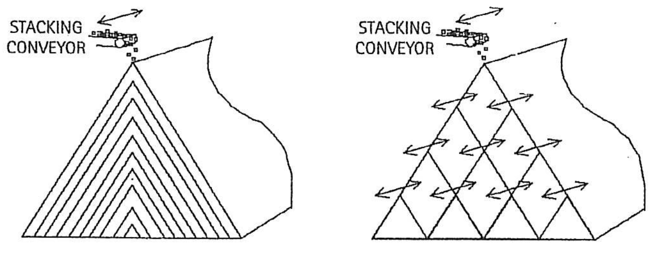
If the limestone and clay/shale are both of high consistency, or if their differences in form would result in segregation, there may be justification for handling each separately up to raw mill feed and proportioning them with the mill feeders. More commonly, material variation can be mitigated, and buffer storage more economically provided, by a mixed pre-blend pile, either longitudinal or circular. The limestone (and clay/shale if premixed) are metered and fed simultaneously to a feed conveyor. There are two major stacking systems:

Chevron – stacking in layers along a single axis with the feed conveyor sweeping back wards and forwards along the length of the pile (Figure 2.2.1.).

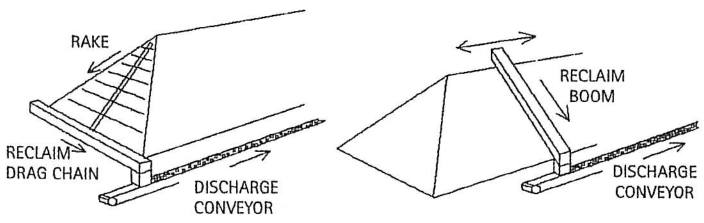
Windrow – stacking in longitudinal strips side be side and then in successive layers; this avoids the segregation which characterizes chevron stacking (Petersen, WC Bulk Materials Handling Review, 1994, p. 30) but otherwise there is little difference in blending efficiency. The windrow system requires a more complex and expensive stacking belt arrangement (Figure 2.2.2.).

Chemical analysis of the material, especially if premixed, as fed to storage is essential. This may be either by continuous sampling followed by periodic conventional analysis (Anderson & Pedersen, WC, 12/2004, p. 89; Narayanan & Hoenig, WC, 11/2004, p. 93), or by neutron activation analysis (Chidambaram & Mokrin, WC, 12/2003, p. 59; Foster & Bond, IEEE/PCA Technical Conference, 5/2005) which can continuously analyse the material flow falling though a chute or, better, travelling on a belt conveyor. The latter method has the obvious advantage of rapidity and of avoiding a very difficult sampling problem.

Figure 2.2. – Stacking and Reclaiming



2.2.1. Chevron stacking *2.2.2. Windrow stacking*



2.2.3. End reclaim *2.2.4. Side reclaim*

Pre-blend effectiveness (the ratio of estimated standard deviations [s] for feed and product) is approximately related to the number of layers [N] by:

$$\text{Blending ratio} = s_{(feed)} / s_{(product)} = \sqrt{N/2}$$

Typically a pile is built of 100-400 layers yielding a blending ratio of 3-6 for raw data and 6-15 if the variations within each reclaimed slice (which should be eliminated by normal handling) are ignored (Labahn).

Recovery to mill feed is either by end or side reclaim:

End Reclaim - various systems to scrape an entire end face with a transverse scraper at floor level moving material to a discharge conveyor (Figure 2.2.3.).

Side Reclaim - a boom mounted scraper working end to end along the side of the pile. This gives less effective blending and recovery flow rate is not constant (Figure 2.2.4.).

Two piles are normally operated with one being built while the other is recovered. Length-to-width ratio should be at least 5:1. The pre-blend pile is usually the principal buffer storage between quarry/crushing and raw milling; it should be minimally sized to maintain mill feed during the longest anticipated interruption in the supply of new rock which may be weekend shut-down of the quarry, or the time required for maintenance of crushing equipment. Total capacity of 7-10 days consumption is normal. Pre-blend piles are usually covered, both to prevent rainfall on the fine and, therefore, absorbent material, and to contain fugitive dust.

Circular pre-blends are sometimes employed where space is limited, but do not allow subsequent expansion.

Pre-blending should be monitored for compositional variation of feed and discharge to determine a blending ratio.

3. Raw Milling and Blending

3.1. Raw Milling

Raw milling, as one of the major cement process steps, must produce sufficient kiln feed meeting targets for fineness, chemical composition, and moisture to sustain required kiln production. With multiple tasks to be accomplished in a single operation, process flow and control become quite complex. Operators should have a clear understanding of both basic principles and interrelationships in order to achieve the optimum operation of the system.

Usually each raw component is stored separately in mass flow storage bins equipped with weighing and dosing systems for mix control to the grinding mill. The product from the raw mill is sampled and analysed to facilitate adjustment of feed proportions to maintain target mix chemistry. Modern on-line analytical techniques, primarily cross-belt neutron activation analysis of mill feed, allow rapid correction and greatly reduced variation of product. Any control system requires reliable and uninterrupted transportation and feeding of raw materials and this is often a serious challenge.

Drying of raw materials in the raw mill is preferred due to:

- Less equipment as compared to separate drying.
- Effective use of waste heat from preheater exhaust.
- Scrubbing of SO_2 from preheater exhaust in the raw mill.

The use of preheater gas for raw mill sweep is called compound operation. The operations of raw grinding and clinker burning become inter-dependent and constant kiln operation is required to ensure control of constant mill temperature and pressure. When the kiln is down, the raw mill must also be shut down unless an auxiliary heater is provided. When the mill is down, preheater exhaust must be diverted to a gas cooling device to maintain gas flow and temperature to the dedusting system. Gas conditioning is usually effected by a water spray tower but, if water is scarce, ambient air tempering can be used. Air to air heat exchanger is not commonly used for dust-laden preheater exhaust gas due to its tendency to build-up and plugging.

The classifier for the raw milling circuit is now predominantly of rotating cage type or the so-called high efficiency separator. While holding the gas flow constant, the product fineness is controlled by the speed of cage rotation.

The production rate of a milling system is determined by the grinding power available and the grindability of the feed material. For high moisture feed, the drying capacity of the system may become the limiting factor.

More than 80% of new raw mills are vertical roller mills, though many ball mills are still in use. Roll presses are also used, particularly in upgrading existing ball mill circuits either to increase production or to reduce specific power consumption. The presses are employed as pre-grinders with or without a disagglomerator to strip fines from the pressed cake before feeding to a ball mill. Roller mills can typically handle raw materials with an aggregate moisture of up to 15%. Ball mills, providing they are equipped with a drying compartment and are adequately air swept with hot gas (2.5-3.5m/s above the ball charge), can handle 8%. Centre discharge mills (Double Rotator) and fully air-swept mills (5-6m/s) can dry to 12-14% moisture. Normally drying is effected by ducting part of the kiln exhaust gas through the mill with inlet temperatures of up to 300°C. Obviously a high drying requirement may

be inconsistent with maximising the thermal efficiency of the kiln; generally five- and six-stage preheaters are only employed where subsequent drying by the exhaust gas is minimal. Alternatively, but more expensively, dedicated hot gas generators can be used for drying in the raw mill. Drying is also aided by heat dissipation from mill drive power which equates to approximately 1t moisture per 1000kWh.

Older ball mill systems often have silos for storage of individual raw materials. These are prone to feed problems if the material is too wet or has been pre-dried. For modern mill systems where drying is effected in the mill circuit, hoppers are generally employed.

Ball mill operation is described in more detail under finish milling (Section 5.2.).

Roller mills have a lower specific power consumption than ball mills. Loesche mills (Figure 3.1.) comprise two to four conical rollers which are hydraulically pressed onto a horizontal rotating grinding table. The roller axis is inclined at 15° to the table and, as axes of rollers and table do not intersect in the plane of the table, the relative motion involves both rolling and sliding which enhances comminution. Feed material is directed onto the centre of the table and is thrown outward by rotation under the rollers and into a rising air current at the periphery which is directed by means of a louvre ring. The air sweep passes through an integral rotary classifier; fines pass out with the air current while coarse material falls back onto the feed table.

Material drying occurs in air suspension between table and classifier. Circulating load is typically 800%. Roller mills are prone to vibration due to an unstable grinding bed. A major cause of material instability is fine, dry mill feed which can usually, be mitigated by spraying water directly onto the bed.

A recent innovation, the LV high efficiency classifier (ICR, 7/2003, p. 49), gives a higher velocity profile above the grinding table which effectively reduces the concentration of suspended material and the pressure drop across the mill. Significantly coarser classifier rejects also result in a more stable grinding bed. Mill throughput increases of 12-30% and system power savings of 1.5-5kWh/t are claimed. The technology has now been successfully applied to vertical coal mills (Nielsen, WC, 1/2002, p. 75) and to ball mill high-efficiency separators (ICR, 7/2003, p. 49). Other means of increasing roller mill capacity are described by Jung (CI, 2/2004, p. 52).

The mill is started either with the rollers lifted away from the table, or with the hydro-pneumatic system at low pressure. In grinding mode, actual metal to metal contact should be prevented by limit switches or a mechanical stop and by consistent feed. Material which is not carried upwards by the air stream falls from the table to a rejects trap, but every effort should be made to exclude tramp metal which can damage the grinding surfaces. Some designs incorporate an external circulating system to elevate and return rejects to mill feed and these deliberately reduce the gas velocity through the peripheral inlet around the mill table from 80-85m/s to 45-60m/s. The benefit of this system is a reduced mill pressure drop.

It is important that a roller mill be capable of drawing its designed power and this is controlled by adjustment of the roll pressure and of the height of the dam ring holding material on the table. Excessive rejects may be the result of too low a dam ring. The inherently high static pressure in the roller mill system requires tight control of air in-leakage; no more than 10-15% in-leakage between mill inlet and dust collector should be permitted.

Loesche mills are defined by grinding table diameter (dM) and number of grinding rollers; e.g. LM46.4 is 4.6m in diameter with four roller modules. 2+2 for the roller modules refers to two grinding rollers preceded by two small compression rollers as used for cement milling.

system power is generally some 30% lower than for ball mills.

Mill product is monitored either by continuous, on-line analysis or by laboratory analysis of hourly grab or composite samples. Computer control is employed to effect feed corrections in order to maintain the desired optimum average composition.

Raw mills are monitored by:

- Production rate, tonnes/hour
- Operating hours
- Involuntary downtime hours
- kWh/tonne (mill motor)
- % of connected power (relative to mill motor rating)
- Product fineness, -170# & -50#
- Feed moisture, %
- Product moisture, %
- Limestone, %
- Clay/shale, %
- Additives, %

Additional operating parameters required periodically include:

- Circulating load, %
- Steel usage, g/t
- Chemical analysis of +170# fraction

The coarse (+170#) fraction may be lime-rich or, more likely, silica-rich relative to the total sample but should show constant bias. Coarse particles should be limited to avoid burning problems:

Silica	>200μ	Not more than 0.5% of KF
	90-200μ	Not more than 1.0%
	> 45μ	Not more than 2.0%
Calcite	>125μ	Not more than 5.0%

Separators are covered in more detail under finish milling (Section 5.3.).

3.2. Blending

There are various blending silo designs. The two major types involve turbulence (in which the material is tumbled about by the injection of high volume air through air-pads on the silo floor) and controlled flow (where sequenced light aeration of segments of air-pads causes layers of material in the silo to blend by differential rates of descent within the silo). Controlled flow silos may have multiple discharge chutes, or an inverted cone over a centre discharge within which the meal is fluidised. Compressor power consumption is approximately:

Turbulent mixing (airmerge)		1.5-2.5 kWh/t
Controlled flow,	inverted cone	0.25-0.50
	multi outlet	0.10-0.13

(Bartholomew, ICR, 9/1995, p. 66)

Turbulent mixing can be operated batch-wise or continuously. The former involves either a filling cycle corrected progressively to average the target mix, or a sequence of filling, mixing, sampling and analysing, correcting, remixing, and then feeding to kiln. Continuous blending involves simultaneous feeding of the silo, overflow to a second silo and discharge to kiln feed.

Modern blending silos are generally of continuous, controlled flow type with each silo having capacity of more than 24 hours kiln feed and yielding a blending ratio of 3-5; older silos are more like 2-3. Note that a given silo will show a lower blending efficiency if the feed is itself consistent. The retention time of raw meal in a blending silo affects blending ratio and may be easily monitored by addition of zinc oxide or fluorescein to silo feed (see Section B4.8). Apart from power savings, the effective capacity of a CF silo is some 20% greater due to the higher bulk density of meal which is not heavily aerated. The design of modern blending silos is described by Halbleib (ZKG, 10/2003, p. 44).

Blending silos should be monitored by:

- Blending ratio ($s_{feed} / s_{product}$)
- Compressor kWh/tonne throughput

Blending silos are prone to internal build-up of dead material, particularly if raw meal is wet or if aeration is defective, and periodic (one to two years) internal inspections and maintenance are necessary. As raw meal is liable to solidify if left inactive (during a kiln shutdown for example), blending silos may require emptying or re-circulating when not in use.

With the availability of real-time, on-line analysis of mill feed or product, it is possible to maintain chemistry within narrow limits and modern plant designs frequently dispense with kiln feed blending.

3.3. Kiln Feed

Both the chemical composition and the rate of feed of raw meal to the kiln must be consistent to avoid kiln instability and to minimise fuel consumption. Short-term feed fluctuation (eg hunting of feeder control) as well as average feed rate should be monitored.

Air-suspension preheater kilns lose a fraction of kiln feed by entrainment in exhaust gas. As this fine fraction is usually of atypical composition, kiln feed analysis must be biased to yield the desired clinker composition. The dust loss, some 5-12% of kiln feed, is not usually collected until after the exhaust gas passes through a raw mill or dryer, so that dust catch is not the same quantity or composition as preheater dust loss. Thus, even if the dust collector catch is returned directly to the kiln, it must still be compensated. Likewise, care must be exercised to minimise the chemical disturbance due to dust return, particularly when the raw mill is down and dust collector catch changes from mill discharge to kiln discharge. The least bad option is feeding to the blend silo or to a separate storage tank for controlled return.

If the kiln exhaust passes directly and continuously to dust collection, then the dust may be returned directly to the kiln with kiln feed or, sometimes, by insufflation at the hood or at the feed-end of the kiln which minimises re-entrainment of the fines. Either way, the rate of return should be controlled.

Kiln feed is monitored by:

- Chemical analysis on four- or eight-hourly grab samples to determine statistical variation (see Section 6.5.). Analysis is conventionally for major oxides with variation monitored statistically in terms of C_3S or LSF.

Kiln feed should typically have an estimated standard deviation for grab samples of less than 3% C_3S or 1.2% LSF (Halbleib, ZKG;, 10/2003, p. 44). It should be borne in mind that standard deviation is not a perfect measure of variation as, simply applied, it does not distinguish between a steady trend and constant fluctuation.

Kiln feed is normally conveyed by bucket elevator to the top of the preheater to minimise power consumption. If this conveying is effected pneumatically, de-aeration is desirable before injection as

the entraining air otherwise adds to the kiln ID fan load and may reduce kiln capacity.

Although about 1.55t raw materials are required to produce 1t clinker, kiln feed-to-clinker ratio is typically 1.65-1.75 as weighed due to the loss of dust entrained with exhaust gas. This dust, however, is collected and returned. The ratio should be periodically reconciled with clinker and cement inventories and with measured dust loss in the preheater exhaust.

$$\text{Kiln feed} = \text{Clinker} + \text{LoI} + \text{Bypass dust} + \text{Downcomer dust} - \text{Coal ash}$$

where both bypass dust and downcomer dust are converted to ignited basis.

4. Burning and Cooling

The basic cement kiln system comprises a preheater in which feed material is prepared by heat exchange with hot exhaust gas, a fired rotary kiln in which the clinkering reactions occur, and a cooler in which the hot clinker exchanges heat with ambient combustion air.

Kiln feed is subject to successive reactions as its temperature increases (Lea, The Chemistry of Cement and Concrete):

100°C	Evaporation of free water
> 500°	Evolution of combined water
> 900°	$CaCO_3$ -> $CaO + CO_2$ (this reaction is called calcination)
> 900°	Reactions between CaO and Al_2O_3, Fe_2O_3 and SiO_2
> 1200°	Liquid formation
> 1280°	Formation of C_3S and complete reaction of Ca

Spahn (ZKG, 7/2004, p. 72) reviews the chemistry and mineralogy of clinker formation and concludes:

- The dimensions of alite (C_3S) crystals are largely determined by the particle size of limestone/marl in kiln feed.
- The size and distribution of SiO_2 particles in raw meal have a decisive influence on alite and belite (C_2S) formation.
- The Bogue calculations for cement compounds assume chemical equilibrium which, of course, is not realised under kiln conditions.

Cyclone preheater kilns have developed rapidly since the 1950s. The first units were four-stage preheaters. Relative to the previous technology of long wet and dry kilns (Section 11), air suspension in the cyclone system greatly increased the efficiency of heat exchange between hot gas and feed material over the temperature range of ambient to about 800°C and also allowed about 10-15% calcination to occur before the hot meal entered the rotary kiln. Kiln gas is cooled from, typically, 1100°C to 350°C. The feed material is preheated by what appears to be counter-current flow but is, in fact, a series of parallel flow processes in each successive duct and cyclone (see Figure 4.1). Heat transfer in each cyclone stage is completed in less than one second.

During the early 1970s, IHI of Japan developed the Flash Calciner process which employed an additional vessel placed between the first (lowest) and second cyclone stages. The calciner is fired with about 60% of the total fuel input and the combustion air is taken directly from the clinker cooler via a separate (tertiary) duct. The material discharged from stage 2 is fed to the calciner and the reaction product is collected in the stage 1 cyclone. The material discharged from the stage 1 cyclone, over 90% calcined, is fed to the rotary kiln. Variations of the precalciner process have since been developed by the major equipment suppliers. Due to the reduced calcinations required in the rotary kiln, the size of kiln has been reduced by about half. While the fuel efficiency of precalciner kilns is not improved, they have allowed the scaling up of production to over 10,000tpd from a single kiln and facilitated the reduction of NO_x emissions. (Note that while cyclone stages are here counted from the bottom for clarity, the usual convention is to count from the top.)

By appropriate arrangement of fuel and tertiary air injection points, a reducing zone can be created between the calciner and kiln which effects the reduction of entrained NO_x back to N_2. This Lo-NO_x calciner is the most cost effective means of reducing thermal NO_x generated in the kiln. The precalciner, which does not require a high temperature flame, has also proved very effective for disposal of waste fuels. The precalciner kiln has, for these reasons, been virtually the only type of cement kiln installed over the past 30 years.

In the mid-1980s, FLSmidth developed low pressure drop cyclones and these were quickly reproduced by other major suppliers (Hose & Bauer, ICR, 9/1993, p. 55). The innovation was primarily the enlargement of the inlet spiral and elimination of the horizontal inlet shelf together with replacement of the meal distribution splash plate by an external splash box. The overall result was a reduction of pressure drop by 40-50%. This allowed the enlargement of the preheater from four- to five- or six-stages with enhanced heat recovery. The selection of number of stages depends upon optimising construction cost against waste heat required for raw material drying, fuel cost, and power cost.

Unfortunately it is now almost universal to count cyclone stages in order of material flow with the first stage at the top. With the proliferation of preheaters having other than four stages, it is believed that counting in order of gas flow from the bottom would allow more meaningful correlation from kiln to kiln.

Single string (precalcining) preheaters are available up to about 6000tpd (with up to 10ϕ cyclones) and larger kilns now have two- and even three-strings allowing unit capacities in excess of 10,000tpd. Heat recovery has also been improved, where heat is not required for drying raw materials, by using five- and six-stages of cyclones. Exit gas temperatures, static pressures, and specific fuel consumptions for modern precalciner kilns are typically:

Six-stage	260°	550mm H_2O	750kcal/kg (NCV)
Five-stage	320°	500mm	775
Four-stage	350°	350mm	800

Temperatures are 20-30° lower without precalciners and older systems are usually 20-30° higher than the above. Early four-stage cyclone preheater kilns commonly have pressure drops of 700-800mm (higher if ID fans have been upgraded without modifying cyclones and ducts) and specific fuel consumptions of 850-900kcal/kg. Large modern kilns are designed to 700kcal/kg and below. A typical temperature and pressure profile with cyclone efficiencies is shown in Figure 4.1.

In cyclone preheater kilns without precalciners, the feed is 20-40% calcined at the kiln inlet. Riser firing increases this, and addition of a precalciner allows up to 90% calcination before the meal enters the kiln. Although calcination could be completed in air suspension, this must be avoided as the endothermic dissociation of $CaCO_3$, which buffers material temperature at 800-850°C, is followed by exothermic formation of cement compounds and an uncontrolled temperature rise in the preheater could lead to catastrophic plugging.

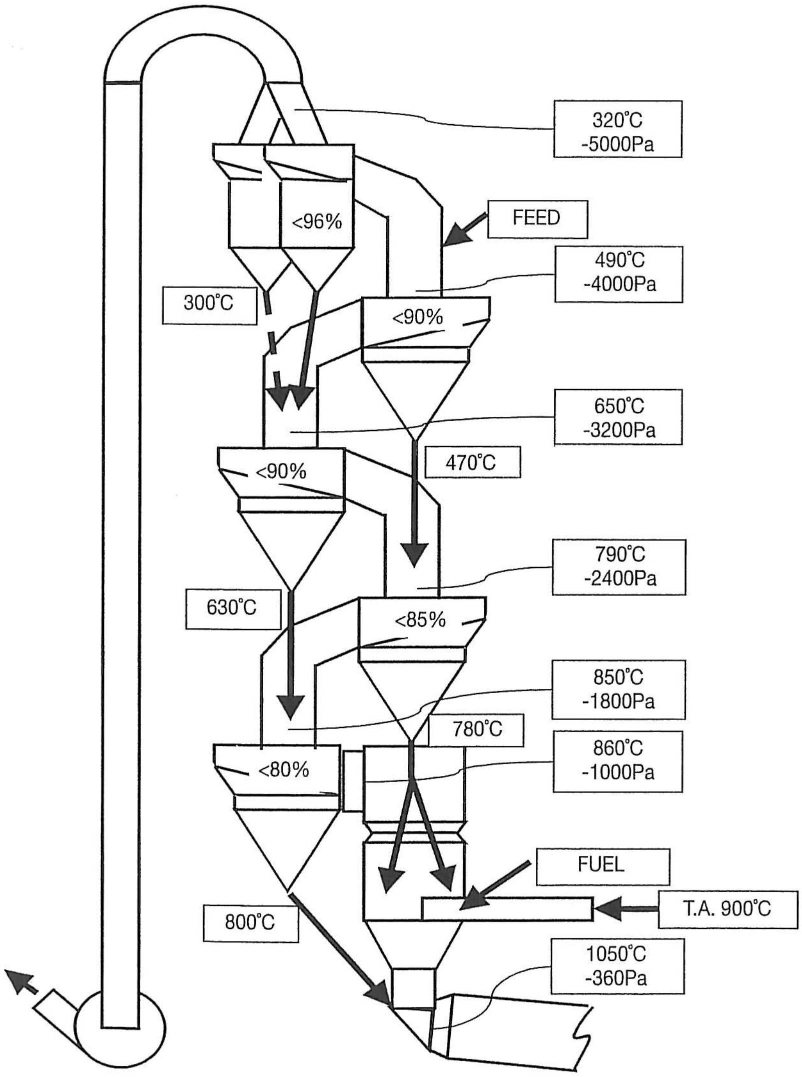
The major cyclone preheater configurations are shown in Figure 4.2. Other terms frequently encountered include:

NSP (New Suspension Preheater) – Precalciner technology which was developed in Japan in the early 1970s.
AT (Air Through) – Precalciner or riser firing using combustion air drawn through the kiln.
AS (Air Separate) – Precalciner using tertiary air.

ILC (In-Line Calciner) – AS precalciner in which kiln exhaust and tertiary air are premixed before entering the calciner vessel.
SLC (Separate Line Calciner) – AS precalciner vessel in parallel with the kiln riser and fed only with gas from the tertiary duct.
SF (Suspension Preheater with Flash Furnace) – IHI precalciner design which is an AS/ILC system.
RSP (Reinforced Suspension Preheater) – Onoda design of precalciner vessel which is an AS/SLC system.
MFC (Mitsubishi Fluidised-Bed Calciner)

Recent developments in burning technology are reviewed by Gasser & Hasler (CI, 3/2003, p. 34 & CI, 6/2003, p. 98).

Figure 4.1. – Precalciner Preheater (5-Stage) Typical Profile



4.1. Kiln Burning

Kiln operation is monitored by:

- Production rate, tonnes/hour clinker
- Operating hours (feed-on)
- Involuntary downtime hours
- Total fuel rate, tonnes/hour
- Proportion of fuel to precalciner/riser, %
- Specific heat consumption, kcal/kg
- Secondary air temperature, °C
- Kiln feed-end temperature, °C
- Preheater exhaust gas temperature, °C
- ID fan draft, mm H_2O
- Kiln feed-end O_2, %
- Downcomer O_2, %
- Kiln feed-end material
    - LoI, %
    - SO_3, %
    - Cl, %
    - K_2O, %
    - Na_2O, %
- Kiln drive power, kW.

There are, of course, numerous other process parameters which should be logged, both to observe trends which may indicate problems, and to provide necessary mean data for process analyses such as heat balances.

Other kiln performance factors include:

- Primary air flow and burner tip velocity, m/s
- Specific kiln volume loading, %
- Specific heat loading of burning zone, kcal/H per m^2 of effective burning zone cross-section area
- Cooler air, Nm^3/h per m^2 grate area
- Cooler air, Nm^3/kg clinker
- Cooler, t clinker/day/m^2 grate area
- Temperature, pressure and oxygen profile of preheater.

Modern kiln operation and maintenance should aim for at least 90% run factor (7884 hours/year), not more than 3% lost time per month (22 hours) between planned outages, and continuous operations exceeding 100 days (Buzzi, WC, 11/2003, p. 92). The best performers achieve 95% year after year.

Note *primary air* is air entering through the main burner, *secondary air* is hot air recovered from the clinker cooler to the kiln, and *tertiary air* is cooler air ducted to the precalciner.

Excessive heat consumption should be investigated immediately and may be indicative of incorrect feed-rate measurement or feed chemistry, fuel or burner abnormality, insufficient or excess oxygen, air in-leakage at kiln seals or preheater ports, low temperature of secondary air, and distortion or collapse of preheater splash-plates.

Clinker free-lime should be as high as possible to avoid the inefficiency of hard burning, but safely below the onset of mortar expansion; typically between 0.5% and 2%. Having established the target, free-lime should, if possible, be maintained within a range of about 0.5%. Variation of kiln feed rate or composition makes this control more difficult. It should be appreciated that over-burning – a common solution to variable kiln feed chemistry or operator circumspection – wastes fuel, stresses refractories, increases the power required for cement milling, and reduces cement strength. Sasaki & Ueda (ICR, 8/1989, p. 55) found a 14kcal/kg heat penalty for each 0.1% reduction in free-lime though other references vary.

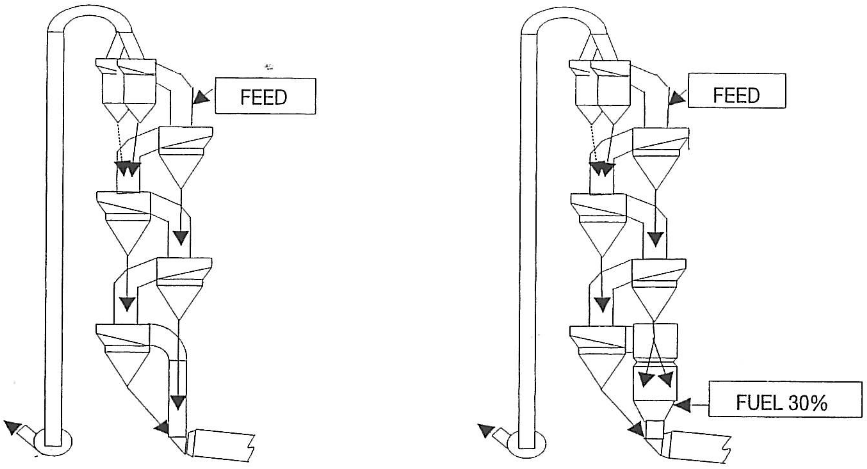
Obviously, if the clinker reactions can be achieved at reduced temperature there will be energy savings. Fluxes, which reduce melting point, and mineralizers, which increase reactivity by incorporation in a solid phase, are reviewed by Hills (ICR, 9/2002, p. 79) and by Kerton (ICR, 9/2003, p. 73). The addition of up to 0.5% CaF_2 in kiln feed has been found to reduce specific fuel consumption by 25-60kcal/kg clinker (Clark, ICR, 5/2001, p. 34) while higher levels can cause preheater build-ups and cement retardation.

A convenient supplement for free-lime measurement is the more rapid determination of litre-weight. This involves screening a sample of clinker from the cooler discharge to approximately +5/-12mm and weighing a standard 1 litre volume. Litre-weight is typically 1100-1300g/l (varying inversely with free-lime) but the target range should be determined with a minimum equivalent to the established free-lime upper limit. A surrogate for litre-weight can be obtained on-line by passing a small stream of screened clinker in front of a gamma radiation source and measuring its attenuation.

Secondary air temperature should be as high as possible in order to recover the maximum heat, usually 800-1000°C. Maximising secondary air temperature involves optimising clinker bed depth and cooling air distribution to the recouperating zone. A common misconception is that increasing the air flow to the hot end of the cooler will cool the clinker rapidly and recover more useful heat. In fact, contact time between cooling air and hot clinker is reduced with consequent lowering of secondary air temperature. Good clinker granulation is essential as fine, sandy clinker results in uneven air distribution and, commonly, a red river of hot clinker extending well down the cooler. Good granulation requires a sufficient liquid phase, typically 23-25%, with high surface tension (Timaschev, Proceedings International Congress on the Chemistry of Cement, Paris, 1980). High alumina ratio and low alkali increase surface tension of the melt while a low burning zone temperature will result in increased liquid viscosity and small crystal size (Section 6.7). Secondary air temperature has been difficult to measure unless there is a hot-gas take off from the hood for tertiary or coal mill air. Recently, however, an acoustic pyrometer has been successfully introduced to the cement industry; this is a low cost and low maintenance instrument which integrates the temperature across the hood and is not affected by entrained dust concentration (ICR, 6/2002, p. 49). The availability of reliable secondary air temperature offers potential for cooler grate speed control to be, at least partly, directed to maintaining constant secondary air temperature rather than the less important function of maximising clinker cooling. Fluctuating secondary air temperature will inevitably cause cycling of kiln operation.

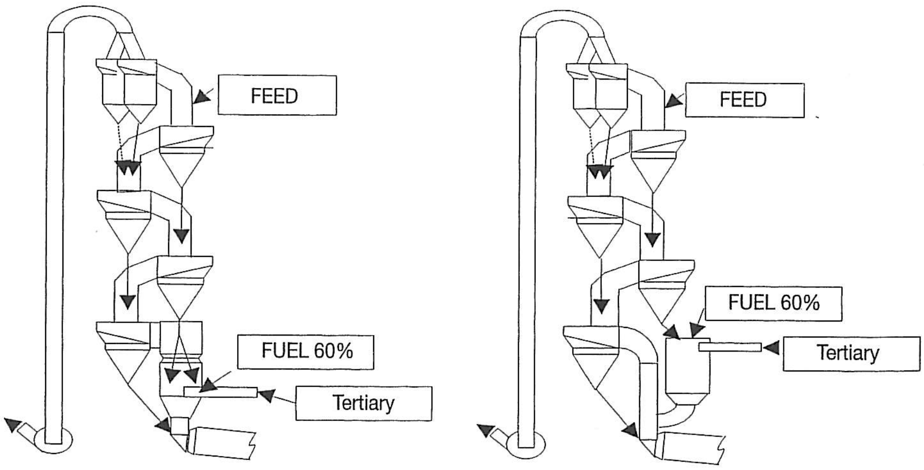
Precalciner kilns are designed to maximise the heat input to the calciner and, typically, 60% of fuel is fed to the calciner while 40% is burned in the kiln. This serves to minimise the size of the rotary kiln and its heat loading; it does not reduce specific fuel consumption. It has been widely found that preheater kilns without precalciner vessels can also benefit from feeding 10-20% of total fuel to the kiln riser. Kiln operation is noticeably more stable and brick life is extended. This is also a useful means of consuming low grade fuels or waste materials. The limit to fuel injection at the riser depends upon its size and consequent gas retention time, and upon fuel-air mixing characteristics; over-fuelling results in preheater operating problems, an increase in exit gas temperature, and CO in the exhaust.

Figure 4.2. – *Major Configurations of Cyclone Preheater Kilns*



SP - Cyclone preheater with or without riser firing

AT (Air Through) Calciner – In-line calciner using excess air from kiln



ILC – In-Line Calciner fed with gas from both kiln exhaust and tertiary

SLC – Separate-Line Calciner fed with combustion gas from tertiary only

All kilns, by definition, have a capacity limitation or “bottleneck” (Section 14.4.) which is, most commonly, the ID fan. Increasing fan capacity is always possible but may lead to excessive pressure drop or inadequate dust collection. An alternative which may well be cost effective, especially for short-term production increase to meet peak market demand, is oxygen enrichment.

Traditionally this involved oxygen enrichment to the kiln burner but the difficulty of maintaining the lance and the danger of overheating refractory largely outweighed any benefits. More recently, injection of oxygen to the tertiary duct of precalciner kilns has been proposed (Tseng & Lohr, ICR, 5/2001, p. 41). This involves a maintenance-free injection port and does not cause refractory stress. A typical addition rate is 2% of total combustion air or 10% of available oxygen, and some 3.5t incremental clinker are obtained per tonne of oxygen. The economics will depend on the cost of cryogenic oxygen or, for more permanent systems, the installation of an on-site Vacuum Swing Adsorption unit which can greatly reduce oxygen cost.

The vortex finders (dip tubes) of lower stage cyclones were for many years prone to collapse and, usually, were not replaced. During the 1990s, a new segmented design in high-temperature alloy became standard (WC, 10/1994, p. 39) and, more recently, a fibre-reinforced monolithic refractory construction is being tested (Gasser & Hasler, CI, 3/2003, p. 34). However, these are still subject to failure and the effectiveness of vortex finders in lower cyclones should be carefully assessed by review of preheater temperature and pressure profile and of specific fuel efficiency both before and after the tubes are removed or fall out, in many cases there is scant justification for reinstallation and the penalties of either distortion or failure far outweigh any trivial margin of efficiency.

For kilns with grate coolers, the burner tip should be in the plane of the kiln nose (hot) or slightly inside the kiln providing it does not suffer damage from falling clinker. The burner should normally be concentric with, and on the axis of, the kiln. Some operators prefer to hold the burner horizontal and even tilted into the load. Such orientation may result in reducing conditions and should be avoided. Clinker produced under reducing conditions causes reduced cement strength and abnormal setting. It should be appreciated that both burner position and tip velocity are intimately related to hood aerodynamics and can not be considered in isolation (see Section 9.3.).

Kiln rings are sections of heavy coating, usually in the burning zone, though sometimes also near the back of the kiln, which can grow to restrict both gas and material flow and eventually force shutdown. Conversely, ring collapse causes a flush of unburned material. Ring formation in the burning zone is commonly attributed to operational fluctuations though a low coal ash-fusion temperature or high mix liquid phase will increase the risk (Bhatty, Proc ICS, 1981, p. 110). Early detection is possible with a shell scanner and rapid reaction is essential. Such ring growth may be countered by varying kiln speed or by small movements (10cm) of the burner in and out. Rings at the back of the kiln are usually associated with volatiles cycles, particularly excessive sulphur at the kiln inlet. It is evident, though of little help, that rings are structurally more stable in small diameter kilns. Recurrence merits an investigation of cause(s) (Hamilton, ICR, 12/1997, p. 53).

Certain plants have raw materials which contain significant proportions of hydrocarbons (kerogens), typically up to 3%, or may wish to dispose of oil contaminated soils. If fed conventionally to the top of the preheater, the hydrocarbons will tend to distil at intermediate temperatures and exit with the flue gas – if they do not explode in the EP (Ryzhik, WC, 11/1992, p. 22). To prevent the resulting pollution, which is frequently in the form of a detached plume or blue haze, and to make use of the heat potential, kerogen-containing materials should be injected at above 800°C; usually to a one-stage preheater with a short kiln if the hydrocarbons are present in the limestone. The high temperature exhaust may then be used for drying or for power cogeneration (Onissi & Munakata, ZKG, 1/1993, p. E7). If the hydrocarbons occur in a minor constituent, this component may be ground separately and fed to the kiln riser. Petcoke, or the residual carbon in fly ash used as raw material, being involatile, can be added conventionally with kiln feed and yield useful heat without a polluted exhaust (Borgholm, ZKG, 6/1992, p. 141). Note, however, that some fly ash contains high and variable carbon (1-30%) and, unless pre-blended, can seriously destabilise kiln operation.

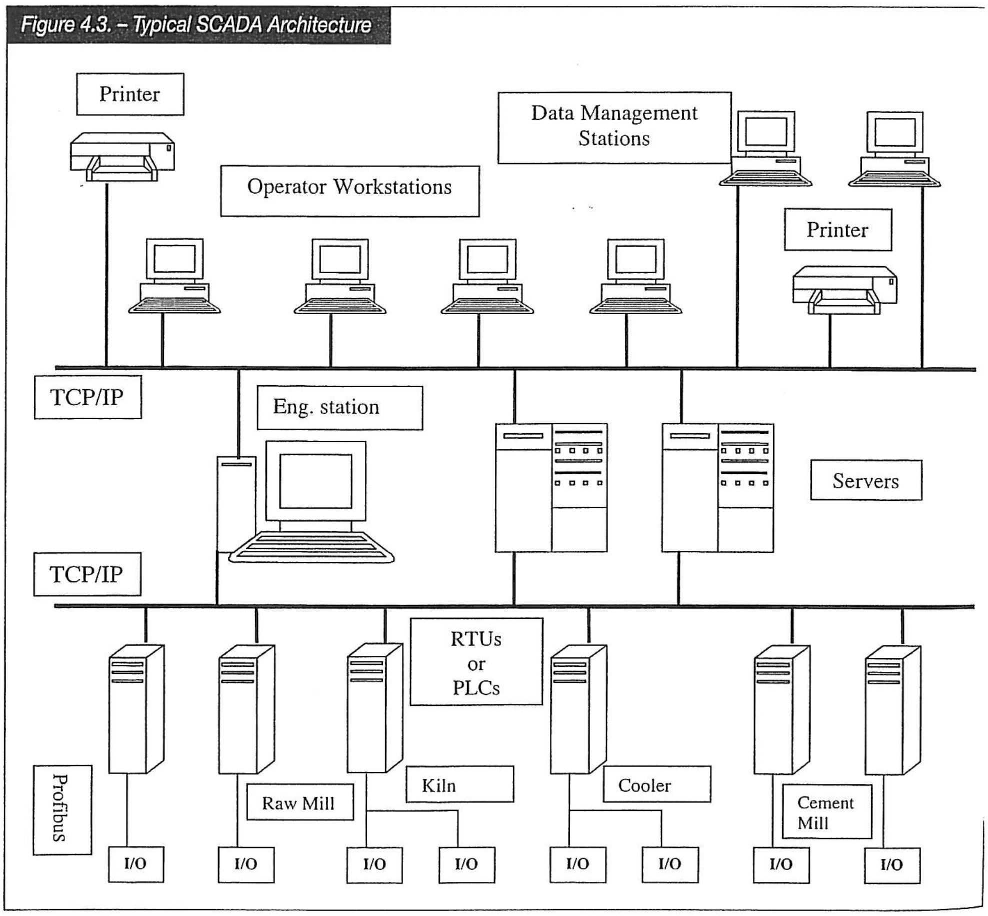
4.2. Control Systems

For the past 20 years the structure of control systems for cement plants has been based on the Distributed Control System (DCS), in which Programmable Logic Controllers (PLCs) are distributed throughout the process system, each to control a certain section of the process. The entire system may be networked for communication and monitoring. The fully developed DCS comprises SCADA (Supervisory Control And Data Acquisition.) software plus HMIs (Human Machine Interfaces) which are usually PCs. A typical architecture of a SCADA system for a cement manufacturing process consisting of input/output signal hardware, controllers, HMI, networks, communication, database server, and software is shown in Figure 4.3.

The process conditions for each production section are monitored and the resulting signals transmitted by wire or optical cable to each corresponding RTU (remote terminal unit) or PLC via I/O (input/output) devices. The control signal generated from the RTU or PLC then returns through I/C to each execution device. The cost of the network is proportional to the number of I/O points.

The RTU or PLC performs PID control loop functions, data compiling and formatting, manages alarm logic, and creates alarm signal and interlocking control.

Figure 4.3. – Typical SCADA Architecture



The Master Station refers to the servers and software responsible for communicating with the field equipment (RTUs and PLCs), and then to HMI software running on workstations in the central control room. Due to advances in microcomputer capability, the software used is developing rapidly. The applications cover many areas, such as production management, model based control, real time optimisation, plant asset management, and real-time performance management tools.

The Human Machine Interface (HMI) refers to operator and engineer workstations where process flowsheets, operating conditions, and logic diagrams are shown on the CRT to facilitate control by operators of equipment start/stop and set points. The software also allows archiving of data and display of data trending for process analysis and trouble shooting.

The communication between HMI, Master Station, and controllers uses an open communication protocol such is Ethernet or TCP/IP.

The reliability of the SCADA system is vital so that backup hardware and redundant signal transmission systems are usually designed in. Both the rapid advances of control and communication technologies and the globalisations of the cement industry serve to drive the industry to adopt open software platforms and open communication protocols. These allow easy system expansion and networking, and incorporation into company information management systems (Garza et al, CI, 1/2003, p. 51 & 5/2003, p. 38). On the other hand, complexity renders the system more vulnerable to cyberterrorism and a robust security shield becomes essential.

Process Optimisation

Various expert systems, now usually called "optimising" systems, are available. More than 75% of the market, however, is held by Linkman (Expert Optimiser Version 4.0) and FLSA's Fuzzy Logic (ECS ProcessExpert Version 4.0) both of which now use neural networks, soft sensors, and model-based control (MPC) technologies. There has been a plethora of one-off, PC-based MPC systems in recent years. There is an excellent series of review papers by Haspel covering developments in Expert System technology, the latest of which is ICR, 4/2007, p. 81. MPC is well established for mill control and gaining credibility for kilns. Overall use, however, is still limited. In 2001, Haspel estimated that only about 15% of worldwide clinker production was subject to expert control (Haspel, ICR, 8/2001, p. 45) and, while this will have increased, systems in use are certainly fewer than installations.

Ultimately, however, these systems require that adequate and reliable instrumentation is in place and that kiln operation is basically stable. Process alarms should be carefully designed and maintained. Critical alarms (eg excess CO in exhaust) should be designed so that cancellation is impossible until the problem is corrected. Interlocks are not uncommonly jumpered (either by hard wiring or by programming) to allow maintenance to cope with a temporary abnormality or for operator convenience; such jumpering must be strictly controlled and frequently reviewed.

4.3. Kiln Control

Kiln operation is a complex art of which the principal control variables are:

	Typical Aim
1 – Burning zone temperature (pyrometer or indirectly from kiln drive power or NO_x)	1500°C
2 – Feed-end gas temperature	1000°C
3 – Feed-end oxygen	2.0%

Control is effected by adjustments to kiln feed, fuel rate, and ID fan speed. Whether normal operation is manual or automated, most kilns are still liable to upset periods due to ring building, coating loss, etc and, while every effort should in any case be made to minimize such instability, effective computer control must be able to cope with the situation.

Kiln feed and speed are usually controlled with a fixed linear relationship and unilateral variation of kiln speed should be avoided. However, a given correlation set up at commissioning may no longer be optimum and it is an important process engineering task periodically to validate the operating graph (Clark, WC, 3/1994, p. 43).

Kiln speed should be such that volumetric loading is within the range 7-12% (Section B5.10). Typically cyclone preheater kilns rotate at 2-2.5rpm (50-70cm/s circumpherential speed) and have material retention times of 20-40mins. Precalciner kilns rotate at 3.5-4.5rpm (80-100cm/s). Material retention in the preheater is 20-40s. It has been asserted by Scheubel (ZKG, 12/1989, p. E314) that CaO, upon calcination, is highly reactive but that this reactivity decreases rapidly so that slow heating between 900-1300°C can result in increased heat of formation of cement compounds. Keeping the same kiln retention time with increasing degree of calcination of the material entering the kiln resulted in extending this transition and there is evidence that the introduction since 1998 of short, two-pier, kilns has led to the reduction of material residence time before entering the burning zone from some 15 minutes to six minutes with resulting improvement in clinker mineralogy and grindability. Two-pier kilns have length:diameter ratios of 11-12 vs 14-16 for three-pier kilns.

Kilns are frequently operated to the limit of the ID fan. In this case, low oxygen must be corrected by reducing both fuel and feed.

Precalciner kilns burn fuel at the kiln hood using combustion air mainly drawn from the hot end of the (grate) cooler, and in the calciner using combustion air drawn from either the hood or the mid-section of the clinker cooler via a tertiary duct. Most precalciner kilns have dampers in the tertiary duct, and some have fixed or adjustable orifices in the riser, to control relative air flows to the two burners in order to maintain the desired fuel split. Frequently these dampers fail and it is then essential to adjust the fuel flows to the resultant air flows. This is effected by maintaining oxygen at the kiln feed-end at, say, 2%. The gas probe at the kiln feed-end should project inside the kiln to avoid the effect of false air in-leakage at the kiln seal. This is a difficult location for gas sampling and an adequate probe is essential (Falligren, ICR, 3/2007, p. 91). CO should, and NO_x may, also be measured at the kiln inlet.

The oxygen level required at the kiln inlet will depend upon kiln stability and combustion efficiency. With a good flame, 1-2% O_2 should result in less than 200ppm CO while an unstable flame may yield in excess of 1000ppm CO with 3% O_2. In a cyclone preheater kiln without riser firing, the downcomer oxygen analyser serves both as backup to the kiln inlet unit and to monitor air in-leakage across the tower, an increase in O_2 of more than 2-3% suggests excessive in-leakage. In a precalciner kiln, an additional gas analyser is required in the outlet duct from the bottom cyclone and, again, this should be operated at as low an oxygen level as is consistent with less than 100ppm of CO.

Note that traditional O_2 operating levels must be modified if staged combustion (Section 9.6.) is employed to reduce NO_x emission.

Useful information on kiln operation can be obtained from frequent (two-hourly) analysis of clinker for SO_3, and periodic (eight-hourly) sampling of the underflow from the bottom cyclone stage(s) for LoI, SO_3, Cl, and alkali determination. Normal SO_3 levels (typically 0.6% in clinker and 2-3% in underflow) should be determined and maintained. In precalciner kilns, retention time and heat loading are particularly low and alkalis (K, Na) tend to pass through to clinker while sulphur is volatilised and builds a cycle at the back of the kiln exacerbated by the deficiency of alkalis. If the kiln is burned too hot or if the flame impinges on the load, this cycle increases excessively until build-up or cyclone plugging occurs. This is matched by abnormally low SO_3 and free-lime contents in the clinker which should be taken as a warning. Eventually, if the kiln is allowed to cool, this sulphur is released and transient high clinker SO_3 results. Such variation in clinker SO_3 will also give rise to varying grindability in the finish mill.

In order to minimise volatile cycles, hard burning mixes should be avoided, the sulphur:alkali ratio should be maintained between 0.8-1.2, and Cl should be limited to not more than 1% and SO_3 to 3% in hot meal entering the kiln.

It cannot be over-emphasised that kiln stability, fuel efficiency, finish grinding power consumption,

and cement quality all depend greatly upon the provision of kiln feed and fuel with minimal variation both of chemistry and feed rate. Healthy scepticism should be nurtured towards both instrument signals and manually reported data. Particular areas for mistrust are:

- False instrument signals of which pressure sensors and gas sampling probes are particularly liable to failure.
- Short term variations masked by electronically damped signals.
- Feeder variations especially when the material is either sticky or fine and dry.
- Chemical variations hidden by faulty analytical methods, statistical mistreatment, or outright fraud.

Variations in kiln behaviour always have a cause, any variations which cannot be explained by observed feed deviation or known operational disturbance should alert to the possibility of faulty data.

Automated kiln control seems, unfortunately, to have reduced operators' habits of looking in the kiln and inspecting the clinker produced. Modern kiln and cooler camera systems, however, are excellent tools (Prokopy, RP-C, 5/1996, p. 38) for observing flame shape and position of the load in the kiln (dark interface of unburned material), "snowmen" (build-up on grates below the hood), "red rivers" and excessive blow-through in the cooler. The appearance of clinker can also be instructive; preferably black with surface glitter, dense but not dead burned, dark grey cores, and absence of excessive fines. Brown cores are usually due to reducing conditions in the kiln but can also be due to the decreased permeability of clinker resulting from high belite and sulphate concentrations which inhibit oxidation of ferrous (Fe^{2+}) iron to ferric (Fe^{3+}) during cooling. This in turn is due to chemical variation of kiln feed and to low volatilisation of sulphur in the kiln (Scrivener & Taylor, ZKG, 1/1995, p. 34). Other causes have also been proposed (Jakobsen, WC, 8/1993, p. 32) and colour is discussed by Clark (ICR, 9/2006, p. 71). Brown clinker is associated with increased heat consumption, reduced grindability, cement strength loss, and rapid setting.

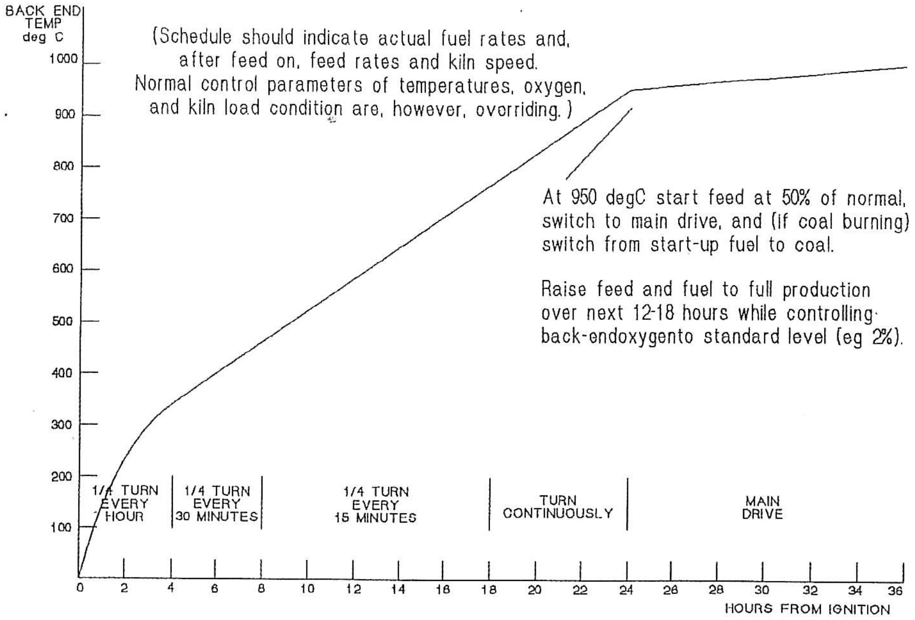
Certain alarms on the kiln control system are critical. Apart from normal mechanical alarms and the routine monitoring of kiln shell for refractory failure, the potential for explosion requires particular care. Gas analysis is conventional at the feed end of the kiln, at the down-comer, and at the dust collector entrance. CO above 1% should cause alarm, and above 2% should cause fuel, and EP if so equipped, to shut off. Flame detection is particularly vital during warm up of the kiln and fuel should be shut off by interlock if the flame is lost. When the kiln is up to temperature it is common to deactivate the flame detector but it should be impossible to start a kiln without this protection.

The light-up of kilns is potentially dangerous as there is insufficient temperature in the system to ensure continuous ignition. Unburned gas, either natural or volatile hydrocarbons from solid fuels, accumulates rapidly in the kiln and, if then re-ignited, will probably explode. It is important that ignition be achieved as soon as the fuel is injected and, if the flame fails during warm-up, the kiln should be purged with five times the volume of kiln, preheater, ducting, and dust collector (probably some 3-5 minutes) before re-ignition is attempted. A simple and reliable ignition system has been described by Davies (ICR, 9/1996, p. 77).

4.4. Kiln Start-up and Shutdown

Detailed schedules should be provided to operators to ensure that what, one hopes, are infrequent occurrences do not result in undue stress to kiln components.

Figure 4.4. – Typical Kiln Warm-up Schedule



(Schedule should indicate actual fuel rates and, after feed on, feed rates and kiln speed. Normal control parameters of temperatures, oxygen, and kiln load condition are, however, overriding.)

At 950 degC start feed at 50% of normal, switch to main drive, and (if coal burning) switch from start-up fuel to coal.

Raise feed and fuel to full production over next 12-18 hours while controlling back-endoxygento standard level (eg 2%).

Warm-up follows agreement by production and maintenance management that all work is completed, that all tools and materials have been removed, and that all doors are closed. Work may, with discretion, continue in the cooler during warm-up but no workers should remain in the cooler at the time of ignition. Commonly, warm-up from cold takes 24 hours from ignition to feed-on, but may be increased if extensive refractory work requires curing. A typical chart is shown (Figure 4.4.) indicating the desired rate of increase in back-end temperature (this may also be set out in terms of fuel rate), the kiln turning program, the introduction of feed (usually 50% of full rate), and the increase of fuel, speed and feed to normal operation which should take another eight hours from feed-on. For PC kilns, fuel is supplied to the calciner at the same time as, or soon after, feed-on. ID fan should be operated to approximately 10% O_2 at the back of the kiln to feed-on whereupon the normal O_2 target is adopted.

For coal fired kilns, warm-up almost invariably employs gas or oil with switch-over to coal at the time of feed-on. If the coal mill uses hot gas from the cooler, there may be a delay before heat is available from the clinker.

Before and during warm-up, equipment checks should be performed to ensure that each unit is ready to operate when required.

Warm-up from shorter stops where the kiln is still hot, say stops of less than 24 hours, are conventionally accelerated to half the shutdown time.

Shutdown may be either:

- Emergency, in which case all equipment upstream of the failure must be stopped immediately, or
- Controlled, in which case feed bin and coal system should be emptied, the kiln load run out as far as possible, and the cooler emptied. The burner pipe is withdrawn, or cooling air is continued through the burner, and the kiln is rotated on a standard schedule for about 12 hours with the ID fan running at reducing speed.

Suggested inching is as follows:

0-2 hours – continuous
2-4 hours – 1/4 turn every 15 minutes
4-12 hours – 1/4 turn every hour

If the shut down is for less than 24 hours and does not involve entering the kiln or preheater, then heat should be retained either by stopping the ID fan immediately and shutting the preheater dampers after two hours, or (if there are no dampers) shutting down the fan after two hours.

4.5. Kiln Refractories

A typical arrangement of brick types and Refratechnik's reported "average best service lives" in Japanese cyclone preheater kilns (without precalciners) is as follows:

Discharge - 1D	70-85% alumina	8 months
1D - 8D	Basic, dolomite, or spinel	6-10 months
8D - 10D	70% alumina	21 months
10D - feed end	40% alumina	21-37 months
(D = kiln diameter)		

Kilns with precalcination average significantly longer brick life.

A detailed historical record of refractory replacement and review thereof are important to minimise cost and service interruption. Typically, brick from the kiln nose to the back of the high-alumina brick section should be replaced if found to be 10cm or less in thickness, but such a rule-of-thumb is subject to much variation depending upon operating considerations. A useful practice is to drill through the brick every metre whenever the kiln is down and coating has been stripped (wider spacing and lesser frequency is adequate in the low alumina brick area). Such drilling requires discretion to locate the shell and to identify irregular circumpherential wear. Alternatively, a taught line may be strung between two drilled points some 6m apart and held, say, 20cm in from the shell. Then brick thickness can be measured in from the line at intermediate positions. Non-intrusive instruments to measure brick thickness are also available (eg Hoganas Linometer). The extent of coating should be observed whenever the kiln is entered and, roughly, basic brick should extend back to the top of the coated zone.

Changes in fuels, feed, or burning conditions will affect the location of the burning zone. Coating location and refractory condition are usually monitored during operation with a shell scanner (Slot & Yazdi, ICR, 8/2004, p. 59). Kiln shells should also be inspected visually, particularly under tyres where small hot spots may be concealed from the shell scanner.

Warm areas of shell can be controlled by use of a fixed fan array or of movable fans which can be directed at the area. Strategies vary, but it is suggested that fans be started either automatically from the shell scanner or manually when the corresponding shell temperature exceeds, say, 250°C. "Red spots", when the kiln shell reaches incandescence, should always be a cause for alarm and should not be allowed to persist for any length of time. If the hot spot is a dull red and is in the burning zone it may be possible to recoat the area and continue operation. Specifically, a small sharp hot spot, relating to the loss of one or two bricks, occurring in the burning zone can be "repaired" by stopping the kiln for 2-5 minutes under the load with an air lance cooling the spot. However, response must be rapid and the long-term problems caused by warping of the shell should always be borne in mind. Red spots on surfaces other than the kiln may be temporarily secured by building a steel box on the

outside to cover the hot area and filling the box with castable refractory; the box should be cut off and permanent repairs effected during the next kiln shutdown.

There is an extensive literature on kiln brick types and performance of which the following is a brief selection:

- Selecting refractories – Cox, WC, 3/2000, p. 48 & 4/2000, p. 76.
- Refractory installation – Karlgren: WC, 12/1999, p. 42.
- Benchmarking refractory performance – Shepherd, ICR, 12/2000, p. 43.
- Lining for critical areas – Kassau, ICR, 5/2001, p. 115.
- Mechanical & thermochemical stress analysis – Klischat & Tabbert, ICR, 9/1998, p. 58.

For plant cost tracking, both net and gross brick consumption should be recorded. Gross consumption is the mass of refractory installed per unit of clinker production (g/t) while net consumption subtracts the mass of brick removed for replacement. Comparison between gross and net figures indicates the wastage of potential refractory life. In strongly seasonal market areas, it may be preferred to remove and replace brick with several months of anticipated life in order to avoid shutting down during periods of peak demand. In more uniform markets, it may be more cost effective to plan on relatively short outages every three or four months; this strategy allows thinner brick to be left in the kiln and has been observed to be the practice at some plants with particularly low operating costs. Gross brick usage averages 850g/t of clinker produced for cyclone preheater kilns and 500g/t for precalciner kilns (Scheubel & Nactwey, ZKG, 10/1997, p. 572). Chromium-containing basic brick is no longer used due to the toxicity of Cr^{6+}.

There are two principal metric brick configurations, ISO and VDZ. Both are described by a three digit code, eg 418, where the first digit gives kiln diameter in M and the last two give brick thickness in cn
With considerable variation, installed brick thickness is related to internal kiln diameter:

<4.2m diameter	180mm
4.2-4.7m	200mm
4.7-5.2m	225mm
>5.2m	250mm

and brick specific gravities are approximately:

Magnesite	3.05	70% Alumina	2.70
Spinel	2.95	40% Alumina	2.25
Dolomite	2.80		

Then brick weight in tonnes per metre of kiln length,

$$W = \rho\pi\left((R^2 - (R - t)^2\right)$$

where ρ = brick specific gravity, g/cm^2
R = inside radius of kiln shell, m
t = brick thickness, m

The two major bricking techniques are the epoxy method and the "ring-jack" method (Mosci, Brick Installation in Rotary Kilns, RefrAmerica 1995: www.info@reframerica.com). Both have their place, the ring-jack is usually faster for long installations but does not allow turning of the kiln which may be important if other maintenance is to be performed on the shell, drive, or seals. Typically, installation after clean-out is at the rate of 0.5m/hour.

In addition, monolithics, which comprise castable and plastic refractories, have various uses from the rapid gunning of large areas or complex shapes to the moulding of burner pipes and distorted kiln nose rings (Fraser, Proceedings IKA, Toronto, 1992).

Castables are concretes with refractory aggregate and a high-temperature resistant (high Al_2O_3) hydraulic binder. Castables may be "heavy" or "lightweight insulating" and are classed:

- standard (>2.5% CaO)
- low cement (1.0-2.5% CaO)
- ultra-low cement (<1.0% CaO).

They are mixed with water, moulded or gunned in place, and allowed to cure for 24 hours before heat is applied. The low cement types are very sensitive to water content and water addition must be strictly minimised, this results in poor workability and requires intense vibration for installation. Excessive vibration, however, can cause aggregate segregation and loss of quality.

Plastics have the consistency of modelling clay and are usually rammed into place though they can be gunned if so formulated. They are classed:

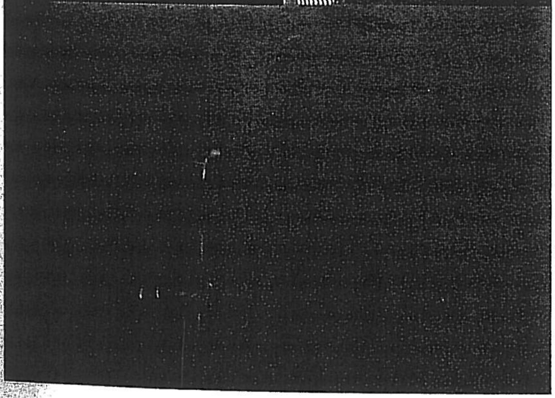
- standard
- air-setting
- chemically-bonded (usually phosphate-bonded).

Plastics have to be heated at specified rates to cure which can allow immediate kiln warm-up to begin.

Castable and plastic refractories require steel anchors to hold them in place. The design and array of these anchors is critical to allow necessary movement of refractory against steel, and expansion joints are also required. The choice of refractory is dictated by the required service temperature, the potential for chemical attack, and the abrasion to which it will be subjected.

Provision is normally made for expansion when installing refractories. Large sections of castable refractory are frequently laid without expansion joints which depend upon anchors to accommodate expansion and contraction, this is not recommended. While brick walls in coolers, feed hoods and firing hoods are designed to accommodate brick expansion, such flat walls do still fail for lack of adequate allowance. Brick walls are also prone both to dust infiltration and to heat distortion of the steel backing which can cause failure. The kiln shell should be provided with a steel retaining ring 1m uphill of the nose casting to resist the thrust resulting from rotation and inclination of the kiln. Experience with large numbers of bricked kilns indicates that no additional retaining rings are necessary if the kiln is reasonably straight and does not have excessive ovality. If further retaining rings are installed, they must not be located within one kiln diameter of a tyre because of the tendency to initiate shell cracks (see Figure 4.5). Conventional rings are rolled plates 40-50mm high and slightly less than the width of one brick positioned such that the uphill brick is in tight contact, while the ring itself is covered by a slightly raised row of brick. Connection to the shell should be by plug welding (Refratechnik Manual). The low profile of

Figure 4.5. – Kiln Shell Crack



the ring results in massive loading of the contact surface with the uphill brick and consequent risk of failure. Gortan et al (ZKG, 10/1994, p. E281) have developed a wedge-shaped retaining ring comprising a honeycomb of special alloy steel filled with castable refractory. Although specially shaped bricks must be laid on the rings, the system is claimed to produce a dramatic improvement in refractory life in difficult cases.

4.6. Insufflation

Insufflation is the injection of dust into the kiln flame. The dust may be either metered into the primary air (using a venturi), or it may be separately conveyed pneumatically and injected adjacent to the main burner.

The effects of insufflation are to:

- return fine dust to the kiln with maximum chance of incorporation into clinker rather than re-entrainment with the exhaust gas
- increase the luminosity, and hence the heat transfer, of oil and gas flames
- reduce flame temperature and, hence, thermal NO_x production
- increase volatilisation of alkalis in the dust to facilitate production of low alkali clinker
- allow production of small quantities of special clinkers by adjustment of mix design without transitioning the whole blending and kiln feed systems. Alternatively, if oil or gas are temporarily substituted for coal as kiln fuel, the effect of coal ash upon clinker chemistry can be maintained by injection of an appropriate correcting mix
- facilitate the addition of small quantities of hazardous waste which must be fed directly to the high temperature zone of the kiln.

It should be recognised that material entering the kiln at the hood uses high-grade heat for preheating which is less efficient than conventional kiln feeding, though this may be offset by better heat transfer. With coal as fuel, however, the emissivity may well be reduced and fuel-air mixing may deteriorate.

Insufflation should not normally contribute more than 5% relative to clinker weight or flame temperature will be excessively reduced. Note that it has been suggested that insufflation of either water or $CaCO_3$ may effect NO_x reduction (Haspel, ICR, 1/2002, p. 63), subsequent tests known to the author confirmed a 10-20% NO_x reduction using water spray.

4.7. Kiln Bypass

If excess volatiles (K_2O, Na_2O, sulphur or chloride) are present in kiln feed or fuel, they will vaporise in the burning zone and condense in the preheater causing a cycle to build up at the back of the kiln. This causes accretions of material in ducts and plugging of preheater cyclones with dire operating results. Where volatile components are unavoidable, or of significant cost saving, a bypass is installed which extracts a proportion of kiln exhaust gas from the feed-end housing for separate quenching with 2-3 times its volume of ambient air, conditioning with water to 150-200°C, dust collection, and release.

Typically, the bypass takes 1-5% of the kiln exhaust gas for chloride control and 10-70% for other volatiles (see Section B5.16). The location of the take-off is critical to ensure that the maximum volatiles are extracted
with the minimum of gas volume, as the latter involves a significant heat loss. Normally the feed housing and meal chute are designed to minimise dust entrainment but, occasionally, high volatiles concentrations which may otherwise cause build-ups may be diluted by the addition of a splash pla to the bottom stage meal chute. Nominally, the dust loss to the bypass is about 2% of clinker weight for each 10% of kiln exhaust gas bypassed. Heat penalty is approximately:

- Cyclone preheater kilns ca. 5kcal/kg x % bypass

- Precalciner kilns ca. 2kcal/kg x % bypass

Conventional bypasses generate relatively large quantities of dust which constitute a disposal problem. Taiheiyo Cement has developed a bypass system for chlorides which separates coarse dust in a cyclone for return to the kiln while the fine dust, containing chloride, is of greatly reduced quantity (Sutou et al, ZKG, 3/2001, p.121).

The criteria for bypass operation are complex depending upon input concentrations, stoichiometric balance between alkalis and sulphur, intrinsic volatility, kiln retention (the heat loading and retention time is shorter in precalciner kilns), and upon cement specification (Farag & Kamel, ZKG, 10/1994, p.586). Automated control of the bypass has been proposed based upon sampling and analysis of the hot meal (Triebel et al, ZKG, 10/1994, p.E272). Chloride circulation should be limited to 5-10g/kg clinker depending upon the degree of precalcination (Farag & Abbas, ZKG, 1/1995, p. 22). The sum of SO_3 and Cl in hot meal entering the kiln should be less than 4%.

Much cement must now meet low-alkali specification ($Na_2O+0.658K_2O < 0.6\%$) while excessive SO_3 in the clinker inhibits C_3S formation. Typically, however, kiln problems may be expected if total alkalis or SO_3 in kiln feed exceed 1%, or chloride exceeds 0.015% relative to kiln feed weight or 0.025% relative to clinker. Alkali cycles are discussed by Clark (ICR, 8/2003, p. 43).

It should be noted that natural SO_2 scrubbing in cyclone preheaters is very efficient. SO_2 found in the stack originates from pyrites or organic sulphur in the raw materials, not from sulphate or from fuel sulphur.

4.8. Preheater Cleaning

Most preheaters are prone to build-up, primarily around the kiln feed-end seal and in the riser. Only the best or the luckiest operators will avoid occasional plugging of cyclone vessels which are caused either by the "stickiness" of condensing volatiles (K,Na,S,Cl) or by temperature excursions. The hot meal at the back of the kiln and in the lower cyclone stages is particularly prone to freeze and build up at cold spots, especially at air in-leakage. Thus, such in-leakage should be specifically prevented at the kiln feed-end seal, at cyclone discharge tipping valves and at all other points in the lower preheater.

It is customary to clean build-up material during operation by use of air-lances, jack-hammers, and high pressure air and water-blasters. Appropriate operator protection and training are mandatory, and ports should only be opened for cleaning after advising the kiln operator and ensuring that people are excluded from below the working area. Claims have been made for both ZrO_2 and SiC containing refractories to reduce build-up tendency (Anagnostopoulos, WC, 3/2001, p. 45). Air cannons are frequently installed in areas of persistent build-up with a discharge cycle which is optimised for coating removal (Zimmer, ZKG, 6/2001, p. 316).

Smooth finishing of feed-chute and riser refractory is helpful. Refractory insulation should be maintained to avoid unnecessary cooling of hot surfaces.

Cyclone clearing is a major operation requiring kiln shutdown and is normally effected through angled ports in the cyclone by long air lances. High-pressure (700kg/cm^2 /10,000psi) water-blasters can be very effective but these are machines of potential danger to both man and materials if improperly used. It is essential during cyclone clearing that all personnel are aware of the potential for release of a large quantity of hot dust with the flow characteristics of water, ports and doors below the cyclone should be closed, no one should stand in front of the hood, and no one should be allowed in the cooler (Renmer et al, ZKG, 1/1996, p. 14).

4.9. Fossil Fuels

Traditional kiln fuels are gas, oil or coal. The choice is normally based on price and availability. It must be noted, however, that fuels are usually priced in terms of gross heat (heat available assuming water in combustion product is condensed to recover latent heat of vaporisation). In practice, only the net heat is employed (assumes that water in combustion gas is released as vapour). The difference varies with fuel:

	Gross kcal/kg	*Net kcal/kg*	*Difference*
Coal	5500-7100	5400-7000	2%
Oil, #6	10200	9700	5%
Natural Gas (kcal/m^3)	6200	5600	10%

It should also be noted that the gas flame is of lowest emissivity and, requiring more combustion air per unit of heat, is the least efficient. Kiln production typically increases by 2-3% when gas is replaced by coal. On the other hand, gas is the cheapest and easiest fuel to handle and is conventionally billed after use rather than requiring advance purchase and inventory cost. Assuming 850kcal/kg clinker and 2% excess O_2:

	Flame temperature °C	*Combustion gas Nm^3/million Cal*	*Total exhaust gas Nm^3/tonne clinker*
Coal	2250	1.23	1360
Oil	2350	1.31	1420
Gas	2400	1.45	1550

Coal, much more than oil or gas, is liable to compositional variation. The nature of production and handling by major suppliers should minimise short-term fluctuation while long-term variation can be compensated by analysis and normal kiln control procedures. If, however, the supply is from small-scale or multiple suppliers, adequate blending must be effected prior to use.

Stockpiling of coal requires vigilance as spontaneous combustion is common, particularly with wet, low rank, or pyrites-containing coal. Smouldering coal should be dug out, the site spread with limestone dust, and the coal then compressed. If long-term storage is necessary, the pile should be compacted and sealed with coal tar emulsion. Thermocouples embedded 1-2m below the surface allow monitoring for combustion.

Coal is usually dried, ground so that the residue on 200# (75m) is not more than 0.5 x % volatiles, and injected with carrying air at a pressure of 120-150g/cm^2 and tip velocity of 60-80m/s. A more precise determination of optimum fineness according to coal type has been described by Seidel (ZKG, 1/1995, p.18). % retained on 50# should be <0.2% and on 100# <0.5%.

Oil may require preheating to reduce viscosity and is injected with a nozzle pressure of about 20kg/cm^2 except for pressure atomised systems which employ pressure to 100kg/cm^2.

Gas is usually received at 10-70kg/cm^2. Primary air is not essential and the gas is injected as axial, or a mixture of axial and swirl, flow at 3-10kg/cm^2 and a tip velocity of 300-400m/s (injection will normally be limited by sonic velocity – 430m/s for methane at 0°C). Gas requires turbulent diffusion and its heat flux tends to be released more slowly than with oil or coal, peak heat release is usually about 20m into the kiln against 5-10m for oil. This results in slower response to control changes which makes for more difficult control of the kiln. It should also be noted that, with a higher ignition temperature than oil or coal, natural gas cannot be reliably re-ignited off hot kiln lining.

4.10. Waste Fuels

In recent years the cost of fuel, which, for most plants, is the largest single cost factor, has stimulated a search for low cost alternatives. Gaebel & Nachtwey (WC, 4/2001, p. 59) review fossil fuel reserves and the future of alternative fuels. Other reviews include Batra et al for petroleum coke (WC China, 3/2005, p. 31), Stoppel for tyres (WC, 8/2004, p. 61), Angelo for animal meal (ENR, 23 Jan/2006) and Brachthaeuser for other wastes (GCL, 10/2003, p. 48).

Petcoke has certain advantages, particularly its very high heat content, but increasing price in some markets has reduced its attraction. The usually high sulphur content (3-6%) also limits rate of addition. It should be noted that there are two main types of petcoke: "delayed" and "fluid". The preponderant type comes from the delayed batch process in which feedstock is heated under vacuum to about 500°C, the residue, "green delayed coke", has typically 8-16% volatiles while calcining at about 1700° yields less than 1% volatiles. Delayed coke may be "sponge" or "shot", can be milled with coal, and is now commonly used up to 100% of total fuel. The burning of petcoke involves finer grinding than coal and higher excess oxygen to complete combustion; this usually results in some de-rating of the kiln (Roy, WC, 4/2001, p.71). Fluid coke consists of small spherical particles resulting from a continuous coking process at about 650°. Volatiles are typically 5-10% and the coke is too hard for conventional milling. Fluid coke is injected un-milled at 10-20% of total fuel (ICR, 10/1993, p.55).

Numerous other by-product and waste fuels have been used and many command disposal fees. Progressively, however, source reduction is diminishing the supply of easily handled liquid solvents and waste oils, and the available materials are, increasingly, solids, aqueous sludges, or scheduled hazardous materials involving onerous regulation. With such materials both consistency and possible contaminants must be monitored. Tyres are potentially attractive though shredding or pyrolisis eliminates much of the cost benefit while fuels added discontinuously, such as whole tyres or containerised waste, de-rate the kiln since sufficient oxygen must be maintained to support the peaks of combustion.

Spent pot-liners from the aluminium industry are another potentially valuable fuel source (Kohnen, GCL, 6/2001, p.8) comprising some 650,000t/y worldwide. Their use hitherto has been constrained by a typically neurotic fear of fluoride and cyanide residues. A more significant limit to use for many plants will be the sodium content. Typical analysis is:

C	55%	Al_2O_3	11%	GCV	4700kca/kg
Na_2O	14%	F	12%		

Other wastes include:

Liquid waste fuels:
- tar
- distillation residues
- used oil
- petrochemical waste
- paint waste
- chemical wastes
- waste solvents
- wax suspensions
- asphalt slurry
- oil sludge

Solid waste fuels:
- paper waste
- pulp sludge
- petroleum coke
- plastic residues
- domestic refuse
- refuse derived fuel
- oil-bearing earths
- animal meal
- rubber residues
- used tyres
- battery cases
- wood waste
- rice chaff
- nut shells
- sewage sludge
- carpet waste

Gaseous waste: landfill gas pyrolysis gas

Kilns employing alternative fuels have detailed specifications to prevent operating or environmental problems, and each shipment is sampled and checked before unloading to ensure compliance. Typically, heat value should not be less than 4000kcal/kg, chloride is limited to 1% and most plants decline fuels with PCB content exceeding 50ppm. If waste fuel is introduced to the hot end of the kiln, all organic compounds, whether hazardous or not, can be completely destroyed (>99.99%) while all trace metals except Se, Cd, Tl, and Hg can be contained in clinker or kiln dust (>99.8%) (Von Seebach & Tompkins, RP, 4/1991, p. 31). Non-hazardous wast fuel may be burned in the riser or the precalciner, but some compounds may volatilise before combustion. Solid waste may also be pyrolysed before injection (Section 14.4. – Fuel Supply).

4.11. Coal Firing

Coal firing for cement kilns falls into two basic systems (Figure 4.6.). Direct firing involves grinding of coal and feeding directly to the burner with all of the drying/carrying air entering as primary air (typically 15-30% of total combustion air). Indirect firing involves intermediate storage of ground coal and separate cleaning and venting of the drying/carrying air. There are several variations on the two basic coal firing systems.

There is a common assumption that indirect firing yields higher thermal efficiency by reducing primary air and by excluding the water vapour from coal drying. Such claims may be invalidated due to the poor fuel/air mixing of low primary air burners while water vapour in the flame has a catalytic effect on combustion. Of more importance is the ability of an indirect system with a single mill to supply two or more burners where a pure direct system requires one mill per burner. Note that significant volatile matter and, hence heat content of the fuel (up to 280kcal/kg), may be lost by venting the milling system.

Coal can be ground in most types of mill but roller and pendulum mills probably predominate. It may be noted that the Claudius Peters EM mill (also known as the Babcock E-mill) comprising large steel balls compressed between fixed and rotating grooved rings, although less common in cement applications, is used widely for coal grinding in the power industry (Floter & Thiel, ICR, 7/1992, p. 22).

Generally roller mills are designed with integral static classifiers though dynamic classifiers may be employed, dynamic classifiers allow fineness adjustment using rotor speed. Roll separation from grinding table should be maintained at 5-10mm and coal feed size should be 100% -25mm with approximately 30% +10mm. Rock and metal rejects fall from the table into the hot air plenum and are swept by a rotating scraper for discharge through an air-locked chute. Abnormal spillage (i.e. more than 2% of mill feed) may be due either to roll clearance of more than 15mm or to excessive clearance between table and louvre ring, if this clearance exceeds about 10mm, the required 25m/s air velocity through the louvre vanes cannot be maintained.

Roller mills can dry coal of up to 10% moisture beyond which the mill is de-rated according to manufacturers design data. Similarly mills are normally designed for 55 Hardgrove index and harder coals (lower HGI) will result in de-rating. Finally, a 10% fall in capacity between maintenance is assumed and allowed for in sizing a coal mill. Mills with common table and fan drives may be given separate drives and capacity can then often be increased by raising the table speed.

Mill inlet temperature should not exceed 350°C and coal should not be dried to below 1% surface moisture. Mill discharge temperature is limited to 65°C for indirect systems and 80°C for direct. Carrying air velocity must be maintained above 20m/s to avoid dust settlement (Recommended Guidelines for Coal System Safety, PCA, May 1983).

Vendors specify a minimum airflow, typically 1-1.5kg (0.8-1.2NM^3) air per kg coal, which must be maintained even when the drying requirement is negligible. This airflow is required to ensure that coal does not remain above its ignition temperature long enough for auto-ignition. Fires are usually the result of rags or wood lodging within the mill and may be detected by an increasing discharge air temperature unrelated to increased inlet temperature or reduced feed rate. Fires in direct firing mills are extinguished by adding feed to act as a heat sink and lowering the mill inlet temperature. Mills in indirect firing systems conventionally employ CO monitoring to detect combustion (thermocouples are too slow to respond), extinction is effected either by water injection or, better, by CO_2 or N_2 with CO_2 the more common.

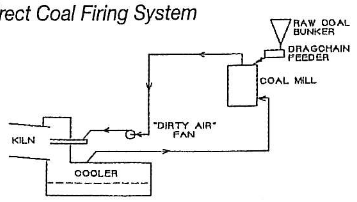
Hot air for drying coal can come from cooler exhaust (normal air) or preheater exhaust (low oxygen). The inlet temperature to the mill is controlled to maintain the outlet temperature as described above and de-dusted in a cyclone. Tempering to about 370°C is effected by bleeding in cold air between cooler and cyclone. If preheater exhaust is used it will typically be at 300-350°C with 5% O_2 and 6% moisture, the temperature and moisture must be considered in the system design. Although most mill fires occur on start-up or just after shut down, the low oxygen atmosphere does reduce the risk of a mill fire during normal operation.

In the United States all coal mill systems are designed in accordance with NFPA Standard 8503 which requires the equipment to withstand pressures of 2.5 times the absolute working pressure, this equates to 3.5kg/cm^2. For conveying systems, this requirement may be proportionately higher because the transfer system operating pressure can be 1.5kg/cm^2 (20psi).

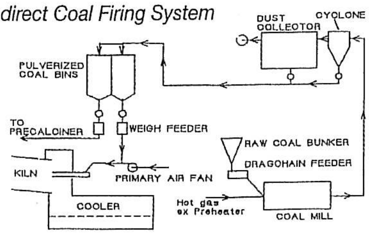
Direct coal firing involves a single pipe burner through which the mill carrying air together with entrained coal are injected at a tip velocity of approximately 80m/s. Tip velocity must always be substantially greater than the flame propagation velocity which may be up to 25m/s. Flames produced by coal nozzle velocities in excess of 80m/s are susceptible to severe instabilities. The pipe is usually narrowed near the tip to minimise parasitic pipe losses and to convert the flow into the desired static pressure. For indirect firing, multi-channel burners of various designs are employed. One annulus is used for conveying pulverised fuel from the mill and one or more separate streams are used to supply

Figure 4.6. – Coal Firing Systems

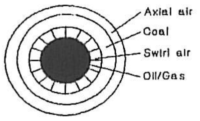
Direct Coal Firing System



Indirect Coal Firing System



Three-Channel Burner



primary air for controlling the flame (Figure 4.6.). Typical specifications used by vendors for burners with indirect firing systems are as follows:

	FLS "Duoflex"	*Pillard "Rotoflam"*	*KHD "Pyro-jet"*
PF conveying air	2%	2%	3.8%
Total primary air (axial+swirl)	6-8%	8%	4.3%
Axial velocity,m/s	140–160	200-230	350-450
Swirl velocity, m/s	(combined)	100-200	100-200

Coal firing almost inevitably involves a normal operating condition where pulverised coal is in contact with air before reaching the burner; this is a condition that should, theoretically, be avoided due to its inherent risk of fire and explosion. The design of a coal firing system is, therefore, critically important and should consider numerous factors including the following:

- Minimum ignition temperature of air/fuel mix; coal ignition temperatures range 200-750°C though the ignition temperature of volatile products may be lower.
- Minimum explosive concentration of fuel in air is about 40g/m^3 (note that dust suspensions are not homogeneous).
- Maximum permissible oxygen concentration to prevent ignition is 12%.
- The entire pulverised coal system must be designed to contain 3.5kg/cm^2 (NFPA 85F) or with explosion relief as appropriate (NFPA 68).

Coal dust explosive tendency increases with volatiles content and with fineness, and decreases with water content and with inert dust diluent (eg limestone).

Coal dust is liable to spontaneous ignition which increases in risk with thickness of dust layer and with the presence of easily oxidised contaminants such as pyrites (more than 2%). A smouldering fire can become explosive if disturbed.

Coal obviously can be handled safely but pulverised coal should always be considered as a potential explosive. Safety considerations are reviewed in a publication, *Coal Fires and Explosions: Prevention, Detection & Control*, EPRI Research Project 1883-1, Final Report, May 1986.

4.12. Clinker Cooling

The clinker cooler serves to cool clinker from the 1200°C at which it leaves the kiln to less than 100°C by exchange of heat with ambient air which is, thereby, preheated before entering the kiln (or precalciner) as combustion air.

By far the preponderant type is the **Reciprocating Grate Cooler** first introduced by Fuller Company. This comprises a series of under-grate compartments with separate fans which allow individual control of pressure and volume of injected cooling air. There may be up to eight under-grate compartments and two or three separately driven grate sections. Fuller coolers are defined by width and length of each grate in series, eg 1035S/1242 has a first grate 10ft wide x 35ft long (S=sloped) and a second grate 12ft x 42ft. Under-grate pressures are approximately 600mm in the first compartment reducing progressively to about 200mm in the last compartment. Modern grate cooler performance is reviewed by Wallis (ZKG, 7/2004, p. 40) and by Lorimer (ICR, 5/2002, p. 67).

Frequently, grate coolers are "upgraded" over the life of the kiln by increasing under-grate pressures and air volumes. It should be confirmed that such ad hoc modifications do not cause significant deterioration such as fluidisation of clinker on the first grate, reduced secondary air temperature, and increased heat loss with cooler exhaust.

For conventional grates, there should be a uniform gap of 3-5mm below the nose of each plate. Distortion of the support frame translates into uneven gaps and plate-to-plate contact which significantly reduces cooling effect. Realignment minimally involves shimming of each support girder in turn starting from the discharge end.

On traditional grate coolers, some 2-2.5Nm^3 of cooling air per kg clinker are required to achieve clinker discharge at below 100°C. This compares to approximately 0.9Nm^3/kg required for combustion so that the balance is exhausted taking with it more than 100kcal/kg clinker. Such coolers were typically rated at 35-40t/m^2/day and operated with clinker bed depths of 200-400mm. Air beam designs are now available from all cooler manufacturers which, with individual variation, provide for the direct ducting of cooling air to hollow grate support beams whence the air is directed into the clinker bed more efficiently than is possible from the former under-grate plenum and grate array. This type of cooler allows 1.5-2NM^3/kg to be used with a rating of 45-55t/m^2/day and bed depth of 800mm. The grates used with air beams, "controlled flow grates" (CFG) are of a higher resistance design with air passing horizontally through slots into recesses in the grate surface; this largely eliminates fall through of clinker fines and renders the air flow less dependent upon bed resistance. Similar grate designs are also available for the non-recouperating zone but with lower resistance as required by plenum aeration, these are referred to as "reduced fall through" (RFT) plates (Roy & Brugan, ICR, 2/1995, p. 47).

Numerous refinements include a pendulum frame for the moving grates which is claimed to minimise drive maintenance, a static inclined arrangement of the first 6-9 air beams which serves to distribute more evenly the clinker entering the cooler, and a hydraulically operated heat shield at the end of the secondary (or tertiary) air recouperation zone which reduces high-grade heat loss to exhaust (von Wedel, ICR, 4/1995, p. 75). The static grate section has been found particularly effective and, typically, costs US$0.5-1m installed. Note that coarse clinker predominates below the kiln-down, or unloaded, side of the kiln and fines below the kiln-up, or loaded, side.

A recent development of cooler design by FLS-Fuller is the **Cross Bar Cooler** (ZKG, 11/1999, p. 608). This consists totally of static grates with clinker transport effected by reciprocating pusher bars above the grate surface. This cooler also incorporates an ingenious flow regulator on each grate which maintains a constant air flow through the clinker bed regardless of bed porosity. Wear of the reciprocating bars is significant but is predictable; cooler efficiency is excellent and much less subject to deterioration over an operating cycle than is the case with conventional grate coolers. More recently, Polysius have introduced the modular Polytrack, with a static cooler floor and above-floor transport (Morgenroth et al, ICR, 5/2004, p. 58), Claudius Peters the η-Cooler (GCL, 1/2004, p. 15), and KHD the Pyrofloor (Klöckner, ICR, 10/2006, p. 60). Progressively, too, designs have become more conservative with specific air flows of 1.8-2.2m^2/kg clinker and 40-50t/d/m^2 (Gasser & Hasler, CI, 6/2003, p. 98).

As kiln discharge rate and clinker size can vary with kiln operation, the pressure drop across the clinker bed on the cooler will also vary. Under-grate air pressure is maintained constant by closed loop control of the grate speed using under-grate pressure as the process input variable. The second drive is commonly controlled at 10% faster than the first if of equal width, but the ratio may be varied as required.

As large clinker nodules and blocks of coating cannot effectively be cooled, grate coolers incorporate clinker breakers. Size reduction is more effective if performed at a mid point in the cooler using a roll crusher. However, due to the difficulties of maintenance at high temperature, both hammer breakers and roll crushers are more often located at the discharge.

Visual inspection of the cooler is important, in particular persistent "red rivers" indicate a problem with segregation of fine and coarse clinker nodules, grate geometry or air distribution, and "blow through" indicates excessive air flow to a particular compartment. Cooler cameras (Sagar, ICR, 9/2007, p. 131) and thermal monitors (Idoux, WC, 12/2002, p. 57) are available.

Acoustic horns have also been suggested as a means to improve cooler heat recovery (Andersch & Kramer, WC, 1/1995, p. 26).

Clinker coolers are monitored by:

- Secondary air temperature,°C
- Tertiary air temperature, °C
- Discharge air temperature, °C
- Discharge air volume or fan amps
- Clinker discharge temperature, °C.

For process analysis, a detailed record is also required of individual cooler compartment under-grate air pressures and of grate drive speed and power. The tuning of under-grate pressure control requires skill and experience. In particular, rings or heavy coating in the kiln may affect bed depth and confuse the control correlation.

Broken grates can allow excessive leakage of hot clinker to the under-grate compartment with risk of major damage. Thermocouples or level indicators placed below the grate drive permit an alarm for clinker filling up within a compartment.

Clinker production is not usually measured directly and is calculated from kiln feed with ultimate confirmation from cement shipment. However, it is helpful to have a point in the clinker transfer system from which clinker production can be loaded to a truck to allow weight checks.

Other coolers are occasionally encountered which avoid the need for separate dust collection:

Rotary coolers are simple rotating drums which lift the clinker to fall through the incoming combustion air stream effecting heat exchange. These coolers are limited to small kilns.

Planetary coolers comprise a ring of tubes attached to the kiln shell and turning with the kiln, which serve as multiple rotary coolers. These coolers can cause mechanical problems on the kiln, they tend to complicate flame aerodynamics, and it is difficult to balance clinker flow to the cooler tubes. Such coolers are, therefore, limited to special applications.

Rotary and planetary coolers are less effective than grate coolers, being limited in cooling air to that which can be consumed in the kiln for combustion. However, with efficient internals, clinker can be cooled to about 180°C (Steinbiss, ZKG, 8/1992, p. E210) and they do not require a separate vent or dust collector. Replacement of planetary coolers by grate coolers is not uncommon (Cohrs, ICR, 9/1995, p. 40).

Peters G-Coolers are supplied by Babcock Materials Handling Division and are secondary systems usually installed in series with planetary coolers or with grate coolers which are being run beyond their design rating. The G-Cooler comprises a number of standard modules stacked vertically as required for temperature reduction and horizontally according to throughput. Clinker from the primary cooler and clinker breaker enters the top of the G-Cooler and settles at a rate of about 5cm/minute with heat exchange to air cooled tubes (Harder et al, ICR, 9/1994, p. 56). These units tend to require little maintenance and little control once the column discharge gates have been adjusted to ensure smooth

outflow and the filling of all columns. There is no direct contact between clinker and air so that no dust collection is necessary.

For increasing cooler capacity, enlarging an existing grate cooler will probably be of lower cost. However, where space or downtime are constrained, the G-Cooler may be attractive.

4.13. Kiln Mechanical

The kiln shell is designed to provide a gas tight support to the refractory lining of the kiln. The shell also imparts rotary motion to the refractory lining in order to convey the raw meal through the kiln and discharge the clinker produced. Since refractory replacement is the major cause of kiln downtime in most cement plants, it is critical to manage the factors which affect refractory life. Similarly, if the kiln shell is designed, maintained, and operated so as to maximise refractory life, then the shell itself will be preserved.

Although a kiln appears to be a straight, cylindrical, steel tube it does, in fact, sag between support piers and deform in cross-section. While the cross section is generally considered elliptical, the orientation of the long axis usually varies from one area of the kiln to another. Near the tyres and in cantilevered overhanging sections the long axis tends to be oriented horizontally, while between the tyres it is vertical. These shell deflections impose significant and, due to rotation, constantly varying mechanical stresses on the refractory lining which is also exposed to thermal stresses. The lining absorbs these stresses through minute amounts of deflection within the individual refractory units (bricks) themselves and through relative motion between refractory units. The material strength of the individual refractory unit together with the strength of the joints between units enables the kiln lining to deform to some extent without failure; however, excessive stresses will lead to lining failure. Though it is extremely difficult to quantify refractory lining stresses, it is easy to conclude that any condition which exacerbates stresses to the kiln shell will increase risk to the kiln lining and should be avoided. Proper alignment of the kiln support rollers and maintenance of tyre pad clearances are the two primary precautions for minimising stress. Roller shafts must be on the same slope as the kiln though they do deflect due to the kiln vertical load by as much as 1-5mm. All shafts on a given pier must be parallel to avoid generation of unnecessary thrust bearing loads. Shafts should be slightly skewed relative to the kiln shell's theoretical axis at each pier to generate some thrust on the kiln tyre that pushes it, and consequently the kiln, uphill. On a properly skewed set of bearings, rollers ride downhill against their thrust bearings. Slight changes in skew can relieve the rollers' downhill thrust. Although all kiln thrust rolls are designed to support the entire kiln's downward load, only certain types, such as those with hydraulic actuators, are designed to operate this way continuously. When all rollers are properly skewed, and under stable process conditions, each support roller imparts a small uphill thrust to the riding rings and, thus, to the kiln shell so that the kiln will contact its thrust rolls only intermittently during each revolution.

Evidence of excessive support roller thrusting can be detected by temperature comparisons of the thrust bearings and from bearing wear rates. Unfortunately, many kilns do not have bearing metal thermocouples so that temperature measurement must be indirect. The two most common methods are to monitor the temperature of the bearing housing where the thrust bearing ("button") is mounted, or to measure the temperature of the roller shaft thrust shoulder using an infrared pyrometer directed through the bearing housing hand-hole. Another excellent way to check for excessive thrusting is carefully to rub the fingers across the surface of each roller noting if the surface feels smooth or rough while wiping in the uphill or downhill direction. This is often referred to as "checking for fish-scales" since the roller surface will feel rough in one direction and smooth in the other. It should be remembered that roller and tyre surfaces can be very hot during kiln operation.

Kiln seals are required at the inlet and discharge ends to exclude false air. The feed end seal must protect against 4-5cm WG differential pressure compared to 1cm or less for the discharge seal. False air at the discharge seal displaces hot (900-1000°C) secondary air so is detrimental to fuel efficiency. However, false air at the feed end displaces potential combustion air on draft limited kilns which reduces production capacity and efficiency. Also, introducing cold air in the middle of any volatile cycle, tends to cause serious build-up which interferes with both material and gas flow.

There are numerous designs of kiln seal and most work reasonably well if they are properly maintained. The most common replacement seal design utilises flexible sheet metal plates arrayed around a steel ring mounted to the kiln shell (Geiger, WC, 12/1995, p. 16). Even within this type there are many variations in design. At the discharge end, the most common cause of poor sealing is overheating of the sheet metal plates. Many kiln designers have found that it is important to protect the internal surfaces of the seal plates from exposure to radiant heat. This is usually achieved by appropriate provision for radiant heat shielding and cooling air flow.

Apart from overheating, the most common cause of seal failure is material loss resulting from stationary and rotating component contact. Since wear between these components is unavoidable, regular inspection and opportune maintenance is essential. It is also important that the rotating component run-out be held within the seal's capability.

Kiln shell design has historically been based on consideration of the kiln shell as a beam of cylindrical cross section. Effective designs are characterised by tyre locations that balance the load uphill and downhill on mid-kiln tyres and yield shell overhangs between one and two kiln diameters from feed and discharge end tyres. The shell thickness is selected to maintain calculated material stress levels well within the steel's capabilities and manufacturers utilise historically proven stress limits that accommodate variation from design assumptions.

Attention should be given be to the weld joints between plates of different thickness. There is ample evidence that for acceptable fatigue life, no step change should exceed 20mm while the thicker plat should be scarfed to provide at least a 3:1 taper down to the thickness of the thinner plate.

All steels used in kiln shell construction lose significant strength when their temperatures exceed 400°C. In fact, at 500°C most steels have only about half the strength relative to ambient temperature so that it is essential that shell temperatures be monitored continuously, recorded and alarmed. Infrared imaging systems are used increasingly for this purpose and also to provide information on refractory condition, coating thickness, and tyre creep. Creep is the relative movement of kiln shell and tyre. This can be measured manually by marking both with chalk and observing the displacement after one or more revolutions, or is conveniently monitored automatically by most modern shell scanners. Creep should never be zero and may typically be up to about 2cm per revolution. Any shell temperatures in excess of 350°C should be monitored closely and if corrective action is deemed necessary to establish or modify the coating it should be commenced before the shell temperature reaches 500°C. Continued operation at or above this temperature will generally result in permanent shell deformation or crack initiation.

Generally there are two thicknesses of shell at each tyre; the thicker plate directly under the tyre is known as the tyre course and the thinner plates uphill and downhill are known as flanking plate. The plate between tyre sections is even thinner than the flanking plate. The most common location for shell cracking is at the transition between the flanking plate and the thin shell plate that spans between piers. Failures generally occur at the toe of the weld joint on the thin plate side. It is often acceptable simply to mark the extent of these cracks while continuing to operate until an opportune shutdown. Drilling a "crack stopper" hole at the end of a crack is a common practice but it will

generally not be effective unless it is at least 25mm in diameter. The crack should be removed by burning or gouging from the outside a groove with sides making an angle of about 60°. Magnetic particle or ultrasonic inspection should be used to determine that the entire crack has been removed. Welding should be performed with appropriate filler metals laid down in straight beads. Each bead should overlap the one below it by about half the bead width to provide heat for relief of weld shrinkage stresses in the underlying bead. Beads should be laid in straight parallel lines, weaving of the beads should not be allowed as this technique usually results in poor quality and high residual stresses (the width of the weld pool should be no more than three times the filler metal diameter). The layers of weld bead should be built up until the toe of each weld bead lies on the line connecting the lips of the groove, ie the weld should form a tapered surface from the thicker plate to the thinner. There is no need to stress-relieve after welding but the shell plates must be maintained above about 5°C during the entire process. At the first opportunity, the repair should be accessed from the inside and the full length again burned or gouged to remove the root pass and re-weld as described for the initial repair.

Apart from overheating, the most common cause of shell cracking is probably fatigue due to excessive alternating stress generated from kiln rotation with high tyre pad clearance. Though tyre creep should be logged daily for each tyre, creep is not the best indication of tyre pad clearance. This should be measured with a device commonly known as the Obourg Pen Tester (Chapman, Recommended Procedures for Mechanical Analysis of Rotary Kilns, Fuller Company 1985, Figure 10) which yields a trace of the relative motion between tyre and shell during several revolutions. More importantly, regular ovality measurements should be taken at each tyre to verify the actual shell deflection. Generally, when percentage ovality exceeds a value:

Shell diameter in metres / 10

corrective action should be considered both to limit the negative impact on refractory life and also to prevent excessive shell stress. On tyres adjacent to the kiln gear, it is common to find normal or even low ovality while tyre pad clearance is high. When tyre pad clearance on these tyres reaches a level which is associated with excessive ovality on the other tyres, corrective action should be instigated. Although the gear is able to hold the kiln shell with minimal ovality, it is not designed to handle the stress caused by this situation. Tyre creep varies with shell temperature and the continuous measurement offered by some shell scanners is valuable. Some creep is essential at all times and it should not normally exceed about 1cm per revolution, correction is effected by shimming of tyre pads.

Tyre thrusting or excessive thrust loading on a tyre is indicated by hard contact between a kiln tyre and its retaining mechanism. The most frequent cause of excessive tyre thrust loading is a slope difference between support rollers and the kiln axis through the tyre. Other, less common, causes are conical wear on tyre or roller and excessive tyre pad clearance. Axial thrusting of a kiln tyre should be corrected before significant metal is removed from the tyre sides because the resulting undercut is difficult to remedy. It is generally ineffective to attempt to relieve tyre thrust by cutting support rollers, a thrusting tyre requires one or more corrective actions: the support rollers returned to design slope, the rollers and/or tyre reground to cylindrical, or the tyre pad clearance reduced.

Kiln drives generally utilise girth gears and pinions designed to give over 20 years of continuous service if lubrication and alignment are maintained. Lubrication should be in accordance with the gear supplier's recommendations for viscosity at operating temperature. An infrared pyrometer is used to measure the gear and pinion tooth flank temperatures and at least three measurements across the gear face should be recorded monthly. A properly aligned gear set will vary no more than 5°C across the gear mesh. Alignment corrective action should be considered when temperature variation exceeds 10°C.

The tip to root clearance between gear and pinion should be measured routinely and after any refractory failure which may have resulted in shell damage. This measurement should be made on the uphill and downhill end of a pinion tooth every 60° of kiln rotation to ensure that clearance and gear run-out are adequate for continued operation. Any changes in support roller position should be considered for their effect on gear alignment and it is, therefore, normal to move tyre axes on all piers except the drive pier. The renovation of mechanical kiln drives is discussed by Hankes (ICR, 7/2004, p. 92).

Over 80% of the horsepower generated by the kiln drive motor is, by design, intended to elevate the kiln charge. However, excessive kiln misalignment will greatly increase the base load power required. Motor designs provide for short term loading of up to about 2.5 times the motor rated current and torque to overcome inertia and static friction for starting. Precalciner technology and the desire to minimise retention time of material between calcination and sintering have resulted in a trend to increased kiln rotation speed. As older kilns are upgraded it is common for drive speeds to be increased and this is usually accomplished in one of three ways:

- gear reducer ratio changes
- weakening of d.c. motor field strength
- utilising higher than rated motor frequency on a.c. induction motors with variable frequency drives.

After the kiln speeds up, it should be verified that the drive motor has adequate reserve capacity to accommodate increased torque demands from process upsets. If a motor is run close to its current limit, load changes resulting from operating conditions may cause uncontrolled changes in kiln speed which will seriously exacerbate the problems of kiln control. Thus, a marginal drive motor should be replaced before attempting to increase kiln speed.

Kiln alignment. Most kiln designers limit bearing pressure between tyre and support rollers to around 35kg/cm^2 though some designs allow for higher pressures if the bearings are self-aligning. Allowable bearing pressures are determined by the bearing materials used, either brass or babbitt, so that the length of the bearing journal must be selected to keep these pressures within design limits. The lubricant film generated by rotation of the support roller journal is usually not over 125m thick so that anything which impacts this film thickness is of consequence. It is essential to inspect the shaft surface routinely during shut downs and re-machine before excessive circumpherential scoring may cause penetration of the oil film, temperature elevation and bearing failure.

Kiln designs make generous allowances for the weight of the kiln charge, refractories, and accretions. To operate reliably the support roller journal surface and bearing clearances must be adequate as must the viscosity, supply, and cleanliness of the bearing lubricant. Variations in coating pattern resulting in non-uniform shell temperature distributions can cause temporary changes in the shell's theoretical axis of rotation. These temporary, process induced, bends in the shell axis combined with other permanent shell deformations can cause excessive bearing pressures and subsequent bearing failures.

The lowest risk operating state then, is the one that allows the kiln to accommodate as much transient shell misalignments as possible. To achieve this state the kiln shell's theoretical axis must be measured while the kiln is operating. This is actually done somewhat indirectly by measuring the position of the riding rings, assuming the rings and shell are perfectly round, and calculating the location of the kiln axis at each tyre. Then imaginary straight lines are drawn between each tyre axis to determine if the tyres are high, low, left, or right of a theoretical straight line representing the kiln shell's axis. If the kiln shell is relatively straight, the location of each tyre's measured axis should lie within about 3mm left or right of the theoretical kiln axis when viewed from above the kiln. The vertical

locations of the tyre axes relative to the theoretical straight line kiln axis may vary significantly from pier to pier among kilns. Generally, on a three-support kiln, the middle support rollers bear the highest load and often must be set low relative to the theoretical kiln axis in order to avoid excessive bearing pressures. The most effective means to make the final alignment adjustments is to utilise an ovality gauge to measure the shell deflections at each tyre. The support rollers should then be adjusted to give equal deflection between left and right rollers on a given pier as well as between different piers. This is not, however, accomplished by obtaining the same ovality percentage at each pier because ovality is significantly influenced by the tyre pad clearance. Bearing temperatures also are indicators of the support roller loading and, consequently, should be monitored closely. Hot kiln alignment is described by Deventer (ICR, 7/2004, p. 43).

Kiln support rollers are designed to bear the weight of the kiln as well as some of the downhill thrust acting along the axis of the inclined kiln. The support rollers on each pier should have their axes aligned parallel to the theoretical axis of the kiln between each pier and slightly cut to impart an upward thrust to the tyre on each pier. This will result in the support rollers lightly touching against their thrust bearings. The sum of all support roller thrusting should keep the kiln from contacting its thrust rollers continuously when internal coating is normal and uniformly distributed. Note that shell expansion at operating temperature is approximately 20cm and it should be confirmed that the tyres are centred on the rollers when hot.

Corrosion of the kiln shell is not normally a serious problem unless high levels of sulphur or chloride are present. Corrosion is reviewed by Mosci (ICR, 6/2004, p. 104).

4.14. Emergency Power

Electricity supply is always prone to interruption and this can result in significant damage unless precautions are taken. The best system is a standby diesel generator (of ca 1MW) which starts automatically upon failure of the main power supply. The generator is connected to a bus feeding power to selected equipment which would include:

- Kiln inching drive
- Primary air fan
- Water supply pump for gas analyser probe
- Cooler first (and second) compartment fans
- Emergency lighting
- Control system monitors
- Pack-house and load-out.

Alternatively, at the very least, there should be a gasoline engine to drive the kiln at low speed. Certain manual procedures are then essential such as withdrawing the burner pipe from the kiln hood and opening doors at the top of the preheater if there are no automatic vents.

The more reliable the main power supply and the more infrequent the use of the emergency system, the more important is maintenance of the standby equipment and rehearsal of the procedure. Serious distortion of the kiln shell will result if it is not turned within 15-30 minutes of a crash stop.

5. Cement Milling

5.1. Clinker Storage

Clinker on transit to the finish mills should have a temperature not exceeding 100°C. It is, however, liable to be, at least intermittently, at a considerably higher temperature as discharged from the cooler. The standard transport system is a steel deep-pan conveyor which can handle high temperature clinker and can elevate the clinker at up to 45° to the top of the clinker storage without intermediate transfer.

In combination with cement storage, there should be adequate clinker capacity to maintain cement shipments during kiln maintenance shutdowns and, if the market is cyclical, to bridge low and high shipping periods. While total clinker capacity should be equivalent to at least 14 days of kiln production, there must also be separate storage for different types of clinker (if produced) and for high free-lime clinker. Clinker storage serves also to blend the clinker and a silo should not be filled and discharged to milling at the same time unless the silo has multiple discharge points to avoid short-circuiting.

It is inadvisable habitually to run clinker silos down to the steel cone as abrasion will eventually cause structural failure. Loading of hot (>150°C) clinker to concrete silos should also be avoided as spalling and de-lamination will progressively weaken the silo walls.

High free-lime clinker must be blended into mill feed with circumspection to ensure that the cement is not expansive. A maximum composite free-lime for the cement should be established and used as a control parameter. Where an autoclave expansion test is employed (ASTM C151), the control target will need to be reduced for clinker containing more than 4% MgO.

There are obvious economies of scale for clinker storage and single silos up to 150,000t capacity have been constructed (Heine, ICR, 6/1991, p. 48). The desirability of large capacity storage buildings has resulted in a range of structures including sheds, silos, conical buildings, and domes of various shapes either clad steel structures or concrete shells sprayed on inflatable forms. Peter (ZKG, 7/1999, p. 371) reviews various designs. Flat-bottomed circular storage with gravity discharge may allow up to 80% recovery without manual assistance while mole reclaimers can recover 100%. Such storage can cost as little as US$33/t (Conroy, GCL, 5/2001, p. 8). Montoya (ICR, 1/2004, p. 48) reports a cost of US$50-60/t for silo construction. Certain clinker silo designs leave substantial quantities of dead material without the possibility for access, such silos, upon construction, are better pre-filled with limestone and run down to refusal to avoid a perpetual inventory of high-value clinker.

Outside stockpiling of clinker is not usually worthwhile if subject to rainfall; it is also prohibited in some locations. Partial hydration of clinker before grinding seriously reduces strength and increases setting time. Recovery from the stockpile should not contribute more than 1.0% to cement LoI and can seldom exceed 10% of clinker fed to the mill while cement produced is of higher unit cost due to added handling costs for recovered clinker, and due to milling to increased fineness to avoid strength loss. Although the Blaine fineness may need to be raised, the -325# fraction should be maintained at the normal level. If clinker must be put outside, it is advisable first to screen out fines (-4#) to minimise fugitive dust and hydration.

5.2. Cement Milling

Finish milling is the grinding together of clinker with some 3-5% gypsum, natural or synthetic, for set control, and other additives (eg pozzolan, slag, and limestone) where appropriate for performance and permitted by specification. The process and equipment circuits are similar to those employed for dry raw milling.

In recent years there have been numerous plant capacity increase projects as well as the construction of new lines. Raised milling capacity has often been achieved by adding pre-grinding, principally roll presses, to existing ball mills. New lines have increasingly incorporated vertical roller mills due to improved reliability, lower specific power consumption, and due to their ability to grind blast furnace slag both separately and in blended cements. Progressively the fineness achievable for Portland cement and slag has been increased to above 6000cm^2/g and it has been demonstrated that the particle size distribution and cement performance of ball mill and roller mill cements are compatible (Schafer, ZKG, 3/2003, p. 56). Ball mills, however, are still the most common and will be discussed here (roller mills were covered in Section 3.1).

Cement strength and consistency has been increasing over the years. Though some single-compartment and open-circuit mills remain, two-compartment ball mills in closed circuit with separators predominate. The diaphragm separating the mill compartments allows the first to be charged with large media appropriate to raw clinker while the second contains small balls which more efficiently achieve fine grinding.

Finish grinding involves the largest unit consumption of power in cement manufacture and should be optimised. It must, however, be recognised that clinker grindability is largely governed by clinker chemistry and burning conditions so that kiln and finish mill should be considered together. The principal cause of hard grinding is the presence of excessive belite (C_2S) in clinker; conversion of belite to alite (C_3S) is maximised if silica is not present in raw mix as coarse quartz, if the lime saturation factor is high (97-99%), if the liquid phase is relatively high (>24%), and if the silica ratio is relatively low (<2.5). It has also been noted that higher K_2O (>0.5%) correlates with ease of grinding though this may be due to the incidental correlation between alkalis and clay which, being of fine particle size, contributes to easy burning mixes. It is important to avoid variable or hard burning mixes as the harder burning and longer retention time involved in controlling free-lime result in large alite and, worse, large belite crystals which cause poor grindability. These large crystals tend to cause dusty clinker (a normal clinker should contain less than 2% -1mm) which, in turn, causes a high re-circulating load between cooler and kiln, can exacerbate "snowman" formation in the cooler, and overload the cooler dust collector. The final major factor is the rate of reaction in the kiln. After calcination is complete, transition to melt formation should be as rapid as possible to minimise growth of belite and CaO crystals. This transition is delayed by a long, lazy flame which may be due to poor fuel-air mixing, coarse coal, or insufficient burner tip momentum (Weihrauch, Influences of Burning Process on Clinker Grindability, Polysius Corporation Technical Seminar, October 1997).

Higher clinker SO_3 gives harder grinding and higher free-lime gives easier grinding (increasing FL by 0.2% reduces finish mill power requirement by ca 1%). Any reduction in power which can be obtained by varying mix design and burning practice must be considered in the context of overall manufacturing cost and cement quality.

Ball mills are cylindrical shells designated by their inside diameter, length, and connected drive motor power. The shell is protected by steel liners and usually divided by a diaphragm into two compartments of which the first is typically 30-33% of the overall length. High-chrome alloys are now almost exclusively used for balls and liners and ball usage should not exceed 50g/tonne cement. The first compartment is primarily to break feed clinker nodules which may be up to 30mm;

lifting liners and balls from 50mm up to 90mm are employed to effect impact. Clinker passing into the second compartment is typically less than 2-3mm and second compartment balls range 15 to 50mm (Fuhrmann, ZKG, 9/1991, p. E199). While coarse grinding benefits from a range of ball sizes, greater efficiency of fine grinding, which involves mainly attrition, may be achieved with single sized small (20-25mm) balls. If a range of ball sizes is used in the second compartment, classifying liners are employed to retain the larger balls close to the diaphragm and the small balls at the discharge. If a single ball size is used, simple wave liners are appropriate. Note that the discharge screen slots must be at least 3mm wider than the diaphragm slots and 5mm less than the smallest ball. Charges should be dumped and sorted, preferably once each year, to maintain the optimum size profile and to remove tramp metal.

The quantity of balls, the type and condition of the shell liners, and the mill speed determine the power draw of the mill. Ball mills typically operate at about 75% of critical speed (the speed at which centrifugal force will just hold charge to the shell during rotation), and 25-35% volumetric charge loading. A charge of different sized balls is best designated by its specific surface area (m^2/tonne charge). Following are general guidelines for mill operation:

2-Compartment Closed-Circuit Ball Mill		
	First Compartment	*Second Compartment*
Power draw	9-10kWh/t cement	24-25kWh/t
Liner type	Continuous lifting	Wave or classifying
Charge specific surface	9.5-10.5m^2/t charge	35-38m^2/t
Closed-Circuit Ball Mill preceded by Roll Press		
Power draw	5-6.5kWh/t cement	22-25kWh/t
Liner type	Continuous lifting	Wave or classifying
Charge specific surface	12-14m^2/t charge	38-40m^2/t

Ventilation of mills has tended to increase since about the 1970s. Air flow serves to remove water evaporated from wet feed materials, from dehydration of gypsum, and from water injected to the mill for cooling. Air sweep also assists with transportation of material within and out of the mill and with the direct removal of heat due to hot clinker and to mill power dissipation. Normal air velocity is 0.8-1.2m/s related to the open mill cross section above the ball charge. Fully air-swept mills operate with 5-6m/s. Static pressure drop across the mill during operation is a good indirect measure of air sweep.

Cement discharge temperature is maintained between about 90° and 115°C; if too cold, gypsum is insufficiently dehydrated and cement strength is lost, if too hot, excessive dehydration of gypsum occurs resulting in false set. Water sprays are used to control mill temperature but it is essential that the water is evaporated and does not give rise to cement hydration or to build up on liners or screens. It is easier to add water to the first compartment but, where the maximum cooling is required, the best effect is achieved by spraying at the diaphragm co-currently into the second compartment.

Circulating load is typically 100-200% for raw milling, 200-300% for finish mills with mechanical separators, and 150-200% with high efficiency separators. Specific power consumption usually decreases with lower charge loading but so does production rate. Since other circuit power is essentially fixed, the design charge loading and rated mill power should normally be maintained.
A useful test of mill condition is to shut the mill down on load after steady state has been maintained for 8-10 hours. The fan should be stopped immediately to avoid sweeping fines out of the mill. The mill needs to be positioned so that the doors can be opened, preferably using an inching drive. On entering the mill following appropriate lock-out, it should be observed that the top layer of balls in the first compartment half floats in feed; exposed balls require more smaller balls to fill voids, or the

blanking of an outer annulus of the diaphragm, or the closing of scoops on an adjustable diaphragm (Gudat & Albers, ZKG, 3/1992, p. 73). In the second compartment, the balls should be covered to a depth of 20-25mm. Samples should be taken every metre through both compartments starting and finishing about 20cm from the end walls and diaphragm. At each interval, 1-2kg samples should be taken from just below the surface along a line perpendicular to the mill axis. The samples should then be screened:

First compartment:	1/2", 1/4", 4#, 8#, 16#, 30#, 50#, 100#
Second compartment	8#, 16#, 30#, 50#, 100#, 170#

Graphing of screen fractions should show a progressive reduction of size from feed to discharge. The samples taken before each screen should be within the range:

Before diaphragm	15-25% +30#	7-14% +16#	<5% +8#
Before discharge	7-14% +170#	<5% +100#	

Any divergence from progressive size reduction indicates flow or charge problems.

Ball charge gradation is also determined during the crash stop by collecting totals of approximately 250 balls from the first compartment and 400 from the second. Equal numbers are taken at each of the locations used for screen samples and random sampling may be assured by spray painting the balls in a line across the width of the mill. Ball diameters are measured using callipers and each ball is either weighed or its weight calculated using a specific weight of 7.9g/cc (or other actual density if known). The data is then sorted:

Fraction	*Mid F*	*# Pieces*	*Unit Wt*	*Total Wt*	*% of Total*
90-85mm	87.5mm	___	___kg	___kg	___
85-75	80	___	___	___	___
75-65	70	___	___	___	___
65-55	60	___	___	___	___
55-45	50	___	___	___	___
45-35	40	___	___	___	___
35-25	30	___	___	___	___
25-20	22.5	___	___	___	___
<20		___	___	___	___

This process allows both an estimate of overall ball gradation and, where classifying liners are installed, an assessment of classifying efficiency.

If negligible particles in the second compartment are greater than 3mm, then the ball size and/or loading in the first compartment may be reduced. Diaphragm and discharge screen should be inspected for wear, holes, and plugging.

The energy efficiency of ball mills is very low, particularly for coarse grinding. In recent years, various circuits have been introduced incorporating hammer mills, roll-presses, and roller mills for pre-grinding ahead of the ball mill. An impact crusher in closed circuit can yield product with 100% -3mm; this approximates to the size of material passing from first to second compartments of a two-compartment mill so that the large ball charge can be replaced by ca 20-25mm balls with a potential mill capacity increase of 20% and a combined system specific power reduction of 5-10% (Stoiber et al, ZKG, 3/1994, p. E94).

Roller mills were originally used for relatively soft materials such as raw and coal grinding but, in recent years, improved metallurgy for wear parts and modified roll configurations to allow handling of fine material have led to their increasing use for cement and slag grinding (Schafer, ZKG, 3/2003, p. 56).

Roll presses have been used extensively for pre-grinding in a variety of circuits and they have also been used as stand-alone cement mills (Liedtke, WC, 9/2000, p. 41). Many early roll presses suffered from roll surface and bearing failures but, progressively, operating pressures have been reduced, roll sizes increased, and metallurgy improved to achieve satisfactory performance. Roll press grinding efficiency (kWh/t) is approximately 0.625 times ball mill for raw grinding and 0.55 times for cement. The most common circuits are now pre-grinding with slab re-circulation and semi-finish grinding "S". In the former (Figure 5.1.1.), the compacted slab discharge of the roll press is split between re-circulation of 80-100% relative to fresh feed, and transfer to ball mill for finish grinding. This system allows a production increase of up to 40%. Greater capacity increase can be achieved if fines from roll press product are removed using a VS-Separator with subsequent selection by a high-efficiency separator between final product and finish grinding in a ball mill (Figure 5.1.2.). The VS-separator both disagglomerates and classifies in a static configuration of stepped plates down which the material cascades through a cross flow of air (Strasser & Bembla, ICR, 2/2001, p. 46). A further development, the VSK separator combines the functions of the VS separator with an integral high efficiency separator (Suessegger, WC, 2/2003, p. 84).

Figure 5.1. – Roll Press Circuits

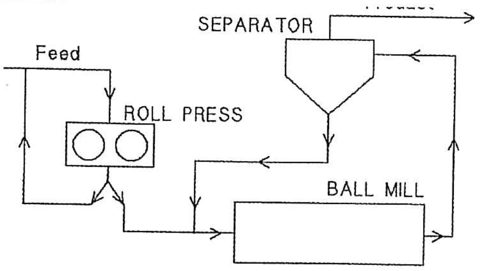


Figure 5.1.1. – Pregrinding with Slab Recirculation

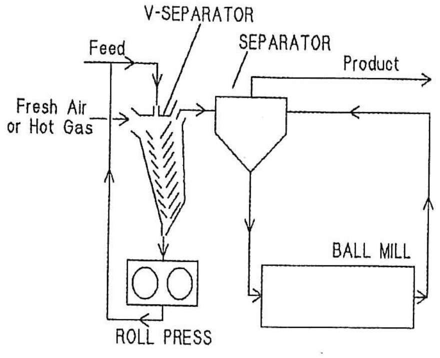


Figure 5.1.2. – Semifinish grinding "S"

For efficient operation of a roll press:

- The feed size should be limited to approximately 50mm (not more than 2x roll gap width).
- Tramp metal ingress must be prevented.
- An optimum head of feed material must be maintained across the whole width of the rolls and the fresh feed and recycled material should be well mixed.
- Feed should contain neither hot clinker nor wet raw materials.

Throughput can be expressed:

$$M = 3.6 \times S \times L \times U \times D$$

where M = total feed (t/h)
S = gap between rolls (mm)
L = length of rolls (m)
U = roll circumferential speed (m/s)
D = product density [clinker = 2.5, raw mats = 2.3, slag = 2.45]

(von Seebach, Proceedings IEEE, 1995).

FCB has addressed the reconciliation of efficient compression grinding with operating simplicity by developing the Horomill. This is a cylindrical mill shell, rotating above critical speed, with a single idler roller and internal fittings to control the flow of material (de La Fouchardiere, ZKG, 3/2003, p. 44). The roller is in free rotation but is hydraulically pressed against the shell. The mill may operate either in closed circuit with a separator or be used for pre-grinding in a ball mill circuit. The Horomill is approximately one third the length of the equivalent ball mill and, when standing alone, power savings of 30-50% are claimed.

The overall performance of a milling circuit is best summarised by its specific power consumption. This is numerically denoted by the ratio of the mill motor power and the corresponding production rate (kWh/t) at a given Blaine fineness (cm^2/g). Specific power consumption for clinker/gypsum grinding in a ball mill should be approximately:

3000cm^2/g	24.4kWh/t
3200	26.8
3400	29.4
3600	32.0
3800	34.7
4000	37.5

Combined power can be reduced by some 20% with pre-grinding. Abnormally high power consumption may be due to mill inefficiency, but is also likely to be caused by over-burned clinker.

Laboratory grinding techniques to determine standard grindability indices (kWh/t) are ubiquitous. Theisen (WC, 8/1993, p. 17) has developed an empirical formula to correlate grindability to alite (C_3S) and belite (C_2S) crystal sizes determined by microscopy:

$$\text{kWh/t (@ 3500cm}^2\text{/g Blaine)} = 33.4 + 0.32C_3S^N + 0.27C_2S - 13.2TEA$$

where C_3S^N = mean alite size, m
C_2S = % belite
TEA = % grinding aid

The effect of changing fineness on mill production rate and specific power consumption is approximated by:

$$(\text{Blaine 1 / Blaine 2})^n = (\text{t/h 2 / t/h 1}) = (\text{kWh/t 1 / kWh/t 2})$$

where Blaine = cm^2/g
n = 1.3 for high-efficiency separators
= 1.6 for conventional separators

As mentioned previously, cement discharge temperature should be kept below about 110°C but should allow some 60% dehydration of gypsum to optimise cement strength without excessive false set (Jorgensen, ZKG, 10/1988, p. 497). High mill temperatures also exacerbate material agglomeration and coating of balls and liners, significantly increasing specific power consumption.

Grinding aids (usually ethylene glycol-based or triethanolamine-based) may be added to the ball mill to reduce such electrostatic agglomeration (Scottili, ICR, 9/2002, p. 91). The optimum addition rate should be determined which balances enhanced grinding against power savings to minimise cost. Grinding aids also serve to reduce *pack set* which is a cement handling problem – see Section 6.22). More recently, composite grinding aids, "quality improvers" have been developed which also yield significant strength increase in both mortar and concrete (Sumner & Gianetti, ICR, 9/2006, p. 101). ASTM specifies that grinding aids and other processing additions must meet the performance criteria of C465.

Ball mill monitoring should include:

- Production rate, tonnes/hour
- Operating hours
- Involuntary downtime hours
- kWh/tonne mill drive
- Connected power, % of mill motor rating
- Cement temperature, °C
- Grinding aid usage, grammes/tonne
- Ball usage, grammes/tonne
- Circulating load, %

Additional monitoring will be required if pre-grinding is effected; specifically the combined kWh/tonne of both units will be of importance.

Particle size may be determined by a number of techniques:

- Blaine or specific surface area, (cm²/g), measured by air permeability through compressed powder.
- +325# residue by wet sieving.
- Laser particle size distribution (PSD) systems which can be reported as the Rosin-Rammler function (Section B4.2).

It is considered that the particle size fraction below 3µ contributes most to one-day strength though it also increases water demand, while 3-25µ governs 28-day strength. Note that hydration only penetrates 3-4µ in 28 days. Particles larger than 25µ make a negligible contribution to strength. Typical particle size distribution for a cement of 3600cm²/g Blaine is:

Below	%	Below	%	Below	%
180µ	100	30µ	81.6	5µ	24.9
150	99.9	20	64.8	4	21.2
100	99.3	15	53.4	3	17.2
80	98.5	10	40.4	2	12.8
60	96.0	8	34.6	1	7.1
40	88.2	6	28.4	0.5	1.9

This particle size distribution corresponds to a Rosin-Rammler function of 0.965.

Obviously +325# particles contribute little to cement strength and should be minimised. A mechanical separator would typically yield 7-8% +325# and a high efficiency separator 4-5% at 3600cm²/g. Fine cement with a narrow particle size range (as is possible with high efficiency separators) gives high mortar strengths but, it should be noted, may also give rise to high water demand which will yield low concrete strength. Concrete strength at fixed workability, not fixed water, is ultimately critical, and standard mortar strengths are significant for process control and specification only – not directly for concrete product design. This should be considered in retrofitting a high efficiency separator to an existing mill; this is usually justified by increase in production due to reduced Blaine requirement for a given mortar strength and the benefits do not always carry through to concrete (Detwiler, ZKG, 7/1995, p. 384 & 9/1995, p. 486).

5.3. Separators

Several types of separator are employed in mill circuits and there are numerous variations of each type:

Grit separators (Figure 5.2.1.) are used to de-dust mill air-sweep. They have no moving parts and effect separation by the cyclonic air flow induced by guide vanes. Radial setting of the vanes gives minimal, and near tangential gives maximum, dust separation. Grit separators should be adjusted so that dust collector catch has the same 325# as product; the dust collected can then be conveyed directly to product cement. These units are typically used in circuits with high air sweep.

Mechanical separators (Figure 5.2.2.) are the traditional classifiers of mill product. The material is fed onto a rotating dispersion plate whence it is spun off into a rising air stream. Coarse particles either fall directly from the dispersion plate or are rejected between the auxiliary fan blades and the control valve. Fine dust remains entrained through the main fan and becomes detrained as the gas flows downwards with both decreasing velocity and diversion through the return vanes. Operating adjustments are the number of auxiliary blades, the clearance between auxiliary blades and control valve, and the radial position of the main fan blades. These adjustments determine the material load in the separating zone of the classifier and are critical to separation efficiency. The main fan blades establish the air flow, while the dispersion plate effects the distribution of material into the rising air flow. The height from the return vanes to the dispersion plate defines the classifying zone. For good operation, the optimum mass flow of material per unit volume of air (kg feed/m^3 air) ought to be established for each product fineness.

High efficiency cyclone separators were introduced to improve on the mechanical separator's low efficiency in fines recovery (Onuma & Ito, ZKG, 9/1994, p. 535). A simplified process flow for the O-SEPA (Figure 5.2.3.) is as follows. Material is fed onto a rotating dispersion plate whence it is dispersed into the classifying air stream which is sucked from tangential inlet ducts through fixed guide vanes. Separator loading is up to about 2.5kg feed/m^3 air flow. A horizontal vortex is formed by the rotor which classifies particles between centrifugal force and the inward air flow. The fine fraction exits upwards with the air exhaust for subsequent dust collection while the coarse fraction falls and is discharged from the bottom of the vessel. Fineness is increased controlled solely by increasing rotor speed; increasing speed increases fineness. Air flow is separately controlled by the separator ID fan, effective material dispersion is assured by buffer plates around the periphery of the dispersion table, and a uniform distribution of incoming air is assured by the design of the incoming air ducts and guide vanes. The height to diameter ratio of the rotor controls the retention time in the separating zone.

Dynamic classifiers, used integrally with a roller mill (Figure 5.2.4.), involve the upward flow of dust-entrained air into a segregating area above the grinding table where, with decreasing air velocity, coarse particles fall back to the ill table while fines leave with the exhaust for external de-dusting. Design developments have yielded a progressively steeper Rosin-Rammler distribution of mill product and an increasingly coarse reject fraction returned to the table which gives a more stable grinding bed.

Figure 5.2. – Mill Separators

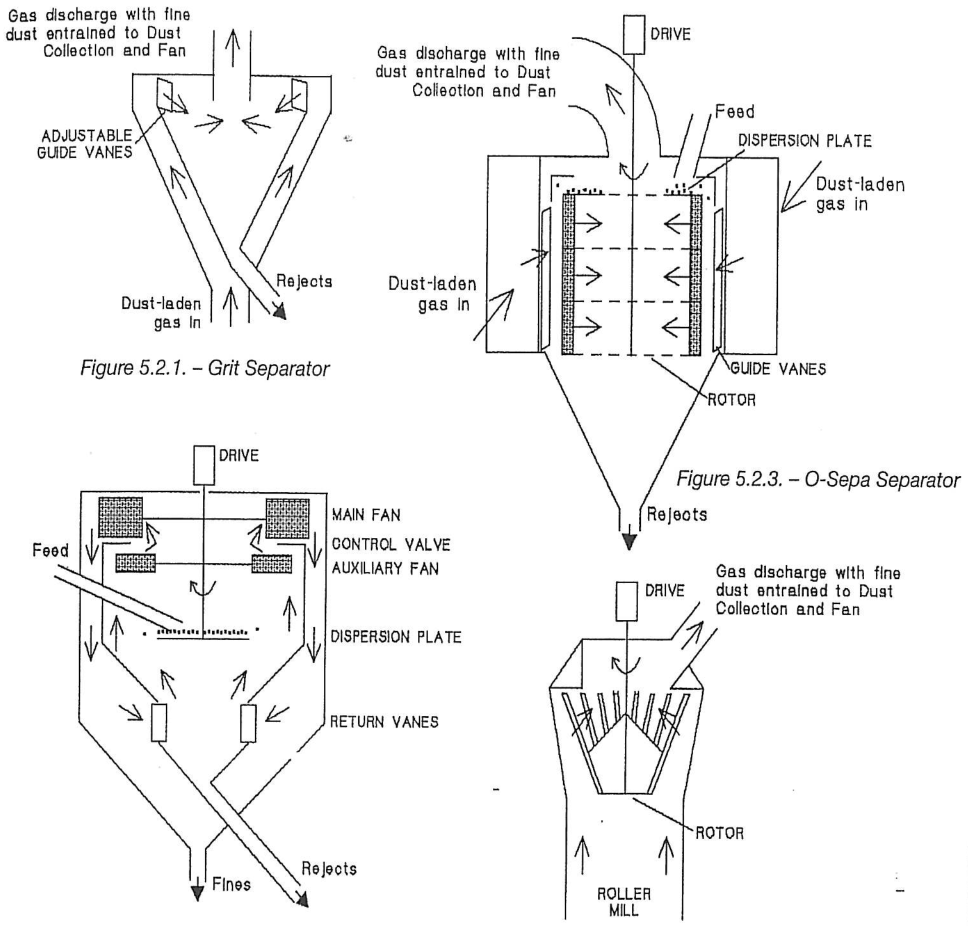


Figure 5.2.1. – Grit Separator

Figure 5.2.3. – O-Sepa Separator

Figure 5.2.2. – Mechanical Separator

Figure 5.2.4. – Roller Mill Classifier

The Rosin-Rammler distribution graphs cumulative particle size fractions against particle size so that a narrow distribution results in a higher slope.

High efficiency classifiers yield R-R slopes of	0.95 – 1.20
Dynamic classifiers	0.80 – 0.85
Static classifiers	0.65 – 0.75

Increasing classifier efficiency has been particularly effective in raising the production capacity of raw mills. And a recent innovation, the LV classifier (see Section 3.1.), has also achieved significant pressure drop reduction, system power reduction and increased throughput on roller mills.

Possible mill circuits are legion, from a single open-circuit mill to a combination of mills and separators (Onuma & Ito, WC, 9/1994, p. 80). The situation is rendered particularly interesting when increased performance is required from existing equipment and new units are combined with old. The only generalisation which can be made is an obvious caution that balance and control are essential.

5.4. Ball Mill Circuit Control

Comminution and classification should be considered as two separate but interconnected unit operations. Optimum grinding conditions depend upon mill feed rate, net power draw, air flow, and mill internal temperature. Classification depends upon classifier feed rate, air flow, and either rotor speed or vane/diaphragm setting.

Mill feed control maintains the quantity of fresh feed and the proportions of individual components. The feed rate determines the ratio of feed to grinding media in the mill, the optimum steel to clinker mass ratio is typically 8-12.
Mill power control depends upon the weight of grinding media. Periodically, make-up charge is added to maintain maximum power draw.
Mill ventilation in a two-compartment mill with L:D ratio of 3.2 is limited to about 1m/s and is normally monitored by static pressure at the discharge which should be typically -100 to -150mm WG. Ventilation effects cooling of the mill and removal of fines.
Mill internal temperature affects material transport and, in cement, the controlled dehydration of gypsum. Material discharge is typically below 100°C for raw milling and 105°-115°C for cement.
Mill sound level is monitored by microphones located externally approximately 1m from each end of the mill.
Classifier control determines product fineness and circulating load (rejects returned to the mill), and usually remains constant for extended periods. In mechanical separators, air flow is determined by the diameter and number of blades on the main fan wheel. Cyclone separators allow independent adjustment of air flow and rotor speed.

Control philosophy is, primarily, to maintain the optimum material level in the mill. The total feed is the combination of fresh feed and classifier rejects and the fresh feed rate is controlled either from a flow meter on the rejects stream or from discharge elevator power. The control loop should be timed to minimise fluctuations. Decreasing microphone sound amplitude and increasing mill drive power indicate that the mill is filling up, and while this will normally reflect excessive feed, it may also indicate a problem such as blinding of the discharge screen. Conversely, increasing sound and decreasing power show emptying of the mill. Mill temperature is controlled by water spray addition rate or, where drying is effected, by control of inlet gas temperature.

5.5. Cement Storage

Combined storage capacity was discussed in Section 5.1. Minimally, there need to be sufficient silos or silo compartments (in multi-compartment silos) to keep different types of cement without simultaneous filling and discharge, and capacity should also be adequate to complete at least one-day strength testing before shipment. With cement storage typically costing US$150-300/tonne, there is strong pressure to minimise both total storage capacity and the number of silos for a given capacity. Dagnan (Proceedings IEEE Cement Industry Conference, Salt Lake City, May 1986, p. 131) suggests a guideline of 30 days production and details the factors involved. He also specifically addresses the economics of small volume special products.

Cement should, if possible, be below 60°C when conveyed to silos. The storage of cement at greater than 80°C causes dehydration of gypsum and reaction of the released water with fine cement particles resulting in loss of early strength and promotion of false set in the cement as well as build-up in the silos. If high temperature is an intractable problem, the effects can be mitigated by reducing gypsum addition and replacing up to half the gypsum by anhydrite (Reid, WC, 4/1997, p. 104). After the mill, cement can be cooled using a dedicated cement cooler (Kochmann & Ranze, ZKG, 10/1997, p. 556) or by water jacketing a pneumatic conveying line.

Silos for most materials are prone to developing dead material which is both wasteful of capacity and misleading as to inventory. This is a particular problem with cement. Periodically silos should be examined and cleaned, either manually from inside or using the various remotely controlled mechanical techniques now available (Laing, WC, 6/2002, p. 65). Build-up and the dubious performance of many silo level detectors may lead to erroneous inventories; methods and problems of level detection are reviewed by Little (ZKG, 8/2004, p. 48) and new radar equipment is offered as a solution. Erratic flow of cement is described by Dudley (WC, 6/2002, p. 69) while Graham (ICR, 2/1997, p. 51) describes a low-frequency acoustic technology for flow enhancement and silo cleaning. Manual measurement should be made at a point 2/3 of the radius out from the centre and measurements should be reconciled with metered filling and discharge of silos to detect anomalies.

Dome storage of cement has presented a challenge in discharging at larger dome diameters; discharge systems are either mechanical (Hunter & Wood, WC, MHR/1999, p. 46) or use a live floor of air slides (ICR, 1/2004, p. 52). For satisfactory performance of a live floor, air slide separation must not be excessive and it is advisable to maintain a low pack set using grinding aid and to minimise the formation of syngenite in storage. The air slide control strategy is also critical to ensure that cement, once dislodged, is discharged.

Dome storage has also presented a problem of inventory determination due to the large surface area and wide and erratic surface geometry. Possible solutions may be to use a laser surveying device which constructs a grid across the surface from the feed point at the top of the dome, or radar as described above.

5.6. Cement Dispatch

In general, shipments of cement range from almost wholly bulk in developed markets to almost wholly sack in less developed countries. The economics of distribution are complex and have important effects upon plant locations and capacities. It has usually been accepted that a plant can effectively serve a radius of about 300km using road transport. Betty & Scott (ICR, 7/1993, p. 36) give formulae for estimating unit bulk shipment cost in the United States) :

23-25t road truck = \$8/t/day + [\$0.022/t x km (one way)]
2000t rail shipment = \$0.50/t + [\$0.024/t x km]
35,000t sea shipment = \$3.00/t + [\$0.002/t x km]

Thus, a 320km (200 mile) haul would cost approximately US\$15/t by road and US\$8.20/t by rail. This, of course, may not reflect door-to-door cost where road haulage normally offers an advantage. Road transport also is usually more flexible than rail for small, irregular, or short-notice shipments.

By comparison, 6000km trans-Atlantic sea freight would cost about US\$15.00 (subject to the cyclical variation of shipping costs). It is obvious, therefore, that coastal areas are very exposed to low cost imports unless protected by tariffs. And certain routes can be served by ships on back-haul at even lower rates. Similarly, it can be seen that plants with deep water docks have almost unlimited market potential. Polansky (ICR, 12/1995, pg 47) reviews the operation and economics of bulk carriers of greater than 25,000dwt and Thomas (WC, 6/2004, p. 15) assesses future trends in ocean shipping. Economic bulk distribution requires minimal demurrage costs for filling and discharging bulk vessels. Such costs, if incurred, should periodically be compared to the cost of upgrading loading/unloading equipment.

Sacking adds approximately US\$4-6/t to production cost and can, of course, increase shipment costs due to the additional handling involved. Sacks are most efficiently handled either on pallets or shrink-wrapped (Remmert, ZKG, 12/2000, p. 690). Alternative packaging and storage systems to minimise deterioration are reviewed by Offenbecher (ICR, 3/2007, p. 70).

Cement sacks are typically 94lb, 50kg or 40kg net weight. The European Commission, however, based upon their extensive experience of handling cement sacks, have ordained that European labourers could not healthily and safely lift bags of more than 25kg (Reitmeyer, WC Bulk Materials Handling Review, 1993, p. 32).

Sack weight standard deviation should be determined in order to minimise overweight. ASTM C150 Section 11.3. specifies that packages more than 2% underweight may be rejected, and that if 50 random sacks average less than the stated weight, the whole shipment may be rejected. Thus, with an average of the stated weight and a standard deviation of less than 1%, fewer than 1 in 20 sacks could be rejected. Any margin of designed overweight is at management discretion, but a conservative target would be the stated net weight plus the mean bag weight (200-250g for 40/50kg bags) plus 1x standard deviation.

The major cost element in sacking is the kraft paper and this has historically been a commodity volatile in cost and, occasionally, in short supply. If a plant has a significant sack market, therefore, some effort is justified in purchasing and stockpiling paper and in optimising sack configuration against breakage (Grundy, ICR, 7/1993, p. 50). In-plant sack breakage of less than 0.3% should be maintained and this can be assisted by careful attention to sack handling and load-out conveyors. Where acceptable to the market, woven polypropylene bags offer both strength and economy (Worff, ICR, 12/2000, p. 60).

Bulk loading of road or rail trucks is normally about 400tph, and of ships and barges considerably faster. Twenty-five-tonne road trucks can be loaded at six-minute intervals from a single loading spout with appropriate equipment and procedures. The filling spout is telescopic and has a conically tapered nozzle to seal against the tank opening. The filler comprises a central cement delivery pipe surrounded by an outer suction tube which vents at least $1000m^3/h$ and is ducted to a dust collector. Load weight is determined either from tear and gross weights of the truck (preferably on dedicated truck scales under each loading point), or by loading from a hopper on load cells which is itself weighed before and after loading and which can be programmed for a specific delivery (Brix & Pauer, WC, 3/1995, p. 106). To prevent spillage, the loading spout should be fitted with a level sensor and cut-off switch. The design of modern road tankers is reviewed in ICR, 4/2003, p. 83.

A third mode of shipment is the "big bag", a 0.5-2t flexible bulk container comprising a woven polypropylene outer ply and a polyethylene liner. The bags may be single or multiple use and can be handled by slings or forklifts. There tends to be little, if any, cost advantage over sacks, and to date the use of big bags has been limited to special situations (Lasheen, ICR, 4/2003, p. 69) or as temporary "silos" for small batch plants.

Automated dispatch systems as described by Bauer (ZKG, 6/2004, p. 54) are increasingly employed.

5.7. Quality Assurance and Customer Service

In Europe, EN 197-2 stipulates external evaluation of standard conformity and product certification while in North America, ASTM C183 is a voluntary control. Quality control at most North American plants ends at the testing of daily composite cement production. Thus, potential problems such as temporary gypsum starvation or contamination during silo feeding or discharge may not be detected. It is advisable to institute quality assurance procedures to monitor cement as shipped and it is recommended to retain shipping samples for, say, three months to help investigation of customer complaints.

It is customary for cement manufacturers to provide technical support for cement users. The more common cement and concrete problems are described briefly in Section 6.22. Apart from contamination or the shipment of an incorrect cement type, it is very unusual for cement to fail to meet specification. In many cases, advice on concreting practices or the use of admixtures can amicably solve the problem. There are, however, a number of consistency and performance questions upon which it may be diplomatic to help or compensate customers in the interest of long-term relationships.

Particularly in markets with a high proportion of sack shipments and, possibly, primitive storage facilities between plant and customer, it is important that all sacks should be date-marked, albeit in code. It is not unusual for complaints to be received of hydrated or low strength cement which may be years old. The counterfeiting of cement by substituting ground grey dirt has also been known. Records allowing the tracing of questioned shipments to silo and production data are desirable.

It has been found valuable to record and analyse all customer complaints and to identify settlement costs resulting from each class of problem. Frequently a known problem such as hot cement or contamination can be corrected, but the remedy needs to be paid for and it is the compensation to customers or avoidance of lost sales which provide the cost justification.

6. Quality Control

The plant quality department is responsible for:

- Search, selection, specification, and analysis of raw materials and fuel.
- Mix design which should reflect individual raw material costs, production costs (principally drying, grindability, and burnability), and product quality.
- Selective mining of limestone.
- Preblending of raw materials and blending of kiln feed to minimise chemical variation into the kiln.
- Process control which involves raw materials moisture, raw meal chemistry and fineness, clinker free-lime, proportioning of clinker and gypsum, and fineness of milled cement.
- Chemical analysis and physical testing of cement to confirm and certify compliance with specification.
- Quality assurance and certification of shipped cement.
- Investigation and solution of process and product quality problems.
- Miscellaneous functions such as water treatment and laboratory measurements associated with process engineering and pollution control. Frequently, the laboratory maintains and reports production and inventory data.

6.1. Sampling

Sampling, which is the reduction of a large quantity of material to a small portion which is representative of the whole, is as important as analysis, particularly with raw materials where large particle (rock) size and heterogeneity require that massive samples are taken and reduced systematically to the quantity actually analysed.

Whenever possible, samples should be taken from a flowing stream (transfer point or drop chute) or a stopped conveyor belt where the entire stream should be cut. Reduction involves successive crushing and splitting of samples to achieve the final analytical sample; the ASTM procedure for aggregate is described in C702. Theoretically, the required sample is estimated using standard formulae involving predetermined factors for material heterogeneity, particle size, density, shape factor, and required precision (Pitard, Sampling Theory and Sampling Practice, 2nd Ed., CRC Press, 1993). The Minimum Representative Weight (MRW) is given by:

$$\text{MRW, g} = 18\,\lambda f d^3 / s^2FE$$

where λ = density, g/cm^3 (see Section B7.2)
f = shape factor (dimensionless; cube – 1.0, sphere – 0.523, rock – 0.5, coal – 0.45, quartz – 0.47, mica – 0.1)
d = maximum particle size, cm (defined as the square mesh opening which retains 5%)
s^2FE = particle heterogeneity factor (practical limits are +/-5%)

There are also various simpler guidelines for sample size such as:

- Between 0.2% and 2% of total flow depending upon heterogeneity and particle size (Labahn).

■ Max Particle Size	Minimum Sample
10mm	10kg
20	25
30	60
40	80
50	100
60	120
75	150
90	175

(*ASTM C75; Standard Practice for Sampling Aggregates*)

Sampling procedures for hydraulic cement are specified in ASTM C183 for contractual purposes and in C917 for strength uniformity evaluation. The sample size is approximately 5kg.

Sampling procedures for coal are given by ASTM D2234, D4915 and D2013. The sample size is determined by the maximum particle size and the testing required.

Note that grab samples must be used for calculation of standard deviation (Section 17). The use of composite samples or averaged data as input to statistical analysis is common but meaningless.

6.2. Chemical Analysis

X-ray fluorescence

Process control requires on-line or rapid off-line analyses in order to adjust raw material feed proportions to the raw mill. X-ray fluorescence is now almost universally employed using either pressed powder or fused disk samples. Numerous XRF instruments are available but can be classified broadly into wavelength dispersive machines which may count elements of interest either simultaneously or sequentially, and energy dispersive machines. A primary beam of x-rays stimulates the emission of secondary x-rays from each element present in the sample with intensity proportional to their quantitative concentration. The secondary radiations, each with characteristic wavelengths, can be separated geometrically using appropriate analysing crystals for individual detection and measurement (i.e. wavelength dispersion) or can be collected as mixed radiation and analysed into energy bands electronically (i.e. energy dispersion). The latter mode of operation provides poorer theoretical dispersion but, if one trusts the computer to resolve overlapping bands against a high background, offers a much cheaper, simpler, and more reliable system than wavelength dispersion (Field & Hornung, WC, 9/1995, p. 67). X-ray fluorescence becomes less sensitive as atomic weight decreases and sodium analysis near 0.1-0.2% levels is frequently marginal; one should beware of the computer-generated calibration with a zero slope and an intercept equal to the previously unvarying result.

It should be noted that x-ray fluorescence is not normally a primary method of analysis and must be calibrated from standard samples though standardless XRF is useful for approximate analysis of alternative raw materials, or special elements in common raw materials, for which no calibration has been established (Moller, WC, 9/1995, p. 75 and Boumans & Blaakmeer, WC, 8/1995, p. 67). Also, even more than wet analysis, the actual sample analysed is of the order of microgrammes and requires considerable faith in the sample reduction procedures which may start with several hundred tonnes of process material. This is why the recently applied neutron activation system which analyses a total process stream is intuitively preferable.

X-radiation only penetrates a few microns into the sample surface and powder samples must be ground to -20µ before being pressed into disks for analysis. Some minerals are prone to render the surface non-representative; examples in raw materials are quartz which may be present in large

particles and platy micreous minerals. Such materials may require fusion to avoid analytical bias. Manual fusion is time consuming and requires skill. Most laboratories now use automatic fusion equipment available from suppliers such as Philips, Krupp Polysius, Herzog, Claisse, and Phoenix (WC, 4/2003, p. 73). Sample fusion is usually carried out near 1000°C heated by a propane flame or an induction furnace with a mixture of one part sample (on ignited basis) to 5-10 parts flux (lithium meta- or tetraborate or a mixture). The melt is quickly quenched in a platinum casting dish. Oxidising and releasing agents may be added during fusion. Fused samples are essential for accurate analysis of raw materials while pressed powder pellets are appropriate for raw mix, hot meal to kiln, clinker, cement, and kiln dust. Fusion of hot meal, kiln dust and bypass dust may lead to partial volatilisation of Cl and SO_3. For volatile mass balance studies, Cl is better analysed by $AgNO_3$ titration (ASTM C114) and sulphur by high-temperature combustion (ASTM D4239). Low levels of chloride, fluoride, sulphite (in some synthetic gypsum), and sulphate in raw materials can also be analysed as anions in solution by ion chromatography (IC). These ions are separated by ion-exchange and detected sequentially by conductivity cell. Trace metals in waste fuels, waste raw materials, and kiln dust can be analysed in solution by induction-coupled-plasma (ICP) emission spectroscopy (Dirken & van der Broeck, WC, 6/2004, p. 67) or atomic absorption (AA) spectroscopy. A corporate central laboratory or a commercial testing laboratory often provides such analytical services.

Pedersen & Wismann (WC; 5/1997, p. 34) review XRF sample preparation and make an interesting point in estimating relative contributions to total analytical error as:

XRF measurement	10-15%
Sample preparation	25-30%
Sample collection	55-65%

A fused sample eliminates errors due to particle size and mineralogy. However, each element in the sample attenuates or enhances the x-radiation of other elements to varying degrees so that inter-element corrections are necessary and calibrations are not linear. While wide matrix calibration programs are available, most calibrations used for process control apply only to a narrow range of composition. If analysis is performed on pressed pellets, discretion should be exercised to avoid the use of inappropriate calibrations; for example, the use of a kiln feed calibration to analyse bypass dust.

EN 196-2.2 is the current European standard for XRF analysis of hydraulic cements. It serves as the basis of the corresponding ISO worldwide standard being developed. ASTM committee C01 is also in the process of developing a similar standard method. Presently in North America, all analytical methods, including XRF, which are used to issue a mill certificate must comply with the accuracy and precision requirements of ASTM C 114 (see below).

Prompt gamma neutron activation analysis (PGNAA) uses an isotope source of neutrons to radiate samples resulting in characteristic secondary gamma radiation. Gamma radiation being of much higher energy than x-radiation (0.5-5mev vs 1-50kev) is less subject to attenuation so that bulk material (10-100cm thick stream) can be analysed where only the sample surface is sensitive to XRF. The gamma spectrum is resolved by energy dispersion allowing rapid, on-line analysis of process streams without sample preparation. This technique is capable of analysing Si, Ca, Al, Fe, Ti, Mg, K, and S as well as Cl, N and H. Many plants now use cross-belt PGNAA analysers to control the composition of pre-blend piles and raw mix. Presently some 20 per cent of installations are for preblend and 80 per cent for raw mix control (Foster, ICR, 9/2007, p. 115). If layered materials are to be measured, as on a raw mill feed belt, it is advisable to pass the mixture over a transfer point to homogenise the stream. Accuracy and precision now approach that of XRF and sampling error is drastically reduced (Macfadyen, WC, 8/2000, p. 70). However, it is still common practice to verify on-line analyses by laboratory XRF analysis. XRF is still the standard technique for clinker and cement analyses (Foster & Bond, IEEE/PCA Technical Conference, 5/2006).

It should also be remembered that most analytical methods measure elemental concentration; thus Ca is measured but, by convention, CaO is reported.

ASTM specifies that all analytical methods and instruments used to certify cement should meet the following precision and bias criteria (ASTM C114, Table 1). Certified standards are required for verification purposes but note that calibration standards should not be used to test bias.

	Precision	Bias/Accuracy
SiO_2	±0.16%	±0.2%
Al_2O_3	±0.20%	±0.2%
Fe_2O_3	±0.10%	±0.10%
CaO	±0.20%	±0.3%
MgO	±0.16%	±0.2%
SO_3	±0.10%	±0.1%
LOI	±0.10%	±0.10%
Na_2O	±0.03%	±0.05%
K_2O	±0.03%	±0.05%
TiO_2	±0.02%	±0.03%
P_2O_5	±0.03%	±0.03%
ZnO	±0.03%	±0.03%
Mn_2O_3	±0.03%	±0.03%
S (sulphide)	±0.01%	
Cl	±0.003%	
Insol. Res.	±0.10%	
Free CaO	±0.20%	

These limits are important in themselves and indicate that major oxides should not be reported with greater precision than one place of decimals, though raw data for statistical calculation should retain two places. Thus, if a laboratory reports an oxide to two or more places, or a cement compound to more than the nearest whole number, this would suggest that they do not understand precision and may well not understand much analytical chemistry either. Chatterjee tabulated standard estimates of error, sample preparation errors, and instrument measurement errors for XRF analysis of 12 oxides in cement related materials (Innovations in Portland Cement Manufacturing, PCA, 2004, Chapter 2.3, p. 130). Other applications are described in Chapter 8.1. of the PCA book.

Indeed, to meet the requirement of rapid analysis and adjustment for efficient control of a continuous operation, increasing use is being made of automatic systems for sampling, sample conveying and preparation, and for chemical and physical measurements (Riedhammer, ZKG, 1/1994, p. E16 – Moller, ZKG, 6/2004, p. 24 and Hepper, ICR, 8/1996, p. 35).

A plant laboratory should possess a flame photometer for reference alkali determination and a bomb calorimeter for testing solid fuel. An automatic sulphur analyser (for fuel and raw materials) may be useful, and clinker microscopy should at least be available as a service if the plant laboratory is not itself equipped.

It should be noted that a cement plant making blended cement with slag is required to perform separate determinations of sulphur in sulphide and sulphate forms (C-595, and C-114; Section 15.2). In this test method, sulphide S is evolved as H_2S from an acid solution of cement and dissolved into a solution of ammoniacal zinc sulphate or cadmium chloride. The sulphide S is then titrated with a standard solution of potassium iodate (KIO_3). Alternatively, total S and sulphate S can be determined by other methods, and sulphide S is the differece between the two.

Since alternative raw materials are increasingly being employed, these raw materials may contain organic carbon and elemental carbon. Such carbon could introduce a significant heat value to the kiln system, but it could also cause environmental problems due to incomplete combustion and volatilisation during passage through preheater. To quantify such carbon, a sample can be digested with acid solution to remove carbonate and the residual carbon determined using a carbon analyser.

6.3. Evaluation of Clinker Mineralogy, Microstructure and Clinker Quality

For simplicity, cement chemists consider Portland clinker to comprise four pure compounds, C_3S, C_2S, C_3A, and C_4AF, also known as Bogue compounds as their concentrations can be calculated arithmetically from the major oxides using formulae derived by R.H. Bogue (see Section B5.1.).

C_3S	Tricalcium silicate	Alite	$3CaO.SiO_2$
C_2S	Dicalcium silicate	Belite	$2CaO.SiO_2$
C_3A	Tricalcium aluminate	Aluminate	$3CaO.Al_2O_3$
C_4AF	Tetracalcium aluminoferrite	Ferrite	$4CaO.Al_2O_3.Fe_2O_3$

In reality, the kiln burning process causes the minor elements in raw materials to be incorporated in these four compounds as solid solutions and the elemental compositions of these phases are quite different from that of pure compounds. Each phase also exists in several crystalline forms (polymorphs). The high-temperature polymorphs are stabilised by the minor elements on cooling, and usually are more reactive than the lower-temperature polymorphs. Thus clinker made in an industrial kiln should be of better quality than that of pure Bogue potential compounds; how much better will depend on the raw materials employed, the comminution and blending equipment used to prepare kiln feed, and the kiln conditions and fuel used. Industrial clinker also contains minor phases such as free-lime, free magnesia (periclase), and different forms of sulphate compounds, all of which affect clinker quality and cement performance in concrete. Thus it is important directly to determine clinker phases, to examine crystalline characteristics and microstructure, and to relate these to clinker granulometry, grindability, and reactivity. Reactivity is with water, with calcium sulphate in different forms, and with concrete admixtures. Such direct determination is facilitated by X-ray diffraction (XRD), thermal analysis (TA), microscopy, isothermal calorimetry, and by techniques for testing kiln feed burnability and clinker grindability. These procedures are not required for routine process and quality control and may be provided as a service by corporate or commercial testing laboratories.

The effects of minor elements introduced with waste materials on quality and process must be assessed together with their economic benefits (PCA ibid, Chapter 3.6. – Bhatty, Role of Minor Elements in Cement Manufacture and Use, PCA, 1995 – Moorthy, WC, 12/2003, p. 73 and Klaska et al, CI, 4/2003, p. 89).

6.4. X-Ray Diffraction

X-ray powder diffraction (XRD) offers the ability to identify and quantify chemical compounds such as $CaCO_3$, $CaSO_4$, $CaSO_4.1/2H_2O$, $CaSO_4.2H_2O$, quartz, free lime, free magnesia (periclase), clinker phases and other mineral phases in conventional and alternative raw materials.

Powder diffraction used to be of relatively low precision and was limited to comparative studies in kiln chemistry (Beilmann, ICR, 8/1993, p. 46). It is now increasingly used for rapid determination of free-lime in clinker (Bonvin et al, WC, 8/1994, p. 11) and ASTM C1365 provides a standard XRD method for phase analysis of Portland cement and clinker. Recent advances in total XRD pattern analysis, ie the Rietveld standardless method in combination with selective phase extractive procedures have improved the speed, precision and accuracy of quantitative XRD analysis (Fullmannn, ZKG, 5/2003, p. 45 and Lundgaard & Jons, WC, 4/2003, p. 59) and offers capability of online analysis (Storer &

Manias, WC, 2/2003, p. 49 and Rapson & Storer, WC, 3/2006, p. 67). This, in combination with on-line particle size analysis, facilitates the prediction of cement strengths and setting times.
It has been noted that QXRD and microscopic point-count usually give higher alite and lower belite than calculated Bogue potential C_3S and C_2S. Commercial integrated XRF-XRD units have been available for several years (see Section 6.8.), which allow simultaneous elemental and phase analyses (e.g. free-CaO and C_3S) on the same pressed powder sample (Yellepeddi & Bonvin, WC, 2/2003, p. 67).

6.5. Particle Size Analysis

There are many techniques for measuring particle size distribution (PSD). These analysers are mostly computerised, but based on different principles such as laser light diffraction by particles in random motion in a dynamic liquid or air, settling of particles in a still liquid, the effect of particles on the impedance of a sensing zone of an electrolyte solution, electron or optical microscopic image measurements, or sieving. Results from different techniques should not be expected to agree. Due to speed and ease of use, laser diffraction analysers seem to be the most widely used in the cement industry. However, there are many variables that can affect a PSD curve obtained by a laser PSD analyser. These were revealed by two "round robin" tests sponsored by ASTM. The most important variables are:

1. Powder dispersion (dry aerosol or ultrasonic alcoholic dispersion; isopropyl alcohol is the best for liquid dispersion).
2. Choice of optical models (Mie or Fraunhofel) and the values of particle refractive indices for converting light diffraction data into PSD.
3. Solid concentration.

The researchers at US NIST also reported a novel technique to extract the PSD of the gypsum fraction from the total PSD of cement. This knowledge is useful for understanding and predicting cement performance properties. The reference PSD was first available for SRM 114p and lately for the new issue SRM 114q, thanks to the co-operation of US NIST and ASTM. In the near future, laser PSD analysis will be approved as an ASTM standard method.

Commercial automatic on-line PSD analysers for kiln feed and cement have been used for several years (Kaminaga; WC; 6/1997, p. 42) with dry analysis being the more practical (Kalkert; WC; 9/2001, p. 42). On-line analysers are principally justified by shortening the transition time when multiple types of cement are ground on a single mill.

6.6. Thermal Analysis

Differential scanning calorimetry (DSC) is a simple and fast method to quantify $CaSO_4.1/2H_2O$ and $CaSO_4.2H_2O$ in cement, and to determine degree of gypsum dehydration in a cement mill. Although not yet widely practiced, this is an important aspect of cement milling process control. Gypsum or gypsum/anhydrite addition rates, and the degree of gypsum dehydration greatly influence cement performance in setting, strength, and interaction with concrete admixture. DSC is also useful in quantifying needle-shaped syngenite ($CaSO_4.K_2SO_4.H_2O$) and ettringite ($CaSO_4.K_2SO_4.H_2O$ and $3CaSO_4.C3A.32H_2O$ respectively). These compounds at low concentrations are responsible for cement dry flowability and pack-set problems. Precipitations of these compounds and of secondary gypsum during cement hydration cause early stiffening of paste, mortar, or concrete. Another useful analytical method is thermo gravimetric analysis (TGA) which determines losses of sample weight as a function of temperature up to 1600°C. TGA is used to detect pre-hydration of cement in a mill caused by over, or incorrect, spraying of water. It is also used to analyse $Ca(OH)_2$ and $CaCO_3$ in cement or weathered clinker. When trouble-shooting volatile build-ups in a kiln system, TGA of kiln process materials alone, or in combination with differential thermal analysis (DTA), provides insight into the thermal behaviour of volatile compounds such as KCl, various forms of sulphate, carbospurrite ($2C_2S.CaCO_3$), and sulphospurrite ($2C_2S.CaSO_4$) (Proceedings 7th Int. Conference on Thermal

Analysis, 1982, p. 1303). A few cement plants today are equipped with DSC. Most corporate central laboratories have TGA and DTA, either stand-alone or integrated for simultaneous analyses. Sample sizes for thermal analyses are fairly small, usually from a few mg to tens of mg. Sample preparation is, therefore, very important (PCA ibid, Chapter 8.4.).

6.7. Microscopy

Clinker microscopy is long established, but became a widespread tool for process control and problem solving with the introduction of the Ono powder-mount method outside Japan in the 1970s. There are mixed opinions of the Ono method as a routine procedure for kiln monitoring and cement strength prediction, but the technique, together with reflected light microscopy, constitutes a valuable resource for process optimisation.

The Ono method is described in his book "Ono's Method, Fundamental Microscopy of Portland Cement Clinker", Chichibu Onoda Cement Corporation, 1995. A review has been made by Campbell in a 1992 PCA report, "A Summary of Ono's Method for Cement Quality Control with Emphasis on Belite Colour". Campbell has also published a standard text "Microscopic Examination and Interpretation of Portland Cement and Clinker", 2nd Ed, PCA, 1999.

The Ono powder method, in gross simplification, involves crushing clinker and screening to -140μ/+325μ (106-44μ). 1.715 refractive index immersion oil is used to mount the sample between a glass slide and cover glass for examination at 200-400x magnification using transmitted polarised light. Measurement is made of alite (C_3S) size, alite birefringence, belite (C_2S) size, and belite colour. Belite colour is observed under un-polarized light corrected with a light-blue filter. Since alite and belite crystals are often coated with brown matrix phases, a hot aqueous solution of sugar and KOH can sometimes be used selectively to dissolve matrix phases to facilitate determination of belite colour and alite birefringence. Measuring birefringence is rather time consuming, and determining size ranges of alite and belite is somewhat subjective. However, meaningful results can be obtained with experience. Alite size (AS) indicates the rate of temperature rise of kiln charge between ca 1200-1450°C where C_2S is combining with CaO to form C_3S. Rapid heating is desirable and is indicated by alite of 15-20μ, while slow heating produces alite of 40-60μ or larger. Numerical ratings assigned to AS are:

4 = 15–20μ, 3 = 20-30μ, 2 = 30-40μ, 1 = 40-60μ.

Alite birefringence (AB) is the difference between the refractive indices of blue and red light. It is related to maximum kiln charge temperature. High temperature up to 1500°C is desirable and is indicated by birefringence of 0.008-0.010 against 0.002 for a cool burning zone. Numerical ratings assigned to AB are:

4 = 0.008-0.010, 3 = 0.006-0.007, 2 = 0.005-0.006, 1 = 0.002-0.005

Belite size (BS) reflects retention time of kiln charge in the portion of burning zone above 1400°C. A long retention is preferred and is indicated by average crystal length of 25-40μ while short retention yields 5-10μ. Numerical ratings assigned to BS are:

4 = 25-40μ, 3 = 20-25μ, 2 = 15-20μ, 1 = 5-10μ

Belite colour (BC) relates to the rate of initial clinker cooling to below 1000°C (ie cooling in the kiln before entering the cooler). Rapid cooling is desirable and results in clear belite crystals while slow cooled clinker gives yellow to amber colour. Numerical ratings assigned to BC are:

4 = clear, 3 = faint yellow, 2 = yellow, 1 = amber

Ono proceeded to correlate these four measured parameters with 28-day mortar compressive strength and reported the following multiple-variable linear correlation formulae:

$$\text{28-day strength, kg/cm}^2 = 258.1 + 5.23AS + 20.7AB + 3.41BS + 22.2BC$$

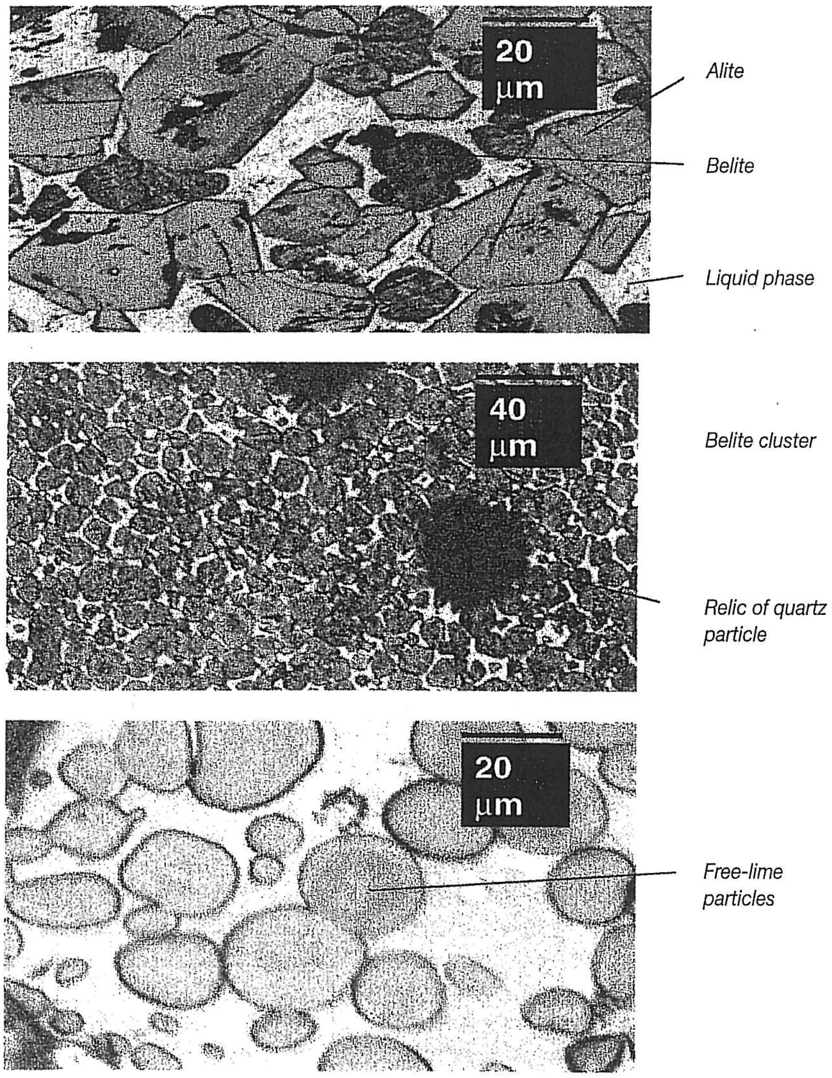
The standard deviation is 17kg/cm^2 for a range 300 – 450kg/cm^2. Or, in psi:

28-day strength, psi = 3672 + 74.4AS + 294AB + 48.5BS + 316BC

It is significant that belite colour and alite birefringence have much greater influence on strength than alite or belite size. The above correlation can apply when burning conditions are reasonably constant. Distortions, however, can result from variation of alite and belite, free-lime and alkali sulphate concentrations, gypsum dehydration, and cement fineness. Note also that hard burning and coarse kiln feed can cause large alite and belite and, thus, less reactive clinker. Under-burning can result in low quality clinker with very small crystals (15μ). Thermal profiles of long wet and long dry kilns produce less reactive clinker having large alite and belite crystals, while shorter and faster kilns with preheater and precalciner make more reactive clinker, having smaller crystals. High SO_3 in clinker tends to increase alite size and clinker dust.

Ono also related relative clinker grindability to the same set of parameters (Proceedings International Cement Microscopists Association, 1981, p.198-210):

Figure 6.1. - Clinker Micrographs



$$\text{Blaine, cm}^2\text{/g} = 1251 + 218AS + 239AB + 287BS + 2.1BC$$
$$r^2 = 0.83 \quad \text{std dev} = 87.8$$

Parallel techniques are the inspections of polished sections prepared by:

- embedding the same 106-44µ powder fraction in epoxy resin, cutting the cast, and polishing and etching the surface for examination under reflected light.
- mounting whole or crushed nodules of clinker in epoxy resin, grinding, polishing and etching, also for reflected light examination. Typical photomicrographs are shown in Figure 6.1 (supplied by Dr Hugh Wang).
- cutting, grinding and polishing a 10-20µ thin section of an epoxy resin mount for examination using transmitted light.

These techniques allow much additional data to be collected but are not as simple or rapid to perform as Ono's method. Direct estimates of cement compounds can be made (as opposed to Bogue calculations), free-lime can be estimated, crystalline MgO can be identified, and certain phenomena such as belite nesting can be observed (Chromy, ZKG, 12/1992, p. E325). The latter correlates to coarse silica particles in kiln feed and has an adverse effect upon clinker grindability, and 28-day and later strengths. Presence of different crystalline forms (polymorphs) of alite, belite, and aluminate, and other microstructures often reflect the history of kiln feed preparation, burning conditions, and clinker quality. The standard method for quantitative analysis of clinker phases by microscopy is given in ASTM C1356M. See also PCA ibid, Chapter 8.3.

A mineralogical comparison of wet and dry process clinkers produced from the same raw materials is described by Mill et al (WC, 4/2004, p. 127).

6.8. The Virtual Cement and Concrete Testing Laboratory – VCCTL

Cement manufacture involves extensive physical testing to assure product performance, and conventional tests incur delays of 28 days or longer. Since the 1980s, computational models have been developed using available experimental data of clinker, cement, aggregate, and concrete to predict cement and concrete physical properties and performance. In 2001 a consortium was established between US NIST and international partners in cement, concrete, aggregate, and admixture industries to develop a software tool named Virtual Cement and Concrete Testing Laboratory (VCCTL). A new version of this software has been issued every year. VCCTL uses computing power, and materials science models for cement hydration and microstructure, to simulate production and testing of concrete, and to predict and optimise concrete properties and performance. VCCTL will help the concrete industry's long term drive to convert from prescription-specification-based production to performance-specification-based production. It is claimed that the system is also applicable to blended cements and that, through prediction of diffusion coefficients, concrete durability can also be modelled. (Haecker et al, CI, 3/2003, p. 86).

The value of the technique is to facilitate investigation of alternative materials and process changes rather that to replace conventional testing for control and certification.

6.9. Calorimetry

An isothermal or conduction calorimeter monitors and determines the rate of heat release of hydrating cement at constant temperature over one to two days. An example of a heat release profile is shown in Chapter 10, Cement Hydration. This technique is most useful in studying cement reactivity. When several samples are run simultaneously, it provide a good comparison of reactivities of cements, and allows observation of the variation in reactivity with cement fineness, SO_3 content, the forms of calcium sulphate in cement, clinker composition, and mineral and organic admixtures. ASTM has a draft "Standard Practice for Monitoring Cement – Admixture Interactions Using Isothermal

Calorimetry". The development in using this technique as a standard method for measuring two-, three-, and seven-day heats of cement hydration was reported in the Proceedings of 11th ICCC, 2003, p. 228.

It should be noted that isothermal calorimetry cannot simulate the heat effect of massive hydrating concrete, which is nearly adiabatic. The temperature of mass concrete increases during hydration and the hydration is accelerated by the higher temperature. Thus, semi-adiabatic and adiabatic calorimeters are more suitable for concrete and these calorimeters are easy to assemble, cost much less than isothermal units, and can be used in the field (RILEM 119-TCE, RILEM Report 15, 1999, p. 315 and Wang et al, FHWA DTF 61-01-00042, January 2006).

Automatic calorific analysers for fuel are nowadays widely used in the cement industry.

6.10. Burnability

The burnability of kiln feed is often determined experimentally by burning pelletised samples for a fixed period of time, e.g., 15 or 30 minutes, in a gas or electric furnace set at different temperatures; commonly 1350°, 1400°, 1450°, 1500°, and 1550°C. After cooling and pulverising, free-lime is determined by titration or XRD. A programmable furnace may be used to approximate the thermal profile of an industrial rotary kiln and tends to yield more reproducible data than manual control of the furnace. Before changing kiln feed fineness, chemistry, or using a new raw material, the effect on laboratory burnability should be checked. A correct amount of coal ash should be included in the trial mix. Even with a correction for coal ash, clinkering in a laboratory furnace usually produces a higher free-lime than burning in a rotary kiln, probably due to the lack of mixing action in a stationary furnace and the tendency of a kiln operator to over-burn. There are also arithmetic formulae for estimating burnability factors considering only the influence of raw mix chemistry as described in Section B5.3. but such formulae ignore the effects of fineness and mineralogy.

6.11. Grindability

Grindability is a measure of the specific power needed to reduce unit mass of a material from a given initial size to a smaller product size. A comprehensive survey of literature on clinker grindability has been reported by Hills (PCA RP331, 1995). The Bond ball mill is widely used as a standard laboratory grindability test. It simulates closed-circuit grinding with a 250% circulating load. The testing procedure is described by Moore (American Ceramic Society Bulletin, 74 (8), 1995, p. 80). The test result is a Bond Work Index which is the kWh/t required to reduce a material from infinite size to 80% passing 100 micron (Section B4.9.). Some data are given in Section B7.2. and a number of factors that affect clinker grindability are described in Sections 5.2. and 6.7. above.

A number of other tests are also practiced including those of Schiller & Ellerbrock (ZKG, 6/1994, p. E164) and Theisen, (WC, 8/1993, p. 17).

The grindability of coal is conventionally tested using a Hardgrove mill (ASTM C408). Some data is again given in Section B7.2.).

It should be noted that most grinding tests involve screening of the product using 45µ (325#) or 75µ (200#) screens. It would seem apparent that both clinker and quartz would have natural grain sizes in this range so that a discontinuity in apparent grindability may occur either just above, or just below, the test product fineness. It is also possible that certain materials, such as petroleum coke, may not give rise to the crystalline fractures expected by the tests; a certain malleability may occur which, again, would seriously distort the test results.

Most grindability tests involve considerable resources and are justified only for equipment sizing or

major process engineering studies. A simple relative test involving, say, 10g of sample screened to -8#/+50# and then ground in a sample preparation ring mill for a fixed time yields a product whose Blaine fineness (cm^2/g) is proportional to grindability. Such testing rapidly rates the grindabilities of clinker from multiple kilns and can be used to assess the effects of mix design or kiln process changes.

6.12. Physical Test

Cement physical test equipment is closely defined by the cement specification(s) to which the plant is manufacturing. Cement fineness is measured by air permeability or, sometimes, by laser scattering. The other tests, which are applied to daily or batch composites, include fineness, compressive strength (1-,3-,7-,28-day), setting time, air content, and soundness. There are also numerous other tests for particular cements or particular specifications.

Different test methods, particularly for strength, can give widely different results from a single cement so that the method must be known for a strength value to have meaning.

It is possible for bias to creep into the results of even the most conscientious physical tester and, to a lesser extent, the results of classical analysis. It is strongly recommended that every plant laboratory engage in a programme of comparative testing with other laboratories (eg the semi-annual proficiency test samples of the US CCRL), so that anomalies can be identified and corrected before they affect kiln operation or cement quality.

Some automated physical testing equipment is now commercially available and is reviewed by Morrison & Walden (WC, 8/2006, p.73). An automated Vicat time-of-set test has already been approved by ASTM as an alternative method which must be calibrated against the manual method (ASTM C191). Automatic tamping machines and vibration tables are also allowed for making cubes for mortar compressive testing (ASTM C 109). Here again, the automated methods must be calibrated against the manual method. Automatic Blaine testers have been used for on-line or off-line quality control for many years but not yet for cement certification. Automatic mortar and paste mixers and automatic air-void image analysis for hardened concrete have been developed and are commercially available, but are not yet approved by ASTM.

6.13. Process Control Analysis

The function of laboratory staff extends beyond pure measurement to close co-operation with production staff and responsibility for kiln mix design and for cement quality. This involves:

- Exploratory investigation of the limestone and shale quarries to anticipate raw materials variation. This initially involves chemical analysis of drill cores during exploration and, during mining, the analysis of blast-hole cuttings. Fused standards and samples are essential for accurate analysis of raw materials.
- Monitoring and evaluation of bought-in raw materials and fuels to ensure contractual compliance and consistency.
- The optimal use of analysis and storage facilities to blend the kiln feed to minimal variation.
- The management of clinker and cement inventories.

Given the plethora of specialised analytical equipment available, a clear distinction should be drawn between those methods which give useful data necessary for process control and more exotic systems which may or may not be useful for research or problem solving, but which are inappropriate for routine plant laboratory use.

For convenience, process control usually employs composite terms such as Bogue compounds, lime saturation factor, silica ratio, etc (defined in Section B5.1.) rather than individual oxides for monitoring deviations and calculating statistical variation.

Testing frequencies of raw, intermediate, and product materials should be related to the uniformity of raw materials and process conditions. Typical frequencies are:

Quarry drill cores	- Chemical analysis, including S and Cl, every 2m.
Blast hole cuttings	- Chemical analysis of composite of each blast.
Imported raw materials	- Chemical analysis of shipment composite.
Raw mill product	- Chemical analysis of one- or two-hour composite; 200# screening of eight-hour composites. Raw mills are increasingly controlled by on-line analysis.
Kiln feed	- Chemical analysis of four- or eight-hour grab samples.
Hot meal to kiln	- K_2O, Na_2O, SO_3, Cl and LoI (800°C) on four- or eight-hour grabs.
Clinker	- SO_3, free-lime (or litre-weight) on two- or four-hour grabs. Complete chemical analysis on daily composites.
Cement mill product	- Blaine and SO_3 on two-hour composites, complete chemical, gypsum dehydration, and physical tests on daily composites.
Cement shipped	- 7- and 28-day strength testing on two or three grabs per week per C917.

It is a good practice to retain samples of cement shipped for one to three months for further testing in the event of field performance problems.

6.14. Cement Strength

The strength of cement depends upon six principal factors:

- Chemical composition
- Clinker mineralogy and burning conditions
- Fineness; Blaine and particle size distribution
- Retardation; mainly sulphate form and activity (note that most, but not all, synthetic gypsums are satisfactory for set control)
- Dilution of active components
- Pre-hydration of hydraulic components.

The simple means of increasing early strength is to grind cement finer. This, however, may have a negative effect upon concrete strength due to increased water demand. Achieving a target mortar or concrete strength should be considered as the culmination of the chemical and physical composition of raw materials, mix design, and process control and should not be related solely to finish milling parameters (Gebhardt, P&Q, 4/1995, p. 26).

C_3S primarily governs strength gain up to 28 days while C_2S has more effect on later strength. Work reported by Lea (Chemistry of Cement & Concrete, 1998) indicates that at six months, strength is largely independent of C_3S:C_2S proportions. Higher SO_3 tends to increase 28-day strength and higher alkalis tend to increase early strength and reduce 28-day strength.

Multiple regression analyses using data from round-robin tests on a large suite of all types of Portland cements from different sources, provide the following table, showing the effect of changing a quality control parameter on C109 mortar cube strengths at different ages.

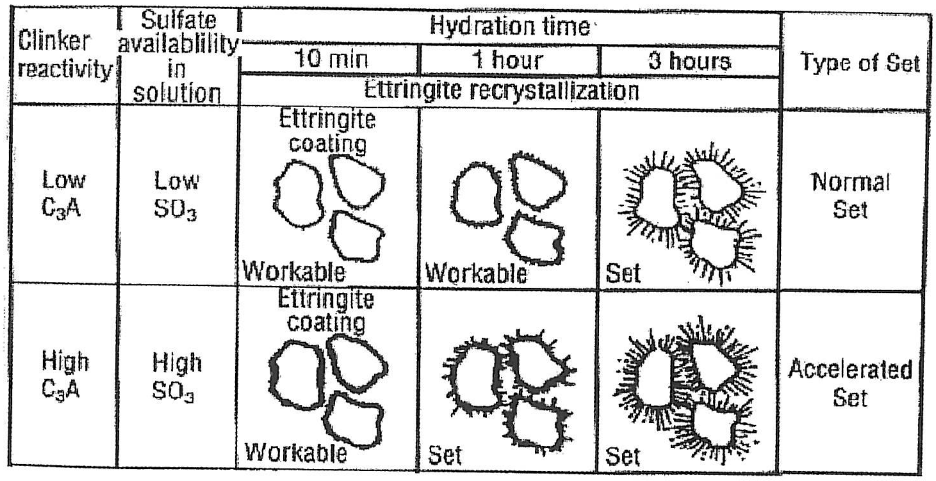
Parameter	*Change*	*Effects on C109 Mortar Cube Strengths (Mpa)*			
		1-day	*3-day*	*7-day*	*28-day*
Blaine, m²/kg	+20.	0.9	0.9	0.6	0.5
-45µm, %	+2.0	0.0	0.4	0.7	0.8
Na_2O eq, %	+0.1	1.2	0.8	0.4	-0.8
SO_3, %	+0.5	0.9	1.0	1.2	1.0
free-lime, %	+0.3	-1.2	-0.5	-0.4	-0.6
C_3A, %	+1.0	0.0	0.1	0.0	0.0
C_3S, %	+5.0	1.5	0.7	0.0	0.0
LoI, %	1.0	-0.8	-1.0	-1.2	-1.0
Mortar air, %	1.0	0.0	0.0	-0.3	-0.5

The above correlations do not include the effects of clinker burning conditions, clinker mineralogy, forms of calcium sulphate, and particle size distribution. These parameters are also significant, but are not routinely monitored and controlled at a cement plant.

6.15. Setting Time

Setting of cement is caused by the development of cross-linking structure of hydration products shortly after the dormant period (see Chapter 10, Hydration of Portland Cement). This is illustrated by Figure 6.2. from Locher et al (ZKG, 6/1980, p. 271). Good control of setting is achieved by matching C_3A reactivity with sulphate availability in solution.

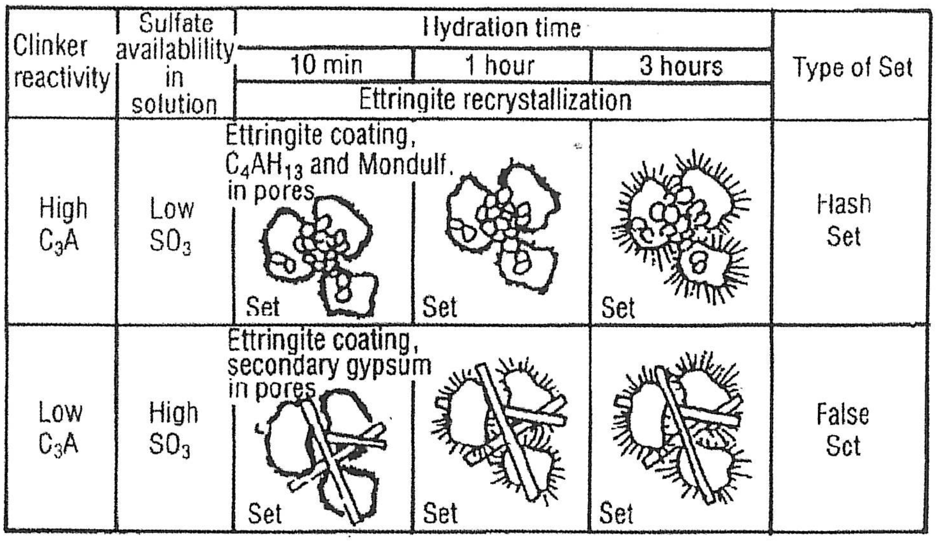
Figure 6.2. – Cross Linking of Hydration Products



Clinker reactivity	Sulfate availability in solution	Hydration time 10 min	1 hour	3 hours	Type of Set
		Ettringite recrystallization			
Low C_3A	Low SO_3	Ettringite coating Workable	Workable	Set	Normal Set
High C_3A	High SO_3	Ettringite coating Workable	Set	Set	Accelerated Set

Unbalance between C_3A reactivity and sulphate availability can cause flash set or false set as shown in Figure 6.3.

Figure 6.3. – Flash and False Set



The primary factors that affect normal setting time are free-lime and the forms of calcium sulphate (see Section B7.4.). The secondary factors are fineness and C_3A content. Figure 6.4. from Hanson et al (Gypsum & Anhydrite in Portland Cement, 3rd Ed., p. 57) shows the effects of various forms of calcium sulphate:

The effects of other parameters on Vicat initial setting time (VIST) can be appreciated by the following correlation:

$$\text{VIST (min.)} = 216 - 44 \times (\%\ \text{free-lime}) - 0.125 \times (\text{m}^2/\text{kg}) - 2.22 \times (\%\ C_3A)$$

Thus, it is apparent that free-lime has much greater effect on setting time than do Blaine or C_3A. The impact of gypsum dehydration is shown in Figure 6.5.

Figure 6.4. – Effects of Gypsum Form on Setting Time

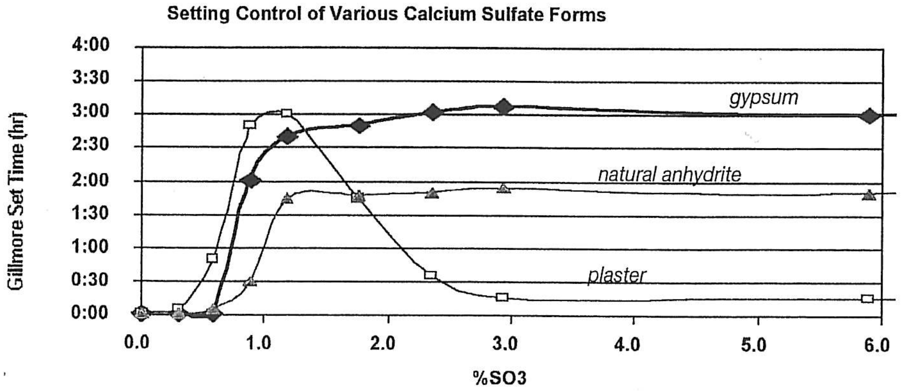
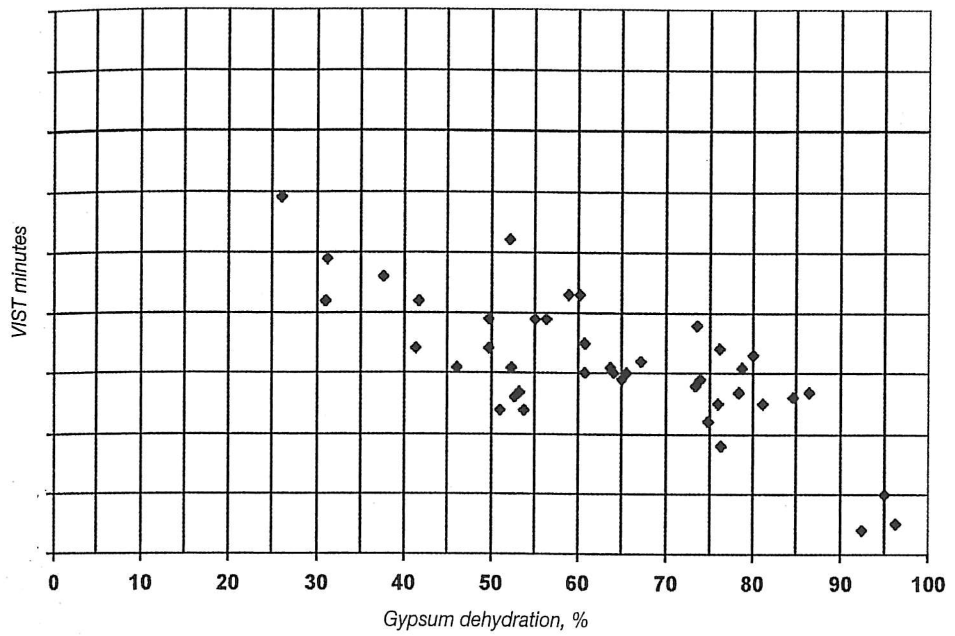


Figure 6.5. – Effects of Gypsum Dehydration on Setting Time



6.16. ASTM Cement Types and Specifications

There are numerous national and international cement specifications together with the American Petroleum Institute specification for oilwell cements (API Specification 10A) which latter has virtually universal recognition (Bensted, WC, 2/1995, p. 40). The European specification for construction cements, EN 197 is quite different from those of North American; ASTM C150, C595, C1157, C91, C1328, C1329; CSA A3001, A3002. The repective cement types and associated testing methods are listed below.

It is a curiosity that the trading of clinker cannot relate to a general specification; chemical ranges on the clinker, and physical performance of the cement produced from the clinker, can be used contractually but are of limited real value. And clinker sales, especially if not to a regular recipient, provide plant managers with an irresistible temptation to dispose of old or hydrated rubbish.

It should be noted that the results of different specification tests may not be directly comparable. In particular, different tests for compressive strength may use differing cement contents, water-cement ratios, and mixing and moulding methods; all of which can lead to wide disparity in quoted strengths from identical cement samples.

ASTM C150 SPECIFICATION FOR PORTLAND CEMENT

Type I	General purpose
Type II	Moderate heat/moderate sulphate resistance
Type III	High early strength
Type IV	Low heat of hydration (rare)
Type V	High sulphate resistance
A	Air-entraining
LA	Low alkali

Associated ASTM test methods & specifications:

C109	Compressive strength
C114	Chemical analysis
C151	Autoclave expansion
C185	Mortar air content
C186	Heat of hydration
C187	Normal consistency
C191	Vicat setting time
C204	Fineness by air permeability
C265	Water extractable sulphate
C266	Gillmore setting time
C359	Early stiffening/false set/flash set (Mortar method)
C451	Early stiffening/false set/flash set (Paste method)
C452	Mortar expansion of Portland cement – gypsum mixture in water
C465	Processing additions
C563	Optimum SO_3
C917	Evaluation of cement strength uniformity
C1012	Mortar expansion in sulphate solution
C1038	Mortar expansion of cement in water
C1565	Pack Set Index (from 2004)

Chemical Limits (%):

	I *IA*	*II* *IIA*	*III* *IIIA*	*IV*	*V*
SiO_2, min		20.0			
Al_2O_3, max		6.0			
Fe_2O_3, max		6.0		6.5	
MgO, max	6.0	6.0	6.0	6.0	6.0
SO_3, max (C_3A<8)	3.0	3.0	3.5	2.3	2.3
(C_3A>8)	3.5		4.5		
LoI, max	3.0	3.0	3.0	2.5	3.0
Insol res, max	0.75	0.75	0.75	0.75	0.75
C_3A, max	8	15			
C_4AF+2C_3A				25	
(Alternative)					
C_3A, max				7	5
C_4AF+2C_3A				25	
C_3S, max				35	
C_2S, min				40	
(Optional)					
C_3S+C_3A, max (moderate heat)		58			
$Na_2O+0.658K_2O$, max (LA)	0.60	0.60	0.60	0.60	0.60

Physical Limits:					
Air content, max	12	12	12	12	12
(A), max		22	22	22	
min		16	16	16	
Fineness, min, m²/kg	280	280	280	280	280
Autoclave exp, max, %	0.80	0.80	0.80	0.80	0.80
Strength, min, MPa					
1-day			12.0		
(A) 1-day			10.0		
3-day	12.0	10.0	24.0		8.0
(A) 3-day	10.0	8.0	19.0		
7-day	19.0	17.0		7.0	15.0
(A) 7-day	16.0	14.0			
28-day				17.0	21.0

(note A cements have lower strength requirements)
Gillmore IST/FST, min - 60/600 minutes all types
Vicat, IST/FST, min - 45/375 minutes for all types

Typical cement compound concentrations for the five cement types are:

Type	I	II	III	IV	V
C_3S	55	51	56	28	48
C_2S	19	24	19	49	30
C_3A	10	6	10	4	4
C_4AF	7	11	7	12	10

Additions are limited to limestone (under certain specifications, including ASTM C150 up to 5%), water, calcium sulphate (and air entraining agent). At the option of the manufacturer, process additions (primarily grinding aids) may be used which meet C465. C465 requires comparative testing with/without additives; difference shall not exceed:

1. Water requirement for normal consistency shall not increase by more than 1%
2. Setting time (Vicat initial) shall not increase/decrease by more than the lesser of 1h or 50%
3. Autoclave expansion shall not increase by more than 0.10
4. Grand average of strengths at all ages shall not decrease by more than 5%
5. Difference in SO_3 <0.3%
6. Difference in Blaine <13m²/kg
7. Drying shrinkage <0.025%.

Under C465, various additives such as blast furnace slag have also been used.

White cement is a special Portland cement with very little ferrite phase (<1%) and relatively high aluminate and silicates contents. Its whiteness is achieved by using raw materials containing very little iron, manganese, and chromium, using gas or other low-ash fuel, burning clinker under reducing conditions, quenching in water, and grinding the cement to a high fineness. Cement whiteness is impaired if Fe_2O_3 is >0.50%, Mn_2O_3 >0.02%, and Cr_2O_3 >0.01%. The use of pure sand and limestone and the low liquid content of the raw mix require clinker burning at 1600°C, usually with a mineraliser (CaF_2). However, specific fuel consumption is considerably greater than for grey cement. Quenching

CEM III	Blast furnace cement	CEM III/A	35-64	35-65	0-5
		CEM III/B	20-34	66-80	0-5
		CEM III/C	5-19	81-95	0-5
CEM IV	Pozzolanic cement	CEM IV/A	65-89	11-35	0-5
		CEM IV/B	45-64	36-55	0-5
CEM V	Composite cement	CEM V/A	40-64	18-30 + 18-30	0-5
		CEM V/B	20-38	31-50 + 31-50	0-5

Siliceous fly-ash is <10% CaO; calcareous fly-ash >10% CaO.
Limestone L has <0.5% organic carbon; LL has <0.2% organic carbon.
Composite cements include both slag and silica-fume/pozzolan/fly-ash.

Physical Requirements

Strength class	*Compressive Strength, MPa* *2 day*	 *7 day*	 *28 day*	*Initial set time,* *min*	*Soundness,* *Mm*
32.5 N		>16.0	32.5 – 52.5	>75	
32.5 R	>10.0	-			
42.5 N	>10.0	-	42.5 – 62.5	>60	<10
42.5 R	>20.0	-			
52.5 N	>20.0	-	>52.5	>45	
52.5 R	>30.0	-			

N = Ordinary cement R = High early strength cement

Chemical Requirements

Property	*Test reference*	*Cement type*	*Strength class*	*Requirements*
Loss on ignition	EN 196-2	CEM I, III	all	<5.0%
Insoluble residue	EN 196-2	CEM I, III	all	<5.0%
SO_3	EN 196-2	CEM I, II, IV, V	32.5N, 32.5R, 42.5N	<3.5%
			42.5R, 52.5N, 52.5R	<4.0%
		CEM III	all	<4.0%
Cl	EN 196-21	All	all	<0.10%

clinker with water and trying to recover the waste heat pose challenges to the process and equipment (Pekin, ICR, 12/2000, p. 35 – Clark, ICR, 7/2001, p. 43 and Schulz, ZKG, 2/2003, p. 82).

Various pastel-coloured cements can be made by mixing or, preferably, intergrinding pigments with white cement (Bensted, WC, 2/1993, p. 13). Darker colours such as red and brown may also be produced from grey cement. Pigments are usually inorganic and should be durable to light and weathering, non-soluble, and not reactive with cement. Common pigments, added as 5-10% by weight of cement, are:

Iron oxides	red, yellow, brown
Manganese oxide	black, brown
Chromium oxide	green
Cobalt oxide	blue
Carbon	black

Masonry Cement is used for mortar in masonry construction where good workability and rapid hardening are required. Various inter-ground ingredients may be used, but commonly 20-50% limestone is incorporated together with an air-entraining agent. Masonry cement is typically ground to 5000-6000cm²/g and ASTM specification C91 defines three grades: N, S, and M with increasing strength requirements. ASTM C1328 specifies plastic (stucco) cement for plastering applications. There is very little difference between C1328 and C91. ASTM C1329 specifies mortar cement, which is basically masonry cement with a lower air content limit and with additional bond strength requirement.

6.17. European EN 197 Cement Specification

EN 197 – 1 :2000 Cement – Part 1 : Composition, specifications and conformity criteria for common cements.

Type	*Notation*	*Symbol*	*Composition*		
			Clinker	*Additive*	*Gypsum etc*
CEM I	Portland cement	CEM I	95-100%		0-5%
CEM II	Portland slag cement	CEM II/A-S	80-94	6-20%	0-5
		CEM II/B-S	65-79	21-35	0-5
	Portland silica-fume cement	CEM II/A D	90-94	6-10	0-5
	Portland pozzolana cement	CEM II/A-P	80-94	6-20	0-5
		CEM II/B-P	65-79	21-35	0-5
		CEM II/A-Q	80-94	6-20	0-5
		CEM II/B-Q	65-79	21-35	0-5
	Portland burnt shale cement	CEM II/A-T	80-94	6-20	0-5
		CEM II/B-T	65-79	21-35	0-5
	Portland fly-ash cement	CEM II/A-V	80-94	6-20	0-5
		CEM II/B-V	65-79	21-35	0-5
		CEM II/A-W	80-94	6-20	0-5
		CEM II/B-W	65-79	21-35	0-5
	Portland limestone cement	CEM II/A-L	80-94	6-20	0-5
		CEM II/B-L	65-79	21-35	0-5
		CEM II/A-LL	80-94	6-20	0-5
		CEM II/B-LL	65-79	21-35	0-5
	Portland composite cement	CEM II/A-M	80-94	6-20	0-5
		CEM II/B-M	65-79	21-35	0-5

Associated Testing Standards:

EN 196-1	Methods of testing cement – Part 1: Determination of strength
EN 196-2	Methods of testing cement – Part 2: Chemical analysis of cement
EN 196-3	Methods of testing cement – Part 3: Determination of setting time and soundness
EN 196-5	Methods of testing cement – Part 5: Pozzolanity test for pozzolanic cements
EN 196-6	Methods of testing cement – Part 6: Determination of fineness
EN 196-7	Methods of testing cement – Part 7: Methods of taking and preparing samples of cement
EN 196-8	Methods of testing cement - Part 8: Heat of hydration, solution method
EN 196-9	Methods of testing cement - Part 9: Heat of hydration, semi-adiabatic method
PrEN 196-10	Methods of testing cement - Part 10: Soluble Cr^{+6} in cement
EN 196-21	Methods of testing cement – Part 21: Determination of chloride, carbon dioxide and alkali content of cement

EN 197-2 Cement – Part 2: Cement Conformity Evaluation

This standard specifies the scheme of evaluation of cements to their corresponding product specifications in EN 197-1. A cement manufacturing facility is required to implement a documented production quality management system quite similar to ISO 9000, but with additional auto-control of released cement. For auto-control, rules are laid out for cement sampling, preparation, retention, testing, and statistical evaluation of test results. Criteria are set for conformity to the applicable standards. Specified corrective actions must be taken in case of nonconformity. If in conformity, the product is certified and the CE mark is applied to the product. ASTM C183 is somewhat similar to EN 197-2, but it is not as comprehensive and is not mandatory.

6.18. Cement Intergrinds

Apart from natural pozzolan, there are two waste materials, which are commonly mixed, or inter-ground, with cement under ASTM C595: pulverised coal fly ash (PFA) and ground granulated blast furnace slag (GBFS). Silica fume, a by-product of electric-arc manufacture of ferro-silicon and silicon metal, is also used to a much lesser extent. Pozzolans are materials that are not themselves cementitious but are latently hydraulic; i.e. they react with lime (CaO) released during hydration of Portland cement to form compounds with cementitious properties. Although generally slower to develop strength than straight Portland cements, these blended cements reduce the porosity of concrete and, in the presence of moisture, promote self-healing of cracks. The result is cement with reduced heat of hydration, reduced alkali-aggregate reactivity, and increased sulphate resistance. Slag and fly ash are considered pozzolanic for the present purposes though, containing some CaO, they may themselves be slightly cementitious. Blast furnace slag is hydraulic if activated by sodium hydroxide, silicate, or calcium sulphate.

C595 requires that the manufacturer shall, on demand, state the source and amount of pozzolan added.

Although C595 refers only to blended cements, a common practice is to add the same pozzolanic materials at the concrete batch plant. Pozzolanic materials employed thus are covered by ASTM C618 (fly ash and natural pozzolan) and C989 (blast-furnace slag).

Natural pozzolans are mainly volcanic materials but include some diatomaceous earths. They are soft to grind. ASTM C618 requires that $SiO_2+Al_2O_3+Fe_2O_3$ > 70% and addition rate is 15-40%.

Fly ash is obtained in large quantities from coal-fired power generation. The ash is principally an aluminosilicate material in the form of small hollow glass spheres; this results in very low bulk density (ca 0.8t/m^3) which impacts transportation and storage costs. ASTM C618 classifies fly ash as "F" with SiO_2+Al_2O_3+Fe_2O_3 >70% and "C" if >50%. Some Class C fly ashes contain sufficient CaO to be themselves appreciably cementitious. Addition rate is 15-40% of total. Alternatively to inter-grinding, much fly ash is used as a concrete admixture. Ash is increasingly liable to contain unburned carbon due to the use of low-NO_x burners in power boilers. For inter-grinding or mixing in concrete, ASTM C618 requires that loss on ignition (carbon) should be less than 6%. High carbon ash may be suitable as a kiln raw material.

Typical compositions, with wide variations, are:

	Cement	*Pozzolan*	*Fly-Ash*	*Slag*
CaO %	65.0	5.0	5.0	40.0
MgO %	1.5	2.0	2.0	5.0
SiO_2 %	21.0	60.0	45.0	35.0
Al_2O_3 %	5.0	15.0	20.0	15.0
Fe_2O_3 %	3.0	5.0	10.0	0.3
Bulk density, t/m^3	1.5	1.6	1.0	1.8

Granulated Blast Furnace Slag for cement inter-grinding must be water quenched in order to fracture the glass structure and expose active surface. The activity of slag increases with the ratio (CaO+MgO)/(SiO_2+Al_2O_3) and with glass content; the two, however, appear to be inversely related. Addition rate is typically 25-70%. Ground GBFS is specified in ASTM C989.

ASTM C595 SPECIFICATION FOR BLENDED HYDRAULIC CEMENTS

Pozzolan or slag used for inter-grinding must satisfy requirements for pozzolanic activity; primarily that a standard inter-grind shall have not less than 75% of the 28-day strength of the control cement mortar.

Types covered are:

Type IS	Portland blast furnace cement containing 25-70% slag.
Type IP	Portland pozzolan cement containing 15-40% pozzolan.
Type P	Portland pozzolan cement containing 15-40% pozzolan but with slower strength development than IP.
Type S	Slag cement containing at least 70% slag normally used with hydrated lime for making masonry mortar.
Type I(PM)	Pozzolan modified Portland cement containing less than 15% pozzolan.
Type I(SM)	Slag modified Portland cement containing less than 25% slag.

All of these types can be specified for moderate sulphate resistance (MS), air entrainment (A), or for moderate heat of hydration (MH).

Chemical Limits (%):	*I(SM),I(S)*	*S*	*I(PM),P,IP*
MgO, max			6.0
S as SO_3, max	3.0	4.0	4.0
Sulphide S, max	2.0	2.0	
IR, max	1.0	1.0	
LoI, max	3.0	4.0	5.0

Physical Limits:		*I(SM),IS,* *I(PM),IP*	*IS(MS)* *IP(MS)*	*S*	*P*
Autoclave expansion, max, %		0.80	0.80	0.80	0.80
contraction, max, %		0.20	0.20	0.20	0.20
Vicat IST, min, minutes		45	45	45	45
FST, max, hours		7	7	7	7
Air content, max, %		12	12	12	12
Water requirement, max, %					64
Drying shrinkage, max, %					0.15
Strength, min, MPa	- 3 day	13.0	11.0		
	- 7 day	20.0	18.0	5.0	11.0
	- 28 day	25.0	25.0	11.0	21.0

In 1992, ASTM approved an alternative, performance-based, specification for blended cements, C1157. This standard was later revised to include Portland cements. Adoption has so far been minimal but it is the intention of ASTM that performance based specification will eventually displace the present mix of compositional and performance criteria.

Rejection is permitted under ASTM C150 if any requirements are not met.

6.19. Supersulphated Cement

Highly resistant to seawater, sulphate attack, and acid attack, this cement is made by inter-grinding 80-85% granulated blast furnace slag, 10-15% anhydrite and 5% Portland cement clinker to about 450m²/kg. Ettringite is the primary hydration product, which develops strength. Therefore, a slag high in alumina (15-20) is preferred. Some European standards require a minimum slag content of 75% (Lea, Chemistry of Cement and Concrete, 4th Ed. [1998], p. 187 and Taylor, Cement Chemistry, 2nd Ed. [1997], p. 272).

6.20. Calcium Aluminous Cement (CAC)

A cement with high resistance to acid or sulphate attack can be made using a mix with high alumina and high iron contents. Typical composition of Ciment Fondu is:

SiO_2	3.5-5.5%
Al_2O_3	37-41%
CaO	35-39%
Fe_2O_3	9-10%
FeO	5-6%

A mixture of limestone or chalk and bauxite is charged into the upright stack section of an L-shaped reverberatory open-hearth furnace. The mixture is fired with pulverised coal or oil. The charge is preheated and calcined by combustion gases and then passed into the hearth section of the furnace where it is melted above 1450°C and homogenised. The molten product is continuously withdrawn by means of a pan conveyor where it cools and solidifies as clinker. The clinker is crushed and then ground to 250-400 m²/kg in a ball mill. A setting regulator is not added during the grinding stage. Bauxite, fuel and power costs are all high. Other high alumina cements with 70-80% alumina can be sintered in a rotary kiln and are used both for refractory and construction purposes. CAC is graded by Al_2O_3 content. The higher the alumina content, the more rapid the hardening and the better the refractory properties. The typical chemical compositions are:

	Low-grade	*Medium-grade*	*High-grade*
SiO_2	3-8%	5-8%	0-0.5%
Al_2O_3	37-42%	49-52%	68-80%
CaO	36-42%	39-42%	17-20%
Fe_2O_3	11-17%	1.0-1.5%	0.0.5%

The clinker minerals are CA, CA_2 (in high grade only), $C_{12}A_7$, C_2AS, C_6A_4S, and C_4AF.

Before the hydration sequence of aluminous cement became well understood, there were a few well-documented failures of buildings constructed with high alumina cement concrete beams. The major phase of the cement, CA, initially hydrates into metastable CAH_{10}, and eventually converts into stable C_3AH_6. This conversion is accompanied by a reduction in volume of the hydrate and the evolution of water. This increases the porosity of the hydrated mass and reduces strength. Failure could be prevented, if a low water:cement ratio were used, because un-hydrated cement continues to react with the available water and keeps filling the new pores with new hydration product. Unlike Portland cement, calcium hydroxide is not a hydration product of aluminous cement. Therefore, aluminous cement provides rapid setting and strength development (one-day strength higher than Portland 28-day), good resistance to abrasion, to high temperature, and to sulphate and weak acid attacks (Clark, ICR, 1/2002, p. 57). The disadvantage is price, which is four to five times that of Portland cement. The primary application of CAC is for refractories. The secondary uses are concrete repair, self-levelling floor screeds, fireplaces, sewer pipe lining, mining, and fertilisers (Clark, ICR, 1/2002, p. 57).

6.21. Shrinkage-Compensating Cements (SCC)

Shrinkage-Compensating cement is used to minimise or eliminate the cracking of Portland concrete due to drying shrinkage. This is achieved by modifying Portland cement by adding more calcium sulphate and an expansive agent that produces controlled expansion on hydration. The most widely used expansive agents are $3CA.CaSO_4$ (kleinite compound), CA, $C_{12}A_7$, and C_3A. On hydration, these compounds all expand due to formation of ettringite ($C_3A.3CaSO_4.32H_2O$). The cement containing kleinite compound is called Type K in ASTM C845, that containing CA and $C_{12}A_7$ is Type M, and Portland cement high in C_3A and SO_3 is Type S. Less commonly used expansive agents are dead-burned free MgO and free CaO. ACI 223 covers standard practice for concrete from shrinkage-compensating cements.

Single SCC clinkers can be made, but it is more flexible to produce mixtures that are blended or interground with normal Portland cement or clinker. Blending has the advantage that fineness and addition rate of SCC admixture can be optimised.

The composition of kleinite expansive clinker can be varied considerably to utilise the raw materials most economically available; usually gypsum, bauxite, limestone, or industrial by-products. The raw mix can be sintered in a rotary kiln at temperatures not higher than 1300°C. Clinkers made in the USA typically contain 8-50% kleinite compound, with anhydrite, alite, belite, and some free-lime. (Lea ibid, p. 830: and Taylor ibid, p. 31). If 10-20% of expansive clinker is ground with 75% Portland cement and 10% granulated blast furnace slag, the shrinkage of Portland cement is compensated. Both proportioning and curing are critical (Zivica, WC, 11/1999, p. 105).

Approximately 90% of SCCs consist of the constituents of conventional Portland cement. The kind of aluminate used influences the rate and amount of ettringite formation at early ages and thus the expansion. Total potential expansion is governed by the amount of aluminate and calcium sulphate, and by the rate at which they form ettringite (ACI 223).

The following are the typical compositions of Type K SCCs made in the USA (Taylor ibid, p. 314):

	#1	#2
SiO_2	21.6%	18.8%
Al_2O_3	4.8%	5.7%
CaO	62.4%	61.4%
Fe_2O_3	2.8%	1.9%
MgO	0.7%	3.4%
SO_3	6.2%	6.5%
LoI	1.1%	1.5%

Type M cement is made by blending calcium aluminate cement, or aluminous slag plus lime, with Portland cement.
Type S cement is made from Portland cement clinker with a high C_3A content. This cement has limited applications.

6.22. ISO 9001:2000 Quality Management System

The previous edition of ISO 9000, issued in 1994, has been replaced and significantly revised. The standards, developed by the International Standards Organization in Geneva, remain, however, an instrument to help an organisation to implement a process of quality management and continuous improvement. They are not, themselves, product or service quality standards.

The simplified ISO 9002 and 9003 standards have been withdrawn and cement plants now come under ISO 9001:2000 although some elements are not applicable (e.g. design & development, custody of customer property).

- ISO 9000 contains fundamentals and vocabulary
- ISO 9001 contains system requirements
- ISO 9004 contains guidelines for performance improvement
- ISO 90011 contains guidelines on system auditing

There are some 250 requirements in ISO 9001 under eight headings:

- 0 Introduction (includes the process approach; plan, do, check, correct)
- 1 Scope (includes reasons for implementing ISO 9001)
- 2 Normative Reference
- 3 Terms and Definitions
- 4 Quality Management System (includes identification of processes, quality manual, control of documents)
- 5 Management Responsibility
- 6 Resource Management
- 7 Product Realization – Manufacturing (includes quality objectives, testing activities, compliance of purchased materials & services, traceability, control and calibration of testing equipment)
- 8 Measurement, Analysis, and Improvement (includes monitoring of customer satisfaction,

internal quality audits, continual improvement)

Hoyle (ISO 9000:2000 An A-Z Guide, Butterworth, 2003) condenses the process into five linked tasks:

1. Identify the needs of customers.
2. Establish policies, objectives, and an organisational environment conducive to meeting these needs .
3. Design and manage the process of implementation.
4. Monitor the effectiveness of the process.
5. Seek continual improvement by raising standards and improving the process.

Establishing the programme involves extensive documentation of operating procedures and training to ensure that such procedures are understood and implemented. Certification is conducted by nationally accredited organisations and is the process whereby (normally) the plant is deemed to meet specified requirements. Re-certification is required every three years.

A number of claims are made for the efficiencies and cost savings which stem from implementation; this is more evident for widget manufacture if the rejection or repair rate is reduced, but is less convincing for cement manufacture. This is not to contest the benefits of appropriate procedures and training but these are, in any case, good management practices and predated formalised quality management. More persuasive, however, is a recognised worldwide trend towards adoption of ISO 9000 with the increasing likelihood that major engineers and contractors will require certification of cement manufacturers in order to meet standard procurement terms.

6.23. Concrete Problems

The design of concrete mixes and their correct preparation is an extensive and complex subject which is not of direct concern to the cement producer. The manufacture of cement is more sophisticated than the production of concrete and present cement quality demanded by the market is substantially higher than specification minima. However, while cement as shipped is seldom out of specification, variation in cement properties due to inadequate materials and process control can seriously compromise concrete quality. Cement manufacture allows of sufficient intermediate and final product storage and monitoring to permit correction, if necessary, before shipment while concrete producers are entirely dependent upon the quality and consistency of their raw materials, including cement. Concrete test data is usually available after the concrete has been placed so that cement variability can only be compensated by designing for the lowest probable strength – a wasteful and expensive recourse for the concrete producer. Consistent cement strength allows a concrete producer to design concrete with a smaller safety margin using a lower cement factor. This materially reduces the cost of concrete production. Virtually all concrete mixes involve additives to modify cement performance and cement-admixture incompatibility is a potential problem. It is believed that close technical communication between cement and concrete producers is invaluable for both sides.

A typical concrete mix may be:

Water	7% by weight
Cement	13%
Fine aggregate, -6mm (dry)	32%
Coarse aggregate, +6mm (dry)	48%
Water reducer	0.25% of cement weight]

which yields a concrete with:

Slump (ASTM C143)	7-10cm
Density	ca 2300kg/m^3 (3900lb/yd^3)
	28-day strength (ASTM C39) ca 250kg/cm^2 (3500psi).

For more detail on concrete mix design and performance see Kosmatka & Panarese, Design and Control of Concrete Mixtures, PCA. Virtually all concrete mixes involve admixtures (see Ramachandran, Concrete Admixtures Handbook – Properties, Science and Technology; 2nd Ed. [1995]). Admixtures fall into three groups, which are self-explanatory:

- Air -entraining agents
- Accelerators
- Water reducing and set retarding agents also known as plasticisers and superplasticisers

Air entraining agents may be added either by inter-grinding in cement or directly to the concrete mix. Desired air content is usually 4-7% by volume, the higher contents being employed where the concrete is subject to freeze-thaw cycles. Increasing air content, however, yields lower strength.

It is a curiosity of cement manufacture that the product is tested and certified either as a paste of cement and water or as a mortar containing standard sand. Such testing can give only a limited indication of performance in concrete. Specifically, cement strength tests ignore both water demand, which varies significantly with fineness, and possible interactions with the wide range of concrete additives. Optimum SO_3 (C563) is the level which maximises mortar strength at one-day. However, higher SO_3 contents are required to maximise strengths at later ages or at elevated temperatures. Higher cement fineness, C_3A and alkali contents also demand more SO_3 for maximum strength. Concrete containing water reducer usually requires a higher optimum SO_3 (Tagnit-Hamou & Aitcin, WC, 8/1993, p. 38). It is also known that high alkali cement requires less air-entraining additive so that alkali variation in cement can cause strength variation in concrete. Helmuth has suggested that the optimum ratio of SO_3:Al_2O_3 is about 0.6 for Portland cement with a normal particle size profile, and 0.8 for cement with a narrow particle size range (Detwiler, ZKG, 7/1995, p. 384).

In general, strength decreases with increased water content and the minimum water allowing satisfactory workability should be used. This is a common source of problems as concrete workers find the addition of excess water a convenience. Water reducers are widely used to improve concrete workability. Normal water reducers may decrease the water requirement of concrete by 10-15% and more effective superplasticisers are capable of reducing it by up to 30%. These additives may be used to increase concrete strength at the same slump, or to increase slump for easy placing or pumping without loss of strength. Water reducers are more effective for concretes incorporating low-alkali and low-C_3A cements.

Potential problems with concrete are numerous and their investigation and solution is a specialist task. However, as cement is frequently, and usually incorrectly, blamed for defects, some common areas of contention may be summarised.

Alkali-silica reactivity (ASR) or alkali-aggregate reactivity, which leads to the habitual specification of low alkali cement by engineers in some countries, is of concern to, but outside the control of, cement producers. While alkali-aggregate reactivity is a serious problem, reactive aggregates are not ubiquitous and the unnecessary specification of low alkali cement frequently entails additional production cost and the disposal to landfill of process dust. Mather (e.g. Proceedings PCA-GTC, 9/1982) has argued that cement should be specified with more discretion, as the 0.6% (ASTM C150) limit may be unnecessary for many aggregates while being still too high for others. Furthermore, the known effect of certain pozzolan and slag inter-grinds in reducing alkali reactivity should be taken into account.

Cracks – deep can be due to:

- high slump concrete with consequent high shrinkage
- poor aggregate gradation
- rusting of re-bar too close to surface

- structural settlement from lack of footings
- restrained concrete or inadequate relief joints
- alkali-aggregate reactivity
- freeze/thaw of non-air-entrained concrete

Cracks, when stable, can be repaired by injecting epoxy.

Cracks – plastic shrinkage are random, relatively deep cracks which form away from the edge of the slab while still plastic. They are caused by:

- rapid drying of surface
- low bleeding characteristics (mitigate with chemical admixture or by adjusting fine aggregate gradation)
- sub base, aggregate, or formwork not pre-saturated.

Cracks – surface may be due to:

- rapid or premature surface drying of finished concrete
- excessive working or premature floating which causes a high cement-content surface
- high slump (excessive water content)
- dusting on of dry cement to hasten drying.

Delayed ettringite formation (DEF) has recently achieved some notoriety. Ettringite ($3CaO.Al_2O_3.3CaSO_4.31H_2O$) forms early in the hydration of cement leading to setting and hardening of cement. It appears that under certain circumstances, principally curing temperatures above about 70°C, ettringite decomposes and subsequently reforms in the presence of water to cause expansion and cracking of hardened concrete. The search for causes has become somewhat emotional but, to date, there is no evidence that this is the result of anything but incorrect concrete practices (Stark & Bollmann, ZKG , 4/2000, p. 232 & 5/2000, p. 298).

Efflorescence is usually due to passage of water through cracks carrying un-reacted lime to the surface where it reacts with CO_2 to precipitate $CaCO_3$. It is prevented by ensuring that the concrete is watertight. Occasionally, other water-soluble salts such as alkali sulphates can be involved. Efflorescence can be cleaned by dilute solutions of hydrochloric, acetic, or phosphoric acid.

False set is the result of gypsum dehydration due to overheating of cement ($CaSO_4.2H_2O$ -> $CaSO_4.1/2H_2O$). Upon mixing with water, crystallisation of reformed gypsum causes stiffening of the mix within a few minutes. Usually false set is only a problem with very rapid mixing systems as false set can be broken up after a few minutes and no subsequent problems result. Rapid precipitation of ettringite ($3CaO.Al_2O_3.3CaSO_4.31H_2O$) and/or syngenite are also possible causes of false set. The desirable form of ettringite is that which forms on the surface of C_3A thereby retarding its hydration. Aerated or carbonated cement has a strong tendency to false set and strength loss and, while false set is not usually accompanied by significant release of heat, setting time (C191) is often lengthened.

The standard penetration test methods for false set are C359 for mortar and C451 for paste. Although C451 is the optional test under C150, it is widely held that it has little relationship to field performance of concrete. C359 using mortar gives better correlation though it is sensitive to water:cement ratio and to mortar temperature.

Flash set or quick set is the result of uncontrolled hydration of C_3A before the formation of surface ettringite. It is accompanied by rapid release of heat and loss of workability and is caused either by insufficient gypsum or of insufficient sulphate solubility. If the latter, partial dehydration of gypsum to hemihydrate in the cement mill may solve the problem. Flash set, which is not accompanied by heat release, can be caused by excessive precipitation of gypsum, syngenite or ettringite.

It should be noted that false set or flash set may be caused by certain concrete admixtures, particularly water reducers.

"Hot cement" is a common complaint. It must be admitted that cement over about 60°C can be unpleasant to handle, but, as it constitutes only 10-15% of the concrete mix, its heat contribution is seldom critical. Heat generated by hydration in mass pours, of course, is a different issue and reflects the type of cement employed.

Low strength of concrete cylinders can result from:

- high air content
- incorrect mix, most commonly high water cement ratio
- incorrect sampling or moulding
- improper curing
- incorrect capping

Cement content of hardened concrete can be confirmed (ASTM C1084) by arithmetic comparison of, usually, CaO analyses of cement, aggregates and concrete. SiO_2 analysis may be employed when limestone aggregate is present. SO_3 content of segregated mortar may also be used to calculate cement content if cement SO_3 is known.

Lumpy or low-strength cement is usually caused by protracted or improper storage allowing partial hydration and carbonation.

Pack-set is a phenomenon of electrostatic charging of cement particles which, upon storage and compaction (particularly in bulk distribution vessels), results in resistance to flow. Another cause of pack-set is dehydration of gypsum in hot cement resulting in syngenite ($K_2SO_4.CaSO_4.H_2O$) formation and partial cement pre-hydration. Unlike hydration, however, once pack-set is broken, the cement flows freely. The electrostatic charging is prevented by addition of cement grinding aids. A standard method for the determination of pack-set is available from WR Grace & Co, Construction Products Division [Cambridge, MA, USA, www.wrgrace.com]. This test method recently became ASTM C-1565. Thermal scanning techniques (DTA/DSC/TGA) are useful for investigating gypsum dehydration and syngenite formation.

Pop-outs are usually conical, 2-10cm in diameter at the surface, and 1-5cm deep. The cause is expansion of aggregate after setting and may be due to:

- freezing of water in porous aggregate
- alkali-aggregate reactivity
- contamination by burnt lime or dolomite or by broken glass
- oxidation of sulphide or magnetite in aggregate
- presence of soft particles such as clay lumps, shale, chert and coal.

Retarded Concrete Setting – Fresh concrete containing under-sulphated Type V cement and a lignosulfonate water reducer does not set for several days when cold. This is an example of severe cement and admixture incompatibility. The concrete has to be dug out and wasted when this occurs.

Slump loss is a normal phenomenon, which takes place with prolonged mixing. Typically, a 12cm slump will fall to 10cm after 15 minutes and 6cm after 60 minutes. Higher slump losses will occur with porous aggregates, with elevated temperatures, and with superplasticiser incompatibility. Slump loss due to admixtures may be related to:

- insufficient sulphate and high alkali contents in cement
- accelerated formation of ettringite
- an excess of soluble sulphate causing gypsum precipitation

- inadequate C_3A to control sulphate released into solution.

Delayed incremental additions of a superplasticiser is an effective way of extending workability. Other methods may include modifying sulphate content during cement manufacture, changing the type or dosage of admixture, or reducing concrete temperature.

Sulphate attack on hardened concrete often leads to expansion, cracking and spalling. In more advanced stages of attack, concrete may be softened and disintegrated. Sulphates may come from ground or seawaters, soils, road de-icing salts, and from flue gas emissions and give rise both to chemical reactions and to physical processes. Sodium sulphate reacts with calcium hydroxide to form calcium sulphate, which, in turn, reacts with hydrated calcium aluminates to form ettringite. These reactions are all accompanied by large expansions since the molar volume of ettringite [715mL] is much greater than those of the reactants $Ca(OH)_2$ [33mL], $CaSO_4$ [74mL], C3AH6 [150mL], and C4AH19 [369mL] (Lea ibid, p. 310). The attack on concrete by magnesium sulphate is more severe than that of sodium or calcium sulphates since, in addition to reacting with hydrated aluminates and calcium hydroxide, it also reacts with hydrated calcium silicates to form gypsum and magnesium hydroxide. Long-term concrete exposure studies conducted by PCA indicate that sulphate attack under alternating wet and dry conditions is particularly destructive due to crystallisation pressure. In general, Type V Portland cement, blended cements, or high alumina cements should be used to improve sulphate resistance of concrete.

Surface brown discolorations are caused by a very small concentration of Fe^{2+} ions which diffuse from inside of concrete through capillary pore solution to the surface and are retained there in the lime efflorescence (calcite crystals). The Fe^{2+} ions can originate from any of the components of concrete. However, many simultaneous conditions are required for discoloration to occur in addition to Fe^{2+} ion concentration. To reduce the potential for discoloration, it is suggested to:

- Produce a dense concrete
- Use cement containing 30-40% slag
- Use cement which has been burned harder to increase C_3S formation at the expense of C_2S and free-lime.

A new test method has been developed to test the brown discoloration potential of cements (Haerdtl et al, CI, 4/2003, p. 101).

Surface dusting of floor slabs is caused by:

- high slump concrete
- premature finishing
- surface drying
- water adsorbing formwork.

Surface scaling is the breaking away of a 1-5mm surface layer and may be caused by:

- unsound aggregate
- freezing and thawing
- premature finishing
- excessively fine aggregate (-100#).

Unsoundness of cement is the expansion of hardened paste due to hydration of hard-burned free-lime and/or hard-burned free magnesia. Cement specifications typically do not limit free-lime but do limit MgO (to 6.0% under ASTM C 150). Some specifications also limit expansion under autoclave conditions (to 0.8% as tested under ASTM C151). It is however, well established that free-lime and periclase do not affect the field performance of modern concrete and that the autoclave test is a quite unnecessary restriction on the use of MgO containing limestone resulting in additional cost and wasted reserves.

In summary, it is believed that while cement quality may occasionally be the cause of concrete problems, this is much less likely than other causes. The most common cement problem is the shipping of an incorrect cement type or contamination during transportation.

6.24. Domestic Water Treatment

Bacterial treatment is typically by the addition of 0.5-2.0ppm Cl_2 from chlorine gas or NaOCl. The effectiveness is reduced if pH is greater than 8 or in the presence of turbidity or ammonia. Residual Cl_2 should be 0.1-0.5% and can be removed, if required, by the addition of stoichiometric quantities of SO_2 or sodium thiosulphate:

$$Cl_2 + SO_2 + 2H_2O = 2HCl + H_2SO_4$$
$$Cl_2 + 2Na_2S_2O3 = 2NaCl + Na_2S_4O_6$$

Turbidity is reduced by flocculation using e.g. 1ppm alum $[Al_2(SO_4)_3.nH_2O]$, a clarifier, and filtration. pH should be not less than 7.

Hardness may be temporary (Ca/Mg-HCO_3) which can be eliminated by boiling or addition of CaO, or permanent (Ca/Mg-SO_4/Cl) which is eliminated by addition of Na_2CO_3. Treatment may also be by ion exchange. Hardness is expressed in mg/l $CaCO_3$ and water is considered hard if greater than 150mg/l.

Consumption for domestic usage varies from about 50l/person/day for lower class housing in developing countries to 180l/person/day for upper class housing in advanced economies (Twort, Law & Crowley, Water Supply, 3rd Ed., Edward Arnold, London, p. 7.)

7. Maintenance

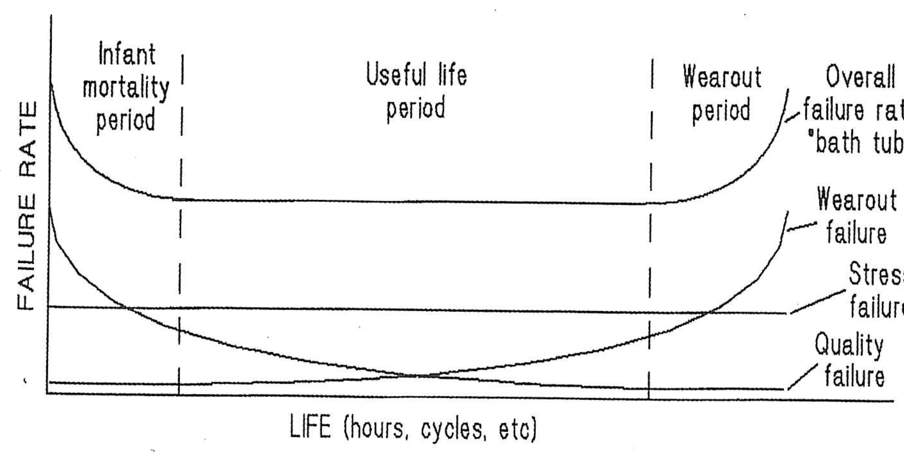
As maintenance ranges across chemical, electrical, mechanical, civil and structural engineering, and involves numerous arcane skills, no attempt is made here to address the subject in practical detail. Planned maintenance is reviewed by Patzke & Krause (ZKG, 5/1994, p. E135), condition based maintenance by Rudd & Wesley (WC, 6/2003, p. 43) and plant engineering by Guilmin (ZKG, 5/1994, p. E131). Some general concepts are considered here, however, as it is being increasingly recognised that there are more failure modes than old age and that an appropriate analysis of equipment can lead to both greater reliability and reduced maintenance cost.

7.1. Failure Modes

The purpose of maintenance is to ensure maximum availability and efficiency of plant equipment using limited resources of manpower, cost, and equipment downtime.

Maintenance is the preservation of equipment condition while repair means restoring the equipment to pristine condition. Patching is inadequate repair to less than new condition. Historically, maintenance in the cement industry involved running to failure followed by repair or replacement. In the 1950s, the concept of preventive, or operating-time-related, maintenance was developed which attempted to predict equipment life expectancy and allowed repair or replacement just ahead of anticipated failure. It became evident, however, that, apart from items subject to wear, corrosion, or fatigue, failure is not generally related to operating life and that many items of equipment or their components are subject to a high risk of early failure (infant mortality) followed by an extended life with high reliability. Combination of these two precepts led to the "bath tub" concept (Fig. 7.1). Within the last few years it has become evident that there are at least six different characteristic curves.

Figure 7.1. – Equipment Failure Profiles



Thus, it has been accepted that inappropriate time-scheduled maintenance may increase risk of failure by reintroducing infant mortality to stable systems. This led to the idea of condition-based maintenance (Hackstein, ZKG, 11/2000, p. 636) which depends upon recognition that most failures give advance warning through such parameters as temperature or vibration. The optimum maintenance programme should, therefore, identify and adapt to the failure modes for each equipment item or component. Because much equipment is actually damaged through mis-operation, regular surveillance to detect abnormal conditions is essential. The surveillance is best performed by operations personnel appropriately trained and provided with clear visual guidelines for normal operating parameters. The concept is often termed 5S+1 in maintenance literature.

Maintenance represents, typically, 15% of total manufacturing cost and, since the 1970s, maintenance departments in the cement industry have shared the need for increasing cost discipline along with increased plant reliability. This has tended to result in smaller staffs with multiple craft skills and an increasing use of information and measurement technologies.

7.2. Computerised Maintenance Management Systems (CMMS)

A single-line cement plant may have more than 500 pieces of motor driven equipment together with numerous other items which require maintenance. Effective monitoring demands computerised data processing and there are now hundreds of proprietary systems in addition to systems developed in-house by some operations (Plant Services Magazine Annual Review of CMMS). Common to all systems are:

- an equipment database which stores descriptive and specification information on all pieces of equipment.
- a database for preventive maintenance tasks together with a scheduling function related to operating time or to throughput.
- a system for generating work orders for repair or maintenance and for logging work orders received from operating departments.
- a database recording the maintenance history of all items of equipment and it is desirable also to address:
- store room inventory management and procurement.
- labour and overtime tracking.
- safety record-keeping.

Essential to the CMMS is an examination of the logic underlying every preventive maintenance task to determine whether it is cost effective, whether the task is in fact effective to prevent the subject failure, and to ensure that all known failure modes are addressed with appropriate preventive practices. Implementation of a CMMS without this scrutiny will only perpetuate previous system defects with greater efficiency. It is desirable that the CMMS interfaces with the company's accounting systems to avoid the inefficiency and potential inconsistency of multiple data entry.

7.3. Reliability-Centred Maintenance

The systematic review involved in establishing the CMMS constitutes what is generally referred to as reliability centred maintenance and comprises:

- Failure mode and effect analysis (FMEA)

and

- consequent prevention or control task definition,

which should be carried out by the maintenance people, operators, engineers and supervisors responsible for each major equipment system. FMEA involves the most likely, the most expensive, and the most hazardous modes of failure and, not infrequently, it is recognised that certain failure modes cannot be cost effectively prevented either because there is no advance warning or because monitoring is impractical. Such situations are usually remedied by redesign or by changes in operating practice. FMEA also requires the definition of function for each equipment system to avoid such ambiguities as the maintenance department considering that a machine is functioning because it is running while the production department considers that its operation is defective. Management

must, therefore, establish for each system the required:

- *Capacity* (eg production rate) together with feed and product specifications,

and

- *Reliability* in terms of % scheduled operating time when the equipment is actually available

based upon historical production achieved, equipment design specifications, standard industry performance, and business objectives for the year. Usually these parameters must be maximised for the kiln but may be relaxed for oversized ancillary equipment. The same priorities should be employed by the maintenance department in allocation of the scarce resources at their disposal (manpower, expense, and down-time), and these may vary from time to time depending upon equipment and inventory situations.

Reliability-centred maintenance was developed in the aerospace industry and its effectiveness is attested by the very low failure rates achieved there. The concept also underlies the various quality management systems (Ireson & Coombs, Handbook of Reliability Engineering and Management, McGraw Hill, 1988).

It has generally been accepted in the cement industry that the greatest run-factor and maintenance cost efficiencies are achieved by running kilns to failure subject, of course, to analysis of such on-line equipment monitoring as is possible (eg ID fan vibration). Planned maintenance is conventional on all major equipment with the scheduling of the shut-down being either fixed by management plan or determined by equipment failure. The specific job list should incorporate feedback on deficiencies from both production and quality departments. Shut-down tasks should include:

- Lubrication, filter cleaning, etc.
- Attention to problem areas identified by operators, e.g. frequent alarms, vibration, etc.
- Inspection and measurement of wear parts
- From previous inspections, or at fixed periods, part replacement or service should be performed
- Inspection of highly stressed equipment that cannot be inspected during operation should be done at every opportunity (eg clinker cooler).

7.4. Maintenance Cost Management

Manpower costs comprise direct maintenance staff costs and, sometimes, labour from other departments which may or may not be captured as maintenance. Work by outside contractors obviously must be captured either as a maintenance expense or as a capital project cost. The CMMS accumulates costs under equipment codes but practice varies in the detail to which equipment systems are broken down and coded. The less the detail, the greater will be the need for individual investigation of cost variances and the greater the risk that a perennial fault involving minor direct cost but, perhaps, significant impact on reliability may be overlooked. The most useful systems apply costs directly to each work order.

Maintenance labour costs are essentially fixed though overtime and outside contractors may constitute a significant variable element. Studies have shown as little as 28% of an eight-hour shift may be spent actually working on equipment and, more than with other jobs in a plant where a routine is established, productivity requires efficient scheduling and supervision.

Equipment downtime is another major maintenance resource which should be used as efficiently as possible. For kilns, which are run to failure the best use must be made of the unavoidable downtime and it is essential to plan in advance for all routine checks and for accumulated non-critical repairs (Lowell, GCL, 7/2001, p. 12 and 8/2001, p. 24). This may also involve contingency planning for

possible tasks that are not confirmed until access is obtained. Once the required job list has been established and the critical path determined the work should be prioritised and scheduled to minimise the outage. Ancillary equipment which can be made available for scheduled preventive maintenance can accommodate other considerations; for example, maintenance on mills may be best performed during peak power tariff periods, and maintenance on crushers should be done when the quarry is not operating. A final consideration is that scheduled maintenance should keep as closely as possible to the schedule so that the production department may have confidence that the equipment will be available when they need it; loss of maintenance credibility is a major factor in operators' refusal to shut down equipment for routine attention and a downward spiral into breakdown maintenance.

Inputs to the CMMS come from *work orders* which should:

- Identify the equipment by code which determines the cost centre for charging maintenance costs
- Describe the problem to be investigated or the repair to be made
- Classify the type of work performed, components involved and reason for work.

An important function of plant management is to determine operating strategy with respect to maintenance and work order coding allows a large number of orders to be classified, thus providing a basis for strategic planning. Some flexibility is appropriate to respond to periods of high market demand when non-essential maintenance can be deferred and to periods of low market demand when the time may be available for deferred maintenance or equipment modification but, frequently, costs are under pressure. Plant management should define what is required from maintenance, their responsibility and authority, and the indices and metrics by which maintenance performance will be assessed.

7.5. Maintenance Organisation

In most other industries, the maintenance department comprises both mechanical and electrical functions. Separating electrical and mechanical groups and having them both report to plant management is peculiar to, though not universal in, the cement industry. We have seen no explanation, other than tradition, why this should be so. Mechanical maintenance crafts are primarily mechanics (or repairmen) and machinists; electrical crafts are electricians and instrument technicians. Labourers, dust collector crew, and mobile equipment mechanics may also come within the (mechanical) maintenance department. Conventionally, supervisors and artisans are grouped under functional headings though many plants find it effective to dedicate groups of maintenance people to functional areas of the plant as this can lead to increased familiarity with, and "ownership" of, the equipment.

Over the past 15 years there has been much discussion and implementation of self-directed workers. This can take various forms but, in the cement industry, the most common has been the work team of 3-5 artisans under a team leader who, in turn, reports to a salaried supervisor or manager. This concept can and has worked well, but it must be recognised that such teams should not be called upon to make decisions beyond their expertise and the information available to them. An appropriate system of both commercial and technical information flow must be developed and institutionalised.

7.6. Role, Planning and Control

Traditionally, maintenance has responded to problems and failures identified by operators. Increasingly, however, with the development of more sophisticated monitoring techniques, the maintenance department is taking responsibility for detecting abnormal equipment conditions as well as for their remediation. This frequently involves a dedicated inspector though a significant amount of diagnostic data can be collected and analysed using electronic data processing.

Analysing the proportion of maintenance time spent on breakdown work is a useful measure of the effectiveness of a preventive programme. In badly maintained facilities, up to 80% of maintenance man-hours are spent on unscheduled repairs. The prime reasons for such situations are:

- Equipment unreliability due to unsuitable design, defective construction, or mis-operation
- Inadequate prioritisation of scheduled work.

It is essential to identify and eliminate the root causes, otherwise the department will begin to organise for perennial breakdowns and will become entirely reactive. Inevitable results will be either excessive manning levels or excessive overtime, and the expansion of parts inventories to cover any eventuality. With carrying costs accounting for 10-25% of inventory value depending upon taxation, depreciation, and the cost of capital, a significant expense can be generated which may well be ignored by plant management if it does not impact plant operating earnings. Risk analysis can be effective to establish optimum stock levels but an analysis of equipment failure and of procurement cycles must be integrated with the analysis. In this context it should be noted that considerable savings in inventory cost may be achieved if suppliers can agree to carry stock in the plant and charge only upon use. There may also be potential for selling back to a supplier stock which is no longer required.

Prioritisation of maintenance tasks is essential in order to allow timely inspection and repair of equipment before failure. Resources will never be considered adequate so that priorities must be set by plant manager, maintenance manager, and the heads of operating departments. The bases for priority are largely self evident and include:

- The kiln must be kept in operation to avoid production loss which cannot be made up; ancillary equipment can often be allowed to shut down without permanent loss; redundancy reduces priority
- Equipment failures which impact the ability to load out product and satisfy the customer
- Safety items must be either corrected or temporarily neutralised (eg by locking out or roping off)
- Environmental or regulatory items may sometimes be deferred if a plan is communicated to, and accepted by, the regulator.

Priorities and their justifications should be communicated to shift supervision so that they are not lost in the heat of new crises. Ultimately, manning levels with or without outside resources must be sufficient to cope with all tasks deemed necessary. A list of deferrable, non-critical jobs is required to achieve efficient employment of a work force with a varying load of priority work.

7.7. Mobile Equipment Maintenance

Vehicle maintenance requires specialised skills not normally associated with conventional cement plant maintenance. Traditionally, therefore, mobile equipment is under the control of the quarry department. With increasing equipment size, the number of units is frequently inadequate to allow efficient management of a dedicated shop and workforce so that many plants have resorted to outside shops for all but the routine checks and adjustments which are, in any case, conducted by operators. Frequently such shops belong to the equipment dealer who can offer efficiencies of familiarity, technical support, parts inventory, and the ability to apply varying levels of resource according to need. In-house mobile equipment facilities are now largely associated with remote locations or an abnormal fleet composition.

However, whether or not maintenance is conducted in-house, a system must be established to track the maintenance and service history of each piece of mobile equipment and this may be either the quarry department's production record keeping programme or the maintenance department's CMMS. Operating hours are logged at each routine inspection so that preventive maintenance activities can be initiated when appropriate. Mobile equipment is subject to an increasing risk of wear related to operating hours while the condition of the machinery can be closely monitored through analysis of the lubricating and hydraulic fluids which come into contact with the various components. Oil monitoring has become a sophisticated tool employing spectrochemical analysis and particle concentration and size distribution which, both from absolute levels and from trending, allow identification of components subject to abnormal wear or nearing failure. The optimum frequency of oil analysis is related to service conditions and the characterisation of failure types to be managed.

8. Environment and Pollution Control

8.1. Dust Collection

The principal types of dust collection used in cement plants are:

Cyclones are typically 95% efficient dropping to ca 60% for particles less than 5µ. Normal inlet/outlet velocity is 10-20m/s, pressure drop 50-150 mm H_20, and aspect ratio (height:diameter) 3-5.

Electrostatic precipitators (EPs or ESPs) comprise an array of discharge wires at 50-100kV negative potential and earthed collecting plates. Pressure drop is about 15-20 mm H_2O and power consumption 0.2-0.3 kWh/1000m³. Efficiency is typically 80% per field so that a four-field unit should capture up to 99.85% of entrained dust. Efficiency varies with particle size and operation is described by the Deutsch formula:

$$n = 1 - e^{-(\omega.A/Q)}$$

where n = efficiency (%)
ω = particle migration velocity (m/s)
A = area of collecting plates (m^2)
Q = gas flow rate (m^3/s)

Migration velocity reflects dust resistivity, particle size, field intensity, gas viscosity, and other design parameters and should range 0.07-0.10m/s. Dust resistivity should be 107-10^{11}Ωcm. H_2O, Cl and SO_3 reduce resistivity of basic dusts. Conditioning of inlet gas is normally required with about 15% moisture in kiln exhaust and 3% in cooler exhaust; gas temperature should be 120-150°C, though EPs can be designed for higher temperatures. Although modern units are of high reliability and can be guaranteed to achieve below 20mg/Nm³, they are still liable to total shutdown due either to kiln interlock or to electrical failure, and this must be acceptable under emission regulation. Richter et al (ICR, EHB/2003, p. 91) reviews developments in the design and control of EPs and Vandelli (WC, 10/2003, p. 87) describes hybrid dust collectors where an electrostatic precipitator is followed by a fabric filter to achieve optimum efficiency.

Bag filters comprise filters of either woven fabric (which employ bag shaking or reverse air flow for cleaning) or needle felts (which are cleaned by reverse air pulse). Air to cloth ratio (m³/min/m²) should be 0.5-0.9m/min (1.6-3.0ft/min) for woven fabrics and 1.7-2.3m/min (5.5-7.5ft/min) for needle felts. Pressure drop is typically 150-250 mm H_2O and efficiency 99.95%. The normal fibre is polyester which can operate up to 150°C, while polyamide can be used to 230° and glass fibre to 280°. The relative installation and operating costs of reverse air and jet pulse dust collectors are reviewed by D'Lima (ICR, 2/2000, p. 51).

A potentially useful development is a flexible metal fibre filter material resistant to 1100°C (ZKG, 7/2004, p. 36).
Pulse air should be dried and of 6-8kg/cm² pressure, and bag tension approximately 0.4kg/cm of bag circumference. Bag house pressure drop can be reduced and, thus, capacity increased by use of acoustic horns (Cameron, WC, BMH/1998, p. 67).

While the risk of explosion with unburned fuel or reducing conditions is obvious for electrostatic precipitators, it should also be recognised that, unless anti-static bags are employed, a similar risk attaches to bag filters.

Gravel-bed filters – Used for de-dusting clinker cooler vent air, these filters are proof against temperatures up to 450°C and forgiving of temporary overloading. They are, however, relatively inefficient and are being phased out. The gravel bed comprises quartz grains of 2–5mm supported on a wire mesh and requires reverse air cleaning (Labahn).

Conditioning of kiln exhaust gas may be necessary before dust collection by. The exhaust gas from many kilns is used for drying raw materials and this process serves both to cool the gas and to raise its humidity before dust collection. The gas temperature should be below 170°C for both bag-houses and EPs and, for the latter, a moisture content of ca 15% (v/v) is required in exhaust gas. It is important, however, that the gas should not be allowed to pass through the dew point before release, especially if significant S or Cl is present. A conditioning tower is usually provided for periods when raw mill or dryer are bypassed to allow gas cooling by water spray. The alternative is to add tempering air but this significantly increases total gas flow and fan power consumption, and may lead to de-rating of the kiln while bypassing the raw mill or dryer. The operation of conditioning towers is notoriously problematical due to the dirty atmosphere in which the water sprays must function and to the large turn-down necessary for control (Richter & Taylor, WC, 1/1999, p. 46).

8.2. Pollution Control

Environmental regulation is, of course, very much a matter of national and local ordinance. However, certain generalisations can be made about air, water, solid, and noise pollution. Cement plants are primarily concerned with air emissions.

Water discharge should not be a concern beyond handling normal domestic waste and storm water run-off with its potential for leaching from stockpiles and spillage. The temperature of discharged cooling water may also be subject to control.

Solid waste is frequently confined to kiln brick. Chromium, once common in basic brick, has largely been eliminated due to its alleged toxicity and all used refractory may now be incorporated into kiln feed (after crushing and grinding) or sent to landfill. Note that if the limestone quarry also produces aggregates, recovery through the crushing system must be carefully managed as basic brick contamination of aggregate can be catastrophic due to expansive hydration.

The only other likely solid waste will result if kiln dust is discharged to relieve a volatile cycle in the kiln. Hitherto, land-filling such dust has involved minimal expense, but increasing regulation, particularly in the United States (Weiss, CA, 3/2000, p. 21), may eventually encourage processes to reduce kiln dust generation (Section 4.7.) or to recycle rather than landfill. Kiln dust remediation technologies involve either leaching or heat treatment but none are presently both economic and environmentally acceptable.

Material spillage can be collected by either fixed or mobile heavy-duty vacuum systems (Haugen, WC, 2/2006, p. 99) and, preferably, returned to the process.

Noise originates primarily from mills, fans/compressors and screw/drag-chain conveyors. Noise affecting plant workers can be controlled by ear protection. Noise at the property boundary is best considered at the design stage but can be mitigated by enclosure, insulation and, for fans, sound attenuators (Fuchs, ZKG, 7/1993, p. E185). Although most fan problems occur with short exhausts, some tall chimneys have been found to resonate unless silenced. Quarry blasting is a specific problem which may involve charge design and timing to minimise disturbance to neighbours. Noise sources, regulation and control are discussed by Dupuis (ZKG, 12/2003, p. 52 and 1/2004, p. 41).

Air pollution control is becoming progressively more onerous world wide as regulators lower limits on particulates, CO, SO_2, NO_x, etc. and add new prohibitions on metals, dioxins, and other trace chemicals. (1990 Amendments to the U.S. Clean Air Act, Title III, Section 301).

Typical *particulate* limits for kilns are now 40-100mg/NM^3 and will probably continue to decrease; this progressively favours baghouses over EPs, especially given that baghouses are not prone to the total failure which may afflict an EP. Where opacity is also used to monitor emissions, *detached plumes* resulting from hydrocarbons or ammonium compounds can present a problem which is not solved by conventional dust collection. Considerable understanding of detached plumes has been acquired by afflicted plants and some information has been published (Wilber et al, ICR, 2/2000, p. 55 and Tate, ICR, EHB/2003, p. 97).

European environmental policy and emission standards are reviewed by Locher & Schneider (ICR Environmental Handbook, 2003, p. 43). Limits for rotary cement kilns are:

Particulates	30mg/m^3
SO_2	50mg/m^3 (combustion products only)
NO_x	800mg/m^3 (400mg/m^3 for new plants)

US emission limits are discussed by Salmento & Shenk (ZKG, 11/2004, p. 52).

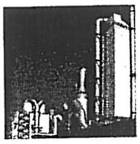
While emission regulations stipulate particulate levels, actual measurement is a protracted procedure involving isokinetic sample collection. Automated systems are available for continuous opacity monitoring which measure light attenuation across the stack (Stromberg & Puchta, WC, 10/1996, p. 66). Some regulators also recognise visual estimations such as the Ringelmann Smoke Chart. The Ringelmann chart is a series of cards with increasingly dense cross hatching representing opacity from 0-100%; where applicable, emissions should not normally exceed 10% opacity or between Ringelmann 0-1 (see eg Duda, 3rd Ed., Vol. 1, p. 579).

Dust suppression can be important in both quarry and plant where dry materials are handled and where unpaved surfaces are used by mobile equipment. Various engineered and chemical-spray systems are reviewed by Carter (RP, 5/1995, p. 19). Archer (WC, 5/2003, p. 109) describes methods to control fugitive dust from crushers, screens and transfer points. Respirable dust at locations within and at the perimeter of plants is subject to regulation in many countries including the United States (Cecala et al, CA, 1/2000, p. 20 and 3/2000, p. 28).

CO is formed by the incomplete combustion of carbonaceous materials. Oxidation to CO_2 takes place in the presence of excess oxygen at temperatures above about 680°C. CO in stack emissions is usually attributed to overall deficiency of oxygen in the burning zone or to poor fuel/air mixing. However, Sadowsky et al (ZKG, 5/1997, p. 272) have found that many cement raw materials contain 1.4-6g organic carbon per kg clinker, that these oxidise below 680°C, and that 10-20% of the oxidation results in CO irrespective of the level of excess O_2 (Note that 2g C/kg clinker with 15% conversion yields about 250ppm CO at 5% excess oxygen). Obviously this is too low a temperature for post oxidation to CO_2 and suggests that some CO observed at the stack may not be due to combustion problems and may not be easily rectifiable.

See also Combustion Section 9.6 for more detailed description of CO, NO_x and SO_2 control.

NO_x is formed during fuel combustion by oxidation of nitrogen compounds in the fuel (fuel NO_x) and of the nitrogen from combustion air (thermal NO_x) and comprises about 95% NO and 5% NO_2 (Salmento, ZKG, 11/2004, p. 52). Thermal NO_x predominates and increases with flame temperature above 1200°C, with retention time, and with increasing free oxygen.



Haspel et al (ICR, 1/1991, p. 30) working with NO_x as a control parameter for kiln operation, have discovered that, although the NO_x produced is mainly thermal, it is a good indication of burning zone condition only with burners which provide stable flames with good recirculation. With non-recirculatory, low primary air burners, there are interferences with its correlation to burning zone temperature. Specifically, non-robust burners can yield CO of more than 1000ppm (0.1%) with 2-3% O_2 at kiln inlet and this can totally confuse logical kiln control responses. Petcoke combustion, too, is particularly sensitive to secondary air temperature. Thus, additional factors affecting NO_x at kiln exhaust include:

- kiln atmosphere (NO_x is degraded by CO > 3000ppm)
- alkali cycle which increases rapidly with burning zone temperature and with reducing conditions
- secondary air temperature

NO_x emissions for normal operation may be 1000-1500mg/NM^3. Flame quenching, low-NO_x burners, or staged combustion (for precalciner kilns only) should approach 500mg/NM^3. Selective catalytic reduction (SCR) or, the increasingly preferred, selective non-catalytic reduction (SNCR) would be required to get significantly lower (Haspel, ICR, 1/2002, p. 63 and Horton et al, ICR, 8/2006, p. 85).

SO_2 is produced in the kiln both by oxidation of fuel S and by decomposition of sulphates. SO_2 thus produced is almost totally scrubbed by K_2O, Na_2O and CaO in the cyclone preheater. The lower volatility of the alkali sulphates leads to their predominantly exiting with clinker (unless relieved by a gas bypass) while $CaSO_4$ will largely re-volatilise in the burning zone and results in a sulphur cycle building up at the back of the kiln and the lower preheater cyclones. In extreme cases, this cycle will cause accretion and blockage problems unless relieved by a gas (or meal) bypass. This is exacerbated by the hard burning required for low alkali clinker and frequently leads to strict sulphur limits on feed and fuel.

Sulphides and organic sulphur in raw materials, however, oxidise in the preheater and largely exit with exhaust gas. SO_2 can theoretically oxidise to SO_3 at low temperature but, in practice, more than 99% of gaseous sulphur will be SO_2. With SO_2 emissions being increasingly regulated (the US Clean Air Act mandates limits of 1.2lb/million BTU or 2.2kg/million kcal by 2000), the only solution if such raw materials cannot be avoided is to scrub the exhaust gas (Marechal, ICR Environmental Handbook, 2003, p. 15). The low temperature adsorption of SO_2 as kiln exhaust gas passes through drying and grinding systems is investigated by Krahner & Hohmann (ZKG, 1/2001, p. 10 and 3/2001, p. 130).

An old, but still interesting, review of NO_x and SO_2 production and control is by Kupper (WC, 3/1991, p. 94). Models for predicting emissions are discussed by Salmento & Shenk (ZKG, 11/2004, p. 52).

Unfortunately, regulators have not yet extended their jurisdiction to natural phenomena such as the 1991 eruption of Mount Pinatubo which is estimated to have injected into the atmosphere SO_2 equal to between 10 and 100 times the present world-wide annual production (Economist; 21 Nov 1992, p. 97).

Gas analysis for process control and emission monitoring involves a wide range of proprietary instruments (Gumprecht et al, WC, 10/2003, p. 103). For occasional measurement of most gases of interest, absorption tubes with a small syringe pump provide a simple, accurate, and low cost method (MSA, National Draeger, Sensidyne). Continuous emission monitors (CEMs) are reviewed by Tarodo (WC, 10/2003, p. 67).

Dioxin emissions are of particular concern where alternative fuels are burned. It is generally accepted that dioxins and furans will not be released in significant quantity if burned under the following conditions:

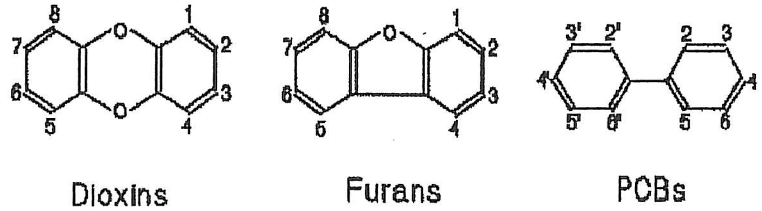
Minimum temperature		1200°C
Minimum retention time		2s
Minimum excess oxygen	- liquid fuels	3%
	- solid fuels	6%
Maximum CO (@ 11% O_2)		40ppm

(Krogbeumker, ICR, 5/1994, p. 43)

Cement kilns can easily meet these requirements with adequate process control and well-designed burners. The emissions associated with alternative raw materials and fuels are reviewed by Scur & Rott (ZKG, 11/1999, p. 596). The US EPA proposed emission standards for dioxins and furans from cement kilns call for <0.20ng TEQ/dry standard m^3 (Knotts et al, WC, 10/2004, p. 89).

It may be noted that there are 75 chlorinated dibenzo-p-dioxins and 135 chlorinated dibenzofurans, all with different toxicities. Many regulators recognize a scheme of "Toxic Equivalents" (TEQ). Following are the basic structures:

Figure 8.1. – Basic structures of dioxins, furans and PCBs



Toxic heavy metals of common concern, in decreasing order of volatility, are Hg, Tl, Cd, Se, Sn, Zn, Pb, Ag, Cr, Be, Ni, Ba, As, V. They are ubiquitous in trace quantities, and the manufacture of cement from natural minerals is usually of negligible consequence as most metals are retained in the clinker. A protocol for trace metal mass balance is described by Gossman (ICR, EHB/2003, p. 117). The introduction of alternative raw materials and fuels requires monitoring both of inputs (Dirken, WC, 4/2003, p. 64) and of distribution between cement, kiln dust (if discharged), and gaseous exhaust (Gossman & Constans, ICR, EHB/2003, p. 117).

Regulation varies with jurisdiction but has, unfortunately, frequently lost sight of the value of the cement kiln to destroy organic wastes and to encapsulate waste metals in concrete. Also, the legal position has become confused by consideration of waste burning kilns as incinerators, by considering the products of processes incorporating hazardous waste materials as themselves automatically hazardous, and by assumptions that any measurable toxic metal is dangerous even when lower than its natural elemental occurrence in the earth.

An unfortunate malady encountered by a small proportion of brick layers is cement eczema attributed to Cr^{+6} in cement. Water soluble chromate is now conventionally limited to 2ppm (Kerton, ICR, 5/2004, p. 65 and Olmix, ICR, 9/2006, p. 105).

8.3. ISO 14000

A standard which "provides a framework for the development of an environmental management system (EMS) and the supporting audit programme". The suite of standards comprises:

ISO 14001	Specifies the framework of control for an EMS against which an organisation can be certified.
ISO 14004	Guidance on the development and implementation of EMS.
ISO 14010/11/12	Principles of, and guidance on, environmental auditing (superseded by ISO 19011).
ISO 14013/5	Audit programme review.
ISO 14020	Labelling.
ISO 14030	Guidance on performance targets and monitoring.
ISO 14040	Life cycle issues.

8.4. Sustainable Development and Climate Change

The World Business Council for Sustainable Development defines sustainable development as "forms of progress which meet the needs of the present without compromising the ability of future generations to meet their own needs" (www.wbcsd.ch). Ten major cement companies committed in 2002 to an initiative comprising: climate protection, resource efficiency, employee health & safety, emissions reduction, local impacts and internal business practices. For those interested in all this, or for those who must fake an interest, the concept is well summarised by Kerton (ICR Environmental Handbook, 2003, p. 11). A more critical view is presented by Portland Bill, ICR, 4/2007, p. 101 and by Baetzner, ZKG, 7/2004, p. 2.

The concept is broad and in most substantive instances, such as energy efficiency and emissions reduction, simply restates practices which have been common pursuits of the industry for many years. Particular emphasis is placed upon the putative responsibility of greenhouse gasses – primarily combustion products – for climate change. Of interest, therefore, may be the outcome of the Copenhagen Consensus as reported in the Economist (3rd June 2004, p. 59). An expert panel of economists attempted to prioritise various global challenges which may be addressed with limited resources and concluded that the Kyoto protocol and carbon tax were of the lowest priority and rated as bad investments. Significantly, an international youth forum which heard the same submissions ranked climate change as ninth out of ten global challenges. As the Economist commented, "perhaps this should give pause to governments dedicated, or claiming to be dedicated, to Kyoto's implementation".

Politicians and the press repeat the mantra that the debate is over and that the evidence is overwhelming for imminent disaster from global warming. It is suggested that this is not the case and that much sophistry and misrepresentation is employed to make the case. Lomborg (The Skeptical Environmentalist, Cambridge, 2001.) showed in detail how this is done while the Stern Report's (2006) use of a 0.1% discount rate to estimate the cost of climate change indicates that the situation is not improving.

9. Combustion and Heat Transfer Processes

Kiln operators have a number of areas for consideration when firing fuel in a kiln or calciner. Those areas that are within their control are:

- Fuel Type
- Fuel Handling
- Heat Transfer
- Burner Momentum
- Excess Air
- Emissions

Other areas that are typically outside operator control and can have a dramatic effect on combustion are:

- Secondary Air Momentum
- Tertiary Air Momentum
- Kiln Aerodynamics
- Calciner Aerodynamics

Unfortunately each area is integral to one system. As such, it is difficult to review one area independently of another. The section on combustion attempts to address to the broad subject of combustion and how it applies to making cement.

History of Combustion as a Science

Although combustion has a long history (since Prometheus' stealing of fire from the Gods!) and great economic and technical importance, its scientific investigation is of relatively recent origin. Combustion science can be defined as the science of exothermic chemical reactions in flows with heat and mass transfer. As such, it involves thermodynamics, chemical kinetics, fluid mechanics, and transport processes. Since the foundation of the second and last of these subjects were not laid until the middle of the 19th century, combustion as a science did not emerge until the beginning of the 20th century. In recent years, great improvements in understanding of combustion processes have arisen through advances in computer capability, in experimental techniques, and in asymptotic methods of applied mathematics.

Technological developments in an area often precede the emergence of the area as a firmly established science. This seems to have been especially true in combustion, and in many respects it remains true today.

9.1. Chemistry of Combustion

Combustion is a specific group of chemical reactions where a fuel and oxygen burn together at sufficiently high temperature to evolve heat and combustion products. The fuel can be a gas (e.g. H_2 or natural gas), a liquid (e.g. alcohol or oil), or a solid (e.g. Na, pure carbon, or coal). Combustion can vary in rate from a very slow decay to an instantaneous explosion. The objective of the combustion engineer and plant operator is to obtain a steady heat release at the required rate.

Most industrial fuels are hydrocarbons, so called because their elements carbon and hydrogen are oxidised to release heat during combustion. The chemistry of this oxidation process is a very complex chain reaction. However, for our purposes we can reasonably simplify the chemistry to four basic reactions.

The Complete Oxidation of Carbon

$$C + O_2 \rightarrow CO_2 + 394 \text{ kJ/mole (94kcal/mole)}$$

The Complete Oxidation of Hydrogen

$$2H_2 + O_2 \rightarrow 2H_2O \begin{cases} + 572 \text{ kJ/mole (137kcal/mole) – water condensed (GCV)} \\ + 484 \text{ kJ/mole (116kcal/mole) – water as steam (NCV)} \end{cases}$$

The difference in the physical states of the water produced as a result of the oxidation of hydrogen is the reason for the complexity of the gross (GCV) and net calorific values (NCV) for hydrocarbon fuels. The gross heat release is that which is released when the hydrogen is oxidised and the water condensed, while the net calorific value is the heat which is released while the water remains as steam. The former is also referred to as the high-heating-value (HHV) and the latter low-heating-value (LHV).

The Incomplete Oxidation of Carbon

In the event of imperfect combustion, not all of the carbon in the fuel will be oxidised to carbon dioxide but some will be partially oxidised to carbon monoxide. The main effect of carbon monoxide production is to reduce the heat release from the fuel.

$$2C + O_2 \rightarrow 2CO + 221 \text{ kJ/mole (53kcal/mole)}$$

It can be seen that only just over half of the heat is released in the production of carbon monoxide, compared with the complete combustion of carbon. Thus any burners producing carbon monoxide as a result of bad fuel/air mixing will cause a significant drop in combustion efficiency. It is therefore absolutely essential to prevent the production of significant levels of carbon monoxide in any combustion system.

The Oxidation of Carbon Monoxide

Carbon monoxide is the unwanted repository of considerable combustion energy in inefficient combustors, and also an important air pollutant, a poisonous gas in high concentrations. In many instances where hydrocarbons are burnt, the oxidation reactions proceed rapidly to the point where CO is formed and then slow greatly until CO burnout is achieved. Carbon monoxide may be further oxidised to carbon dioxide according to the following reversible chemical reaction:

$$2CO + O_2 \leftrightarrow 2CO_2 + 173 \text{ kJ/mole (41kcal/mole)}$$

The combustion of dry carbon monoxide is extremely slow, however, if H-containing radicals are present in the flame, the combustion rate of (wet) carbon monoxide increases significantly.

9.2. Fuels

Hydrocarbon fuels may be solids, liquids or gases. Gases may be natural or manufactured, generally from oil or coal. Both natural and manufactured fuels vary widely in chemical composition and physical characteristics. Each of these fuels is considered below.

Gases

Natural gas has been known for many years and utilised for much of this century. The characteristics of some typical gases are given in Section B6.3. It can be seen that while the basic constituent of all is methane, the presence of other gases affects both the calorific value and the density. Methane has narrow flammability limits and the presence of higher hydrocarbons widens these limits and assists with flame stability. Owing to the low carbon content of natural gas, conventional burners have low emissivity flames. This has a detrimental effect on the radiant heat transfer from the flame and can seriously affect the efficiency of the plant. The high hydrogen content means that natural gas requires more combustion air per kJ of heat released than most other fuels, and produces more exhaust gases, though these have a smaller proportion of CO_2.

Oil Fuels

Oil fuels are produced by the refining of crude oil or can be manufactured from coal. Waste lubricating oil is currently being used as a supplementary fuel in a number of plants, but supplies are limited. Oil fuels are classified as distillate fuels, such as kerosene and diesel oil or residual fuels. The latter come in a range of viscosities and are classified differently in different countries. Typical characteristics of oil fuels are given in Section B6.2. Residual fuels have to be heated to render them pumpable and to reduce the viscosity to enable atomisation. The heavier the fuel, the more it has to be heated. Owing to the tendency of these fuels to solidify when cold, great care has to be taken with the design of oil fuel handling systems to minimise 'dead legs'. Since the lighter 'white' oil products have a higher value than black fuel oils, refineries increasingly manufacture more light products, leading to heavier and heavier back fuels containing increasing quantities of asphaltenes. These augmented refining processes involve 'cracking' the oil and produces black oils which have different characteristics from the former residual oils. These cracked fuels vary in character, depending on the source of crude and the refining process and are not necessarily compatible with each other. Under some circumstances, fuel oils from different sources can form 'gels' in tanks and fuel handling systems with disastrous results. Proposed fuels should therefore always be tested for compatibility with the existing fuel before purchase.

Atomisation of fuel oil is important because the initial drop size determines the size of the cenosphere which is formed and hence the length of time taken for the particle to burn. The oxygen diffusion is dependent on the surface area but the oxygen demand is dependent on the mass of the particle. Since the surface area is dependent on $diameter^2$ and the mass on $diameter^3$ it follows that the larger the drop the longer it takes to burn.

Droplet sizes are normally measured in microns, a micron being 10^{-6} of a metre. This means that a 100 micron drop is 0.1mm and 1000 micron drop is 1mm diameter. Most atomisers produce a range of drop sizes with the smallest being in the order of a few microns in diameter, and the largest anything from 100 micron to 1000 micron or even more. A 100 micron particle takes about half a second to burn in a typical industrial flame, therefore a 500 micron particle takes about five times as long and a 1000 micron particle 10 times as long. Since the residence time of a droplet in a flame is typically one second or less, it follows that drops larger than about 200 micron will not be fully burnt out at the end of the flame and will either drop into the product as un-burnt fuel or end up in the dust collector. Anyone who makes a light coloured product but sees discoloured dust – dark grey or black – is suffering from just this sort of problem.

For optimum combustion performance, an oil sprayer with a range of drop sizes is ideal, fine drops to facilitate ignition and flame establishment and then some larger drops to maintain a controlled burning rate. However, for the reasons outlined above, there should be a limit on the largest drops in the spray. Depending on the particular application, this upper limit should be in the order of 100-250 micron to minimise the risk of unburned fuel at the tail of the flame.

Equally as important as the drop size is the angle of the spray. Essentially, most sprays are conical and two types are common, hollow cone and solid cone. Hollow cone atomisers are generally preferred, since this enables the air to mix most effectively with the fuel. The small number of drops in the core of a hollow cone spray allows the establishment of an internal recirculation zone which assists in maintaining a stable flame front.

Most burners are required to operate over a range of heat liberation and therefore fuel flow-rates. It is especially important that the atomiser performance is satisfactory over the entire operating range, since cement plants do not operate consistently at full load all the time. The drop size of many types of atomiser increases rapidly as the fuel flow-rate is turned down and this can present special problems for plant operation. The turndown performance varies with different types of atomiser and is an important consideration when choosing an atomiser for a particular application.

Coals

Great care has to be taken handling and burning coal owing to the risk of spontaneous ignition, fire and explosion. As a result, the design and operation of coal firing systems requires greater specialist knowledge than gas and fuel oil systems. The characteristics of coals vary even more widely than other fuels, from anthracite which has a high calorific value and very low volatile and moisture content, to the lignites with moisture and volatile contents of up to 60%. Typical properties of some commonly traded coals are given in Section B6.1. The characteristics of the coal and its ash have a dramatic effect on the performance of the plant in which it is burnt and on the plant maintenance requirements. Relevant properties include:

Volatile content – The higher the volatile content the more rapidly the coal ignites and burns. High volatile coals (above 35%) tend to present a significantly higher explosion risk than those below 25%. Coals with volatile contents above 45% require special precautions.

Swelling properties – Once the volatiles have been driven off a coke particle is left behind. If this is larger than the original particle then it is more open and the particle will burn more rapidly than if it shrinks.

Moisture content – Coals have two types of moisture, surface moisture and inherent water. Generally the higher the inherent water the greater the reactivity of the coal and the higher the consequential fire and explosion risk. For pulverised coal firing the surface moisture has to be removed when grinding. Removal of the inherent water should be minimised otherwise moisture from the atmosphere recombines with the coal and causes spontaneous heating which can result in a fire or explosion.

Ash content – The chemical composition of the ash has a significant effect on some processes and appropriate coals have to be selected accordingly.

Hardness and Abrasion Indices – The hardness of the coal affects the capacity of coal mills, the harder the coal the less can be ground and/or the coarser the resulting pulverised coal. The abrasion index is mainly dependent on the ash characteristics. Very abrasive coals with high silica ashes cause high wear rates in coal mill grinding elements.

9.3. Physics of Combustion

None of the chemical reactions can take place until the oxygen in the air is brought into contact with the gas.

All combustion processes therefore take place in the following stages:

MIXING > IGNITION > CHEMICAL REACTION > DISPERSAL OF PRODUCTS