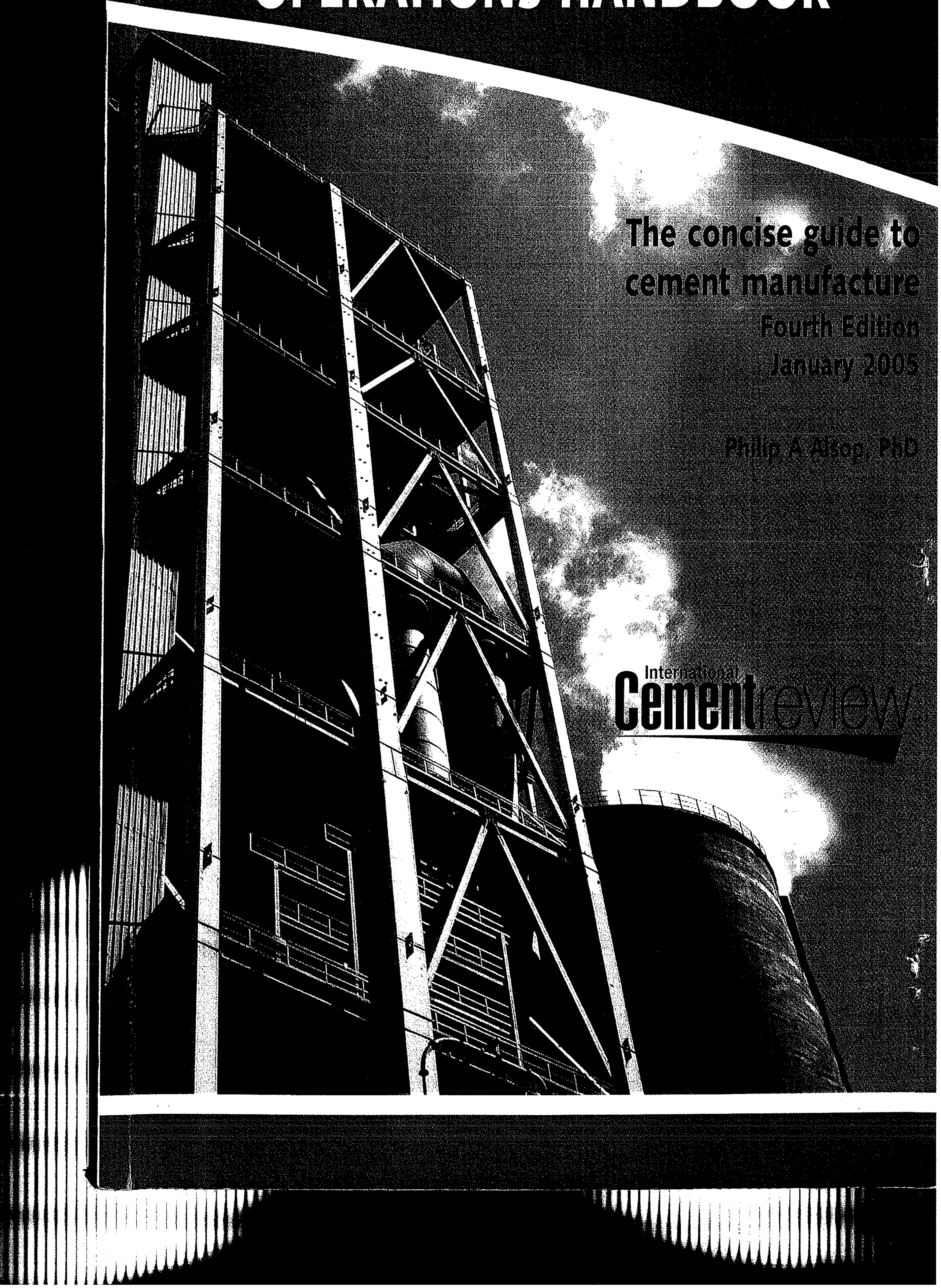
THE
CEMENT PLANT OPERATIONS HANDBOOK



The concise guide to cement manufacture

Fourth Edition
January 2005

Philip A Alsop, PhD

International Cement review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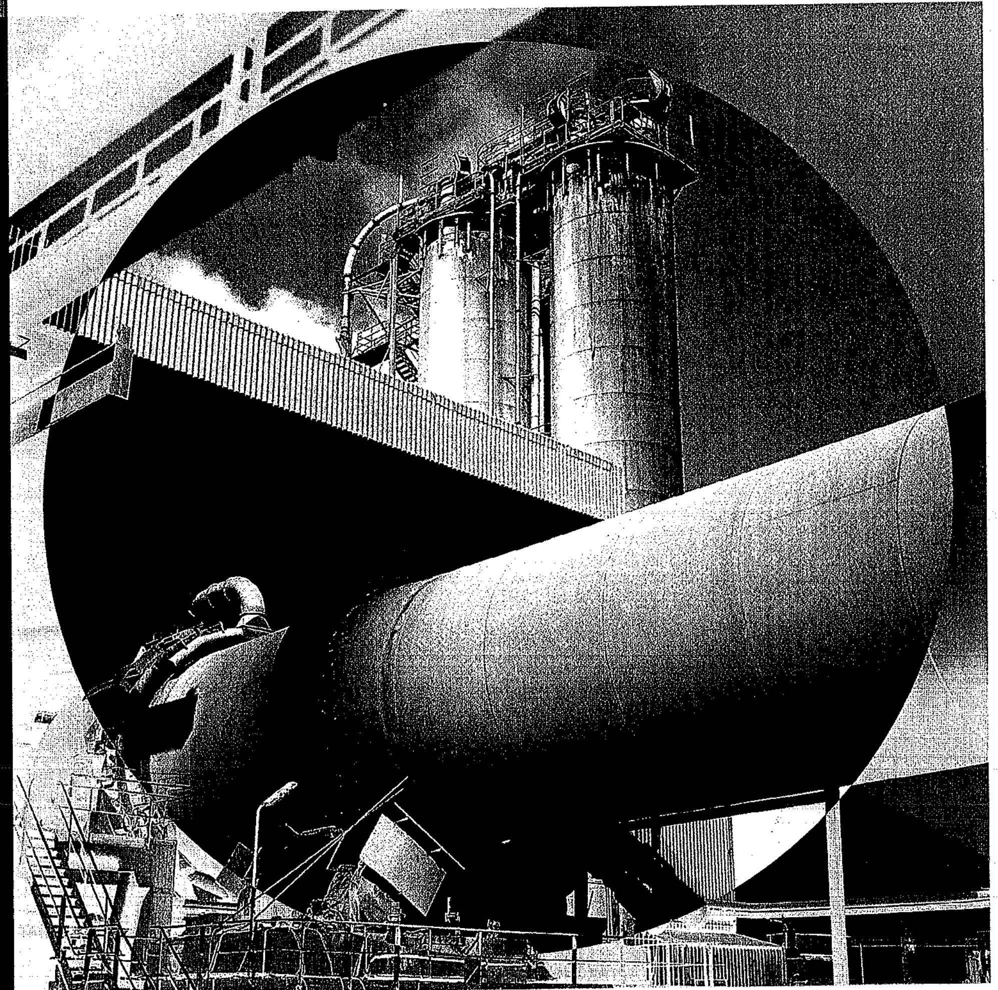

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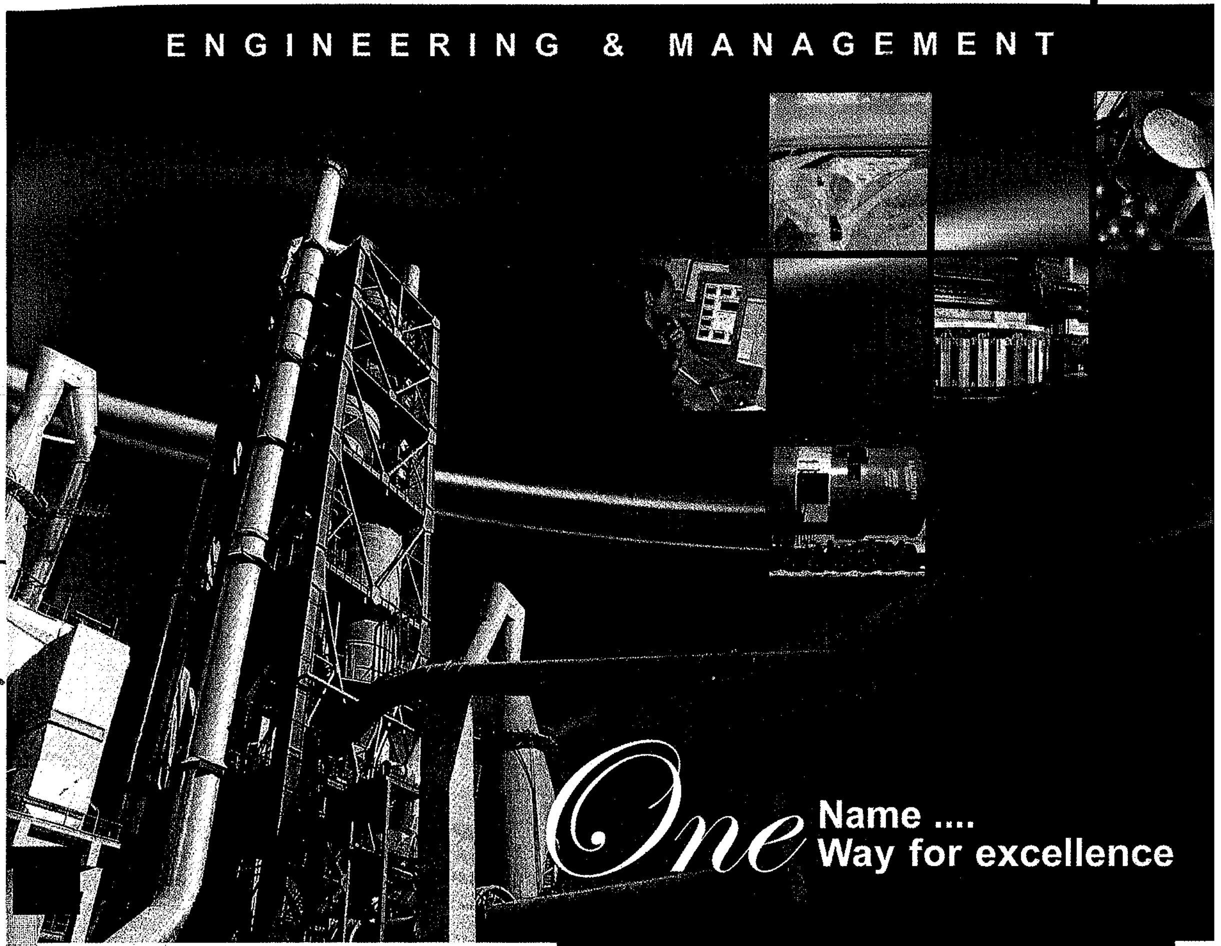

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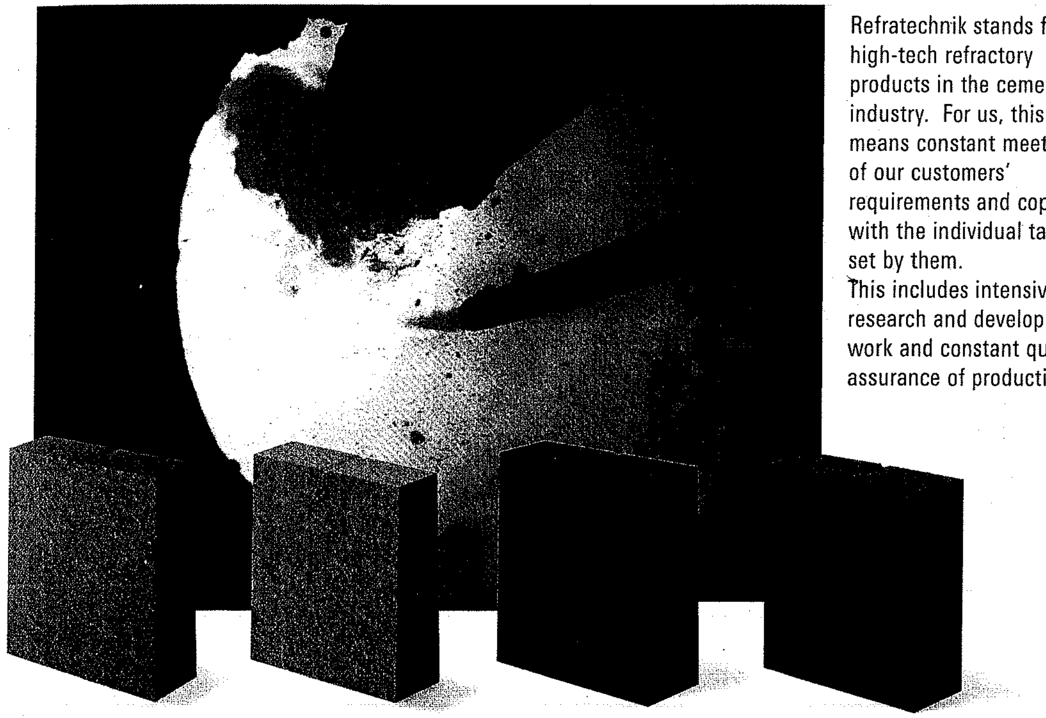


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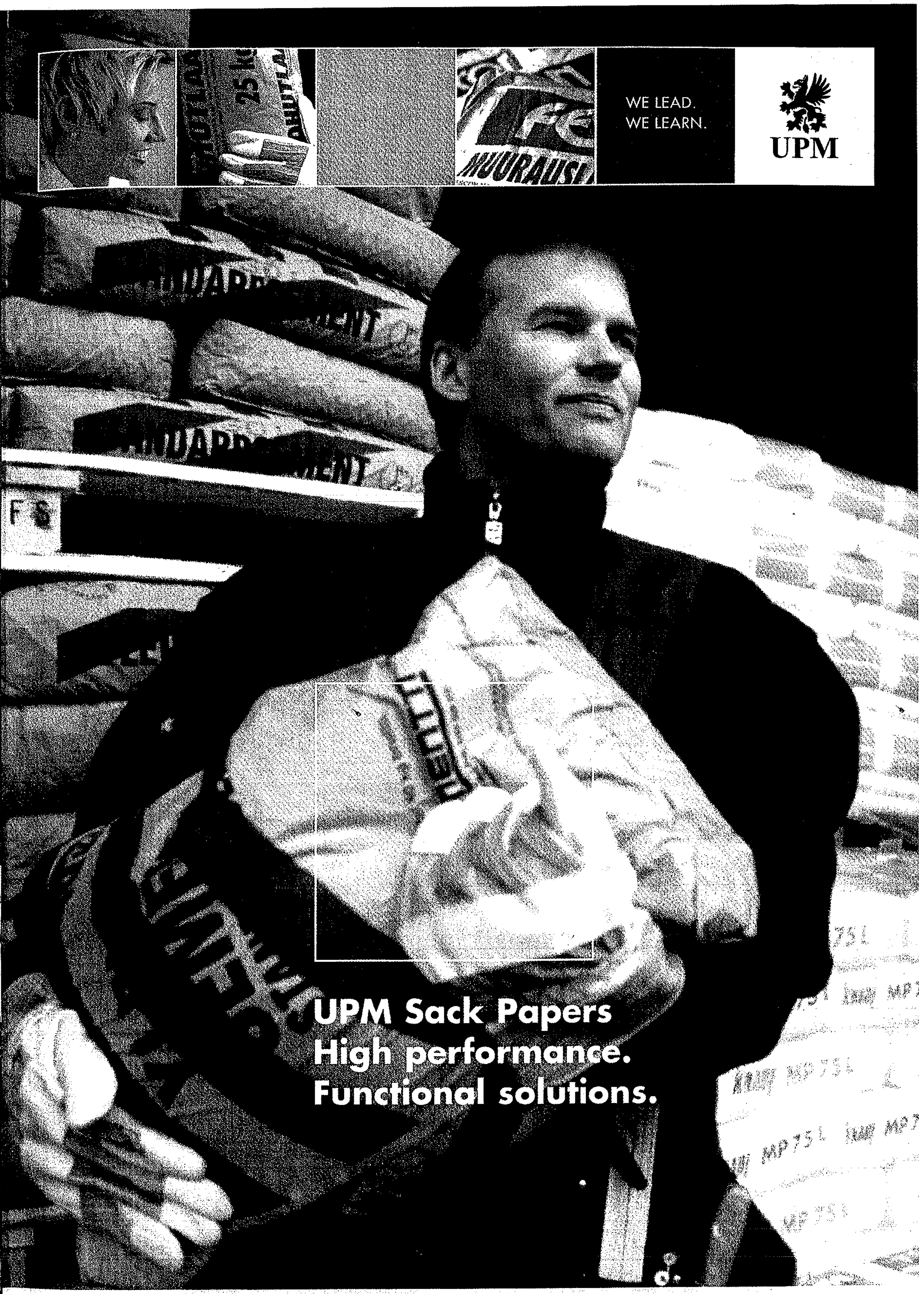
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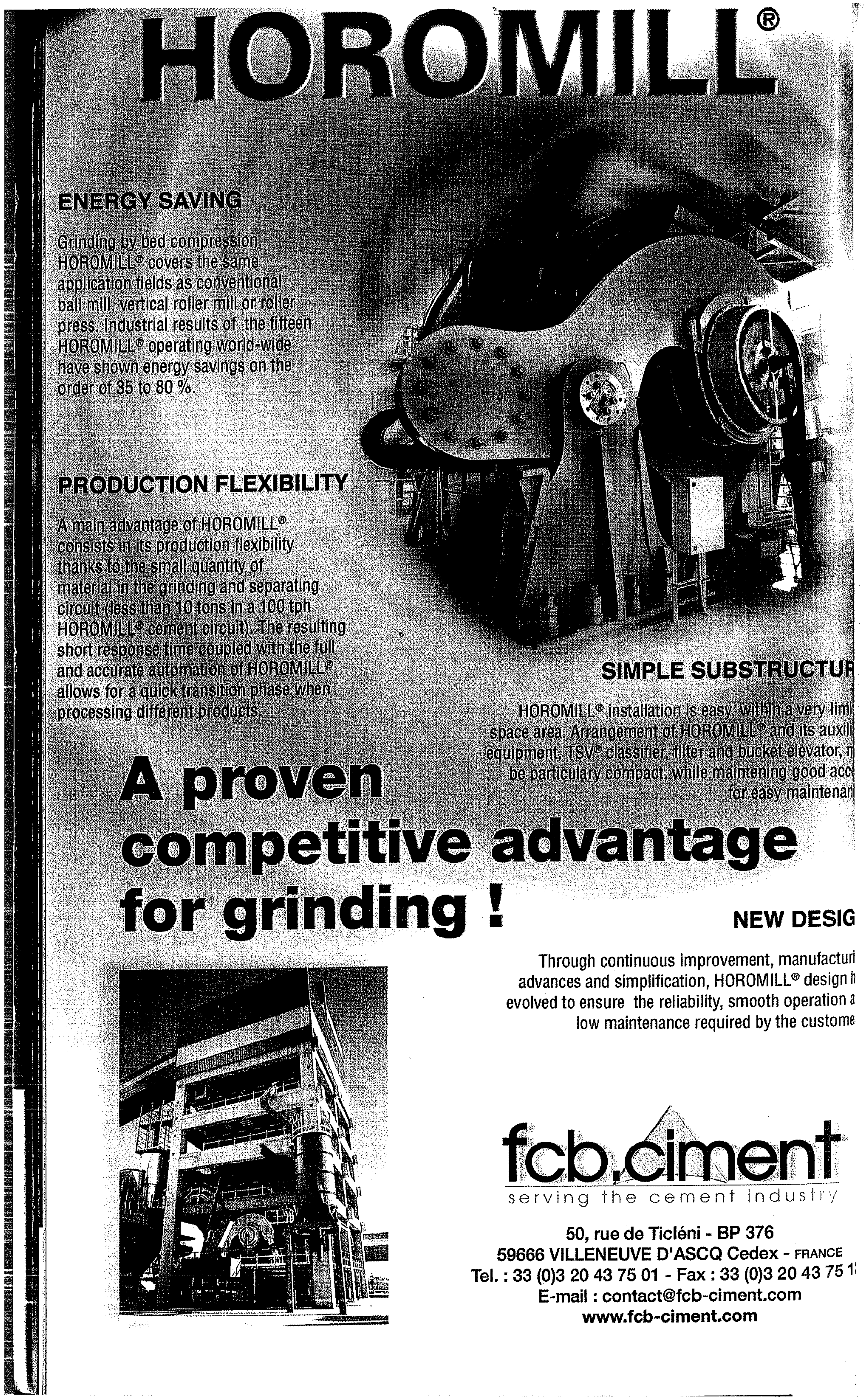
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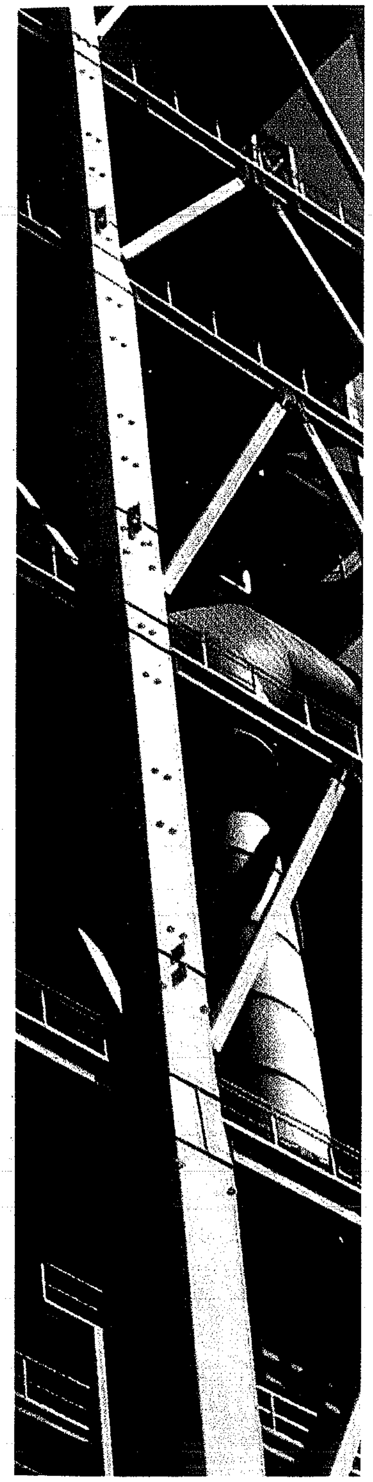

Cement Plant Operations Handbook

for Dry Process Plants

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Preface

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

The scope attempted comprehends:

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

The contents are somewhat idiosyncratic and the authors once again claim proprietary rights over all expressions of ignorance and opinion.

Philip Alsop
Bern, Switzerland.
December 2004

Obiter Dicta

- We are not in business to make cement, we are in business to make money.

- The primary task of operations management is to maximize profit. This is not the same as maximizing production or minimizing cost as either may incur a disproportionate sacrifice of the other.

- "We are not running out of energy or natural resources. There will be more and more food per head of the world's population. Fewer and fewer people are starving. In 1900 we lived for an average of 30 years; today we live for 67. According to the UN we have reduced poverty more in the last 50 years than we did in the preceding 500, and it has been reduced in practically every country. Nor will we lose 25-50% of all species in our lifetime – in fact we are losing probably 0.7%. Acid rain does not kill the forests, and the air and water around us are becoming less and less polluted. Mankind's lot has actually improved in terms of practically every measurable indicator."
  Bjorn Lomborg;
  The Skeptical Environmentalist;
  Cambridge 2001

- "State industrial planning doesn't work, but we seem to ignore this lesson when it comes to the environment. We are in a world of 'targets', just as self-defeating as old Soviet five-year plans. The assumption, highly debatable, is that the Earth is being destroyed by climate change. The solution, highly improbable, is that the Kyoto treaty will make a difference to this threat. The effect, absolutely certain, is that voters will be made to pay."
  Charles Moore;
  Daily Telegraph; 13 Nov 2004

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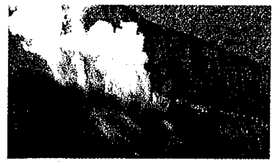

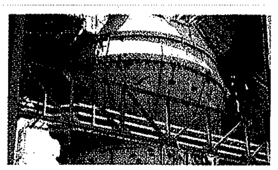

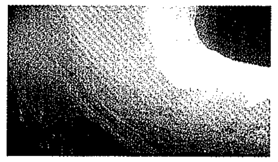

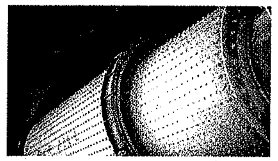

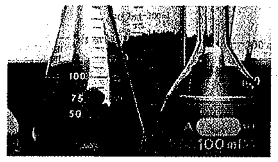

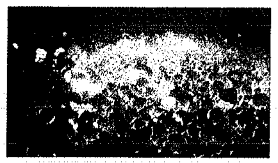

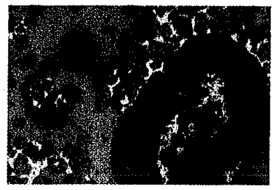

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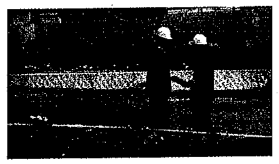

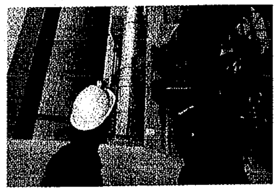

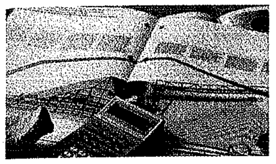

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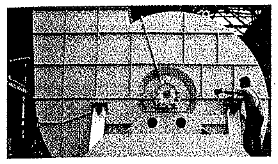

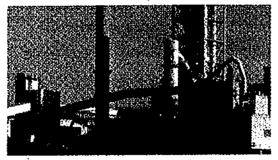

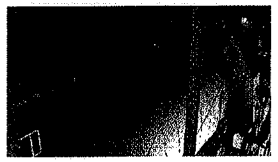

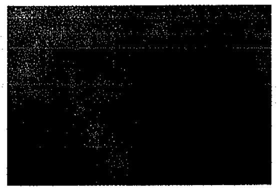

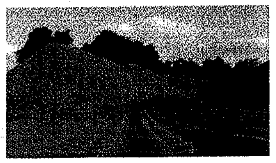

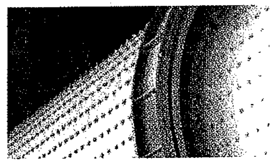

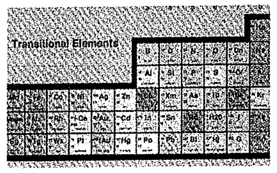

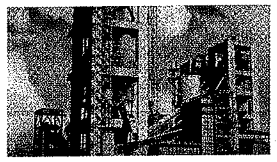

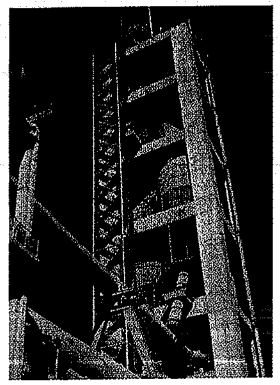

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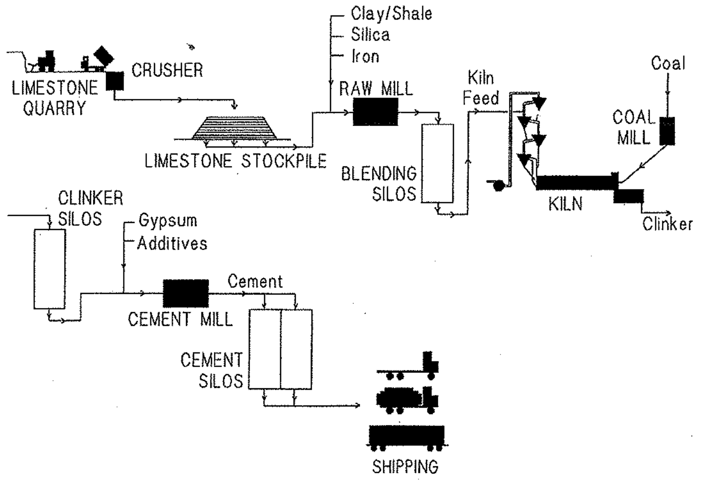
1 Introduction

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

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

Figure 1.1: Cement Plant Schematic Process Flow



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

1.1 History of Cement Manufacture

The ancient history of hydraulic mortars is extensive but becomes appreciable with the widespread use of mixtures of natural pozzolans and burned lime by both Greeks and Romans. The Pantheon in Rome is the only perfectly preserved building from this period; it was constructed in 27BC and rebuilt 117-125AD and is of pozzolan-lime concrete with an unsupported dome spanning 45M. Portland cement was developed in the 19th century and is so called due to its resemblance in colour and character to the naturally occurring stone of Portland Bill, off the south coast of England.

Following are some of the more significant dates in the development of Portland cement manufacture (Alsop; ICR; 7/2002; pg 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	30 million t/Y	1.5 billion	20kg
1925	150 million t/Y	2.0 billion	75kg
1940	400 million t/Y	2.2 billion	180kg
1955	600 million t/Y	2.7 billion	220kg
1974	1,000 million t/Y	4.0 billion	250kg
2000	1,500 million t/Y	6.0 billion	250kg

This shows a long-term growth rate of 2-3% per year accelerating to a little under 4% for the past decade (Portland Bill; ICR; 7/2001; pg 92).

Total international cement trade was 122 million tonnes in 2001 with 80 million tonnes seaborn (Thomas; WC; 11/2003, pg 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, pg 25). World consumption is projected at 3,060 million tonnes per year by 2020 (Betts, ibid).

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

2 Raw Materials

2.1 Raw Materials

The composition of Portland cement varies from plant to plant due both to cement specifications and to the mineralogy of available materials. In general, however, an eutectic mix is sought which minimizes 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, pg 31.)

Cement mixes vary from 'cement rock', a single component which, as mined, contains appropriate proportions of all the required minerals, to 4- or 5- 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 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, pg 262). Additives are usually brought in, albeit in small quantities.

The conventional explosive used for limestone quarrying is ANFO (ammonium nitrate activated with ca5% 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	grams 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 is also a factor in selecting raw materials. In particular, silica additives containing large-grain quartz are very difficult to grind and can result in hard burning and high fuel consumption. If quartz silica is employed it should, preferably, have a natural grain size of less than 50µ.

In recent years, cement kilns have been increasingly employed to utilize 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	coal fly ash	metallurgical slags
agents:	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 (Sec 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, pg 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, pg 34). Mobile crushers are common in aggregate quarries but rare for cement (RP; 9/1994, pg 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, pg 12).

Hopper and silo flow processes are reviewed by Bresler (WC; 2003/BMH, pg 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, pg 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.

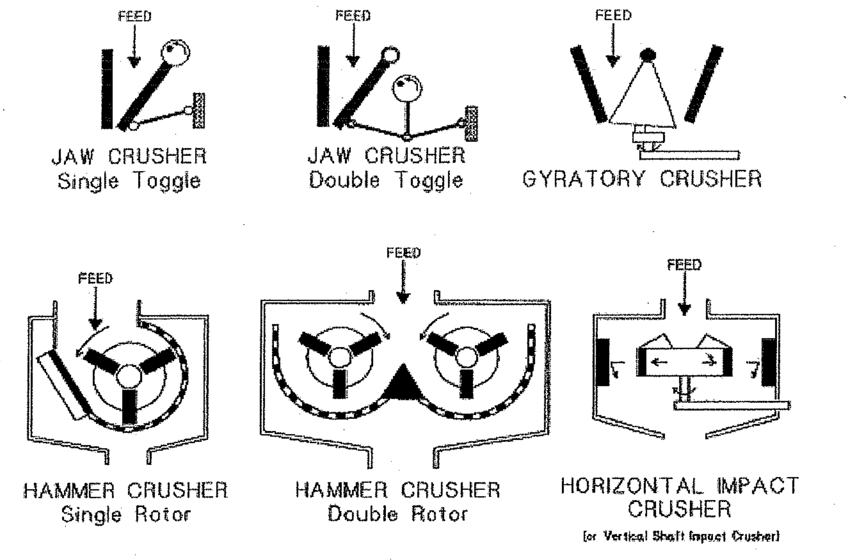
Impact machines may be either hammer mills or impact crushers. These are usually preferred for limestones with quartz contents of less than 10%. Wear is greater than for compression crushing and there is a larger proportion of fines, but moisture contents up to about 12% can be accepted and reduction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

Figure 2.2: Stackers and Reclaimers

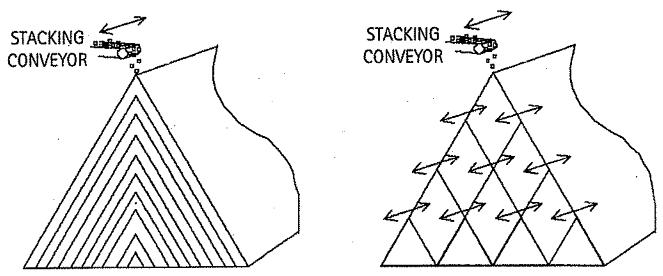


Figure 2.2.1: Chevron Stacking

Figure 2.2.2: Windrow Stacking

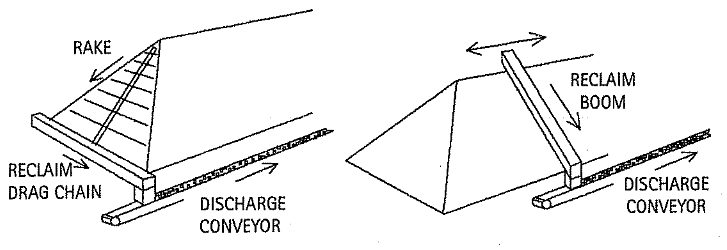


Figure 2.2.3: End Reclaim

Figure 2.2.4: Side Reclaim

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, or by neutron activation analysis (Chidambaram & Mokrin; WC; 12/2003, pg 59) 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.

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

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

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

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

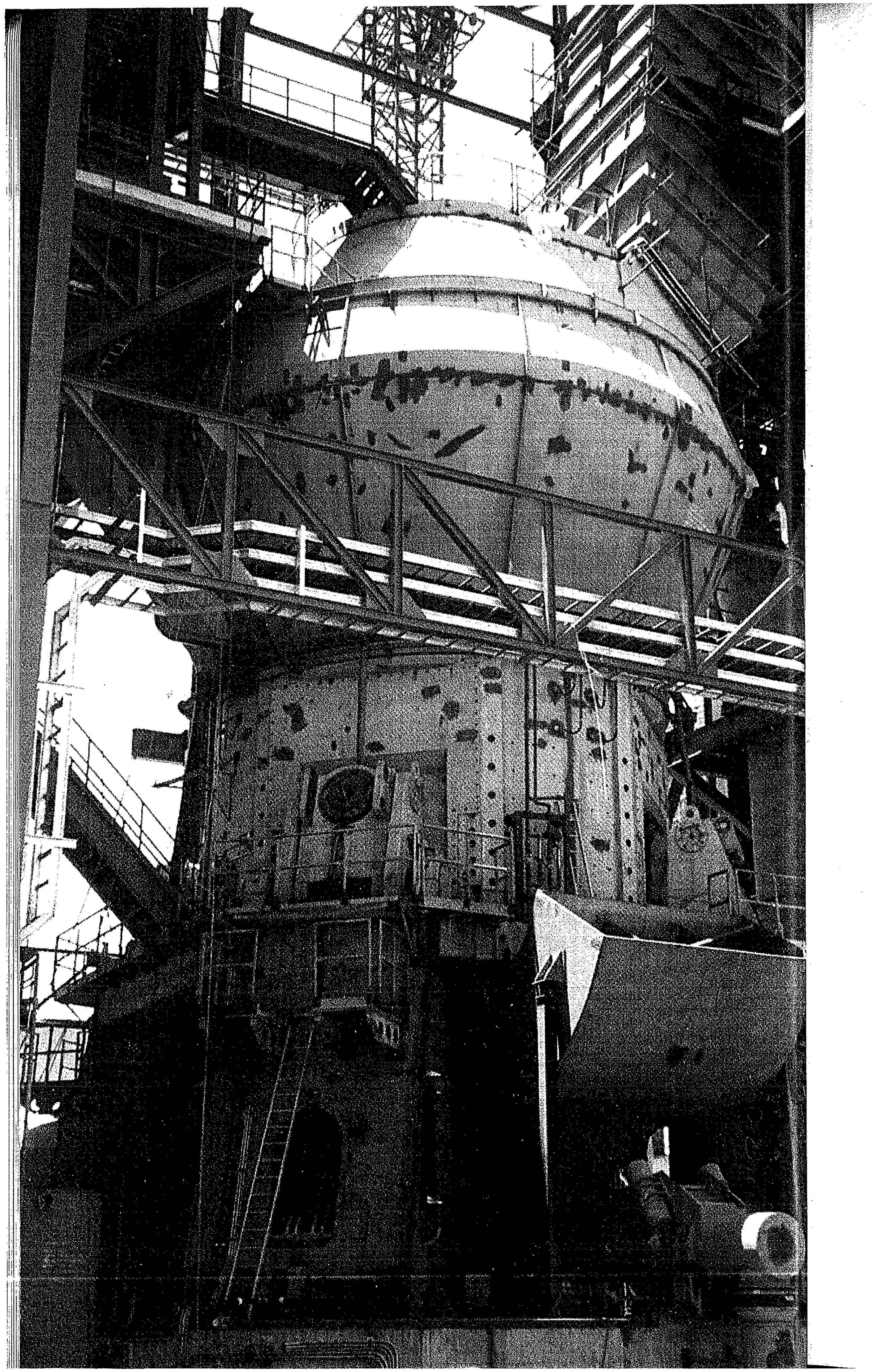
End Reclaim – various systems to scrape an entire end face with a transverse scraper at floor level moving material to a discharge conveyor (Figure 2.2.3).

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

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

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

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



3 Raw Milling and Blending

3.1 Raw Milling

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/sec above the ball charge), can handle 8%. Centre discharge mills (Double Rotator) and fully air-swept mills (5-6M/sec) 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 maximizing 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.

In the absence of pre-blend/storage of limestone and shale, or if the materials as fed to the raw mill are effectively dry, silos are appropriate for the feed materials. Where wet materials are fed for drying in the raw mill, silos are unnecessary.

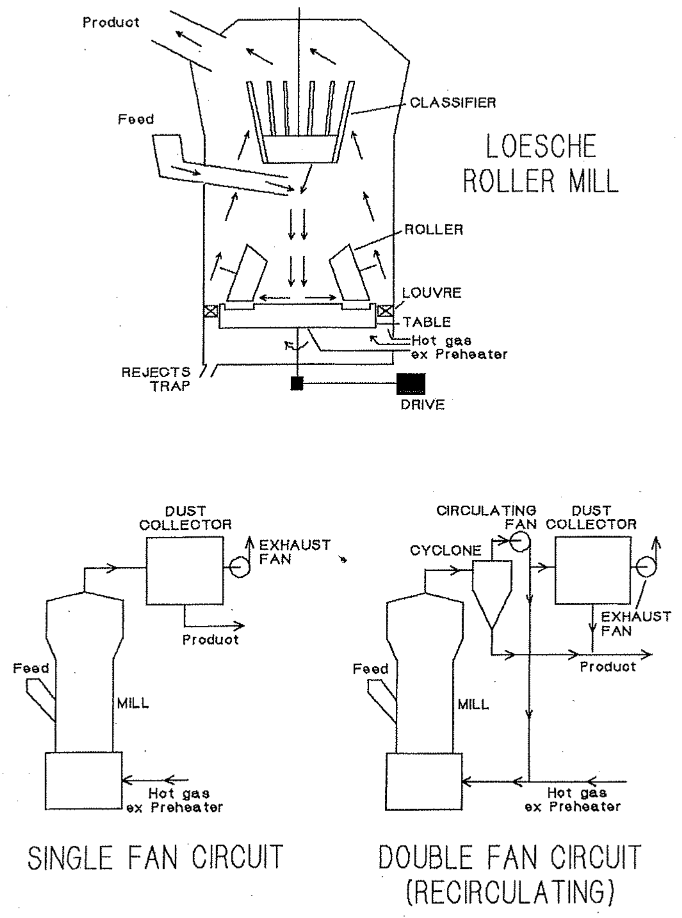
Ball mill operation is described in more detail under finish milling (Section 5.2).

Roller mills have a lower specific power consumption than ball mills. Loesche mills (Figure 3.1) comprise 2-4 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, pg 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, pg 75) and to ball mill high-efficiency separators (ICR; 7/2003, pg 49). Other means of increasing roller mill capacity are described by Jung (CI; 2/2004, pg 52).

Figure 3.1: Roller Mill



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; eg LM46.4 is 4.6M in diameter with 4 roller modules.

Primary roller mill controls are:
- Mill drive power or mill differential pressure which control mill feed rate
- inlet gas temperature (up to 600°C)
- outlet gas temperature
- outlet gas flow

It should be noted that the material cycle time of a roller mill is usually less than a minute against several minutes for a ball mill. Thus, control response time should be appropriately faster. In addition, the roller mill feed system must be capable of responding to low material flow within no more than 45 seconds or excessive vibration will cause mill shut down.

Optimum fineness of kiln feed must be determined empirically and should be as coarse as the kiln will tolerate. Typically, raw meal should be ground to 15% +170# (88µ) and 1.5 – 2.5% +50# (300µ). A narrow particle size distribution is optimal because fines tend to increase dust loss by entrainment in exhaust gas, while coarse particles are harder to react in the kiln and result in high free-lime, and/or excessive fuel consumption.

Specific power consumption depends upon material hardness and mill efficiency. For ball mills, the range is approximately 10kWh/t (mill drive only) for soft, chalky limestone to 25kWh/t for hard materials. For roller mills the range may be 4.5 – 8.5kWh/T and, although ID fan power is increased, system power is generally some 30% lower than for ball mills.

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

Raw mills are monitored by:
  Production rate, tonnes/hour
  Operating hours
  Involuntary downtime hours
  kWh/tonne (mill motor)
  % of Connected Power (relative to mill motor rating)
  Product fineness, -170# & -50#
  Feed moisture, %
  Product moisture, %
  Limestone, %
  Clay/shale, %
  Additives, %

Additional operating parameters required periodically include:
  Circulating load, %
  Ball usage, g/t (for ball mills)
  Chemical analysis of +170# fraction

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

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

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

3.2 Blending

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

Turbulent mixing (airmerge)		1.5 - 2.5 kWh/t
Controlled flow:	inverted cone	0.25 - 0.50
	multi outlet	0.10 - 0.13
(Bartholomew, ICR, 9/1995, pg 66)		

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

Modern blending silos are generally of continuous, controlled flow type with each silo having capacity of more than 24 hours kiln feed and yielding a blending ratio of 3-5; older silos are more like 2-3. Note that a given silo will show a lower blending efficiency if the feed is itself consistent. 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, pg 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 (1-2 years) internal inspections and maintenance are necessary. As raw meal is liable to solidify if left inactive (during a kiln shut-down 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 minimize 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 minimize the chemical disturbance due to dust 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 minimizes re-entrainment of the fines.

Kiln feed is monitored by:
  Chemical analysis on 4- or 8-hourly grab samples to determine statistical variation (see Section 6.5). Analysis is conventionally for major oxides with variation monitored statistically in terms of C3S or LSF.

Kiln feed should typically have an estimated standard deviation for grab samples of less than 3% C3S or 1.2% LSF (Halbleib; ZKG; 10/2003, pg 44). It should be born 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.

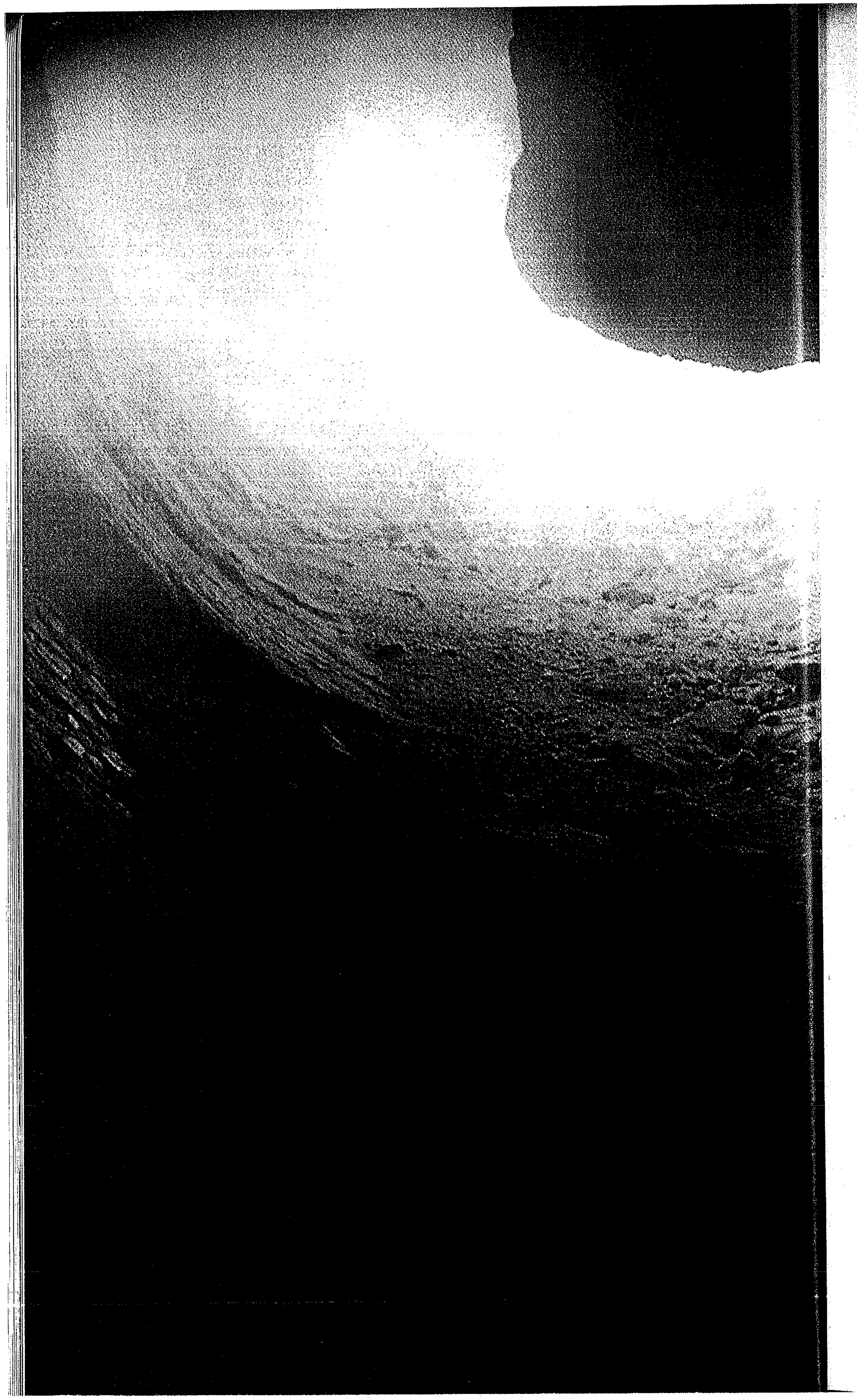
Kiln feed is normally conveyed by bucket elevator to the top of the preheater to minimize power consumption. If this conveying is effected pneumatically, de-aeration is desirable before injection as the entraining air otherwise adds to the kiln ID fan load and may reduce kiln capacity.

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

Kiln feed = Clinker + LoI + Bypass dust + Downcomer dust - Coal ash

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

* * *



4 Burning and Cooling

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

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

100°C	Evaporation of free water
> 500°	Evolution of combined water
> 900°	$CaCO_3 \rightarrow CaO + CO_2$ (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 CaO

Spahn (ZKG; 7/2004, pg 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 and have been virtually the only type of cement kiln installed over the past 30 years. The first units were 4-stage preheaters. Relative to the previous technology of long wet and dry kilns (Sec 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 significant 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 1 second.

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 4 stages, it is believed that counting in order of gas flow from the bottom would allow more meaningful correlation from kiln to kiln.

Precalcination is the addition of a second firing point and combustion chamber at the base of the preheater with separate ducting of hot air from the clinker cooler through a 'tertiary' air duct. This system allows an approximate doubling of production from a given rotary kiln size.

Single string (precalcining) preheaters are available up to about 6,000t/day (with up to 10Mϕ cyclones) and larger kilns now have two- and even three- strings allowing unit capacities in excess of 10,000t/day. Heat recovery has also been improved, where heat is not required for drying raw materials, by using 5-and 6-stages of cyclones, and redesign of cyclone vessels has allowed pressure drop to be reduced without loss of efficiency (Hose & Bauer; ICR; 9/1993, pg 55). Exit gas temperatures, static pressures, and specific fuel consumptions for modern precalciner kilns are typically:

6-stage	260°	550mm H_2O	750kcal/kg (NCV)
5-stage	300°	450mm	775
4-stage	350°	350mm	800

Temperatures are 20-30° lower without precalciners and older systems are usually 20-30° higher than the above. Early 4-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 (Figure 4.1). Large modern kilns are designed to 700kcal/kg and below.

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

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

NSP (New Suspension Preheater) – Precalciner technology which was developed in Japan in the early 1970s.

AT (Air Through) – Precalciner or riser firing using combustion air drawn through the kiln.

AS (Air Separate) – Precalciner using tertiary air.

ILC (In-Line Calciner) – AS precalciner in which kiln exhaust and tertiary air are premixed before entering the calciner vessel.

SLC (Separate Line Calciner) – AS precalciner vessel in parallel with the kiln riser and fed only with gas from the tertiary duct.

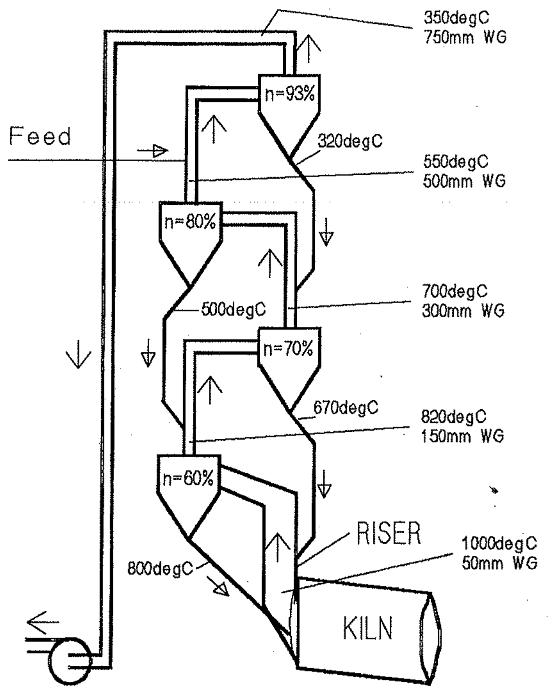
SF (Suspension Preheater with Flash Furnace) – IHI precalciner design which is an AS/ILC system.

RSP (Reinforced Suspension Preheater) – Onoda design of precalciner vessel which is an AS/SLC system.

MFC (Mitsubishi Fluidised-Bed Calciner)

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

Figure 4.1: Cyclone Preheater Typical Temperature and Pressure Profile and Cyclone Efficiencies



4.1 Kiln Burning

Kiln operation is monitored by:

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

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

Other kiln performance factors include:

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

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

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

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

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

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

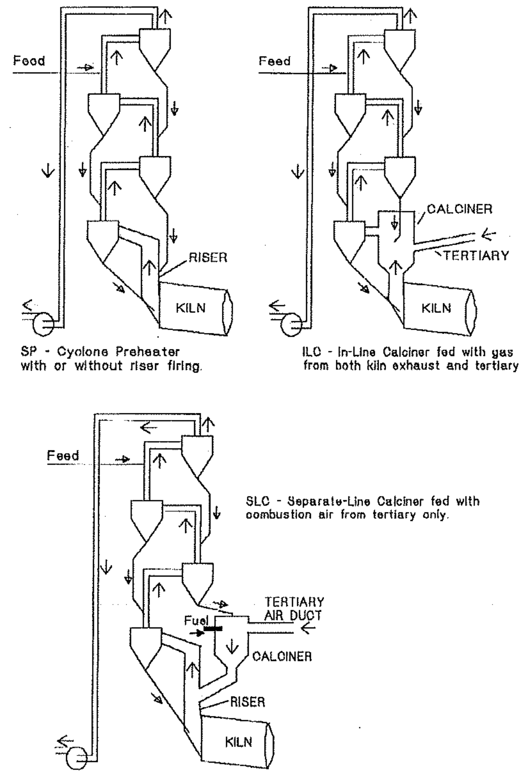
A convenient supplement for free-lime measurement is the more rapid determination of litre-weight. This involves screening a sample of clinker from the cooler discharge to approximately +5/-12mm and weighing a standard 1 litre volume. Litre-weight is typically 1100-1300g/L (varying inversely with free-lime)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

Secondary air temperature should be as high as possible in order to recover the maximum heat; usually 800-1000°C. Maximising secondary air temperature involves optimising clinker bed depth and cooling air distribution to the recouperating zone. A common misconception is that increasing the air flow to the hot end of the cooler will cool the clinker rapidly and recover more useful heat. In fact, contact time between cooling air and hot clinker is reduced

with consequent lowering of secondary air temperature. Good clinker granulation is essential as fine, sandy clinker results in uneven air distribution and, commonly, a red river of hot clinker extending well down the cooler. Good granulation requires a sufficient liquid phase, typically 23-25%, with high surface tension (Timaschev; Proc International Congress on the Chemistry of Cement; Paris, 1980). High alumina ratio and low alkali increase surface tension of the melt while a low burning zone temperature will result in increased liquid viscosity and small crystal size (Sec 7.3). Secondary air temperature has been difficult to measure unless there is a hot-gas take off from the hood for tertiary or coal mill air. Recently, however, an acoustic pyrometer has been successfully introduced to the cement industry; this is a low cost and low maintenance instrument which integrates the temperature across the hood and is not affected by entrained dust concentration (ICR; 6/2002, pg 49). The availability of reliable secondary air temperature offers potential for cooler grate speed control to be, at least partly, directed to maintaining constant secondary air temperature rather than the less important function of maximising clinker cooling. Fluctuating secondary air temperature will inevitably cause cycling of kiln operation.

Precalciner kilns are designed to maximize the heat input to the calciner and, typically, 60% of fuel is fed to the calciner while 40% is burned in the kiln. This serves to minimize the size of the rotary kiln and its heat loading; it does not reduce specific fuel consumption. It has been widely found that preheater kilns without precalciner vessels can also benefit from feeding 10-20% of total fuel to the kiln riser. Kiln operation is noticeably more stable and brick life is extended. This is also a useful means of consuming low grade fuels or waste

Figure 4.2: Major Configurations of Cyclone Preheater Kilns



SP - Cyclone Preheater with or without riser firing.

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

SLC - Separate-Line Calciner fed with combustion air from tertiary only.

It cannot be over-emphasized that kiln stability, fuel efficiency, finish grinding power consumption, and cement quality all depend greatly upon the provision of kiln feed and fuel with minimal variation both of chemistry and feed rate. Healthy scepticism should be nurtured towards both instrument signals and manually reported data. Particular areas for mistrust are:

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

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

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

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

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

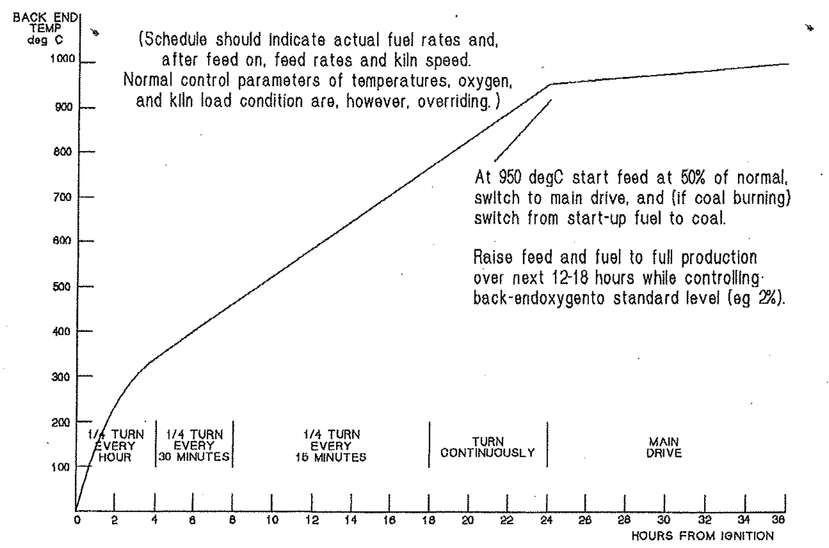
4.4 Kiln Start-up and Shut-down

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

Warm-up follows agreement by production and maintenance management that all work is completed, that all tools and materials have been removed, and that all doors are closed. Work may, with discretion, continue in the cooler during warm-up but no workers should remain in the cooler at the time of ignition. Commonly, warm-up from cold takes 24 hours from ignition to feed-on, but may be increased if extensive refractory work requires curing. A typical chart is shown (Figure 4.3) indicating the desired rate of increase in back-end temperature (this may also be set out in terms of fuel rate), the kiln turning program, the introduction of feed (usually 50% of full rate), and the increase of fuel, speed and feed to normal operation which should take another 8 hours from feed-on. For PC kilns, fuel is supplied to the calciner at the same time as, or soon after, feed-on. ID fan should be operated to approximately 10% O_2 at the back of the kiln to feed-on whereupon the normal O_2 target is adopted.

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

Figure 4.3: Typical Kiln Warm-up Schedule



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

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

Shut-down may be either:

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

Suggested inching is as follows:

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

If the shut down is for less than 24 hours and does not involve entering the kiln or preheater, then heat should be retained either by stopping the ID fan immediately and shutting the preheater dampers after 2 hours, or (if there are no dampers) shutting down the fan after 2 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 precaliners) is as follows:

Discharge - 1 D	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 minimize 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 10cms 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 meter 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, pg 59). Kiln shells should also be inspected visually, particularly under tires 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 born 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, pg 48 & 4/2000, pg 76.
Refractory installation – Karlgren; WC; 12/1999, pg 42.
Benchmarking refractory performance – Shepherd; ICR; 12/2000, pg 43.
Lining for critical areas – Kassau; ICR; 5/2001, pg 115.
Mechanical & thermochemical stress analysis – Klischat & Tabbert; ICR; 9/1998, pg 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, pg 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 meter of kiln length,

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

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; Proc 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 minimized; 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 tire because of the tendency to initiate shell cracks (see Figure 4.4). 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 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, pg 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.

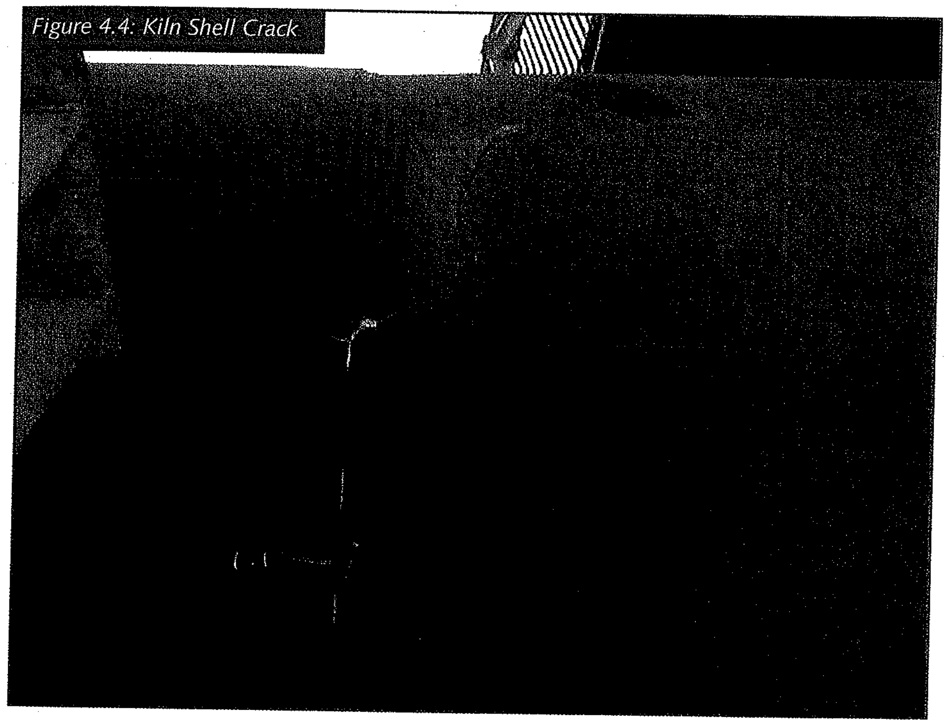


Figure 4.4: Kiln Shell Crack

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 NOx 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 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 recognized 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 NOx reduction (Haspel; ICR; 1/2002, pg 63).

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 5-15% of the kiln exhaust gas for chloride control and 10-70% for other volatiles. 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 minimize 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 new 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, pg 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, pg 586). Automated control of the bypass has been proposed based upon sampling and analysis of the hot meal (Triebel et al; ZKG; 10/1994, pg E272). Chloride circulation should be limited to 5-10g/Kg clinker depending upon the degree of precalcination (Farag & Abbas; ZKG; 1/1995, pg 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, pg 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, pg 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, pg 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 shut-down 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, pg 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	*Combustion gas NM^3/million Cal*	*Total exhaust gas NM^3/tonne clinker*
Coal	2250°C	1.23	1360
Oil	2350°C	1.31	1420
Gas	2400°C	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 minimize 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# (75µ) 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/sec. A more precise determination of optimum fineness according to coal type has been described by Seidel (ZKG; 1/1995, pg 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 atomized 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/sec (injection will normally be limited by sonic velocity – 430M/sec 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, pg 59) review fossil fuel reserves and the future of alternative fuels.

Petroleum coke 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 pet-coke: '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 pet-coke 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, pg 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, pg 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. Tires are potentially attractive though shredding or pyrolisis eliminates much of the cost benefit while fuels added discontinuously, such as whole tires 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, pg 8) comprising some 650,000T/Y world-wide. Their use hitherto has been constrained by a typically neurotic fear of fluoride and cyanide residues. A more significant limit to use for many plants will be the sodium content. Typical analysis is:

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

Other wastes include:

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

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

Gaseous waste:
- landfill gas
- pyrolysis gas

Kilns employing alternative fuels have detailed specifications to prevent operating or environmental problems, and each shipment is sampled and checked before unloading to ensure compliance. Typically, heat value should not be less than 4000kcal/kg, chloride is limited to 1% and most plants decline fuels with PCB content exceeding 50ppm. Waste fuels may be burned in the kiln, the riser, or the precalciner. Solid waste may also be pyrolysed before injection (Sect 14.4, Fuel Supply).

4.11 Coal Firing

Coal firing for cement kilns falls into two basic systems (Figure 4.4). 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, page 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 (ie 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/sec 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/sec to avoid dust settlement (Recommended Guidelines for Coal System Safety; PCA; May 1983).

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

Hot air for drying coal can come from cooler exhaust (normal air) or preheater exhaust (low oxygen). The inlet temperature to the mill is controlled to maintain the outlet temperature as described above and de-dusted in a cyclone. Tempering to about 370°C is effected by bleeding in cold air between cooler and cyclone. If preheater exhaust is used it will typically be at 300-350°C with 5% O_2 and 6% moisture; the temperature and moisture must be considered in the system design. Although most mill fires occur on start-up or just after shut down, the low oxygen atmosphere does reduce the risk of a mill fire during normal operation. In the United States all coal mill systems are designed in accordance with NFPA Standard 8503 which requires the equipment to withstand pressures of 2.5 times the absolute working pressure; this equates to 3.5kg/cm^2. For conveying systems, this requirement may be proportionately higher because the transfer system operating pressure can be 1.5kg/cm^2 (20psi).

Direct coal firing involves a single pipe burner through which the mill carrying air together with entrained coal are injected at a tip velocity of approximately 80M/sec. Tip velocity must always be substantially greater than the flame propagation velocity which may be up to 25M/sec. Flames produced by coal nozzle velocities in excess of 80M/sec are susceptible to severe instabilities. The pipe is usually narrowed near the tip to minimize parasitic pipe losses and to convert the flow into the desired static pressure. For indirect firing, multi-channel burners of various designs are employed. One annulus is used for conveying pulverized fuel from the mill and one or more separate streams are used to supply primary air for controlling the flame (Figure 4.5). Typical specifications used by vendors for burners with indirect firing systems are as follows:

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

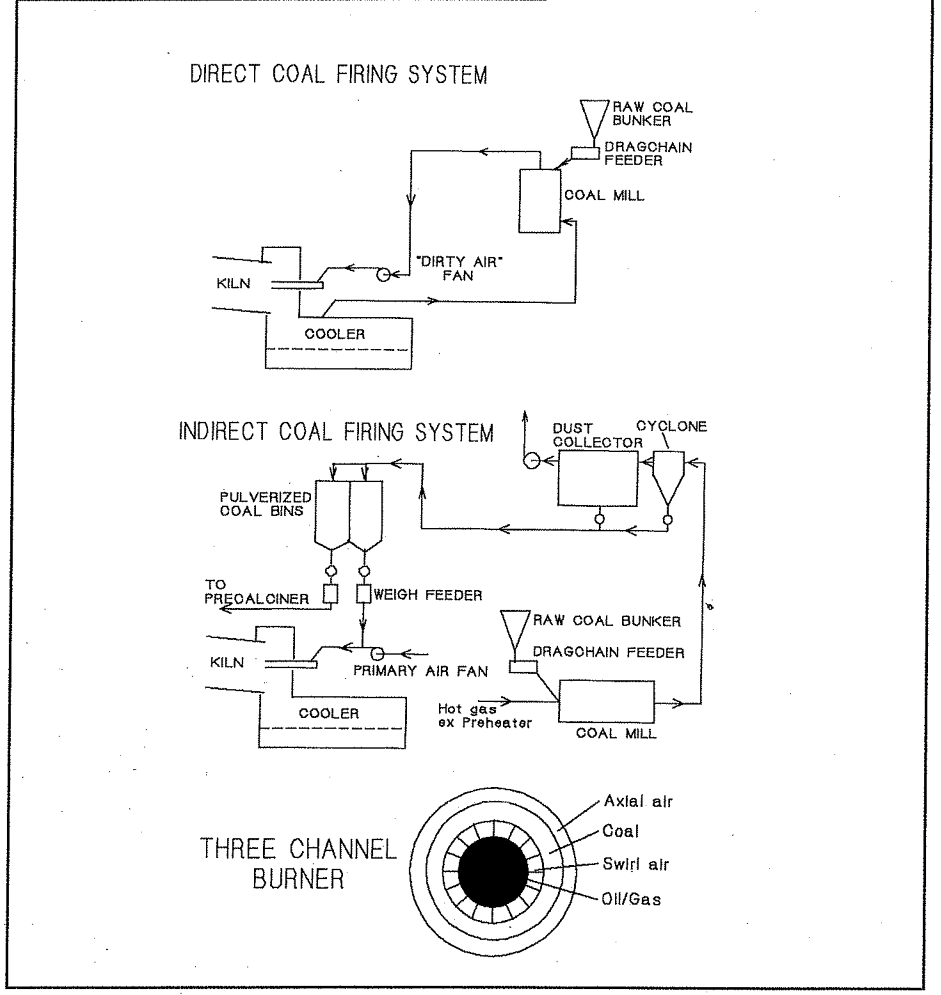
Coal firing almost inevitably involves a normal operating condition where pulverized coal is in contact with air before reaching the burner; this is a condition that should, theoretically, be avoided due to its inherent risk of fire and explosion. The design of a coal firing system is, therefore, critically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

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

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

Coal dust is liable to spontaneous ignition which increases in risk with thickness of dust layer

Figure 4.5: Coal Firing System



and with the presence of easily oxidized contaminants such as pyrites (more than 2%). A smouldering fire can become explosive if disturbed.

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

4.12 Clinker Cooling

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

By far the preponderant type is the **Reciprocating Grate Cooler** first introduced by Fuller Company. This comprises a series of under-grate compartments with separate fans which allow individual control of pressure and volume of injected cooling air. There may be up to 8 under-grate compartments and two or three separately driven grate sections. Fuller coolers are defined by width and length of each grate in series, eg 1035S/1242 has a first grate 10ft wide x 35ft long (S=sloped) and a second grate 12ft x 42ft. Under-grate pressures are approximately 600mm in the first compartment reducing progressively to about 200mm in the last compartment. Modern grate cooler performance is reviewed by Wallis (ZKG; 7/2004, pg 40) and by Lorimer (ICR; 5/2002, pg 67).

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

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

On traditional grate coolers, some 2-2.5NM^3 of cooling air per kg clinker are required to achieve clinker discharge at below 100°C. This compares to approximately 0.9NM^3/kg required for combustion so that the balance is exhausted taking with it more than 100kcal/kg clinker. Such coolers were typically rated at 35-40t/M^2/day and operated with clinker bed depths of 200-400mm. Air beam designs are now available from all cooler manufacturers which, with individual variation, provide for the direct ducting of cooling air to hollow grate support beams whence the air is directed into the clinker bed more efficiently than is possible from the former under-grate plenum and grate array. This type of cooler allows 1.5-2NM^3/kg to be used with a rating of 45-55t/M^2/day and bed depth of 800mm. The grates used with air beams, 'controlled flow grates' (CFG) are of a higher resistance design with air passing horizontally through slots into recesses in the grate surface; this largely eliminates fall through of clinker fines and renders the air flow less dependent upon bed resistance. Similar grate designs are also available for the non-recouperating zone but with lower resistance as required by plenum aeration; these are referred to as 'reduced fall through' (RFT) plates (Roy & Brugan; ICR; 2/1995, pg 47).

Numerous refinements include a pendulum frame for the moving grates which is claimed to minimize drive maintenance, a static inclined arrangement of the first 6-9 air beams which serves to distribute more evenly the clinker entering the cooler, and a hydraulically operated heat shield at the end of the secondary (or tertiary) air recouperation zone which reduces high-grade heat loss to exhaust (von Wedel; ICR; 4/1995, pg 75). The static grate section has been found particularly effective and, typically, costs $0.5-1.0 million installed. Note that coarse clinker predominates below the kiln-down, or unloaded, side of the kiln and fines below the kiln-up, or loaded, side.

A recent development of cooler design by FLS-Fuller is the **Cross Bar Cooler** (ZKG; 11/1999, pg608). This consists totally of static grates with clinker transport effected by reciprocating pusher bars above the grate surface. This cooler also incorporates an ingenious flow regulator on each grate which maintains a constant air flow through the clinker bed regardless of bed porosity. Wear of the reciprocating bars is significant but is predictable; cooler efficiency is excellent and much less subject to deterioration over an operating cycle than is the case with conventional grate coolers. More recently, Polysius have introduced the modular Polytrack, with a static cooler floor and above-floor transport (Morgenroth et al; ICR; 5/2004, pg 58) and Claudius Peters the η-Cooler (GCL; 1/2004, pg 15). Progressively, too, designs have become more conservative with specific air flows of 1.8-2.2M^2/kg clinker and 40-50t/D/M^2 (Gasser & Hasler; CI; 6/2003, pg 98).

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

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

Visual inspection of the cooler is important; in particular persistent 'red rivers' indicate a problem with segregation of fine and coarse clinker nodules, grate geometry or air distribution, and 'blow through' indicates excessive air flow to a particular compartment. Cooler cameras (Stein; WC; 12/1999) and thermal monitors (Idoux; WC; 12/2002, pg 57) are available.

Acoustic horns have also been suggested as a means to improve cooler heat recovery (Andersch & Kramer; WC; 1/1995, pg 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, pg 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, page 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, pg 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 maximize 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 tires and in cantilevered overhanging sections the long axis tends to be oriented horizontally, while between the tires 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 tire 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 tire 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 tire 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, pg 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 characterized by tire locations that balance the load uphill and downhill on mid-kiln tires and yield shell overhangs between one and two kiln diameters from feed and discharge end tires. The shell thickness is selected to maintain calculated material stress levels well within the steel's capabilities and manufacturers utilize historically proven stress limits that accommodate variation from design assumptions.

Attention should 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 tire creep. Creep is the relative movement of kiln shell and tire. 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 tire is known as the tire course and the thinner plates uphill and downhill are known as flanking plate. The plate between tire sections is even thinner than the flanking plate. The most common location for shell cracking is at the transition between the flanking plate and the thin shell plate that spans between piers. Failures generally occur at the toe of the weld joint on the thin plate side. It is often acceptable simply to mark the extent of these cracks while continuing to operate until an opportune shutdown. Drilling a 'crack stopper' hole at the end of a crack is a common practice but it will generally not be effective unless it is at least 25mm in diameter. The crack should be removed by burning or gouging from the outside a groove with sides making an angle of about 60°. Magnetic particle or ultrasonic inspection should be used to determine that the entire crack has been removed. Welding should be performed with appropriate filler metals laid down in straight beads. Each bead should overlap the one below it by about half the bead width to provide heat for relief of weld shrinkage stresses in the underlying bead. Beads should be laid in straight parallel lines; weaving of the beads should not be allowed as this technique usually results in poor quality and high residual stresses (the width of the weld pool should be no more than 3 times the filler metal diameter). The layers of weld bead should be built up until the toe of each weld bead lies on the line connecting the lips of the groove; ie the weld should form a tapered surface from the thicker plate to the thinner. There is no need to stress-relieve after welding but the shell plates must be maintained above about 5°C during the entire process. At the first opportunity, the repair should be accessed from the inside and the full length again burned or gouged to remove the root pass and re-weld as described for the initial repair.

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

Shell diameter in metres / 10

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

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

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

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

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

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

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

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

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

The lowest risk operating state then, is the one that allows the kiln to accommodate as much transient shell misalignments as possible. To achieve this state the kiln shell's theoretical axis must be measured while the kiln is operating. This is actually done somewhat indirectly by measuring the position of the riding rings, assuming the rings and shell are perfectly round, and calculating the location of the kiln axis at each tyre. Then imaginary straight lines are drawn between each tyre axis to determine if the tyres are high, low, left, or right of a theoretical straight line representing the kiln shell's axis. If the kiln shell is relatively straight, the location of each tyre's measured axis should lie within about 3mm left or right of the theoretical kiln axis when viewed from above the kiln. The vertical locations of the tyre axes relative to the theoretical straight line kiln axis may vary significantly from pier to pier among kilns. Generally, on a three support kiln, the middle support rollers bear the highest load and often must be set low relative to the theoretical kiln axis in order to avoid excessive bearing pressures. The most effective means to make the final alignment adjustments is to utilize 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, pg 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 tire 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 tires 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, pg 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 & load-out

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

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

■■■■■

5 Cement Milling

5.1 Clinker Storage

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

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

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

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

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

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

5.2 Cement Milling

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

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

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

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

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

Ball mills are cylindrical shells designated by their inside diameter, length, and connected drive motor power. The shell is protected by steel liners and usually divided by a diaphragm into two compartments of which the first is typically 30-33% of the overall length. High-chrome alloys are now almost exclusively used for balls and liners and ball usage should not exceed 50g/tonne cement. The first compartment is primarily to break feed clinker nodules which may be up to 30mm; lifting liners and balls from 50mm up to 90mm are employed to effect impact. Clinker passing into the second compartment is typically less than 2-3mm and second compartment balls range 15 to 50mm (Fuhrmann; ZKG; 9/1991, pg E199). While coarse grinding benefits from a range of ball sizes, greater efficiency of fine grinding, which involves mainly attrition, may be achieved with single sized small (20-25mm) balls. If a range of ball sizes is used in the second compartment, classifying liners are employed to retain the larger balls close to the diaphragm and the small balls at the discharge. If a single ball size is used, simple wave liners are appropriate. Note that the discharge screen slots must be at least 3mm wider than the diaphragm slots and 5mm less than the smallest ball. Charges should be dumped and sorted, preferably once each year, to maintain the optimum size profile and to remove tramp metal.

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

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

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

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

Circulating load is typically 100-200% for raw milling, 200-300% for finish mills with mechanical separators, and 150-200% with high efficiency separators. Specific power consumption usually decreases with lower charge loading but so does production rate. Since other circuit power is essentially fixed, the design charge loading and rated mill power should normally be maintained.

A useful test of mill condition is to shut the mill down on load after steady state has been maintained for 8-10 hours. The fan should be stopped immediately to avoid sweeping fines out of the mill. The mill needs to be positioned so that the doors can be opened, preferably using an inching drive. On entering the mill following appropriate lock-out, it should be observed that the top layer of balls in the first compartment half floats in feed; exposed balls require more smaller balls to fill voids, or the blanking of an outer annulus of the diaphragm, or the closing of scoops on an adjustable diaphragm (Gudat & Albers; ZKG; 3/1992, pg 73). In the second compartment, the balls should be covered to a depth of 20-25mm. Samples should be taken every meter through both compartments starting and finishing about 20cm from the end walls and diaphragm. At each interval, 1-2kg samples should be taken from just below the surface along a line perpendicular to the mill axis. The samples should then be screened:

First compartment:	½", ¼", 4#, 8#, 16#, 30#, 50#, 100#
Second compartment	8#, 16#, 30#, 50#, 100#, 170#

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

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

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

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

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

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

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

The energy efficiency of ball mills is very low, particularly for coarse grinding. In recent years, various circuits have been introduced incorporating hammer mills, roll-presses, and roller mills for pre-grinding ahead of the ball mill. An impact crusher in closed circuit can yield product with 100% -3mm; this approximates to the size of material passing from first to second compartments of a 2-compartment mill so that the large ball charge can be replaced by ca 20-25mm balls with a potential mill capacity increase of 20% and a combined system specific power reduction of 5-10% (Stoiber et al; ZKG; 3/1994, pg E94).

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

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, pg 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 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, pg 46). A further development, the VSK separator combines the functions of the VS separator with an integral high efficiency separator (Suessegger; WC; 2/2003, pg 84).

For efficient operation of a roll press:

- The feed size should be limited to approximately 50mm (not more than 2 x 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/sec)
D = product density [clinker = 2.5; raw mats = 2.3; slag = 2.45]
(von Seebach; Proc IEEE; 1995)

Figure 5.1: Roll Press Circuits

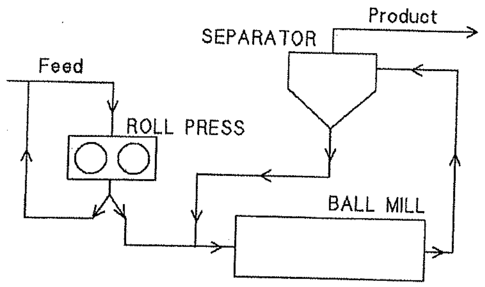


Figure 5.1.1: Pregrinding with Slab Recirculation

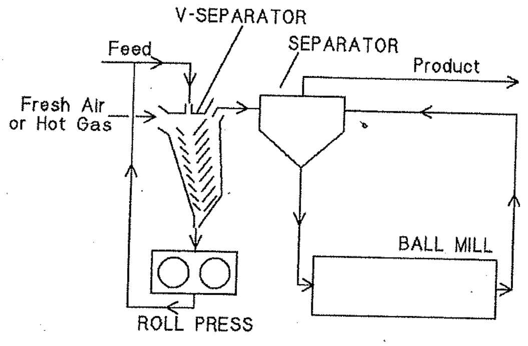


Figure 5.1.2. Semifinish Grinding 'S'

FCB have addressed the reconciliation of efficient compression grinding with operating simplicity, and have developed 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, pg 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 summarized 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, pg 17) has developed an empirical formula to correlate grindability to alite (C_3S) and belite (C_2S) crystal sizes determined by microscopy:

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

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:

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

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, pg 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, pg 91). The optimum addition rate should be determined which balances enhanced grinding against power savings to minimize 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, ICR; 7/2003, pg 64). 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, grams/tonne
- Ball usage, grams/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 1-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 minimized. 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, pg 384 & 9/1995, pg 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 per 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, pg 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 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.

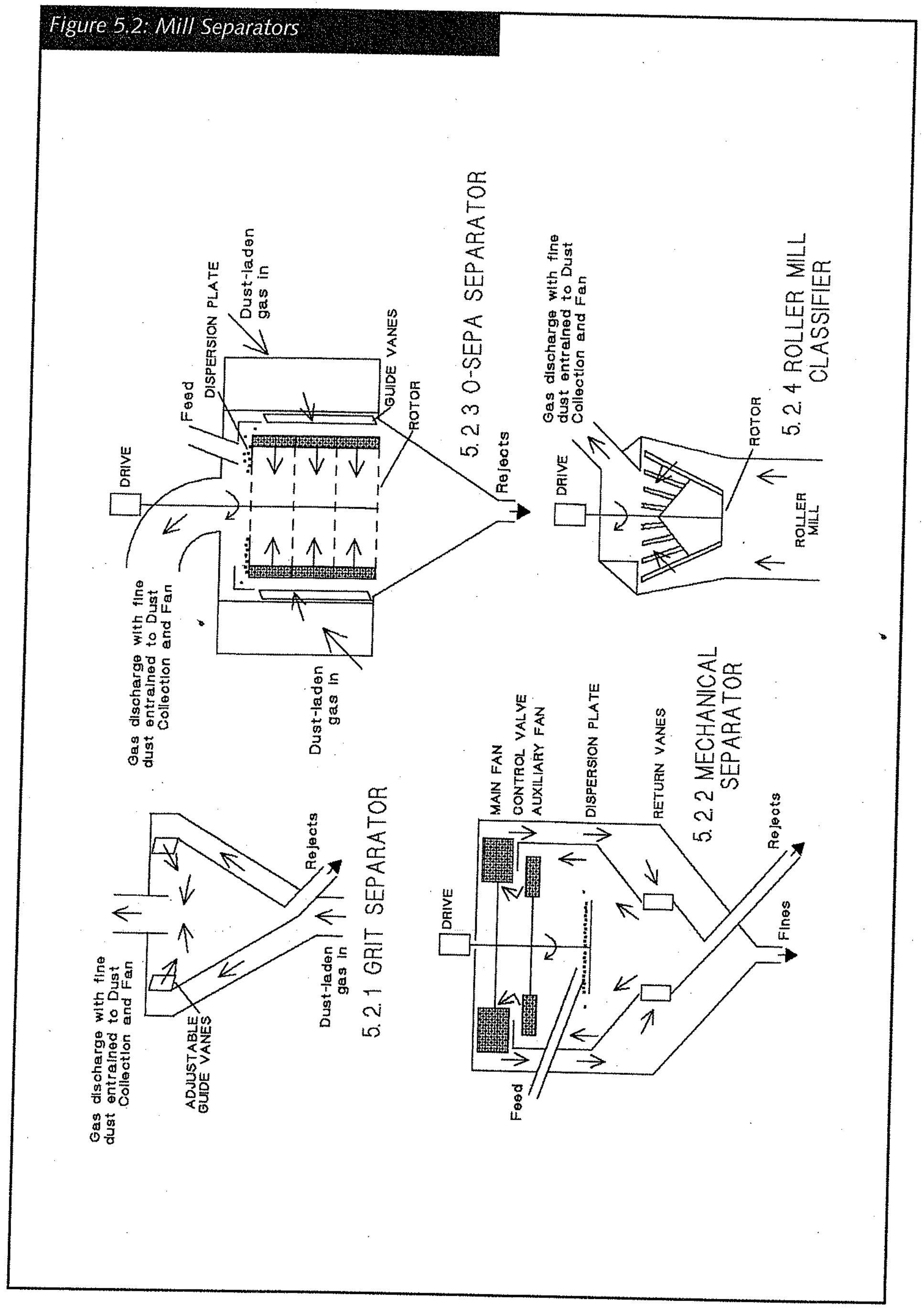
The Rosin-Rammler distribution graphs cumulative particle size fractions against particle size so that a narrow distribution results in a higher slope.

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, pg 80). The situation is rendered particularly interesting when increased performance is required from existing equipment and new units are combined with old. The only generalization which can be made is an obvious caution that balance and control are essential.

Figure 5.2: Mill Separators



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 2-compartment mill with L:D ratio of 3.2 is limited to about 1M/sec 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 1-day strength testing before shipment. With cement storage typically costing $150-300 per tonne, there is strong pressure to minimize both total storage capacity and the number of silos for a given capacity. Dagnan (Proc IEEE Cement Industry Conference; Salt Lake City, May 1986, pg 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, pg 104). After the mill, cement can be cooled using a dedicated cement cooler (Kochmann & Ranze; ZKG; 10/1997, pg 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, pg 65). Build-up and the dubious performance of many silo level detectors may lead to erroneous inventories. Erratic flow of cement is described by Dudley (WC; 6/2002, pg 69) while Graham (ICR; 2/1997, pg 51) describes a low-frequency acoustic technology for flow enhancement and silo cleaning. Manual measurement should be made at a point 2/3 of the radius out from the centre and measurements should be reconciled with metered filling and discharge of silos to detect anomalies.

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

Dome storage has also presented a problem of inventory determination due to the large surface area wide and erratic surface geometry. Possible methods may be to use a laser surveying device which constructs a grid across the surface from the feed point at the top of the dome, and radar devices (WC; 2/2004, pg 91).

5.6 Cement Dispatch

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

23-25tonne road truck = $8/t/day + [$0.018/t x kM(one way)]
2000t rail shipment = $0.50/t + [$0.02/t x kM]
35,000t sea shipment = $3.00/t + [$0.0015/t x kM]

Thus, a 320kM (200 mile) haul would cost approximately $14/t by road and $6.90/t by rail. This, of course, may not reflect door-to-door cost where road haulage normally offers an advantage. Road transport also is usually more flexible than rail for small, irregular, or short-notice shipments.

By comparison, 6000kM trans-Atlantic sea freight would cost about $12.00 (subject to the cyclical variation of shipping costs). It is obvious, therefore, that coastal areas are very exposed to low cost imports unless protected by tariffs. And certain routes can be served by ships on back-haul at even lower rates. Similarly it can be seen that plants with deep water docks have almost unlimited market potential. Polansky (ICR; 12/1995; pg 47) reviews the operation and economics of bulk carriers of greater than 25,000DWT and Thomas (WC; 6/2004, pg 15) assesses future trends in ocean shipping. Economic bulk distribution requires minimal demurrage costs for filling and discharging bulk vessels. Such costs, if incurred, should periodically be compared to the cost of upgrading loading/unloading equipment.

Sacking adds approximately $4-6/t to production cost and can, of course, increase shipment costs due to the additional handling involved. Sacks are most efficiently handled either on pallets or shrink-wrapped (Remmert; ZKG; 12/2000, pg 690).

Cement sacks are typically 94Lb, 50kg or 40kg net weight. The European Commission, based upon their extensive experience of handling cement sacks, first ordained that European labourers could not healthily and safely lift bags of more than 25kg (Reitmeyer; WC Bulk Materials Handling Review; 1993, page 32). Our masters subsequently discovered that workers were shouldering two bags at a time and, ever active in protecting us from ourselves, they now plan to increase the official bag weight to 35kg (Frenzel; ZKG; 11/2003, pg 52). Packer manufacturers are doubtless delighted while customers must remember to order sacked cement in multiples of 1.015t.

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

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

Bulk loading of road or rail trucks is normally about 400t/h, and of ships and barges considerably faster. 25T road trucks can be loaded at 6-minute intervals from a single loading spout with appropriate equipment and procedures. The filling spout is telescopic and has a conically tapered nozzle to seal against the tank opening. The filler comprises a central cement delivery pipe surrounded by an outer suction tube which vents at least 1000M^3/h and is ducted to a dust collector. Load weight is determined either by tear and load weights of the truck (preferably on dedicated truck scales under each loading point), or by loading from a hopper on load cells which is itself weighed before and after loading and which can be programmed for a specific delivery (Brix & Pauer; WC; 3/1995, pg 106). To prevent spillage, the loading spout should be fitted with a level sensor and cut-off switch. The design of modern road tankers is reviewed in ICR; 4/2003, pg 83.

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

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

5.7 Customer Service

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

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

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

■ ■ ■ ■ ■

6 Quality Control

The plant quality department is responsible for:

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

6.1 Sampling

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

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

MRW, g = $18\,\lambda f d^3 / s^2_{FE}$
where λ = density, g/cm^3 (see Section B7.2)
f = shape factor (dimensionless; cube – 1.0,
sphere – 0.523, rock – 0.5,
coal – 0.45, quartz – 0.47, mica – 0.1)
d = maximum particle size, cm (defined as the
square mesh opening which retains 5%)
s^2_{FE} = particle heterogeneity factor
(practical limits are +/-5%)

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

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

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

(ASTM C75; Standard Practice for Sampling Aggregates)

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

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

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

6.2 Chemical Analysis

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

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

X-radiation only penetrates a few microns into the sample surface and powder samples must be ground to –20m before being pressed into disks for analysis. Some minerals are prone to render the surface non-representative; examples in raw materials are quartz which may be present in large particles and platy mica minerals. Such materials may require fusion to avoid analytical bias; Philips offers the Soled Perl'X system, and Claisse Fluxer (WC; 4/2003, pg 73). Sample fusion is usually carried out near 1000°C on a mixture of 1 part sample (on ignited basis) to 5-10 parts flux (lithium meta- or tetraborate or a mixture). The melt is quickly quenched into a glass disk. Fused samples are essential for accurate analysis of raw materials while pressed powder pellets are appropriate for raw mix, hot meal to kiln, clinker, cement, and kiln dust. Fusion of hot meal, kiln dust and bypass dust may lead to partial volatilisation of Cl and SO_3. For volatile mass balance studies, Cl is better analysed by $AgNO_3$ titration (ASTM C114) and sulphur by high temperature combustion (ASTM D4239). Low levels of chloride, fluoride, sulphite (in some synthetic gypsum), and sulphate in raw materials can also be analysed as anions in solution by ion chromatography (IC). These ions are separated by ion-exchange and detected sequentially by conductivity cell. Trace metals in waste fuels, waste raw materials, and kiln dust can be analysed in solution by induction-coupled-plasma (ICP) emission spectroscopy or atomic absorption (AA) spectroscopy. A corporate central laboratory or a commercial testing laboratory often provides such analytical services.

A cement plant making blended cement with slag is required to perform separate determinations of sulphur in sulfide and sulphate forms (C-595 and C-114).

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

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

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

Neutron activation using an isotope source of neutrons results in characteristic secondary gamma radiation. Gamma radiation being of much higher energy than x-radiation (0.5-5mev vs 1-50kev) is less subject to attenuation so that bulk material (10-100cm thick stream) can be analysed where only the sample surface is sensitive to XRF. The gamma spectrum is resolved by energy dispersion allowing rapid, on-line analysis of process streams without sample preparation. If layered materials are to be measured, as on a raw mill feed belt, it is advisable to pass the mixture over a transfer point to homogenize the stream. Accuracy and precision now approach that of XRF and sampling error is drastically reduced (Macfadyen; WC; 8/2000, pg 70)

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

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

It should be recognized that the maximum acceptable difference between the average of duplicate analytical results and the SRM certificate values (ASTM C114, Table 1) are:

SiO_2	±0.2 %
Al_2O_3	±0.2 %
Fe_2O_3	±0.10%
CaO	±0.3 %
MgO	±0.2 %
SO_3	±0.1 %
LOI	±0.10 %
Na_2O	±0.05%
K_2O	±0.05%
TiO_2	±0.03 %
P_2O_5	±0.03 %
ZnO	±0.03 %
Mn_2O_3	±0.03%

These limits are important in themselves and indicate that major oxides should not be reported with greater precision than 1 place of decimals, though raw data for statistical calculation should retain 2 places. Thus, if a laboratory reports an oxide to 2 or more places, or a cement compound to more than the nearest whole number, this would suggest that they do not understand precision and may well not understand much analytical chemistry either. Chatterjee tabulated standard estimates of error, sample preparation errors, and instrument measurement errors for XRF analysis of twelve oxides in cement related materials (Innovations in Portland Cement Manufacturing, PCA, 2004, Chapter 2.3, pg. 130). Other applications are described in Chapter 8.1 of the PCA book.

ASTM specifies that all analytical methods and instruments used to certify cement should meet the precision and bias criteria listed in Table 1 of C114. One or two additional certified standards are required for verification purpose. Note that calibration standards should not be used to test bias. Re-qualification is required every two years.

Indeed, to meet the requirement of rapid analysis and adjustment for efficient control of a continuous operation, increasing use is being made of automatic systems for sampling, sample conveying and preparation, and for chemical and physical measurements (PCA ibid, Chapters 2.3, 5.1, 5.2: Moller; ZKG; 6/2004, pg 24).

6.3 Evaluation of Clinker Mineralogy, Microstructure, and Clinker Quality

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

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

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

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

6.4 X-Ray Diffraction

X-ray powder diffraction (XRD) offers the ability to identify and quantify chemical compounds such as $CaCO_3$, $CaSO_4$, $CaSO_4.½H_2O$, $CaSO_4.2H_2O$, quartz and clinker phases. Powder diffraction used to be of relatively low precision and was limited to comparative studies in kiln chemistry (Beilmann; ICR; 8/1993, pg 46). It is now increasingly used for rapid determination of free-lime in clinker (Bonvin et al; WC; 8/1994, pg 11) and ASTM C1365 provides a standard XRD method for phase analysis of Portland cement and clinker. Recent advances in total XRD pattern analysis using the Rietveld method have improved speed, precision, and accuracy of quantitative XRD analysis (Fullmannn; ZKG; 5/2003, pg 45) and offers capability of online analysis (Storer & Manias; WC; 2/2003, pg 49). It has been noted that QXRD and microscopic point-count usually give higher alite and lower belite than calculated Bogue potential C_3S and C_2S. Commercial integrated XRF-XRD units have been available for several years (ss Sect 6.8), which allow simultaneous elemental and phase analyses (e.g. free-CaO and C_3S) on the same pressed powder sample (Yellepeddi & Bonvin; WC; 2/2003, pg. 67). XRD is also widely used for raw material mineralogy and for the identification of kiln process materials. Rietveld QXRD and cheap computing power have made real-time analyses of clinker and cement streams possible which, in combination with particle size analysis, facilitates the prediction of cement strengths and setting times (PCA ibid, Chapter 8.2).

6.5 Particle Size Analysis

Commercial automatic analysers for kiln feed and cement are now available based on laser scattering (Sect B4.2). On-line as well as laboratory systems are available and the dry process now predominates (Kalkert; WC; 9/2001, pg 42). On-line analysers are principally justified by shortening the transition time when multiple types of cement are ground on a single mill.

6.6 Thermal Analyses

Differential scanning calorimetry (DSC) is a simple and fast method to quantify $CaSO_4.\frac{1}{2}H_2O$ and $CaSO_4.2H_2O$ in cement, and to determine degree of gypsum dehydration in a cement mill. Although not yet widely practiced, this is an important aspect of cement milling process control. Gypsum or gypsum/anhydrite addition rates, and the degree of gypsum dehydration greatly influence cement performance in setting, strength, and interaction with concrete admixture. DSC is also useful in quantifying needle-shaped syngenite and ettringite ($CaSO_4.K_2SO_4.H_2O$ and $3CaSO_4.C_3A.32H_2O$ respectively). These compounds at low concentrations are responsible for cement dry flowability and pack-set problems. Precipitations of these compounds and of secondary gypsum during cement hydration cause early stiffening of paste, mortar, or concrete. Another useful analytical method is thermo-gravimetric analysis (TGA) which determines losses of sample weight as a function of temperature up to 1600°C. TGA is used to detect pre-hydration of cement in a mill caused by over, or incorrect, spraying of water. It is also used to analyse $Ca(OH)_2$ and $CaCO_3$ in cement or weathered clinker. When trouble-shooting volatile build-ups in a kiln system, TGA of kiln process materials alone, or in combination with differential thermal analysis (DTA), provides insight into the thermal behaviour of volatile compounds such as KCl, various forms of sulphate, carbospurrite ($2C_2S.CaCO_3$), and sulphospurrite ($2C_2S.CaSO_4$) (Proc. 7th Int. Conf. on Therm. Analysis, 1982, Pg. 1303). A few cement plants today are equipped with DSC. Most corporate central laboratories have TGA and DTA, either stand-alone or integrated for simultaneous analyses. Sample sizes for thermal analyses are fairly small, usually from a few mg to tens of mg. Sample preparation is, therefore, very important (PCA ibid Chapter 8.4).

6.7 Microscopy

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

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

The Ono powder method, in gross simplification, involves crushing clinker and screening to -140#/+325# (106-44_). 1.715 refractive index immersion oil is used to mount the sample between a glass slide and cover glass for examination at 200 – 400x magnification, using transmitted polarized light. Measurement is made of alite (C_3S) size, alite birefringence, belite (C_2S) size, and belite colour. Belite colour is observed under un-polarized light corrected with a light-blue filter. Since alite and belite crystals are often coated with brown matrix phases, a hot aqueous solution of sugar and KOH can sometimes be used selectively to dissolve matrix phases to facilitate determination of belite colour and alite birefringence. Measuring birefringence is rather time consuming, and determining size ranges of alite and belite is somewhat subjective. However, meaningful results can be obtained with experience.

Alite size (AS) is a measure of kiln temperature rise between ca 1200-1450°C where C_2S is combining with CaO to form C_3S. Rapid heating is desirable and is indicated by alite of 15-20μ, while slow heating produces alite of 40-60μ or above. Numerical ratings assigned to AS are:

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

Alite birefringence (AB), the difference between the refractive indices of blue and red light, is related to maximum kiln temperature. High temperature up to 1500°C is desirable and is indicated by birefringence of 0.008-0.010 against 0.002-0.005 for a cool burning zone. Numerical ratings assigned to AB are:

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

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

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

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

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

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

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

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

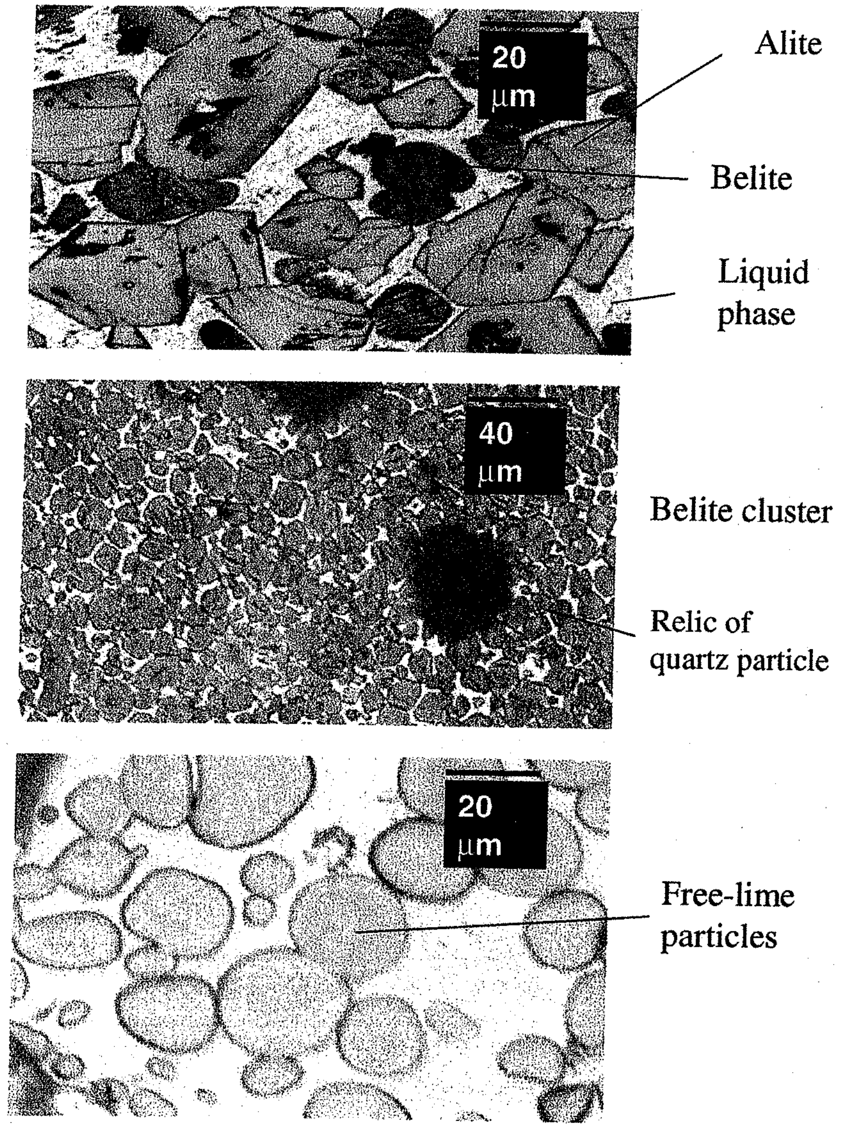
$$\text{28-day strength, psi} = 3672 + 74.4AS + 294AB + 48.5BS + 316BC$$

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

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

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

Figure 6.1: Clinker Micrographs



Parallel techniques are the inspections of polished sections prepared by:

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

- cutting, grinding and polishing a 10-20μ thin section of an epoxy resin mount for examination using transmitted light.

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

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

6.8 The Virtual Cement and Concrete Testing Laboratory - VCCTL

Cement manufacture involves extensive physical testing to assure product performance, and conventional tests incur delays of up to 28 days. Since the 1980s, computational models have been developed using available experimental data of clinker, cement, aggregate, and concrete to predict cement and concrete physical properties by "virtual testing". Virtual testing employs techniques such as particle size analysis, scanning electron microscopy, and x-ray microanalysis of cement, and the results are processed mathematically to simulate cement hydration, to model cement paste and concrete microstructures, and to compute physical properties. Good correlation has been achieved for cement compound concentration (C_3S, C_2S, C_3A, C_4AF) and cement physical properties such as degree of hydration, chemical shrinkage, setting time, and compressive strength. It is claimed that the system is also applicable to blended cements and that, through prediction of diffusion coefficients, concrete durability can also be modelled. (Haecker et al; CI; 3/2003, pg 86).

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

6.9 Isothermal Calorimetry

This method continuously monitors the rate of heat release of hydrating cement for one to two days. An example of a heat release profile is shown in Chapter 10, Cement Hydration. This technique is most useful in studying cement reactivity. When several samples are run simultaneously, it provide a good comparison of reactivities of cements, and allows observation of the variation of reactivity with cement fineness, SO_3 content, the forms of calcium sulphate in cement, clinker composition, and mineral and organic admixtures. ASTM has a draft "Standard Test Method for Monitoring Cement – Admixture Interactions Using Isothermal Calorimetry". The development in using this technique as a standard method for measuring 2-, 3-, and 7-day heats of cement hydration was reported in the Proc. of 11th ICCC, 2003, pg 228.

6.10 Burnability

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

6.11 Grindability

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

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

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

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

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

6.12 Physical Test

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

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

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

6.13 Process Control Analysis

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

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

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

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

Quarry drill cores – Chemical analysis, including S and Cl, every 2M
Blast hole cuttings – Chemical analysis of composite of each blast
Imported raw materials – Chemical analysis of shipment composite
Raw mill product – Chemical analysis of 1- or 2-hour composite; 200# screening of 8-hour composites. Raw mills are increasingly controlled by on-line analysis.
Kiln feed – Chemical analysis of 4- or 8-hour grab samples
Hot meal to kiln – K_2O, Na_2O, SO_3, Cl and LoI (800°C) on 4- or 8-hour grabs
Clinker – SO_3, free-lime (or litre-weight) on 2- or 4-hour grabs. Complete chemical analysis on daily composites.
Cement mill product – Blaine and SO_3 on 2-hour composites, complete chemical, gypsum dehydration, and physical tests on daily composites.
Cement shipped – 7- and 28-day strength testing on 2 or 3 grabs per week per C917.

It is a good practice to retain samples of cement shipped for 1-3 months for further testing in the event of field performance problems.

6.14 Cement Strength

The strength of cement depends upon six principal factors:

- Chemical composition
- Clinker mineralogy and burning conditions
- Fineness; Blaine and particle size distribution

- Retardation; mainly sulphate form and activity (note that most, but not all, synthetic gypsums are satisfactory for set control)
- Dilution of active components
- Pre-hydration of hydraulic components

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

C_3S primarily governs strength gain up to 28 days while C_2S has more effect on later strength. Work reported by Lea (Chemistry of Cement & Concrete) indicates that at 6 months, strength is largely independent of C_3S:C_2S proportions. Higher SO_3 tends to increase 28 day strength and higher alkalis tend to increase early strength and reduce 28-day strength.

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

Parameter	*Change*	*Effects on C109 Mortar Cube Strengths (Mpa)*			
		1-day	*3-day*	*7-day*	*28-day*
Blaine, M^2/kg	+20	0.9	0.9	0.6	0.5
-45 µm, %	+2.0	0.0	0.4	0.7	0.8
Na_2O eq, %	+0.1	1.2	0.8	0.4	-0.8
SO_3, %	+0.5	0.9	1.0	1.2	1.0
free-lime, %	+0.3	-1.2	-0.5	-0.4	-0.6
C_3A, %	+1.0	0.0	0.1	0.0	0.0
C_3S, %	+5.0	1.5	0.7	0.0	0.0
LoI, %	+1.0	-0.8	-1.0	-1.2	-1.0
Mortar air, %	+1.0	0.0	0.0	-0.3	-0.5

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

6.15 Setting Time

Setting of cement is caused by the development of cross-linking structure of hydration products shortly after the dormant period (see Chapter 10, Hydration of Portland Cement). This is illustrated by Figure 6.2 from Locher et al (ZKG; 6/1980, pg 271). Good control of setting is achieved by matching C_3A reactivity with sulphate availability in solution. Unbalance between C_3A reactivity and sulphate availability can cause flash set or false set as shown in Figure 6.3.

The primary factors that affect normal setting time are free-lime and the forms of calcium sulphate (see Sec B7.4). The secondary factors are fineness and C_3A content. Figure 6.4 from Hanson et al (Gypsum & Anhydrite in Portland Cement; 3rd Ed, pg 57) shows the effects of various forms of calcium sulphate, while Fig 6.5 demonstrates the effect of gypsum dehydration:

Figure 6.2: Cross Linking of Hydration Products

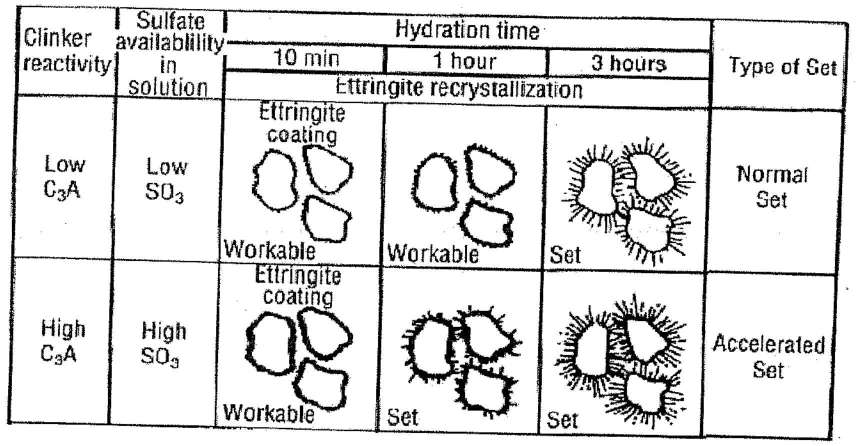


Figure 6.3: Flash and False Set

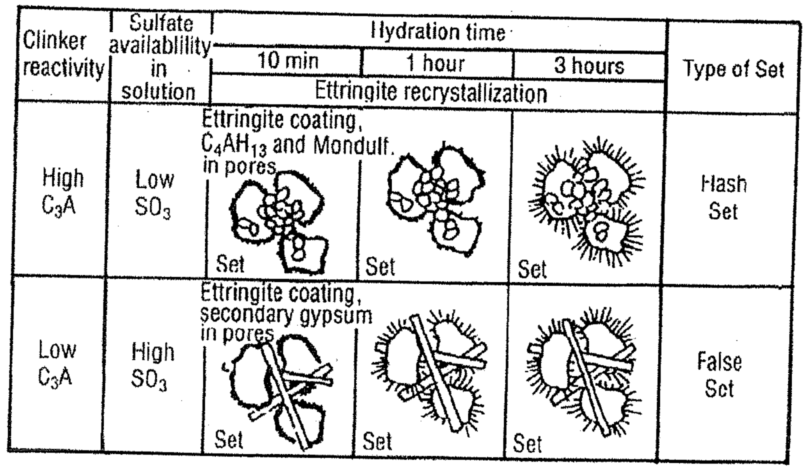


Figure 6.4: Effects of Gypsum Form on Setting Time

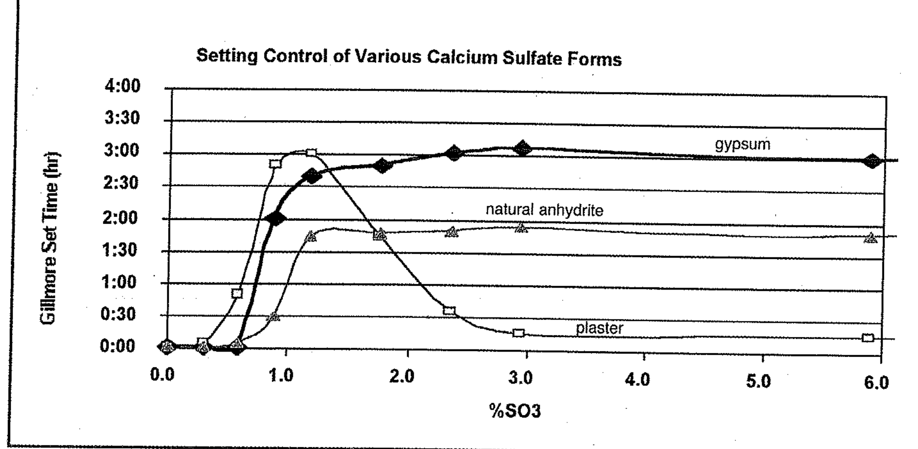
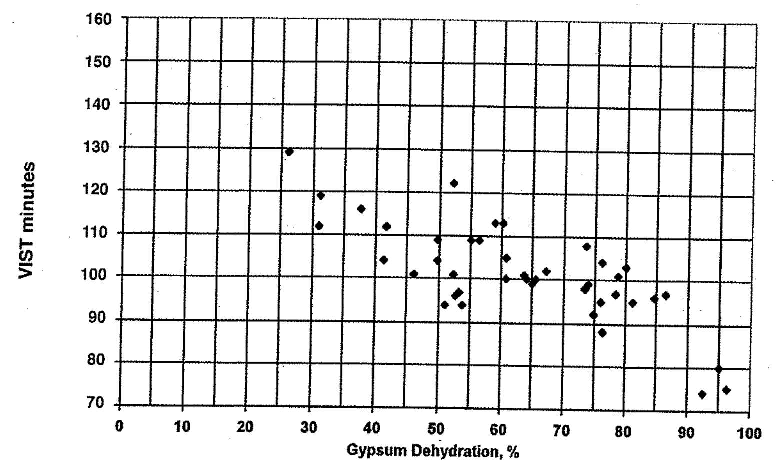


Figure 6.5: Effect of Gypsum Dehydration on Setting Time



6.16 Cement Types and Specifications

There are numerous national and international cement specifications together with the American Petroleum Institute specification for oilwell cements (API Specification 10A) which latter has virtually universal recognition (Bensted; WC; 2/1995, pg 40). As an example, a summary of the ASTM specification is given below.

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

It should be noted that the results of different specification tests may not be directly comparable. In particular, different tests for compressive strength may use differing cement contents, water-cement ratios, and mixing and moulding methods; all of which can lead to wide disparity in quoted strengths from identical cement samples. To avoid disparity and facilitate cement trade in the EU common standards EN-196 and EN-197 have been adopted.

ASTM C150 Specificaton for Portland Cement

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

Associated ASTM test methods & specifications:

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

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

Physical Limits:	I	II	III	IV	V
Air content, max	12	12	12	12	12
(A), max		22	22	22	
min		16	16	16	
Fineness, min, m^2/kg	280	280		280	280
Autoclave exp, max, %	0.80	0.80	0.80	0.80	0.80
Strength, min, MPa					
– 1 day			12.0		
– 3 day	12.0	10.0	24.0		8.0
– 7 day	19.0	17.0		7.0	15.0
– 28 day				17.0	21.0

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

Typical cement compound concentrations for the five cement types are:

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

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

1. Water requirement for normal consistency shall not increase by more than 1%.
2. Setting time (Vicat initial) shall not increase/decrease by more than the lesser of 1H or 50%.
3. Autoclave expansion shall not increase by more than 0.10.
4. Grand average of strengths at all ages shall not decrease by more than 5% for mortar, and no more than 10 per cent for concrete.
5. $SO_3 < 0.3\%$
6. Blaine $< 13m^2/kg$
7. Drying shrinkage < 0.025%
8. Flexural strength of concrete <10%

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

White cement is a special Portland cement with very little ferrite phase (<1%) and relatively high aluminate and silicates contents. Its whiteness is achieved by using raw materials containing very little iron, manganese, and chromium, using gas or other low-ash fuel, burning clinker under reducing conditions, quenching in water, and grinding the cement to a high fineness. Cement whiteness is impaired if Fe_2O_3 is >0.50%, Mn_2O_3 >0.02%, and Cr_2O_3 >0.01%. The use of pure sand and limestone and the low liquid content of the raw mix require clinker burning at 1600°C, usually with a mineraliser (CaF_2). However, specific fuel consumption is considerably greater than for grey cement. Quenching clinker with water and trying to recover the waste heat pose challenges to the process and equipment (Pekin; ICR; 12/2000, pg 35: Clark; ICR; 7/2001, pg 43: Schulz; ZKG; 2/2003, pg 82).

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

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

Masonry Cement is used for mortar in masonry construction where good workability and rapid hardening are required. Various inter-ground ingredients may be used, but commonly 20-50% limestone is incorporated together with an air-entraining agent. Masonry cement is typically ground to $5000\text{-}6000cm^2/g$ and ASTM specification C91 defines three grades: N, S, and M with increasing strength requirements. ASTM C1328 specifies plastic (stucco) cement for plastering applications. There is very little difference between C1328 and C91. ASTM C1329 specifies mortar cement, which is basically masonry cement with a lower air content limit and with additional bond strength requirement.

6.17 Cement Intergrinds

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

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

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

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

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

Typical compositions, with wide variations, are:

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

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

ASTM C595 SPECIFICATION FOR BLENDED HYDRAULIC CEMENTS

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

Types covered are:

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

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

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

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

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

European Standards are issued by the European Committee for Standardization. EN 197-1 covers composition, specifications and conformity criteria. EN 196 covers sampling and both physical and chemical testing of cement.

Rejection is permitted under ASTM C150 if any requirements are not met. BS 12 and EN 197 employ 'auto-control' by the producer, which generates statistical test data on cement as shipped; a finite though small incidence of defects is permitted.

6.18 Supersulphated Cement

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

6.19 Calcium Aluminous Cement (CAC)

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

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

A mixture of limestone or chalk and bauxite is charged into the upright stack section of an L-shaped reverberatory open-hearth furnace. The mixture is fired with pulverized coal or oil. The charge is preheated and calcined by combustion gases and then passed into the hearth section of the furnace where it is melted above 1450°C and homogenized. The molten product is continuously withdrawn by means of a pan conveyor where it cools and solidifies as clinker. The clinker is crushed and then ground to 250-400 m^2/kg in a ball mill. A setting regulator is not added during the grinding stage. Bauxite, fuel and power costs are all high. Other high alumina cements with 70-80% alumina can be sintered in a rotary kiln and are used both for refractory and construction purposes. CAC is graded by Al_2O_3 content. The higher the alumina content, the more rapid the hardening and the better the refractory properties. The typical chemical compositions are:

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

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

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

6.20 Shrinkage-Compensating Cements (SCC)

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

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

The composition of kleinite expansive clinker can be varied considerably to utilize the raw

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

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

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

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

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

6.21 ISO 9001:2000 Quality Management System

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

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

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

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

0 Introduction (Includes the process approach; *plan, do, check, correct*)
1 Scope (Includes reasons for implementing ISO 9001)
2 Normative Reference
3 Terms and Definitions
4 Quality Management System (Includes identification of processes, quality manual, control of documents)

5 Management Responsibility
6 Resource Management
7 Product Realization - Manufacturing (Includes quality objectives, testing activities, compliance of purchased materials & services, traceability, control and calibration of testing equipment)
8 Measurement, Analysis, and Improvement (Includes monitoring of customer satisfaction, internal quality audits, continual improvement)

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

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

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

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

6.22 Concrete Problems

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

A typical concrete mix may be:

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

which yields a concrete with:

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

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

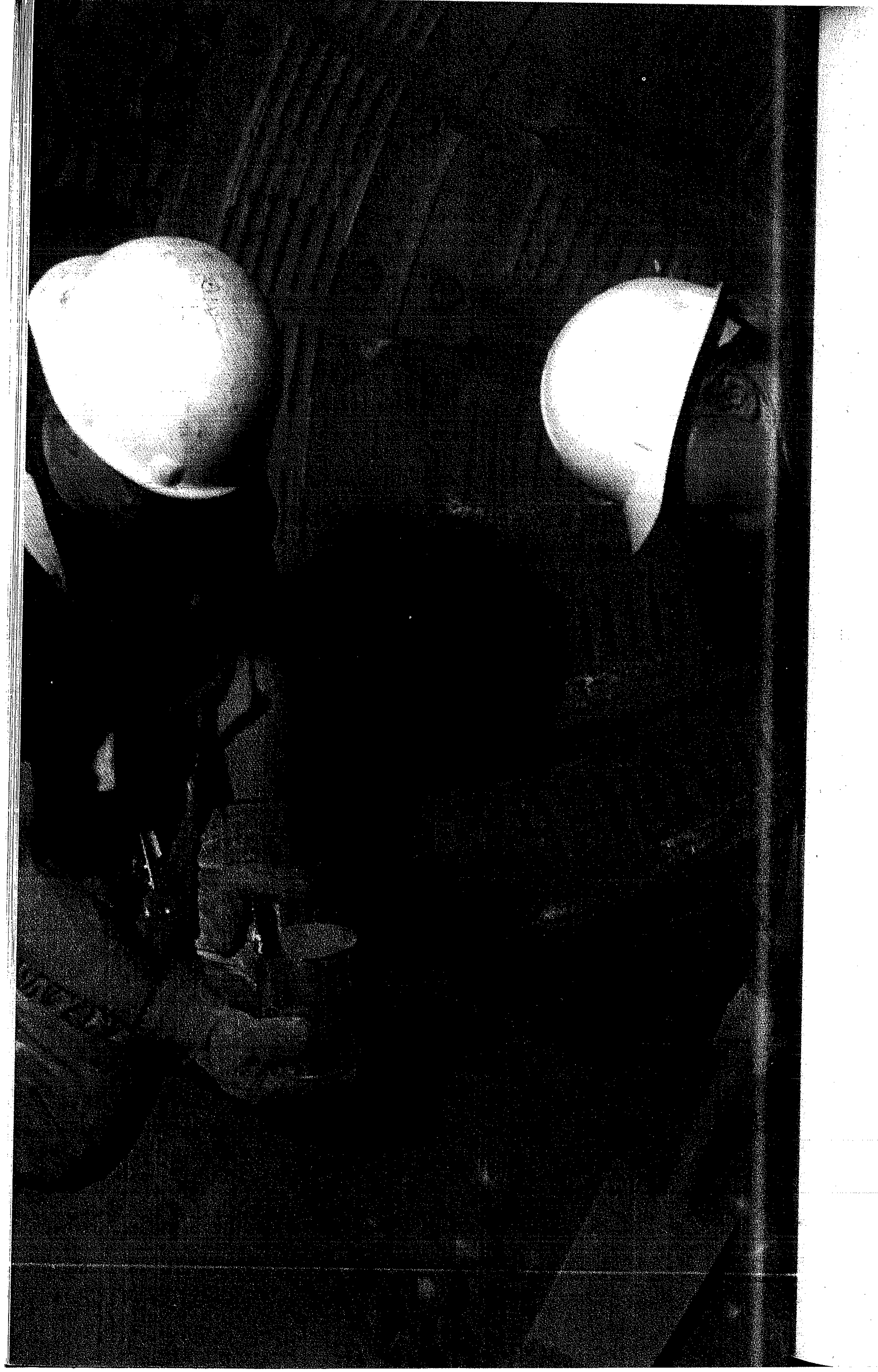
- Air entraining agents
- Accelerators
- Water reducing and set retarding agents also known as plasticizers and superplasticizers

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

It is a curiosity of cement manufacture that the product is tested and certified either as a paste of cement and water or as a mortar containing standard sand. Such testing can give only a limited indication of performance in concrete. Specifically, cement strength tests ignore both water demand, which varies significantly with fineness, and possible interactions with the wide range of concrete additives. Optimum SO_3 (C563) is the level which maximizes mortar strength at 1-day. However, higher SO_3 contents are required to maximize strengths at later ages or at elevated temperatures. Higher cement fineness, C_3A and alkali contents also demand more SO_3 for maximum strength. Concrete containing water reducer usually requires a higher optimum SO_3 (Tagnit-Hamou & Aitcin; WC; 8/1993, pg 38). It is also known that high alkali cement requires less air-entraining additive so that alkali variation in cement can cause strength variation in concrete. Helmuth has suggested that the optimum ratio of SO_3:Al_2O_3 is about 0.6 for Portland cement with a normal particle size profile, and 0.8 for cement with a narrow particle size range (Detwiler; ZKG; 7/1995. Pg 384).

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

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



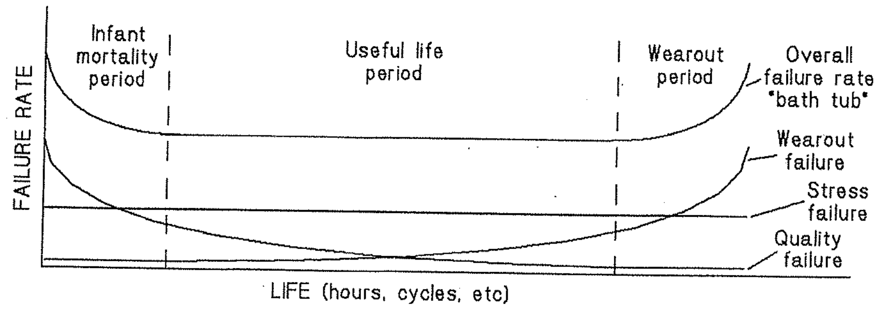
7 Maintenance

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

7.1 Failure Modes

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

Figure 7.1: Equipment Failure Profiles



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

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

7.2 Computerized Maintenance Management Systems (CMMS)

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

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

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

7.3 Reliability Centred Maintenance

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

- Failure mode and effect analysis (FMEA) and
- consequent prevention or control task definition

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

department considers that its operation is defective. Management must, therefore, establish for each system the required:

- *Capacity* (eg production rate) together with feed and product specifications, and
- *Reliability* in terms of % scheduled operating time when the equipment is actually available based upon historical production achieved, equipment design specifications, standard industry performance, and business objectives for the year. Usually these parameters must be maximized for the kiln but may be relaxed for oversized ancillary equipment. The same priorities should be employed by the maintenance department in allocation of the scarce resources at their disposal (manpower, expense, and down-time), and these may vary from time to time depending upon equipment and inventory situations.

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

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

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

7.4 Maintenance Cost Management

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

Maintenance labour costs are essentially fixed though overtime and outside contractors may constitute a significant variable element. Studies have shown as little as 28% of an 8-hour shift may be spent actually working on equipment and, more than with other jobs in a plant where a routine is established, productivity requires efficient scheduling and supervision.

Equipment downtime is another major maintenance resource which should be used as efficiently as possible. For kilns, which are run to failure the best use must be made of the unavoidable downtime and it is essential to plan in advance for all routine checks and for accumulated non-critical repairs (Lowell; GCL; 7/2001, pg 12 & 8/2001, pg 24). This may also involve contingency planning for possible tasks that are not confirmed until access is obtained. Once the required job list has been established and the critical path determined the work should be prioritised and scheduled to minimize the outage. Ancillary equipment which can be made available for scheduled preventive maintenance can accommodate other considerations; for example, maintenance on mills may be best performed during peak power tariff periods, and maintenance on crushers should be done when the quarry is not operating. A final consideration is that scheduled maintenance should keep as closely as possible to the schedule so that the production department may have confidence that the equipment will be available when they need it; loss of maintenance credibility is a major factor in operators' refusal to shut down equipment for routine attention and a downward spiral into breakdown maintenance.

Inputs to the CMMS come from work orders which should:

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

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

7.5 Maintenance Organisation

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

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

7.6 Role, Planning and Control

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

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

- equipment unreliability due to unsuitable design, defective construction, or mis-operation
- inadequate prioritisation of scheduled work

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

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

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

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

7.7 Mobile Equipment Maintenance

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

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

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8 Environment & Pollution Control

8.1 Dust Collection

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

Cyclones are typically 95% efficient dropping to ca 60% for particles less than 5µ. Normal inlet/outlet velocity is 10-20M/sec, pressure drop 50-150 mmH_20, and aspect ratio (height:diameter) 3-5.

Electrostatic precipitators (EPs or ESPs) comprise an array of discharge wires at 50-100kV negative potential and earthed collecting plates. Pressure drop is about 15-20 mmH_2O and power consumption 0.2-0.3 $kWh/1000M^3$. Efficiency is typically 80% per field so that a 4-field unit should capture up to 99.85% of entrained dust. Efficiency varies with particle size and operation is described by the Deutsch formula:

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

where n = efficiency %
w = particle migration velocity (M/sec)
A = area of collecting plates (M^2)
Q = gas flow rate (M^3/sec)

Migration velocity reflects dust resistivity, particle size, field intensity, gas viscosity, and other design parameters and should range 0.07-0.10M/sec. Dust resistivity should be 10^7-$10^{11}\Omega$cm. H_2O, Cl and SO_3 reduce resistivity of basic dusts. Conditioning of inlet gas is normally required with about 15% moisture in kiln exhaust and 3% in cooler exhaust; gas temperature should be 120-150°C, though EPs can be designed for higher temperatures. Although modern units are of high reliability and can be guaranteed to achieve below $20mg/NM^3$, they are still liable to total shut-down due either to kiln interlock or to electrical failure, and this must be acceptable under emission regulation. Richter et al (ICR; EHB/2003, pg 91) reviews developments in the design and control of EPs and Vandelli (WC; 10/2003, pg 87) describes hybrid dust collectors where an electrostatic precipitator is followed by a fabric filter to achieve optimum efficiency.

Bag filters comprise filters of either woven fabric (which employ bag shaking or reverse air flow for cleaning) or needle felts (which are cleaned by reverse air pulse). Air to cloth ratio ($M^3/min/M^2$) should be 0.5-0.9M/min (1.6-3.0ft/min) for woven fabrics and 1.7-2.3M/min (5.5-7.5ft/min) for needle felts. Pressure drop is typically 150-250 mmH_2O and efficiency 99.95%. The normal fibre is polyester which can operate up to 150°C, while polyamide can be used to 230° and glass fibre to 280°. The relative installation and operating costs of reverse air and jet pulse dust collectors are reviewed by D'Lima (ICR; 2/2000, pg 51).
A potentially useful development is a flexible metal fibre filter material resistant to 1100°C (ZKG; 7/2004, pg 36).

Pulse air should be dried and of 6-8kg/cm^2 pressure, and bag tension approximately 0.4kg/cm of bag circumference. Bag house pressure drop can be reduced and, thus, capacity increased by use of acoustic horns (Cameron; WC; BMH/1998, pg 67).

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

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

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

8.2 Pollution Control

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

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

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

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

Material spillage can be collected by either fixed or mobile heavy-duty vacuum systems (Haugen; WC; BMH/2003; pg 9) and, preferably, returned to the process.

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

Air pollution control is becoming progressively more onerous world wide as regulators lower limits on particulates, CO, SO_2, NOx, etc. and add new prohibitions on metals, dioxins, and other trace chemicals. (1990 Amendments to the U.S. Clean Air Act; Title III, Sec. 301). Under the Kyoto protocol, some countries intend to try to control CO_2 emissions; the EU will introduce a cap-and-trade scheme in 2005 involving emissions trading (Christiansen; ICR; 2/2003, pg 55) the cost of which will be substantial (von Landsberg; ZKG; 7/2004, pg 50).

Typical *particulate* limits for kilns are now 40-100mg/NM^3 and will probably continue to decrease; this progressively favours bag-houses over EPs, especially given that bag-houses 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, pg 55: Tate; ICR; EHB/2003, pg 97).

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

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

US emission limits are discussed by Salmento & Shenk (ZKG; 11/2004, pg 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, pg 66). Some regulators also recognize 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, pg 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, pg 19). Archer (WC; 5/2003, pg 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, pg 20 & 3/2000, pg 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, pg 272) have found that many cement raw materials contain 1.4-6g organic carbon per kg clinker, that these oxidize 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 Sec 9.6 for more detailed description of CO, NOx and SO_2 control.

NOx is formed during fuel combustion by oxidation of nitrogen compounds in the fuel (fuel NOx) and of the nitrogen from combustion air (thermal NOx). Thermal NO increases with flame temperature above 1200°C, with retention time, and with increasing free oxygen.

Haspel et al (ICR; 1/1991, pg 30) working with NOx as a control parameter for kiln operation, have discovered that, although the NOx 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. Petroleum coke combustion, too, is particularly sensitive to secondary air temperature. Thus, additional factors affecting NOx at kiln exhaust include:
- kiln atmosphere (NOx is degraded by CO > 3000ppm)
- alkali cycle which increases rapidly with burning zone temperature and with reducing conditions
- secondary air temperature

NOx emissions for normal operation may be 1000-1500mg/NM^3. Flame quenching, low-NOx burners, or staged combustion (for precalciner kilns only) should approach 500mg/NM^3. Selective non-catalytic reduction (SNCR) or Selective catalytic reduction (SCR) would be required to get significantly lower (Haspel; ICR; 1/2002, pg 63).

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, oxidize in the preheater and largely exit with exhaust gas. SO_2 can theoretically oxidize 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, pg 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, pg 10 & 3/2001, pg 130).

An old, but still interesting, review of NOx and SO_2 production and control is by Kupper (WC; 3/1991, pg 94). Models for predicting emissions are discussed by Salmento & Shenk (ZKG; 11/2004, pg 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, pg 97).

Gas analysis for process control and emission monitoring involves a wide range of proprietary instruments (Gumprecht et al; WC; 10/2003, pg 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, pg 67).

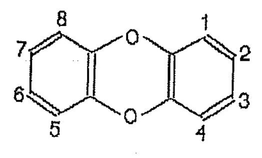
Dioxin emissions are of particular concern where alternative fuels are burned. It is generally accepted that dioxins and furans will not be released in significant quantity if burned under the following conditions:

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

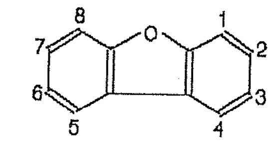
(Krogbeumker; ICR; 5/1994, pg 43)

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, pg 596). The US EPA proposed emission standards for dioxins and furans from cement kilns call for <0.20g TEQ/dry standard M^3 (Knotts et al; WC; 10/2004, pg 89).

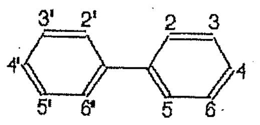
It may be noted that there are 75 chlorinated dibenzo-p-dioxins and 135 chlorinated dibenzofurans, all with different toxicities. Many regulators recognize a scheme of 'Toxic Equivalents' (TEQ). Following are the basic structures:



Dioxins



Furans



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, pg 117). The introduction of alternative raw materials and fuels requires monitoring both of inputs (Dirken; WC; 4/2003, pg 64) and of distribution between cement, kiln dust (if discharged), and gaseous exhaust (Gossman & Constans; ICR; EHB/2003, pg 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 bricklayers is cement dermatitis attributed to Cr^{+6} in cement. Water soluble chromate is now conventionally limited to 2ppm (Kerton; ICR; 5/2004, pg 65).

8.3 ISO 14000

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

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

8.4 Sustainable Development and Climate Change

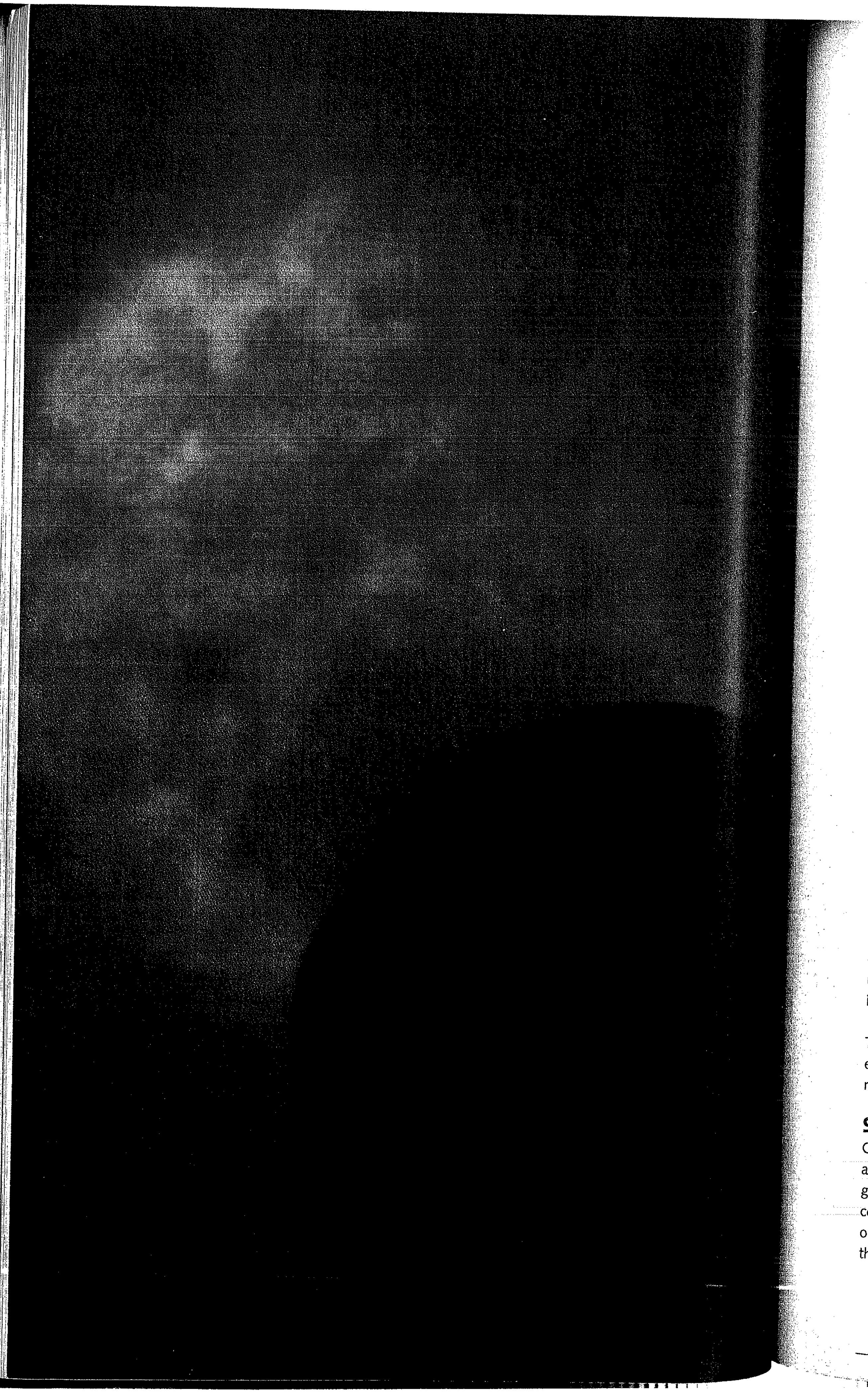
The World Business Council for Sustainable Development defines sustainable development as "forms of progress which meet the needs of the present without compromising the ability of future generations to meet their own needs" (www.wbcsd.ch). Ten major cement companies committed in 2002 to an initiative comprising: climate protection; resource efficiency; employee health & safety; emissions reduction; local impacts; internal business practices. For those interested in all this, or for those who must fake an interest, the concept is well summarised by Kerton (ICR Environmental Handbook; 2003, pg 11). A more critical view is presented by Portland Bill; ICR; 12/2002, pg 53 and by Baetzner; ZKG; 7/2004, pg 2.

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

It is not surprising that politicians and bureaucrats endorse the concept of control. It is not surprising that researchers should seek to maintain the flow of funding by spreading alarm. It is also not surprising that industries should capitulate to the pressure of activists. However, the futility of the exercise, the wilful or ignorant misrepresentation of the scientific data upon which it is based, and the astronomic cost are documented and discussed by Lomborg (The Skeptical Environmentalist [2001]) and by Okonski (Adapt or Die [2004]). It may, of course, be significant that the Kyoto initiative, like the European Union and the United Nations, has little democratic legitimacy and less accountability. Almost inevitably, it's administration will become as corrupt. The Clean Development Mechanism (CDM), which is part of the Kyoto Protocol, will allow rich countries to meet their own emission reductions by funding projects to reduce emissions in poor countries. As pointed out by Mitra and Kenny (Adapt or Die), efficient, safe energy supplies are one of the most desperate needs of subsistence societies, and frustrating their provision while promoting quite inappropriate wind and solar power projects evidences hubris verging on the criminal. For comparative costs of power sources see Sect B1.6.

If and when it transpires that the present attempt to manage our climate is based on false premises, is it conceivable that the legions of regulators, monitors, carbon traders, and others who will certainly make off with a significant part of the funding will apologise, close down their operations, and find more useful employment?

* * *



9 Combustion and Heat Transfer Processes

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

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

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

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

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

History of Combustion as a Science

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

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

9.1 Chemistry Of Combustion

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

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

The Complete Oxidation of Carbon

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

The Complete Oxidation of Hydrogen

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

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

The Incomplete Oxidation of Carbon

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

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

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

The Oxidation of Carbon Monoxide

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

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

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

9.2 Fuels

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

Gases

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

Oil Fuels

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

Atomisation of fuel oil is important because the initial drop size determines the size of the cenosphere which is formed and hence the length of time taken for the particle to burn. The oxygen diffusion is dependent on the surface area but the oxygen demand is dependent on the mass of the particle. Since the surface area is dependent on diameter2 and the mass on diameter3 it follows that the larger the drop the longer it takes to burn.

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

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

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

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

Coals

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

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

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

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

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

Hardness and Abrasion Indices – The hardness of the coal affects the capacity of coal mills, the harder the coal the less can be ground and/or the coarser the resulting pulverised coal.

The abrasion index is mainly dependent on the ash characteristics. Very abrasive coals with high silica ashes cause high wear rates in coal mill grinding elements.

9.3 Physics of Combustion

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

All combustion processes therefore take place in the following stages:-

MIXING → IGNITION → CHEMICAL REACTION → DISPERSAL OF PRODUCTS

The rate of combustion is dependent on the slowest of the above stages. In most industrial combustion systems, the mixing is slow whilst the other steps are fast. The rate and completeness of the combustion process is therefore controlled by the rate and completeness of fuel/air mixing. Insufficient fuel/air mixing produces un-burned CO in the flue gases, wasting fuel energy potential. For good combustion, it is necessary to ensure that adequate air is supplied for complete mixing and that the burner is designed to mix the fuel and air streams effectively and efficiently. The rate and completeness of combustion is controlled by the rate of completeness of the fuel air mixing. Hence, the saying of combustion engineers:

"If it's mixed, it's burnt"

Fuel/Air Mixing

For most burners, fuel/air mixing occurs as a result of jet entrainment.

Figure 9.1: Entrainment of Secondary Air into a Free Jet

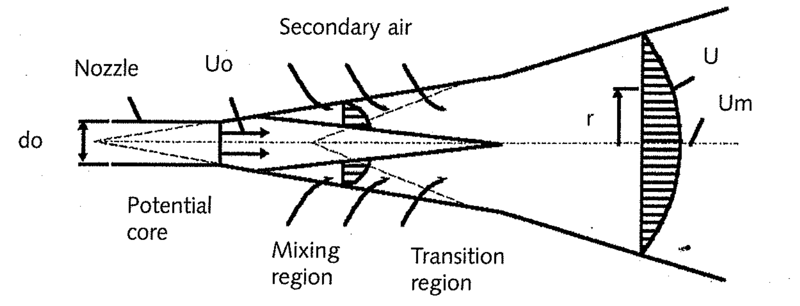
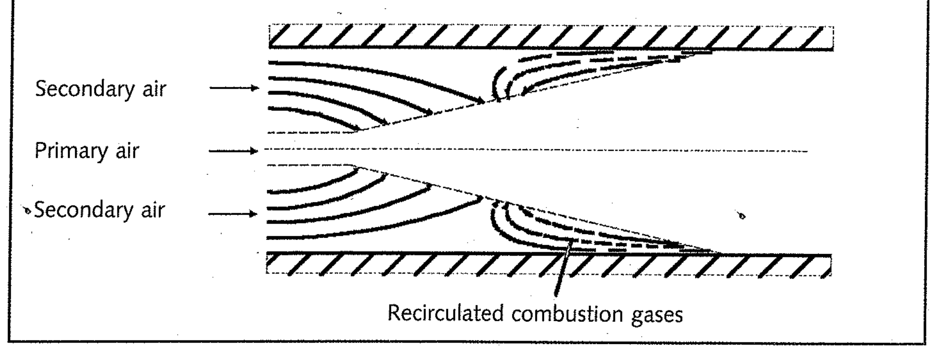


Figure 9.1 shows a free jet issuing from a nozzle in an ambient medium. Friction occurs between the boundary of the jet and its surroundings, causing the surrounding fluid to be locally accelerated to the jet velocity. The accelerated air is then pulled into the jet thus expanding it. This process is momentum controlled and continues until the velocity of the jet is the same as that of its surroundings. The greater the momentum of the jet, the more of the surrounding fluid that will be entrained into it. In the case of the free jet it can entrain as much of its surrounding medium as it requires to satisfy its entrainment capacity and it is able to expand unimpeded in doing so.

In the case of a confined jet however, such as in a rotary kiln flame, the jet is now constrained in two ways. The quantity of surrounding fluid being fed to the kiln ie., the secondary air, is controlled and limited. In addition, the expansion of the jet is now bound by the physical presence of the kiln shell.

If the confined jet has momentum in excess of that required for the complete entrainment of the secondary stream, then jet recirculation will occur. The secondary air stream is initially pulled into the jet as described above for a free jet. A point is then reached, however, when all the available air has been entrained into the flame. At this stage, the jet will pull back exhaust gases from further up the kiln and draw them into the flame in order to overcome this excess momentum. This phenomenon, known as external recirculation, is illustrated in figure 9.2.

Figure 9.2: Idealised Recirculation of a Confined Jet



The Role of Primary Air

Primary air has two major roles in burners:-

1. It controls the rate of fuel/air mixing.
2. It assists with flame stability.

The Effect of Primary Air on Fuel/Air Mixing

The primary air itself mixes very rapidly with the fuel at the nozzle, but the remaining air (secondary air) must be entrained into the primary air and fuel jet as described above. The rate of entrainment is dependent on the ratio of the momentum between the combined primary air and fuel jet and the momentum of the secondary air. Thus, the higher the flow-rate and velocity of primary air, the more rapid the fuel/air mixing. The flame characteristics are determined by this momentum ratio, and burners can be designed to give specific flame characteristics by the use of combustion modelling.

The presence or absence of recirculation has a great effect on the characteristics of the flame. A moderate degree of recirculation is a positive indication that fuel/air mixing is complete, whilst its absence is a clear indication that not all of the secondary air has been entrained into the primary jet. In this case, the production of significant levels of carbon monoxide is normal.

Furthermore, in the absence of recirculation there is a tendency for the flame to expand until it impinges on the brickwork. Hot reducing gases will then be in direct contact with the refractory brick, tending to wash them away and causing their subsequent failure. The recirculating gases from a high momentum ratio flame, however, provide a 'cushion' of cooler neutral gases which prevents this direct impingement of the flame on the brickwork.

Table 9.1: Characteristics of Flames with and without External Recirculation

	Flame with Recirculation	Flame without Recirculation
Fuel/Air Mixing	Good	Poor
Reducing/Oxidizing Conditions	Oxidizing conditions exist throughout the flame.	Strongly reducing conditions occur in fuel rich parts of the flame. Oxidizing conditions exist elsewhere.
Flame Impingement	None. Recirculating gases protect bricks and clinker from flame impingement.	Flame impingement occurs on the brickwork/clinker at the point where the jet expands to hit the kiln (11°). Impingement is severe where a low primary air/secondary air ratio occurs.
Carbon Monoxide Level	CO only produced at levels of oxygen below 0.5%	CO produced at levels of oxygen as high as 2-4%
Heat Release Pattern	Rapid mixing gives high flame temperature near the nozzle and a short burning zone.	Poor mixing gives gradual heat release with long flame.
Flame Stability	Good flame shape with stable heat release pattern	Heat release pattern considerably effected by changes in secondary air temperature, excess air, fuel quality, etc.

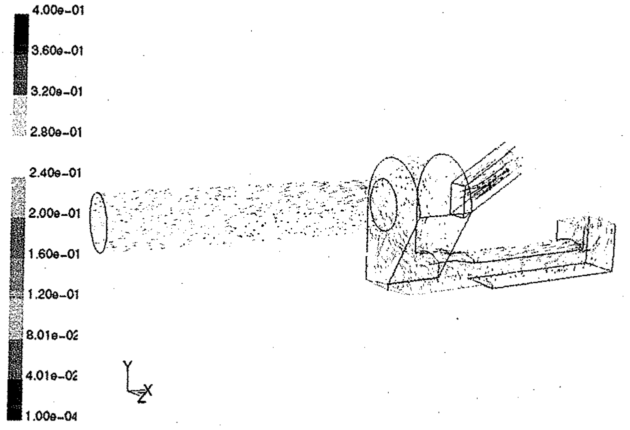
A high momentum recirculatory burner jet will also produce a more responsive and stable flame that is more controllable, hence making the operation of the plant itself easier. The characteristics of kiln flames with and without external recirculation are summarized in table 9.1.

Secondary Air Aerodynamics

Since the secondary air has to be entrained into the fuel/primary air jet the secondary air aerodynamics can have a huge effect on the fuel/air mixing. The secondary air flow patterns are considerably affected by the inlet ducting.

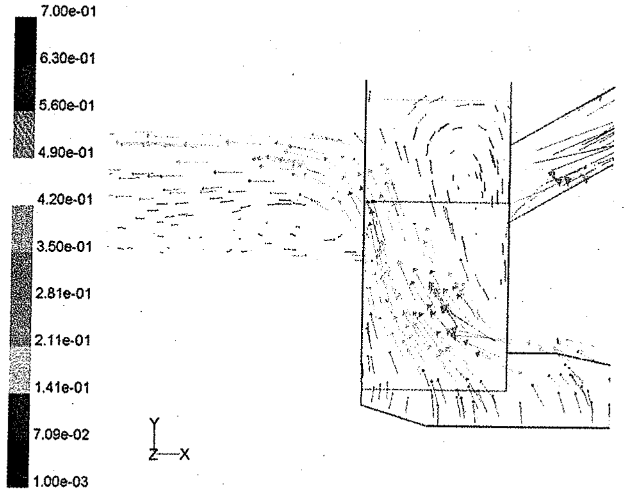
In the case of rotary kilns the flow is considerably determined by the design of the cooler uptake and hood system, or in the case of planetary coolers, by the cooler throats. To obtain the optimum potential performance from any kiln, it is absolutely essential that the aerodynamic characteristics of the kiln are taken fully into account when designing the burner. Extensive tests of kiln aerodynamics have been conducted by using water/bead model tests, computational fluid dynamics and full size investigations. One example of the aerodynamics for a grate cooler kiln is illustrated in figures 9.3 and 9.4. The asymmetry in the airflow pattern can be clearly seen.

Figure 9.3: Typical Aerodynamics from a Grate Cooler



Velocity Vectors coloured by Velocity Magnitude (m/s)

Figure 9.4: Close-up of Aerodynamics in the Burning Zone



Velocity Vectors coloured by Velocity Magnitude (m/s)

In most cases burner design cannot overcome certain air-flow patterns and, therefore, modification is often required to the geometry of the equipment. For example, in figures 9.3 and 9.4 the tertiary air off-take on the rear of the hood has to be relocated to eliminate the poor aerodynamics.

However, in some cases, the solution may be as simple as changing the location of the burner tip as shown in figures 9.5 and 9.6.

Figure 9.5: Physical model of secondary air pushing the flame to top of kiln

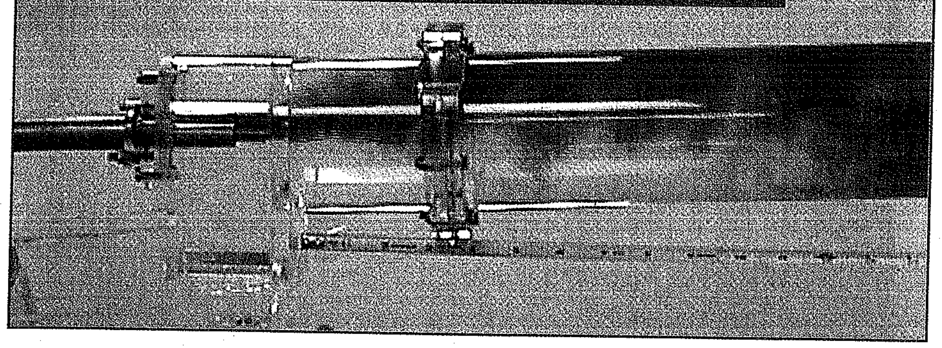
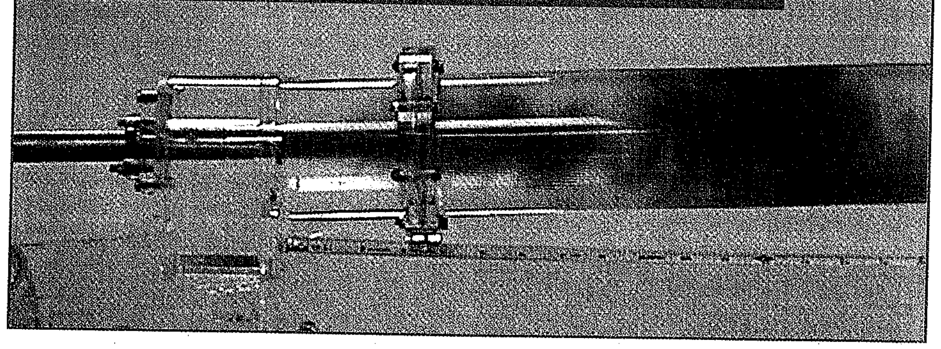
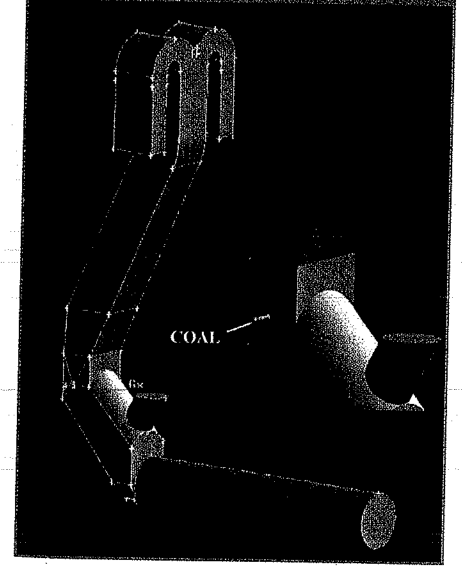


Figure 9.6: Same system of 9.5 after inserting burner further into kiln



Similar effects are observed with riser ducts and flash calciners, figures 9.7 and 9.8.

Figure 9.7: Riser Duct firing in a Preheater



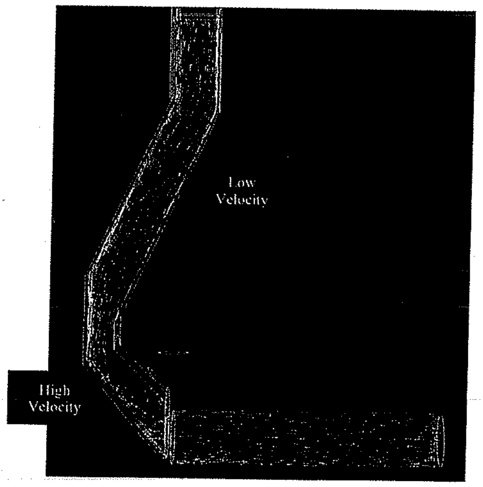


Figure 9.8: Velocity Profile in Riser Duct

Figure 9.9: Particle tracking showing skewed distribution

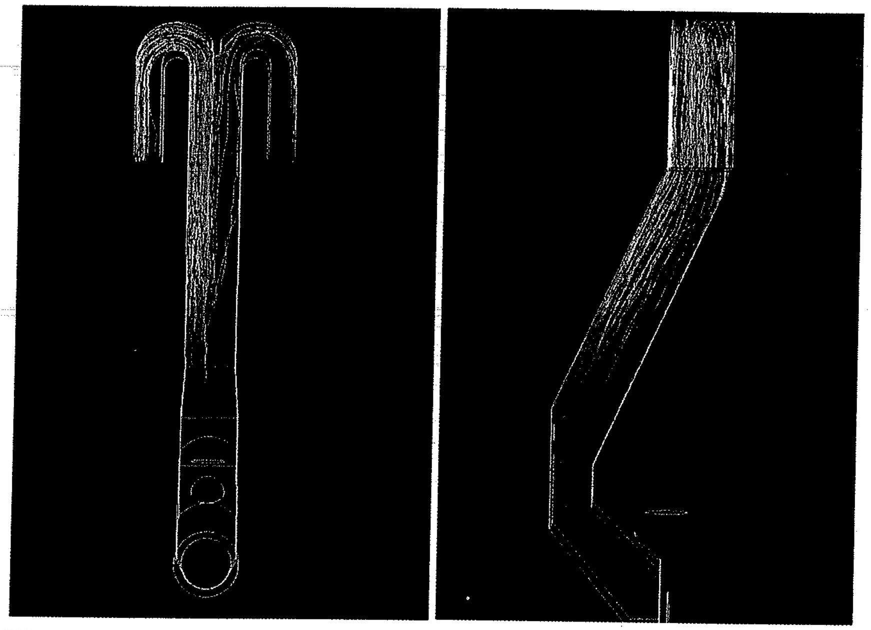
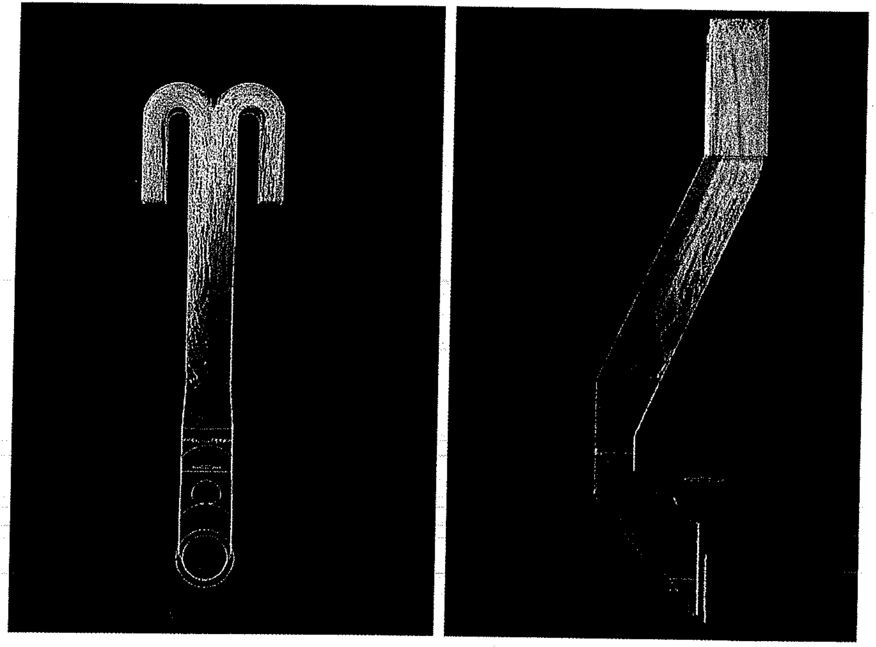


Figure 9.9: Particle tracking showing even distribution



The sharp angle entry from the kiln to the riser gives a highly asymmetric airflow giving poor fuel/air mixing and an intense recirculation zone on one side. Figure 9.9 shows how the velocity profile from the gases exiting the kiln dominates the distribution of particles injected in the riser. CO and temperature measurements confirmed these predictions. Figure 9.10 shows how the problem was corrected by tailoring the fuel injection to suit the aerodynamics.

For complete combustion and uniform heat transfer, it is essential to have an even distribution of fuel throughout the cross-section of a furnace. Modelling techniques like the ones shown above are becoming an important tool in the cement industry for process optimisation.

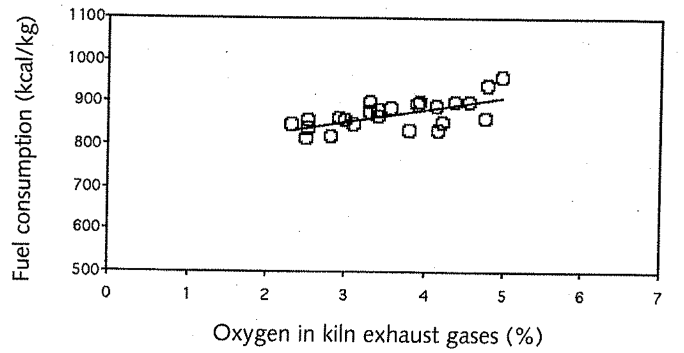
Effect of Excess Air on Fuel Consumption

Although the effect of excess air level on overall efficiency for thermal processes has been understood for many years, it is surprising how little attention is paid to this matter even today. The reduction in efficiency which occurs as the oxygen level is increased is caused by the requirement to heat the excess oxygen and nitrogen passing through the system firstly to flame temperature and ultimately to exhaust gas temperature.

In cement plants, the increased air flow through the coolers causes a reduction in the secondary air temperature, and therefore a reduction in the flame temperature, thus requiring even more fuel to heat the charge to the required process temperature. The total increase in fuel consumption is much greater than that necessary to heat the excess air to back-end temperature alone.

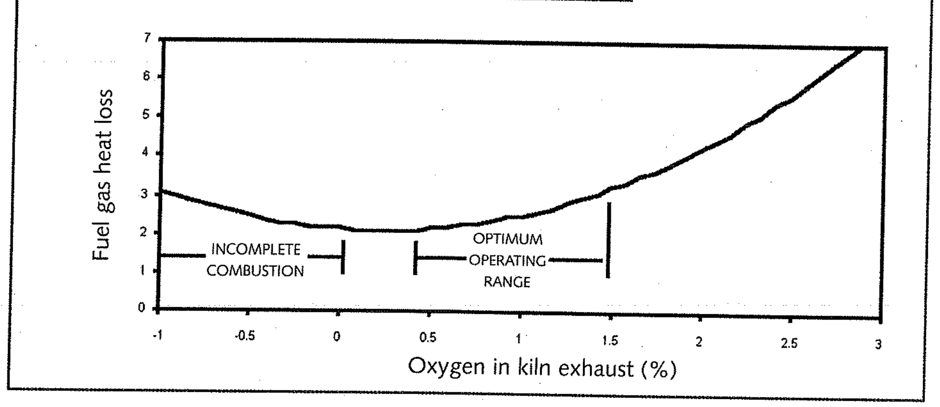
The effect of excess air on kiln thermal efficiency is very considerable. Figure 9.11 shows the relationship between the oxygen level and the measured daily heat consumption for a cement kiln. A clear trend is apparent and increasing the oxygen level from 1% in the kiln to 5% causes an increase in the heat consumption of more than 10%.

Figure 9.11: Effect of Excess Air on Heat Consumption



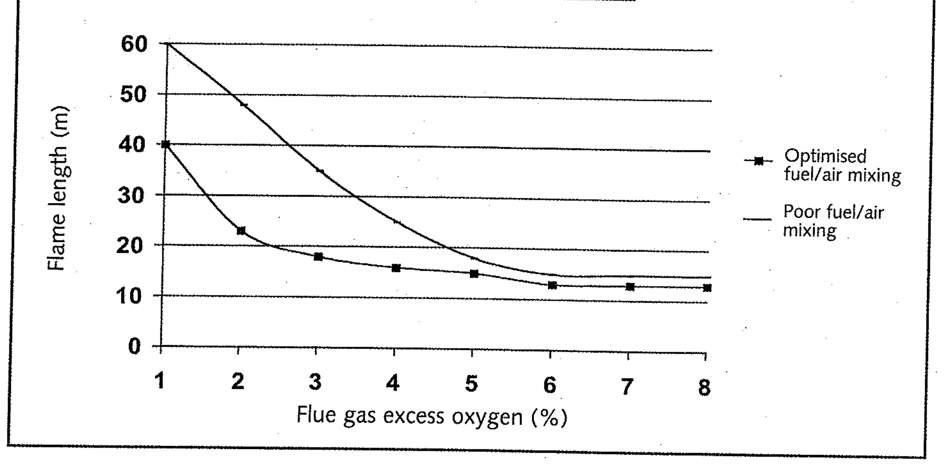
If the excess air level in a flame is reduced below a certain level, carbon monoxide is produced. This in turn also leads to an increase in the fuel consumption, due to the incomplete combustion of carbon, figure 9.12. The better the fuel/air mixing, the lower the excess air level at which these emissions occur.

Figure 9.12: Effect of Kiln Oxygen on Flue Gas Heat Loss



Excess air also has a dramatic effect the flame length and heat profile in the kiln. Many operators tend to believe the flame length increases as the draft from the ID fan increases. The opposite is true. Figure 9.13 shows a typical relationship between flame length and excess air for an optimised kiln with good fuel/air mixing and one with poor fuel/air mixing.

Figure 9.13: Flame Length Versus Excess Air



Two important characteristics of figure 9.13 are:
(1) Magnitude of flame length to excess air levels.
(2) Responsiveness of the flame to excess air levels.

The optimised flame can produce a length of 30 meters at 1.5% excess oxygen whereas the poor flame produces the same flame length at 3.5% excess oxygen. In addition, the responsiveness of the optimised flame allows the operator to fine tune the length of the flame with minor adjustments to the excess air, whereas the poor flame requires a much broader range of excess air.

9.4 Burners Design

Turbulent Jet Diffusion Flames

The flame in the rotary cement kiln, riser, or calciner is for the most part produced by a turbulent diffusion jet. During the past century, scientists have paid far less attention to diffusion flames than they have to premixed flames, despite the fact that the majority of industrial flames involve the simultaneous mixing and combustion of separate streams of fuel and air. The problem with analysing diffusion flames is that there is no fundamental property, like flame speed, which can be measured and correlated, even the mixture strength has no clear meaning.

When any jet mixes into its surroundings, steep concentration gradients are set-up in the neighbourhood of the orifice. Further downstream, turbulent mixing causes these gradients to become less severe but then rapid and random oscillations and pulsations occur. Only after the jet has largely decayed can any approximation to homogeneity be seen.

The particular fuel/air mixing pattern is determined by the mechanical and diffusion flux. The rates of chemical reaction are of little importance except in the tail of the flame where chars can take a significant time to burn.

Rotary Kiln Burners

Rotary kiln burners are different from most other industrial burners in that only a proportion of the combustion air passes through the burner and is therefore under the control of the burner designer. Most of the air comes from the product cooler and the aerodynamics of the flow is dependent on others.

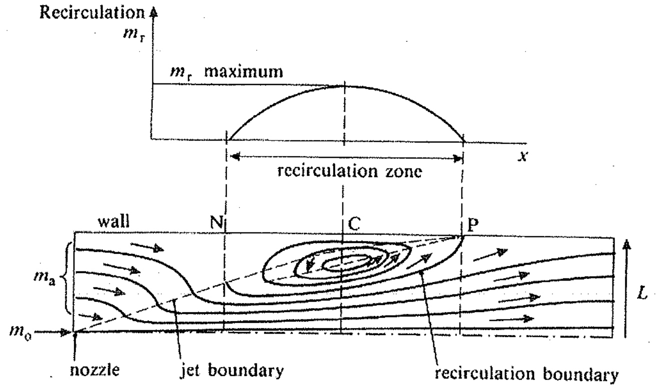
The most commonly used methods for designing rotary cement kiln burners are:
- Kinetic Energy: where the cross-sectional area of the burner nozzle or nozzles are generally based on the formula: PAV^2 , Primary Flow x $(\text{Velocity})^2$
- Momentum Flux: where the cross-sectional area of the burner nozzle or nozzles are generally based on the formula: Primary Air Flow x Velocity expressed as %m/s. % = primary airflow as a percentage of the stoichiometric air requirement.
- Jet Entrainment, see figure 20, where the cross-sectional area of the burner nozzle or nozzles are derived from more complex calculation but generally related to:

$$\frac{m_e}{(m_o + m_a)}$$

m_e = mass flow of entrained secondary air.
m_o = mass flow-rate of fuel and primary air through the burner
m_a = mass flow of secondary air.

The first two approaches assume the mixing between the fuel and air is unaffected by the secondary air and confinement of the rotary kiln. The jet entrainment approach determines the degree of external recirculation as the burner fuel jet mixes with the secondary air as shown in figure 9.14.

Figure 9.14: Mixing and recirculation downstream of a confined jet



All kiln and calciner burners except precessing jet burners are jet entrainment burners. The first two methods for designing burner usually results in a very high primary air velocity (>300 m/s) that employ 5-10% of the stoichiometric air requirement. The jet entrainment method usually requires more mass flow of primary air at a lower velocity to provide enough momentum for external recirculation.

The mass flow and velocity of primary air is a central debate in the cement industry. Heat balance calculations suggest that incremental increases in primary air at a low temperature reduce the thermal efficiency of the kiln by displacing hotter secondary air from the cooler. However, this argument assumes that the amount of excess air required to obtain the same production rate without flame impingement or carbon monoxide emission is held constant. In practice, if the burner momentum is insufficient to effectively mix the fuel with the secondary air, the heat consumption increases by 2% for every 1% increase in excess oxygen. This is one of the main reasons why NOx emissions are reduced with low primary air burners. Hence, there are competing forces between minimizing the amount of primary air and excess air that must be taken into consideration when designing a kiln burner.

Flame Stability

Good flame stability is important for safe and efficient combustion. A stable flame has a constant point of ignition very close to the burner nozzle, but with an unstable flame the point of ignition fluctuates up and down the kiln. This is potentially dangerous, since there is a high risk of flame out and in any case, the substantial quantities of un-burned gas between the gas nozzle and the ignition point form an explosion risk.

Obtaining good flame stability with natural gas is quite difficult because of its high ignition temperature, narrow flammability limits, and slow flame speed. Stabilization of oil flames requires proper atomisation plus some means of local recirculation. Stabilization of pulverized fuel flames (e.g. coal/coke) is effected by grind size, ash properties, volatile content, and conveying velocity. Flames produced by coal nozzle velocities in excess of 80 m/s are susceptible to severe instabilities. Despite all these potential hazards, few kiln burners have adequate means of ensuring good flame stability. The most effective technique is to form an internal recirculation zone just in front of the gas nozzle. Burning gas is carried back from further down the flame and constantly ignites the incoming fuel, hence anchoring the flame to the nozzle.

The internal recirculation zone can be achieved by a number of methods:

- Bluff body
- Swirl on the fuel
- Swirl on the primary air
- Swirl on both fuel and primary air

Bluff bodies suffer overheating caused by the flame and therefore tend to be unreliable over long periods. Swirl on the gas can give good results, but tends to be less effective with high primary air flows and velocities. Swirl on the primary air is a very effective way of ensuring flame stability, but quite high levels of swirl are required to achieve effective stability, and this can have adverse side effects on the overall flame characteristics such as causing flame impingement on the refractory.

The most effective method of ensuring flame stability is to use limited swirl on the both the fuel (in the case of gas only) and primary air. This ensures excellent flame stability and predictable burner performance over a wide range of operating conditions.

Except on some relatively primitive burners, such as some commercially available rotary kiln burners, re-radiation from the hot walls should rarely be used as the primary means of flame stabilisation, more positive methods are preferred.

Flash Calciner Burners

Many calciner burners are simply open ended pipes projecting through the walls of the vessel. The burning fuel is in intimate contact with the product. Flame stability is not normally an issue, since the incoming combustion air is normally preheated to above the gas ignition temperature. Hence, sophisticated swirl and bluff body devices are generally unnecessary. However, such simple burners can suffer from a number of disadvantages including poor fuel/air mixing and uncontrolled heat transfer, which can adversely affect product quality. Most calciner burners of this type produce large quantities of CO, typically over 1000 ppm and sometimes up to several per cent.

More sophisticated calciner burners are scaled down kiln burners, and like kiln burners should be matched to the calciner aerodynamics to optimise performance. Some calciner aerodynamic flow patterns give serious combustion problems because of poor combustion airflow. In these cases the burner alone cannot ensure the fuel/air mixing is optimised. The airflow must also be improved to ensure optimum combustion and heat transfer efficiency.

Gas Burners

Natural gas burners range from open ended pipes to multi-jet adjustable orifice designs. Owing to the narrow limits of flammability and high auto-ignition temperature, burning natural gas safely throughout a wide range of flow requires some means of stabilization. The most effective method of ensuring flame stability is to use limited swirl on the both the gas and primary air. This ensures excellent flame stability and predictable burner performance over a wide range of operating conditions.

A relatively recent development in gas burning uses the patented Precessing Jet (PJ) nozzle developed at the University of Adelaide, Australia, in combination with other jet flows, to provide a high radiation, low NOx flame tailored for a given application. To date its principal application has been in gas-fired rotary lime, alumina, cement and zinc oxide kilns but new developments are in progress and it is anticipated that it will soon find application using other fuels and in other processes. The term precessing refers to a gyro-scopic motion. The stirring motion of the flow field creates large fuel-rich structures that reduce the methane molecules into a carbon-laden laden environment. The result is a highly radiant low temperature flame that promotes high heat transfer with low NOx formation.

Oil Burners
Oil is an excellent fuel for rotary kilns owing to its high emissivity, which results in high rates of heat transfer to the charge. However, to burn oil efficiently it must be atomized and sprayed into the kiln in a controlled manner. If the drop size is too coarse the larger drops take a long time to burn and some of the largest may not burn out at all. This can adversely affect product quality, production rates and fuel consumption.

Rotary kiln oil burners are similar to conventional gas burners with an oil sprayer replacing the gas gun. Some primary air is always used. Atomizers commonly employed include simple pressure jet, duplex pressure jet, spill return, twin fluid (air blast or steam) and high efficiency twin fluid. Some burners employ bluff bodies for flame stabilisation while others employ swirl. Like a few gas burners, some oil burners rely entirely on re-radiation from hot refractory, for achieving flame stability.

Because the cement kiln requires a precisely controlled heat up, operators should use a high performance twin fluid atomizer with a wide turndown (8:1) and excellent flame stability is achieved using an aerodynamic swirler flame stabilizer.

Flash Calciner Oil Burners
Oil burners for flash calciners vary from open ended pipes spewing oil into the vessel to sophisticated burners employing twin fluid atomizers.

The open ended pipes tend to produce large drop sizes (over 1000 micron) which cannot burn-out during their residence time in the vessel. This un-burned fuel either contaminates the product or ends up in the duct collector. It results in both efficiency losses and product deterioration. The open ended pipe may well have been satisfactory for the lighter fuels of the 1960's but they are totally unsuitable for the high asphaltene oils of today. High performance, internal mixing, twin fluid atomizers are essential if all the fuel is to be burnt within the vessel. As for gas burners, the oil burners should be designed and optimised using modelling.

Coal Burners
Where the ash contamination can be tolerated, coal is the best fuel for rotary kilns owing to its very high emissivity, which results in high rates of heat transfer to the charge. Generally, the lower cost of coal gives it a significant economic advantage compared with other fuels. Coal is however, rather more difficult to handle than oil or gaseous fuels, since it is a solid material of varying composition and calorific value. Regardless of the design of burner, coal must be dried and ground before being supplied to the kiln. As for oil and gas burners, a coal burner is a critical component in a rotary kiln. The variable nature of pulverized coal requires a flexibility of burner design to allow the use of differing grades of fuel.

Many rotary kiln coal burners are simple open ended pipes, and apart from the inconvenience of having to insert a temporary oil burner to warm-up the kiln, an open ended pipe can given an excellent performance. Unlike oil and especially gas, it is quite safe to rely on re-radiation from the kiln walls when coal firing owing to the low ignition temperature of most coals.

With the conversion of many rotary kilns from oil firing to coal firing in the 1970's, new coal burners were developed from existing oil burner designs. These generally use less primary air than the open ended pipes and are suitable for use with indirect firing systems. However, in many cases their performance is very poor as a consequence of inadequate fuel air mixing resulting from the low jet momentum. In many cases these burners are sophistication for its own sake, with no benefits accruing from the extra cost. However, benefits can result from the use of multi-channel burners, especially with more difficult low volatile fuels such as petroleum coke, provided the burner is matched to the kiln, as described above.

Dual and Multi-Fuel Burners
Dual and multi-fuel burners combine the essential features of the single fuel burners described above. True multi-fuel burners give a real flexibility in fuel choice, a major advantage given that the current instability in world fuel prices which makes any medium term prediction of fuel costs, and hence investment decisions, very difficult. A true multi-fuel installation allows the plant to utilise the most economical fuel currently available. In many cases by-product fuels such as pulverised petroleum coke or wood waste may be used in place of, to in addition to, the primary fuel.

9.5 Heat Transfer

Normally, the sole purpose of burning fuel is to heat a product or generate steam. For this to occur requires that the heat is transferred from the flame to the process, thus heat transfer plays a vital role in the system. If there is a temperature difference (ie., a driving force) between two parts of a system, then heat will be transferred by one or more of three methods.

> **Conduction** – In a solid, the flow of heat by conduction is the result of the transfer of vibrational energy from one molecule to the next, and in fluids it occurs in addition as a result of the transfer of kinetic energy. Conduction may also be created from the movement of free electrons (viz; metals).
>
> **Convection** – Heat transfer by convection is attributable to macroscopic motion of a fluid and is thus confined to liquids and gases. Natural convection arises from density differences caused by temperature gradients in the system. Forced convection occurs due to eddy currents in a fluid in turbulent motion.
>
> **Radiation** – All materials radiate thermal energy in the form of electro-magnetic waves. When radiation falls on a surface it may be reflected, transmitted, or absorbed. The fraction of energy that is absorbed is manifest as heat.

The general equation for the heat transfer rate between two parts of a system is:-

$$Q = \Phi\ (dt^{n}) \qquad (7.1)$$

The origin of Φ and the value of **n** vary according to the mode of heat transfer.

	Φ	n
Conduction	kA/x	1
Convection	hA	1
Radiation	$\sigma\varepsilon A$	4

where k = thermal conductivity
x = separation distance of t
h = convective h.t.c
σ = Stefan-Boltzmann constant
ε = emissivity
A = surface area

Heat transfer is a very complex subject worthy of several books in itself. The subject is covered here only in sufficient detail to allow a reasonable understanding of the relative importance of the three methods of heat transfer.

Conduction

Equation (7.1) can be written more correctly as:

$$Dq/d\theta = -kA\ dt/dx \quad \text{(Fourier Equation)} \quad (7.2)$$

There are two types of conduction to consider:

- steady state – dt/dx independent of θ
- transient – dt/dx depends on θ

Steady State

The temperature gradient through the system remains constant, thus (7.2) becomes:-

$$Q = \frac{k.A.\ \Delta t}{x}$$

This is the simplest type of problem involving calculations on heat loss through walls, optimum lagging thickness etc.

Transient Conduction

The temperature gradient through the system varies with time ie, Equation (7.2) holds:

$$dQ/d\theta = -kA \quad dt/dx \quad (7.2)$$

This involves problems including heating and cooling of objects (steel-slab reheating, ingot soaking, space heating, thermal inertia of buildings, vulcanising, glass cooling, etc). Solution of problems is more complex. Two techniques are commonly used:-

- Mathematical solution of Equation (2) for given boundary conditions using calculus.
- Numerical solution by finite difference methods (graphical).

Convection

Convective heat transfer is normally divided into natural convection where the fluid motion is caused by density differences and gravity and forced convection. In the latter, the fluid motion is caused mechanically by a fan, pump, etc.

$$Q = hA\ (t) \quad (7.3)$$

Convection contributes relatively little to the heat transfer at flame temperatures, but has considerable importance in product preheating and cooling.

Radiation

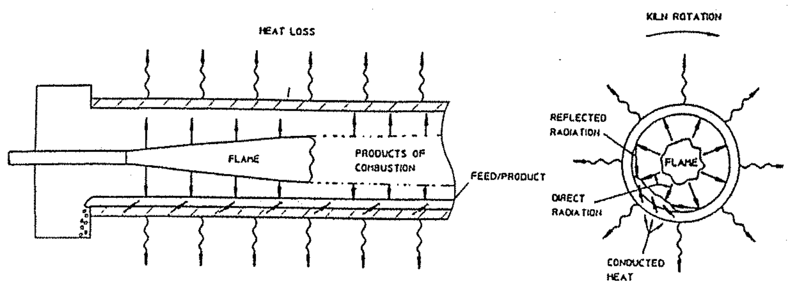
Radiation is the dominant mechanism of heat transfer in cement kilns with over 95% of the heat transferred this way in the burning zone. It can be seen from figure 9.15 that the process is more complex than the basic equation would suggest:

$$Q = \sigma\varepsilon A\ (T_F^4 - T_\rho^4) \quad (7.4)$$

where Q = the heat transferred
σ = Stefan-Boltzmann constant
ε = emissivity
A = surface area
Tf = flame temperature °K
$T\rho$ = Product temperature °K

The rate at which heat is transferred to or from a flame is controlled predominantly by the radiative exchange in the combustion chamber. The factors which effect this exchange are the temperatures, emissivity and relative geometry of the flame and surroundings.

Figure 9.15: Heat Transfer Paths in the Rotary Kiln



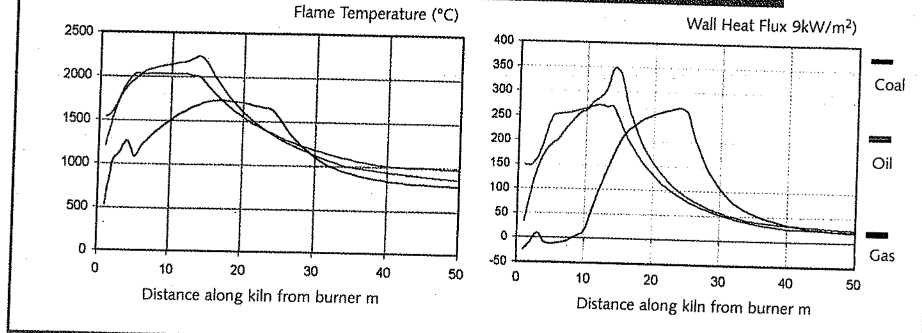
Effect of Fuel Type on Heat Transfer

The most observable difference between gas, oil and coal flames is the brightness or emissivity:

gas flame emissivity	~ 0.3
oil flame emissivity	~ 0.5b
coal flame emissivity	~ 0.85

The emissivity varies along the length of the flame as shown in Figure 9.16.

Figure 9.16: Heat Transfer rates from different fuel types

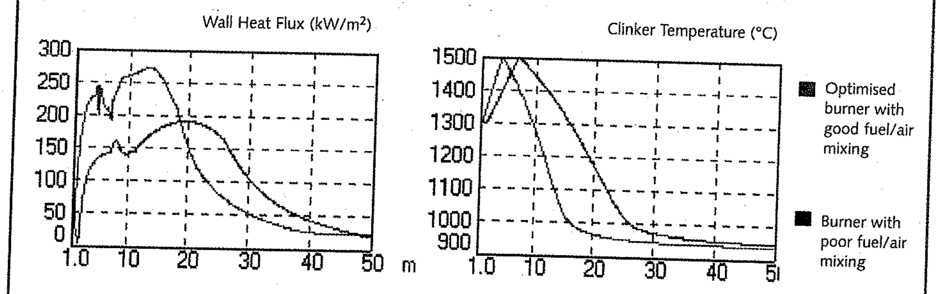


The higher emissivity of the oil and coal flames result in higher heat transfer rates in the near flame region, and a 'peakier' heat flux profile.

Effect of Burner Design on Heat Transfer

The burner aerodynamics also significantly modify the flame shape and heat transfer profile as shown in Figure 9.17. A long, flat heat flux profile equates to a long burning zone. This is usually detrimental to clinker quality as this gives rise to larger alite and belite crystals

Figure 9.17: Heat Transfer rates from different burner designs



The optimised heat flux was achieved by producing a recirculatory flame with all the benefits as outlined in Table 1. This was achieved by optimising the aerodynamics in the kiln and by tailoring the momentum of the burner relative to the momentum of the secondary air through physical and mathematical modelling techniques.

9.6 Pollution Formation and Control

NOx Formation

The NOx formation in flames is generally by both thermal and fuel routes (for coal, oil and petroleum coke). Owing to the very high temperatures which occur i.e.., above 2000°C (3600°F), thermal NOx is generally the dominant mechanism. In gas fired plant, fuel NOx is absent so all the NOx is thermal NOx. However, it should be noted that the absence of fuel NOx in gas fired plant does not necessarily lead to a reduction in NOx emissions, since flame temperatures are often higher.

Thermal NOx is formed by the combination of atmospheric nitrogen and oxygen at very high temperatures. The high temperatures are required because of the high activation energy of the reaction, it is therefore highly temperature dependent. The reaction takes place between oxygen radicals, nitrogen radicals and molecular nitrogen in the Zeldovich reaction couple. Apart from temperature, the in-flame oxygen concentration and the residence time in the high temperature zones influence the final thermal NOx emissions.

Most fuels, other than gas contain nitrogen bound as an organic compound in the structure. When the fuel is burnt this organic nitrogen becomes converted into a range of cyanide and amine species which are subsequently oxidized to NOx, depending on the local oxygen availability, but this mechanism is less dependent on temperature.

A third mechanism of NOx formation has been identified by some workers which involves the fixation of nitrogen by hydrocarbon compounds in fuel rich areas of the flame. This mechanism is known as prompt NOx. The formation mechanisms of prompt NOx, thermal NOx and fuel NOx are described in more detail below.

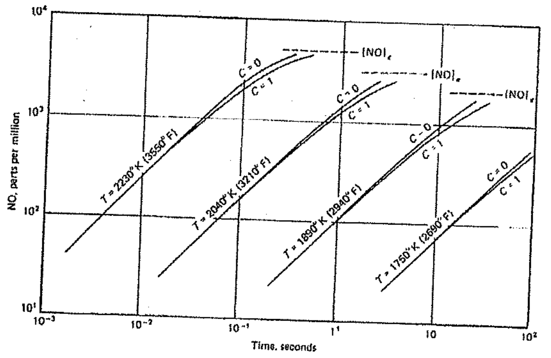
Thermal NOx Formation

These reactions are highly dependent upon temperature as shown in Figure 9.18.

Fuel NOx Formation

The mechanisms by which NOx is formed from the chemically bound nitrogen in the coal is extremely complex, even the structure of the nitrogen in the coal is subject to considerable conjecture. The nitrogen is believed to be in the form of pyridine, pyrrol and amine type structures. The actual structure in any coal is believed to be strongly dependent on coal type. The predominant forms of nitrogen in most coals were the pyrrolic and pyridine forms and that the former tended to decrease with increasing coal rank. However, at present, the importance of the structure of the nitrogen in the coal on the final NOx emissions is not well established.

Figure 9.18: Concentration of NO formed from a 40:1 ratio of N_2/O_2 as a function of time at various temperatures



When coal is burnt in suspension it is heated very rapidly to high temperatures and pyrolysis occurs, producing solid and gaseous products. The nitrogen present will divide between these with typically 20% of the nitrogen in the char and 80% in the gaseous phase, the latter both as the light fractions and tars. For any coal, the distribution of nitrogen between the gaseous phase and char is heavily dependent on the conditions in the flame such as heating rate, peak temperature, and residence time at high temperature.

Most of the gaseous nitrogen pyrolyses either directly or indirectly to HCN. This complex process is not instantaneous but dependent on the conditions in the flame. The HCN then oxidises to NO with this reaction being both temperature and time dependent.

Prompt NO Formation

In low temperature fuel rich flame zones, NO is found to form more rapidly than predicted from considerations of the thermal NO mechanism. The difference is due to the so called 'Prompt NO' formation mechanism. Prompt NO is formed by the rapid fixation of atmospheric nitrogen by hydrocarbon fragments. Prompt NO is formed in all combustion system but its contribution to the total NOx emission in coal-fired cement kilns is minimal.

NOx Control

The table below gives an overview of the techniques that have had a positive effect on, ie reduce, the emissions of NOx arising during the manufacture of cement.

Technique	Kiln systems applicability	Reduction Efficiency	Reported emissions		Reported Costs[3]	
			Mg/m^3 [1]	Kg/tonne [2]	Investment	Operating
Flame Quenching	All	0-50%	400-	0.8-	0.0.-0.2	0.0-0.5
Low-NOx Burner	All	0-30%	400-	0.8-	0.15-0.8	0
Staged Combustion	Preheater & Precalciner	10-50%	<500-1000	<1.0-2.0	0.1-4	0
Mid-kiln firing	Long	20-40%	No info	-	0.8-1.7	No info
Mineralised clinker	All	10-15%	No info	-	No info	No info
SNCR	Preheater & Precalciner	10-85%	200-800	0.4-1.6	0.5-1.5	0.3-0.5
Pilot SCR	Possibly All	85-95%	100-200	0.2-0.4	Ca. 2.5-4.5	0.2-0.4

Source: 1999 European Commission Reference Document on Best Available Techniques in the Cement and Lime Manufacturing Industries

[1] Normally referring to daily averages, dry gas, 273K, 101.3 kPa, and 10% O_2.

[2] Kg/tonne clinker: based on 2000 m3/tonne of clinker

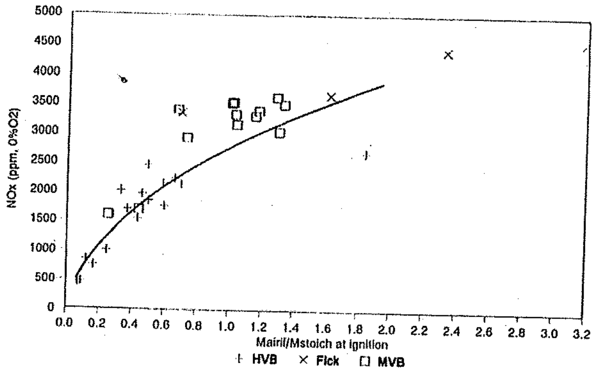
[3] Investment cost in 106 euros and operating cost in euros/tonne clinker, normally referring to a kiln capacity of 3000 tonne clinker/day and initial emission up to 2000 mg NOx/m^3.

Low-NOx Burner

By definition a low-NOx burner injects fuel and air into the kiln through concentric tubes. The primary air proportion is reduced to some 6-10% of the Stoichiometric air requirement. Axial air is injected at a high velocity, i.e. >250 m/s. Coal is normally blown through a middle channel at a lower velocity, 20-30 m/s. A third channel is used for swirl air, its swirl being induced by vanes and/or bluff body. This type of burner design requires an indirect firing system when using pulverized fuel. Great care must be taken when designing a low primary burner as discussed in earlier sections.

The net effect of this burner is to produce very early ignition, especially of the volatile matter in the fuel, in an oxygen deficient atmosphere, and this will tend to reduce NOx formation. Figure 9.19 below shows the results from a research program in 1992 called CEMFLAM which was funded by the cement industry and conducted at the International Flame Research Foundation (IFRF).

Figure 9.19: Effect of Ignition on NOx Formation



The research produced a comprehensive analysis of how NOx is formed in cement kiln flames. The effects of burner design and fuel type we extensively tested and documented in the research report. In the figure above, data is shown for high volatile bituminous coal, flexicoke, and medium volatile bituminous coal.

Staged Combustion

Staged combustion is carried out in preheater and precalciner systems where the fuel can be introduced in a reducing atmosphere. Typically the most effective reducing atmosphere found in the process is from the vitiated gases exiting the kiln as shown in figures 9.20 and 9.21. The strategy employed is to decompose a portion of the nitrogen oxides generated from the sintering zone. Further attempts are made by introducing the calcining fuel in direct contact with the raw meal. The raw meal acts as a heat sink and minimises local temperatures to produce "flameless combustion". Some modern well optmised pants achieve emission levels below 500 mg/Nm^3 with multi-staged combustion.

Figure 9.20: KHD Pyroclon Calciner

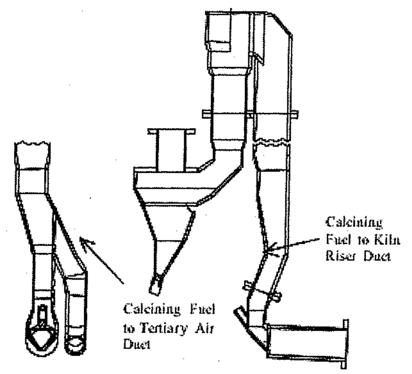
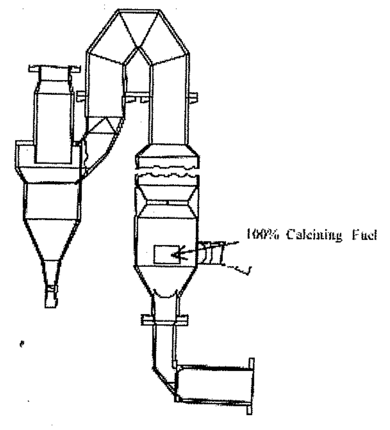


Figure 9.21: FLS ILC Calciner



Catalytic Reduction of NO using ammonia or urea is described by Steuch et al (ZKG; 1/1996, pg 1) while Polysius have developed activated carbon filter technology (Polvitec) to scrub NOx, SO_2, NH_3, and other pollutants from kiln exhaust (Rose & Bentrup; ZKG; 4/1995, pg 204).

Carbon Monoxide Formation

Carbon monoxide (CO) is highly toxic and is the result of incomplete combustion. There is always some CO because mixing processes are not perfect and the reaction between CO and oxygen to produce CO_2 is reversible. CO may also be formed as one of the primary products of fuel pyrolysis during rapid heating of the fuel, before being oxidised. However, in good combustion systems CO should be limited to a few ppm, normally in the range of 20-50ppm. Slightly higher levels may be tolerated in specific circumstances, but levels of 1,000ppm or more usually result from poor fuel/air mixing or starving air (high fuel/air ratio) and are the symptoms of serious problems within the combustion processes. In addition to its toxicity, there is also a heat penalty when CO is emitted; carbon burning to CO yields only 56% the heat released by oxidation to CO_2 (see Sec 9.1 "Incomplete Oxidation of C", and Figure 9.12).

Unfortunately, CO tends to increase when NOx reduction techniques are applied, since these generally achieve their objectives by delays in fuel/air mixing.

Spasmodic injection of fuel is intuitively offensive but is sometimes driven by cost considerations as in the burning of whole tires. Inevitably the effect is either de-rating of the kiln in order to ensure adequate oxygen, or pulses of CO as peaks of combustion exceed available oxygen. Neither option is desirable.

Sulphur Compounds

Sulphur compounds are emitted where sulphur bearing fuel (oil, coal etc.) is burnt or where sulphur bearing materials are processed. Sulphur dioxide (SO_2) is the most common sulphur compound emitted. It is acidic and corrosive and causes major problems with corrosion in cooler areas of the plant and on the surrounding environment. Where conditions are favourable (high oxygen concentrations and moderate temperature) sulphur trioxide (SO_3) can be formed. This is even more corrosive than SO_2 and leads to sulphuric acid formation with the water from combustion, at the so called acid dew point. This leads to severe corrosion in flue gas ducting.

Hydrogen sulphide (H_2S) is extremely toxic and has a nauseating smell. It is formed by reducing conditions in the flames of sulphur bearing fuels, or is driven off from sulphur bearing feeds, such as some clays.

This chapter on combustion and heat transfer is based upon material developed by, and proprietary to, the International Kiln Association. For further information, or for assistance with combustion related questions, please contact the IKA (address is given in the appendix).

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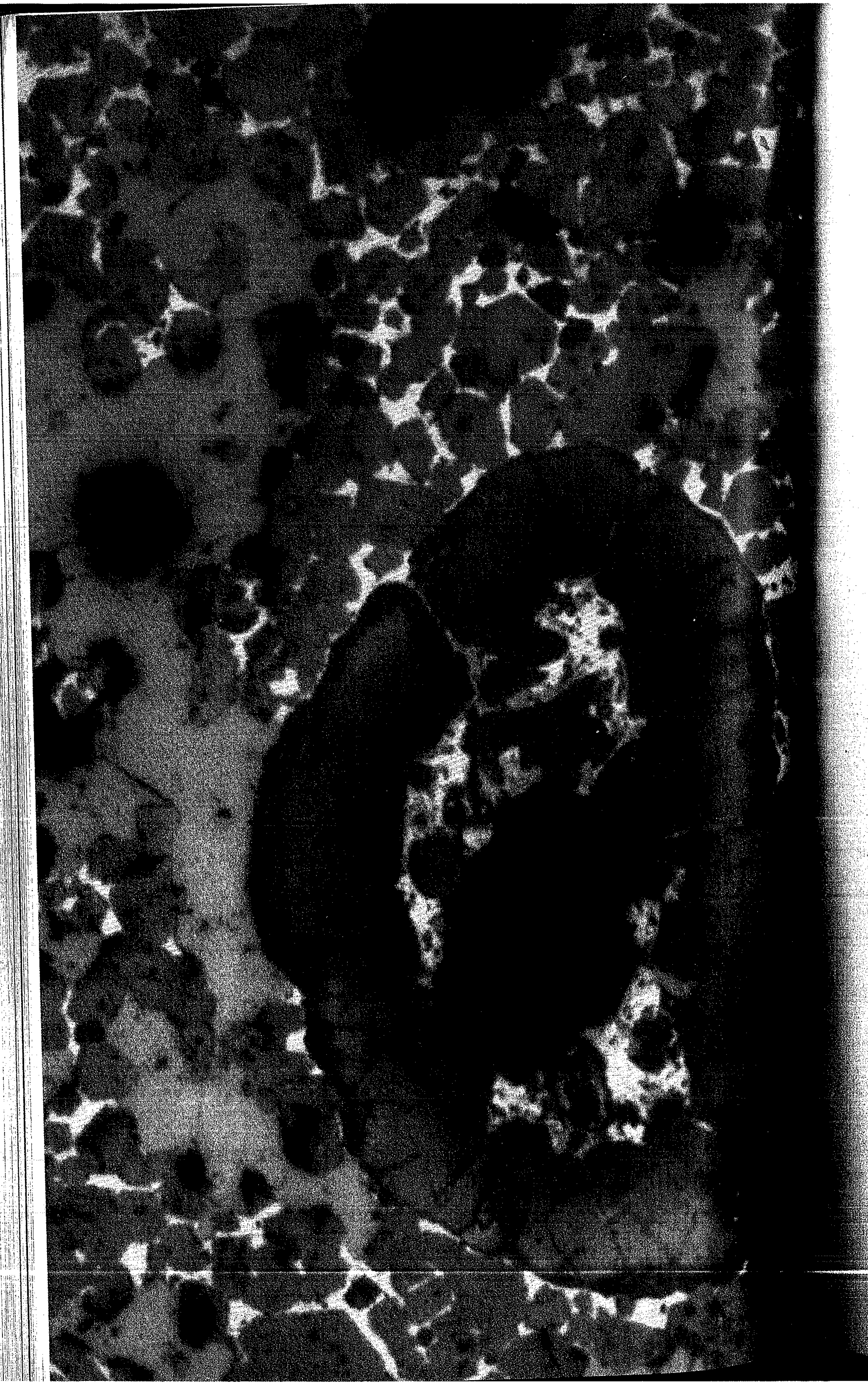


10 Hydration of Portland cement

Portland cement sets and hardens due to exothermic hydration reactions. Hydration products include colloidal gels and sub-micron crystals. The crystals may be fibres, needles and plates which form an interlocking microstructure that can be observed with the aid of a scanning electron microscope (SEM). Both chemical and physical processes are, therefore, involved with the two standard explanations of cement setting and strength development: Le Chatelier's interlocking crystal theory and Michaelis's colloidal gel theory (Benstead; WC; 8/1991, pg 21: Chen & Odler; Cem and Conc Res; Vol 22; 1992, pg 1130).

The reactivity of cement phases with water are in the order aluminate > alite > ferrite > belite. C_3A is the most soluble of the major compounds and appears to dominate early hydration. Aluminate and, particularly, silicate hydration reactions are extremely complex and many undoubtedly contribute to setting and strength gain of cement (see Lea ibid, pg 241).

Cement hydration can be approximately divided into four stages (see Fig 10.1).

Initial Stage – Within seconds of contact with water, alkali sulphates dissolve and, within minutes, calcium sulphate reaches saturation. Dissolved aluminate and sulphate react to form ettringite ($C_3A.3CaSO_4.32H_2O$) which is precipitated, normally on the surface of cement particles. Alite dissolves slightly and a calcium silicate hydrate gel (C-S-H) forms on the surface of alite particles. These hydration product coatings block further reaction and initiate a dormant period. This initial period is characterized by heat release. The rate of dissolution of sulphate relative to aluminate is critical to prevent early stiffening of cement paste. A deficiency of sulphate in solution leads to uncontrolled hydration of C_3A to $C_4A.14H_2O$ and $C_2A.8H_2O$ which crystallize to cause flash set. An excess of soluble sulphate leads to precipitation of gypsum, syngenite, or ettringite in pore solution causing flash or false set. A discussion of sulphate properties and their effects upon paste and concrete performance is given by Hills (WC; 7/2004, pg 115).

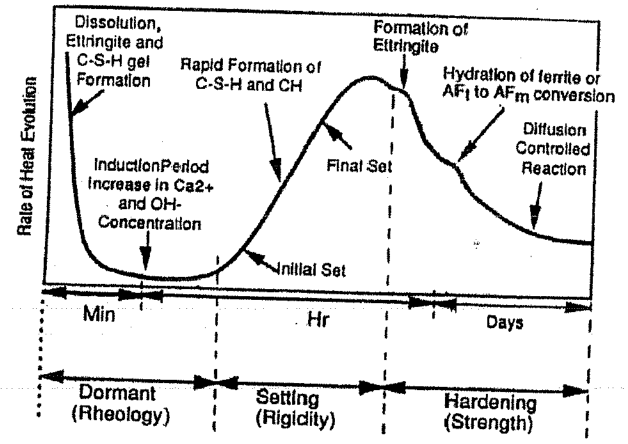
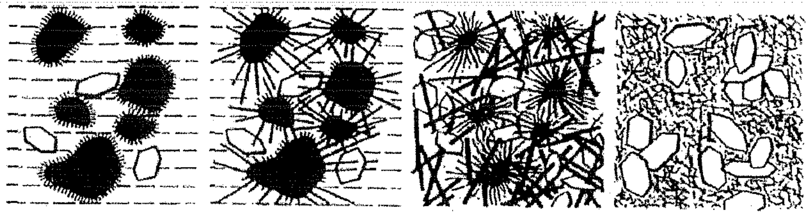


Figure 10.1: Rate of Heat Evolution and Structure Development of Portand Cement Hydration



Induction or Dormant Stage – This stage lasts 1 to 3 hours while the hydration of all clinker phases progresses slowly. In the early stages, Ca^{2+} in pore solution reaches super-saturation with respect to $Ca(OH)_2$, and nucleation and growth of both C-S-H and $Ca(OH)_2$ begin. At this stage of structure development, thin shells of C-S-H and a few ettringite rods develop around clinker particles. The subsequent decrease in the concentration of Ca^{2+} in solution and the rupture of coatings trigger renewed acceleration of alite dissolution and heat release. The continuous deposition of C-S-H, $Ca(OH)_2$ and other hydration products causes bridging between particles and reduces paste porosity. This signals the onset of setting.

Acceleration Stage – This stage begins with initial set and continues with rapid hydration of alite into C-S-H and Ca(OH)2. These reactions are accompanied by intense heat release which reaches a peak approximately 12 hours into hydration. More ettringite is formed as gypsum is depleted and the ettringite subsequently dissolves and reacts with Al(OH)4- to form monosulphoaluminate ($C_3A.CaSO_4.12H_2O$). Depending upon the relative amounts of gypsum and aluminate, monosulphoaluminate formation may be completed before or after the peak of alite hydration. Final set is reached before peak heat release from alite hydration.

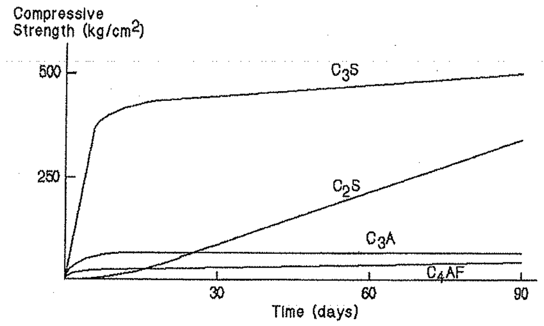
Deceleration Stage – This spans many days and is characterized by low heat evolution and a decreased overall rate of reaction as the reacting species become used up and diffusion slows with decreasing porosity. Belite becomes the primary hydrating phase. C-S-H is believed to undergo polycondensation of the SiO_4 tetrahedral chains with progressively increasing strength.

It has been observed that setting is largely independent of C_3A concentration and it is now believed that both setting and strength development are largely caused by hydration of C_3S to tobermorite, a C-S-H gel of variable composition.

Hydration of cement typically involves combined water of about 22% relative to clinker weight. Given that a normal water:cement ratio in concrete is 0.4-0.5, it is clear that excess water must be used for workability and that this excess water causes strength loss. The relative contributions to strength development are shown in Fig 10.2.

Pozzolanic activity is the result of the reaction of soluble SiO_2 from the pozzolan with CaO in solution. As free CaO will always be present in solutions in contact with hydrated cement, pozzolanic reactions provide "self-curing" of cracks in pozzolanic concrete.

Figure 10.2: Strength Gain of Cement Compounds



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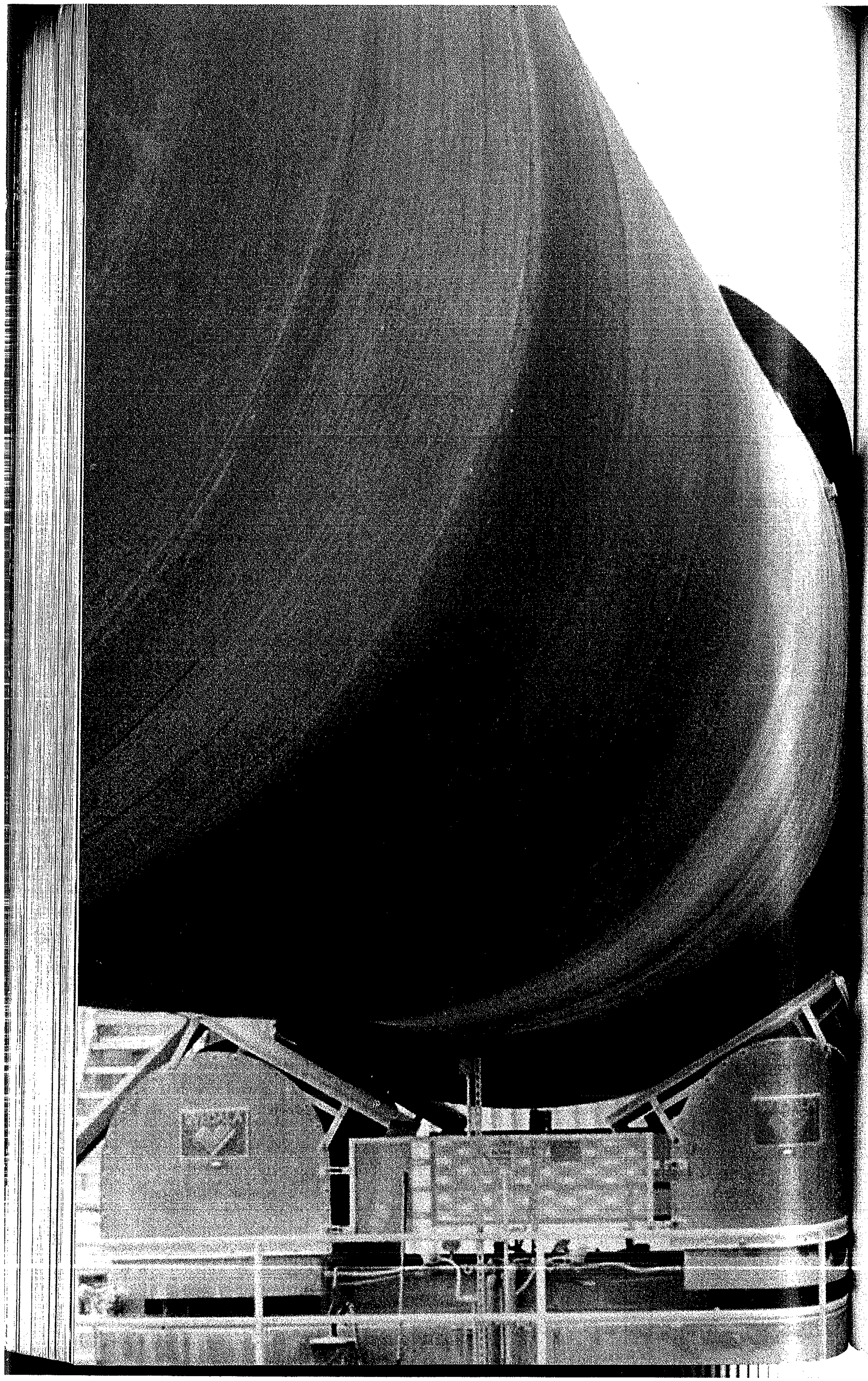


11 Other Kiln types

- Long Wet
- Grate Preheater
- Fluidised Bed
- Long Dry
- Vertical Shaft

The earliest cement kilns were vertical shafts in which mixtures of raw materials and solid fuel were burned in a natural draft of combustion air. Ransome introduced the rotary kiln in the 1880s and this allowed more uniform heat transfer and controlled clinker burning. Initially rotary kilns used slurry feed – the wet process – as this facilitated raw material grinding and homogenizing. In certain areas of the United States, shortage of water led to a variant, the long dry kiln, which required, and resulted in, improved pneumatic blending systems. It was acknowledged that, while the rotary kiln was an excellent device for heat transfer and materials handling at clinker burning temperatures, it was inefficient for preheating and calcination. The first alternative approach to preheating was the Lepol, or grate preheater, system where nodulised raw materials are conveyed on a travelling grate permeated by hot kiln exhaust gas; with appropriate raw materials this process is successful. Ultimately, however, it was determined that the most efficient low temperature heat exchange and calcination can be effected in air suspension and this led first to the cyclone preheater and later to the addition of separately-fueled precalcination. These last two systems now predominate and have been the substance of this book.

Typical comparative data, with considerable variation, is as follows:

	Maximum rating (t/d)	Specific fuel (kcal/kg)	Length: Diameter
Shaft kiln	200	900-1000	
Long wet kiln	2,000	1200-1500	32-38
Long dry kiln	2,000	900-1200	32-38
Lepol kiln	2,000	800-900	14-16
Cyclone preheater kiln	2,000	800-900	14-16
Precalciner kiln	11,000	700-850	11-16

It has also been observed that the grindability of cement differs significantly with kiln type. Relative power consumptions for clinker types are:

Lepol kiln	100 (softest)
Cyclone preheater kiln	107
Long wet kiln	112
Long dry kiln	117 (hardest)

11.1 Long Wet Kiln

Long wet kilns were predominant until the appearance of cyclone preheaters in the 1950s. They are now obsolescent though they may still justify their existence where they are fully depreciated, where the market demands only a small production capacity, and where fuel is cheap. Wet kilns also avoid the need for drying of naturally wet raw materials and the homogenizing of slurry is still usually more effective than the blending of dry raw meal.

Raw materials are milled with addition of water to a total of, typically, 30-35% by weight, to form a slurry which is stored and blended in tanks with continuous agitation (rotating rake augmented with air jets) before feeding to the kiln. Water is adjusted to produce a consistency which allows ease of conveying without segregation. As evaporation of the water involves a considerable heat penalty, use of water reducing agents may be justified; 1% water reduction is equivalent to about 15kcal/kg clinker. An approximate correlation of slurry density to water content is:

30% water by weight	=	$1220kg/M^3$
32%	=	1160
34%	=	1100

To enhance evaporation of water by increasing surface area for heat exchange, to facilitate the handling of feed as it transitions from slurry through plastic material, and to detrain dust from kiln exhaust gas, chain systems are hung within the kiln shell (Figure 11.1). A typical system would comprise one to two diameters of bare shell followed by one diameter of curtain chains as a dust curtain (curtain chains are lengths of chain about 75% the diameter of the kiln and attached at one end only in successive circles around the circumference of the shell). Next come some five diameters of spiral curtain chain to break up and convey the drying (plastic) feed down the kiln (Figure 11.1). While curtain chains are easier to manage, garland chains have been claimed to give better efficiency (garland chains are attached to the shell at both ends; the attachments should be 90° apart in a spiral down the axis of the kiln and the chain should hang slightly below the centre line). Usually there is a second section of bare shell near the down-hill end of the chains to reduce circumpherential imbalance in gas temperature and material conveying. Duda recommends chain design parameters of 12% of daily clinker production for total chain loading and surface area of $6\text{-}8.5M^2/M^3$ of chain section volume; de Beus (ICR; 12/1997, pg 41) suggests 15% and $6\text{-}10M^2/M^3$ respectively. Chain consumption is about 100g/t clinker.

The feed material leaves the chain section at 5-10% moisture and proceeds to the preheating, calcining, and burning zones of the kiln. Total material retention time in a long wet kiln is approximately 3 hours and gas is discharged at 150-200°C.

Dust loss with exhaust gas should ideally be 8-10% but is often much higher as kiln production is increased with resulting increase in gas velocity. Return of dust to the slurry system is inadvisable as it frequently causes agglomeration and sedimentation. Up to 5% relative to clinker weight can be returned by insufflation into the kiln flame; beyond this quantity, flame cooling becomes unacceptable. Alternatives are separate slurrying in a vortex mixer and parallel injection with the main feed, and return using scoops which inject the dust slightly downhill from the chain section or into a bare section of kiln near the downhill end of the chains. The basic causes of high dust loss, however, are gas velocity and chain design and condition.

11.2 Long Dry Kiln

Long dry kilns differ from wet kilns primarily in raw grinding and handling and in their lower specific fuel consumption. Within the kiln itself, dry kilns use only curtain chains as the requirement is for heat exchange and dust detrainment rather than for conveying. Usually, 6-7 diameters of curtain chain are employed below about 2 diameters of bare shell at the feed end; approximately half is hung in rings perpendicular to the kiln axis and the lower half is hung in a spiral arrangement. Chain loading is some 10% of daily kiln production. The gas discharge temperature of long dry kilns is typically in excess of 300°C and, if available, water is sprayed into the feed end to reduce gas temperature before dust collection.

11.3 Lepol (Grate Preheater) Kiln

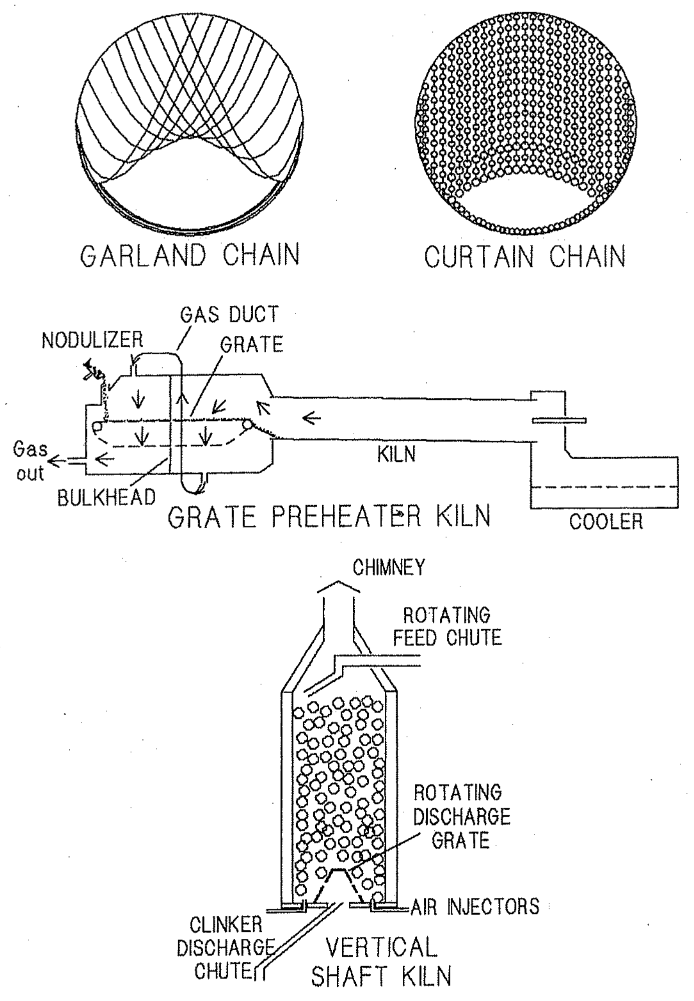
Polysius introduced this system during the 1930s and achieved a dramatic reduction in specific fuel consumption from the wet process. Nodulised feed is conveyed on a travelling grate through which the hot kiln exhaust gas is passed, originally once but, in a later development, twice. The material was preheated to approximately 900°C before entering the kiln while the exhaust gas was cooled to below 150°C, humidified for dust collection, and filtered by the material bed to a low dust concentration. Raw materials may be either wet milled and filter-pressed to yield a cake of about 20% moisture, or dry milled and nodulised in an inclined rotating pan with a water spray to a moisture content of 11-15%. Wet milling offers the possibility of extracting soluble salts such as chloride which may then be removed with the filtrate. If the cake or nodules do not possess good mechanical and thermal stability - usually associated with clay ingredients - there is excessive disintegration on the grate and loss of efficiency. The second (low temperature) exhaust gas pass through the grate dries and preheats the material. The first pass involves an initial gas temperature of about 1000°C and a final temperature below 500°C; this serves to condense volatiles exhausted from the kiln and it was found that if the gas were passed through cyclones between the first and second pass, the collected dust contained a high concentration of volatiles, thereby providing an effective bypass. The material is discharged to the kiln inlet at incipient calcination and the short kiln thereafter operates similarly to a cyclone preheater kiln (Figure 11.1).

11.4 Vertical Shaft Kilns

Shaft kilns originally constituted the only available technology from the beginnings of lime burning which can be traced at least to Greece in the 5th century BC. Since the beginning of the 20th century they have been largely superseded by rotary kilns. However, there remain areas where, due to lack of infrastructure, very small production units are appropriate and where relatively simple construction methods do not demand high cement quality. Such conditions can still favour shaft kilns and many are to be found in China, India, and in a number of developing countries (Reiter; AC; 11/1997, pg 23).

Traditional shaft kilns were basically holes in the ground using mixtures of un-ground feed roughly mixed with solid fuel and burned in batches with natural draft. The lack of feed homogeneity together with non-uniform ventilation gave rise to widely varying temperature and oxidizing conditions so that quality was low and erratic. Considerable advances have been made and Rajbhandari (WC; 1/1995, pg 65) describes Spohn's black meal process as one of the most advanced shaft kiln technologies presently available (Figure 11.1). Practical unit capacities are 20-200t/d.

Figure 11.1: Kiln Types and Chain Systems



Raw materials and solid fuel are ground together and nodulised (black meal process). Alternatively, but less effectively, raw mix and fuel can be ground separately and then blended and nodulised (white meal process). As with the Lepol kiln, stable nodules are important and usually require both a clay component and a solid fuel with less than 16% volatile content. The fuel may be coal, charcoal, coke, or petroleum coke. The kiln shaft is filled with the prepared mix and air is blown into the shaft at and near the base. The material is in turn heated, calcined, and burned at progressively higher temperatures as it moves down the shaft counter-current to the combustion air. Near the base the clinker, with fuel already consumed, is rapidly cooled by the injected air and is discharged through a gate. Production is continuous with new feed added at the top to balance discharge.

The process is, therefore, basically similar to that of rotary kilns. The principal difference is in uniformity; the rotary kiln ensures that the material is constantly agitated and that all material is subject to the same retention time and heat transfer. In the shaft kiln, however, there is a definite thermal gradient with the core material reaching a maximum temperature of ca 1450°C, some 200° higher than material at the walls. Differential melting of the material tends to increase air flow velocity at the walls which reinforces heat loss though the walls to exacerbate the difference. More sophisticated shaft kilns can compensate for this with increased peripheral fuel concentration and reduced wall heat loss. Retention time above 1250°C is typically 30 minutes. Increasing the air flow through the bed both increases production rate and clinker quality; the necessary air injection pressures (1000-2500mm WG) require an efficient air lock on the clinker discharge; either a triple gate or a controlled choke flow (seal leg). In practice, the seal leg is too dependent upon clinker bulk density and porosity to be effective and the triple gate is preferred. In recent years there has also been a trend to increased diameter and reduced height to an aspect ratio of 2.5-3.0 but the diameter is usually limited to about 2.4M as increasing diameter makes uniform air distribution more difficult to achieve. Modern shaft kiln designs can be fully instrumented and PLC controlled.

11.5 Fluidised Bed Kiln

Kawasaki has been working to develop a fluidised bed kiln and has reported on the operation of a 200t/day unit (Hashimoto; ZKG; 1/1999, pg 1). The equipment comprises a cyclone preheater, calciner, granulation fluidized bed , clinker fluidised bed, and packed bed cooler all in the same tower. The granulation bed is controlled at 1300°C and the clinkering bed at 1400°C. Advantages of the process are:

- Low heat consumption due to high cooler efficiency and low radiation loss
- Low NOx emission
- Low CO_2 emission due to reduced specific fuel consumption
- Low capital cost
- Flexibility for the production of different clinker types

Problems are still being addressed in controlling the granulation process, in avoiding coating build-up, and in achieving stable operation. If these obstacles can be overcome, commercial application may be attractive.

■ ■ ■ ■ ■

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 either for producing blended cements or for grinding bought-in clinker.

Kiln run-factor – Various definitions have been encountered including fire-on time and running time exclusive of planned shut-downs. 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, 5 consecutive days of operation. A base run of other than 5 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 maximizing 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). Utilization facilitates 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 if feed is off for more than, say, 3 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

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.

12.3 Inventories & 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 weighfeeder. 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, pg 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 one of which may cause the kiln to be shut down. It is much easier though, unfortunately, seldom as rewarding, to keep an old wet kiln in operation.

The purpose of downtime analysis is not only to monitor overall availability but, more importantly, to identify and quantify reasons for breakdown. It has, however, a pragmatic intent and pedantic accounting is not necessary. Usually, there will be an overriding reason for shutting down, though other tasks will be performed during the same period. It is suggested that the main reason only need be recorded though the other tasks should be noted in order to explain anomalous totals. For example, if a kiln is shut down for four weeks due to high cement inventory, and major re-bricking and maintenance are completed at the same time, the ascription should be voluntary downtime, but the subsequent low annual downtime must not be incorrectly construed.

Any kiln voluntary downtime should be entirely due to excess clinker/cement inventory due, in turn, to depressed shipment. By design, the kiln should be the limit on plant production and it should never be slowed or shut down either for lack of kiln feed, or for lack of clinker and cement space. If this does occur, then either maintenance or the capacity of secondary equipment is inadequate and should be reviewed. Likewise, there should be substantial voluntary downtime or spare capacity on all equipment other than the kiln.

12.5 Miscellaneous Reporting

Exception reporting should be incorporated into automated data handling so that any abnormalities of process rate, efficiency, down-time, spare part consumption, and cost are quickly identified.

Operator's data logging is useful even when data is recorded automatically as it helps to bring the operator's attention to process changes. Either the operator or supervisor should also keep a narrative log of equipment shut-downs (with detailed explanation), alarm faults, requirements for maintenance or process investigation, process inventory situation, personnel accidents, and any other matters that require management attention.

Equipment numbering should allow easy identification and location. A typical system comprises area (eg kiln = 4), equipment type (eg bucket elevator = BE), and a serial number; thus 4BE2. All maintenance and cost records should refer to equipment number.

12.6 Typical Daily Production Report

(All data should be entered for both day and month-to-date)

Materials Balance:	**Opening**	**Prod/Recd**	**Cons/Disp**	**Closing**
Stockpiles:				
Limestone				
Shale				
Silica				
Iron ore				
Gypsum				
Coal				
Silos:				
Limestone				
Shale				
Silica				
Iron ore				
Raw meal				
Coal				
Clinker				
Gypsum				
Slag/fly ash				
Cement, total				
Type 1				
Type.....				

Plant Operation:	**Prod**	**Op Hrs**	**t/H**	**kWh/h**	**kWh/t**	**kcal/kg**	**DT-hrs**	**DT reason**
Primary crusher				–	–	–		
Crusher-dryer				–	–	–		
Raw mill						–		
Kiln				–	–			
Finish mill						–		

12.7 Typical Process Summary Data

Crushing & Drying	**Month**	**Year to date**	**Previous year**
Primary Crusher, t/H			
Crusher – Dryer, t/H			
Operating time, %			
Fuel usage, kcal/kg CL			
Production, Limestone			
Shale			
Feed material moisture, %, Limestone			
Shale			
Product moisture, %, Limestone			
Shale			
Raw Milling			
Production, t, Raw			
Operating time, %			
Production rate, t/H			
Mill, kW			
kWh/t			
Feed moisture, %			
Meal fineness, % -170#			
Meal moisture, %			
Material usage, % Limestone			
% Shale			
% Silica			
% Iron ore			
Ball usage, g/t			

Process Summary Contd.

Burning and Cooling	**Month**	**Year to date**	**Previous year**
Production, t clinker			
Operating time, %			
Production rate, t/H			
Specific heat, kcal/kg clinker			
Coal, t/H			
Precalciner fuel, %			
Kiln exhaust gas, °C			
Preheater exhaust gas, °C			
Average clinker free-lime, %			
Feed end, % LoI			
% SO_3			
Coal, % moisture			
% ash			
kcal/kg (NCV, adb)			

Finish Milling			
Production, t, total			
Type 1			
Type....			
Operating time, %			
Production rate, t/H			
Mill, kW			
kWh/t, total			
Type 1, etc			
Gypsum usage, %			
Grinding aid usage, g/t			
Ball usage, g/t			

Process Summary Contd.

Electrical Power Summary	**Month**	**Year to date**	**Previous year**
Crushing & drying, kWh/Eqt			
Raw milling			
Blending			
Burning & Cooling			
Finish milling			
Packing & shipping			
Utilities & Miscellaneous			
Total			

Quality Summary			
Kiln feed, Type 1, C_3S			
C_3A			
SiR			
Clinker, Type 1, C_3S			
Free CaO			
SO_3			
Cement, Type 1, C_3S, %			
C_3A, %			
SO_3, %			
Blaine, cm^2/g			
+325#, %			
Initial set, hrs:mins			
Strength, 1 day, kg/cm^2			
3 day			
7 day			
28 day			

One of the key practical differences between Financial accounting and Management accounting is the process from revenue to operating profit.

Financial Accounting	**Management Accounting**
Revenue	*Revenue*
Less Manufacturing Costs	Less Variable Costs
(Variable and Fixed Manufacturing)	(Variable Manufacturing, Variable SG&A*)
= Gross Margin	*= Contribution Margin*
Less SG&A*	Less Fixed Costs
(Variable and Fixed SG&A)	(Fixed Manufacturing, Fixed SG&A*)
= EBITDA	*= EBITDA*

(*SG&A = Sales, General & Administrative)

Manufacturing costs, also referred to as Cost of Goods Sold (COGS), represent direct materials, direct labour, and plant rent. They can be classified as either fixed or variable.

Overhead represents costs that are neither direct materials nor direct labour. In a sense, overhead is the 'garbage can' of cost accounting. The allocation of overhead can be controversial. Under conventional costing systems, unit overhead costs are typically allocated to products as, say, $/Ton calculated from (total overhead cost/quantity produced). When multiple products are being made, eg, Type I, II, III, oil-well cement, etc. overhead allocation becomes more arbitrary resulting in cross subsidy of one product by another using political rather than technical considerations.

With computerized information systems and increasingly competitive markets, there is a good argument for improving cost allocation using methods such as activity based costing.

Activity based costing derives the cost of a product as the sum of the activities that occur to make this product. This is not a new concept but rarely seen in the cement industry due to the relative similarity of the products. However, as plants become more responsive to customer demands, a true determination of cost-by-type becomes essential.

Variable costs, by definition, increase with each unit of clinker or cement produced and sold. These typically include fuel, power, raw materials, and refractory.

Fixed costs are loosely defined as 'all other costs' that are committed one year ahead. These are typically salaries and wages, burden, over time, maintenance and supplies, external labour and services, and other plant overhead such as rent. The inclusion of depreciation in fixed costs is not universal so that care should be exercised in comparing total manufacturing costs.

Full Absorption Cost is the variable cost plus an allocated portion of fixed costs. Using full costs to make short-term decisions involves the implicit assumption that the full cost figure estimates the incremental impact of the decision. This is usually wrong and referred to as the full cost fallacy.

SG&A (Sales, General, & Administrative) expenses include sales commissions, order processing costs, office rent, CEO's compensation, and corporate jet.

Depreciation
The recognition that an asset has a multi-year life and that, due to wear and tear or obsolescence, its value diminishes with time. A deduction from annual profit is taken to balance this loss in value. The usual calculation is by the straight line method; the estimated residual or scrap value is deducted from the original cost and the difference is divided by the number of years of useful life. Alternatives are *the reducing balance method* and *the inflation-adjusted method*. All of these are somewhat arbitrary.

EBITDA, earnings before interest, taxes, depreciation, and amortization, is used interchangeably with operating profit.

The profit equation which the cement plant operator is managing can be shown as:

$$\pi = (P - V) \text{ x } (X - F)$$

where,

π = profit (including or excluding SG&A allocation)
P = price
V = unit variable cost
X = quantity produced
F = total fixed costs
$P - V$ = contribution margin

For the cement plant operator and sales force, *the contribution margin* is a critical number to track.

The target profit volume then becomes written as

$$X = (F + \pi) / (P - V)$$

The break-even volume at which profit = 0:

$$X^{\text{break even}} = F / (P - V)$$

Operations managers often fail effectively to apply the above equations. One example would be the conversion of a kiln to a lower cost fuel which de-rates the kiln (eg gas, high moisture, high sulphur). Assuming the kiln is ID fan limited, such a decision can result in reduced clinker capacity and, in the case of high sulphur fuel, to reduced gypsum addition. As a result, while the variable manufacturing cost (V) may be reduced, cement quantity is also reduced and profitability (π) may suffer.

There is no practical alternative use for a cement plant so that it is reasonable to argue that fixed costs (F) are essentially sunk and should be spread over as large a production as possible. However, while maximizing the contribution margin may involve minimising variable cost in a depressed market, in a sold-out market it may dictate accepting a higher variable cost in order to achieve a higher production rate. Indeed, oxygen enrichment and the use of slag as a raw material, both of which can significantly increase kiln capacity, may be profitable strategies even though they raise the incremental variable cost above the break-even point.

Price wars are self-defeating. This used to be based on the argument that while we lost a dollar on every Ton, we could make it up with volume. Perhaps we have learned, but perhaps not. If a company's strategy is to increase its market share it should do so by:
- investing to reduce manufacturing cost, or

– increasing supply by plant construction, plant acquisition, or by importing cement to the market

An operating budget should be prepared before the start of each financial year. Management estimates budget by month and by cement type; the limitation or 'principal budget factor' is usually either market or equipment capacity. Each department estimates its own detailed costs, and the combined estimate should be reviewed in the light if previous actual costs and anticipated process and cost changes. Few manufacturers operate in sufficiently stable environments to avoid significant variances. Flexible budgeting allows provision for different levels of production.

Comparison of actual results with budget will indicate any variances. Variances will be due either to change in volume or change in cost/price and, whether favourable or adverse, must be explained, and, if necessary, corrected.

1) *Volume Effect* which is the effect of a change in plant production on sales revenues and costs. This is normally defined as:
   Net Volume Effect = Volume Sales \$ - Volume Cost \$ where,
   where: Volume Sales \$ = Current Year (CY) Sales @ Prior Year (PY) \$ - Prior Year Sales
   Volume Change % = CY Sales @ PY\$/PY Sales
   Volume Cost \$ = Volume Change % x PY Cost

2) *Price Effect* is the effect of output prices on total sales revenue, defined as,
   Price Effect = CY Sales - CY Sales @ PY\$

3) *Cost Inflation* is the effect of changes in input prices on total cost, defined as,
   Cost Inflation = CY Cost - CY Cost @ PY\$

4) *Productivity* is technically a plug number (ie the number required to effect balance) as shown in the model below.

All operating profit variances from one period to the next can be explained by a combination of changes in price, inflation, volume, and productivity :

Current year operating profit = Prior year operating profit + net volume effect
+ price effect
+ cost inflation effect
+ productivity (plug)

While budget reconciliation is a valuable operations management tool, an accurate budget is also essential for cash flow management.

Cost by type of cement should be estimated as accurately as possible in order to establish the net margin for each type. This requires identification and quantification of all associated cost elements including transition, product transfer, and storage costs. and involves overhead allocation as discussed above. Special cements involving only finish milling and additives can be quite easily estimated, special clinkers which involve intangibles such as kiln refractory life and mechanical wear and tear can not.

Costing of spare parts and purchased materials inventories can employ various methods (FIFO, LIFO, standard cost, etc); with modern data processing, weighted average costing is probably the most satisfactory.

Order levels and order quantities should be reviewed frequently. A perpetual warehouse inventory audit should be conducted and, periodically, obsolete items identified and separated from the stock-list. In plants which have acquired a variety of equipment at different times and from different suppliers, it is common to find a given part separately stocked under two or more warehouse numbers; data bases are available for identifying duplications.

While automatic order generation is desirable, any items costing more than some predetermined minimum amount (say \$1,000) should be directed for management approval prior to order. An acceptable turnover rate for maintenance parts and supplies, on a cost basis, is 1.5 per year. This does not include critical, slow moving spares often carried to prevent significant business interruptions; such spares are better considered insurance and assessed using risk management principles. The value of warehouse inventory should typically be \$2-\$3 per tonne of annual clinker production.

Major spare parts should be kept under review. Slow wearing parts such as mill diaphragms should be monitored and ordered an appropriate time ahead of anticipated failure. Parts subject to catastrophic failure such as motors, kiln rollers, mill girths, and fan impeller shafts require judgment to balance their high inventory cost against potential production loss. If possible, equipment should be standardized so that one spare can be borrowed from a less critical unit (e.g. finish mill to raw mill); a record should be kept of which gears have been turned and how many hours they have operated and contingency plans should be made for emergency repair of large shafts and rollers. Reciprocal agreements between plants with similar equipment may also be possible but advanced planning is desirable.

An annual capital budget should be prepared covering items and projects exceeding a predetermined minimum (say \$50,000) which are not included in routine operating or maintenance costs. Justification is conventionally based on capacity increase, cost reduction, maintenance, safety, quality improvement or compliance with environmental regulations.

An annual capital budget may be of the order of \$2-\$3 per tonne of production. Whether long-term maintenance items such as mill linings are considered, capital or expense will depend upon company policy and tax treatment. Large projects (exceeding, say \$2,000,000) are conventionally considered separately.

A long-term plan, usually for 5 years and updated each year, is a valuable precursor and basis for programming and approving capital investment. The principal inputs are economic, market, price, and competition projections which are required for estimating future benefits and, therefore, the IRR of an investment. The plan should also review raw materials reserves, plant obsolescence, the availability and projected cost of fuel and power, the development of local infrastructure and surrounding property, and any anticipated changes to use permits or environmental regulation.

Any discussion on managerial accounting in the cement industry would be inadequate without reference to transfer prices. Globalisation has made this aspect of accounting more relevant and controversial than ever. Transfer prices split profits among divisions, so if total revenues are fixed, transfer prices do not affect profits (although they can distort the performance of divisions). However, profitability can be significantly enhanced by the appropriate transfer of funds between countries with different rates of taxation. In international companies, effecting this balance within the legal framework is an important task of the finance department.

13.2 Investment Justification

Plant investment may be justified by:

- Safety
- Environmental regulation or permit compliance
- Payback

Payback projects require cost reduction or capacity increase. Capacity increase is, in fact, usually essential to justify significant projects but requires that the market can accept the increase without reducing selling price. A project proposal conventionally requires:

- Description of present situation with capacity limitation, operating problems, excessive energy consumption, excessive maintenance, equipment downtime, quality problems, etc.
- Proposed remedy indicating scope, anticipated benefits, compatibility with possible future plant changes or capacity increases, requirement of permits, etc.
- Alternatives considered.
- Budget.
- Risk assessment (technical risk, cost over-run, etc).
- Schedule.
- IRR, NPV, payback, etc.

Justification of plant modification and new equipment costs involves cash flows, estimation of uncertainty and risk, and the opportunity cost of alternative investments. Four ways to reaching a yes/no decision are:

1) Simple Payback
2) Net Present Value (NPV)
3) Internal Rate of Return (IRR)
4) Real Options

Each has its own advantages and disadvantages. In practice, NPV and IRR can be easily calculated using a business calculator or a standard computer program such as Excel.

Simple payback, also known as straight-line payback, is convenient for a back-of-the-envelope calculation of the time a company must wait before recouping its investment. It ignores the time value of money.

Payback Period (years) = Total Cash Outflow / Forecast Annual Cash Inflow

Payback is a crude measure but is useful for periods of less than 2 or 3 years.

The remaining methods take into account the time value of money and are useful if the investment is expected to produce benefits beyond 3 years.

Present value reflects the notion that a dollar today is worth more than a dollar in the future. It takes into account reduced purchasing power in the future due to inflation, uncertainty involved in the receipt of a forecasted dollar, and most importantly, the opportunity cost of not investing in the next best alternative. It does this by discounting forecasted cash flows.

So many conflicting definitions of *cash flow* exist today that the term has almost lost its meaning. For the purpose of investing in plant equipment, two types of cash flow are relevant: *free cash flow* and *discounted cash flow.*

Free cash flow recognizes the importance of reinvesting cash back into the plant in the form of capital expenditures and working capital. Here is a working model:

EBITDA
less Taxes paid
Change in net fixed assets (capital expenditures – depreciation) (+/-)
Change in working capital (current assets – current liabilities) (+/-)
= Free Cash Flow

Discounted cash flow (DCF), used to estimate the present value of a project, is a sum of money today having the same value as the projected free cash flows.

$$PV\,(CF_t) = \sum_{t=1}^{T} \frac{CF_t}{(1+k)}$$

Net present value (NPV) takes into account the initial investment, I_0.

$$NPV = -I_0 + \sum_{t=1}^{T} \frac{CF_t}{(1+k)^t}$$

Discount rate, k, for the purpose of this formula is the company's weighted average cost of capital (WACC) which is a function of its capital structure, e.g. debt and equity, and tax rate. Firms in the cement industry typically carry a lot of debt, mostly rated just above basic investment grade. WACC usually ranges between 11% and 15% depending on the company's credit rating.

In theory, value is created with positive-NPV projects and so the general rule has been to accept a project with a NPV exceeding zero by a margin reflecting the perceived risk of the project (technical, budget, schedule, etc).

Note that the discount rate at which the NPV is zero is called the Internal Rate of Return (IRR).

Use of APV is considered appropriate for projects that incur high initial debt which is paid down rapidly.

Internal Rate of Return (IRR) takes the sum of future benefits, after tax, and calculates the interest at which the total equals present project or acquisition cost. There is an IRR Rule that supports investment in a project only if the IRR is at least as great as the opportunity cost. Typically, if IRR > WACC, then the project should be accepted. The WACC is often referred to as the minimum 'hurdle rate' for investments. The required margin of excess is related to the perceived risk involved.

IRR and payback are calculated (using typical numbers):

Capital investment	$4,000,000
Avoided cost or benefit/year	$1,000,000
Operating cost	($200,000)
Depreciation (say 10 years)	($400,000)
Pre-tax savings	$400,000
Less tax (say 35%)	($140,000)
Add back deprecation	$400,000
After-tax cash flow	$660,000

Then payback = $4,000,000 / $660,000 = 6.1 years
IRR = 10.3%

Some projects are essentially discretionary (paving, office refurbishment, etc) where benefits are virtually impossible to cost justify. Another class of projects (on-line analysers, blending upgrades, etc) involve a recognisable benefit but with a difficult quantitative correlation. Wherever possible, it is suggested that an IRR should be calculated even if the benefits are 'soft' such as an estimation of a 1% capacity increase or a 2% cost reduction. This also applies to replacement and major maintenance projects where it is usually possible to estimate an avoided loss. If such a process is not applied it becomes very difficult to prioritise a large group of unjustified projects.

Real Option Pricing recognises that a manager has the 'option' to make any of the following decisions at any time:
- Defer the investment to a later date.
- Abandon the investment if cash flows do not meet expectations.
- Modify the scale of operations as demand varies.
- Alter the mix of inputs such as change in raw materials prices.
- Make follow-on investments if the initial investment is successful.

Examples of the above are known as real options in recognition of their formal equivalence to traded financial options (See Dixit & Pindyck; Investment Under Uncertainty; Princeton University Press, 1994). One of the most useful applications of this approach is the decision to invest in a single pyro-processing line as stage 1 with the option to make a follow-on investment in a second line as stage 2. Management has the option to defer stage 2 until after seeing what the results are from stage 1. The expected value of success or failure of stage 1 can be factored into the NPV of the investment.

NPV of Investment = NPV (base case) + Value of Real Options

It is entirely possible that the value of the real options more than compensates for a negative base-case conventional NPV project. The moral of this approach is clear: failure to recognize and value real options implicit in corporate investments will make executives unnecessarily timid in the face of high-risk, high-return opportunities (Economist; 8 Jan 1994). Conversely, excessive valuation of real options can lead to acceptance of bad projects.

13.3 Capacity Increase by Process Change

There are a number of methods for increasing kiln capacity which involve incremental operating costs rather than capital investment (eg oxygen enrichment [Sect 4.1] and the use of calcined raw materials [Sect 2.1]). The use of activated pozzolanic extenders may also increase plant capacity by reducing the clinker content of cement.

There are various ways to assess the economics of such systems:
- Ideally the process change should not increase average cost of production. While the variable cost will almost certainly rise, unit fixed cost will decrease.
- Even when the incremental unit cost exceeds the average cost of base production, it may still be justified if there is market demand and the incremental production cost is less than the selling price.
- There is usually an alternative of making equipment changes to achieve the same production increase without increasing variable cost (eg larger kiln ID fan, cyclone modifications to reduce preheater pressure drop, increased cooler efficiency). However, such projects would involve lead-time, down-time for tie-in, and would result in depreciation cost whether or not the incremental production was required. A process change can be attractive when employed for a short-term or to meet brief peaks of demand.

13.4 Project Cost Estimation

Cost estimating ranges from preliminary conceptual estimates to definitive contract pricing. Conceptual estimates can be obtained by techniques such as cost-capacity scaling based on the known cost of a similar project of different capacity:

$$[\text{cost } 1/\text{cost } 2] = [\text{size } 1/\text{size } 2]^R$$

An R-value of 0.65 may be used for first approximations, though a comprehensive list of values for various types of equipment and tasks is given by Remer & Chai (Chemical Engineering Progress; 8/1990, pg 77). Alternatively, if equipment cost is known it can be doubled (or some other experience based factor applied) to yield an approximate installed cost. At this level of refinement, annual maintenance costs equal 10% of equipment cost may be assumed. Another useful reference, unfortunately not recently revised, is *Spon's International Construction Costs Handbook* (E & F Spon, London, 1988), which gives unit, and comparative costs for 32 countries.

More detailed costing is obtained from materials and equipment lists to which unit costs can be applied with appropriate estimates for engineering, permitting, erection, commissioning, etc.

Ultimately definitive lists are developed based on specific bids by equipment suppliers and contractors. Except for the simplest projects, even with a firm turn-key price, there will be scope changes and additions which demand contingency funding.

13.5 Financial Statements

Three basic statements are used by decision-makers outside of the company:

1) Balance Sheet
2) Income Statement (Profit and Loss)
3) Cash Flow Statement

In addition to these, a statement of shareholder's equity (retained earnings) is often provided.

The Balance Sheet reflects the resources the company owns at a point in time, most commonly at year-end. It is a snapshot of assets, liabilities and owner's equity:

Assets – Current	*Liabilities – Current*
Cash	Accounts payable
Accounts receivable	Wages payable
Inventories	Taxes payable
Other current assets	Other current liabilities
– *Non-current*	– *Non-current*
Land & buildings	Deferred taxes
Plant, property & equipment	Long-term debt
Intangible assets	– Owners' Equity
Other non-current assets	Capital (Common) stock
	Retained earnings

Assets	=	Liabilities	+	Owners' Equity
(Resources of the Company)		(External claims against the Company)		(Shareholders' claims against the Company)

With some variation in terminology (income/earnings/profit), statements are expressed:

Income Statement:

Sales Revenues
  less Cost of goods sold
Gross Profit
  less Selling, general & administrative expenses
    Research & development and Engineering expenses
EBITDA
  less Depreciation / depletion / amortization
    Other income / expenses (+/-)
    Interest income / expenses (+/-)
    Foreign exchange adjustments (+/-)
Earnings Before Tax
  Provision for income taxes
Net Income
  less Extraordinary items (+/-)
    Dividends to preferred shareholders
Income Available for Distribution to Common Shareholders

Cash flow is an important concept because a company can be profitable and insolvent at the same time. Cash flow is variously defined and is determined from the 'Statement of Cash Flows'. Sources and uses of cash are classified into three categories: *operating, financing, and investing*

Operating Activities – activities related to the production and sale of goods and services that enter into the determination of the company's income.

Cash Inflows	*Cash Outflows*
Collections from customers	Payments to suppliers
Receipts of interest and cash dividends	Payments of taxes and interest
All other cash receipts	Payments of business expenses
	All other cash payments

Investing Activities – activities that involve investment of the firm's resources.

Cash Inflows	*Cash Outflows*
Disposal of plant assets	Acquisition of long term assets
Receipt of loan repayment	Loans made or purchased
Sale of other companies' securities	Purchase of other companies' securities

Financing Activities – activities that provide the firm funds whether from its owners (issue of equity) or from its creditors (issue of debt).

Cash Inflows	*Cash Outflows*
Borrowing cash from creditors	Paying off loan capital
Issuing of bonds	Stock repurchase
Issuing of stocks	Distribution of cash dividends

Cash flows from operations and investing establish the basis for calculating *free cash flow,* the key variable used in the Discounted Cash Flow (DCF) valuation models for cement companies.

Assessment of Financial Statements

Management accounts contain proprietary information that need to be kept out of the hands of competitors so their distribution is internal and limited. As a result, financial statements are usually the only source of information for outsiders. Five areas to scrutinize when reading financial statements are:

- *Inventory Levels.* When a plant's stockpile of unsold cement or clinker is rising faster than sales, it can mean quality has slipped, market share has declined, or production costs are excessive.
- *Receivables.* Although bad debts are unusual in the cement industry, channel stuffing is not an uncommon practice where last minute sales are made to ready-mix suppliers just before a quarter's end.
- *Extraordinary Expenses.* A big charge isn't the result of a sudden occurrence. More often, it is a sudden recognition of a longstanding problem. Alternatively, extraordinary expenses are used to smooth earnings and avoid the embarrassment of a down year.
- *Asset Sales.* Management may say it is divesting certain 'non-core assets' to focus limited resources on its core business when really it is running out of cash.
- *Reduced Capital Spending.* In theory, capital expenditures should keep up with depreciation charges. When capital expenditures are below depreciation for extended periods of time, owners are probably implementing a 'milk it and kill it' strategy or are struggling to reach targets such as free cash flow as discussed above.

References:

Homgren, Datar & Foster; Cost Accounting 11/e (2003). Prentice Hall Business Publishing.
Stickney & Weil; Financial Accounting: An Introduction to Concepts, Methods and Uses, 10th Ed (2003). Thompson, South-Western.
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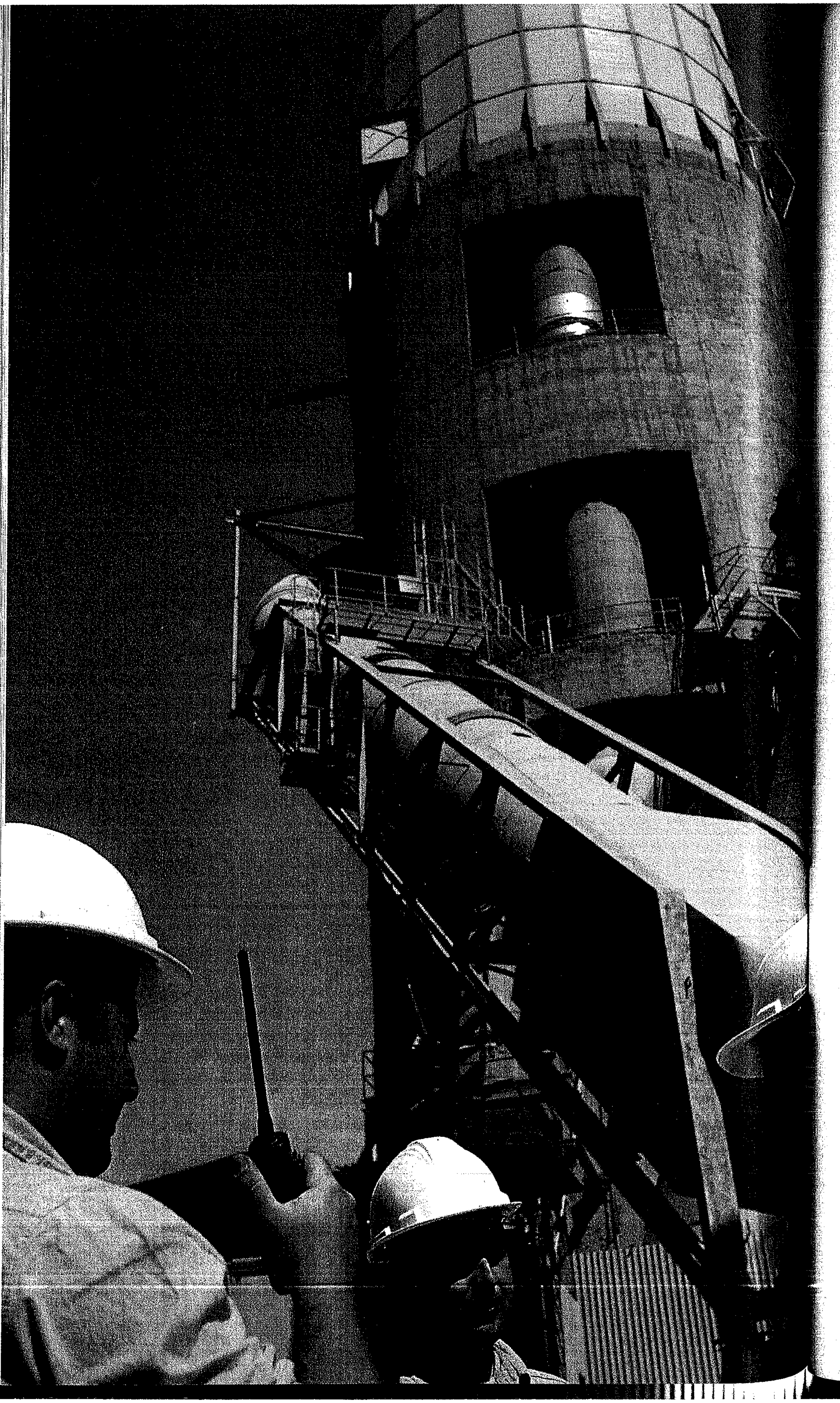

14 Technical and Process audits

Technical audit is a review of all units of plant equipment to determine their operating capacities and efficiencies with relation to original design and to subsequent modifications. Current performance is compared to what is realistically achievable and for this purpose 'rated' or guarantee capacities should not be taken uncritically. The purpose is to establish the efficiency with which the plant is being operated, to identify bottlenecks and to propose means of eliminating them (McLeod; ICR; 4/1996, pg 49). 'De-bottlenecking' is a cost effective technique for increasing productivity.

Process audit is an analysis of plant operations with the purpose of increasing production, reducing unit cost, and improving product quality by establishing and controlling optimum mix parameters. This program must be implemented on a comprehensive basis rather than by trouble-shooting individual deficiencies. Elements include but are not limited to:

- Establish optimum mix design with respect to cost, handling, grindability and burnability
- Produce constant chemical and physical kiln feed characteristics
- Operate kiln with minimum process variations and optimum free-lime
- Finish grind to a product having mid-market concrete strength with minimal variation (market position is, of course, a subject for management judgement but it is suggested that policy be based upon concrete performance, not that of mortar)

14.1 Kiln Specific Fuel Consumption

The specific fuel consumption of a kiln is arguably the most important operating parameter. Most kilns are limited by ID fan capacity which limits the amount of fuel which can be burned and, hence, the amount of clinker which can be produced. Any reduction in specific fuel consumption will result in the potential to increase production rate (4% reduction in specific fuel will allow about 3% more clinker production; the discount is due to the calcined CO_2 from increased production using some of the gas handling capacity).

The first consideration is the optimisation of mix design for both burnability and product quality. A considerable amount of trial and error may be involved in establishing an optimum mix design but there are a number of rules of thumb:

C_3S	58-65 (clinker basis)
LSF	0.92-0.97
S/R	2.2-2.6
Liquid (1450°C)	23-26%

Large particle quartz in raw materials should be avoided if at all possible as grinding to reactive size, say <30μ, by simple raw milling will result in excessive power consumption, in over-grinding the softer components, and in increasing cyclone preheater dust loss. Laboratory grindability tests should be made for each raw material (Kannewurf, Grindability Standard; PCA Technical Report MP-8; 1956). Laboratory burnability tests can also be helpful (Kirsch; ZKG; 1/1994, pg E27) as, to a lesser extent, can empirical formulae such as given in Section B5.3.

A fairly modest average for an early vintage 4-stage cyclone preheater kiln is 850kcal/kg

(NCV). Modern large precalciner kilns coupled with high-efficiency coolers are capable of less than 700kcal/kg. Starting from an appropriate, or the designed, efficiency, corrections can be made for various factors to give a realistic target value:

- Precalciners vs cyclone preheaters have several effects:
    - Heat exchange is slightly less efficient and exhaust gas temperatures 10-20° higher
    - For a given kiln size, the PC will have up to twice the production so that the unit radiation loss will be less
- Large production units have lower radiation heat losses
- Other than 4 cyclone stages affects heat exchange by approximately 20kcal/kg per stage
- Bypasses waste high grade heat; a 10% gas bypass equates to heat loss of approximately
  6-8% for a cyclone preheater kiln
  3-4% for a precalciner kiln
- High primary air from a direct fired coal system forgoes some heat recouperation from secondary air. However, low primary air indirect burners may result in non-recirculatory flames with poorer heat transfer.
- Fuel in raw materials may or may not contribute useful heat but must be included in the heat balance

A heat balance should be constructed for the preheater, kiln, and cooler and the 'heat out' values compared to standard values such as those given in Section B5.6 adjusted for the above factors. Significant abnormal heat loss is almost invariably associated with preheater exhaust and/or cooler exhaust. As can be seen from the typical heat balance given in Section B5.6, of a total heat consumption of 850kcal/kg, 420kcal/kg is for chemical reaction and 430kcal/kg is due to heat loss of which 325kcal/kg or 75% is due to losses through kiln and cooler stacks. The challenge is to reduce these exhaust gas heat losses.

High exhaust heat loss may be remedied by:

1 Reduce losses due to CO formation

- Good combustion in the main burner to avoid local reducing conditions in the presence of excess oxygen (Sec 9.1)
- Sufficient air-fuel mixing and retention time in the calciner for complete combustion

2 Reduce preheater exhaust gas quantity

- Maximize heat recouperation from the cooler
- Minimize excess air without compromising combustion
- Avoid over-burning with consistent kiln feed chemistry and with constant feed and fuel rates. Note that every 0.1% free-lime below the optimum wastes up to 14kcal/kg (see Section 4.1).
- Minimize false air at kiln seals and preheater ports.

3 Reduce preheater exhaust gas temperature

- Good meal distribution in gas ducts by design and maintenance of splash plates and splash boxes.
- High cyclone efficiency so that hot meal is not carried up the preheater

4 Reduce cooler exhaust

- Maximum heat recouperation by control of air flow, clinker distribution and clinker granulometry
- Minimise false air into firing hood and kiln discharge seal

The specific fuel efficiencies of 700kcal/kg and below achieved by modern pyrosystems are largely due to their high efficiency coolers and the addition of 5th and 6th preheater stages.

It should be noted that, if heat is required from kiln exhaust for drying of raw materials, improving heat exchange in the preheater may be counterproductive; there is still, however, a benefit in optimizing burning conditions and mix design.

If cooler exhaust or clinker heat loss is excessive, this will correlate to low secondary air temperature and poor cooler operation.

Excessive kiln radiation may indicate poor kiln coating which may suggest an unstable flame.

It has been shown (Roseman et al; ZKG; 12/1987, pg E280) that the effective heat loss varies with the source of loss due to the grade of heat involved. For a 4-stage cyclone preheater with a 2-stage grate cooler:

1 kcal/kg heat loss from-		
Preheater exhaust must be compensated by		0.87kcal/kg in additional fuel energy
Radiation from	4th (top) stage	0.22
	3rd stage	0.44
	2nd stage	0.76
	1st (bottom) stage	1.18
Kiln radiation loss		1.18
Feed material reactions (eg over-burning)		1.18
Cooler radiation, vent, and clinker		1.46

This suggests that heat losses from the cooler are the most serious as heat lost from secondary or tertiary air must be compensated by additional fuel burning compounded by the heating of additional combustion air.

14.2 Cement Mill Specific Power Consumption

A cement mill is usually limited by drive power so that any reduction in kWh/t translates into increased production capacity as well as a reduced unit power cost. A reasonable standard efficiency is 32kWh/t at 3600cm^2/g which can be adjusted for other finenesses as shown in Section 5.2. Excessive specific power consumption may be due to:

- Hard clinker due to over-burning
- Poor separator adjustment and high or low circulating load
- Incorrect tuning of grit separator (if installed)
- Unnecessary return of dust collector catch to milling circuit
- Poor ball charge
- Inadequate air sweep
- Inadequate or poorly injected grinding aid

The need to balance mill and separator was addressed in the Grace 'Umbrella Technique', a valuable tool for optimizing mill circuits (Welch; The Grace Factor – A New Tool for Cement Process Engineers; ASTM; 1984).

14.3 Other Systems

Kilns and finish mills are the highest priorities for process optimisation as they are the highest consumers of energy and are usually the plant capacity limiting units. Apart from their potential for energy savings, however, other systems may also have major impacts on overall process efficiency:

Crushing	Production of appropriate raw mill feed size
Raw milling	Production of appropriate kiln feed particle size
Blending	Achievement of adequate blending ratio
Coal milling	Production of appropriate coal fineness. Minimal primary air in direct firing systems.

14.4 De-bottlenecking

This repulsive but descriptive term is a mandatory focus for cement plant engineers. Every process has one or more capacity limitations. Identification of the limiting equipment is the first step, but cost-effective removal of a limit requires that subsequent limits also be analysed in order that a comprehensive plan for an optimum capacity increase may be designed and implemented.

Raw Material Supply

Many waste materials are available to replace traditional raw materials in order to reduce cost or allow capacity increase. Quartz sand causes both milling and burnability problems, clay may be a handling and drying problem; fly ash or silica fume, if available, may perform better and can reduce raw mill capacity limitations. Synthetic gypsum is frequently cheaper than natural gypsum but is difficult to handle; processing with slag, kiln dust, or other materials may facilitate handling and even improve cement quality. Note that cement specifications may limit or preclude additions by the cement producer.

Fuel Supply

Various waste and by-product materials such as petroleum coke, tires, and waste oil may be used to reduce fuel cost. Each opportunity must be considered in the light of local supply cost, compatibility with the process, and possible contaminants such as alkalis, heavy metals, PCBs, etc. Whole tires are quite commonly added to the kiln riser; their combustion is, however, intermittent and requires either that a high feed-end oxygen level is maintained thereby de-rating the kiln, or the acceptance of high CO spikes in the preheater exhaust. Pyrolysis is a technically attractive development but does increase the capital cost and complexity of tire burning (Schmidthals & Rose; ZKG; 2/1999, pg 88). A further development is FLS's 'Hotdisk' which forms an integral part of the kiln riser or calciner and appears to solve most of the problems associated with the burning of whole tires (ICR; 1/2003, pg 64).

Raw Milling

Traditionally raw mills were oversized relative to kilns. However, increasing kiln capacity is often relatively simple (bigger ID fan), and some raw mills have thus become limiting (see options under Sects 3.1 & 5.2). More commonly the limit is in drying capacity. Remedies include the reduction of false air, increase of hot gas temperature, or increase of gas handling capacity (fan or dust collector).

Cyclone Preheater

Cyclone gas duct velocity is usually designed for about 15 – 16M/sec. An increase of 20% is acceptable and can be achieved by installing a larger or more efficient ID fan; this will, however, increase pressure drop by 40%. Replacement of preheater cyclone vessels is possible but expensive. PMT's Hurriclon technology allows the modification of upper stage vortex finders resulting in pressure drop reductions of 30% at modest cost (Kroihs; ZKG; 1/1998, pg24).

Rotary Kiln

Kilns are usually oversized and can accommodate specific loads of 1.8 – 2.6t/day/M^3 for preheater kilns and up to 6t/day/M^3 for precalciner kilns. A common upgrade is to increase speed to about 4rpm.

Clinker Cooler

The first purpose of the clinker cooler is heat recouperation and most common upgrades for grate coolers are the replacement of under-grate fans with higher static pressure units which can support deeper bed-depths, and the installation of a static section of 6-9 rows of inclined grates at the hot end of the cooler which improves heat recouperation and bed distribution (Sect 4.12). It is essential for satisfactory performance of any grate cooler that the clinker is reasonably granular and free of excessive fines. The second function of the cooler is to cool clinker before discharge. The acceptable discharge temperature will depend upon downstream equipment and upon cement quality effects. Where cooling is inadequate the options are either to extend the grate cooler or to add a G-cooler (Sect 4.12). While the BMH G-cooler is probably the more expensive option, it does avoid the extensive kiln down-time inevitable with a major cooler modification and the unit is both efficient and requires minimal maintenance.

Finish Milling

The options are primarily the replacement of mechanical separators with high-efficiency units for a capacity increase of 5-10%, or the addition of pregrinding. Pregrinding applies only to ball mills and may comprise a horizontal impact crusher for 10-20% increase or a roll press for an additional 20-100%. Vertical roller mills are not normally considered amenable to capacity increase. However, LV classifier modification can yield a 10-20% increase and various means to achieve up to a 45% increase are discussed by Jung (CI; 2/2004, pg 52).

Coal Milling

Roller mills can be upgraded by replacing a static classifier with a dynamic classifier which reduces the return of fine particles to the mill and hence allows increased feed rate. In some instances the table speed and/or roller size may be increased but for this the manufacturer should be involved. Ball mills can also be fitted with high efficiency separators or the separator rejects may be ground in a small, secondary mill.

Process Fans

Fan impellers can often be tipped or their speed increased to increase air flow though this may be limited by housing, by shaft size, or by drive power. If an existing fan has a low-efficiency 'paddle wheel', it may be replaced with a backward curved impeller retaining the same drive and, sometimes, even the same housing. If a major fan must be replaced, it should obviously be sized for any possible future system capacity.

Bag-house

Gas flow can be increased by changing filter material (eg to membrane coated fabric), by using pleated bags to increase cloth area, or by using longer bags in the existing or an extended housing.

Air-to-Air Heat Exchanger
Cooler dust collector capacity can be increased by addition of a heat exchanger.

Conveying
Conveying capacities frequently become limiting and must be assessed according to type. Replacement of a conveyor gives the opportunity to change system in order to reduce either power cost or maintenance from, say, air slide to screw, or from pneumatic conveying to mechanical.

14.5 Project Audit

It is unfortunately true that more effort is usually invested in justifying a capital project than in making it work; certainly in making it work after the acceptance or guarantee tests are complete. Consideration should be given to a program to audit the performance of projects one or more years after completion. Not only should it be of interest to confirm that the investment was justified, but it may save repeating a disappointing project. To avoid unnecessary antagonism from plant staff, this may be made part of the standard company internal audit process.

It must, however, be recognized that process and equipment changes are often continuous and cumulative and that it is usually difficult to isolate the effect from a specific modification. Many kiln equipment modifications are effected during the annual outage during which the preheater will be cleaned, false air reduced, and cooler efficiency recovered so that it is usual for production rate to receive a temporary boost. To claim an increase in production rate from immediately before the outage to immediately after the outage as wholly due to a specific modification may be quite spurious.

14.6 Risk Assessment

Even where present and historical plant production and cost may be satisfactory, it is prudent to identify and assess a number of risk factors which can seriously disrupt plant operations or economics. Such factors include:

Risk due to factors under management control
1 **Maintenance** – Inadequate maintenance practices which enhance risk of unplanned major equipment failure (eg gear failure, fan impeller collapse) or of structural failure (eg silo cracks, dust build-up on roofs).
2 **Operation** – Operating practices which enhance risk of major equipment damages (eg overheating of kiln shell, high or unmeasured CO in kiln exhaust).
3 **Availability of critical spare parts** – Provision for very low frequency failures either by cost justifying and having spares on hand, or by contingency planning for pooled spares or expedited procurement.
4 **Product quality** – Failure of product to meet specification or, far more common, contamination or shipping of wrong type of cement can result in costly liability and loss of customers.
5 **Transgression of regulation** – Failure to comply with operating permits or emission regulation can result in significant fines or even plant closure.
6 **Safety** – Failure to maintain safe equipment and practices will result in accidents, penalties, and increased insurance cost.
7 **Succession** – Having suitably trained candidates available to cover for loss of a critical operator or line manager.
8 **Industrial action** – Where appropriate, contingency planning for continuing plant operation in the event of striking workforce.

Risk due to inherent factors
9 **Obsolescence** – Operation of equipment which has become unreliable and for which spare parts are difficult to obtain.
10 **Reserves** – Inadequate limestone (or clay) reserves or shift in chemistry or rock type which increase stripping, recovery or comminution cost, or increase cost of corrective additives.
11 **Design provision for natural disasters** – Probability of earthquake, wind or flood for which inadequate or out-of-date design standards have been used. Lightening protection.
12 **Interruption of supply or distribution routes** – Lack of redundancy in road or rail access to the plant; probability of high or low water causing prolonged interruption of water access.
13 **Interruption of power supply** – Lack of redundancy in HV supply line or transformers; dependence upon inadequate or unreliable utility power supply.
14 **Disruption of information system** – Failure of process control system and plant information system, or breach of computer security by outsiders.
15 **Sabotage** – Risk of sabotage by employees or damage due to local political instability.

Project risk
16 **Project cost and schedule** – Possibility that committed investment or plant down-time may over-run.
17 **Project technical risk** – Possibility that committed investment may not yield projected benefits due to inherent system inadequacy.

Market risk
18 **Competitor action** – Opening of new plant or terminal within market area. Irresponsible price cutting by existing competitor to gain market share.
19 **Alternative product** – This is normally limited by the constraints of the Portland cement specification and the long lead time that would be required for even a miracle alternative to be accepted.

Intellectual property
20 **Proprietary cost advantage** – The possibility that a competitor discovers, or gains control of, a patented process or piece of equipment which gives him a significant manufacturing cost advantage.

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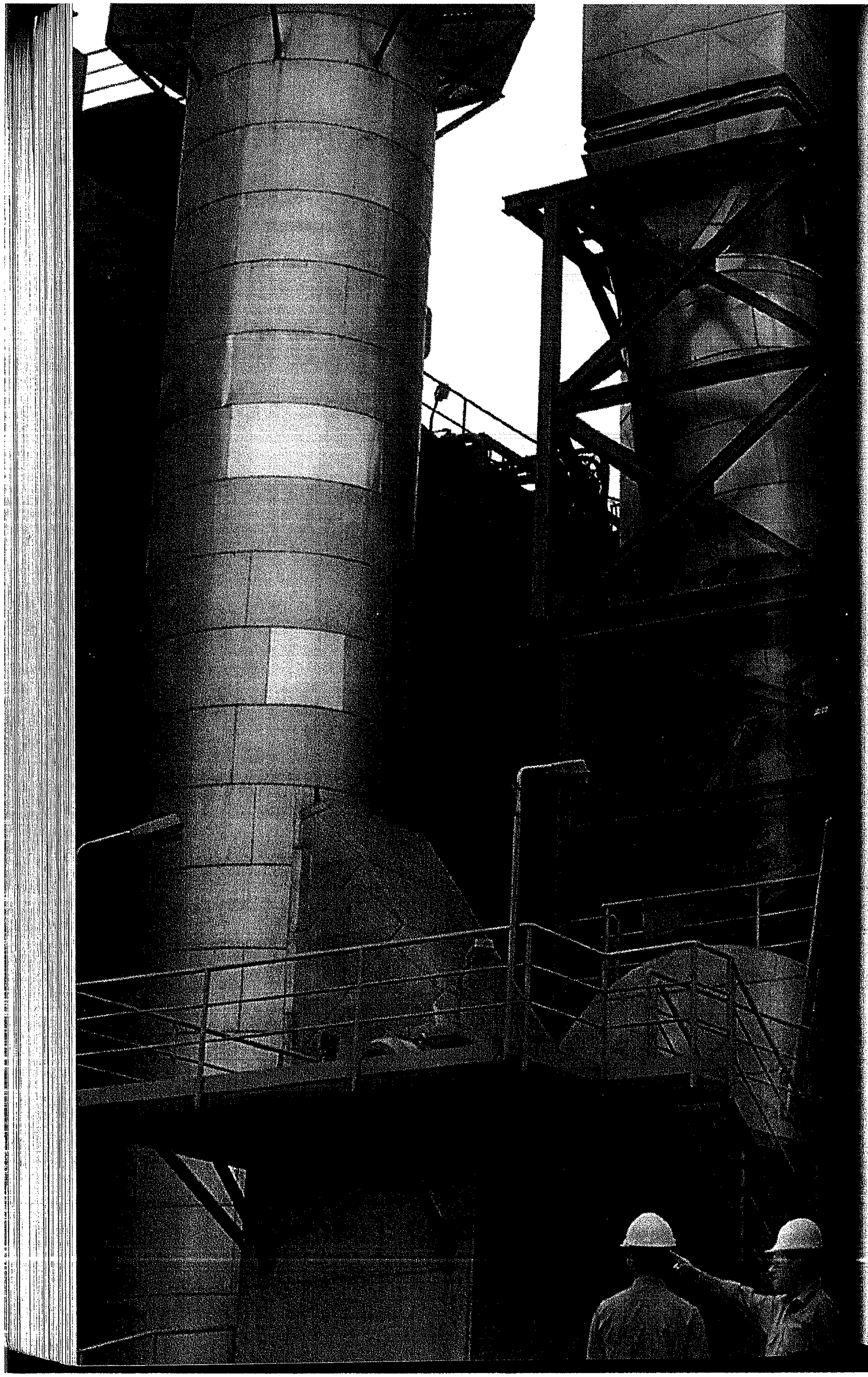

15 Cement Plant Assessment Data List

Plant ____________________ Date ____________

15.1 General

1 Property map

2 Land title

3 Surrounding land use

4 Plant lay-out

5 Plant age

6 Space & scope for expansion

7 Foundation conditions, earthquake zone

8 Regulatory agencies & permits

9 Process flow

10 ISO 9000, ISO 14000

11 Rainfall

12 Temperature high/low

13 Paving

14 Housekeeping

15 Insurance

16 Power supply

17 Power tariff on/off peak

18 Emergency generator

19 Water source & consumption

20 Availability of waste materials

21 Cement market & seasonal variation

15.2 Administration & Commercial

1 Manufacturing costs by area & natural expense

2 Manufacturing cost by cement type (bulk/sack)

3 Price by cement type (bulk/sack); price trends; price regulation

4 Data processing, information systems

5 Employee numbers (including permanent contract labour)

6 Organization chart

7 Employee benefits/housing/transport

8 Safety records

9 Purchase procedure, local/import

10 Warehouse value

11 Capital investments approved or planned

15.3 Quarry

1 Materials type

2 Limestone blasting/ripping, loading, hauling

3 Limestone crushing

4 Limestone reserves

5 Clay mining, transporting

6 Clay reserves

7 Bought in materials

8 Preblend type, ratio, capacity

9 Mobile equipment type, capacity, age, condition

10 Mine planning

15.4 Drying

1 Raw materials moisture

2 Drying method

15.5 Raw Milling

1 Raw materials storage capacity

2 Mill type & dimensions, rated/absorbed kW

3 Separator type

4 Mill capacity, rated/actual

5 Mill kWh/t

6 Feed & product fineness

7 Discharge conveyor

15.6 Blending

1 Silo type, dimensions, capacities

2 Blending ratio

3 kWh/t

4 Discharge conveying

15.7 Kiln

1 Kiln feed conveying & metering

2 Kiln type & dimensions

3 Kiln capacity, rated/actual

4 Run factor & downtime analysis

5 Kiln seals, tire & shell condition

6 Kiln gas analysis

7 Kiln brick life & consumption (kg/t)

8 Kiln roof

9 Precalciner/kiln fuel split, tertiary take-off location

10 Preheater pressure drop

11 Preheater efficiency (dust loss), kiln feed:clinker factor

12 Kiln rings, preheater build-up

13 ID fan build-up

14 Bypass

15 Cooler type & dimensions, clinker breaker

16 Clinker discharge temperature

17 Coal mill/raw mill/tertiary take off locations

18 kcal/kg clinker

19 Auxiliary drive

20 Preheater discharge temperature, conditioning tower, kiln de-rating with raw mill down

15.8 Fuel

1 Fuel types including waste fuels

2 Start-up/back-up fuel

3 Firing system, burner type, % primary air

4 Coal proximate analysis

5 Coal mill type & capacity

6 Coal fineness

15.9 Clinker

1 Clinker conveying from cooler

2 Clinker types

3 Storage capacity covered/outside

15.10 Finish Mill

1 Mill type & dimensions; rated/absorbed kW

2 Mill capacity rated/actual

3 Mill kWh/t

4 Separator type

5 Product fineness

6 Gypsum type & purity

7 Grinding aid

8 Cement discharge temperature

9 Cement conveying

15.11 Cement

1 Cement types

2 Silo capacity

15.12 Quality Control

1 Cement specifications

2 Analytical methods

3 Physical test methods

4 Release age

5 Laboratory equipment

6 Quality control sampling methods and frequencies

15.13 Packing & Distribution

1 Shipment by month

2 Shipment % bulk vs bag

3 Distribution by road, rail, water

4 Point of sale, distribution by plant/customer/contractor

5 Cement temperature at load-out

6 Packers

7 Bag type, bag weight, bag breakage, palletizing

15.14 Dust Collectors

1 Types

2 Capacity & condition

15.15 Maintenance

1 PM schedule

2 Predictive maintenance

3 Shop facilities

4 Availability of cranes, machining, & motor rewinding

5 Maintenance man hours planned vs break-down

15.16 Process

1 kcal/kg total & non-kiln

2 kWh/t total

3 Process engineering resources & records

4 Equipment redundancy

5 Plant bottlenecks

6 Emission limits, dust

7 Emission limits, NOx, SO_2, CO, etc

9 Process control systems

15.17 Materials Analyses

1 Raw materials

2 Mix design (reconcile with annual consumptions of raw materials)

3 Gypsum (mineral/synthetic)

4 Clinker

5 Cement

6 Cement test certificates

7 Variation – raw materials, pre-blend, raw mill product, kiln feed, cement

15.18 Plant Capacity Summary

	t/H	Eqt/Y
Crushing		
Raw Milling		
Coal Milling		
Kiln		
Finish Milling		
Packing		
Load-out		

Note Eqt refers to cement equivalent tonnes as described in Sec 12.1.

	t	Days usage
Storage capacity of limestone available to raw mill		
Storage capacity of clay available to raw mill		
Storage capacities of other raw materials		
Storage capacity of blending/kiln feed silos		
Storage capacity of clinker (excluding outside)		
Storage capacity of cement		

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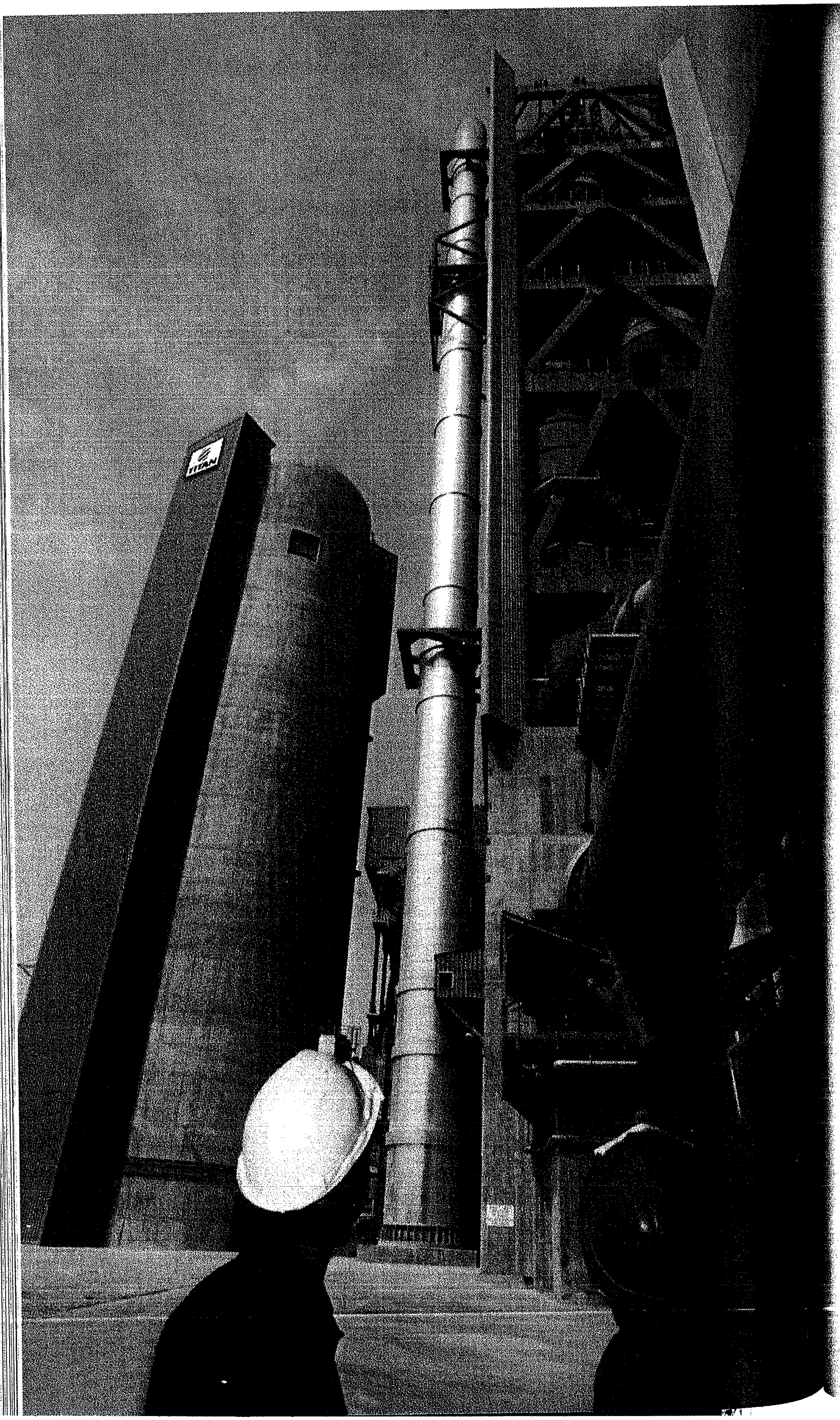

16 Cement Plant Valuation and Construction Cost

16.1 Plant valuation

Plant/Company Valuation can be determined by three general approaches: cost, income, and market comparison. It must also be recognised, however, that many cement industry acquisitions are by existing competitors and are primarily motivated by strategic or market considerations which involve models of much greater complexity that those outlined here.

Cost or Financial approach uses replacement as an indicator of value. This is employed for financial reporting and for insurance purposes rather than acquisition.

Book Value is the historical cost of the asset less depreciation and depletion and is simple to obtain.

Present value = [(Historical cost - Residual value) x (Remaining life/Total life)] + Residual value

For financial reporting purposes (as opposed to tax write-off which may be accelerated), depreciation schedules are usually straight-line over the total useful life which is, typically:

Mobile equipment	3-10 years
Machinery & equipment	10-20 years
Buildings	20-40 years

Financial Value is the balance sheet estimate of net assets of a company.

Assets = Liabilities + Equity
or:
CA + FA + OA = CL + LD + E

where CA = current assets (cash equivalents & accounts receivable)
FA = fixed assets (original cost less depreciation and depletion)
OA = other assets (non producing assets & intangible assets)
CL = current liabilities (payables and current portion of LD)
LD = long term debt (maturity greater than one year)
E = equity
or:
Valuation = LD + E = FA + OA + (CA-CL)
where (CA-CL) = net working capital

Income approach estimates the present value of future net benefits and is the primary method for valuing acquisitions. There are several variants for estimating income.

Market Capitalisation is probably the simplest valuation with the share price reflecting market assumptions of future earnings :

Market Capitalisation = Current share price x Number of shares outstanding

Enterprise Value (EVA) is a refinement which also takes into account long-term debt and net cash on hand (www.fool.com/dripport).

EVA = Market Capitalization + Long-term debt - Net cash & investments

Net Earnings (after interest, amortization, depreciation, and tax) multiplied by the average ratio of market-share-price/earnings-per-share (P/E) yields a valuation based only on present operation. For acquisition purposes, it is assumed that a premium must be paid which reflects the additional value to the purchaser of potential improvements or synergies. Thus:

Valuation = Net earnings x P/E + Premium

P/E ratios and acquisition premiums vary considerably with time and market. Recent averages are:

Europe	13.0
United States	13.6
Mexico	9.1
Japan, Taiwan, Thailand	10.8
India	20.5

(ICR; 6/2004, pg 21)

The premium is typically 30 - 40%.

Net Present Value (NPV) attempts to estimate revenues during the life of the investment and is based upon discounted cash flows (DCF) and residual salvage value (RV) of the asset. Typically a period of twenty years is considered. The weaknesses of the method are the estimation of discount rate (k), which is the assumed interest rate for discounting future incomes (see Sec 13.2), and the uncertainty of projecting volumes and prices into the future.

$$\text{Valuation} = \text{NPV} [\, k\,(CF_{yr1}, CF_{yr2} \dots CF_{yrX} + RV)]$$

Return on Invested Capital (ROIC) is defined as the cash rate of return on capital invested in operations (www.fool.com/school/roic):

ROIC = Net operating profit after taxes (NOPAT) / Invested capital

where: NOPAT = Reported net income + Amortisation & Depreciation + Non-recurring costs + Interest expense + Tax paid - Investment income

and: Invested capital = Total assets - Excess cash - Investments - Non-interest-bearing current liabilities

ROIC must exceed the weighted average cost of capital (WACC) for the company to create value (see Sec 13.2), and the spread (ROIC - WACC) is a prime indicator of performance.

Valuation = NPV of (ROIC - WACC)

Calculation of ROIC is complex in practice and is still subject to manipulation by management, is influenced by accounting changes, and is affected by inflation and currency exchange movements.

Apart from the weaknesses of long-term projections, various other concerns apply to specific situations:

- If the subject plant is part of a multi-plant company, determination or confirmation of appropriate profit-centre figures is necessary as cost allocation or transfer pricing practices may cause distortion.
- If the potential acquisition is in a country subject to subsidies or price controls which may be discontinued, 'shadow pricing' is an attempt to impute free market costs and prices.
- In developing countries with unstable exchange rates, local currency finance, if available, is usually prohibitively expensive. Devaluation will, almost certainly, be followed by direct or indirect control of product prices and, thus, revenues while debt service, imported spare parts and supplies and, usually, fuel remain payable in hard currency. Hedging is, of course, possible but expensive and a number of companies in South East Asia suffered from this exposure to an extreme degree after 1997.

Market Comparison approach takes recent sales of similar assets and adjusts prices to reflect known differences in technology, reserves, market, etc. This is unlikely to offer an adequate valuation on its own, but does constitute a useful sanity test. In the cement industry, a common number used for reference is the unit cost in $/annual tonne of production.

16.2 New Plant Construction

New green-field cement plants cost $150-200 per tonne of annual production, while kiln expansions cost $80-150 depending upon the usable excess capacity of existing ancillary equipment and storage. Indian and Chinese plant construction costs are much lower, perhaps $50-80 per tonne. Permitting of heavy industrial facilities becomes ever more difficult and expensive, so that considerable intangible value may attach to an existing operation and its market, even if the equipment is antiquated and inefficient.

Obviously, profitable markets suck in imports from low cost producers and, unless protected geographically, by tariffs or by other regulation, high margins tend to be ephemeral. Also, the lead time for new capacity and the cyclical nature of most cement markets can enhance both risk and reward for building new cement plants.

A typical project cost for adding a new 1.5 million tonne/year line to an existing plant is:

Equipment	Quarry equipment	US$ 3,000,000
	Raw milling & blending	13,000,000
	Kiln, preheater & cooler	18,000,000
	Coal system & storage	5,000,000
	Clinker storage	3,000,000
	Cement milling	9,000,000
	Cement storage and packing	3,000,000
	Electrical & control	9,000,000
Sub-total		**$63,000,000**
Civil, structural & erection		$38,000,000
Engineering, construction management, freight, commissioning		$19,000,000
Owner's capitalised cost		$ 5,000,000
Construction interest		$14,000,000
Spares		$ 4,000,000
Contingency (5%)		$ 7,000,000
Total project cost		**$150,000,000**

(Adapted from Conroy; Proc IEEE, May 1998)

There is, of course, considerable variation due to specification, location, degree of local fabrication, labour and materials costs, and cost of capital. The alternatives available for project execution, from turnkey to multiple contracting, are reviewed by Ireland (ICR; 6/2000, pg 35) and Muckley (ICR; 7/2004, pg 92).

There is also considerable variation in schedule for construction. Conventionally, process and preliminary engineering design, and permitting are completed before procurement and contract bidding begin. However, where expedition is essential the risks of overlapping engineering with construction are sometimes accepted though the cost penalty is unpredictable and usually severe. Site preparation is also frequently performed by plant forces or separately contracted. Then assuming permitting does not impact the critical path, a typical schedule would be:

	Cumulative
Start detailed engineering	0 months
Issue tender documents	2
Contracts for equipment and construction	5
Ground breaking	7
Complete civil work	12
Major equipment delivered	12
Complete mechanical erection	18
Complete electrical and instrumentation	20
Begin commissioning	21
Commercial operations	24

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17 Statistics

Accuracy is the closeness between the observed value and the "true value". The difference is usually due to both random and systematic errors. When random errors are excluded, the term 'bias' is used (ASTM E456 "Standard Terminology Relating to Statistics").

Precision is the closeness between randomly selected individual experimental values; this is the subject of statistics.

Average or Mean is usually used for the arithmetic mean:

$$x_{bar} = (x_1 + x_2 + x_3 \ldots + x_n) / n$$

Median is the middle value of a distribution.

Mode is the value of a variable which occurs most frequently.

Variance is a measure of the squared dispersion of observed values or measurements. The variance, s^2, of a sample of n observed values or measurements is:

$$s^2 = \Sigma (x_i-\bar{x})^2 / (n-1)$$

where x_i = individual test value

$\bar{x}$ = arithmetic mean of n determinations

Standard deviation, s, is a measure of the dispersion of a set on n observed values expressed as the positive square root of the variance:

$$s = \sqrt{(x_i-\bar{x})^2 / (n-1)}$$

Variances may be added. Thus the variance of test results is the sum of the actual sample variance and the test method variance:

$$s^2_{observed} = s^2_{actual} + s^2_{test}$$

or, in terms of standard deviation:

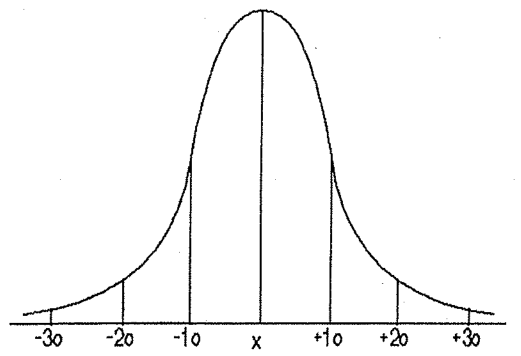
$$s_{actual} = \sqrt{s^2_{observed} - s^2_{test}}$$

It follows from this that the precision of any test method must be significantly greater than that of the process variation being monitored. In employing statistical analysis to quantify material and process variation, it is important that the input data should be grab samples or spot measurements – not composites or averaged results. Input data period should also reflect actual cycle timing.

The number of *degrees of freedom*, φ = n-1 since two values are required to define a line and only values in excess of two provide statistical information. Where n is finite, an approximation only of standard deviation is obtained, the estimated or sample standard deviation, s. As n increases, it becomes closer to the true or population standard deviation, σ. Likewise, the sample mean, x, becomes closer to the population mean, μ.

Random errors give rise to Gaussian (normal) distribution. Normal distribution is defined by two parameters, the population mean μ and the standard deviation σ. This distribution is represented by the familiar *bell curve*. Note, however, that not all bell shaped curves are normal distributions. There are other theoretical distributions such as Poisson, binomial, and rectangular, but most process and analytical measurements can be handled by Gaussian statistics.

Figure 17.1: Gaussian Distribution



For Gaussian distribution:

- 49.7% of values lie within $+0.67s$ and -0.67σ of the population mean, μ.
- 68.3% of values lie within $+1\sigma$ and -1σ
- 95.4% of values lie within $+2\sigma$ and -2σ
- 99.7% of values lie within $+3\sigma$ and -3σ

To convert a sample estimated standard deviation, s, to % confidence limits, it is multiplied by an appropriate 't' factor (see table). Thus, from a sample n of 11, there will be a 99% probability that a single test result will lie within 3.169s of the population mean.

This gives rise to the *Six Sigma* (6σ) concept of quality management. 99.99966% of values lie between $+6\sigma$ and -6σ, or 3.4 rejects per million. While this has an obvious cost benefit for widget manufacture where rejects are wasted, it seems to be of lesser importance to cement manufacture where out-of-specification material is almost always blended into product with minimal cost and negligible impact on quality. This is not to say that stable plant operation and consistent product quality are not of the utmost importance.

Range, R, is the difference between the extreme values in a group of data. It is a simple and approximate measure of the dispersion of a group of data and allows an estimate of precision without the need for a calculator. Assuming that the data have normal distribution, the range is related to sample standard deviation:

$$S = R / d$$

where d = a constant depending upon sample size, n:

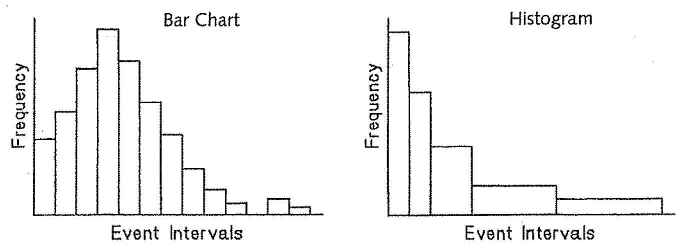
n	2	3	4	5	6	7	8	9	10	16	25	50	100	250
d	1.13	1.69	2.06	2.33	2.53	2.70	2.85	2.97	3.08	3.5	4.0	5.0	5.5	6.0

Bar chart is a graphical representation of frequency vs interval using constant intervals.

Histogram is a bar chart of frequency vs interval in which the width of each bar is proportional to a variable class interval and the area of each bar is proportional to the frequency it represents. Typically, less than 100 data will be divided into 7-10 bars increasing to, say, 20 bars for more than 200 data points. A plot using percentage of total frequency instead of numerical frequency is referred to as a relative histogram.

Bar charts and histograms for many process variables should approximate a bell curve and distortions or multiple troughs will reflect process abnormalities.

Figure 17.2: Bar Chart and Histogram



Outliers are exceptionally high or low values among a group of data. Often such values are due to gross error and can be eliminated prior to statistical treatment. If the cause of error cannot be identified, there are objective statistical criteria for their elimination (eg ASTM E 178 'Standard Practice for Dealing with Outlying Observations'). Discarding data without such justification is fraud.

Moving or rolling average is a plot of the averages of sequential groups of n data moving one point each time (eg 1-10, 2-11, 3-12, etc). Thus ASTM C 917 requires a 5-day (n=5) moving average for 7-day and 28-day mortar strength. The plot of moving averages more clearly reveals trends free of short term fluctuations. The greater n, the greater is the smoothing effect. Similar moving averages are used to analyse stock price movements.

F test allows estimation of whether two values of s are from the same population. If calculated $F = s_A2/s_B2$ (where $s_A > s_B$) is equal to, or less than, the corresponding F value given in tables for a specified probability, then identity can be deduced.

X^2 test (chi squared)determines fit of data to theoretical distribution. For present purposes this will be confirmation of normal distribution about a mean. Data are collected into sub-groups and the percentage of observed values falling within each sub-group (O) is compared to theoretical probabilities (E), eg:

sub-group range	observed (O)	theoretical (E)
$<-1\sigma$	....	15.9%
-1σ to -0.5σ	....	15.0
-0.5σ to 0	....	19.1
0 to $+0.5\sigma$	....	19.1
$+0.5\sigma$ to $+1\sigma$	....	15.0
$>+1\sigma$	....	15.9

B1 Power

B1.1 Specific Power Consumption

Power consumption rose significantly with the introduction of dry process kilns and has continued to rise with conversion to coal, increased fineness of cement, and with the demands of environmental protection. Typically, power consumption is presently 100-120kWh/t cement which may be broken down:

Quarrying & pre-blend	6kWh/t	5%
Raw milling	28	24
Blending	7	6
Burning & cooling	25	22
Finish milling	44	38
Conveying, packing & loading	6	5
Total	116kWh/t	

(Ellerbrock & Mathiak; ZKG; 11/1994, pg E296)

B1.2 Power Conservation

Conservation of electrical power should first address such areas as:

Blending (if turbulent)	– convert to CF
Pneumatic conveying	– convert to mechanical
Milling (ball)	– install pregrinding
ID fans	– eliminate air in-leakage, use high efficiency impellers.
Cooler fans (with outlet dampers)	– convert to inlet vane or variable speed
Plant air compressors (if central)	– minimise system loss and convert to distributed system. Compressed air requires some 7-8kWh/M^3
Plant lighting	– basic lighting can be augmented by additional lighting as required with timed shut-off.

For most large fan applications, frequency controlled variable speed drives offer significant power savings over damper or inlet vane controlled fixed speed ac drives albeit at higher capital cost. A technical and cost comparison is made by Bosche (WC; 6/1993, pg 2).

Energy efficiency should obviously be built in to original plant design. For existing plants, most remedies involve significant capital cost which must be justified by anticipated power savings. (Shenoy & Chacko; ICR; 10/1997, pg 60)

B1.3 Three-Phase Power

$$kVA = \sqrt{3} \times A \times V / 1000$$

$$kW = \sqrt{3} \times A \times V \times \sigma / 1000$$

Power factor (σ) is a measure of inefficiency and is defined as the cosine of the phase angle between currant (A) and voltage (V).

Actual voltages vary with country and with plant age and design. Typically, incoming power may be at 60-70kV transformed to 4160-6300V for large motors (>1000kW), then to 380-460V (3-phase) for intermediate equipment (3-1000kW), and 110-220V (1-phase) for small equipment and lighting.

B1.4 Motor Power Output

$$\text{Shaft kW} = \sqrt{3} \times A \times V \times \sigma \times E / 1000$$

where E = motor efficiency

B1.5 Peak Power Tariffs

These tariffs allow significant cost savings in exchange for shutting down all or some equipment during designated peak demand periods. This requires the availability of excess capacity (eg to allow cement mills to catch up), production and maintenance scheduling, and the assiduous attention of operators. Similarly there may be a penalty for operating at a power factor below the utility's stated minimum (0.8 – 0.95). A low power factor can be corrected using large synchronous (mill) motors or automatic capacitor control supplying reactive power to each individual load. Load management is discussed by Foster (ICR; 1/1996, pg 44).

B1.6 Power Generation

Major fossil-fueled power generators typically operate at 35% efficiency. Thus 1 tonne of coal with 6000kcal/kg net heat value will produce approximately 2.5MWh of power. Efficiency is 40-45% for pulverised coal plants.
Note that coal burning produces on average 10% ash of which 70% is fly ash and 30% bottom ash.

B1.7 Cogeneration

The generation of electric power from kiln waste heat is not usually considered viable. Heat not required for drying raw materials is better conserved using 5-or 6-stage preheaters and high efficiency clinker coolers. Certain exceptions may apply, however, when power tariffs are particularly high or where cogeneration is subsidized or mandatory (Huckauf & Sankol; ZKG; 3/2000, pg 146: ICR; 9/2000, pg 92).

B1.8 Cost of Electric Power Generation

The present cost of generating electricity in the UK (p/kWh) excluding subsidies, including decommissioning, assuming 7.5% discount rate, £30/t coal and £2.30/MMBTU gas is:

	Basic cost	*With back-up*	*With £30/t CO_2 Tax*
Nuclear	2.3	–	–
Gas, (CCGT)	2.2	–	3.4
Coal, pulverised	2.5	–	5.0
Coal, fluidised bed	2.6	–	5.1
Onshore wind	3.7	5.4	–
Offshore wind	5.5	7.2	–

Royal Academy of Engineering; Costs of Generating Electricity (2004)

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B2 Fans and Air Handling

Fans are essential and ubiquitous gas handling equipment in modern cement plants and consume some 30% of total electrical power used for cement manufacture. Rotation of the fan impeller increases gas pressure at the discharge and the resulting pressure drop across the fan causes gas flow. The design of a fan involves the required ranges of gas flow and static pressure, gas density and the optimisation of efficiency.

The fan operation point is at the intersection of the fan curve and the system curve (Fig B2.1) and control of fan performance is achieved by changing the fan curve and/or the system curve. The fan curve can be changed by varying fan speed or by using an adjustable inlet vane. Change in gas density also affects the fan curve but is not a means of control. The system curve can be changed using a damper at the inlet or outlet. Fan performance can be controlled in decreasing order of efficiency by adjustment of rotation speed, inlet vane, inlet box damper and, discharge damper. The operation point should be significantly to the right of the curve apex as the fan will become unstable if operated to the left of the apex.

The most generally useful rule of thumb for gas handling is that system pressure drop is proportional to the square of gas volume.

B2.1 Fan Laws

Shaft kW = Q x δp / 367,000 x η
where Q = gas flow, AM^3/H
δp = fan static pressure, mmH_2O
η = fan efficiency - ca 0.68 for radial tip
- 0.80 for backward curved

Selection of fan motor size should consider in addition to base 'test block' power, reserves for possible change in gas density, dust loading, increase of fan speed or subsequent tipping. Typically, some 33% may be added to base power.

For a fan with fixed inlet and discharge resistance, the effects of changing rpm are:

$Q2/Q1 = (rpm_2 / rpm_1)$ $\qquad p_2/p_1 = (rpm_2/rpm_1)^2$ $\qquad kW_2/kW_1 = (rpm_2/rpm_1)^3$

For a fan with fixed speed and fixed resistance, the static pressure and kW increase in proportion to gas density (ρ):

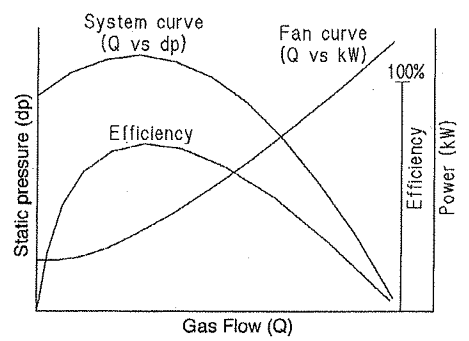
$$p_2/p_1 = (\rho 2/\rho 1) \qquad kW_2/kW_1 = (\rho_2/\rho_1)$$

For a fan with fixed speed and fixed resistance, the effects of changing impeller diameter (D) are:

$$Q_2/Q_2 = (D_2/D_1)^3 \qquad p_2/p_1 = (D2/D1)^2 \qquad kW_2/kW_1 = (D_2/D_1)^5$$

Fan curves are determined by rotor speed, impeller size, and gas density only. Curve shape depends upon impeller blade shape. Two identical fans in series will give a doubling of static pressure for the same flow while two fans in parallel will double the flow at the same static pressure. A typical fan curve is shown in Figure B2.1.

Figure B2.1: Fan Curves

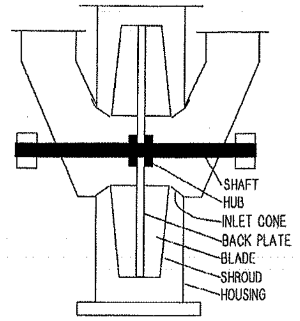


The two impeller geometries most common for large fans in the cement industry are:

- *Radial blade* which show relatively little variation of static pressure with flow over the operating range. Efficiency is 55-60% increasing about 5% if the impeller is closed sided, and decreasing by 5% with dust entrainment.
- *Backward curved blade* which provide the relatively low static pressure for a given wheel diameter. They are, however, typically 75-80 efficient.

Due to rising power cost, high efficiency backward curved fan wheels are largely replacing the traditional paddle wheel design (Saxena; WC; 6/1995, pg 62). Experienced fan suppliers can design a high efficiency impeller to withstand temperature surges and the heavy dust loading pertaining to many cement plant applications. The outline of a double inlet fan is shown in Figure B2.2.

Figure B2.2: Double Inlet Fan



B2.2 Fan Mechanical

Fan shaft diameter is generally designed to give a rigid rotor critical speed at least 1.4 times the maximum operating speed which, in most cases, will provide adequate shaft strength. Anti-friction bearings with a bore size to match such a shaft will usually have load carrying capacity far in excess of that required for fan service and would have too high a peripheral speed for bearing service. Often, therefore, the diameter of the shaft at the bearings is reduced significantly to reduce shaft peripheral speed resulting in less heat generation and easier lubrication. While this may give bearings loaded only to about 2% of their dynamic rating, it is considered that loading of less than 1.5% of the bearing's rating may yield abnormal rolling element motion and excessive heat generation. This problem is of most concern with the spherical roller bearings generally used on heavy duty fans to accommodate the combined thrust and radial loads.

It is also common to encounter fan bearings mounted on tapered adapters; these are easily maintained but can be prone to failure. The inner race run-out measured axially on these bearings can be up to 200μ which may cause excessive vibration, heat generation and abnormal roller motion even though the bearing loading is light. The risk of this problem is most severe with spherical roller bearings. Hot bearings with early failure may sometimes be remedied by switching to split race cylindrical roller bearings though the bearing manufacturer may need to be consulted for help to accommodate the thrust loading.

Larger fans are normally supplied with fluid film bearings (eg Dodge Sleeveoil) which are reliable and have high load carrying capacity. Such bearings, however, do not like low speed operation where the lubricant film between shaft and bearing may be deficient. Higher viscosity lubricant may help for low speed operation when drafting a mill or kiln during maintenance, but bag-house and kiln ID fans should not normally be operated below the bearing suppliers recommended minimum speed.

Lubricated couplings are not usually suitable for kiln system ID fans as they require periodic recharging of lubricant which may not be feasible with long kiln runs. Such couplings should be replaced by elastomeric or disk-pack types which require only occasional inspection.

B2.3 Impeller Build-up

Deposits can be a serious problem for fan impellers driving dust-laden gas, particularly kiln ID fans where hard build-ups can form and spall throwing the fan into catastrophic imbalance. The problem can usually be corrected by consideration of impeller material, blade design, or the addition of guide vanes (Krift; ZKG; 9/1994, pg E235: Gutzwiller & Banyay; CA; 11/2000, pg 39). Backward curved impellers are significantly less prone to build-up than are radial. Failing prevention, BalaDyne offers an automatic balancing system which mounts to the fan shaft and continuously monitors and corrects balance (Rizzo; ICR; 1/1999, pg 61).

Fan vibration which cannot be attributed to build up or loss of balance weights, may be caused by cracks in the welding of armour plate to the impeller. Such cracks may either allow dust to become trapped under the armour plate or, more probably, can change the structural stiffness of the fan wheel resulting in asymmetric deflections under load. This type of problem is extremely difficult to diagnose and should be referred to a fan expert.

B2.4 Gas Properties

	Density g/L	*SH cal/g/°C*	*LH Evap cal/g*
Air	1.293	0.237	
CO_2	1.977	0.199	
O_2	1.429	0.218	
N_2	1.250	0.248	
SO_2	2.927	0.154	
H_2O (100°C)	0.581	0.482	539.5

B2.5 Plant Air Distribution

The production of compressed air is energy-intensive and air is best conserved using a decentralised system of screw compressors. Recommended pressures are:

Air-slides	0.4-0.6kg/cm²	(6-8psi)
Controls	6-8kg/cm²	(80-110psi)
Cleaning	8-12kg/cm²	(110-170psi)

For controls (instrument air) and for bag-house pulse cleaning, the air should be cooled, dried, and de-oiled (Guilman; ZKG; 5/1994, pg E131). Jack-hammers and air-lances should have a separate air supply.

B2.6 Pitots, Orifices and Venturis

Gas velocity, V (M/sec) = $4.43\ K\sqrt{\delta p/\rho}$
where K = coefficient
δp = differential pressure, mm H_2O
ρ = gas density, kg/M³

Gas flow, Q (M³/H) = 3600.A.V
where A = orifice area, M²

K values for conventional pitot tubes are 0.98-1.00. For gas flows with high dust loading, 'S-type' tubes may be used with K values of about 0.8 (Perry; Chemical Engineers' Handbook, 6th Ed, pg 5-10). K values should be provided by the pitot manufacturer. K values for orifice/venturi with constriction diameter d and pipe diameter D:

d/D	*Orifice K*	*Venturi K*
0.7	0.731	
0.6	0.683	
0.5	0.658	1.012
0.4	0.646	0.993
0.3	0.640	0.984

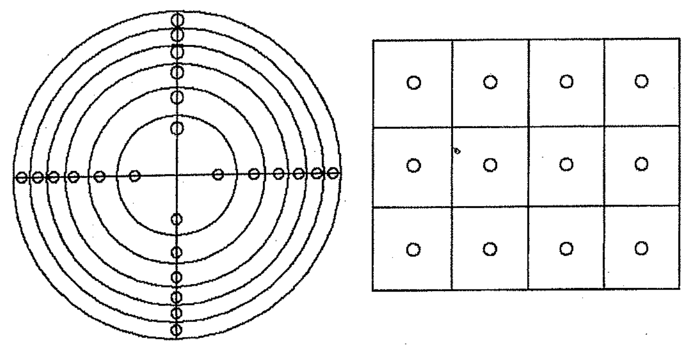
(Note: venturi recovery cone of 15° assumed)

Determination of air flow in ducts using a pitot tube involves measurement at several points across a section of the duct to allow for non-uniform flow and is described in detail for regulatory practice in the United States by EPA 40 CFR Part 60, Appendix A (Note EPA Reference Methods are available at www.epa.gov/ttn/emc/promgate.html). Measurement should be, if possible, at least 8 diameters down-stream and 2 diameters up-stream from any bend or irregularity. Measurements are traversed across two diameters at right angles, with 12 points per traverse for diameters greater than 0.6M, and 8 points for diameters 0.3-0.6M. Closer proximity to disturbances is compensated by additional traverse points. This procedure cannot be used when the flow is cyclonic and no point should be less than 2.5cm from the wall. Note that flow should be calculated from the arithmetic mean of all velocity measurements, not the mean of pitot velocity pressures. For approximate process, as opposed to regulatory, measurements fewer points may be used.

Insertion distances should be:

12 points/traverse		*8 points/traverse*		*6 points/traverse*	
0.021 D	0.644 D	0.032D	0.677 D	0.044 D	0.704 D
0.067	0.750	0.105	0.806	0.147	0.853
0.118	0.823	0.194	0.895	0.296	0.956
0.177	0.882	0.323	0.968	*4 points/traverse*	
0.250	0.933			0.067 D	0.750 D
0.356	0.979			0.250	0.933

Figure B2.3: Pitot Traverse



PITOT TRAVERSE FOR GAS FLOW MEASUREMENT IN CIRCULAR AND RECTANGULAR DUCTS LARGER THAN 0.6M DIAMETER

Rectangular duct of section length L and width W yields an equivalent diameter:

$$D = 2LW/(L+W)$$

and this diameter is used to determine the number of measurement points (12/traverse above 0.6M, 9 below). The section is then divided into that number of equal areas and measurement made at the centre of each area.

B2.7 False Air

Air in-leakage through an aperture of area A (M2) with pressure differential δp (mmH_2O) can be approximately calculated from:

$$\text{Volume, } M^3/h = 8{,}900 \times A \times \sqrt{\delta p}$$

Air in-leakage between two points in the kiln exhaust system can be determined by oxygen measurement. Then, relative to initial volume:

In-leakage, % = $100 (G_2 - G_1) / (20.9 - G_2)$
where G_1 = initial % O_2
G_2 = final % O_2

B2.8 Dust Loading

Bin vent	5-15g/NM^3
Belt transfer point	5-20
Preheater exhaust	50-75
Long kiln exhaust	<30
Short kiln exhaust	<60
Bulk loading	40-60
Ball mill, gravity discharge	20-80
air swept	300-500
Roller mill internal	650
FK pump vent	15-200

(Duda; Vol 1; 3rd Ed; pg 576)

B2.9 Stack Draft

Theoretical draft, mm H_2O = $0.46HP(1/T_a - 1/T_s)$
where H = stack height, M
P = atmospheric pressure, mm Hg
T_a = ambient temp, °K
T_s = average stack temp, °K

Note: there will be some reduction for friction losses.

B2.10 Dew Point of Moist Air at Atmospheric Pressure

		Moisture	
	mg H_2O/L	*% of Dry Vol*	*% of Wet Vol*
10°C	9.35	1.16	1.18
20	17.23	2.14	2.19
30	30.18	3.76	3.90
40	50.69	6.31	6.71
50	82.14	10.22	11.27
60	128.88	16.04	18.61
70	196.13	24.41	30.36
80	290.07	36.10	49.13

H_2O content of saturated air:

$$mg/L = 3.84 \times 10^{-6} \times T^4 + 2.93 \times 10^{-5} \times T^3 + 0.014 \times T^2 + 0.29T + 4.98$$

where T = temperature, °C

B2.11 Spray Cooling of Gas

$\text{Water L/h} = Q.\rho.S\ \delta T/[(100\text{-}T_W)+539]$
where Q = gas flow, M^3/h
ρ = gas density, kg/M^3
S = specific heat of gas
δT = gas cooling, °C
Tw = water temperature

More approximately, for kiln exhaust gas with 5% O_2, water required to reduce gas temperature by δT°C is:

$$\text{L/min water} = \text{t/h clinker} \times \text{kcal/kg}/1000 \times \delta T/66$$

B2.12 Abrasion Resistance

Air entrained particulates are liable to abrade fan impellers and housings, particularly in cooler exhaust or recirculation fans. Apart from hard facing, the impeller vanes can also be protected by guide vanes located near the hub which act to deflect particulates away from the main vanes (Godichon & Revillot; WC; 1/1999, pg 57).

Fan housings together with cyclones and pneumatic conveying lines can be protected by lining either with fused cast basalt or, for high temperatures, materials made by fusing alumina, zirconia and silica (Edwards; ICR; 9/2004, pg 109).

Pneumatic conveying lines are prone to failure at elbows. It has long been standard to embed the elbow in a concrete-filled box. However, as the pipe wears, increased turbulence accelerates and extends the wear. T-bends have been found to simplify construction of pneumatic lines, reduce wear, and do not increase pressure drop but should not be used close to the pump where velocity is low.

■ ■ ■ ■ ■

B3 Conveying

B3.1 Comparative Power Consumption for Lift

	kWh/t/100M
Screw pump	1.20
Airlift	1.10
Dense phase pump	0.59
Bucket elevator	0.41

(Fujimoto; WC; 7/1993 pg 25)

B3.2 Pneumatic Conveying

Fuller Kinyon (FK) Pump. This is a pneumatic conveyor comprising a decreasing pitch screw which compresses a material seal inside a barrel section followed by injection of fluidising and conveying air. Power consumption is 0.8-1.5kWh/tonne per 100M combined for screw drive and compressor; screw speed is 750-1500rpm and air pressure 1-2.5kg/cm^2 (15-35psi). The original 'H' pump has a screw which is unsupported at the discharge end and requires constant material flow. The 'M' pump has a side discharge and a fully supported screw; the 'Z-flap' is a retrofit outboard bearing for 'H' pumps. Capacity to approximately 500t/H is possible but the sizing of pneumatic conveying systems is complex depending principally upon line length and configuration as discussed by Hunlich & Bogdanski (ZKG; 6/1997, pg 307). Design air volume is typically 0.012NM3/kg material. Material flow in pneumatic conveying lines can be measured using radar technology (ZKG; 7/2004, pg 38).

Air Lift (Aeropol). The air lift conveyor comprises a vertical cylindrical pressure vessel, with aeration pad at the bottom and conveying air nozzle passing upwards through the centre of the pad. A rising pipe (conveying line), open at the bottom, is situated just above the conveying air nozzle. Material is fed in near the top of the vessel and controlled by a level indicator; the material column overcomes the back pressure of the conveying line and material is forced into the rising air stream. An alleviator is fitted to the top of the rising pipe to separate carrying air from material. The quantity of material conveyed increases with vessel height, and blower pressure is a surrogate for material flow rate. The conveying height is limited by, and is approximately 10 times, the vessel height. Design air volume is typically 0.048NM^3/kg material.

De-aeration of pneumatically conveyed kiln feed. If not de-aerated, conveying air enters the preheater with the feed material and, if the kiln is ID fan limited, will de-rate kiln production. Approximately, the gas flow at the top of a preheater with 850kcal/kg and 5% excess oxygen is 1.53NM^3/kg clinker.

An FK pump will deliver 0.012NM^3 x 1.6 = 0.019NM^3/kg clinker or 1.25% of preheater exhaust.
An Air Lift will deliver 0.048NM^3 x 1.6 = 0.077NM^3/kg clinker or 5% of preheater exhaust.
A separate dust collector to avoid injecting carrying air to the preheater may, therefore, be justified.

B3.3 Bucket Elevator Power

kW = k.C.H/367
where C = load, tonnes/hour
H = height, M
k = coeft. varying from 1.2 for fed buckets to 2.0 for nodular material with high scooping resistance

Chain elevators can handle material up to 100mm and to 250°C, but should not exceed 60M in height.

Belt elevators are normally limited to 3mm material and 120°C, but can exceed 100M in lift. Elevator speed up to 0.7M/sec results in gravity discharge while above 1.5M/sec the load is ejected centifugally (Labahn).

B3.4 Belt Conveyor Power

Calculation for belt conveyors is complicated by belt inclination and the number of rollers, scrapers, etc. (Labahn). Belt speed is typically 2M/sec; less for fine material, more for coarse material and wide belts. The economics of long belt runs is described by Azan & Clot (ICR; 7/1990, pg 30).

B3.5 Screw Conveyor Power

kW = 2.25(L+9)(C/530)
where L = length, M

B3.6 Air-slide

Air slides are inclined at 4-10o with aeration compartment lengths of up to 5M. Air pressure is typically 400mm H_2O, air volume is $2M^3/M^2$/min and power consumption approximately 0.01kW/t/10M (Labahn). Dust collection is required to maintain upper compartment under negative pressure.

B3.7 Drag Chain Power

kW = (CL/220)+0.8

This is a simplistic equation for a horizontal drag using a theoretical loading; ie the maximum load before slip occurs within the material column.

B3.8 Tube Belt Conveyor

Relative to traditional troughed belt conveyors, tube conveyors shape the flat belt into a closed tube between loading and discharge points. Advantages are intrinsic protection from wind and rain and prevention of fugitive dust, together with ability to climb steeper gradients and turn tighter corners. There are variations of design (www.sicon-roulunds.com & www.beumer.com), some of which involve special belting with cables embedded in reinforced edges. Installation cost and power consumption are comparable to troughed belts but maintenance is less. Tube conveyors up to 850mm diameter have been built carrying 2500t/H of material up to 400mm in size.

B3.9 Sandwich Conveyor

Troughed belt conveyors are limited in elevating loads to angles of less than 15°. This limitation can be overcome by use of a sandwich belt or a high-angle conveyor which allows conveying angles up to 90° (Dos Santos; WC Bulk Materials Handling Review; 1996, pg 34).

B3.10 Modified Belt Conveyor

Magaldi Corporation offer a line of conveyors for original installation or for retrofit to belt systems in which steel plates (for horizontal transport) or steel pans (for inclines) are fastened to steel mesh belt. The belt can run on an existing belt conveyor frame and can usually employ the same drive. Benefits include high temperature capability and much reduced risk of belt failure (www.magaldi.com).

B3.11 Capsule Conveyor

An innovative conveying system is Sumitomo's installation of a 3.2kM pneumatic capsule conveyor for limestone which is claimed to involve lower capital cost, lower maintenance, and lower noise than a belt conveyor (ICR; 10/1994, pg 52).

B3.12 Water Pump Power

kW = QH/6.1e
where H = head, M
Q = flow, M^3/min
e = pump efficiency (typically 0.6-0.9 for positive displacement pumps increasing with pump size)

■ ■ ■ ■ ■

B4 Milling (Ball Mills)

B4.1 Sieve Sizes (ASTM E11)

4#	4.75mm	50#	300µ	200#	75µ
8#	2.36mm	100#	149µ	325#	45µ
16#	1.18mm	120#	125µ	400#	38µ
30#	0.60mm	170#	90µ		

B4.2 Particle Size Analysis

There are now a number of laboratory and on-line laser particle size instruments which provide data of particle size distribution in the range 0.1–200µ of interest for kiln feed and 0.1-100µ for cement (Bumiller & Malcolmson; WC; 6/1998, pg 50). Samples are taken from the separator or classifier product stream. While these instruments directly determine volume proportions, they normally have software to convert the results to a weight basis. Particle size distributions may be plotted:

1. As weight percent of particles within successive increments of particle size (say, 5µ groups) against the mid-point of each particle size group. A logarithmic scale is conventional for particle size (Figure B4.1).

2 As cumulative weight percent smaller (or larger) than a particle size against (logarithmic) particle size (Tromp curve). The *Tromp curve* is a tool for assessing classifier performance and is described in Section 4.5.

3 There are numerous empirical equations which attempt to convert size distribution to a straight line for easier characterization (Perry; Chemical Engineers' Handbook, 6th Ed, pg 8.5). The *Rosin-Rammler function* is widely used in the cement industry:

$$\ln(\ln(100/Wr)) = b\,(\ln(D) - \ln(a))$$

where Wr = cumulative % retained at size D
D = particle size, µ
a & b = constants

The relationship is linear and can be described by the slope, 'b' (often referred to as the RRSB number) and the y-intercept, '-b ln(a)'. For Portland cement, 'b' typically ranges 0.6 for a broad distribution to 1.1 for a narrow range. If D=a, Wr is numerically equal to 36.8% retention. The size distribution can then be described by 'b' and the particle size, Dm, at which 36.8% of particles are retained. A typical Rosin-Rammler graph is shown in Figure B4.2)

Figure B4.1: Particle Size Distribution

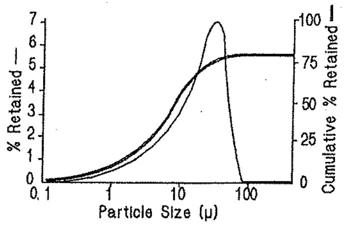
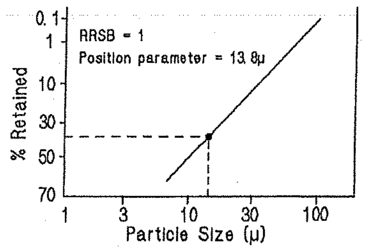


Figure B4.2: Rosin-Rammler Graph



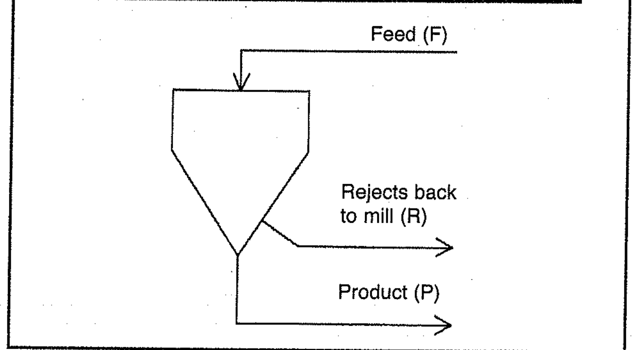
B4.3 Circulating Load

Circulating load refers only to the classifier and is defined as the ratio of the mass of material rejected back to the mill to the mass removed as fine product. Sometimes the ratio of feed to product is used but this should be referred to as Circulating factor. The ratios are defined as mass flows but are usually estimated from sieve residues of the material streams.

All of the following analyses require that samples of materials are representative of the respective streams and taken during stable operation. Each sample should be a composite of at least three grab sub-samples taken of a period of 15-20 minutes. If necessary, the samples should be reduced using a riffle or other valid method of sample division.

Feed to classifier	F t/h	f % passing
Product from classifier	P t/h	p % passing
Rejects back to mill	R t/h	r % passing

Figure B4.3: Separator Feed and Discharge



$$\text{Circulating Load } C_L = R / P = (p - f) / (f - r)$$

$$\text{Circulating Factor } C_F = F / P = (p - r) / (f - r)$$

$$\text{Separator Efficiency } E = p(f - r) / f(p - r)$$

Samples are taken at the separator during stable operation. Each sample is screened on 170# or 200# for raw milling and 325# for cement milling. The specific surface area of cement samples may also be used.

B4.4 Classifier Recovery

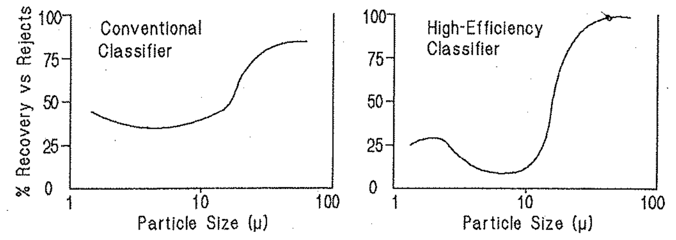
Classifier recovery is defined as the fraction of fines present in the feed which is recovered with product. This is sometimes referred to as the *classifier efficiency* but, being determined from a single mesh size, it is an inadequate representation of overall efficiency. Using the nomenclature from B4.3:

$$\% \text{ Recovery} = 100 \times f(p - r) / p(f - r)$$

B4.5 Tromp Curve

The Tromp is a graphical representation of the probability of a particle in the classifier feed exiting with the rejects. The probability can also be expressed as probability of exiting with the product but this is not conventional in cement industry usage. Using particle size distributions of each of the three streams, a mass balance for incremental size fractions from 1 to 100μ for cement, or 1 to 200μ for raw meal is performed. Typical curves are shown for a mechanical and a high-efficiency separator respectively.

Figure B4.4: Tromp Curves



The selectivity for each increment is plotted against the mid-point of each increment of particle size. This is conveniently plotted on semi-log paper from which the particle diameters corresponding to 25% (S_{25}), 50% (S_{50}) and 75% (S_{75}) probability of rejection are determined.

Apparent bypass (Bp) is the proportion of fines which are rejected. A perfect separator would have 0% bypass and an efficient unit less than 20%.

Classifier cut size (D_{50}) is the particle size corresponding to a 50% probability of rejection.

Sharpness index (D_{25}/D_{75}) represents the sharpness of cut between product and rejects. The perfect classifier has an index of 1 and the smaller the index the poorer the classification. More simply, the steeper the curve, the narrower the particle size distribution of the product.

Many Tromp curves exhibit a characteristic tail at the bottom of the curve. This is often an indication of poor dispersion of the feed in the classifying zone which may be caused either by agglomeration of the feed or by non-uniform distribution of feed in the classifying zone.

B4.6 Mill Critical Speed

Critical speed (C_s) is the rotational speed in rpm of the mill relative to the speed at which centrifugal force just counters gravitation and holds the charge against the shell during rotation. Raw mills usually operate at 72-74% of critical speed and cement mills at 74-76%.

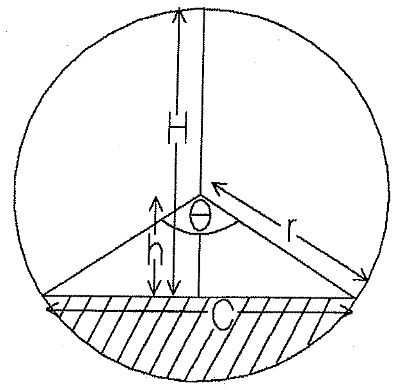
$C_s = 42.3/\sqrt{D}$
where D = effective mill diameter, M

$\% C_s = 100 n / C_s$
where n = actual mill speed, rpm

B4.7 Charge Volume Loading

Loading, $V_L \% = 100[(\pi r^2(\theta/360)) - (h\sqrt{r^2-h^2})] / \pi r^2$
where r = effective mill radius, M
H = free height, M
h = H-r
C = width of charge surface
$= 2\sqrt{r^2-h^2}$
θ = angle subtended at mill axis by charge surface
$\cos ½\theta = h/r$
(Note only the free height, H, and internal diameter, D, need be measured directly.)

Figure B4.5: Mill Charge Volume



For normal charge levels, this gives

h/D	V_L %	h/D	V_L %
0.211	24%	0.143	32%
0.202	25%	0.135	33%
0.194	26%	0.127	34%
0.185	27%	0.119	35%
0.177	28%	0.110	36%
0.168	29%	0.102	37%
0.160	30%	0.094	38%
0.151	31%	0.086	39%

B4.8 Grace Factor and Other Mill Parameters

Instantaneous clinker charge, M_I kg = $M_R R(C_L+100)/6000$
Clinker bulk volume, M_V M^3 = M_I/M_D
Ball charge volume, G_V M^3 = $\pi r^2 L V_L/100$
Ball charge weight, G_W kg = $4550G_V$
Steel:Clinker ratio, S_C = G_W/M_I
Ball charge void space, G_S M^3 = $0.42G_V$
Void fill by clinker, V_F % = $100M_V/G_S$
Grace Factor, G kg/min/M^3 = $M_R(C_L+100)/6000G_S$

where r = Mill inside radius, M
L = Mill inside length, M
V_L = Charge loading, %
M_R = Mill production rate, kg/h
R = Mill retention time, min
M_D = Mill discharge bulk density, kg/M^3
C_L = Mill circulating load, %

An analysis of 79 cement mills gave an optimum Grace factor of 263 and an acceptable range of 180-300. Void fill should be close to 100% (Welch; The Grace Factor – A New Tool for Cement Process Control Engineers; ASTM; 1984).

Retention time is determined using fluorescein (7g / 10t/h of combined mill feed and circulating load - see Mardulier & Wightman; RP; 8/1971). Samples taken at discharge at 30 second intervals are shaken with water, filtered, and the solutions observed under UV light to determine peak concentration. If XRF is available which is equipped for zinc analysis, ZnO can be used as tracer; addition should be such as to give sample concentrations of greater than 20ppm, more if the milled material contains significant Zn.

B4.9 Mill Power

Bond's Third Theory of Comminution allows an approximate estimation of specific power consumption (kWh/Ton) from the work index, Wi of the material being ground (Section B7.2). The value of Wi determined from Bond's Ball Mill Grindability test conforms to the mill motor output power of an 8ft diameter wet grinding overflow ball mill in closed circuit.

$$kWh/T = (10W_i/\sqrt{f_1} - 10W_i/\sqrt{f_2})C_1C_2$$

where f_1 = the μ size which 80% of feed passes
f_2 = the μ size which 80% of product passes
C_1 = 1.3 for dry grinding
C_2 = $(2.438/D)^{0.20}$
D = effective mill ID, M

The Bond work index (W_i) is related to Hardgrove grindability (Hg):

$$Wi,\ kWh/T = 435/Hg^{0.91}$$

(Duda; Cement Data Book 1, 3rd Ed, pg 129)

The approximate power consumption of a mill can also be calculated from:

$$kW = 7.5G_W\sqrt{D}$$

where G_W = ball charge, t
D = effective mill ID, M

Traditionally, ball mills have been driven by single girth gear & pinion drives which are satisfactory up to about 5000kW. Larger mills are driven by girth gears with double pinions, central drives with epicyclic gears, or ring motors (Ranze & Grothaus; ICR; 8/1994; pg 57).

B4.10 Ball Weight & Surface Area

φmm	kg/ball	#balls/t	SA M^2/t
20	0.033	30,600	38.46
25	0.064	15,700	30.77
30	0.110	9,100	25.64
40	0.261	3,830	19.23
50	0.511	1,960	15.38
60	0.882	1,130	12.82
70	1.40	710	10.99
80	2.09	480	9.61

Steel density is assumed 7.8g/cm^2.
Bulk density of a mixed ball charge may be taken as 4550kg/M^3.

B4.11 Maximum Ball Size Required

Ball diameter, mm $= 25.4 \sqrt{(f_1 W_i / 200 C_s) \sqrt{(s/3.28D)}}$

where s = specific gravity of feed

C_s = % of critical speed

Interestingly, this suggests that the second compartment of a cement mill with f_1 of 3mm requires a maximum ball size of about 25mm.

B4.12 Measurement of Wear

The constant task of wear monitoring, ball sorting, and charge make-up has been greatly reduced by use of high-chrome alloys with charge loss of approximately 20-50g/t.

Ball wear can be measured by weighing an entire charge before and after grinding a known tonnage of material. With new balls a sample of given size can be weighed before and after grinding a given tonnage and, assuming uniform wear per unit of ball surface, the total charge wear can be estimated. More simply, and quite adequate, is to keep a record of material ground and of charge added periodically to recover a given level of mill power.

Mill liners should be measured periodically to determine wear rate and, more importantly, to anticipate replacement. Thickness can usually be measured relative to the shell which can be probed between liner plates.

Mill diaphragms are prone to excessive localized wear which must be monitored to anticipate replacement. Note that diaphragms normally have about 15% open area (slots plus centre screen) and that the slots are tapered outward on the discharge side to prevent plugging.

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B5 Kilns and Burning

B5.1 Cement Compounds and Ratios

C_3S = 4.071Ca - 7.600Si - 6.718Al - 1.430Fe - 2.852S

C_2S = 2.867Si - 0.754C_3S

C_3A = 2.650Al - 1.692Fe

C_4AF = 3.043Fe

LSF = (Ca + 0.75Mg*)/(2.85Si + 1.18Al + 0.65Fe)

S/R = Si/(Al + Fe)

A/F = Al/Fe

Alkali equivalent = Na + 0.658K

Liquid, % (1450°C) = 3.00Al + 2.25Fe + Mg + K + Na

Note Ca, Si, etc represent oxides: CaO, SiO_2, etc

* if MgO exceeds 2%, use constant 1.5 for this term

B5.2 Coating Tendency

Coating tendency = $C_3A + C_4AF + 0.2C_2S + 2Fe$

Note index <28 indicates light coating

>30 indicates heavy unstable coating, rings & snowmen

(RefrAmerica Clinker Reference Data Sheet)

B5.3 Burnability Factor

Numerous factors have been proposed over the years from the simplistic Kuehl Index (C_3S/C_4AF+C_3A) to complex formulations involving laboratory burning tests. A useful compromise is Miller's empirical formula for burning at 1400°C:

$\%\text{Free-lime}_{1400} = 0.31(\%\text{LSF}-100) + 2.18(S/R-1.8) + 0.73Q + 0.33C + 0.34A$

where Q = +45μ residue after acid wash (20% HCl) identified by microscopy as quartz

C = +125μ residue which is soluble in acid (ie coarse limestone)

A = +45μ residue after acid wash identified by microscopy as non-quartz acid insoluble

Note Q, C & A are expressed as % of total raw mix sample (DuToit, WC, 3/1997, pg 77)

B5.4 Required Burning Temperature

Burning Temperature, °C = 1300 + 4.51C_3S - 3.74C_3A - 12.64C_4AF

B5.5 Theoretical Heat of Formation of Clinker

Heat in:	Heat raw materials 20-450°C	170 kcal/kg
	Dehydration of clay at 450°	40
	Heat materials 450-900°	195
	Dissociation of $CaCO_3$ at 900°	475
	Heat materials 900-1400°	125
	Net heat of melting	25
		1030
Heat out:	Exothermic crystallisation of dehydrated clay	10
	Exothermic formation of cement compounds	100
	Cooling clinker 1400-20°	360
	Cooling CO_2 900-20°	120
	Cooling and condensing steam 450-20°	20
		610

Net theoretical heat required to form 1kg clinker, Q = 420kcal

Heat of formation for specific mix designs can be calculated:

$$Q = 4.11Al + 6.48Mg + 7.646Ca - 5.116Si - 0.59Fe$$

B5.6 Kiln Heat Balance (referred to 20°C)

	kcal/kg	*Typical (4 stage SP)*
Heat in:		
Kiln feed (__kg x __°C x 0.22)	___	15
Fuel (__kg x __kcal/kg NCV)	___	835
Total air (__NM³ x 1.293 x 0.237 x __°C)	___	0
Total kcal/kg clinker in	___	**850**
Heat out:		
Theoretical heat of clinker burning	___	420
Moisture evaporation (__kgKF x __% x 5.40)	___	5
Kiln exhaust (__NM³ x D x SH x °C)	___	210
Entrained dust (__kg x 0.22 x °C)	___	3
Bypass gas (__NM³ x D x SH x °C)	___	-
Bypass dust (__kg x SH x °C)	___	-
Cooler exhaust (NM³ x 1.293 x 0.237 x __°C)	___	115
Clinker (1kg x 0.20 x __°C)	___	14
Radiation, preheater	___	35
kiln	___	45
cooler	___	3
Total kcal/kg clinker out	___	**850**

Note:

1 All weights and gas volumes are relative to 1kg clinker.
2 Temperatures (°C) are relative to 20°C (or ambient).
3 Specific heats for kiln feed, kiln dust, and clinker can be calculated from standard values for components and temperatures, confirmed empirically, or typical values may be used (kcal/kg/°C):

	20°	*20-500°*	*20-700°*	*20-900°*	*20-1100°*	*20-1400°*
kiln feed	0.21	0.249	0.259	0.265		
clinker	0.19	0.220	0.229	0.236	0.242	0.262

(Lee; Chemistry of Cement & Concrete; 3rd Ed, pg 125)

4 Fuel may be on 'as received' or dried basis, but weight and kcal/kg should be consistent, and 'net' heat content should be used.
5 Clinker cooling air + primary air + false air + net combustion gas + calcination CO_2 + moisture = kiln exhaust + cooler exhaust.
6 Evaporation is kiln feed (kg) x % moisture x latent heat; if coal is direct fired, its moisture must also be considered.
7 Exhaust gas density (D) and specific heat (SH) can be calculated from its components (Section B2.4) or, approximately, from the typical gas composition given in Section 5.13. Specific heat varies with temperature:

	20°	*20-200°*	*20-400°*	*20-600°*	*20-800°*	*20-1000°*
air	0.237	0.242	0.245	0.247	0.250	0.252
CO_2	0.209	0.218	0.233	0.244	0.252	0.259
N_2	0.250	0.251	0.253	0.256	0.259	0.261

8 Bypass gas and dust analyses are required for acceptable estimates of gas density and specific heats.
9 Radiation losses can be determined by integrating measurements of surface temperature and emissivity for sub-areas.
Alternatively, approximations such as those employed above are frequently used.

B5.7 Kiln Gas Velocities

Upper limits:	
Through cooler grate	5 AM/sec
Hood	6
Under cooler bull-nose	15
Burning zone (1450°C)	9.5
Feed end transition (1000°C)	13
Riser	24
Preheater gas ducts	18
Lower limit:	
Tertiary duct	25
Pulverised coal conveying	20

Note that entrained dust increases with (gas velocity)³

B5.8 Kiln Specific Heat Loading (SHL)

Heat loading, kcal/M^2/h = F x GCV / π(D/2)^2
where F = kiln fuel rate, kg/h
GCV = gross calorific value of fuel, kcal/kg
D = effective kiln diameter, M

Note that heat loading calculation requires gross calorific value and excludes fuel burned in riser or calciner.

Standard heat loading values are:

LD (long dry kiln)	1.2kcal/M^2/h
SP (preheater kiln)	2.0
AT (air-through precalciner)	3.0
PC (precalciner)	4.0
NPC (new precalciner)	5.0

B5.9 Kiln Retention Time (t)

The US Bureau of Mines formula for retention time (mins) gives:

t = 11.2L/rDs
where L = kiln length, M
r = kiln speed, rpm
D = effective diameter, M
s = slope,°

Note that an accurate estimate of kiln retention time depends upon the physical properties of the material. The slope of cyclone preheater kilns is usually 3-3.5%.

Slope° vs %

°	1.0	1.2	1.4	1.6	1.8	2.0	2.2	2.4	2.6	2.8
%	1.75	2.09	2.44	2.79	3.14	3.49	3.84	4.19	4.54	4.89

B5.10 Kiln Volume Loading

Kiln volume loading is intended as a relative rather than an accurate value. It depends upon retention time and bulk density both of which are rather arbitrary.

% loading = 1.67 x FR x W x t / L x A
where FR = Feed ratio (kg material entering kiln to give 1kg clinker)
W = clinker production, t/H
A = effective cross sectional area, M^2

	SHL	FR	L/D	rpm	% Vol Loading
Typically, LD kiln	1.2	1.55	35	1.6	16.8
SP	2.0	1.45	16	2.0	9.2
AT	3.0	1.22	15	3.0	7.3
PC	4.0	1.11	15	3.5	7.6
NPC	5.0	1.05	12	3.5	7.2

Alternatively, the specific kiln capacity is expressed in relation to the effective kiln volume as kg clinker produced/M^3/hour with typical ranges:

SP without riser firing	80- 90kg/M^3/h
ILC or SP with riser firing	100-130kg/M^3/h
SLC	120-150kg/M^3/h

Very large SLC kilns are now being designed with over 180kg/M^3/h.

B5.11 Kiln Drive Power

$$kW = \pi L(D/2)^2/4.7$$

Kiln power during normal operation should be about half the installed power.

B5.12 Cooler Efficiency

E, % = [C_1 - (V + C_2 + R)] / C_1
where C_1 = heat content of clinker from kiln
C_2 = heat content of clinker out
V = heat content of cooler vent air
R = cooler radiation

Typical efficiencies are 60-70% for a conventional grate cooler, 70-75% for an air-beam, and 75-78% for a cross-bar system.

B5.13 Kiln Exhaust Gas (Coal)

Assume a typical bituminous coal with ultimate analysis (air dried basis):

C	80.0 %
H	5.0
S	1.0
O	5.0
N	0
Ash	8.0
Net kcal/kg	7400

with indirect firing, and specific fuel consumption of 850kcal/kg clinker. Then coal consumption is 0.127kg/kg clinker and combustion gases produced are:

C	0.80 x 127	→	101.6g x 22.4/12	=	190L CO_2	= 271g O_2
H	0.05 x 127	→	6.35g x 22.4/2	=	71L H_2O	= 51
S	0.01 x 127	→	1.27g x 22.4/32	=	0.9L SO_2	= 1
O	0.05 x 127	→	6.35g			
						323g O_2

Then added O_2 required for combustion = 323g – 6.35g = 317g = 222L or 0.222NM³
Then equivalent N_2 from air = 222L x 79/21 = 835L or 0.835NM³

CO_2 from calcination of raw meal to yield 1kg clinker (assuming kiln feed LoI of 35%):
[(1000/0.65) - 1000] = 538g = 274L or 0.274NM³
Then total CO_2 in exhaust gas = 274L + 190L = 0.464NM³

H_2O from kiln feed (assuming 1.65 kiln feed:clinker factor and 0.5% H_2O):
1kg x 1.65 x 0.005 = 8.25g = 10L or 0.01NM³
Then total H_2O in exhaust gas = 10L + 71L = 81L or 0.081NM³

Then exhaust gas with no excess air, is:

CO_2	0.463NM³	=	33.5%
H_2O	0.081	=	5.9
SO_2	0.001	=	0.1 (1000ppm)
N_2	0.835	=	60.5
	1.38NM³		

Estimation of **net exhaust gas volume, NM³/kg clinker @ 0% excess O_2**
= (kcal/kg x 0.00129) + 0.284
Estimation of **gross exhaust gas volume, NM³/kg clinker with n% O_2**
= Net NM³/kg x (1 + (n / (21-n))

Physical data of exhaust gas with various levels of (dry) excess air:

	Density g/L	*SH cal/g/°C*	*Dew point °C*
0% O_2	1.487	0.216	38
2% O_2	1.469	0.218	36
5% O_2	1.441	0.221	33
10% O_2	1.395	0.226	26

Typical gas volumes for combustion products (with no excess air) are related to specific fuel consumption and vary with fuel:

NM³/kg clinker - Coal = kcal/kg x 0.00129
- Oil = kcal/kg x 0.00119
- Natural gas = kcal/kg x 0.00132

In each case add 0.284NM³/kg clinker for calcination and moisture

B5.14 Circulation of Volatile Components

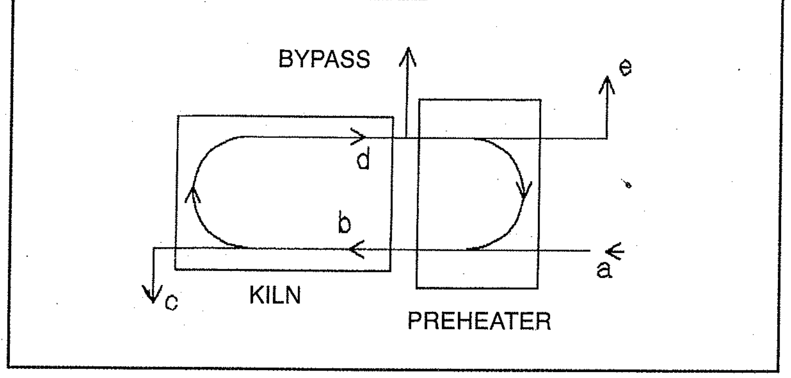
K, Na, S and Cl are all subject to partial evaporation at kiln burning zone temperatures:

	KCl	**NaCl**	**$CaCl_2$**	**K_2SO_4**	**Na_2SO_4**	**$CaSO_4$**	**K_2CO_3**	**Na_2CO_3**	**$CaSO_4$**
M Pt, °C	776	801	772	1069	884	1450	891	851	896^D
B Pt, °C	1500^S	1413		1689					

Eutectics have even lower melting points.

Volatilisation in burning zone and condensation in preheater may be represented as shown (Norbom; Proc IEEE; Miami 1973). The external cycles through dust collector are not here considered; if dust is not wasted, then virtually all 'e' is returned to the kiln.

Figure B5.1: Kiln Volatile Cycle



Conventional factors defining the circulation and typical values for SP kilns without bypass are:

		K_2O	Na_2O	SO_3	Cl
Primary volatility,	E1 = b/d	0.67	0.53	0.55	0.99
Secondary volatility,	E2 = b/d	0.88	0.92	0.80	0.97
Valve,	V = e/d	0.10	0.40	0.1/0.4	0.05
Circulation factor,	K = b/a	3.20	1.40	3.3/1.4	45.0
Residue,	R = c/a (=1 without bypass)				

(Ritzmann; ZKG; 8/1971, pg 338)

Note primary volatility applies to raw feed and secondary volatility to recirculated material. SO_3 volatility increases with stoichiometric excess over alkalis

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B6 Fuels

The data for solid, liquid and gaseous fuels is taken from various sources including Jenkins, 'The Changing Market of Fuel for Cement Manufacture'.
A summary of world fuel reserves is given on the BP website: *www.bp.com*

B6.1 Typical Data for Solid Fuels (as received)

	Lignite	Sub-bitum.	Bituminous	Anthracite	Pet-coke	Oil Shale	Dry Sludge	Refuse	Mixed Plastics
C, %	66.9	68.0	74.4	87.0	87.2	41.2	33.4	39.8	60.0
H, %	4.7	4.8	4.9	1.7	3.7	5.0	4.9	5.4	7.2
N, %	1.30	1.40	1.40	0.60	1.50	0.10	3.10	1.00	0
S, %	0.80	0.40	1.00	0.90	5.50	0.90	0.50	0.20	0
O, %	19.04	16.1	9.3	2.0	1.7	5.7	21.1	32.9	22.8
Ash, %	7.3	9.3	9.0	7.8	0.4	47.1	37.0	20.7	10.0
H_2O, %	33.7	18.3	10.9	2.8	2.0	2.0	2.0	28.2	2.0
Volatiles, %	29.2	31.0	32.1	1.2	10.8	51.4			
Fixed C, %	29.8	41.4	48.0	88.2	86.8	1.5			
GCV kcal/kg	6335	6555	7365	7555	8365	4840	3615	3870	6475
NCV kcal/kg	6085	6295	7100	7460	8165	4575	3355		6090
Air required*	8.5	8.8	9.9	10.5	11.4	6.2	8.4	4.6	5.0
Hardgrove	>100	65	55	45	35				

Note: *Coal is ranked according to the degree of conversion of original plant material into carbon (lignite < sub-bituminous < bituminous < anthracite).*

Coal heat contents calculated on dry basis. Oil shale and refuse heat contents calculated on 'as received' basis.

Green delayed petroleum coke as produced has a moisture content of 10-20% which drains to about 2-3%.

**Air required is theoretical mass ratio.*

Proximate analysis: H_2O + Volatiles + Fixed C* + Ash = 100% (* normally obtained by difference)

Ultimate analysis: C + H + N + S + O* + Ash = 100%

Gross Heat, kcal/kg = 80.8C + 22.45S + 339.4H - 35.9O
Net Heat, kcal/kg = 80.8C + 22.45S + 287(H-O/8) - 6W where W is H_2O content, %
Gross - Net = 51.5H* where H* is total % H_2 including H_2O

Gross Heat is the theoretical heat of combustion which assumes that water produced is condensed. In practice, water is usually released as vapour so that only Net Heat is recovered.

B6.2 Typical Data for Liquid Fuels

	Kerosene	Gas Oil	Heavy Fuel Oil	Vacuum Residue	Orimulsion	Blended Waste
C, %	85.8	86.1	85.4	86.8	61.4	70.1-83.3
H, %	14.1	13.2	11.4	9.9	6.5	7.1-8.4
S, %	0.1	0.7	2.8	1.0 – 5.5	2.9	2.8-3.3
O, %				0.5		0.0-15.0
N, %			0.40	0.46	0.40	
Cl, %						4.0-6.7
Ash, %			0.04	0.20	0.22	
H_2O, %			0.30		29.8	
V, Ni, etc, ppm		5 – 70	70 – 500	460		
SG (water = 1)	0.78	0.83	0.96	1.00-1.05	1.01	0.80-1.00
Viscosity, cSt	1.48 @38°C	3.3 @38°C	862 @38°C	1000-3200 @100°C	600 @50°C	15-50 @38°C
GCV, kcal/kg	11,100	10,250	10,250	10,200	7,260	
NCV, kcal/kg	10,390	9,670	9,670	9,610	6,740	5-10,000
Solids						100%-200μ
Air required	14.7	13.8	13.8	13.5	9.3	9.3-12.6

API Gravity = (141.5/SG) - 131.5 1 BBL oil = 42 gals(US)

B6.3 Typical Data for Gaseous Fuels

	North Sea NG	W Aust NG	LPG	Blast Furn Gas	Coke Oven Gas	Digester Bio-gas	Landfill Gas
O_2					0.5		
CO_2	0.2			17.5	4.0	38.0	14-17
CO				24.0	4.0		
H_2				2.5	30.0		
N_2	1.5	1.1		56.0	4.0		0-47
CH_4	94.4	98.6			52.0	57.0	25-60
C_nH_m	3.9	0.3	100		5.5		
H_2S					5.0		
GCV kcal/M^3	9050	8930	22,430	760	8070	5730	2400-5730
NCV kcal/M^3	8270	8050	20,640	740	7260	5180	2150-5160
SG (air = 1)	0.6	0.6	1.5	1.0	0.6	1.0	0.9-1.0
Air required*	9.8	9.4	23.8	0.6	8.2	5.8	1.9-5.7
Flame speed	34.0	33.7	39.7	10.5	59.8	23.9	22.3-24.6

**Air required is theoretical Volume ratio*
Flame speed, M/sec

$$\text{Gross kcal}/M^3 = 90.3CH_4 + 159.2C_2H_6 + 229C_3H_8 + 301.9C_4H_{10} + 373.8C_5H_{12} + 57.6H_2S$$

Most natural gas is free from sulphur but, if it occurs, is usually removed before delivery.

Liquified gases yield the following:

1 litre liquid	Methane	→	606 litres gas
	Propane	→	139 litres gas
	Butane	→	119 litres gas

B 6.4 Tyres

Tyres may be used whole, shredded or pyrolysed. Typical data for used tyres is:

Weight:			**Composition:**	
Passenger tyre,	used	9-10kg	Rubber	84%
	new	13kg	Steel belt & bead	10%
			Fabric	6%

Proximate (dry basis, less steel):		**Ultimate** (dry basis, less steel):	
Ash	5%	Ash	5.0%
Volatiles	67%	C	84.2%
Fixed carbon	28%	H	7.1%
		N	0.25%
		S	1.25%
		O (by difference)	2.2%

Heat:			
	GCV, whole tyres	7,500kcal/kg	(13,500BTU/Lb)
	TDF (shreds)	8,900kcal/kg	(16,000BTU/Lb)

Ash (less steel):

ZnO	35.0%	Fe_2O_3	2.4%
SiO_2	35.0%	K_2O	1.0%
Al_2O_3	13.0%	Na_2O	0.9%
SO_3	6.7%	Cl	1.2%
CaO	3.0%	Mg,P,Cr,Cd,F,Pb	Trace

Pyrolysis is the heating of carbon or hydrocarbon containing material in an oxygen deficient atmosphere:

$$C + CHn + [O] \rightarrow CO + CH_4 + C_2H_6 + \ldots\ldots + \text{char}$$
$$C + [O] = CO + 26.5\text{kcal/mole}$$
$$C + O_2 = CO_2 + 93.7\text{kcal/mole}$$

The total heat content of the tyres (or other solid waste) can be utilised and supplied to the kiln continuously as gas (mainly CO) or char (mainly fixed C). The resulting combustion is more controlled than is the intermittent heat release from injecting whole tyres direct to the kiln.

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B7 Materials

B7.1 Bulk Densities of Materials for Silo Storage (kg/M^3)

Aggregate, fine	1500	Fly-ash	550
coarse	1600	Gypsum	860
Bauxite	720	Iron ore	2700
Cement	1500	Limestone	1400
Clinker	1360	Raw meal	1250
Coal, bituminous, bulk	850	Sand	1600
Coal, anthracite	920	Sand, foundry	1600
Coal, pulverized	450	Shale/clay	1000
Coke, petroleum	880		

Angle of repose:		
	Clinker & dry rock	30-35°
	Cement	20°
	Gypsum	30°
	Bauxite	30°

B7.2 Specific Gravities and Grindabilities

	SG	**Bond Wi**	**Hardgrove Hg**
Bauxite	2.38	9.5	
Cement raw materials	2.67	10.6	43-93
Clay	2.23	7.1	97
Clinker	3.09	13.5	30-50
Coal, - anthracite			30-53
- bituminous	1.63	11.4	44-85
Feldspar	2.80	43	
Gypsum rock	2.69	8.2	
Iron ore	4.50		38
Limestone	2.68	10.2	54-78
Sandstone	2.68	11.5	
Silica sand	2.65		24-55
Blast furnace slag	2.39	12.2	

(Perry's Chemical Engineers' Handbook; 6th Ed., pg 8-13)

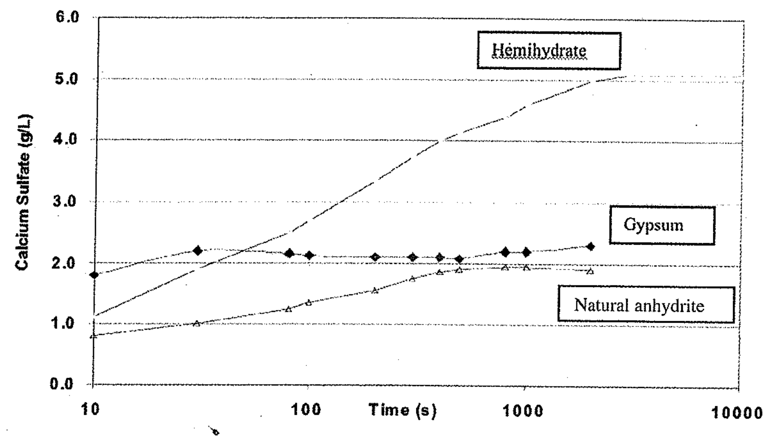
B7.3 Solubilities of Sulphates

Gypsum	$CaSO_4.2H_2O$	24g/L
Hemihydrate	$CaSO_4.\frac{1}{2}H_2O$	ca 6
Soluble anhydrite	$CaSO_4.0.001\text{-}\frac{1}{2}H_2O$	ca 6
Natural anhydrite	$CaSO_4$	21
Syngenite	$K_2Ca(SO_4).2H_2O$	25
Potassium sulphate	K_2SO_4	136

(Hanson et al; Gypsum & Anhydrite in Portland Cement; 3rd Ed, pg 54.)

B7.4 Rates of Dissolution of Different Forms of Calcium Sulphate at 20°C

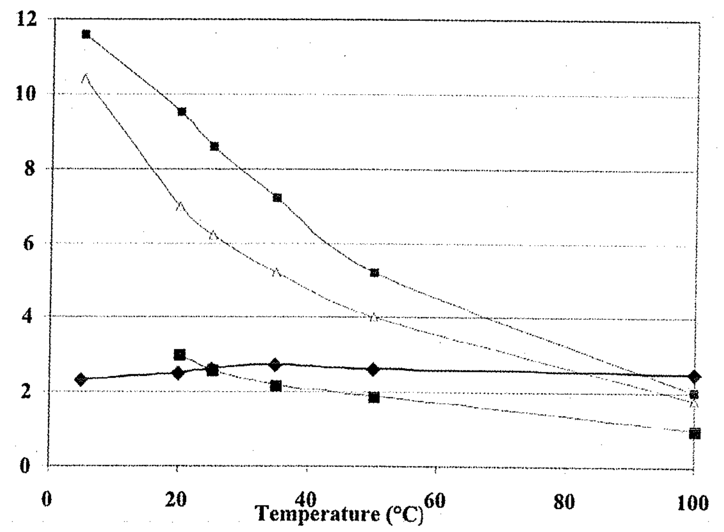
Figure B7.4: Rates of Dissolution of Different forms of Calcium Sulphate at 20°C



(Hanson et al; Gypsum & Anhydrite in Portland Cement; 3rd Ed, pg 37)

B7.5 Influence of Temperature on the Solubilities of Various Sulphate Forms

Figure B7.5: Influence of Temperature on the Solubilities of Various Sulphate Forms



(Hanson et al; Gypsum & Anhydrite in Portland Cement; 3rd Ed, pg 37)

B7.6 Chemical Formula Weights

Al_2O_3	102	BaO	153	CO	28	CO_2	44
CaO	56	Cr_2O_3	152	Fe_2O_3	160	FeO	72
Fe_2O_3	232	H_2O	18	K_2O	94	MgO	40
Mn_2O_3	158	Na_2O	62	O_2	32	P_2O_5	144
SO_2	64	SO3	80	SiO_2	64	SrO	104
TiO_2	80	ZnO	81				

CA_2	260	CA	158	$C_{12}A_7$	1386	FeS_2	120
C_2AS	278	C_4AS	406	C_3S	228	C_2S	172
C_3A	270	C_4AF	486	C_2F	272	CaF_2	78

$CaCO_3$	100	$CaSO_4$	136	$CaSO_4.2H_2O$	172	$CaSO_4$. ½H_2O	145
$MgCO_3$	84	K_2SO_4	174	KCl	75	$CaCO_3.MgCO_3$	184
$Ca(OH)_2$	74	Na_2SO_4	142	$2C_2S.CaCO_3$	444	$2C_2S.CaSO_4$	480

$3K_2SO_4.Na_2O$	664	$2CaSO_4.K_2SO_4$	446
$C_3A.3CaSO_4.32H_2O$	1254	$C_3A.CaSO_4.13H_2O$	640

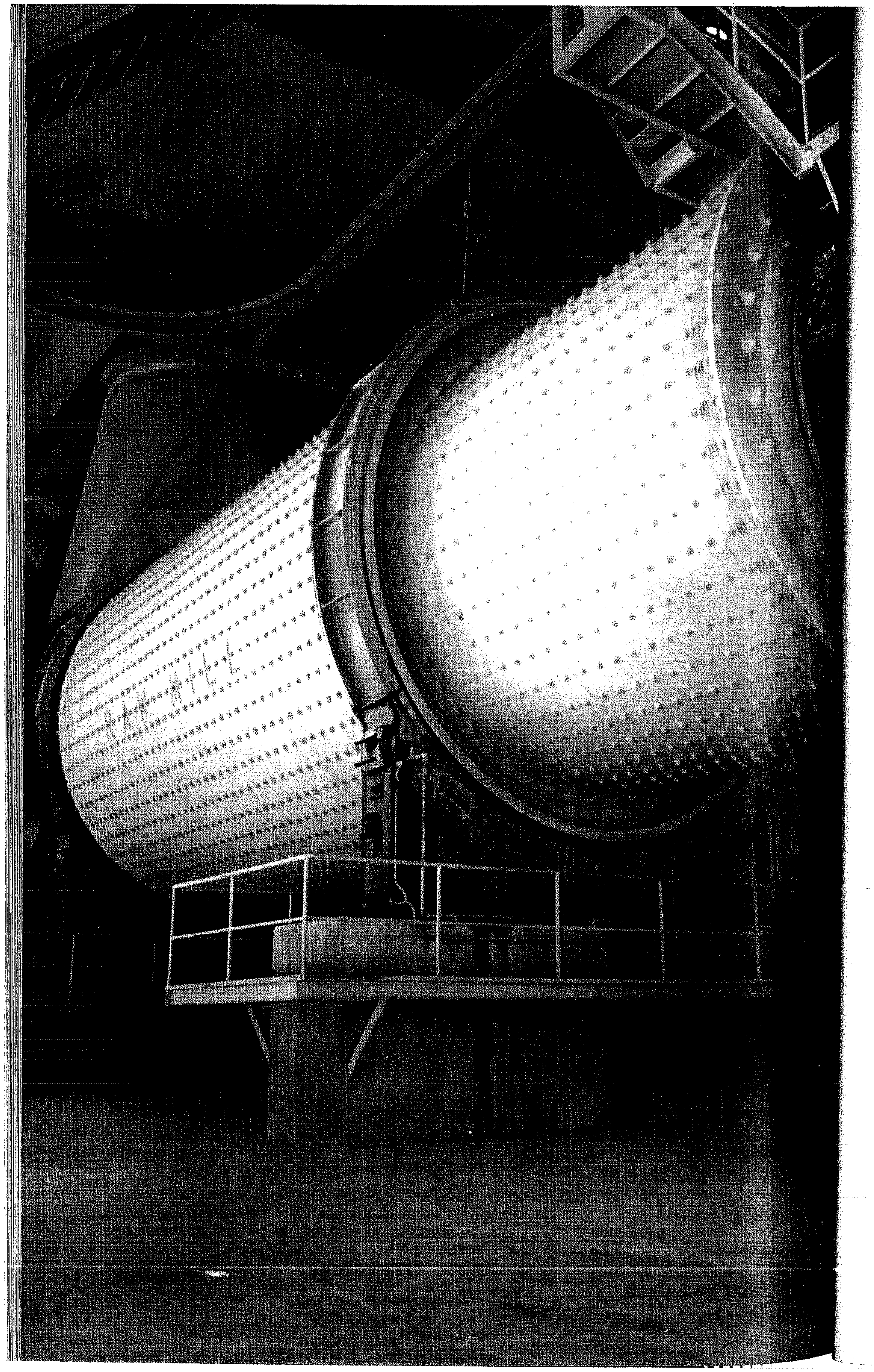
SO_3/K_2O molar ratio = 0.85 x mass ratio
SO_3 in $CaSO_4.2H_2O$ = 46.5%
SO_3 in $CaSO_4$ = 58.8%
H_2O in $CaSO_4$. ½H_2O = 6.2%

SO_3/Na_2O molar ratio = 1.29 x mass ratio
SO_3 in CaSO4. ½H_2O = 55.2%
H_2O in $CaSO_4.2H_2O$ = 20.9%

B7.7 Coefficients of Linear Expansion (μ/M/°C)

Aluminium	22.7
Brick	6.4
Concrete	14.6
Copper	16.5
Steel	11.6

■ ■ ■ ■ ■



B8 Miscellaneous Data

B8.1 Geometrical and Trigonometrical Formulae:

π = 3.14159 26535 89793 23846 26433 83279 50288

Cone, surface area = $\pi r(l+r)$ where l = slant height
volume = $1/3\pi r^2 h$

Segment of circle Area = $\pi r^2(\phi/360) - Ch/2$
where r = radius of circle
ϕ = angle subtended by chord at centre
C = length of chord
h = perpendicular distance from chord to centre

Trigonometrical Tables

	sin	cos	tan		sin	cos	tan		sin	cos	tan
0°	0.0000	1.0000	0.0000	**30°**	0.5000	0.8660	0.5774	**60°**	0.8660	0.5000	1.7321
1	0.0175	0.9998	0.0175	**31**	0.5150	0.8572	0.6009	**61**	0.8746	0.4848	1.8040
2	0.0349	0.9994	0.0349	**32**	0.5299	0.8480	0.6249	**62**	0.8829	0.4695	1.8807
3	0.0523	0.9986	0.0524	**33**	0.5446	0.8387	0.6494	**63**	0.8910	0.4540	1.9626
4	0.0698	0.9976	0.0699	**34**	0.5592	0.8290	0.6745	**64**	0.8988	0.4384	2.0503
5	0.0872	0.9962	0.0875	**35**	0.5736	0.8192	0.7002	**65**	0.9063	0.4226	2.1445
6	0.1045	0.9945	0.1051	**36**	0.5878	0.8090	0.7265	**66**	0.9135	0.4067	2.2460
7	0.1219	0.9925	0.1228	**37**	0.6018	0.7986	0.7536	**67**	0.9205	0.3907	2.3559
8	0.1392	0.9903	0.1405	**38**	0.6157	0.7880	0.7813	**68**	0.9272	0.3746	2.4751
9	0.1564	0.9877	0.1584	**39**	0.6293	0.7771	0.8019	**69**	0.9336	0.3584	2.6051
10	0.1736	0.9848	0.1763	**40**	0.6428	0.7660	0.8391	**70**	0.9397	0.3420	2.7475
11	0.1908	0.9816	0.1944	**41**	0.6561	0.7547	0.8693	**71**	0.9455	0.3256	2.9042
12	0.2079	0.9781	0.2126	**42**	0.6691	0.7431	0.9004	**72**	0.9511	0.3090	3.0777
13	0.2250	0.9744	0.2309	**43**	0.6820	0.7314	0.9325	**73**	0.9563	0.2924	3.2709
14	0.2419	0.9703	0.2493	**44**	0.6947	0.7193	0.9657	**74**	0.9613	0.2756	3.4874
15	0.2588	0.9659	0.2679	**45**	0.7071	0.7071	1.0000	**75**	0.9659	0.2588	3.7321
16	0.2756	0.9613	0.2867	**46**	0.7193	0.6947	1.0355	**76**	0.9703	0.2419	4.0108
17	0.2924	0.9563	0.3057	**47**	0.7314	0.6820	1.0724	**77**	0.9744	0.2250	4.3315
18	0.3090	0.9511	0.3249	**48**	0.7431	0.6691	1.1106	**78**	0.9781	0.2079	4.7046
19	0.3256	0.9455	0.3443	**49**	0.7547	0.6561	1.1504	**79**	0.9816	0.1908	5.1446
20	0.3420	0.9397	0.3640	**50**	0.7660	0.6428	1.1918	**80**	0.9848	0.1736	5.6713
21	0.3584	0.9336	0.3839	**51**	0.7771	0.6293	1.2349	**81**	0.9877	0.1564	6.3138
22	0.3746	0.9272	0.4040	**52**	0.7880	0.6157	1.2799	**82**	0.9903	0.1392	7.1154
23	0.3907	0.9205	0.4245	**53**	0.7986	0.6018	1.3270	**83**	0.9925	0.1219	8.1443
24	0.4067	0.9135	0.4452	**54**	0.8090	0.5878	1.3764	**84**	0.9945	0.1045	9.514
25	0.4226	0.9063	0.4663	**55**	0.8192	0.5736	1.4281	**85**	0.9962	0.0872	11.43
26	0.4384	0.8988	0.4877	**56**	0.8290	0.5592	1.4826	**86**	0.9976	0.0698	14.30
27	0.4540	0.8910	0.5095	**57**	0.8387	0.5446	1.5399	**87**	0.9986	0.0523	19.08
28	0.4695	0.8829	0.5317	**58**	0.8480	0.5299	1.6003	**88**	0.9994	0.0349	28.64
29	0.4848	0.8746	0.5543	**59**	0.8572	0.5150	1.6643	**89**	0.9998	0.0175	57.29

SIN = Opp/Hyp
COS = Adj/Hyp
TAN = Opp/Adj
COSEC = Hyp/Opp
SEC = Hyp/Adj
COT = Adj/Opp

B8.2 Greek Alphabet

Α α	alpha	(a)	Η η	eta	(h)	Ν ν	nu	(n)	Τ τ	tau	(t)
Β β	beta	(b)	Θ θ	theta	(q)	Ξ ξ	xi	(x)	Υ υ	upsilon	(u)
Γ γ	gamma	(g)	Ι ι	iota	(i)	Ο ο	omicron	(o)	Φ φ	phi	(f)
Δ δ	delta	(d)	Κ κ	kappa	(k)	Π π	pi	(p)	Χ χ	chi	(c)
Ε ε	epsilon	(e)	Λ λ	lambda	(l)	Ρ ρ	rho	(r)	Ψ ψ	psi	(y)
Ζ ζ	zeta	(z)	Μ μ	mu	(m)	Σ σ	sigma	(s)	Ω ω	omega	(w)

B8.3 Atmospheric Pressure and Density vs Altitude (0°C)

Alt M	mmHg	kg/M^3	Alt M	mmHg	kg/M^3	Alt M	mmHg	kg/M^3
0M	760	1.293	700	699	1.189	1600	624	1.062
100	751	1.278	800	691	1.176	1800	610	1.038
200	742	1.262	900	682	1.160	2000	596	1.014
300	733	1.247	1000	673	1.145	2200	582	0.988
400	724	1.232	1100	664	1.130	2400	569	0.968
500	716	1.218	1200	655	1.114	2600	556	0.946
600	707	1.203	1400	639	1.092	2800	543	0.924

$$p = 760\,(1 - 0.0065H / 288)^{5.255}$$

where p = atmospheric pressure, mmHg
H = altitude, M

B8.4 pH and Normality

Pure water dissociates slightly into hydrogen ions and hydroxyl ions:

$$H_2O = H^+ + OH^-$$

the concentration of each being 10^{-7} moles per litre.

pH is a measure of acidity of a solution defined by:

$$pH = -\log[H^+]$$

so that pure water has a pH of 7 which is taken to represent neutrality. If acid is added to water, the H^+ concentration increases and pH decreases. If alkali is added, the concentration of OH^- increases causing H+ to decrease and pH to increase. Thus pH below 7 indicates acidity and above 7 indicates alkalinity.

Normality is a measure of solution concentration and is equal to the number of gram-equivalents of electrolyte per litre of solution. If an acid were completely dissociated, a normal solution would have a pH of 0.

A similar solution of alkali would have pH of 14. The degree of dissociation increases with the strength of acid (or alkali) and with dilution but is never 100%.

B8.5 Laboratory Reagents (aqueous solutions)

Hydrochloric acid	HCl	37 %	12 N	1.18 SG
Nitric acid	HNO_3	70	16	1.42
Sulphuric acid	H_2SO_4	97	36	1.84
Phosphoric acid	H_3PO_4	85	41	1.69
Acetic acid	$CH_3.COOH$	99.5	17	1.05
Ammonia	NH_3	27	14	0.90

B8.6 Sea Water Composition

Cl	2.05%
SO_4	0.23%
Na	1.10%
Mg	0.14%
K	0.05%
Ca	0.03%
Total	3.60%

B8.7 Abundance of Elements in Earth's Crust

Element	At No	Wt %	Element	At No	ppm
O	8	46.60	Sr	38	300
Si	14	27.72	Ba	56	250
Al	13	8.13	Zr	40	220
Fe	26	5.00	Cr	24	200
Ca	20	3.63	V	23	150
Na	11	2.83	Zn	30	132
K	19	2.59	Ni	28	80
Mg	12	2.09	Cu	29	70
Ti	22	0.44	Sn	50	40
H	1	0.14	Pb	82	16
P	15	0.12	Be	4	6
Mn	25	0.10	As	33	5
S	16	520ppm	Tl	81	0.6
C	6	320	Hg	80	0.5
Cl	17	314	Cd	48	0.15
Rb	37	310	Ag	47	0.10
F	9	300	Se	34	0.09

(Handbook of Chemistry & Physics; Chemical Rubber Co.)

B8.8 Hardness of Materials

a) Moh's scale:

Talc	1	Feldspar	6
Rock-salt	2	Quartz	7
Calcite	3	Topaz	8
Fluorite	4	Corundum	9
Apatite	5	Diamond	10

b) Metal hardness is usually determined by:

Brinell – A known load is applied to a surface using a spring-loaded hammer and a standard impression point; the depth of the impression is then measured. Range is approximately 60 for aluminium to 6-700 for hard alloy steels.

Rockwell – Two loads, one large and one small, are applied by compression to the surface; the difference in impression depths is measured. Values are approximately 1/10th those for Brinell. Both methods employ standard impression points which may be steel balls for soft materials but should be diamond for hard surfaces.

B8.9 Earthquake Scales

The Richter Scale is a measure of the energy released by an earthquake:

1-3 Generally not felt
3-4 Often felt but no damage
5 Widely felt with slight damage near epicentre
6 Damage to poorly built structures within 10kM of centre
7 Major earthquake causing serious damage up to 100kM from centre
8 Extreme earthquake with great destruction over several hundred kM (eg San Francisco 1906)
9 Rare disaster causing major damage over 1000kM (eg Chile 1960)

Modified Mercalli Scale measures the effects of an earthquake which vary with location and geology:

I Not felt
II Occasionally felt
III Usually perceived indoors with movement of hanging objects
IV Doors rattle, parked cars rock
V Significant movement, people are awakened, doors swing, objects fall
VI Difficulty walking, furniture moves, slight building damage
VII Difficulty standing, cars shake, considerable damage to poorly built structures
VIII Houses shift on foundations, tree branches break, tall structures can twist and fall
IX Well built structures significantly damaged, underground pipes break
X Most buildings and foundations seriously damaged, landslides, large ground cracks
XI Most buildings collapse, bridges destroyed, railway tracks badly bent
XII Total destruction, large rock movements

B8.10 World Cement Production (2001)

	Major Producers	Annual Production	% of World Production	Per capita Consumption
1	China[(1)]	620 Mt	36.2%	429kg
2	India	93.6	5.5	89
3	USA	88.8	5.2	408
4	Japan	79.5	4.6	540
5	South Korea	52.1	3.0	951
6	Spain	40.9	2.4	1051
7	Italy	39.8	2.3	682
8	Brazil	38.9	2.3	217
9	Russia	35.3	2.1	193
10	Thailand	34.5	2.0	293
11	Indonesia	33.6	2.0	113
12	Germany	31.8	1.9	370
13	Mexico	30.4	1.8	278
14	Turkey	30.0	1.8	380
15	Iran	26.0	1.5	350
16	Egypt	25.7	1.5	229
17	Saudi Arabia	21.7	1.3	818
	(EU/25)	224.2	13.1	485
	(Lafarge)	87.6	5.1	
	(Holderbank)	84.3	4.9	
	(Cemex)	61.2	3.6	
	(Heidelberg)	59.8	3.5	
World total		**1,714 Mt[(2)]**		

Notes: (1) Chinese clinker production was 367Mt and 70% of production in China still comes from shaft kilns.
(2) Total world capacity estimated at 1,790Mt from 1447 plants

(ICR, Global Cement Report, Fifth Edition)

B8.11 New Cement Capacity

An analysis by OneStone Consulting (Harder; WC; 10/2001, pg 89) indicated:

New capacity ordered:		
	1996	48.6 million tonnes (Mt)
	1997	25.5Mt
	1998	20.4Mt
	1999	24.5Mt
	2000	23.1Mt

B8.12 Ship Capacities

	Deadweight t	Length, M	Beam, M	Depth, M
River barge	2,000	50	14	4
River barge	5,000	85	20	6
Minibulk	<15,000	155	21	9
Small handy	15-25,000	170	23	10
Handy	25-40,000	200	29	11
Handymax	40-55,000	220	32	12
Panamax	55-80,000	240	38	13.5

(Lithgart; ICR; 2/2002, pg 64. Dimensions are given as approximations only)

Road bulk truck (U.S.)		23 tonnes (25T)
Rail bulk car,	small (U.S.)	68 tonnes (75T)
	large (U.S.)	91 tonnes (100T)

B8.13 Patents

A patent is an agreement under which legal protection for a limited time is given to the inventor in return for publishing the invention. Patent conditions vary with jurisdiction but the GATT approved term is for 20 years from the date of filing. It is a right to exclude others from making use of the invention rather than a specific right of use. Rights under the patent can be assigned, sold, or licensed for an agreed royalty payment.

To be patentable, an invention must be:

- a process, a machine, a manufactured product, a composition of matter, or a new use for any of these
- useful as opposed to aesthetic, and cannot be merely a theoretical phenomenon
- novel relative to prior published or widely practiced art
- not obvious to a person having ordinary skill in the art

A patent may cover more than one country but must be separately filed for each. Most countries are signatories of the 1883 Paris Convention and this protects an invention in all countries from the date it is filed in one. The Patent Cooperation Treaty (PCT) includes a mechanism that allows a preliminary international search for prior art before applying to the specific countries where protection is sought. There are presently about 120 states in contract with the PCT.

The first decision following inspiration is whether to patent or whether the idea can be sufficiently well protected so that patenting, and thus public revelation, is unnecessary (the Coca Cola method). Alternatively, printed publication will suffice to protect one's right to practice the invention and will prevent or invalidate subsequent patenting by someone else. The most efficient channel for such publication is as a Research Disclosure (see www.researchdisclosure.com).

It is also recommended to retain notarised records of significant discoveries and developments as these may help to establish precedence in the event of patent prosecution by another inventor.

The cost of obtaining a patent in the United States using outside attorneys may be typically $25-50,000. Patenting in additional countries involves translation, application and maintenance fees, and may require active "working". Before going to this expense, it is important to consider what is to be done with the patent. The obvious purposes are either to earn royalties from licensing or to prevent use by competitors. The former requires a strategy to market the invention and both require the resources to police infringement.

Much useful information for the uninitiated is available from the website of Oppedahl & Larsen LLP (www.patents.com).

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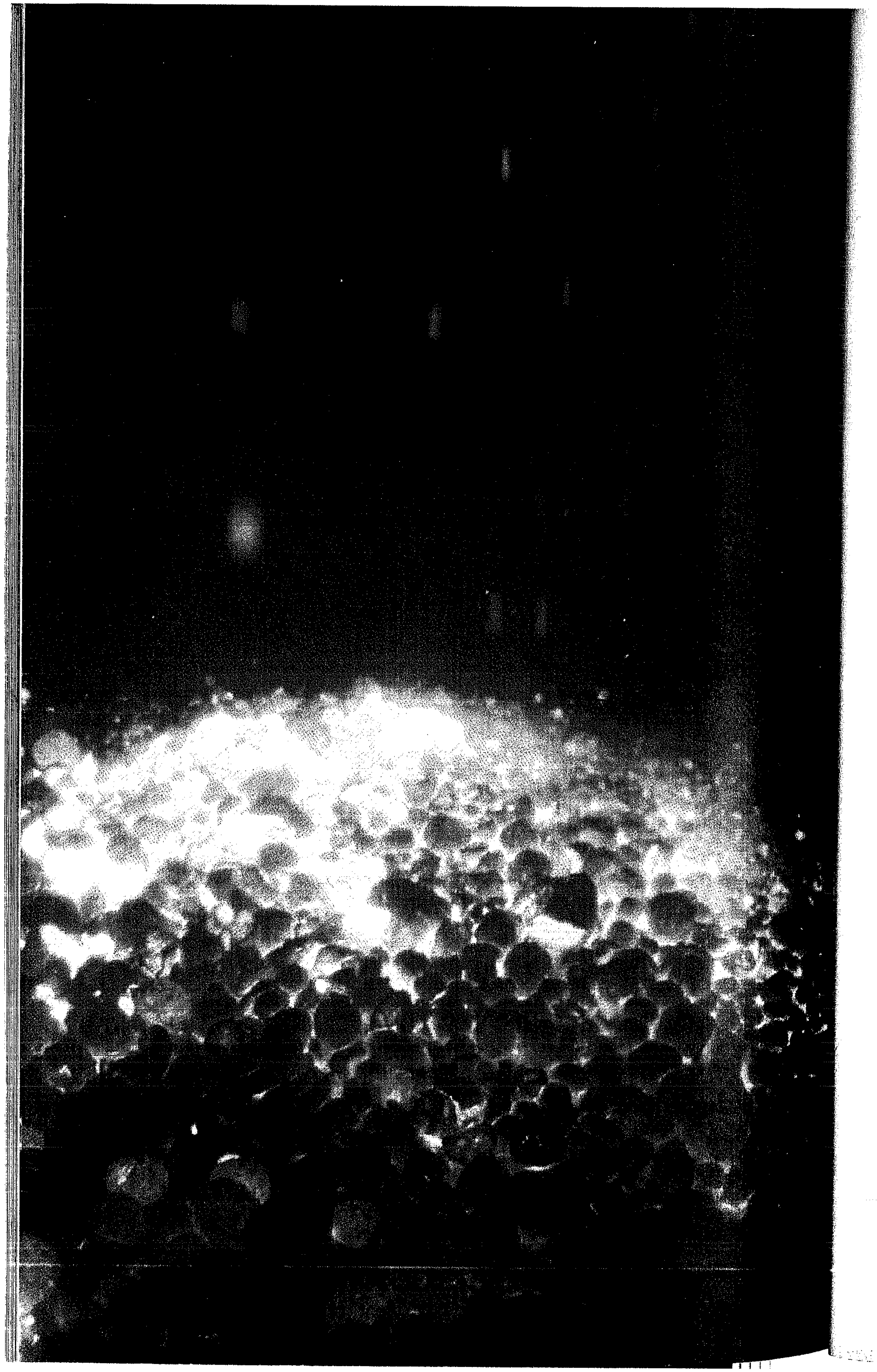

B9 Conversion Tables

B9.1 Length

1 mm = 40 thou
25.4 mm = 1 inch
1 M = 3.28 ft
1.61 kM = 1 mile

B9.2 Area

1 M^2 = 10.76 ft^2
1 acre = 4840 yd^2
1 HA = 2.47 acres
259 HA = 1 square mile

B9.3 Volume

1 M^3 = 35.31 ft^3
28.32 L = 1 ft^3
3.785 L = 1 gal (US)
1 ft^3 = 7.48 gal (US)
4.546 L = 1 gal (Imp)

B9.4 Density

1 g/cm^3 = 62.4 Lb/ft^3
1 t/M^3 = 262Lb/ft^3
1 gal(US) water = 8.345Lb
1 gmole gas = 22.4 L
1 Lbmole gas = 359 scf
1 cm^3 mercury = 13.6 g

B9.5 Pressure

1 Atmos = 1.034 kg/cm^2
= 14.7 psi
= 1.013 bar
= 101.8 kPA
= 76.0 cm Hg
= 1033 cm H_2O (WG)
= 10.13 N/cm^2
1 kPA = 101.4mm WG
1 MPa = 144.4psi
1" WG = 2.49 mbar
= 250 PA
1 N/mm^2 = 145 psi

B9.6 Energy

1 kWh = 860 kcal
= 3413 BTU
= 1.34 HP
= 3.60 MJ
1 kcal = 4.187 kJ
1 kcal = 3.698 BTU
1 kcal/kg = 1.80 BTU/LB
= 3600 BTU/T

B9.7 Force

1 kNewton = 224.8 Lb

B9.8 Weight

1 g = 15.43 grains
31.1 g = 1 oz(Troy)
28.35 g = 1 oz(Avdp)
1 kg = 2.205Lb
1 tonne = 1.102 short Tons

B9.9 Miscellaneous

2.29 mg/NM^3 = 0.001 grain/scf
1 BBL(oil) = 42.0 gal(US)
1 BBL(cement) = 376 Lb
1 M^3/h = 0.589ft^3/min
1 M^3/min = 35.32ft^3/min

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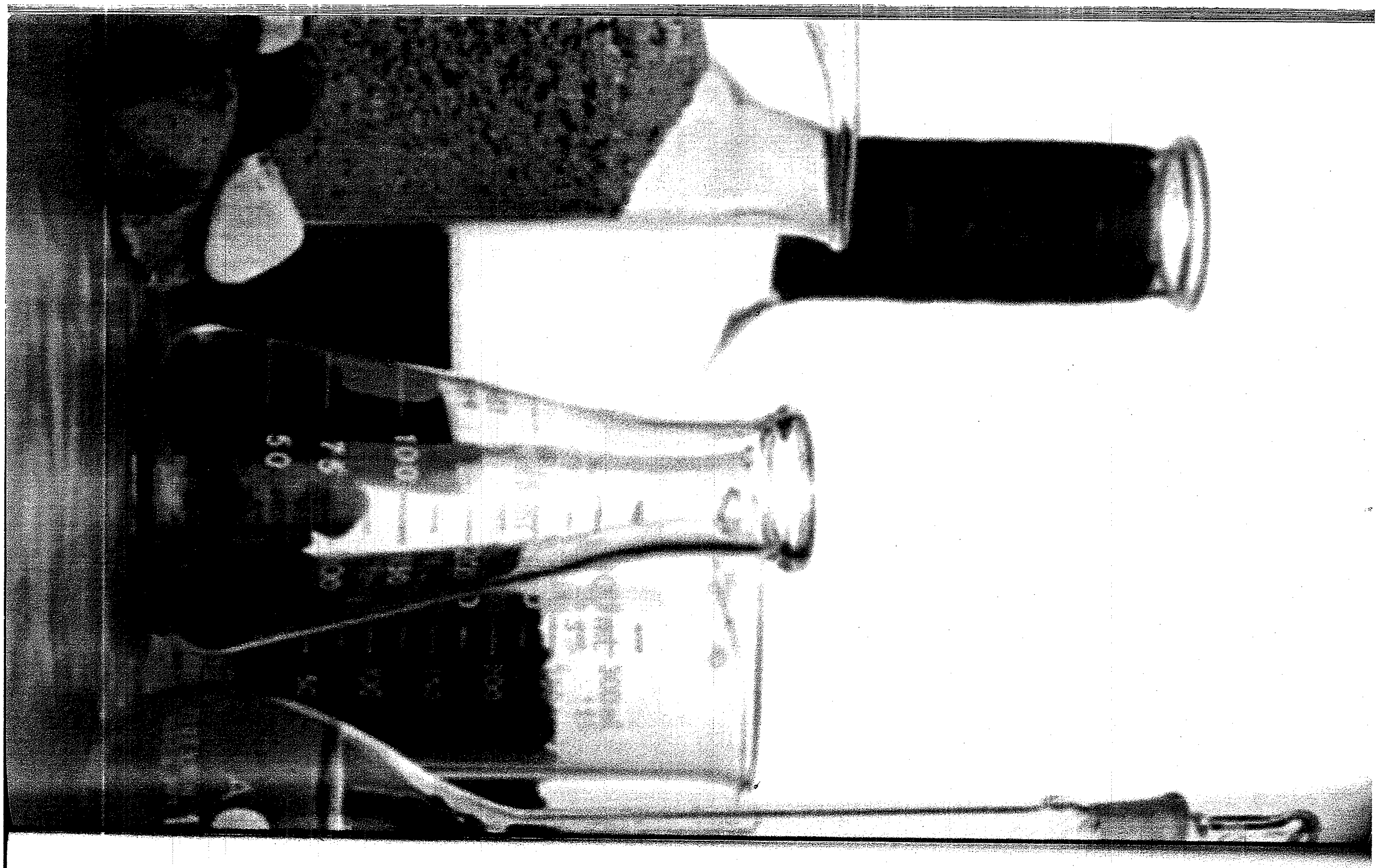
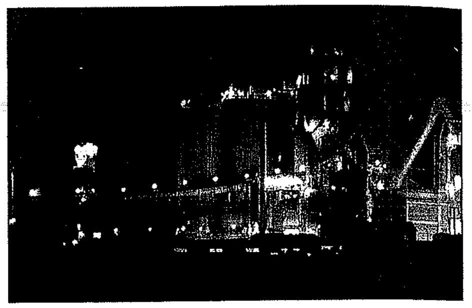
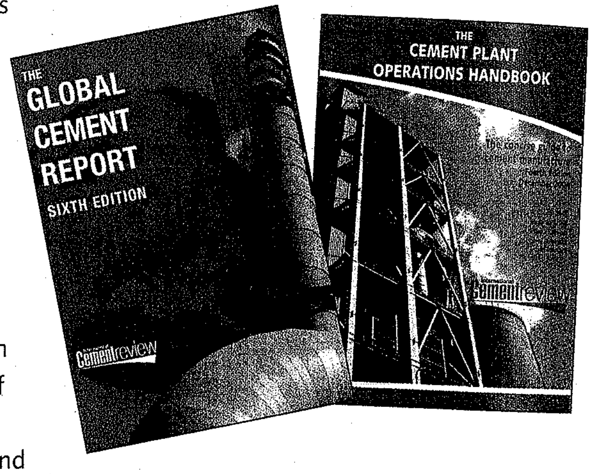
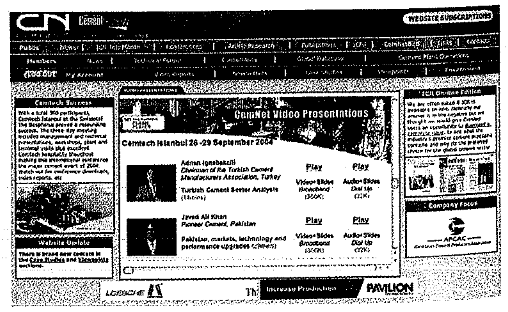


B10 Periodic Table

1 H 1.008																	2 He 4.003
3 Li 6.941	4 Be 9.012											5 B 10.81	6 C 12.011	7 N 14.007	8 O 15.999	9 F 18.998	10 Ne 20.179
11 Na 22.990	12 Mg 24.305											13 Al 26.982	14 Si 28.086	15 P 30.974	16 S 32.06	17 Cl 35.453	18 Ar 39.948
19 K 39.098	20 Ca 40.08	21 Sc 44.956	22 Ti 47.90	23 V 50.941	24 Cr 51.996	25 Mn 54.938	26 Fe 55.847	27 Co 58.933	28 Ni 58.71	29 Cu 63.546	30 Zn 65.38	31 Ga 69.72	32 Ge 72.59	33 As 74.922	34 Se 78.96	35 Br 79.904	36 Kr 83.80
37 Rb 85.468	38 Sr 87.62	39 Y 88.906	40 Zr 91.22	41 Nb 92.906	42 Mo 95.94	43 Tc 98.906	44 Ru 101.07	45 Rh 102.905	46 Pd 106.4	47 Ag 107.868	48 Cd 112.40	49 In 114.82	50 Sn 118.69	51 Sb 121.75	52 Te 127.60	53 I 126.904	54 Xe 131.30
55 Cs 132.905	56 Ba 137.34	57 *La 138.905	72 Hf 178.49	73 Ta 180.948	74 W 183.85	75 Re 186.2	76 Os 190.2	77 Ir 192.22	78 Pt 195.09	79 Au 196.966	80 Hg 200.59	81 Tl 204.37	82 Pb 207.2	83 Bi 208.980	84 Po (210)	85 At (210)	86 Rn (222)
87 Fr (223)	88 Ra 226.025	89 ⁑Ac (227)	104	105													
*Lanthanides				58 Ce 140.12	59 Pr 140.908	60 Nd 144.2	61 Pm (147)	62 Sm 150.4	63 Eu 151.96	64 Gd 157.25	65 Tb 158.925	66 Dy 162.50	67 Ho 164.930	68 Er 167.26	69 Tm 168.934	70 Yb 173.04	71 Lu 174.97
⁑Actinides				90 Th 232.038	91 Pa 231.036	92 U 238.029	93 Np 237.048	94 Pu (244)	95 Am (243)	96 Cm (247)	97 Bk (247)	98 Cf (251)	99 Es (254)	100 Fm (257)	101 Md (258)	102 No (255)	103 Lr (255)




THE GLOBAL CEMENT REPORT
SIXTH EDITION
THE CEMENT PLANT OPERATIONS HANDBOOK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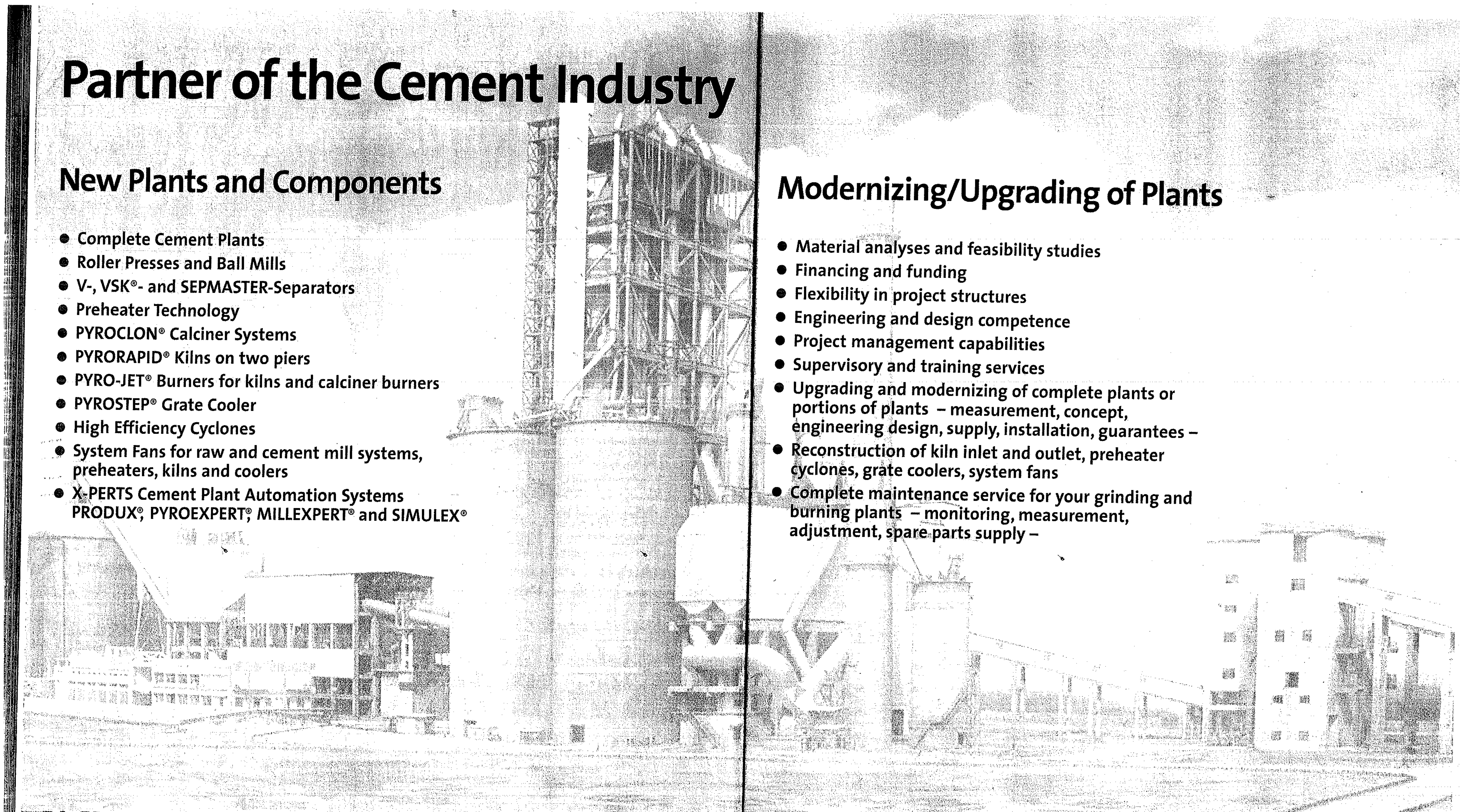

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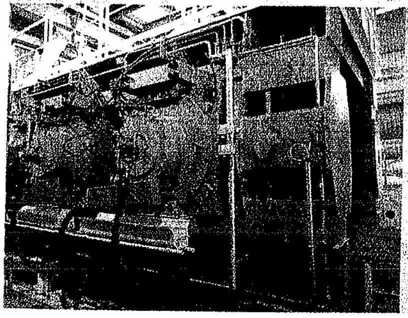
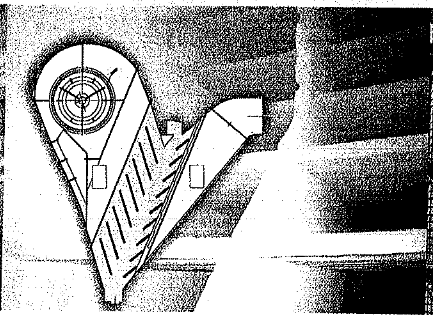
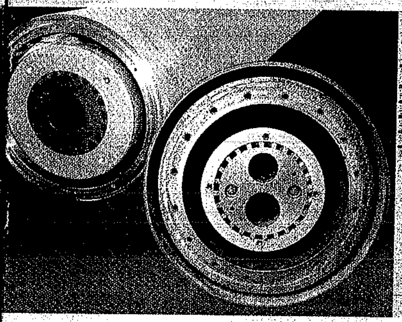


Roller Press
for grinding of raw materials, clinker, slags, additives in pregrinding, semi-finish- and finish-grinding mode.

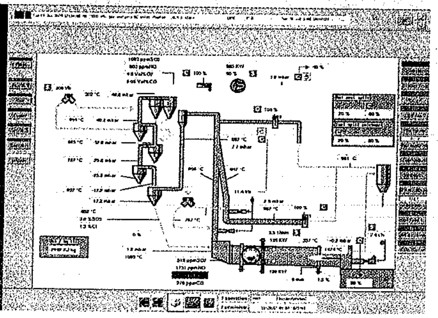


VSK®-Separator
combining the advantages of static separators and high efficiency, dynamic separators in one compact unit.

PYROCLON® Technology
the efficient modular solution for low NO_X emissions / **LowNO$_X$-Calciner** and complete burnout of primary and secondary fuels / **PYROTOP® Compact** mixing chamber.



PYRO-JET® Kiln Burner
Main design features – the Sandwich-Concept – with internal swirl body, one or two annular channels for secondary or other dust-like fuels and the surrounding jet nozzle ring.



X-PERTS Automation Systems
intelligent solutions for powerful cement plant automation with focus on cost-effectiveness, product quality and environmental protection.

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Section B

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