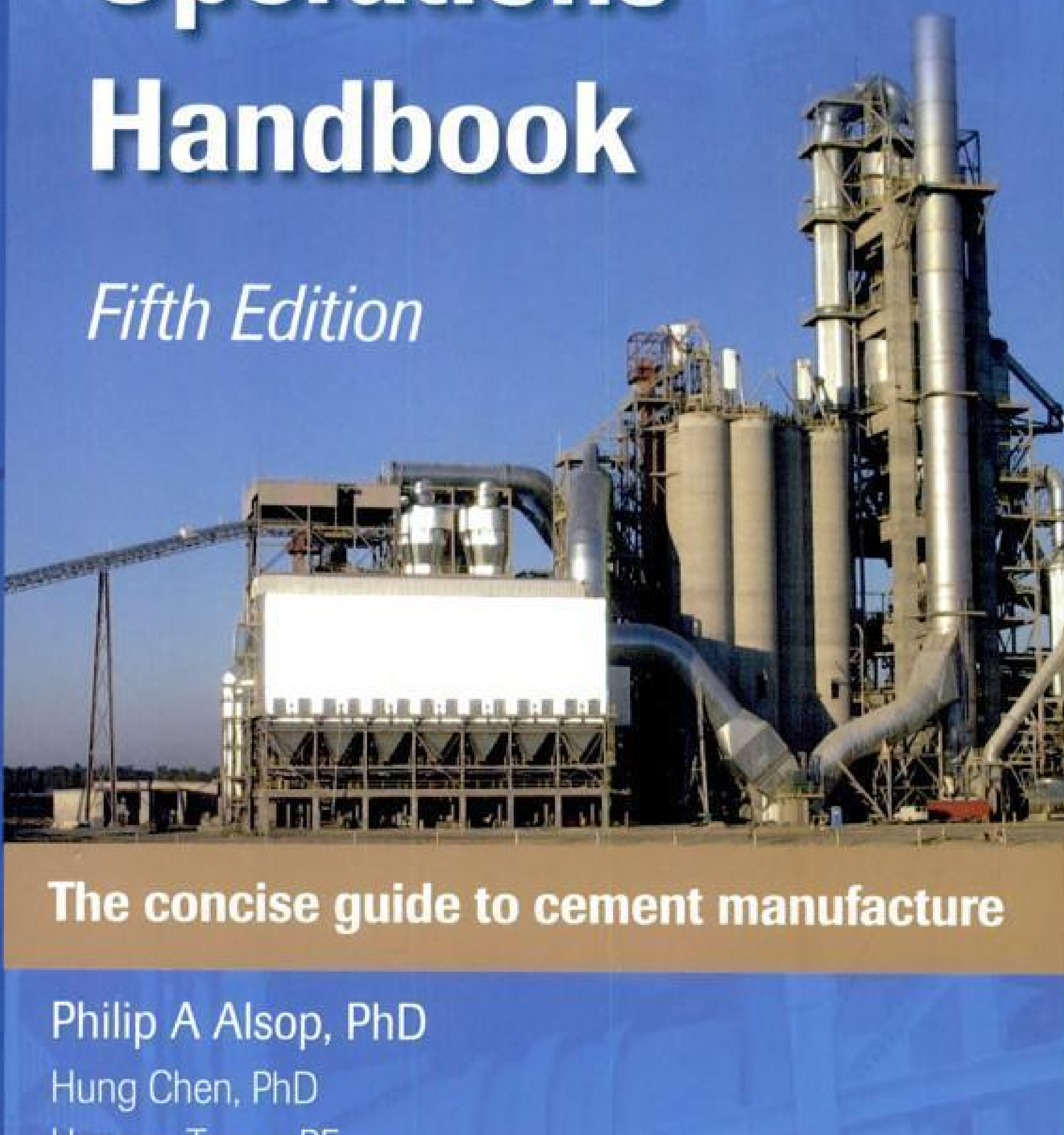
The Cement Plant Operations Handbook

Fifth Edition



The concise guide to cement manufacture

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International Cementreview

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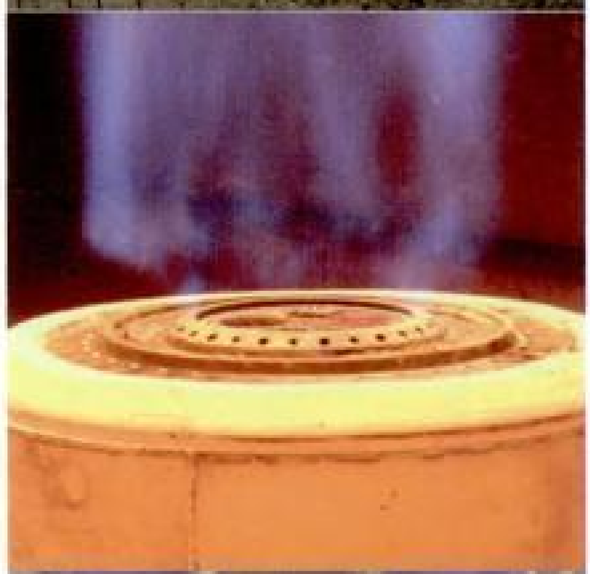

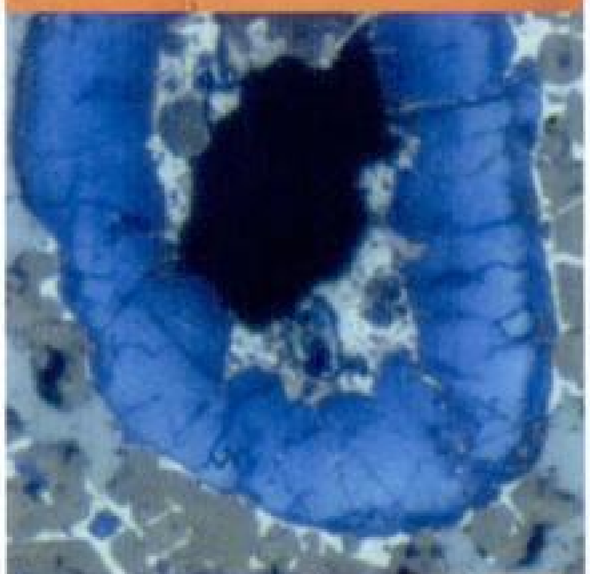



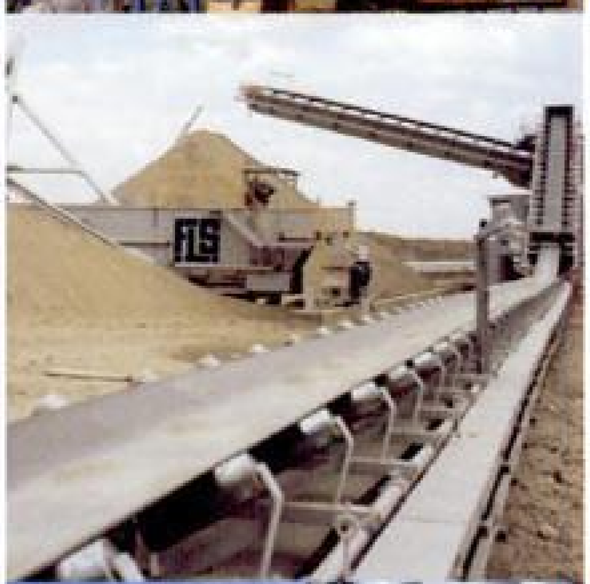
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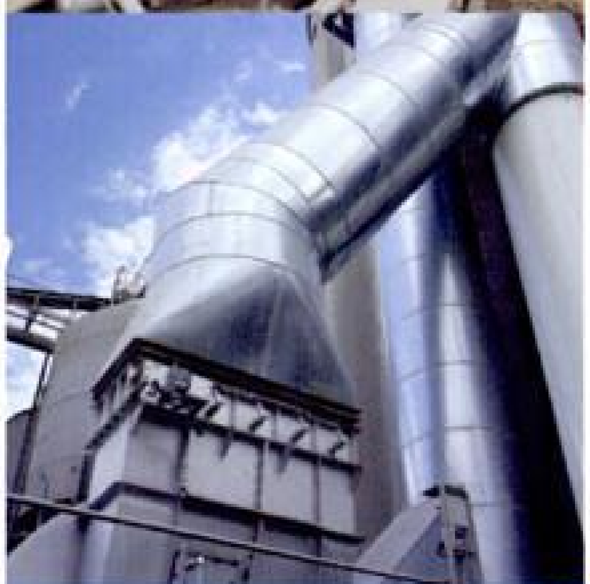




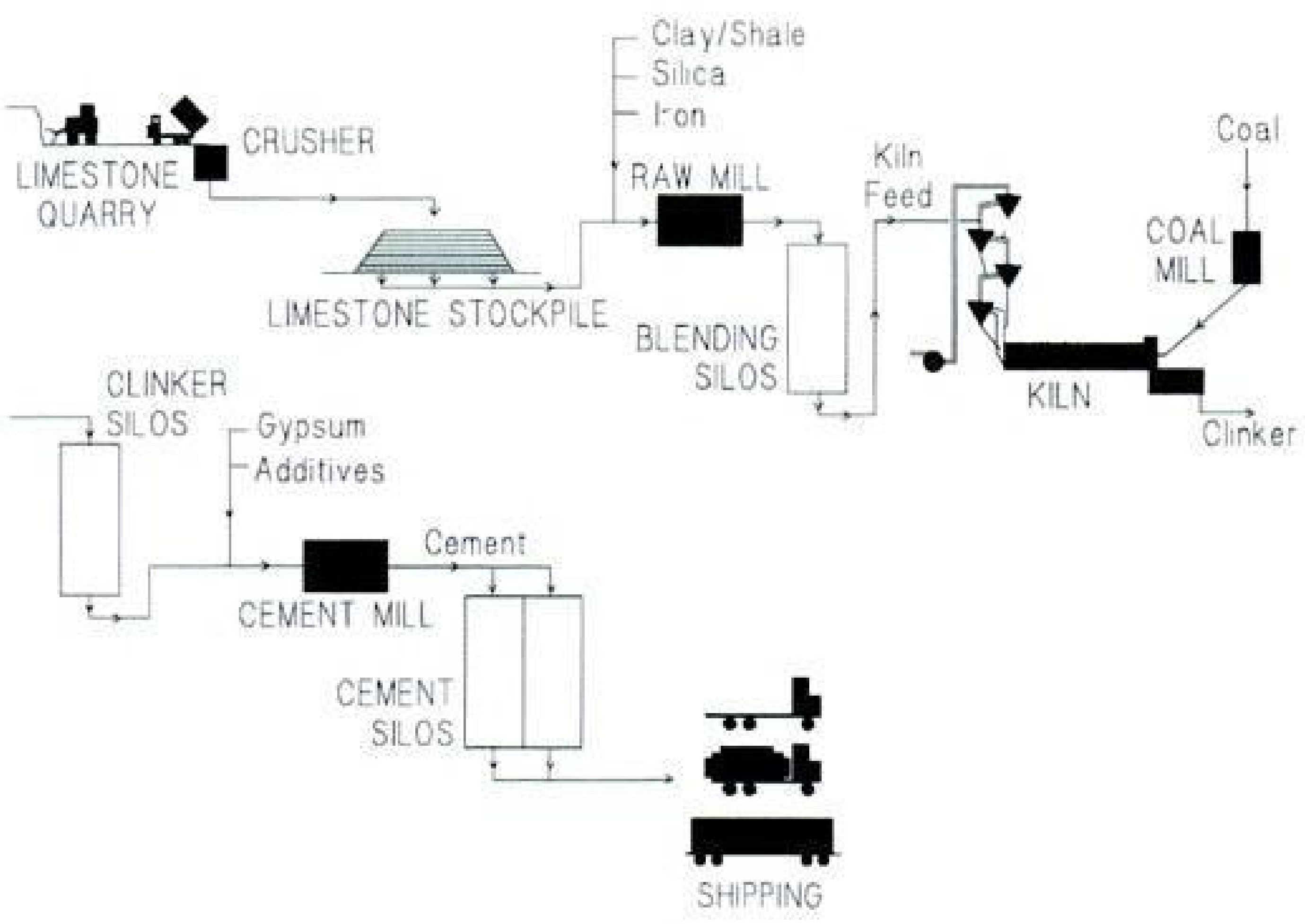
PETRON

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.

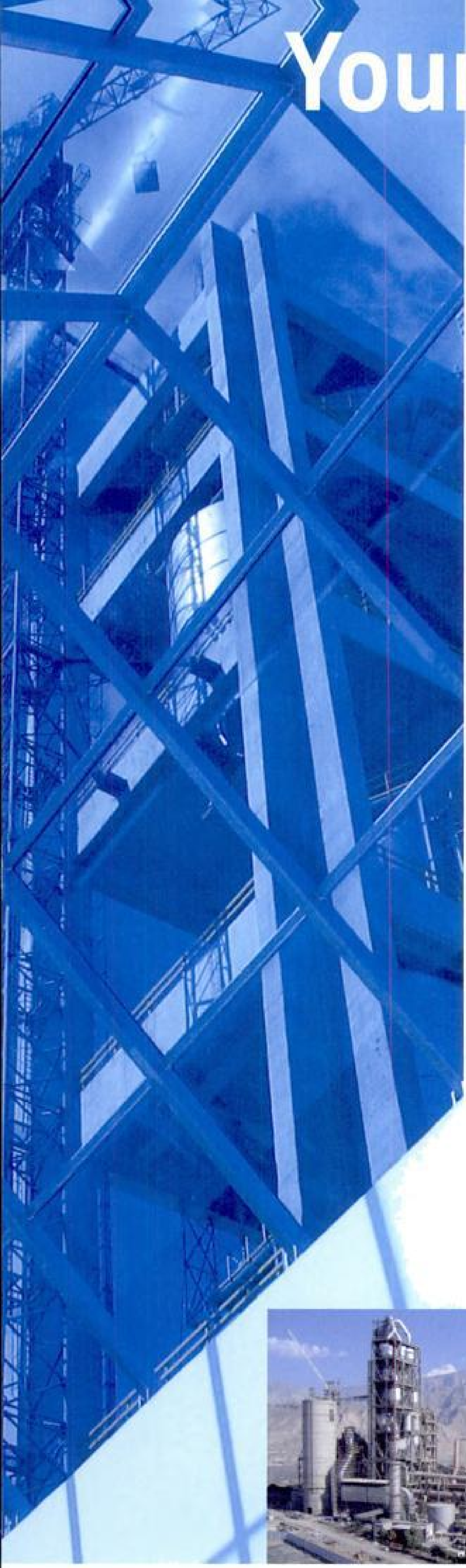
The world consumption of Portland cement has grown:

	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

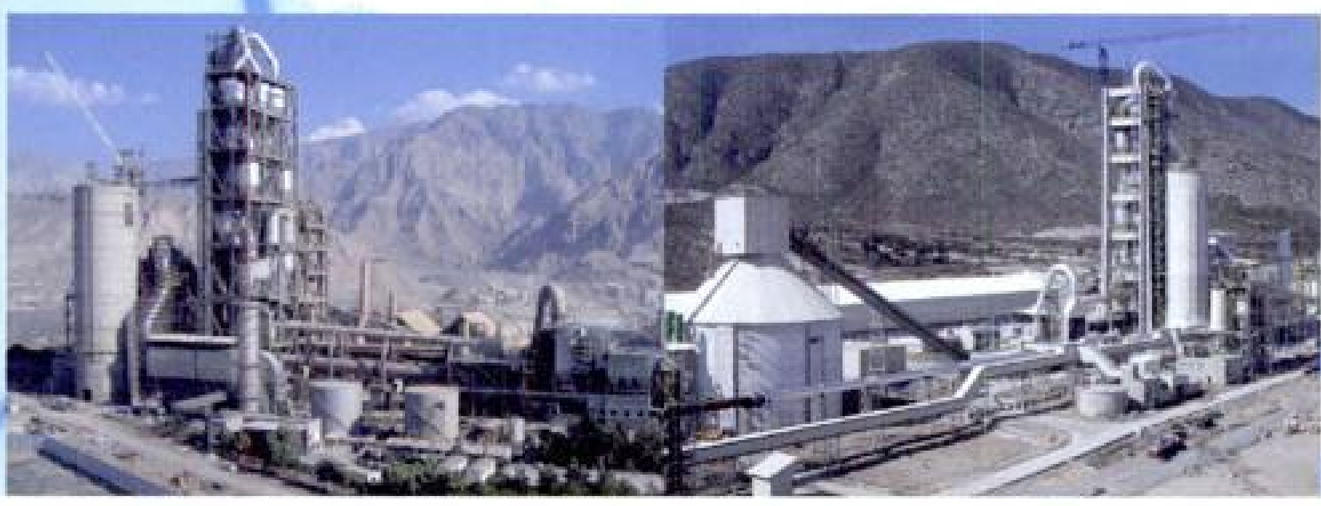
This shows a long-term growth rate of 2-3% per year accelerating to a little under 4% for the decade up to 2000 (Portland Bill, ICR, 7/2001, p. 92). Consumption since 2000 has increased by nearly eight per cent per year (ICR Global Cement Report, 7th Ed., p. 8)

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.



Complete cement plants made by Polysius.



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.

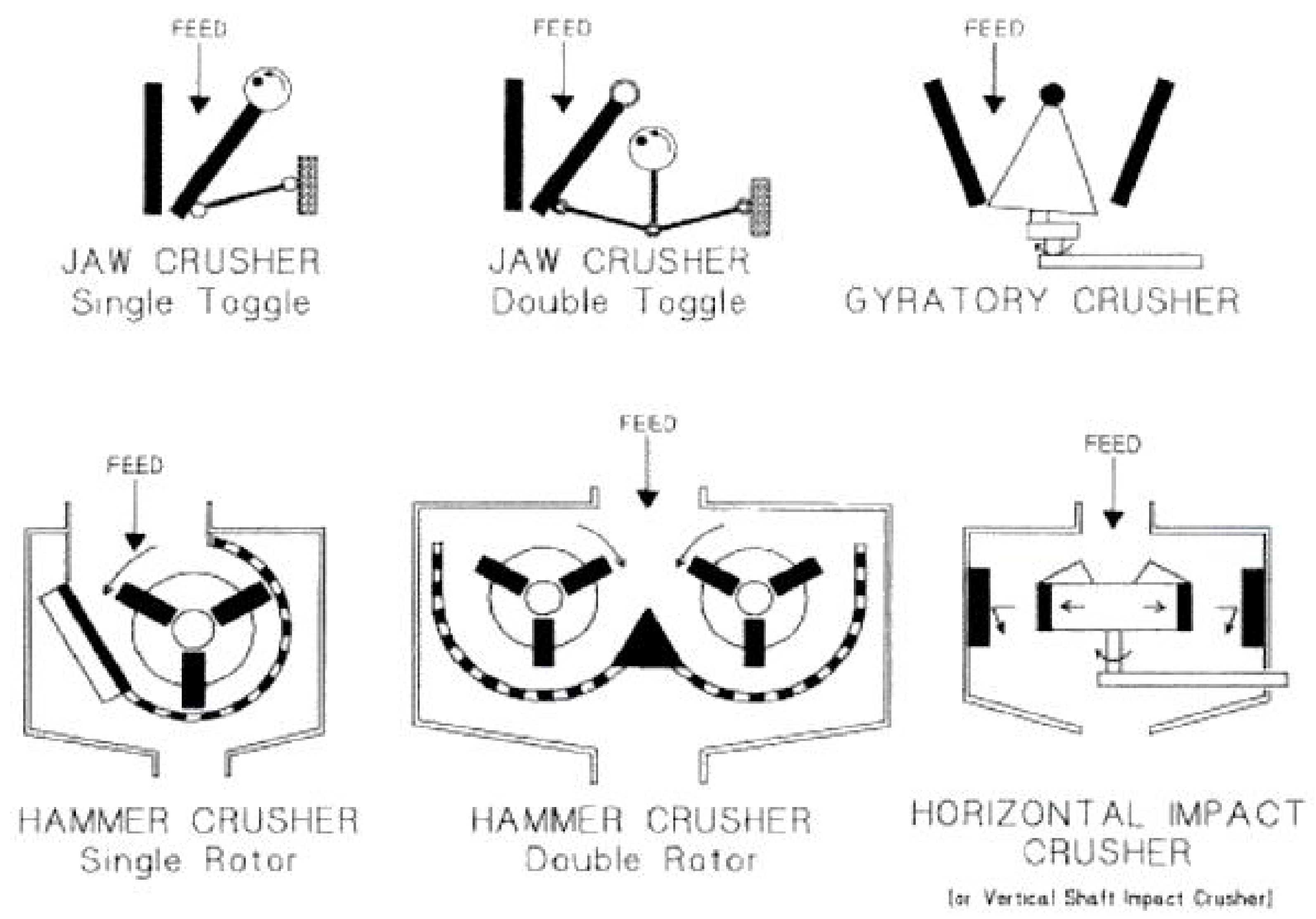
Impact machines may be either hammer mills or impact crushers. These are usually preferred for limestones with quartz contents of less than 10%. Wear is greater than for compression crushing and there is a larger proportion of fines, but moisture contents up to about 12% can be accepted and reduction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

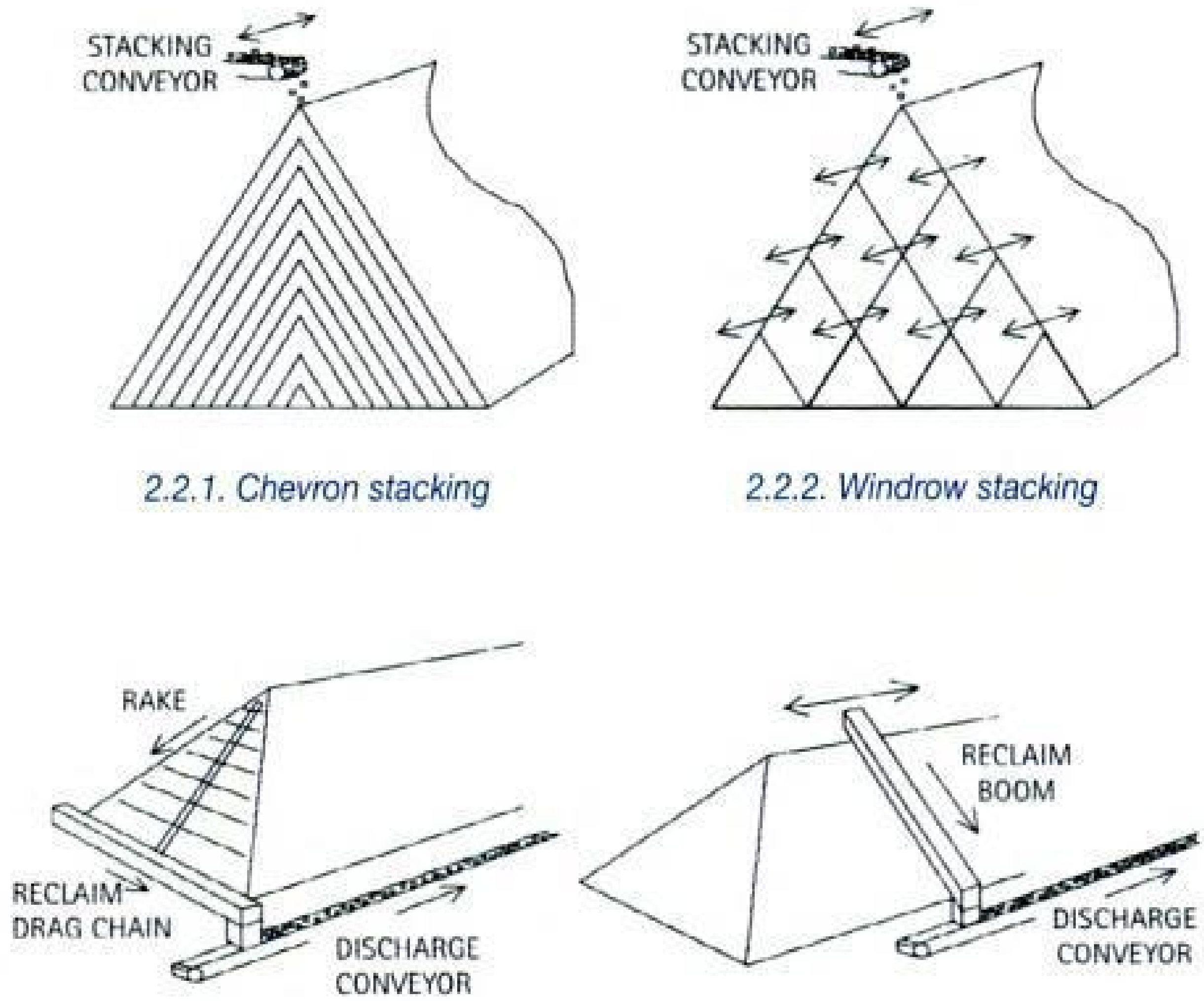
2.5. Pre-blending

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.


LOESCHE

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.

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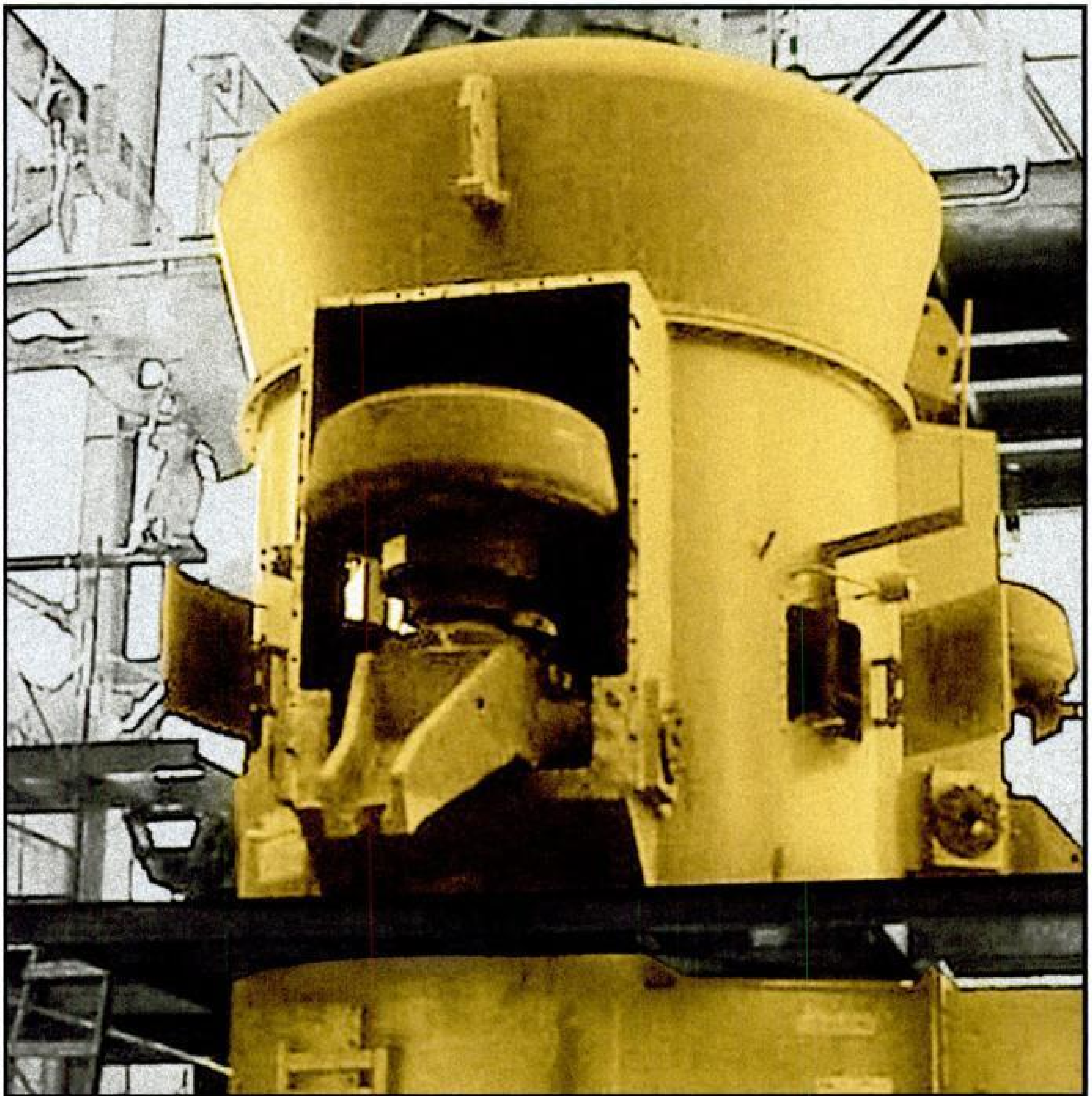


Modern blending silos are generally of continuous, controlled flow type with each silo having capacity of more than 24 hours kiln feed and yielding a blending ratio of 3-5; older silos are more like 2-3. Note that a given silo will show a lower blending efficiency if the feed is itself consistent. The 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

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.

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.

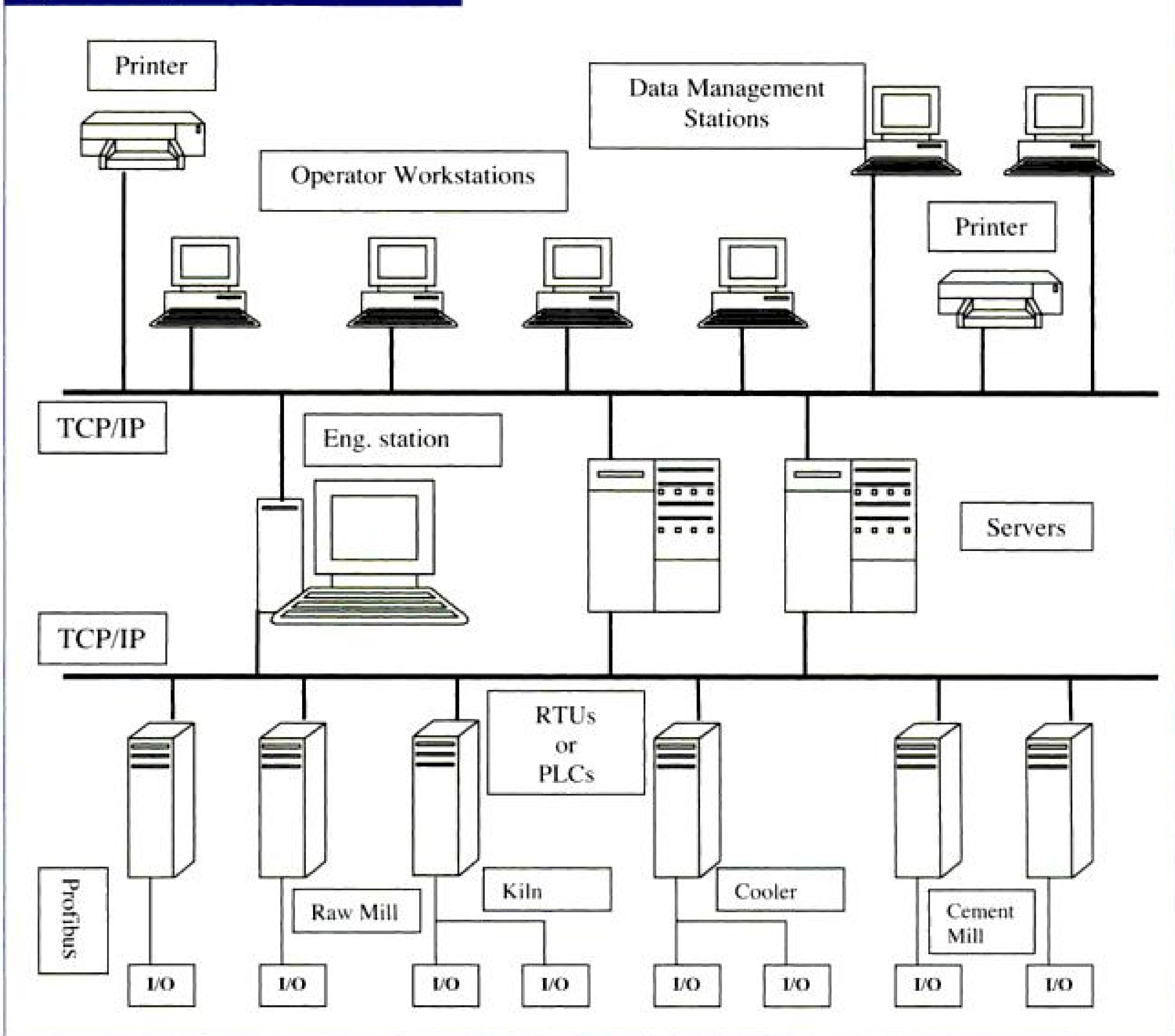
4.2. Control Systems

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/O 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

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 cm. 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:

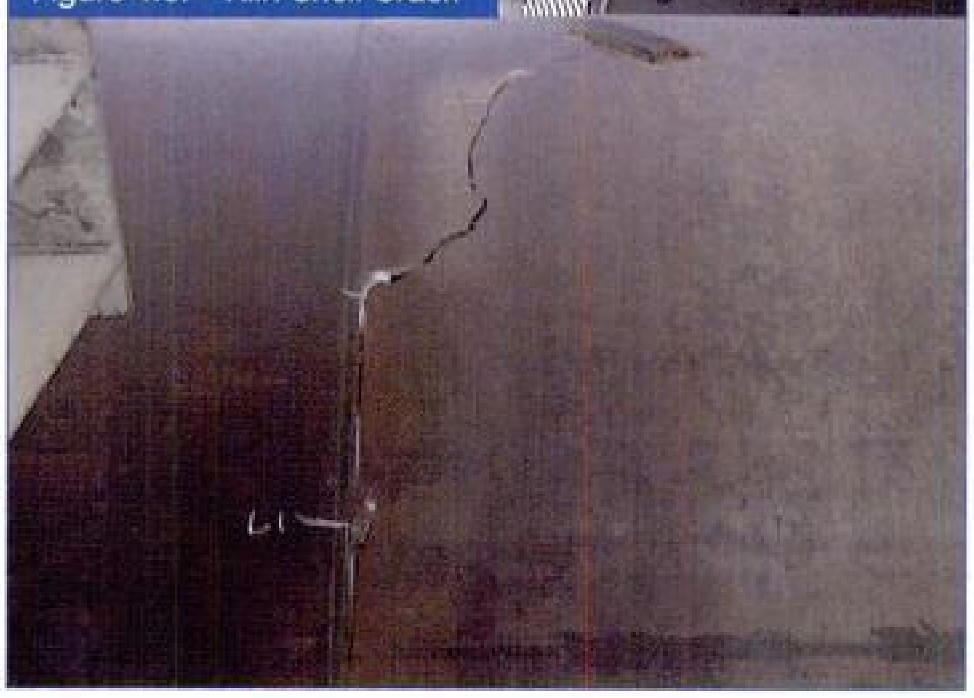
- standard
- air-setting
- chemically-bonded (usually phosphate-bonded).

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 plate 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 Nm³/million Cal*	*Total exhaust gas Nm³/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	chemical wastes
	distillation residues	waste solvents
	used oil	wax suspensions
	petrochemical waste	asphalt slurry
	paint waste	oil sludge
Solid waste fuels:	paper waste	rubber residues
	pulp sludge	used tyres
	petroleum coke	battery cases
	plastic residues	wood waste
	domestic refuse	rice chaff
	refuse derived fuel	nut shells
	oil-bearing earths	sewage sludge
	animal meal	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).

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 plate 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

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.

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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

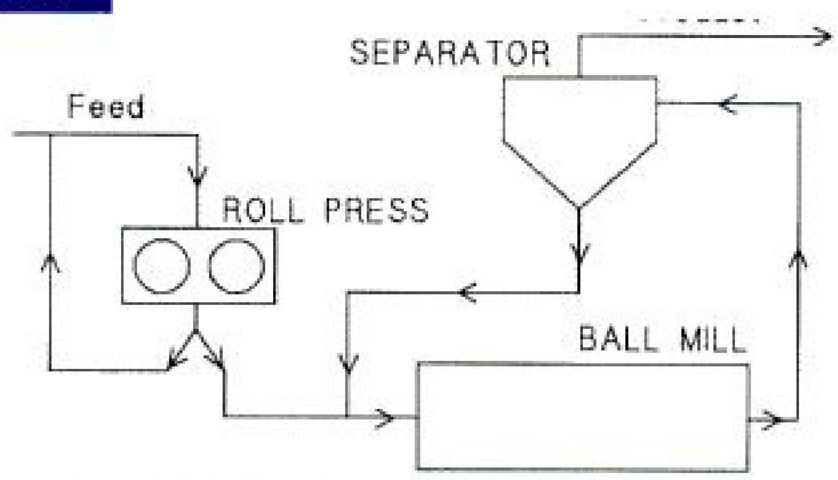


Figure 5.1.1. – Pregrinding with Slab Recirculation

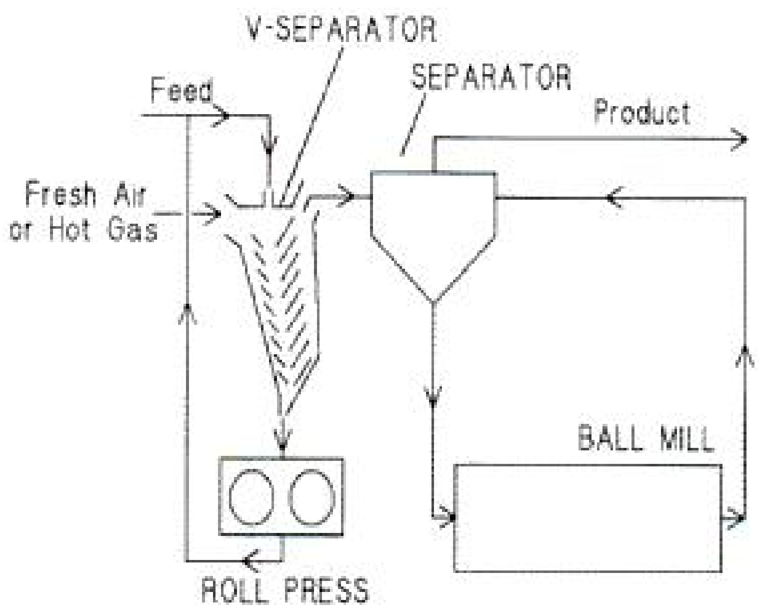


Figure 5.1.2. – Semifinish grinding "S"

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:

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

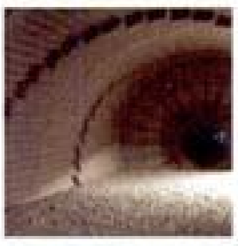
where C_3S^N = mean alite size, m
C_2S = % belite
TEA = % grinding aid

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, (cm^2/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 3600cm^2/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 3600cm^2/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

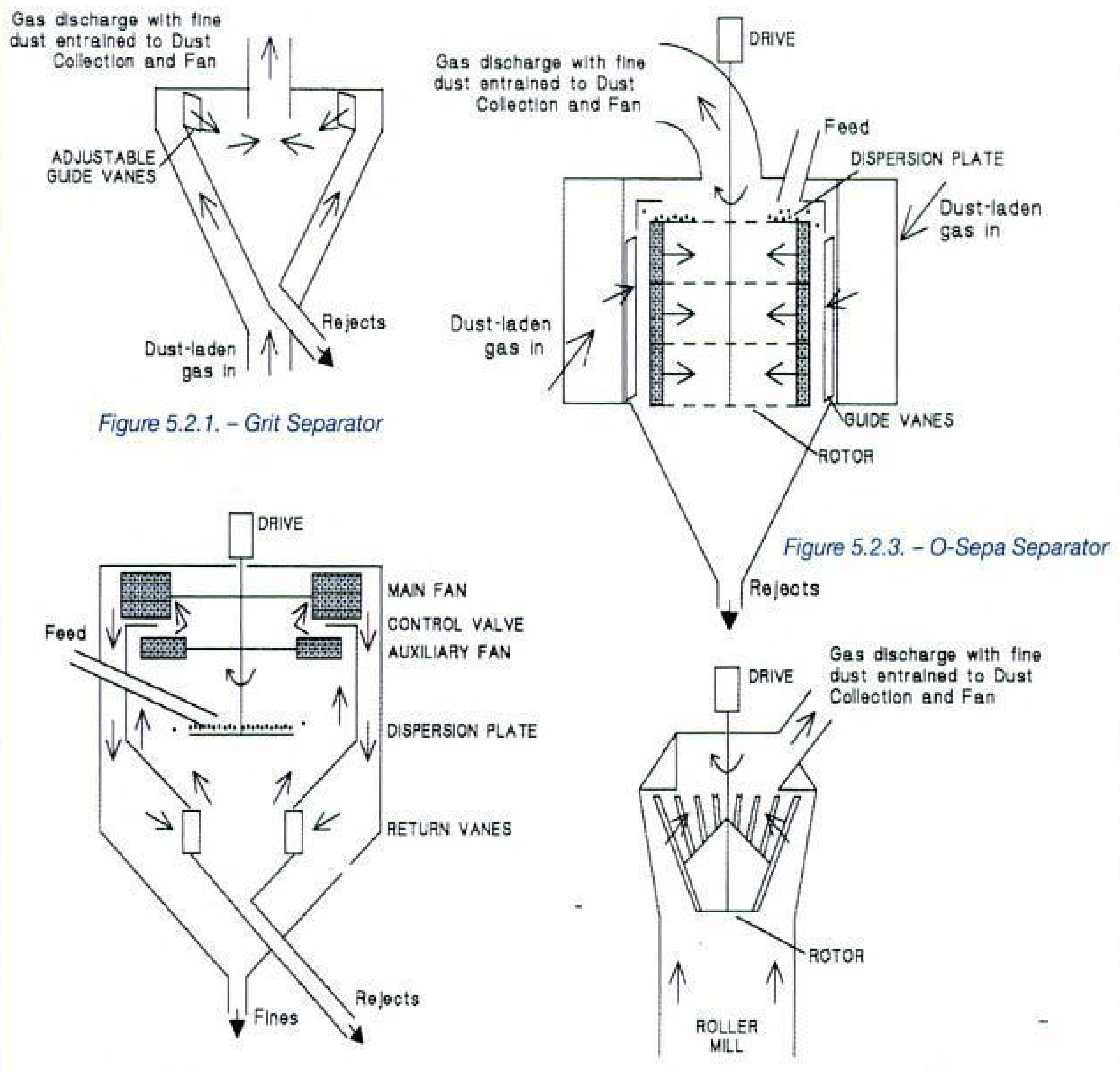


Figure 5.2.1. - Grit Separator

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

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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/NM³. Flame quenching, low-NO_x burners, or staged combustion (for precalciner kilns only) should approach 500mg/NM³. 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:

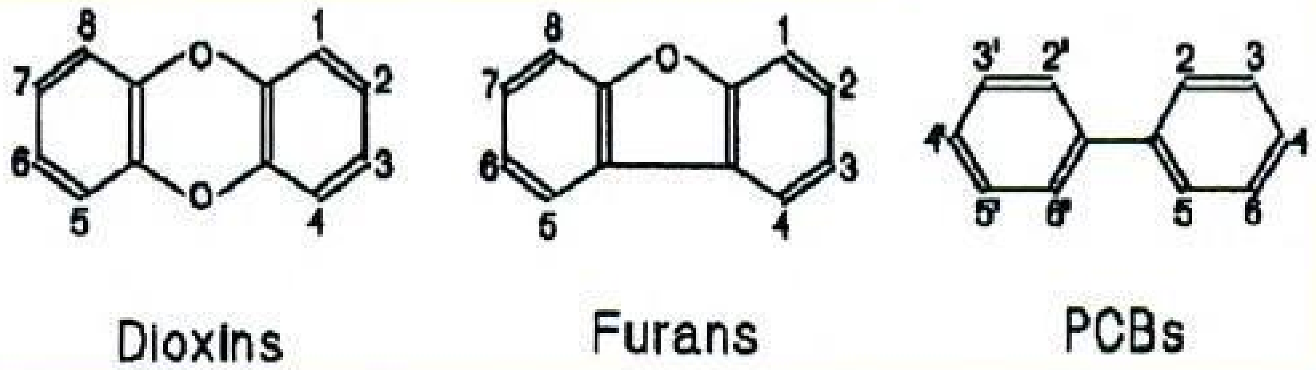
Minimum temperature		1200°C
Minimum retention time		2s
Minimum excess oxygen	- liquid fuels	3%
	- solid fuels	6%
Maximum CO (@ 11% O_2)		40ppm

(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:

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12. Plant Reporting

12.1. Definitions

The assessment of cement plant equipment and operations involves numerous terms and numbers, many of which are prone to varying definitions.

Plant capacity – Annual capacity can relate to various assumptions for kiln operation and cement inter-grinding. A reasonable standard is the designed, or established, daily clinker production assuming 90% annual run factor and 5% cement additives:

Annual cement capacity = Clinker t/day x 365 x 0.9 / 0.95

Some elaboration is still required for a plant which has excess cement milling capacity for producing blended cements or a deficit based on shipping clinker elsewhere for grinding.

Kiln run-factor – Various definitions have been encountered including fire-on time and running time exclusive of planned shutdowns. Feed-on time is suggested as the most significant parameter and should be expressed as a percentage of 8760 hours per year.

Kiln utilisation – Utilisation on a period basis is the average actual production rate in t/h divided by the base rate. The base rate is determined as the average over the best, say, five consecutive days of operation. A base run of other than five days may be used and it is expected that, with continuous improvement, the base rate will rise over time. While most kiln operators have long concentrated on maximising kiln up-time, much production may also be lost through operating at reduced rate due to instability or the limitations of peripheral equipment (eg reduced gas flow when raw mill is down). Utilisation allows focus on production rate as well as on operating time.

Mean time between stops (MTBS) – The total number of operating hours over a period divided by the number of kiln stops, expressed in days. The kiln is considered stopped for this metric if feed is off for, say, five minutes.

To facilitate materials balance for reporting, certain conventions are desirable though not essential.

Dry tonnes are used for production and inventory of raw materials though reversion to wet tonnes may be necessary in assessing quarrying, crushing and conveying efficiencies. Also, certain materials such as coal are usually bought with a standard moisture so that adjustment is required to reconcile inventory with purchased quantities and consumption.

Equivalent tonnes facilitate the compilation of materials and process cost contributions to the unit (tonne) of cement produced. An equivalent tonne of cement is usually assumed to be 950kg clinker and 50kg gypsum.

Then, for example, if there is 80% limestone in the raw mix and a kiln feed:clinker factor of 1.6:

1 Eqt Limestone = 950 x 1.6 x 0.8 = 1216kg

Also, if raw mill operation costs $3.00/tonne of kiln feed ground:

Unit cost of raw milling = $3.00 x (950x1.6)/1000 = $4.56 per tonne cement

The system requires adjustment when pozzolanic or masonry cements are produced which differ significantly from 95% clinker.

12.2. List of Reports

Minimal reports to plant management for monitoring operations include:

Daily

- *Production Report* with production, downtime, utilisation, and inventory by area (milling, burning, etc).

Monthly

- *Production Report.*
- *Quarry Report* with production figures and details of drilling, blasting, loading and hauling.
- *Process Summary* with operating data and efficiencies for each area.
- *Downtime Report* with total downtime and detailed breakdown for each area.
- *Shipping Report* with total cement shipped broken down by type, by bulk vs sack, by conveyance (road, rail, etc), and by destination market.
- *Paper Sack Inventory* reconciliation and sack breakage.
- *Quality Summary* with raw material, process and product analyses, and statistical variation.
- *Mobile Equipment Report* with availability, fuel consumption, and details of major downtime.
- *Manufacturing Cost Summary* with total unit cost and detailed breakdown by area, by individual equipment and by grouping (power, fuel, labour, etc).
- *Inventory Schedule* valuing product, process, fuel and warehouse inventories.
- *Order Status* itemising deliveries which have been rescheduled or are overdue.
- *Manpower Report* comparing actual numbers with establishment by department, and including overtime, hiring and terminations.
- *Safety Report* detailing all accidents and total days worked (to month-end) since last lost-time accident.
- *Projects Report* covering planning, ordering, progress, and budget of capital projects managed by plant staff.

While computerised data processing allows instantaneous calculation of much data, it must be recognised that, due to the crudity of, particularly, weight measurement, period averages will be more meaningful.

12.3. Inventories and Feeders

Stockpile inventories are often calculated from production and consumption figures. At least monthly, all piles should be surveyed and their capacity calculated from standard bulk density assumptions. For large, disorderly piles, flyovers are particularly valuable; aerial digital imaging is now accurate to 1m horizontally and 15cm vertically. The base of stockpiles when constructed on soft ground becomes uncertain and errors are usually encountered when the pile is recovered. "Creative grading" of the base usually solves the problem.

Weighfeeders should be calibrated regularly and cross-checked against inventories and indicated feed rates at other stages of the process. Fine material should be de-aerated before loading to a belt weigh-feeder. Impact flow meters are particularly liable to instability and error if located in a moving air flow.

12.4. Downtime Reporting

One of the most useful and revealing reports of plant operations is the downtime summary. It is believed that SP and precalciner kilns should be available to operate not less than 90% or 330 days/year (Buzzi, WC, 11/2003, p. 91). The dictates of thermal efficiency and of sophisticated process control have led to large numbers of items of ancillary equipment and of control signals, failure of any

	Managers	Supervisors & Professional	Hourly
Plant Manager	1		
Secretary			1
Administration Manager	1		
Accountant		1	
Clerical (3 – purchasing, HR accounting)			3
Mining Manager	1		
Foreman		2	
Drill/Blast Operators (2)			22
Mobile Equipment Operators (10)			
Crusher Operators (2)			
Helpers (2)			
Garage Foreman			
Mechanics (6)			
Production Manager	1		
Process Engineer			
Automation Engineer		6	
Shift Supervisor (4)			
Operators (4)			
Helpers (4)			
Labourers (12)			
Packhouse Manager	1		
Operators (4)			
Helpers (4)			8
Maintenance Manager	1		
Mechanical Engineer		6	
Electrical Engineer			
Supervisors (4)			
Maintenance Clerk			
Warehouse Clerks (2)			
Mechanics (15)			
Electricians (6)			
Instrument Technicians (3)			27
QC Manager	1		
Technicians (7)			7
Environment, Health & Safety Manager	1		
Helper			1
Sub-totals	*8*	*15*	*93*
Total			*116*

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The Cement Plant Operations Handbook

Fifth Edition

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EDITORIAL

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